

The Operational History of Mines in the Northwest Territories, Canada



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An Historical Research Project by Ryan Silke

- 2009 -

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(www.nwtgeoscience.ca)*



Front cover photos (clockwise from top-left)

- 1) Ruth Mine, 2000. (Ryan Silke photo)**
- 2) Discovery Mine gold pour, 1968. (Discovery Mines Ltd. photo)**
- 3) Bullmoose Lake pilot mill, 1986. (Terra Mines Ltd. photo)**
- 4) Indore uranium miners, 1952. (Henry Busse - N.W.T. Archives photo)**
- 5) Crestaurum Mine office, 1946. (N.W.T. Mining Heritage Society collection)**
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Introduction

The preparation of this report has been an intensive task. It actually started early in 1999 when a curious and slightly younger version of myself took upon the monstrous and ambitious task of learning all that there was to know about the mining history of his hometown, Yellowknife, N.W.T.

I was never happy with my first draft published in November 1999. I created a webpage ‘database’ with entries for each of the old mines, and the published version was nothing but a print off of the internet resource, bound in cirlocks, and dished out at 20 bucks a copy. I didn’t complain at the time. I made a bit of money by selling the rights to the government and I was able to penetrate the local mining industry ranks and amateur historians ‘club’. But the report was quickly researched and prematurely released. It contained many mistakes and a lack of information that was not physically possible for me to complete at the time. My interest did not peter out in 1999, it only grew...so much that within a year I was already drafting a revision. Post-secondary education got in the way of completing this project, but I am thankful that I waited this long. I would never have been able to compile such a mass of research had I not let life take me on the directions that it has in the last eight years.

It is designed as a one-stop source for information regarding the N.W.T.’s current and past mining operations. Production statistics, descriptions of mine workings, geology, maps, details of operation and equipment used, employees, site layouts, and general history is all laid out for the interested reader. Those who find the remains of these old sites all ask the same question: what happened here? History buffs in mining districts throughout Canada and the United States would have loved to collect this data before time blew away the answers. Luckily the N.W.T.’s mining history is relatively recent, and much of that information can still be tracked down today.

This is an historical documentation. As the old mines wind down and their remains vanish from the landscape, I hope that this report will provide a lasting imprint on what was an important stage in the development of the N.W.T.

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Brief History of Mining in the N.W.T.

including Nunavut Territory

The Canadian north was settled because of its mineral resources. When the Canadian Dominion Government first purchased the arctic from the British Crown in 1870, the Northwest Territories was seen as a vast stretch of cold and inhospitable land and was practically ignored by the authorities at the time. First minerals were reported in the 16th century by British explorer Martin Frobisher, but his gold ores turned out to be types of pyrite, or better known as 'fool's gold'.

The original inhabitants of the north, the natives, once mined copper ores along the Arctic Coast for use as tools, implements of war, and objects of trade. This copper was the target of fur trader and explorer Samuel Hearne, who in the 18th century sought out the legendary mountains of the mineral along the Coppermine River. Those copper deposits proved vastly exaggerated and to this day no mine has entered production in this area despite periodic copper strikes and aerial staking rushes.

The Klondike Gold Rush of 1897-1898 was the turning point in mineral exploration in the north. While no major gold strikes were made in the boundaries of the modern day N.W.T., some of the Yukon stampeders did make their journey through the subarctic, including the prospectors who first reported gold at Yellowknife River and lead and zinc ores at Pine Point on the shores of Great Slave Lake. In 1900, exploration of the N.W.T. was underway by students of the Geological Survey of Canada and the first mapping was done at Great Slave Lake and Great Bear Lakes. As Canada entered the 20th century, it was abundantly clear that the Barrenlands North of 60 held immense potential for mineral resources.

Numerous gold strikes were reported in the Nahanni River-Mackenzie Mountain ranges and also at Great Slave Lake (Wilson Island) during this time, but the first real mineral rush into the N.W.T. occurred as a result of Gilbert LaBine's radium and silver discoveries at Great Bear Lake in 1930. This new and exciting development not only brought forth the first productive mines, but it also spurred production of oil resources at Norman Wells, discovered in 1921.

By this time aerial exploration was playing a big part in exploring the vast stretches of the N.W.T. Parties fanned outwards from Great Bear Lake during the 1930s and high-grade gold was discovered up the Yellowknife River in 1933 by Herb Dixon and Johnny Baker. An increase in the price of gold that year caused great interest, and by 1936 several hundred claims had been staked around Yellowknife Bay on Great Slave Lake and a settlement was being built. The first gold mine to go into production was the Con Mine in 1938, followed by Negus Mine in 1939, and several smaller mines before conditions attending to World War II mothballed most projects and companies.

The silver and radium mines at Great Bear Lake also ceased operating during the war. However, the demand for wartime minerals such as uranium brought new life back into many of these mines. Tungsten and some high-tech metals such as tantalum and columbium were also the target of exploration during this time period in the Yellowknife area, but no substantial mines were discovered.

Post-war N.W.T. was a very active one. Federal powers that oversaw administration of the northern regions understood that development of the north hinged upon the harvesting of its resources. They invested huge capital into the modernization of communities and infrastructure. Fantastic gold discoveries at Yellowknife brought forth the producing reigns of Giant Mine and Discovery Mine, and the community of Yellowknife grew into a modern pioneer town. Its first municipal town council was elected in 1953.

Another major event during this time period was the establishment of an immense lead and zinc orebody at Pine Point. To mine those ores economically, a railroad to Great Slave Lake was necessary. The Canadian government recognized the value of this new mine and provided funds for the project. Pine Point Mine and the community around it operated from 1965 to 1988. Mining very valuable base metal products, mineral production in the N.W.T. increased tenfold with the sale of Pine Point's ores.

Nickel deposits were exploited at Rankin Inlet in Nunavut from 1957 to 1962, and silver production from the Great Bear Lake region recommenced in the late 1960s chiefly from Echo Bay Mine and Terra Mine. Lead and zinc ores were also mined from the arctic islands at the Polaris and Nanisivik Mines in the 1980s. Large-scale production of tungsten was achieved at Cantung Mine between 1962-1986.

A steady climb in the price of gold during the 1970s brought new life to Yellowknife's gold mines. Con Mine began major expansion programs and Giant Mine developed its open pits. Yellowknife became the Territorial Capital in 1967 with the transfer of government from Federal authority to local officials.

During the 1970s, Canadians were achieving greater awareness of the effects of mining on the environment and aboriginal ways of life, and mine development in subsequent years was dictated largely by these factors. Mineral development was also impacted by new political developments, including land claims, which continue to have an effect on the mineral industry today. Stricter environmental conditions were imposed on operations, and older abandoned mines were targeted as part of remedial efforts.

The most recent mineral rush in the N.W.T. was the 'Diamond Rush of 1992', in which prospectors and companies staked large tracts of land in response to Chuck Fipke's diamond discoveries in 1991. Two mines have so far emerged from this exciting new field north of Yellowknife and deep in the 'Barrenlands' – Ekati Mine and Diavik Mine, with a third mine, Snap Lake, ready for production soon.

In April 1999, the Northwest Territories was split into two with the creation of Nunavut Territory. The western section retains the old name. As a result of this division, this report does not report on any of the mines that exist within Nunavut boundaries, although all of the mines were actually developed during the years as part of the N.W.T.

Today, the mineral industry struggles in the north, where costs are high and political forces continue to put pressure industrial development. Although the old gold mines have closed in Yellowknife (Giant Mine was the last to close in 2004), diamonds have been very successful and other commodity prices have turned around. Other mines, including a large poly-metallic deposit called the NICO, and reactivated gold projects such as Tyhee's Ormsby project and Seabridge's Tundra project, are also gearing up for major developments and probably production. The N.W.T. is large and still full of geological potential. Only the surface has been scratched. There are some difficulties that prospectors and companies face in the effort to develop and open up new mines. Land claims issues have not been settled and mineral exploration companies face incredible uncertainty when they come north to explore for metals. There are also miles of environmental regulations and other forms of 'red-tape' that have to be navigated. Due to geography, many deposits are written off as uneconomic. With new transportation corridors, improved markets, and better sources of local labor, hopefully those deposits will become productive mines.

If there were a final chapter in the story of mining up to this point it would be the effort of enthusiasts to preserve that history. The N.W.T. lacks a mining museum although there are many interesting artifacts, documents, and structures that could be put to a good use. Since 2000 a group now known as The N.W.T. Mining Heritage Society has been active in establishing a mining museum in Yellowknife. It will be located at Giant Mine and will cover the history of mining and mineral exploration in the N.W.T. and Nunavut. The author of this report is amongst the founding members of the group.

Your support in that effort would be appreciated. Please visit www.nwtminingheritage.com.



Report format



Name of Mine

Primary Commodities

Stage of Development (Current Status)

Years of Primary Development: xxxx-xxxx

Mine Development: shaft, adit tunnel, decline, winze, open pit, etc...

Years of Production: xxxx-xxxx

Mine Production: xxxx tons milled = xxxx oz/lbs/kg Au, Ag, etc...

Years of Bulk Sampling: xxxx-xxxx (if applicable)

Bulk Sampling: xxxx tons = xxxx oz/lbs/kg Au, Ag, etc... (if applicable)

Years of High-Grading: xxxx-xxxx

High-Graded: xxxx tons high-grade ore = xxxx oz/lbs/kg Au, Ag, etc...

Notes on Name, Stage of Development, and Current Status:

These old mines have been known as many names over the years. The most popular and accepted form of **Name** is usually used. Many deposits over which mines have been developed have no formal names, or are named after boring 'claim' references (eg: AL#1). In many cases I have chosen names for sites based on what bodies of water they are near, or anything in its history to suggest a good name (maybe an old company or prospector).

I have attempted to assign designations to each individual minesite. These designations are reflective of the **Stage of Development** up until this point (2008). The following are used and are ranked from least to greatest stage:

Minor Exploration: A few sites in the N.W.T. have completed some underground development, but the work completed is not significant. A short 30 foot shaft sunk by a small crew using hand-steel during a brief period of time is an example. I wanted to include these sites because they are technically underground mines, but in normal classification they practically fall off the radar.

Advanced Exploration: These sites have underground development that is more extensive than that completed under Minor Exploration. They have sunk deep shafts and completed extensive underground workings and are at a much more advanced stage of development.

High-Graded: These mines have produced small amounts of minerals by high-grading a surface exposure. They are characterized by their short term of operation (one season) and the relatively small amount of ore mined.

Bulk Sampled: Mine sites that have operated very briefly, usually for the purpose of high grading a deposit or processing an existing stockpile. No mine of any longevity is witnessed during this period. These sites are sometimes characteristic of not having any facilities or plants installed to process ores; rather, ores are shipped elsewhere in a large batch to be milled. In some cases a plant is installed on site, but it is usually for a very short period of time and the operations are not extensive enough. In more recent times, companies have mined bulk samples to test milling capabilities of ore deposits. A full production decision will be made based on the results, so this period of development is not seen as production. Older operations would not have distinguished the difference, but new designations are needed.

Minor Producer: Operations that have produced ores at their own facilities can be considered Minor Producers, but they have not been as extensive nor of longevity to be considered a complete Producer.

Satellite Producer: The existence of some mines relies on a larger operation within proximity. These satellite operations ship ores to a central plant normally on the property of another mine that the company owns where a mill is already established. This is not bulk sampling because the level of production is usually large and ongoing. The relationship between Ptarmigan and Tom Mines, Terra and Norex or Smallwood Mines, or that of Con and Giant Mines in recent years is an example of Satellite Production.

Producer: Mines that have operated for very long periods of time, self sufficiently operated with their own facilities, and produced massive amounts of ore.

The **Current Status** of the mine is its current operational status. *Abandoned* mines have not been remediated in any way since they closed. *Remediated* sites have been cleaned up either by the companies themselves or through the policies of the Federal Government. *Under Remediation* suggests that (as of 2006) work is under way in cleaning up a shutdown or abandoned minesite. *Operational* is a mine that is currently in operation, whether through an Advanced Exploration stage or through Production. *Non-Operational* is a mine that has been temporarily suspended of operations, or the company currently owning the property has publicly made plans to continue with work (eg: Nicholas Lake Mine, which hasn't been operated since 1994 and yet is still maintained by its owners). A non-operational mine becomes abandoned usually only if the current company goes defunct (eg: Ptarmigan Mine's former owner Treminco Resources Limited), since in this day-and-age closed mines are normally remediated by their former operators. *Under Construction* suggests that a mine, previously in the exploration stage, is gearing up for full production operations and is in the process of construction (eg. Prairie Creek Mine).

Introduction

This first paragraph gives the reader an introduction to the minesite, with a brief description of where it is located, when it operated, and whether or not the author has visited the site. Site visits to abandoned mines help to determine certain aspects of each former mining operation.

Brief History

A summary of staking, development, and production for each mine, where applicable. This paragraph emphasizes the pre-development stage of each property and lists any significant development completed since closure of the operation.

Geology and Ore Deposits

This provides a brief summary of the basic geological setting of the property, along with descriptions of the developed or productive deposits. The information is taken directly from the online database of the NWT Geoscience Office, but other sources are also used.

Operating Company Name (date to date)

This is the bulk of the information to be supplied in the description of each mining operation. The company or group that was in charge of operations is always listed above the section. Often, each site was owned by several different groups in joint-ventures or through parent companies, but only those directly involved in the development or production of each mine is listed. Occasionally, the parent company will be listed in brackets since it may be the case that this certain group is better known to be involved with the operation. ie: **Pine Point Mines Limited [Cominco Limited] (1965-1988)**. Other details are referenced in footnotes.

The life of the mine begins to be described once major developments get underway, whether it is the start of underground development or the start of milling. That is why for Discovery Mine the date of operation for the starts in 1946 even though the company actually first acquired the claims in 1945. Underground development did not proceed until the following year. This report documents the mining development at these sites, not the entire record of exploration history. Property staking and previous exploration is summarized in **Brief History**.

The goal of this project is to describe in some detail the facets of each mining operation. It is not possible to give the reader a complete explanation over how a mine operates, what the difference is between different mining methods, or the detailed metallurgical function of a milling plant The **Glossary of Terms** section of this report will help the layman understand some of these concepts. Internet resources, and the very interesting "Mining Explained", published by the Northern Miner Press Ltd., are recommended.

The following sectional headings that deal with specific aspects of each mine will be the most common:

Power Plant

A description of the power generation facilities and other machinery installed onsite to power the mining developments.

Milling Plant

A description of the flowsheet and process used to recover the desired commodity. Includes a listing of major machinery employed in the circuit and its specifications. Please refer to the original source for a complete explanation of each flowsheet.

Mining Equipment

Equipment and machinery in use for development operations, from surface haul trucks to underground mucking machines, or drills.

Camp and Plant Facilities

A description of the service buildings in use during the operation. Most of these facilities are quite common for any mine, therefore a complete listing is not normally necessary for each site, but in some cases building dimensions on some individual sites will be given.

Mine Production/Development

A summary of mine production and total development at the end of each operation. This data will also be displayed in tables, usually to give data year by year. A total mine summary which combines data from all years of operation may be given at the very end (also listed in the top title bar).

Others Categories

There will also be categories on crews working at the site, costs involved in the operation, transportation, general operating data, plus others depending on the site. The report is set in chronological order, so certain headings will highlight 'turning points' in the history of operation of the mines. For example: ***1938 Operations*** or ***New Shaft is Sunk on Vein***.

Please note that for the most part the report is written in a flowing, chronological format with these bits of information tagged into the story where necessary. Headings may repeat themselves decade by decade as the operating details of the mine changed.

2nd Operating Company Name (date to date)

Old mining properties that were first developed in earlier years sometimes reopened again by a new company. Take, for example, the Ptarmigan Mine. Here we can see that there were two mining operations that took place at this property, one that occurred in the early 1940s by Cominco Limited, and a second operation in the 1990s by Treminco Resources Limited. These operations, while built over the same hole in the ground, were two entirely different mining operations built 40 years apart. Since the buildings, equipment, and mining trends of each era were totally different, separate descriptions of both operations are required.

A mining operation will often change hands and come under the management of a new company. Although operations may remain quite similar despite a shift in company management, the advent of a new company will be considered the birth of a new period for the mine. An example is the Con Mine, which was owned by Cominco Limited from 1938 to 1986, when the mine was sold to Nerco Minerals Incorporated, then again sold in 1993 to Miramar Mining Corporation Limited.

Changes in ownership can often drastically alter the nature of mining operations at a mine. When Royal Oak Mines Incorporated went bankrupt and Giant Mine closed at the end of 1999, an entire gold mining operation closed. When it reopened in 2000 under the management of Miramar Mining Corporation, operations were cut back and a self supporting mine with its own production and management facilities was not seen again (ores were trucked and processed at Con Mine).

Companies have also changed their corporate structure over the years (mergers, new share capital), which usually results in a change in company name. To avoid confusion a new section will begin at the date that the company's name is changed. For example, **Discovery Yellowknife Mines Limited** began developing the Discovery Mine in 1946. The name of the company was changed in 1954 to **Consolidated Discovery Yellowknife Gold Mines Limited**, and then again in 1964 to **Discovery Mines Limited**. It is important to highlight these changes in the report at chronological points, so starting a new section is a practical solution. However, with Con Mine, the name 'Cominco' as the operating company is used from the beginning, although original the company was known as the

Consolidated Mining and Smelting Company of Canada until the 1960s. Cominco is a well known organization and it was felt there was no need to distinguish the change in name during the mine's narrative. A small footnote denotes the original company's name in the mines that apply to this company.

Exploration Since Mine Closure

After these mines closed, interest in their potential did not cease. In fact, many of the mines in this report never did achieve production so there is still a possibility that one day they will. Exploration on these old properties has been ongoing in order to determine their economic worth. Other mines that did produce are being reinvestigated because certain deposits, uneconomic 20 years ago, now have new potential with higher metal prices, new technology, etc. This section provides a brief summary on some of this work to date.

References and Recommended Reading

A listing of the sources used to research this description of each mine, referencing the citations used within the document. It also provides the reader with a source for additional reading material if more information is desired. Extensive use of newspaper articles has been used to provide data on the operation of old mining companies and their mines, and direct citations are provided within the document as to the exact newspaper and issue that I referenced. I also made extensive use of published sources and other books, most of which are referenced. I should also note that interviews with individuals that worked at these sites were very useful in my research.

I have collected an extensive amount of documents over the past decade which document the history of mineral development in the north. Much of this was acquired locally by the mining companies and through our small but interesting collections at government libraries and archives, including the *N.W.T. Archives*, and the *N.W.T. Geoscience Center*. Other information was found at national institutions, including the *National Archives of Canada* in Ottawa, *University of Alberta Libraries*, *Glenbow Archives* in Calgary, and private papers.

My archival and library collection has provided valuable information in the preparation of this report. Please contact me if you desire other information on the operations of these mines. If I do not have the information available in my files then I may know where to find it.



Glossary of Terms

The following mining related terms are used frequently in this report. This glossary provides a general explanation for the layman reader. Most of these entries are from “Mining Explained”, produced by the Northern Miner Press Ltd., 1996 Edition.

Adit	A horizontal mine tunnel opening, usually driven into the side of a hill.
Agitator (agitation)	Tanks in a milling plant which stirs a solution to aid in the separation of minerals from waste, usually with the introduction of compressed air.
Anomaly	Any departure of the norm which may indicate (in geophysical analysis or surface prospecting) the presence of a mineralized area.
Anticline	An upward fold or arch of rock strata.
Assay	A chemical test performed on a sample of ore to determine its mineral content.
Assessment Work	The amount of work, specified by laws of mining, which will keep a claim or property in good legal standing.
Autoclave	A high pressure and temperature vessel for oxidizing refractory ore. Ore or concentrate is fed into the strong vessel and placed under high pressure and temperature conditions with elevated oxygen levels to liberate the gold or base metals.
Back	The roof or upper part in any underground mining cavity.
Backfill	Mine waste rock or tailing sands used to support the stope roof after ore removal.
Ball Mill	A steel cylinder filled with steel balls that is rotated at great speeds. Mine ore is added into the mill and the balls are used as a crushing and grinding medium. See also: Rod Mill
Banded Iron Formation	A bedded deposit of iron minerals.
Barren (Barren Solution)	Said of rock or vein material containing no minerals of value, and of strata without coal, or containing coal in seams too thin to be workable. Barren solution in the milling circuit is the clear solution left over after the gold has been precipitated or filtered out.
Box Hole	A short raise driven up into a stope from a drift to permit the removal of ore from a stope. See also: Draw Point
Break	A large scale regional structural fault or regional shear zone.
Breast	The working face of the stope.
Broken Reserve	Ore which has already been blasted from a stope and is ready to be removed and milled.
Bulk Sample	A large tonnage of ore that is sent to be processed for the purposes of testing its metallurgical characteristics and to determine if recovery of the desired minerals is economic or feasible. The decision to bring a mine into commercial production is hinged on these results.
Bullion	Metal (gold, silver, lead, zinc, copper) which has been formed into refined bars or ingots.
Bunkhouse	Crew quarters for the employees of a mine when private accommodations are not available (usually in an isolated mining camp).

Byproduct	Other minerals which are produced from an ore and are not the primary mineral of interest.
Cage	In a mine shaft, the device, similar to an elevator car, that is used for hoisting personnel and materials.
Calcine	Concentrate that is ready for smelting (typically arsenic ores which have had arsenic and sulphur minerals roasted off)
Carbon-In-Pulp (CIP)	A method of recovering gold and silver from a pregnant cyanide solution by absorbing the precious metals to granules of activated carbon.
Chalcopyrite	A sulphide mineral of copper and iron.
Channel Sample	A sample composed of pieces of vein or mineral deposit that have been cut out of a small trench or channel.
Chip Sample	A method of rock sampling whereby a regular series of small chops is broken off along a line across the face.
Chute	A chute structure through which ore is drawn. Chutes are constructed below stopes to load ore cars, and they are also used to transport ores between bins in the milling circuit.
Claim	A portion of land held by a prospector or mining company under the authority of Federal or provincial laws.
Clarification (Clarifier)	Process of clearing dirty water from gold-bearing solution by removing suspended material.
Classification (Classifier)	Process of separating minerals and ore material according to size and density.
Closed Circuit	A loop in the milling process wherein a selected portion of the product of a machine is returned to the head of the machine for further processing because it does not meet the required finishing specification.
Collar	A timber or concrete structure built around the top of a mine shaft for structural support.
Concentrate	A rich mixture of minerals (in the form of a fine powder) that is produced from the milling process. This material requires further processing in the form of smelting or roasting to recover the desired mineral content. See also: Precipitate , Calcine
Conglomerate	A sedimentary rock consisting of rounded, water worn pebbles or boulders cemented into a solid mass.
Contact	The contact between two different rock formations. The heat from the intrusive rock meeting the existing rock forms a metamorphic reaction which often creates mineralization within the contact area, usually radioactive minerals.
Cone Crusher	See: Crusher
Contact	The place or surface where two different kinds of rocks meet. Applies to sedimentary rocks, as the contact between a limestone and a sandstone, for example, and to metamorphic rocks; and it is especially applicable between igneous intrusions and their walls. Mineral deposits are often formed in contact zones.
Core (Drill Core)	A long, cylindrical piece of rock brought to surface by diamond drilling.
Crosscut	A horizontal mine tunnel that is driven perpendicular to the strike of a vein or deposit. Crosscuts are typically driven to cut across to another deposit.
Crusher	A machine for crushing rock or other materials. Jaw Crusher: Rock is crushed by

steel plates pounded against one another. Cone Crusher: Rock is crushed between a gyrating crushing head and a truncated cone.

Custom Milling	An agreement where a company with a milling plant agrees to process ores from another company's mine at a negotiated price.
Cut-and-Fill (Stoping)	A method of stope mining where ore is mined in slices, or lifts. The ore is then removed completely from the stope. In order to reach the next slice in the stope, the excavation is filled with waste rock or backfill and formed with a cement floor to support heavy machinery. The next slice is mined and the process is repeated until the stope is completely mined to the above level.
Cyanidation	A method of extracting gold grains from crushed ores by dissolving it in a weak cyanide solution.
Decline	An underground ramp that spirals down to a depth, usually with a -10% grade. Declines are a cheaper method to developing an underground deposit than vertical shafts. Also called a ramp.
Deposit	A mineralized body which has been intersected by sufficient closely spaced drill holes and/or sampling to support sufficient tonnage and average grade of metal(s) to warrant further exploration-development work. Does not necessarily qualify as a commercially mineable ore body.
Development	The type of work performed to access and mine a deposit, either underground or on surface. Includes shaft sinking, crosscutting, drifting, and raising.
Diabase	A common basic igneous rock usually occurring as dykes or sills.
Diamond Drill	A rotary machine that drills into a deposit to recover a core sample. A diamond-drill effectively allows geologists to probe a deposit and determine its size and mineral content.
Dilution	Waste rock from unmineralized walls outside of a vein or shear zone, which is by necessity removed along with the mineralized ore during the mining process, subsequently lowering the grade of the ore.
Diorite	An intrusive igneous rock composed chiefly of sodic plagioclase, hornblende, biotite, or pyroxene.
Dip	The inclination of a geologic structure (bed, vein, fault, etc.) from the horizontal; dip is always measured downwards at right angles to the strike.
Dividend	Cash or stock awarded to the shareholders of a company. Companies which are financially well off and whose operations show a modest profit will usually declare a dividend in a show of appreciation to the shareholders for their original investments.
Dore Bars (Gold or Silver Bricks)	Unrefined gold bars that have been poured in the final stages of a gold mining operation. These bars contain many impurities that require additional refining to recover 99.9999% gold. Dore bars are the final product of a gold mine and are shipped, usually to the Royal Canadian Mint, to become gold bullion.
Drag Fold	The result of a plastic deformation of a rock unit where it has been folded or bent back on itself.
Drawpoint	An underground opening at the bottom of a stope through which broken ore from the stope is extracted. See also: Box Hole
Drift	A horizontal mine tunnel that follows the strike of a vein or deposit. See also: Crosscut
Drill Bit	The hardened and strengthened device at the end of a drill rod that transmits the energy of breakage to the rock. The size of the bit determines the size of the hole.

	A bit may be either detachable from or integral with its supporting drill rod.
Drill-Indicated Reserves	An ore reserve calculation based on widely spaced drill holes. More detailed work is required to classify the ore as probable or proven.
Dry (Mine Dry)	The mine facility where workers change into work clothes. Clothes are hung up on hooks and baskets to dry.
Dump	A pile of broken rock on the surface.
Dyke	A long and thin body of igneous rock that intruded a fissure in older rock. Can contain pegmatite minerals or kimberlite (diamond bearing) ore.
Exploration	The work performed before mining is undertaken which establishes a deposit's size, character, and grade.
Extrusive	A body of igneous rock formed by lavas extruding onto surface through volcanoes.
Face	The end of a drift, crosscut or stope in which work is taking place.
Fault	A break in the crust caused by tectonic forces which has moved the rock apart.
Feldspar	A group of common rock-forming minerals that includes microcline, orthoclase, plagioclase and others.
Felsic	Term used to describe light-coloured rocks containing feldspar and silica.
Flotation	A milling process in which valuable mineral particles are induced to become attached to bubbles and float, and others sink.
Flowsheet	An illustration or description which outlines the sequence of operations, step by step, by which ore is treated in a milling plant.
Flux	A chemical substance that reacts with gangue minerals to form slags, which are liquid at furnace temperature and low enough in density to float on the molten bath of metal or matte.
Fold	Any bending or folding of a rock strata.
Footwall	The rock on the underside of a vein or ore structure. See also: Hangingwall
Free-Milling	Ores of gold and silver from which the metals can be recovered by concentrating or cyanidation methods without resorting to pressure leaching or roasting treatment.
Gabbro	A dark, coarse-grained igneous rock.
Galena	Lead sulphide, the most common ore mineral of lead.
Geophysics	The study of the physical properties of rocks and minerals. Types of geophysical surveying include magnetism, specific gravity, induced polarization, electrical conductivity, and radioactivity.
Gossan	The rust-coloured staining of a mineral deposit, generally formed by the oxidation or alteration of iron sulphides.
Grab Sample	A sample from a rock outcrop, usually picked from the best looking material and is not intended to be a representative sample of the deposit.
Grade	A calculation of average mineral content in a single unit of ore (calculated as ounces per ton, or grams per tonne).
Granite	A coarse-grained intrusive igneous rock consisting of quartz, feldspar, and mica.
Greenstone Belt	An area underlain by metamorphosed volcanic and sedimentary rocks.

Grizzly	A steel grate placed over top of a chute or ore pass for the purpose of stopping large pieces of rock or ore that will jam the ore pass or crusher. The large pieces are broken down by hammer or drill.
Grouting	The process of sealing off a water flow in rocks by forcing a thin slurry of cement into crevices through diamond drill holes.
Hangingwall	The rock on the upper side of a vein or ore deposit. See also: Footwall
Head Grade	The average grade of ore fed into a milling plant.
Headframe	A structure built over-top of a shaft that functions as part of the hoisting system.
Hematite	An oxide of iron.
High Grade	Rich ore.
Hoist	An item of machinery that is used primarily to service a mine shaft with an elevator type of function for man-cage and skip handling.
Host Rock	The rock surrounding an ore deposit.
Igneous Rock	Rocks formed by the solidification of molten material from far below the Earth's surface.
Induced Polarization	A method of geophysical exploration employing an electrical current to determine indications of mineralization.
Intrusive	A body of igneous rock formed by the consolidation of magma intruded into other rocks. This is opposite of Extrusive .
Ion Exchange	An exchange of ions in a crystal with ions in a solution. Used as a method for recovering valuable metals, such as uranium, from a solution.
Jaw Crusher	See: Crusher
Jig (Mineral Jig)	Milling equipment used to concentrate ore on a screen submerged in water, either by the reciprocating motion of the screen or by the pulsation of water through it.
Kimberlite	A volcanic rock that hosts diamonds.
Lagging	Increasing the diameter of a hoist drum by coiling the drum surface with wood. Also a method of support the ceiling of drifts, crosscuts, and the roof of a stope by using wood timbers as a crib structure.
Launder	A chute or trough for conveying pulp, water, or powdered ore in a milling plant.
Leaching	A chemical process for the extraction of valuable minerals from ore.
Lens	A body of ore that is thick in the middle and tapers towards the ends.
Level	A horizontal opening underground consisting of drifts and crosscuts. They are driven off of shafts or decline ramps and are spaced at regular intervals.
Mafic	Igneous rock composed mostly of dark, iron- and magnesium-rich minerals.
Metallurgy	The study of extracting minerals from rocks.
Metamorphic Rocks (Metamorphism)	Rocks which have undergone a change in texture or composition as the result of heat and/or pressure.
Mill (Processing Plant, Concentrating Plant)	A processing plant that uses a variety of chemical and mechanical techniques to breakdown ore from a mine and recover its mineral content. Products from a mill usually require further treatment (refining or smelting) to fully recover the desired metals. Mills produce concentrates or precipitates, and tailing wastes. Processing plants at diamond mines do not require any further treatment to recover the rough diamond products, although diamond cutting is required to market a diamond

much like specialized refining is necessary to market gold bars.

Mineable Reserves	Ore reserves that are known to be extractable using a given mining plan.
Minerals	A naturally occurring substance having definite physical properties and chemical composition and, if formed under favourable conditions, a definite crystal form.
Mineralization	A mineralized body which has been intersected by sufficient closely spaced drill holes and/or sampling to support sufficient tonnage and average grade of metal(s) to warrant further exploration-development work. This deposit does not qualify as a commercially mineable ore body.
Muck	Ore or rock that has been broken by blasting.
Net Smelter Return	A share of the net revenues generated from the sale of metal produced from a mine. NSR agreements are often negotiated between the original vendors or prospectors of a mineral claim and the company that purchases the right to the property.
Open Pit, Open Cut	Large surface pits that are mined to extract shallow or low-grade ores at economical costs.
Option	An agreement to purchase a property between the property vendor and a party or company who wishes to explore the property.
Ore	A mixture of ore minerals from which at least one of the metals can be extracted at a profit.
Ore Car	A railway car adapted to carrying coal, ore, and waste underground. Different styles include end-dump, side-dump (rocker or Granby style), or bottom dump.
Ore Pass	Vertical or inclined raise opening underground for the downward transfer of ore connecting a level with the lower level where the ore can be loaded onto skip for hoisting to surface.
Orebody	A natural concentration of valuable material that can be extracted and sold at a profit.
Ore Reserves	The calculated tonnage and grade of mineralization which can be extracted profitably; classified as Drill Indicated Reserves , Possible/Inferred Reserves , Probable/Indicated Reserves , and Proven/Measured Reserves according to the level of confidence that can be placed in the data.
Ore Shoot	The portion, or length, of a vein or other structure, that carries sufficient valuable mineral to be extracted profitably.
Outcrop	An exposure of rock that can be seen on surface and is not covered by soil or water.
Oxidization	A chemical reaction caused by exposure to oxygen that results in a change in the chemical composition of a mineral.
Pegmatite	A coarse-grained, igneous rock, generally coarse but irregular in texture and similar to a granite in composition. Usually occurs in dykes or veins.
Pillar	A block of solid ore or other rock left in place to structurally support the shaft, walls or roof of a mine.
Pitchblende	An important uranium ore mineral, highly radioactive.
Plant	The operational facilities of a mine. (Milling Plant, Power Plant)
Porphyry	An igneous rock in which large crystals are set in a fine-grained groundmass.
Portal	A mine tunnel opening that identifies the start of an adit or a decline.

Possible Reserves (Inferred Reserves)	Valuable mineralization not sampled enough to accurately estimate its tonnage and grade. Also called “Inferred Reserves”
Precipitate	A rich mixture of minerals that is produced from the milling process. This material requires further processing in the form of smelting or roasting to recover the desired mineral content. See also: Concentrate
Probable Reserves (Indicated Reserves)	Valuable mineralization not sampled enough to accurately estimate the terms of tonnage and grade. Also called “Indicated Reserves”
Production	The amount of ore milled and the amount of minerals recovered from a mining operation. Commercial production is the stage in which a mine enters once operations achieve a fluent state.
Project	A mining property in the stage of exploration and development.
Prospect	A mining property, the value of which has not been determined by exploration.
Proven Reserves (Measured Reserves)	Reserves that have been sampled extensively by closely spaced diamond drill holes and developed by underground workings in sufficient detail to render an accurate measurement of grade and tonnage.
Pulp	In a milling circuit, the pulverized or ground ore in solution.
Pyrite	A yellow iron sulphide mineral, normally of little value. Also known as “Fool’s Gold”.
Quartz	Common rock-forming mineral consisting of silicon and oxygen.
Quartzite	A metamorphic rock formed by the transformation of a sandstone by heat and pressure.
Radioactivity	The property of emitting alpha, beta, or gamma rays by the decay of the nuclei of atoms.
Raise	A vertical or incline mine tunnel that is driven up from a mine working to tap into a deposit in preparation for certain types of stope mining. Raises are commonly driven to connect mine levels, to break-through to the surface for ventilation, or as escape-routes, man-ways, or ore passes.
Rake	The trend of an orebody along the direction of its strike.
Rare Earth Elements	Scarce minerals such as niobium and yttrium.
Reclamation	The restoration of a mining site after mining or exploration activity has ceased. To return the site to a natural state as it was before mining disturbance.
Reconnaissance	A preliminary survey of ground.
Recovery	The percentage of valuable metal in the ore that is recovered by metallurgical treatment.
Refinery	The plant in which precipitate or concentrates from the gold milling process are smelted and poured into the form of rough gold dore bars.
Refractory Ore	Ore that resists the action of chemical reagents in the normal treatment processes and which may require pressure leaching or other means to effect the full recovery of the valuable minerals.
Resuing	A method of stoping in narrow-vein deposits whereby the wall rock on one side of the vein is blasted first and then the ore.
Roaster	A plant designed to heat a refractory ore to drive off volatile substances or oxidize the ore. The oxidation of the ore liberates the gold. Typically produces poisonous gases and arsenical wastes that must be disposed of properly or treated.

Rockbolting	The act of supporting openings in rock with steel bolts anchored in holes drilled especially for this purpose.
Rockburst	A violent release of energy resulting in the sudden failure of walls or pillars in a mine.
Rod Mill	A rotating steel cylinder that uses steel rods as a means of grinding ore. See also: Ball Mill
Room and Pillar Stopping	A method of mining flat-lying ore deposits in which the mined-out area, or rooms, are separated by pillars of the same size.
Sample / Sampling	A small portion of rock from a mineral deposit, taken so that the metal content can be determined by assaying.
Scaling	The act of removing loose rock from the backs and walls of an underground opening using a hand-held scaling bar or mechanized hammers.
Scarp/ Escarpment	A cliff or steep slope along the margin of a plateau or hill.
Schist	A foliated metamorphic rock, the grains of which have a roughly parallel arrangement, generally developed by shearing.
Sedimentary Rocks	Rocks formed from material derived from other rocks and laid down under water and cemented over time.
Shaft	A vertical or inclined mine opening that is used as a hoisting compartment to service the underground workings of a mine. Headframes are built over-top of a shaft as a function of the hoisting operation.
Shearing	The deformation of rocks by lateral movement along innumerable parallel planes, generally resulting from pressure and producing such metamorphic structures as cleavage and schistosity.
Shear Zone	A zone in which shearing has occurred on a large scale.
Sheave Wheel	A large, grooved wheel in the top of a headframe over which the hoisting rope passes.
Shrinkage Stopping	A stopping method which uses part of the broken ore as a working platform and as support for the walls of the stope.
Silicia	Silicon dioxide. Quartz is a type of silicia.
Sill	An intrusive sheet of igneous rock of roughly uniform thickness that has been forced between the bedding planes of existing rock.
Skarn	Metamorphic rock surrounding an igneous intrusive where it comes in contact with a limestone or dolostone formation. Common host of tungsten/scheelite ores.
Skip	A self-dumping bucket used in a shaft for the hoisting of ore.
Slag	The mass separated from the fused metals in the smelting and refining process.
Slashing	The method of enlarging or widening a lateral underground working so that larger machinery can be used in the tunnels.
Station (Shaft Station)	An enlarged opening at the start of a mine level, blasted off a shaft for the storage and handling of equipment on that level.
Stockpile	Broken ore stored on surface in preparation for milling.
Stope	An excavation in a mine from which ore is extracted. See also: Shrinkage, Cut-and-Fill, Room-and-Pillar Stopping.
Strike	The direction or bearing (measured by angle on the horizontal surface from true

	north) of a vein or rock formation.
Stringer	A narrow vein or irregular filament of a mineral traversing a rock mass.
Strip	To remove the overburden or waste rock overlying an orebody in preparation for mining by open pit methods.
Sub Level	A level or working horizon in a mine between main working levels.
Sulphide	A compound of sulphur and some other element.
Sump	An underground excavation where water accumulates before being pumped to surface.
Syncline	A down-arching fold in bedded rocks.
Tailings (Tailings Ponds)	Material rejected from a mill after most of the recoverable minerals have been collected. These wastes are impounded in protective ponds, which are blocked off by dams and dikes to prevent the (sometimes) hazardous material from entering the natural watershed. Sometimes tailings will contain a small mineral content that may be economical to re-process to recover the previously un-recovered metals.
Thickener	A large, round tank used in milling operations to separate solids from liquids; clear fluid overflows from the tank and rock particles sink to the bottom.
Tons/Tonnes per Vertical Foot/Meter	Common unit used to describe the amount of ore in a deposit, ore length is multiplied by the width and divided by the appropriate rock factor to give the amount of ore for each vertical foot (or meter) of depth.
Tram	To haul cars of ore or waste in a mine using a line of ore cars hauled by locomotive.
Trench	A long, narrow excavation dug through overburden, or blasted out of rock, to expose a vein or ore structure.
Uncut Assay Value	The actual assay value of a core sample as opposed to a cut value which has been reduced by some arbitrary formula.
Vein	A fissure, fault or crack in a rock filled by minerals that have traveled upwards from some deep source.
Visible Gold	Native gold which is visible to the naked eye.
Volcanic Rocks	Igneous rocks formed from magma that has flowed out or has been violently rejected from a volcano.
Vug	A small cavity in a rock, frequently lined with well-formed crystals. Amethyst commonly forms in these cavities.
Wall Rocks	Rock units on either side of an orebody. See also: Footwall, Hangingwall
Waste	Unmineralized rock. Also mineralized rock that cannot be mined at a profit.
Winze	An internal shaft which is collared from an underground heading rather than on surface.
Zone	An area of distinct mineralization.



Mine Site Location Maps

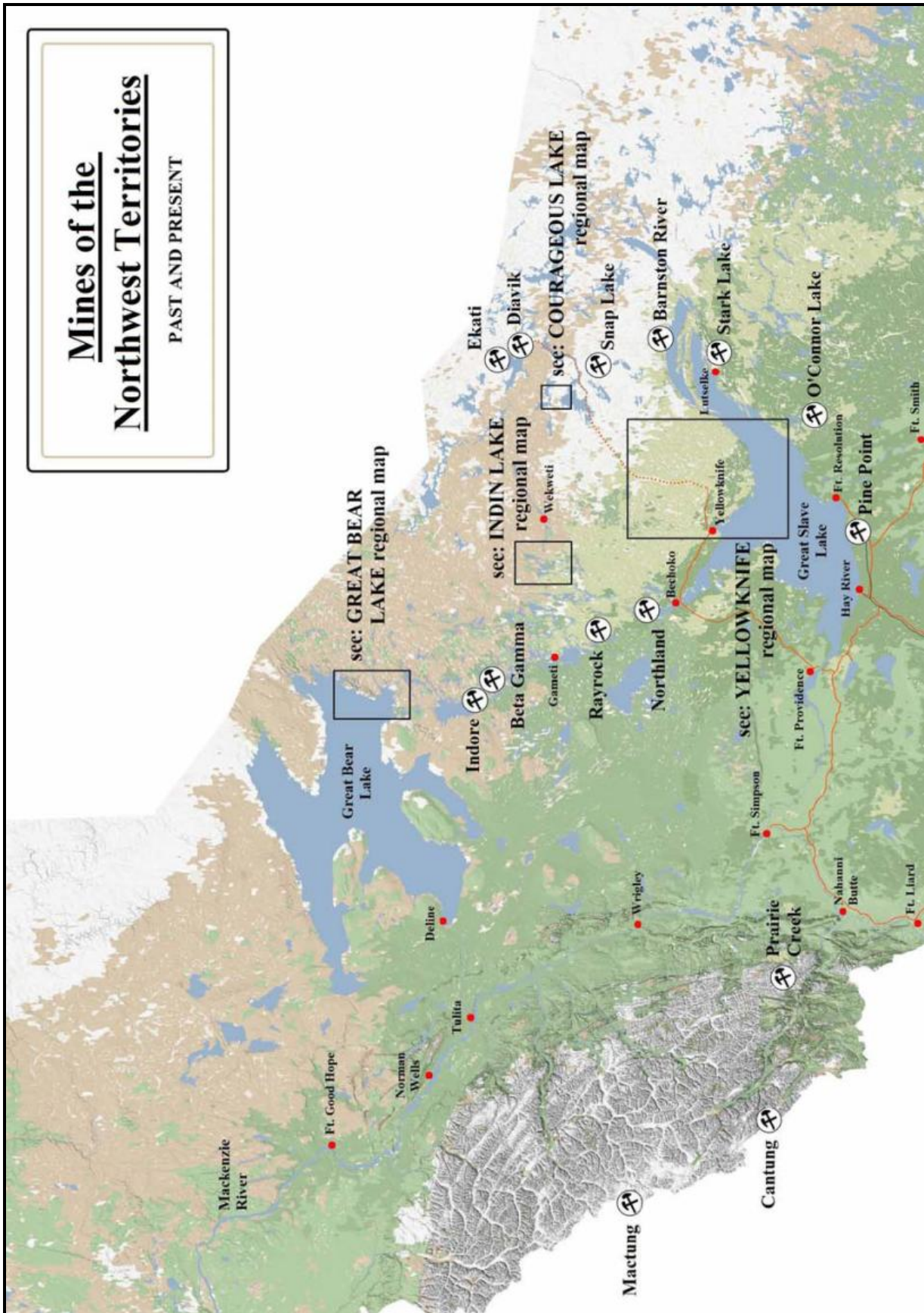


Figure 1. Location of mine sites in the Northwest Territories. See Figure 2 for Yellowknife Region detail, Figure 3 for Great Bear Lake Region detail, Figure 4 for the Indin Lake area, and Figure 5 for Courageous Lake Region detail.

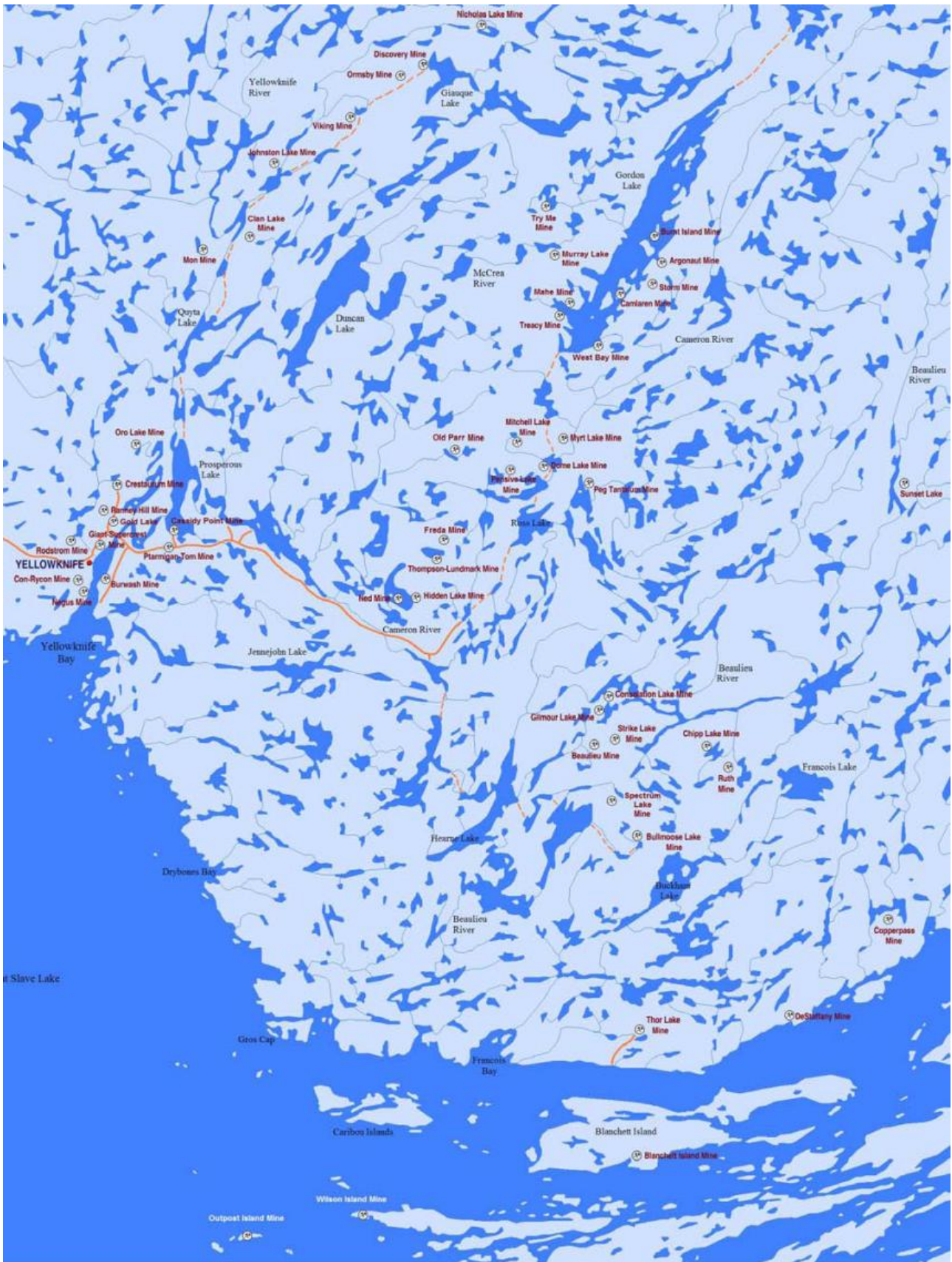


Figure 2. Yellowknife regional map showing locations of mines.

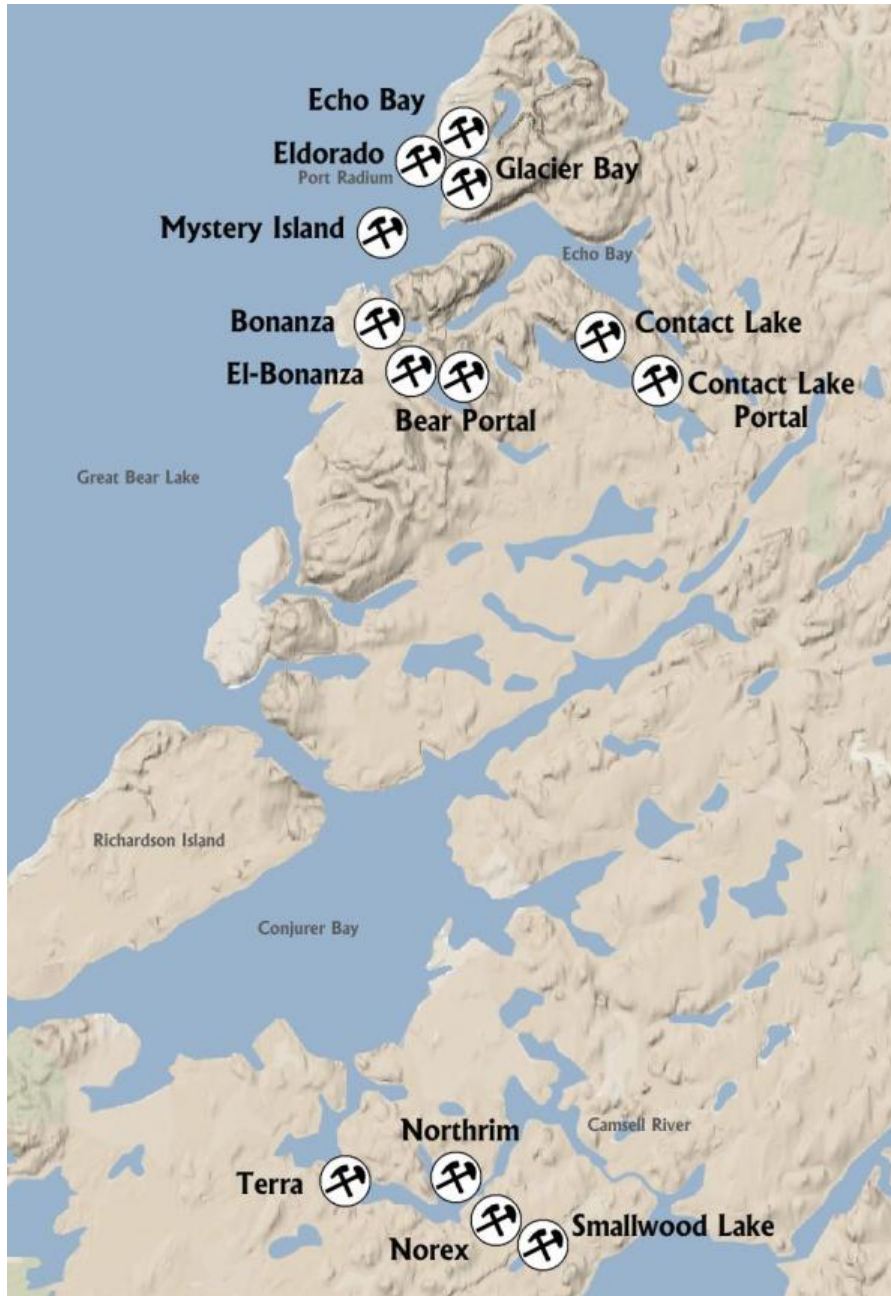


Figure 3. Great Bear Lake Region showing location of mines.

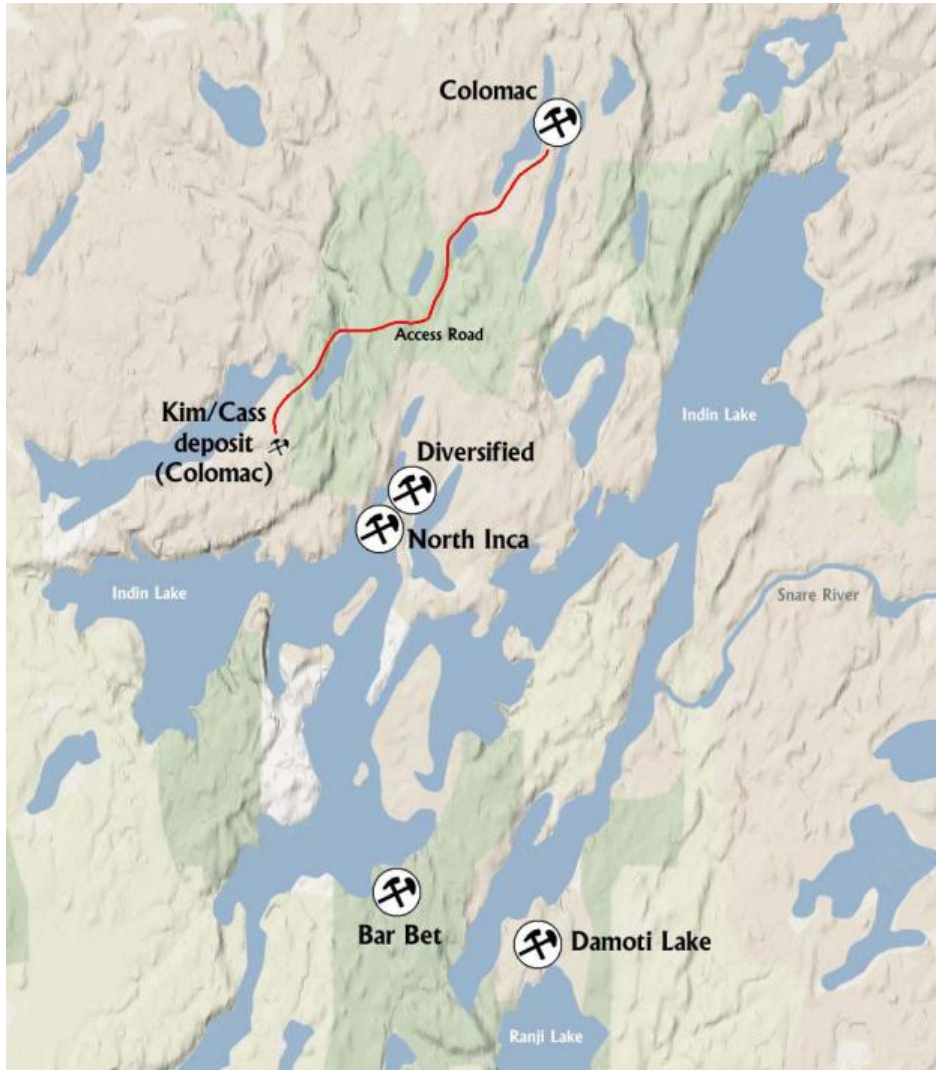


Figure 4. *Indin Lake Region showing location of mines.*

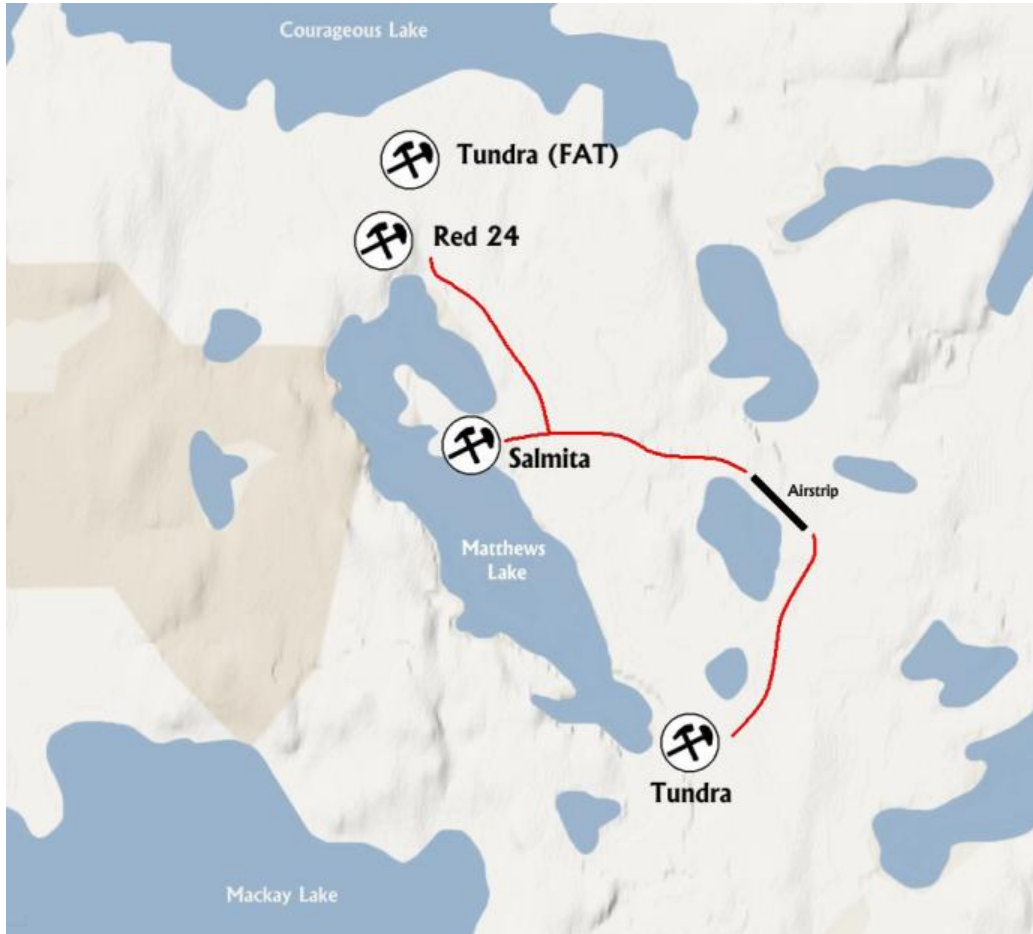


Figure 1. *Courageous Lake Region showing location of mines.*



The History of the Mines



ARGONAUT

Minor Producer (Abandoned)

Gold, Tungsten

63° 02' 00" 113° 08' 00"

Years of Primary Development: 1938-1943

Mine Development: 40' vertical shaft, open cuts

Years of Production: 1942-1943

Mine Production: 720 lbs tungsten concentrates

Introduction

The Argonaut Mine is located on the east shore of Gordon Lake, about six kilometers southeast of Sandy Point Lodge, and 90 kilometers northeast of Yellowknife, NWT. This former tungsten mine, which closed around 1943, burned down in a 1998 forest fire, the remains of which were visited by the author in September 2001.

Brief History

The original claim groups, including the 'Pormac' claims, were staked in 1937 and 1938. Prior to 1942, work was focused on the gold potential of the claims, but with a high demand for tungsten attention turned to producing a concentrate of this valued metal. Minor production was attained between 1942 and 1943 by Goodrock Gold Mines Limited. The original claims lapsed in 1958. In 1997, Joe McByran staked the 'Argonaut' claim overtop of the area. It lapsed in 2000. Walt Humphries staked the 'Argo' claim in September 2004.

Geology and Ore Deposits¹

Gordon Lake is underlain by greywackes and slates of the Yellowknife Supergroup. These thin interbeds were deposited as a turbidite sequence. In places, the greywacke beds become thick to massive and any bedding may be obliterated by metamorphism. Quartz veins, which are abundant throughout the sedimentary rocks, consist of high-temperature glassy quartz with tourmaline and a few feldspar crystals in schists and hornfels. The #1 Vein on the property, which strikes northeast, occurs as lenses and stringers with minor scheelite reported.

Galloway Gordon Lake Mines Limited (1938)

Galloway Gordon Lake Mines Limited was organized in March 1938 to explore the 'Pormac' and adjoining claim groups through the sinking of a 40 foot shaft, trenching, and diamond drilling, all on the #1 vein. At this time, the property was regarded as a gold prospect. The advent of war and the inability of the company to raise additional finances resulted in a cessation of work (Lord, 1951).

Goodrock Gold Mines Limited (1942-1943)

In March 1942, Goodrock Gold Mines Limited acquired a controlling interest (60%) in the claims from Galloway Gordon Lake Mines Limited. A crew was assembled headed by Claude Watt and arrived on the property on April 20th 1942 (The Toronto Star, Apr. 22nd 1942). Exploration of the surface veins by several pits and trenches over a strike length of 2,000 feet revealed quantities of scheelite ores, mostly within the #1 Vein (The Toronto Star, May 14th 1942). In July 1942 it was reported that diamond drilling encountered scheelite ores within the vein to a depth of 150 feet (The Toronto Star, July 3rd 1942). Sampling of the ore material revealed tungsten and gold in commercial quantities, with assays ranging from 1% to 9% tungsten oxide (WO₃) and up to 1 ounce per ton gold (The Globe and Mail, July 14th 1942). Work was focused on an ore shoot south of the shaft, 60 feet long and up to 5 feet wide (The Yellowknife Blade, July 6th 1942).

Company directors decided to install a small 10 ton per day concentrating plant at the Gordon Lake property during the summer of 1942. Material and equipment was airlifted to the site from Yellowknife in September 1942. The milling plant was installed by November 1942, and production began (Lord, 1951). Mill feed was derived from a number of large trenches east of the plant and ore was pushed by wheelbarrow to the ore bins. The mill was a timber structure, small around the base, but about 3-stories high. The old shaft was fitted with a tripod timber A-frame for hoisting (Bill Holden, pers. comm.).

In December 1942, it was reported that mill heads were 2.24% WO₃ but the recovery rate was a very low 62%. It was hoped that finer grinding of the ores would achieve better recoveries. Also in that month additional power plant machinery was installed with the hope of increasing production (The Toronto Star, Dec. 18th 1942).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Milling Plant

Little information is available on how this plant operated, but it is known that a jaw crusher was employed in the crushing circuit. A small ball mill received crusher discharge, and a number of Wilfley tables recovered a concentrate that was bagged, and loaded into an ore cart. A rail line connected the mill-site with the small bay on Gordon Lake, where the concentrate was loaded onto boat or airplane. The 10 ton per day plant was powered by a number of 12-horsepower Lister and Fairbank-Morse diesel engines, which drove the mill equipment by drive shaft and belt. There was also a small assaying facility on the property (W.R. McBryan, pers. comm.).

Camp Site

The camp consisted of a log bunkhouse, cookery, shop, and up to four residences with a capacity for 25 persons. Around five to ten men were employed with families residing on site. Power was supplied by a small Lister genset. Claude Watt was mine manager, W.R. "Red" McBryan was a mechanic or surface foreman, and A.L. Schneider was a consulting engineer in 1942. A trail connected the log cabin camp to the #1 vein and mill site, about ½ kilometer southeast near a small pond (W.R. McBryan, pers. comm.).

Mining Operations

Mining development was selectively completed on sections of the vein with high scheelite assays. Drilling was done by portable Warsop drills and hand-steel methods (W.R. McBryan, pers. comm.).

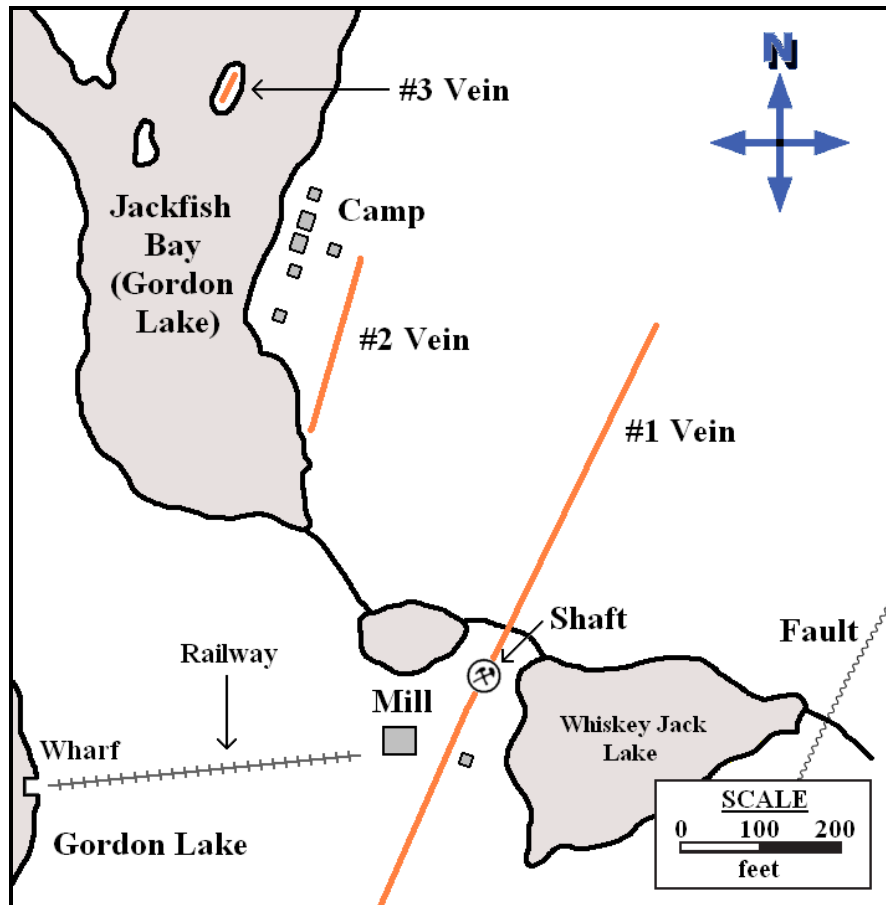


Figure 1. *Argonaut Mine property map, c.1944.*

Operations at the Argonaut tungsten property continued into 1943 after which no further information is available. In January 1943, it was reported that production was progressing satisfactorily and that a substantial amount of concentrate was ready for shipment, either by winter road or aircraft. There were also reports of extremely high-grade tungsten ores being mined from the depths of the open cuts, including a sample that assayed 16% WO_3 (The Toronto Star, Jan. 27th 1943). It is assumed that milling at Argonaut Mine ceased sometime in 1943. The operation was run on a sporadic schedule and the mill probably was only able to operate during the warmer months of the year.

Production

First shipment of tungsten concentrate from the property was reported in June 1943 (Canadian Mines Handbook, 1943). No definite record of production is known to exist, however statistics available for 1943 indicate that 720 pounds of tungsten concentrate were produced from the Northwest Territories during that year (Statistics Canada, 1957). The Argonaut Mine was the only tungsten mine reported to operate during 1943, so it is assumed that either all or the bulk of this production was from this property. Actual tungsten oxide content of these concentrates is unknown. Equipment was dismantled in 1945-1946 and sent to other mining operations in the Gordon Lake area (W.R. McBryan).

Exploration Since Mine Closure

Argonaut Yellowknife Mines Limited was formed in 1944 and acquired the property. They were primarily interested in its value as a gold property. As indicated, it is possible that they conducted some development and perhaps even ran the mill as a bulk sampling plant, but no record exists. No extensive exploration has been conducted. In 1997, Joe McBryan staked the 'Argonaut' claim overtop of the area. It lapsed in 2000. Walt Humphries staked the 'Argo' claim in September 2004.

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National Mineral Inventory (Goodrock). NTS 85 P/3 Au 7.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085PSW0024

Personal communication: W.R. McBryan; Jim McAvoy Jr.; Bill Holden



BAR-BET

High-Graded (Abandoned)

Gold

64° 09' 15" 115° 13' 15"

Years of Primary Development: 1938-1941

Mine Development: high-grade trenching

Years of High-Grading: 1938-1941

High-Grading: 331 oz Au

Introduction

This small high-grading gold operation was located on the south side of Indin Lake, 193 kilometers northwest of Yellowknife, N.W.T. It has not been visited by the author of this report.

History in Brief

Gold was discovered on the Barker vein in 1938 by prospectors with Territories Exploration Limited and the 'Anna' claims were staked. Some gold was high-graded from the showing by their crews during 1938-1939. In 1940, Peter and Charles Schwerdt re-staked the property and hand-cobbed more gold at these claims. It was rumored that they had actually stolen the gold from the Con and Negus Mines in Yellowknife.

Geology and Ore Deposits¹

The area is underlain by foliated, dark green, Archean andesitic metavolcanic rocks of the Yellowknife Supergroup. The main showing, the Barker vein, is a 40 foot long, quartz-carbonate vein. The vein is truncated on the southern end by a fault and ends abruptly in soft grey schist in the north. The vein is one foot wide in the southern portion and widens to 3 to 6 feet to the north. All of the gold was extracted at the intersection of the south end of the vein with the truncating fault.

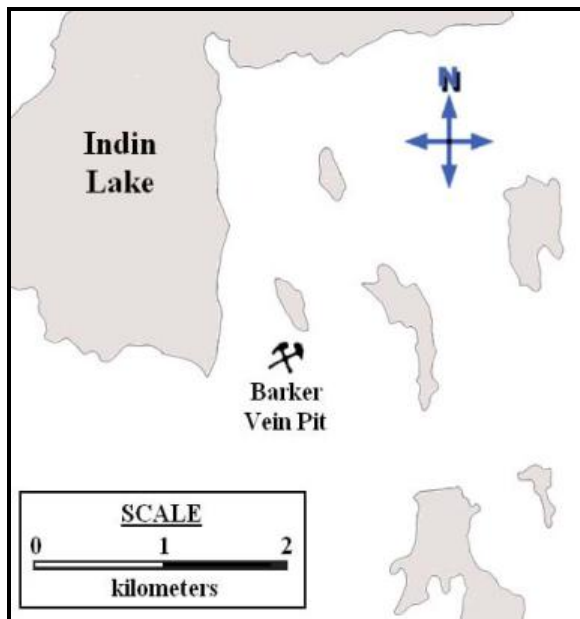


Figure 1. Bar-Bet Mine property map.

Territories Exploration Ltd. (1938-1939)

Crews with Territories Exploration Limited were at work during 1938-1939 trenching the Barker vein. They hand-cobbed 1600 pounds of ore and were able to extract 83 ounces of gold by mining a pipe-shaped oreshoot 1 foot in diameter and 10 feet in length. The company abandoned the claims in 1940 because of war conditions (Lord, 1941).

Pete and Charles Schwerdt (1940-1941)

In 1940, the claims were re-staked by Pete and Charles Schwerdt, and an additional 248 ounces of gold were recovered from the Barker vein by 1941.

Exploration Since Mine Closure

The claims were diamond drilled and trenched by American Yellowknife Mines Limited in 1945-1946, returning interesting gold values (National Mineral Inventory). The property was re-staked as the 'Vidie' claims by G.E. Swanson in 1982, followed in 1985 as the 'Barker' claims, and in 1992 as part of the 'Hela' claims. In 1984, the old trenches were mapped and sampled, and it was reported that further work was merited (Day, 1984; Flood, 1994).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

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BARNSTON RIVER

Minor Exploration (Abandoned)

Copper

62° 52' 30" 109° 31' 00"

Years of Primary Development: 1941

Mine Development: three short shafts (20 feet)

Introduction

The Barnston River is located on McLeod Bay of the East Arm of Great Slave Lake. The old copper workings are located near the mouth of the river, south of the old Arctic Star Lodge.

History in Brief

A small crew sank a few short shafts near the mouth of the Barnston River in 1941. It was a small copper prospect known as the 'Ryan' claims.

Geology and Ore Deposits

No information available.

Ryan Exploration & Development Company Limited (1941)

The 'Ryan' group of nine claims were staked in August 1939 by J. Russell and T.O. Evans for the Ryan Exploration & Development Company Limited of Edmonton. No work was done in 1940, but in the summer of 1941 five men were employed for about six weeks under the direction of T.O. Evans. Copper was the target of exploration. The crews sank three small shafts, each six-feet square, to depths ranging from 14 to 21 feet. The shafts were all about 1,000 feet apart. The shafts exposed considerable chalcopyrite across widths of several feet near the surface, but none below a depth of about 12 feet. (Lord, 1951)

Exploration Since Mine Closure

No known work.

References and Recommended Reading

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.



BEAR PORTAL

Minor Exploration (Abandoned)

Silver

66° 00' 30" 118° 01' 30"

Years of Primary Development: 1933-1934

Mine Development: adit tunnel

Introduction

The Bear Portal Mine is located in the Echo Bay region of Great Bear Lake, 9 kilometers south of LaBine Point (Port Radium) on the northeast side of Miles Lake and 2 kilometers east of the El-Bonanza Mine. It is 433 kilometers northwest of Yellowknife, NWT. The author has coined the name of this site, as the original claims that covered this ground were known as the 'Bear' claims. The area was viewed from the air by the author in July 2005, but no evidence of this mine could be seen.

History in Brief

The 'Bear' claims were staked by Charles Sloan in 1931 for the Great Bear Syndicate, operated by J.J. Byrne of Toronto. A small crew under the direction of newly formed Great Bear Lake Mines Limited blasted out this short tunnel during 1933-1934 as part of assessment work on this group of claims. They were looking for silver and radium minerals. No other work is reported.



Figure 1. Regional map of the Dowdell Point area, showing the location of the Bear Portal Mine.

Geology and Ore Deposits¹

The property is situated within the Great Bear Magmatic Zone, a part of the Bear Structural Province of the Canadian Shield. Mineralization occurs within a narrow strip of altered volcanic and sedimentary rocks that are part of the Port Radium Formation of the Apehbian Labine Group. The northwest striking volcanic-sedimentary sequence is bordered by the Late Archean granite body of the Bear Batholith to the south and by Archean granodiorite of the Bertrand Lake Pluton to the northwest. The contact of the Port Radium Formation with the granite to the south is very sharp suggesting that the magma was relatively dry. The major contribution of this granitic magma to the surrounding rocks was in the form of aplites and possibly quartz veins. The mineralization of the deposit is presumed to occur in one or more hydrothermal quartz-carbonate veins.

Great Bear Lake Mines Limited (1933-1934)

In 1933, Great Bear Lake Mines Limited owned the 'Bear' group of claims, adjacent to the El-Bonanza Mine property. The 'Bear' claims were a silver property and underground exploration of the deposit was warranted based on the interesting values encountered on the neighboring claims. Gold and some radioactive material were also

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

reported in 1933, but no economic values of uranium minerals were located. J.P. Dolan was field manager in charge of work in 1933.

In October 1933 it was reported that the company had intersected silver veins 110 feet from the portal entrance. At this point the vein had a width of 10 feet with massive mineralization, including bornite, chalcopyrite, chalcocite, and manganese (The Toronto Star, Oct. 16th 1933). The tunnel was apparently driven a total length of 125 to 130 feet, but no drifting was performed on the vein (Cummings, 1934; The Northern Miner, May 24th 1934). The company was also actively exploring their silver claims at Glacier Bay during these years (see Glacier Bay Mine) and as other properties were of more interest, work ceased at the 'Bear' claims.

Exploration Since Mine Closure

No known work.

References and Recommended Reading

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The Northern Miner newspaper articles, 1933-1934.

The Toronto Star newspaper articles, 1933.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) 086KSW0027



BEAULIEU

Minor Producer (Remediated)

Gold

62° 25' 00" 112° 54' 30"

Years of Primary Development: 1942, 1946-1948

Mine Development: 320' vertical shaft, 2 levels (681' dev.), small pits

Years of Production: 1942, 1947-1948

Mine Production: 477 tons milled = 30 oz Au

Introduction

The Beaulieu Mine is located 74 kilometers directly east of Yellowknife, NWT. All buildings at the former gold mine were destroyed in 1994 during a government cleanup effort.

Brief History

The 'Norma' claims were staked by Sam Hansen in 1939. A small mill went into operation in 1942 but work ceased for the duration of World War II. After the war, Beaulieu Yellowknife Mines Limited was formed to develop the deposit. Initial high-grade assays provided much enthusiasm and the company, spurred on by the wave of favourable publicity surrounding post-war gold potential in the Yellowknife area, championed a highly publicized exploration campaign on the claims in 1945-1946. Fantastic but erroneous and misleading ore reserves were announced and in 1947 the mine was financed into production. However, the deposit proved to be without economic value and the mine shut down in 1948, folding in chaos and bankruptcy.



Mike Piro

Figure 1. *Beaulieu Mine headframe, 1980s.*

Geology and Ore Deposits¹

The Norma gold prospect is situated in the Yellowknife Basin, a supracrustal belt within the Archean Slave structural province. Gold occurs in quartz veins hosted by turbiditic metasediments belonging to the Burwash Formation which is part of the Archean Yellowknife Supergroup. The sediments include medium-bedded greywackes rhythmically interbedded with argillite and occasional layers of grey to black, thin-bedded phyllite. The strata are strongly deformed into folds with axes plunging steeply to the northeast. Regional metamorphic grade is lower greenschist.

The Norma vein has been traced on surface for about 550 meters. It averages 15 centimeters in thickness and is concordant with bedding, following an argillite layer in the turbidites. Several minor nearby quartz veins parallel it. The veins are composed mainly of rusty weathering, grey quartz containing a variable amount of wall rock inclusions along with minor carbonate, feldspar, biotite, and chlorite. Metallic minerals constitute less than 1% of the vein by volume and include: arsenopyrite, galena, chalcopyrite, marcasite, scheelite, and native gold. Sphalerite and

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

pyrrhotite have been tentatively identified. Native gold tends to occur with galena and the other sulphide minerals, particularly near the wall rock, near inclusions of wall rock, and in chloritic slips.

Most of the hangingwall argillite has been strongly chloritized, while silicification of the footwall greywacke appears to have taken place. Also in the footwall is a 20 meter wide zone in which the sedimentary strata have been strongly contorted, broken and infilled with numerous stringers and irregular bodies of quartz.

The bulk of better grade material is present in two main shoots, named A and B. The A shoot plunges down the dip of the vein 70° to the north-northeast. At surface, the A shoot was reported to be 2.5 meters long and 50 centimeters wide with a grade of 1.64 ounce per ton gold. At the 175-foot level, it was reported to have a drift length of seven meters, width of 50 centimeters and grade of 0.55 ounce per ton gold. Shoot B was reported to be about 10 meters long, 30 centimeters thick and to contain in excess of one ounce per ton gold. Shallow diamond drilling revealed only erratic values of gold.

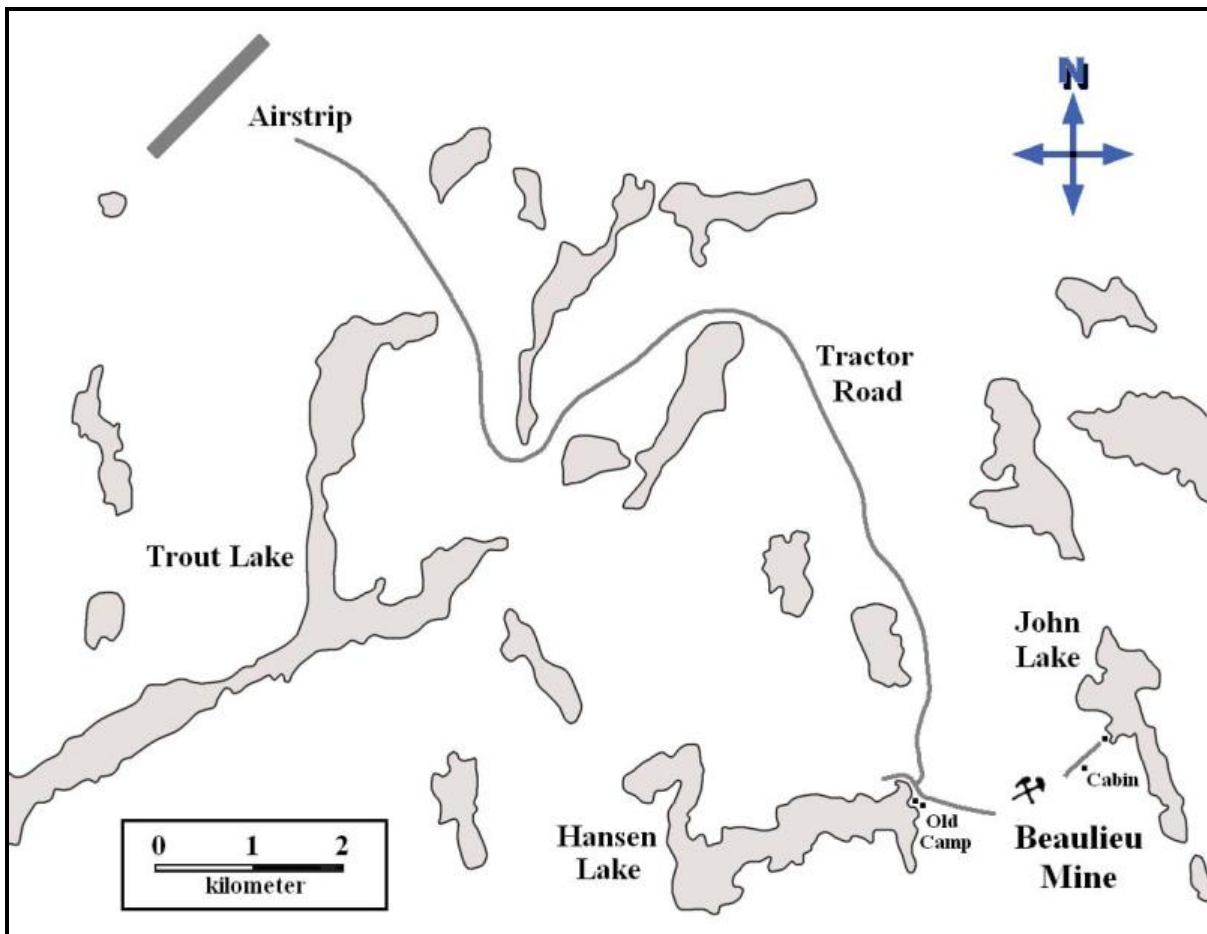


Figure 2. Beaulieu Mine property map.

Norma Tungsten and Gold Mines Limited (1942)

In July 1941, Norma Tungsten and Gold Mines Limited was formed to undertake exploratory development on the 'Norma' claims. Gold was the mineral of interest, but traces of tungsten were also reported. Sam Hansen was in charge of work, together with assistance from Harry Ingraham and Angus MacKinnon (The Yellowknife Blade, July 20th 1941). Beginning in June-July 1942, 15 tons of ore grading 4 to 5 ounces per ton gold from the Norma vein was mined by pit and processed in a small Gibson milling plant. The Norma vein had been explored over a strike length of over 1,800 feet and ore was derived from two main pits, the A and B-pits (A-zone and B-zone), the A-pit being about 18 feet deep (Messer, 1945; Lord, 1951). The mill recovery was very low because of faulty equipment, and conditions attending World War II forced a cessation of work (Lord, 1951). According to records available from the Royal Canadian Mint, a shipment from the 'Norma' claims in 1943 was refined to produce 7 ounces of gold and 2 ounces of silver (National Archives of Canada).

Beaulieu Yellowknife Mines Limited (1946-1948)

Major gold-bearing ore zones were discovered on the Giant Mine property in Yellowknife in 1944-1945, invigorating interest in new gold deposits in the Yellowknife region. Beaulieu Yellowknife Mines Limited was formed in August 1945 to acquire and explore the 'Norma' property, with Emil Schnee in charge of work. High-grade gold was discovered in the nose-fold section of the Norma vein during early work. Typically, these occurrences are very high-grade and continuous at depth. By 1946, it was felt that the drill results easily justified major developments. The drilling suggested 14,000 tons of ore grading 1.00 ounces per ton gold to a depth of 250 feet in the A-zone of the Norma vein. An ore reserve of this type was small by the standards of the day, but company officials believed that more ore would be found of equivalent grade with additional exploration. The B-zone was traced for a length of 130 feet, but more work was required. Exploration also traced the Norma vein swinging to the north as it followed a drag fold in that direction, whereas previously it was believed that the vein pinched out. (The Northern Miner, April 11th 1946) An ambitious development program, supported by the climate of the post-war gold boom, proceeded at the Beaulieu property. Assay results obtained from the vein included 17 ounces per ton gold intersections, and it was because of these high-grade assays that the company could easily promote the property.

Fraudulent Campaign

Since the money needed for major developments is raised primarily on the stock market, it is possible that the Beaulieu company seized the publicity opportunity surrounding post-war Yellowknife to raise funds for a property that was actually of little value. Ore reserves were extrapolated based on questionable diamond-drilling campaigns. Through promotional tactics, the company painted a picture to the investing public that Beaulieu was going to be the next big tonnage gold producer in the Yellowknife region. In early 1946, company president Samuel Ciglen was quoted as saying that Beaulieu Mine would be of greater importance than the gold strikes in South Africa. (The Toronto Star, June 4th 1946)

Company stock was raised to great heights in the spring of 1946 as a result of this wild promotion and the great results that they reported. The stock reached a high of \$2.60 on May 16th 1946, but then plummeted to great lows by June. This was due, perhaps, to an editorial in *The Northern Miner* newspaper which called into question the sampling methods and promotional tactics of Beaulieu Yellowknife Mines Limited (The Northern Miner, May 16th 1946). The main culprits, according to company officials, were short-selling interests who raided the market to make money on shares they didn't own, but not all the evidence backed up this theory (The Toronto Star, June 4th 1946). Rumours circulated that Ciglen and other directors of the company were in on a conspiracy. The company was hit hard with accusation of stock market manipulations, and an investigation by the Ontario Securities Commission and the Toronto Stock Exchange began in July 1946 (The Toronto Star, July 27th 1946). The investigation did find fault on the part of company officials, but there was no Canadian law to punish them (The Northern Miner, October 31st 1946) The commission recommended stronger regulations in the Toronto Stock Exchange to prevent short-selling deals. The affair did not hurt the Beaulieu company financially but it was bad for publicity. Some new directors were elected and H.G. Hutchings replaced Samuel Ciglen as president (Ciglen was 'demoted' to vice-president). Management was still determined despite the tarnish on their reputations, and they remained confident that a gold mine could be brought into production.

1946 Operations

A mining crew was hired during the summer of 1946, headed by Major Art Ames, and mining machinery was brought to the site. Company engineer Dr. A.F. Banfield recommended the sinking of a 2-compartment shaft to a depth of 300 feet to explore the A-zone at that horizon. He believed that underground exploration would be less costly than diamond drilling and provide more concrete information, because of the complex folding of the vein in this area. Underground development of the B-zone was also recommended, but only after the first stages of development in the A-zone. Banfield also recommended the installation of a 25 tons per day mill to treat high-grade ores from the A-zone for bulk-sampling purposes, so that a profit could be made while bringing the mine into full production (The Northern Miner, June 27th 1946).

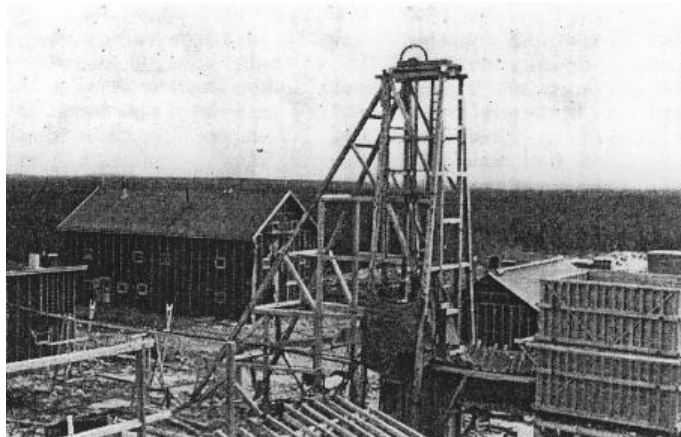
The company announced that the shaft would be sunk and a mill installed at the same time; a very unorthodox procedure. Because of the short transport season and the difficulties in getting freight up north, preordering equipment was not unusual, but it was risky since only limited ore had been outlined. A deal was made with Aerofall Mills Company for the installation of a 35 tons per day plant at the Beaulieu Mine, a unit that could be increased to 55 tons per day with minor modifications. Metallurgical testing during the fall of 1946 suggested that 76% recovery could be attained with the Aerofall mill, with tailings to be stockpiled for future cyanide treatment.

The shaft was collared in October 1946 to a depth of 28 feet and concrete was poured for the headframe, hoist, and mill buildings. No development was planned over the winter, but the company wanted to ensure that preparations for shaft sinking would be made for the spring of 1947. Diamond drilling results by this time maintained a 14,000 ton ore reserve, with the A-zone ore shoot pinching out at 300 feet depth and the widest section lying below the 150 foot horizon (The Northern Miner, June 27th 1946).

1947 Operations

Heavy freight was brought to the site by Cat train early in 1947, and a shaft-sinking contract was awarded to Frank MacKinnon. A Cummins diesel generator was installed in April 1947, purchased used from the Canol project the year previous. Much equipment was purchased used from local mining companies or from war assets, and provided for a substantial cost-saving benefit. Shaft sinking commenced in May 1947, and by the end of the month a good set of buildings, built from locally sawn and milled spruce, had been erected (The News of the North, March 28th 1947; May 30th 1947).

The 2-compartment shaft was sunk to 320 feet depth and two levels were opened up at the 175- and 300-foot levels. Lateral development thence commenced under a contract with Miners Inc. of Noranda, Quebec. Gold bearing veins were intersected on both levels late in August 1947 and the vein had a reported width of 14 feet on the 1st level. On the 2nd level intersection, it was reported that visible gold was found and that several smaller vein faces were encountered while driving towards the A-zone. This was the cause for concern earlier in exploration when diamond drilling intersected a narrow section of vein below 300 feet; now, it was faithfully believed that the Norma vein had not pinched out, and that diamond drilling had intersected these smaller veins. It was also announced that a new vein (Easton vein) had been discovered 1,900 feet west of the shaft. It was stripped by bulldozer for a length of 50 feet (The Northern Miner, Sept. 4th 1947).



The Western Miner magazine

Figure 3. *Beaulieu Mine, September 1947.*

While shaft sinking was still in progress, the company decided to clear an airstrip near the property. This decision was made when it was realized that a large amount of supplies and equipment necessary to start production by the fall of 1947 would not be available due to an early winter break-up. Therefore, using construction equipment on hand, a 4,000 foot x 200 foot airstrip was cleared on a sand-deposit about nine kilometers northwest of the mine site. A rough tractor-road was cleared from the mine to airstrip. This made it possible to employ DC-3 wheeled aircraft contracted from U.S.C.A.N. Engineering Corporation Limited for freight transport to Beaulieu (Lord, 1951). It also allowed for the targeted start-up in August 1947. This date was actually delayed two months due to problems with the installation in the mill. Also, the shaft sinking crew quit in July, and a new crew of miners was brought in to finish work and begin lateral development. A stockpile of ore was therefore not ready until sometime in September.

In September 1947, two drift faces were being advanced on both levels of the Beaulieu Mine, and raises were being driven between the 2nd and 1st levels, and the 1st level and surface. A developed ore reserve of 17,500 tons assaying 1.25 ounces per ton gold was announced. Indicated probable ore was estimated by the company at 105,000 tons of ore grading 0.87 ounces per ton gold (The Western Miner, Nov. 1947).

Production Begins

Production commenced on October 23rd 1947 after a brief trial run of muck-grade ore (Lord, 1951). A 73° raise (following the dip of the vein) being driven from the 2nd level was apparently one of the first sources of ore, although records as to what areas the company was mining are incomplete. Samples from this raise during October 1947 included 2.3 ounces per ton gold and 1.14 ounces per ton gold over 30 inches from each wall of the raise at a distance of 92 feet. A raise was also driven from the 1st level to surface during the month of October, but results of this work were not published at the time. Management reported that the raises had contributed little to giving a clearer understanding of the location of high-grade ore shoots, admitting that the underground picture was not comparable to the favourable results on surface (The Northern Miner, Nov. 13th 1947).

Milling Plant

Principle design of the mill was an amalgamation circuit with some special design features to accommodate Beaulieu ores. Ore from a 75 ton ore bin, equipped with a grizzly, was conveyed into a 7 foot x 4 foot Aerofall autogenous mill. Mill feed was regulated by an electric ear. The mill was capable of grinding 35 tons per day when using the ore itself as a grinding medium, but maximum grinding was achieved when using an appropriate ball charge (tungsten-carbide balls). The mill product was drawn out by an exhaust fan, and through a cyclone dust-collector unit. Gold-bearing dust particles were passed into a single-cell 18 inch Denver jig, and then over twelve blanket tables, each 3 foot x 8 foot, to produce a high-grade concentrate. All concentrates were treated in a 24 inch x 48 inch Denver amalgamation barrel and a gold amalgam was recovered. Fine dust from the cyclone was exhausted from the mill, and tailings were discharged to the tailings pond (Lord, 1951).

Power Plant

Electric power for the camp, hoist, and mill was provided by a 125 KVA Cummins-General Electric diesel-electric unit, and a smaller 35 KVA Waukesha-General Electric unit. Two Cat D-13,000 diesel engines operated Sullivan compressors for a total output of 900 cubic feet per minute. Fuel oil was stored in small tanks aggregating 35,000 gallons. Shaft operations were serviced by a single-drum Sullivan hoist, driven by a 30 horsepower (hp) electric motor. A low-pressure 35 hp boiler supplied heat to the mine and camp. Additional heat was supplied through a waste-heat recovery system using the powerhouse diesel engines. Fresh water was pumped from John Lake for purposes of camp consumption and for the mill. The camp maintained radio contact with the R.C.C.S. station at Yellowknife on channel CJ4E (Lord, 1951; The Western Miner, Nov. 1947).

Mining Operations

It was planned to use shrinkage stoping methods at the Beaulieu Mine, using locally cut timber as sill supports. Underground equipment included four I-R leyner drills, four I-R stoppers, three Sullivan plugger drills, two Sullivan tugger hoists, ore cars, and an Eimco mucking machine. Cars were caged and hoisted to the orebin deck, where waste and ore were separated into huge piles of rock at the back of the headframe (Lord, 1951; The Western Miner, Nov. 1947).

Crews and Camp

The Beaulieu camp was centralized at the mine itself, and included a 3-story staff house (which was also the office/warehouse), and bunkhouse/cookery. Several small cabin and tent dwellings were located a kilometer away at the Hansen Lake camp and a cottage near the John Lake pump-house served as a guest house for company officials. The staffhouse structure also provided schooling facilities for young children at Beaulieu. Lumber for practically all buildings, including the headframe, were sawn and milled on property from local stands of spruce. In August 1947, there were 45 men employed at the mine, 7 of which had families residing at the property. In September 1947, total payroll was 66, of which 6 were staff, 40 were employed on surface and construction, and 20 were underground. Mine staff included: Art Ames, mine manager; Charles Stocking, assistant manager; Ken McGinley, assayer; Frank McKinnon, mine captain; Joe Williams, surface foreman; L.E. Johnson, mill superintendent; Alf Scott, mechanic; Art Fortens, accountant; Ken McGinley, assayer; and Alf Wilmot, geologist (Lord, 1951; The Western Miner, Nov. 1947).

End of 1947 Operations

Operations at the mine were not going well during November 1947. Mill recoveries were terrible and the underground development program within the A-zone ore shoot had not uncovered anything spectacular. It was clear that the gold recovered from the mill would not pay for the capital spent on putting the property into production. To November 30th 1947, it is reported that \$795,000 had been spent on the property. This total included the expenses of supplies and equipment, construction of the mill, airstrip, and camp, and underground development to date (Lord, 1951). In order to keep the public interested in the first post-war gold producer at Yellowknife, the mine manager decided to call for a

gold pour, one month into production. The event was highly publicized and was scheduled for December 7th 1947. Production of ores stopped on November 30th 1947 when the staff had to admit defeat. The mill was not recovering enough gold. It was not the fault of the mill – there just was not enough gold in the ores. Major Ames was fired by the company and the gold pour was canceled (Price, 1967). Later in December, the Beaulieu company (under orders from the Ontario Securities Commission) hired A.D. Hellens to evaluate the deposit. He gave the final blow to the operation with his estimate of only 1,200 tons of ore reserve grading 0.63 ounces per ton gold remaining, enough for two-weeks of mill feed (Hellens, 1948). This material was above the 225-foot level; development below that depth had not outlined ore grade material (Lord, 1951).

1948 Operations

A.D. Hellens made several recommendations in his report. He recommended a limited amount of lateral development on the 1st and 2nd levels to complete development of the A-zone ore shoot, followed by milling of the remaining tonnage. He also suggested diamond drilling from the 2nd level to 600 feet depth to test the A-zone vertical extent, geological mapping of the entire property, and a thorough investigation of the Easton vein (Hellens, 1948).

Mining recommenced in January 1948 and the mill was restarted on January 20th. Development ore was being milled, but a 30-foot section of the A-shoot on the 1st level was exposed and ready for stoping operations. A sub-level was developed below the 1st level, driven off the raise. Mill heads from January 20th to February 5th 1948, inclusive, were 0.38 ounces per ton gold from 188 tons of ore milled. Milling was halted in February because of severe cold temperatures which froze up water lines. Meanwhile, surface exploration and underground diamond drilling was to continue as per A.D. Hellen's recommendations (The Northern Miner, Feb. 19th 1948). Two holes were drilled, showing the continuation of the Norma vein to the 400-foot horizon, but they failed to cut economic ore. Five x-ray holes were drilled on the Easton vein at surface, but failed to pick up the continuation of the vein. Also, geological mapping of the property failed to uncover anything of importance (The Northern Miner, Feb. 17th 1949).

The mill was brought back on line in August of 1948 for a short period while mining the A-shoot, but all work ceased on October 1st 1948 with the advent of winter. 10 men were employed in August 1948. Samuel Ciglen, company president, reported that it was planned to reactivate the property in 1949 to mine the B and Easton-shoots (The Northern Miner, Sept. 2nd 1948; Feb. 17th 1949). The property was abandoned in 1949 and most of the mill equipment was removed. Production to 1948 and mine development as of December 31st 1947 are given in **Tables 1 and 2**, and a plan of the underground workings is shown in **Figure 4**.

Level:	Drifting:	Crosscutting:
1 st (175')	151'	162'
2 nd (300')	274'	94'

Table 1. Beaulieu Mine underground development to December 31st 1947. (source: Lord, 1951)

Year:	Ore Milled:	Rough Gold Produced:	Royal Canadian Mint Recovery:
1942	15 tons @ 5.00 oz/ton	?	7 oz Au, 2 oz Ag
1947	252 tons @ 0.03 oz/ton	7.5 oz	23 oz Au, 4 oz Ag
1948	210 tons @ 0.298 oz/ton	41 oz	
Total:	477 tons	50 oz	30 oz Au, 6 oz Ag

Table 2. Beaulieu Mine production. (source: Lord, 1951; National Archives of Canada)

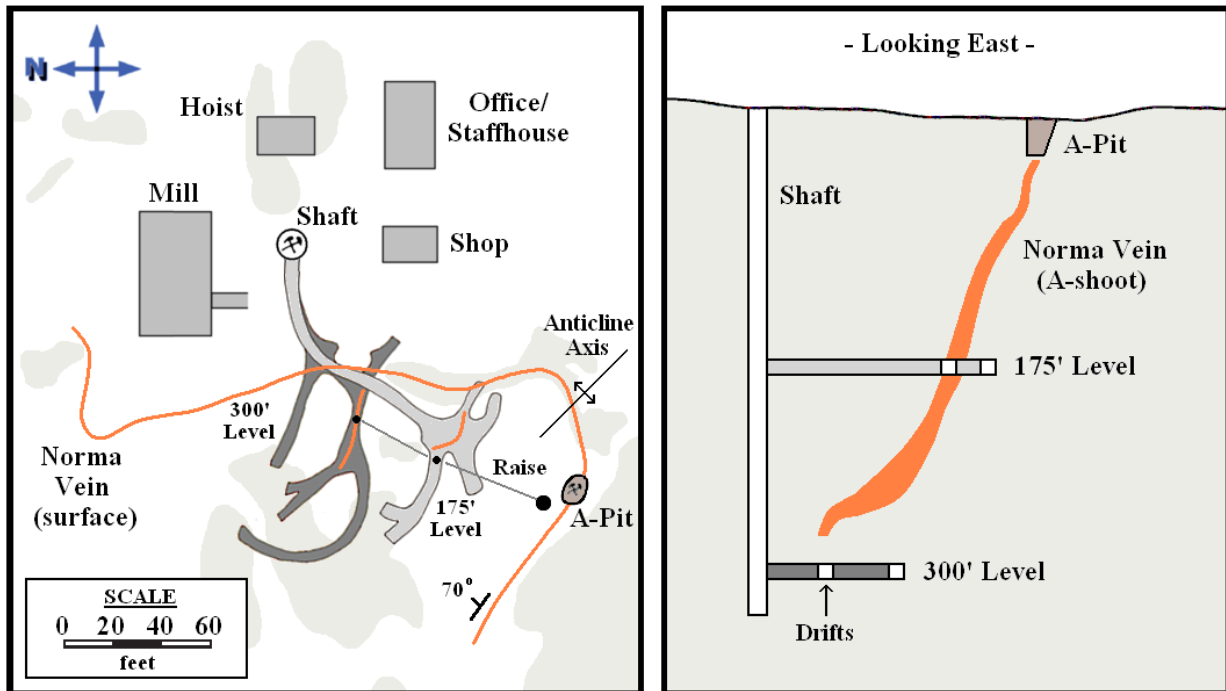


Figure 4. Surface plan of the mine site area and underground plan, 1947.

Exploration Since Mine Closure

The property remained dormant until 1983 when the 'Brandy' claims were staked by Genesis Resources Corporation Limited. In 1984, two holes were diamond drilled and in 1985 the property was geologically mapped and chip sampled. Magnetometer and VLF-EM surveys were also conducted and three anomalous gold assays were recorded, but in 1989 the claims were allowed to lapse (Cremonese, 1984). Robert Carroll staked the 'Irene' claim in 1992 over the old mine, but no work was done and the claim lapsed in 2000 (Robert Carroll).

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0017

Personal communication: John Clark (Vista Engineering Limited); Robert Carroll; Mike Piro



BETA GAMMA

Bulk Sampled (Abandoned)

Uranium

64° 44' 00" 118° 11' 00"

Years of Primary Development: 1934, 1956

Mine Development: 65' vertical shaft + adit tunnel (992' dev.)

Years of Bulk Sampling: 1934

Bulk Sample: 1 ½ tons cobbled ore shipped @ 41% U₃O₈

Introduction

The Beta Gamma property is located at Beaverlodge Lake, 316 kilometers northwest of Yellowknife, NWT. It was an underground uranium prospect in 1934 and 1956. The site is only accessible by floatplane, although an old winter road to Great Bear Lake once passed through the area. It was visited by the author in August 2006. A forest fire destroyed all buildings some time ago.

Brief History

The first work was done in the 1934 when a shaft was sunk on the 'Tatee' and 'Bee' claims. About 1½ tons of cobbled pitchblende ore was sent for assaying and was found to contain 41% uranium oxide. Work stopped after finishing the shaft and the claims were dropped. In 1943, the property was re-staked as the 'Cormac' claims by DeStaffany Tungsten Gold Mines Limited, who later reformed into Transverse Longlac Mines Limited. During the 1950s, the claims were surveyed and additional uranium occurrences noted. The property was enlarged and a program of drilling was conducted. In 1956 underground work resumed by Consolidated Beta Gamma Mines Limited through the use of a long adit. No major mining development has been done since 1956.

Geology and Ore Deposits¹

The geology of the area consists of a belt up to 1.5 kilometers wide consisting of northeast-trending Proterozoic Snare Group interbedded sediments, volcanics, and quartz-feldspar porphyries that form a northeast-trending ridge crossing the mine site. Quartzite, with thin beds of conglomerate and slate or talc-sericite schist striking northeast and dipping 75 to 80° northwest, outcrops along the ridge. It is bound on the southeast by the Beaverlodge thrust fault, where altered, fine-grained porphyry overlain by a massive dacite flow is in contact with the quartzite, and in the northwest by granitic intrusions. Granitic intrusions flank the belt and are cut by white, barren or hematite-bearing, quartz veins. The granite is thought to underlie most of the strata north of the Beaverlodge Thrust, and borders the exposed sedimentary and volcanic rocks to the south. At the contact, the porphyry is brecciated, a 1 to 4 inch gouge, and quartz veins in either unit terminate at the contact. The northeast corner of the property is underlain by a folded, cherty argillite, and a massive, medium-grained, altered and fractured gabbroic body is found east of the main mineralized area. Vertical diabase dykes also intrude the strata. Pitchblende is deposited in open cavities and fractures that are spatially related to the regional fault system. Four main deposits of this pitchblende have been developed. Pitchblende is closely associated with hematite, often with chlorite, and with minor cobaltite.

Hottah Lake Mines Limited (1934)

The property was staked as the 'Tatee' and 'Bee' claims in January 1934 by Darcy Arden Sr. following the identification of brown surface oxide staining in this area. Edward Hargreaves of Great Bear Developments Limited optioned the claims in March 1934 after finding a lens of solid pitchblende. The break was said to be traceable for over 15 kilometers. The discovery was widely acclaimed and promoted following its discovery, and Hargreaves credited the property as one of the richest radium finds in the region (Lord, 1941). Great Bear Developments Limited formed a new public company in order to raise funds for a mine. In April 1934, Hottah Lake Mines Limited was incorporated to development the property. A six-man crew began work in April with the construction of a camp. Trenching of the showings was then undertaken. On the #3 deposit (the original discovery, including the #1 and #2 veins), two pits were dug. One pit was 6 feet deep from which 13 bags of ore were mined. The second pit was on the #2 vein, which was discovered in May 1934, parallel to the #1 vein and 125 feet south. The pit was sunk 8 feet and 18 bags of ore were mined. Samples were sent to the Eldorado Mine on Great Bear Lake, assaying 48.4% uranium oxide (U₃O₈) (Hargreaves, 1934).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

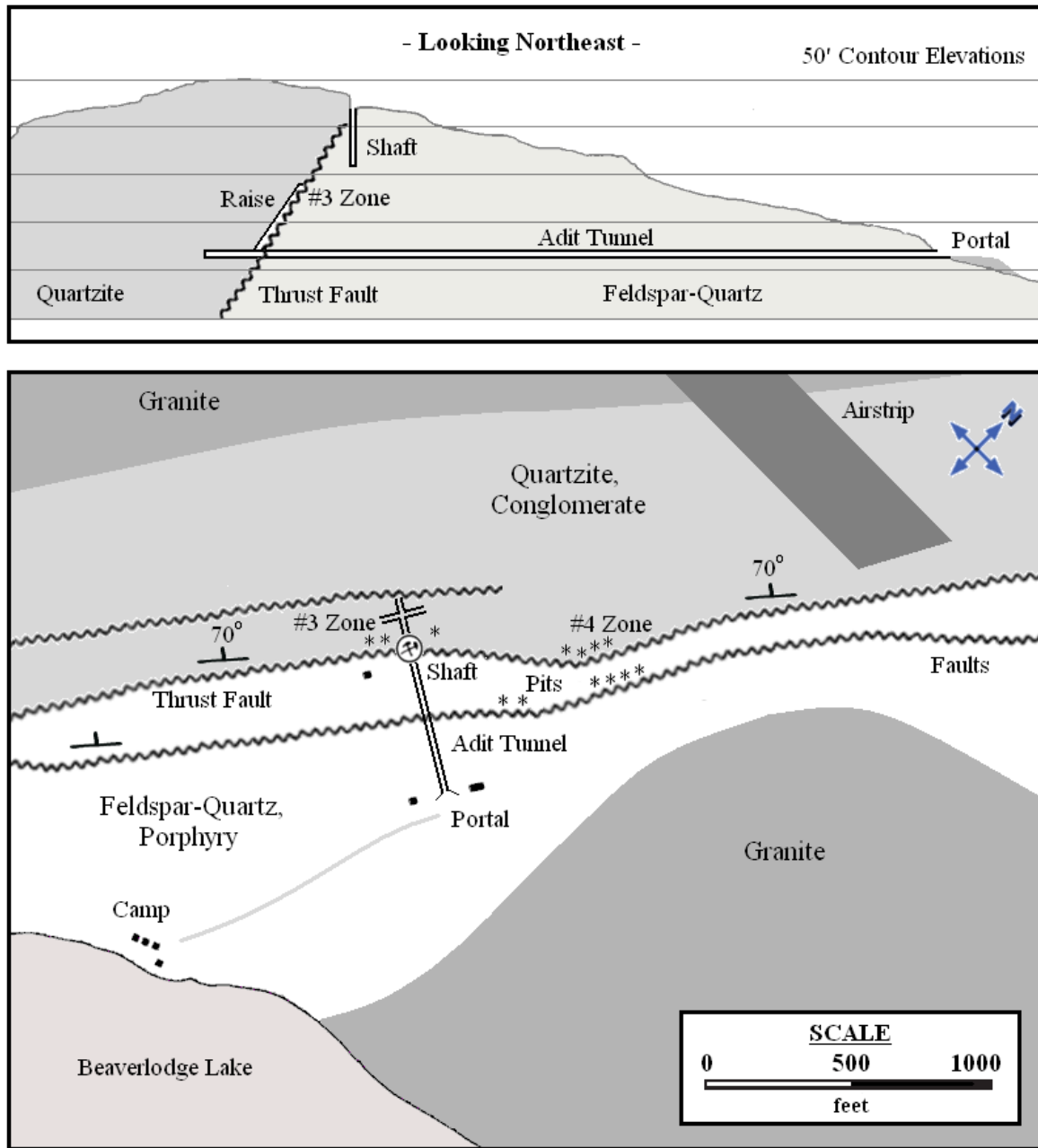


Figure 1. Surface and underground plan of the Beta Gamma Mine, c.1956.

Cobbed Shipment

About 1½ tons of ore was removed from pits on the #3 and #4 deposits and shipped south in June 1934. Tests found these to contain 41% U_3O_8 (McDonald, 1943). Fifty additional tons were stockpiled on the property in August 1934. Other small samples were tested for radium content. One of these was assayed at the University of Alberta Geology Department, indicating 50 milligrams of radium per ton.

The Hottah Lake company made a tentative deal with Eldorado Gold Mines Limited in which ores containing over 50% U_3O_8 would be purchased by Eldorado. The company made arrangements to sink a shaft on the #2 vein in order to obtain information about the persistence of the deposit's grade. Unfavourable weather conditions and freighting problems created delays in getting supplies and equipment to the property. During July-August 1934, a 65 foot shaft (6 feet x 8 feet) was sunk on the #2 vein and a 60 foot crosscut (5 feet x 7 feet) was excavated (N.W.T. Geoscience Office Assessment Report #015222). A small headframe was built, and rock was hoisted by an ore bucket made from an oil drum, and windlass. It was then dumped by ore car southeast of the shaft (site evidence). The ore left the shaft

within the first 10 feet of excavation, and there was difficulty bagging ore for shipment because of the lateness of the season. Lack of supplies for the winter season and the difficulty of the company to raise the needed funds to continue exploration resulted in a shutdown of operations late in 1934 (Hottah Lake Mines Ltd., 1934). The company went bankrupt early in 1935 and despite the effort to refinance and renew operations, the mine did not reopen at this time. It is reported that no ore was found in the shaft and that the discovery was too small to be of economic value (Lord, 1941).

Consolidated Beta Gamma Mines Limited (1956)

In 1956, underground development resumed at the long dormant uranium property. The 1950s uranium boom resulted in the re-evaluation of several deposits in the Northwest Territories. A newly formed company called Consolidated Beta Gamma Mines Limited began work on an adit that would crosscut into the #3 deposit under the old shaft at a vertical depth of 150 feet. A rough airstrip was cleared to handle large Bristol aircraft. The power plant included a Cat D-13,000 diesel engine driving a 365 cubic feet per minute Gardner-Denver air compressor, powering two Copco rock drills for adit blasting.

Development completed to December 6th 1956 consisted of a 750 foot adit driven westerly into the contact zone, followed by two drifts, one 158 feet north and one 84 feet south along the vein. A raise designed to intersect the old shaft was advanced 90 feet from the end of the crosscut, but did not reach its objective, although the face remained in ore. Additional equipment was purchased including units for a milling plant, but a production decision was deferred pending a review of work done. John H. Parker and Norman W. Byrne were the consulting engineers during this program. The claims were allowed to lapse in 1965 or 1966 and no major mining development has been undertaken since (Baykal, 1967; McGlynn, 1971).

Exploration Since Mine Closure

In 1966, the 'Tin' and 'Atom' claims were staked by M. McGuire, and in 1967 Syracuse Oils Limited optioned the property to conduct some exploration. The property was again re-staked in 1970 as the 'Joe' group of claims for Nemco Explorations Limited, who in 1974 conducted a ground scintillometer survey that failed to outline any new ore zones. In 1975, New Pyramid Gold Mines Inc. optioned the claims and initiated diamond drilling, surveying, and geophysical and geochemical work. Work was focused on the two high-grade lenses previously explored. Diamond drilling consisted of 14 holes (3,633 feet total) and failed to find any extensions to the zones or intersect significant mineralization. The two lenses were reported to contain 200 tons of material grading 3% U₃O₈ (Laporte et al., 1978).

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BLANCHET ISLAND

High-Graded (Abandoned)

Cobalt, Nickel

61° 59' 45" 112° 23' 45"

Years of Primary Development: 1969-1970

Mine Development: 75' adit tunnel, open cuts

Years of High-Grading: 1969-1970

High-Grading: 330 tons high-grade ore shipped @ 15 % Ni + 11% Co

Introduction

This site is located 115 kilometers southeast of Yellowknife, N.W.T. on the southern shore of Blanchet Island. It briefly operated as a high-grading cobalt and nickel operation in 1969-1970.

History in Brief

Occurrences of cobalt and nickel on Blanchet Island were noted by Alfred V. Giauque in 1968 and the 'Lux' claims were staked. In 1969, the claims were optioned to Jason Explorers Limited, who staked additional claims adjoining the original showing. High grading of cobalt-nickel ores was undertaken in 1969-1970. Pits were blasted and a short adit was driven into the contact zone. Small amounts of ore were shipped to France for processing. No commercially economic deposit was uncovered at the time.

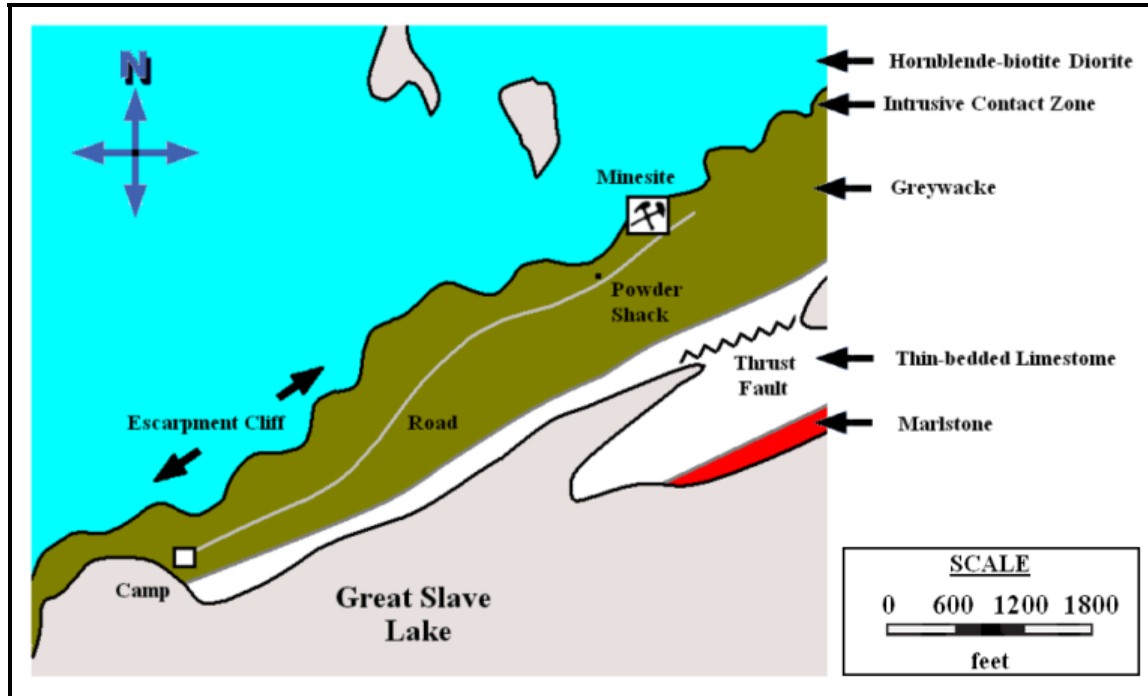


Figure 1. Blanchet Island Mine geology and surface plan.

Geology and Ore Deposits¹

Blanchet Island lies at the western end of the East Arm of Great Slave Lake in a graben of little deformed and metamorphosed Archean sedimentary rocks and volcanics. These rocks have undergone thrusting and later intrusion by more than twenty plugs and laccoliths of diorite-monzonite composition scattered throughout the belt. The showings (cobalt-nickel arsenide) that occur on Blanchet Island generally are found at the brecciated contact between the sediments and the overlying diorites. A prominent escarpment up to 90 meters above lake level runs northeasterly through the area. This contains the contact between the diorite and sediments, which undulates up and down the escarpment with a wavelength of about 100 meters and an amplitude of approximately 30 meters. The showings occur in the dolomite/limestone breccias within five meters of the contact with the diorite. Massive cobalt-nickel

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

mineralization appears to be confined to the breccia zone, in areas not replaced with magnetite. Often only minor veinlets are found between breccia clasts, but elsewhere matrix fillings of 30% to 60% coarse dolomite or grey cobalt-nickel arsenide are found. The mined vein was reported as 30 to 35 feet long and 6 inches to 5 feet wide, with an ore shoot 10 to 15 feet long averaging 4 feet in width. The vein has a northeast strike and a dip of 15 to 40° to the northwest.

Jason Explorers Limited (1969-1970)

An open-cut was blasted on the showing early in 1969 by crews working for Jason Explorers Limited. Jim D. Mason was in charge of the work. By October 1969 the crews had mined and bagged about 300 tons of high-grade arsenide ore. In September, crews encountered radioactive mineralization in the open cut (Kelly, 1969b). An agreement was signed for the sale of ores to Ugine-Kuhlmann SA of Paris, France, and in October the first shipment of 86 tons of ore was made (Kelly, 1969c). This shipment of ore had considerably good values and the smelter reported assays of 11% cobalt and 12% nickel (Cabrol, 1971). Surface exploration in 1969 totaled 627 feet of diamond drilling on the LUX claims, plus trenching on the LUX and DL claims. The DL claim was staked by David Lent adjacent to the east of the main property and optioned to Jason Explorers in August 1969.

In November or December 1969, an adit was started at the face of the open cut, because the dip of the oreshoot precluded further pit mining. This was because a 5 to 6 foot lift would be required to reduce the elevation of the pit floor to bring the oreshoot back to workable elevation on the face. Hence a five foot wide trench was excavated on the floor of the pit up to the working face and an adit was driven into the face as an extension of this trench (Kelly, 1969d). Total length of this adit was reported to be 75 feet, with dimensions of 5 feet width and 6 feet height. The open cut 30 feet long, 20 feet wide, and 18 feet deep. A rail cart was used to dump ore from the adit down a 60 foot steel-tube “chute” and into barrels (Lecouteur, 1988). A high-line system was also in use at one point to zip sacks of ore to the valley below.

Jason Explorers continued exploring the property in 1970 as part of a joint-venture agreement with Ugine-Kuhlmann. This agreement was signed in May 1970 and called for an investment of \$132,000 for the 1970 summer program, \$50,000 of which was provided initially by the Ugine-Kuhlmann company, with the remaining \$82,000 to be shared 50-50. (Canadian Financial Journal, May 12th 1970) Exploration continued and 1,983 feet of diamond drilling was completed in 1970 on the main zone, 228 feet on the West zone, and 543 feet on the DL claims. (NWT Geoscience Office Assessment Report #015063)

Underground mining appears to have ceased in 1970. The adit work was not able to find additional mineralization and it appeared that the lens was mined out. Two men were still on property extracting cobalt ore in January 1970 (Kelly, 1970). A second shipment of 243 tons of ore was sent to France in November 1970. This batch was not as high-grade, with cobalt assaying at 5%, and nickel at a higher 15% (Cabrol, 1971). Thirty tons of ore in forty 45-gallon barrels were left on the shore and not shipped at that time (Murphy, 1971; Lecouteur, 1988).

Exploration Since Mine Closure

Dave Smith restaked the property circa 1980 and collected eight tons of stockpiled ore, grading 6.0% nickel-cobalt, and shipped them out for treatment. In 1984, Highwood Resources Limited staked the ‘HRL’ claims on Blanchet Island covering the known areas of mineralization. The showings were examined and sampled in 1987 (Lecouteur, 1988).

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BONANZA

Advanced Exploration (Abandoned)

Silver

66° 00' 30" 118° 05' 45"

Years of Primary Development: 1937-1938

Mine Development: 100' vertical shaft, one level (300' dev.)

Introduction

The Bonanza Mine is located in the Echo Bay area of Great Bear Lake, nine kilometers south of LaBine Point (Port Radium) on the west side of Miles Lake. It is 434 kilometers northwest of Yellowknife, NWT. The property was visited in July 2005 by the author. The old headframe is the only building left.

History in Brief

The 'Bonanza' claims were staked by Spud Arsenault in 1931 for Eldorado Gold Mines Limited to cover what was originally quite a spectacular silver vein. In 1934 the eastern half of the claims were sold to El-Bonanza Mining Corporation Limited (see El-Bonanza Mine); the western half occupying the original discovery was retained by Eldorado. Eldorado sank a short shaft on the vein in 1937-1938 to explore the silver potential. A significant drop in the price of silver halted all work on this property and no production was attained. The area was later re-staked by Hugh Arden in 1982 and some surface exploration was conducted.

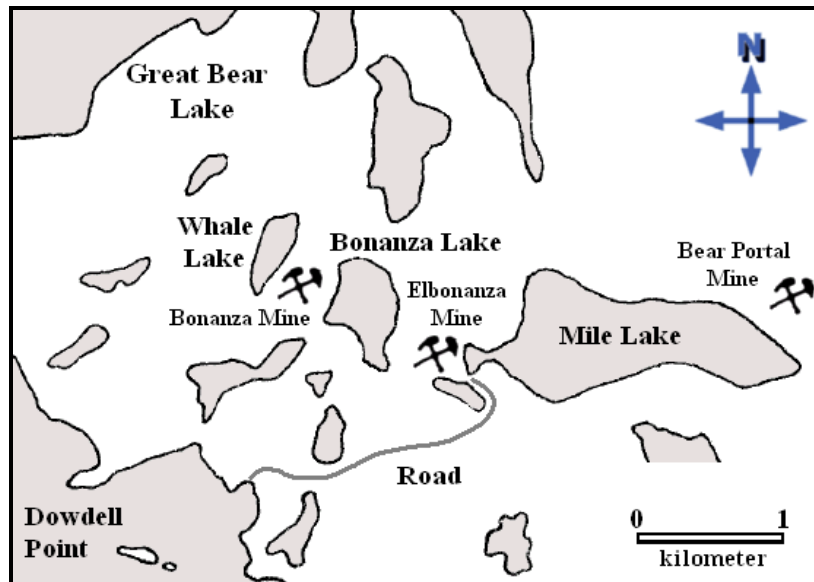


Figure 1. Dowdell Point area, Great Bear Lake, showing the location of Bonanza Mine.

Geology and Ore Deposits¹

The mine is situated within the Great Bear Magmatic zone, a part of the Bear Structural Province. The oldest rocks in the area belong to the Port Radium Formation and the Lower Echo Bay Formation, both part of the Early Proterozoic (1.87) Ga LaBine Group. The Bonanza deposit is hosted within a narrow, northwest-southeast striking, 10 meter to 100 meter wide band of fine grained, thin-bedded quartzites, dipping steeply to the northeast. The altered zone is irregular, but follows the general strike of the host rocks and coincides closely with the silver mineralization. The altered zone shows a variably developed schistosity, which is also conformable with the bedding of the sediments.

The silver mineralization is confined to shears and fractures in a chloritic alteration zone and also to carbonate veinlets cross cutting the chloritized sediments. The zone consists dominantly of chlorite with calcite stockworks and specular and massive hematite. Minerals identified at the showing to date are: native silver in wires, chalcopryrite, pyrite, pyrrhotite, magnetite, and galena, as well as traces of erythrite and malachite. Evidence exists that the

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

fractures and shears have been reopened several times and that there were several stages of mineralization. Early minerals were brecciated and then cemented by successive mineralizing solutions.

Eldorado Gold Mines Limited (1937-1938)

Crews and equipment from the nearby Eldorado Mine were mobilized in August 1937 to start a shaft on the Bonanza property. An 80-hp diesel power plant was installed and a headframe was erected and closed in for winter operations. A road was cleared to Dowdell Point and other buildings erected included a powerhouse (annexed to the headframe), blacksmith shop, and bunkhouse/cookhouse.



Ryan Silke

Figure 2. Bonanza Mine headframe, July 2005.

The one-compartment shaft had reached a depth of 25 feet by November 1937 when work temporarily ceased due to mechanical problems with the diesel power plant. Crews had to wait for additional supplies and equipment that could only be flown in after the freeze-up period. Twenty men were employed in November 1937 (The Northern Miner, Dec. 16th 1937; Jan. 6th 1938).

By March 1938 the shaft had reached a depth of 100 feet and the 1st level was advanced into the silver vein. Some 300 feet of lateral advance was performed when operations ceased in April 1938. Crews were brought back to Eldorado Mine to concentrate on radium and uranium mining operations there. Underground work at the Bonanza Mine in 1937-1938 on the 100-foot level showed the existence of a strong sheared and altered zone carrying disseminated silver values.

Silver grades were not encouraging. The drop in the price of silver on international markets and the need for Eldorado Gold Mines Limited to focus on pitchblende mining at its main Eldorado Mine resulted in the cessation of silver exploration at Great Bear Lake and at the Bonanza property (The Northern Miner, Apr. 1st 1938).

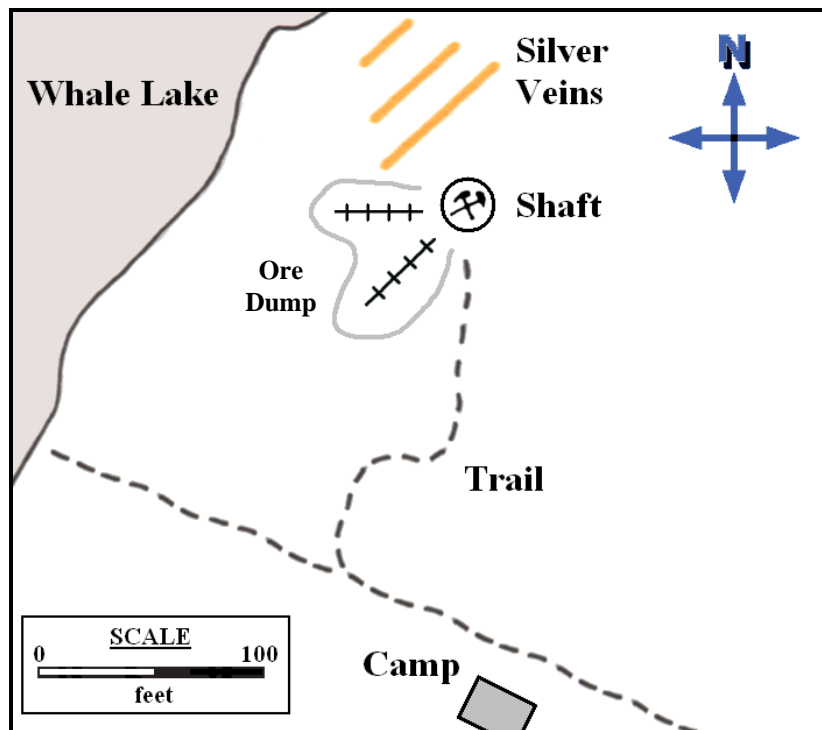


Figure 3. Surface plan of Bonanza Mine shaft area.

Exploration Since Mine Closure

No work was conducted until 1982 when Hugh Arden and partners re-staked the property as the 'El Bonanza' claim. In 1984 the property was optioned to O.P. Resources Limited. Work carried out by this group included prospecting as well as detailed magnetic, VLF-EM, and geological surveys in the area of the old Bonanza Mine. This exploration confirmed the existence of high-grade silver in a zone of strong chloritic alteration over a length of 300 feet and 35 foot width (Magrum, 1988). In 1988, the property was optioned to Octan Resources Incorporated. During the year, 15 holes (3,661 feet) were drilled on the property to investigate the extent of alteration and mineralization on the property. Additional magnetic and VLF-EM surveys were conducted outlining a magnetic horizon associated with the alteration zone that continued northwest under Whale Lake. A 1,500 pound bulk sample (presumably from trenching) was removed, assaying 352 ounces per ton silver (Hoefler and Magrum, 1988). Octan Resources went defunct in 1990 and the property reverted back to Hugh Arden. So far as is known, no further work has been done on the claims.

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BULLMOOSE LAKE

Producer (Remediated)

Gold

62° 20' 30" 112° 44' 45"

Years of Primary Development: 1941, 1975-1976, 1983-1987

Mine Development: 56' incline shaft + main decline to 700' depth (7 levels), secondary decline to 100' depth

Years of Bulk Sampling: 1941

Bulk Sample: 14 tons shipped = 228 oz Au

Years of Production: 1986-1987

Mine Production: 74,600 tons milled = 503 kg Au

Introduction

The Bullmoose Mine is located 84 kilometers east of Yellowknife, NWT. between Campbell and Buckham Lakes. Although it was open for a short time as an underground prospect in the 1940s, the main development period for this property was in the 1980s when a small mill went into production. The site was remediated and all buildings removed following 1987 closure. The site was viewed from the air in 2000.

History in Brief

The area around Bullmoose Lake was staked as the 'TA' claims in 1939 by Spud Arsenault and C.S. McDonald for Cominco Limited. The amount of gold found on the claims warranted a small mining operation consisting of the sinking of a shaft and the shipment of a bulk sample of ore. By 1961 only half of the original claim group remained in good standing, and these were acquired by William McDonald. The property was acquired by Duke Mining Limited in 1967.

The next period of mining work started in 1975 when Terra Mines Limited, in partnership with Duke Mining Limited, drove a decline into the gold deposit. Low gold prices kept the project mothballed until 1983 when develop resumed, and in 1986 the mine was placed into production. This mining and milling was not entirely successful and in early 1987 Terra Mines Limited shut the operation down and sold the property for \$40,000.

Geology and Ore Deposits¹

The Bullmoose gold deposit is situated in the Yellowknife Basin, a supracrustal belt within the Archean Slave structural province. Gold occurs in quartz veins hosted by turbiditic metasediments belonging to the Burwash Formation, which is part of the Archean Yellowknife Supergroup. The sediments include mainly medium-bedded greywackes rhythmically interbedded with argillite. Both the bedding and an early, bedding-parallel cleavage are isoclinally folded. The major fold structure is a shallowly north plunging, north to northwest trending syncline. A later cleavage consisting of quartz-filled fractures strikes parallel to the axial trace of the syncline.

Auriferous, bedding-parallel quartz veins are associated with the nose of an open fold on the west limb of the syncline. The major quartz veins, #1 - 4, contain pyrite, arsenopyrite, scheelite and gold. Sulphides present in other veins in the system also include any or all of pyrrhotite, chalcopyrite, sphalerite, galena and molybdenite. Gold is unevenly distributed and tends to be associated with pyrite and/or pyrrhotite, particularly near the contacts between the quartz veins and host sediments. Gold is generally found as fracture fillings and disseminations within the veins. Ore shoots plunge 60° to 80° north with strike lengths ranging up to 40 meters and averaging 15 meters. Drilling has defined gold mineralization to a depth of at least 400 meters.

Cominco Limited (1941)

In 1940-1941, Cominco crews explored the 'TA' claims by conducting trenching and diamond drilling 17 holes. They also sank a short inclined (70° to the northeast) two-compartment shaft on the #4 vein a length of 56 feet. About 92 feet of drifting was performed at a depth of 50 feet (N.W.T. Geoscience Office Assessment Report #015069). Apparently this shaft was sunk entirely by handsteel without the aid of power tools or a hoisting plant, which was not uncommon for this kind of limited development (Lord, 1951). A bulk sample of high-grade ore, from an open cut on the #1 vein, was also shipped from the property in the spring of 1941. Records found at Con Mine report that 14 tons of ore grading 16.8 ounce per ton gold were treated at the Con mill in March and April 1941 to produce 228 ounces of gold and 58 ounces of silver (Cominco Ltd., 1941). Cominco crews built a small log cabin camp near Bullmoose Lake, the ruins of which are believed to still be there today.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

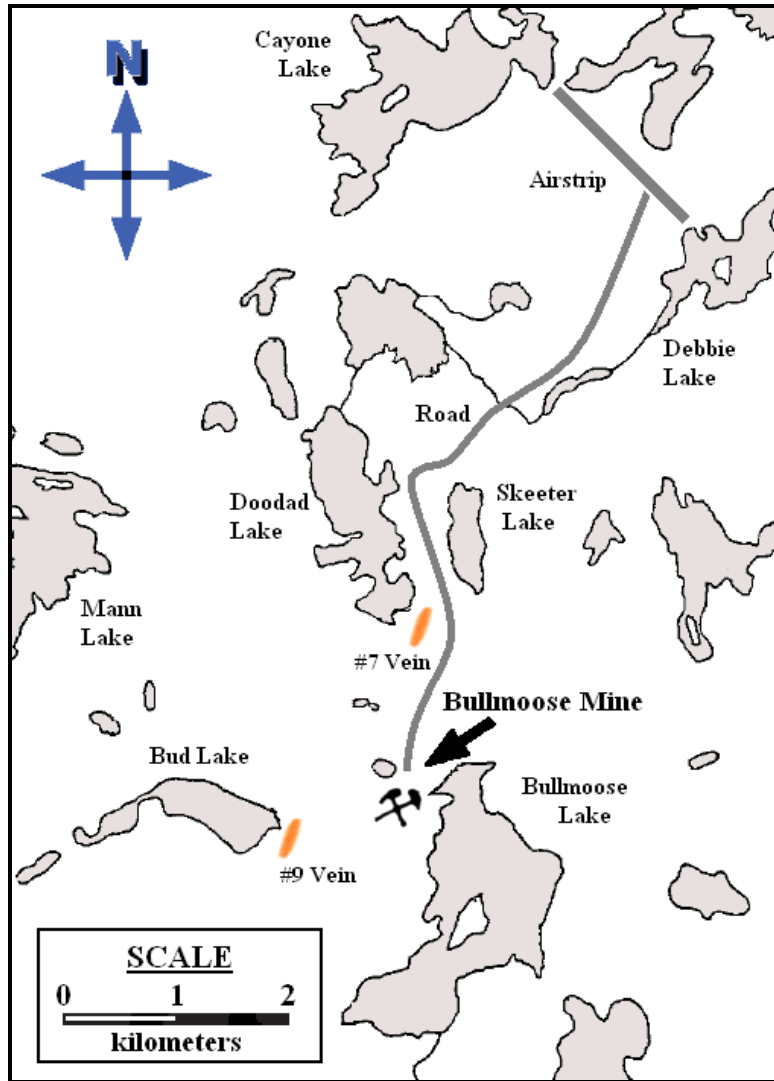


Figure 1. Map of the Bullmoose Lake Mine area.

Terra Mines Limited / Duke Mining Limited (1975-1976)

In December 1972, Terra Mines Limited signed a joint-venture agreement with Duke Mining Limited for the development of the Bullmoose property. Underground work was authorized, and equipment and supplies were mobilized to the site over winter road early in 1975. During the summer of 1975, an 8 foot x 8 foot wide decline was driven a length of 500 feet. At the 50-foot level, lateral work was performed on the #2 vein, where 390 feet of drifting and 110 feet of raising were completed. Another 220 feet of drifting was performed to intersect the #1 vein, but this work failed to locate a significant gold deposit. A total of 1,281 feet of underground diamond drilling in 9 holes was also completed during the 1975 program. 1,430 tons of ore grading 0.31 ounces per ton gold were stockpiled from stoping operations in the #2 vein. This material, plus the development ore removed, resulted in a total stockpile of 2,017 tons grading 0.33 ounces per ton gold at the end of 1975. Work ceased for the winter season and to await the transportation of additional equipment for an expanded underground program in 1976 (Mitchell, 1976; Laporte et al., 1978; Terra Mining & Exploration Ltd. Annual Report, 1975).

The #4 vein was the focus of exploration in 1976. An extensive development program was planned, including drifting, raising, and stoping on the #4 vein and parallel systems in the West zone. More equipment and supplies were marshaled to the property via winter road in early 1976. Underground development was scheduled to commence in May 1976, but a series of mechanical breakdowns delayed work until June and the program was not completed until well into the winter freezeup (October 24th 1976). As a result of the delays, not all the recommended work was completed. Drifting and crosscutting within the West zones and other parallel veins and raising on two ore shoots within the #4 vein was not completed. However, the following underground development was reported during 1976:

913 feet of 9 feet x 11 feet wide decline plus the slashing and widening of the 1975 decline, 70 feet of crosscutting, and 466 feet of drifting, all within the #4 vein at 220-foot depth. Three mineralized sections were encountered and found to be highly erratic, but very high-grade in sections. 320 tons of drift muck grading 0-19 ounces per ton gold were removed and stockpiled to add to the 1975 stockpile. A 350 pound bulk sample was sent to Calgary for metallurgical tests (Mitchell, 1976; Terra Mining & Exploration Ltd. Annual Report, 1976).

1976 Equipment, Camp, and Employees

During 1976, seven employees were working at the Bullmoose Mine. A. Montgomery was manager, M.A. Mitchell was geologist, and there were three miners, one surface labourer, and one cook. The camp consisted of plywood buildings and portable trailers (kitchen, mine dry, and three bunkhouses). Mining equipment included two Wagner scooptrams, an Atlas-Copco 600 cubic feet per minute air compressor, a D.H. Crawler 100C loader, and a 25 KVA Deutz genset (Mitchell, 1976).

Summary of Underground Work 1975-1976

During the development program of 1975-1976, 1,400 feet of decline was driven. On the 50-foot level, crosscuts were extended to access the #1 and #2 veins and a total of 610 feet of lateral work was performed on these veins in 1975. 110 feet of raising was also performed within the #2 vein with two surface breakthroughs. On the 220-foot level, the #4 vein was probed by 536 feet of lateral work during 1976. Total lateral development during 1975-1976 on both levels was 1,146 feet of drift and crosscuts. It was planned to resume operations in 1977 and complete the development program on the #4 vein, but a lack of funds halted work at the end of 1976. Terra Mines Limited earned a 50% interest in the project as a result of this work.

Terra Mines Limited (1983-1987)

Terra Mining & Exploration Limited re-initiated exploration at the Bullmoose property in 1981, consisting of a diamond drill program to a depth of 1,000 feet. A drill-indicated ore reserve of 25,000 tonnes of ore grading 0-34 ounces per ton gold to a depth of 210 feet was reported. (Terra Mining & Exploration Ltd. Annual Report, 1981). In November 1981, Terra acquired control of Duke Mining Limited gaining 100% interest in the project. The name of the company was changed to Terra Mines Limited. Extensive diamond drilling was accomplished during 1983 and 11 new veins were discovered, bringing total gold deposits to 15 by the end of the year. The #11 zone was discovered while scouting for a possible location for a mill, and drilling proved the depth continuation of the #4 vein and #11 zone orebodies to 1,000 feet depth (Terra Mines Ltd. Annual Report, 1983).

An underground program designed to extend the decline past the 220-foot level was begun in April 1983. By year-end 1983, the decline had been driven an additional 1,270 feet to the 350-foot level. The 200- and 350-foot levels were developed through 1,635 feet of advance by year-end. Several raises encountered spectacular gold showings. Almost all work during 1983 was done on the 300-foot level on the #4 zone. Drilling early in the year encountered grades of 0-60 ounces per ton gold over a 4 foot mining width on the 600-foot level. A 64 kilometer winter road was established from Reid Lake on the Ingraham Trail, east into the property via Harding-Hearne and Campbell Lakes. Complete camp and plant facilities were brought to the site over this route (Terra Mines Ltd. Annual Report, 1983).

Mining Equipment

Power was supplied to the operation by a 125 kilowatt Cat D-3306 diesel generator and a smaller 40-KVA Deutz generator. Air was supplied by two 600 cubic feet per minute Atlas-Copco air compressors, and mining development was performed using Copco drills, and Jarvis-Clark scooptrams. A Tamrock single-boom Jumbo drill and two JDT 15-ton ore trucks were later added to the mining fleet. In 1984, power generation was increased to 400 kilowatts by the addition of a 250 kilowatt generator unit (Terra Mines Ltd. Annual Report, 1984). Later in 1985, a single Cat D-379 diesel unit supplied 400 kilowatts of power with three other units kept as standby. A third Atlas-Copco compressor was also brought to Bullmoose. It was estimated that the mine would need an input of 2,800 cubic feet per minute when mill operations begin. Fuel was stored in tanks totaling 185,000 gallons.

The decline was advanced during 1984 another 780 feet and the opening enlarged to 11 feet x 15 feet dimensions to allow for larger mining equipment underground. This slashing work equaled 562 feet of development footage. Other development work per year up to 1987 is given in **Table 2**. Most work done in 1984 was on the 200- and 300-foot levels, although a major program involving drifting northeast into the #11 zone on the 150-foot level was completed. Surface exploration led to the discovery of the #7, #9, and #12 veins during 1984 (Terra Mines Ltd. Annual Report, 1984).

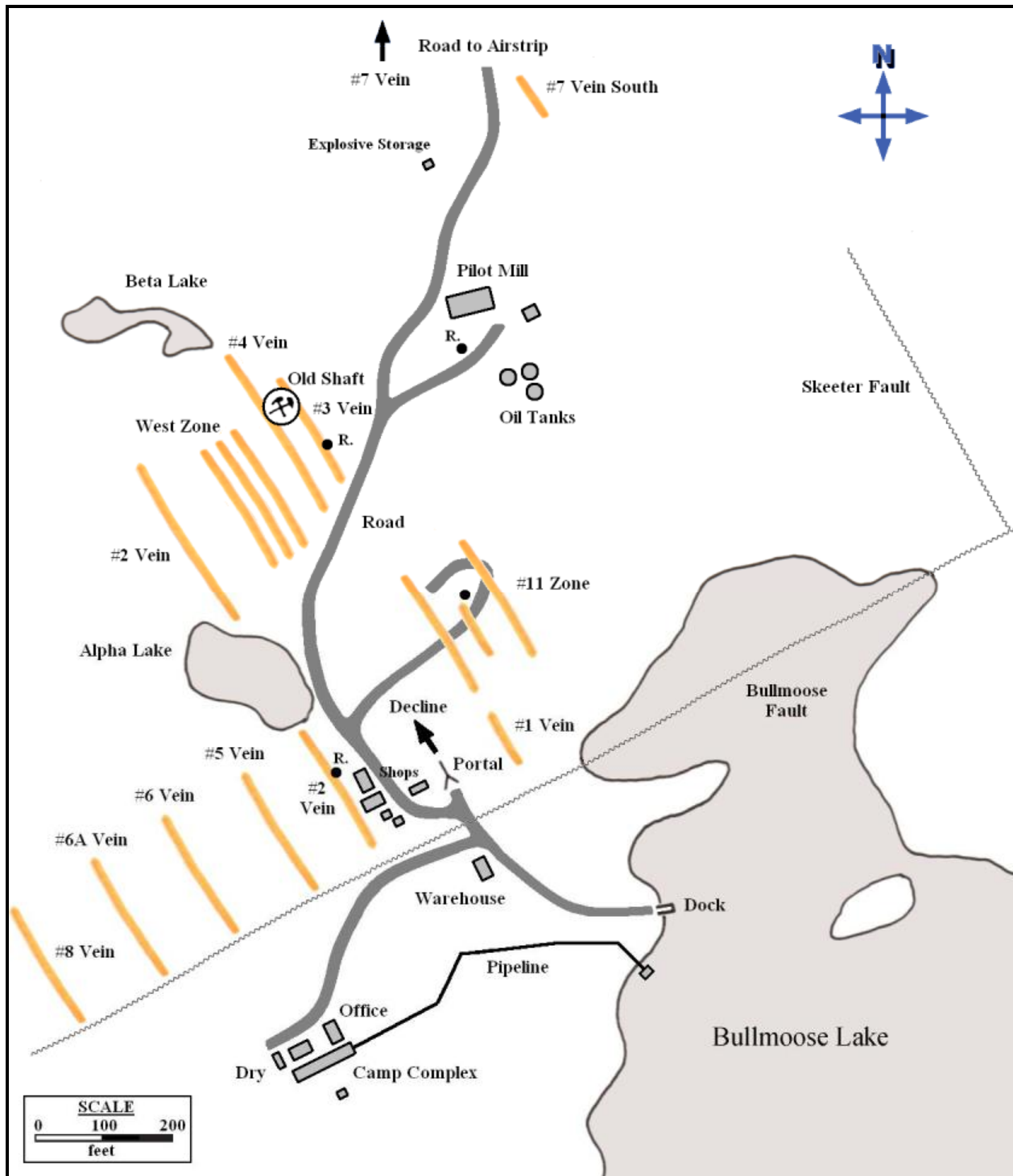


Figure 2. Bullmoose Lake Mine surface plan c.1987, showing surface vein systems.

Camp and Crew

A 100 man capacity Atco trailer camp was in use by 1985. Water was pumped from Bullmoose Lake and a proper septic treatment system was in place. On average, the operation employed about 40 to 45 persons during these years. In charge of work in 1984 were mine manager Jim McCormack, general manager Fred Sveinson, and mine captain Ray Gagnon.

Underground work was suspended for the first half of 1985, but resumed in July 1985. The onsite facilities (mine services and camp) were upgraded early in the year in anticipation of continued development and ultimate

production. The 600-foot (5th) level of the mine was reached in 1985, from where deep drilling of the deposit could begin. A 258 foot long Alimak raise was blasted from the 600-foot level to surface to provide adequate ventilation. The focus of development at this time was to create a clear picture of the deposit, incorporating all known vein structures located thus far. Raising between levels and a vigorous program of drift development was underway in 1985. Some of the best assays were attained during raise development on the 300-foot level. Early in 1986, an ore reserve was released (see **Table 1**). These reserves included mainly the #4, #11, and West zones. The #7 vein may have been part of this reserve (Terra Mines Ltd. Annual Report, 1985).

Economic Reserves:	Tons and Grade:	Gold:
<i>Proven:</i>	27,679 tons @ 0.43 opt	11,923 oz
<i>Probable:</i>	35,433 tons @ 0.36 opt	12,862 oz
<i>Indicated:</i>	72,408 tons @ 0.49 opt	36,150 oz
<i>Inferred:</i>	25,723 tons @ 0.45 opt	11,435 oz
<i>Total Economic:</i>	<i>161,243 tons @ 0.45 opt</i>	<i>72,400 oz</i>
Sub-Economic Reserves:	Tons and Grade:	Gold:
<i>Proven:</i>	41,998 tons @ 0.10 opt	4,190 oz
<i>Probable:</i>	4,772 tons @ 0.13 opt	642 oz
<i>Indicated:</i>	35,359 tons @ 0.14 opt	5,014 oz
<i>Inferred:</i>	20,731 tons @ 0.15 opt	3,209 oz
<i>Total Sub-Economic:</i>	<i>102,860 tons @ 0.32 opt</i>	<i>13,055 oz</i>

Table 1. Ore reserves February 11th 1986. (source: Terra Mines Limited, Annual Report 1985)

To better provide an idea of the grade of ore being mined, it was decided to install a portable pilot mill. Overall, it was agreed that the installation of such a plant would be a good investment, as the anticipated pilot mill could be expanded at a later date if results warranted full production at Bullmoose Lake, but this would require added mine support facilities. Planning for a possible long-term high-tonnage gold producer was underway during 1985. The creation of this mine was important for the managers of Terra Mines Limited, since the silver operations at Great Bear Lake were on the verge of closing due to depressed prices of the commodity. The Bullmoose operation would provide the company with a source of income (Terra Mines Ltd. Annual Report, 1985).

Towards Production

Work completed during 1985 towards putting Bullmoose into production included the clearing of a 5,000 foot long airstrip (capable of to handling both DC-3 and C-130 Hercules aircraft), the enlargement of the camp to house 100 persons, and the purchase of a 75 tons per day portable pilot mill from Vancouver. This mill was designed by Inlet Metal and Machinery Limited and Nelson Machinery Company Limited, incorporating used machinery to build a cost effective milling plant. The compact design and affordable price tag was important for the uncertain Bullmoose project. The mill arrived onsite in late 1985 and was ready for operation in March 1986. Production began in April 1986 (Terra Mines Ltd. Annual Reports, 1985-1986; Inlet Metal and Machinery Company, 1986).

Mill Operations

Ore was crushed two-stage by jaw and cone crushers before being processed in a 7 foot x 7 foot Marcy ball mill in closed circuit with a cyclone classifier. About 60% of the gold was caught as a concentrate in jigs, and tailings were sent to a bank of four Denver flotation machines. Flotation concentrates were thickened, dried, and sacked for shipment to Cominco's Trail smelter in B.C. where the remainder of the gold was recovered. Tailings were jettisoned into a nearby lake. The mill processed about 150 tons per day with recoveries as high as 95%, but averaging about

93%. The mill was powered by two Cat D-379 diesel generators of 400 kilowatt each, which also provided heat for the building. These units also supplied the camp with power (Inlet Metal and Machinery Company, 1986).

Mining Operations 1986

Ore was drawn from the #4 zone, which contained many vein structures including the #3-4 veins. Testing of different veins was underway to compare assay tests with mill recoveries. Three stopes were being developed in 1986. Underground work during the year included the driving of the main decline to about the 700-foot level (Terra Mines Ltd. Annual Report, 1986).



Terra Mines Ltd.

Figure 3. Bullmoose Lake Mine mill, 1986.

The highlight of the year in development was the start of a new decline collared on surface at the #7 zone, north of the main mine workings. This development was favored over deepening the main decline past 700 feet, since new ore could be provided for the mill. By the end of 1986, this decline had reached the 100-foot level where high-grade gold was encountered. The #7 zone decline is reported to be about 600 feet long. A record amount of mine development was done during 1986, the result of three operational programs: decline development to the 700-foot level, decline driving on the #7 zone, and the development required on all levels to gain access to veins for the mill operation. No known development was performed during 1987 (Terra Mines Ltd. Annual Report, 1986).

Year:	Decline:	Drifting and Crosscutting:	Raising:	U/G Drilling:	Surface Drilling:
1975-1976	1,400'	1,146'	110'	1,281'	-
1977-1982	-	-	-	-	4,916'
1983	1,270'	1,635'	574'	9,626'	10,945'
1984	800'	6,768'	2,893'	19,197'	30,939'
1985	592'	2,994'	1,608'	10,673'	5,000'
1986	3,229'	9,685'	3,580'	21,854'	9,572'
1987	-	-	-	-	-
<i>Total:</i>	<i>7,271'</i>	<i>22,451'</i>	<i>8,655'</i>	<i>62,631'</i>	<i>61,372'</i>

Table 2. Total mine development to the end of 1986. (source: Terra Mines Ltd. Annual Reports)

Year:	Ore Milled:	Gold Produced:
1986	47,200 tons	343 kg
1987	27,400 tons	160 kg
<i>Total:</i>	<i>74,600 tons</i>	<i>503 kg</i>

Table 3. Bullmoose Lake Mine production, 1986-1987.

Closure in 1987

The year-long mill test showed that the grades of milled material and those of mine assays did not compare and that dilution caused a loss of gold content. Due to a lack of sufficient tonnage and lower than anticipated mill results, it was decided to cease operations at Bullmoose Lake. The plant closed down in June 1987 and all equipment was removed by the following summer. It is believed that considerable ore reserves remain on the property, but they have not been proven to be economic. The entire site was remediated during the summer of 1987, and all equipment and the portable mill were trucked off site by winter road early in 1988 (Terra Mines Ltd. Annual Report, 1987).

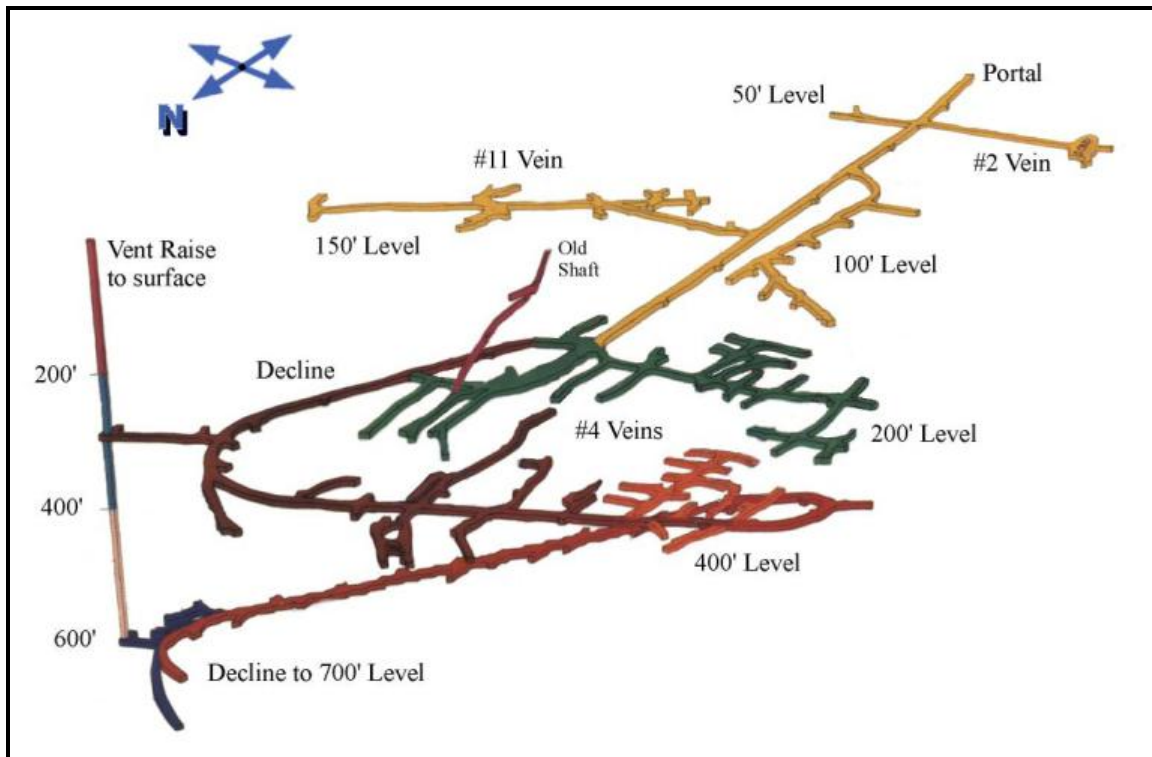


Figure 4. Bullmoose Lake Mine underground plan, main decline.

Mine Development Summary

Total mine development is given in **Table 2** and **Figure 4** shows the extent of underground workings to the 600-foot level. Underground work from 1975 to 1987 included two declines. The main decline was collared south of the #4 zone, and extended to a depth of about 700 feet to provide access to seven levels at 50-, 100-, 150-, 200-, 400-, 600- and 700-foot depths. Sub levels at 250- and 300-foot depth were also developed. An Alimak raise connects the 600-foot (5th) level to the surface and had been used for the heat and ventilation plant. The second decline was collared in 1986 on the #7 zone and reached a depth of 100 feet. Total extent of workings in this area are unknown but the decline itself is approximately 600 feet long. Some mill feed was derived from the #7 zone in 1987.

Exploration Since Mine Closure

In 1989, Pickwick Explorations Ltd mapped the property and collected 135 samples (Brophy et al., 1995). Also in that year, it was reported that Mount Grant Mines Limited optioned the property and undertook a small exploration

program, including some geophysical work. Results of this work were unknown, but the company was hopeful that an economic mining operation could be established. Ore reserves were announced as 36,718 tons of proven and probable ore grading 0.47 ounces per ton gold, and an additional 84,937 tons of drill indicated ore grading 0.39 ounces per ton gold (Northwest Territories Mining Industry News, Dec. 1989). No other work has been reported.

The property was then acquired by Avance International Incorporated in 1994 and performed small amounts of assessment work to keep the lease in good standing. This included a magnetometer survey, probably in search for kimberlite anomalies, in 1996.

In 2006, the property was optioned to NWT Gold Corporation. No known work has been done.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0019



BURNT ISLAND

Minor Producer (Abandoned)

Gold

63° 03' 45" 113° 09' 45"

Years of Primary Development: 1940s, 1984, 1989-1990 *Mine Development:* 43' incline shaft; decline to 100' (1 level)

Years of Production: c.1944, 1990

Mine Production: ~1,800 tons milled @ 1.1 oz/ton Au.

Introduction

Burnt Island is located on Gordon Lake, 90 kilometers northeast of Yellowknife, NWT. Two periods of development, one in the 1940s and the other in the 1980s, explored the small high-grade ore body underground and even had small mills installed to recover some gold. The Island was visited in September 2001 by the author.

Brief History

The property was first staked as part of the 'Ardogo' claim group in 1936 by Mining Corporation of Canada Limited. These lapsed and were re-staked in 1942 as the 'Good Hope' group. Some development was done during the 1940s through sinking a shaft and installing a small mill. About 60 tons were milled to recover 36 ounces of gold. Jake Woolgar re-staked the property as the 'Goo' claims in 1958. In 1980, the lease was sold to Grover George of Spokane, Washington. The claims were subsequently turned over to Burnt Island Gold Incorporated. A decline was started in 1984 but the underground target was not reached due to lack of funds. Small-scale mining and minor production was attained in 1990 under the direction of Cameron Mining Limited. Walt Humphries re-staked the property as the 'Isle' claims in 1998.



Ryan Silke

Figure 1. *Burnt Island Mine old headframe, 2001.*

Geology and Ore Deposits¹

Gordon Lake is underlain by greywackes and slates of the Yellowknife Supergroup. These thin interbeds of dark slate and greywacke have been deposited as a turbidite sequence. In places, the greywacke beds are thick and massive and locally bedding structures have been obliterated by metamorphism. Quartz veins, which are abundant throughout the sedimentary rocks, consist of high-temperature glassy quartz with tourmaline and a few feldspar crystals in schists and hornfels. This contrasts with lower temperature, milky quartz with carbonate in the slate and greywackes underlying Gordon Lake. The sedimentary rocks underlying Burnt Island are typical bedded greywackes, siltstones and slates of turbidite sequences. The vein mineral assemblage includes, pyrite, arsenopyrite, pyrrotite, galena, chalcocopyrite, and free gold.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office and Knutsen (1989)

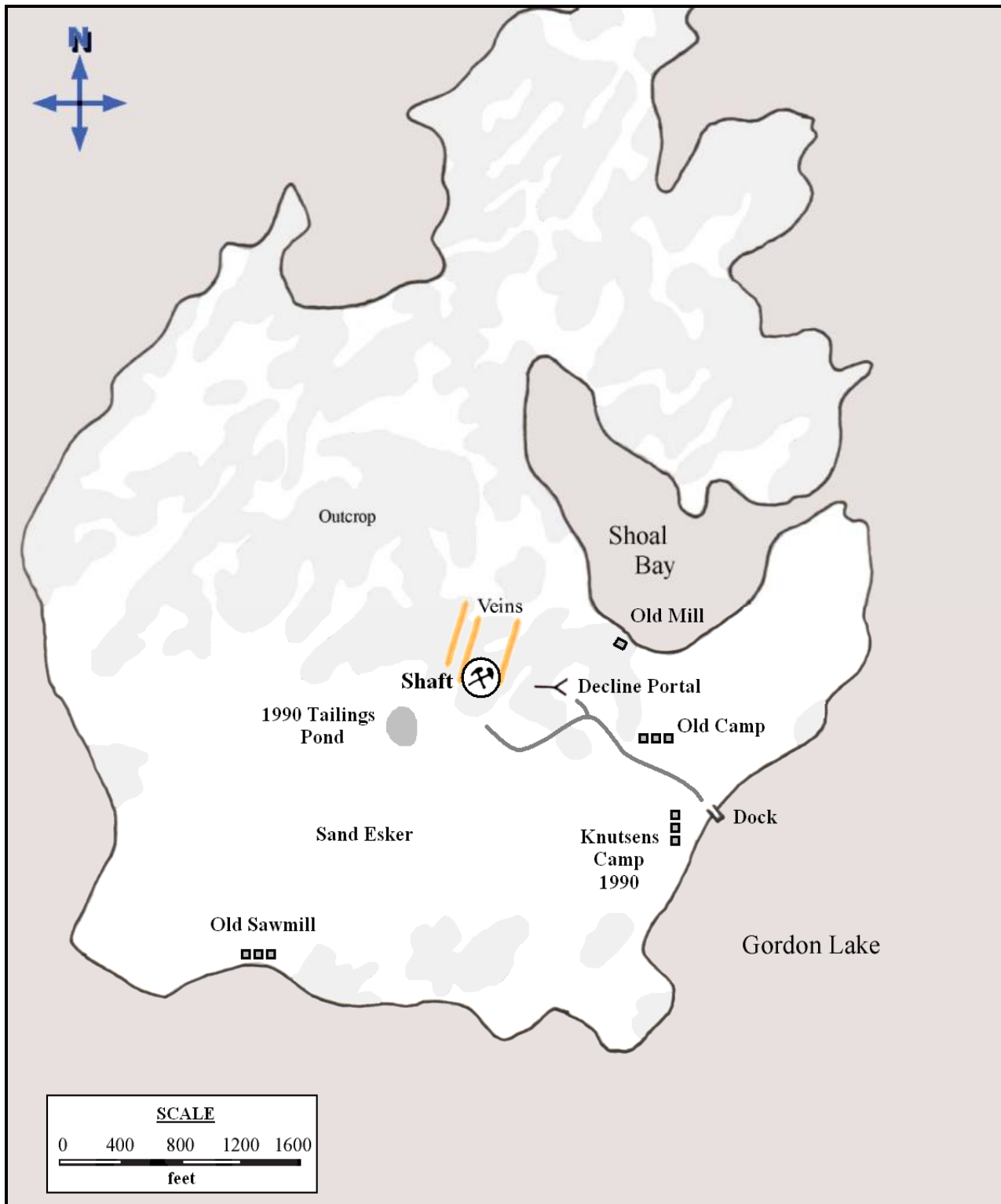


Figure 2. Burnt Island Map.

Veins are seldom more than 100 feet long and extend to unknown depths. Though high-grade, it is believed the veins are small in tonnage. Most work has focused on the #1 vein. This vein is reported to be up to 40 feet long, 4 feet wide, and dipping 85° to the southeast. Underground development and diamond drilling has outlined an M-shaped vein extending to at least the 250-foot level (Knutsen, 1989).

Good Hope Mining Syndicate (1942-1945)

Some of the first major development at Burnt Island was reported soon after the ground was re-staked as the 'Good Hope' group in 1942. Jim McAvoy Sr. was in charge of most work at the property between 1942 and 1945, during which time a small sampling mill was erected and the prospect shaft sunk. Early mine development consisted of trenches and pits on a number of closely spaced quartz veins in a zone up to 6 feet wide. This work showed the zone to extend 100 feet where the vein narrowed to two feet (National Mineral Inventory). About 120 tons of ore were stockpiled from workings and crushed preparatory to milling (Knutsen, 1989).

Test Mill

A sampling plant was erected at a small bay on the east side of the Island (Shoal Bay). This plant did not operate for a long time and was designed utilizing equipment on-hand. Ore was probably carried to the mill on sled or by wheelbarrow, a distance of 800 feet on rugged terrain. As this material was already crushed to an appropriate size, it was immediately fed down an inclined ore bin and through a feeder into a Gibson pulverizer, in place of a ball mill. Ground product passed down a concentrating table and gold was recovered using a Denver amalgamation pan. Tailings were discharged into Shoal Bay. The plant was powered by gas or diesel engine and was housed in a log building 17 feet x 14 feet in dimensions (site evidence; Jim McAvoy Jr., pers. comm.).

Shaft

In 1945, the owners sank a 5 foot x 8 foot inclined prospect shaft a length of 43 feet directly on the #1 vein. A short timber headframe was built to service the prospect shaft. No known lateral development was conducted at the bottom of this shaft (Knutsen, 1989).

Camp

A small crew was housed in a camp consisting of three log tent frames and a log cookery. Burnt Island was once covered with large trees along the sand esker that makes up much of the south side. These trees provided building material for the headframe, mill, and camp buildings. A sawmill operated on the south end of the Island during these years (site evidence).

1940s Production

The only record of gold production was in 1944, when a shipment of three gold bars to the Royal Canadian Mint from the Good Hope Mining Syndicate contained a total of 36 ounces of gold. (National Archives of Canada) This ore would have been mined from surface pits and trenches since the shaft was not sunk until the following year. Based on the original 120-ton stockpile, and the reported 60 tons of high-grade ore left near the shaft stockpile in later years, it can be suggested that approximately 60 tons of ore were ultimately milled (Knutsen, 1989).

It has also been noted that some of the original stockpile was stolen in the 1940s, so it is difficult to say exactly how much ore was milled all together (Walt Humphries, pers. comm.). It is possible that more gold was recovered after 1944, perhaps even from the shaft, but no record has been found to suggest this. Samples of the tailings material, collected by Walt Humphries from the shallow shores of Shoal Bay, have assayed 1.05 ounces per ton gold (Humphries, 1999).

The property was spectacularly rich and high-grade, but lack of finances resulted in a cessation of work. Zolota Yellowknife Gold Mines Limited acquired the property in 1946 through a deal with Jim McAvoy. This company contemplated deepening the shaft in 1950 but no work was done (The News of the North, Nov. 8th 1946; Apr. 21st 1950). The claims later lapsed and but were again re-staked as the 'Goo' group by Jake Woolgar in 1958.

Burnt Island Gold Incorporated (1984)

This company acquired the property in 1980 and conducted extensive exploration and evaluation of the gold deposits in the Burnt Island vicinity, focusing on the old mine. Drilling indicated that the #1 vein extended to a depth of 200 feet. An underground decline was collared in March 1984 on the shaft zone with the objective of driving the ramp a length of 600 feet. A lack of funds stopped the project before the target could be reached, and it is told that the 325-foot decline stopped 80 feet short of the #1 vein (Knutsen, 1989). Jake Woolgar was apparently involved in this work through his association with the company.

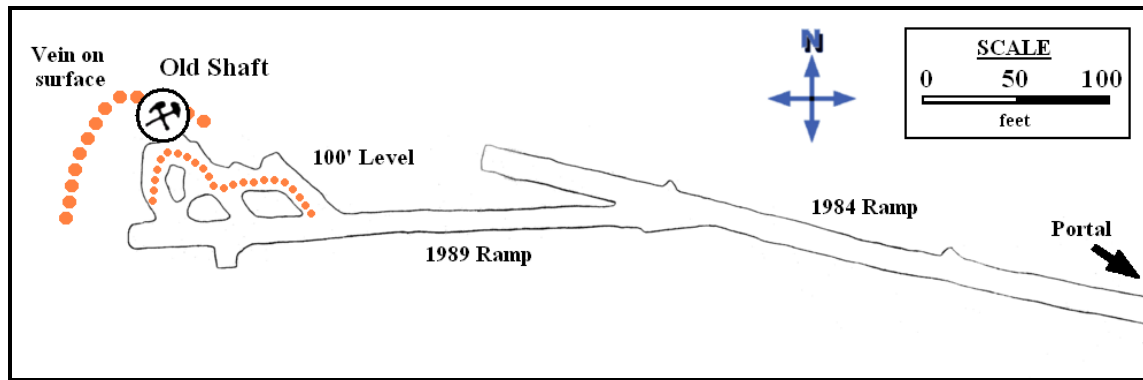


Figure 3. Burnt Island Mine underground plan, 1989.

Cameron Mining Limited (1989-1990)

New Era Developments Limited acquired the property in 1983 but performed no work. In 1986, Cameron Mining Limited, a small mining outfit headed by William Knutsen, optioned the Burnt Island property subject to a royalty interest to New Era. Reserves were calculated in 1989 as 1,300 tons grading 2.11 ounces per ton gold to the 100-foot level (Knutsen, 1989). Underground work resumed in the summer of 1989 when the decline was continued for a distance of 250 feet to a depth of 100 feet below shaft collar. A program of drifting, slashing, and raising was conducted on the 100-foot level of the #1 vein. A raise was brought up to tap into the bottom of the old shaft. A 1,000-ton bulk sample was mined from the 100-foot level to 30 feet below the surface. 2,000 tons of broken ore remained to be removed at the end of the 1989 program. Thirteen holes were diamond drilled (1,000 feet) to test the vein to the 200-foot level. It was initially planned to truck ore to Yellowknife for processing in 1990. This plan was changed late in 1989 when it was considered feasible to install a portable milling plant on Burnt Island to process the ores. Metallurgical testwork indicated a 92% recovery through the use of a gravity milling process. If the testing went well, the mill would also be used to process ores from the nearby West Bay Mine and could be kept onsite for long-term mining operations in the Gordon Lake area. Supplies and equipment were mobilized onsite over the winter road in March and April 1990. Dewatering of the underground workings commenced in April and limited mining development recommenced (NWT Water Board Files – Water License N1L3-1566; Cameron Mining Ltd., 1990a).

Milling Plant

The 35 tons per day plant was housed in portable trailer units for easy erection and dismantling. It was a simple gravity-process mill, and there was no cyanide circuit employed. Crushing was completed three-stage using a 10 inch x 20 inch Sheffield jaw crusher, 8 inch x 10 inch Denver jaw crusher, and 18 inch Kue-Ken cone crusher in sequence. Final crushed product was about ¼ inch, which was sent to a 5 ton fine ore bin. A rubber-lined 4 foot x 5 foot Marcy ball mill was used in the grinding circuit, in closed circuit with a 6 foot spiral classifier and 8 inch Dorr cyclone. The rubber lining in the mill proved to hamper its effectiveness. A concentrate was recovered off a duplex mineral jig placed at the end of the ball mill and also from the spiral and cyclone classifier, and concentrated on two Wilfrey tables. Gold pouring was done on site with a pour once a week. Bars weighed 60 to 70 oz containing 96% gold with about 12 to 14 bricks poured during the year (William Knutsen, pers. comm.; Cameron Mining Ltd., 1990a).

Tailings were supposed to be deposited in the underground mined-out stope. Tailings were pumped from the plant up to the old shaft where a 10 foot diameter Dorr thickener was positioned to separate the suspended solids with the clear overflow water (which was reused in the milling circuit). The solid tailings were then pumped down the shaft into the mined-out stope, but only two-weeks of tailings production was impounded underground when it was decided to divert tailings to a surface storage area located in a natural depression south of the old shaft (NWT Water Board Files – Water License N1L3-1566; Cameron Mining Ltd., 1990a).

The milling plant was powered by a 250 kilowatt Cat diesel generator. Underground mining was powered by Atlas-Copco drills and portable Atlas-Copco air compressors. A crew of 16 to 18 men were employed and resided in a tent camp on the east shore of Burnt Island. William Knutsen was in charge of operations (William Knutsen, pers. comm.).

Exploration to define the continuation of the deposit continued during the milling operation. Gold was reported in drill results down to 250 feet, and it was planned to extend the decline to access these ores. The decline may have been extended beyond the 100-foot level, but no mining was accomplished.

Production

Production from June to September 16th 1990 came from from the 1989 surface stockpile and previously broken underground ore (NWT Water Board Files – Water License NIL3-1566; Cameron Mining Ltd., 1990a). William Knutsen says that he milled 1,800 tons with a grade of 1.1 ounces per ton gold (R.H. Hauser, 1996 reported 0.75 ounces per ton, but his sources are unknown). The operation was considered uneconomic as no profit was made on the venture. Financial and management problems, not to mention the high cost of operating the small camp and plant, were the primary reasons for the cessation of work. Ore stored at the property from the West Bay Mine was sold to Knut Rasmussen and was trucked to Ptarmigan Mine in Yellowknife for milling. William Knutsen, with the funds recovered from the sale of gold bars and the West Bay ore, was able to dismantle and cleanup the operation. The mine site was cleaned up and abandoned by 1993. (William Knutsen, pers. comm.).

Exploration Since Mine Closure

Walt Humphries re-staked the property as the ‘Isle’ claims in 1998 and has conducted surface sampling. Samples of tailings from the original 1940s mill have assayed 1.05 ounces per ton gold. Four samples from the 1990 tailings pond averaged 0.081 ounces per ton gold. A sample of ore at the 1940s mill site averaged 0.82 ounces per ton, and four samples of muck from around the shaft area ran 1.744 ounces per ton, 0.425 ounces per ton, 0.9 ounces per ton, and 6.141 ounces per ton gold (Humphries, 1999).

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085PSW0076

Personal communication: William Knutsen; Knut Rasmussen; Walt Humphries; Red McBryan; Jim McAvoy Jr.



BURWASH

High-Graded (Abandoned)

Gold

62° 27' 45" 114° 19' 00"

Years of Primary Development: 1935-1936

Mine Development: 125' vertical shaft, 1 level (177' dev.)

Years of High-Grading: 1935

High-Grading: 16 tons shipped @ 13.6 opt Au = 200 oz Au

Introduction

Burwash Mine was the first underground mining operation in the Yellowknife area and opened in 1935. It is located on the east side of Yellowknife Bay, south of Burwash Point. The area has been visited numerous times by the author, most recently in the summer of 2004.

History in Brief

Johnny Baker discovered gold near Burwash Point in September 1934 while prospecting the Yellowknife Bay region for Yellowknife Gold Mines Limited, a subsidiary of Bear Exploration and Radium Limited. The discovery was made by pure chance when Baker and Hugh Muir camped at Burwash Point to wait out a storm. The claims were aptly named the 'Rich' group.



Yellowknife: An Illustrated History

Figure 1. *Burwash Mine in 1936.*

In 1935, a small shaft was sunk to gain access to the vein at depth. The high-grade nature of the gold vein did not persist at depth and the property was closed in 1936. Exploration in 1945 and as recently as 1997 have been unsuccessful in attempts to locate a significant gold resource at the old Burwash Mine. There are no structures remaining at the site; the old headframe was knocked down in 1969.

Geology and Ore Deposits¹

The Burwash Mine is within the western part of a belt of Archean turbiditic sediments, the Yellowknife Metasedimentary Basin. The Basin is flanked to its west of the property by the Yellowknife Volcanic Belt, exposed one kilometer west across Yellowknife Bay. Various evidence suggests that the Yellowknife Volcanic Belt represents the original basin margin. The metasedimentary basin is dominated by Yellowknife Supergroup Burwash Formation greywackes and argillites, which host the showing; Duck Formation mafic volcanics also occur in the southwest part of the Basin and are exposed a few kilometres southeast of the deposit. The supracrustals were folded and faulted during two or three phases of Archean deformation, and invaded by several generations of Archean granitoids, mainly north and south of the showing. The rock are metamorphosed to upper greenschist facies in the vicinity of the deposit and to lower amphibolite facies six to eight kilometers farther east.

The sediments underlying the showing area strike 060 to 120 degrees and dip steeply south to vertically but face north. The Proterozoic Akaitcho Fault trends north-northwest a few hundred metres east of the deposit, while the host

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

shear zone is oriented 015/75 degrees west. Movement along the host shear/fault has been judged as west-side up and slightly south. The shear is exposed along 70 meters. Its width is not much greater than that of the quartz lenses it hosts, from a few centimetres of chlorite schist where there is no quartz, to 70 centimeters wide at the discovery site. Quartz lenses occur at intervals along the shear and are up to 25 centimeters wide (except at the discovery site, which is about 14 meters from the south end of the exposed shear). The lens which hosts the ore is up to 70 centimeters wide and is 7.6 meters long at surface. At 38 meters depth, it ranges from zero to 80 centimeters wide along a 16 meter length. The host lens on surface is offset up to one meter by several cross-faults.

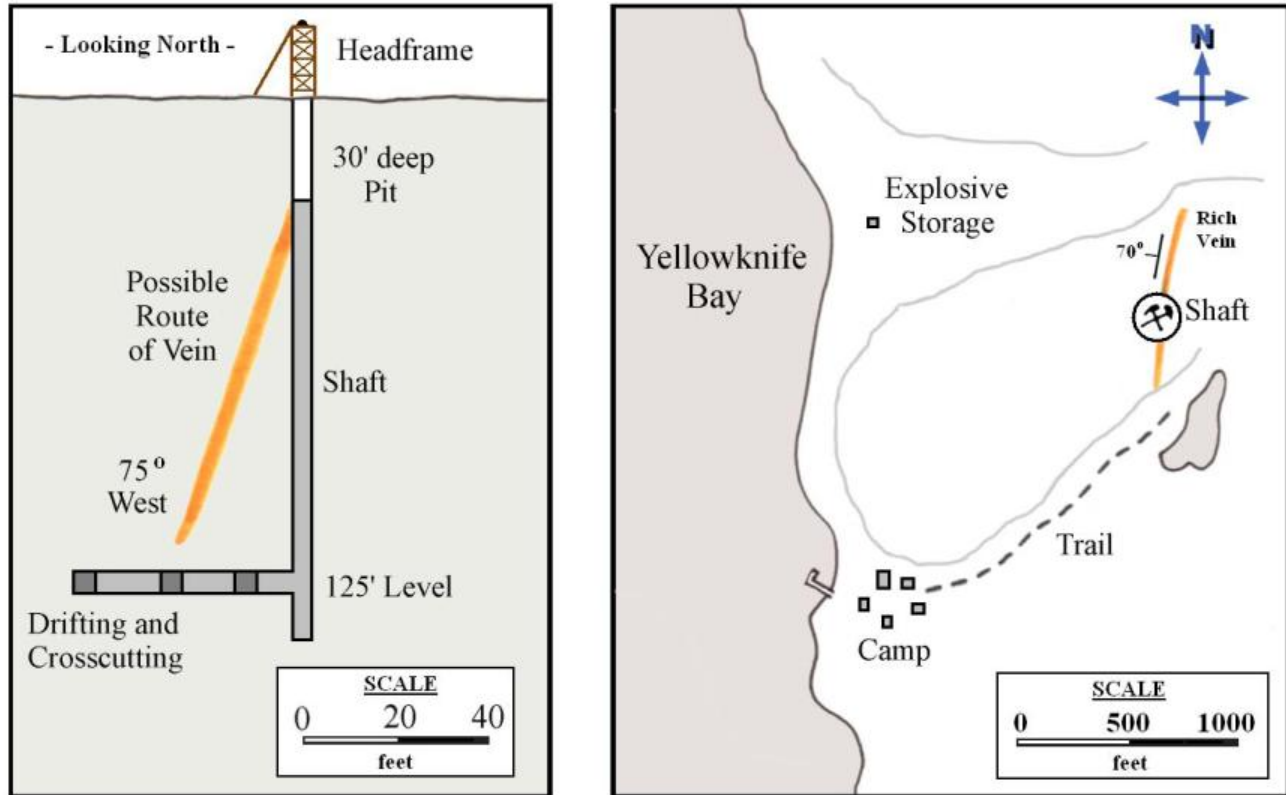


Figure 2. *Burwash Mine surface and underground plan, c.1936.*

Burwash Yellowknife Gold Mines Limited (1935-1936)

Organized for the acquisition of the Yellowknife property from its parent company Yellowknife Gold Mines Limited, Burwash Yellowknife Gold Mines Limited dispatched a development crew to the 'Rich' claims in March 1935. Johnny Baker was in charge. First work involved the erection of a suitable prospecting camp. This camp originally consisted of log-tent frames with four bunk-tents and a cookery tent. In April, construction of a permanent log-cabin camp began. Logs were brought in by local native people under contract. The most impressive building was a large 20-man cookhouse made of manually cut and peeled timber logs. By October 1935, the camp also included a 15-man bunkhouse and warehouse; all log buildings.

In April 1935, Major Lockie Burwash arrived at the property to act as manager. Starting April 29th 1935, a small ten man crew began excavation of a 30 foot deep pit on top of the discovery vein (The Northern Miner, June 6th 1935). In June, the pit was 23 feet in length and 12 feet deep. Channel samples were taken at two foot intervals, assaying 13 ounce per ton gold over a 13 inch width (The Toronto Star, June 25th 1935). Value increased in the pit during July 1935, where channel samples at 17 feet depth on the pit floor at 2 foot intervals produced 40 ounces per ton gold assay across a 6 inch width. Samples of the wall rock also returned high values. But the highest gold values of all came from a quartz calcite vug at the north end of the pit at a depth of 15 to 17 feet. The assays returned 300 ounces per ton gold and 74 ounces per ton silver (The Toronto Star, July 25th 1935).

The pit when completed was 25 feet long, 7 feet wide, and 30 feet deep. Through a vigorous job of hand mucking about 30 tons of high-grade ore (comprised of both vein quartz and some wall-rock) were stockpiled (Lord, 1941).

While developing the pit, it was noted that the vein remained almost constant at a 75° dip to the west so that at the bottom of the pit, the vein struck through the middle. A decision was made to follow the vein underground by sinking a shaft.

Shaft Sinking

In July 1935, a 2-compartment vertical shaft (12 feet x 6 feet dimensions) was collared at the bottom of the pit utilizing hand steel methods (The Toronto Star, July 25th 1935). The pit was then backfilled and a headframe was erected at the shaft site. Shaft sinking did not get underway until October 15th 1935, when mining equipment arrived at the property. 12 men were employed during September installing machinery, erecting headframe, and performing assessment work on the claims. By the end of the month of October, the shaft was 30 feet deep. Construction was wrapped-up during the month with the completion of headframe, compressor and hoist house, shop, bunkhouse, and cookery. During November 1935, shaft sinking continued on a 2-shift basis at 2.5 feet per day. It was 70 feet deep by the end of the month. It was noted that the vein left the shaft at 33 feet depth. The shaft was completed to its first level objective of 125 feet on December 24th 1935, and crosscutting to reach the vein was begun (Cummings, 1935).

Bulk Shipment

About 30 tons of ore were mined and stockpiled during the summer of 1935, of which 16 tons were shipped by boat to Trail, B.C. for assaying in September 1935. Content was 13.6 ounces per ton gold (Lord, 1941).

Employees

On average, 15 to 20 people were employed at the Burwash property during 1935-1936. The monthly payroll for March 1936 suggests that 14 men were employed. The highest paid employed was Ollie Hagen, mine captain, who received wages of \$175 per month (NWT Archives). Major Lockie T. Burwash was in charge of operations, assisted by field manager Johnny Baker. Tom Payne was mechanic; Ollie Hagen and George Goodwin were mine captains; Noel Barlow was camp cook; Jock McMeekan was a field prospector (McMeekan, 1984).

Mining Plant

Operations were serviced by a 210 cubic feet per minute Canadian Ingersoll-Rand air compressor engine and a small 6x5 air hoist. A small blacksmith shop was also in use to sharpen drill steel. No assay lab was on site during 1935; samples were flown to Contact Lake Mine at Great Bear Lake for assaying. However, in 1936, an assay lab operation was set up at the Burwash Mine (NWT Archives).

Underground work continued into 1936. From the 125-foot level, a 70 foot crosscut was driven west in an effort to intersect the westerly dipping vein. Three drifts then followed a mineralized section of the vein for distances totaling 107 feet (Lord, 1941). First drifting missed the ore shoot, and in April 1936 more work was required to reach the vein (The Northern Miner, April 16th 1936). The vein was reported up to 30 inches wide, but averaging 16 inches. This is in contrast to the surface where the vein was 2 to 13 inches wide (The Toronto Star, July 22nd 1936). It was later reported that the break was intersected at a deeper horizon by diamond drilling in an area 595 feet south of the shaft. A second diamond drill rig was operating underground, following the vein with a series of short holes from the point where it dips west (The Toronto Star, Aug. 12th 1936). The accuracy of these newspaper reports are unknown, as it was later reported that the lateral development underground intersected a number of quartz stringers but the mineralized zone was not as spectacular underground as it had been on surface (Lord, 1941). It has also been suggested that underground work missed the vein entirely because its dip and strike changed (Walt Humphries). More than 2,000 feet of diamond drilling and 300 feet of trenching were conducted up until September 1936 when the camp was closed (Lord, 1941). Trenching 130 feet north of the shaft in the vein encountered some high-grade ore samples (The Toronto Star, Aug. 12th 1936).

It was then decided to forego any further development in favor of exploration of the Giant property. All equipment and most of the buildings on site were sent to Giant for the development of the Brock vein. The property was largely abandoned by 1938.

Exploration Since Mine Closure

Rich Group Yellowknife Mines Limited was formed in 1945 to acquire the property. Diamond drilling in 29 holes was conducted but results were disappointing (National Mineral Inventory). In 1987, the claims were acquired by Rayrock Yellowknife Resources Limited, through a merger of Yellowknife Bear Resources Inc. (previously Yellowknife Bear Mines Ltd., the successor of Bear Exploration and Radium Ltd.) and Rayrock Resources Limited. In 1991, Walt Humphries and Dave Smith of Yellowknife purchased a controlling interest in the claim group. They conducted surface prospecting and sampling throughout the 1990s (Walt Humphries, pers. comm.). In 1997, Roberts

Bay Resources Limited optioned the claims from Humphries and Smith and conducted some surface sampling of the primary vein, obtaining some high values. They were interesting in diamond drilling the deposit. The significant drop in the price of gold later that year forced the company to drop the option, and no work has been done since (Roberts Bay Resources Ltd. Press Release, July 17th 1997).

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Personal communication: Walt Humphries



CAMLAREN

Producer (Remediated)

Gold

62° 59' 15" 113° 12' 15"

Years of Primary Development: 1937-1938, 1962, 1974, 1980-1981

Mine Development: 1,055' vertical #1 shaft, 5 levels (5,267' dev.); 200' vertical #2 shaft

Years of Production: 1963, 1980-1981

Mine Production: 64,137 tons milled = 36,661 oz Au

Introduction

The Camlaren Mine is located on Gordon Lake, 83 kilometers northeast of Yellowknife, NWT, on the south-end of a narrow island unofficially known as Muir Island. The #2 shaft of Camlaren Mine is located on the northeast shore of Zenith Island, about 2 kilometers southwest of the main mine-site. The mine closed in 1981 and the site was cleared of all structures in 1991. The old mine was visited by the author in September 2001.

Brief History

Gold was found in a massive quartz vein by Don W. Cameron in the early summer of 1936 and the 'Camlaren' claims were staked covering Muir and Zenith Island on Gordon Lake. This and other discoveries by Cameron's crew sparked a massive staking rush to Gordon Lake in the fall of 1936. Development began at Camlaren Mine in 1937 but work stopped in 1938, due primarily to the economics of the project. It reopened in 1962-1963 and a large amount of ore was removed for milling. Additional mine development was completed in 1974-1975. Full production was achieved in 1980-1981 when a portable milling plant was installed to recover the known gold reserves down to the 1,000 foot level. The mine was cleaned up and all old buildings were burned down in 1991. The property was re-staked as part of the 'Joliff' claims by Walt Humphries in 1997-1998.

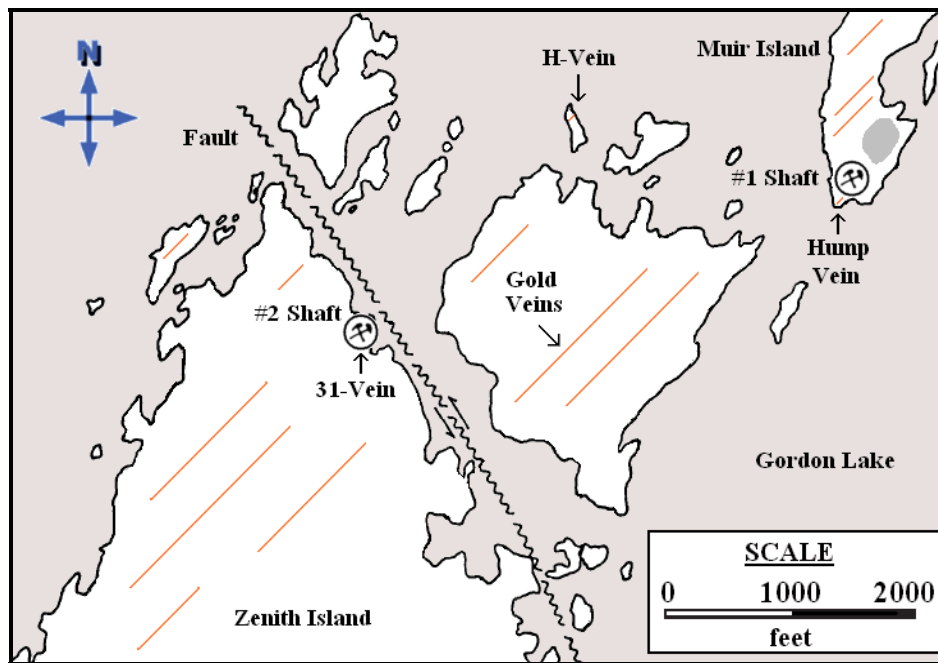


Figure 1. *Camlaren Mine geology and ore deposits.*

Geology and Ore Deposits¹

The showing area is underlain by thinly-bedded slate and greywacke of the Archean Yellowknife Supergroup. The sediments strike northeasterly and are steeply dipping. The Camlaren Mine is developed on the Hump vein, which occurs on the nose of an anticlinal fold that plunges about 50° north. The vein is white quartz, weakly mineralized with sulphides and native gold. The sulphides include pyrite, pyrrhotite, marcasite, arsenopyrite, galena and

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

sphalerite. Chalcopyrite has been noted in drill core but is not common. The area around the mine hosts two other significant showings the 31-vein and the H vein. These veins are similar in appearance to the Hump vein. The 31-vein is discontinuous and the H vein is only 38 centimeters wide. The H vein averaged 42 grams per tonne over a length of 33 meters.

Camlaren Mines Limited [Mining Corporation of Canada Limited] (1937-1938)

Camlaren Mines Limited was formed in 1937 to develop the gold claims at Gordon Lake. Mining Corporation of Canada Limited formed the company with a 65% controlling interest, the remaining share being held by the original stakers. Construction and development of a gold mine began during the summer of 1937 under the direction of Ken Muir, for Mining Corporation of Canada Limited. Diamond drilling performed early in the year provided encouraging indications that the Hump vein held promise for underground exploration (Lord, 1941).

Construction of buildings was speeded during the summer of 1937 in preparation for freezeup. Sinking of the #1 shaft began in September 1937 by handsteel methods. Orders were placed for permanent mining equipment to develop the shaft, and a 96-kilometer winter road was surveyed between Yellowknife and Gordon Lake. Because this machinery would not arrive until January 1938 at the earliest, and as the company wished to expedite the shaft in order to advance a production decision, the decision was made to bring in a small gasoline powered plant in the interim. Sinking of a 2-compartment shaft using this machinery started on October 13th 1937 with a four man shaft crew and continued to the 200-foot level in December 1937. A station was then cut and two crosscut rounds were taken out. Total shaft advance at year-end 1937 was 221 feet depth. (The Northern Miner, Feb. 17th 1938).

Equipment

Machinery used during the initial stages of shaft sinking in the winter of 1937-1938 included a Canadian Ingersoll-Rand 190 cubic feet per minute portable compressor, a Canadian Ingersoll-Rand 6 inch x 5 inch air hoist, two Canadian Ingersoll-Rand N69 rock drills, a 20 cubic foot mine car, a Canadian Ingersoll-Rand R34 drill sharpener, and a Canadian Ingersoll-Rand 7F oil furnace. An interesting method of cost reduction in the operation of the power plant was to mix gasoline and fuel oil (The Northern Miner, Feb. 17th 1938).

Personnel and Camp

In the summer of 1938 an average of 50 employees worked at Camlaren. Women and children of employees were also living on the property, being housed in small tent shacks and log cabins in the area. Ken Muir was mine manager, Hugh Fraser was geologist, A. Van Raalte was engineer, and Fred Prout was master mechanic. Camlaren Mine was a well-built camp, consisting of two bunkhouses and a cookery. Ken Muir, mine manager, had his own house (The Northern Miner, Feb. 17th 1938; The Globe and Mail, Aug. 19th 1938; Ward, 1957).

A 50 foot timber headframe was erected in February 1938, the permanent power plant was installed, and sinking of the #1 shaft to 380-foot depth was continued. The 1st level was opened up at 200-foot depth in January 1938, with ore being encountered early in the next month. Initial results were very encouraging, and company directors decided to proceed immediately with the purchase of milling equipment so as not to delay a production decision. The company planned for a 50 ton per day mill to process ores from two levels, the second of which had not yet been opened up by this point. But if the company had waited until results of the 2nd level work were known, there would have been a two-year delay in getting the mill on property (The Globe and Mail, May 14th 1938). Crosscutting on the 2nd level at 350-foot depth was started in May 1938, with ore being reached in June. Work continued throughout the summer on both levels in the effort to prove gold resources. Initial results on the 1st and 2nd levels suggested that a 50 tpd or larger mill was warranted for the Camlaren Mine. However, further work on the 2nd level failed to find enough ore to justify the installation of a mill, therefore orders for the mill was cancelled and plans were made to shut down operations at the end of 1938 (The Globe and Mail, July 7th 1938; The Toronto Star, Nov. 2nd 1938; Ward, 1957).

Zenith Island Shaft

In an effort to locate additional ore reserves, the company put down #2 shaft on the 31-vein, located on Zenith Island about two kilometers southwest of Muir Island. The 31-vein is part of a wider zone of quartz veins, traceable along strike for 150 feet before disappearing under Gordon Lake to the north. Starting in August 1938, the #2 shaft was sunk 200 feet to explore the 31-vein, and one level was developed with 220 feet of driving. 2,000 feet of diamond drilling was also reported from here. Work was suspended at the #2 shaft in November 1938 (The Globe and Mail, Aug. 19th 1938; Lord, 1941). At the same time, the H-vein was discovered on a small island between Zenith and Muir Islands (see **Figure 1**). As this deposit was mostly underwater, only diamond drilling was carried out. The entire claim group was geologically mapped during the summer of 1938 in the hopes of delineating further gold veins (Lord, 1941).

Development Summary 1937-1938

On the Hump vein, the two-compartment #1 shaft was sunk to 380 feet with two levels at 200- and 350-foot depths. On the 1st level, 322 feet of drifting exposed three ore shoots within the Hump vein, and on the 2nd level, 352 feet of drift exposed four ore shoots. Widths of the vein ranged from 1 to 7 feet. 530 feet of raising between levels was also completed. This included two surface breakthroughs from the 1st level. This comprehensive underground program indicated 13,177 tons of ore to a depth of 350 feet containing 0.62 ounces per ton gold in the Hump vein. Total diamond drilling at the Camlaren property during the 1937-1938 development program amounted to 8,823 feet. Total cost of the 1937-1938 development program was \$400,000 (Lord, 1951).

Consolidated Discovery Gold Mines Limited (1962-1963)

In 1958, Consolidated Northland Mines Limited conducted some assessment work on the old Camlaren property. It was the view of Norman W. Byrne, engineer, that it would be economical to re-mine the deposit and truck ore for milling (Byrne, 1958). Late in 1961, an agreement was signed between Consolidated Discovery Yellowknife Mines Limited and Camlaren Mines Limited, whereby Discovery Mine would mine and mill the known Camlaren ores at their own expense with profits split 50-50%. Discovery Mines Limited would also earn a share interest in the Camlaren company. This deal was a good one for both parties. For Discovery, whose own reserves were dwindling at its Giauque Lake property, operating the Camlaren would give the company time to conduct their own exploration programs and extend the life of the Discovery Mine another four months. For Camlaren, it meant re-opening the long dormant gold property that, thus far, had not turned in any profit or revenue (Consolidated Discovery Yellowknife Mines Ltd. Annual Report, 1961; Barrager and Hornbrook, 1963).

A large amount of equipment, men, and supplies were transferred from Discovery operations to Gordon Lake for the development program. The first objective for the program was to de-water the #1 shaft on the Hump vein, which was blocked with ice to a depth of 30 feet (Consolidated Discovery Yellowknife Mines Ltd. Annual Report, 1962).

Power and Hoisting Plant

As all equipment had previously been removed from the property, a new power plant was required. A cost savings was met by the acquisition of used equipment. Electricity was generated by a 125-KVA Ruston-Hornsby 10HRGE diesel generator set and a 75-KVA Cat D-13,000 diesel generator. Air was supplied by two compressors, one Cat diesel driven Chicago Pneumatic compressor of 500 cubic feet per minute and an electrically driven Gardner-Denver air compressor of 300 cubic feet per minute. The old 50 foot timber headframe was found to be in good shape and required little or no repairs. A Canadian Ingersoll-Rand 2-drum 36 inch x 24 inch electric hoist was installed, which was then lagged with wood to equal a diameter of 40 inches for greater rope speed (Hales, 1963).

Stoping operations between levels began in the summer of 1962. Drilling in the raises and stopes were done by Holman and Atlas-Copco air leg drills and a vent-fan was installed on the top of the upper stope raise to ventilate the mine. Before stoping could begin, an additional 468 feet of raising was required in the 2-03 and 3-03 stope areas. Shrinkage stoping was the preferred method of mining because of its simplicity and economy of operation. The hanging-wall and footwall of the Hump vein were easily defined because of a graphitic shear plane, resulting in minimal dilution. Several 50° angle ore shoots were outlined, and it was decided to mine these using open-stoping methods. Broken ore was collected by Eimco 12B mucking machines and transported to the shaft by one-ton side-dump Hudson ore cars. These cars were hoisted to the surface and hand trammed from the headframe along a 380 foot trestle tramway to the surface stockpile.

Over 14,000 tons of ore was mined and stockpiled on the surface by a 25 man operating crew under the direction of Fred W. Hales, engineer in charge. Cost of the 1962 operation was almost \$330,000 (Hales, 1963; Barrager and Hornbrook, 1963).

1963 Production

In 1963, 12,174 tons of stockpiled ore were transported by winter road west to the Discovery Mine, where it was milled during July and August of that year. 13,885 ounces of gold and 3,738 ounces of silver were produced, the product of 23 gold bricks poured at Discovery. The 1962-1963 program essentially proved the grade and mining characteristics of Camlaren's Hump vein structure, but a decision on full production was shelved until a substantial improvement in gold prices (Consolidated Discovery Yellowknife Mines Ltd. Annual Report, 1963).

In 1965, Consolidated Discovery Yellowknife Mines Limited was renamed Discovery Mines Limited. No work was done at Camlaren because of high costs and low gold prices. Some mobilization of equipment and supplies, and rehabilitation of some buildings was conducted in 1968 (Discovery Mines Ltd. Annual Report, 1968).

Discovery Mines Limited (1974-1975)

Underground development resumed in 1974 as a result of stronger gold prices. The shaft was deepened to 840 feet level and 509 feet of lateral advance was performed on the new 800-foot level. A level station was also excavated at the 600-foot level. The vein was reported to have similar characteristics and dimensions as on the upper levels. The deepened shaft was enlarged to 3-compartments. The aim of this work was to explore the conditions of the Hump vein below the 350-foot level and to provide a horizon for further deep drilling to test the gold vein at 1,000-foot depth. Work was done under the direction of Bill Case. Work temporarily ceased early in 1975 and the underground was allowed to flood. Preparations for future operations continued with the re-supply by winter road of supplies and some equipment. Other work included site development and engineering studies. The company announced plans to place a 75 to 125 tons per day mill on the property by 1976, but these plans were shelved because of gold price weakness and rising costs (Discovery Mines Ltd. Annual Reports, 1974-1975).

In 1978, total reserves to a depth of 1,000 feet were estimated as 56,000 tons grading 0.62 ounces per ton gold. Proven and indicated reserves (based on a cut-off grade of 0.35 ounces per ton) were 36,600 tons. The Hump vein was credited with 28,000 tons and the narrow #11 vein with 8,600 tons of ore. There were also 5,700 tons of stockpiled development ore on the surface with grades of 0.42 ounces per ton gold. Total ore reserve, including surface stockpile, was 42,300 tons. (Mining Corporation of Canada Ltd., 1979) Work since 1962 had allowed Discovery to acquire a 66.66% interest in the Camlaren project. Discovery Mines Limited's total direct and indirect interest in Camlaren Mines Limited by this point was 89% (Discovery Mines Ltd. Annual Report, 1978).

Mining Corporation of Canada Limited [Discovery Mines Limited] (1979-1981)

In light of rising gold prices during 1978-1979, attention in the local gold mining industry was focused on small high-grade projects with possibilities for a quick profit. Through terms of agreements between Discovery Mines Limited, Pamour Porcupine Mines Limited, and Noranda Mines Limited, Mining Corporation of Canada Limited was commissioned as contract developer and operator for the Camlaren Mine. Much of the ore reserves (44%) at Camlaren were below the 800-foot level, so it was necessary to deepen the shaft to 1,000-foot depth. Diamond drilling exploration below the 1,000-foot level was planned to outline depth extension of the deposit. Lateral development was planned for the 600-, 800-, and 1,000-foot levels. Meanwhile, a portable cyanidation mill would be shipped to Yellowknife for a startup in production during the summer of 1980 (Mining Corporation of Canada Ltd., 1979). Work on the project began in November 1979 with the start of detailed engineering work, acquisition of equipment, permits, and licenses. A crew was mobilized on property on January 20th 1980 with the completion of a winter road from the end of the Ingraham Trail. A trailer camp was erected, fuel storage facilities installed, and mining equipment brought to site (McCormack, 1980).

Production Begins

Commercial production of surface stockpiles began July 11th 1980. Shaft sinking to 1,055-foot depth was completed by September 1980 and production from the new levels started in late 1980 (Brophy et al., 1984).



R. Angus

Figure 3. *Camlaren Mine, 1981.*

Camp and Plant Complex

A 70 man camp was erected on Muir Island in 1980 consisting of 25 Atco portable trailers, arranged into two forty-man bunkhouses, one cookery complex, a recreation hall and commissary, and a 70 man dry. In 1979, the operation was considered to have a 16-month life span, and even though the project was extended a few more months, the short life of the Camlaren operation required that the site be totally portable. Thus, diesel generators and the explosive storage shed were all portable trailer-van units, and the mill building was fully portable. Power for the operation was supplied by three (and later four) Cat 3412 diesel generators each of 500 kilowatt output. Compressed air was supplied by two Worthington-Rolair electro-screw com-pressors that combined to deliver 2,000 cubic feet per minute. A tank farm consisting of ten 40,000-gallon fuel bladders were installed in a lined containment area. Propane for underground heaters and for the camp heaters was stored in six 15,000-gallon tanks. Mobile equipment for ore and waste handling included a Cat D-6 tractor, and two Cat 930 loaders (McCormack, 1980; Larkin, 1982).

Hoisting Operations

One of the only original features of the Camlaren operation was the old hoist and headframe plant, which required only a small amount of repair and maintenance to keep operational (McCormack, 1980).

The bottom levels of the mine intersected the Hump vein in similar conditions as the previously developed horizons. The vein maintained a constant northeasterly dip down to 1,000 feet as was hoped. Development of the 1,000-foot level was completed in September 1980, where it was reported that conditions of the vein (size and grade) had actually improved compared to the 800-foot level (The Northern Miner, Nov. 20th 1980).

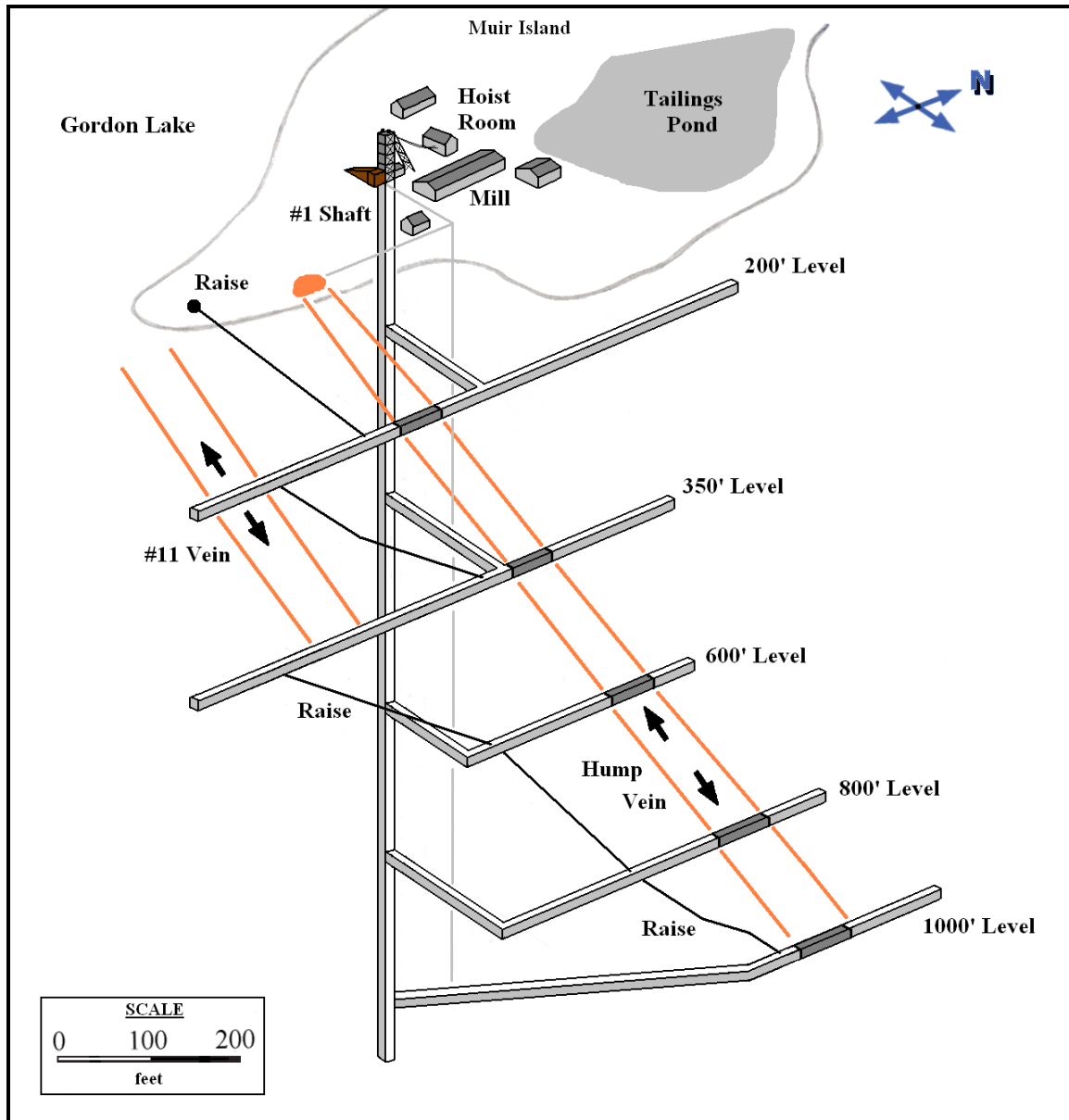


Figure 2. Camlaren Mine underground plan, c.1981.

Mining Operations

It was originally planned to use shrinkage stope mining methods, but a method of flat-breasting was put to use instead, which was later found to create dilution problems. Mill feed was originally from a small surface stockpile, but in late 1980, the bottom levels of the mine were opened up through northerly drifting and new underground ore was introduced. During the project, all levels were under development (see Table 1). Along with stoping of the Hump vein, some production came from the #11 vein, a thin collection of quartz lenses positioned 200 feet south of the Hump vein and parallel to its northeasterly plunge. Dilution in the #11 vein was high due to its narrow width and

schistose wall rock. Raising from the bottom level followed the vein. **Table 2** gives a listing of stopes in production during the project.

Milling Plant

The 150 tons per day mill was completely portable, having been designed specially for the Camlaren project. It was housed in a large 117 foot x 40 foot Atco steel building, with agitator tanks housed outside the structure. Ore was fed from a 30 ton coarse ore bin to an 18 inch x 30 inch Soya Massey jaw crusher, reducing ore to –2 inch, which was then fed up through a 42 foot high bucket elevator, and sorted through a Dillon screen. Oversize was passed into a 2 foot Nordberg cone crusher, which discharged back into the bucket elevator. All undersize material went for grinding in a 7 foot x 6 foot Allis-Chalmers ball mill, then through a 10 inch cyclone. A Denver Duplex jig was later added to send undersize cyclone material back into the ball mill. Cyclone product was sent to agitation, using four 15 foot x 15 foot wood-stave tanks with 24-hour agitation time. Filtering was accomplished with a 108 square foot Denver belt filter. Due to the graphite slime content of the ore feed, soluble losses using the belt filter were high. Pregnant solution from the filter was pumped into a 5 foot x 8 foot ten-leaf clarifier, and then vacuumed into a Merrill-Crowe de-aeration tower, then through one of two sock presses. Precipitate was packaged and made ready for shipment down south for refining into gold bullion (Ross, 1981; Kilborn Engineering Ltd., 1981).

The mill averaged about 130 tons per day, but in some months averages were as high as 210 tons per day. This step up in production was due to lower than expected ore grades. The plant had terrible recoveries during the first few months of production (50%) due to graphite contamination and equipment problems. Recovery was improved to 80% following a review of the milling circuit in 1981 (Ross, 1981; Kilborn Engineering Ltd., 1981).

Shaft Sinking:	216'
1 st Level (200'):	108'
2 nd Level (350'):	273'
3 rd Level (600'):	541'
4 th Level (800'):	362'
5 th Level (1,000'):	1,233'
Raising:	2,222'

Table 1. Summarized development, 1980-1981.

(source: Brophy et al., 1984)

1 st Level (Crown Pillar):	1,990 tons @ 1.364 oz/ton
2 nd Level (3-11 Stope):	3,283 tons @ 0.502 oz/ton
3 rd Level (6-H Stope):	9,663 tons @ 0.603 oz/ton
4 th Level (8-H Stope):	8,951 tons @ 0.529 oz/ton
5 th Level (10-11 Stope):	4,284 tons @ 0.404 oz/ton
5 th Level (10-11A Stope):	4,187 tons @ 0.393 oz/ton
5 th Level (10-H Stope):	13,372 tons @ 0.457 oz/ton

Table 2. Stope production, 1980-1981.

(source: Jarvi, 1981)

Depth Exploration

While production was underway, it was planned to conduct drilling programs from the 1,000-foot level to find additional ore reserves. Good grades were encountered on the 1,120- and 1,155-foot horizons. Significant ore reserves were indicated below the 1,000-foot level (10,841 tons with a cut-grade of 0.58 ounces per ton gold to 1,200 feet within the Hump vein), but it was considered uneconomic to deepen the shaft at that time. Such development would require headframe improvements and a new hoist; expenses which were not feasible. Originally, it was planned to end operations in March 1981, when ore reserves were scheduled to be depleted. Complications in the milling plant reduced daily tonnage, and an increase in ore reserves on the 1,000-foot level required a longer operational period. The original contract was revised with a new projected end-date of September 1981 (The Northern Miner, Nov. 20th 1980).

Final Closure

By September 1981, ore above the 1,000-foot level was largely mined out and the mine was closed. Cost of operations was extremely high and the capital cost associated with bringing Camlaren into production exceeded the revenue from gold production. The mine apparently also encountered a diabase dyke cutting across the vein thereby reducing ore reserves, and there were problems with the milling plant (Brophy et al., 1984). All portable buildings and equipment were mobilized for removal over the 1982 winter road.

Year:	Ore Milled:	Gold Produced:	Silver Produced:
1963	12,174 tons	13,885 oz	3,738 oz
1980	12,282 tons	5,196 oz	?
1981	39,680 tons	17,580 oz	?

Table 3. Camlaren Mine production. (source: Jarvi, 1981; Canadian Mineral Yearbooks)

Production and Development Summary

Between July 11th 1980 and September 20th 1981, it was reported by company officials that 51,963 tons (47,140 tonnes) of ore was milled at a mill-head grade of 0.463 ounces per ton (35.98 grams per tonne) gold. This ore was derived from both underground stope production and older surface stockpiles (Jarvi, 1981). According to the Canadian Mineral Yearbook, production for those years was 645.7 kilograms of gold (22,776 ounces). A breakdown of production by year is listed in **Table 3**. Total underground development at the main Camlaren deposit (Hump vein, #11 vein) consists of a 1,055-foot shaft (#1 shaft) developed with five levels, 5,267 feet of lateral workings, and 3,220 feet of raising, two of which break through to the surface. The 31-vein was developed with a 200-foot shaft (#2 shaft) in 1938.

Exploration Since Mine Closure

The property was re-staked as part of the 'Joliff' claims by Walt Humphries in 1997-1998. Many samples were taken from the tailings pond, the muck piles at #1 and #2 shafts, and the H-vein and 31-veins. Significant values were obtained. A sample from the tailings pond at three feet depth assayed 0.153 ounces per ton gold (Humphries, 1999). An underground resource still exists and was reported in 1986 to contain 9,979 tonnes grading 19.8 grams per tonne (Discovery West Corporation Ltd. Annual Report, 1986).

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	CANTUNG <i>Producer (Operational)</i>	Tungsten
		61° 58' 15" 128° 14' 00"

Years of Primary Development: 1960-1986, 2001-2003, 2005-current *Mine Development:* open pit; adit tunnel and decline ramps

Years of Production: 1962-1986, 2002-2003, 2005-current *Mine Production:* ongoing (see **Tables 2 and 3**)

Introduction

Cantung Mine, when in operation, has been Canada's only tungsten producer, and the largest in North America. It opened in 1962, closed in 1986, and was reopened in 2002 for a brief period before again closing due to financial problems with the operating company. The mine was reopened in September 2005.

Cantung, and the old townsite of "Tungsten", is located on the NWT-Yukon border in the Mackenzie Mountains. A 305-kilometer all-weather road connects the site to the Alaska Highway in the Yukon. It is 200 kilometers north of Watson Lake, Yukon and although the mine is within the boundary of the NWT, the property and former community have historically been tied to the economic and social interests of the Yukon.

History in Brief

The claims were first staked in 1954-1955 based on findings of copper. The deposit was uneconomic at the time and the claims were dropped, but the property was re-staked for its tungsten possibilities in 1958 by Karl Springer of the Mackenzie Valley Syndicate. Extensive exploration followed in the next three years. Canada Tungsten Mining Corporation Limited was formed in 1959, and construction of the mine complex was completed in 1962. Open pit mining continued until 1974 at the site of the original discovery. Underground mining of the newly discovered E-zone began in 1974. Milling rate was expanded in 1979, but the mine was forced to close twice during the early 1980s due to a strike and poor economic conditions. Owing to a collapse in the market for tungsten in 1986, the mine was closed but maintained with the hope of reopening in the future.

North American Tungsten Corporation Limited bought the property in 1997 and began to investigate reopening the mine. With a steady increase in the value of tungsten oxide and the financial backing of private buyers, the mine was reactivated early in 2002. The mine closed temporarily in December 2003 when the tungsten market took a brief nosedive and the company lost its tungsten buyer. The company was able to reorganize itself and find a new buyer, and the mine was reactivated during the summer of 2005.

Geology and Ore Deposits¹

The region is underlain by a thick sequence of late Proterozoic to Mississippian Selwyn Basin sediments, deposited primarily under subaqueous environments along the western margin of the North American craton. Deformation of sediments in the Cantung Mine area resulted in formation of a broad northwest-southeast trending syncline, with tight, overturned folding along the southwest limb. The Flat River extends along the fold.

The mine area is located along the southwest limb of the syncline, underlain by argillite and minor carbonate to the southwest and dolomite and quartzite to the northeast. These have been intruded by a small Late Cretaceous granodiorite stock. Intrusion of this stock resulted in metasomatism and resulting skarn mineral formation within two units of the country rock: the Ore Limestone and the Swiss Cheese Limestone, the former being the main ore host lithology. The Ore Limestone consists of blue-gray finely laminated recrystallized limestone and marble; the Swiss Cheese Limestone is a dolomitic siltstone with pods and lenses of impure limestone. These units, as well as enclosing upper and lower argillite units, have been folded into a recumbent anticline.

Major mineralized zones consist of massive replacement-style pyrrhotite within a gangue of calc-silicate skarn minerals, hosted within the Ore Limestone. Tungsten occurs as disseminated fine anhedral grains of scheelite within pyrrhotite and calc-silicates. Mineral emplacement was apparently controlled by bedding plane slips and cross fractures, particularly along hanging wall and footwall contacts of the Ore Limestone.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office. See **Figure 1**

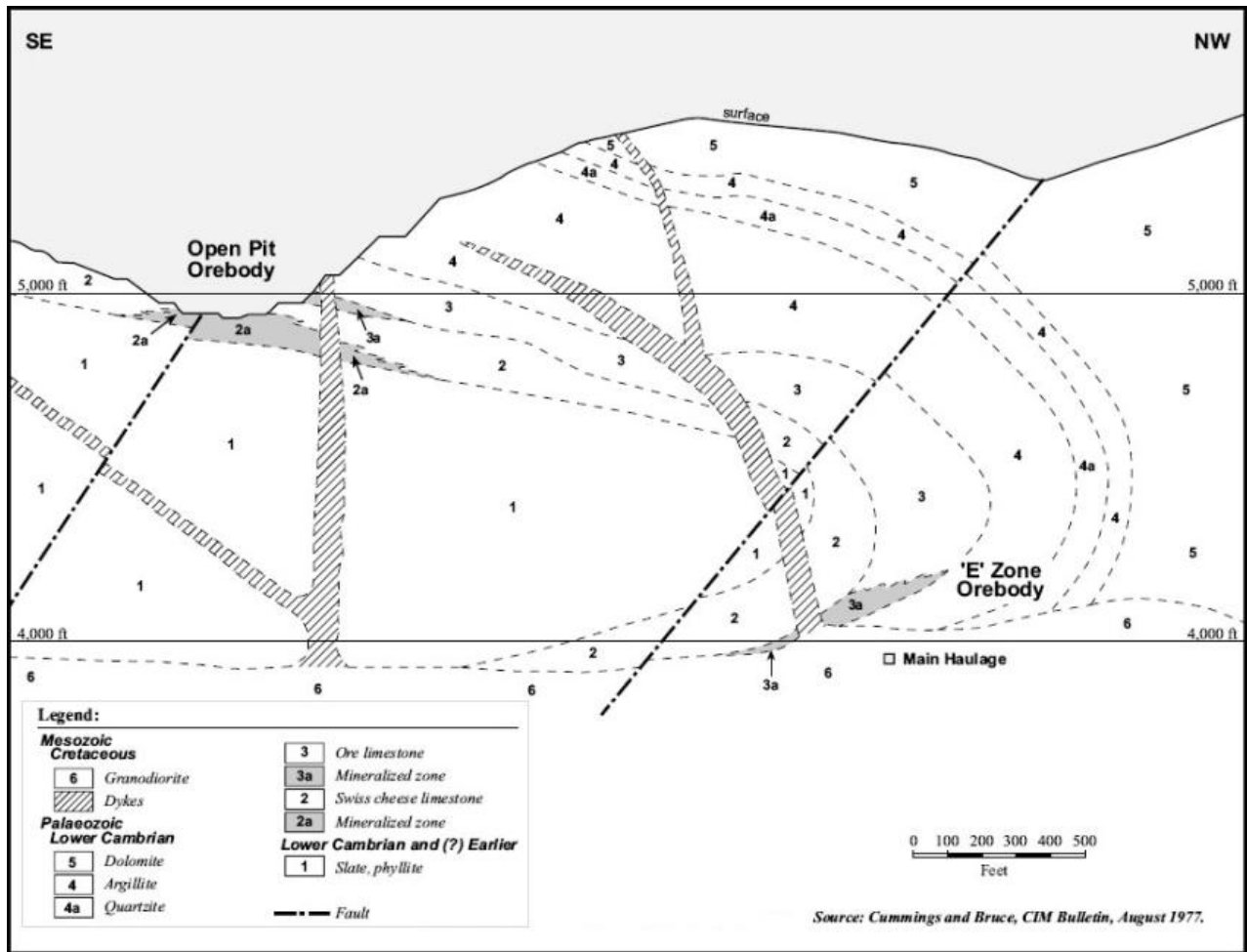


Figure 1. Geology longitudinal plan, Cantung Mine.

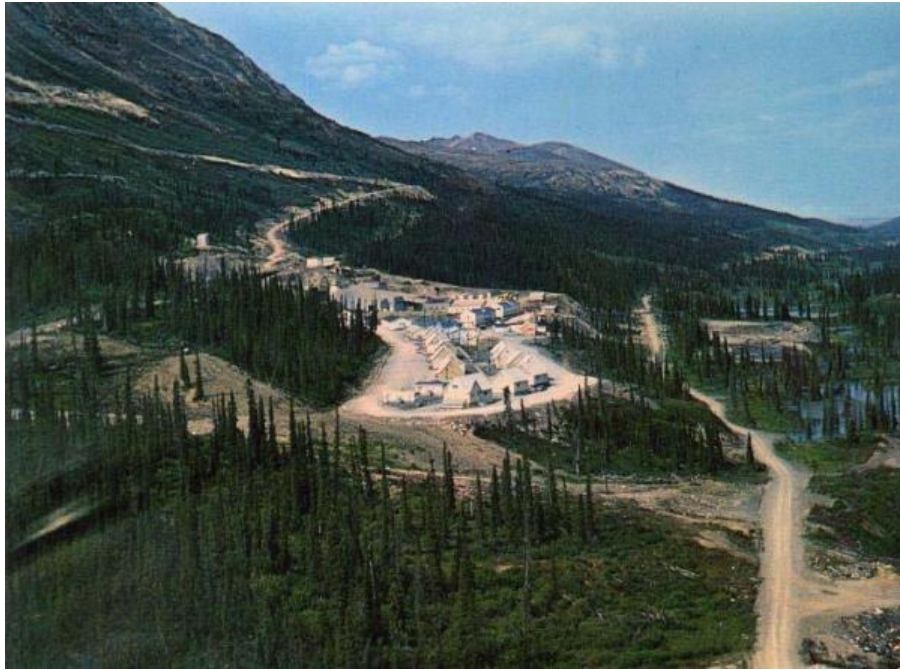
Two major ore bodies have been mined: the Pit ore body along the upper limb at an elevation of 1524 meters (5,030 feet), and the E-zone, about 550 meters to the north along the lower limb at an elevation of 1220 meters. The Main zone of the north-dipping Pit orebody measured roughly 200 x 90 x 20 meters thick. This was hosted by both Ore and Swiss Cheese limestones, is cut by a fault on its south side and pinches out to the north with no apparent structural cause. Pyrrhotite and scheelite occur within diopside-hedenbergite-garnet skarn with quartz, calcite and microcline. Ore mineralization includes chalcopyrite, with grades estimated at 0.50% copper; minor zinc as sphalerite, and minor gold and bismuth.

The E-zone occurs as an east-west striking tabular mass about 850 meters long and 12 meters wide, dipping at 20° to the south towards the Mine Stock. The contact with the stock is sharp, and the tenor of mineralization decreases up-dip from the contact. Mineralization occurs as numerous pods, lenses and sheets consisting of up to 80% pyrrhotite within barren limestone. Scheelite is variably disseminated within pyrrhotite zones, with grade generally proportional to pyrrhotite concentration. The E-zone is cut by four northeast striking, steeply dipping faults, with resultant displacements up to six meters, and some local drag folding.

Canada Tungsten Mining Corporation Limited (1960-1986)

Under a 1961 agreement involving American Metal Climax Incorporated, Dome Mines Limited, and Ventures Limited, money was raised to construct a tungsten mine in the Northwest Territories. Early exploration had indicated a reserve of 1,176,400 tons of ore grading 2.47% tungsten oxide (WO₃). During 1961, a short adit was driven on the B-zone, which is the east limb of the main ore deposit and a section that was difficult to probe by surface diamond drilling (The Canadian Mines Chronicle, June 16th 1961). Open pit mining began during the summer of 1962, and ore was stockpiled in preparation for milling operations. The mining contractor was Isbell Construction Company of

Canada Limited (incorporated October 1961, name changed in 1963 to Isbell Mining Company of Canada). Construction on the 200 mile access road between Watson Lake and Cantung began in 1961 and was completed in 1962. Until the road was completed, supplies, materials and equipment were transported by 'Cat Train' over a temporary winter-road from Watson Lake in 1961. An airstrip was also constructed along the Flat River for wheeled aircraft such as DC-3's and C-46's.



Canada Tungsten Mining Corporation Ltd.

Figure 2. *Cantung Mine and townsite, 1968.*

Stan Hunter was the first mine manager, hired in 1961. Staff under him, who were largely responsible for bringing the mine from construction to production, included Ken Sanders, geologist; Bill Burton and Rolf Lovlin, mill superintendents, Frank Jackson and Trevor Hancock, engineering; and Ted Maestretti, mine superintendent. Jack Crowhurst was general manager of Canada Tungsten Corporation Limited and Karl Springer was company president, both of whom also played dedicated roles in the opening of Cantung Mine (Jean Connor, pers. comm.).

First Stage of Operations

The plant was completed at the end of October 1962, and recovery for the year (November 1st to December 31st) amounted to 6,379 short ton units (stu) of tungsten oxide concentrates. The plant operated at a rate of 270 tons per day. Production operations during 1962-1964 were wrought with several difficulties, during which period the company conducted detailed metallurgical testing to refine and otherwise improve its processing flowsheet. Production was of a testing nature and several trial-lot batches were produced and shipped. Because of metallurgical problems and due to depressed conditions in tungsten markets, which saw prices drop as low as \$7.75 (US) per stu, the company suspended development in August 1963. The contract with the Isbell company was terminated as mining operations ceased. During the first stage in operations, tungsten recoveries were well below expectations and only in one month, July 1963, were the predicted recoveries and grade attained. The flotation circuit, which started in March 1963, performed very poorly, largely due to the high calcite contents of the ore. In 1963, an attempt was made to produce a copper concentrate from flotation machines, but this proved fruitless (Beaumont, et al., 1965).

Second Stage of Operations

The shutdown period provided additional time for the company to refine and redevelop its milling process. Mining resumed in July 1964 and, after extensive alterations (including the addition of several banks of gravity tables), test milling resumed at the end of September 1964. Production during the first four months were limited to the recovery of a gravitation concentrate as little headway was made in the production of an adequate flotation concentrate. Production

remained low at a recovery of 60%, primarily due to the problems with the flotation circuit and the inability to upgrade gravity concentrates to the desired level. Production of high-grade concentrates could only be achieved by sacrificing recovery. The lack of experienced crews and a high turnover rate plagued the operation, and the company endeavored to improve camp life in an attempt to stabilize its workforce (Beaumont, et al., 1965).

Commercial Production Attained

By June 1965, recovery rates were reaching acceptable levels and commercial production commenced. Both gravity and flotation concentrates were produced at grades satisfactory to meet market demands. Production of the low-grade reject concentrates ceased in July 1965 with the implementation of a satisfactory re-treatment system. Flotation concentrates were stockpiled to await a leaching plant to process them. The copper circuit was not functional until June 1966. During the mining season, 200,088 tons of waste and overburden were removed from the open pit area and 127,097 tons of ore grading 2.33% WO_3 was mined. Ore reserves at December 31st 1965 were reported as 919,917 tons of ore grading 2.49 % WO_3 . Construction in 1965 included additions to the townsite, consisting of renovations to houses and staff quarters and a recreation hall. These changes were hoped to improve living conditions and promote a sustainable workforce. A two-year labour contract was also negotiated with the miner's union (Canada Tungsten Mining Corporation Ltd. Annual Report, 1965).

Mining and milling operations attained a full year of commercial production in 1966. Production reached near capacity during the year at a rate of 300 tons per day, and recoveries and grades were on the rise. This was attributed to revisions in the ore-dressing processes and a more stable and skilled workforce. Copper production commenced on June 22nd 1966. Mining operations at the open pit continued normally during the summer months, when 404,702 tons of waste and overburden were removed and 102,877 tons of ore grading 2.40% WO_3 were mined and stockpiled. A study was conducted under the direction of A. Soderberg, an open-pit engineer, to determine the most economical mining of the orebody (Canada Tungsten Mining Corporation Ltd. Annual Report, 1966).

Vancouver Leaching Plant

A leaching plant was constructed in 1966 near Vancouver, BC, to treat the flotation concentrates from the mine. It was originally planned to erect the plant at Cantung, but the freight costs of shipping the required acids north, and the substantial manpower and power requirements dictated that the leach plant be constructed in the south. The plant was completed by year-end 1966, and received its first flotation concentrates in early 1967. They were treated in a hydrochloric acid leach process followed by filtering to remove calcite and apatite. Roasting of the leached concentrate at 800° F removed reagents and sulphur. In 1969, the plant was upgraded and flotation concentrate rose from 31% to 68% WO_3 content. Overall recovery at the Vancouver leaching plant was 97% in 1972 (Canada Tungsten Mining Corporation Ltd. Annual Reports).

Destruction of Mill by Fire

Production at Cantung Mine was going smoothly and with notable performance when, on December 26th 1966, the mill building burned down. The fire was believed to have been started by an over-heated air dryer in the concentrate drying section, and quickly consumed the original wooden building. No other buildings were damaged. The plant was fully covered by insurance, and the company immediately announced plans to rebuild.

Mining operations continued normally during the summer of 1967; meanwhile, construction of the new milling plant proceeded quickly. The new fireproof mill was completed in November 1967 and had a capacity of 350 tons per day. (Canada Tungsten Mining Corporation Ltd. Annual Report, 1967) Production resumed in late November 1967 with the completion of the new milling plant. During 1968, tungsten and copper recoveries saw significant improvement over previous years. The plant performed well during the year; however, beginning in September 1968 and throughout the remainder of the year, ore feed consisted of a mixture of chert and skarn ore, the hardness of which was greater than anticipated. This negatively affected the grinding rate. In order to fix this problem, a secondary ball mill was installed in the mill in December 1968. Open pit mining operations continued normally during the summer of 1968, with 250,000 tons of waste and overburden removed and 131,000 tons of ore grading 1.71% WO_3 and 0.25% copper mined and stockpiled (Canada Tungsten Mining Corporation Ltd. Annual Report, 1968).

Mining Operations

Because of the cold and inhospitable climate, mining operations at the open pit were restricted to the warmest months of the year between June and October. Operations afforded a stockpile of ore that could last during the winter, and the mill operated year round. Ore was mined using three Gardner-Denver air-track drills operated by Atlas-Copco air

compressors and collected in two Bucyrus-Erie (71B and 80D models) diesel shovels. Ore was hauled in six 22-ton Euclid haul trucks and two Mack trucks (Cummings and Bruce, 1977).

The original mining plan at Cantung called for both open pit and underground mining of the open-pit orebody. This plan was changed in 1967 to eliminate the underground mining, necessitating a reduction in the ore reserve at the end of the year. It was believed that new benching methods would allow for the mining of ore that would previously have been accessible only through underground workings. Of course, with the later discovery of the E-zone in the early 1970s, the mine would witness a period of underground mining anyway (Canada Tungsten Mining Corporation Ltd. Annual Report, 1967).

Power Plant

In 1968, the mine was powered by three D398 Caterpillar 600 kilowatt diesel generators. In the mid 1970s, four additional D398 units were added due to the increased power needs brought forth by underground mining operations. Heat was supplied by a 1,900 horsepower (hp) Cleaver-Brooks electric boiler (Cummings and Bruce, 1977).

Employees

During the 1960s, the population fluctuated on a seasonal basis. 60 employees operated the milling and other surface facilities year-round, but in the summer months 40 additional positions were created for open pit mining. Married employees lived in family housing and single men lived in bunkhouses. While the mine suffered from high turnover in the mid to late 1960s, by 1970 improved living conditions and benefits resulted in the creation of a more stable workforce (Cummings and Bruce, 1977). A list of mine managers throughout the history of Cantung is in **Table 1**.

Years:	Manager:	Years:	Manager:
1961-1962	Stan Hunter	1976-1978	Hal E. Pawson
1962-1966	Roy Lambert	1978-1982	Martin Swizinski
1966-1969	Ed Hoddinott	1982-1986	Dave Armstrong
1969-1972	John Keily	1986	Jack Devitt
1972-1974	John K. Kervin	2002-2003	Mike Redfern
1975-1976	Kelson G. Collins	2005-2007	Dennis Bergen/Wayne Osborne

Table 1. Mine managers at Cantung Mine. (source: Canada Tungsten Mining Corporation Ltd. Annual Reports)

Tungsten Townsite

A townsite, which became known as “Tungsten”, was under construction during the summer of 1961 to accommodate workers and a small amount of families. It originally consisted of several small bungalow houses. Total population of Tungsten during the 1960s was approximately 120 persons, including about 27 families. In 1968, families were housed in 28 units (single and duplex housing). Because of extremely good wages and benefits, turnover rates for the entire operation were quite low by 1968. Families benefited from the K-8 Grade school, and later a K-9 system. During the summer months, because of the open pit operation, manpower and townsite population grew to 160. In the mid 1970s, the townsite expanded to include a trailer court, three condominiums, bunkhouse trailers, and in 1982, a modern recreation complex. The town and mine were serviced with an all-weather road to Watson Lake, Yukon and a 3700 foot x 100 foot airstrip (Cummings and Bruce, 1977).

The company enjoyed a profitable year in 1969 as tungsten demands remained very strong. A slight increase in the price of tungsten allowed for the economical mining of lower grade ores in the open pit. Increased milling rates were also made possible by the addition of new grinding equipment in 1968, resulting in an increased tungsten recovery rate of 78.8% in 1969. During the mining season, 146,258 tons of ore were mined, crushed and stockpiled, containing an average of 1.56% WO₃. In addition, 265,720 tons of waste rock was removed from the open pit area. Ore reserves at December 31st 1969 were 733,823 tons grading 1.68% WO₃. There was also 84,058 tons of ore in the stockpile grading 1.56% WO₃ (Canada Tungsten Mining Corporation Ltd. Annual Report, 1969).

E-Zone Discovered

In 1971, the E-zone was discovered underneath the Open Pit, extending the Cantung reserves to greater depth. The mine carried out deep surface drilling during the summer of 1972 to define the new orebody. The E-zone lies about 1,800 feet north and 500 feet below the open pit (which bottoms at 4,900-foot elevation). Development of an underground adit at 3,950-foot elevation was begun in October 1972 and had advanced 437 feet by year-end 1972. Canadian Mines Services Limited was the development contractor. The portal was 14 feet x 16 feet in dimensions. It was planned to extend the adit a distance of 4,000 feet to intersect the E-zone, but the zone was encountered much sooner than expected at only 2,000 feet distance. The adit was driven 4,800 feet during 1972-1973. An incline was driven to 4,410 feet elevation to act as a ventilation passageway. The E-zone was found to extend over a length of 1,700 feet at an elevation 150 feet above the portal level. It was mineable in widths of 500 feet or more. Underground drilling indicated a reserve of 4,242,000 tons of ore grading 1.68% WO_3 and 0.22% copper. The deposit was still open to the west, but full investigation of this area was not to be initiated until later. (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1971-1972).

Production operations in 1972 were focused on the mining of ore in the open pit. Increased chert-type ore of refractory nature was encountered on the lower benches, causing metallurgical difficulties and limited milling capacity. Tungsten grades also dropped because of the inclusion of low-grade chert material. Some mill circuit changes were made, but grades did not improve very well. No copper concentrates were sold during 1972; material was stockpiled to await higher commodity prices (Canada Tungsten Mining Corporation Ltd. Annual Report, 1972).



Canada Tungsten Mining Corporation Ltd.

Figure 3. *Underground mining, 1983.*

Open Pit Mining Ceases

As a result of the new underground exploration and the lower-grade chert ores encountered in the open pit, it was decided in 1973 to divert all production operations at Cantung from open pit to underground mining (Canada Tungsten Mining Corporation Ltd. Annual Report, 1973). The open pit provided 1,346,000 tons of mill feed between 1962 and 1974, with a grade of 1.64% WO_3 . 1,700,000 tons of overburden and waste rock was also removed during open pit operations. Dimensions of the pit are estimated as 300 meters x 150 meters and 100 meters deep. Mineable reserves in the open pit, calculated in 1985, were estimated at 966,000 tons of ore grading 0.66 % WO_3 (Roscoe Postle Associates Inc., 2001).

Production in 1973 was focused on the mining of remnant high-grade skarn-type ores in the open pit in preparation for the switching to underground mining. Driving of the main adit tunnel ceased early in 1974 after 4,300 feet of advance. Underground work continued with the preparation of stopes, a ventilation raise, and access ramps into the hanging wall of the deposit (Canada Tungsten Mining Corporation Ltd. Annual Report, 1973).

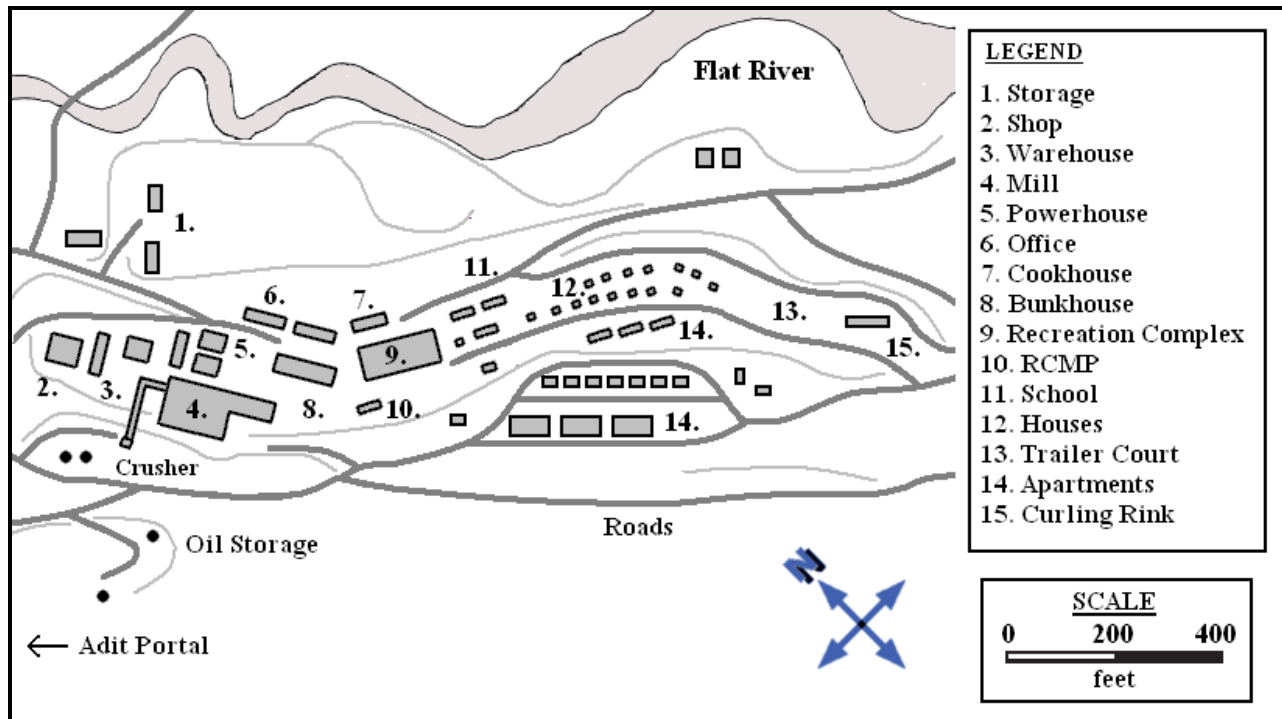


Figure 4. *Cantung Mine property plan, c.1982.*

Initiation of Underground Production

First production of ore from the E-zone through the underground workings was accomplished in February 1974. Milling of stockpiled open pit ore from the 1973 mining season was completed on June 15th 1974, after which all production was from underground operations. Mining operations at Cantung were now a year-round affair, unlike the previous open pit operations. E-zone ore was found to be high in talc content and as such the mine experienced problems in tungsten recovery. Research began on how to best combat this problem. Two illegal strikes in November 1974 disrupted operations briefly, but they were settled with the negotiation of a new labour agreement on November 28th 1974 (Canada Tungsten Mining Corporation Ltd. Annual Report, 1974).

It cost about \$5 million to bring the underground workings into production, which included expanding the mill to 600 tons per day capacity. Ore was mined through room and pillar method (drifting of the hanging wall to outline 30 foot rooms and 40 foot pillars, with sublevels at 50 foot vertical intervals). 20 foot benches were blasted, with completed areas filled with waste. Stope pillars were mined by longhole stoping from below. In 1977, the mine operated at 600 tons per day, 6 days per week to keep the mill running at 500 tons per day, 7 days a week. Mining was concentrated between the 4,100- and 4,150-foot levels with six stopes in production during 1977. Underground ventilation was provided by 140,000 cubic feet per minute intake at the ventilation portal at 4,410-foot elevation. Air was heated with a propane burner. An underground conveyor was excavated during 1973-1974 to bring ore to the surface portal at faster rates (Cummings and Bruce, 1977). Total underground development to 1979 is listed in **Table 2**.

Equipment 1970s

Equipment associated with underground production and ore haulage at Cantung featured two 35 ton Euclid R22 trucks, two 2-boom Atlas-Copco drill jumbos, two Jarco RBM-11 rock bolt jumbos, three ST-5A Wagner 5 yard scooptrams, one HST-1 Wagner scooptram, two G-D airtrac drills, and three 900 cubic feet per minute Atlas-Copco electric compressors (housed in the underground portal shop) (Cummings and Bruce, 1977).

Construction during 1974 included 18 additional townhouses, a reagent storage building, firehall, mine office and dry, powerhouse extension, and an oil tank farm dyke. Construction in 1975 included 12 mobile homes, a 48 man bunkhouse, new assay office, new cold-storage warehouse, and upgrading of the site roads (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1974-1975).

The underground ore was considered to have better milling characteristics than the open pit deposit and the company hoped to show a greater profit. Higher metal prices and higher-grade ores from this zone provided initial enthusiasm. During 1975, however, recovery dropped significantly (71%) due to separation problems in the concentration circuit as a result of treating talc-rich underground ore. This material affected the recovery rates of flotation concentrates. While the problem was investigated, mining shifted to areas of the mine lower in talc. Lower tungsten sales and increasing costs associated with the more expensive methods of underground mining also had a negative effect on operations during 1975. Other notable events of 1975 included the issuing of the company's first Water License as per new rules and regulations concerning water and land use in the Northwest Territories. (Canada Tungsten Mining Corporation Ltd. Annual Report, 1975)

Improvements to the mill during 1976 solved the low recovery problems, which increased to 82% from 71%. Higher commodity prices of tungsten also improved operational conditions, which allowed for the mine to economically operate at 98% capacity as opposed to only 92% capacity in 1975. Extensive diamond drilling and drifting on the western extension of the orebody during 1976-1977 proved additional tonnage, and ore reserves at December 31st 1977 were 4,200,000 tons grading 1.55% WO₃. The copper circuit was operated periodically during 1976 after having been shut down during 1975. However, metallurgical problems in the circuit dictated the permanent cessation of copper recovery at the end of the year, and after 1976 no more copper was being produced at the Cantung Mine.

Construction in 1976 included four additional mobile homes, another powerhouse extension, addition to the school, addition to the local store, renovations to the post office, a new laundry building, a new warehouse, a garage, and new roofs for ten of the houses. An oil-fired boiler was installed and the powerhouse diesel engines were overhauled and repaired (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1976-1977).

Mine Expansion

During 1976, the company announced a CDN \$10 to 12 million program to double the operational capacity of the mine and mill at Cantung and the leach plant in Vancouver. Operations in 1977 were focused largely on the mine expansion program, which started in June of that year. Other construction in 1977 consisted of the start of a new powerhouse building, a new water system, improvements to the cookery and warehouse, a new maintenance shop, a 60 man bunkhouse, portable classroom, a church, and new trailers. Mining operations were normal with the exception of a short wildcat strike between August 25th 1977 and September 6th 1977. Operations resumed on September 15th 1977 (Canada Tungsten Mining Corporation Ltd. Annual Report, 1977).

Construction during 1978 consisted primarily of the mill expansion program, which necessitated a new primary crusher, 1,000 ton coarse ore bin, 500 ton fine ore bin, secondary crusher, and new screens and conveyors. The powerhouse building was completed in 1978, and other construction included nine new houses, and additions to the recreation hall and cookery (Canada Tungsten Mining Corporation Ltd. Annual Report, 1978).

The new mill was completed and in operation on July 25th 1979 with rated capacity reached by October 1979. The mine produced lower-grade ores during the year, but the increased tonnage and higher recoveries helped offset the input grades. The mine expansion program was completed at a cost of CDN \$30 million. Mill additions included a new primary crusher, a new rod mill, more cyclones, modern flotation cells, two thickeners, and a new 100 foot diameter multiple hearth roaster. Capacity was now 1,000 to 1,200 tons per day. Other construction in 1979 consisted of a new mine office and dry, an underground repair shop, two 150,000 gallon oil tanks, and the placing of seven houses on permanent basement foundations. 800 feet of the main adit tunnel were paved and a 150,000 cubic feet per minute ventilator fan was installed. Other notable work in 1979 was the paving of 2,500 feet of road at the townsite and the inauguration of "live" television service (Canada Tungsten Mining Corporation Ltd. Annual Report, 1979).

New Power Plant

A new power generating facility was commissioned in March 1979 consisting of two 2500 kilowatt Mirrlees diesel generators and a 1,500 cubic feet per minute air compressor.

Employees and Staff

Management staff at the Cantung Mine in 1979 included Peter Cain, vice-president of operations; Martin Swizinski, mine manager; M.F. Lindsley, production superintendent; P. Bouma, mill superintendent; J.G. Connor, plant superintendent; D.A. MacKinnon, underground superintendent; L.G. Daigle, exploration manager; and L.T. Chrisholm, purchasing agent (Canada Tungsten Mining Corporation Ltd. Annual Report, 1979). In 1982, mine operations employed 52 men, mill operations 51 men, with a total of 240 employees (The Western Miner, July 1982).

Upgraded Milling Plant 1979

Underground ore was dumped in a 30 ton receiving bin and primary crushed in a 42 inch x 48 inch jaw crusher. Crushed ore was transferred to a 1000 ton coarse ore bin for secondary crushing in a 4.25 foot Symons cone crusher set at 1.5 inch. Crusher discharge was screened on a single-deck Dillon screen with 5/8 inch x 5 inch slots. Screen oversize was crushed in a second 4.25 foot Symons cone crusher set at 3/8 inch, in closed circuit with the Dillon screen. Screen undersize was conveyed to two 800 ton fine ore bins. Fine ore was fed to a 9 foot x 12 foot Allis-Chalmers primary rod mill charged with 3 to 4 inch diameter rods. Mill discharge was fed to two 15 inch diameter cyclones, the overflow passing to talc flotation and the underflow to 24 trommel screens equipped with 48 mesh cloths. Screen oversize returned via a screw classifier for secondary grinding in a 7 foot x 10 foot Hardinge rod mill that discharged (along with classifier overflow) back into the primary grinding circuit. Screen undersize was de-watered in a 40 foot triple-tray thickener and sent to sulphide flotation.

The talc flotation circuit was comprised of eight #24 Denver DR flotation cells. Fine sulphides and serpentine contamination were floated and rejected to tailings, while the remainder of the material passed to two 40 foot triple tray thickeners and sent to scheelite flotation. The sulphide flotation circuit consisted of 13 Agitair 100 cubic-foot flotation cells, which received contaminated material from secondary grinding. Tails were forwarded to gravity separation while flotation concentrates containing sulphides were removed and rejected as mill tailings. Sulphide flotation tails were distributed to ten Deister triple-deck shaking tables. Rougher table concentrate was cleaned on seven single-deck tables, while rougher table middlings were fed to three single-deck tables and re-cleaned on one single-deck table. Rougher table tails from the center part of the table were classified in a 10 inch cyclone, the overflow going to the two 40-foot thickeners ahead of scheelite flotation, and the underflow rejected as mill tailings. The outer portion of the rougher table tails was classified in two secondary 15 inch cyclones, the overflow going to the scheelite flotation-feed thickeners and the underflow to a 7 foot x 6 foot Marcy ball mill. Ball mill discharge was treated in eight #21 Denver DR flotation cells for removal of sulphides. Flotation tails were distributed to four triple-deck rougher tables and cleaned on four single-deck tables. Primary and secondary cleaner table tailings passed through the secondary cyclones and middling products were treated over individual single-deck tables. All cleaner table concentrate products continued to a de-watering screw ahead of roasting and magnetic separation.

The operation produced high quality gravity and flotation grade scheelite (CaWO_4) concentrates, which contained WO_3 as the marketable product. The mine shipped high-grade scheelite concentrates to customers in the United States, Western Europe and other destinations. Lower grade concentrates were shipped to a plant in Fort Madison, Iowa for further processing into ammonium paratungstate (APT). In addition, the company purchased outside concentrates, depending on market conditions, for conversion to APT. The company sold APT, ammonium metatungstate (AMT), and minor amounts of other products to customers in the United States (Canada Tungsten Mining Corporation Limited, 1985).

Operations 1980s

A 6-month labour strike, beginning in November 14th 1980 and continuing until May 14th 1981, affected operations during the early 1980s. Tungsten production in 1980 was at record rates despite the month and a half of lost operations, due to the expanded mine and milling plant. Construction in 1980 included the start of a new recreation center, the start of a new assay lab, and townsite paving. New mining equipment purchased during 1980 included an electric hydraulic jumbo drill, a powder loader, and two scooptrams. The strike was settled on May 14th 1981 and production resumed in late June 1981 (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1980-1981).

During 1981, the mine facilities were expanded and improved. New mobile mining equipment was placed into operation. The new recreation center was completed together with three new apartments. The shops were upgraded, a new assay lab was built, a chlorinating plant was installed to correct tailings discharge pollution, and the mill was given an improved copper recovery circuit. Copper was produced as a byproduct and was stored pending better prices. Other changes to the mill in 1982 allowed for the reduction of impurities, the elimination of cyanide as a reagent, and increased recoveries from lower-grade ores. Canada Tungsten Mining Corporation Limited entered into an agreement with AMAX Incorporated in 1980 to supply AMAX with low-grade tungsten concentrates. Shipments of concentrates to AMAX's APT refinery in Fort Madison, Iowa commenced in 1981 and accounted for approximately 30% of sales during the year (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1981-1982).

'Tungsten' Townsite 1980s

In 1981, the townsite was modernized through the opening of a new recreation center (consisting of an indoor pool, curling rink, bowling alley, racquet court, gym, store, lounge) and three new 21-unit apartment buildings. A new 80

man bunkhouse was built in 1983. In 1979, it was estimated that there were 506 people living at Tungsten, 200 of which were employees. There were 450 residents of Tungsten in 1982 (The Western Miner, July 1982). About 100 children were enrolled in the school in 1982. By 1986, it was estimated that only 280 people were living at Tungsten.

Depressed tungsten markets caused problems at the mine in 1982. At mid-year production was reduced about 15% in response to the poor conditions, and on January 22nd 1983 the company laid off the employees and shutdown the mine. The action was temporary, but lasted throughout most of the year. During this time the company upgraded the mine operation to 1,200 tons per day capacity by instituting new mining plans aimed at improving efficiency and reducing costs. On November 30th 1983 reactivation of the plant began and Cantung Mine restarted operations at about half capacity on December 1st 1983. Full capacity was achieved in August 1984. During the 1983 closure, programs were initiated to improve operating efficiencies, including high productivity mining methods, which resulted in a 20% reduction in the work force upon reopening (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1982-1983).

Mining Operations 1980s

In 1982, the mining methods at Cantung changed. The original room and pillar method proved unfeasible in new areas of the mine because of ground conditions. The E-zone was found to become steeper as it moved west, which resulted in fragile pillars that were too costly to maintain. The mine switched to cut-and-fill methods, pumping mill tailings underground. This change was also favorable to milling operations as it meant that 70% of the tailings could be deposited underground rather than on the surface. (The Western Miner, July 1982) The mill was modified to allow for the pumping of backfill underground.

A raise was driven from the underground workings up to the open pit elevation in 1982, and waste rock from the open pit was used for additional stope fill. Ore was brought out of the mine by two 35 ton DJB and two 22 ton Euclid trucks. In 1982, other underground equipment included four Atlas-Copco jumbo drills, three Jarvis-Clark rock-bolting jumbos, and five Wagner 5 yard scooptrams (The Western Miner, July 1982). Highest workings at Cantung were at 4,200-foot elevation. During 1983-1984, some underground units were converted into remote controlled units and drilling was conducted using large diameter drills as part of a bulk-mining initiative. In 1985, 80% of millfeed was derived from the remote controlled machinery using bulk-mining methods. Other methods used, where required, consisted of cut and fill and top-slice horizontal bench method (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1982-1985).

Exploration Programs

Beginning in 1983, extensive surface and underground diamond drilling was undertaken to evaluate ore reserves in the West Extension zone. 16,000 feet of drilling was accomplished in 1983; in 1984, an additional 37,100 feet of drilling was performed in a 700 foot extension of the zone. Diamond drilling intersected a water-bearing fault zone in 1985 and exploration within the West Extension zone was scaled back, and then halted following 1986 closure (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1983-1985).

Events Leading to Closure

Bad market conditions again had an effect on Cantung during 1985. Lower-grade ores were encountered in the first half of the year, and while higher-grade ores were being mined in the later half, operating costs continued to escalate together with the worsening state of tungsten prices. A new management team was hired to help solve these problems, and a revised mine plan was completed in December 1985. Production was reduced to 65% of plant capacity through the layoff of 50 workers. In early 1986, cheap Chinese tungsten products flooded the open market, driving the price of tungsten down to record lows. The company was forced to make cutbacks including 12% wage cuts and 25% benefit cuts. The Union took strike action once again, forcing a stop to production on May 20th 1986. The company decided against settling the strike and laid off the entire 210-person workforce and mothballed the townsite (Canada Tungsten Mining Corporation Ltd. Annual Reports, 1985-1986).

Production and Development Summary 1962-1986

Cantung Mine production between 1962 and 1986 was 4,386,837 tons milled to produce 5,313,159 stu of tungsten oxide concentrates and 1,325 tons of copper (see **Table 2**).

The mine is serviced by underground workings that total approximately 20,000 feet on two main levels (3,950-foot portal elevation and 4,200-foot elevation). A ramp has been driven above the workings to provide a vent portal and escape way that surfaces at the 4,410-foot elevation above the main portal entrance. Seven stopes were mined by open stoping, cut and fill, and mechanized room and pillar methods. The West Extension zone was explored on the portal

level and on a sub-level. There is also a small entrance at 3,750-foot elevation that serves workings within what is known as the B-zone, an area to the east of the open pit. This adit was apparently driven in 1960.

Year:	Ore Milled:	Ore Grade:	Tungsten Concentrate Produced:	Copper Produced:
1962-1963	71,824 tons	2.74 % WO ₃	125,820 stu	-
1964	33,543 tons	2.57 % WO ₃	51,400 stu	-
1965	107,651 tons	2.53 % WO ₃	200,000 stu	-
1966	115,568 tons	2.43 % WO ₃	213,022 stu	167 tons
1967	7,778 tons	2.36 % WO ₃	13,380 stu	24 tons
1968	116,558 tons	1.98 % WO ₃	180,000 stu	298 tons
1969	167,389 tons	1.54 % WO ₃	203,174 stu	233 tons
1970	176,816 tons	1.39 % WO ₃	186,340 stu	183 tons
1971	181,596 tons	1.19 % WO ₃	164,420 stu	121 tons
1972	172,828 tons	1.15 % WO ₃	158,706 stu	113 tons
1973	164,900 tons	1.23 % WO ₃	161,430 stu	99 tons
1974	170,614 tons	1.45 % WO ₃	177,880 stu	82 tons
1975	179,032 tons	1.28 % WO ₃	162,892 stu	-
1976	188,934 tons	1.55 % WO ₃	238,998 stu	5 tons
1977	185,629 tons	1.65 % WO ₃	251,813 stu	-
1978	194,740 tons	1.96 % WO ₃	318,085 stu	-
1979	271,559 tons	1.62 % WO ₃	361,015 stu	-
1980	350,208 tons	1.55 % WO ₃	441,664 stu	-
1981	234,126 tons	1.40 % WO ₃	277,249 stu	-
1982	360,618 tons	1.28 % WO ₃	394,202 stu	-
1983	39,685 tons	1.19 % WO ₃	38,584 stu	-
1984	338,872 tons	1.43 % WO ₃	386,400 stu	-
1985	381,211 tons	1.35 % WO ₃	409,868 stu	-
1986	151,324 tons	1.56 % WO ₃	196,817 stu	-
Total:	<u>4,363,003 tons</u>	<u>1.68 % WO₃</u>	<u>5,313,159 stu</u>	<u>1,325 tons</u>

Table 2. Cantung Mine production, 1962-1986. Copper recovery ceased in 1976. (source: Canada Tungsten Mining Corporation Ltd. Annual Reports) *stu* = short ton units

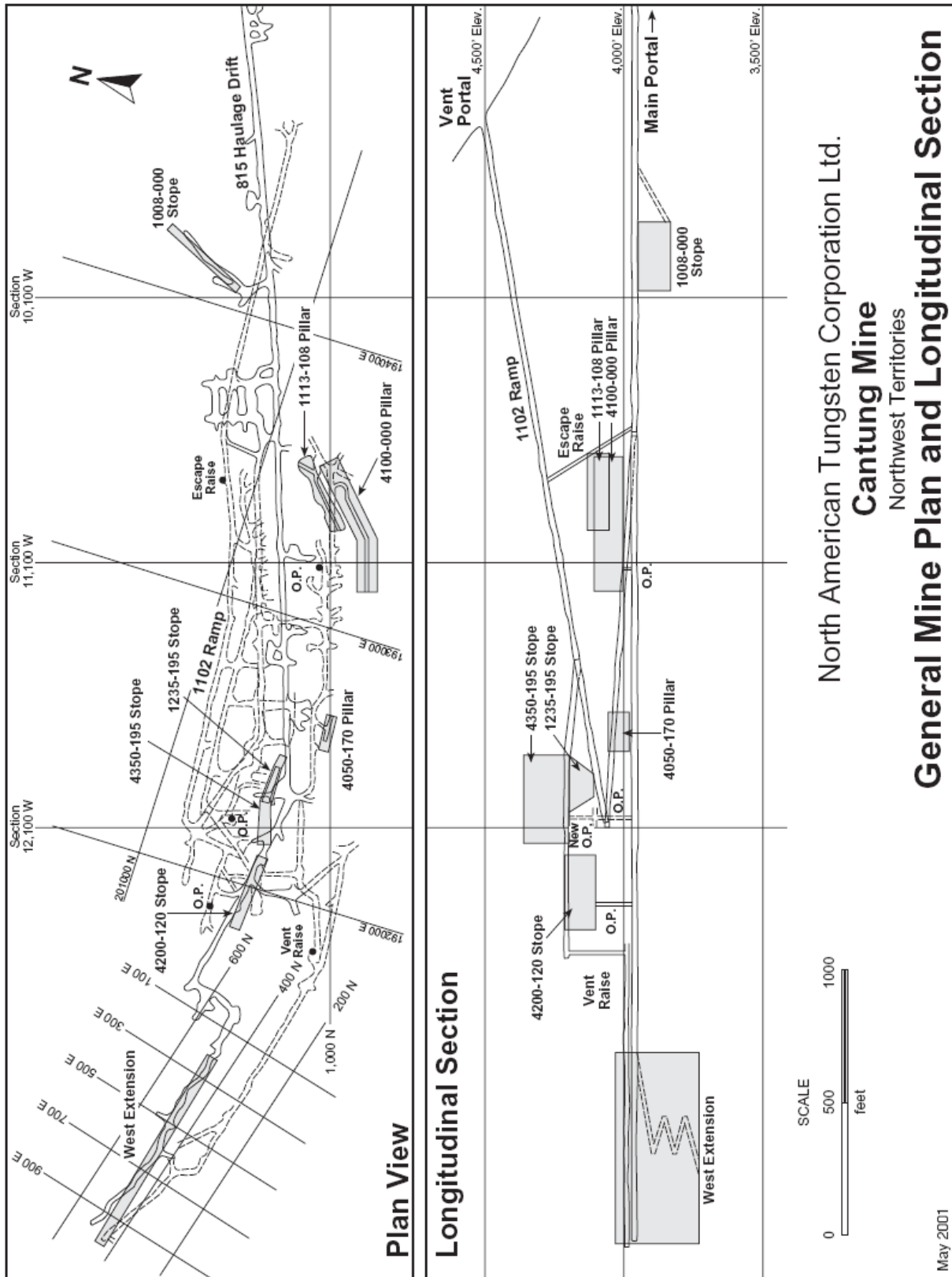


Figure 5. Cantung Mine underground plan, 2001. (Roscoe Postle Associates Inc., 2001)

North American Tungsten Corporation Limited (2002-current)

In July 1997, North American Tungsten Corporation Limited purchased the tungsten assets of Aur Resources Limited (acquired control of Canada Tungsten Mining Corporation in 1992). These assets included Cantung Mine, and North American Tungsten continued to maintain the site in care and maintenance mode pending the recovery of tungsten markets. During 2000-2001, world tungsten prices began to increase in response to limits placed on Chinese tungsten exports and the perception that world demand would not be met with this decrease in supply. In response to this, the company made plans to bring the long-dormant Cantung Mine back into production. A buyer for tungsten concentrates was found in March 2001. AB Sandvik Coromant of Stockholm, Sweden and Osram Sylvania Products Incorporated of Towanda, Pennsylvania were to purchase tungsten concentrates from North American Tungsten Corporation Limited, at a projected rate of 300,000 metric tonne units (MTU) per year and at prices of \$60 per MTU, well above the market price for tungsten products. Reserves at the time suggested enough ore for 900,000 MTU tungsten concentrates. Sandvik and Osram advanced a CDN \$4.5 million repayable loan to North American Tungsten that was applied towards the start-up of the mine (North American Tungsten Corp. Ltd. Quarterly Report, Mar. 31st 2001).

Rehabilitation of the mine complex began in July 2001, leading to a restart of the operation in December 2001. A crew of 130 employees was mobilized in December. Originally it was planned to contract all work out, but due to operational and economic issues the company hired its own workforce and its own fleet of equipment (North American Tungsten Corp. Ltd. Quarterly Report, Sept. 30th 2001). First ore from the mine was being crushed on January 21st 2002 and milling operations were at a rate of 500 tons per day. On February 18th 2002, the first batch of concentrates (1,745 MTU WO₃) was shipped to outside markets. During the start-up period from January 21st to March 30th 2002, 40,141 tons of ore were milled to produce 36,348 MTU of tungsten oxides. The mine achieved full production at 950 tons per day in April 2002, producing over 1,000 MTU of concentrate per day. Recovery was increased to 70% and was anticipated to increase even higher once the equipment and employees were 'broken-in' (North American Tungsten Corp. Ltd. Press Releases, Apr. 23rd 2002; June 4th 2002).

Mining Operations

During the 1980s, the mine utilized open stoping methods. It was found that this method increased dilution in the ore feed and reduced recovery, therefore during 2002-2003 the company employed cut-and-fill methods. Long-hole mining methods were introduced late in 2005 to recover old stope pillars.

Milling Operations

The mill operated with the same flowsheet as it did during previous operations at Cantung. It had a maximum capacity of 1,200 tons per day. It employed a gravitational and flotation flowsheet that included primary and secondary crushing following by grinding and classification, and talc-sulphide flotation. Gravity concentrates were roasted and exposed to magnetic separation, followed by re-tabling to separate high and low-grade gravity concentrates. After the start of commercial period of production at Cantung in April 2002, low-grade concentrates were returned to the mill circuits and not shipped separately. The mill produced two marketable concentrates: a high-grade gravity concentrate and a flotation concentrate, which made up 80% and 20% of the final product respectively on average. Concentrates were trucked from the mine to Edmonton, Alberta, and then shipped to a smelter in Pennsylvania.

Employees

Total workforce early in 2002 was approximately 140, 50% of which were northerners, and of that, 35% were aboriginal. The company also estimated that 20 direct contact jobs were created by company mining operations. In May and June, direct employment at the mine site was increased to 164 employees. The old Tungsten townsite remained largely vacant and only the old 80 man bunkhouse complex was put to use. Demolition of old houses and redundant structures were to proceed during the operation in anticipation of future closure. Mine manager at Cantung during 2002-2003 was Mike Redfern.

During the summer of 2002, new power-generating units were purchased to accommodate increased production, and several pieces of the mining fleet were replaced. Power was now supplied by six Cat generators, rating 5.3 megawatts in total. Diamond drilling into the West Extension zone was planned and it was hoped that new high-grade resources in this section would add life to the mine (North American Tungsten Corp. Ltd. Press Release, Nov. 22nd 2002). Late in the year, production of concentrates exceeded that of the negotiated contract with Sandvik and Osram, and concentrates were stockpiled. One solution was to sell the excess products on the open market, but the market price for tungsten was much lower than the negotiated contract price. A reduction in production was deemed the wisest course of action in order to preserve the mine's resources and maximize onsite employment. In order not to exceed the sales quota and to optimize overall site costs, the mine suspended production on March 6th 2003 for a period of five weeks.

The downtime was utilized to maintain and clean fleet equipment. Production resumed April 10th 2003. During the summer, progress was being made on an extensive diamond-drilling program to probe the West Extension zone. The effect of a strong Canadian dollar against the U.S. dollar had a negative effect on profits for the company during 2003. In addition to this, tungsten markets failed to improve and remained largely fixed at \$46 US per MTU (North American Tungsten Corp. Ltd. Quarterly Reports Mar. 31st, June 30th, Sept. 30th 2003).

2003 Closure

In November 2003, Sandvik and Osram terminated the 2001 agreement to purchase the companies tungsten concentrates and demanded repayment of the advanced loans. This effectively terminated all operations at the Cantung Mine, and mining ceased on December 2nd 2003. Processing of ores continued until December 4th. 170 employees were laid off. Remaining concentrate stockpiles were sold to third party interests early in 2004 (North American Tungsten Corp. Ltd., Quarterly Report Dec. 31st 2003). Production during the period 2002-2003, including the pre-commercial production stage (January to March 2002), was 614,935 tons milled from E-zone mining operations at a grade of about 1-65% tungsten oxides, to produce concentrates totaling 772,778 stu (701,052 MTU)

Restructuring of the company took place during 2004 with hopes of finding a new market and resuming production operations. In November 2004, North American Tungsten Corporation Limited entered into an agreement with Kaska Minerals Corporation Limited in which Kaska Minerals, an aboriginal company in the Yukon, would invest nearly CDN \$3 million to bring the mine back into production. The company will become a major stakeholder in the economic and social benefits relating to the operation of the tungsten mine and an envisioned smelting facility, which is to be built near Watson Lake.

Reactivation 2005

In response to the continued strength of the tungsten market and expressions of interest from tungsten consumers, the company announced plans to reopen Cantung Mine. Reactivation of the property began in June 2005 with rehabilitation of plant and camp facilities and the underground workings. Three new Cat diesel generators were installed in the power plant. As many crews as were laid off in 2003 were returned to work, but it was found many of the management positions needed to be re-filled. Dennis Bergen was brought aboard as new mine manager in June 2005. In August, ore was being stockpiled from both the open pit and underground development in preparation for production. When production began on September 6th 2005, 35,000 tons of ore were ready for milling. Commercial production commenced on September 22nd and the first shipment of concentrate was made on October 1st 2005. (North American Tungsten Corp. Ltd., Quarterly Report Sept. 30th 2005)

Ramp-up to full production continued through to the end of the year. Full-scale production was delayed due to the inability to drive mine development into the higher-grade areas, and a Yukon based mining contractor was hired to assist the company in underground development. Mining operations were enhanced with the introduction of long-hole methods and the addition of new equipment. Access to seven long-hole pillars which were hoped to increase tonnage and grade in mill-feed was expected by March 2006. Mill feed was derived from surface stockpiles of underground ore and old open pit ore. Improvements in mill operation were made by automating a number of components and process controls (including pumps and flow lines). These automated features increased efficiency by monitoring circuit variables such as ore density, and by optimizing flow rates, resulting in an overall increase in tungsten grades. (North American Tungsten Corp. Ltd. Press Release, Feb. 8th 2006; Quarterly Report Dec. 31st 2005) Tungsten markets continued to remain strong early in 2006, highlighting the wise choice to place the mine into production before the winter season. The improved market also encouraged North American Tungsten Corporation to begin plans for new exploration in the Cantung Mine area with hopes of extending the life of the mine.

2006-2007 Operations

Production was impacted in fiscal year-end September 30th 2006 by lower grades, higher costs of fuel, and power generating capabilities. Significant mine development was required to access zones for production. Some pillar recovery was also initiated in early 2006. Long-hole stoping of pillar areas were part of the reason for lower grades during 2006, and measures were in force by the end of the year to mitigate these effects and apply greater grade control. A low-grade stockpile of ore was established for lower-grade ores that will be economic to process at a future date, towards the end of the mine's life. A new tailings pond was commissioned in the summer of 2006.

In the calendar year 2007, production improved at the Cantung Mine. Grade control was effective in minimizing dilution from waste and hydraulic backfilling. In February the main power generating plant (Cat 3612 engine) was

overhauled to increase its capacity. Several smaller power units were decommissioned resulting in improved fuel consumption, and a waste heat recovery circuit for the camp was installed. Modifications were made to the mill and a large Knelson gravity concentrator was installed to treat the scheelite flotation tailings and recover fine and middling tungsten particles. Higher tungsten recovery was expected. The mill experienced some downtime early in 2007 because of frozen orebins, an unscheduled repair to the Allis-Chalmers ball mill, and a switch-over to disposal in the new tailings pond. A small underground diamond drill program near the underground shop area ('Shop' zone) was performed early in the year to confirm indicated resources. In September 2007, the diamond drill was moved to test the West Extension zones on the 3700 level, an area that would receive significant attention in 2008. (North American Tungsten Corp. Ltd. Annual Reports September 30th 2006 and 2007)

Year: *	Ore Milled:	Ore Grade:	Tungsten Concentrate Produced:	Tungsten Concentrate Produced: (stu)
2002	206,557 tons	1.62 % WO ₃	230,448 MTU	-
2003	348,177 tons	1.69 % WO ₃	423,181 MTU	-
2004	60,183 tons	1.50 % WO ₃	64,815 MTU	-
2005	-	-	9,140 MTU	-
2006	339,743 tons	1.09 % WO ₃	237,869 MTU	-
2007	367,939 tons	1.16 % WO ₃	286,031 MTU	-
<u>Sub-Total:</u>	<u>1,322,599 tons</u>	<u>1.41 % WO₃</u>	<u>1,251,484 MTU</u>	<u>2,759,050 stu</u>
<u>Total 1962-2007</u>	<u>5,685,602 tons</u>	<u>1.55% WO₃</u>	<u>3,661,492 MTU</u>	<u>8,072,209 stu</u>

Table 3. Cantung Mine production 2002-2007. Production continues in 2008. (source: North American Tungsten Corporation Ltd. Annual Reports) **MTU = metric tonne units stu = short ton units**

* Company fiscal year-end September 30th

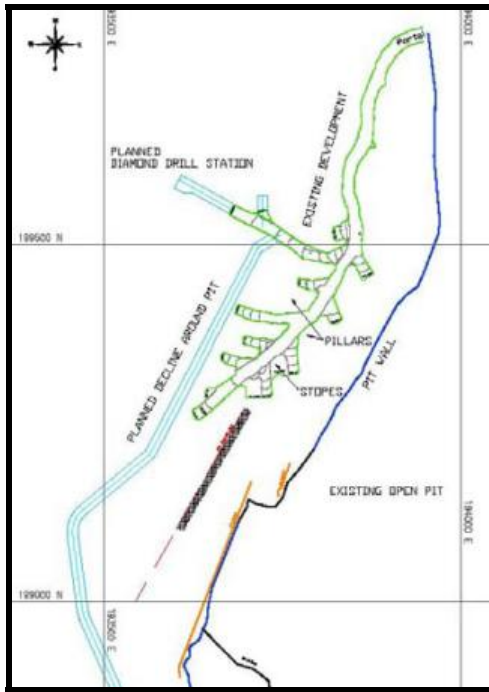


Figure 6. Underground development on the PUG zone, 2007-2008. (courtesy NA Tungsten Corp.)

Mining Operations 2007

In 2007 there was considerable effort and focus on grade control, with some success. Adjustments were made to the drilling, blasting and extraction procedure of the pillars in the E-Zone, where long-hole mining was taking place. The installation of additional ground support in the form of 30 foot long cable bolts was initiated in wide stoping areas and areas requiring rehabilitation. Long-hole stoping accounted for 69% of the ore mined during the fiscal year end September 30th 2007. Minimal ore (13%) was mined by cut and fill methods, while development work in ore, principally in the E-Zone and South Flats, provided the remaining 18% of the mined ore. The surface low-grade stockpile contained (as of June 30th 2007) 8,023 tons. An underground exploration program was started in the PUG (Pit UnderGround) zone to expand indicated resources and develop a mine plan (see **Figure 6**). By the end of 2007, additional cut-and-fill areas were being mined in the E-Zone to provide a more balanced mill feed with less reliance on long-hole stoping. Grade control was being maintained in part by chip and muck sampling and close visual observations of ore faces and muck piles by geologists. (North American Tungsten Corp. Ltd. Annual Report 2007, Quarterly Reports June 30th 2007 and December 30th 2007)

Equipment 2008

The following mining and surface mobile equipment is in use at Cantung Mine during 2008, many of which are items that were originally used at the mine in the 1970s-1980s: five 6-yard scooptrams, two 3.5-yard EJC scooptrams, one 2-yard JCI-200 scooptram, one 1.5-yard Wagner scooptram, two 26-ton EJC ore trucks, three 30-ton EJC ore trucks, three 2-boom Tamrock jumbo drills, one 1-boom CMD jumbo drill, one Maclean rockbolter, one Stopemaster long-hole machine, one D3 Cat dozer, one D6D Cat dozer, two graders, two Cat loaders, and a variety of other surface mobile equipment. All power at the mine is generated by five Caterpillar 3512 diesel-electric units, and backup units consisting two Caterpillar 3516 units and three other smaller Caterpillar diesels. Compressed air is produced by two Sullair LS-25 200hp electric compressors, one Atlas-Copco compressor, and two Sullair 1600-PDQ portable diesel-driven unit. (Reid, etal, 2008)

Grade control has played a significant toll on the economics of the Cantung Mine. In the quarter ending March 30th 2008, the mine did not meet its targeted production and North American Tungsten reported a loss of \$5.5 million for the quarter. Long hole stoping only accounted for 50% of the ore being mined. Exploration was continuing on the 3700 level West Extension Zone where over 10,200 feet had been drilled at quarter's end. The future of the mine depends on the company's ability to control dilution of pillar recovery and the success of exploration in the West Extension and other zones. (North American Tungsten Corp. Ltd. Quarterly Report March 31st 2008)

area	resource	Tons	% WO3	STU's
west extension Total		250,053	1.05	262,859
west-below-3700 Total		231,011	1.24	287,136
E-zone Total		48,748	1.23	59,940
shop-zone Total		8,155	0.70	5,709
main zone pillar Total		184,203	1.43	263,641
central flats Total		159,395	1.23	195,573
south flats Total		71,021	1.42	100,662
Stockpile Total		9,091	0.62	5,636
Total E-Zone		961,677	1.23	1,181,156
area	resource	Tons	% WO3	STU's
pit-pug Total		468,606	1.30	609,188
Total Pit Pug		468,606	1.30	609,188
Grand Total		1,430,283	1.25	1,790,344

Table 4. Cantung Mine Indicated Ore Reserves, September 30, 2008 (from Reid, etal, 2008)

Zone	Tons	Grade (WO ₃ %)	STU'S
West Extension	163,087	1.09	177,654
West Extension Below 3700el	148,449	1.06	157,148
E-Zone	20,505	1.37	28,175
Shop Zone	8,155	0.70	5,709
Main Zone Pillars	246,523	1.14	282,085
Central Flats	59,896	1.09	65,062
Stockpile	9,091	0.62	5,660
TOTAL Probable Reserves	655,706	1.10	721,492

Table 5. Cantung Mine Probable Ore Reserves, September 30, 2008. (from Reid, etal, 2008)

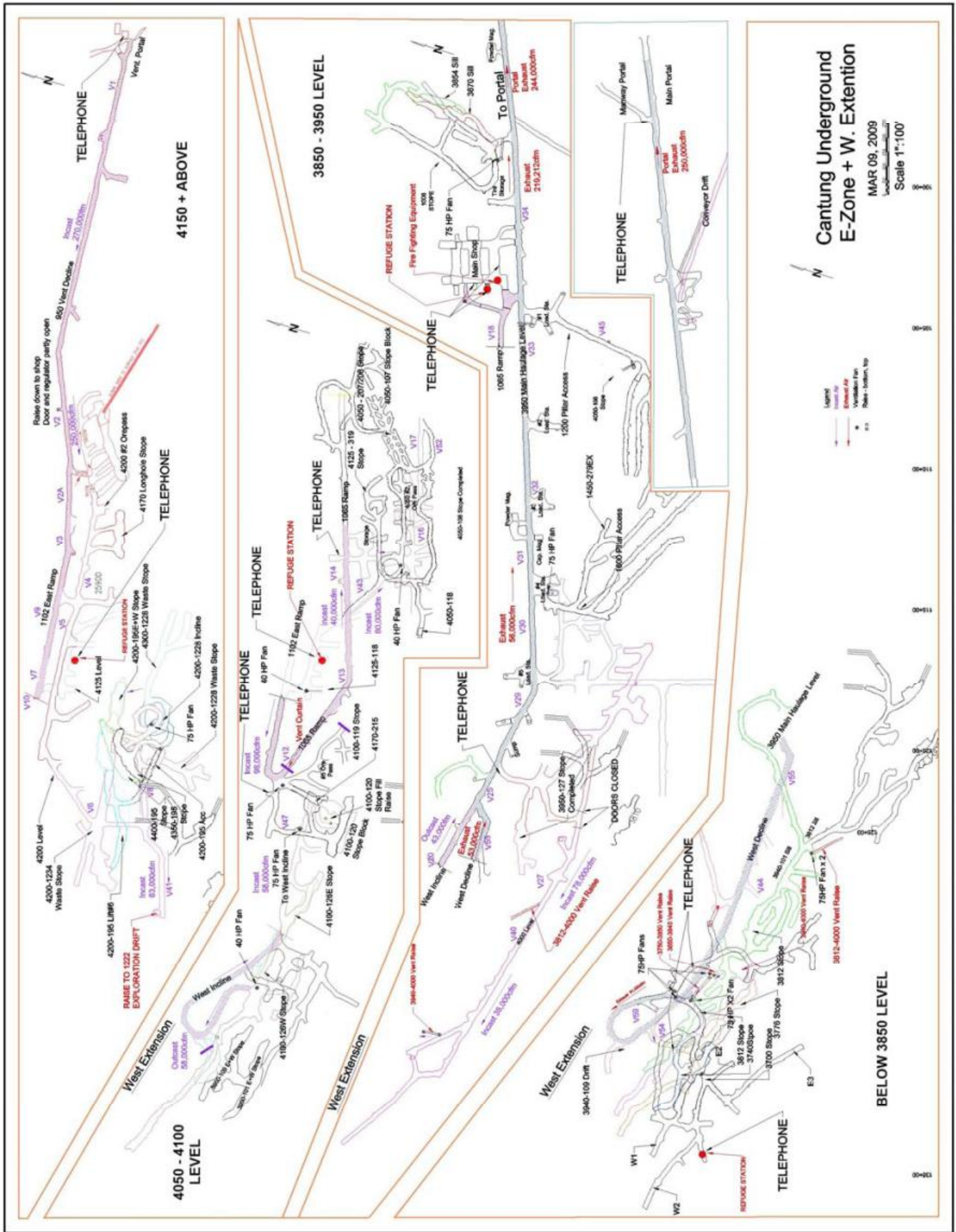


Figure 7. Cantung Mine Underground Ventilation Plan, March 2009. (courtesy North American Tungsten Corporation)

Exploration Since Mine Closure

Not Applicable.

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geology from NORMIN.DB (www.nwtgeoscience.ca) Showing 105HNE0006

	<h2>CASSIDY POINT</h2>	Gold
	<i>Bulk Sampled (Abandoned)</i>	62° 33' 00" 114° 11' 00"

Years of Primary Development: 1950, 1986

Mine Development: pit and tunnel

Years of Bulk Sampling: 1950, 1986

Bulk Sample: 30 tons @ 0.78 oz/ton Au (1950)

1,226 tons @ 0.37 oz/ton Au (1986)

Introduction

Cassidy Point is located 13 kilometers northeast of Yellowknife, NWT, and was a small mining operation in 1950 and 1986. The area is home to many cabins, but the old mine opening still exists at the very northern tip of Cassidy Point. The area has been visited numerous times by the author.

History in Brief

The area was once part of the 'Tin' claims and were owned by Tom Cassidy in the late 1930s. The vein zone at the tip of Cassidy Point was the target of exploration for many years, and 30 tons of ore may have been shipped to Giant Mine for treatment in 1950 or 1951. In 1981, Consolidated Five Star Resources Limited drilled to 65 feet depth and outlined an ore reserve of 1,544 tons grading 0.45 ounces per ton gold. This material was recoverable by open cut methods, and in 1986 Knud Rasmussen drove a pit and tunnel to a depth of 46 feet to recover the known reserves.

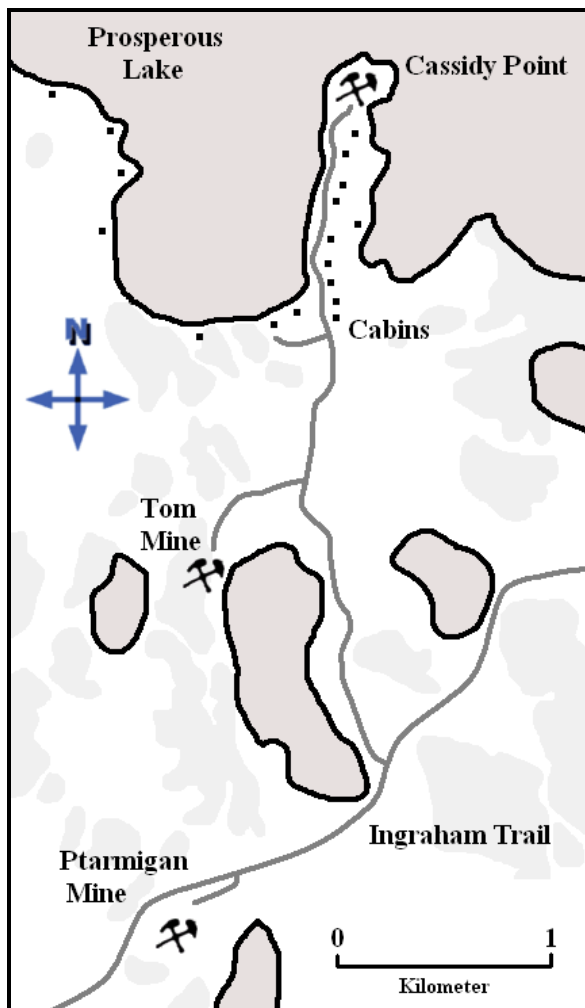


Figure 1. Cassidy Point Mine location.

Geology and Ore Deposits¹

The Cassidy Point deposit lies near the western edge of the Burwash Formation greywacke-argillite turbidites belonging to the Yellowknife Supergroup. The deposit is an auriferous quartz vein system within a shear, cutting folded metagreywackes and incipiently nodular schists of the Burwash Formation. The sediments in the southern Prosperous Lake area have been folded about moderately to gently southeast-plunging and moderately to steeply northwest-plunging axes, while the auriferous zone trends northeast 40° northwest. The veins occupy a shear folded into a syncline plunging gently along an east-west axis; this plunge, and the dips of the shear limbs, are not conformable with bedding, suggesting that the shear cross-cut the beds prior to folding. The #1 zone, comprising sheared quartz-mica schists, quartz lenses, sphalerite and pyrite (with minor pyrrhotite, galena, and visible gold), strikes northwest to north and dips 60° west. It is 60 centimeters wide in the northern most 20 meters and thickened in the nose of a small syncline to 4-5 meter wide in the southern 15 meters. There are also a #2 zone a meter or so to the west, striking 77° and dipping north, and a #3 zone to the south striking northwest.

Tom Cassidy (1950-1951)

It has been reported that 30 tons of ore were removed from the trenches at Cassidy Point and delivered for processing at either Con or Giant Mines. Grade of the ore was 0.78 ounces per ton gold with mill recoveries of 93%. It is presumed that Tom Cassidy was still the owner of the property at this time (Brophy, J.A. et al., 1984). Del Curry was the contractor that hauled the ore to Yellowknife, using the old Ptarmigan Mine road.

Knut Rasmussen (1986)

Knut Rasmussen obtained a lease on the mineral claim in 1986 and sank a pit and short tunnel in order to recover the known reserves. The pit was driven to a depth of 46 feet over a length of 60 feet of vein strike. (Webb, 1993) Rasmussen mined and extracted 1,226 tons of ore grading 0.37 ounces per ton gold during the year. This material was trucked to Con Mine in Yellowknife for processing in October-November 1986 (Cominco Ltd., 1986).

Exploration Since Mine Closure

In 1991, Rosemary Webb staked the 'RIN' claim at Cassidy Point. A reserve of proven plus drill indicated ore was reported in 1992 as 11,000 tons grading 0.40 ounces per ton gold to a depth of 165 feet. During 1992, the owners conducted a geophysical EM-VLF survey and detected three conductors, two of which appear to be related to the mineralized zone. A small tonnage of ore remained within the main zone, which was recoverable by small-scale open pit mining. The potential to develop ore at depth was considered good, and it was recommended that an EM survey be conducted to better define this mineralization. Detailed geological mapping, a property VLF survey, and shallow diamond drilling was also suggested (Webb, 1993).

References and Recommended Reading

Brophy, J.A., Gibbons, W.A., Laporte, P.J., Lord, C.C., Padgham, W.A., Seaton, J.B., 1984. Mineral Industry Report 1980/1981 Northwest Territories. EGS 1984-5. Indian and Northern Affairs Canada, Minister of Supply and Services Canada, 1984.

Cominco Ltd., 1986. Con Mine production records, 1986.

Webb, D.R., 1993. Geological and Geophysical Assessment of the RIN 1 Claim - Cassidy Point Area. for Rosemary Webb, March 1993. (*N.W.T. Geoscience Office Assessment Report #083166*)

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085JNE0002



CHIPP LAKE

Minor Exploration (Abandoned)

Gold

62° 28' 00" 112° 38' 45"

Years of Primary Development: 1942

Mine Development: 32' vertical shaft

Introduction

The Chipp Lake property is located 88 kilometers east of Yellowknife, NWT. Only a short shaft, a log cabin, and trenching remains of this mining operation. The author flew over the site in July 2000, but has not actually visited the ruins.

History in Brief

The 'Eileen' claims were staked by Jimmy Blaisdell for Emil Skarin of Edmonton in 1941. Mining developments began in 1942, during which time a shaft was sunk 32 feet. Some 20 tons of ore were removed from the shaft and stockpiled. Although a small 9-ounce crude bar was smelted down from material obtained from this property in 1942 (containing 2 ounces of gold and 3 ounces of silver), the stockpiled ore was still on the property in 1946. It was probably removed soon thereafter.

Geology and Ore Deposits¹

The Chipp Lake showing is located within the Slave structural province, and is hosted within massive greywackes and quartzites of the Yellowknife Supergroup. Within the showing area, these strata strike at roughly 35° and have a steep to vertical dip. A number of quartz veins are developed locally within the metasediments, and in the showing area these appear to strike northeast and dip about 80°. These veins are generally sulphide poor, hosting only small amounts of pyrite, chalcopyrite, bornite, and arsenopyrite.

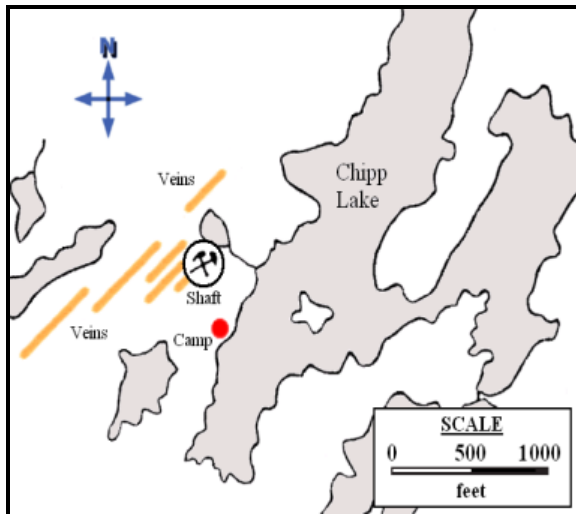


Figure 1. *Chipp Lake area.*

Emil Skarin (1942)

The claims were owned by Emil Skarin and Harry Cohen of Edmonton. During the 1942 season, it was reported that Jimmy Blaisdell, Peter Porter, Tom Hansen, and Hilding Ahlfrith were working at Chipp Lake sinking a shaft. By July 1942, this 8' x 6' shaft was 32 feet deep (The Yellowknife Blade, April 27th 1942; July 1942). Some 10 to 20 tons of material was removed from the shaft, bagged in burlap sacks, and stockpiled to await removal (National Mineral Inventory).

In September 1942 the Royal Canadian Mint received a crude 9-ounce gold bar from the "Chipp Lake claims", and smelted it down to recover 2 ounces of gold and 3 ounces of silver (National Archives of Canada). The gold was probably recovered from some high-grade surface exposures and should not represent production or bulk sampling operations. No further work was done for the remainder of World War II. A small log cabin on the shore of Chipp Lake dates back to operations of the 1940s.

In 1946 the owners (Emil Skarin and Harry Cohen) suggested to Tom Payne that it might be advisable to ship the stockpiled ore to a mill for treatment. It is not known if this material remains at Chipp Lake today and if not, when it was removed.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Exploration Since Mine Closure

Exploration continued in 1949, with four diamond drill holes totaling 457 feet. Dave Nickerson acquired the claims sometime in the 1970s. Andex Mines Limited performed some diamond drilling (10 holes) in 1975, but results are unknown. (National Mineral Inventory)

References and Recommended Reading

National Archives of Canada: Royal Canadian Mint Collection (*RG 120*).

N.W.T. Geoscience Office Assessment Report #082163

The Yellowknife Blade newspaper, 1942 articles.

National Mineral Inventory (Eileen). NTS 85 I/7 Au 11.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0120

	CLAN LAKE <i>Bulk Sampled (Abandoned)</i>	Gold
		62° 55' 00" 114° 14' 15"

Years of Primary Development: 1967

Mine Development: open cut

Years of Bulk Sampling: 1967

Bulk Sample: 1,141 tons milled = 515 oz Au, 115 oz Ag

Introduction

The property is located 53 kilometers north of Yellowknife, NWT, on the southeast side of Clan Lake, on the Yellowknife River. A forest fire swept through this area in 1998, believed to have started when lightning struck one of the old buildings at the mine site.

History in Brief

Gold was discovered at Clan Lake in August 1964 by Jack D. Curry, and the 'Nose' group of claims was staked on behalf of the Earl-Jack Syndicate and Gunnex Limited. An extensive program of exploration was carried out in subsequent years in which three zones were outlined. In 1967 a bulk sample of ore was extracted and was shipped to the nearby Discovery Mine for processing.

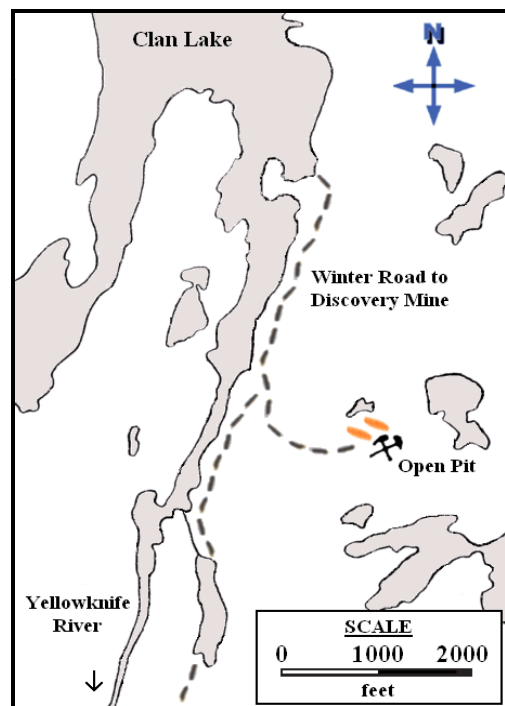


Figure 1. *Clan Lake area and mine location.*

Geology and Ore Deposits

Three principal zones of vein hosted gold mineralization occur at the apex of a fold in the Clan Lake volcanics of the Yellowknife Supergroup. Numerous smaller showings occur nearby which may indicate a discontinuous strike length of 1,500 feet (National Mineral Inventory).

Shield Resources Limited (1967)

Jack and Earl Curry purchased the property from the former financial backers of the Earl-Jack Syndicate in October 1966 and transferred title to Shield Resources Limited. It was decided to conduct bulk sampling of the #1 zone ore shoot to determine its grade and continuity. In February 1967 equipment was mobilized over the winter road and open cutting of the zone commenced under the direction of Jack Curry and Frank Mistal. The entire surface operation was carried out in temperatures ranging from -25 to -50° Celsius (News of the North, July 13th 1967).

A 1,141 ton sample was trucked to Discovery Mine for milling. Mining equipment was returned back to Yellowknife before spring breakup 1967. This sample was thought to only be 70% of the ore mined; the remainder was left on site and is believed to be lower-grade material (News of the North, July 13th 1967).

Milling between July 2nd and July 8th 1967 at Discovery produced a gold bar with 515 ounces gold and 115 ounces silver. The bar was worth \$17,000 and the grade indicated was 0.42 ounce per ton gold. It was believed a commercial operation could be established (News of the North, July 13th 1967). Shield Resources Limited was busy developing other projects at the time but made plans to one day return and continue with development of the property.

Exploration Since Mine Closure

In 1973, Shield Resources Limited entered into a joint agreement with Numac Oil & Gas Limited to undertake exploration at the Clan Lake property. Diamond drilling (4 holes) was conducted in 1974 within five zones and while visible gold was reported, assays were low. Sampling of the ore stockpiles and the open pit and some diamond drilling (two holes) was undertaken in 1980; in 1983, the claims were optioned to Pacific Copper Mines Limited (National Mineral Inventory). The property was re-staked as the new 'Nose' claim in 2003 by Pat Hungle and optioned to Tyhee

Development Corporation in 2007. Tyhee has confirmed gold mineralization in gold-bearing veins hosted within an altered igneous breccias. Elevated gold values have been identified and diamond drilling was initiated in spring 2008. One hole returned 78.8 metres grading 1.94 grams per tonne gold including 33 metres grading 4.29 grams per tonne. Exploration is continuing in the summer of 2008. (Tyhee Development Corporation Press Release, May 13, 2008)

References and Recommended Reading

Curry, J.D., 1964. Geological Report for Parts of the Nose Group, Clan Lake, N.W.T. November 1964. (*N.W.T Geoscience Office Assessment Report #017226*)

Tyhee Development Corporation. www.tyhee.com

The News of the North newspaper articles, 1967.

National Mineral Inventory (NOSE – Clan Lake). NTS 85 J/16 Au 4.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085JNE0071



COLOMAC

Producer (Under Remediation)

Gold

64° 24' 00" 115° 06' 00"

Years of Primary Development: 1946, 1989-1991, 1994-1997

Mine Development: 3 open pits

Years of Production: 1990-1991, 1994-1997

Mine Production: 12,294,352 tons milled = 535,708 oz Au

Introduction

The Colomac Mine is a closed gold mining and milling operation located near Indin Lake, 220 kilometers northwest of Yellowknife, NWT. It produced gold from 1990-1991 and from 1994-1997 when low gold prices bankrupted the company and shut down the mine for good. Responsibility for cleaning up the hazardous site fell into the hands of the Federal Government (D.I.A.N.D.) and reclamation activities are ongoing during 2009. The property was visited by the author in June 2001.

Brief History

The Indin Lake area saw some prospecting activity in 1938-1939, but gold deposits were not found at Colomac until 1945. There was interest in a long gold-bearing dyke that ran through several mineral claims. In the fall of 1945 exploration began on the sections of the dyke owned by Colomac Yellowknife Mines Limited and Indian Lake Gold Mines Limited through a joint-venture agreement. A series of drifts and crosscuts were cut into the zone via an adit drive and a bulk sample was pulled in 1946. This work indicated a large but low-grade gold deposit.

The property switched hands many times and by 1968 it was thought feasible to open-pit the deposit. Several interested companies optioned the claims, but none could arrange a profitable operation with such a low-grade, high tonnage deposit.

In the fall of 1986, Neptune Resources Corporation Limited, a Peggy Witte-managed company, optioned the property from Johnsby Mines Limited, and initiated an exploration program in 1987 to establish ore continuity in the Zone 2-0 area. They proposed mining gold through a vat-leach process and erected a testing facility in 1987. Conventional gold recovery methods proved more attractive however, and in 1988 plans made laid to bring the mine into production at a 10,000 tons per day rate. Gold production commenced May 1990, but the operation was hampered with technical difficulties associated with the installation of the plant. High operating costs and financial trouble resulted in a temporary closure in June 1991.

In April 1993, the assets of Neptune Resources Corp. Limited were purchased by Royal Oak Mines Incorporated and the Colomac Mine reopened in 1994. The Kim and Cass zones were purchased in 1994 from Echo Bay Mines Limited, and plans were made to mine these deposits as satellites. Low gold prices hampered profitable operations and as a result, economic gold reserves were mined out in December 1997. Royal Oak went bankrupt early in 1999 and the property reverted to the Federal government.

Geology and Ore Deposits¹

The Indin Lake area is situated in the southwest portion of the Slave Structural Province and is located within the Indin Lake Belt of Archean supracrustal metasedimentary and metavolcanic rocks of the Yellowknife Supergroup. The rock types and metamorphic zones form a series of broad, northeast-trending belts that extend south from the south side of Truce Lake to the southern portion of the Snare River. Much of the Indin Lake area is underlain by metamorphosed greywacke-argillite turbidites, and a hornblende-biotite granodiorite complex composes the basement. Evidence for four deformational events has been observed and the Indin Lake Supracrustal belt has been metamorphosed to lower greenschist facies with local amphibolitic facies. An intrusive sill/dyke complex is found west of Baton Lake. The complex intrudes a 4-kilometer thick belt of greenschist-grade intercalated mafic-intermediate flows and intermediate-felsic volcanics, with 800 meters of underlying turbidites. The host strata and dyke complex are strongly deformed, and the mafic volcanics have a steeply-dipping foliation and a steeply-plunging lineation. The complex strikes northeast and dips steeply east, subparallel to the host strata. It consists of a series of multiphase, medium-grained diorite to quartz-diorite and gabbroic sills, with about 15% of the complex occupied by elongate, andesitic enclaves.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

The Colomac dyke orebody, the largest of several bodies believed to be rotated sills, occurs near the top of the complex in contact with, or nearby, andesitic volcanics. A talc-carbonate schist found near the dyke (Zone 2-0) is presumably altered andesite. Gold is found in several zones within the dyke, in association with parallel sets of glassy, tensional quartz veins that consist of lenses of smoky grey quartz within white quartz. The veins are up to one meter wide but average 5 centimeters thick, and they commonly contain up to 15% carbonate. The quartz veins generally terminate at the dyke margins but a few small, barren quartz-carbonate veins occur locally in the andesite. A late set of white, barren quartz veins is also present. Gold occurs as fine grains along contact margins, in fractures, and in quartz vein selvages/halos, but not commonly in the quartz veins themselves. Gold is spatially associated with pyrite (in quartz veins and vein selvages), chlorite, pyrrhotite, tourmaline, arsenopyrite, and magnetite. Sericitic alteration is found in the immediate vicinity of mineralization, accompanied by silicified and chloritized selvages containing the above minerals. Alteration halos also contain albite, quartz, sericite-muscovite, various carbonates, Ti-oxides and epidote-group minerals, and are enriched with minor chalcopyrite, marcasite, galena, and sphalerite.

The Kim and Cass zones occur within a steeply dipping mafic intrusive body, northwest of Indian Lake. Gold occurs associated with swarms of quartz veinlets containing minor amounts of sulphides.

Colomac Yellowknife Mines Limited (1946)

During 1945, diamond drilling on the properties held by Colomac, Indian Lake Gold Mines Limited, Indyke, and Nareco Mines Limited totaled 47,000 feet. Due to the erratic distribution of the gold, the drilling gave inconclusive results and it was decided that underground work and bulk sampling was required. Before breakup of 1946, a Cat train of supplies and equipment arrived at the property. It was decided to drive an adit into a massive ridge, then crosscut into the Zone 2-0 orebody. This operation was a joint venture project between Colomac Yellowknife Mines Limited and Indian Lake Gold Mines Limited; therefore it was necessary to start the adit at a position with an advantage for both, but also at the shortest distance to reach the orebody. The ideal situation would have been to drive two adits on each property, perhaps one at a second elevation, but costs and logistics involved in mobilizing a second mining crew prohibited this plan.



photo from Creelman et al (1946)

Figure 1. Colomac bulk sampling plant, 1946.

A tent camp occupied by Colomac and Indian Lake crews was erected on Baton Lake. A power plant consisting of a Cat D-13,000 diesel engine driving a 365 cubic feet per minute Gardner-Denver air compressor was used during underground operations. These units, plus a small service building, were housed in temporary frame structures.

Development

The adit was collared on the Colomac property about 200 feet north of the Indian Lake claim boundary and about 190 feet below the ridge. Development began May 1st 1946 by a crew of contract miners from Ontario (Central Mining Services Limited). Up to November 1946, when work stopped, development consisted of a 570 foot adit-crosscut, two main drifts trending north and south, and six other crosscuts trending west-east branching from these drifts to explore the west contact of the zone. Underground workings totaled 2,500 feet and remained within the zone.

Bulk Sampling

During development, all ore was trammed out the adit and processed through the bulk-sampling plant. As many as twelve samples for assay were obtained for each round advanced underground. 5,000 tons were mined. The ore was crushed to 1 inch size, reduced by splitters to a 3,000 pound sample, re-crushed, and split to provide eight 47 pound samples for assaying.

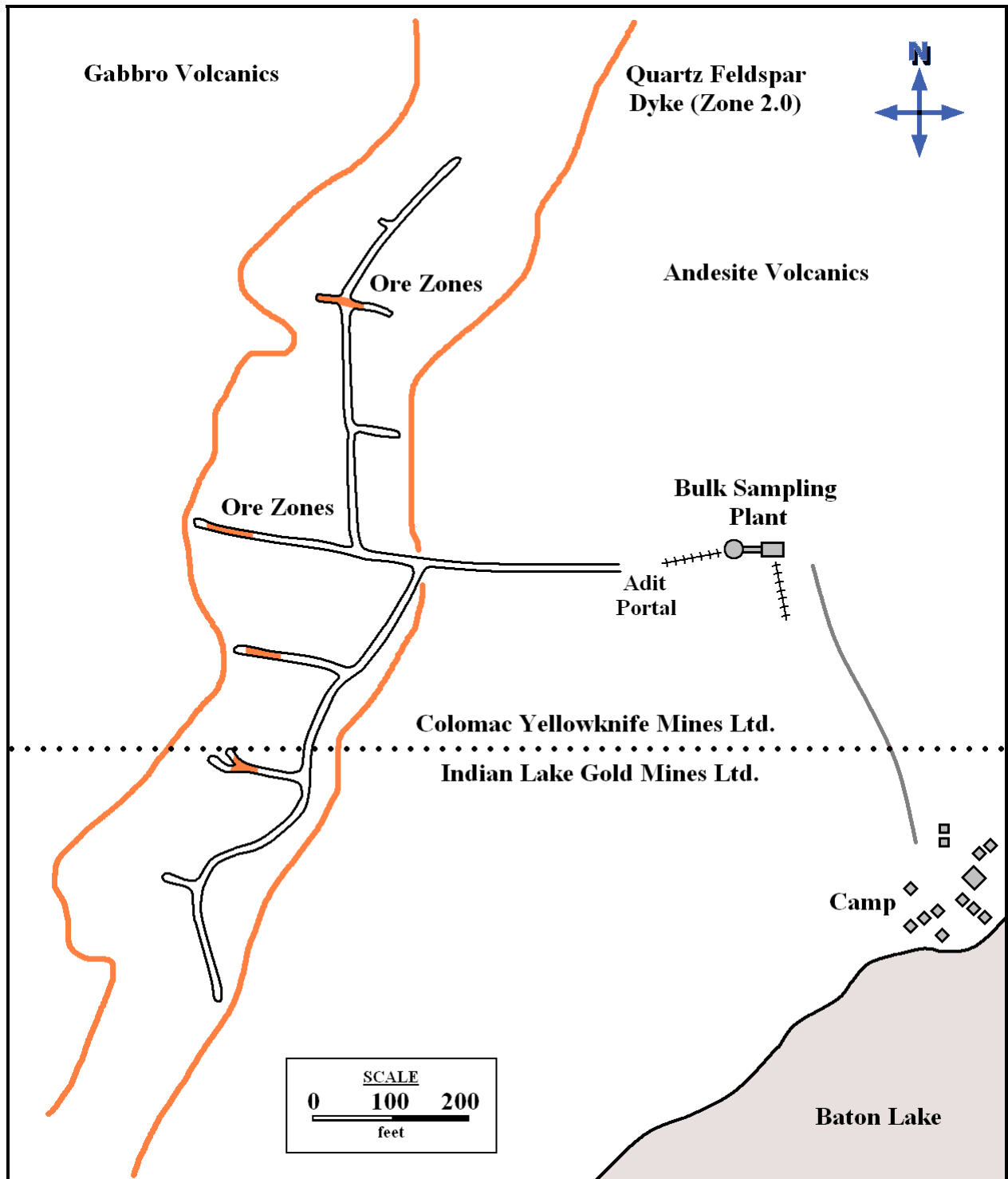


Figure 2. *Underground and surface plan at Colomac Mine, 1946.*

A crew of 20 men were employed during underground and sampling operations. Resident manager of the operation was Norman Edgar, with E.C. Creelman and Harry W. Darling on property as consulting engineers. Other staff included R.A. Fee, Frank Anderson, Fred Garbutt, and E.O. Lilge (Creelman et al, 1946; Stanton, 1947). Work at the Colomac property during 1946 indicated 21,000 tons per vertical foot with a grade of about 0.086 ounces per ton gold. A refinancing and a better idea of how to extract gold profitably at such a low-grade deposit was necessary to continue development at Colomac (Creelman et al, 1946; Stanton, 1947).

Neptune Resources Corporation Limited (1987-1989)

As part of its 1987 exploration program at the Colomac Mine, Neptune Resources planned to test gold recovery using a vat-leaching process. Colomac ore was known to be free-milling. Preliminary blasting of the future open pit on Zone 2-0 began in June 1987 to extract ore for the leaching test. Two benches were mined to 40-foot depth. This early mining established that the ore body was economic and that the ore occurred in large blocks suitable for open pit mining methods. The leaching test began in October 1987. 1,500 tons of ore were mined and crushed and placed into the single vat. The vat was designed to hold 15,000 tons of ore, but the crusher and screening plant experienced mechanical difficulties resulting in the low tonnage. The ore was dissolved in cyanide and other reagents to extract the gold. The test lasted 35 days and an 80% recovery of gold was attained, with a 108 ounce gold bar produced. The test was considered successful, and Neptune Resources began plans to erect a commercial vat-leaching plant of 10,000 tons per day at Colomac Mine.

Wright Engineering Limited was hired to perform a feasibility study based on an open pit mine feeding a 10,000 tons per day vat-leach process with 79% gold recovery. It was determined that this process, with only a 79% gold recovery, would not be economical. Vat-leaching technology was also not considered suitable for northern climates. A revised Wright Engineering Limited study was released in February 1988 based on a conventional milling and carbon-in-pulp gold recovery circuit. This method required higher capital costs, but gold recovery would be much higher at 94%. The plan was approved by directors of Neptune Resources in 1988 and the mine was authorized for production (Scales and Werniuk, 1990).



Royal Oak Mines Inc.

Figure 3. *Colomac Mine, 1991.*

NorthWest Gold Inc. [Northgate Exploration Limited] (1989-1991)

Neptune experience financial problems early in 1989, which led to Northgate Exploration Limited acquiring a controlling interest in the company and arranging major financing to continue construction. NorthWest Gold Inc., a Northgate subsidiary, was set up as manager of the mine. Construction of the facilities was started in the spring of 1989. Practically all major equipment had to be brought to site via ice road. Due to certain delays, the winter-road window of 1989 was very narrow resulting in the fast tracking of design, purchase, and delivery of necessary supplies and equipment. As a result of this shortfall, much of the equipment had to be purchased used (including the primary crusher) and expedited to site as soon as possible before the winter road closed. These early problems were often blamed on incomplete and unsupervised work carried out during the Neptune-Northgate transition (Neptune Resources Corporation Ltd., 1992).

Gold reserves before start-up (January 1st 1990) suggested a total of 26,736,000 tons of ore grading 0.055 ounces per ton gold between the three known zone deposits. Other zones did not seem economically viable to mine. The first

open pit was developed on the Zone 2-0 section of the Colomac dyke. The original mine plan suggested that the zones be developed one at a time, with the most southern deposit, Zone 3-0, developed nearing the end of the projected mine-life (Northwest Gold Corp., 1990).

Start of Production

Full-scale construction of a 10,000 tons per day carbon-leaching plant began in March 1989 and open pit stripping at Zone 2-0 began in August 1989. The Colomac mill was commissioned in April 1990, processing waste followed by low-grade ore. Official milling operations began on May 1st 1990 with the first gold bar poured on May 29th 1990. Operations were steady for the first few months of operations, but equipment failures and power outages plagued the operation during the winter season of 1990-1991. The designed plant capacity of 10,000 tons per day was not achieved, with an average daily rate of 6,255 tons realized (Neptune Resources Corporation Ltd., 1992).

Open Pit Operations

The mine produced on average 6,000 tons of ore per day, and up to 40,000 tons of waste. Cat 992 front-end loaders filled 85-ton Cat 777 haul trucks with ore for transport to the crushing plant. Nine of these trucks were in service during 1990-1991, along with two 35-ton Cat 769 haul trucks. Drilling was complicated by hard and abrasive rock and in order to maintain production, it was necessary to operate four primary drills and three secondary drills. Blasting costs were also high because of ground conditions. Haul trucks and loaders had short tire life, again, due to abrasive rock on haul roads and the open pit floor. The Zone 2-0 open pit provided the entire mill feed during operations in 1990-1991, with a total of 11,571,000 tons of combined ore and waste removed. This pit was mined in 30 foot benches with a slope of 60° (Neptune Resources Corporation Ltd., 1992; Scales and Werniuk, 1990).

Milling Operations

Ore was hauled to the crushing plant, which consisted of an Allis-Chalmers 54 inch x 72 inch gyratory crusher. A 900 foot conveyor stockpiled the ore adjacent to the milling plant, and three feeders drew the ore into the primary 32 foot x 12 foot S.A.G. (Semi-AutoGenous) mill, to which ball charges were added to achieve an efficient grind of Colomac ore. Pebbles of two inch size were re-ground in two Allis-Chalmers 15 foot x 28 foot pebble mills, and + 3/8 inch -2 1/2 inch material went through a Nordberg 7 foot pebble crusher. Undersize material was sent back for grinding in the S.A.G. mill. Screened mill discharge was classified by a bank of 20 inch Linatex cyclones. A gravity circuit was employed consisting of spiral classifiers and concentrate tables from which a large amount of free gold was obtained from cyclone underflow. Material passing through the gravity circuit was recycled through the grinding units. Cyclone overflow was thickened in a 125 foot diameter tank, with thickener underflow reporting to twelve 52 inch x 56 inch agitators with a 64 hour retention time in which gold was dissolved in cyanide. The leached product was pumped to five carbon-in-pulp (CIP) tanks, 36 foot x 36 foot, where gold was collected with the carbon. The gold was then stripped from the carbon batches in a high-heated vessel, with gold solution precipitated in two electrowinning cells. Gold was melted in an induction furnace and poured into dore bars (Scales and Werniuk, 1990).

Power Plant

Power for the milling operations was provided by six EMD diesel generators of 2,500 kilowatt capacity. During the period 1990-1991, the average fuel consumption was 1.94 million litres per month. Eight 1 million gallon oil tanks were installed. Fuel was trucked in over the winter road (Scales and Werniuk, 1990).

Camp and Employees

Camp facilities at Colomac Mine consisted of an Atco trailer complex with 276 rooms and 7 suites, including a small recreation room in the kitchen complex. On average, 320 personnel were employed at Colomac Mine. An agreement between local Dogrib communities and Neptune Resources Corporation Limited suggested that 25% of the workforce be aboriginal. Jim Johnstone was general manager of the operation. Other staff in 1990 included Denis Anderchek, human resources manager; Dave Loder, mining manager; Michael Eaid, mining superintendent; Wener Koetter, mill superintendent; and Wayne Valliant, chief engineer and geologist. A 5,000 foot long gravel airstrip was established south of the mine site, capable of handling aircraft up to the size of a Lockheed C-130 Hercules (Northwest Gold Corp., 1990; Scales and Werniuk, 1990).

Operations 1990-1991

The operation was plagued with equipment failures in the mill that resulted in low production rates. Operating costs were also very high (average \$23 per ton milled). In late 1990, it was realized that the \$15 million required to finance operational supply needs for 1991 would not be raised. Operations were therefore scaled back in December 1990 with the reduction of crews. Waste stripping operations were also scaled back. Mill performance improved the following year. Production reached the design rate of 10,000 tons per day in April 1991. Gold recovery reached 95.9% in May

1991 as a result of operational experience. Only a limited amount of fuel and supplies were brought to site over the 1991 winter road, enough to last until June 1991. Milling operations ceased on June 29th 1991. A small crew was retained to cleanup and mothball the plant and camp (Neptune Resources Corporation Ltd., 1992).

Tons Milled:	2,664,636 tons
Daily Tonnage (avg):	6,255 tons per day
Calculated grade:	0.0595 oz/ton gold
Mill Recovery:	92.3%
Gold Produced:	146,400 oz

Table 1. Colomac Mine production, 1990-1991. (source: Neptune Resources Corporation Ltd., 1992)

Colomac Mine Operating Summary 1989-1991

The Colomac Mine produced 146,400 ounces of gold from the milling of 2,664,636 tons of ore from May 1990-June 1991 (see **Table 1**). This figure reportedly does not include 8,700 tons of low-grade material milled in April 1990 before commercial production. Tonnage milled and gold recovered were lower than the original 1988 feasibility study. Mill performance was low because of mechanical and operational problems with equipment. Gold recovery was low because of lower tonnages and the lower mill head grades. Total rock moved from September 1989 to June 1991 was 11,570,800 tons (8,356,800 tons waste and 3,214,000 tons ore) (Neptune Resources Corporation Ltd., 1992).

Royal Oak Mines Incorporated (1994-1997)

In April 1993, Royal Oak Mines Incorporated acquired the property from Neptune Resources Limited and immediately made plans to bring Colomac back into production. Ore reserves pre-startup are listed in **Table 2**. Equipment and supplies were mobilized to the site on the 1994 winter road. Because of scaled-back stripping and overburden removal operations in December 1990, some 6 million tons of waste material had to be removed before full production mining could recommence in the Zone 2.0 open pit. Pit stripping began in March 1994 (Royal Oak Mines Inc. Annual Report, 1993).

Zone:	Ore Reserves:	Grade:	Gold:
2.0	11,036,334 tons	0.053 oz/ton	585,000 oz
2.5 and 3.0	3,412,219 tons	0.070 oz/ton	239,000 oz
<u>Total:</u>	<u>14,448,553 tons</u>	<u>0.057 oz/ton</u>	<u>824,000 oz</u>

Table 2. Ore reserves at Colomac Mine, 1993. (source: Royal Oak Mines Inc. Annual Report, 1993)

Milling began on July 15th 1994 but was delayed a short time later by a fire in the crushing plant. Stripping of the Zone 2.5 and 3.0 open pits were underway during 1994 and mining of these zones began in 1995. Previously used mining equipment was found to be in bad shape and unreliable. Royal Oak purchased brand new production units, including two Hitachi EX-1800 shovels, three Drilltech drills, and eight Euclid R-85B haul trucks. Other equipment in use included an Ingersoll-Rand DM-45E drill, a Gardner-Denver 3800 drill, a GM-CM 45R drill, and a Cat D-9L bulldozer (Royal Oak Mines Inc. Annual Report, 1994; mine records).

Royal Oak altered the open pit mining practice by mining ore in 20 foot benches rather than the 30 foot benches mined by Neptune Resources for better ore definition. The mine later switched to 40 foot benches in the fall of 1995, which was continued until closure in 1997. Mill capacity was increased in 1995-1996 through a modification of the milling circuit and the targeted capacity of 10,000 tons per day was reached. This modification included the installation of a pebble crusher bypass system (Royal Oak Mines Inc., 1998).

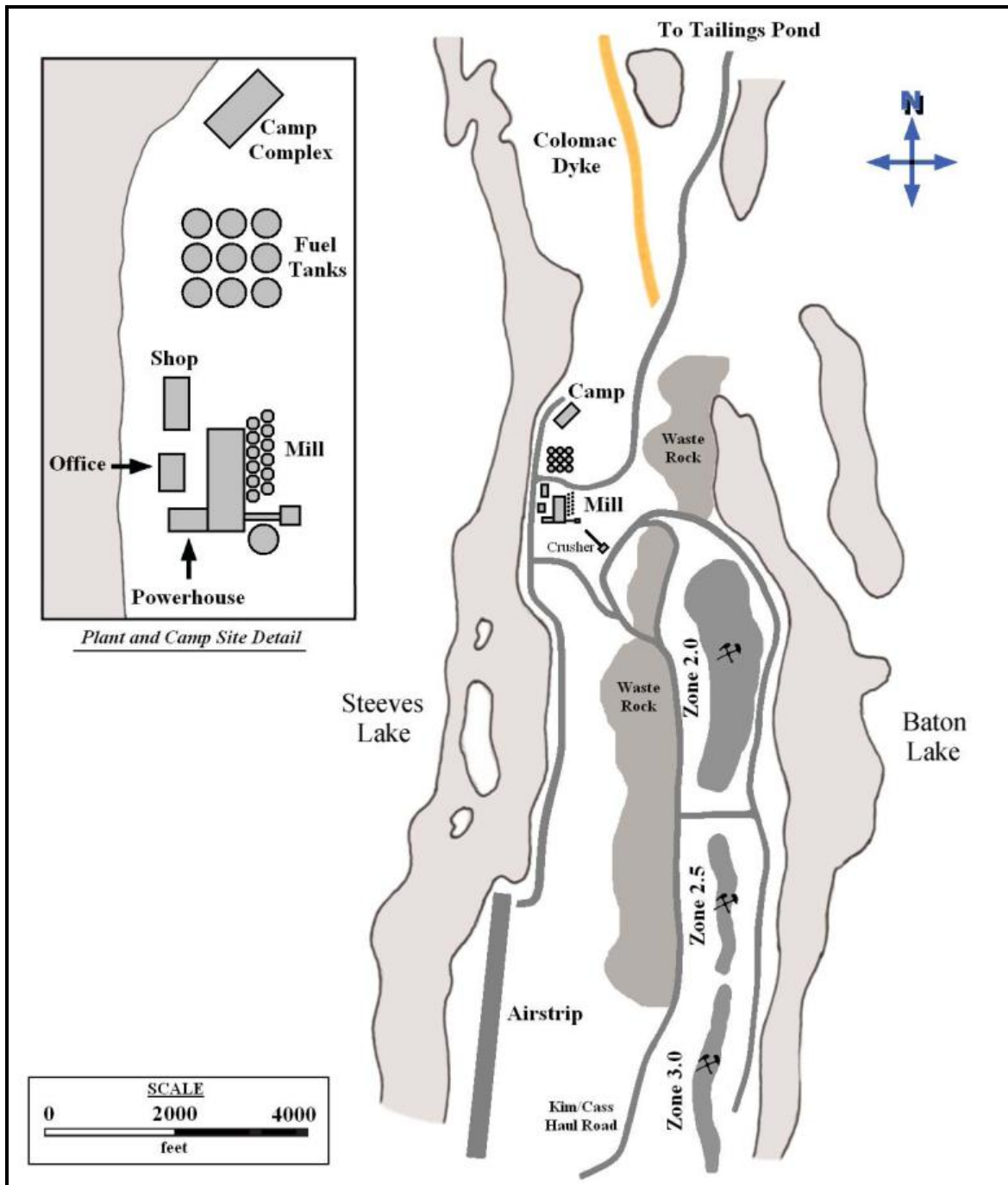


Figure 4. Colomac Mine property plan, c.1997.

At year-end 1994, 258 people were employed at Colomac Mine. This rate was reduced to 194 in 1995 but increased to 222 in 1996. In 1995 the cost of operations was U.S. \$383 per ounce produced with the price of gold at around \$420 per ounce. New reserve calculations indicated reserves with lower than anticipated gold grades. This was bad news in light of increasing costs and the outlook of falling gold prices (Royal Oak Mines Inc. Annual Reports, 1994-1995). During 1996 the cost of operations was U.S. \$370 per ounce. Early in 1997, the company began to plan the cessation of mining activities on properties that were not projected to be generating positive cash flow at a gold price of U.S. \$350 per ounce. A major factor in this plan was that the company was in the process of constructing a low-cost copper mine in B.C. at this time (Kemess Mine). Closure of high-cost gold mines such as Colomac was considered necessary in order to conserve company finances (Royal Oak Mines Inc. Annual Reports, 1996-1997).



NWT & Nunavut Chamber of Mines

Figure 5. *Open pit mining at Colomac, 1995.*

Kim/Cass Project

Development of the Kim and Cass zones was also underway during 1996, located about 15 kilometers southwest of Colomac. The property was purchased from Echo Bay Mines Limited in August 1994, under an agreement in which Royal Oak was to place the property into production within four years to earn a 100% interest. Reserves for all properties (Colomac, Kim, and Cass) at year-end 1994 were 1,161,000 ounces of gold. An all-weather road was cleared to the Kim and Cass sites in the winter of 1996-1997 with the idea of hauling ore to Colomac. A bulk sampling program was undertaken in March and April 1997 when 35,000 tons of ore were mined, 16,500 tons of which were trucked to the Colomac mill for test processing. Mining plans were affected by projected high-water levels in the planned ultimate limit of the Kim pit. Additionally, the company was unable to acquire the needed permits to begin development of the deposits, and by the summer of 1997 it was obvious that mining of the Kim and Cass zones was economically unfeasible (Royal Oak Mines Inc., 1998).

New Zone Discovered

Early in 1997, exploratory diamond drilling uncovered a new zone within the Colomac dyke: Zone 3-5. The zone was found to maintain grades of 0.35 ounces per ton gold for depths of over 1,500 feet. Zone 2-0 was also found to extend to greater depths than previously calculated, indicating a continuation of the gold mineralized system to at least 2,200 feet vertical depth and the presence of significant grades over mineable widths (0.353 ounces per ton over 135 feet). These findings suggested the possibilities of underground mining operations at the Colomac Mine. This exploration program was preliminary in nature, and there was no opportunity to follow-up on the 1997 results before the mine closed at the end of the year (Royal Oak Mines Inc. Annual Report, 1997).

Year:	Tons Milled:	Grade:	Gold Produced:	Mill Recovery:
<i>1994</i>	985,091 tons	0.047 oz/ton	40,568 oz	87.1%
<i>1995</i>	2,725,388 tons	0.047 oz/ton	117,646 oz	92.3%
<i>1996</i>	3,013,156 tons	0.046 oz/ton	122,416 oz	87.3%
<i>1997</i>	2,906,081 tons	0.044 oz/ton	108,678 oz	85.4%
<i>Total:</i>	<i>9,629,716 tons</i>	<i>0.046 oz/ton</i>	<i>389,308 oz</i>	<i>88.0%</i>

Table 3. *Colomac Mine production, 1994-1997. (source: Royal Oak Mines Inc. Annual Reports)*

Cost of operations for the year 1997 was U.S. \$354 per ounce against an average price of U.S. \$330 per ounce for the year. It was announced by Royal Oak on August 6th 1997 that the mine would close due to low gold price, high operating costs, and diminishing ore reserves. The shutdown was planned to be temporary, with the possibility of reopening within a few years if the price of gold were to improve (Royal Oak Mines Inc. Annual Report, 1997).

1997 Closure

Open pit mining ceased in September 1997. Milling of stockpiled ore, including low-grade material which had previously been stockpiled in the early 1990s during Neptune Resources' term, continued until December 1997. The plant was mothballed and placed on care and maintenance pending an increase in the price of gold (Royal Oak Mines Inc. Annual Report, 1997). A total of 576 gold bricks were poured at Colomac Mine by year-end 1997. An additional 25 bricks were poured from clean-up material sent to the Giant Mine in Yellowknife early in 1998 (mine records). The subsequent bankruptcy of Royal Oak early in 1999 reverted ownership of the Colomac property to the Federal Government (D.I.A.N.D.), leaving a major environmental problem and ruling out the possibility of ever resuming gold mining operations at the mine. Federal funded remediation of the property is ongoing as of 2005.

Development and Production Summary

The Colomac Mine was developed with three open pits. The Zone 2-0 open pit contributed 95% of production (11,621,300 tons of ore mined). Open pits on the Zone 2-5 and 3-0 zones were much smaller and contributed together the remaining 5% of production (343,400 tons and 351,600 tons of ore mined, respectively). The Zone 2-0 open pit is 3,000 feet long, 1,000 feet wide, and 400 feet deep. The Zone 2-5 open pit is 1,600 feet long, 450 feet wide, and 93 feet deep. The Zone 3-0 open pit is 2,100 feet long, 300 feet wide, and 110 feet deep (Royal Oak Mines Inc., 1998). Total Colomac Mine production from 1990-1991 and 1994-1997 is 12,294,352 tons milled to produce 535,708 ounces of gold. (see **Tables 1 and 3**)

Exploration Since Mine Closure

No work has been conducted since mine closure in 1997.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 086BSW0004

	CON including Rycon and Vol <i>Producer (Under Remediation)</i>	Gold
		62° 26' 00" 114° 22' 00"

Years of Primary Development: 1937-2003

Mine Development: two primary shafts, depth 6,200', two internal winzes, vent shafts

Years of Production: 1938-2003

Mine Production: 12,195,585 tons milled = 5,276,363 oz Au, 2,933 tons marketable arsenic

Years of Remediation: 2000-current

Mine Remediation: 55,105 tons waste reclaimed = 13,547 oz Au

Introduction

The Con Mine first achieved gold production in 1938 and was the founding industry of Yellowknife, NWT, the city that grew up nearby. Con closed in 2003 and the mine is now under remediation.

History in Brief

Gold was found on the west side of Yellowknife Bay by government geologists in September 1935. This discovery led to a small staking rush to this area in September and October 1935, during which time the 'Con' claims were staked by Cominco Limited. Staking was performed when winter set in and no prospecting could be done until the following spring. Early work in 1936 disclosed a large group of veins associated with a wide shear zone. Development began right away, and gold production was attained in September 1938 making the Con Mine the first gold mine in the N.W.T.

Operations at Con Mine were associated with production of Rycon Mine ores, an adjacent property in which Cominco had controlling interest. It is because of this that in the early years, Con Mine was known as the Con-Rycon Mine, although they were in effect the same operation. The Rycon property produced gold from 1939 to 1979. The Con Mine property would eventually expand to include several other adjacent properties including the 'N'kana' (Vol), 'Negus', 'PRW' (Yellorex), 'Aye', 'Star', 'Rose', 'Meg', and 'Kam' groups of claims. Most of these were not productive claims, although the 'N'Kana' claims produced briefly in 1964-1967. The 'Negus' claims were productive under a separate company between 1939 and 1952 (Negus Mine), becoming part of Con Mine property in 1953.

Because of conditions attending World War II, production ceased at Con Mine in 1943. During the shutdown, geological evaluation of the regional geology disclosed the potential for a faulted extension of the newly discovered Giant Mine orebody within the Con-Rycon property. This new zone, intersected in 1946, became known as the Campbell shear zone and entered production in 1956. All production at Con Mine after 1963 came from this rich ore zone. The Con Mine became an expensive operation in the late 1960s due to rising costs of production and the static position of gold on world markets. Cominco came very close to closing the mine in 1971, but the sudden ascent of gold values assured the mine's future. New reserves were uncovered to a depth of 6,000 feet or more, so the company decided to sink a new production shaft. The new shaft became known as the Robertson shaft and was placed into operation in 1977.

Cominco sold the Con Mine in 1986 to Nerco Minerals Company Limited. Nerco began an extensive program designed to modernize the aging operation. They reopened some of the older workings in 1990 in order to re-mine refractory ore zones, and installed a state-of-the-art gold recovery plant known as an Autoclave to safely process those ores. Investing much capital into the operation, Nerco increased the value of the mine and put it up for sale in 1993. Miramar Mining Corporation Limited acquired the operation and continued with production and development until 2003 when economic ore reserves were depleted. Remediation of the old mine is ongoing as of 2006.

Geology and Ore Deposits¹

The mine is located within the northerly trending Yellowknife volcanic belt. This belt is a sequence of northwest striking, steeply dipping Kam Group mafic flows and tuffs which have been intruded by numerous dykes and sills. To the east, the volcanic belt is overlain by a turbidite basin and to the west, by the Western Granodiorite. In the upper part of the Kam Group, the metamorphic grade is greenschist and grades westward to amphibolite grade near the contact of the Western Granodiorite.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

The Yellowknife Bay Formation, the uppermost formation of the Kam Group, is cut by the Campbell shear zone. The strike of the Campbell Shear changes from north to northeast as it trends to the south and dips moderately west, volcanic flows strike 050 degrees and dip steeply to the southeast. The shear zones vary in width and contain numerous splays, flexures and blocks of unshered to weakly shered volcanics. The Campbell shear zone ranges from over 500 feet to less than 100 feet in width, has been traced for approximately 8 kilometers, and is truncated to the north by the West Bay fault. The shear is marked by a chlorite carbonate schist with increasing sericite, sulphide and quartz content in zones of better gold mineralization.

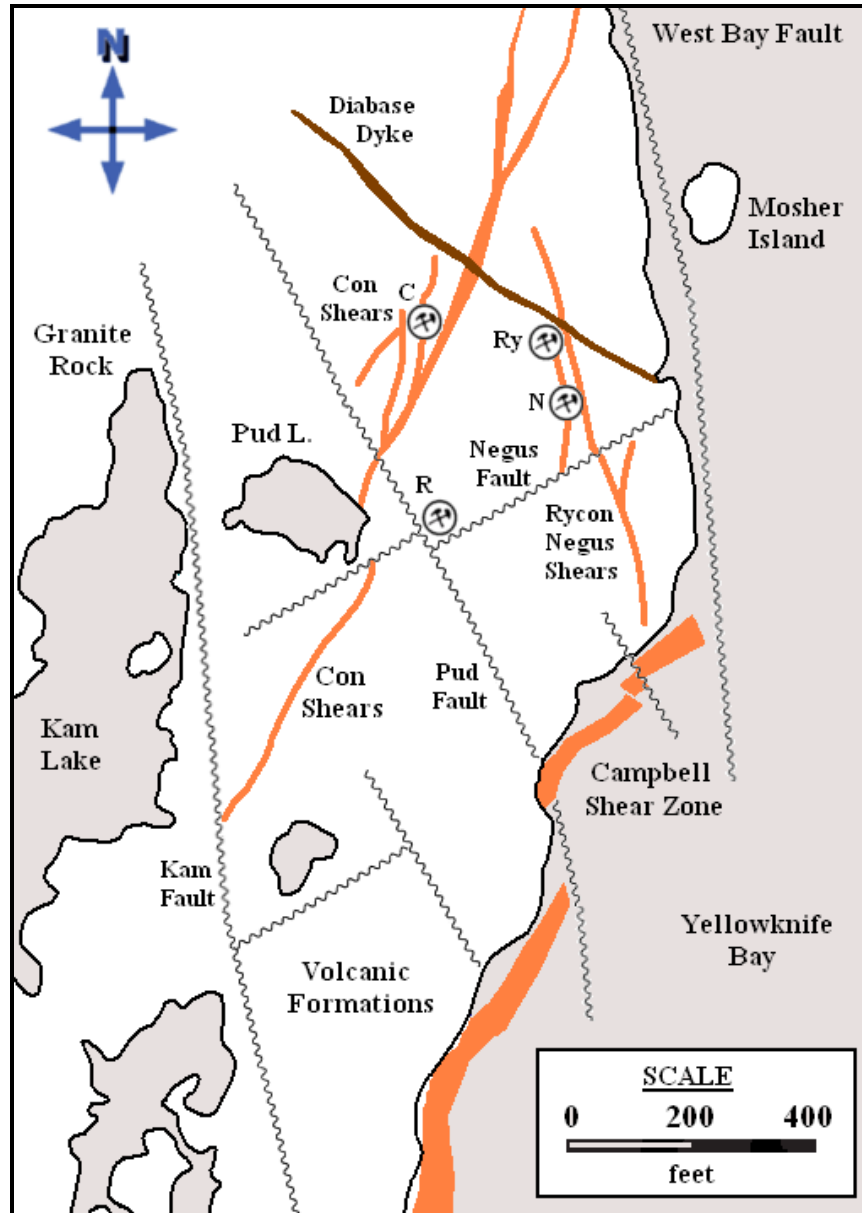


Figure 1. Simplified surface geology plan, Con Mine area.

Cominco Limited (1937-1986)

Early prospecting and development of the 'Con' claims in 1936 disclosed a large group of veins in the vicinity of Rat Lake, east of the north tip of Kam Lake in the middle of the claim group. These veins, collectively known as the Con shear zone, strike northerly and dip between 50 and 70° to the west. Prospecting during the spring and summer of 1936 was conducted by a small crew under the direction of Bill McDonald, Cominco geologist. The original tent camp and located on Kam Lake, but this was moved early in the summer of 1936 to the shores of Yellowknife Bay.

By the end of 1936, the Con shear zone was explored by about 4,500 feet of diamond drilling, trenching, and a 50-foot inclined prospect shaft on the C-10 vein, one of the most promising deposits. Cominco engineers and management were so impressed at the high-grade showings at the 'Con' claims that the decision was made to proceed with shaft sinking and mill construction - a very unorthodox procedure considering the limited development undertaken to that point. Bill Armstrong was in charge of construction during 1936-1938. A large construction program was initiated in July 1937 with the objective to place the mine into production by May 1938. A 100 tons per day milling plant was under construction and a complete mining plant was being installed. A camp site for 200 men was laid out on the shores of Yellowknife Bay across from Mosher Island (Lord, 1941).

Sinking of the main production shaft, the C-1, began in the summer of 1937. By the summer of 1938 the shaft was 250 feet deep and two levels were being mined at the 125- and 250-foot levels. The mine had a delay of production early in 1938 because of a delay in the delivery of fuel from Norman Wells, but a temporary supply was brought from Fort McMurray. Pre-production expenditure was about CDN \$1,100,000 (1939 figures).



Miramar Con Mine Ltd.

Figure 2. *Con Mine 1938-1939.*

Production Starts

Ore from the Con Mine workings was introduced into the mill on July 20th 1938 and the first gold bar was poured on September 5th 1938. The original mill was of 100 tons per day capacity. In the fall of 1939 capacity was increased to 175 tons per day, in part due to introduction of Rycon ore (see below), by the addition of extra grinding and filtering equipment.

Rycon Mines Limited

In August 1937, Cominco obtained a controlling interest in a property adjacent to the 'Con' claims. This property was the 'P&G' claims and had been staked by Tom Payne in August 1936. They were owned by Payne's company Ryan Gold Mines Limited. Cominco sought the claims as an additional ore reserve and purchased a 60% interest through the formation of a new company - Rycon Mines Limited. Shaft sinking on the R-51 vein (part of the Rycon-Negus shear zone) began in December 1938 and the Rycon shaft was sunk to 250 feet for exploratory purposes. In 1939, ore from Rycon Mine was added to a secondary circuit in the Con milling plant. All development into the Rycon area took place underground through crosscuts from the 500-foot level of the C-1 shaft, which had by then been extended to that depth. After 1939 the Rycon shaft was used only for ventilation and emergency escapeway purposes. Raises from the deeper levels of the mine connect up to the 250-foot level of the Rycon workings (Lord, 1941).

Mining Operations

In 1938-1939 all operations underground were labour intensive as no mechanized equipment was in use at the Con Mine. All tramming of ore was done by hand in 16 and 20 cubic foot mine cars, which were loaded directly onto the shaft cage for hoisting and dumping. Ore was mined by shrinkage stoping methods. It was found that lateral workings did not require timber supports in most cases (Lord, 1941).

Milling Operations 1939

Con and Rycon ore was separated into different ore bins upon hoisting to surface. Feed from the bins was crushed at different times and the Con and Rycon ores were sampled to determine head assays, from which gold recovery was established. Feed was crushed in a 10 inch x 20 inch Traylor jaw crusher, passed under a magnet, elevated to a Hummer screen with 3/8 inch openings, undersize being sent for grinding while oversize material was crushed again in a TY gyratory crusher. Secondary crushed product was dropped back onto the conveyor feeding the bucket

elevator for screening thus closing the circuit. Fine ore was then further crushed by a set of Denver rolls, from which Con and Rycon ore samples were sent for assaying to determine a gold content for each mine. Fine ore was then sent for storage in the 160-ton fine ore bin, ahead of the grinding circuit. Grinding was performed in a 6 foot x 4 foot Hardinge ball mill. Discharge from the mill passed through a 6 foot x 25 foot Dorr Duplex classifier, the sands from which were re-ground through a 5 foot x 9 foot Allis-Chalmers secondary ball mill and a 12 inch x 18 inch Denver mineral jig before returning to the classifier. Classifier overflow was pumped over three 8 foot x 4 foot blanket tables, with the tailings entering the cyanidation circuit. The cyanidation circuit consisted of two 30 foot Denver thickeners, four 16 foot x 16 foot Denver agitators, a primary 8 foot x 12 foot Oliver drum filter, and a secondary 11-½ foot x 14 foot Oliver drum filter. Thickener underflow entered the first two agitators, was then filtered, then re-pulped, then pumped into the remaining two agitators, and filtered again. Gold was recovered from the blanket tables and pregnant solution through bullion fired furnaces (Gray, 1940; White et al., 1949).

Power Plant

The mining and milling plant at Con Mine was powered by diesel engines during the early years. This was replaced in 1941 by hydroelectric power. In 1938 the mine had six Ruston-Hornsby diesel engines, three of which drove two-stage Canadian Ingersoll-Rand air compressors (total 1,900 cubic feet per minute output), and three of which drove 550 volt Westinghouse electric generators (total 624 kilowatt output). Heat was supplied by a wood-fired Inglis-Supreme boiler of 125 horsepower (hp) at the campsite. A standby 15 kilowatt generator was housed in the boiler house and could power the mine camp in an emergency. A 50,000 gallon oil tank was erected near the boiler to supply the plant with oil once the boiler units were converted into oil-burning vessels (Gray, 1940; Lord, 1941).

Hoisting Plant

The C-1 shaft, the primary production shaft until 1977, was a 3-compartment vertical shaft that at year-end 1939 was 500 feet deep. It was originally equipped with a 2-drum 42 inch x 30 inch Canadian Ingersoll-Rand electric hoist. The Rycon shaft, not in use after 1939, was also a 3-compartment shaft sunk to a depth of 250 feet using a 1-drum 8x6 Circo 24 inch x 18 inch air hoist. Both shafts were fitted with timber headframes: the C-1 headframe was 52 feet tall and the Rycon headframe was 35 feet tall (Gray, 1940). In 1941-1942, the C-1 headframe was retrofitted with steel framing and raised to 105 feet to service a deepened shaft. A new electric hoist was also installed: a 2-drum Canadian Ingersoll-Rand with 96 inch x 70 inch drums (White et al., 1949). Centered around the C-1 shaft and mill, service buildings included hoist house, powerhouse, assay lab, refinery, warehouses, mine dry, blacksmith shop, mine office, machine and carpenter shop, garage, 15,000 gallon water tower, and six 50,000 gallon oil tanks.

Mine Camp

The camp was completed during 1938 and consisted of three 2-story bunkhouses each housing 44 men, one 1-story bunkhouse for overflow and housing 24 men, cookhouse for 150 men, recreation hall with bowling lanes and gym, a six bed hospital, two houses for senior staff, staff house for single staff members, a small guest house, warehouses, 15,000 gallon water tank, and boiler house. Also erected at the campsite was a cold warehouse for Burn's Meat Company, which was used to service the entire mining district (Gray, 1940; Lord, 1941). In 1939, the mine operation employed 160 people. Annual turnover rate was about 30%. Henry C. Giegerich was mine manager. By the fall of 1943, when the mine shutdown for the duration of World War II, the mine employed 170 people, 80 of which were married, 15 women, and 15 natives. There were also 40 children residing in the camp.

Mine Expansion 1940s

In the fall of 1940, ore containing arsenopyrite (sulphide ores) were encountered underground in the C-1 shear zones. The ore was refractory in nature and untreatable with cyanidation techniques. Flotation machines were installed in April 1941 and flotation concentrates were stockpiled pending the installation of a roasting plant. (Lord, 1951) During 1941-1942 the C-1 shaft was deepened to 950 feet and Rycon ore was being mined at the 900-foot level. A ventilation raise connected the 950-foot level with the bottom of the old Rycon shaft. The deeper C-1 shaft required a new, upgraded steel headframe (raised to 105 feet height), larger hoist (a Canadian Ingersoll-Rand 96 inch x 70 inch 2-drum 300 hp electric unit), and also a modified milling plant as refractory ores were being encountered. Mining operations also became more mechanized and locomotives were introduced to tram ores as distances from stopes and the shaft increased. Equipment included four 2-ton Mancha locomotives, 38 ft³ capacity Wabi ore cars, four Eimco mucking machines, 35 Ingersoll-Rand stoper drills, and eight DA35 Leyner drills. A new skip system was installed in the shaft compartment to allow quicker hoisting and disposal of ore and waste rock. Compressed air was supplied by two Canadian Ingersoll-Rand 1,750 cfm units and two Canadian Ingersoll-Rand 640 cfm units, all driven by electric motors. Thus, a total of 4,700 cfm of compressed air was available for mining operations. A 420 hp Ruston-Hornsby diesel engine driving a 312 KVA generator was kept on standby for emergency power generation in the event of power failure (Lord, 1951; White et al., 1949).

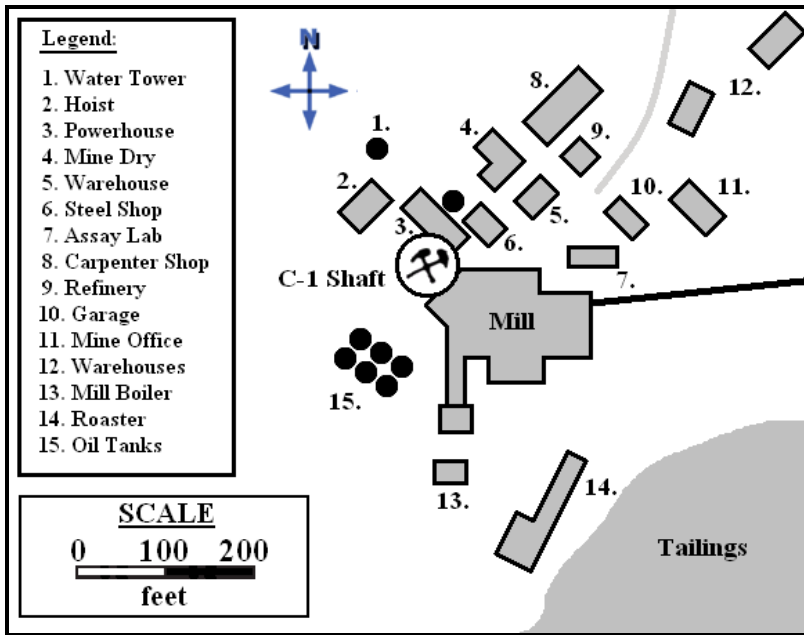


Figure 3. Surface plan of the C-1 Shaft area, c.1960.

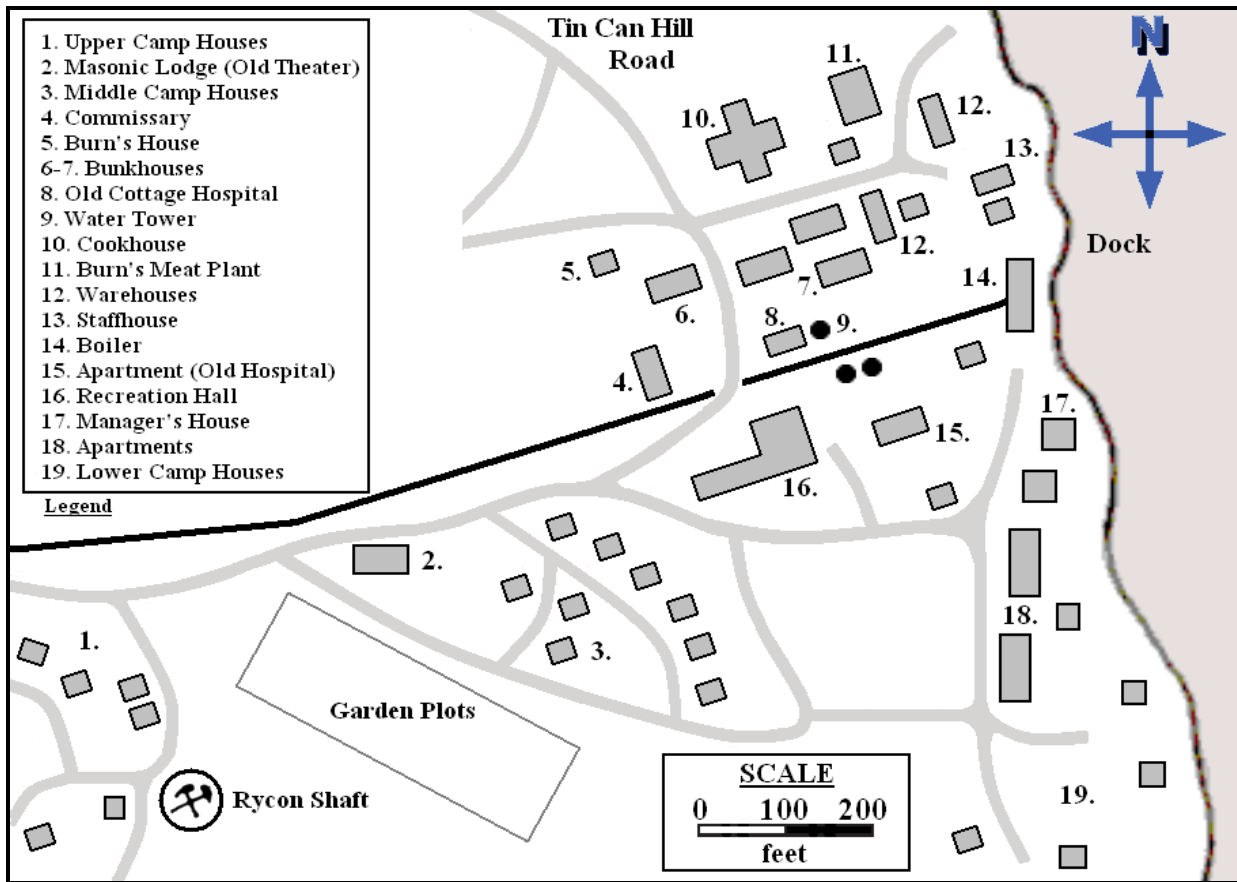


Figure 4. Surface plan of the Con Camp area, c. 1960.

Bluefish Hydro

Construction of the new hydroelectric plant took place during the summer of 1940 at Bluefish and Prosperous Lakes, north of Yellowknife. Harry Ingraham was in charge of the work. Delivery of hydropower commenced in January 1941 and helped to lower the high costs involved with diesel power generation. Con Mine's power plant was gradually converted to operate by electric motors and the diesel engines were phased out of normal service during the 1940s (White et al., 1949).

Milling Upgrade and Roaster

Construction on a further mill extension was under in the fall of 1941 that increased capacity of the plant to 300 tons per day. This program included the installation of a new mill, cyanidation and filtration equipment, a larger flotation unit, and an Edwards roaster unit to treat sulphide concentrates. The expansion program was completed in March 1942. Full capacity of the new plant was limited due to wartime labour restrictions (operating at an average of 175 tons per day), and the roaster only operated between April and November of 1942 before ceasing for the duration of World War II (Lord, 1951; White et al., 1949).

The mine expansion meant an increase in manpower, and the period of 1941-1942 was busy with new construction at both the mine and campsites. A new 44 man bunkhouse was erected, the cookhouse was expanded to accommodate 250 men, several new housing units including a new apartment (4-unit), bungalow houses, and management housing, a new 2-story hospital to replace the original cottage hospital, new commissary, curling rink, and theater building. At the mine site additions were made to the mine shops, office, assay lab, and warehouses (White et al., 1949).

Milling Operations 1942

Although rated at 300 tons per day, the new plant operated at only half-capacity due to war-time labour and supply limitations. Con and Rycon ore were handled and crushed separately, with each ore being milled at different times of the month. Separate gold recovery was based on head assays of feed ore and metallurgical tests on composite head samples. Con ore passed through a 24 inch x 36 inch Telsmith jaw crusher set at 6 inch. Product was stored in a 300 ton fine ore bin feeding a 12 foot x 6 foot Hadsel mill. Mill product was collected in a 10 foot hydro-separator, with overflow being pumped directly to the cyanidation circuit, and underflow sent to secondary grinding.

Rycon ore was crushed to $\frac{3}{4}$ inch in the 10 inch x 10 inch Traylor jaw crusher and a TY gyratory crusher in closed circuit with a 3 foot x 6 foot Niagara screen. Screen undersize was sent for grinding in the 6 foot x 4 foot Hardinge ball mill, which discharged to a 6 foot x 22 foot Dorr Duplex classifier in closed circuit with a 5 foot x 9 foot Allis-Chalmers ball mill and Denver mineral jig. Jig concentrate was amalgamated.

Classifier overflow from the Rycon circuit and hydro-separator overflow from the Con circuit were combined into the cyanidation circuit, composed of 40 foot and 30 foot Dorr thickeners in parallel. Both thickeners underflowed to feed three 20 x 20 Denver agitators in series. The last agitator discharged directly to an 11- $\frac{1}{2}$ foot x 14 foot Oliver drum filter. Filter cake was re-pulped with barren solution and sent to another bank of agitators (three 16 foot x 16 foot tanks). The final filter, a 14 foot x 16 foot Northern Foundry unit, washed the lime and cyanide from the material. Thickener overflows were sent for the precipitation process (see below).

Final filter cake was sent for flotation. First it was conditioned in a 14 foot x 8 foot Denver agitator, then processed in ten #24 Denver flotation cells. Flotation concentrate discharge was pumped to the roaster building and treated in another cyanidation circuit, consisting of a 20 foot Denver thickener and 6 foot x 6 foot Northern Foundry drum filter. Filter cake was processed through the 13 spindle Allis-Chalmers Edwards-type roaster unit. Gas was processed through a cyclone to recover gold-bearing dusts, which were sent back into the cyanidation circuit. Other gas was expelled from the plant through the steel roaster stack.

Roaster calcines were classified in a 14 foot Dorr Simplex classifier. Sands were re-ground in a small 4 foot x 4 foot Denver ball mill. Classifier overflow was thickened and washed in a 20 foot Denver thickener and filtered in a 6 foot x 6 foot Northern Foundry drum filter, and thickener overflow was rejected as mill tailings. Cake from the washing filter was re-pulped with water and passed through two 12 foot x 12 foot Denver agitators. Cyanide was added to the agitators. Discharge was diluted with barren solution and again thickened in a 15 foot unit, then again filtered in two 4 foot American leaf filters. Overflow from the thickener was sent back into the milling circuit to act as pregnant solution. Filter cake was sent for precipitation.

Filter cake and overflow from the 1st stage cyanidation process was sent into the precipitation process, which consisted of a 1000 tons per day Merrill-Crowe unit and a 36 inch x 36 inch Perrin press. Gold was refined using two

King bullion furnaces and the product shipped to the Royal Canadian Mint. Rycon settlement was made through cash payment to the interests of Ryan Gold Mines Limited (White et al., 1949).

War Closure

Mining operations were focused on the 375-foot and 500-foot levels during 1942-1943 although extensive development had taken place on the lower levels of the mine. Operations at Con Mine were strained due to wartime conditions, especially after an official order by the Government of Canada that asked the mines to reduce their production rates. In the summer of 1943 it was decided to suspend normal operations for the foreseeable future. Hoisting of ores ceased August 15th 1943 and milling stopped September 11th 1943, although the plant was put into a position where production could recommence as soon as warranted. An adequate crew was retained to put the mine back into production at a moments notice. Most of the crew was transferred to other Cominco operations, specifically the Pinchi Lake mercury mine in B.C. that had a greater contribution to the war effort. (Lord, 1951)

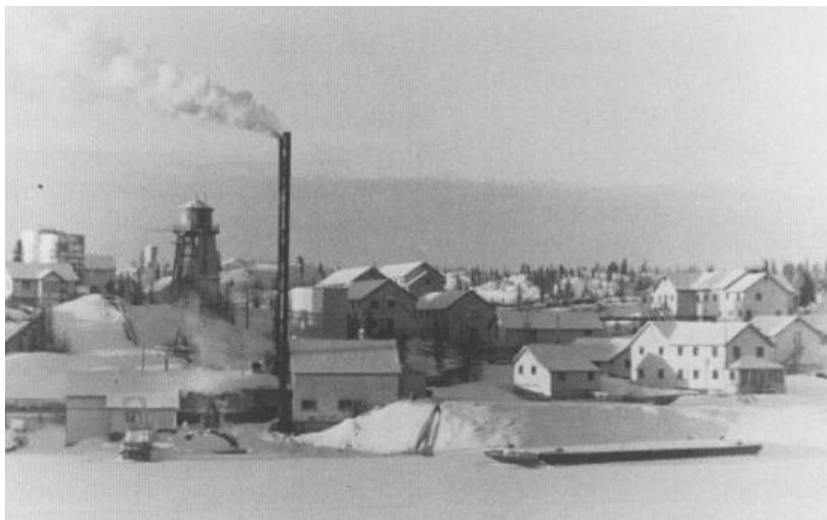
During 1944-1945, extensive underground development was performed in order to keep ore reserves developed. The C-1 shaft was deepened to the 1,400-foot level and diamond drilling probed the extensions of the Con shear zone. New levels were established at 1,100-, 1,250-, and 1,400-foot depths (Lord, 1951).

Geological Investigation

Following the discovery of important gold deposits at the Giant Mine in 1944, some of which terminated against the West Bay fault, a geological study was initiated by Cominco to locate a possible faulted extension of that ore. Mapping undertaken by Neil Campbell, a Cominco geologist, revealed several similarities between the Con/Negus and Giant properties that were interpreted to be displaced by faulting. In January 1946, a joint venture program was initiated between Con and Negus Mines Limited to drill eastward from Negus Mine to intersect a theorized ore zone. A single hole, 2,092 feet in length, was drilled on the 1,250-foot level of the Negus Mine and intersected the structure. The C-1 shaft was subsequently sunk an additional 1,000 feet to the 2,300-foot level, and a crosscut was driven easterly to intersect the new shear zone. It was named after geologist Neil Campbell whose work was instrumental in its discovery (Lord, 1951).

Con Mine Reopens

Although the first year after the war was difficult because of continued labour shortage, Cominco was able to restart operations at Con Mine in August 1946. The mill was put back into operation and the first post-war brick was poured on September 18th 1946 (Lord, 1951). Extensive lateral work was completed into the Campbell shear zone during the following years; however no production would come from the new ore zones until the mid 1950s. Production was therefore focused on the stopes within the Con and Rycon shear zones during the late 1940s and early 1950s. Deeper development into the zone started in 1950. During this period, the B-3 winze was sunk from the 2,300-foot level to a depth of 2,825 feet at a location 3,500 feet east of the main shaft.



Miramar Con Mine Ltd.

Figure 5. *Con Mine camp, 1940s.*

Milling resumed in August 1946, but the roaster was not placed back into operation until July 1948. Scrubbers were installed in 1949 on the roaster stack to help reduce toxic emissions. In anticipation of the processing of new ores from the Campbell shear zone, the mill was expanded during 1949-1950. The Hadsell mill was replaced with a standard ball mill. Further expansion was completed in 1955 and 1956 that introduced new crushing circuits and an expanded cyanidation circuit. Mill capacity was now 500 tons per day.

Unionization

Con Mine became unionized in 1947 when the Local 802 of the International Union of Mine, Mill and Smelter Workers became organized. Con management disputed the organization, citing unreasonable wage demands during a difficult period for the mine (White, et al., 1949).

Camp Expansion

A few additions were made to modernize the Con Mine camp. In 1948, a new regional hospital was built at the Yellowknife townsite and the Con Mine hospital building was renovated into a 4-suite apartment. A new recreation hall (known as Jewitt Hall) was built in 1947 to replace the one that burned down in 1946. The original section was a gymnasium and bowling alley. In 1957, a large curling rink was added, replacing the old, decrepit curling rink. More housing was erected for employees and their families, and the Con Mine camp became subdivided into three parts known as Lower, Middle, and Upper (“Rycon”) camps.

Purchase of Negus Mine

In September 1952 the Negus Mine, adjacent to the southeast of Con Mine, closed due to lack of economic ore and financial reasons. In March 1953 Cominco bought the mineral rights to the entire property, a parcel of the surface lease, and all the underground workings, including the headframe and hoisting plant. A raise was driven from the 2,300-foot level of Con Mine to connect to the lower workings of Negus Mine, and the old Negus workings were used for ventilation. Some 7,300 tons of material grading 0-30 ounces per ton gold was trammed from the Negus workings in 1957. This was ore that had been broken during the mine’s previous life and was easily recoverable (mine records).

Mining Operations 1950s

The early 1950s saw the rapid completion of mining from the old Con and Rycon shear zones in preparation for the opening up of Campbell shear zone workings. The Rycon property only contributed 10% of total production at Con Mine during this period, due primarily to lack of ore, insufficient manpower and resources, and legal issues surrounding the Rycon deal. The mill circuit was constantly tied up because Con and Rycon ores were treated separately, and lots of labour was used to clean out the circuit for rotation of ores every month (mine records).

Ore from the upper levels of the new Campbell shear zone were being introduced into the mill by 1956, and by 1958 practically all ore had been mined out from the Con and Rycon shear zones. Small-scale mining for the purpose of recovering stope pillars continued until 1963. Early mining of the Campbell shear zone indicated a zone of lower and more erratic grades than the Con shear zones. Con geologists had assumed that the shear would be a continuous orebody more amenable to older mining methods, but early work in 1956-1957 showed that the shear was in fact a series of erratic lenses with steeper dips. As time passed mine productivity increased as more knowledge about the new zones was gained and applied to mining operations. Mining methods changed from the original shrinkage stoping to cut-and-fill methods, using mine tailings as backfill material (mine records).



Miramar Con Mine Ltd.

Figure 6. *Con Mine C-1 Shaft area 1960s.*

The following underground equipment was in use during 1952: Eimco mucking machines (Models 12, 12B, 20, 21B), one Atlas locomotive, five Mancha "Little Trammer" locomotives, one Mancha "Titan" locomotive (used on 2,300-foot level haul), two Universal locomotives, 70 Dominion side and end-dump ore cars (16 cubic foot), 14 Vulcan side and end-dump ore cars (20 cubic foot), 10 E.Long rocker-dump ore cars (35 cubic foot, used on 2,300-foot level haul), and 18 Wabi rocker-dump ore cars (38 cubic foot). Further additions to the mining fleet took place during the 1950s. The B-3 winze was originally sunk using a Canadian Ingersoll-Rand PSR 8x6 air hoist. The old Negus Mine hoist, a Canadian Ingersoll-Rand 48 inch x 36 inch 2-drum electric unit, was installed in the B-3 winze for production operations in the mid-1950s. A 40 cubic foot skip was in use at the B-3 winze by 1958. The B-3 was progressively sunk to further depths in the 1950s-1960s, attaining a final depth of 4,900 feet in 1966. (mine records)

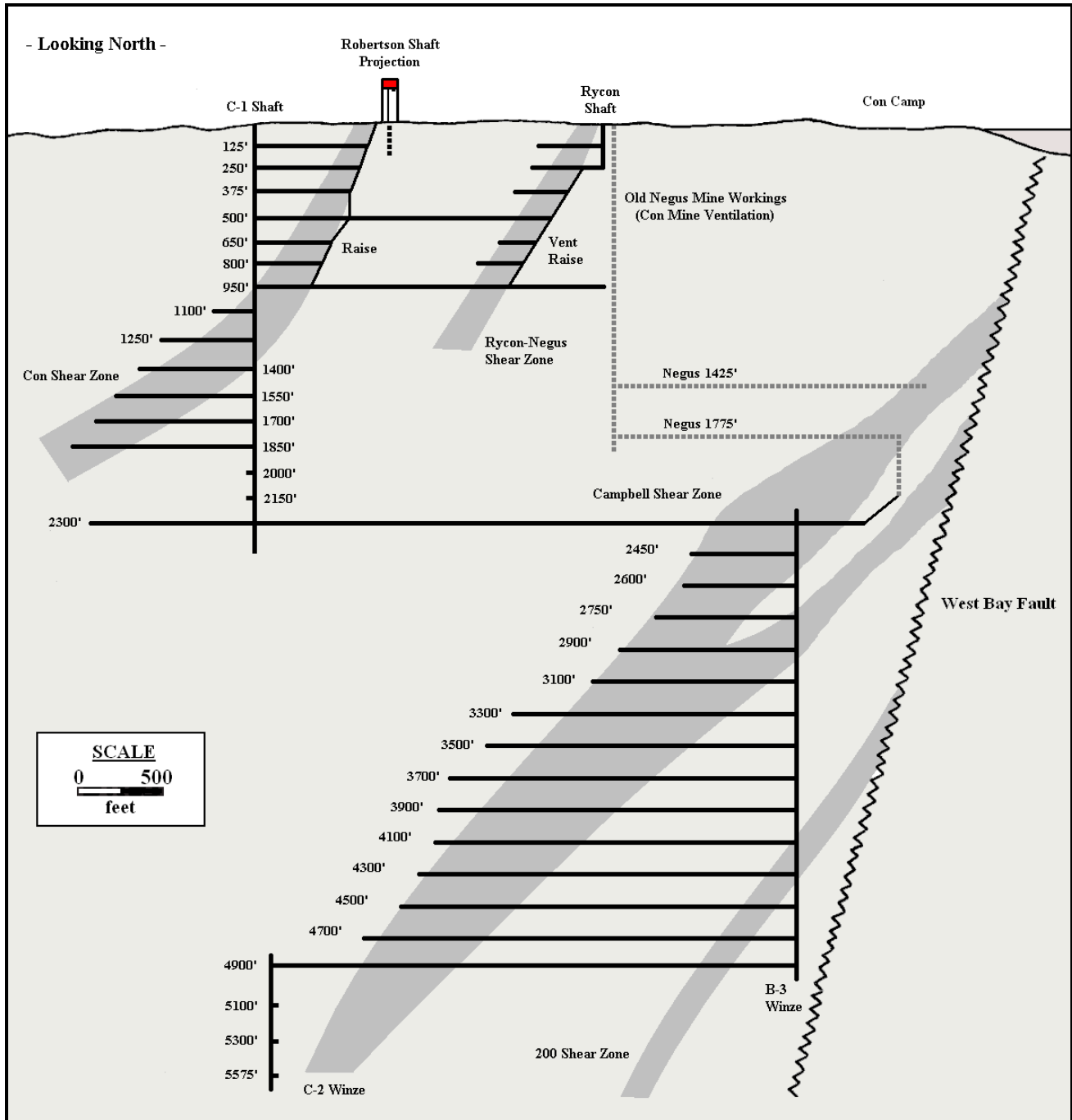


Figure 7. Simplified cross section of underground workings at Con Mine, circa 1972, showing a projection of the future Robertson Shaft. Looking north.

Employees 1950s

In 1956 it was estimated that 45% of the general roll at Con Mine were recent Italian or German immigrants to Canada. For many years immigrants would form a large portion of Con's workforce. During 1957-1959, Jay Colthorp was mine manager, and under him were George Clayton, mine superintendent; John Giovanetto, mill superintendent; George McVittie, chief geologist; Joe Scarborough, chief engineer; H. Johnstone, chief mechanic; and J.H. Winter, accountant. In 1960 the company worked with the federal government to promote native employment at the Con operations, but there was a lack of willing participants. Also in 1959, Con maintained 58 housing units (33 bungalows, four houses and 21 apartment units) at the mine camp, plus four houses in Yellowknife. Most of these units were fully serviced with water and sewage. There were also two private houses on the property. About 50 single-men were living in the bunkhouses. Twelve single-staff members lived in staffhousing. Total employment at Con Mine during the late 1950s was approximately 190 (mine records).

Mining Operations 1960s

Mining and production focused in the upper levels of the Campbell shear zone in the late 1950s and throughout the 1960s in an area just below the Con Mine camp and Tin Can Hill. A large portion of production came from the Rycon property. In 1953, only 3% of production was derived from the Rycon property. This was increased to 40% in 1960, and by the late 1960s over half of total mine production was coming from Rycon ground (mine records). In 1965, production was being maintained within the 101, 102, and 103 zones within the Campbell shear zone, located between the 2,900- and 3,900-foot levels. Mine development and exploration on the 4,100- and 4,300-foot levels were driven west through the Campbell shear, and then crosscuts were driven south to test its extents. Many ore shoots were being located, some of which were low-grade. Milling capacity was decreased to 470 tons per day during this period when the regrind ball mill was removed from service. Amalgamation also ceased in 1968 (Egli, 1977).

Yellorex Exploration

Exploration also focused on the Yellorex or PRW property to the southeast of Con Mine. In 1967, Cominco optioned the 'PRW' claims from Yellowknife Bear Mines Limited and began a drive on the 2,300-foot level in the footwall of the Campbell shear zone. By 1971 over 2,700 feet of drifting had been completed, of which 1,500 feet was within the Yellorex property. Diamond drilling at 200 to 400 foot intervals along the drive in search of gold-bearing ore shoots related to the shear zone was unsuccessful (mine records).

Hoistroom Fire

Disaster struck the mine in 1964. A fire on March 15th completely destroyed the C-1 hoisting plant and knocked out production operations at Con Mine for a period of four months. Many employees were laid off although other regional mines offered to find work for them on a temporary basis (News of the North, Mar. 19th 1964). A new hoist, a 2-drum 96 inch x 80 inch Nordberg 800 hp electric unit, was installed and operations resumed on June 10th 1964. Shortage of crews after the fire caused production delays during the remainder of the year.

Vol Production

During the late 1960s, Con mined and produced ores from the 'N'kana' claims property, located to the north of Con Mine camp and underlying much of downtown Yellowknife and all of Old Town. The area being mined was located under Tin Can Hill and School Draw Avenue. Development began in 1959 when Cominco optioned the claims from owners Conwest Exploration Company Limited. Two drives on the 2,900- and 3,100-foot levels explored the hanging wall of the shear zone. Production of development ore took place during 1962-1964 during which time 4,572 tons of ore were milled. In December 1964, Cominco incorporated Vol Mines Limited to acquire the property, pursuant to the option agreement with Conwest. From 1964-1967 when the property was being 'commercially produced', 21,961 tons of ore was mined and milled to produce 10,120 ounces of gold. More development was undertaken in 1968, 1978-1979, and 1981 on the 2,900-, 3,100-, and 4,300-foot levels, although no further production came from this area (mine records).

Years:	Vol Ore Milled:	Grade:	Gold Produced:
1964-1967	21,961 tons	0.46 oz/ton	10,120 oz

Table 1. Vol production, 1964-1967. ('N'Kana' claims)(source: mine records)

Expensive Production

Con Mine suffered from low gold prices (\$38 per ounce), escalating costs of operations, and lack of suitable labour during the mid to late 1960s. The B-3 winze was viewed as a bottleneck to the entire operation, as its hoisting capacity was half that of the C-1 shaft. That meant that the B-3 complex was unable to keep up with the rate of production and supply the mill with steady ore. If the mine wanted to develop new ore below the 4,900-foot level, deepening the B-3 winze was not a practical solution. One solution suggested in 1966 was to deepen the C-1 shaft to 6,000 feet, but this would impose other financial and logistical difficulties. Another option was to sink a second winze below the 4,900-foot level. Neither option was feasible for maintaining production in the long-term, so management decided in 1968 that the best plan of action was to sink a winze below 4,900 feet for exploratory purposes. An exploratory winze would not be expensive, and at least there would be a chance of further defining the new ore zones in the deeper reaches of the Campbell shear zone (mine records).

Depth Development

Sinking of the new winze began in April 1969. The C-2 winze was collared on the 4,900-foot level, directly underneath the C-1 shaft. It was completed in 1971 to 5,650 feet depth with three new levels at 5,100-, 5,300- and 5,575-foot depths, exploring the westerly dip of the Campbell shear zone. No extensive development was undertaken from the C-2 workings at this time. The C-2 winze, although not put into production use, was fitted with the old B-3 winze hoist. In 1970, the B-3 winze was equipped with a 2-drum Canadian Ingersoll-Rand electric hoist 72 inch x 58 inch size, a 2-ton capacity skip, and a nine-man capacity cage (Jones, 1976). By 1968 the following machinery was being used underground at Con Mine operations: Canadian Ingersoll-Rand, Holman, and Copco drills, six Atlas-Copco LM-56 mucking machines, Eimco 12-B mucking machines, 10 Mancha "Titan" locomotives, 6 Mancha "Little Trammer" locomotives, 12 Hudson side-dump ore cars (used on 2,300-foot level haul), 83 Wabi side-dump ore cars, 3 Rex ore cars, and 24 E.Long ore cars. Many of these units were reaching their age and were due for replacement. By this time the mine was planning for the use of Granby-type automatic-dump ore cars of 85 cubic foot capacity. During 1968, 10 Granby cars had been purchased by Con Mine, three of which were in use on the 4,900-foot level by November 1968 (mine records). Ores within the Campbell shear zone became less refractory and more free milling at greater depths, and roasting as part of the Con milling circuit ceased in November 1970 (Jones, 1976).

Future in Doubt

The future of the Con Mine was not looking very bright during the early 1970s with increasing operating costs fighting against a fixed price of gold. It was found that costs associated with producing through the C-2 winze would be too expensive because of the transportation distance from the surface. Therefore, mining below the 4,900-foot level was not warranted at the time. Lateral development within the C-2 winze area ceased during 1971. Cominco reported that the future of the mine rested on the gold market and the ability of the company to acquire emergency funding through the Emergency Gold Mining Assistance Act. Conditions further worsened by the destruction of the Bluefish Hydro by fire in January 1971. Repairs brought the plant back online by November 1971.

1970s Turnaround

In 1972, gold began to rise in value as the marketplace turned away from the U.S. dollar. World governments allowed gold to find its own level on the markets and by 1975 the price of gold was at a record \$200/oz. Con Mine initiated exploration within the C-2 winze area in 1972, locating additional ore that brought the mine's ore reserves to the highest point ever. Proven and potential reserves to the 5,575-foot level were 2.1 million tons grading 0.62 ounces per ton gold with the 101 zone extending beyond depths previously theorized and with the possibility of a continuation of the Campbell shear zone to 7,100 feet or more (Cominco Ltd., 1972).

Production via the C-1/B-3/C-2 shaft complex was viewed as uneconomic, so Cominco created an all-new mine plan which called for a new shaft system and increased milling capacity. The new shaft would make ore more accessible and also speed up transportation of men, equipment, and ore from underground to surface, reducing operating costs and permitting a longer working day in the headings through reduced travel time. The mill would be increased to 650 tons per day as a result of the new productivity.

The Robertson Shaft

When commissioned in 1977, the new shaft was called the Robertson, after a long-time Con Mine employee who retired in that year. Site work and excavation at the shaft site began during the summer of 1973. The shaft was collared in November 1973 to 50-foot depth and all necessary sinking equipment was installed, including a sinking headframe (The Yellowknifer, Nov. 7th 1973). Sinking began in January 1974 under a contract with Thyssen Mining and Construction Company of Canada. By March 1975 the shaft was 2,900 feet deep, with a projected completion depth of 5,800 feet (early plans suggested a future depth of 7,200 feet if warranted).



Ryan Silke

Figure 8. *The Robertson Shaft, 2000.*

Construction of the permanent headframe also began during the summer of 1975. The 250-foot tall headframe featured a German-made friction hoist. The Yellowknifer, Mar. 27th 1975). In 1976, it was decided to sink the shaft to only the 5,425-foot level, in order to have ore available sooner.

The mine was betting on the new shaft to cut enough costs that operations could survive a bad gold market. Sinking was completed in December 1976, and construction of shaft services took place during 1977. Operations commenced in June 1977. The Robertson shaft had three compartments with a total dimension of 9 feet x 25 feet. Two cages and one skip operated in the shaft. The main cage was a double-deck unit capable of holding 34 men, or a 20 ton payload. The bottom-dump skip had a capacity of 15 tons. The main cage and skip were operated by a 17.5 foot diameter Koepe-type friction hoist, driven by a 2,000 hp motor. Designed hoisting speed was 2,474 feet per minute, with the cage and skip operating in balance. An auxiliary cage used for emergency purposes held 6 men and was controlled by a separate Allis-Chalmers 1-drum electric hoist (92 inch x 72 inch). A diesel generator was

Other Robertson Shaft Facilities

A new mine office and dry complex were erected at the Robertson shaft site in 1979. In addition, all new shops and warehouse space were provided. Two Joy compressors and one Airways compressor supplied an additional 5,000 cubic feet per minute of air power to mining operations, supplementing 5,000 cubic feet per minute available through older units at the C-1 shaft. A new and improved ventilation system was established using the Robertson shaft as an intake, with the C-1 and Negus shafts as exhaust. A 150 hp Joy model 72 fan was mounted at the Robertson end, and two 250 hp Joy model 72 fans at the Negus end. Intake air was heated by propane burner during the winter months (The First Boston Corporation, 1986).

Mining Operations 1970s-1980s

The Con Mine became a more mechanized affair during the late 1970s and early 1980s, due in part to both the need to lower costs and because of the lack of skilled miners. Ore was being mined between the 4,300- to 5,100-foot levels during this period, in areas 2,000 feet north and south from the Robertson shaft. Levels were 200 feet apart. During the period 1979-1983, mechanized production (mechanized cut-and-fill using flat breasting) accounted for 75% of production, while conventional cut-and-fill mining accounted for 17% of production and development ore accounting for the remaining 8%. After 1983, conventional cut-and-fill mining increased as a result of depletion of the heart of the mechanized stopes and by 1986, conventional methods accounted for 56% of the total production (McKenzie, 1991).

Mechanized stopes were mined within the 101 zone on the 4,700-, 4,900-, and 5,100-foot levels where a number of ore lenses could be mined more effectively. Mechanized mining involved driving ramps up from the level and accessing the ore by a number of parallel crosscuts. In 1976, drilling was done using two Atlas-Copco wagon drills. Mucking was with three Jarvis-Clark scooptrams (1-yard), dumping ore into chutes for loading into ore cars on the main level (Jones, 1976; Fish, 1981).

Equipment was augmented upon the completion of the Robertson shaft. By 1980, drilling was done using one Long-Tom 2-boom jumbo drill, one Tamrock 1-boom hydraulic jumbo drill, and two Jarvis-Clark MJM 20B 2-boom jumbo drills. Mucking was done using two Wagner ST2 scooptrams and three Jarvis-Clark LHN scooptrams, dumping ore into chutes for loading onto ore cars on the main level (Fish, 1981). By 1986, 13-ton capacity trucks were used underground to haul ores to chutes in areas where distances were greater.



Miramar Con Mine Ltd.

Figure 9. *Mechanized mining, 1980s.*

Ore from all stope development, conventional and mechanized, were dropped into 4 ton Granby ore cars or 2 ton Wabi ore cars. The Granby cars were introduced during the 1970s because of their large capacity and the ability for automatic dumping. Cars were pulled by 8 ton Clayton, 2 ton Mancha, or 4 ton Titan locomotives, and ore was dumped to a loading pocket on the 5,100-foot level. Ore was loaded into the skip and hoisted to surface (Fish, 1981; The First Boston Corporation, 1986). By 1986, the mine had a large fleet of underground mining vehicles. This included one 1 yard scooptram, eight 2 yard scooptrams, three 2-boom jumbo drills (tracked), two 2-boom jumbo drills (trackless), two 1-boom jumbo drills (trackless), two 13 ton haul trucks, two longhole drills, two Long-Tom drills (tracked), three Long-Tom drills (trackless), 20 locomotives (Clayton, Mancha, Titan, and Goodman), 16 mucking machines (Atlas-Copco and Eimco), and 16 pumps on 13 mine levels (The First Boston Corporation, 1986).

Rycon Mining Ends

The C-1/B-3 shaft complex was shutdown in 1979 following completion of Rycon mining in January. It was maintained as an emergency escapeway and for ventilation purposes during the 1980s, but all mining was by then serviced from the Robertson shaft. This complex remained closed until 1989 when Nerco Minerals Company Limited rehabilitated the old workings (see below). Total Rycon production from 1939 to 1979 is listed in **Table 2**. (mine records)

Years:	Rycon Ore Milled:	Grade:	Gold Produced:
1939-1979	1,204,122 tons	0.71 oz/ton	850,908 oz

Table 2. *Rycon Mine production at Con Mill 1939-1979, which accounted for 42% of production from the Con operation during this period. Total percentage of Rycon production from the total of ounces produced at the Con Mine operation between 1938 and 2003 is 20%.*

Milling Operations 1980

The mill was upgraded to 650 tons per day capacity during 1977-1978 to accommodate greater productivity from the new Robertson shaft (see **Figure 12**). Haul trucks of 30 ton capacity were loaded at the Robertson shaft house and trucked to the mill, where it was crushed two-stage in a Telsmith 25 inch x 36 inch jaw crusher and Symons 5.5 foot shorthead cone crusher. This product was distributed into two 300 ton fine orebins by a shuttle conveyor and fed into the Allis-Chalmers 8 foot x 12 foot ball mill in closed circuit with a 78 inch Akins classifier. Overflow from the classifier proceeds to another stage of classification using a bank of three Kreb cyclones, overflow of which was sent for cyanidation, and underflow was reground in a Allis-Chalmers 5 foot x 9 foot ball mill before proceeding to cyanidation.

There were three thickeners in operation in 1980, overflow from which was removed as pregnant solution. Underflow was agitated in four tanks where sodium cyanide and lime were added. The solution from the agitators were filtered, with pregnant solution removed and filtrate processed through another series of agitators. Another stage of filtering was completed using four units to recover more pregnant solution, with filtrate being rejected as a final mill tailing. Tailings were classified in the backfill plant with larger material being pumped underground for impoundment, and smaller material impounded in the tailings pond. Pregnant solution was combined with lead acetate and clarified, followed by a Merrill-Crowe vacuum tower. Zinc dust was added to precipitate the gold from the solution. Two filter presses removed the precipitated gold and the sludge was fire refined to produce gold bars (Fish, 1981; The First Boston Corporation, 1986).

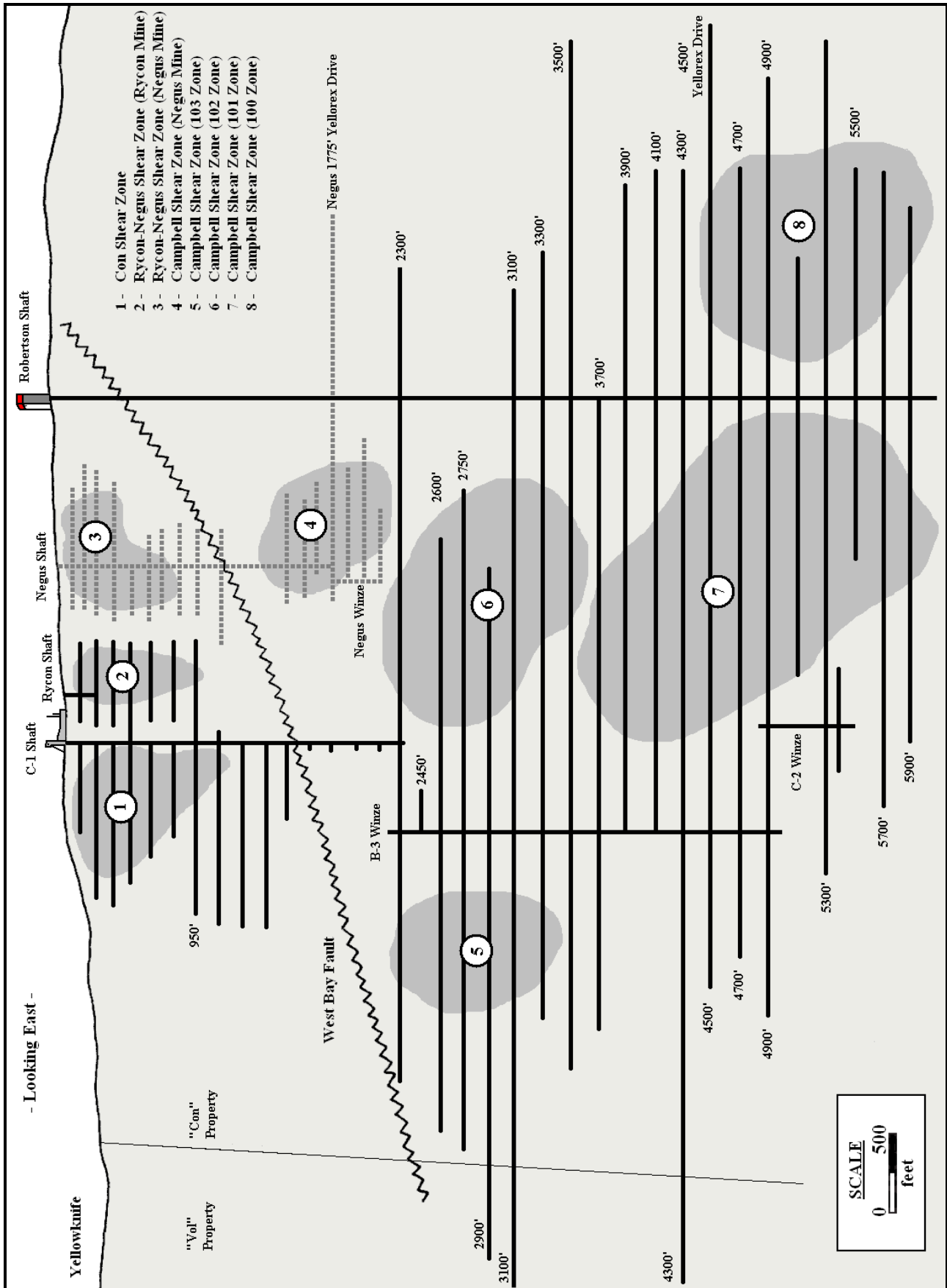


Figure 10. Simplified cross section of underground workings at Con Mine, 1990s.

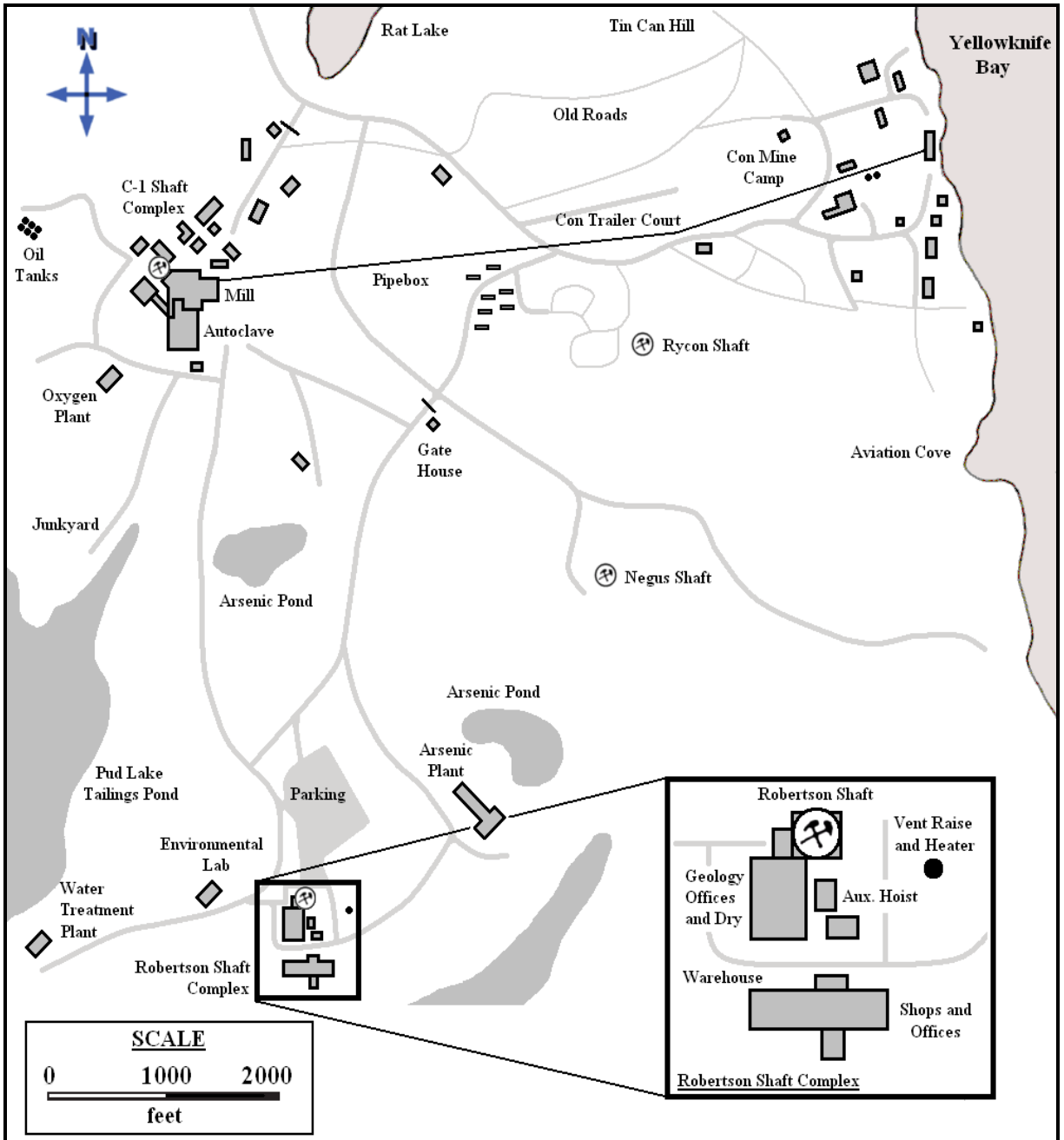


Figure 11. Property plan of Con Mine, 1990s.

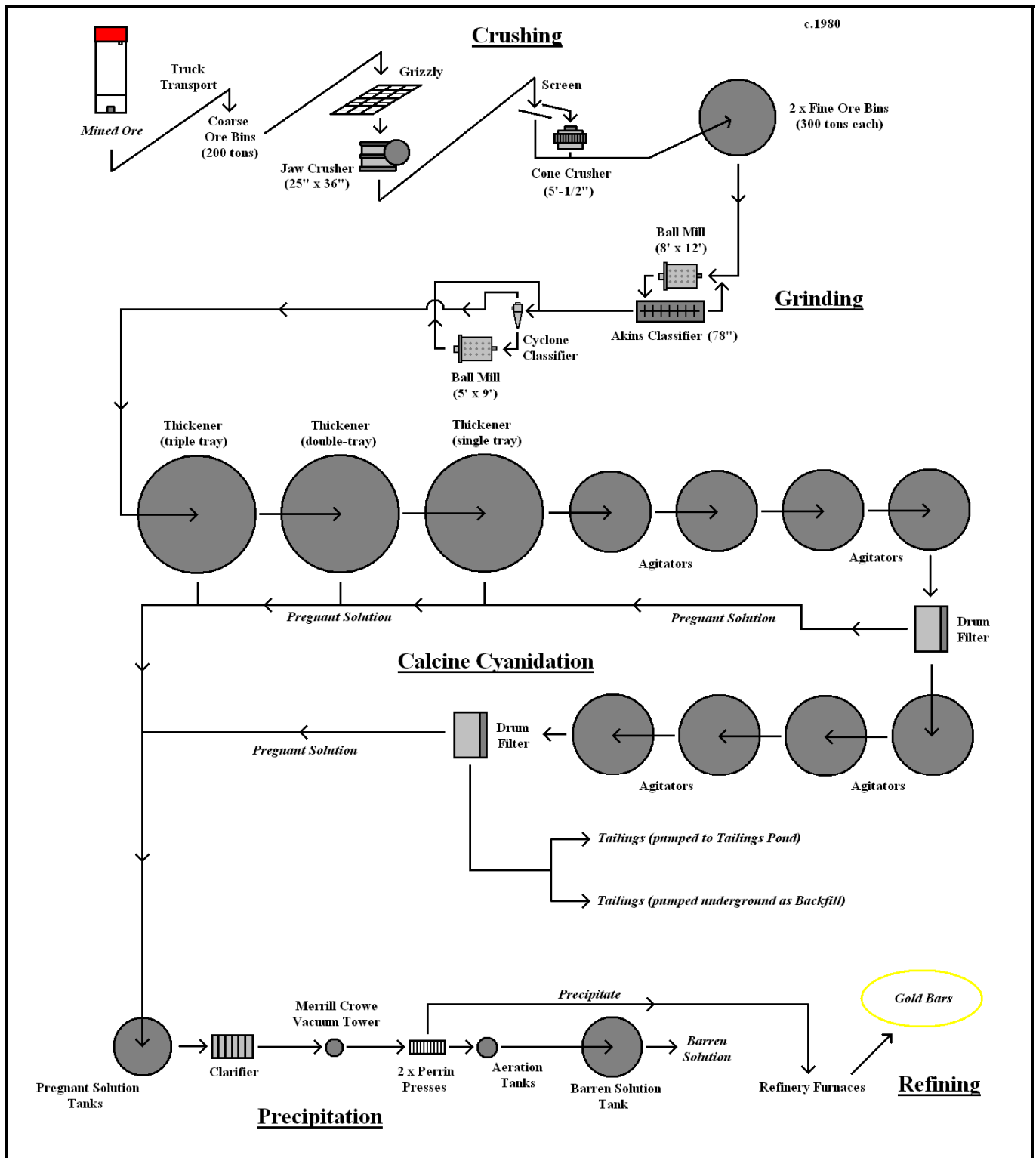


Figure 12. Con Mine milling flowsheet, c.1980. (modified from Fish, 1981)



NWT Mining Heritage Society

Figure 13. *Aerial view of Con Mine, 1980.*

Mill Changes 1980s

In order to improve efficiencies in the milling plant, various equipment additions, changes, and modifications were carried out in 1984-1985. This included the installation of a new Kopper tower mill to provide regrind capacity to keep the mill running at high rates, the replacement of the Akins classifier with a cyclone unit, the replacement of two agitators, larger vacuum pumps, and new conveyor belts (The First Boston Corporation, 1986).

Con Camp 1980s

In September 1976, the mine closed its bunkhouse services at Con Mine camp due to high costs associated with the bunkhouse and cookery operation and the physical condition of the old buildings. (The Yellowknifer, Sept. 16th 1976) Many of the older houses at Con camp were demolished during this period because they were in poor shape, and it was easier for the company to arrange for housing in Yellowknife. In 1986, Cominco reported providing housing for 60% of its employees. This included nine houses and eight apartment suites at Con Mine camp, and seven houses in Yellowknife. The company also leased 151 units consisting of trailers, houses, and apartment suites, renting them to employees at nominal rates (The First Boston Corporation, 1986).

Arsenic Plant

The arsenic plant was installed in 1982 to remove the environmentally hazardous arsenic material that had accumulated at the Con Mine between 1942 and 1970 when the roaster was operational. Arsenic sludge was impounded in special concrete ponds in two locations at the property. Starting late in 1982 the mine began to reclaim these wastes in the arsenic plant to comply with new environmental regulations that called for their removal. The plant produced high purity arsenic trioxide, a product used in the wood preservative industry during the 1980s. A small amount of gold was also recovered from these wastes (see **Table 3**).

Reclamation of the ponds could only be conducted during the summer due to the arsenic ponds freezing in the winter. Sludge was pumped into three holding pits capable of holding a years worth of sludge (recovery operations were year-round). After two stages of leaching, the solids were free of arsenic trioxide, but were still rich with gold, which was recovered in another process. The hot arsenic trioxide was fed through four evaporative cooling crystallizers. The crystals were pumped to the de-watering and drying circuit. Dried arsenic trioxide was then collected in a silo, packaged into 1,000 pound steel drums, and trucked to markets. Operation of the plant ceased in December 1985 due to high amounts of arsenic escaping through the water discharge (McKenzie, 1991).

Year:	Sludge Reclaimed:	Gold Feed Grade:	Gold Produced:	Arsenic Produced:
1983	?	1.02 oz/ton	288 oz	276 tons
1984	?	1.27 oz/ton	557 oz	1,399 tons
1985	?	1.06 oz/ton	864 oz	1,258 tons

Table 3. Arsenic Plant production, 1983-1985.

Robertson Shaft Deepening

The Robertson shaft was deepened during 1984-1985 under a contract with J.S. Redpath Limited. The 6,240-foot level was reached in June 1985, and the lower levels went into production during 1986. A new system of ore and waste passes were established, with all ore being dumped by chutes down to the 6,100-foot loading pocket, where the skip was loaded and hoisted to surface (The First Boston Corporation, 1986).

During the mid-1980s, the Cominco company was seeing an overall financial loss in its worldwide operations. Its new long-term business strategy did not include Con operations. Cominco put the Con Mine up for sale in April 1986 in hope of improving its financial picture (The First Boston Corporation, 1986; The Yellowknifer, April 6th 1986).

Nerco Con Mine Limited [Nerco Minerals Company Limited] (1986-1993)

Upon sale of Con Mine to Nerco Minerals Company Limited in December 1986 (cost was U.S. \$47 million), the new management initiated a number of measures to help make the mine a profitable one. This plan included a program of capital projects to modernize the aging operation. In 1987, Nerco funded a study by Kilborn Engineering Limited to evaluate the practices of the property and recommend improvements to increase gold production. As a result, a number of operational, administrative, productivity, and environmental improvements were enacted between 1988-1991. Nerco Con Mine Limited was formed as a subsidiary to manage the Con Mine (McKenzie, 1991).

C-1 Shaft Upgrade

The original C-1/B-3 shaft workings went offline in 1977 upon completion of the new Robertson shaft. In 1988, Nerco began rehabilitation of these workings to provide access to old ore reserves and to provide a secondary escapeway out of the mine as the Robertson shaft was developed without a manway compartment. The plan called for a complete refurbishment of the shaft structure to allow for production operations. The old C-1 headframe was torn down and replaced with a new headframe, the C-1 and B-3 shafts were rehabilitated and fitted with new manways, new Kimberley skip-over-cage conveyances were installed in both shaft compartments, and a new dumping system was arranged that stockpiled ore outside the headframe rather than immediate conveyance into the mill plant. The existing Nordberg hoist was maintained. The first waste was hoisted in March 1990 and, after extensive rehabilitation of the 2,300-, 2,600-, and 2,900-foot levels, the first ore was hoisted on June 8th 1990 from the new C-1/B-3 shaft complex (McKenzie, 1991).

Rycon Buyout

Mining of the C-1/B-3 complex centered around the production of ores from the Rycon property, inactive since 1979. Rather than form a new deal with Ryan Gold Mines Limited, Nerco bought out the company in 1990 and became sole shareholder in Rycon Mines Limited, which it then abolished (mine records).

Ventilation Upgrades

The ventilation system was upgraded to eliminate delays in production caused by ventilation plant failures. A new primary centrifuge fan of 600 hp was installed at the Negus shaft to replace the double 250 hp Joy fans, other fans were replaced throughout the mine workings, new airlocks and vent doors were installed, and underground pressure resistance was reduced by driving new borehole raises between levels (McKenzie, 1991).

Many of the drifts and other underground workings at the mine were in poor shape and incapable of handling the increased production traffic. Drifts were slashed to increase their dimensions (9 feet x 9 feet) so that they could handle larger tramming equipment. The drifts were screened and rock-bolted for extra safety, new mine rail was laid, and new service lines (electrical, compressed air, water) were installed (McKenzie, 1991).

Mining Operations During Nerco

By 1986, conventional cut-and-fill mining had been an increasing factor in mining operations at Con Mine. This was unacceptable because of the higher costs of conventional stopes compared to mechanized. Unfortunately, the irregular nature of the Con Mine orebodies, combined with slayed veins and nugget effects, did not allow for a standard bulk mining procedure, and Nerco management realized that selective mining based on many factors was required. A process of long-hole stoping using remote controlled mechanized machinery was commenced within orebodies regularly shaped. This also proved effective in areas of stope pillar recoveries where ground conditions were dangerous, and allowed ore to be mined that was previously inaccessible. Shrinkage mining was also implemented as a cost-saving measure, which allowed for greater control over mining widths. Nerco's budget for the year 1991 allowed for the following production split: conventional cut-and-fill, 39%; mechanized cut-and-fill, 9%; mechanized long-hole, 24%; shrinkage, 14%; development ore, 14% (McKenzie, 1991).

New Surface Facilities

The maintenance shop at the Robertson shaft was modified to implement a centralized maintenance program. The mine dry at the Robertson shaft complex was also expanded to accommodate the planned increase in mining crews (including addition of a 10 person woman's dry), and space within the geology and engineering offices was increased through the construction of a trailer-complex addition to the miner's dry. Management, safety, and the mine rescue station was moved into this new office complex. A site wide electrical upgrade was also initiated (McKenzie, 1991).

New Underground Fleet

Nerco replaced much of Cominco's antiquated mining machinery. After an expenditure of \$2 million, the mine was fitted with nine new scooptrams, ten 4 and 6 ton locis, mucking machines, and over 70 ore cars (McKenzie, 1991).

New Water Treatment Plant

Prior to 1983, the production process at Con Mine did not produce an overly toxic effluent. With the opening of the Arsenic Plant in 1983, it became necessary to investigate treatment of these wastes. Cominco failed to find a workable solution, but in 1987 Nerco proceeded with the construction of an advanced water treatment plant to deal with the effluent requirements imposed by the regulation agencies. The plant was completed in August 1987 using a copper sulfate and hydrogen peroxide, lime, and ferric sulfate process to help break down cyanide, copper, and arsenic contents of the mill effluent. In 1988, Nerco also erected a building to house an expanded environmental department and upgraded the staff's equipment and funding (McKenzie, 1991).

Years:	Manager:	Years:	Manager:
<i>1937-1946</i>	Henry C. Giegerich	<i>1986-1988</i>	Darrell E. Spilde
<i>1946-1953</i>	Claire E. White	<i>1988-1990</i>	Clint R. Naumen
<i>1954-1955</i>	J.E. McGlynn	<i>1991-1993</i>	Jason B. McKenzie
<i>1955-1966</i>	E. Jay Colthorp	<i>1993</i>	Nick Burt
<i>1966-1968</i>	John J. McKay	<i>1993-1994</i>	Robert Spengler
<i>1969-1973</i>	Alec J. Richardson	<i>1995-1996</i>	Ian Berzins
<i>1974-1980</i>	Dave McPhail	<i>1996-1997</i>	Bob Steinke
<i>1980-1983</i>	Jim R. Greenhalgh	<i>1997-1999</i>	Gordon McDermit
<i>1985-1986</i>	Anthony J. Keen	<i>2000-2004</i>	John Stard

Table 4. *Con Mine managers. (source: mine records)*

Mill Expansion 1990

During the summer of 1989 it was obvious that mining operations were outperforming the capacity of the milling plant. Several reforms were made to the milling circuit to allow for an increased daily production rate. The obsolete 5 foot x 9 foot ball mill was put back into service and installed in parallel to the existing ball mill. Crushing size was reduced from ½ inches to 7/16 inches to increase grinding rates (McKenzie, 1991).

Further work modified the plant to treat both free-milling ores from the Robertson shaft and refractory ores from the C-1/B-3 shaft complex. Additions included a new 10 foot x 15 foot ball mill, a new 40 foot triple-tray thickener, a new drum filter, and a new flotation circuit. The new grinding circuit was placed into operation on October 22nd 1990, and the new cyanidation and flotation circuit was placed into production in December 1990. The rate of the milling facility was now 1,200 tons per day using a mix of free-milling and refractory ore. Some structural supports in the mill building were replaced and sections of the roof were rebuilt (McKenzie, 1991).

Autoclave Plant

Flotation concentrates were stockpiled during 1991-1992 pending the installation of technology capable of treating the material to recover gold. Studies were completed and it was decided the best way to treat these concentrates was with a pressure oxidizing leach system, also known as an Autoclave. It used high temperature and pressure to liberate gold from refractory ores and neutralize the hazardous arsenic trioxides by converting them into stable ferric arsenate. The Autoclave was an environmentally friendly replacement for the old roasting technology and would also contribute to future reclamation activities by reclaiming the old arsenic wastes from roasting between 1942 and 1970. Construction began in April 1991. The plant went into operation on August 26th 1992 although its first few months of operation were quite sporadic. It could operate at a rate of 90 tons of concentrate per day. Total cost of the Autoclave project was \$19 million. In 1991, Nercos reconditioned the old arsenic plant into a blending plant to mix the old arsenic sludge and calcine wastes into a 50/50 ratio before being sent for processing in the Autoclave. It was planned to treat 9,000 tons of arsenic sludge and 9,000 tons of calcine waste per year, neutralizing the hazards of this old mine waste and removing a major environmental liability (Wright, 1992).

Mining and hoisting operations at C-1 shaft were suspended briefly in 1991 but resumed during 1992 and 1993. Lack of refractory ore was a primary reason for these shutdown periods. Nercos experienced several problems with the Autoclave plant during the first year of operation, principally due to high chloride levels in the refractory feed which led to extreme corrosion of oxygen and vent pipe lines in the Autoclave plant. This led to a significant rise in maintenance costs and made the Autoclave practically uneconomic (The Northern Miner, July 25th 1994).



NWT Mining Heritage Society

Figure 14. *Con Mine 50th Anniversary celebration, 1988.*

Exploration

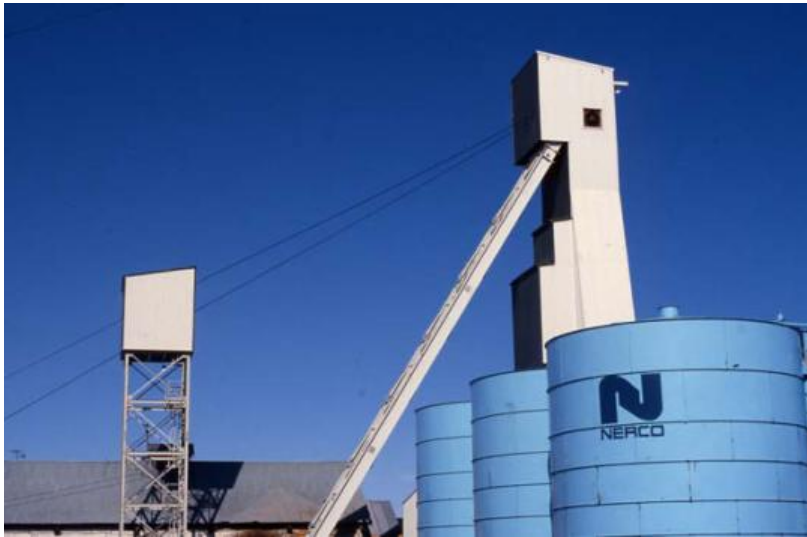
By 1985, exploration at Con Mine had largely ceased. Work resumed following Nercos' advent, and by 1990 the ore reserve was nearly double that in 1985. An aggressive exploration program in the C-1/B-3 area was conducted to outline reserves that had previously been below the cut-off grade of 0.50 ounces per ton gold. Nercos developed a new mining plan to access these reserves at a profit. On October 1st 1986, ore reserves were stated as 1,429,700 tons grading 0.41 ounces per ton containing 583,500 ounces of gold. On January 1st 1991, a preliminary ore reserve statement listed 3,527,240 tons of ore grading 0.29 ounces per ton containing 1,039,354 ounces of gold. (McKenzie, 1991) The company proposed developing ore within the Campbell Shear zone beneath the city core of Yellowknife and announced in 1989 a plan to sink a new shaft at Niven Lake. This new shaft would allow the extraction of a large block of previously undeveloped ore at cheaper rates. This plan was later cancelled on account of economics and

negative public opinion in Yellowknife surrounding the expansion of mine facilities elsewhere in the city (The Yellowknifer, Dec. 1st 1989).

It has been estimated that between 1986 and 1993, Nerco invested U.S. \$123 million in operational improvements at the Con Mine, including the rehabilitated C-1/B-3 shaft complex, expanded mill, and Autoclave construction (The Financial Post, July 3rd 1993). In early 1989, Nerco announced a layoff of more than 80 at Con Mine as part of a cost-cutting program probably in response to an economic recession at the time and the high costs of the expansion programs (The Yellowknifer, Jan. 25th 1989). In 1993, operational costs were very high at U.S. \$355 per ounce.

End of Nerco Tenure

The assets of Nerco Minerals Company were acquired by Kennecott in the summer of 1993, and the Con Mine was put up for sale. Although several companies put in bids for the mine (including Royal Oak Mines Incorporated, owners of the adjacent Giant Mine), a deal was made in June 1993 for the sale to Red Lion Management Limited, who in turn sold the mine to Miramar Mining Corporation Limited in October 1993.



NWT Mining Heritage Society

Figure 15. *New C-1 headframe, 1989.*

Miramar Con Mine Limited [Miramar Mining Corporation Limited] (1993-2004)

On October 14th 1993, Miramar Mining Corporation Limited acquired the Con Mine from Red Lion at a cost of U.S. \$25 million. The subsidiary company Miramar Con Mine Limited was formed. Miramar addressed many of the operating difficulties that had conspired to make the mine a money-losing venture, and set about a cost-cutting agenda to improve operations. The first initiative that Miramar undertook was shutting down the Autoclave plant until the high chloride problems could be fixed. The autoclave was shutdown in late 1993 pending a review of operations and a complete rehabilitation of the current milling flowsheet. For the first year (1993-1994) Miramar focused on implementing operational changes rather than invest in capital projects. Among the changes included a switch of mining method to long-hole and shrinkage stoping instead of expensive cut-and-fill methods, which made stopes more productive. Some cut-and-fill stopes continued to be used. Increased mechanization and bulk mining was key to this strategy. At January 1st 1994, total ore reserves (proven and probable) at Con Mine were calculated as 3.7 million tons grading 0.31 ounces per ton gold, about a third of which was refractory ore (The Northern Miner, July 25th 1994).

Milling Plant Expansion

During 1994-1995 the mill was expanded to 1,400 tons per day capacity to provide separate circuits for the treatment of free-milling and refractory ores, whereas Nerco had blended the two products. The capital cost of upgrading the plant was estimated to be \$3 million and included the conversion of the gold recovery circuit to a carbon-in-leach system and modifications to the grinding and sulphide circuits. The changes were projected to increase gold recoveries by 2% and reduce milling costs by 10 to 20% (Miramar Mining Corp. Ltd. Annual Report, 1994; The Northern Miner, July 25th 1994). In reality, production records for 1995 and 1996 suggest that recovery dropped significantly compared to the early 1990s.



NWT Mining Heritage Society

Figure 16. *Bluefish Hydro, 1990s.*

The Bluefish Hydro Plant was also upgraded from 3.5 megawatts to 7.5 megawatts at a capital cost of \$11 million to help lower costs associated with the mining operation (Miramar Mining Corp. Ltd. Annual Report, 1994). The C-1 shaft was re-commissioned as a production shaft during April 1995 to handle between 300 to 400 tons per day of refractory ore (The Northern Miner, June 12th 1995). Autoclave operations resumed in February 1995. Miramar solved the problem of pipe corrosion in the Autoclave by using fresh water instead of mine water in the plant. An important part of the Autoclave function was the elimination of stockpiles of old arsenic and calcine sludge, remnants of the roaster operation before 1970. Processed through the Autoclave, this hazardous material would be broken down into harmless ferric arsenate. In 1994, calcine sludge reserves totaled 48,000 tons grading 0.45 ounces per ton gold while arsenic sludge amounted to 45,000 tons grading 0.50 ounces per ton gold (The Northern Miner, July 25th 1994). This material would not be processed until 2000.

Milling Circuit 1990s

Gold was recovered through two types of ores: refractory and free-milling. Each ore was processed through separate circuits after 1995. Refractory ore was sent through two-stage crushing circuit reducing ore to -5/8 inches then ground in an Allis-Chalmers 8 foot x 12 foot ball mill. Material then reported to sulphide flotation, and was roasted through pressure-oxidation in the autoclave. Gold bearing pulp from the autoclave was then sent to the cyanidation circuit where it was combined with free-milling ore. Free-milling ore was also crushed in two stage, being processed in a 10 foot x 15 foot ball mill where cyanide was added. This ore was not subjected to flotation nor autoclave roasting, and was sent directly into the cyanidation tanks where it was leached with carbon and cyanide. All products were filtered and precipitated in conventional fashions, followed by refining to produce gold bars (mine records).

During 1996 the company made a new discovery at depths of 3,500 feet to 5,300 feet within the Robertson shaft area (the 104-zone). No progress was made on developing these resources at the time on account of falling gold prices. During 1996 the mine employed about 350 workers (Miramar Mining Corp. Ltd. Annual Report, 1996).

Cost Cutting and The Strike

During 1997, low gold prices and high operating costs had an adverse effect on operations. Autoclave operations ceased again in 1997 due to lower prices of gold and the cessation of refractory production. These problems necessitated a revised mine plan that called for a shift in production to free-milling areas of the mine that were accessible and already developed with an attempt to lower production costs. The new plan resulted in a lay off of 120 employees and the reduction of production to 600 tons per day early in 1998. As a result of these measures the mine union went on strike starting May 14th 1998 and all operations ceased at Con Mine (Miramar Mining Corp. Ltd. Annual Report, 1998).

During the strike, company geologists reevaluated the potential for refractory resources. They uncovered a block of ore in the Robertson shaft area that they believed could be cheaply exploited. This resource was estimated to contain 500,000 tons of ore grading 0.31 ounces per ton gold (Miramar Mining Corp. Ltd. Annual Report, 1998).

Operations Resume

The strike was settled in April 1999 and operations resumed under the new mine plan. Commercial production resumed July 1999. Development of the refractory reserves commenced in the summer of 1999 and production of these ores commenced February 2000 using the re-commissioned Autoclave plant. Also during this time period, Miramar Mining Corporation bought the neighboring Giant Mine and began to truck ores to the Con plant. Giant Mines ores were refractory, therefore Giant Mine production was used to bolster the rate of refractory production for the company and maintain the operational requirements of the Autoclave. Running at 300 tons per day, the Autoclave increased production at Con Mine to 900 tons per day during 2000 (Miramar Mining Corp. Ltd. Annual Reports, 1999-2000). Although it was intended to utilize the C-1 shaft for production of refractory ores, it was instead decided to use the Robertson shaft as the primary production shaft. During 2000 modifications were made to the underground ore handling system to accommodate this demand on the Robertson shaft. The C-1 shaft was maintained as secondary access to the mine workings (Miramar Mining Corp. Ltd. Annual Report, 2000).

Sludge Reclamation

Gold was also being produced from old arsenic sludges stored in the calcine ponds at Con Mine. These were processed in the Autoclave on a sporadic schedule, but by 2003 this material had afforded 10,840 additional ounces and also neutralized the hazardous arsenical content of the sludges, an important factor in the company's reclamation activities at Con Mine (see **Table 5**) (Miramar Mining Corp. Ltd. Annual Report, 2003). During 2000 grades of free-milling ores at Con Mine steadily improved. Cost-saving programs also improved the situation, including a 13% reduction in development. However, the years 2001-2002 were difficult for Con Mine as the company recognized that the old mine was nearing the end of its long life. Grades were lower than anticipated in all areas of the mine. Two minor setbacks hurt the mine during 2002: the collapse of the roof on the oxygen plant in March and mechanical difficulties with the Robertson shaft hoist. The roof collapse temporarily suspended refractory ore operations as the oxygen plant was used in Autoclave operations. Free-milling ores were supplemented as the sole source of mill-feed during the four-month period in which the autoclave was down (Miramar Mining Corp. Ltd. Annual Reports, 2000-2002).

Year:	Sludge Reclaimed:	Grade:	Gold:	Recovery:
2000	3,260 tons	0.46 oz/ton	1,210 oz	81.1%
2001	9,187 tons	0.47 oz/ton	3,806 oz	88.1%
2002	5,307 tons	0.55 oz/ton	2,524 oz	86.8%
2003	11,904 tons	0.35 oz/ton	3,300 oz	79.1%
2004	18,209 tons	?	2,707 oz	?
2005	7,238 tons	-	-	-

Table 5. *Reclamation of Arsenic and Calcine Tailings at Con Mine 2000-2005. Gold recovery ceased December 2004 and all treatment operations ceased in April 2005. (source: Miramar Mining Corp. Ltd. Annual Reports)*

Late in 2002 the company implemented a new mine plan that called for accelerated mining of free-milling ores and shutting down of the Robertson shaft complex. It was planned to temporarily cease refractory production and divert all manpower to the development and mining of free-milling ore zones. However, mechanical problems with the Robertson shaft hoist impacted this plan (Miramar Mining Corp. Ltd. Annual Report, 2002). There was no underground diamond drilling in 2002. 39 free-milling stopes were in production and nine refractory stopes were in production until refractory production ceased on November 1st 2002 (Miramar Mining Corp. Ltd., 2003). Production shortfalls occurred throughout 2003. Operation costs inflated to nearly CDN \$450/oz up from \$270/oz in September 2000 proving that gold mining at Con Mine was becoming very expensive. Financially, Miramar also suffered from the effects of a stronger Canadian dollar. Gold reserves were stretching thin and although there were many zones in the mine that indicated possible grades, it was not feasible to attempt to develop them in light of an uncertain financial climate (Miramar Mining Corp. Ltd. Annual Report, 2003).

Economic Reserves:	Tons and Grade:	Gold:
<i>Proven:</i>	188,178 tons @ 0.33 oz/ton	61,664 oz
<i>Probable:</i>	374,743 tons @ 0.34 oz/ton	126,497 oz
<u>Sub Total:</u>	<u>562,921 tons @ 0.33 oz/ton</u>	<u>188,161 oz</u>
Sub-Economic Reserves:	Tons and Grade:	Gold:
<i>Measured and Indicated:</i>	1,413,591 tons @ 0.33 oz/ton	462,695 oz
<u>Sub Total:</u>	<u>1,413,591 tons @ 0.33 oz/ton</u>	<u>462,695 oz</u>
<u>Grand Total:</u>	<u>1,976,512 tons @ 0.33 oz/ton</u>	<u>650,856 oz</u>

Table 6. Ore reserves January 1st 2003. Free-milling ore represented 50% of the tonnage. Of the proven and probable reserves, 58% of the tonnage was within developed stopes. (source: Miramar Mining Corp. Ltd., 2003)

In August 2003 it was announced that mining operations were phasing down to a November closure. On October 24th 2003, a first batch of layoffs resulted in a 67 man reduction in the workforce. Ninety-four men were laid off on November 28th closure, leaving about 40 to 50 men to stay and continue with surface and milling operations. (Miramar Mining Corp. Ltd. Annual Report, 2003)

Final Closure

Mining operations ceased on November 28th 2003, but milling of refractory stockpiles continued into early 2004. Processing of arsenic sludges in the autoclave continued. Gold recovery through this process ceased in December 2004, and all Autoclave activities temporarily stopped in April 2005. The Autoclave is scheduled to resume operations in 2006 once the company has identified all locations of arsenic wastes and is able to stockpile them in one location. Several buildings at the mine site and the old Con camp were demolished in the summer of 2005, and surface cleanup is to continue. (Miramar Mining Corp. Ltd. Annual Reports, 2004-2006)

Exploration Since Mine Closure

No work has been done since mine closure in 2003. Based on the ore reserve figures of January 1st 2003 and the amount of ore mined and milled during the calendar year 2003, there is still significant resources remaining at Con Mine (proven and probable ore of 563,000 tons minus 2003 production of 194,000 tons equals 369,000 tons of known ore containing over 130,000 ounces of gold, plus over 1 million tons of measured and indicated ore). Whether or not this resource will ever be economical to recover is unknown. (Miramar Mining Corp. Ltd., 2003) Record high gold prices in 2008 and a new owner, Newmont Mining Corporation, has led to some speculation that the mine could reopen to recover the last known reserves, meanwhile reclamation work proceeds as the mine floods and surface facilities are removed.

Year:	Ore Milled:	Grade:	Gold:	Recovery:
1938	13,834 tons	0.53 opt	6,794 oz	91.9%
1939	43,464 tons	0.88 opt	35,172 oz	91.9%
1940	58,606 tons	0.64 opt	33,796 oz	90.3%
1941	59,834 tons	0.76 opt	40,904 oz	88.3%
1942	72,205 tons	0.63 opt	42,111 oz	92.9%
1943	38,321 tons	0.68 opt	23,024 oz	88.6%
1946	27,057 tons	0.43 opt	9,469 oz	81.7%
1947	94,515 tons	0.54 opt	42,285 oz	83.6%
1948	100,697 tons	0.62 opt	55,253 oz	91.4%
1949	111,399 tons	0.56 opt	59,618 oz	91.2%
1950	107,580 tons	0.57 opt	57,290 oz	90.9%
1951	121,938 tons	0.57 opt	62,642 oz	88.2%
1952	128,824 tons	0.57 opt	67,144 oz	90.4%
1953	148,291 tons	0.55 opt	73,784 oz	90.2%
1954	166,337 tons	0.55 opt	82,008 oz	89.7%
1955	170,129 tons	0.53 opt	80,501 oz	89.0%
1956	183,894 tons	0.54 opt	89,165 oz	89.3%
1957	183,914 tons	0.52 opt	84,726 oz	88.8%
1958	188,497 tons	0.59 opt	104,183 oz	92.3%
1959	191,299 tons	0.53 opt	92,918 oz	91.6%
1960	190,626 tons	0.59 opt	104,640 oz	92.9%
1961	192,702 tons	0.60 opt	106,204 oz	91.3%
1962	196,339 tons	0.57 opt	103,611 oz	92.3%
1963	191,579 tons	0.62 opt	111,959 oz	94.4%
1964	132,282 tons	0.64 opt	77,904 oz	93.9%
1965	169,198 tons	0.69 opt	110,640 oz	95.2%
1966	156,040 tons	0.71 opt	107,246 oz	95.6%
1967	157,103 tons	0.71 opt	104,890 oz	95.0%
1968	152,122 tons	0.66 opt	91,763 oz	95.3%
1969	145,962 tons	0.69 opt	98,882 oz	96.7%

Year:	Ore Milled:	Grade:	Gold:	Recovery:
1970	147,285 tons	0.62 opt	89,521 oz	95.7%
1971	158,480 tons	0.61 opt	90,544 oz	94.2%
1972	164,776 tons	0.67 opt	106,060 oz	95.8%
1973	168,696 tons	0.56 opt	90,730 oz	95.5%
1974	145,205 tons	0.59 opt	83,095 oz	95.9%
1975	148,482 tons	0.55 opt	77,630 oz	95.7%
1976	151,058 tons	0.62 opt	89,987 oz	95.9%
1977	157,295 tons	0.62 opt	92,897 oz	95.6%
1978	219,981 tons	0.55 opt	114,492 oz	95.3%
1979	216,570 tons	0.46 opt	95,012 oz	95.5%
1980	211,978 tons	0.48 opt	96,946 oz	95.3%
1981	193,982 tons	0.43 opt	74,772 oz	94.7%
1982	234,178 tons	0.36 opt	79,450 oz	94.6%
1983	209,239 tons	0.36 opt	70,229 oz	92.3%
1984	243,655 tons	0.39 opt	88,552 oz	91.4%
1985	223,419 tons	0.39 opt	80,359 oz	93.1%
1986	239,999 tons	0.46 opt	88,993 oz	94.0%
1987	212,011 tons	0.41 opt	83,149 oz	96.4%
1988	213,895 tons	0.38 opt	77,927 oz	96.3%
1989	268,148 tons	0.37 opt	94,872 oz	95.7%
1990	322,416 tons	0.39 opt	117,116 oz	94.5%
1991	366,292 tons	0.36 opt	123,092 oz	93.6%
1992	402,636 tons	0.32 opt	120,686 oz	93.2%
1993	403,479 tons	0.33 opt	119,318 oz	91.1%
1994	351,232 tons	0.38 opt	125,519 oz	93.4%
1995	447,682 tons	0.31 opt	122,010 oz	87.7%
1996	414,113 tons	0.30 opt	111,021 oz	87.3%
1997	384,398 tons	0.29 opt	94,410 oz	85.7%
1998	86,026 tons	0.30 opt	23,447 oz	92.1%
1999	119,539 tons	0.35 opt	38,678 oz	93.4%

Table 7. Con Mine production 1938-1943, 1946-1999. (including Rycon and Vol production, but does not include milling of custom ores from Ptarmigan Mine (1983), or Cassidy Point Mine (1986)) opt = ounces per ton

(Gold production statistics for 1938 to 1972 are gold sales from the Con-Rycon Mine and may conflict with other data, but those numbers were the only reliable source of gold production that the author could find for both Con and Rycon (mine records). Statistics after 1972 is gold produced based on other mine records. Tons milled and grade and recovery is based on monthly or yearly mine operation records during both Cominco's and Nerco's term. Production during Miramar's term of operation is based on company annual reports.)

Year:	Ore Milled:	Grade:	Gold:	Recovery:
2000	300,516 tons	0.38 oz/ton	101,670 oz	89.5%
2001	298,455 tons	0.36 oz/ton	100,992 oz	88.7%
2002	279,638 tons	0.35 oz/ton	95,513 oz	89.4%
2003	193,754 tons	0.31 oz/ton	56,700 oz	86.7%
2004	2,489 tons *	0.18 oz/ton	448 oz	80.9%

Table 8. *Con Mine production 2000-2004, not including recovery of gold in arsenic or calcine wastes (see Table 5) (source: Miramar Mining Corp. Ltd. Annual Reports) (* Refractory stockpiles)*

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geology from NORMIN.DB (www.nwtgeoscience.ca) Showing 085JSE0056



CONSOLATION LAKE

Bulk Sampled (Abandoned)

Tungsten

62° 30' 15" 112° 56' 30"

Years of Primary Development: 1942

Mine Development: primary trench

Years of Bulk Sampling: 1942

Bulk Sample: 11 tons milled = 1,917 lbs tungsten concentrate (35% WO₃)

Introduction

This small operation was located on the southeastern shore of Consolation Lake, 72 kilometers east of Yellowknife, NWT. It milled a few tons of ore in a test mill in 1942. The site has not been visited by the author.

Brief History

The 'Storm' group of 6 claims was staked in 1940 by Jimmy Irwin and Hank Lange, and in 1941 was controlled by Storm Yellowknife Syndicate. The discovery was originally that of gold, but during the year a vein was found to contain appreciable tungsten. In 1942, a small mill was erected to recover a tungsten concentrate.

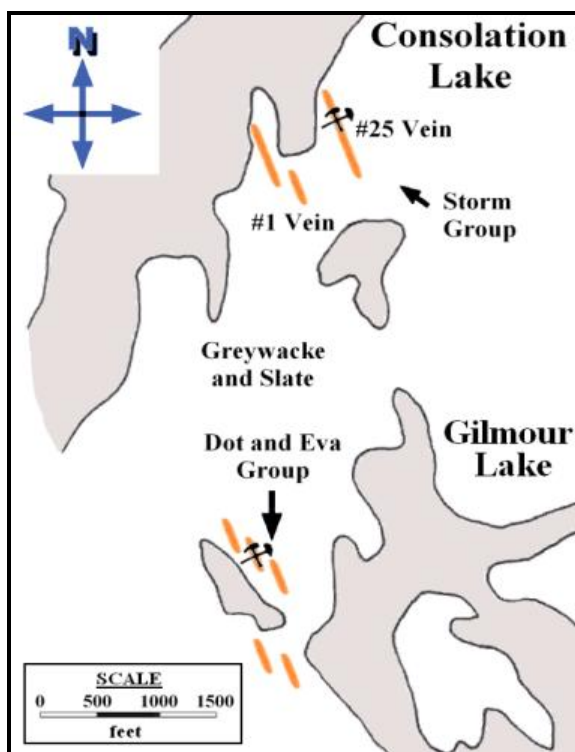


Figure 1. *Consolation/Gilmour Lake area.*

Geology and Ore Deposits

The area is underlain by greywacke and slate, impure quartzite, and argillite of the Archean Yellowknife Supergroup. Quartz veins in the area are typically 6 inches to 4 feet in width, containing visible gold, pyrite, and tungsten as scheelite. The veins strike northwesterly and dip vertical (Lord, 1951).

Tungsten Developers Limited (1942)

This small company was formed by Jimmy Mason and Bill McDonald to erect a milling plant at Consolation Lake to extract a tungsten product from the #25 vein, a 60 foot long structure with an ore shoot 15 feet long. The Yellowknife region was the focus of some attention as a possible producer of the war metal and one operation at Outpost Island was already extracting a concentrate on a steady basis. There was, therefore, considerable interest in tungsten prospects. Mason and McDonald leased the claims from the Anpara Syndicate, for whom the 'Storm' claims were originally staked. A small mill was flown to Consolation Lake and installed. Bill McDonald was in charge of the plant. The operation was run by four people, Jimmy Mason, Bill McDonald, W.L. Johnson, and Mrs. Bill McDonald, as camp cook (The Yellowknife Blade, Aug. 18th 1942; Oct. 4th 1942).

Operations

Material was extracted from a pit cut along the #25 vein located on 'Storm #5' claim on the east side of a small bay jutting south from the lake. This pit was sunk 8 to 12 feet in depth. Before being processed, the ore was burned to prevent sliming during the tabling process. The ore was then broken into three inch pieces and run through a 4½ inch x 5½ inch jaw crusher, then a 24 inch x 12 inch Straub ball mill. This machinery was powered by a 7 horsepower Wisconsin gas engine. Two sluice-boxes were built to receive the ball mill product, one of which was floored by 16 feet of rubber matting, and the other with 24 feet of corduroy cloth. These boxes were fitted side by side, and product from the mill was switched into either circuit after a few minutes when a visible concentrate was seen on the tables. This helped to reduce metal losses in the tailings (The Yellowknife Blade, Oct. 4th 1942).

A total of 11 tons were milled to produce 1,917 pounds of tungsten concentrates, grading 35% tungsten oxide (WO₃). (Lord, 1951) Operations were suspended in October 1942 due to the advent of cold weather, which resulted in

mechanical difficulties. The freeze-up of the lake also hampered transportation to and from the site. At the end of the 1942 season, it is told that six tons of roasted ore were stockpiled and ready for milling the following year (The Yellowknife Blade, Oct. 4th 1942). So far as known, no further mining operations were conducted at the 'Storm' claims.

Exploration Since Mine Closure

In 1945, the property was optioned to Carlmac Gold Mines Limited and some diamond drilling was undertaken on the #1 vein (Cavey and Helgason, 1985). In 1972, Jimmy Irwin re-staked the property as the new 'Storm' claims and in 1974 Delphi Resources Limited examined the claims. Samples from the veins were assayed for gold, and the best assay was 1.08 ounce per ton gold (Nickerson, 1975). Twenty-one trenches were excavated in 1980 by the owners (Irwin and Irwin, 1980). More samples were taken from the new trenches and a few percussion holes were drilled in 1981, with the best sample being 0.15% WO₃ (Turner, 1982). During 1985, geological mapping, rock and soil geochemistry, and a VLF-EM survey were carried out on the property. Six samples from quartz veins returned around 0.01 ounces per ton gold and less than 0.01% of WO₃ (Cavey and Helgason, 1985).

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CONTACT LAKE

Minor Producer (Abandoned)

Uranium, Silver

65° 59' 30" 117° 48' 00"

Years of Development: 1932-1939, 1947-1949, 1969, 1979-1980

Mine Development: 190 vertical shaft, 3 levels, adit, internal winze

Years of Production: 1935-1939, 1975-1977, 1979-1980

Mine Production: 16,995 tons milled = 678,000 oz Ag, 6,933 lbs U₃O₈

Years of Bulk Sampling: 1947

Bulk Sample: 21,200 lbs ore shipped = 29,000 oz Ag

Introduction

The Contact Lake Mine is located in the Echo Bay region of Great Bear Lake, 15 kilometers southeast of LaBine Point (Port Radium) on the north side of Contact Lake. It is 427 kilometers northwest of Yellowknife, NWT. It was mined for its silver and uranium content beginning in 1934. It was last operated in 1980. The old mine was visited by the author in July 2005 courtesy of Alberta Star Development Corporation, owners of the current mineral claims.

Brief History

The 'M', 'S', and 'E' claim groups were staked in July 1931 by Tom Creighton and others for the Northern Aerial Minerals Exploration Company. An Ontario mining interest headed by William Wright and Harry Oaks acquired the property and financed it into development through the creation of Bear Exploration and Radium Limited. Showings of high grade silver on surface and via a short adit showed the potential for a profitable production operation. A 25-ton per day mill was shipped to Contact Lake with the purpose of recovering a silver concentrate. This mill operated intermittently between late in 1936 to early 1938, and then for a steady year until a drop in the price of silver resulted in a June 1939 shutdown. Minor uranium was also recovered.



Ryan Silke

Figure 1. *Contact Lake Mine headframe, July 2005.*

The property was reopened in 1946 and development ensued with the intent of mining uranium ore bodies through three shaft levels. Although some ore was sent out as bulk shipments, no production was attained. Development and exploration during this period indicated a sizable orebody, but work ceased in 1949.

Silver again became the focus of work in 1969 when Ulster Petroleum Limited reactivated the underground workings. Previous work suggested that both the ore reserves and the old mill tailings could contain a large amount of silver values. The purpose of the 1969 program was to verify the underground deposit. The property was last opened in the late 1970s when Echo Bay Mines Limited mined and milled some underground ore and surface tailings from Contact Lake. New mineral claims were staked in 1996 by Lane Dewar and Trevor Teed. In April 2005, Alberta Star Development Corporation acquired the property to undertake a regional geophysical survey.

Geology and Ore Deposits

The property is underlain by Aphebian granodiorite intruding Echo Bay Group volcanics. A small granite body is in contact with the granodiorite southwest of the mine. Numerous breaks, including shear zones and tensional openings,

disrupt the granodiorite. Shear zones are locally filled with quartz-hematite and quartz-carbonate. The #1 zone is a tensional opening characterized mostly by quartz-carbonate with small rich ore shoots containing silver, pitchblende, and sulphide minerals. The #3 zone is a shear zone filled with quartz-hematite with small ore shoots carrying cobalt-nickel arsenides, silver, pitchblende, chalcopyrite, and pyrite. Another mineralized shear zone (#2 zone) is composed of altered granodiorite and chlorite with chalcopyrite, bornite, pyrite, and minor silver (Lord, 1951).

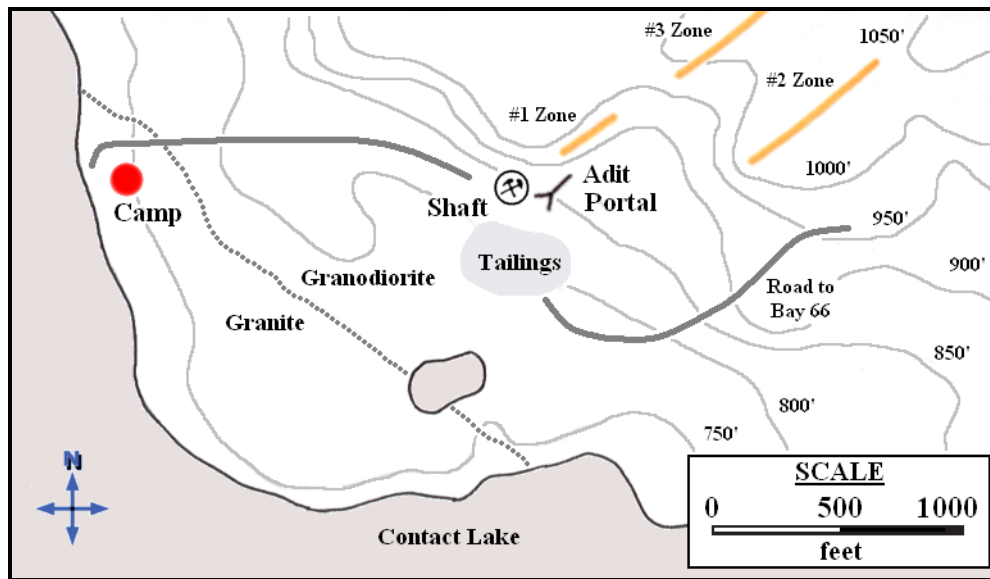


Figure 2. Contact Lake Mine area, showing location of camp, mine site, and vein deposits.

Bear Exploration and Radium Limited (1932-1939)

The company acquired the property in June 1932 and immediately dispatched Major Bernhard Day to oversee developments. By the freeze up period, crews had built a log cabin camp and had collared an adit entrance into the side of the targeted silver zone. Trenching over a length of 26 feet and to a depth of 10 feet revealed high-grade pitchblende and silver along the #1 zone. Tunneling via handsteel then commenced from the adit portal and continued for about 90 feet when winter conditions forced a temporary stoppage in December 1932. The adit level is 100 feet below the surface exposures (Day, 1933).

Work continued in 1933 and a portable Canadian Ingersoll-Rand type-20 gas-powered air compressor was flown to the site in March 1933 to aid in mining developments (The Toronto Star, Apr. 1st 1933). Twenty-four men were employed in August 1933 with W. Bert Airth in charge of work. A five tons per day pilot milling plant was reported to be on site, but there was no intention of using it, as the company now believed that it was too small. A larger plant of 25 tons per day was ordered. Three lenticular zones had been identified by the summer of 1933, with the #1 zone being the focus of all underground development thus far (The Northern Miner, Aug. 24th 1933). This development drifted along the strike of the main vein and explored it for a distance of 450 feet from the adit entrance, with 115 feet of crosscutting reported in December 1933 (Meikle, 1933). Numerous diamond drill holes also probed the vein at 80 feet depth below the adit level, uncovering good values. A vein length of 1,700 feet was proved, with another 1,000 feet as part of the west extension 125 feet below the adit. Underground work in this area was expected to give good values as the vein reached the granite contact. A winze would have to be sunk in order to reach this area (The Northern Miner, Aug. 24th 1933).

Pitchblende

The property was originally staked for its possible radium content, which was most concentrated within the #1 zone. In 1933, the pitchblende ore was known to occur sporadically and in a friable form – difficult to recover in a concentration plant. More research was needed before a pitchblende concentrate could be economically recovered, therefore practically all mining development in the 1930s was dedicated to the mine's silver content (The Northern Miner, Aug. 24th 1933).

Development during the first three months of 1934 focused on the sinking of the #1 winze from the adit level to the 2nd level, at a depth of 125 feet below the adit. Crosscutting and drifting along both directions of the vein then

commenced (The Toronto Star, Apr. 18th 1934). A milling decision was postponed in July 1934 to allow for increased ore reserves. During the winter of 1934-1935 a vertical raise was driven towards the surface from the 2nd level, which would later be developed into the #1 shaft. A headframe and hoist room was constructed during bitter cold temperatures to service this new shaft. The shaft was widened into a 2-compartment opening. Development on the west extension of #1 zone at 125-foot depth identified a 95-foot ore shoot with values of 260 ounces per ton silver across widths of 21 inches. Gold values of 0.30 ounces per ton were also identified in this area. The #1 shaft was advanced towards the 3rd level during the summer of 1935 and high-grade ore was identified in the new workings. The majority of equipment necessary for mill erection was on hand during July 1935 and construction began (The Northern Miner, Jan. 10th 1935; June 27th 1935; The Toronto Star, July 25th 1935).

Year:	Ore Milled:	Pounds of Concentrate:	Silver:	Uranium Oxides: (¹)	Notes
1935/1936	776 tons	19,200 lbs	24,000 oz	-	<i>For the period November to December 1935 and May to August 1936</i>
1936	948 tons	13,936 lbs	33,341 oz (²)	-	<i>For the period November to December 1936</i>
1937	1,299 tons	28,880 lbs	70,546 oz	-	<i>Mill only operated for 10 weeks of the year.</i>
1938	1,174 tons	23,704 lbs	34,700 oz	-	<i>For the period January 1st 1938 to June 30th 1938</i>
1938/1939	6,658 tons	198,265 lbs	195,333 oz	6,933 lbs	<i>For the period July 1st 1938 to June 30th 1939</i>
Total:	<u>10,855 tons</u>	<u>283,985 lbs</u>	<u>357,920 oz</u>	<u>6,933 lbs</u>	

Table 1. Contact Lake Mine production, 1935-1939. (source: Lord, 1941; The Northern Miner, Mar. 18th 1937)

(¹) The original milling plant was not equipped to recover uranium oxides and no record of its content was kept until the last year of operation

(²) Silver recovered from concentrates was 24,541 ounces and silver recovered from high-grade nuggets was 8,800 ounces.

Mining Equipment

Equipment reported in November 1935 consisted of a 25 horsepower steam engine, 60 horsepower vertical boiler operating a 532 cubic feet per minute air compressor, and a gas powered 20 horsepower Canadian Ingersoll-Rand air compressor for backup power. A 7x6 steam hoist was used in shaft operations (Meikle, 1935). The headframe, hoist room, shop, and adit portal were interconnected by corridors to utilize the waste heat produced from the boiler and steam hoist.

Production Starts

Tuning in of the mill was reported in November 1935 with full operations expected to commence at the end of the year. Operations appear to have ceased a short time later in December, due to mechanical difficulties brought forth by cold weather. Inadequate power supply was also a major issue, and a diesel engine was ordered to replace the steam engines (The Northern Miner, Aug. 27th 1936).

Milling resumed in May 1936 after a prolonged startup delay. The mill was again shutdown on August 19th 1936 in order to install diesel power, a new Wilfrey table, and an Allis-Chalmers vibrating screen. By November 2nd 1936, all alterations were complete and operations resumed. Difficulties were also experienced with the ball mill being unable to handle enough tonnage. It was believed that once higher-grade ore was introduced this problem would cease, as the higher-grade ore was softer and more amenable to treatment. Production during December 1935 and from May 1936 to August 1936 was reported as 776 tons of ore treated to produce 19,200 pounds of concentrate with a content of approximately 24,000 ounces of silver. Recovery averaged about 66%. Production from November to December 1936 was 948 tons milled to produce 13,936 pounds of concentrate with a content of 24,541 ounces of silver. Total silver production during that period was actually 33,341 ounces, which includes the silver content of 550 pounds of silver nuggets (The Northern Miner, Mar. 18th 1937).

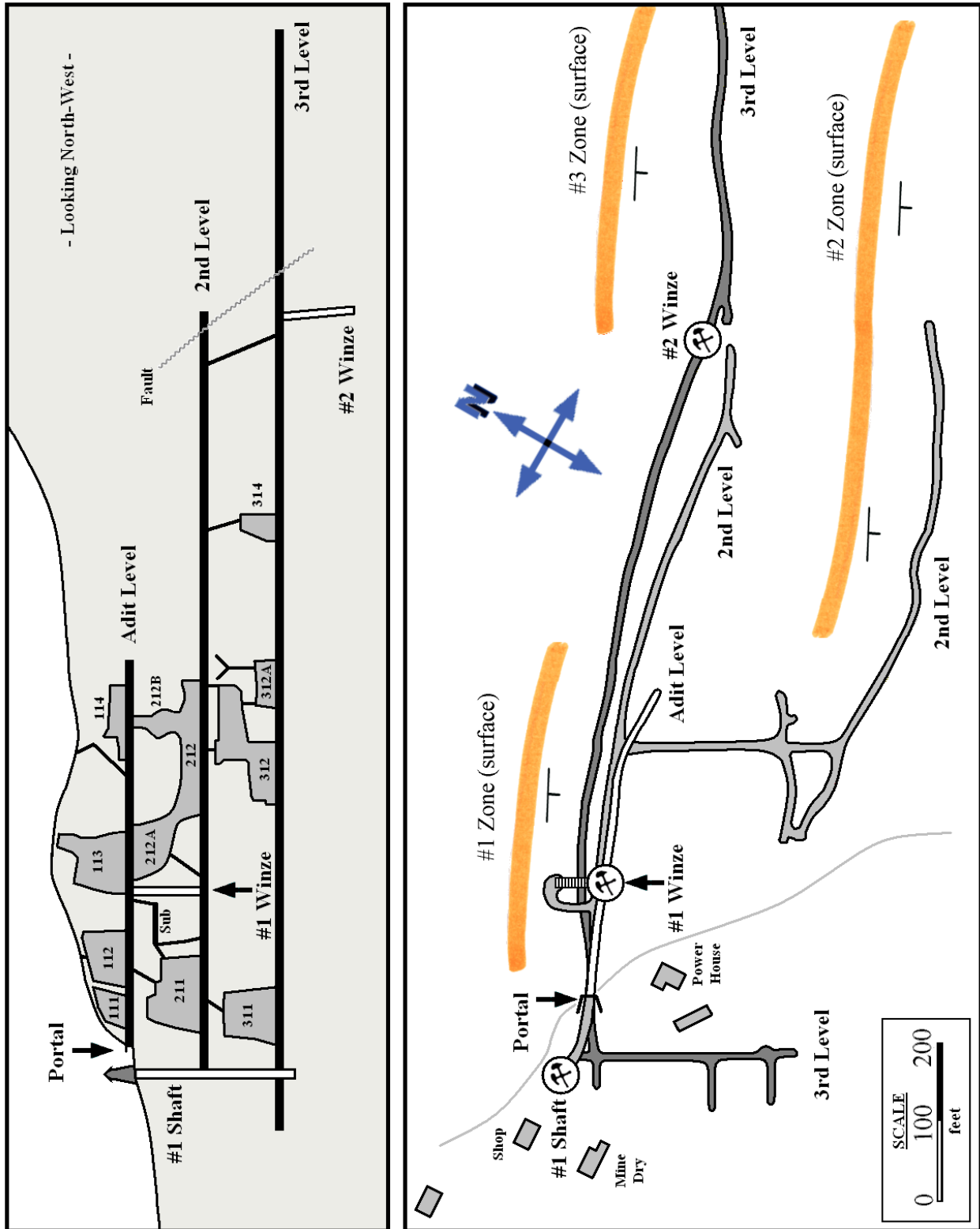


Figure 3. Contact Lake Mine underground plan, 1980.

Milling Operations

The Contact Lake Mine had a simple gravity concentration plant, of 25 ton per day capacity. Ore was crushed to an appropriate size in a 7 inch x 12 inch Telsmith jaw crusher, then brought to a 3 foot x 5 foot Allis-Chalmers ball mill by bucket elevator. A silver concentrate was recovered by two Wilfley tables and possibly through jiggling.

Concentrates were shipped by plane and barge to Trail, B.C. for smelting (Hershman, 1942). Tailings were deposited out from the mill and downhill. Power to the mill was supplied by a small diesel engine.

Underground development during 1936-1937 was largely confined to developing known reserves within the eastern section of the zones. Development during 1936 consisted of 443 feet of lateral work, 175 feet of raising, and the opening up of two new stopes on the 2nd and 3rd levels in June and October, respectively. Drifting to the west on the 3rd level was abandoned in 1936 when no ore was found (The Northern Miner, Aug. 12th 1937). The mining method used was shrinkage stoping. Equipment included an Ingersoll-Rand #R51 stoper drill, an Ingersoll-Rand #69 drifter drill, and a Sullivan T-10 drifter drill. Four ore cars were employed using hand-tramming methods (Hershman, 1942).

Milling again ceased during the summer of 1937 due to lack of power. The mill only operated during the first 10 weeks of the year, it was reported. All new diesel engines were ordered to fully replace the steam power, which before 1937 had been augmented by a single diesel unit. All machinery was found to be inadequate. Lack of quality wood fuel in the area was the biggest problem for steam generation. Two 140 horsepower Polar diesel engines were installed in August 1937 together with an additional 380 cubic feet per minute Ingersoll-Rand air compressor. Milling again resumed in December 1937 (The Northern Miner, June 17th 1937; Aug. 12th 1937; Dec. 30th 1937).

A new management crew was hired in July 1938 to reorganize the operation. Charles Hershman was hired in charge of the property, replacing Major Richard M. Treloar, manager since 1935. A focus of operations at this point became the recovery of a uranium oxide concentrate from pitchblende ores. However, the mill was not adequately equipped for such recovery. Stoping continued into 1939 for the purpose of obtaining information regarding the occurrences and values on the partially developed ore shoots on the three levels, and to prospect for possible pitchblende deposits. Exploration development completed from July 1938 to closure in June 1939 consisted of 563 feet of drifting on the 2nd level and 134 feet on the 3rd level. Raising totaled 65 feet advance. Total work to this point through 1,200 feet of work on the 2nd level and 640 feet on the 3rd level indicated a downward extension of the #1 zone through numerous lenses. Seven stopes were in production during this period, four on the adit level, one on the 2nd level, and two on the 3rd level. A drop in the price of silver from 42 cents per ounce to 35 cents per ounce resulted in the shutdown of the operation in June 1939. Ore reserves were calculated as 3,403 tons with content of 68,060 ounces of silver. Tailings were estimated to contain 292,833 ounces of silver and 16,933 pounds of uranium oxide (U₃O₈) (Bear Exploration & Radium Limited Annual Report, 1939).

# 1 Winze (Adit to 2nd):	110'
# 2 Winze (3rd to 4th):	105'
# 1 Shaft (Surface to 3rd):	210'
# 1 (Adit) Level:	486'
# 2 Level:	2,309'
# 3 Level:	2,377'
# 4 Level:	0'
Raising:	375'

Table 2. Summarized development, 1934-1949. (source: Mining Inspection Services; Lord, 1951)

International Uranium Mining Company Limited (1946-1949)

In 1942, this company took over the property from Bear Exploration and Radium Limited. A program of diamond drilling (15,326 feet), geological mapping, and prospecting was completed during 1944 and 1945. The primary aim of exploration was to test the uranium content of the deposit (Lord, 1951). An 800-pound ore sample was shipped for metallurgical testing in the summer of 1946, with assays of 12.40 pounds of U₃O₈ and 65 ounces per ton silver (The Toronto Star, June 18th 1946). In January 1946 de-watering of the mine began. A program of work involved developing the 2nd and 3rd levels to explore the #2 zone east of the #1 zone. Underground work began in October, but was halted due to a fire that destroyed the old mill and power plant in November. The program was continued in 1947 when 2nd level development reached the #2 zone (Lord, 1951).

Power Plant

The previous plant was destroyed by the fire in November 1946. Quickly, a temporary building was set up and new equipment consisting of a Cat D-13,000 diesel engine and a 365 cubic feet per minute Gardner-Denver compressor were installed. An air hoist (Ingersoll-Rand single drum) was in use for the shaft (Lord, 1951).

1947 Bulk Sample

Mining of the 212 stope on the 2nd level provided a bulk sample shipment of 10.6 tons of high-grade ore during 1947. Returns suggested an average content of 2,767 ounces per ton silver (approximately 29,000 ounces of silver). An aggressive program of drifting on the 3rd level easterly through the #1 zone showed considerable pitchblende. It was also planned to rebuild the former milling plant, mine the underground workings, and recover the old tailings material.

Development in 1948 focused primarily on the 3rd level where high-grade stopes were mined. East drifting indicated strong radioactive zones, heavy copper mineralization, but little silver concentration. It was decided early in the year to explore a possible downward extension of the #1 and #3 zones and a winze was started from the 3rd level to the 4th level. The #2 winze reached the 4th level in September 1948 at 300-foot depth. No lateral development was undertaken on the 4th level, with operations confined to 3rd level drifting and raising during 1949. (Mining Inspection Services) It was planned to ship 6 tons of high-grade silver ore and 1 ton of pitchblende ore from the property during 1949, and if results warranted, install a 50 tons per day concentrating plant in 1950 (The Northern Miner, Dec. 30th 1948). Work ceased in August 1949 (Mining Inspection Services).

Ulster Petroleums Limited (1969)

New claims were staked ('Sam' group) in 1961 by L. Peckham, and in 1965 they were optioned to Pinnacle Petroleums Limited. James Millar & Associates Limited wrote a brief property evaluation report and sampled the tailings pile in that year. Pinnacle's interest in the property was then assigned to Ulster Petroleums Limited through a corporate merger. Increased prices of silver and copper and new geological interpretations, plus the availability of a nearby mine for custom milling purposes, led to the decision to reopen the mine and initiate an exploration program in 1969 (Fingler, 2005).

The purpose of the 1969 exploration and development program was to verify the tonnage and value of ore previously blocked out by work done on the #1 vein. It was required to upgrade the camp and plant buildings to support a crew of 20 men from Triangle Mine Services Limited, contractor. Former owners had removed all useable equipment from the mine in 1956, therefore, it was also necessary to install a new mine plant. Work began at the end of April 1969 under the direction of engineer Norman Byrne of Precambrian Mining Services Limited.

Underground operations were supplied with air from two 600 cubic feet per minute Gardner-Denver Rotary-screw compressors. The shaft was re-timbered in places and equipped with a 30 inch diameter air hoist. A small fleet of underground equipment consisted of an Eimco mucking machine, two ore cars, and a Mancha "Little Trammer" battery locomotive. Power to the camp and other services was through a 15 KVA diesel generator. Fuel was stored at Bay 66 in a 50,000 gallon tank, where a small tanker wagon was employed hauling daily requirements of fuel to the mine. A total of 31,500 gallons of fuel were consumed during the program.

Following de-icing and pumping operations, it was found that the 2nd level was in bad shape. The 3rd level was in much better condition and it was decided therefore to concentrate work on this level. Prior to the commencement of exploration, the 3rd level was enlarged by slashing operations that totaled 8,900 cubic feet excavation. Then, an 18 foot raise was driven on the #1 vein. Samples were taken all along development headings to replace inadequate assay information from previous operators. Other work included 967 feet of diamond drilling and showed no new ore. Reserves were stated as 8,738 tons of ore in place underground, 660 tons of broken ore in stopes, 1,425 tons in the surface ore dump, and 5,000 tons tailings, with an average grade of 46 ounces per ton silver. At the end of the program, it was recommended that a deal be made with Echo Bay Mines Limited for the milling of known ores at their nearby plant. Operations stopped in August 1969 when negotiations with Echo Bay stalled, and the contractor removed all equipment (Byrne, 1969).

Echo Bay Mines Limited (1975-1980)

In 1973, Bill Knutsen reviewed the tailings resource and suggested a reserve of 2,264 tons grading 27 ounces per ton silver. It was recommended to reopen negotiations with Echo Bay with the view of selling the tailings pile to them to process (Knutsen, 1973). A deal was made between Ulster Petroleum Limited and Echo Bay Mines Limited early in 1975, in which Echo Bay was to perform exploration work to acquire full interest in the property. This work was to

be completed by September 1977. Production of tailings and unprocessed ores from the Contact Lake Mine at the Echo Bay mill would be subject to a 3% smelter-return charge agreement with Ulster. During 1975 Echo Bay Mines Limited removed 381 tons of stockpiled ore and 2,085 tons of tailings (Way, 1976). By 1977, 1,200 tons of tailings and surface stockpile had been milled at Echo Bay grading 30 to 50 ounces per ton silver (estimated 50,000 ounce silver content). This material was depleted in 1977 and no work was done in 1978 (Brophy, et al., 1983).

Year:	Ore Milled:	Silver:
1975-1977	1,200 tons	~50,000 oz
1979	4,900 tons	270,000 oz
1980	2,676 tons	129,204 oz

Table 3. Contact Lake Mine production, 1975-1980. Ore was processed at the Echo Bay/Eldorado Mine. Ore mined in the summer of 1980 was probably transported to Echo Bay for milling in the winter of 1980-1981.

In 1979, the underground workings were reopened again and 4,900 tons of ore were removed, producing about 270,000 ounces of silver. Mining was conducted during the summer months, and ore was hauled to Echo Bay Mine on the winter road (Brophy, et al., 1983). Ore reserves at November 16th 1979 were reported as 800 tons proven ore, 200 tons probable ore, and 1,300 tons possible ore for a total of 2,300 tons undeveloped ore reserves grading 18 ounces per ton silver, plus 6,050 tons of broken ore grading 42 ounces per ton silver in stopes. Total mineral inventory was 294,160 ounces of silver. Additional mining development was conducted in 1980. During June-August 1980, 2,676 tons of ore was mined in stopes on the 1st and 2nd levels (sources of ore: 111, 112, 113, and 211 stopes) and hoisted to surface for stockpiling. This ore was estimated to grade 48.3 ounces per ton silver and containing 129,204 ounces of silver. Last available records suggest an end of mining on August 27th 1980. During the month of August, 71 feet of raising was performed plus 70 feet of sub-drifting within the 211 stope area. **Table 4** gives ore reserves as of September 1st 1980. It is possible that the stockpiled and broken ore mentioned was removed for processing during the winter of 1980-1981, but there is no record of this.*

Location:	Proven Ore:	Possible Ore:	Total Ore:	Ounces Silver:
111 Stope:	50 tons @ 113 oz/ton	-	50 tons @ 113 oz/ton	-
113 Stope:	200 tons @ 37 oz/ton	-	200 tons @ 37 oz/ton	-
211 Stope:	850 tons @ 87 oz/ton	500 tons @ 11 oz/ton	1,350 tons @ 59 oz/ton	-
311 Stope:	150 tons @ 16 oz/ton	-	150 tons @ 16 oz/ton	-
Sub Total:	1,250 tons @ 72 oz/ton	500 tons @ 11 oz/ton	1,750 tons @ 54 oz/ton	94,900 oz
Tailings:	-	-	750 tons @ 16 oz/ton	12,384 oz
Surface Stockpile:	-	-	3,000 tons @ 48 oz/ton	129,204 oz
Broken Ore:	-	-	1,950 tons @ 52 oz/ton	101,400 oz
Grand Total:			7,450 tons @ 45 oz/ton	337,888 oz

Table 4. Contact Lake ore reserves, September 1st 1980.

* The above information concerning 1980 mining operations and ore reserves was found in records within the collection of the N.W.T. Geoscience Office, currently in bulk storage.

It is not believed that any further mining was undertaken, but during 1981, 210 feet of lateral work, 2,600 feet of surface diamond drilling, and 1,330 feet of underground diamond drilling was conducted (Brophy et al., 1984). Another source for ore reserves, in 1981, reports 700 tons undeveloped, along with 7,350 tons of underground broken ore, grading 43 ounces per ton silver and containing 350,000 ounces of silver (National Mineral Inventory). The accuracy of this report is unknown considering that a large amount of mining and stoping development would have been required to produce 7,350 tons of broken ore, especially in view of the fact that only 1,750 tons of undeveloped ore and 1,950 tons of developed, broken ore was available underground in September 1980.

Exploration Since Mine Closure

New claims were staked in the area in 1996 by Lane Dewar, Trevor Teed, and T. Burylo. The 'Contact' claim occupies the old Contact Lake Mine, while the 'SC' group of 4 claims is located to the east of the mine, and the 'Cobalt' claims northwest of the mine. The owners conducted prospecting and trenching in 1996-1997, and sampled numerous showings and the old tailings dump. Tyhee Development Corp. Limited optioned the claims in 2000-2003. While they did not perform any work around the original mine deposits, they did investigate what has become known as the K1, K2 and D2 deposits, located on the 'Cobalt' claims northwest of the mine (Fingler, 2005).

Alberta Star Development Corp. Limited acquired the 'Contact' and 'SC' claim groups in April 2005 and also staked new 'Cobalt' claims to protect the interesting mineral deposits (K1, K2, D2) on the lapsed claims of the same name. The main interest currently is the potential for a high-grade, bulk-tonnage, silver-uranium or polymetallic IOCG-type (iron-oxide, copper, gold, plus cobalt) mineral deposit in the area. J. Fingler was commissioned to assemble a report on the mining and geological history of the Contact Lake area (Fingler, 2005). In 2006, Alberta Star was awarded a five-year diamond-drilling permit for their extensive claim holdings in the Great Bear Lake area. Exploration plans for 2006 include a 15,000 meter diamond drilling campaign on the Contact Lake property, ground and airborne geophysics (induced polarization/resistivity), mapping, and line-cutting (Alberta Star Development Corp. Ltd., Press Release May 9th 2006).

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CONTACT LAKE PORTAL

Minor Exploration (Abandoned)

Silver, Copper

65° 58' 45" 117° 44' 45"

Years of Primary Development: 1934

Mine Development: adit tunnel

Introduction

The Contact Lake Portal is located in the Echo Bay region of Great Bear Lake, 18 kilometers southeast of LaBine Point (Port Radium) on the east end of Contact Lake. It is 424 kilometers northwest of Yellowknife, NWT. The mine is not related to the Contact Lake Mine, which is located 4 kilometers to the northwest. The site was viewed from the air briefly in July 2005.

History in Brief

The 'VIE' claims were part of a large group of claims staked in 1931 by John Michaels of the Michaels Syndicate. In the summer of 1932, native silver was located on 'Vie #2' claim and Contact Lake Mining Company Limited was formed to begin exploration. Driving of an adit tunnel was performed during 1933-1934.

Geology and Ore Deposits

The deposit was reported to contain promising amounts of sulphides showing copper, cobalt, bismuth, and silver. The silver is found in a narrow shoot of schist material in a quartz and calcite vein, one foot wide. The wall rock contains considerable chalcopyrite and pyrite, and small stringers of quartz paralleling the vein give a mineralized width of about 5 feet. Cobalt bloom and manganese stain also occurs (The Northern Miner newspaper, July 7th 1932).

Contact Lake Mining Company (1933-1934)

In the summer of 1933, Contact Lake Mining Company began driving an adit tunnel by handsteel on the 'Vie' claims, upon which a deposit of silver and copper sulphides was reported. A set of camp buildings was erected. The vein was intersected 59 feet in from the portal entrance, and drifting to the east began. Seven men were employed during 1933. Forty feet of drifting was accomplished in 1933, and total development has been reported as 115 feet of lateral work on one level (Cummings, 1934; N.W.T. Mining Inspection Services).

Exploration Since Mine Closure

The property was re-staked as the 'Bell' claims by the 1970s, but no assessment work has been reported.

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The Northern Miner newspaper articles, 1932-1933.



COPPER PASS

Bulk Sampled (Abandoned)

Nickel

62° 24' 30" 111° 51' 45"

Years of Primary Development: 1957, 1969-1970

Mine Development: three trenches

Years of High-Grading: 1969-1970

High-Grading: +450 tons (grades of 33-35% Ni reported)

Introduction

This nickel deposit, which was briefly mined in 1969-1970, is located 129 kilometers east of Yellowknife, NWT at Sachowia Lake.

History in Brief

The original claims were staked in 1940 by Cominco. In 1950 they were re-staked by Chuck McAvoy. In 1957 the claims were optioned to Venture Resources Limited who conducted some surface mining from a primary trench and may have shipped some ore. The claims again lapsed and were re-staked twice, finally being incorporated into the 'Gogo' group of claims in 1969 by Jim McAvoy. In association with Copper Pass Mines Limited, McAvoy high-graded surface showings during the summer of 1969. No other development is reported.

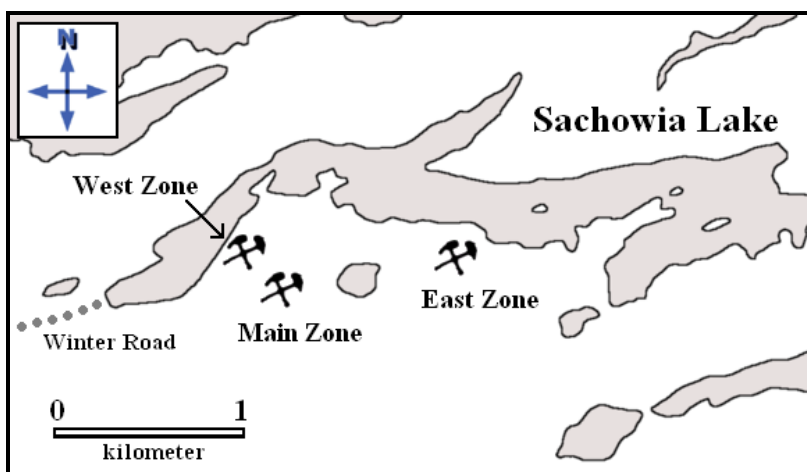


Figure 1. *Copper Pass Mine area showing location of ore zones.*

Geology and Ore Deposits¹

The area is underlain by Archean Yellowknife Supergroup rocks, including mafic flows and fragmentals together with metasediments. A large granitic intrusion is present to the north; various small granite pegmatite and aplite dykes within the layered rocks are probably related to it. Ferruginous sediments and lava flows of the Kahochella Formation (Great Slave Supergroup) overlie Archean basement approximately 1 kilometer south of the occurrences. Niccolite, skutterudite, gersdorffite, rammelsbergite, arsenopyrite, pyrite and chalcopyrite occur in structurally controlled quartz-carbonate veins and vein breccias. The latter are generally less than one meter in width but locally swell to 3 meters. Metallic mineral content varies considerably within the veins.

Venture Resources Limited (1957)

This company blasted a deep trench on the Main Zone showing that consisted of about 260 cubic yards of rock during or after 1957. Some hand-cobbed nickel ore was apparently shipped out. No record exists. Some seven tons of ore remained bagged on site (Thomas, 1969).

Copper Pass Mines Limited (1969-1970)

The company acquired the 'Gogo' claims in March 1969 from Jim McAvoy. Additional claims were staked soon thereafter. It was reported that 24 barrels of ore were mined during the spring of 1969 and shipped to Edmonton and

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

processed by two separate firms to establish grades. These samples returned grades of 25% to 43% nickel (33% average) with traces of cobalt, bismuth and silver (Copper Pass Mines Ltd. Report to Shareholders, 1969). Some of this ore likely included the seven tons of ore leftover from the previous operators. By October 1969, 200 tons of ore had been mined from the open pits (Kelly, 1969).

Plans were made to mine 500 tons of ore by open pit method and barge ore across Great Slave Lake to the railhead at Hay River. Preliminary agreements were made with smelting firms for the purchase of ores from this property. These plans did not materialize. Small-scale mining of vein exposures was conducted and up to November 1969 the following work had been completed:

Main Zone: 260 cubic yards of excavation were made into the existing trench on the Main zone. No ore was bagged from this zone. *East Zone:* 155 cubic yards of rock were removed from the trench. About 18 tons of niccolite ore were hand-cobbed and bagged for shipment. *West Zone:* 216 cubic yards of rock were removed from the trench. About 5 tons of nickel ores were hand-cobbed and bagged for shipment (Nickerson, 1969).

Ore bagged in the above references were likely mined after the initial shipments of ores for testing earlier in the year (the 24 barrels, for which tonnage is not available). The stockpiled ore was shipped to European smelters during the winter of 1969-1970. In January 1970, it was reported that a winter road had been completed from the mine to Yellowknife, and that two truckloads of ore had been shipped (destination and quantity unknown). Open pit extraction was continuing in the 'main pit' (Main zone?) (Kelly, 1970).

In 1970, the company was reported to have continued hand-cobbing high-grade ores. One open pit was completed in the spring, and another was started in September 1970. A total of 250 tons of niccolite ore was shipped over winter road in March 1970 and 200 tons by barge in October 1970. The refinery in France was still the customer of this ore. The grade of this ore was reported to be 35% nickel. It was also reported that an adit was collared at lake-level (Sachowia Lake) during 1970, but no decision had been made to proceed with underground development. The company planned to resume operations in the second open pit in 1971 (Mines and Mineral Activities, 1970). No work was reported in 1971, and the company went defunct soon after.

Exploration Since Mine Closure

No known work. New claims have been recorded in recent years, including the 'Coni 1' by Dave Webb in 1998 and 'MK14' by M. Knutsen in 2004.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 075LSW0001



CRESTAURUM

Advanced Exploration (Abandoned)

Gold

62° 35' 00" 114° 21' 15"

Years of Primary Development: 1946-1947

Mine Development: 420' vertical shaft, 3 levels (360' dev.)

Introduction

The Crestaurum Mine site is located 14 kilometers north of Yellowknife, NWT at the very end of the Vee Lake road, between Ryan and Daigle Lakes. It was a short-lived post-war gold prospect that never attained production. The remains of this operation have been visited numerous times over the years by the author, most recently in the summer of 2004.

History in Brief

This area had originally been staked prior to World War II but no ore deposits were located. The 'Midas' and 'Goldcrest' claims were then staked in 1944 by two individual prospectors, Harry Weaver and C. Duncan Campbell, respectively. The #1 shear zone was discovered soon after staking. The two claim groups were acquired by Jimmy Mason whose organization conducted extensive exploration including the sinking of a shaft in 1946-1947. For numerous reasons, the project was put on hold in 1947 and never reopened. Buildings were later destroyed by arsonists in 1957.

Geology and Ore Deposits¹

The Crestaurum deposit occurs in the Yellowknife Volcanic Belt. Regional metamorphism in the belt is lower to middle greenschist facies, and up to lower amphibolite on the west side of the belt. The Kam Group comprises four formations, predominantly mafic volcanics with minor amounts of dacitic volcanics and tuffaceous sediments, intruded by metagabbro sills and sheeted dykes. The Crestaurum Formation lies conformably on the basal Chan Formation. It is characterized by massive and pillowed basalts or andesites with a number of laterally continuous felsic tuffs and thin rhyodacitic flows. At the Crestaurum Mine, thinly bedded fine grained cherty tuffs are strongly deformed by folding. Higher in the formation the tuffs are coarser grained and become graded suggesting turbidite sedimentation.

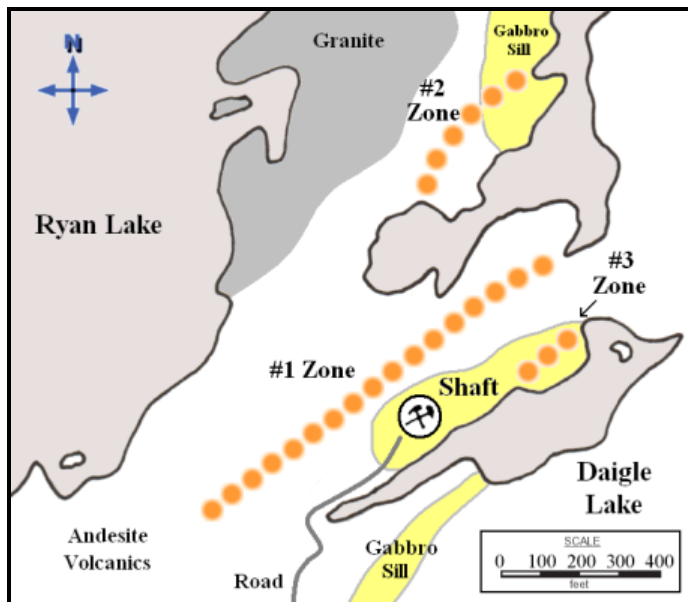


Figure 1. *Crestaurum Mine local geology.*

Metagabbro dykes form a prominent northerly trending swarm that comprises at least 10% of the Crestaurum Formation. Metagabbro sills at the base and at three levels within the formation are cut by many of the dykes and are probably synvolcanic.

A discontinuous zone of whitish-weathering silicified basalts extends irregularly southwesterly from Banting Lake. This may mark a zone of submarine hydrothermal alteration in the Crestaurum formation.

A complex zone of anastomosing shears, and quartz veins of varying orientations, cut the Crestaurum Formation in the area east of Ryan Lake and underlying Daigle Lake, where andesitic volcanics strike northeast and dip steeply. The main western shear zone includes three northeast-trending branches known as the #1, #2 and #3 shears.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

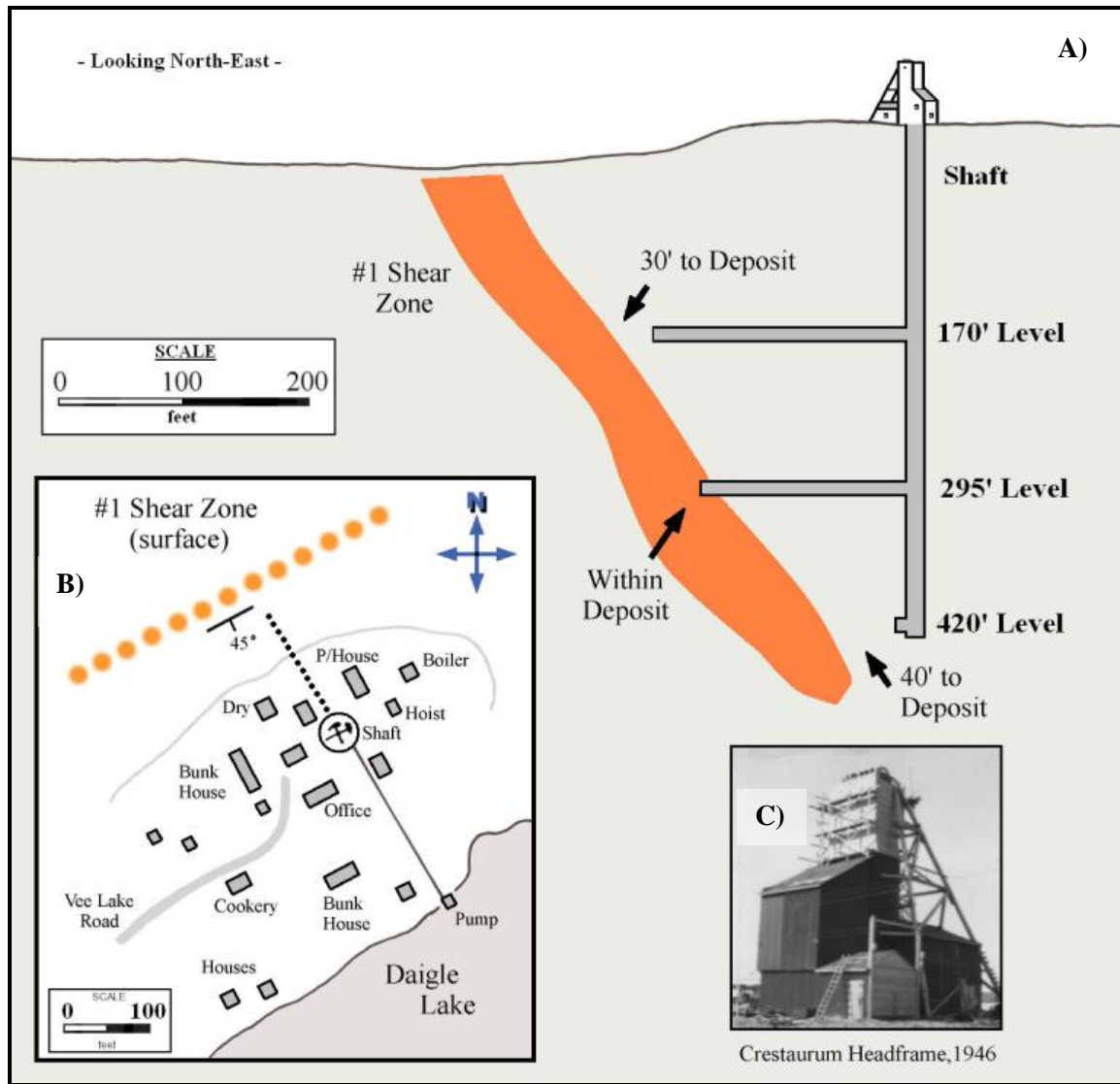


Figure 2. A) Crestaurum Mine underground section. B) Surface plan of mine structures. C) Headframe, 1946. (N.W.T. Mining Heritage Society collection)

The #1 shear hosts the main orebodies. It is over 1500 meters long and generally 120 meters deep, although it has been intersected as deep as 300 meters. Ore shoots within the shear have been traced to 120 meters depth, and there are two main shoots at approximately 30 meters depth: the south shoot is 275 meters long, of which a 140 meter by one meter section grades 17.48 grams per tonne gold; and the north shoot is 229 meter by 80 centimeter grading 20.23 grams per tonne gold. Both shoots probably have a gentle northeasterly pitch. The ore consists of subparallel quartz +/- carbonate veins and seams interfoliated with the schist. Metallic minerals are sparse and comprise primarily arsenopyrite, pyrite, stibnite, and jamesonite in the quartz, although much of the arsenopyrite occurs in the schist as fine needles.

The #2 shear (490 meters north-northeast of the shaft) cuts pillowed andesites and a quartz-feldspar porphyry dyke approximately six meters wide. It strikes 70° southeast, is 100 meters long and 75 meters deep, and ranges from 10 centimeters to two meters wide, averaging 60 centimeters. It contains quartz seams much as in #1 shear. It bends at one point and widens just southwest of the bend, where the quartz also achieves its greatest width of 30 centimeters, and contains several ounces of gold per ton along 18 meters strike length.

The #3 shear (75 meters east of the shaft) strikes 45° southeast for 150 meters, ranges from 30 centimeters to 1.8 meters wide and averages 80 centimeters, and grades 17.1 grams per tonne gold across approximately 30 centimeters for a 55 meters length. Its mineralogy is also similar to #1 shear zone.

Crestaurum Mines Limited (1946-1947)

During 1945, Transcontinental Resources Limited, an exploration outfit under Jimmy Mason, completed 30,000 feet of diamond drilling and numerous trenches on the #1 shear zone. Some of the first pits blasted in 1945 quickly unveiled a high-grade gold system extending through this shear zone. Drilling results, published in September 1945, reported three high grade ore shoots in the #1 zone, which had been drilled for a length of over 3,000 feet and to a depth of 375 feet. Drilling in the fall intersected the zone at a depth of 1,000 feet and followed it for a distance of 5,000 feet. The results were encouraging, and Crestaurum Mines Limited was formed in October 1945. Plans were immediately made for an underground prospect (Lord, 1951).

Some used equipment was purchased and other cost-saving measures implemented to assure freight delivery by spring breakup of March 1946. The company also purchased several buildings left over from U.S. Military Canol operations at Mills Lake on the Mackenzie River. These were dismantled and re-erected at Crestaurum (The Yellowknife Blade, Feb. 16th 1946). A company owned Cat train of eight sleighs pulled by a Cat D-7 tractor arrived at Daigle Lake March 18th 1946 with 80 tons of equipment and supplies (The News of the North, Mar. 23rd 1946). Great difficulty was experienced in getting all supplies to the property due to an early breakup (Ames, 1946). By March 1946 a camp had been erected and a location for the shaft was picked out. An all-weather road was cleared from the north end of Giant Mine to Crestaurum; a joint venture project between the company and the Federal Government that totaled about \$37,000. This road had a length of 10 kilometers and was one of Yellowknife's first out-of-town 'highways' (The News of the North, July 1946).

Shaft Development

Shaft sinking began on May 15th 1946. (Ames, 1946) The 3-compartment shaft (9 feet x 17 feet) was sunk to a depth of 420 feet and three level stations were cut at the 170-, 295-, and the 420-foot level. Some lateral development commenced from the 1st and 2nd levels during the remainder of 1946. This work was restricted to only a total of 360 feet of development. Crosscuts extended northwesterly towards the #1 zone, where it is reported that an intersection was made in the 160-foot crosscut on the 2nd level. The deposit remained 30 feet from the face of the 200 foot crosscut on the 1st level, while being 40 feet northwest of the 3rd level station. Two ore shoots were identified within the #1 shear zone (see **Geology and Ore Deposits** for detailed description) (Lord, 1951).

Mining Plant and Camp

A large mining plant was installed at Crestaurum for the sinking program, and included two Canadian Ingersoll-Rand air compressors of 680 cubic feet per minute each (driven by Cat D-17,000 diesel engines), a portable 360 cubic feet per minute air compressor, a 44 KVA General Electric generator driven by a Cat D-4,400 diesel engine), a Canadian Ingersoll-Rand 2-drum air hoist (size 9 inch x 8 inch), and a wood-fired boiler of 150 horsepower. The mining fleet included an Eimco 12-B mucking machine, a Mancha locomotive, and ore cars. Buildings included a dry, shop, 65 foot timber headframe, assay lab, powerhouse, 2-story office/warehouse, two 1-story bunkhouses, a cookery, garage, and two cottages, plus several tents (Lord, 1951; The Western Miner, Nov. 1947).

Employees

In August 1946, there were 54 men employed at Crestaurum Mine. Management employees at that date included: Ev C. Rudd, mine manager; W.B. Blair, engineer; and Donald Foote, master mechanic (Lord, 1951; The Northern Miner, Aug. 1st 1946). In September 1947, by which time underground operations had ceased, there were 20 men employed. Jim D. Mason was in charge as manager, together with E.A. Haigh, accountant and J. Stevenson, surface foreman (The Western Miner, Nov. 1947).

The first phase of development was completed early in 1947 and the underground was allowed to flood in February 1947. (Lord, 1951) Additional geological investigation of the property was accomplished during the year and further gold discoveries were made at the #2 shear zone, north of the shaft area. Diamond drilling extended the #2 shear zone an additional 1,500 feet and lengthened the north ore shoot an additional 200 feet. The potential to develop more resources on the property was regarded as excellent (The Western Miner, Nov. 1947).

Cost of operations to September 1947 amounted to \$415,300 (Lord, 1951). It was believed that if production were to begin, a cheap source of power would be required. Therefore, the company decided to wait until the Snare River Hydro plant was completed. Even after this power was available starting in late 1948, and after Giant Mine had roared into gold production, it was felt that the industry was not strong enough to risk further developments at Crestaurum. Labour shortages, high costs of supply, and the low price of gold as a result of the dollar parity of July 1946 were also issues.

Abandonment

Meanwhile, most equipment and supplies at the mine were liquidated and a permanent caretaker (Sam Daigle) was stationed at the camp. It would appear as though the property was abandoned in the mid 1950s, at which time the site was vandalized and pillaged of remaining supplies and building materials. Then, in October 1957, arsonists set the buildings on fire and burned down almost the entire site (The News of the North, Oct. 31st 1957).

Exploration Since Mine Closure

No further work was done until 1964 when Crestaurum Mines Limited transferred the claims back to Transcontinental Resources Limited. Northbelt Yellowknife Mines Limited was formed with equal ownership split between Transcontinental Resources and Falconbridge Nickel Mines Limited. In 1966, a large group of claims up the Yellowknife volcanic belt, including the old Crestaurum property, were amalgamated into Northbelt's holdings with exploration under the management of Giant Yellowknife Mines Limited. Diamond drilling was undertaken on the Crestaurum deposit in the 1970s, indicating a resource of 100,000 tons of ore grading 0.55 ounces per ton gold to a depth of 400 feet. It was considered a viable open pit operation. Treminco Resources Limited optioned the claims from Giant Yellowknife Mines Limited in 1989, but a period of economic recession in the early 1990s resulted in a cessation of all work (Treminco Resources Ltd. Annual Report, 1990).

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085JNE0014



DAMOTI LAKE

Advanced Exploration (Non-Operational)

Gold

64° 08' 30" 115° 06' 45"

Years of Primary Development: 1996

Mine Development: 1,420' decline to 132' depth, 2 levels (495' dev.)

Introduction

The site is located between Damoti and Ranji Lakes in the Indin Lake region, located 192 kilometers north-northeast of Yellowknife, NWT. A brief underground program in 1996 explored a mineralized gold zone within the local banded-iron formations.

Brief History

The area around Damoti Lake was first prospected in the mid 1940s, but the discovery of gold at this property was not made until 1992 when government geologist John Brophy reported gold on BIF Island within the banded-iron formations. The 'DAM' and 'SUF 200' claims were staked by Lou Covello in 1993 and serious exploration efforts began in 1994. An underground exploration program on the Horseshoe zone was completed in 1996 by Quest International Resources Limited. Other work has since consisted of additional drilling exploration and geological mapping.

Geology and Ore Deposits¹

Damoti Lake is located within the 20 kilometer wide Indin Lake Supracrustal Belt of Archean metasedimentary and metavolcanic rocks of the Yellowknife Supergroup. Much of the region is underlain by folded, metamorphosed greywacke-argillite turbidites. Volcanic rocks of the area are mainly mafic units including pillowed flows (with minor andesite) and local interflow breccias. Occurrences of quartz diorite and gabbro sills throughout the Belt are thought to be related to the volcanic units. Local granodiorite bodies and pegmatites also intrude the belt. Within the turbiditic sediments are strongly folded, locally auriferous, banded-iron formation units. These banded-iron formations occur as lens/pods or conformable beds up to several kilometres in length with widths up to tens of metres. Quartz veins often cut the exhalative and turbiditic sediments, with increased concentrations in the fold noses and the banded-iron formation units.

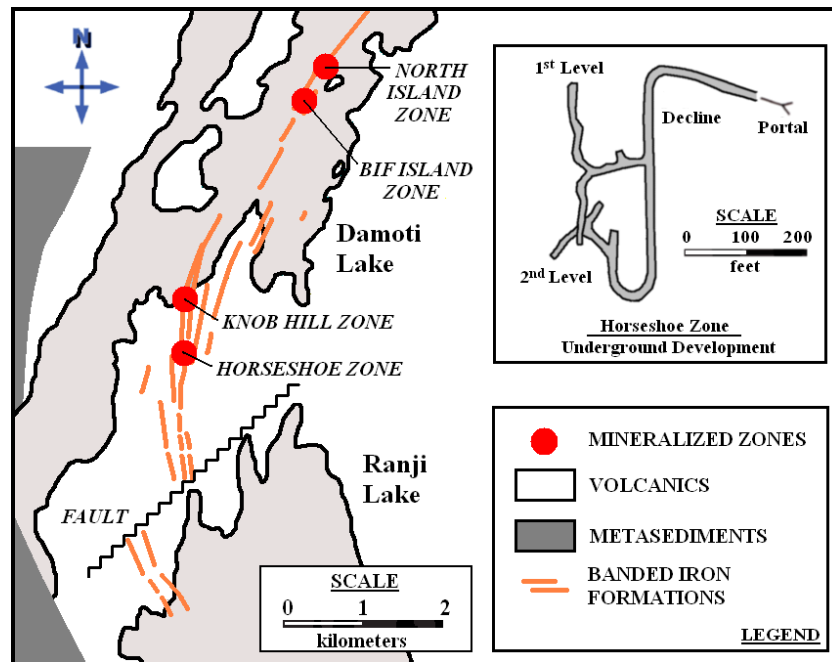


Figure 1. Damoti Lake regional geology and underground plan for the Horseshoe zone.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

The Damoti Lake property covers a portion of a metasedimentary belt on the eastern edge of a narrow band of northeast trending volcanics. The metasediments are lower greenschist grade turbidites with graded beds and load structures, and a planar north-northeast trending fabric dipping 75 to 80° west. The turbidites are repeatedly folded into a series of tight folds and contain interstratified bands of iron formation. Several dikes and generations of quartz veins, and a small granodiorite body, intrude the metasediments and iron formation, and late faults cut the fold structures as well as some of the early intrusions and the zones of mineralization. Diabase dykes intrude all units and lie along faults.

The Horseshoe zone (300 meter strike length) is located near the south shore of Damoti Lake and is associated with several other gold showings that are very similar geologically. The gold showings in the banded-iron formation unit have a close spatial association with quartz and/or chlorite veins (or shear zones). The banded-iron formation showings tend to consist of chloritic sulphide facies and cherty-grunerite facies in the cores, with silicate facies and silicate-amphibole facies along the margins. The intensity of alteration and gold grades generally diminishes outward from the center of the banded-iron formation units and gold is strongly associated with sulphides. In the Horseshoe zone, pyrrhotite is the dominant sulphide. Gold mineralization is strongly concentrated in the eastern limbs of synclinal folds and in the fold closures. Additional gold mineralization of significant grade has been found within the Horseshoe zone in several deep-seated zones.

Quest International Resources Limited (1996)

In March 1996 the decision was made to go underground on the Horseshoe zone to test the extensive mineralized zones within the banded-iron formation. A decline was collared in June, driven at -15%, and driven to a depth of 132 feet. Total length of the decline and associated crosscuts are 1,420 feet, with two levels established at 82-foot (1st level) and 132-foot (2nd level) depth. Drifting on the 1st level totals 396 feet and workings on the 2nd level totals 100 feet. Underground development was completed on October 15th 1996 (Russell, 2004).

Gold values on the 1st level were encountered over a strike length of over 150 feet. Mineralization on the south end of this section was reported to be wider and at higher grades. Both levels have promising mineralized zones and are almost identical, suggesting continuity of a high-grade deposit. Diamond drilling from the underground workings was completed in 35 holes. Mineralization within the Horseshoe zone was encountered in 23 of these holes at seven locations. The underground was extensively mapped and sampled. Ore from decline development was stockpiled on surface next to the decline portal, and is estimated to consist of 3,810 tonnes grading 15 grams per tonne gold (Russell, 2004).

Exploration Since Mine Closure

Diamond drilling continued in 1997 to upgrade the gold resources within the inferred category and to search for other ore zones. Sixty-nine holes were drilled totaling 38,000 feet on the BIF, Horseshoe, Arc, and Octopus zones, as well as undefined areas of the claim group. Work then ceased due to a drop in gold prices in 1997-1998. Quest International reorganized into Standard Mining Corporation, a subsidiary of Doublestar Resources Incorporated, in 1999. In June 2002, Canadian Zinc Corporation Limited entered into an option agreement with Doublestar to earn a 50% interest in the Damoti Lake project, but this option was cancelled in May 2003. A new option agreement was presented by Anaconda Gold Corporation in 2003 giving Anaconda the right to earn a controlling interest by spending CDN \$2.5 million in five years (Russell, 2004). Diamond drilling was conducted in during 2004-2005 by Anaconda. A total of 52 holes were drilled totaling 4,191 meters, delineating high-grade gold resources in the Horseshoe zone. An ore reserve of 40,600 tonnes (measured and indicated) grading 26.17 grams per tonne and 17,800 tonnes (inferred ore) grading 16.38 grams per tonne was reported for the Horseshoe zone (Anaconda Gold Corp. Press Release, Nov. 17th 2005). Exploration continues and permits for a resumption of underground exploration are planned.

References and Recommended Reading

Anaconda Gold Corp. Ltd. Press Releases. 2005-2006.

Russell, R.H., 2004. Evaluation Report on the Damoti Lake Property, N.W.T., Canada. Doublestar Resources Ltd., January 2004.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>)



DESTAFFANY *including Best Bet*

Minor Producer (Remediated)

Tantalum, Lithium

62° 10' 45" 112° 12' 30"

Years of Primary Development: 1942-1954

Mine Development: 3 primary open cuts, 75' vertical shaft, open pit (Best Bet)

Years of Production: 1944, 1947-1948, 1953-1954

Mine Production: 18,991 tons milled = 43,060 lbs TaCb conc., 105 tons Li conc. (824 lbs Ta, 329 lbs Cb, 25,108 lbs Li) + 45 tons high-grade amblygonite shipped

Introduction

The DeStaffany Mine was a rare-metal production operation located 115 kilometers southeast of Yellowknife, NWT on the shores of Great Slave Lake (Hearne Channel). It was one of only two post-war mines of its type in the region. The site has not been visited by the author. It is known that the Federal Government conducted a cleanup of the site in the 1990s.

The above coordinates are for the main mine and mill site on the shores of Great Slave Lake. The Best Bet open pit is located on the northwest shore of Drevor Lake at coordinates 62° 13' 35" 112° 18' 10".

Brief History

The Moose #1 dyke was discovered by Gus DeStaffany and Al Greathouse in June 1942, being originally staked as a tungsten deposit. The 'Moose' claims were staked. Following the encouragement of the Geological Survey of Canada towards the development of rare-metal deposits, DeStaffany reviewed the mineralogy of his claims and found an abundance of tantalite-columbite showings. First development occurred during 1943 and 1944. The first plant to recover a concentrate went into operation in 1947, and after a short period of inactivity, reopened in 1953 with a newer plant. The mine closed in 1954.

Geology and Ore Deposits¹

The Moose #2 pegmatite dyke and smaller Moose #1 dyke occur in the Yellowknife Sedimentary Basin within the Slave Structural Province. They are members of the "Yellowknife pegmatite field", a series of pegmatites east of Yellowknife that are probably associated with a number of late Archean granitoid intrusions of the Prosperous Granite suite. The pegmatites are most abundant in Burwash Formation sediments where the sediments have been metamorphosed to lower amphibolite facies. There is a broad zonation in the distribution of pegmatite varieties relative to the larger intrusives in the area: an inner zone dominated by beryl-rich pegmatites is closest to the intrusion, and is successively enclosed by zones dominated by tantalum-columbium/nobium-rich varieties and then by lithium-rich varieties. The Moose dykes are lithium-rich varieties and are members of the Faulkner Lake series.

The two dykes are 1600 meters apart and appear to be aligned along the same structure. The #2 dyke strikes northerly and dips steeply to the west. It is about 420 meters long and up to 60 meters wide at surface. A 221 meter strike length was drilled to a 30 meter depth, showing the average thickness at that level to be 7.5 to 9 meters. The #1 dyke is slightly less than 300 meters long with a maximum thickness of 10 meters. Both are internally zoned as follows: core is quartz-spodumene-amblygonite; wall zone is Kspar-quartz-muscovite-cleavelandite; border zone is quartz-muscovite. Other minerals present include schorl tourmaline, beryl, cassiterite, petalite and lithiophyllite.

Lithium mineral rich and columbium-tantalum mineral rich zones are largely mutually exclusive. The former are concentrated in or near the central core while the latter are concentrated in two feldspar-rich zones near the walls and in the fine-grained border zone. The distribution of columbite-tantalite is also zoned, with a blocky tantalum -rich variety concentrated in the border zone and a platy columbium-rich variety elsewhere. Beryl tends to occur as white irregular crystals and masses on the order of 20 centimeters across within the wall zone bordering the core. The quartz core is particularly well-developed in the #2 dyke, reaching a width of about 4 meters. Large crystals and blocks of spodumene and amblygonite reach sizes of up to 2 meters and 1 meter respectively. Patches and stringers of metasomatic albite and secondary muscovite are found throughout the body.

The Best Bet pegmatite dyke is a lithium-rich variety belonging to the Faulkner Lake series, located seven kilometers northwest of the Destaffany Mine site on the northwest side of Drevor Lake. It is approximately 100 meters long and

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

up to 8 meters wide. It strikes north with a steep westerly dip and is roughly concordant with foliation in the enclosing schist. Some shearing has taken place along the hangingwall contact.

Gus DeStaffany (1942-1944)

Early work done on the 'Moose #1' and 'Moose #2' claims was the exploration of tungsten deposits. Several small pits and a short shaft (#1 shaft) were sunk on 'Moose #1' claim by Gus DeStaffany and Al Greathouse during 1942 and 1943. About 100 tons of scheelite ore was stockpiled. (DeStaffany Tantalum Beryllium Mines Ltd. Annual Report, 1947) After the discovery of tantalite and columbite occurrences in the area during 1943 and 1944, DeStaffany enlarged the claim group to cover two pegmatite dykes and also arranged for the purchase of nearby groups including the Best Bet and Big Hill properties (The Nor-West Miner, Jan-Feb. 1945).

Dr. Fred Jolliffe of the Geological Survey of Canada examined the Moose #2 dyke in August 1943 and his enthusiasm of the dyke as a tantalum orebody caused Gus DeStaffany to shift his interest from a tungsten mine to a tantalum and other rare-earth mineral producer. At this time, tantalum and columbium products were in high demand around the world. The rare columbium metal, when refined and alloyed with steel, would allow for much greater tension characteristics for the steel industry and by the military for use in war weapons. Tantalum was valuable for medicinal and technological purposes (The Nor-West Miner, Jan-Feb. 1945).

First Mining

During September 1943, work got underway to stripping of the Moose #2 dyke's southern section. Considerable beryl and much tantalite crystals were noted during this initial development. In May 1944 mining was shifted to the northern section of the Moose #2 dyke, 700 feet north of the southern section. Preliminary stripping was conducted on Moose #1 dyke during September 1944 and great tonnages were estimated. DeStaffany claimed that the ores were highly concentrated and could be cheaply exploited (The Nor-West Miner, Jan-Feb. 1945).



The Western Miner Magazine

Figure 1. DeStaffany milling plant, 1947.

Mill Construction and Test Production

Erection of the first stages of the milling plant was started in November 1943, and some crude machinery was installed. New camps and a dock were also erected (The Nor-West Miner, Jan-Feb. 1945). It was later reported that in 1944 a test-run of the mill was accomplished, during which period 30 tons of ore from the Moose #2 dyke was milled to produce 660 pounds of concentrate, grading 70% combined tantalite-columbite. The product was shipped to U.S. markets (The Northern Miner, May 14th 1953).

DeStaffany Tantalum-Beryllium Mines Limited (1945-1948)

Some public financing was required to get the mine officially started. The company DeStaffany Tantalum-Beryllium Mines Limited was incorporated in February 1945 and a share offer was made available to the public. During the summer of 1945, surface mining was underway and the mill was being improved in preparation for production. The company announced plans to mill ores and ship concentrates out over the ice of Great Slave Lake to Hay River. As a result of previous stripping operations, plenty of ore was exposed and ready to be blasted during winter to keep the mill running. But due to scarcity of needed equipment, a production decision was again delayed in 1946 (The Nor-

West Miner, Sept. 1945). In the summer of 1946, a small crew was at work on the mill and sinking #2 shaft on the Moose #2 dyke. This shaft was sunk using drills operated by a Cat D-2 tractor engine with compressor, and a gasoline powered hoist engine (Meikle, 1946). By early 1947 it was down 68 feet and may have obtained a final depth of 75 feet with a short amount of lateral work at that depth (N.W.T. Geoscience Office Assessment Report #82159). Lateral work intersected the footwall of the dyke deposit (The Nor-West Miner, Jan-Feb. 1947).

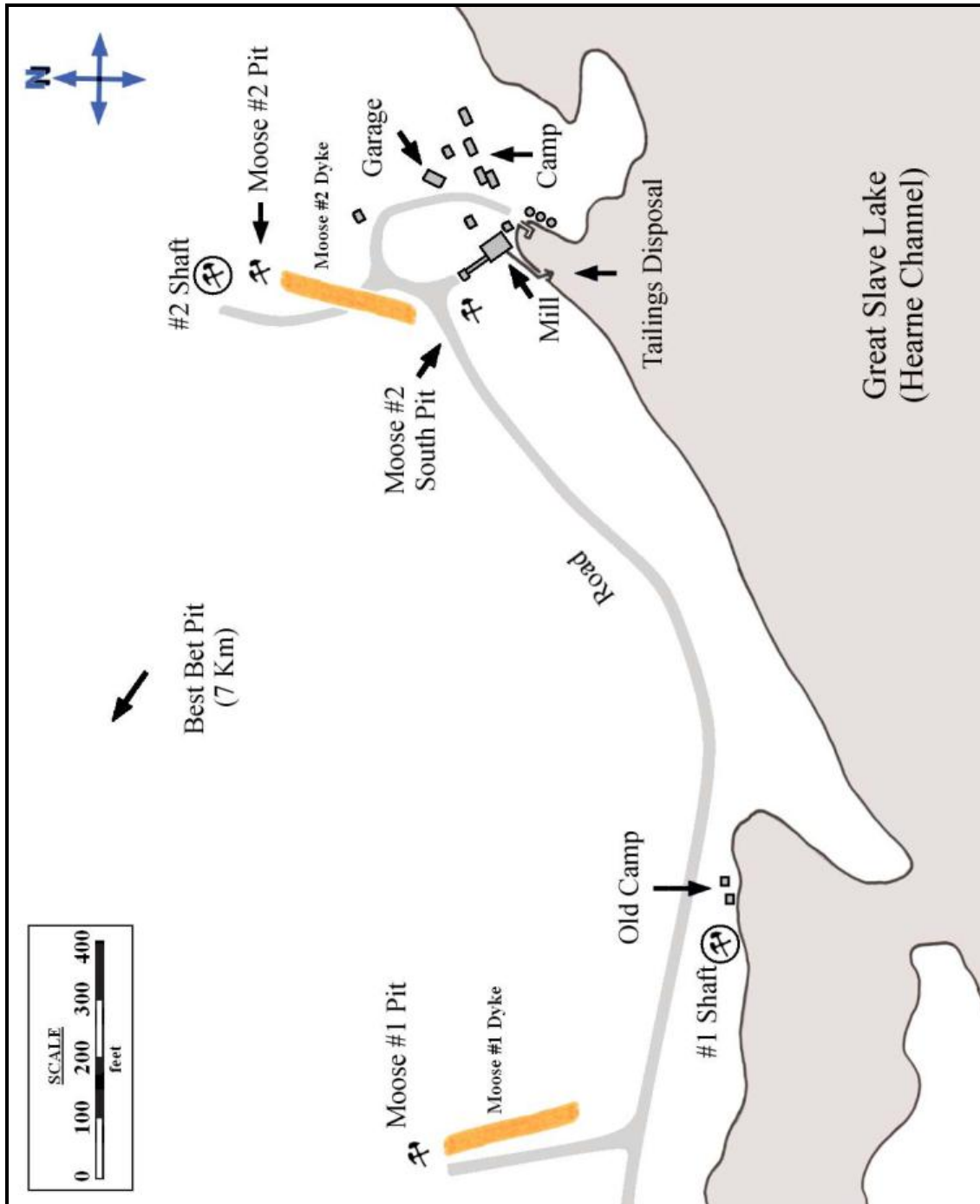


Figure 2. DeStaffany Mine surface plan, c.1955.

Milling Plant

The simple concentrator, erected during 1943-1947, had a limited capacity of 25 tons per day. This mill operated with jaw crusher, rolls, and a two-stage classification system and was operated by a Cat D-3,400 diesel generator. It was not a very effective recovery plant and had been erected from supplies on hand or acquired very cheaply (Lord, 1951).

1947 Production

The mill operated between September and October 1947 as part of an expanded trial run. Mill feed of 30 tons was derived from the Moose #2 pit. The Best Bet pit, located 7 kilometers north of the site, provided 2 tons of mill feed. Concentrate production for the year amounted to 1,200 pounds of tantalum-columbium concentrate of unknown grade (DeStaffany Tantalum Beryllium Mines Ltd. Annual Report, 1947; Lord, 1951).

1948 Production

Some further production was attained in 1948, when more material was mined from a high-grade section at the Best Bet pit and milled. About one ton of ore was processed to recover 1,400 pounds of concentrate of unknown grade (Lord, 1951). A period of inactivity occurred between 1948 and 1952 while the company underwent reorganization.

Boreal Rare Metals Limited (1953-1954)

This company was formed in October 1951 with the purpose of acquiring the rare-metal property from Gus DeStaffany. Negotiations were completed in early 1953. Primary focus was on the northern section of the Moose #2 dyke, where officials believed the grade of ore to be rich enough to justify the expenditure of reactivating the plant. It was also hoped that beryl or lithium production would be viable at some point in the future (Boreal Rare Metals Ltd. Prospectus, 1952). A total of 11 dyke deposits had been discovered at the extensive property by 1953 but not enough exploration had been conducted to form an adequate reserve figure (The Northern Miner, Mar. 5th 1953).

Quebec Refinery

The Boreal company also saw it reasonable to erect its own refinery plant to refine the tantalum and columbium concentrates produced at the DeStaffany Mine. A site was chosen at Cap de la Madeline, Quebec (Boreal Rare Metals Ltd. Prospectus, 1952).

Rehabilitation of the camp and plant buildings at DeStaffany was undertaken during 1952-1953 and the mill building was enlarged to accommodate a plant of 125 tons per day. A new diesel generator was installed (Caterpillar D-13,000 and 83 kilowatt generator). New camp facilities were also constructed to replace old log cabins. A smaller camp was established at Drevor Lake for crews working on the 'Best Bet' claims. Construction was speeded during October-December 1953 in preparation for production.

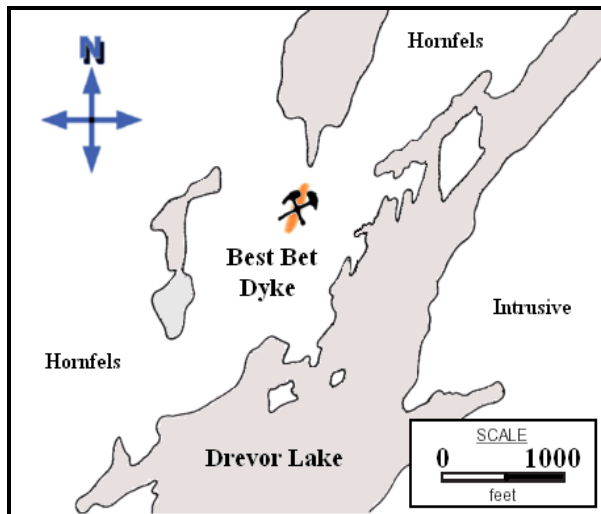


Figure 3. Best Bet open pit location.

Production commenced on December 9th 1953 using the Moose #2 pit as mill feed. A test run of ore (225 tons) was processed during the first few weeks to yield 1,100 pounds of concentrate, indicating a grade of 5 pounds per ton. Initial production of ore was 50 tons per day (The Northern Miner, Feb. 11th 1954). To March 1st 1954, the mine produced 8,645 pounds of concentrate from 3,226 tons milled, equivalent to 2.68 pounds per ton (The Northern Miner, June 24th 1954). By April 30th 1954, the mine had milled 8,689 tons of ore to produce 17,925 pounds of concentrate (News of the North, June 18th 1954).

High tailing losses during initial stages of milling were blamed for the lower grades. 1,610 tons of additional ore had been stockpiled, ready to be processed. Tonnage during this period averaged 97 tons per day. Operations early in 1954 were greatly hampered by cold weather but the work crew was able to keep a steady supply of ore to the mill (The Northern Miner, June 24th 1954).

This provided a learning experience for the company, and from then on it was decided to focus on milling during the summer months and mining the ore during the winters. The Best Bet deposit was mined during the early winter of 1954 with ore being trucked to the milling plant by winter road. Thirty men were employed in January 1954.

Milling Plant

The plant had a capacity of 125 tons per day but it is reported to only have operated at a rate of 40 to 90 tons per day during 1954. Ore was received into the ore-bin by truck and tractor and crushed in a 30 inch x 15 inch Kue Ken jaw

crusher. Grinding was performed in a Denver rod mill, and then the minus 65-mesh material was classified in a Gibson unit and then sized on a Dorrco sizer, followed by tabling on five Wilfley tables. Re-tabling was required for some products in order to eliminate gangue. Two concentrates were produced: tantalum-columbium/nobium and lithium/amblygonite (Driscoll, 1955).

Diamond drilling operations were conducted during the summer and fall of 1954 to outline the deposits at Best Bet and Moose #2 dykes. Drilling showed that mineralization at depth (100 feet) was similar to that on surface. The Moose #2 dyke was drilled a length of 725 feet and showed a width of 25 to 30 feet at depth. The Best Bet dyke was drilled over a length of 300 feet and showed a progressive widening at depth, up to 32 feet. This work, plus the concentrates recovered from milling, also suggested economic lithium and beryllium deposits (The Northern Miner, Oct. 7th 1954). In the Best Bet dyke alone, 187,800 tons reserves to a depth of 300 feet were reported. Total reserves, reported later in 1955, were 2,027,500 tons grading over 1% lithium, in five deposits within economical distance to the central milling plant (The News of the North, Nov. 18th 1955).

Production

The mill was shutdown in August 1954 after processing 18,928 tons of ore from the Best Bet and Moose #2 dykes. Metallurgical difficulties in the plant were cited as a reason for the closure (National Mineral Inventory).

Year:	Ore Milled:	Concentrate Produced:	Metals Recovered:
1944	30 tons	660 lbs TaCb conc.	unknown
1947	32 tons	1,200 lbs TaCb conc.	*
1948	1 ton	1,400 lbs TaCb conc.	*
1953-1954	18,928 tons	39,100 lbs TaCb + 105 tons Li conc.	824 lbs Ta, 329 lbs Cb, 17,052 lbs Li
<i>Total:</i>	<i>18,991 tons</i>	<i>43,060 lbs TaCb + 105 tons Li conc.</i>	<i>824 lbs Ta, 329 lbs Cb, 17,052 lbs Li</i>

Table 1. DeStaffany Mine production. (sources: Lord, 1951; The Northern Miner, May 14th 1953; National Mineral Inventory) (*) concentrates produced in the 1940s were not refined until 1954

The company planned to install new equipment to increase recoveries and efficiencies of the milling plant. Milling during 1953-1954 afforded a recovery of 39,100 pounds of tantalum-columbium concentrate and 105 tons of lithium concentrate (amblygonite) (Statistics Canada, 1957). The lithium material was shipped for bulk testing purposes to Foote Minerals Limited who, upon favorable results (25,108 pounds lithium recovered according to Statistics Canada), offered to purchase further quantities of the lithium ores from Boreal. Some of the ore material from the Best Bet pit (which apparently totaled 4,000 tons) was high-grade and did not require milling. Forty-five tons of high-grade lithium ores (amblygonite) were hand picked from the mill feed and sacked for direct shipment (Boreal Rare Metals Ltd. Annual Report, 1955).

Refinery Operations

The refinery at Cap de la Madeline, Quebec went into operation in December 1953 treating older DeStaffany concentrates. Metals were produced during trial runs with some complications, but the process was somewhat perfected during 1954-1955. The refinery reportedly processed 20,000 pounds of DeStaffany tantalum concentrate in 1954-1955 and recovered 824 pounds of tantalum and 329 pounds of columbium/nobium (National Mineral Inventory).

Post-Closure

The Boreal company planned to place the DeStaffany Mine back into production by the fall of 1955 with new milling units to increase the capacity of the plant to 500 tons per day, treating tantalum-columbium and lithium ores from the known deposits. Tailings from previous operations, containing high mineral values (19,500 tons containing 24% lithium minerals), could also be treated (The News of the North, Nov. 18th 1955). Disaster struck on January 9th 1955 when the mill and power plant burned down. This was only a temporary blow: the plant was insured and most of the equipment was to be replaced with the expanded mill anyways (The Northern Miner, Feb. 10th 1955). Operations at the Quebec refinery continued as the company reported overcoming previous metallurgical difficulties. However, as the company could acquire outside sources of tantalum-columbium concentrates at lower costs compared to mining

its own, plans to reopen the DeStaffany Mine were delayed. The company also felt that bulk production of concentrates would best be postponed until the envisioned railroad to Great Slave Lake was built (The Northern Miner, Feb. 9th 1956).

The final blow to the future of the mine was the termination of U.S. purchasing of worldwide tantalum-columbium stockpiles, along with the uncertainty of lithium prospects. Owing to these events, Boreal Rare Metals Limited went bankrupt in 1956 and no further work was done (The News of the North, Nov. 18th 1955; Boreal Rare Metals Ltd. Annual Report, 1955).

Mine Development Summary

Mining developments at DeStaffany Mine has focused on three primary pits. The #1 pit was mined on Moose #1 dyke during early operations. Its mined dimensions were 60 feet long, 18 feet wide, and 15 feet deep. The #2 pit was 240 feet x 60 feet x 30 feet deep and was mined on the southern section of the Moose #2 dyke, just north of the mill. The Best Bet pit was an open cut 260 feet x 26 feet x 27 feet deep. Sinking of a 75 foot shaft (#2 shaft) in 1946-1947 on the northern section of Moose #2 dyke did not contribute to mill feed.



Henry Busse - NWT Archives – N-1979-052-3279

Figure 4. *DeStaffany Mine from Great Slave Lake, c.1955.*

Exploration Since Mine Closure

Boreal Rare Metals Limited went bankrupt in May 1956; in 1957, the assets of the company were acquired by Beauport Holdings Limited who held 67 claims in the area including the old DeStaffany Mine. Some surveying of the claim group was performed in 1963. The property was re-staked as the 'Elk 1' claim by Charles O'Sullivan in 1978. Hemisphere Development Corporation acquired a 50% interest in the claims and prepared an evaluation of the economic potential of the Moose #2 dyke. The tantalum-columbium zones were estimated to contain 246,755 tonnes and the amblygonite and spodumene zones 388,275 tonnes to a depth of 70 meters. Hemisphere Development continued work in the 1980s, but there has been no significant work since (National Mineral Inventory).

References and Recommended Reading

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0002 / 0108


DIAVIK
Producer (Operational)
Diamond

64° 30' 00" 110° 17' 00"

Years of Primary Development: 1995-1997, 2000-current

Mine Development: primary open pit

Years of Production: 2003-current

Mine Production: ongoing (see Table 3)

Introduction

The Diavik mine is a diamond producer located on East Island of Lac de Gras, 300 kilometers northeast of Yellowknife, N.W.T. It began production in 2003 as Canada's second diamond mine. The mine is currently managed by Diavik Diamonds Mines Inc., a Rio Tinto company. The mine is a joint-venture between Rio Tinto plc (60%) and Harry Winston Diamond Corporation Limited (40%). Under the current life of mine plan, underground diamond production will begin in 2010 and continue beyond 2020. Open pit mining is expected to cease in 2012.

History in Brief

The claim block surrounding the producing mine was staked during the exciting "Diamond Rush" of 1991 and 1992 by prospectors working for Covello, Bryan & Associates, under contract with Aber Resources Limited. ¹ Exploration of the property took place in 1992-1993 to assess its kimberlite potential. In June 1992, Kennecott Canada Incorporated (a subsidiary of Rio Tinto plc) acquired a right to earn 60% interest through the advancement of funds for exploratory programs. The primary kimberlite pipes, A-154 South, A-154 North, A-418, and A-21 were discovered in 1994-1995. Underground development began in 1995 to extract bulk samples from the A-154 and A-418 pipes. This work identified the A-154 pipes as being the target of first development, followed by mining of the A-418 and A-21 pipes.

Diavik Diamond Mines Inc. was formed as a Rio Tinto plc subsidiary in December 1996 to act as manager for the joint-venture project. During 1997-1998, feasibility and various assessment studies were compiled to fulfill the company's obligation to provide detailed information on the construction and operation of the diamond mine for government and local interests. In the summer of 2000, the company received all the necessary government permits to place the property into production. Construction and open pit preparation was underway in 2001-2002. Diamond production began January 2003 and commercial production was achieved in August 2003. The mine has a projected life of up to 22 years and operations are ongoing as of 2009.

Geology and Ore Deposits ²

The Lac de Gras area is located in the central Slave Geological province. The Slave craton contains deformed and metamorphosed, Archean aged metaturbidite and lesser metavolcanic rocks of the Yellowknife Supergroup. These supracrustal rocks have been intruded by extensive Archean granitoids, and are in turn intruded by undeformed, late Archean granites and diabase dike swarms. The three main Archean lithologies on East Island are greywacke-mudstone metaturbitites, quartz diorite, and granodiorite. The Diavik kimberlites are Eocene in age (54 to 58 million years) and were formed by volcanic surface eruptions and near surface injections of kimberlite magma and volcanoclastic debris into the granitic country rocks and into mid Cretaceous to Tertiary mudstones. The A-154 and A-418 pipes are carrot-shaped and begin to narrow at a depth of 320 meters below the surface.

Kennecott Canada Incorporated (1995-1996)

Exploration during 1994 outlined a sizeable diamond resource within the A-154 pipe on the property of Aber Resources Limited. The A-154 South pipe contained a drill indicated resource of 8.4 million tonnes to a depth of 250 meters, with reserves to 650 meters exceeding 20 million tonnes. The A-154 North pipe had indicated reserves of 15 million tonnes to a depth of 650 meters. Further exploration early in 1995 outlined two additional kimberlite pipes – the A-418 and A-21. Grades and accurate values were not obtainable until large bulk sampling of the deposits could be undertaken. As a result, Aber Resources Limited began plans to initiate a joint-venture underground exploration program (Aber Resources Ltd., Annual Report 1994).

¹ Now known as Harry Winston Diamond Corporation Limited.

² Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

On March 23rd 1995, Aber Resources Limited and Kennecott Canada Incorporated signed the “Diavik Joint Venture Agreement” in which Kennecott earned a 60% interest in the diamond project at Lac de Gras for a \$300,000 payment to Aber and the expenditure of \$9.7 million on exploration and development. Kennecott became manager of the project (Aber Resources Ltd., Annual Report 1995).

Bulk Sampling

The A-154 South and A-418 pipes were targeted for bulk sampling. The bulk samples would be obtained through the driving of underground ramps into the pipes at depths of 150 meters below the water surface (lake depth is 15 meters in this area). Mobilization of equipment, supplies, and crews was made over the winter road early in 1995 and construction began in March-April. A campsite was erected with capacity for 75 people. A 550 meter all-weather airstrip was cleared to permit the usage of Twin Otter planes (Aber Resources Ltd., Annual Report 1994).



Diavik Diamond Mines Inc.

Figure 1. Decline ramp portal, Diavik 1999.

Underground Ramps

During 1995-1996, a portal was excavated on East Island and the A-154 South and A-418 pipes were tunneled into through two branches of ramps. Drifts were cut in 2 x 2 meter dimensions. The A-154 pipe was intersected and mined at a depth of 155 meters below lake level in 1995 through 670 meters of workings (152 meters of drifting), excavated between August and October 1995. The eastern half of the pipe was explored and sampled. In 1996, the A-418 pipe was mined at 145 meter depth through 194 meters of drifting in three headings. In this pipe, the western section was explored and sampled. Underground workings are said to total 2 kilometers. The ramp was collared in permafrost but continued into unfrozen ground under the lake and in parts of the kimberlite pipes.

Ground support using rock bolting, screening and shotcreting was required to control water inflow and erosion of the kimberlite, especially in unfrozen ground (Aber Resources Ltd., Annual Report 1996; *The Northern Miner*, June 10th 1996; N.W.T. Mine Inspection Services).

Kimberlite ore from the A-154 South pipe was split into two batches. One was sent to the Ekati Mine bulk sampling plant, 30 kilometers north, and the other to Yellowknife, N.W.T, where Kennecott had erected a bulk sampling plant (Aber Resources Ltd., Annual Report 1996). A total of 2,587 tonnes of ore were mined and milled in 1996 to recover 12,688 carats of diamonds (Bryan and Bonner, 2003). The bulk sample from the A-418 pipe was stockpiled to await trucking to Yellowknife, N.W.T. over the winter road early 1997. Large diameter core (L.D.C.) drilling was undertaken on all four pipes during this term and the material recovered was mini-bulk sampled (see **Table 1**). This drilling established mineralized depths of at least 390 meters for all the pipes.

Employees

Richard Lock was in charge of the 1995-1996 underground development and bulk sampling program. In the fall of 1995 there were 25 contractors employed with the underground program at A-154 South. Total workforce during the winter of 1995-1996 was 70 persons, housed in a tent and later in March 1996 a trailer camp.

Diavik Diamond Mines Incorporated³ (1997-current)

On November 29th 1996, Rio Tinto plc formed a new subsidiary company to continue with exploration at the Lac de Gras property - Diavik Diamond Mines Inc. It assumed management of the project in December 1996. Aber Resources Limited continued to maintain a 40% share in the joint venture and pay 40% of the exploration funds. Appraisal of the carats recovered from the A-154 South bulk sample was performed early in 1997 by independent appraisers in Belgium. Values of US \$67 carats per tonne were estimated at current markets. Appraisers were

³ A joint-venture company owned 60% by Rio Tinto plc and 40% by Harry Winston Diamond Corporation Ltd. Production royalties apply.

unanimous in their view that the diamonds were attractive, of high quality and readily marketable (Aber Resources Ltd., Annual Reports 1996-1997).

Year Mined:	Kimberlite Pipe:	Type of Sampling:	Year Processed:	Ore Sampled:	Diamonds Recovered:	Grade:
1995	A-154 South	Underground	1996	2,587 T	12,688 c	4.41 c/T
1996	A-21	L.D.C. Drilling	1996	7 T	22 c	3.10 c/T
1996	A-418	L.D.C. Drilling	1996	62 T	248 c	4.20 c/T
1996	A-154 North	L.D.C. Drilling	1996	72 T	157 c	2.19 c/T
1996	A-418	Underground	1997	3,350 T	8,325 c	2.76 c/T
1997	A-21	L.D.C. Drilling	1997	24 T	65 c	2.73 c/T
1997	A-10	L.D.C. Drilling	1997	7 T	0.11 c	0.02 c/T
2003	A-154 North	Open Pit	2003	19,342 T	11,771 c	0.61 c/T
2004	A-154 North	L.D.C. Drilling	2004	853 T	2,109 c	?

Table 1. Bulk sampling programs at Diavik Mine. (*T* = metric tonnes, *c* = carats, *L.D.C* = Large Diameter Core). (source: Canadian Minerals Yearbook, 1998; Diavik Diamond Mines Inc. Press Release, July 27th 2004)

1997 Bulk Sampling

Sampling and exploration continued during 1997. Kimberlite ore mined from the A-418 pipe in 1996 (3,350 tonnes) was trucked to the Yellowknife plant in early 1997 to recover 8,325 carats (Bryan and Bonner, 2003). The diamonds from A-418 were said to have the same attractive characteristics as those recovered from the A-154 South pipe and would be readily accepted by the market. Mini-bulk sampling of the A-21 pipe also continued during 1997 through L.D.C. drilling efforts. A small sample from the newly discovered A-10 pipe proved to be of low grade (see Table 1). Also discovered during 1997 was the A-840 and T-107 pipes (Aber Resources Ltd., Annual Reports 1996-1997). As a result of the 1996-1997 work, resource estimates were revised to 37,300,000 tonnes of ore within the four sampled pipes with an average grade of 3.18 carats per tonne. The A-154 South pipe contained the most ore and was believed to have the highest grades. The project moved from the pre-feasibility stage to the feasibility stage and the long process to obtain necessary land and water use permits began.

Although exploration operations continued during the following summers, development activity ceased in the fall of 1997 following the successful completion of bulk sampling. Future operations were dependent on the approval of the companies feasibility reports and the environmental assessments that would scrutinize the plans. Total cost of the bulk sampling stage of operations between 1995-1997 were estimated at US\$80 million (Aber Resources Ltd., Annual Reports 1996-1997).

Mining Plan

The basic mining plan suggested a mine life of 16 to 22 years through the mining of four pipes - the A-154 South, A-154 North, A-418, and A-21. All pipes were located under the waters of Lac de Gras, so specialized rock dikes would have to be constructed to surround the future open pits. The A-154 pipes would be the first mined. 400 jobs were expected as a result of operations (Aber Resources Ltd., Annual Reports 1997-1998).

In November 1999, the Federal government approved the Diavik Mine project and the company moved into the permitting stage. Mobilization of equipment and supplies for winter road transportation to the site began, although permits were not granted until March 2000 (Aber Resources Ltd., Annual Report 1999). As a result of the delay in permitting, only 1,000 truckloads were moved across the winter road to Diavik. An important part of this shipment was 23 temporary fuel tanks to supply the site during the construction program of 2000. Only 16 were brought to the site, resulting in a delay in completing the construction program (Aber Resources Ltd. Press Release, June 27th 2000).

Starting in June 2000, several kilometers of access roads were laid, airstrip built, a construction camp for 450 persons was built, erection of two of the permanent fuel tanks was finished, excavation started at the plant site, and several of the foundations were being prepared. A large quarry was being mined in the northern part of the island to provide rock for site development (Aber Diamond Corp. Ltd. Press Release, June 26th 2001).

Open Pit Dike

Specialized equipment was brought in over the 2001 winter road season to begin the extensive task of creating the protective dike surrounding the A-154 pipes. First, dredging of the dike and pit footprint was performed to remove the lake bed sediments preventing a stable floor for the dike. The process of dike construction then began (July 9th 2001) by placing rock into the lake through slip displacement to minimize silt. Crushed rock of –5 centimeter size was placed in the core of the dike, followed by –20 centimeter size rock, and then topped off with larger rocks. The material was compacted. A 0.8 meter concrete wall was then placed into the center of the dike down to bedrock to make it water proof. Equipment from Germany was brought to the site for this specialized task. Jet and curtain grouting sealed the dike to glacial till and bedrock below. The dike was completely closed in October 2001. Concrete placement ceased for the winter months but resumed in the summer of 2002 so that water could be pumped from the area to permit open pit mining of the A-154 pipes. Fish within the dike were returned to the lake. Clean water was pumped to the lake while silty water was stored onland for settling then remaining silt was removed in the water treatment plant. The engineering accomplishment of the A-154 dike was significant and it won Canada's top engineering award. The A-154 dike is four kilometers in circumference and contains approximately 6 million tonnes of rock (Aber Diamond Corp. Ltd., Annual Report 2001/2002).



Photo by Jiri Hermann, courtesy Diavik Diamond Mines Inc.

Figure 2. *Diavik Mine camp and plant facilities under construction, summer 2002.*

Site Construction

To provide additional accommodation for increased construction workers that would eventually total over 1,000 people, a 700 man capacity temporary complex known as the “South Construction Camp” was built (Aber Diamond Corp. Ltd. Press Release, June 26th 2001). Earthwork and building erection progressed rapidly at Diavik during 2001-2002. Steel framing and cladding of most of the plant buildings was completed during the summer of 2001. The third permanent fuel tank was erected and the truck shop was completed. Equipment in the mill was being put in place during the winter of 2001-2002; however, installation of the diesel engines in the powerhouse was awaiting the arrival of the units in early 2002. Construction of the dams for the tailings ponds, and the foundations for the permanent camp complex were started (Aber Diamond Corp. Ltd., Annual Report 2001/2002).

The winter road season of 2002 saw over 3,100 truckloads of buildings supplies, equipment, and mining machinery delivered to Diavik. Prefabricated units for the bunkhouses and camp complex were put into place and interior finishing was completed during the summer. The final units of machinery were installed in the mill, and the powerhouse was fitted with its new diesel engines. The sewage, water treatment, boiler, and power plants were also commissioned during this period. An addition was being started on the truck shop to house offices, warehouse, and other mechanical shops.

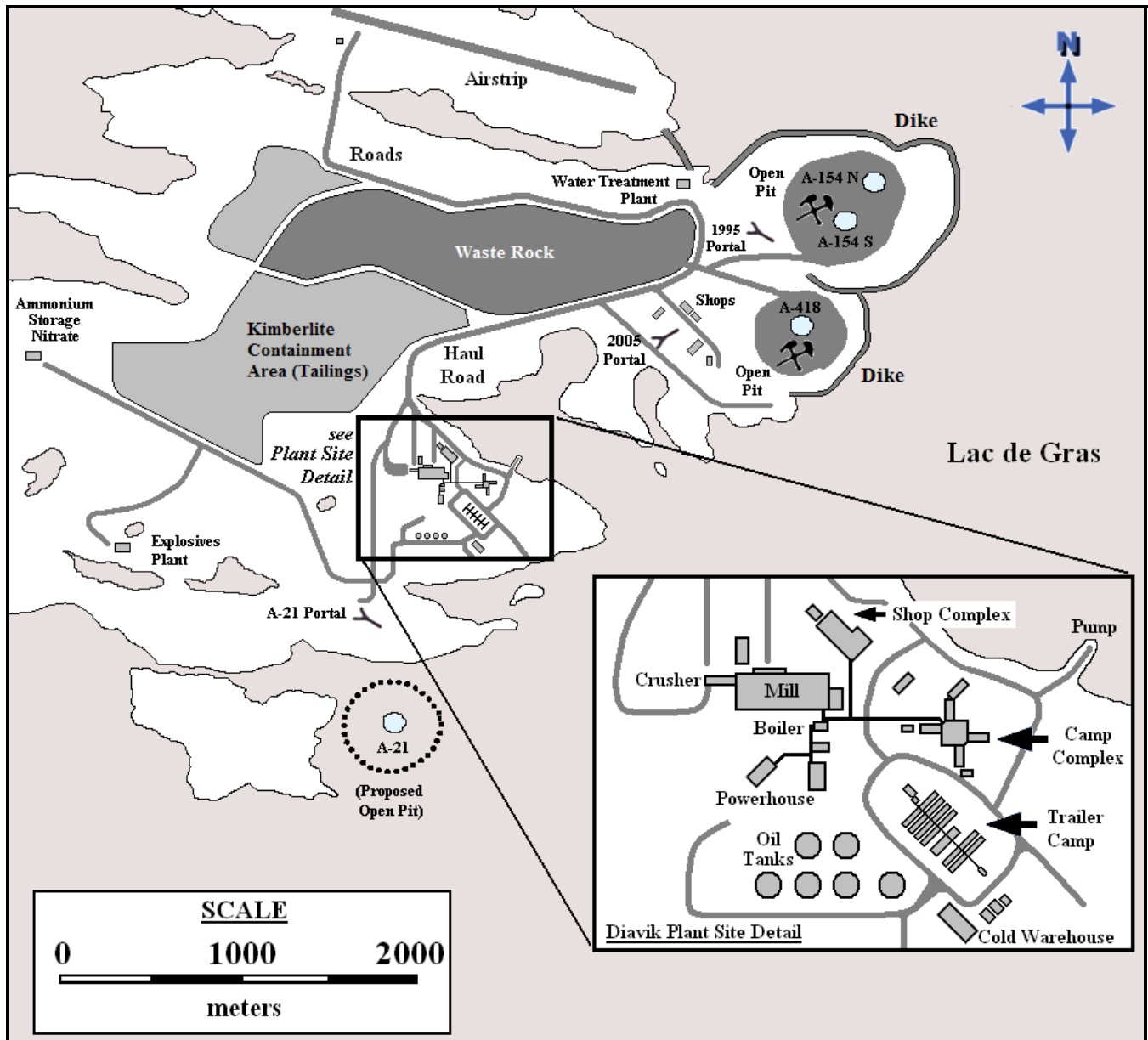


Figure 3. Diavik Mine property plan, 2009.

Exploration 2002

Continued prospecting of the Diavik claim block resulted in the discovery of five additional kimberlite pipes, bringing the total to 63 pipes, about half of which were known to be diamondiferous. Mini-bulk sampling and core drilling was underway to evaluate the new finds.

Pit Stripping

Draining of the A-154 open pit area started in July 2002 and was completed in September after the safe removal of 10.2 million cubic meters of water. Removal of lake bottom sediments, glacial till, and bedrock was then started. This overburden was up to 30 meters deep and needed removal to access the kimberlite pipes. On November 23rd 2002, first kimberlite ore was mined and trucked to the mill for sampling and breaking-in of processing equipment.

Production Begins

Production ramped up during December 2002 with initial mill-feed derived from kimberlite overburden (fine-grained ash and mudstone). Commercial operations began in January 2003 with the kimberlite orebody stripped from the A-154 South pipe (production during November and December 2002 is included in the annual production figures for

2003). On July 19th 2003, the mine celebrated its grand opening as the operation reached rated production levels. Removal of overburden wastes was completed. Processing of the lower-grade kimberlite ash and mud was completed and the higher-grade pipe was now the sole source of mill-feed. The Diavik project entered production ahead of schedule and within budget (Aber Diamond Corp. Ltd., Annual Report 2003/2004).

The winter road season of 2003 saw the arrival of 1,500 loads of equipment, fuel, and supplies to the mine, but also saw the departure of 800 loads of back-hauled equipment and supplies used during the construction phase of 2001-2002. Workforce at October 2003 was approximately 625 persons, of whom 73% were northerners and 36% were aboriginal (Diavik Diamond Mines Inc. Press Release, October 24th 2003).

Mining Operations

Mining has been by conventional open pit methods. The A-154 and A-418 are the two open pits in operation at present (2009). Benches are mined at 10 meters and ore is loaded into haul trucks via hydraulic excavators and loaders. Primary surface mining equipment includes (2009): eleven Komatsu 830E haul trucks (218 tonne), eight Komatsu HD785 haul trucks (90 tonne), one Hitachi EX3600 excavator (20m³ bucket), one Hitachi EX1900 excavator (12m³ bucket), one Terex RH200 excavator (21m³ bucket), three Drilltech D75EX blast drills, one Drilltech D90K blast drill, one LeTourneau L1400 front end loader (20m³ bucket), plus an assortment of auxiliary service machinery (Diavik Fact Book).

Milling Operations

The Diavik Mine processes up to two million tonnes of ore annually. The kimberlite ore is trucked to a storage area outside of the process plant. A primary sizer reduces the ore prior to processing, where it is mixed with water and crushed to less than 30 millimeters. The ore is then conveyed to the dense medium separation circuit where a fine grained, heavy and magnetic ferro-silicon (FeSi) sand is added to the crushed ore and water mixture. The FeSi magnifies the gravity effect and enhances diamond and other heavy mineral separation. A large magnet recovers the FeSi, which is recycled. Water is also recycled. The less dense waste kimberlite fraction is ejected from the plant to the processed kimberlite containment area (tailings pond). The heavy mineral concentrate (containing diamonds, garnet, diopside, spinel and olivine) is conveyed to the recovery circuit. In the recovery building, the diamonds are separated from the waste heavy minerals using X-rays to trigger a unique characteristic of diamonds. Diamonds glow under this kind of light, and photoelectric sensors direct strategically placed air blasts to blow the diamonds off the conveyor belt into diamond collection receptacles. The diamonds are then transported to the product splitting facility in Yellowknife where they are cleaned using a strong alkaline, divided 60/40 between the joint-venture partners (Rio Tinto and Harry Winston), and undergo evaluation for government royalty purposes. Waste minerals are re-crushed or ejected to the tailings pond. The process plant is housed in a 152 meters x 40 meters x 35 meters high structure (Diavik Fact Book).

Power Plant

The site's power generation fleet consists of five Caterpillar diesel generators each with a rated capacity of 4-4 megawatt. Three engines operate at any time with one held on reserve for times of peak demand and one under routine maintenance. The building they are housed in is 60 meters x 36 meters in dimensions. Fuel is stored in six 18 million litre tanks for total capacity of over 100 million litres of fuel. The mine has three 700 horsepower oil-fired boilers on reserve that are only to be used in the event of a power failure to prevent the site buildings from freezing (Diavik Fact Book).

Shops

Most of the mechanical departments are based out of one building, 127 meters x 60 meters in dimensions. The same building also houses administration offices and the warehouse. The maintenance shops have 10 bays and can accommodate 218-tonne haul trucks. Included are a wash bay, welding bay and tire repair bay. Separate shops are also included for plate welding, machining and electrical repair. Bulk lubrication is stored in a separate building for fire protection (Diavik Fact Book).

Camp Complex

The main camp complex at Diavik was completed in January 2003. Staff accommodations are arranged into four wings and have a total capacity for 330 persons in single-rooms. The camp complex houses a cafeteria, gymnasium, running track, and squash court. At the end of one of the wings is a small trailer camp known as "E-Wing", used for temporary employees and contractors. "South Construction Camp" is a large trailer complex for contractors and contains its own cafeteria and recreation facilities. An expansion of "South Camp" was completed in the summer of 2005. (Diavik Fact Book)

The plant and main camp complex are all inter-connected with sheltered tunnels known as “Arctic Corridors”, a series of prefabricated modules that join all the major buildings at the mine. They transport all the utilities such as power, water, sewage, and communication, together with supplying heated and safe walkways for workers commuting to work from the camp complex (Diavik Fact Book).

Water Treatment Plant

Commissioned in March 2002, the water treatment plant is a state-of-the-art facility that treats silty water generated by mine operations. Through a system of sumps, piping, storage ponds, and reservoirs, Diavik collects runoff water which can be reused in processing or cleaned in the water treatment plant before release back to Lac de Gras (Diavik Fact Book).



Diavik Diamond Mines Inc.

Figure 4. *Aerial view of the A-154 open pit and the water retention dikes in Lac de Gras, 2003.*

Airstrip

At 1600 meters long, the airstrip is capable of accepting Boeing 737 jet service and Hercules transport aircraft (Diavik Fact Book).

Exploration 2003-2004

Bulk sampling of the A-154 North pipe during the summer of 2003 revealed higher quality diamonds than previously forecast. 19,342 tonnes were bulk sampled to produce 11,771 carats of diamonds (News/North, August 6th 2003). L.D.C. reverse circulation drilling in 2004 on the deposit extracted an 853 tonne sample producing 2,109 carats. This, plus the increase in diamond grades in mining operations, gave consideration to a possible change in mine plan that included underground mining sooner than planned. It also spurred further exploration of the Diavik property and evaluation of other kimberlite pipes. In 2004 two additional pipes were located, one of which is located 4 km east of the minesite (Diavik Diamond Mines Inc. Press Release, July 27th 2004).

2004 Operations

Strong demand of Diavik’s valuable diamonds caused the company to raise its production target for 2004 to 8.2 million carats, compared to the 3.8 million produced in 2003. Three new Komatsu 830E haul trucks were ordered for winter delivery 2004. A new wing was also added to the accommodation complex in anticipation of an increase in workforce. The open pit was extended to develop the A-154 North pipe resource and mining of the deposit was underway. Workforce at this time was 730 persons (Diavik Diamond Mines Inc. Press Release, July 27th 2004). Engineering planning was begun on the construction of the A-418 pipe dike with plans to put the pit into production

Personnel

Total workforce reported in the first quarter of 2005 was 750, of which 521 were northern and 279 were aboriginal. Twenty-two workers were involved in the new A-418 dike and underground projects.

The mine experienced lower production in the 3rd quarter of 2005 as a result of the mining of lower-grade portions of the A-154 North pipe and smaller amounts of high-grade ore from the A-154 South pipe. The A-418 open pit dike was closed off in October 2005 and work during 2006 will make the dike impermeable with the plan to begin dewatering of this section at the end of 2006 (Aber Diamond Corp. Ltd. Press Release, Dec. 2nd 2005). The A-154 and A-418 underground ramp was excavated a total of 800 meters to year-end 2005 in preparation for initiating the final feasibility study for underground mining.

A-21 Exploration

In 2005, Diavik began an exploration program to evaluate the A-21 kimberlite pipe. A decline was advanced 200 meters by year-end 2005 in preparation for an underground bulk-sampling program; by year-end 2006, the decline had advanced 1,300 meters can the mine was planning a 10,000 tonne bulk sample. The bulk sample was later completed but as of 2009 the A-21 deposit is not part of the mine plan and will be further evaluated.

<i>Deposit</i>	<i>Proven</i>		<i>Probable</i>		<i>Proven and Probable</i>	
	<i>Tonnes</i>	<i>Carats</i>	<i>Tonnes</i>	<i>Carats</i>	<i>Tonnes</i>	<i>Carats</i>
<i>A-154 South</i>						
<i>Open Pit</i>	1.2	7.1	0.7	5.0	1.9	12.2
<i>Underground</i>	-	-	3.0	14.8	3.0	14.8
<i>A-154 North</i>						
<i>Open Pit</i>	0.1	0.3	-	-	0.1	0.3
<i>Underground</i>	2.8	6.3	5.9	12.7	8.7	19.0
<i>A-418</i>						
<i>Open Pit</i>	4.3	14.6	-	-	4.3	14.6
<i>Underground</i>	0.5	2.1	3.5	14.2	3.9	16.3
<i>Total</i>	9.0	30.3	13.0	46.7	21.9	77.1
<i>Open Pit</i>	5.7	20.0	0.7	5.0	6.3	27.0
<i>Underground</i>	3.3	8.4	12.3	41.7	15.6	50.1

Table 2. Diavik Mine ore reserves at December 31st 2007. (source: Harry Winston Diamond Corp Annual Report, 2008)

(*) Tonnes and carats are in million units. Total reserves includes “Indicated” and “Inferred” resource categories. A-21 resource is based only on “Indicated” and “Inferred” resource categories. Underground reserves make up 52% of the tonnage and 42% of the carats.

2006 Operations

Unseasonably warm temperatures in the early winter of 2005-2006 led to a very short ice road season, and the diamond mines in the north were unable to move all of their material from Yellowknife to Lac de Gras. Diavik was particularly innovative as they had to bring in a large Terex RH200 production shovel to replace one of the original Hitachi EX-3600 shovels destroyed by fire in December 2005. To move this important piece of machinery, and other loads, to Diavik, the company commissioned a Russian Mi-26 helicopter to airlift the various components. Other materials were flown using 737 jets and a Russian Antonov AN12 aircraft. Several million litres of fuel were flown in

using three commercial Hercules transports. This three-phase airlift program successfully concluded in January 2007 in time for the start of the next winter road season.

Despite these logistical challenges, the Diavik Mine produced a record 9.8 million carats in 2006, an 18% increase over 2005 production. Production was from the A-154 North and South pipes. Dewatering of the A-418 pit commenced in the summer of 2006. The underground development to the A-154 and A-418 kimberlite pipes continued during the year and a feasibility study was underway. Mining tests using continuous mining technology was successfully completed at A-418 pipe while production headings continued to the A-154 pipes. At the A-21 pipe on the south side of East Island, a decline ramp reached the kimberlite and a bulk sample was expected in 2007. By the end of September 2006, the A154 tunnel had advanced 1,400 meters, and the branch tunnel to the A418 pipe had advanced an additional 870 meters (Diavik Dialogue Newsletters, 2006).

2007 Operations

Diamond production in 2007 marked another great record when nearly 12 million carats were recovered at the Diavik Mine. The winter road season of 2007 was successful and over 10,700 loads (327,000 tonnes) were moved to the four diamond mines, Ekati, Diavik, Jericho (Nunavut), and Snap Lake. Overburden removal at the now dewatered A418 open pit area commenced in the summer of 2007. Late in the year the owners of the mine, Rio Tinto and Harry Winston, announced approval to bring an underground mine to full production, representing a US \$563 million investment, bringing total underground mine capital investment to US \$787 million. Considerable construction was planned including new fuel storage, new crusher, paste backfill plant, new power generators, and expanded camp facilities. Under the current life of mine plan, underground diamond production will begin early in 2010 and continue beyond 2020. Open pit mining is expected to cease in 2012 (Diavik Dialogue Newsletters, 2007).

Year:	Ore Milled:	Ore Grade:	Diamonds Produced:
2003 *	1,175,000 tonnes	3.21 carats/tonne	3,833,000 carats
2004	1,950,000 tonnes	3.88 carats/tonne	7,575,000 carats
2005	2,200,000 tonnes	3.72 carats/tonne	8,270,000 carats
2006	2,332,000 tonnes	4.21 carats/tonne	9,840,000 carats
2007	2,400,000 tonnes	4.97 carats/tonne	11,910,000 carats
2008	2,414,000 tonnes	3.82 carats/tonne	9,225,000 carats

Table 3. Diavik Mine production, 2003-2008. (source: Roscoe Postle Associates Inc., 2005; Diavik press releases; Aber Diamond Corp Ltd. press releases; Harry Winston Diamond Corp. Ltd. press releases)

(*) includes minor production during November and December 2002

(40% production owned by Harry Winston Diamond Corporation Limited and 60% production owned by Rio Tinto Plc)

2008 Operations

For 2008, Diavik open-pit mined ore from all three kimberlites, the majority from the A154 South pipe. By mid-year, open-pit mining of the A154 North pipe concluded. As the year progressed, ore mined from the A418 pipe, in the newer A418 open pit, increased to match ore mined from the A154 South pipe, in the well advanced A154 open pit. Ore amounts mined from the three pipes reflect that the new open pit is advancing and that, overall, Diavik is moving closer to its underground mining phase, which is expected to begin in early 2010. In total, Diavik produced 9.2 million carats in 2008. In the latter months of 2008, the global economic situation became volatile and, in December, to lower costs Diavik announced its initial response to the global economic crisis which included deferring underground production (ultimately the 2009 announcement would defer underground startup until early 2010), placing its small diamond recovery project on hold, and a reduction in exploration (Diavik Dialogue Newsletters, 2008).

Proven and probable ore reserves at year end 2008 stand at 20 million tonnes grading 3.1 carats per tonne. The A-21 pipe is not included in the current mine plan as it has not yet been upgraded to reserves status.

2009 Operations

In early 2009, volatile market conditions continued and in March, Diavik would announce further actions to lower costs and reduce diamond production, including a summer 2009 and a winter 2009-2010 production shutdown, each scheduled for six weeks. The summer shutdown concluded successfully and safely in August but by September the market had recovered sufficiently enough to allow Diavik to cancel the planned winter shutdown. In terms of operations, open pit mining of the A154 South and A418 pipes continues. By September, surface and underground works associated with the underground mine were well advanced in preparation for the early 2010 start of diamond production from the underground mine (Diavik Dialogue Newsletters, 2009).

Exploration Since Mine Closure

Not applicable.

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DISCOVERY

Producer (Under Remediation)

Gold

63° 11' 15" 113° 53' 45"

Years of Primary Development: 1946-1969

Mine Development: 4,060' vertical shaft, 27 levels (50,473' dev.)

Years of Production: 1950-1969

Mine Production: 1,044,450 tons milled = 1,031,575 oz Au

Introduction

The Discovery Mine and townsite (**Figure 1**) is a substantial abandoned property located 84 kilometers north of Yellowknife, NWT. The mine produced gold between 1950 and 1969, producing one million ounces from one million tons of ore. It has been visited numerous times over the years by the author. The old townsite and mine buildings were demolished in the summer of 2005 during a cleanup project managed by D.I.A.N.D.



Ryan Silke

Figure 1. *Discovery Mine in 2000, destroyed in government cleanup summer 2005.*

Brief History

The 'Lux' claims were staked in 1944 by Alfred V. Giauque and sons after the discovery of high-grade gold in the Giauque Lake area. The property was acquired by Toronto promoters and Discovery Yellowknife Mines Limited was formed in February 1945¹. The Byrne family acquired control of the company soon thereafter. A surface exploration program commenced and two favourable gold zones were discovered during the year – the North and West zones. Diamond drilling commenced in 1946, and the results of this work prompted the company to initiate shaft sinking and underground development. Gold production commenced in 1950. Production and development continued throughout the 1950s and a large community was established. The mine reached a depth of 4,000 feet in 1960 and produced at a rate of 250 tons per day. Economic ore reserves were depleted in 1969 and the mine and village closed.

Geology and Ore Deposits²

The showing area is underlain by metasediments and volcanics of the Archean Yellowknife Supergroup. The showing lies within the Yellowknife Metasedimentary Basin, a vast area underlain largely by Burwash Formation greywacke-argillite turbidites belonging to the Yellowknife Supergroup. These were intricately folded and faulted during at least two phases of Archean deformation, and metamorphosed to lower amphibolite grade in a 40 kilometer wide, north trending zone. The Burwash sediments were invaded by plutons, plugs and stocks of the Prosperous granite. These discordant and predominantly north-south elongate bodies lie within the lower amphibolite grade metamorphic zone.

¹ The company reorganized as Consolidated Discovery Yellowknife Mines in 1954, and then as Discovery Mines Limited in 1964.

² Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

The largest and most continuous vein is the North zone (#1 vein) deposit. This vein is hosted by sediments a few hundred feet east-northeast of the pinch out of volcanic rocks. The vein occurs in the form of an anticline, the axis of which plunges 80° north. The limbs strike south and dip west, with the western limb dipping 75° and the eastern limb dipping 85°. The outcrop is U-shaped, open to the south with the limbs approximately 70 feet apart. The wall rocks that host the vein are fine-grained schistose greywackes, argillites and phyllites. In places, especially on the west limb of the vein structure, numerous veinlets branch from the main vein and extend as far as several feet into the adjacent rock. The vein is generally sinuous or intricately contorted, but on the east limb of the structure, it is nearly straight. Most of the quartz is medium grained, and banded in shades of grey. The North zone has biotite and sulphides including pyrite, pyrrhotite, chalcopyrite, arsenopyrite, pentlandite, magnetite, hematite, ilmenite, gold, feldspar and a carbonate mineral. The #1 vein on the 125-foot level, is reported to be 93 feet long and average 5.5 to 6 feet wide and have a grade between 1.21 to 1.37 ounces per ton gold.

The West zone occurs 350 feet southwest of the North zone and is composed of scattered small quartz bodies cutting host volcanic rocks. The volcanic rocks are recrystallized to a massive to gneissic quartz-feldspar-hornblende rock which is in part garnetiferous. The zone has been explored by trenching and drilling on surface and development work underground. The quartz is both black and milky in colour and mottled or banded. The vein typically carries a little gold, pyrite, pyrrhotite, chalcopyrite, sphalerite and white feldspar. An orebody was outlined by Discovery Mine and reported to contain 32,000 tons at a grade of 0.37 ounces per ton gold from the 125-foot to the 365-foot levels. The South zone occurs at the contact between volcanic rocks and argillite approximately 700 feet southwest of the North vein. The zone was explored on surface with trenching and diamond drilling with encouraging results. Development underground tested the zone on the 125-foot level and indicated a grade less than 0.30 ounces per ton gold over an average width of 36 inches.

Discovery Yellowknife Mines Limited (1946-1954)

The results of the 1945-1946 diamond drilling program, in which over 21,000 feet of drilling was completed, showed such promise to Discovery project managers that immediate plans for a production shaft on the North zone were put into play. Equipment and supplies were ordered in June 1946 for immediate delivery to the claims. Later that summer, the Canadian government lowered the value of gold from \$38 per ounce to \$35 per ounce to help restore the value of the Canadian dollar. This move was damaging to the post-war gold mining industry in Canada and specifically to developing gold projects like Discovery who relied on market trading to raise money, but the high-grade nature of the claims combined with company tenacity and cooperation from underwriters allowed operations to continue with a modified budget (Byrne, 1971).

In November 1946, shaft sinking got underway using a small portable Holman air and hoist plant that was flown to Giauque Lake by Canso floatplane. A winter road was cleared from Yellowknife to the mine site in January 1947 (via Duncan Lake) and 500 tons of freight arrived, including the permanent mining plant (see below). A permanent camp for 60 men was built and other necessary mine buildings were erected during 1946-1947, including a 75 foot timber headframe, powerhouse, blacksmith and machine shop, carpenter shop, combined office/warehouse, assay office, bunkhouse, boiler house, and cookery. Forty men were employed in the summer of 1947. Tom D. Anderson was in charge as mine manager and Norm W. Byrne was consulting engineer. Other staff included Robert Fee, assistant manager; M.J. Piloski, geologist; Ed Drummond, accountant; Ed Wilcox, warehouseman; Robert Lindgren, surface foreman; and Bob Maguire, expeditor (Lord, 1951: *The Western Miner*, Nov. 1947).

By the end of September 1947, the shaft was 275 feet deep with levels stationed at 125-foot and 250-foot depths. The shaft had 3 compartments, with balance hoisting in two compartments using two mine cages. Openings on both levels totaled 1,287 feet of drifts and crosscuts. This work was confined largely to the North zone where the most important deposit discovered to date was the #1 vein. Of the 456 feet of drifting accomplished in the North zone on the 1st level, 285 feet (62%) was in ore grading 1 ounces per ton gold. Similar results were encountered on the 2nd level. On the 1st level, a 240-foot crosscut extended into the West zone (Discovery Yellowknife Mines Ltd. Annual Report, 1947).

Power Plant

In 1947, a larger diesel plant arrived at property to push development into a production stage. This plant consisted of two 100 horsepower Cat D-13,000 diesel engines each operating 420 cubic feet per minute Gardner-Denver air compressors. Hoisting was achieved using an 8x6 Canadian Ingersoll-Rand 2-drum air hoist, and the camp was powered by a 15 kilowatt Cat D-3,400 diesel generator. Heat was supplied by a 40 horsepower wood-fired boiler. Permanent plant facilities installed in 1947 gave the mine a capacity to explore at 500 feet depth (Lord, 1951). Mine equipment in 1947-1950 consisted of four side-dump and ten end-dump ore cars of 20 cubic foot capacity, six Gardner-Denver leyner drills, two Ingersoll-Rand stopper drills, and a Gardner-Denver 9H mucking machine. Mine

cars were hoisted directly in the cage. Surface rolling stock consisted of an Allis-Chalmers HD-10 bulldozer and a 2-ton Chevrolet truck (Lord, 1951; *The Western Miner*, Nov. 1947).

Development during 1947-1948 suggested an ore reserve of 47,700 tons ore averaging 0.86 ounces per ton gold in the North zone, and 32,000 tons averaging 0.37 ounces per ton gold in the West zone to a depth of 375 feet. Metallurgical testing in 1947-1948 indicated that conventional cyanidation would be suitable. North zone ores responded with 93% recovery using gravity concentration and amalgamation, and 98% through cyanidation of tailings. Testing of West zone ores under similar methods saw recoveries only slightly lower due to ore grades (*Discovery Yellowknife Mines Ltd. Annual Report*, 1948).

Ordering the Mill

These ore reserves would only supply a 100 tons per day mill for two years, but the company decided to order a milling plant in hopes of developing additional reserves at depth. Negotiations were completed for a milling plant to be delivered to the property in the fall of 1948. This mill was ordered from the United States, but while being barged across Great Slave Lake to Yellowknife, the barge sank during a night storm. Development was now under immense pressure due to the gold marketing crisis and the high costs of transportation. Undaunted by this situation however, the Discovery company quickly ordered a replacement mill from British Columbia, Canada (Byrne, 1971).

Also during 1948, a new winter truck road was established between Yellowknife and Discovery. This route followed the Yellowknife River valley where several other potential mines were being prospected and developed. For this reason, the Federal government provided financial assistance towards winter road building (Byrne, 1948).

A production objective for January 1950 was met after an 8-month delay. Deferring underground development until gold was being produced reduced the cost involved in getting the mine operational. At year end 1949, underground development on the 1st and 2nd levels consisted of 1,497 feet of drifting, 764 feet of crosscutting, 622 feet of raising, 11,694 feet of diamond drilling, and almost 3,000 tons of slashing. Mining operations during 1949 consisted of two stopes in the North zone on the 1st level and two stopes on the 2nd level. Stope preparations were also started in the West zone on the 2nd level. A surface ore stockpile amounted to 4,301 tons averaging 0.50 ounces per ton gold. Stopes were mined using shrinkage methods and ore was hand-trammed from the workings (*Discovery Yellowknife Mines Ltd. Annual Report*, 1949).

Production Begins

The tune-up period for the mill commenced January 1st 1950; regular mill feed was initiated January 13th 1950 and the first gold bar was poured in early February 1950 (Byrne, 1950).

Mill Description 1950-1953

The mill was based on amalgamation and cyanidation as the Discovery ores were free milling. In 1950, the plant had a capacity to produce 100 tons per day and was powered by a 250 hp Buckeye diesel generator of 250 kilowatt. Ore was hoisted through the shaft in mine cars. Hand sorting of mine-run ore was initiated in 1953 to help eliminate waste rock from the mill feed. Originally, crushing was done in one stage using a 10-inch x 20-inch Traylor jaw crusher with manganese plates to reduce ore to one inch size. Single-stage crushing restricted mill operations to a rate of 90 tons per day. Grinding was performed in a 6-foot x 5-foot Traylor ball mill in closed circuit with a 4 foot Dorr Simplex classifier. A Denver duplex jig was used in the grinding circuit to recover coarse gold. Product from the jigs was amalgamated, representing 60% of the gold recovered in 1950.

The overflow from the classifier was pumped into the cyanidation circuit. First stage cyanidation involved a 26-foot x 10-foot thickener, whose overflow reported to precipitation. Underflow solution was processed in two 18-foot x 14-foot agitators and then diluted with barren solution before flowing into a second 26-foot x 10-foot thickener. Underflow from this thickener was discharged directly in an 8-foot x 8-foot Oliver filter where the re-pulped filter cake was discharge from the plant as waste.

Overflow from the thickener tanks was clarified and then precipitated through a Merrill-Crowe press unit. Gold-bearing precipitate and retorted amalgam sponge from the amalgamation process was fed into a Rockwell oil-fired bullion furnace from which gold bars were poured (Byrne, 1950).

In 1953, a two foot Symons cone crusher was installed for secondary crushing, thereby increasing the daily rate of the milling plant. With standard cyanidation practice the gold recovery was 96% in 1950. In 1951, a blanket table was added to the gravity process and a third agitator was installed in the cyanidation circuit, increasing gold recovery to

97%. Tailings were discharged to a marshy area south of the mine site via an 800 foot trestle pipeline (Discovery Yellowknife Mines Ltd. Annual Reports, 1951 and 1953).

During the January 1950 tune-up period, batches of ore from the North and West zones were processed for testing purposes. The North zone test consisted of 680 tons averaging 1.046 ounces per ton gold and the West zone test consisted of 190 tons averaging 0.436 ounces per ton gold. These tests confirmed the grade estimates of pre-production.

New mine equipment was purchased during 1950-1951, including a Mancha "Little Trammer" locomotive, Eimco 12-B mucking machine, and jumbo drill. Construction in 1950 included two residences and a new bunkhouse.



Geological Survey of Canada

Figure 2. *Discovery Mine, 1950.*

Employees and Camp

In 1949-1950, Discovery Mine employed about 60 to 70 men. Men were housed in two bunkhouses each of 35 man capacity. Single-staff members were housed above the combined office/warehouse structure. Six residences were built for staff members with families. The cookhouse had capacity for 65 persons and quarters for a staff of four. Staff in 1949 included: Tom W. Anderson, mine manager; James E. Bryans, mill superintendent; Ed Drummond, accountant; Dan McDonald, mine captain; Don Broughton, mine engineer; Jim Brown, master mechanic; and Ken McGinley, assayer (Byrne, 1950).

Development Program

The shaft was sunk to the 375-foot (3rd) level during 1950. At this depth it became apparent that a fault (Discovery Fault) had displaced the orebody by an unknown distance. Water seepage into the mine was also becoming excessive. The fault was identified by mine geologists as a low-angle plunging post-ore fault with the ore zones being displaced southward. After extensive geological assessment and underground diamond drilling, the ore zones were picked up 240 feet south of the shaft. The ore zones also developed a northward dip as the orebody grew deeper, bringing the mining area closer to the shaft at depth (Discovery Yellowknife Mines Ltd. Annual Report, 1950; Byrne, 1971).

Shaft sinking continued in 1951 with the hope of picking up the orebodies below the Discovery Fault. The 500-foot (4th) level was reached in January 1951, which was immediately below the Discovery Fault and confirmed the 240-foot displacement indicated by diamond drilling. The North and West zones were developed at this level with fair results. In April 1951, shaft sinking resumed and three additional levels were opened up at 650-, 800-, and 950-foot depths (5th, 6th, and 7th levels). Although the mine was a high-grade operation, the future of Discovery depended on the depth of the deposit. Expert geologists suggested that although the deposit was small in dimensions, it had good structure and potential to 2,000 feet depth.

A new vein, the #4, was discovered on the 950-foot level during 1951 while crosscutting within the North zone. This discovery indicated the depth potential of the mine and also bolstered additional lateral exploration programs. The North zone structure and the #1 vein at depth was strong, with very rich orebodies averaging 1.50 ounces per ton gold and widths of 8 feet for lengths of 330 feet on the 7th level. Construction during 1951 included a machine shop, residence, and a recreation hall/school/commissary/curling rink structure. (Discovery Yellowknife Mines Ltd. Annual Report, 1951).

Expanded Power Plant

The 1951 depth development program required an increased plant capacity, necessitating the installation of a 150 horsepower Ruston diesel engine operating a Canadian Ingersoll-Rand 600 cubic feet per minute air compressor and a 340 horsepower Paxman-Ricardo diesel engine powering a 200 kilowatt electric generator. To hoist to a depth of over 500 feet, a Canadian Ingersoll-Rand 48 inch x 36 inch two-drum electric hoist was purchased.

Hydro Power

In 1952, the company decided to build a hydropower line from the Bluefish Lake powerplant owned by Con Mine's Cominco Limited. A transmission line was constructed during the winter of 1952-1953 by contractor Lanky Muyres, and the power was turned on April 1953. The new hydropower electricity saved the mine the cost of installing new diesel engines to take care of increased power needs and also the cost of fuel and its transportation to the property. A 250 kilowatt Babcock-Wilcox electric boiler was introduced to Discovery to replace the old wood-fired boilers (Discovery Yellowknife Mines Ltd. Annual Report, 1953).

Discovery Townsite 1950s

The original camp built in 1946 was constantly expanded to provide accommodation for a work crew of close to 100 by 1952. Along with being a mining operation, Discovery also became a village for the workers and their families. By 1953, seven small dwellings were built, along with a duplex residence, two larger houses, and the mine manager's house. Mine manager of Discovery between 1946 and 1953 was Tom W. Anderson. The number of families living at Discovery was 12 in 1954 and 23 in 1959, with the company operating a small schoolhouse (5 pupils, 1951), recreation hall, curling rink, ice-rink, and other facilities for the citizens (swimming pool built 1956).

Mine Development and Operations 1950s

Mining above the 950-foot level was essentially completed during 1953 with a large amount of broken ore reserves ready for hoisting. Mine development initiated on the new 1,100-, 1,250-, and 1,400-foot levels (8th, 9th, 10th) following the completion of 1952 shaft sinking. Shrinkage stopes were started within the North zone's #1 vein on the 8th and 9th levels, and preparations for cut-and-fill stopes were completed within the #4 vein on the 8th, 9th, and 10th levels. Shaft sinking continued starting in November 1953. New levels were opened at 1,550-, 1,700-, 1,850-, and 2,000-foot depths by June 1954 with development focused on the #1 and #4 veins. Mining operations in 1954 were confined to the block of ore between the 7th and 10th levels through shrinkage stoping methods (Discovery Yellowknife Mines Ltd. Annual Report, 1953).

Consolidated Discovery Yellowknife Mines Limited (1954-1964)

Discovery Yellowknife Mines Limited reorganized as Consolidated Discovery Yellowknife Mines Limited in 1954. During 1954, production increased by 50% due to a 13% increase in tons milled and because of excellent ore encountered in development of the 11th and 12th levels. Waste picking in the crushing plant was instituted late in 1953 and increased mill capacity by 7% through sorting of over 2,400 tons of waste rock in 1954. Operating costs were reduced from \$23 per ton milled in 1953 to \$21 per ton milled in 1954, against a gross profit of \$51.43 per ton milled in gold revenue in 1954. Gold recovery was at an all-time high of 97.8% and mill-head grades were 1.54 ounces per ton gold during 1954. The mill operated at 99% of the year. These great productive records helped score Discovery Mine as one of the most profitable and richest gold mines in Canada during the 1950s. Construction during 1954 included the erection of four new residences, a new boiler building to replace the one that burned down, and enlargements to the miner's dry building. Preparations were also made to the foundation for the new hoist, which was installed in 1955. R.J. (Bob) Kilgour became the new mine manager in 1954 (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1954).

Concern over increased dilution in ores within the North zone led to the gradual switch from shrinkage stoping methods to cut-and-fill stoping below the 1,400-foot (10th) level starting in 1955. This was projected to increase operating costs. The #4 vein was also being mined by cut-and-fill methods. Lateral work in 1955 was concentrated on the 13th and 14th levels within the #1 vein and #4 vein. Mining was extended to the 1,550-foot (11th) level and stope preparations were carried out on 12th, 13th, and 14th levels. The mine witnessed some favourable changing geological conditions at depth. A large quartz mass which first appeared on the 1,400-foot level in 1954 resulted in a change in vein structure on deeper levels. Grade of ore and tonnage per vertical foot increased in 1955, and new productive vein structures were outlined during lateral development and diamond drilling.

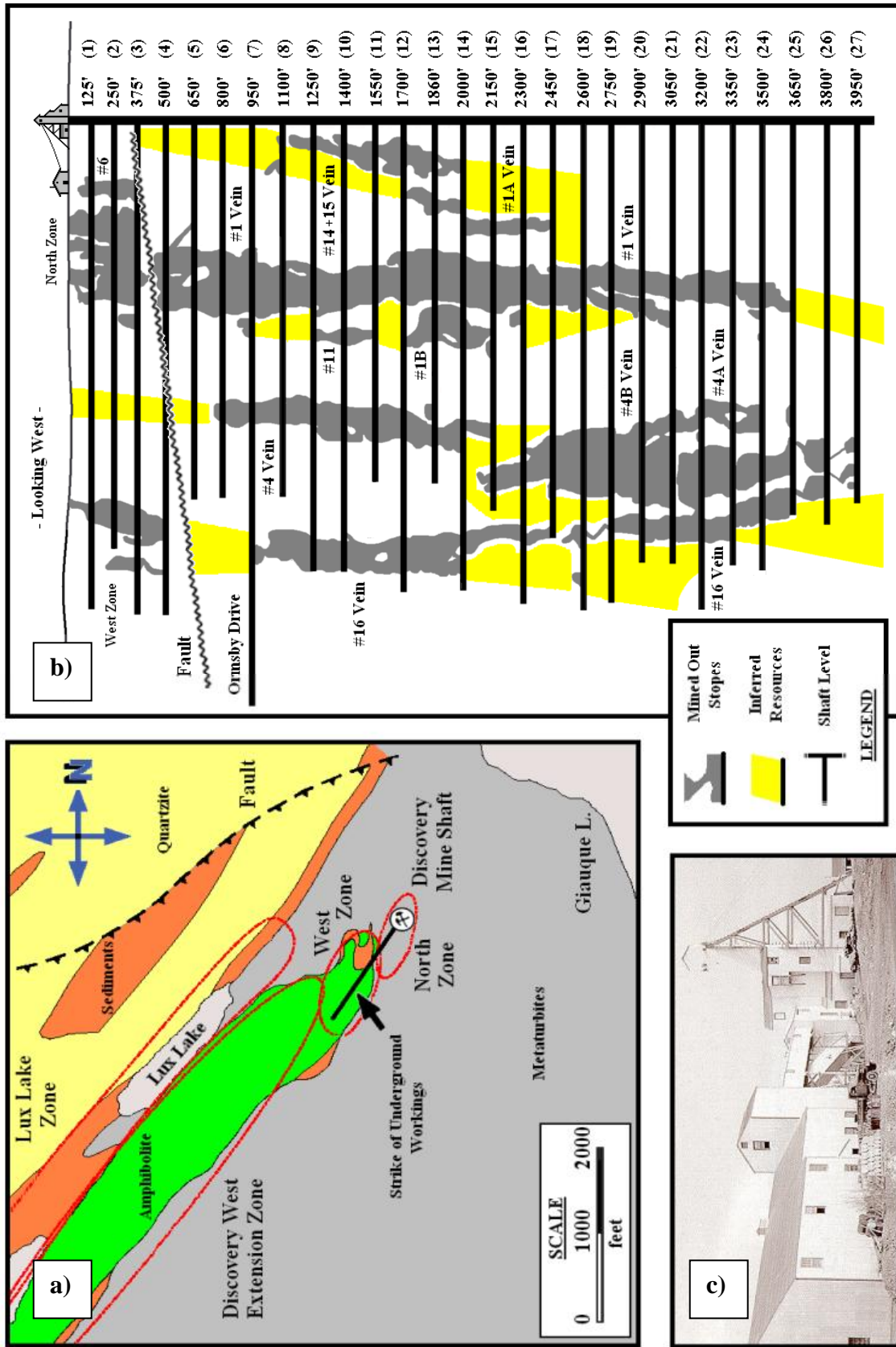


Figure 3. a) Discovery Mine surface geology. b) Underground longitudinal section, c.1968. Inferred resources are based on Nass (2001). c) Discovery mine headframe and mill c.1953. (Henry Busse - NWT Archives Photo – N-1979-052-1896)

This included the #15 vein, which is a narrow, high-grade vein angling northeast from the nose of the #1 vein fold, and was drifted upon on the 1,700- and 1,850-foot levels in 1955. These new sources of ore were scheduled for mining operations upon activation of increased milling rates in 1956. The North zone continued as a strong gold resource where drifting opened up 365 feet of continuous ore averaging 11.5 feet wide, grading 1.78 ounces per ton gold within the #1 vein on the 1,850-foot (13th) level. In May 1955, a new hoist capable of hoisting to a depth to 4,000 feet was installed at Discovery Mine. This was a Canadian Ingersoll-Rand 72 inch x 69 inch 2-drum electric hoist operated by a 400 hp General-Electric motor. A new metal building was erected to house the power plant and hoist facilities in 1955. A new 510 kilowatt electric boiler was also purchased and installed during the year. Other construction in 1955 consisted of three new residences, new refinery, additions to other residences and additions to #3 bunkhouse and the recreation hall (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1955).

Ormsby Option

In 1954, Discovery acquired a substantial share interest in Ormsby Mines Limited, owners of the LaSalle property adjacent to the south boundary of the Discovery Mine claims. A drift drive from the 950-foot level southward was extended over 2,400 feet into the Ormsby property boundaries during 1955-1956. A few flat diamond drill holes were drilled from the end of this drift to the west and southwest to test the sedimentary and volcanic rocks at the north end of the Ormsby claims, but no significant ore zones were encountered.

Operations in 1956 continued at an exceptional rate. A mill expansion program was completed in August 1956 through the addition of a secondary ball mill and expanded cyanidation circuit increasing capacity to 150 tons per day. Hand sorting of waste ores ceased in 1956. Gold recovery rates climbed to 98.4%. The shaft was further deepened in 1956 to 2,786 feet depth, with new level stations cut at 2,150-, 2,300-, 2,450-, 2,600-, and 2,750-foot depths. A loading pocket was cut at 2,660 feet depth. Ore skips were installed in the shaft compartments to improve hoisting efficiency.

Mining operations in 1956 was carried out between the 1,100- (8th) and 1,850-foot (13th) levels. The gradual changeover from shrinkage to cut-and-fill stoping methods was continued. A large block of ore within the #1 vein was mined during the year between the 12th and 13th levels of remarkable grade and dimensions, estimated to ultimately produce 90,000 ounces of gold. At year-end 1956, ore reserves stood at 149,332 tons grading 1.69 ounces per ton gold above the 2,150-foot level. Ore reserves were not as high in 1956 as in previous years because of the limited amount of development work performed due to the focus on shaft sinking (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1956).

Construction in 1956 included mill expansion and additions, a new blacksmith shop, and a swimming pool built into the tailings pond. In 1957, construction consisted of an addition to #2 bunkhouse. C.H. McDonald was hired as mine manager in 1956. R.J. (Bob) Kilgour was promoted to general manager to oversee all operations of the Discovery Mines company, although he was usually stationed at the mine site. C.H. McDonald remained as mine manager until 1964 when he was transferred to the Yukon and Bob Kilgour absorbed his duties. (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Reports, 1956-1957)

Milling Operations 1956-1966

The mill was expanded to produce 150 tons per day in 1956 through the addition of secondary grinding facilities. Ore was hoisted up the shaft in skip conveyance and dumped into a 200-ton coarse ore bin. Hand sorting of ore to eliminate waste ceased in 1956. Crushing was conducted in a two-stage closed circuit using a 10 inch x 20 inch Traylor jaw crusher and a 2 foot Symons cone crusher to reduce ore to -1/2 inch. Crushed product was conveyed into the mill building and stored in a 200-ton fine ore bin. First stage grinding was accomplished in a 6 foot x 5 foot Traylor ball mill, whose product was passed through a 12 inch x 18 inch Denver duplex mineral jig to recover a gold concentrate. Jig tailings were classified in a 4 foot Dorr unit with overflow being passed over a blanket table. Concentrates from the mineral jig and the blanket table were amalgamated in a 4 foot x 3 foot amalgamation barrel and amalgam product was retorted and prepared for refining.

Underflow sands from the Dorr classifier were sent for secondary grinding in a 5 foot x 8 foot Allis-Chalmers ball mill. A 12 inch x 18 inch Denver duplex mineral jig at the end of this ball mill recovered a gold concentrate which was sent for amalgamation. Jig tailings were recycled back into the Dorr classifier.

Blanket table tailings were pumped to the cyanidation circuit. First stage cyanidation involved a 26 foot x 10 foot thickener, whose overflow solution was split into two batches: half was recycled back into the grinding circuit and the half was sent to the clarifier tanks. Underflow solution was processed in four 18 foot x 14 foot agitators and then into

a second 26 foot x 10 foot thickener. Underflow from this thickener was discharged directly in an 8 foot x 8 foot Oliver filter where the re-pulped filter cake was discharge from the plant as waste. Filtrate from the filter and all overflow solution from the second thickener was sent for clarification and precipitation in a Merrill-Crowe unit. Zinc dust was added and the solution was pressed in twelve 24 inch x 24 inch plates. Barren solution was recycled back into the cyanidation circuit. Precipitate was combined with the amalgam sponge and refined in a Rockwell oil-fired bullion furnace, where gold bars were poured (mine records).

In 1957-1958, old mill tailings jettisoned south of the mine were cleared for use as an airstrip, locally named "Little La Guardia". Size was 3,500 feet long by 150 feet wide. The first plane to land was a Bristol air freighter in December 1957. It was originally used for winter use only, but it was upgraded to full season use in 1960.

Mining and development continued routine during 1957. The 500th gold brick was poured at Discovery Mine on August 17th 1957. In 1958, the new #16 vein was located 500 feet south of the workings. It was found by diamond drilling on the 1,700-foot (12th) level and visible gold was evident in the samples. Drilling above and below the level showed vertical continuity and the ore-making potential was regarded as excellent. The shaft was sunk to 3,471 feet depth in 1958 and four new levels were established. Proven ore reserves at December 31st 1958 were 135,000 tons grading about 1-60 ounces per ton gold, enough for three years of production (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1958).



Horst Nendsa

Figure 4. *Discovery Mine truck, Horst Nendsa is the driver.*

Mine development was within the #1 vein on four new levels down the 3,350-foot (23rd) level in 1959. The #16 vein was also being explored on the 1,400- and 2,000-foot (10th and 14th) levels where it remained quite strong and with good concentrations of visible gold. Average grades in the #16 vein, however, were below mine average. A new vein was also encountered on the 23rd level in 1959. Mining operations were extended down to the 2,750-foot (19th) level. Production from the new #16 vein was underway by the end of the year. Minor methane problems were encountered during depth developments. Steps were taken during the year to improve mine ventilation on the lower levels and a series of ventilation raises were driven from the 23rd level to the surface. A large ventilation fan was installed on 23rd level. New safety procedures were instituted in 1960 for the inspection of headings and testing for methane gas before work is conducted. A number of casualties and some fatalities were experienced in the early 1960s due to the methane problem (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1959).

Lower level development of the North zone's #1 vein outlined the persistence of folding in the enclosing sediments where the ore zone continued in a steeply plunging structure. The ore was now occurring on the east limb of this fold rather than on the west limb of the fold as in the upper levels. Quartz vein and mineralization were similar to conditions above the 2,900-foot level. The #16 vein was a steeply plunging structure and suggested a continued and persistent vertical extent. On the 14th level, drifting on the #16 vein during 1959 showed a length of 200 feet of ore grading 0-54 ounces per ton gold across a width of 3 feet. On this level, the concentration of visible gold remained strong, but the grades were below the mine average. Ore at depth did not represent any metallurgical difficulties in the mill. Construction in 1959 consisted of a new blacksmith and steel shop, a new modern school, and enlargement of the refinery for a second bullion furnace (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1959).

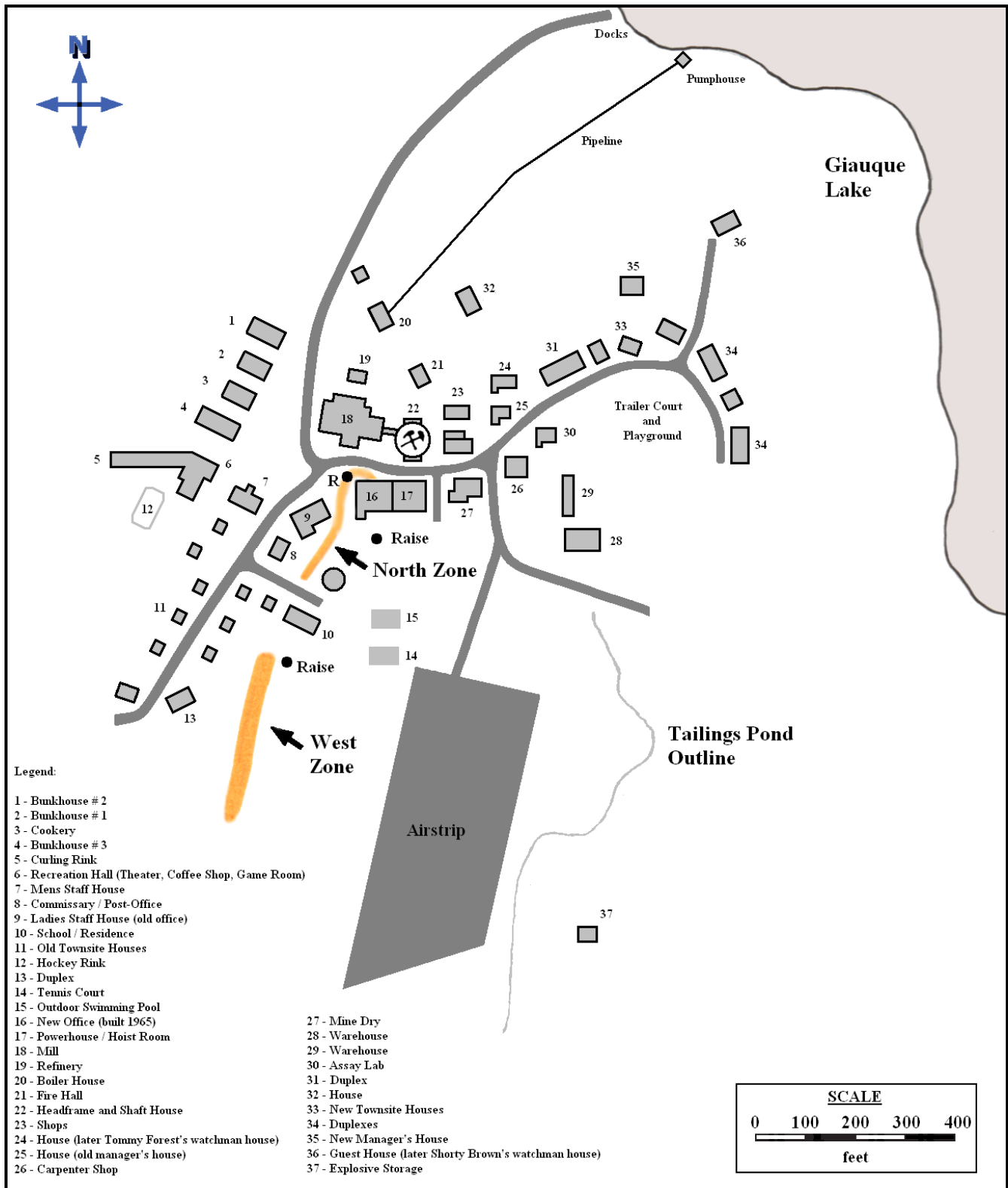


Figure 5. Map of Discovery Mine surface facilities and townsite, c.1965.

Employees 1950s

At year-end 1959, 122 people were employed at Discovery Mine. There were also 23 families living on the site. The following staff members were employed at Discovery in 1959: R.J. (Bob) Kilgour, general manager; C.H. McDonald, mine manager; B.R. Bowes, mine superintendent; C.A. Blaney, mill superintendent; K.B. Culver, geologist; R.J.

Wilson, engineer; R.F. Stedman, accountant; R.N. Lovlin, electrician; G.V. Whitaker, master mechanic; L.R. Lindgren, construction superintendent; Ray D. Falloon, surface foreman; J.C. Gibson, warehouseman; and J.W. Garner, security (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1959).

Mine Operations 1960s

In 1960, the shaft was deepened to 4,060 feet depth and four new levels were established at 3,500-, 3,650-, 3,800-, and 3,950-foot depths (24th, 25th, 26th, and 27th levels). A loading pocket was cut at the bottom level and an ore pass system was completed from the 2,750-foot level to the 3,260-foot loading pocket. Lateral development was sacrificed in favour of shaft sinking, but the limited work was concentrated in the #16 vein area on the 14th and 16th levels. By year end, crosscutting to the #1 vein was underway on the new lower levels. Stope mining was extended down to the 2,900-foot (20th) level and stope preparation to the 3,200-foot (22nd) level. Stopping also began on the #16 vein on the 1,700-foot (12th) and stope preparation on the 1,400-foot (10th) level. The #16 veins were mined using a combination of shrinkage and cut-and-fill stoping methods depending on specific vein characteristics per level. (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1960)

Power Plant 1960s

The mine was power by hydropower (1,300 kilowatt feed) from the Bluefish Lake hydro plant since 1953. Backup diesel generators were in use and were normally run at low rates to augment hydropower. These included a 340 horsepower Paxman-Ricardo diesel engine with 200 kilowatt generator and a 250 horsepower Buckeye diesel engine with 240 kilowatt generator. Compressed air was supplied with two Bellis-Morcom air compressors and two Canadian Ingersoll-Rand air compressors, all operated by electric motor. Total capacity of the compressed air plant was over 2,000 cubic feet per minute. Heat was supplied by an electric-driven 510 kilowatt General-Electric boiler, a Babcock-Wilcox low-pressure boiler, and two 40 horsepower wood-fired boilers used for standby heating (mine records).



Discovery Mines Ltd.

Figure 6. *Aerial view of Discovery Mine townsite, 1965.*

Construction in 1960 included the acquisition of four building shells hauled in from the closed Rayrock uranium mine. They were modified and put to use as residences, a female staffhouse, and a commissary/post office. A new winter road was plowed to the Discovery Mine from Yellowknife in January 1960. This road, cleared and operated by

John Dennison of Byers Transport Limited, allowed for the transport of supplies and equipment at a much lower rate than through the use of aircraft. Construction in 1961 consisted of renovations to the recreation hall and a new carpenter shop to replace the building destroyed by fire (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Reports, 1960-1961).

Development of 24th, 25th, and 27th levels was largely completed during 1961 with little significant new ore encountered. On the 24th and 25th levels, a total of 156 feet of ore was opened up. On the 27th level, no material of ore grade quality was encountered within the #1 vein. Mining operations in 1961 centered on twelve levels down from 2,000-foot depth within the #1 vein where grade yielded more than one ounce per ton gold. Ore from the #4 vein also continued to yield good gold grades. The #16 vein remained below expectations in grade delivered to the mill. A new ore reserve was calculated for the #16 vein between the 9th and 16th levels in a vertical range of 1,050 feet, suggesting 40,000 tons of developed ore grading 0.42 ounces per ton gold.

Total ore reserves at December 31st 1961 were 84,923 tons grading 0.82 ounces per ton gold, a significant drop in previous years due to the inability to locate new tonnages within the #1 vein. Diamond drilling below the 4,000-foot level within the #1 vein revealed nothing of interest. It was believed that ore within the #1 vein of the North zone had pinched out between the 3,700- and 3,900-foot horizons. The 1000th gold brick was poured at Discovery Mine on September 2nd 1961. An exploration drive was driven on the 2,600-foot (18th) level late in 1961 along the east contact of the greenstone formation south of the shaft, for the purpose of exploring the possible downward extension of a favourable zone where early surface diamond drilling had yielded scattered gold values (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1961).

Operations in 1962 assumed a retreating sequence when exploration at the bottom levels of the mine failed to locate additional resources within the #1 vein, reducing the possibilities for the North zone in terms of future ore tonnage. The mine began an extensive review of the vein deposits in the hope of locating additional veins or ore zones that may have been neglected in the history of the operation.

West Zone Development

The West zone became the target of interest in 1962 where volcanic rocks host a number of gold veins similar to the structure of the #16 vein at depth. It was a low-grade source of ore but with large enough mining widths (50 feet or more) to allow for cheap, bulk mining methods. About 4,000 tons of previously broken ore from the West zone were milled in 1962 to yield 2,000 ounces of gold. There were no metallurgical difficulties with this ore, unlike in earlier years. The #5 and #7 veins were located within the West zone.

Stopes between the 2,300- and 3,000-foot levels were mined out during 1962 and all ore was removed. Production was derived 50% from the North zone or #1 vein, 23% from the #16 vein, 7% from the West zone, 4% from development ore, and the remainder from numerous vein-zones on the periphery of the quartz mass. Stopping of the #16 vein was conducted on the 9th, 10th, 12th, 14th, and 16th levels. Stopes were also started on the new #4A vein (parallel but slightly offset to the original #4 vein) on the 21st and 23rd levels. Overall production declined in 1962 because of the greater reliance on ore from the lower grade #16 veins and because of dilution in mining narrow veins on the lower levels of the mine. Problems with high turnover amongst expert miners also affected underground productivity (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1962).

Drifting within the #4A vein on the 20th level during 1963 exposed a good width of 0.80 ounces per ton gold ore which was extended for a length of 110 feet. Milling of Camlaren Mine ore during the year allowed for a relaxation in production requirements from the Discovery Mine workings, and several new ore sources were investigated but with little success. A small ore shoot on the 3rd level was stoped and mined. Minor production also came from the #15 vein located above the 2,000-foot level. The milling rate was gradually increased to 200 tons per day during the year with the object of lowering costs per ton milled. This increase was accomplished without capital expenditures and without additional personnel. Ore reserves at December 31st 1963 were 68,330 tons grading 0.50 ounces per ton gold, enough to provide mill feed for one year of production at Discovery Mine. The future of the operation looked bleak at the end of the year (Cons. Discovery Yellowknife Gold Mines Ltd. Annual Report, 1963).

Discovery of New Vein System

The ore picture at Discovery Mine improved dramatically during 1964 as a result of new discoveries underground. During the last week of 1963, a new vein was encountered on the 2,900-foot (20th) level that at the time was seen as a good source of ore to prolong operations by another few months. Further work during 1964 revealed this as an entirely new orebody, which became known as the #4B vein. The new vein was narrower but had similar

characteristics to the high-grade North zone veins and with grades of one ounce per ton or better. Work continued throughout the year on following this vein and several parallel vein structures were identified bringing mining widths of up to 30 feet. An ore length of 420 feet was developed with an average width of 6 feet. A vertical range of 1,200 feet was proposed, with the deposit existing from the 17th level to the 24th level (Discovery Mines Ltd. Annual Report, 1964).

Discovery Mines Limited (1964-1969)

The company again underwent reorganization during 1964 through the amalgamation of Consolidated Discovery and Ormsby Mines Limited, and became known as Discovery Mines Limited. The #16 vein was encountered on the 3,200-foot (22nd) level in 1964, opening up a block of ore between the 18th and 22nd levels that was still yet to be developed. Lateral development to open up new stopes on these levels was underway during the year. Production during 1964 was derived from the following areas: #1 vein (11%), #16 vein (28%), #4 veins (38%) and West zone (20%). The West zone was a low-grade area but widths were greater and mining costs much lower than other sections of the mine. Mining of the West zone was conducted between the 1st and 4th levels (Discovery Mines Ltd. Annual Report, 1964).

Development of the #4 vein system and the #16 vein occupied the near full attention of crews during 1965. Production was mostly from these two ore zones, but minor production was also derived from the #5 (West zone), #1 and #6 veins (North zone). The #4 veins supplied about 60% of the ore milled in 1965, with the remainder from the West zone and #16 vein. Development of the #16 and #4B veins below the 2,700-foot (19th) level showed that the two veins converged into a single structure. Limited development on the bottom levels of the mine (26th and 27th) was conducted within the #4B vein. Diamond drilling intersected high-grade ore over narrow widths within the #4 vein on the 23rd to 25th levels.

The milling rate was increased during 1965 with the installation of a third ball mill (5 foot x 4 foot Denver) in late August. Capacity of the plant was increased to about 230 tons per day, which helped to offset the lower grade material being treated. Construction in 1965 included a new mine office/warehouse building attached to the west end of the powerhouse building. The site's entire pipebox network was also rebuilt. Trailer units were brought to the site during 1964-1966 to provide expanded and cheap family dwellings. An acute shortage of expert miners during 1965-1966 did not seriously affect mining operations. A large percentage of new hires during this period were fresh immigrants of German, Italian, British, and Eastern European descent (Discovery Mines Ltd. Annual Report, 1965).

In 1966, development of the #4 and #16 vein systems to the bottom level of the mine was completed. Crosscuts were driven to provide a base for a diamond drilling campaign to explore for a depth extension beyond the 4,000-foot level. Extensive exploration of the entire mine continued in search for new deposits or small pockets of un-mined ore. Zones of favourable greywacke beddings, host rock of most production at Discovery, were assessed at shallow horizons. On the 24th level, a series of gold-bearing quartz veins were intersected within fractured greywacke beds. Production was derived from the following zones in 1966: #4B (55%), #16 (30%), West zone (9%), and #3 vein (2%). The #3 vein is located in the North zone section (Discovery Mines Ltd. Annual Report, 1966).

Staff and Employees 1966

An average of 128 were employed at Discovery in 1966. Two women were also on the payroll as assistants in the assay office. The following senior staff were employed during the year: R.J. (Bob) Kilgour, general manager; L.T. (Ted) Vear, mine superintendent; J.W. Platt, geologist; D.P. Sponton, master mechanic; H. Werner, mill superintendent; E.G. Marett, chief cook; J.A. Caswell, purchaser; L.F.G. Borden, accountant; M.V. Taylor, warehouseman; D.R. (David) Crombie, chief engineer; Karl Nendsa, surface foreman; G. Erstling, mine captain; R.F. Stedman, office manager; and M.R. Gromert, security and safety manager (Discovery Mines Ltd. Annual Report, 1966).

Discovery Townsite 1960s

The Discovery Mine townsite was expanded during the 1960s. A Federal post office was opened at the mine in 1960 under the name of "Discovery, NWT". There were 37 family dwellings built by 1966, including seven trailer units. It was estimated that there were 250 people housed at the property in 1966. Townsite facilities included a large recreation hall with curling club, library, coffee shop, theater, and hockey rink, a tennis court, outdoor swimming pool, commissary/post office, hospital, small fire hall, and school (up to Grade 8). Single men were housed in three bunkhouses and ate in a modern cookery. Excellent fishing was provided on Giauque and Thistlewaite Lakes. There was a portage with rail and winch to pull boats between the two lakes. The company built a marine dock for employee yachts in 1965. Some employees built their own small summer cabins out on the lake.

Supply and Transport

The ice road, which was built into Discovery from Yellowknife in 1960, continued to operate through the remaining life of the mine. Most freight was brought to the property over this road, directly from Edmonton, Alberta. About 1,556 tons of freight was trucked to the site in 1966 on 20-ton capacity truck trailers. In the summer, access to the site was by aircraft landing on the gravel airstrip. In 1966, an average of five aircraft landed at the mine each week.

In 1967, it appeared once again as though reserves would be mined out within a short period of time. Although 30,000 tons of material was added to ore reserves at the end of the year, this ore was low grade and exploration of potential area throughout the property did not disclose any appreciable new ore. Mining and production operations continued within the #4B and #16 veins on the lower levels of the mine (17th to 27th levels). Another new vein, the #19, was found on the 3,200-foot (22nd) level, but it was found to be highly erratic in nature although it did provide a modest tonnage of additional ore feed in 1967-1968 (Discovery Mines Ltd. Annual Report, 1967).

Final Year of Operation

A milestone was reached at Discovery in April 1968 when the 1,000,000th ounce of gold was produced. Early in the year, it was announced that a salvaging program would be underway by year-end in preparation for a permanent shutdown. There were many reasons for the closure. Cost of operations was rising including the expense of maintaining a townsite. The price of gold was pegged at low value with no prospect of rising. Narrow gold veins were becoming expensive to mine and no economic reserves were found to replace the mined out deposits.

The majority of ore (70%) in 1968 was derived from the last available working areas of the #4B and #16 veins. Other ore was derived from remnant stopes in the upper levels. Underground exploration and development ceased in mid-1968 after failing to locate additional sources of ore in the mine. To December 31st 1968, a total of 1,018,786 tons of ore had been milled to produce 1,023,575 ounces of gold valued at \$36,417,000 represented by 1,542 gold bars at that date. Total underground development at year-end 1968 consisted of 4,060 feet of shaft sinking (27 levels), 18,065 feet of raises and ore passes, 50,473 feet of drifts and crosscuts, and 265,661 feet of diamond drilling (Discovery Mines Ltd. Annual Report, 1968).

Remaining ore reserves at December 31st 1968, which were reported to be low grade, would carry operations through to the end of March 1969. An additional 25,664 tons of ore was milled from January 1st to April 15th 1969, which was estimated to contain about 8,000 ounces of gold. (mine records; Mines and Mineral Activities, 1969) Total gold bar production could have been close to 1,600 bricks. **Table 1** gives total Discovery Mine production on a yearly basis. The mill accidentally burned down in May 1969, and all operations ceased (mine records).

Exploration Since Mine Closure

The property remained inactive until 1980, when Newmont Exploration of Canada Limited optioned the claims from Discovery Mines Limited. Newmont also owned surrounding claims, and during the year they mapped a 9 kilometer x 2 kilometer area around and including the old Discovery Mine. Geophysical surveys (mag, EM, IP) outlined numerous anomalies. Other work included 4 diamond drill holes testing targets in the Winter Lake area (Brophy et al., 1984). Further work was recommended to investigate the surface anomalies, but there is no other record of work done by Newmont. Canmax Resources Corporation Limited subsequently optioned the claims and conducted additional surface mapping and some diamond drilling.

In 1992, the original claims were allowed to lapse. A new claim covering the old Discovery Mine ('GMC 1') was staked in December 1992 by New Discovery Mines Limited and in 1994 a 50% interest option agreement was entered into with GMD Resources Corporation Limited. Some diamond drilling was done in 1994 to test the West zone, old stope pillars and the west limb vein in the North zone, the Lux Lake shear zone (newly discovered), and the Ormsby zone. Subsequent exploration (1995-1998) focused on the Ormsby zone.

In 1996, GMD Resources Corporation purchased the remaining 50% interest in the Discovery Mine from New Discovery Mines Limited. Ore reserves in the old Discovery Mine consist of un-mined stope pillars, broken ore within stopes, and a large mass of mineralized quartz that was inadequately sampled during mine operations. None of these reserves have been measured, and fall within the indicated and inferred categories. Indicated reserves are 221,000 tonnes grading 19-54 grams per tonne; inferred reserves (mostly within the quartz mass, see **Figure 3**) are 1,206,000 tonnes grading 27-43 grams per tonne gold. Total reserves (indicated and inferred) are 1,427,000 tonnes of ore grading 26-21 grams per tonne gold within the old Discovery Mine underground workings. In 2000, the property was acquired by Tyhee Development Corporation Limited (Naas, 2001). SEE **ORMSBY MINE**

Year:	Ore Milled:	Grade:	Gold Produced:	Recovery:
1950	29,995 tons	0.63 oz/ton	18,030 oz	96.0%
1951	31,915 tons	0.68 oz/ton	20,948 oz	96.7%
1952	33,282 tons	1.12 oz/ton	36,292 oz	97.1%
1953	33,121 tons	1.15 oz/ton	36,919 oz	97.2%
1954	37,389 tons	1.54 oz/ton	56,370 oz	97.8%
1955	38,693 tons	1.76 oz/ton	66,742 oz	98.0%
1956	42,000 tons	1.71 oz/ton	70,688 oz	98.4%
1957	51,273 tons	1.60 oz/ton	81,213 oz	98.6%
1958	47,621 tons	1.78 oz/ton	84,051 oz	99.0%
1959	51,708 tons	1.64 oz/ton	83,988 oz	99.2%
1960	51,776 tons	1.60 oz/ton	81,967 oz	99.2%
1961	55,163 tons	1.18 oz/ton	64,469 oz	99.0%
1962	53,858 tons	0.81 oz/ton	43,011 oz	98.5%
1963	47,924 tons	0.64 oz/ton	30,209 oz	98.3%
1964	77,830 tons	0.62 oz/ton	47,471 oz	98.0%
1965	80,546 tons	0.71 oz/ton	55,865 oz	98.0%
1966	82,848 tons	0.74 oz/ton	60,137 oz	98.3%
1967	85,772 tons	0.61 oz/ton	51,007 oz	98.2%
1968	86,612 tons	0.40 oz/ton	34,172 oz	97.9%
1969	25,664 tons	-	8,000 oz	-
Total:	<u>1,044,450 tons</u>	<u>1.02 oz/ton</u>	<u>1,031,575 oz</u>	<u>98.0%</u>

Table 1. Discovery Mine production, 1950-1969. (source: Discovery Mines Ltd. Annual Reports)

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DIVERSIFIED

Advanced Exploration (Non-Operational)

Gold

64° 16' 30" 115° 12' 45"

Years of Primary Development: 1946-1951, 1980s

Mine Development: 525' vertical shaft, 2 levels (3,200' dev.)

Introduction

The Diversified Mine was an underground gold prospect that operated between 1946 and 1951, and later explored in the 1980s and again during the 1990s. It is located at Indin Lake, 206 kilometers north of Yellowknife, NWT. The mine-site was visited by the author in September 2000.

History in Brief

The Indin Lake region was formerly known as the Wray Lake region in the late 1930s, when gold was first discovered near the Diversified property. Most claims in the area lapsed during World War II. The mine property was staked in 1945 as the 'Arseno' group and gold was found in many of the vein structures. Exploration work paved the way for shaft sinking during 1946 and 1947. Work continued intermittently between 1947 and 1951 with a shaft sunk and two levels opened up.



Mike Piro

Figure 1. *Diversified Mine headframe, 1982.*

Interest in the property was revitalized in the early 1980s when all new buildings were erected and the shaft de-watered and rehabilitated. Preparations for re-mining the deposit continued throughout the 1990s, but the last reported work on the property was in 1997 by Silverspar Minerals Limited. George Stephenson of Calgary is now registered owner of the mineral claims and in 2007 applied for a land use permit to conduct exploration.

Geology and Ore Deposits¹

The Indin Lake area is located within the 20 kilometer wide Indin Lake Supracrustal belt of Archean supracrustal metasedimentary and metavolcanic rocks of the Yellowknife Supergroup. All the rocks within the belt are strongly folded and there is a complicated deformation history associated with the area. Much of the region is underlain by metamorphosed greywacke-argillite turbidites. Volcanic rocks of the Indin Lake area are mainly mafic units including pillowed flows (with minor andesite) and local interflow breccias. Occurrences of quartz diorite and gabbro sills throughout the Indin Lake Belt are thought to be related to the volcanic units in the area. Local granodiorite bodies and pegmatites also intrude the belt. Diabase (to gabbro) dyke swarms occur throughout the Indin Lake area. The Diversified showings are part of a series of gold showings that occur near the north north-east-trending contact between the metavolcanics and the metasediments. The property geology comprises north north-east-trending belts of altered andesitic-dacitic volcanics and pyroclastics (or greenstone), and slate, schistose greywacke-argillite, and fine-

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

grained quartz-mica schist metasediments, which host the auriferous quartz veins. The metasediments are cut by the north-trending Leta fault (parallels the boundary between volcanics and sediments along its southern end) and the north north-west-trending Aztec and Float faults, which may join the Leta Fault just south and north, respectively, of the #1 zone. The direction and displacement of the faults is unknown but thought to be sinistral.

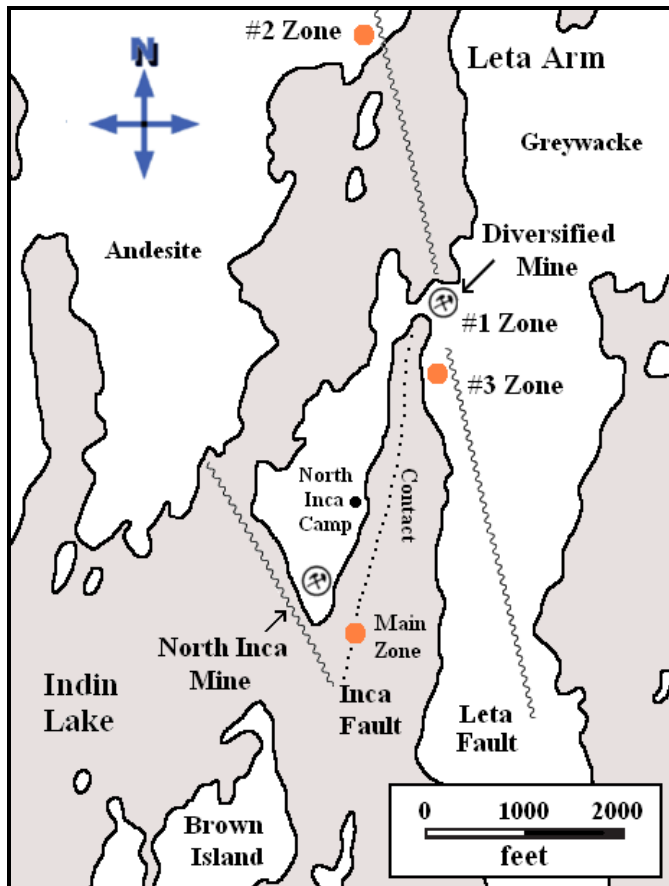


Figure 2. Location of Diversified Mine and ore deposits.

The #1 zone is located underwater in the eastern half of the Leta Arm (Indin Lake) and outcrops where the Inca Peninsula joins the mainland at the Diversified property. The #1 zone is composed of 3 parallel quartz veins (1-A, 1-B, and 1-C) within an area extending 40 meters. The three veins are composed of quartz stringers, lenses, and veins in slate (occasionally rusty and sheared), and greywacke-argillite schist. The southern exposed ends of the zones lie 23 to 61 meters east of the north-trending Leta Fault, from which the exposed veins diverge to the north north-east.

The #2 zone outcrops on the west mainland shoreline just north of the long island that divides the Leta Arm in half from north to south, about 1,065 meters north of the Diversified Mining shaft. The #2 zone consists of quartz lenses and stringers hosted in north-trending, sub-vertical shears in slate and greywacke meta-sediments. The #3 zone outcrops about 457 meters south of the Diversified shaft and terminates to the south at the shore of Indin Lake. The zone consists of quartz lenses and stringers hosted in north-trending, sub-vertical shears, within schistose greywacke-argillite meta-sediments. The #3 zone may be the displaced equivalent of the #1 zone and of the Main zone of the North Inca Mine. The #2 and #3 zones have not been explored underground.

Diversified Mineral Interests of Canada Limited (1947)

All exploration and surface development during 1945 and 1946 was focused on a small neck of land connecting the mainland to the Inca Peninsula near the north end of Indin Lake. Three veins were trenched at intervals on the surface over a length of 730 feet. Drilling from the surface in 51 holes indicated the deposit to have a minimum strike length of 1,500 feet and to extend beyond a depth of 300 feet. Plans for the construction of a mine were underway during the summer of 1946. It was intended to sink down to a first level during the first year of development, and carry the shaft to a further depth depending on results. Jim McDonald was put in charge of shaft sinking. Two other deposits were identified, but development was restricted to drilling and trenching only. The #1 zone on the Arseno #1 claim was the most accessible exploration target at this time (Lord, 1951).

1st Level Development

Shaft development began in April 1947 upon the installation and erection of plant equipment and buildings. The 3-compartment shaft was sunk to 202 feet and the 1st level established at 175 feet depth. Crosscutting to reach the 1-A vein began in July 1947, and drifting north and south for a distance of 941 feet was completed by year-end. Crosscutting consisted of 176 feet of advance to the west and raising totaled 110 feet. All development was confined to the 1-A vein zone, where to the north, gold values increased, and throughout the drifting values remained constant (Lord, 1951). The intention of the 1947 program was to sink to 500 feet depth, but conditions made it advisable to focus work on the 1st level and conduct an underground-drilling program, which totaled 1,729 feet. Less than half of the indicated structure was explored by the underground program, and it was recommended to immediately sink beyond the 1st level to explore the deposit at further depths (Diversified Mining Interests (Canada) Ltd. Annual Report, 1947).

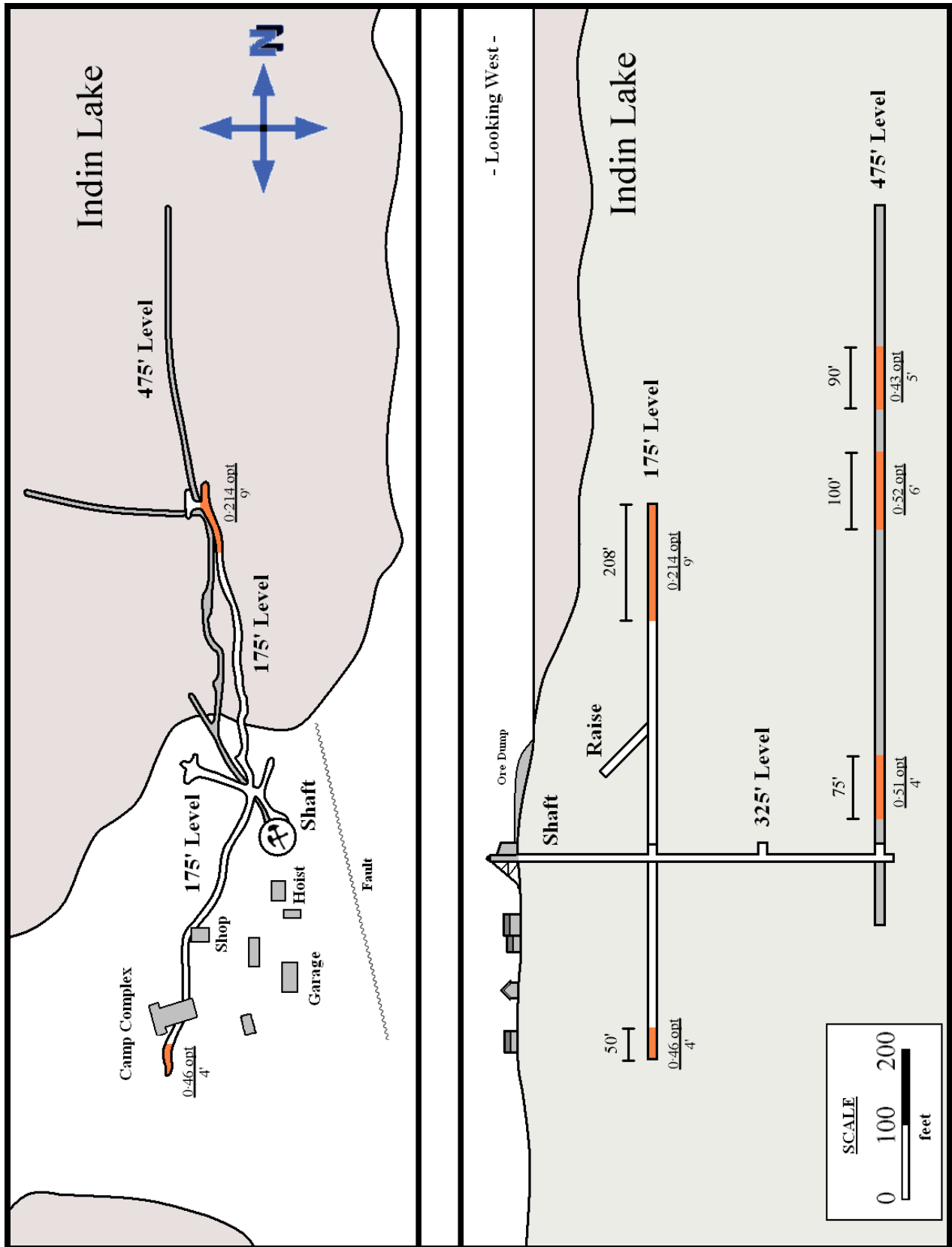


Figure 3. Diversified Mine surface and underground plan, 1990s. opt = ounce per ton

Camp and Plant Facilities

A small camp and plant was erected during the spring of 1947 at the Diversified property. A 2-storey bunkhouse and a cookery were designed for capacity of up to 50 men. A 40 foot headframe, temporary powerhouse, and temporary service shops were adequate for the 1947 program. Diesel fuel was stored in tanks of 28,000 gallon capacity (Lord, 1951). In 1947, J.R. MacDonald was manager, I. MacLean was accountant, and Jack Varty was master mechanic (The Western Miner, Nov. 1947).

Mining Plant

Two air compressors were installed in 1947. These were two 360 cubic feet per minute Canadian Ingersoll-Rand units powered by 90 horsepower Waukesha 145HK diesel engines. A small 35 kilowatt lighting plant was housed in the machine shop, and the headframe was serviced with a small 2-drum air hoist, an 8x6 Canadian Ingersoll-Rand unit. Surface development and site preparation was aided by a Cat D-4 tractor. The shaft was fitted with a bucket for handling ore, supplies, and men. All mucking was performed using a Gardner-Denver mucking machine and ore cars (Lord, 1951).

General Operations and Costs

The cost of 1947 operations for year-end totaled \$179,000. Shaft sinking was done at a contract cost of \$65 per foot. On average, 30 men were employed by the Diversified company during 1947 work. Freight was brought in by DC-3 aircraft during the winter months and by Dakota and a company owned Fairchild plane during the summer. No work other than camp maintenance and supply acquisition was done during 1948 (Lord, 1951).

Progress Diversified Minerals Limited (1949)

During 1948, the Diversified company was re-organized in preparation to re-open its Indin Lake gold property and became known as Progress Diversified Minerals Limited. Work during the previous program in 1947 did not reflect the full potential of the deposit. It was felt that work on lower levels could better sum up ore reserves and unveil a massive gold find. A crew was assembled at Indin Lake in March 1949. By August 1949, the shaft was completed to a target depth of 525 feet and two stations cut at 325 feet and 475 feet for two additional levels. Six diamond drill holes were drilled from the 325-foot level station, but since these were drilled mostly in the un-mineralized area of the shaft, little new information was secured. The operation was then again closed and property subsequently sold to Indigo Consolidated Gold Mines Limited (Campbell, 1951).

Year:	Shaft Sinking:	Drifting:	Crosscutting:	Raising:	Underground Drilling:	Surface Drilling:
1947	202'	941'	176'	110'	1,729'	?
1948	-	-	-	-	-	?
1949	323'	-	-	-	1,971'	?
1950-1951	-	1,265'	800'	20'	8,756'	?
<i>Total:</i>	<i>525'</i>	<i>2,206'</i>	<i>976'</i>	<i>130'</i>	<i>12,456'</i>	<i>22,000'</i>

Table 1. *Diversified Mine underground development summary.*

Indigo Consolidated Gold Mines Limited (1950-1951)

Officials believed that a definite ore reserve could not be calculated without conducting lateral exploration on the bottom level of the mine. In preparation for this winter program, some new buildings and equipment was erected. A permanent powerhouse/hoist room and shops were built, and a complete assay lab outfit was purchased from North Inca Mine, a neighboring operation. Also purchased was a Gardner-Denver air compressor of 365 cubic feet per minute to augment hoist capacity for deep developments (Campbell, 1951).

Work began in August 1950 under the direction of H.J. Logan, mine manager; Dr. J.F. Wright, consulting geologist; and Colin A. Campbell, consulting engineer. The 3rd level program consisted of drifting north to follow the westerly dipping 1-A veins west of the shaft. Two crosscuts were driven from the shaft, and at each end several drill holes were planned to cut down to the 1,000-foot horizon. During January 1951, lateral development was interrupted by the

failure of the mine plant, at which time diamond drilling was initiated. A total of 44 holes (3,064 feet) were drilled east and west from the crosscut to determine the width of the ore zone and to search for parallel ore bodies (Campbell, 1951).

Before May 1951, all drifting was north of the shaft. During the month, drifting south encountered varying averages such as 1.40 ounces per ton gold across 2.5 feet, 2.70 ounces per ton across one foot, and 2.40 ounces per ton across two feet. Work stopped in July 1951, at which time a total of 1,265 feet of drifting and 800 feet of crosscutting were completed on the 3rd level. In addition to this, one short 20 foot raise was put in during the final week of operation. 8,756 feet of diamond drilling was done from the 3rd level crosscuts to reach to deposit at 850 feet depth, suggesting similar structural conditions at lower depths. No work was done on the 2nd level (Campbell, 1951).

At this time, it was recommended that a larger mining plant be installed, including a 1,500 cubic feet per minute compressor and large electric hoist, and the shaft deepened to 1,000 feet depth. Cheap power was available from the nearby Snare River hydroplant. It was also felt that a mill should be installed to treat known ore and provide some revenue (Indigo Consolidated Gold Mines Ltd. Annual Report, 1951). This program would require new financing, and in 1952, Nationwide Minerals Limited acquired the Indin Lake property from Indigo. They did no additional development because the gold markets were too poor to raise needed funds. Work from 1946 to 1951 at Diversified Mine indicated four ore-shoots on each mine level, ranging in width from 3 to 9 feet and in length from 25 to over 200 feet. Grades varied from 0.21 ounces per ton to 0.75 ounces per ton gold (Indigo Consolidated Gold Mines Ltd. Annual Report, 1951).

Exploration Since Mine Closure

The Diversified property remained closed throughout the years until the 1970s when the shaft was de-watered through contract with Stanley Paulson by Ursa Polaris Limited. Ore reserves in the #1 zone were estimated at 72,500 tons of ore with an average grade of 0.358 ounces per ton gold in 1980. Underground rehabilitation and surface construction continued when Indin Gold Limited acquired the property in 1980, but work was interrupted by a fire in March 1981 that destroyed several of the original buildings and equipment. New mining equipment was installed and several of the buildings replaced, including the erection of a 20 man Atco trailer camp. To September 1981, rehabilitation to the 2nd level of the shaft was completed, including the laying of new rail track.

Mining equipment at the Indin Lake project in 1981 included the following: 175-kW Detroit diesel generator, 26-kW Dieseltec generator, 300-kW Detroit diesel generator, two Ingersoll-Rand 'Spiroflow' XL-750 air compressors, a Case loader, a Canadian Ingersoll-Rand 48x36-inch single drum electric hoist, two Mancha electric locomotives, thirteen side-dump ore cars, one Eimco 12B mucking machine, miscellaneous jackleg and stopper drills, and a complete assay lab. Several older, abandoned and damaged mining equipment from the 1940s were also located on site. (Cardy, 1989)

Indin Gold hoped to place the property into production by 1983, but this plan was canceled due to a lack of financing (Indin Gold Ltd. Prospectus, 1981; Globaltex Industries Inc., 1999). Indin Gold amalgamated with Nithex Exploration in May 1983 to form New Lintex Minerals Limited. In 1988, Manson Creek Resources optioned the claims and completed a 10-hole drill program from the ice of Indin Lake, north of the shaft. Ground geophysical surveys were also completed during this program. (Fields, 1993)

In 1993, New Lintex was reformed into Globaltex Industries Incorporated. A new program called for the driving of a -15% decline for a distance of 3,000 feet to intersect the bottom level of the old Diversified Mine shaft. The shaft would be used for secondary access and for ventilation. During the spring of 1994, additional fuel, supplies, and a specialized assay lab was trucked to the property (Globaltex Industries Inc., 1999; Richardson, 2000).

Production plans included the clearing of an all-weather road between the Diversified Mine and the Colomac Mine, which was brought back into production by Royal Oak Mines Incorporated in 1994. Ore from the Indin Lake Project would be milled at Colomac on a custom basis. After developing the Diversified property, development could then switch to the nearby North Inca property, a faulted continuation of the deposit. Globaltex Industries Limited desired a joint-venture partner to assist in development of the Indin Lake project. In February 1997, the company entered into a deal with Silverspar Minerals Limited who would agree to spend at least \$1.8 million on a three-phase exploration and development program in order to earn a 50% interest. The three phases included de-watering of the shaft and workings, mine rehabilitation, underground sampling and diamond drilling, and driving of a decline into the #1 zone orebody. The nature of the program was both to affirm the known ore reserves and to search for new reserves. Aside from the mobilization of fuel and supplies over the March 1997 winter road, construction and repairs to the buildings,

small-scale sampling and assaying, and rehabilitation of some of the workings, the company did little work. Silverspar dropped the option on the property in 1998 citing a lack of finances. A drop in the price of gold and the cessation of operations at the nearby Colomac Mine hampered production plans for the Diversified Mine (Silverspar Minerals Inc. Annual Report, 1997).

No exploration has been conducted since 1997. In 2003, Globaltex Industries reformed as Pine Valley Coal Mining Company and focused on the development of a coal mine in British Columbia. In 2007, Pine Valley sold the mineral and lease rights to George Stephenson of Calgary, Alberta. Mr. Stephenson applied for a land use permit with the Wek'eezhii Land and Water Board late in 2007 for the purpose of mineral exploration and camp rehabilitation.

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DOME LAKE

Bulk Sampled (Non-Operational)

Gold

62° 45' 15" 113° 17' 30"

Years of Primary Development: 1990s

Mine Development: primary open cut

Years of Bulk Sampling: 1992, 1996

Bulk Sample: 408 tons milled = 182 oz Au

Introduction

The Dome Lake Mine is located on the south side of Dome Lake, 64 kilometers northeast of Yellowknife, NWT. Development has focused on the work of Knut Rasmussen, who mined the deposit in 1992 and 1996. Rasmussen is now getting out of the business of mining and has no plans to conduct any further work at Dome Lake. The site was visited in July 2003 by the author.

History in Brief

This property was famous for its 'white gold' showing, staked as the 'C' group in 1938 and explored by Chan Yellowknife Gold Mines Limited and Dome Mines Limited during that fall. In 1973 the property was re-staked as the 'TT' claims by Jim Turner. A grab sample obtained by Mr. Turner assayed in excess of 7.0 ounces per ton gold. A small program of diamond drilling was undertaken by Duke Mining Limited in the 1970s, and in 1987 a company called Lightening Minerals Limited sank a small pit into the Lambert vein. New owner Knut Rasmussen maintained the property during the 1990s with hopes of economically exploiting the small yet interesting deposit. Work was focused on the 14-vein. Some ore was mined by him in 1992 and 1996 and trucked to Yellowknife for processing, but no major work has been undertaken since.

Geology and Ore Deposits¹

Gold mineralization occurs within quartz veins in sediments within the Archean Yellowknife Supergroup. The veins are conformable with the enclosing sediments and have some wall rock inclusions. The quartz is white or grey in color with carbonate and chlorite alteration in places. The most abundant sulphides in the veins are arsenopyrite and pyrite with minor pyrrhotite, trace galena, sphalerite, and chalcopyrite. Visible gold is scattered irregularly throughout the veins. The Lambert vein and the 14-vein have been extensively explored and partially mined. The Lambert vein is 100 feet long by 5 feet wide and has been mined to 10 feet depth. The 14-vein is 110 feet long, 4 feet wide, and has been mined to 25 feet depth. The 18-vein has not been mined but is 320 feet long and 5 feet wide. Pre-1990s exploration suggested an average grade of 0.45 ounces per ton gold within the veins.

Knud Rasmussen (1990s)

Ore was mined by Knud Rasmussen and Dave Nickerson from the Lambert vein pit during the early 1990s, and plans were made to truck the ore over the winter road to Yellowknife where it could be custom milled. In April 1992, 11 tons of ore were sent to Ptarmigan Mine for processing. This shipment yielded 10 ounces of gold (Tremenco Resources Ltd., 1992). Work after 1992 focused on the 14-vein. In 1995, Dave Nickerson calculated 900 tons of ore grading 0.65 ounces per ton gold. During March 1996 a 600 ton sample was mined at the Dome Lake property, awaiting shipment. Mining was performed with standard small-scale pit mining techniques using an air-track drill for drilling 2.5 inch diameter blastholes and a front-end loader and a backhoe for collecting material. Mined dimensions of the 14-vein pit were estimated as 80 feet long, 30 feet wide, and averaging 15 feet deep with a maximum depth of 25 feet (Nickerson, 1997).

Custom Milling 1996

Ore was trucked over the winter road for processing at the Ptarmigan Mine nearing the end of the winter road season in April 1996. 40 truckloads were required to transport 397 tons of material. Tremenco Resources Limited (owners of Ptarmigan Mine) reported milling 397 tons of ore in May 1996 with a head grade of 0.48 ounces per ton gold to recover 172 ounces of gold, representing a 90.6% recovery.

Equipment used included one air-track drill and 600 cubic feet per minute compressor, one 150 cubic feet per minute compressor, a five ton front-end loader, backhoe, and a seven ton dumptruck. Two men, Knud Rasmussen and Dave Nickerson, were employed between March and April 1996 with the operation. A campsite was maintained on the

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

shores of Dome Lake and consisted of two small frame shacks. A network of roads ventured from the camp south towards the mine workings, which are located about a kilometer from the campsite (Nickerson, 1997).

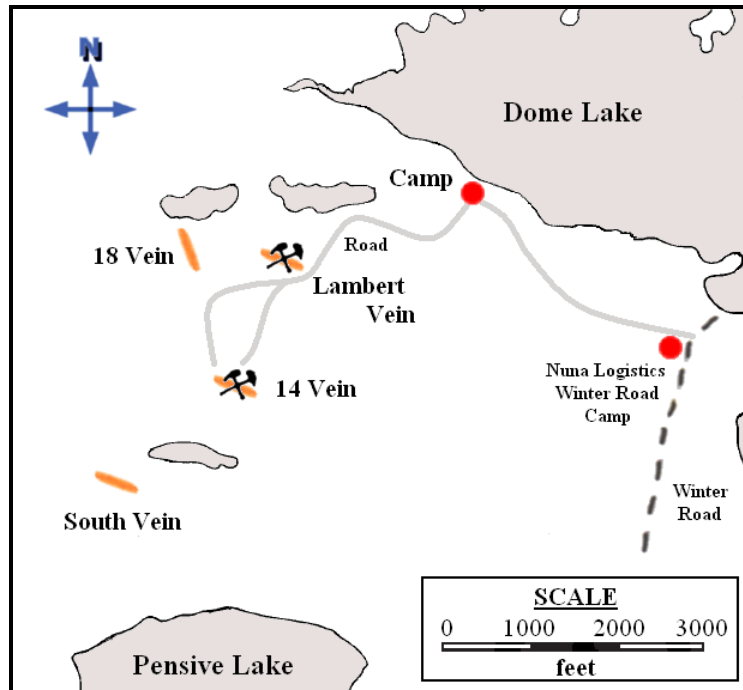


Figure 1. Dome Lake Mine area.

Frank Vein

In March 1995, Dave Nickerson, in association with Knut Rasmussen and Raymond Essery, mined a small amount of ore from the Frank vein, located five kilometers northeast of Dome Lake on separate mineral claims. It was planned to include this ore with the 1996 bulk shipment from Dome Lake, but so far as known this stockpile has not been processed. Forty-five tons are said to be stockpiled there with a grade of 0.64 ounces per ton gold (Nickerson, 1995)

Total cost of the 1996 operation was estimated at CDN \$50,000. Proceeds from the sale of gold exceeded CDN \$61,000 and custom-milling charges with Tremincó totaled \$16,000, representing a \$5,000 loss on the project. Nevertheless, the 1996 program illustrated that economic mining could be performed from Dome Lake's small high-grade gold veins provided that costs were kept low. (Nickerson, 1997) The drop in the price of gold in 1997 and the closure of Ptarmigan Mine effectively made continued operation unfeasible. No other work has been done at the site although Knud Rasmussen still maintains the campsite and equipment at Dome Lake. Barring sale of the property, Rasmussen plans to cleanup the site and abandon it as per current land restoration policies.

Exploration Since Mine Closure

No exploration has been conducted since 1996.

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Personal communication: Knut Rasmussen; Dave Nickerson



ECHO BAY

Producer (Remediated)

Silver, Copper

66° 05' 45" 117° 59' 30"

Years of Primary Development: 1934-1936, 1964-1974

Mine Development: 3 adit tunnels, vertical winze

Years of Production: 1964-1974

Mine Production: 363,140 tons milled = 23,564,461 oz Ag, 4,505 tons Cu

Introduction

The Echo Bay Mine is located near LaBine Point (Port Radium) on Great ear Lake. It is 440 kilometers northwest of Yellowknife, NWT. The mine is adjacent to the famous Eldorado Mine, and the townsite that both mines used was known as Port Radium. First underground exploration of the silver deposits occurred in the 1930s, and in 1964 the mine was brought into production. In 1976, the old Eldorado Mine was reopened and mined for its silver content when Echo Bay ores were depleted. Following the closure of the Port Radium operations in 1982, the Eldorado Mine and Echo Bay Mines were cleaned up. The site was viewed from the air in July 2005, and except for some debris, ruins, and open stopes, there is nothing left of Port Radium.

History in Brief

The 'Echo Bay' claims were staked in 1930 by prospectors with Cominco Limited over a series of silver veins adjacent to Gilbert LaBine's Eldorado Mine. Underground development on two adit levels began in 1933, but work ceased when the price of silver collapsed. The site remained closed until 1964 when Echo Bay Mines Limited reopened the mine workings. Silver and copper was produced from 1964 until 1976 when the orebody was depleted. Echo Bay Mines Limited then reopened the old Eldorado Mine and produced silver from that property until 1982, when all operations at Port Radium stopped.

Geology and Ore Deposits

Silver deposits at Echo Bay occur as 'horsetail' type fracture-filled veins. The veins are hosted in a variety of northeast striking extrusive and intrusive rocks, most of which are fine grained and highly altered. Tuffaceous and aphanitic rocks, in part porphyritic, show various degrees of silicification. Four principal northeast striking veins, ranging from a few hundred feet to over 1,000 feet long, have been explored and mined. They generally dip vertically or steeply north. The veins occupy shear zones and are filled with carbonate and quartz gangue and a wide variety of sulphide minerals. Hydrothermal alteration of country rocks adjacent to the veins is common. The principle minerals are native silver, argentite, argentiferous galena, and acanthite. In addition a number of copper, zinc, nickel, and cobalt sulphides are found. Several ages of mineralization are apparent. More than one type of breccia within the veins indicate disruption and metal deposition during several periods (Schiller, 1965).

Cominco Limited (1933-1936)

The high value of silver in the early 1930s paved the way for some major development on the Echo Bay group of claims by Cominco Limited. Exploration camps were constructed in 1932, and in 1933 a 4 ton bulk sample of silver ore was shipped to Trail, B.C. for the recovery of 3,185 ounces of silver. In addition, 1,500 pounds of pitchblende ore was shipped to produce 1,155 ounces of silver. No economic radium minerals were found in these samples (The Miner, May 1934).

Underground development commenced in 1934 through an adit portal (#1 adit or 1st level adit) collared on the 125-foot level. Transportation difficulties postponed operations until October 1934 when diesel power was installed. During the year, 524 feet of drifting and crosscutting was completed. The adit crosscut intersected a new silver vein parallel to the #2 vein. Rich concentration of silver were found in leaf, wire, and plate form. No pitchblende ores were encountered (The Northern Miner, Apr. 18th 1935). The #1 adit was driven at an elevation of 125 feet from the surface exposures and extended 440 feet southeast into the system of veins. The developed veins included the #4, #2, and #5 veins. About 1,000 feet of drifting was done along these ore structures on the 1st adit level.

Early in 1935, the crosscut reached the #5 vein and sinking of a winze was underway. Expenses involving this winze proved too great, so it was instead decided to drive a new adit portal 200 feet below the original level, or at the 325-foot level. Work on the 2nd level adit began in July 1935 (Spence, 1935). The 2nd level adit portal was located 600 feet northwest of the 1st level adit portal and the crosscut followed a parallel direction to the previous development to intersect the #2 vein, upon which 960 feet of drifting southwest was completed in 1936. No other veins were

intersected on 2nd level. Ten to twenty men were employed during 1935. Dick Powell was field engineer in 1935 and W.G. Jewitt was the Cominco boss overseeing operations (Spence, 1935). Operations ceased in June 1936 upon the exhaustion of supplies (The Miner, May 1937). Although the silver deposit had huge potential, Cominco Limited was unable to justify expending any further money on the claims. This proved to be a wise decision, as two years later in 1938, the price of silver tumbled.

Development Summary 1934-1936

Between 1934-1936, the Echo Bay Mine was developed by two adit levels. The 125-foot level adit was 440 feet long and drifting on three veins (#1, 2, and 3) totaled 1,000 feet. The 325-foot level adit was 1,040 feet long and 960 feet of drifting was accomplished on the #1 vein. A winze was sunk below the 125-foot level and some raising was accomplished. Crosscutting is not included in these statistics (Schiller and Hornbrook, 1964).

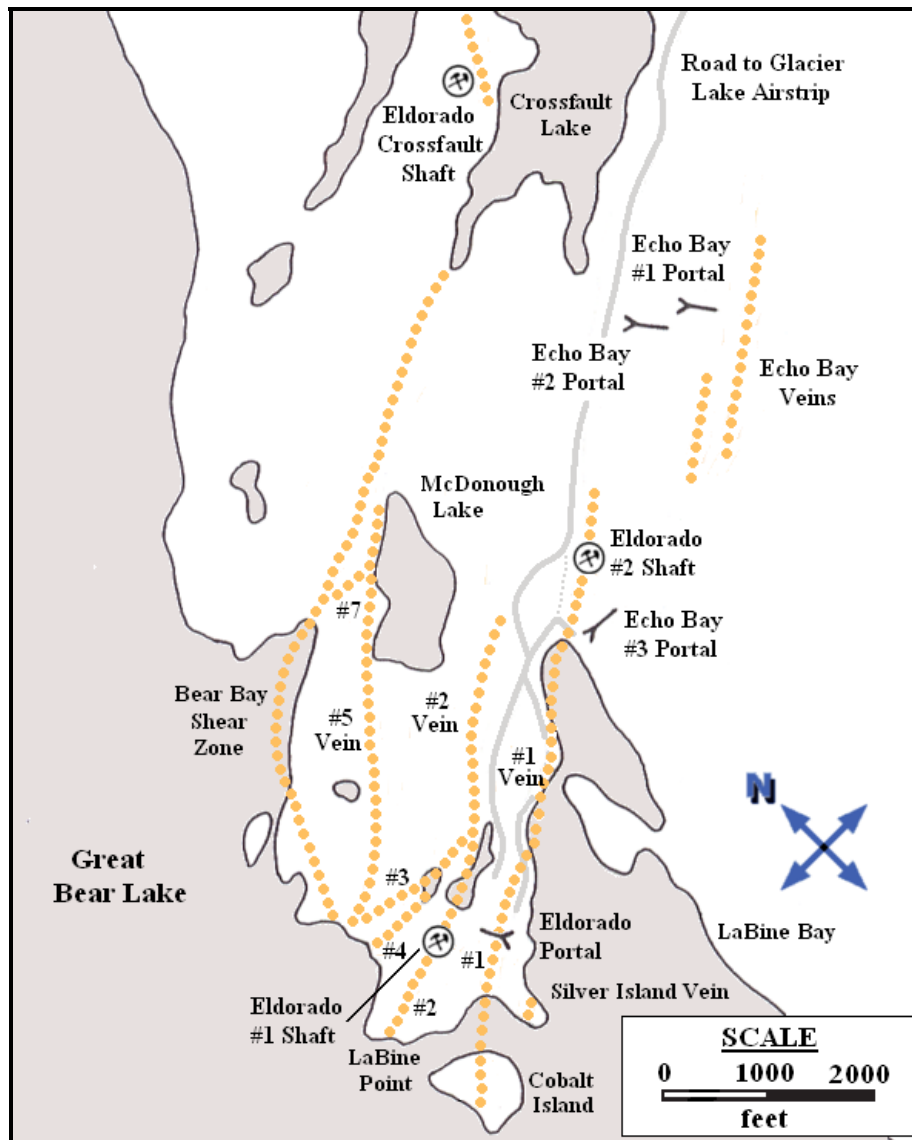


Figure 1. *Echo Bay and Eldorado Mine property plan showing location of shafts and shear zone systems.*

Echo Bay Mines Limited (1964-1976)

In 1963, the long dormant mineral claims were optioned by Cominco Limited to Northwest Explorers Limited of Edmonton, Alberta. The company completed a diamond drilling (11 holes totaling 1,500 feet) and sampling program during the year. A 500 pound sample of ore from the underground workings was sent to Ottawa for testing in 1963, and preliminary results indicated a good silver concentrate could be obtained. Northwest Explorers announced plans

to bring the Echo Bay mine into production during 1964 (Schiller and Hornbrook, 1964). During the summer of 1964, Echo Bay Mines Limited was formed as a subsidiary of Northwest Explorers Limited.

A major justification for bringing this mine into production was the cost savings in leasing the former Port Radium plant and camp buildings from Eldorado Mining and Refining Limited. Without the existing infrastructure in the vicinity of the Echo Bay Mine, it would not have been feasible to contemplate silver production. The mill and surface structures at Port Radium were leased by Echo Bay Mines Limited from Eldorado Mining and Refining Limited, and later purchased in 1966.

Production commenced in October 1964 and a milling rate of 85 tons per day was achieved by the end of the year. The average grade of the ore during 1964-1966 was 35 ounces per ton silver (Thorpe, 1972). Mill feed was from the #2 and #3 veins. Backs were brought down on the 1st adit level on the #2 and #5 veins. Work was suspended on the 1st level during part of the year because of freezing winter conditions. Development at Echo Bay during 1964 included the driving of a raise from 2nd to 1st level (designed as the primary ore pass), 1,050 feet of drifting on the 2nd level, diamond drilling of 16 surface holes and 2 underground holes (total footage unknown), and construction of a combination powerhouse, dry, and machine shop building at the 2nd level adit portal (Schiller, 1965).

During 1965, mining operations were focused within the #2 vein on the 2nd level. Three stopes provided the bulk of the ore mined. Other development during 1965 included the driving of raises to surface, and the driving of a northerly crosscut on the 2nd level to establish a diamond drilling station for exploration of the #2 vein. Deep drilling at the mine resulted in good silver intersections. Plans were made to drive a new adit below the existing workings (Thorpe, 1966).



NWT & Nunavut Chamber of Mines photo

Figure 2. *Port Radium, base of the Echo Bay Mine, 1970s.*

New Adit Entrance

Early in 1966, rich ore containing both argentite and native silver was encountered in the #6 vein. Drilling indicated continuity of mineralization below the 2nd level, and plans were made for the excavation of a 3rd adit level below the old workings. The main object of this adit was to access to the rich silver veins 160 feet vertically below the 2nd level. The portal was excavated on the shores of Great Bear Lake, about 2,300 feet west of the old Echo Bay workings. This new adit would be started closer to the Eldorado milling plant and would eliminate the necessity of trucking over a steep hill to get to the old adits. The #3 adit was advanced 1,686 feet by November 1966. By the end of January 1967, drifting on this level had been accomplished for several hundred feet in either direction along the vein and a raise was started to tap into the #2 adit level workings. A continuation of the #2 vein was found on the 3rd level. Meanwhile, diamond drilling below the 3rd level was started to test for continued depth potential of the deposit (Thorpe, 1972).

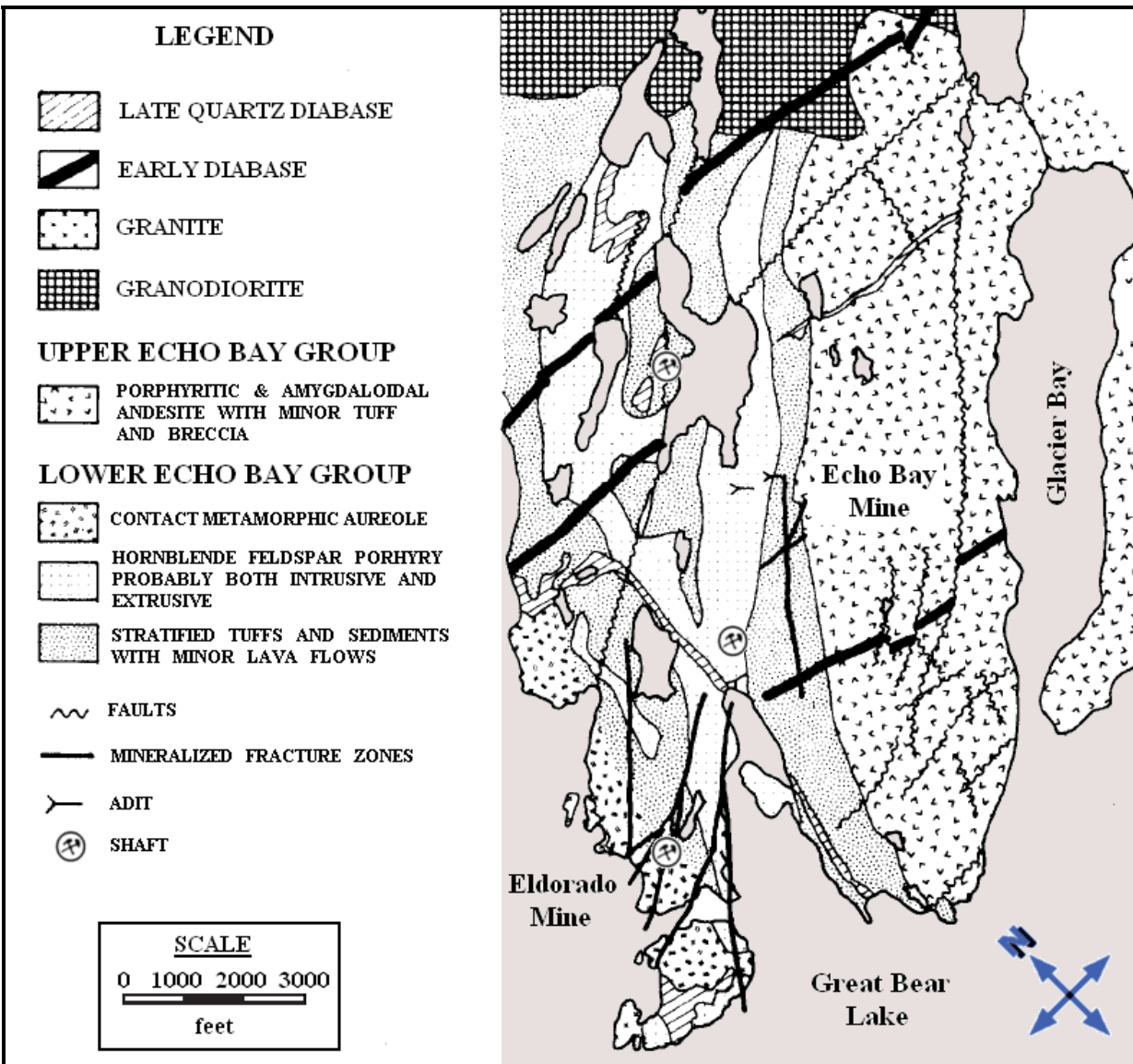


Figure 2. Echo Bay and Eldorado Mine local geology map.

Employees and Camp 1968

In June 1968, Echo Bay Mine staff included the following persons: John Zigarlick, mine manager; R.B. Mason, mine superintendent; Dave Williams, mill superintendent; R. Skode, surface foreman; S. McLean, master mechanic; K. Hawkins, resident engineer; E. Schmidt, chief assayer, and V. St. Amand, accountant. A total of 103 persons were employed at the mine at that date. The camp consisted of two-story bunkhouses, cookery, and recreation hall, previously occupied by Eldorado Mine at Port Radium (The Western Miner, August 1968).

International Utilities Takeover

In July 1967, Echo Bay Mines Limited was acquired by International Utilities Limited through a purchase of a controlling interest (74%) in shares (Thorpe, 1972).

Uranium Potential

The Echo Bay Mine contained small concentrations of pitchblende ore within the silver veins. In late 1967, an effort was made to concentrate the uranium as a separate product in the mill. On the 2nd level in 1967, one stope reportedly assayed 0.50% uranium oxide (U_3O_8), although when this material was shipped from the property as part of the regular concentrate, the company did not receive any payment for the uranium. Additional uranium ore was uncovered on the 3rd level. Ore was stockpiled to await future treatment (The Northern Miner, Oct. 19th 1967).



Figure 3. Echo Bay Mine longitudinal plan along the mined portion of the #2 vein, c.1972. (C/C = cross cut)

Year:	Ore Milled:	Silver Recovered:	Copper Recovered:
1964	4,554 tons	99,631 oz	-
1965	35,608 tons	1,408,245 oz	487 tons
1966	43,839 tons	1,573,752 oz	822 tons
1967	38,998 tons	2,984,643 oz	649 tons
1968	36,982 tons	2,563,499 oz	457 tons
1969	34,797 tons	2,298,372 oz	340 tons
1970	36,925 tons	2,511,267 oz	391 tons
1971	35,985 tons	2,445,709 oz	332 tons
1972	37,291 tons	2,456,386 oz	393 tons
1973	37,393 tons	3,063,820 oz	430 tons
1974	20,768 tons	2,159,137 oz	204 tons
1975-1976	<i>Production statistics are mixed with those of the Eldorado Mine. Total below is up to August 1974.</i>		
Total:	<u>363,140 tons</u>	<u>23,564,461 oz</u>	<u>4,505 tons</u>

Table 1. Echo Bay Mine production 1964-1976. (source: Thorpe, 1972; Mineral Industry Reports Northwest Territories)

Winze Sinking

Preparations for the sinking of an internal winze from the 3rd level down 500 feet began in October 1967. An underground hoist room was excavated and a large electric hoist was installed. Shaft sinking contract was awarded to Haste Mine Development Limited. By July 1968, this winze was completed to a depth of 510 feet below the 3rd level, providing three new levels at 150-foot intervals. Diamond drilling had confirmed the persistence of silver beyond the 4th level (or 1st shaft level. (Thorpe, 1972).

Operations continued very profitably during 1968. Ore on the 3rd level was developed in one continuous ore shoot 520 feet long, with backs of 200 feet, and an average width of 6 feet. Stopping of the vein provided regular and high-grade (75 ounces per ton silver) mill feed (Thorpe, 1972). During 1969 the company reported that the mine was so high-grade that it was causing operational problems. Large pieces of ore were often so rich in silver that the crusher was jammed and the circuits plugged (News of the North, Mar. 20th 1969). A 150 foot thick diabase sill or dyke was encountered between the 4th and 6th levels (1st and 3rd shaft levels) in 1969-1970. The implications of this were not known at the time, and diamond-drilling results indicated that silver mineralization continued below this sill.

Milling Operations 1960s

The following milling circuit was reported in March 1969. Mine ore was trucked from the Echo Bay #3 portal and hauled via 10 ton trucks to the Eldorado #1 shaft where it was dumped and introduced into the Eldorado Mine's underground crusher and mill conveyor. Primary crushing was through a 12 inch x 24 inch Blake jaw crusher. Product was sent to a 150 ton coarse ore bin, then through a Jeffrey feeder, and then conveyed into a 50 ton coarse ore bin. A Syntron feeder and conveyor fed a secondary jaw crusher (9 inch x 15 inch Blake unit). This crushed product was conveyed and dumped over a 3/8 inch screen. Oversize was conveyed back for secondary crushing in a 2 foot Symons cone crusher, and screen undersize was dropped into a 75 ton fine ore bin, then into the grinding circuit. Primary grinding was achieved in a 4 foot x 8 foot Allis-Chalmers rod mill and a 4 foot x 6 foot Allis-Chalmers ball mill. This product was classified in a Rake classifier with underflow being sent for secondary grinding in a 5 foot x 5 foot Allis-Chalmers ball mill. Two Denver duplex jigs (12 inch x 18 inch) at the discharge of the secondary ball mill caught a high-grade jig concentrate. Jig tails were sent back into the classifier. Classifier overflow was pumped to the flotation circuit.

The first stage of flotation was a single agitator tank, feeding a bank of 12 Denver #18 flotation cells, nine roughers, and 3 scavengers. Rougher overflow was cleaned in 3 cleaner flotation cells (Denver #15) and this overflow was sent for two-stage re-cleaning in Denver #15 and Denver #24 flotation cell units, respectively. A final flotation concentrate was recovered from this re-cleaner. Tailings from all flotation units were sent to a bank of four scavenger flotation cells (Denver #15), overflow from which was sent back into the flotation circuit and tailings jettisoned from the plant. Flotation concentrate was sent through a three-disc 4 foot filter to remove water and then dried in a 50 kilowatt drier unit, packed in cardboard boxes, weighed, and shipped.

Products were a silver-copper flotation concentrate, and a silver jigging concentrate. A large portion of the jig concentrate was flown out when feasible. The remainder of the concentrate was trucked over ice roads in the winter or barged out in the summer (The Canadian Mining Manual, 1969). Numerous changes were made in the 1970s to increase tonnage and to increase silver recoveries in the Echo Bay mill. In 1969, a new ball mill was added to bring daily tonnage to 150 tons. Two additional Denver jigs of identical size were added in 1972 to double jigging capacity (Parashyniak, 1977).

Deepened Mine

During 1970 the Echo Bay winze was sunk to a depth of 1,250 feet to provide five additional levels below the diabase sill (Padgham et al, 1978). The company predicted favourable geology below the diabase, and ore expectations were regarded as good. Patrick Harrison and Company Limited was the shaft sinking contractor.

Glacier Lake Airstrip

An airstrip was cleared a few kilometers east of the mine site in 1971 to allow year-round wheeled-plane access to the mine. Concentrates could now be shipped out on DC-3 flights during the summer months.

In the early 1970s, mining operations were focused on mining out the mid levels of the mine. The mill was operating steady at a rate of 100 tons per day with mill heads averaging 70 ounces per ton silver and 0.95% copper. Four large native silver pieces mined in August 1971 consisting of 70% silver and worth \$25,000, were put on display at the Toronto Dominion Bank in Edmonton (The Northern Miner, Jan. 6th 1972).

Diabase Sill Cuts off Mineralization

Additional development and exploration showed that the massive diabase sill cut-off economic mineralization below the 6th level. Exploration failed to locate a continuation of the ore deposits, and operations switched to remnant mining of the upper levels. In 1973 and 1974, production was from the 2nd, 3rd, and 4th levels, with extensive development and diamond drilling on the 5th and 7th levels (The Northern Miner, May 16th 1974; Gibbons et al., 1977).

Eldorado Mine Purchased

In 1974, Echo Bay Mines Limited purchased the mineral rights to the Eldorado Mine from Eldorado Nuclear Limited (previously Eldorado Mining and Refining Limited). Mining of the old silver deposits of the famous Eldorado Mine was considered feasible because of the exhaustion of reserves at the Echo Bay Mine. The hoist was removed from the Echo Bay winze and installed at Eldorado's #1 shaft in 1974, which was then dewatered. All milling ceased at Echo Bay Mine in August 1974 upon exhaustion of reserves, but milling resumed early in 1975 to process cleanup ores. By November 1976, all operations in the old Echo Bay workings had ceased and from 1977 to 1982 all operations at Port Radium were centered on mining silver from the Eldorado Mine (Laporte et al., 1978; Lord et al., 1978).

Exploration Since Mine Closure

Echo Bay Mines Limited conducted some further exploration at the Echo Bay Mine up until 1982, but there is no good record of this work. In 1981, it was reported that 316 feet of lateral exploration development was conducted in the Echo Bay Mine workings, plus 2,700 feet of underground diamond drilling. This exploration was presumably disappointing (Brophy et al., 1984). New claims ('GLAC' and 'Cobalt') were staked in the 1990s by Trevor Teed. In 1997, extensive surface sampling and prospecting was completed in the areas of the old trenches and open cuts above Echo Bay Mine. In addition, six new trenches were excavated and high gold assays were reported (Griep, 1997). The property was optioned to Alberta Star Development Corporation in 2005.

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 EKATI™ <i>Producer (Operational)</i>	Diamond
	64° 43' 00" 110° 36' 30"

Years of Primary Development: 1993-1994, 1997-current

Mine Development: six open pits; underground ramps

Years of Production: 1998-current

Mine Production: ongoing (see **Table 3**)

Introduction

Ekati Mine was North America's first diamond producer, starting operations in October 1998, with a projected life to 2015. The mine is 80% owned by BHP Billiton Diamonds. The remaining 20% interest in the mine is held by discoverers Charles Fipke and Stuart Blusson. Ekati Mine is located 310 kilometers northeast of Yellowknife, NWT, and is accessible by weekly flights and winter road access. The author was casually employed at the mine during 2002-2003. The mine has produced over 40 million carats of diamonds from 1998-2008.

History in Brief

Ekati's discovery was long in the making, and as early as 1981 Charles Fipke and later his company, Dia Met Minerals Limited, were searching for clues across western Canada to find the potential for diamonds. In 1990, Dia Met and BHP Minerals Limited¹ joined forces and narrowed their focus to the NWT's frozen tundra. The following summer, the Point Lake kimberlite pipe was discovered and a massive staking rush ensued. A large claim block of 344,000 hectares was secured by the joint-venture team and in 1992-1993, the Fox, Panda, Misery, and Koala pipes were discovered. The Koala North pipe was discovered in 1998. Many other pipes, most of which are of no commercial value, have been outlined on the massive property. Underground development and bulk sampling of the Fox and Panda pipes commenced in 1994. During 1994-1996, feasibility and various assessment studies were compiled to fulfill the company's obligation to provide detailed information on the construction and operation of the diamond mine for government and local interests. In January 1997, the company received all the required government permits to place the property into production.

Construction began in the spring of 1997 and pre-stripping of the Panda pipe began. The mine was placed into diamond production using the Panda pipe was source of ore in October 1998. Commercial production commenced from the Misery pipe in 2001, the Koala pipe in 2003, and the Koala North pipe was placed into underground production in 2002. The Beartooth pipe was placed into production in 2004 and the Fox pipe entered production in 2005. Underground mining of the Panda pipe commenced in 2005 and is to be followed by underground mining of the Koala pipe.

Geology and Ore Deposits²

The Ekati Mine area is underlain by Archean greywacke, mica schists, and quartzite of the Yellowknife supergroup intruded by Archean granitic rocks. Proterozoic diabase and gabbro dykes crosscut the earlier Archean rocks. Within the main claim block area kimberlite pipes are the only known Phanerozoic rocks. Minor amounts of metasedimentary rocks, comprised mostly of thinly bedded metagreywacke with small rafts of migmatite, occur within the claim area. At least five types of granitoid intrusive rocks occur. A large, homogeneous pluton of equigranular, medium grained quartz diorite comprised of plagioclase, biotite, hornblende, quartz and epidote, occurs in the south-central portion of the main claim block. The pluton is weakly deformed with poor foliation development. Weakly to strongly foliated biotite tonalite and hornblende biotite tonalite have intruded the metagreywackes.

The two most voluminous granitoid rocks in the claim block area include a distinctive two mica granite and porphyritic biotite granite. The two-mica granite typically comprises equal proportions of quartz, plagioclase and potassium feldspar with 5 to 10% muscovite and biotite. Apatite and tourmaline are common. These granites are fine to medium grained and generally equigranular. A tourmaline bearing pegmatitic phase occurs as dykes and small stocks up to 250 meters in diameter. The two mica granite intrudes greywacke and both tonalite units.

¹ now BHP Billiton Diamonds

² Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

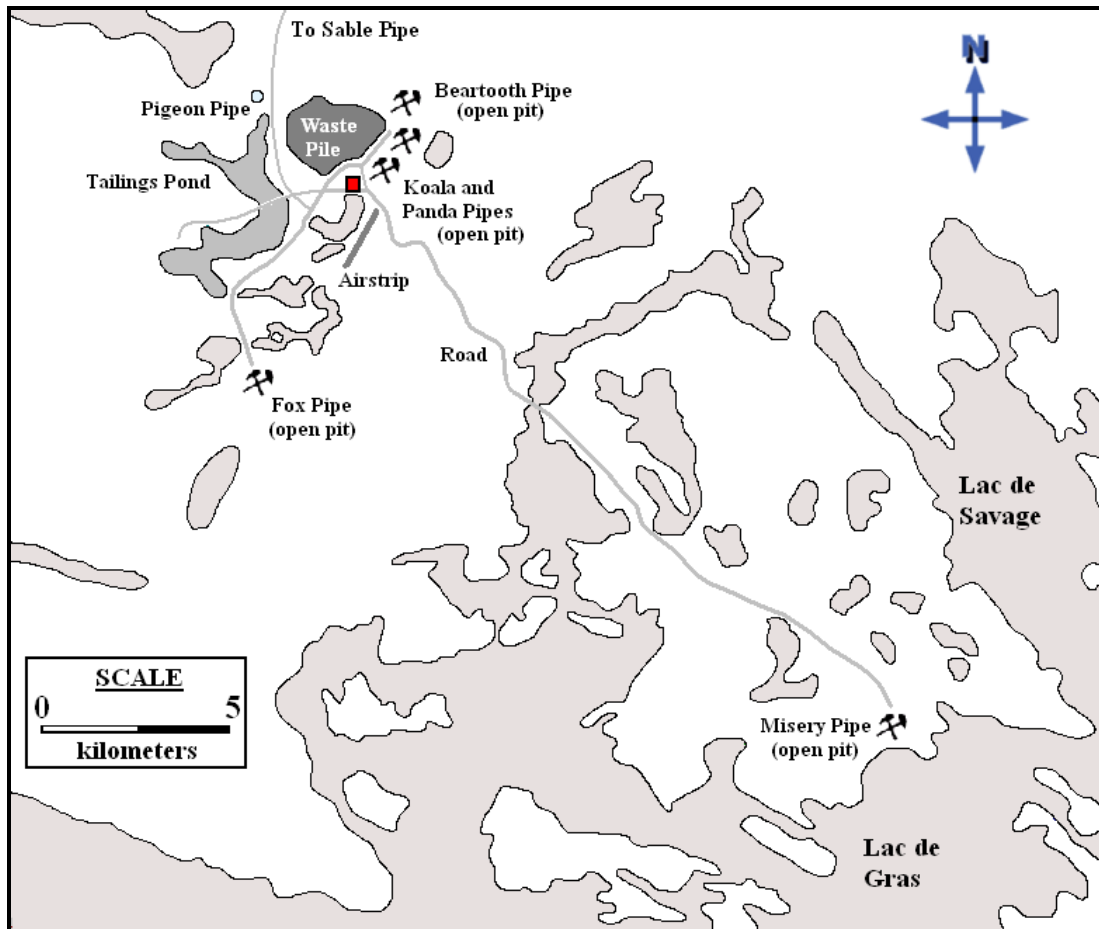


Figure 1. Ekati Mine property plan, 2007.

Five Proterozoic dyke swarms with dyke widths from 15 to 100 meters crosscut earlier Archean rocks. From oldest to youngest these are: the Mackay dykes, the Contwoyto dyke swarm, the Lac de Gras dykes, the Mackenzie dyke swarm, and the '305' dykes. Kimberlite bodies tend to occur in areas of structural preparation, indicated by abundant Proterozoic dyke emplacement. At least 136 kimberlite pipes were identified as of May, 1999. Rb/Sr 3-point isochron age dating yielded an Eocene age of emplacement at 52 ± 1.2 Ma for one of the kimberlite pipes. Fossils within xenoliths carried by the kimberlite indicate that lower Cretaceous to late Paleocene strata, now completely eroded, originally overlaid the Archean stratigraphy at time of kimberlite emplacement. The kimberlite pipes have undergone minimal erosion as evidenced by preservation of crater-facies epiclastic sediments. The kimberlites are typically several hundred metres in diameter, are covered by small circular lakes and are overlain by 5 to 30 meters of sand and gravel. Varieties of kimberlite distinguished in drill core include epiclastic, tuffisitic and tuffisitic kimberlite breccia phases, and massive hypabyssal kimberlite. Only six economic pipes have been outlined on the Ekati Mine property. They are the Koala, Koala North, Panda, Misery, Fox, and Beartooth. The Leslie and Sable pipes are not considered economic at this time.

BHP Minerals Limited³ (1993-1994)

In 1993, BHP Minerals Limited created a wholly-owned subsidiary, BHP Diamonds Ltd., to act as manager of the NWT Diamond Project. Interest was held by BHP Minerals Limited (51%), Dia Met Minerals Limited (29%), Charles Fipke (10%) and Stuart Blusson (10%). Valuations of diamonds from mini-bulk sampling suggesting excellent carat grades from four pipes on the property, and plans were laid to begin larger bulk sampling of the Koala, Panda, and Fox pipes. During the fall of 1993, a large exploration camp suitable for winter habitation was built on the eastern side of Kodiak Lake to house the crew performing the bulk sample and underground development program. Capacity of this camp was 170 to 190 persons, housed in a 'Weatherhaven' facility (Dia Met Minerals Ltd. Annual

³ BHP Minerals Limited was reformed in May 2001 into BHP Billiton Diamonds through the merger with Billiton Incorporated.

Report, 1993/1994). A 1,940 meter airstrip was constructed along a natural esker to accommodate heavy freight brought in by Hercules C130 or Boeing 737 aircraft. The bulk sampling plant, also known as a “dense media separate plant”, was constructed during 1993 and had a capacity to process 10 tonnes per hour. It began operating in January 1994 (Dia Met Minerals Ltd. Annual Report, 1993/1994).

Underground Development at Fox Pipe

Between February and August 1994, a decline was advanced 970 meters into the Fox pipe. Twenty-five meters of raising was also completed. Underground diamond drilling totaling over 1,800 meters was also performed (NWT Mining Inspection Services). An 8,223 tonne bulk sample was extracted from the Fox pipe and processed to produce 2,199 carats of diamonds (Canadian Minerals Yearbook, 1998).

Underground Development at Panda Pipe

Between June and November 1994, a decline was advanced 460 meters into the Panda pipe. 80 meters of raising was also completed. Underground diamond drilling totaling over 2,400 meters was also performed. (NWT Mining Inspection Services) A 3,402 tonne bulk sample was extracted from the Panda pipe and processed to produce 3,244 carats of diamonds. (Canadian Minerals Yearbook, 1998)

An intense period of planning followed in 1995-1996, during which time BHP undertook an intensive environmental review – the first of its kind in the NWT – to ensure that the environment was protected during the course of the mine’s life. BHP also negotiated ground-breaking impact benefit agreements with regional aboriginal communities to maximize northern employment and business opportunities.

BHP Billiton Diamonds⁴ (1997-current)

Authorization to initiate the construction of the mine was granted from government officials in January 1997, upon the positive recommendations of the Federal Environmental Review Panel. 2,022 truck loads of supplies, equipment, and fuel totaling 42,000 tonnes were transported to the property over the winter road. Prefabricated units used to construct the new mine camp were amongst the first to be erected. The mine camp, described below, was completed in May 1997 although accommodation in trailer units was still needed.

Ore Reserves

Ore reserves calculated at April 1st 1997 suggested a mine life of up to 17 years through the production of five pipes by a combination of open pit and underground mining methods. Total ore reserves were 65,900,000 tonnes with grades of 1.09 carats per tonne (see **Table 1**). The possibility of delineating additional economic diamond-bearing pipes within the property was viewed as extremely high, in which case production could be lengthened to 25 years. Beartooth, Pigeon, and Koala North were all viewed as being possible sources of future ore but more bulk testing would need to be done (Dia Met Minerals Ltd. Annual Report 1997/1998).

	Panda Pit:	Misery Pit:	Koala Pit:	Fox Pit:	Sable Pit:	Panda U/G:	Koala U/G:	Total:
Production Years:	1-5	2-15	5-11	11-17	10-15	6-11	11-15	
Proven Reserves (*):	8.6	4.8	10.0	8.1	11.0	-	1.0	43.5
Probable Reserves (*):	4.0	0.7	4.6	8.6	1.9	0.8	1.8	22.4
Total Reserves (*):	12.6	5.5	14.6	16.7	12.9	0.8	2.8	65.9
Grade (c/tonne):	1.09	4.26	0.76	0.40	0.93	0.97	1.63	1.09

Table 1. Ore reserve estimates at April 1997 included a timescale as to which pipes would be mined at what period in the life of the mine. (*) Tonnages are in millions of metric tonnes. (source: Dia Met Minerals Ltd. Annual Report 1997-1998)

⁴ BHP Billiton Diamonds was formed in May 2001 through the merger of BHP Minerals Limited and Billiton Incorporated.

Construction of Mine Facilities

Establishment of foundations for the milling plant, crusher, shop building, power plant, ammonium nitrate storage building, and security building were completed in June 1997 and steel framing was being erected during July. Buildings were enclosed by October 16th 1997 to facilitate the speedy completion of the interiors during the winter months. At the same time the name of the project changed from “NWT Diamonds Project” to “Ekati Diamond Mine”, “Ekati” being the Dene word for “fat lake”, referring to the traditional name of Lac de Gras. As of July 31st 1997, over 4 million tonnes of waste rock had been removed by stripping at the Panda open pit, to be used in road, dyke, and lay-down construction. Workforce associated with construction operations averaged about 750 employees during the summer of 1997. The winter road reopened late in January 1998 and by the closure of the road on April 4th 1998, BHP had transported 2,179 truck loads with a gross weight of 73,000 tonnes, consisting primarily of additional mining equipment, equipment for the mill, and bulk fuel totaling 56 million litres (Dia Met Minerals Ltd. Annual Report 1997/1998).



BHP Billiton Diamonds

Figure 2. Ekati Mine plant and camp complex, 2000.

Panda Open Pit

The Panda pipe was chosen first to be mined because of its high-grade and high quality stones recovered during earlier bulk testing. Stripping of overburden and amounts of waste rock from the Panda open pit location were begun during October 1997. Mining of ores began in August 1998, by which time most of the plant and camp facilities had been commissioned. Official opening of the Ekati Mine was on October 14th 1998 (Werniuk, 1998).

Milling Operations

Originally, primary crushing was accomplished using a 107 centimeter x 188 centimeter Fuller-Traylor gyratory crusher. In the summer of 2000 this unit was augmented with an MMD-1300 mineral sizer to expedite the processing of cohesive, wet, fine, and frozen kimberlite. The mineral sizer has a rated capacity of 1,500 tonnes per hour @ 80% of the ore passing 150 millimeters, with a maximum lump size feed of 1.2 meters. Secondary crushing is performed in a Nordberg-Waterflush unit, reducing ore to -75 millimeter. The 4 meter x 7 meter Fuller-Traylor primary scrubber removes fine clay and disagglomerates friable soft kimberlite.

Kimberlite ore is processed by gravitational processes and no chemicals are used. Heavy minerals are separated from lighter ones by a sequence of careful crushing and scrubbing to loosen the diamonds from the softer rock. A set of KHD Humboldt-Wedag High Pressure Grinding Rolls (1.7 meter x 1.1 meter) is used to create fines (<1 millimeter), liberate diamonds, and break ore down to a -25 millimeter size, followed by secondary scrubbing and screening. Heavy medium separation is used to float lighter minerals, while diamonds and other heavy minerals are allowed to settle. The heavy mineral concentrate then passes through X-ray sorters that selectively recover the diamonds. Both

the X-ray sorting and final diamond cleaning happens in the secure, 6-storey recovery section inside the main plant building. A covered coarse ore stockpile of 20,000 tonnes is used to minimize fluctuations in feed from the open pits. Waste from the plant is in the form of coarse rejects and fine tailings: the rejects are removed by truck to the waste rock dump, while the fine tailings are pumped through an insulated, heat traced pipeline to the Long Lake processed kimberlite containment area. Final cleaning, hand sorting, and government valuation of the recovered diamonds occurs off-site at the Sorting and Valuation Facility, which opened in Yellowknife in February 1999.

Year:	Kimberlite Pipe:	Type of Sampling:	Ore Sampled:	Diamonds Recovered:	Grade:
1992	Point Lake	Core Drilling	160 T	90 c	0.56 c/T
1994	Koala	Underground	1,550 T	1,465 c	0.95 c/T
1994-1995	Panda	Underground	3,402 T	3,244 c	0.95 c/T
1994-1995	Misery	Core Drilling	1,030 T	4,313 c	4.19 c/T
1994-1995	Fox	Underground	8,223 T	2,199 c	0.27 c/T
1995	Leslie	Core Drilling	680 T	233 c	0.33 c/T
1995	Pigeon	Core Drilling	154 T	60 c	0.39 c/T
1996	Sable	Core Drilling	1,096 T	1,070 c	0.98 c/T
1996	Jay	Core Drilling	238 T	477 c	2.01 c/T
1998	Koala North	Core Drilling	202 T	127 c	0.63 c/T
1998	Beartooth	Core Drilling	189 T	227 c	1.20 c/T
1998	Pigeon	Core Drilling	564 T	251 c	0.46 c/T
1999	Gazelle	Core Drilling	241 T	141 c	0.59 c/T
1999	Phoenix	Core Drilling	106 T	149 c	1.41 c/T
1999	Pirahna	Core Drilling	87 T	203 c	2.33 c/T
2000	Cougar	Core Drilling	74 T	1 c	0.01 c/T
2000	Wolverine	Core Drilling	131 T	9 c	0.07 c/T
2000	Zach	Core Drilling	63 T	11 c	0.18 c/T
2000	Lynx	Core Drilling	169 T	141 c	0.83 c/T
2001	Lynx	Core Drilling	173 T	?	?
2001	Fox	Core Drilling	173 T	?	?

Table 2. Bulk samples from the BHP Ekati Mine property. (T = metric tonnes, c = carats) (source: Canadian Minerals Yearbook, 1998; BHP Billiton reports)

The milling plant is housed in a 70 meter x 154 meter x 30 meter high structure. It was described as being the largest building in the N.W.T. The building also contains warehouse space, small shops, and office space for the metallurgical and geology staff. All operations of the plant are monitored in a central control room and much of the equipment is remotely automated. Ore is processed at 9,000 tonnes per day. Expansion to allow for 18,000 tonnes per day expansion is readily available by only a slight modification to the current plant, but the company has no current

(2006) plans to put this into effect. Water for the processing plant is acquired from the Long Lake processed kimberlite containment area. Approximately 8,250 m³ is used daily in the ore treatment process and returned to the lake with fine tailings. Potable water for the camp comes from Grizzly Lake (Werniuk, 1998; Boggis, 2000).

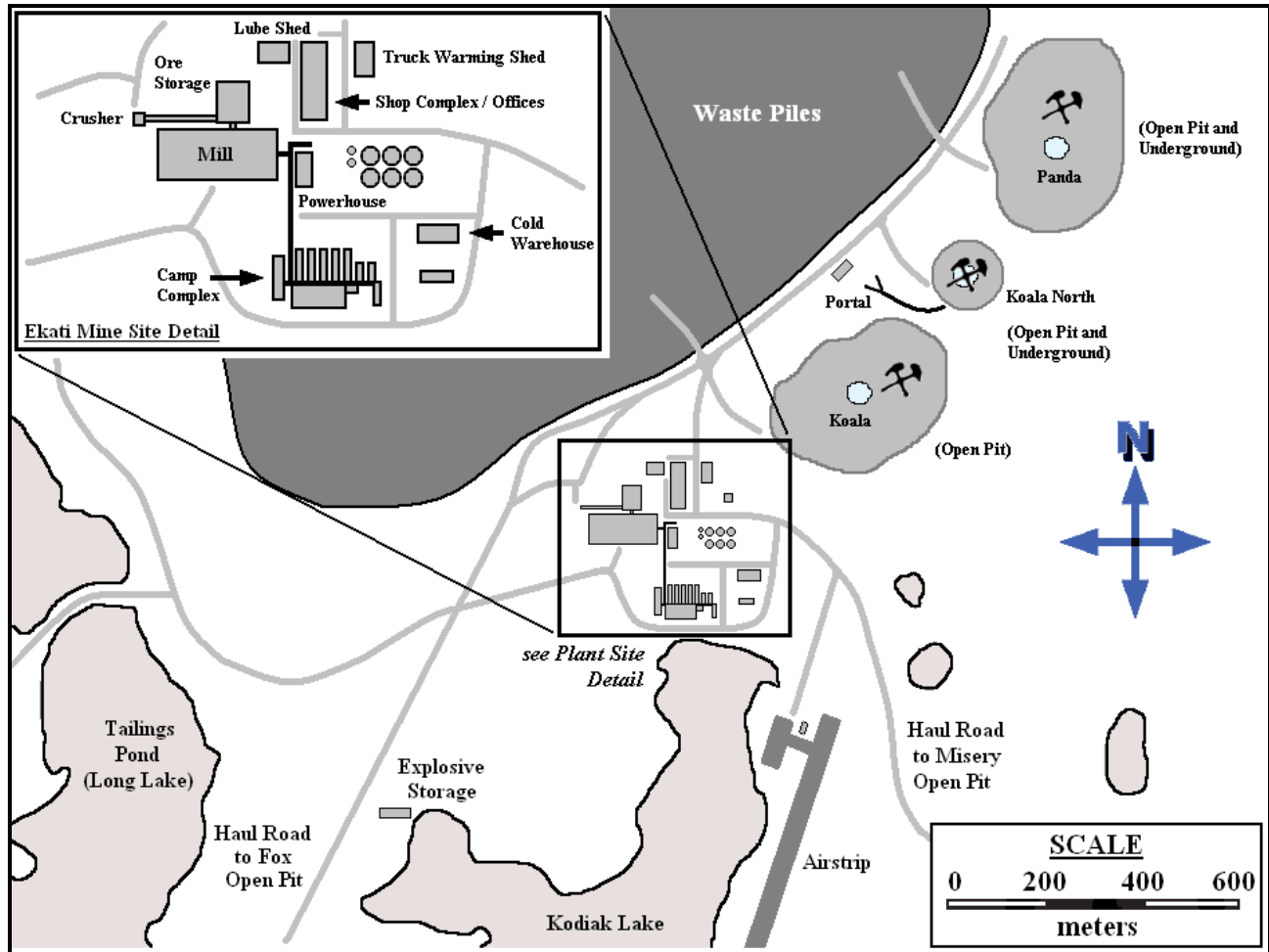


Figure 3. Ekati Mine property plan details, c.2003.

Power Plant

The power generating facility consists of a bank of five V16 Caterpillar 3616 diesel generators, each rated at 4.4 megawatts. Two of the engines operate at any given time and are maintained on a normal rotation to ensure proper maintenance can be performed. The fifth engine is kept as a standby unit during peak demand periods. Waste heat from the diesel engines are removed by means of glycol heat exchanger and is used to heat the mine services and accommodation complex (Boggis, 2000).

Shop Services

Most shop services are performed in the building known as the Truck Shop, a large 3-story structure housing the mechanical departments, heated warehouse, welding bays, automotive/tire repair shop, wash bay, machine shop, and offices for the engineering, human resources, environmental, training, network, and upper management departments. The truck shop itself has 10 main bays, four of which were added in summer of 2000. Each bay is large enough to accommodate the largest haul truck (CAT 793C) in the mine's fleet. A Lube Building, used to fuel up lube trucks, is adjacent to the Truck Shop and was built in 2001 (Boggis, 2000).

Fuel Storage

Fuel is stored in both 9 million litre and 4 million litre tanks that are situated in a central tank farm. 3 new large tanks were completed in 2000 and the site now has a capacity for approximately 77 million litres of fuel. Both the mine operations' mobile equipment fleet and power generation units run on 'arctic-grade' diesel (Boggis, 2000).

Explosives Plant

Due to the isolated location of the mine, the mine constructed a bulk emulsion manufacturing facility in consultation with the explosives supplier. A fully enclosed bulk storage and handling facility was constructed to handle 10,000 tonnes of Porous Prill Ammonium Nitrate. A 30 to 70% AN emulsion explosives blend is used in most blasting applications (Boggis, 2000).

Camp Complex

The main camp complex, completed in May 1997, consists of a two-storey camp support facility for dining, food preparation, laundry and storage. The dining hall has capacity for 250 people at any given time. The original complex consisted of 5 three-storey bunkhouse 'wings' each containing 75 single rooms (occupied by 2-persons during times of peak accommodation). Each suite has its own bathroom. Additional bunkhouse sections were added during 2002-2003 and total capacity is now 683 rooms in seven bunkhouse 'wings'. Older trailer units were phased out of service during 2003. An adjacent camp service building contains a small maintenance shop, the First Aid Clinic, an Emergency Response/Mine Rescue Station, and enclosed parking for an Ambulance, fire tender and rescue vehicle. Recreational facilities include a gymnasium, squash court, weight room, game room and several lounge areas. The plant and camp complex are all inter-connected with sheltered tunnels known as "Arctic Corridors", a series of prefabricated modules that join all the major buildings at the mine. They transport all the utilities such as power, water, sewage, and communication, together with supplying heated and safe walkways for workers commuting to work from the camp complex (Boggis, 2000).

Leslie Pipe Uneconomic

In August 1999, it was announced that the Leslie pipe had been removed from the reserve list at the Ekati Mine property. This was due to lower quality of diamonds that would not be marketable at the time. Focus was instead put on the possible development of the Sable, Beartooth, and Pigeon pipes. These pipes were expected to extend the life of the mine by three years.

Misery Pit Production

In September 2000, stripping was underway to expose the Misery kimberlite pipe, 30 kilometers south of the Ekati plant site. The Misery operation is managed by Nuna Logistics Limited who operates their own camp, equipment, and services. Population of Misery camp was estimated to be 100 employees in early 2001 but this was including the winter road crews. The camp has (2000) capacity for 107 persons. Fuel storage facilities at Misery in 2003 totaled million litres.

Misery Equipment

In 2000, equipment included a Cat 5130 excavator, three Cat 992 front-end loaders, eight Cat 777 haul trucks, two Cat 789 haul trucks, four drill rigs, and a portable crusher. In December 2000, BHP signed a 9-year contract with Kete Whii Limited for the transportation of ore from the Misery open pit to the milling plant, a 30-kilometer haul. The new company was a joint venture between Dogrib Treaty 11 and Akaitcho Treaty 8 Dene Bands. The deal was expected to produce 20 jobs. The company uses (2003) three Kenworth C500 haul trucks with trailers. Two trailers hold 75 tons, and three trailers holds 120 tons. Mining and trucking of ores from the Misery open pit began in October 2001 and recovery of diamonds began in December 2001. The Misery pipe provided a small but consistent level of production. In 2002, the ore grade mined at Misery pipe was higher than that of Panda pipe, but carat value was less. The Misery pipe is known for its multicolored diamonds.

BHP-Billiton Merger

In May 2001, shareholders of BHP approved of the merger between their company and Billiton Incorporated, a mineral resource company based in Britain. The new company is known as BHP Billiton and acquired BHP's 51% interest in production at the Ekati Mine. Further interest was gained in June 2001 when BHP Billiton acquired Dia Met Mineral's 29% share slice, earning itself 80% of the mine. Charles Fipke and Stuart Blusson have retained their 20% combined interests. In October 2002, it was estimated that BHP Billiton employed 650 employees at the mine site. An additional 700 people were employed by on-site contractors including those stationed at the Misery camp. The hiring target has remained constant at about 62% northerners, of which 30% are aboriginal.

Koala North Underground Project

Open pit mining of the Koala North pipe to 70 meters depth was completed during the summer of 2002. Here, the kimberlite pipe is only 75 meters in diameter. The Koala North pipe is too small to be mined entirely by open pit methods, so underground exploitation is the only alternative. Development of the new underground project at Ekati Mine began in February 2002 with the collaring of the decline portal. The portal is 5.5 x 5.5 meters and the decline

extends at -13% grade. The first kimberlite was mined in July 2002, with official opening ceremonies held on November 29th 2002. Commercial production from the Koala North pipe began in December 2002 and became North America's first underground diamond mine project.



BHP Billiton Diamonds

Figure 4. Koala North underground portal, 2002.

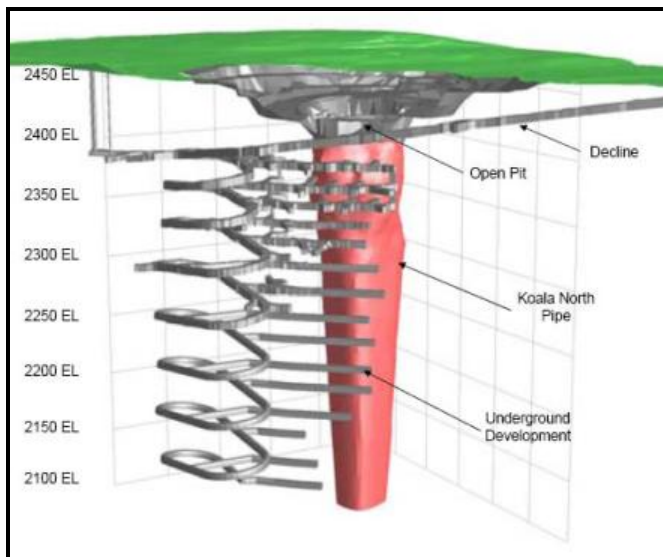


Figure 5. Isometric view of the Koala North mine model.

(from Jakubec et al., 2003)

The mining plan in 2002 involved driving the ramp to 2400 above mean sea level (AMSL) elevation where it would spiral down beside the kimberlite pipe. Access levels were to be driven from the ramp spaced at 15 to 20 meter elevation intervals into the kimberlite ore, from which all material would be removed. The mine was projected to extend to a vertical depth of 200 meters below the open pit, serving 11 levels to 2205 AMSL elevation.

Underground workings need extra support because kimberlite rock is very fragile. Workings are kept small, and walls and ceilings are supported by 2-3 meter long rock bolts, screen, and 50mm thick shotcrete. It is also important to keep the kimberlite ore dry so that it does not freeze. Good ventilation is provided to reduce the moist dust in the atmosphere, and ground water is controlled by sophisticated pumps (Werniuk, 2003).

Koala North Equipment

The following mining fleet was in use at the underground operation in 2002: 1-boom Tamrock longhole jumbo drill, 2-boom Boart jumbo drill, Elphinstone R1700 8-cubic yard scooptram (remote controlled), Procon 200 6-cubic yard scooptram (remote controlled), and an Elphinstone R1700 8-cubic yard scooptram (manual), Procon 200-6 cubic yard scooptram (manual), and two 40 ton Tamrock Toro 40D low-profile haul trucks (Werniuk, 2003).

The development of Koala North was a trial operation to prepare for future underground mining of the lower portions of the Panda and Koala pipes. The experience gained with the Koala North project proved invaluable to those future plans. Procon Mining and Tunneling Limited and Kete Whii Limited joined forces to participate in the contracted mining program on a 50/50 basis. The project employed 11 BHP workers and 65 Procon workers at the end of 2002. Pre-production reserves in the Koala North underground deposit were 1.5 million tonnes of ore grading 0-45 carats per tonne (Werniuk, 2003). Total Ekati Mine ore reserves at December 31st 2002 were 58,000,000 tonnes grading 0-9 carats per tonne (Mining and Exploration in the Northwest Territories Overview, 2002).

2003-2004 Operations

Stripping from the Koala pipe area began in July 2001 and production from the new open pit commenced early in 2003. Open pit mining of the Panda pipe was completed in June 2003. Final dimensions of the Panda open pit are 800 meters diameter and 300 meters depth. Pre-stripping development at the Fox pipe was undertaken during 2003-2004. Mining operations at the Misery pipe ceased at the end of 2003, but were resumed in July 2004. In 2004, the Koala pipe (open pit) was the primary source of ore with minor production from Misery pipe. The Beartooth and Fox pipes were under development in 2004, with the Beartooth pipe achieving production through open pit methods in November 2004. (Mining and Exploration in the Northwest Territories Overview, 2003 and 2004) In February 2004, workers at Ekati voted to join the Union of Northern Workers (a part of the Public Service Alliance of Canada, or PSAC). The new labour agreement under Diamond Workers Local X3050 went into effect June 2004, and in 2005 the union represented 450 workers at Ekati Mine.



Ryan Silke

Figure 6. *Open pit mining operations at Ekati Mine, 2002.*

Panda Underground

Open pit operations at the Panda pipe ceased in June 2003 and in May 2004 it was announced that underground development would commence immediately to achieve production by early 2005. A new underground ramp would be driven as a branch of the Koala North ramp drive. Reserves at Panda underground are expected to last until early 2007. Construction of surface facilities, driving the decline ramp, and blasting four ventilation raises was accomplished early in 2005. First ore production from the Panda underground deposit was made on April 26th 2005, with full production (2,600 tonnes per day) expected to commence early in 2006.

2005-2006 Operations

Lower grade ore was mined during 2005, compared to the mining of higher-grade sections of the Koala pipe in 2004. Ore was derived from the Beartooth pipe, the Koala pipe, the Fox pipe (which entered production in November 2005), the Panda underground, and stockpiled ore from the Misery pipe. In October 2005, 80 mechanics with the contractor Finning (Cat Diesel) went on strike as part of a union move throughout Alberta and the N.W.T. Finning brought in a small selection of replacement workers as a temporary solution. Meanwhile, negotiations for a contract between BHP Billiton and the miner's union commenced in 2005. Despite a 14-month period at the bargaining table, the two sides were unable to reach an agreement surrounding issues of pay increases, seniority, and vacation. On April 17th 2006, the union took strike action.

Mining operations continued at full capacity using existing mine employees, without replacement workers and through the support of 1/3rd of the union members who decided not to support the strike. The strike was settled later that summer. Ore production in 2005 was at a rate of 12,500 tonnes per day and it was planned to ramp up production rates to over 18,000 tonnes per day by 2007 (Mining and Exploration in the Northwest Territories, 2005 Overview).

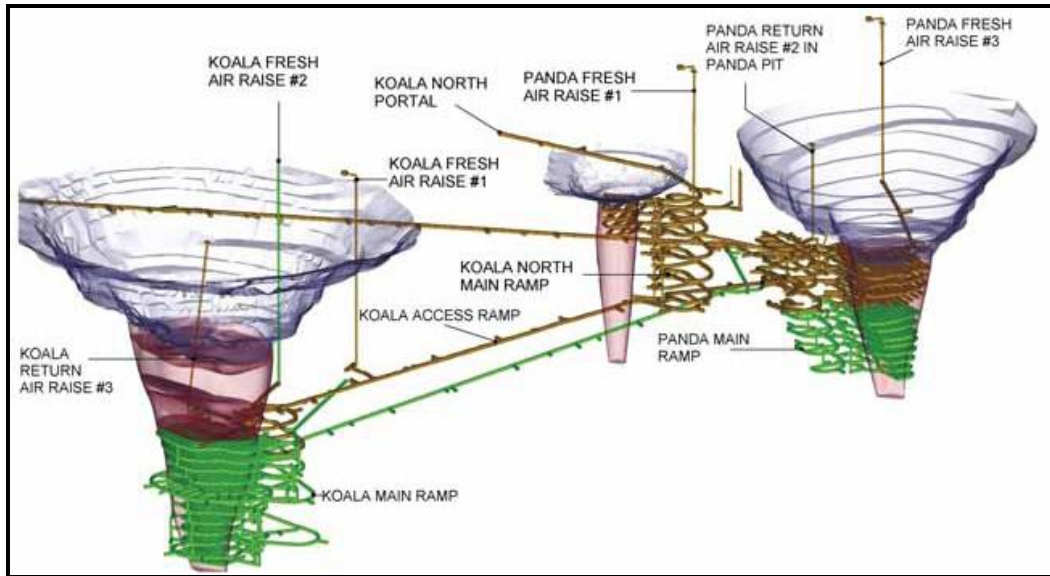


Figure 7. Isometric view of the Koala, Koala North, and Panda underground projects, 2006.

(courtesy BHP Billiton Diamonds)

Year:	Ore Milled:	Diamonds Produced:
1998	586,000 tonnes	278,000 carats
1999	3,175,500 tonnes	2,496,000 carats
2000	3,061,000 tonnes	2,533,000 carats
2001	3,311,000 tonnes	3,691,000 carats
2002	3,732,400 tonnes	4,975,000 carats
2003	5,570,000 tonnes	6,964,000 carats
2004	4,537,000 tonnes	5,096,000 carats
2005	4,437,000 tonnes	4,031,000 carats
2006	4,475,000 tonnes	3,148,600 carats
2007	4,329,000 tonnes	4,581,100 carats
2008	4,425,000 tonnes	3,563,700 carats
<i>Total to end of 2008:</i>	<i>41,639,000 tonnes</i>	<i>41,357,400 carats</i>

Table 3. Ekati Mine diamond production, 1998-2008.

(source: BHP Annual Water License Reports (1998-2002); BHP Billiton plc production reports (2003-2008))

In June 2006, BHP stakeholders approved a third underground mine at Ekati focusing on the Koala Pipe. The project is estimated to deliver 10.6 million tonnes of ore containing 9.8 million carats of diamonds over its 11-year life.

2007-2008 Operations

The period of 2007 to 2008 marked an increasing effort to cut costs at the Ekati Mine in order to prolong the mine life, part of BHP's Vision 2040 campaign. The goal is to lower operating costs to \$50 per tonne

processed. This period also marked the slow transition from open pit mining to underground mining. First production blast of the Koala Underground Project was in October 2007, and the Panda Underground Project was now in full operation. Open pit mining of the Beartooth Pit also began in 2007; open pit production during the year was from the Fox and Beartooth pipes only. Operations continued normally during 2008 from the Fox and Beartooth open pits, and from the Panda Underground Project. Full production of the Koala Underground Project at a rate of 3,300 tonnes per day is expected during 2009. As part of its cost reduction scheme, BHP introduced two new 'surface miner' equipment to the Fox pit operations in early 2009. The 'Wirtgen' 220SM model surface miner is designed for mining soft to medium-hard rock, using a mechanically driven cutting drum. With an approximate cutting speed of 12 meters per minute and a cutting depth of 20 cm, the Wirtgen miner can cut, crush, and load 600 tons of kimberlite per hour.

Ekati Mine diamond production to year-end 2008 is listed in **Table 3**. The mine has produced over 40 million carats to date.

Exploration Since Mine Closure

Not applicable.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 76DNE0021



EL-BONANZA

Bulk Sampled (Abandoned)

Silver

66° 00' 15" 118° 04' 15"

Years of Primary Development: 1934-1936, 1956-1957, 1965

Mine Development: incline shaft to 150' level, 2 levels; adit portal + winze

Years of Bulk Sampling: 1935

Bulk Sample: 6,513 lbs cobbled ore shipped = 30,175 oz Ag

Introduction

The El-Bonanza Mine is located in the Echo Bay area of Great Bear Lake, nine kilometers south of LaBine Point (Port Radium) on the west side of Miles Lake. It is 433 kilometers northwest of Yellowknife, NWT. An old road connects Dowdell Point on Great Bear Lake to the mine. It was a silver prospect dating back to the 1930s. The site was visited in July 2005 by the author and the mine buildings were intact.

History in Brief

Silver was discovered on this property in 1931 by Spud Arsenault, an employee of Eldorado Gold Mines Limited, and the 'Bonanza' and 'St. Paul' claim groups were staked. The eastern half of the 'Bonanza' claim group and the 'St. Paul' claims were acquired by the El-Bonanza Mining Corporation Limited in 1934. Eldorado Gold Mines Limited was a majority shareholder in the new company. Two short shafts were sunk on the deposit during 1934-1936, but development ceased before the mine could be put into production, primarily due to the drop in the price of silver.

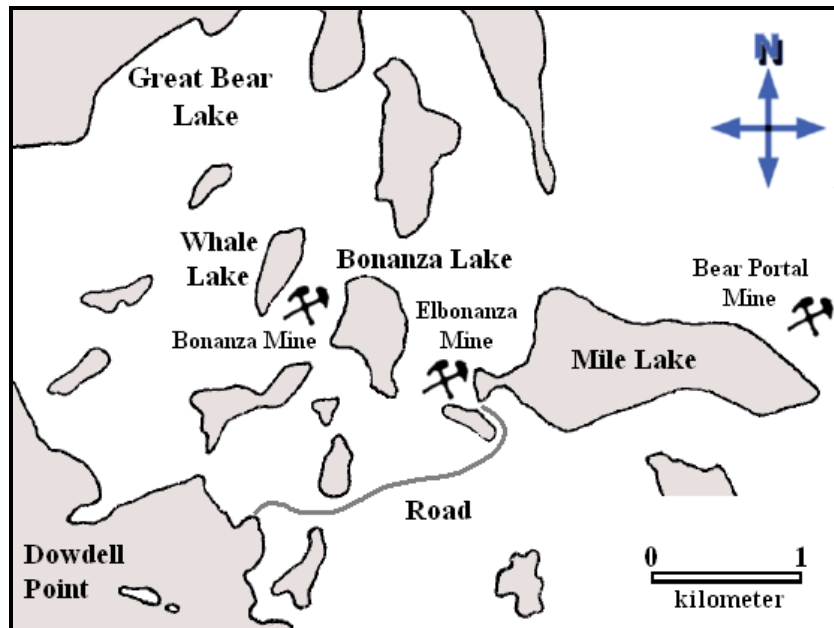


Figure 1. Dowdell Point Area, Great Bear Lake, showing the location of El-Bonanza Mine.

The El-Bonanza company was inactive throughout the 1940s due to its relationship to Eldorado, then a crown corporation, who expropriated the company in 1944 as majority shareholder. Control of the outfit was settled in 1950 when J.J. Gray purchased the company from Eldorado. Plans were made to re-develop the silver mine and investigate the possibilities of uranium ores. Mr. Gray was unable to raise sufficient funds to place the mine into production, but some exploration and development was undertaken. Work included a 1956-1957 underground development program on the 2nd level, mining of stopes and stockpiling of ore in 1965-1966, and numerous de-watering, sampling, and diamond drilling campaigns to prove an economic silver deposit. Last major work was between 1978 and 1980 when Echo Bay Mines Limited tested the silver potential of the mine. They failed to locate mineralization of economic importance, and no further work has been undertaken.

Geology and Ore Deposits¹

The El-Bonanza Mine is situated within the 1.87 Ga Great Bear Magmatic Zone, a part of the Bear Structural Province. Mineralization occurs within a narrow strip of altered volcanic and sedimentary rocks that are part of the Port Radium Formation of the Aphebian Labine Group. Uranium, silver, and cobalt-nickel arsenide veins in the area are spatially but not temporarily associated with intermediate plutons of the Mystery Island Intrusive Suite. The hydrothermal vein systems in the area trend generally northeast and are related to northeasterly striking transcurrent faults. The veins were formed between 1400 Ma and 1450 Ma ago.

The El-Bonanza showing occurs in a drift-filled depression that extends from the west end of Mile Lake approximately 1,500 feet west to Bonanza Lake. South of the depression lies the Dowdell Point granite. The northern contact with fine grained granodiorite is very irregular and appears to dip south at a low angle under the deposit. This may account for the extensive alteration of the intruded rocks. The mineralization of the El-Bonanza deposit occurs in two hydrothermal quartz-carbonate veins, the northerly Spud vein of approximately 800 foot length and the southerly Bonanza vein of approximately 1,000 foot length. The veins are hosted in two sets of fracture zones: one 300 feet long and six feet wide, mainly west-northwest trending, cutting siliceous volcanic rocks 300 feet north of the granite contact. The second zone lies in altered rocks 500 feet southwest of the first and strikes northwest. The fracture zones cut recrystallized and highly altered cherty argillites, siltstones and tuff. Fracture zones and veins are vertical to steeply dipping, generally greater than 65°. Quartz, calcite, native silver, argentite, chalcopyrite, bornite, niccolite, covellite, sphalerite, tetrahedrite, malachite and pitchblende are found in the veins.



Ryan Silke

Figure 2. *El-Bonanza #2 shaft headframe, July 2005.*

El-Bonanza Mining Corporation Limited (1934-1936)

The eastern half of the 'Bonanza' claims and the 'St. Paul' claims in their entirety was acquired by El-Bonanza Mining Corporation Limited in January 1934 through a deal in which Eldorado Gold Mines Limited, the original owners, were to receive a cash payment and majority stock in El-Bonanza (The Toronto Star, Feb. 7th 1934). D.M. Belec was dispatched to the property in March 1934 with a small crew to set up camp and prepare for mining developments. Early work through trenching totaling 3,000 feet was completed on three principle veins at the 'Bonanza' and 'St. Paul' claims. Three veins, the Bonanza, Spud, and Silver Lake, proved most encouraging during this assessment work, revealing high-grade silver showings with the great possibility of grade at depth (Scott, 1935).

In order to economically discern grades underground, 18 diamond drill holes were put down (1,100 feet total) on the Bonanza and Spud veins to shallow depths of 60 to 150 feet along a 700 foot strike length. The results varied in terms of grade and vein widths, but nevertheless it was thought to be an encouraging development. The decision to go underground was almost immediate (Scott, 1935).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Development of #1 Shaft and Adit

In the fall of 1934, an adit tunnel (70 feet below the surface exposure of the vein) was driven for a short distance to explore one of the westerly striking veins, known as the Bonanza #7 vein. At the end of this adit tunnel a 180 foot winze (#1 shaft) was put down. It is reported that a three inch vein of high-grade silver was intersected at a depth of 130 feet in the shaft. No lateral work was completed at the bottom of this shaft. Work stopped at this location to focus on other sections of the property. Grades and width of the vein found in the #1 shaft were reported to average two inches width with assays of 3,000 ounces per ton silver (Scott, 1935).

Sinking of #2 Shaft

Work moved to the #2 shaft early in 1935 on the eastern section of the Bonanza vein. The development of this shaft was interrupted in June 1935 due to the failure of the original power plant. At this time, the #2 shaft was down 84 feet on an incline of 72° south, forcing a decision to crosscut to the vein at this horizon. Spectacular high-grade silver was intersected on the 1st level, at 84 feet depth. Where the crosscut intersected the vein, a width of 14 inches of heavy silver mineralization was encountered, with additional visible silver over a five feet width. Drifting then started in both directions along the vein, and the high-grade section was traced for 34 feet with widths from 8 to 18 inches of native silver. Ore was stockpiled on surface and composite samples assayed 880 ounces per ton silver. The 1st level was ultimately developed by a 90 foot crosscut and 450 feet of drifting, extending westerly towards the #1 shaft (Scott, 1935).

A new air compressor was installed and sinking resumed by August 1935. By November 16th, this shaft was 150' long along the incline and the 2nd level was undergoing lateral work. A third shaft was sunk about 30 feet on the Spud vein north of the #2 shaft during this time as well. Lateral exploration on the 2nd level of the #2 shaft continued into 1936. Approximately 550' of drifting and crosscutting was completed (Scott, 1935).

Bulk Shipment

6,513 pounds of high-grade silver ore were hand picked from the #2 shaft's 1st level and then shipped to Trail, B.C. for smelting during 1935. Silver recovered was an incredible 30,175 ounces (Pasioka, 1977). Another 50 tons of high-grade silver ore were stockpiled at the shaft workings in September 1935 (The Northern Miner, Sept. 19th 1935).

Power Plant

Power was supplied by a Tangye diesel engine and 270 cubic feet per minute Fairbanks-Morse air compressor until August 1935, when a replacement unit was purchased from White Eagle Silver Mines Limited (a portable Ingersoll-Rand engine and 370 cubic feet per minute compressor). Both shafts were fitted with small air hoists (Ingersoll-Rand and Sullivan types), and a diesel generator powered the camp. Fuel was stored on the shores of Great Bear Lake (Scott, 1935).

Employees

A.W. "Sandy" Scott was mine manager in charge of operations during 1934-1936, assisted by D.M. Belec. Engineers and staff of Eldorado Gold Mines Limited consulted on the operation since the company was a major shareholder. 20 men were employed in 1934-1935.

Production Plans

In September 1935, it was reported that 50 tons of ore was stockpiled awaiting shipment or treatment in a mill. The plan was to ship ore to the nearby Eldorado Mine for milling, hauling the ore by tractor and sled at a cost of \$3 per ton. Policy at the time was to utilize the existing infrastructure available at the Eldorado Mine and, if warranted, install a separate milling plant at the Elbonanza workings at a later date. A 150-ton ore bin was constructed at the mine in preparation of this operation (Scott, 1935).

Operations were scheduled to commence during the winter of 1935-1936, using both underground workings to supply mill feed. Equipment was available to service both shafts at the same time if a production decision was made. A drop in the price of silver to about 25 cents per ounce during the following years and the government control on uranium deposits resulted in the closure of the mine throughout the 1940s. It does not appear as though the company released any statement of ore reserves at this time.

El-Bonanza Mining Corporation Limited (1956-1957)

Work in the early 1950s prepared for future underground exploration. The shaft was partially de-iced, some new equipment was installed, sampling of ore stockpiles was performed, and new buildings were erected. The assay tests showed good uranium oxide values and Geiger tests also indicated high radioactivity of the ores. By 1955, indications

that the price of silver may rise and the interest in uranium resulted in a new program of underground work. Starting in September 1956, a 580 foot crosscut was extended from the 2nd level workings north towards the Spud vein. By November, this heading had advanced 448 feet, and in February 1957 the vein was reached. Total drifting in this area consisted of over 300 feet advance along the vein in both directions. Equipment was purchased from the Contact Lake Mine operation in 1955. Known equipment included of a Cat D-13,000 diesel engine, a Gardner-Denver 365 cubic feet per minute air compressor, and an 8x6 Canadian Ingersoll-Rand air hoist for shaft operations. Work was under the direction of consultants John Anderson-Thompson and Norman W. Byrne, and a small crew was employed. Lower level veins were found to be narrower than the surface or 1st level showings. Native silver was identified on the 1st level workings but the mineralization was reported as having no economic importance. Assaying of the 2nd level workings also suggested the same. No important uranium orebodies were disclosed (Anderson-Thompson, 1956; Johnston, 1957).

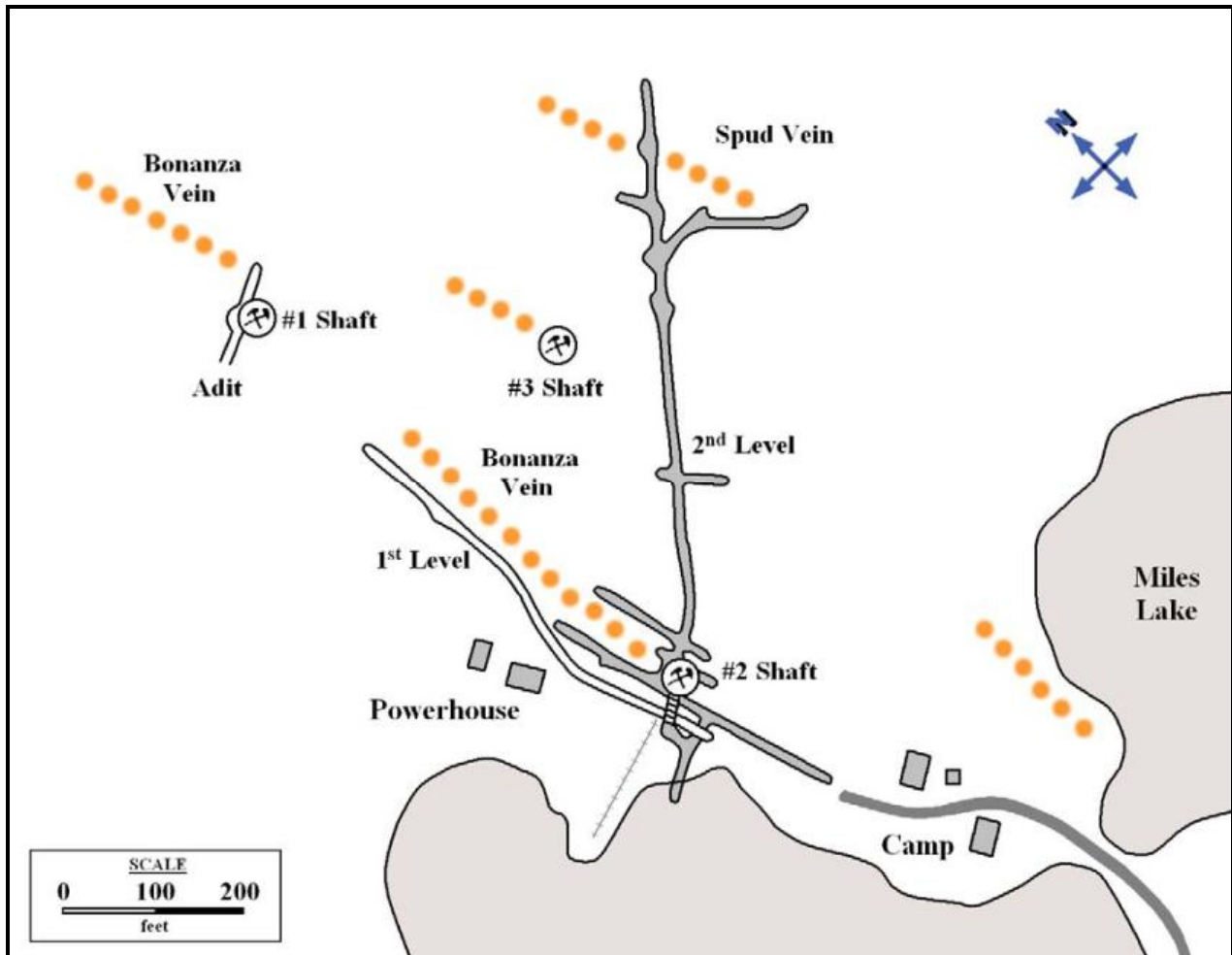


Figure 3. El-Bonanza Mine surface and underground plan, c.1981.

El-Bonanza Mining Corporation Limited (1965)

After a ten-year period of inactivity, the company attempted to put the El-Bonanza Mine into silver production in the mid 1960s. The underground workings were de-watered in 1965 under the direction of George Midgley, with eight men employed during August 1965. Work commenced in March 1965 in de-watering and de-icing the #2 shaft, which was frozen to a depth of 80 feet. Mining of the 1st level stopes began in mid-June and de-watering made the 2nd level accessible by the end of August. A total of 315 feet of drifting was completed on the 1st level. About 300 tons of ore was added to a stockpile from the 1st level mining, now estimated to contain 1,000 tons of material. It was planned to either install a small milling plant at the mine or to ship ore to the nearby Echo Bay Mine. The company was unable to organize the required financing to put in its own mill, and Echo Bay Mines Limited was not interested in custom milling. Work again ceased in 1966 (Thorpe, 1966).

Exploration Since Mine Closure

No further mining developments have been undertaken at the El-Bonanza Mine, but exploration to establish an economic silver orebody continued into the 1980s. A 1968 report by Precambrian Mining Services Limited acknowledged the poor mineralization encountered during the 1956-1957 program, but indicated that good ore had been found on the 2nd level in an ore shoot within the Bonanza vein over a 40 foot length. The extent of this ore had not been fully investigated, but the report suggested 1,000 tons grading 50 ounces per ton silver was a reasonable estimate. Grade of the silver ores were believed to decrease with depth, as had been established at the Eldorado and Contact Lake Mines. However, the pitchblende ores had a tendency to increase with depth at these mines, and the report recommended more work to delineate a uranium orebody at the El-Bonanza Mine. It also recommended additional diamond drilling in the eastern section of the Bonanza silver vein (Gill & McConnell, 1968).

In 1969, the #2 shaft workings were once again de-watered under the direction of Robert Portelance. Five diamond drill holes totaling 900 feet were completed in the 2nd level crosscut area. The breakdown of a pump resulted in the abandonment of the property again, and was said that no visible silver was encountered in the drilling. In 1972, the surface stockpile was reported to contain 1,000 tons grading 50 ounces per ton silver and broken ore available in underground stopes was 1,500 tons grading 70 ounces per ton silver (Sanche, 1972; Pasioka, 1977).

In 1978, Echo Bay Mines Limited leased the property and carried out a major exploration program. The Bonanza and the Spud veins were diamond drilled in 26 holes from the surface. In 1980, the #2 shaft was de-watered and the underground workings were mapped, sampled, and probed by additional diamond drilling. No economic silver was encountered, and as a result no further work was recommended on the property (Moffett, 1981).

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ELDORADO

Producer (Remediated)

Uranium, Radium, Silver, Copper

66° 05' 00" 118° 02' 00"

Years of Primary Development: 1931-1960, 1974-1981

Mine Development: 125' v. shaft, 262' v. shaft,

Years of Production: 1933-1940, 1942-1960, 1975-1982

Mine Production: 1,366,602 tons milled = 13,402,000 lbs U₃O₈, 450g Ra, 13,371,382 oz Ag, 2,389 tons Cu, 140 tons Ni, 250 tons Co, 8 tons Pb

Introduction

The Eldorado Mine is located in the Echo Bay area of Great Bear Lake, on LaBine Point (Port Radium). It is 440 kilometers northwest of Yellowknife, NWT and 265 kilometers east of the community of Deline. Eldorado was the first mine in the Northwest Territories. Radium and silver were the original metals of interest, and production began in 1933. Uranium was the target of work from 1942-1960, followed by renewed interest in silver and also copper from 1975 to 1982 when the mine closed. Port Radium was the name of the settlement which was established at the mine site. All buildings except an old cabin and the Crossfault Lake headframe were razed following closure in 1982. Final remediation was completed by the government in 2007. The site was visited very briefly in July 2005.

Brief History

The discovery of the historic mineral deposits on the eastern shores of Great Bear Lake has been credited to the vision and effort of one man: Gilbert LaBine. The showing he found was that of pitchblende, and he staked the first two claims near Echo Bay in May 1930. Samples of this ore were sent for metallurgical testing where it was found possible to extract a radium product. Mining operations began and in 1933 the property was producing its first radium and silver concentrates.

Eldorado Mine closed in 1940 due to a collapse in the worldwide radium market, but was reopened in 1942 to produce uranium ores for the United States military. The Canadian government expropriated control of the Eldorado company in 1944 for war-time security purposes. Uranium products, used to construct nuclear weapons, were the focus of operations at the Eldorado Mine until 1960, when the uranium ore body was mined out.

Silver became the focus of development in the 1970s when the underground workings at Eldorado were re-opened by Echo Bay Mines Limited. Previously, the camp and mill plant were being used by that company to process ores from the Echo Bay Mine. Production of silver ores from Eldorado Mine took place between 1975 and 1982, when depleted ore reserves closed down the operation for a final time.

Geology and Ore Deposits

The area is situated within the northern half of the Great Bear Magmatic zone, near the western margin of the Wopmay orogen of the Bear Structural Province. The supracrustal component of this Archean complex is the McTavish Supergroup mainly composed of volcanic and minor sedimentary rocks. The McTavish Supergroup is up to 10 km thick, has been metamorphosed under subgreenschist facies conditions and rests on a deformed and metamorphosed 1.92 Ga sialic basement.

The McTavish Supergroup has been subdivided into three groups separated by unconformities: The LaBine, the Sloan, and the Dumas Groups. The LaBine Group is subdivided into the sedimentary Port Radium Formation and the volcanoclastic Mile Lake member of the Echo Bay Formation, itself subdivided into the lower Mine Series and the upper Tuff Series. Sediments and volcanics dip gently to the southeast and are extensively intruded by at least five tabular bodies of feldspar porphyry that have caused deformation and recrystallization of the sediments. Massive granite of the Hogarth Pluton of the Great Bear Batholith lies to the west; its contact trends northeast and dips 70° southeast.

Major faulting and hydrothermal veining occurred between 1,450 and 1,400 Ma. The faults strike northeast and dip 65° northwest, and cut Proterozoic diabase dykes and one diabase sill in the mine area. The diabase dykes, however, do cross cut the earliest quartz veins.

The Mile Lake member of the Echo Bay Formation includes the Mines Series and the Tuff Series. The Mine Series is a sequence of pink and dark green to black, micro crystalline, thin bedded tuffs with interbedded lenticular beds of

massive plagioclase tuff. Bedding is accentuated by ferromagnesian minerals, commonly hornblende and magnetite. Pink coloration is due to dispersed hematite. Diopside, garnet and scapolite occur locally. Most of the Mine series rocks contain at least 20% metasomatic minerals. The overlying Tuff series is pink to brown, microcrystalline, unevenly thin bedded, and characterized by the presence of dacite fragments and occasional dacitic flows. Principal mineral is oligoclase with some albite, and quartz. The fragmental beds are up to 15 meters thick and contain angular, pink fragments in tuffaceous matrix. Near the base is a 100 foot thick bed of conglomerate. Below the Mine Series is the Cobalt Island Formation outcropping on the western tip of Labine Point and on Cobalt Island. The tuff contains 10 to 35% carbonate in rhombs and patches, interbeds of grey quartzite, and also two limestone beds near the base.

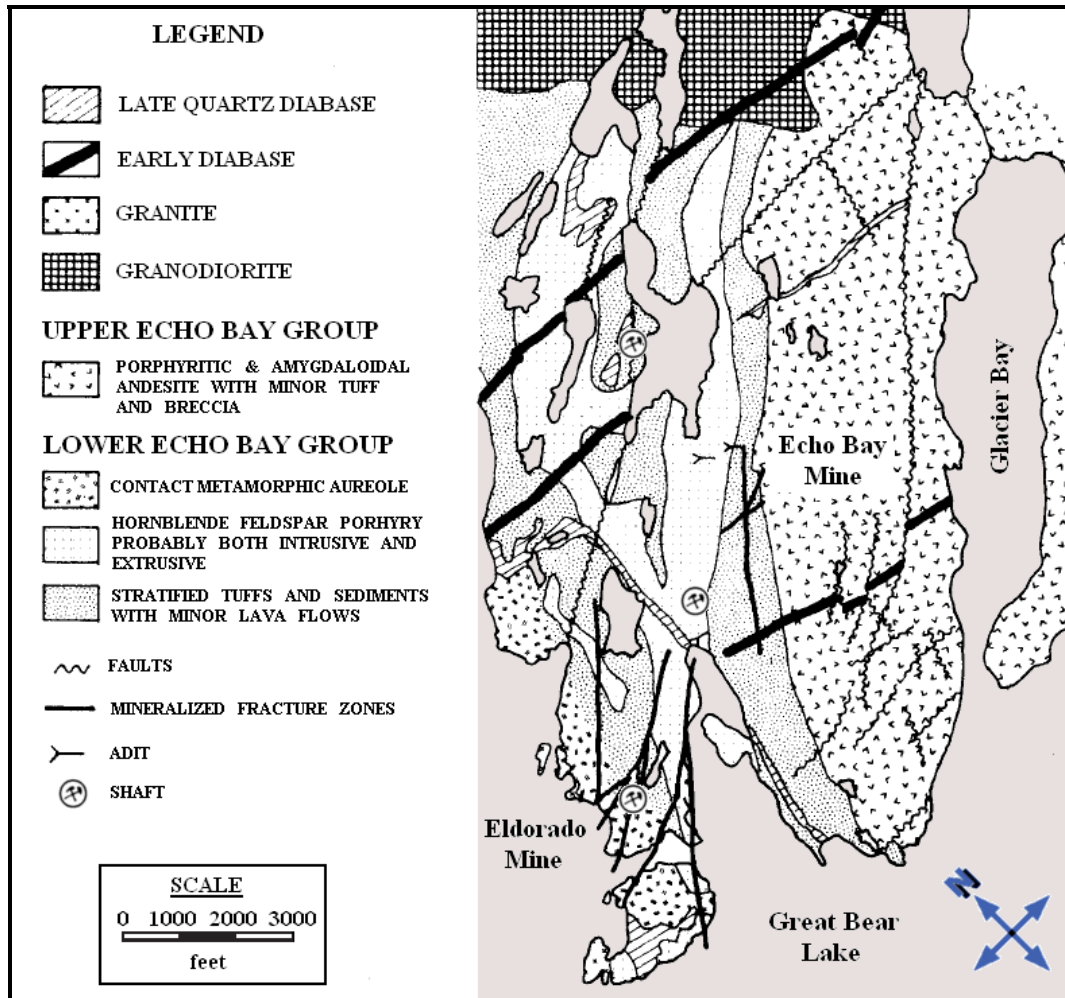


Figure 1. Eldorado Mine area geology.

A sill-like feldspar porphyry body strikes northerly from LaBine Bay to McDonough Lake and beyond. It dips easterly at 20 to 30°. Mine Series sediments to the west have been intruded by numerous small fingers, apophyses, dykes, and irregular bodies of feldspar porphyry. The porphyry is usually maroon or greyish-green to buff in color and composed of 2 to 3 millimeter long, light grey to white oligoclase and dark green hornblende phenocrysts in a matrix of aphanitic oligoclase, hornblende, quartz and magnetite. Phenocryst contents varies from 15 to 50%, and locally hornblende is absent. Chloritization of the phenocrysts is common. Alteration of minerals in the matrix include diopside, magnetite, garnet, and scapolite.

Early Proterozoic, massive, pink, coarse-grained, equigranular biotite granite of the Hogarth Pluton (of the Great Bear Batholith) lies to the west and crops out on Cobalt Island and on LaBine Point. The mineral content is 25% quartz, 35% oligoclase, 25% orthoclase, 10 to 15% hornblende and minor biotite. The contact is fairly regular, striking northeast and dipping at 70° southeast towards the mine workings. It truncates all of the above formations and reduces the shear zones to sparsely mineralized, tight fractures.

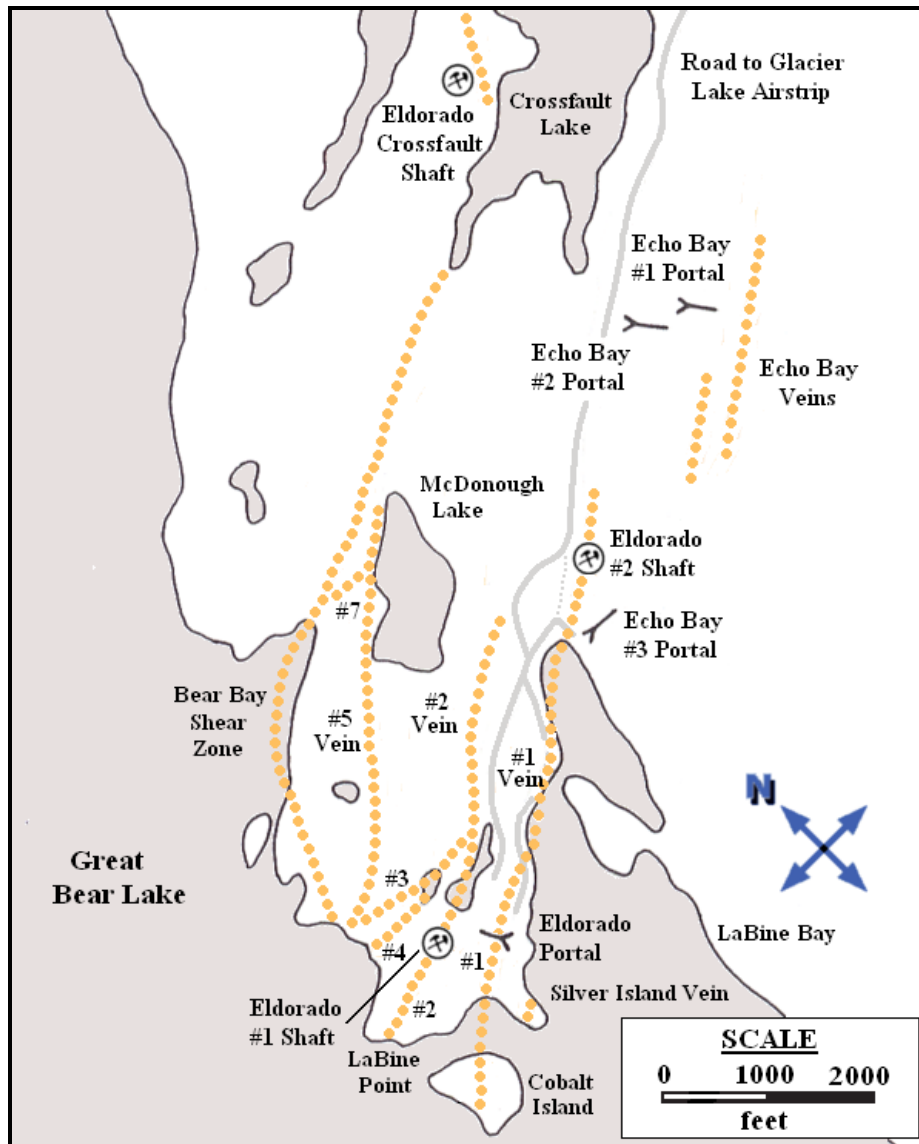


Figure 2. Eldorado Mine property map showing major ore deposits and shaft locations. Includes Echo Bay Mine.

Two diabase dykes and a sill are exposed at the mine. The dykes are 20 to 30 feet wide, dip steeply and strike northwest. The southern diabase cuts across the center of LaBine Point and is well exposed in the mine workings. It crosscuts the #1 vein, has a chilled contact in the vein zones, but was brecciated by later movement in the vein zones and healed by the vein mineralization. The sill is probably younger than all but the latest stage of vein mineralization.

The ore deposits occur in a system of vein-filled fractures and shear zones that trends northeast along LaBine Point and crosscuts stratigraphy. Primary vein minerals are hematite, pitchblende, copper-sulphides, cobalt-nickel arsenides, and native silver. Most orebodies are situated in a 5,000 foot long zone extending from the tip of LaBine Point to the northeast arm of Crossfault Lake. The principal ore deposits are the #1 vein and the Bear Bay Shear zone, encompassing a lenticular area 4,200 feet long and 2,000 feet wide. The rocks between them are extensively dislocated by secondary shear and tension zones. The #2 vein was explored over 5,000 feet strike length and 1,800 feet vertical depth. At LaBine Point both veins are in Port Radium strata then cross-cut into biotite-granite, into granodiorite and finally into Cameron River Bay Formation at the north end of the zone. Other vein systems include the #3, #4, #5, #7, #8, and the Silver Island veins. The Silver Island vein is chiefly silver with little or no pitchblende.

The orebodies are also spatially related to the porphyry bodies intruding the sediments. Where the veins intersect thick porphyry bodies ore is either not deposited or is deposited only close to the footwall contact of the porphyry. Diabase dykes and vein material in places fill the same openings and where adjacent the vein tends to be of higher

grade. Fillings along fractures and shears constitute the main ore-bearing structures. Replacement of the wall rock of these vein zones by some gangue and metallic minerals is not uncommon.

Eldorado Gold Mines Limited (1931-1940)

Following the staking of the 'Cobalt' group of claims, the LaBine prospecting party quickly recorded a number of other claims in the Echo Bay area to protect the main showings. On the 'Cobalt' claims, three main vein systems (the #1-3) were identified on the north side of LaBine Bay. The #1 vein carried high concentrations of pitchblende, the #2 vein was very rich in silver and pitchblende, and the #3 vein appeared to be plentiful with pitchblende and cobalt. A fourth vein called the Silver Island vein (or D-vein) showed very high silver and cobalt content (Bothwell, 1984).

A sample sent to the U.S.A. was found to contain 60% uranium oxides (U_3O_8). However, the main mineral of interest was not the ore's uranium content but its radium content. The world's supply of radium was very low and the price was very expensive due to its limited production in South Africa. If the Great Bear Lake property were to enter production, it would become North America's cheapest supply. Radium was used in the medical industry as a treatment for cancer (McNiven, 1967).

The #2 vein showed the most promise and was the target of first major developments through trenching and stripping. During 1931, a ten man crew headed by Gilbert LaBine hand-cobbed the first shipment of pitchblende ore, which was mined from the #1 vein. This was barged up the Mackenzie River for testing in Ottawa laboratories (The Globe and Mail, Feb. 25th 1932). A total of 20 tons of ore were tested in the spring of 1932 with assays of 39% to 47% U_3O_8 containing an estimated 120 to 140 milligrams of radium salts. Only minor silver was recovered from this ore (300 ounces) because the ore was taken from a section of the vein that was not high-grade silver (The Toronto Star, May 4th 1932). Work at the site ceased over the winter of 1931-1932.

First Underground Work

The camp was reopened in March 1932 and during the summer a crew of 20 men were at work preparing a log cabin camp, trenching the #1 and #2 veins, and initiating underground development. Charles LaBine was in charge of work during the summer of 1932. A 50 horsepower Junkers engine was brought to the site to operate drills for the start of an adit portal, designed to intersect the #2 vein at a depth of 90 feet below the surface exposure. The adit was driven 390 feet to intersect the #2 vein, which was drifted upon for a distance of 600 feet or more. Unusually hard rock formations were encountered while driving the adit (The Toronto Star, Jan. 12th 1933).

In 1933, Emil Walli was placed in charge of mining operations at Eldorado as mine manager. Drifting on the #2 vein revealed an ore shoot 200 feet long and 4 feet wide with assays of 30 ounces per ton silver and important pitchblende content. Drifting in a small vein parallel to the #2 vein did not disclose any commercial deposits (The Toronto Star, Apr. 25th 1934). Total underground development for the year 1933 was 828 feet of drifting on the adit level, along with 253 feet of crosscutting and 60 feet of raising (McNiven, 1967).

For the first year or two of mining operations, a system of hand cobbing ore for shipment was used due to the richness of the ore. As mining underground progressed in 1932-1933, it became apparent that this trend would not continue, and a form of mineral concentration would be required. It was therefore advisable to erect a milling plant at the Eldorado Mine. A refinery to produce uranium and radium materials was also needed, and this was to be built at Port Hope, Ontario (McNiven, 1967).

More ore was shipped from the Eldorado Mine during 1932. A total of 47 tons of ore was barged up the Mackenzie River to Waterways, Alberta where it was sorted into silver and pitchblende batches: 11 tons of silver ore was freighted to Trail, B.C. (recovery of 38,433 ounces silver reported) and the remaining 36 tons was forwarded to the future Port Hope refinery site (The Toronto Star, Oct. 15th 1932; Feb. 22nd 1933). In 1933, another shipment of silver ore was made to Trail, B.C. A grade of 4,329 ounces per ton silver was reported from about 5 tons of ore shipped (The Toronto Star, Nov. 28th 1933).

Original Mill

The first mill of 50 tons per day capacity went into operation late in 1933 with the function of producing pitchblende-silver and silver-copper concentrates. Actual functional rate of the plant was around 25 tons per day. Ore from the coarse ore-bin passed over a stationary grizzly, set at $\frac{3}{4}$ inch, onto a sorting table where high-grade ore was removed. Lower-grade ore requiring concentration was crushed in an 8 inch x 12 inch Dodge crusher set to crush to $\frac{3}{4}$ inch size, which was passed into the main milling circuit by bucket elevator. The fine ore was fed into a 4 foot x 8 foot Allis-Chalmers rod mill, then through a double-deck vibrating screen equipped with 12-mesh and 20-mesh screens.

The plus 12-mesh was sent back for milling, while other material was concentrated on two Wilfley tables. Three products were formed on these tables: a pitchblende-silver concentrate that was removed to a dryer, a middling that was returned to the tables, and a tailing that passed through a conditioning tank and into a two-cell Denver flotation unit, from which the silver concentrate was recovered. The mill was powered by a Ruston-Hornsby diesel engine by line shaft and v-belt. Eldorado pitchblende concentrates were sacked for shipment by barge across Great Bear Lake, over the Bear River portage, and up the Mackenzie and Slave rivers to railhead for transport to Port Hope, Ontario. Silver-copper concentrates were shipped to American smelters (in Tacoma, Washington) for refining (Walli et al., 1938).

Port Hope Refinery

The Port Hope refinery was built during late 1932-early 1933. The first radium was produced in May 1933, and uranium oxide salts, produced as a byproduct, were largely stockpiled, although some of it was sold to the ceramic industry. In the first year of operations 1933, the Port Hope refinery treated 55 tons of concentrates to produce 3 grams of radium salt and 35,000 pounds of uranium salts (The Toronto Star, Apr. 25th 1934).



NWT & Nunavut Chamber of Mines photo

Figure 3. *Eldorado Mine, 1939.*

Transportation

The company also went into the transportation business to ensure transportation links with the outside. In 1936, the Eldorado company purchased control of Northern Transportation Limited and financed a fleet of tugs and barges into service on N.W.T. rivers. All heavy machinery and most supplies were brought to Great Bear Lake by water. Outgoing barges carried mill concentrates to the railhead at Fort McMurray, Alberta. The company also bought an aircraft for transporting concentrates and emergency supplies during the winter season (McNiven, 1967).

1934 Shaft Sinking

The adit level only provided enough ore for the first few months of milling so in late 1934 a two-compartment vertical winze (#1 shaft ¹) was sunk 257 feet (to 347-foot elevation below the surface) below the adit level (1st) to open up levels at 200-foot (2nd) and 340-foot (3rd) vertical depth. A 2-cylinder air hoist was used (McNiven, 1967).

¹ This winze was later (1945) extended to the surface as a regular shaft, so to avoid confusing nomenclature this opening is identified as a "shaft" during this period of operations.

Underground development in 1934 consisted of 467 feet of drifting and 55 feet of crosscutting on the adit or 1st level, and 36 feet of drifting and 27 feet of crosscutting on the 2nd level, plus 257 feet of shaft sinking at #1 shaft. A station was cut on the 3rd level but no development was undertaken. Surface construction in 1934 included a powerhouse building with 196 horsepower Ruston-Hornsby diesel engine driving a 900 cubic feet per minute Ingersoll-Rand air compressor (McNiven, 1967).

Upgraded Milling Plant

The mill was shutdown in October 1934 to undergo major modifications and repairs and was back online in January 1935 with a capacity of 75 tons per day. Although much effort had been made to solve some of the operational difficulties, the plant was unable to keep a steady rate during 1935-1936 (Walli et al., 1938; McNiven, 1967).

Development was focused on the west and east drifts of the 125-foot level during 1935 where good ore was encountered, but late in the year development and stoping proceeded on the 340-foot level and small amounts of ore were mined. Mining operations in 1935 focused on the #2 vein on the 2nd and 3rd levels. Four stopes were opened on the 2nd level and two ore shoots were found on the 3rd level (The Northern Miner, July 9th 1936). Total underground development in year 1935 consisted of 954 feet of drifting and 195 feet of crosscutting on the 2nd and 3rd levels (McNiven, 1967).

#2 Shaft Development 1935

Other development included the #2 shaft, located one kilometer northeast of the main workings. The shaft was sunk using handsteel methods to 50 feet early in 1935 to explore a high-grade section of silver on the Gulch vein, a northerly extension of the #1 vein. A small amount of lateral work (49 feet) was done on the 50-foot level, but development was suspended pending the acquisition of further power requirements. Some early assays of this vein indicated 50 ounces per ton silver. Another assay gave an average of 90 ounces per ton silver over a drift width for a length of 30 feet east of the shaft (The Northern Miner, May 2nd 1935; July 4th 1935; May 13th 1937).

Level:	Depth below Adit:	Depth below surface:
1 st (adit)	0	90'
2 nd	125'	200'
3 rd	250'	340'
4 th	375'	465'
5 th	500'	590'
6 th	650'	740'
7 th	800'	890'

Table 1. Summary of the elevation of underground workings during the 1930s.

Different literature from the 1930s-1940s uses either footage depths when describing the underground levels, so the above table is provided to the reader for reference and to avoid confusion.

Construction of permanent plant and camp facilities were conducted during 1934 and 1935. Previously, crews lived in a number of log structures, but an increased crew of 90 men required the erection of a 2-story bunkhouse and a change house in 1935. The settlement came to be known as Port Radium. Other construction included a cafeteria in 1936 and new assay lab in 1937 (McNiven, 1967).

Employees 1935

The following mine staff were reported in the summer of 1935: E.J. Walli, manager; J.P. Ryan, assistant manager; Bud Wilson, mine captain; R. Bertrand, general foreman; D.G. Smith, mill superintendent; Bob Bowen, Dave Trottier and Gordon Spence, mill foremen; Ted Blundell, radio operator; J.D. Belec and Ray Martin, assay office staff, James St. Marie, cook, and Dr. Burns at the Cameron Bay hospital (McNiven, 1967).

Mining Operations

Ore was mined using shrinkage stope methods, most suitable for the deposits narrow high-grade veins. Vein walls were easily defined in most cases and the wall rock was competent requiring little timber support. Ore was hand-trammed in one ton ore cars. The cars were caged and hoisted to the adit level and again trammed to the mill. Permafrost was found to extend 350 feet below surface, so for mining within this zone, hot water or salt water was used for drilling purposes. Considerable labor was required in keeping drifts free of ice buildup and in re-blasting frozen chunks of ore. There was no mechanized equipment in use at the mine during these years: all ore was hand trammed in small end-dump ore cars. (Walli et al., 1938)

In 1936, the #1 shaft was deepened to 580 feet, and development of the #2 vein was from two more levels (465-foot and 590-foot depth). An enlarged hoist (Canadian Ingersoll-Rand 36 inch x 24 inch 2-drum electric, 75 horsepower) was installed in the underground hoist room. It was planned to open up the #1 vein for underground exploration at these new horizons. Total underground development during 1936 consisted of 1,031 feet of drifting, 635 feet of crosscutting, 255 feet of shaft sinking, and 166 feet of raising. Broken ore reserves at December 31st 1936 totaled 8,177 tons, unbroken 9,332 tons, and indicated ore 8,119 tons. Additional power plant units were secured late in 1936 to provide for the expanded operations (The Northern Miner, Apr. 8th 1937; McNiven, 1967).

Expanded Mill

The mill was expanded to allow for production of 100 tons per day during late 1935-early 1936. Ore was passed over a bumping table where high-grade material was removed. The remainder of the ore was crushed in a 9 inch x 15 inch Blake jaw crusher and processed in the 4 foot x 8 foot Allis-Chalmers rod mill. An elevator discharged over a vibrating screen where oversize entered a James jig to produce a coarse and a hutch concentrates that were dried for sacking. Undersize material from the vibrating screen was distributed over three Wilfley tables to produce a high-grade pitchblende concentrate, a middling, and a tailing. The tailings were classified in an Akins spiral unit, operating in closed circuit with a 4 foot x 6 foot Allis-Chalmers ball mill. Classifier overflow was fed directly into the Denver flotation unit where a silver-copper concentrate was removed. A tailing from the flotation process was sent over shaking tables and then to a bank of four Denver flotation cells to remove a second silver-copper concentrate (Walli et al., 1938).

In 1937-1938, changes were made to increase recovery of pitchblende and silver. These included the enlarging of openings on the primary screen, the additional of a second screen, the increased use of jiggling, a new flotation unit, and the recycling of jig tailings into the grinding circuit (Walli et al., 1938). Other major changes late in 1938 included the addition of a secondary crusher (two foot Symons cone crusher), a vibrating screen, magnetic pulley, new conveyors, and other auxiliary equipment (McNiven, 1967).

Work in 1937 was concentrated on developing and mining the 4th and 5th levels within the western extensions of the #2 vein. It was reported that the #2 vein was uniform in character and grade at the lower levels of the mine. The western sections of the vein were also reported to have stronger pitchblende values than in the eastern sections. Crosscutting south towards the #1 vein at the 590-foot (5th) level was undertaken in the summer of 1937. Drifting encountered 70 feet of high-grade pitchblende, the most important pitchblende ore body discovered to date. Raising began on this ore shoot. Immediately, a crosscut was driven from the 3rd level to intersect the upward extension of the #1 vein. By year-end, two continuous ore shoots had been found on each level with lengths of 130 feet and 70 feet (The Northern Miner, April 8th 1937, May 5th 1937, Dec. 16th 1937, Apr. 21st 1938; May 5th 1938).

Other development in 1937 included a crosscut north on the 3rd level to intersect the #3 vein, which on surface was known to carry strong silver, pitchblende, and copper values. The #4 vein, located in between the #2 and #3 veins, would also be explored by this drive. Construction in 1937 included a new assay office, a new 350 hp Ruston diesel generator, and additional oil storage tanks (The Northern Miner, Sept. 9th 1937). During the year, 4,087 feet of drifting, 1,740 feet of crosscutting, 179 feet of raising, and 4,800 cubic feet of slashing was accomplished in the #1 shaft workings (McNiven, 1967).

#2 Shaft Development 1937-1938

Work continued at the #2 shaft (aka 'Gulch') on the northern section of the #1 vein in 1937. Previous work consisted of a 50-foot shaft sunk using handsteel in 1935. In 1937, a headframe was erected, an electric hoist and diesel compressor installed, and the #2 shaft was deepened to 145 feet and a 2nd level established at 125-foot depth. Lateral development in 1937 at #2 shaft consisted of 458 feet of drifting, 134 feet of crosscutting, 96 feet of shaft sinking, and 708 feet of diamond drilling in a series of short holes. Two short high-grade silver ore shoots were discovered on the 125-foot level. Work continued in early 1938 when 287 feet of drifting was completed. No further ore was

encountered and operations were suspended to focus on the #1 shaft workings. (The Northern Miner, May 13th 1937, Aug. 5th 1937, May 5th 1938; June 15th 1939). It was felt that future exploration of the #2 shaft area could be achieved by crosscutting from the #1 shaft.²

Operations progressed favorably during 1938. Development and exploration of the #3 vein was undertaken on the 3rd and 5th levels disclosing very interesting ore zones. Mining was concentrated on all vein systems on the 3rd to 5th levels. The #1 shaft was sunk to the 890-foot level and two new levels (6th and 7th) were established at 740-foot and 890-foot depth. Other developments during 1938 included mill modifications to increase capacity to 100 tons per day, and the addition of two Bellis-Morcom diesel generators (The Northern Miner, June 15th 1939).

Power Plant

By 1939, the following power units supplied the mine: two Ruston-Hornsby diesel engines and two Bellis-Morcom diesel engines connected to English-Electric 550 volt generators, and a 900 cubic feet per minute Ingersoll-Rand air compressor driven by a 196 horsepower Ruston-Hornsby diesel engine (Lord, 1941).

Employees

In 1938, the Eldorado operation employed 100 men with Emil Walli as mine manager. During these years, Port Radium was a mining camp and not a town site, and the only family residing on property was that of Mr. Walli. Despite the mines isolation and small size, the crew was treated to well-cooked meals and lots of recreation time. Great Bear Lake had many bays and inlets that could be explored (McNiven, 1967).

Silver Production Stops

The steady production of silver from the Eldorado ores was always a problem for the company. Although the ore body was incredibly rich with silver and copper, the costs associated with silver production were too high to warrant full-scale mining. The company by necessity had to focus on pitchblende mining. Development of the lower levels also showed a lower silver grade, and combined with the drop in the commodity price during 1938-1939, the company decided to cease the recovery of a silver-copper concentrate in 1939.

During 1939 development underground continued on the #1, #2, and #3 veins. Consultant Richard Murphy was hired to perform a geological study of the mine and as a result new orebodies were uncovered east of the mine workings on the 5th and 7th levels within the #2 vein. The advent of war in September 1939 brought the collapse of normal markets and a general confusion surrounding the future of radium mining. Some European markets were cut off by the war, but an increased demand from the United States helped to offset this loss of business (The Northern Miner, Apr. 4th 1940).

Sales of radium and uranium products continued to dwindle early in 1940, but developments at the mine continued at a normal rate. Changes were made to the mill flowsheet to increase the recovery rates, and E.J. Bolger was hired to replace the former mine manager. It was also reported that one of the Port Hope refinery's byproducts – polonium – was being utilized in the manufacturing of spark plugs (The Northern Miner, Apr. 18th 1940). Other interesting metallurgical news at this time surrounded the nature of uranium ores. Scientists in the United States were investigating a way to create fission energy using uranium isotopes. Eldorado Gold Mines Limited knew that this research held great importance and value, primarily because their mine at Great Bear Lake was the only source of uranium ores in the Allied Countries. The military applications of this new technology were also being considered (The Northern Miner, May 9th 1940).

1940 Closure

In June 1940, the company ceased operations at Eldorado Mine. Mining stopped on June 16th, and milling on June 18th. It was planned to be a temporary closure. The mine was cleaned, all underground equipment was brought to surface, and extensive grouting was conducted to help slow the mine's flooding. By August 15th 1940, the mine site was mothballed, with two caretakers left on the property (McNiven, 1967). The shutdown was due to a large stock of pitchblende concentrates that had accumulated at the Port Hope refinery. The hoped-for markets from the United States did not pan out as anticipated, and massive inventories of existing concentrate had no buyers. It was estimated that enough concentrated ore for at least three years of refinery production was available (The Northern Miner, June 27th 1940). Mine production and development at Eldorado from 1931 to 1940 is listed in **Tables 2, 3, and 4**.

² In 1956-1957, a long drive from the 5th level of the #1 shaft workings was driven underneath the #2 shaft workings and the 'Gulch' deposit was mined upwards through raises on the 3rd level

Year:	Ore Milled:	Concentrate Produced:	Cobbed Ore Produced and Shipped: ⁽¹⁾
1931	-	-	17 tons
1932	-	-	47 tons
1933	29 tons	20 tons	5 tons
1934	4,023 tons	84 tons	77 tons
1935	14,402 tons	160 tons	-
1936	22,947 tons	402 tons	-
1937	23,827 tons	674 tons	43 tons
1938	27,770 tons	718 tons	40 tons
1939	33,373 tons	1,057 tons	-
1940	15,500 tons	419 tons	-
Total:	<u>141,871 tons</u>	<u>3,534 tons</u>	<u>229 tons</u>

Table 2. Eldorado Mine production, 1933-1940. (sources: Craig, 1944; McNiven, 1967; Eldorado Gold Mines Ltd. Annual Reports, with information reported in newspaper articles of the day. Original Annual Report documents not available) ⁽¹⁾ Products shipped to refineries during the brief summer shipping season.

Year:	Silver:	Copper:	Lead:	Radium:	Uranium Salts:
1932	38,433 oz	-	-	-	-
1933	23,239 oz	-	-	3 grams	34,940 lbs
1934	37,778 oz	-	2 tons	3 grams	27,748 lbs
1935	116,902 oz	-	6 tons	8 grams	73,089 lbs
1936	259,673 oz	-	-	15 grams	160,662 lbs
1937	64,896 oz	-	-	24 grams	211,857 lbs
1938	475,004 oz	38 tons	-	unknown	359,980 lbs
1939	345,685 oz	21 tons	-	unknown	575,125 lbs
1940	47,444 oz	-	-	unknown	200,300 lbs
Total:	<u>1,409,054 oz</u>	<u>59 tons</u>	<u>8 tons</u>	?	<u>1,643,701 lbs</u>

Table 3. Metals produced from Eldorado Mine cobbed ore and concentrate shipments.

Information was obtained from a variety of sources: Lord (1941) reported radium and uranium salt production up until 1938 when this information became classified. Total radium production for this period is unknown. Uranium salt production after 1938 is from SENES Consultants Ltd. (1999) who presented a list of mine production in a chronological format. Their original source is unknown. Silver, copper, and lead production is based on Statistics Canada (1957) by taking yearly commodity production statistics for the N.W.T. and subtracting known production from other mines (Contact Lake, Con and Negus Mines) to get a number for Eldorado Mine.

Year:	#1 Shaft:	Drifting:	Crosscutting:	Raising:	#2 Shaft:	#2 Shaft Lateral:
1932	-	-	137'	-	-	-
1933	-	828'	253'	60'	-	-
1934	257'	502'	82'	-	-	-
1935	-	954'	195'	-	50'	49'
1936	256'	1,031'	635'	166'	-	-
1937	-	4,087'	1,740'	179'	96'	592'
1938	316'	3,372'	1,243'	398'	-	287'
1939	-	3,886'	596'	493'	-	-
1940	-	265'	78'	421'	-	-
<u>Total:</u>	<u>829'</u>	<u>14,925'</u>	<u>4,959'</u>	<u>1,717'</u>	<u>145'</u>	<u>928'</u>

Table 4. Mine development summary, 1932-1940. (source: McNiven, 1967)

Eldorado Mining and Refining (1944) Limited (1942-1960)³

The mine was reopened in the summer of 1942 with a minimum of publicity. The opening was made possible by a new deal with the United States government, hammered out thanks to Eldorado's new partner Boris Pregel, in 1941. The United States was developing atomic weapons as part of its role in World War II and needed a large supply of uranium concentrates. The Eldorado company was recognized as one of the principle producers of the uranium products in the Western World, and a deal was signed in which a 60 ton order of uranium oxide was to be supplied by Eldorado. The LaBine's were more than happy to get the mine reopened, but from this point all operations at the Eldorado Mine were top-secret. Additional orders followed: 350 tons in July 1942 and 500 tons in December 1942. The Eldorado company agreed to supply a total of 910 tons of uranium oxides to the United States by December 31st 1944 (Bothwell, 1984).

Some problems were faced in re-opening the mine. Even after just two years much work was required to bring the place back into operation. The shaft had to be de-watered and de-iced (ice was found to a depth of 15 feet), rotted timbers in the shaft replaced, and workings ventilated of built-up gases. Broken ore was frozen into the ground and large heaters had to be brought in to thaw the material. Ventilation was reduced to prevent underground freezing, but this only added to the hazardous gas (radon and helium) concerns. Production during the war was targeted on the #1 vein, where the high-grade material (5% U₃O₈) could be mined economically and expediently to meet the quota demands of the United States (Bothwell, 1984).

Government Control

Wartime precautions transformed Eldorado Gold Mines Limited (renamed Eldorado Mining and Refining Limited in 1943) from a private company to a public affair. The United States and Britain were both seeking commitments of uranium ores from Eldorado, and to ensure that the orders were met the Canadian government made an offer to buy share control of the company. A price was set, and the LaBine's traveled North America in search of shareholders that would give up control of their shares. This was all done secretly and 'under the table'. The LaBine's then agreed to sell whatever shares they gained control of to the Canadian government. This work was underway during 1942-1943, but it was not until 1944 when the new Crown Corporation was formed - Eldorado Mining and Refining (1944) Limited. The idea that uranium should be a government monopoly also provided justification for a take-over, as it was obvious that other companies in Canada were trying to develop uranium mines (Bothwell, 1984).

³ Called Eldorado Gold Mines Limited until 1943 when it was renamed Eldorado Mining and Refining Limited. In January 1944 the company became a crown corporation with the name Eldorado Mining and Refining (1944) Limited.

The years 1942 and 1943 were very busy at the Eldorado Mine as the company increased capacity of the plant, and erected a number of new facilities. Under new mine manager Ed Bolger, families were allowed to live on property so that by 1944 there were 230 men and a number of wives and children living at the site. Two duplex residences and apartments were built in the years that followed to accommodate these residents. A recreation hall and curling rink was also built. In 1943, new mine shops were commissioned in underground chambers just off the adit portal entrance (McNiven, 1967).

In 1944, production was sacrificed in favor of deepening the mine, a move that was risky considering that in April 1944 the mine was their behind production quota by some 27 tons of uranium oxide (Bothwell, 1984). The winze was sunk to the 1,380-foot level during 1943-1944 with several levels and many veins structures under development. In 1945, the # 1 shaft at Eldorado was extended to the surface. The shaft was enlarged to a 3-compartment opening to provide a manway compartment and a new hoist was installed, a Canadian Ingersoll Rand 48 inch x 36 inch 2-drum electric unit. A headframe was also built overtop of the #1 shaft (McNiven, 1967).

Post-War Operations

When the war ended in August 1945, Eldorado's role in the nuclear program of the United States was well known in the media and the Canadian public. Uranium products produced from the Great Bear Lake mine had been used to produce the bombs dropped on Japan. Operations at the mine continued as normal, because the United States still demanded uranium oxides from Canada. Management of the company was changing, however. The government understood that the LaBine brothers, although capable mining men, were unsuited to continue in leading the company. William J. Bennett, a civil servant under C.D. Howe, minister of munitions, was made president of the company in 1947 (McNiven, 1967).

The company also faced the problem of decreasing ore grades, down to 0.60% from 1% U_3O_8 during the war. Ore reserves were usually maintained at two-year supply and the future of the operation was always bleak. Inadequate exploration and development had been undertaken because the mine management was more focused on high-grading known resources to meet the demand for production. Costs of operation were also rising, equipment was falling into disrepair, local managers were quitting, and the mine was generally regarded as unsafe. The mill at Eldorado was viewed as unproductive, and uranium values were being jettisoned into the tailing waste piles. Eldorado Mine was, of course, the company's only producing uranium venture and if the mine closed they would be out of business quickly. There was therefore an initiative to search for uranium elsewhere in Canada. To improve the safety conditions in the mine, new ventilation fans were installed during 1946 to help ventilate the underground workings where pockets of radon and helium gases were proving hazardous to the miners. New methods of mine assaying were introduced to help control ore grades (McNiven, 1967).



Ryan Silke

Figure 4. *Crossfault Lake shaft, 2005.*

Crossfault Lake Shaft

During the summer of 1945, underground development began at the Crossfault Lake deposit, located on the northwest side of the lake. It was a small vein structure displaying high counts of radioactivity. A 2,200 volt powerline was constructed from Port Radium to the site in August 1945 and a 66 horsepower Mead-Morrison sinking hoist was installed. The shaft was collared to the 30-foot level and a 30 foot headframe was erected. During 1946-1947, the Crossfault Lake shaft was sunk to the 275-foot level and two levels at 150 and 275-foot depths was developed. Two raises were also driven. Equipment included Eldorado's old Canadian Ingersoll-Rand 36 inch x 24 inch electric hoist and a 24 cubic foot ore bucket. Very little ore was found during this operation, but later in the 1950s, 969 tons of material grading 0.45% U_3O_8 was trucked to the Eldorado mill for processing (McNiven, 1967).

The demand for power at the property was great during these years, and as a result two Clark diesel generators were installed in 1946. New construction included two bunkhouses, additions to the existing bunkhouse, staffhouse, and the erection of more town housing during 1947-1949. Other expansion included the construction of a new crushing

plant in 1950, which eliminated some of the bottleneck in mill feed. An expedient underground ore pass was excavated between the crushing plant and the #1 shaft to allow quick transfer of ore from the mine into the mill. This resulted in a 29% increase in tonnage handled in 1950 compared to 1949 (Eldorado Mining & Refining (1944) Ltd. Annual Report, 1950).

An all-weather airstrip was constructed in 1946 to improve summer transport facilities. It was cleared on a sand esker at Sawmill Bay. Incredibly, this strip was cleared 4,500 feet long in nine days using a Cat D-2 tractor. In the summer, a small boat and barge was used to ferry supplies and passengers from Sawmill Bay to the minesite. In the winter, airstrips were cleared on the ice in front of the minesite itself. Eldorado Mining and Refining Limited owned its own fleet of DC-3's and other aircraft to supply the Eldorado Mine year round (McNiven, 1967).

Metallurgical Research

In the mill, flow sheet modifications were made in order to achieve better recoveries of uranium product. It had been recognized for a long period of time that the gravity concentration used in the milling circuit was flawed and that a large amount of uranium exited the plant through the mill tailings. This meant that the tailings pond represented a sizeable ore reserve if a process was developed to process this waste. Another problem was found at depth in the mine, where the ore was becoming lower grade and represented certain metallurgical difficulties (McNiven, 1967).



Eldorado Mining & Refining Ltd.

Figure 5. *Port Radium, Eldorado Mine, 1950.*

A research program was carried out by government scientists in Ottawa between 1945 and 1949. It was determined that acid leaching would be the best approach to treat Eldorado ores. A small pilot plant confirmed this method during 1949, and a larger scale pilot plant was installed at Port Radium in 1950. Equipment was ordered for the erection of the permanent leaching plant during the year (McNiven, 1967).

Mill Fire 1951

A fire that destroyed the old gravity mill on November 9th 1951 almost proved disastrous to the operation. However, a major airlift was conducted to the property in December as soon as the ice was thick enough for DC-3's to land. Supplies and equipment to rebuild the mill were delivered on schedule despite a busy flying season for the Eldorado company, and the new gravity mill together with the new leaching plant was in operation by May 1952 (McNiven, 1967).

Post-Fire Operations

Normal-milling operations resumed in May 1952 in the new gravity plant and the leaching plant. By mid-summer, both plants were operating at near capacity. Although the milling plant was down for a period of 7 months, the new recoveries achieved through the leaching plant allowed for the production of uranium concentrates comparable to 1951 production (McNiven, 1967).

Depth Development

Plans were made in 1951 to sink a winze below the 11th level to explore and ultimately develop the depth extensions of the vein systems at Eldorado. The winze was collared in 1952, northeast of the # 1 shaft, and when completed in 1953 it provided access to five new levels between the 1,175-foot and 1,675-foot levels. Sinking was stopped one level above the objective of 1,800 feet due to excessive water flows. Flooding provided some difficulty in developing these new levels, but new cement grouting techniques using high-pressure pumps and trained personnel helped to solve the problems (McNiven, 1967).

Mining Operations 1950s

Access to the mine was via the #1 shaft to the 1,300-foot level, and through a winze from the 1,175-foot level to the bottom of the mine at 1,675 feet. The headframe installation at #1 shaft was used only for hoisting and handling of ore; access to the underground workings for personnel was through the adit portal to the station on the 1st level of #1 shaft. By 1955 total lateral workings at the mine were estimated to be 23 miles in length (Donald, 1956b).

Production at Eldorado was half underground production and half tailings reclamation in the 1950s, totaling 350-tons per day production. Lateral development was driven by two-man crews using Copco airleg machine drills. Heavy water inflows prevented normal lateral driving techniques. Fractured rock was avoided wherever possible and line drives were carried out in more solid footwall rock. Extensive grouting was carried out before rounds were blasted into the ore zones. Irregular ore deposits encountered at greater depths necessitated the change from shrinkage mining to cut-and-fill in 1952. Cut-and-fill mining was reported to offer flexibility in recovering irregular stringers of ore, reduction in dilution of waste rock, and aiding in the reduction of broken ore in the stopes. Waste fill was supplied from development workings and other waste accumulation through underground work. Equipment used in mining included Gardner-Denver R94 stoper drills and Eimco 12B mucking machines (Donald, 1956b).



Henry Busse - N.W.T. Archives – N-1979-052-3308

Figure 6. *Eldorado Mine #1 shaft headframe, built 1945 when the underground winze was extended to surface.*

Material was hoisted at the winze in a 40 cubic foot aluminum bottom-dump skip and dumped into waste and ore bins of 125-ton capacity above the 1,175-foot level. Material was trammed in Mancha locomotives and 30-cubic foot Wabi side dump cars on the 1,175-foot level, feeding an orepass that loaded ore cars on the 1,300-foot level of #1 shaft. Ore and waste was hoisted directly in mine cars of 20 cubic foot capacity within the mine cage to the surface through #1 shaft. Waste, when not used underground as fill, was sent to surface dumps. Ore was hoisted into the headframe at #1 shaft and dumped over a grizzly in the collarhouse into the primary crusher, from which the crushed material passed into an underground ore pass system and entered the nearby mill building. Both the #1 shaft and the winze used Canadian Ingersoll Rand 36 inch x 48 inch 2-drum electric hoists (Donald, 1956b).

Milling Operations 1950s

Milling at the Eldorado Mine was a complicated process in the 1950s. Run of mine ore was crushed and treated in a gravity concentration plant from which a concentrate was produced. The tailings, those freshly produced and the older piles from former operations, were pumped to a leaching plant for the re-treatment and production of a high-grade uranium precipitate. Sulphuric acid required for the leaching plant was generated in an onsite acid plant. The following is a breakdown of the different circuits involved in producing uranium at Eldorado in 1956: (McNiven et al, 1956)

Crushing

Ore was hoisted via the Eldorado #1 shaft and crushed in a 12 inch x 24 inch Blake jaw crusher set at 2-¾ inch. The crusher discharged into a 250 ton ore pass excavation under the headframe. Ore was conveyed from the ore pass by a 450 foot long conveyor to a 50 ton surge bin in the mill building. Secondary crushing was achieved through a 3-foot Symons cone crusher in closed circuit with a 38 inch x 96 inch Dillon screen until a 3/8 inch size was produced. This fine product was stored in a 75 ton fine orebin (McNiven et al., 1956).

Gravity Concentration

The new gravity plant went into operation in May 1952. With the introduction of the leaching plant, the function of the gravity circuit changed from maximum recovery of uranium concentrates to the preparation of tailings for the leaching process. Material from the fine ore-bin was screened to sort out three sizes, -3/8 inch plus 4 mesh, -4 plus 8 mesh, and -8 mesh. The first two products, comprising about 85% of the feed, were treated in a rougher jig circuit, then in cleaner jigs to produce a concentrate that was dried, bagged and shipped. Tailings from this stage of jiggling was classified, reground in a 4 foot x 8 foot ball mill, and recycled back over the screens for re-treatment (McNiven et al., 1956).

The -8 mesh product from the screens was sized in a hydraulic cone crusher with coarse sands feeding Denver jigs. Another uranium concentrate was recovered from these jigs and shipped. Other sands were again sized and pumped to the leaching plant. The cone overflow was classified in a Dorr bowl unit, and thickened for dewatering prior to leaching. Jig tailings were dewatered in a belt drag classifier, reground in a 4 foot x 6 foot ball mill, and returned to the Dorr classifier. Thickener overflows were pumped to a hydro-separator whose underflows went through a flotation circuit for the partial removal of copper minerals, detrimental to the leaching process. Copper concentrates were sent to waste as tailings. Hydro-separator overflows was pumped to the leaching plant (McNiven et al., 1956).

Dredging Operations

A dredge operated during the winter months to recover tailings from Great Bear Lake where early operations had discarded the material, just off LaBine Point. The dredge unit was a conventional steel-hulled suction-type designed to recover tailings at a depth of 105 feet below water level. The dredge operated year round, although operations were often hampered in the summer months due to storms on the big lake. Winter operations presented other problems that the mine overcame. With massive buildups of ice, dredge movement was restricted. Dredge operations in the summer were conducted in shallow areas, while in the winter it was positioned over top of deeper areas where it could remain in a confined area sucking up an established perimeter of tailings. A small open area of water around the dredge needed to be kept free of ice to provide some mobility. Prior to 1955, jets of water were sprayed over the water surface from a rig set up around the dredge. In 1955 a new system was arranged whereby air was released from pipes on the lake bottom, causing turbulence on the water's surface (McNiven et al., 1956).

Leaching Plant

Plant feed consisted of reclaimed tailings and those tailings pumped from current operations at the gravity mill. Material was subjected to two stages of classification (a 10 foot Dorr hydro classifier and a 54 inch Akins high-weir classifier). Overflow from these units was laundered to a thickener. Underflow from the Dorr classifier was pumped to the Akins classifier, and underflow material from the Akins unit was processed in a closed circuit grinding mill (5 foot x 8 foot Marcy) and recycled back into the Akins for classification. The large Dorr thickener increased the density of the pulp to 61% solids. Clear water overflow from the thickener was discharged from the plant as waste. The underflow from the thickener was then agitated (where sulphuric acid and sodium chlorate oxidant were added to dissolve the uranium value), conditioned, and filtered (two 11½ foot x 16 foot Oliver string filters), with filter cake being discharged as waste and filtrates sent for clarifying and precipitation (filter > vacuum tank > Perrin press). The final precipitate (yellowcake) was dried and packed into drums for shipment (McNiven et al., 1956).

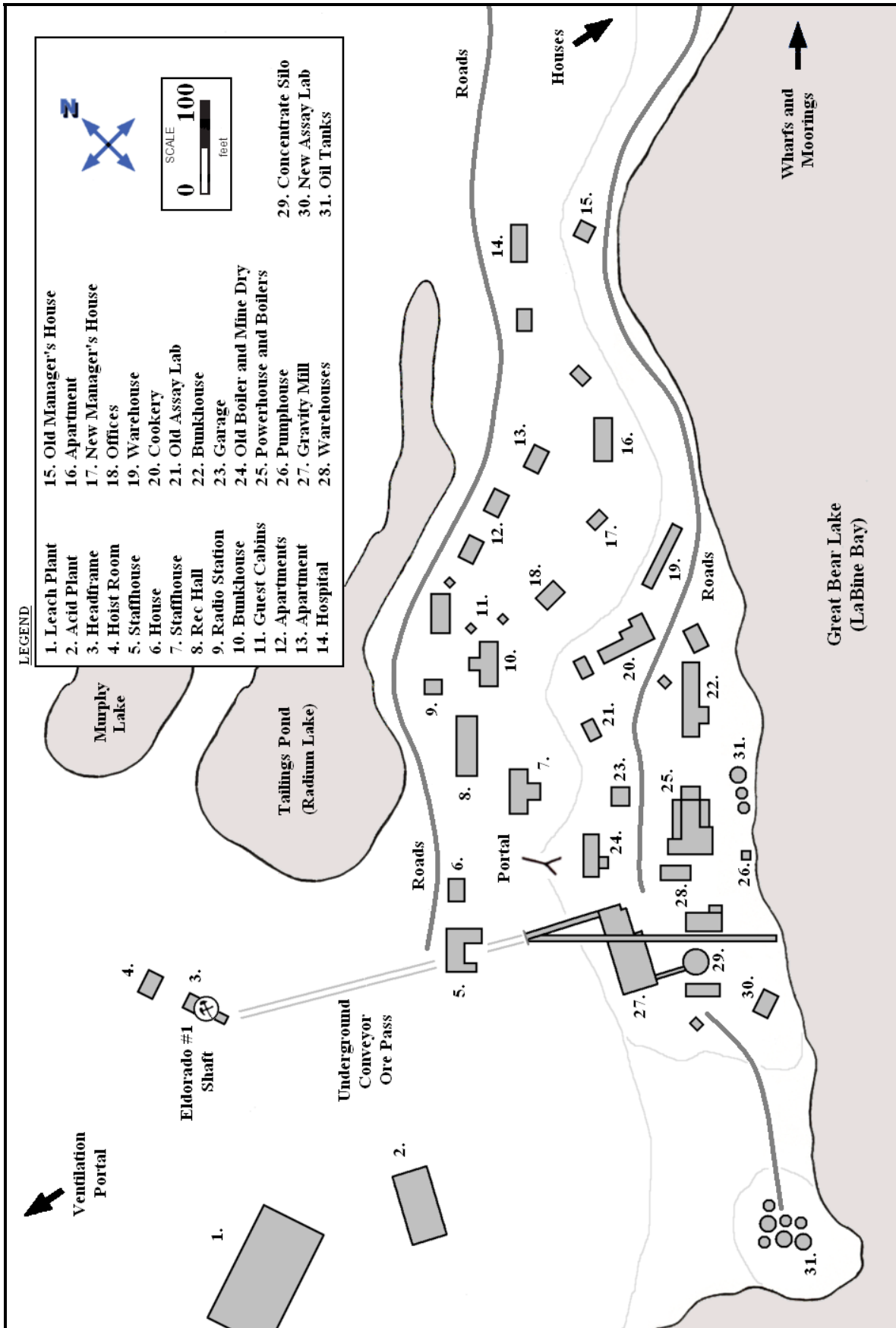


Figure 7. Eldorado Mine / Port Radium townsite surface plan, c.1952.

Ventilation

Ventilation became one of the most important assets of the mine because of radiation concerns, primarily the buildup of deadly radon gas. Powerful fans were installed in 1946 and 1947. Fresh air was brought in from the adit level through the vent portal (fitted with a 30 inch axial vane fan) located to the south of #1 shaft, and down a raise to the 125-foot level. Air circulated through the mine through a raise network and up-drafted through the #1 shaft. Mine air was heated from November to April by means of oil burners capable of raising the temperature of 10,000 cubic feet per minute of air by 70°. A new ventilation plant of increased capacity was installed in 1956 (Donald, 1956b).

Mine Services

All shops for mining operations were housed in excavated underground caverns 100 feet in from the portal entrance on the adit level. These shops included the blacksmith shop, machine shop, electrical shop, pipe fitters shop, and drill repair shop. There was also a small warehouse for shop parts (Donald, 1956b).

Power and Heating Plant 1950s

Electrical power was supplied from a diesel generating plant having a total output of 3,640 kilowatts in 1955. A large portion of this power supply was generated from three Copper-Bessemer units (two 940 horsepower and one 1250 horsepower unit) installed in 1952 and 1953. The introduction of these new units essentially solved the power problems of past years by augmenting the older units. Power costs were also substantially reduced. Backup units included (in 1955) two 200 horsepower Bellis & Morcom units of 150 kilowatts each, one 250 horsepower Ruston-Hornsby of 175 kilowatts, one 300 horsepower Ruston-Hornsby of 200 kilowatts, and two 620 horsepower Clark units of 400 kilowatts each (installed in 1946). All units were housed in the powerhouse building. Diesel oil capacity was 1,700,000 gallons which was equivalent to 14-months supply (Donald, 1956b).

Compressed air of 3,000 cubic feet per minute capacity was supplied from diesel and electric driven units, including: Canadian Ingersoll-Rand 1,000 cubic feet per minute unit (driven by 175 horsepower Ruston-Hornsby diesel), Canadian Ingersoll-Rand 700 cubic feet per minute unit (driven by 150 horsepower Ruston-Hornsby diesel), Canadian Ingersoll-Rand 700 cubic feet per minute unit (driven by electric motor), Canadian Ingersoll-Rand 2,000 cubic feet per minute unit (driven by electric motor), and a Canadian Ingersoll-Rand 700 cubic feet per minute unit (driven by 150 horsepower electric motor). All units were housed in the powerhouse building. Another Canadian Ingersoll-Rand compressor of 250 cubic feet per minute supplied air for the leach plant (Donald, 1956b).

Heat for the entire plant and camp was supplied by one 80 hp Leonard oil-boiler (installed 1946), one 60 horsepower Iron Works wood boiler, one 64 horsepower Foster-Wheeler marine boiler, and three 18 horsepower Foster-Wheeler waste-heat boilers, utilizing cooling water from the diesel units. These units were housed in an annex to the powerhouse building. There was also one 76 horsepower Foster-Wheeler marine boiler in use at the acid plant (Donald, 1956b).

Employees

The highest workforce recorded at the Eldorado Mine was 270 in 1954 (Eldorado Mining & Refining (1944) Ltd. Annual Report, 1954). The operation on average employed about 240 persons. Labour turnover was fairly low considering the isolation of the property. The company combated turnover by attracting married couples and promoting recreational and other social activities at the mine. Mine managers at Eldorado Mine between 1942 and 1960 were Ed J. Bolger (1942-1947), E.B. Gillanders (1947-1952), Harold Lake (1952-1955), and Jock G. McNiven (1955-1960) (McNiven, 1967). In 1956, the following staff was reported: J.G. McNiven, manager; W. Nancarrow, assistance manager; V. Pittson, mine superintendent; N. Hogdson, mine engineer; A.J. Leavenmonth, master mechanic; Harry Bergman, powerhouse operator; Gordon Spence, mill foreman; and K.G. Donald, chief geologist (Donald, 1956b).

Exploration and Development

Exploration work during the 1950s focused on finding sources of pitchblende ore within the existing mine area at both depth and lateral extent. Winze development showed that uranium values of the ore diminished and became more erratic at depth, so the best chance of finding additional ore was by developing the strike continuations of the vein deposits and by searching for new veins. Surface diamond drilling of the Weiner Bay area in 1952, north of the main veins, gave no indication of ore grade material. One of the main underground targets was the #7 vein northeast of the mine, on the 'Uranium' mineral claims owned by Ventures Limited. An agreement was signed in 1950 whereby Eldorado would explore the ground with royalty payments made to Ventures in the event that any ore was produced. Drifting and raising was conducted from the 2nd level into this area and towards the #2 shaft during 1952-1953. Significant values were uncovered to justify the exploration of this area at depth, and in 1954 a 2,000 foot drift

was started from the 5th level towards the #5 and #7 veins. A raise was driven from the 2nd level to surface along the #7 vein, and a raise was proposed from the 5th level to the 2nd level (Eldorado Mining & Refining (1944) Ltd. Annual Reports, 1950-1954). By 1955, three good-grade uranium ore shoots were identified above the 2nd level and a low-grade shoot above the 1,425-foot level. A copper deposit was also outlined by diamond drilling, estimated to contain up to 50,000 tons of ore grading better than 8% copper (The Northern Miner newspaper, Sept. 1st 1955).

In the depths of the mine, the #2 vein was the focus of practically all mining work. Drives on the 15th and 16th levels were driven west towards the granite contact to explore favorable areas indicated by geological studies (Eldorado Mining & Refining (1944) Ltd. Annual Report, 1956). Development on the 14th, 15th, and 16th levels was slow and expensive due to the massive amount of grouting required. There were some interesting findings, however. Ore in short lengths, sometimes of excellent grade, was developed in many parts of the mine, but such erratic behaviour of the deposit became a worrisome issue for geologists and mine planners. Development work succeeded in maintaining ore reserves at appreciable rates, although by 1956 it became clear that development of the 16th level was not manageable due to water issues. Deepening of the winze was no longer considered a feasible option due to the water problems and the apparent ‘pinching out’ of the ore body (McNiven, 1967).

Meanwhile, surface mapping and diamond drilling was unsuccessful in uncovering any further veins of economic importance. In 1955, diamond drilling of the veins in the Crossfault Lake area proved that it would not be of benefit to develop those showings. Other exploration and development included the investigation of an ore-grade intersection within the #1 vein beneath a diabase sill in the area of #2 shaft. A drive was advanced from the 5th level a distance of 1,500’ during 1956-1957. By 1957, exploration work both laterally and at depth showed that all known ore deposits had been well defined and that the chances of new discoveries was remote. The program in 1958 was therefore aimed at identifying which areas of the mine within developed sections could be recovered, with the view that mining operations would enter a retreating stage (Eldorado Mining & Refining (1944) Ltd. Annual Reports, 1955-1958).

Year:	Ore Milled:	Uranium Oxides:	Year:	Ore Milled:	Uranium Oxides:
1942	6,369 tons	115,525 lbs	1952	39,052 tons	399,152 lbs
1943	37,085 tons	585,681 lbs	1953	62,054 tons	754,638 lbs
1944	36,355 tons	714,759 lbs	1954	~62,000 tons	873,878 lbs
1945	39,761 tons	718,998 lbs	1955	62,977 tons	873,613 lbs
1946	42,900 tons	386,281 lbs	1956	62,792 tons	848,492 lbs
1947	49,934 tons	405,034 lbs	1957	63,437 tons	864,603 lbs
1948	49,250 tons	475,354 lbs	1958	66,005 tons	847,830 lbs
1949	47,339 tons	424,331 lbs	1959	65,636 tons	723,518 lbs
1950	61,178 tons	545,538 lbs	1960	46,293 tons	770,561 lbs
1951	52,910 tons	430,574 lbs	Total:	<u>953,327 tons</u>	<u>11,758,360 lbs</u>

Table 5. Eldorado Mine uranium production 1942-1960. Ore milled from 1953-1960 does not include recovery of dredged tailings.

Information is based on McNiven (1967), Eldorado Mining & Refining (1944) Limited Annual Reports, Eldorado Mine production records within the National Archives of Canada (RG 134), and SENES Consultants Ltd. (1999). No official record for 1954 could be found, so 62,000 tons ore milled is an estimate with the uranium production for 1954 based on SENES Consultants Limited. Uranium oxide production for 1942-1951 is from SENES Consultants, and production from 1952-1960 is from production records within the National Archives of Canada and Eldorado Annual Reports. Ore milled is based on McNiven (1967), production records, and annual reports.

Type of Development:	Footage:
<i>Crosscutting:</i>	26,819'
<i>Drifting:</i>	121,497'
<i>Raising:</i>	43,666'
<i>Stations and Sumps:</i>	253,072 cu. ft.
<i>Shaft Sinking:</i>	2,437'
<i>Underground Shops:</i>	52,164 cu. ft.
<i>Conveyor Bypass:</i>	2,292 cu. ft.
<i>Underground Hoist Room:</i>	12,528 cu. ft.
<i>Surface Diamond Drilling:</i>	77,938'
<i>U/G Diamond Drilling:</i>	136,652'
<i>Grout Drilling:</i>	502,329'

Table 6. Total mine development at Eldorado Mine, 1932-1960. Raising and lateral work includes slashing equivalent. (source: McNiven, 1967)

Solvent Extraction Plant

In 1956, Eldorado company researchers proposed the introduction of a solvent extraction process in the leaching plant, designed to replace the precipitation process then in use. Testwork suggested that such treatment would be more economical and would increase uranium recoveries. A pilot plant was built in 1956 and construction of the expansion was completed in February 1958 (Eldorado Mining & Refining (1944) Ltd. Annual Reports, 1956-1958). This process consisted of mixing the uranium-bearing solutions with tri-fatty amine in a fuel-oil solution. The organic compound combined with the uranium and the fuel-amine-uranium phase separated from the barren aqueous solution. The uranium-bearing fuel oil was then combined with an aqueous solution of sodium carbonate, which re-extracted the uranium as a concentrated solution. The final step was the re-precipitation of the uranium as sodium diuranate, which was separated by double filtration. When the idea of the solvent extraction plant was first theorized, it was realized that the life of the mine was short and for that reason the installation of the plant was risky. However, the subsequent improvement in metallurgy (recovery jumped from 87% in 1955 to 95% in 1959), together with the lower cost of production, made the operation more profitable in the last two years of operation (McNiven, 1967).

Cleanup Operations

As the mine entered into the closing stages, cleanup and recovery of old workings began. This consisted of cleaning ore spillages along the main haulage ways and from drift floors under stope chutes. From 1955-1959, this work yielded an additional six-weeks of mill supply (6,862 tons grading 0-60% U₃O₈). Other work included the recovery of surface stockpiles of ore at the Crossfault Lake shaft area (previously mentioned) (McNiven, 1967).

Closure

In 1957, it was announced that current ore reserves, both underground and in the old tailings, would run out by 1960. Exploration had failed to identify additional uranium resources in the mine and the tailings were almost fully recovered. On September 16th 1960, mining operations ceased and milling stopped soon after. Based on ore reserves calculated in 1959 and the amount of ore produced in 1960, very little ore remained as reserves by September 1960; not enough to warrant continued operations under winter conditions. Uranium mining at Eldorado Mine was finished (McNiven, 1967). Mine production from 1942 to 1960 is listed in **Table 5**, and total mine development from 1932 to 1960 is listed below in **Table 6**.

Echo Bay Mines Limited (1974-1982)

Echo Bay Mines Limited purchased the mineral rights beneath the Eldorado Mine from Eldorado Nuclear Limited (previously Eldorado Mining and Refining Limited) in 1974. Echo Bay had actually occupied the old buildings and mill since 1964 when they leased and then bought the plant and buildings from Eldorado for the purpose of mining silver and copper deposits at the Echo Bay Mine, adjacent to the east. In 1973 it was realized that the Echo Bay Mine would be depleted of ore within a year, so Echo Bay Mines Limited investigated the possibility of reopening the Eldorado Mine and recovering the silver mineralization, untapped since the late 1930s. In June 1974, the old Echo Bay winze hoist was installed at the Eldorado #1 shaft, which was then dewatered to the 850-foot level. Small amounts of underground development and large amounts of diamond drilling were conducted during 1974-1975, and first ore was introduced into the mill in February 1975. By November 1976, all mining of Echo Bay Mine ores had ceased and all mill feed came from the Eldorado Mine (Lord et al., 1978).

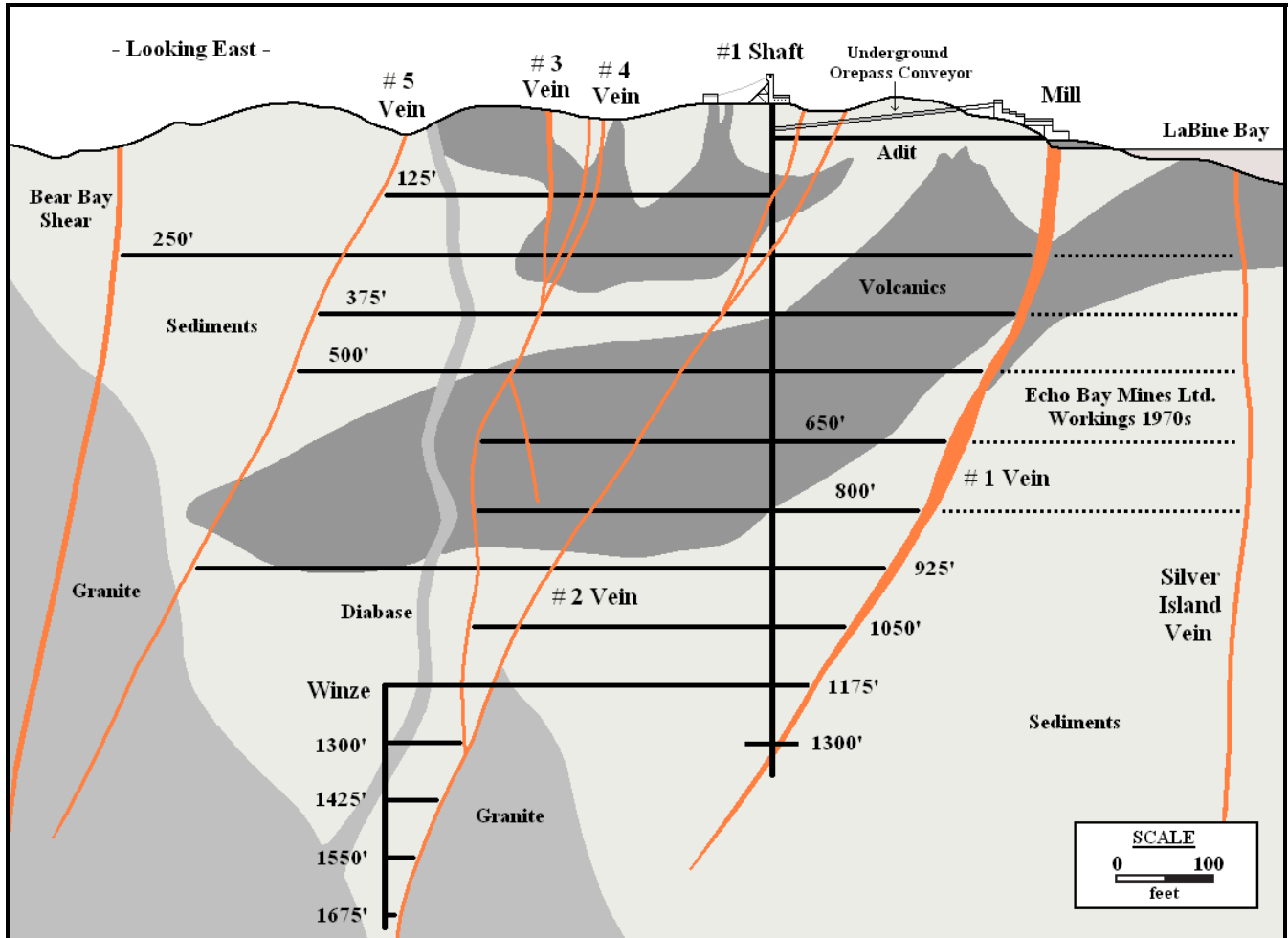


Figure 8. Eldorado Mine underground workings cross-section 1950s. (based on Campbell, 1957 and Jory, 1964)

Milling Operations 1970s

Milling switched over from Echo Bay Mine to Eldorado Mine ore in 1976 and ore was hoisted from the Eldorado #1 shaft and dumped into the mill circuit. In 1978, the flotation circuit was expanded to improve metallurgy for the changing ore and to improve copper recoveries. The following circuit was reported in January 1980: Ore was hoisted via the Eldorado #1 shaft and crushed in a 12 inch x 24 inch Blake jaw crusher set at 2-³/₄ inch. The crusher discharged into a 250 ton ore pass excavation under the headframe. Ore was conveyed from the ore pass by a 450 foot long conveyor to a 50 ton surge bin in the mill building. Secondary crushing was achieved through a 3 foot Symons cone crusher in closed circuit with a 38 inch x 96 inch Dillon screen until a 3/8 inch size was produced. This fine product was stored in a 75 ton fine orebin. Ore drawn from the fine orebin was fed to a 7 foot x 11 foot Allis Chalmers ball mill equipped with a scoop feeder. The ball mill was in closed circuit with two 12 inch x 18 inch

Denver duplex mineral jigs and a 10 inch Krebs cyclone. Cyclone overflow was sent to the flotation circuit. A jig concentrate is recovered from the mineral jigs, and dried on a steam table, sampled and weighed, then packed into half-ton boxes for shipping.

The flotation circuit recovered 37% of the silver from the mill feed. Flotation feed was conditioned in an 8 foot x 8 foot conditioner. Conditioner overflow was fed to a Maxwell flotation cell where a copper-silver concentrate was floated, which went directly to the concentrate filter. This stage of rough flotation was continued in 12 Denver #18 flotation cells, six cells of which were used as scavenger cells. The rough tailings were also scavenged in a bank of four Denver #15 cells where a final tailings was discharged from the mill plant. Scavenger concentrate was recycled back into the flotation conditioner tank. Rougher overflow or concentrate was reground in a closed-circuit cycle with a 6-inch Krebs cyclone and 5 foot x 5 foot Allis-Chalmers ball mill. This material was only reground if the ore contained fine sulphide silver, which was common in Echo Bay and Eldorado Mine ores. Cleaner tailings was scavenged in four Denver #15 flotation cells and then re-cleaned. The final flotation concentrate was thickened in a 12 foot Door thickener. The concentrate was filtered in a three-disc 4 foot filter, dried, sampled and weighed, and packed into 1-ton boxes for shipment to the smelter. In 1980 it was reported that 60% of the mill feed silver was recovered in the jiggling concentrates and that 37% was recovered in the flotation concentrates. 3% was lost to tailings (Karklin, 1980). In 1977, total copper recovery was 88-8% (Parashyniak, 1977).

The power plant consisted of a combination of Cat D-353 and D-398 diesel engines totaling 2500 kilowatts. A standby General Motors MU-20E 2500 kilowatts diesel generator was also available (Karklin, 1980).

Transportation

Winter roads connected the mine with the south during the winter months. This road was cleared annually by Byers Transport Limited starting in 1963. During the summer, wheeled aircraft landed on the Glacier Lake airstrip, cleared in 1971 east of the minesite.

Camp and Accommodations

In January 1980, it was reported that Port Radium had a population of 165 people, including 14 families. Single workers were on 2-month contracts and a normal rotation was 2-months in and 2-weeks leave. Natives consisted of approximately 2% of the workforce. There was no formal agreement with the Government of the NWT for the utilization of native labour. Accommodation for single-men was in the older bunkhouses and later by trailer units. Married families were provided with furnished apartments in duplex and quadriplex buildings. Recreation was provided with gymnasium, two-sheet curling rink, bar and lounge, billiards, television, and library (Karklin, 1980). Employees were infamously known for being rowdy and one former employee swears that most of the men were probably ex-cons enticed off the street, because Echo Bay had problems finding anybody willing to work this far north. Mine managers included Frank Burton (c.1976), Bob Phillips (c.1978), and Fred Sveinson in the 1980s.

Detailed records that outline mining and development operations at Eldorado during Echo Bay Mines Limited period are not available. Production was focused on the upper levels of the mine. The shaft was de-watered in 1979 to the 1,300-foot level, but it is doubted any work was done at that depth (Brophy et al., 1983). In 1977 it was reported that development and production came mainly from the Silver Island or D-vein on the 650-foot level. Higher-grade ores were being encountered in this area (Lord et al., 1981). Exploration drilling on the 800-foot level of the D-vein in 1978 indicated 3,000 tons of silver ore (Lord et al., 1983). Another source of mill feed during this period was from the Contact Lake Mine, but it was not a substantial contribution.

The mine suffered declining grades and lower silver revenue in the early 1980s. In 1980, the average grade of ore was 34 ounces per ton silver down from 50 ounces per ton silver in 1979. Operating expenses rose from \$408 per ton milled in 1979 to \$439 per ton milled in 1980. Silver prices fluctuated during 1980, with a softened market by year-end. At year-end 1980, there was an estimated 12 months of ore reserves with little prospect of adding additional reserves. In light of depleting reserves, rising operational costs, and softening silver markets, Echo Bay Mines Limited began plans for phasing out Port Radium silver mining at the end of 1981. Crews and equipment, wherever possible, would be transferred to Echo Bay's new Lupin gold mine in what is now Nunavut Territory (Echo Bay Mines Ltd. Annual Report, 1980).

Final Closure

Conditions worsened in 1981, when the average grade of ore declined further to 26 ounces per ton silver and silver prices continued to plummet. Operational costs improved as a result of certain cost-saving and cutback programs; however, depletion of economic reserves led to a permanent cessation of mining operations in November 1981.

Milling of stockpiled ore continued until March 1982 (Echo Bay Mines Ltd. Annual Report, 1981). **Table 7** lists mine production from 1975 to 1982, and **Table 8** lists total Eldorado Mine production from start to closure.

Year:	Ore Milled:	Silver Recovered:	Copper Recovered:
1975	31,251 tons	771,332 oz	399 tons
1976	39,387 tons	1,865,775 oz	439 tons
1977	34,243 tons	2,113,967 oz	268 tons
1978	37,278 tons	2,299,798 oz	262 tons
1979	39,800 tons	2,162,611 oz	260 tons
1980	39,800 tons	1,426,302 oz	294 tons
1981	41,600 tons	1,206,863 oz	346 tons
1982	8,045 tons	115,680 oz	62 tons
<u>Total:</u>	<u>271,404 tons</u>	<u>11,962,328 oz</u>	<u>2,330 tons</u>

Table 7. Eldorado Mine production, 1975-1982. (source: Mineral Industry Reports Northwest Territories; Echo Bay Mines Quarter Report, March 31st 1982)

Echo Bay Mines Limited reported that although silver was no longer a significant resource at the Eldorado and Echo Bay Mines, cobalt might one day be a profitable mineral in the mine area. Some surface facilities, including the headframe at #1 shaft, were retained for a short period in light of future mining possibilities in the area (Echo Bay Mines Ltd. Annual Report, 1982). By 1985, Echo Bay had completed its remediation of the property and all buildings at Port Radium were burned down or demolished, leaving only a tiny log cabin. The mine openings were sealed and all moveable equipment, supplies, and other assets were removed from the property, some to the Lupin gold mine.

Year:	Ore Milled:	Silver:	Copper:	Uranium Oxides:	Radium Salts:	Nickel:	Cobalt:
1932-1940	141,871 tons	1,409,054 oz	59 tons	1,643,701 lbs	?	?	?
1942-1960	953,327 tons	-	-	11,758,360 lbs	?	?	?
1975-1982	271,404 tons	11,962,328 oz	2,330 tons	-	-	-	-
<u>Total:</u>	<u>1,366,602 tons</u>	<u>13,371,382 oz</u>	<u>2,389 tons</u>	<u>13,402,061 lbs</u>	<u>450 g</u>	<u>140 tons</u>	<u>250 tons</u>

Table 8. Total Eldorado Mine production during the three-stages of operations. Ore milled, silver, copper, and uranium oxide production is based on the references in **Tables 2, 3, 5, and 7**. Total production of radium salts, nickel, and cobalt is based on Jory (1964) who also reports 110 tons of lead and 1,000 microcuries of polonium production. His figures for uranium oxides, silver, and copper do not match up with other production records.

Exploration Since Mine Closure

The mineral claims in the area are today (2007) owned by Alberta Star Development Corporation and Solitaire Minerals. Exploration of the region is ongoing.

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FREDA

Bulk Sampled (Abandoned)

Tantalum, Columbium

62° 38' 00" 113° 28' 00"

Years of Primary Development: 1946

Mine Development: open cuts

Years of Bulk Sampling: 1946

Bulk Sample: 4 tons milled = 500 lbs tantalite concentrate (46% tantalum oxide, 31% columbium oxide)

Introduction

The Freda was a very small tantalum/columbium (nobium) operation located north of Thompson-Lundmark Mine about three kilometers inland from Thompson Lake and about 50 kilometers northeast of Yellowknife, NWT. The exact location is not available. Forest fires of 1998 swept through this area, so any structural remnants have probably been destroyed. Aerial reconnaissance of the area in August 2004 to search for remains of the mine was unsuccessful.

Brief History

The 'Freda' claim was staked on September 20, 1944 by Ernie Sutherland over a small pegmatite dyke discovered by the Geological Survey of Canada. A small mill went into operation in 1946 during a very brief stage of production.

Geology and Ore Deposits¹

The showing is underlain by the Archean Age greywacke-argillite turbidite belonging to the Burwash Formation of the Yellowknife Supergroup. These rocks are intricately folded and faulted during at least two phases of Archean Age deformation, and metamorphosed to lower amphibolite grade in a 40 kilometer wide, north trending zone. The Burwash sediments were intruded by plutons, plugs and stocks of the Prosperous Lake granite suite. Minor mafic to felsic volcanics have also been noted within the prospect area but are volumetrically insignificant. Discordant and predominantly north-south trending, elongate dykes and veins lie within the lower amphibolite facies metamorphic zone. The primary pegmatite dyke is 300 feet long with width of 15 to 20 feet. The dyke dips from 10 to 33° south. Tantalite occurs in clusters ranging from small crystals to large crystals 15 pounds in weight. Beryl appears to be associated with the tantalite.

Northern Tantalum and Rare Metals Limited (1946)

Following stripping and blasting of the Freda dyke during 1945-1946, the owners judged the content of this rock to be high enough to recover a high-grade tantalum concentrate through the use of a small mill. The claim was acquired by Northern Tantalum and Rare Metals Limited with funding from Laurence-Lee Gold Mines Limited. L.F. Gauvreau was placed in charge of developments. According to claim documents, 160 tons of material was mined between October 12 and November 5, 1946, creating an excavation 24 feet long, 9 feet wide, and 9 feet deep. Ore was mined using gasoline plugger drills. Milling was conducted in November, from which 500 pounds of tantalite concentrate was produced from four tons of cobbled material grading 4% to 6% tantalite. A sample of these concentrates assayed 46% Ta₂O₅, 31% CbO₅, 1.83% tin, and 0.57% TiO₂. In 1946, it was believed that the Freda deposit had good economic possibilities, and it was hoped that a profitable operation could be established if the grades were to be maintained throughout the dyke. A very rough reserve estimate of 100,000 pounds of tantalite concentrates or 45,000 pounds of metal to the 100-foot level was reported. Beryl could be recovered as a byproduct (Lord, 1951; The Northern Miner, Jan. 23rd 1947).

Exploration Since Mine Closure

Unknown.

References and Recommended Reading

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The Northern Miner newspaper articles, 1947.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085INW0034

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.



GIANT / SUPERCREST

Producer (Under Remediation)

Gold

62° 30' 00" 114° 21' 00"

Years of Primary Development: 1937-2004

Mine Development: 3 v. shafts, 2,000' depth, 8 levels; five open pits

Years of Production: 1948-2004

Mine Production: 18,149,578 tons milled = 7,116,345 oz Au (plus 49,337 oz Au recovered from Tailings Retreatment Plant, 1988-1990)

Years of High Grading: 1939-1940

High Grading: 121 tons shipped = 908 oz Au

Introduction

Located on the north side of Yellowknife, NWT, the Giant gold mine was in production between 1948 and 2004. Mining operations ceased July 2004 and the property is now under the management of the Department of Indian Affairs and Northern Development (D.I.A.N.D.), which assumed ownership of the property in 1999. Reclamation activities surrounding the underground arsenic trioxide storage problem are ongoing. Other sections of the property are under ownership of the City of Yellowknife and will be utilized for future recreational purposes.

Brief History

The productive claims are composed of two separate properties, amalgamated into the same operation in 1965 – Giant and Supercrest. The original 'Giant' claims were staked in July 1935 by Johnny Baker. Burwash Yellowknife Gold Mines Limited, a subsidiary of Bear Explorations and Radium Limited, conducted diamond drilling exploration during 1936. The Giant Yellowknife Gold Mines Limited company was incorporated in 1937 and underground exploration by the Consolidated Mining and Smelting Company of Canada Limited (C.M.&S.) commenced that year under an option agreement. Underground operations resumed in 1939 and some ore was recovered from the Brock vein, but no further work was done until after 1943 when the Giant company was acquired by Frobisher Explorations (a subsidiary of Ventures Limited at the time). Spectacular ore deposits were found underlying a valley floor on the claims in 1944, and a program of extensive development began. Gold production was attained in May 1948.



Giant Yellowknife Mines Ltd.

Figure 1. *Giant Mine, 1980s.*

The Supercrest property was staked as the 'AES' claims in 1936 and was explored by Akaitcho Yellowknife Gold Mines Limited after the war. The claims bordered Giant to the north. Although a shaft was collared and several buildings erected in 1948-1950, the property did not become productive until 1965, through a deal in which Giant mined the property from the underground. A new company called Supercrest Mines Limited was formed, in which Giant owned a controlling share. Mining of Supercrest ores continued until 1981, and resumed in 1994.

Open pitting operations at Giant Mine commenced in 1976 as a result of higher gold prices, which allowed for the recovery of lower grade ores near the surface. Open pit operations ceased in 1990 after the development of five primary pits and two smaller pits. The property switched hands twice between 1986 and 1990 in an attempt to make

the Giant mine more profitable during times of economic hardship. Royal Oak Mines Incorporated's producing reign between 1990 and 1999 was a troubled one with one violent strike and many cost-cutting reforms. Royal Oak went bankrupt in February 1999, but operations at the Giant Mine continued under the command of creditors until November 1999 when all operations ceased. The property then reverted back to the Crown (D.I.A.N.D.).

Miramar Mining Corporation Limited signed an agreement with D.I.A.N.D. late 1999 to acquire the Giant mineral rights for a nominal sum and to mine the deposit and truck ore to the local Con Mine, which Miramar owned. A two year agreement was signed, but was extended into 2005. Most exploration and ore feed was being mined from the Supercrest areas. With the failure to make the mine a profitable venture, especially after the closure of underground mining at Con Mine in November 2003, Miramar was forced to cease all gold mining operations in July 2004. Giant Mine is now under reclamation.

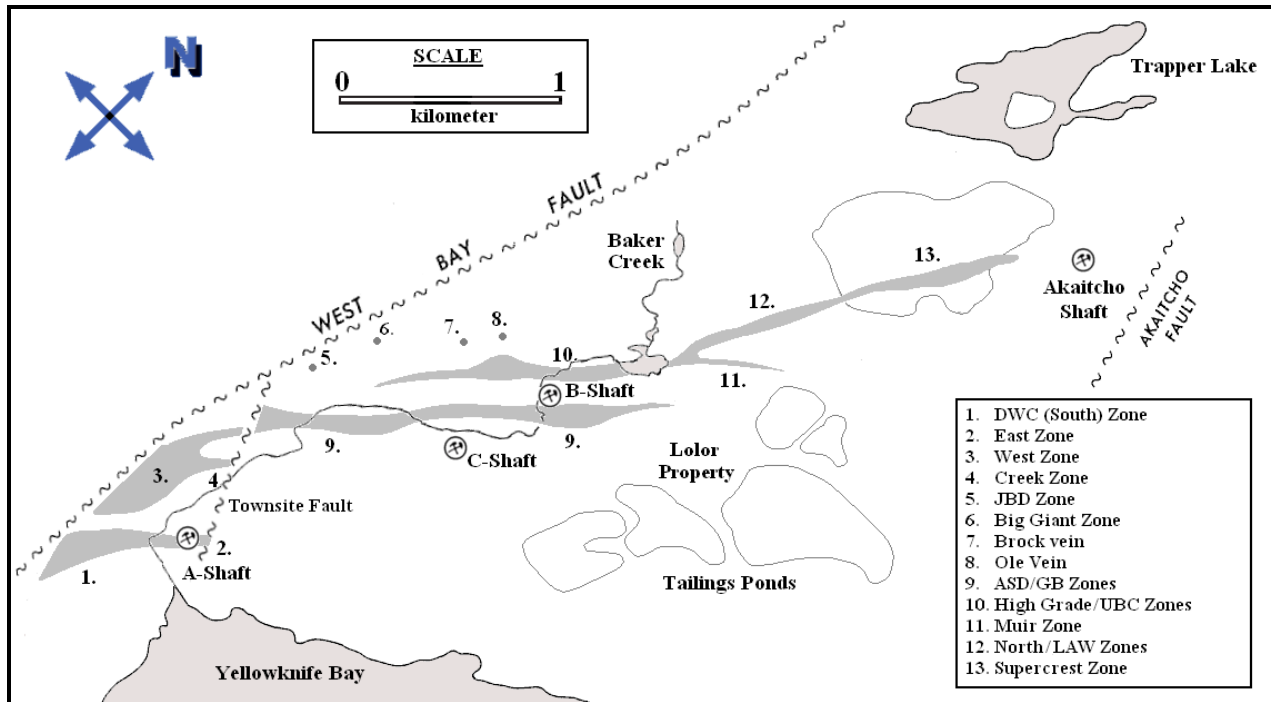


Figure 2. Orebodies and other gold deposits at the Giant Mine.

Geology and Ore Deposits¹

The Giant Mine orebodies occur in the Yellowknife Volcanic Belt. This belt lies along the western margin of a vast basin of turbiditic sediments, the Yellowknife Metasedimentary Basin. Various evidence suggests that the Yellowknife Volcanic Belt represents the actual basin margin. It trends north for roughly 8-5 kilometers. It includes a 10 - 12 kilometer thick, tholeiitic basal sequence, the Kam Group, overlain in angular unconformity by 2 kilometers of dominantly calc-alkaline volcanics and turbiditic sediments of the Banting Group. The volcanic belt is a southeast-facing, steeply dipping homoclinal sequence, intruded to the west by the Western Plutonic Complex, and cut by Proterozoic diabase dykes of the Dogrib and Indin groups. Regional metamorphism in the belt achieved lower to middle greenschist facies, and upper greenschist to lower amphibolite adjacent to the Western Pluton Complex. The entire belt has been subjected to compressional stress that is manifested by anastomosing, high-angle, reverse shear zones. These pre-date the diabase dykes and host most of the gold mineralization discovered to date in the belt. The entire package has been disrupted by a set of Proterozoic faults which post-date the diabase and show sinistral displacement: the West Bay Fault to the south and west of the Giant orebodies; the Akaitcho Fault to the north; and fault splays which crosscut the orebodies.

Shear zones cut various formations within the Kam and Banting Groups, but gold-bearing shears in the Giant area are largely restricted to the Yellowknife Bay Formation, uppermost of the Kam Group. Also in the Kam Group are, from base to top, the Chan, Crestaurum, and Townsite formations. The two basal Formations and the upper (host)

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

formation are predominantly tholeiitic, while the Townsite Formation is rhyodacitic and calc-alkaline; all of them are abundantly intruded by subparallel sheeted gabbroic dykes and local synvolcanic sills. The host Yellowknife Bay Formation comprises massive and pillowed basaltic flows and metagabbro intrusives, and, in the upper parts of the Formation, numerous intercalations of felsic tuff and tuffaceous interflow sediments. Some variolitic pillowed flows and many of the interflow tuffs are continuous for up to ten kilometers along strike and allow correlation across Proterozoic faults. Locally, sheeted dykes feed sills which grade laterally into pillowed flows.

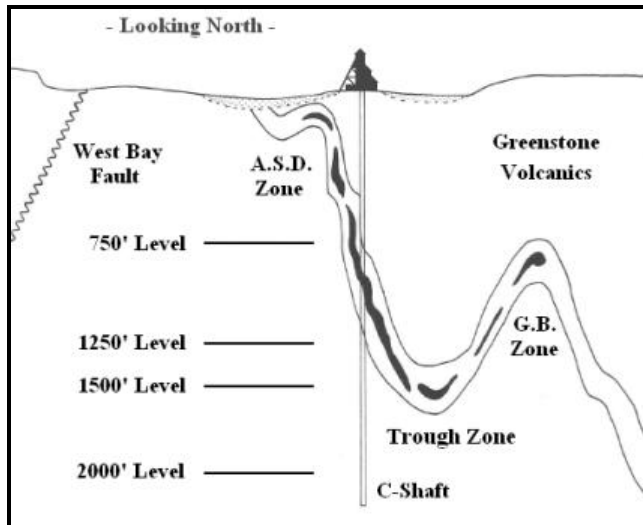


Figure 3. *Simplified cross section of Giant orebodies.*

The shear system which hosts the ore at Giant cuts massive and pillowed basaltic flows and gabbroic intrusives of the lower Yellowknife Bay Formation. The main shear structure strikes north slightly crosscutting the stratigraphy and dips steeply overall. It is traceable northeast for five kilometers and to a depth of 600 meters (2,000 feet). To the south, it is offset six kilometers by the West Bay Fault, then continues southwest as the Campbell Shear zone of the Con Mine, according to popular theory. Individual shear branches in the Giant system have a variety of strikes and dips, and host linear ore zones which comprise irregular lenses and pods of quartz-rich ore enveloped by schist. These tend to occur in dilatant areas such as shear zone flexures and intersections, and are characterized by complex folds and contortions in both quartz and schist. Orebodies range from 1 to 15 meters wide, and dip flatly to steeply.

Sulphides are usually present in accumulations up to 5%. Many of the orebodies throughout the mine host ore material in slightly different ways. The main zones (G.B. and A.S.D.) are generally hosted by quartz-chlorite-sericite-carbonate shears parallel to foliation. In some instances quartz veins exhibiting second and third-order large and small scale folding (eg: L.A.W. zone) are the main host. Free gold is generally more abundant in these ores. Other areas (eg: Lower B zone) host mineralization in tightly folded narrow bands of pyrothite rich chlorite-carbonate-sericite shears. Supercrust ore is located in moderately dipping (30 to 45°) quartz-sulphide-sericite zones.

C.M.&S. Limited [under option from B.E.A.R. Limited] (1937)

Work prior to 1944 on the Giant Mine property was focused on small, high-grade quartz veins, including the Brock, Ole, and South Giant systems, which are unrelated to the major orebodies. The following two sections detail Giant Mine operations prior to World War II when exploration was focused on the small quartz veins.

Consolidated Mining and Smelting Company of Canada Limited * (C.M.&S.) optioned the claims from Bear Exploration and Radium Limited (B.E.A.R.) early in 1937 to conduct extensive exploration of the known deposits. Apparently this deal was verbal in nature, and there was no signed agreement. Work was concentrated on the Ole zone, the South Giant vein, and the Big Giant vein (Lord, 1941). A small camp was occupied near the mouth of Baker Creek, previously built by Burwash Yellowknife Mines Limited in 1936 during their work on property. Additions to the camp included a frame bunkhouse and log cabin warehouse.

Sinking of the Ole shaft (#1 shaft) began in August 1937 using a crew of 4 men and a small air hoist and compressor. A small timber pole headframe and powerhouse were erected. Two other shafts (#2 and #3 shafts) on the South Giant and Big Giant veins were sunk during August and September. These were both about 15 feet in depth (Meikle, 1940). Most work was on the Ole shaft, and when work terminated November 14th 1937 the inclined shaft was down 81 feet. At the bottom of the shaft a crosscut was driven 33 feet, but no drifting was completed. They also cleared the site of #4 shaft on the Brock vein, which later was completed as the Brock shaft in 1939. In September 1937, eighteen men were working on the property. Mike Finland and Alf Wilmot were engineers in charge (Finland, 1937; Wilmot, 1937).

* now Teck-Cominco Limited

C.M.&S. was interested enough in the Giant property that in August 1937 they offered to buy the claims. B.E.A.R. turned down the offer, however, and instead made a deal with Howey Gold Mines Limited for the creation of a new company - Giant Yellowknife Gold Mines Limited (The Toronto Star, Aug. 21st 1937). The Giant property was turned over to the new company in August 1937. C.M.&S. was then asked to vacate the claims, but they insisted their work was blessed by the old verbal agreement with Bear Exploration and Radium Limited (The Toronto Star, Aug. 26th 1937; The Globe and Mail, Aug. 26th 1937). B.E.A.R. threatened to have them forcibly removed with legal action, but to avoid going to court C.M.&S. and B.E.A.R. made a deal in which C.M.&S. was to gain a small share interest in Giant Yellowknife Gold Mines Limited as repayment for the extensive work they had accomplished thus far. C.M.&S. remained on property until November 1937 when their operations ceased as per agreement with B.E.A.R. (The Toronto Star, Nov. 12th 1937).



Giant Yellowknife Mines Ltd.

Figure 4. *Workers at the Ole shaft at Giant Mine, 1937.*

Total cost of work during the 1937 program was \$22,044 for which Cominco earned a share interest in the new Giant company. Cominco considered the Giant property an interesting prospect with many possibilities, but they were unimpressed with the Ole vein and did not encounter high gold values underground.

Giant Yellowknife Gold Mines Limited [B.E.A.R. Limited] (1939-1941)

Diamond drilling and trenching of the South Giant and Big Giant veins was the focus of work during 1938. Work after 1939 focused primarily on the Brock veins, one of the most interesting gold discoveries on the Giant claims. Cominco had worked on clearing the Brock shaft (#4 shaft) site in 1937 but did not do any development. The Brock shaft was designed to be a prospect shaft. A crew of eight men were brought to the property in April 1939 under the direction of Dave Smith. When the Contact Lake Mine closed later in the fall of 1939 the crew was increased to 14 men, including new manager Charles Hershman. The Brock shaft was sunk a length of 126 feet on an incline 33° west, and at a vertical depth of 55 feet a drift was driven 192 feet north to explore the vein. Underground work stopped in August 1939 (Lord, 1941).

Equipment

A sinking plant, previously used at the Burwash Mine, consisted of a portable Canadian Ingersoll-Rand diesel driven compressor of 210 cubic feet per minute capacity, and a small Canadian Ingersoll-Rand 6x5 SSR 2-drum air hoist. The shaft (7 feet x 11 feet) was fitted with a small timber pole headframe and ore bucket, which slid on pole guides up the shaft incline. Other shaft site structures included a log powder magazine, frame blacksmith shop, and powerhouse (Meikle, 1940; Hershman, 1940).

High-grade ores were mined from development workings and hoisted to the surface by bucket. They were loaded into a small ore car and pushed on a trestle-way, dumping down into a large log ore bin. Ore was transported by Cat tractor to the camp on Yellowknife Bay, hand-picked further to obtain the free gold material in the quartz, sacked, and then sent south on barge for treatment at Trail, B.C. Two shipments of Brock vein ore were made, one in 1939 from the development workings, and one in 1940 from limited stoping operations and stockpile reserves (see Table 1).

Year:	Ore Shipped:	Gold Produced:	Silver Produced:
1939	74 tons	647 oz	45 oz
1940	47 tons	261 oz	39 oz

Table 1. *Giant Mine ore shipments, 1939-1940. (source: Lord, 1941)*

Operations continued during the winter of 1939-1940 with a crew of 10 to 12 men under the direction of superintendent Dave Smith and manager Charles Hershman, but work was terminated on June 5th 1940 due to financial restraints as a result of the war. Along with the shipment of further ores (see above table), work during 1940 included the collaring of the #5 shaft at the Brock vein area. This shaft, 14 feet deep when work stopped, was to be a vertical production shaft down to 700 feet. It was never continued. A new powerhouse and new equipment was erected in preparation for this work. None of it was ever put to use at the Giant property during this time period. Also, a small pilot mill was erected on the shores of Yellowknife Bay adjacent to the campsite. It consisted of a small crusher, ball mill, Wilfrey concentration table, blanket table, and amalgam unit. The plant was rated at 5 to 10 tons per eight hours, with power supplied by a diesel tractor. It too was never used (Meikle, 1940).



Polkington Collection - N.W.T. Archives – N-1990-012-0079

Figure 5. *Giant Mine, Brock shaft site c.1941.*

Camp Site

The camp on Yellowknife Bay at the outlet of Baker Creek was expanded and improved during 1939 and 1940. Several new log cabins and frame structures were erected and grew to include two bunkhouses, cookery, manager's house, office, staff house, warehouses, assay lab, garage, and the pilot mill building. Nine structures lined the shores of Yellowknife Bay along what is now Lakeshore Road at the Giant Townsite, across from the Giant Mine Boat Launch (Meikle, 1940; Hershman, 1940). A new mill rated at 25 tons per day was purchased and arrived in September 1941. It was announced during the year that new financing would be sought to install the mill, but these efforts were hampered by wartime limitations. Production at the Brock vein at Giant Mine was put on hold (The Northern Miner, Jan. 16th & Sept. 4th 1941).

Giant Yellowknife Gold Mines Limited [Ventures Limited] (1945-1962)

In 1943, Frobisher Exploration Company Limited (a subsidiary of Ventures Limited ²) purchased a controlling interest in Giant Yellowknife Gold Mines Limited and became manager of the Giant Mine property. The company, under Thayer Lindsley and Glyn Burge, was enthused by the gold potential after conferring with Don W. Cameron, a prospector who had discovered a high-grade shear zone at the south-end of the property in 1938. It was theorized that the long and wide Baker Creek Valley might be underlain by a series of similar ore bodies more important than the narrow gold veins explored by the Giant company before the war. (Giant Yellowknife Gold Mines Ltd. Annual Report, 1943)

² Became Falconbridge Nickel Mines Limited, 1962

Under Frobisher's management, an extensive mapping and prospecting program began in the summer of 1943 under the direction of A.S. 'Stuart' Dadson. This work showed evidence of a massive shear zone following the valley. Diamond drilling was started in January 1944 and Ken Muir was appointed manager of the property. Enough tonnage had been indicated by the summer of 1944 that the company ordered pieces of mining machinery in preparation for the sinking of a shaft so that a more definitive ore reserve could be calculated. Wartime labor and supply restrictions were still being felt at this time, so major development at the property was delayed a year (Giant Yellowknife Gold Mines Ltd. Annual Report, 1944). Even so, the Yellowknife region was seeing the start of the post-war mining boom during the summer of 1944 due to the incredible results from the Giant property. Prospectors and mining companies arrived back in Yellowknife in an attempt to uncover the geological secrets of the Giant orebodies and to correlate those findings with their own gold claims.

Original Camp

The old log cabin campsite was expanded during 1944-1945 to include several larger, framed structures, including bunkhouses, staffhouse, cookery, warehouses, shops and garage, and assay office. Most of these buildings were built in the area where the Giant Mine boat launch parking lot is located today (2006).

Sinking of A-Shaft

Excavation at the shaft site was started on June 25th 1945. The shaft was sunk on the south end of the property to test the East zone and its northern extension. Sinking operations, under contract, commenced on September 24th 1945. It was sunk to a depth of 522 feet by January 23rd 1946 with level stations established at 200-, 325-, and 450-foot depths (it was later deepened to 750 feet). Lateral development commenced on the first two levels, complete with underground diamond drilling and some raising. This first stage of development closely confirmed the extent and grade of the ore in the East zone as indicated by surface diamond drilling (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1945-1946).



Ryan Silke

Figure 6. *Giant Mine A-shaft headframe, 1999.*

Equipment for the A-shaft sinking job included a Canadian Ingersoll-Rand hoist (36 inch x 24 inch), a 500 cubic feet per minute Allen-McLellan electric air compressor, a 210 cubic feet per minute Ingersoll-Rand gasoline air compressor, a 20 horsepower boiler, and an electric generator. A permanent 55 foot timber headframe was built at A-shaft in September 1945. A temporary hoistroom, powerhouse, and shop were erected, and rock excavation had started for the site of the permanent hoist and powerhouse buildings, built in 1947 (Lord, 1951).

Sinking of B-Shaft

Located towards the northern end of the property, the B-shaft was put down to test the North High Grade zone, Muir zone, and North A.S.D. zone, also outlined by 1944-1945 diamond drilling. A separate camp was built at the B-shaft site for the contract crews. Site excavation and construction of buildings were carried out in extremely cold weather late in 1945, and shaft sinking itself did not commence until April 15th 1946. High-grade ore was encountered almost immediately when the shaft unexpectedly passed through a mineralized section of the High Grade zone (Giant Yellowknife Gold Mines Ltd. Annual Report, 1946).

By the end of June 1946 the B-shaft was 328 feet deep with two levels at 100- and 250-foot depths. Equipment used during B-shaft sinking included a Canadian Ingersoll-Rand 9x7 hoist, air compressor, and 40 horsepower boiler. The B-shaft headframe, built in December 1945, was of a permanent nature and the original structure (later upgraded) was 55 feet high (Lord, 1951).

Underground development at A-shaft was suspended in September 1946 after extensive exploration of the East zone on the 1st and 2nd levels. Drifting and crosscutting on both levels totaled 2,656 feet, plus 37 feet of raising and 9,847 feet of diamond drilling. Insufficient raise work had been done to specify an exact ore reserve, but some 300,000 tons of ore grading 0.44 ounces per ton gold were suggested within a single block of the East zone. 4,500 tons of ore grading 0.48 ounces per ton gold was mined from A-shaft and stockpiled. Further work at A-shaft was planned including the West zone and deeper into the East zone (Giant Yellowknife Gold Mines Ltd. Annual Report, 1947). Operations turned towards the development from the B-shaft and construction in preparation for milling operations. B-shaft sinking was advanced to 615 feet depth during 1947 and finally to 780 feet depth in February 1948. The 3rd and 4th levels were established at 425- and 575-foot depths. All work was focused on developing the oreshoots within the A.S.D. and High Grade zones (Giant Yellowknife Gold Mines Ltd. Annual Report, 1948).

Construction 1947

Meanwhile, surface operations were underway during 1947 in an attempt to achieve commercial gold production by the summer of 1948. Excavation of the plant and milling site was conducted during the fall of 1946 and throughout early 1947. Although the initial plant was to be a 500 ton per day capacity unit, allocation was made for future expansion. Framing and sheeting of the mill building was completed November 1947. A pump house and new boiler plant were built and pipeboxes were placed inland a distance of 4,500 feet north to the mill site. A permanent camp was also under construction starting with a 120 man bunkhouse building and new employee dwellings. A temporary diesel power plant was setup at the A-shaft site, which would be used to power the milling plant until hydropower was available (see below). In conjunction with the building of a gold mine the company also took upon the task of building a hydroelectric power project at Snare River. The plant, when complete, would be turned over to the federal government and Giant would be supplied with discounted electricity for a period of time. The Snare River Hydro plant would also be connected to the Yellowknife power grid so that the entire region could benefit from its supply (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1947-1948).

Production Begins

Initial mill operations commenced on May 12th 1948 with ore from the B-shaft stockpiles. Operations were restricted to the amalgamation and flotation circuits pending the completion of roaster and cyanidation circuits. The first gold bar was poured on June 3rd 1948, but mine opening celebrations were held on August 24th and the 'Inaugural Brick' pouring ceremony was held. Hydropower from Snare River was acquired in October 1948 and operations ramped up (Lord, 1951; Giant Yellowknife Gold Mines Ltd. Annual Report, 1948).

Mining Operations

During 1948 all mining and development was being conducted from B-shaft workings within the A.S.D. and High Grade zones. The B-shaft was fitted with a 2-drum Canadian Ingersoll-Rand electric hoist (42 inch x 30 inch). Six stopes within the High Grade zone and one within the A.S.D. zone were in production during the year, with all work focusing on the 100- and 250-foot levels. Mining was conducted by shrinkage stoping with ore being allowed to spill into the haulage drifts through boxholes from stope workings above. Mucking machines were used to load 2-ton and 1.5-ton mine cars, pulled by Mancha battery locomotives. Larger 80 cubic foot Granby-style side-dump ore cars began to be introduced in 1950. In 1948, only the 100- and 250-foot levels were serviced by skip loading, with ore and waste from lower levels being hoisted to surface in mine cars. Ore was trucked from the B-shaft to the mill crusher (Lord, 1951; Giant Yellowknife Gold Mines Ltd. Annual Report, 1948).

Milling Plant 1948

The original mill had a rated capacity of 500 tons per day, although the crusher plant itself could treat up to 2,000 tons per day. A complete description of the plant is given further below since in 1948 the plant was incomplete pending the completion of the roasting and cyanidation circuit. After suitable crushing and grinding, gold was caught in mineral jigs and amalgamated. The remainder of the ore was concentrated by flotation in preparation for roasting. All gold bullion produced during 1948 was by amalgamation; the gold-bearing flotation concentrates were stored pending completion of the roaster plant. Mill tailings were impounded for future cyanidation treatment. The mill operated at only 235 tons per day on average during 1948. It was estimated that only 17% of the feed gold was recovered by the gravity-amalgamation circuit, and that the remainder was either being lost in the tailings or contained in the flotation concentrates (Lord, 1951).

Power Plant

Power for the entire operation was supplied by two 400 horsepower Dominion diesel engines powering English-Electric generators, located in the A-shaft powerhouse. These units were intended to be used only until hydropower was introduced, but even these two units were found to be inadequate for increased power usage at the mine during

late 1947. In February 1948, the Giant company rented two Caterpillar D-17,000 diesel engines powering 650 cubic feet per minute compressors to help ease the electric power consumption (installed at B-shaft site). When milling started in May 1948, the mine again faced a power-supply crunch and a 400 horsepower General-Motors diesel generator set was ordered. In October 1948 hydro power was introduced and diesel power was used only as back-up or to augment power-supply during times of peak consumption. B-shaft operations during 1947-1948 required the use of two Canadian Ingersoll-Rand air compressors (1,500 cubic feet per minute total) driven by electric motor. These were augmented in 1948 with the two rented diesel units described above. Fuel storage was provided in tanks aggregating 700,000 gallons for bunker fuel, heavy diesel, and light diesel fuels (Lord, 1951).

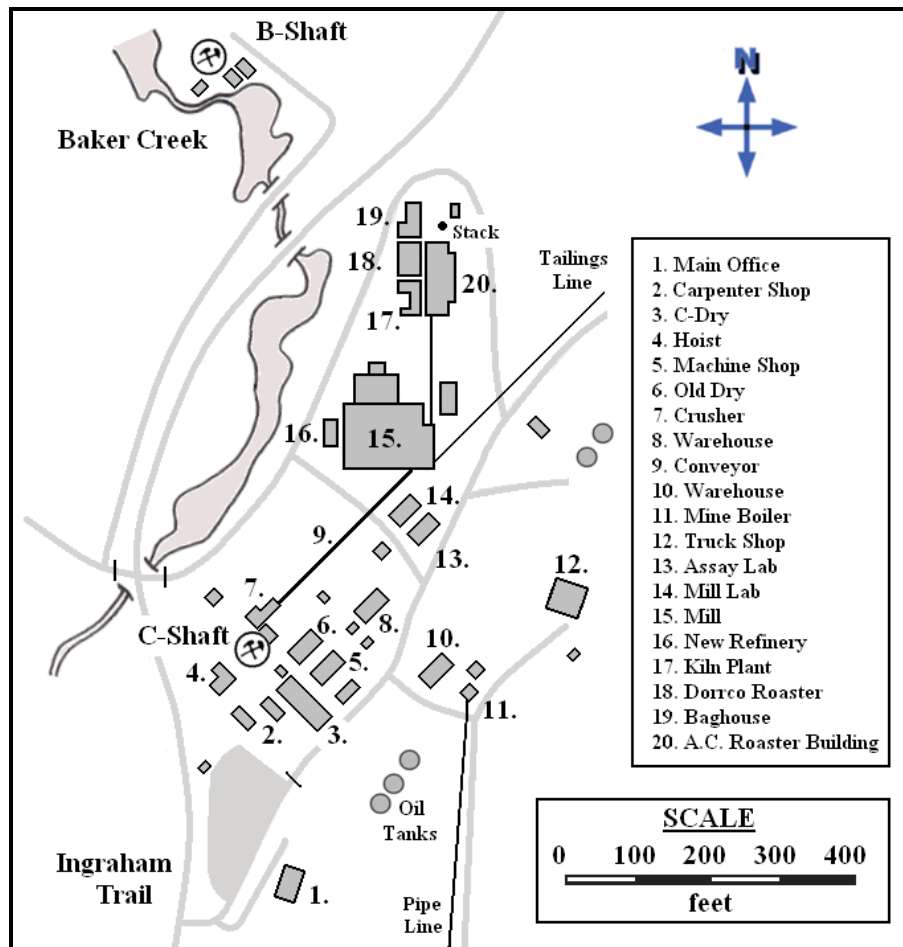


Figure 7. Giant Mine plant site, 1990.

Employees

About 222 persons were employed during 1948, as follows: 156 on surface, 59 underground, and 7 in the mill. Employees were unionized in June 1947 under Local 802 of the International United Mine, Mill, and Smelter Workers (C.I.O.). In 1948, the following management staff was working at Giant Mine: A. Ken Muir, general manager; Archie Freakes, mine superintendent; John D. Bateman, mine geologist; and Ken C. Grogan, mill superintendent (Lord, 1951). In August 1951, the payroll numbered 370, of whom 45 were engaged in construction. Management staff included: Peter N. Pitcher, mine manager; J.C. McCutcheon, mine superintendent; W.C. Toomey, mine captain; D.C. MacDonald, mine engineer; Ken C. Grogan, mill superintendent; J.D. Bateman, resident geologist; Hugh McCorquodale, assayer; Murray Pickard, geologist; C.E. Anderson, geologist with Frobisher Explorations Limited; Chet W. Wilkinson, electrical superintendent; Ned Mockford, shop foreman; L.A. Sparks, purchaser; Gordon Allen, accountant; H.S. Carter, construction foreman, A.S. Dadson, consulting geologists, and E.V. Neelands, consulting engineer (The Western Miner, 1951).

Late in January 1949, the roaster and cyanidation sections of the milling plant were completed and first feed was delivered on January 27th. Feed was derived from stockpiled flotation concentrates in addition to mill-run concentrates. Recovery was reported to be excellent although the company was seeking ways to improve the plant's

recovery rates. Metallurgical testwork would continue into the 1950s. Mill daily tonnage was increased as the milling plant was fine-tuned so that by 1950 the plant was operating at about 400 tons per day (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1949-1950).

Sinking of C-Shaft

The company had planned to sink a centralized shaft since 1947 at a location where future ore could be easily exploited. It was developed adjacent to the crushing plant and was designed as the main production shaft at Giant Mine. Sinking commenced in November 1949 and by July 1950 the shaft had reached its targeted first-stage depth of 1,025 feet. A well-integrated mining plant was being construction around the C-shaft area including hoistroom, shops, warehouse, miners dry, and assay lab. A 105 foot timber headframe with concrete and steel orebins was constructed (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1949-1950).

Stations were cut on the 100-, 250-, 425-, 575-, and 750-foot levels and a chamber for a future underground crusher unit was excavated on the 800-foot level. Workings connected the B and C-shaft areas on the 575- and 750-foot levels during 1950 and 1951. The B-shaft headframe was upgraded to 80 feet height with the installation of new orebins. Underground development was now focusing on the southern reaches of the A.S.D. zone in the area of C-shaft on the 425-, 575-, and 750-foot levels. The Lower B and 409 ore shoots within the High-Grade zone were targeted by a long drive from B-shaft on the 750-foot level and some minor stoping was conducted in 1952. Further attention was on the zones at the southern end of the property around A-shaft. A long haulage drive was driven during 1951-1952 through the length of the A.S.D. zone southward towards A-shaft, which was sunk to 793 feet depth. All three shafts were connected on the 750-foot level by 1952 (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1950-1952). Development waste from this work was hoisted up A-shaft in ore cars, dumped into the A-shaft ore bin, and disposed of by truck, but mining operations themselves would be conducted by tramming ore along the 750-foot level to C-shaft for hoisting. A-shaft area zones would supply ore until late 1957 when operations were suspended.

Operational Plant 1950s

C-shaft was originally serviced with a 2-drum Canadian Ingersoll-Rand electric hoist (60 inch x 36 inch) for both man cage and skip operations. A secondary hoist was installed in 1955 at C-shaft to operate the skips, while the other unit was now used solely for mancage operation and back-up skip operation. The new hoist was a 2-drum Canadian Ingersoll-Rand electric hoist (94 inch x 52 inch). Compressed air at C-shaft was supplied by three Canadian Ingersoll-Rand units supplying a total of 3,500 cubic feet per minute by 1955 (mine records).

A-shaft operations (in 1955) were fitted with a Canadian Ingersoll-Rand 2-drum electric hoist (36 inch x 24 inch), one 1000 cubic feet per minute Canadian Ingersoll-Rand air compressor, and two Gardner-Denver air compressors (as backups). The A-shaft powerhouse also housed the two 400 horsepower Dominion diesel generators used for backup power generation. Increased power demand during the 1950s and the inability to secure a reliable supply of hydropower from Snare River placed these units back into operation for a brief period. They were augmented in 1958 by the installation of a large Macintosh-Seymour diesel generator. Upon completion of mining operations at A-shaft in 1957, a 50,000 cubic feet per minute heating and ventilation plant was installed in the old A-shaft mine dry to circulate through the underground workings (mine records).

B-shaft operations (in 1955) were fitted with a Canadian Ingersoll-Rand 2-drum electric hoist (42 inch x 30 inch), one 1000 cubic feet per minute Canadian Ingersoll-Rand air compressor, one 500 cubic feet per minute air compressor, and one 700 cubic feet per minute Canadian Ingersoll-Rand air compressor. In 1958 the 1,000 cubic feet per minute air compressor from A-shaft was installed at B-shaft. The 500 cubic feet per minute air compressor was moved to the roaster complex to act as a spare in 1958 (mine records).

Boiler Plant

In 1948 the mine operation was heated by 150 horsepower and 100 horsepower Waterous oil-fired boilers. By 1952, two 1000 kilowatts electric-fired boilers were in operation at the millsite boiler and by 1956, a Foster-Wheeler electric boiler was in operation at the mine camp boiler, outright replacing the older oil-fired units (mine records).

Mine Camp and Townsite

By the 1950s the Giant Mine was a self-sufficient townsite for employees and their families. By 1952 construction had ended and most of the mine camp facilities were complete. These included six bunkhouse units with accommodation for 295 men, a large cookery for 300 men, a three-story staffhouse, firehall with accommodations for nine men, curling rink, laundry plant, garage, and a townsite consisting of 24 family units in single and duplex buildings, plus a commissary store where discount goods and foods were available. The company also owned lots and

housing in downtown Yellowknife. A recreation hall was started in 1953 and by 1963, after several additions, the building contained a gymnasium, pool hall and game room, post-office, theater, kitchen, and library. There was also a separate curling rink, built in 1947. In 1952 there were approximately 425 employees at Giant Mine, including crews working on construction jobs. By the 1960s the workforce was cut back to 360 as operations leveled out to normal rates (mine records).

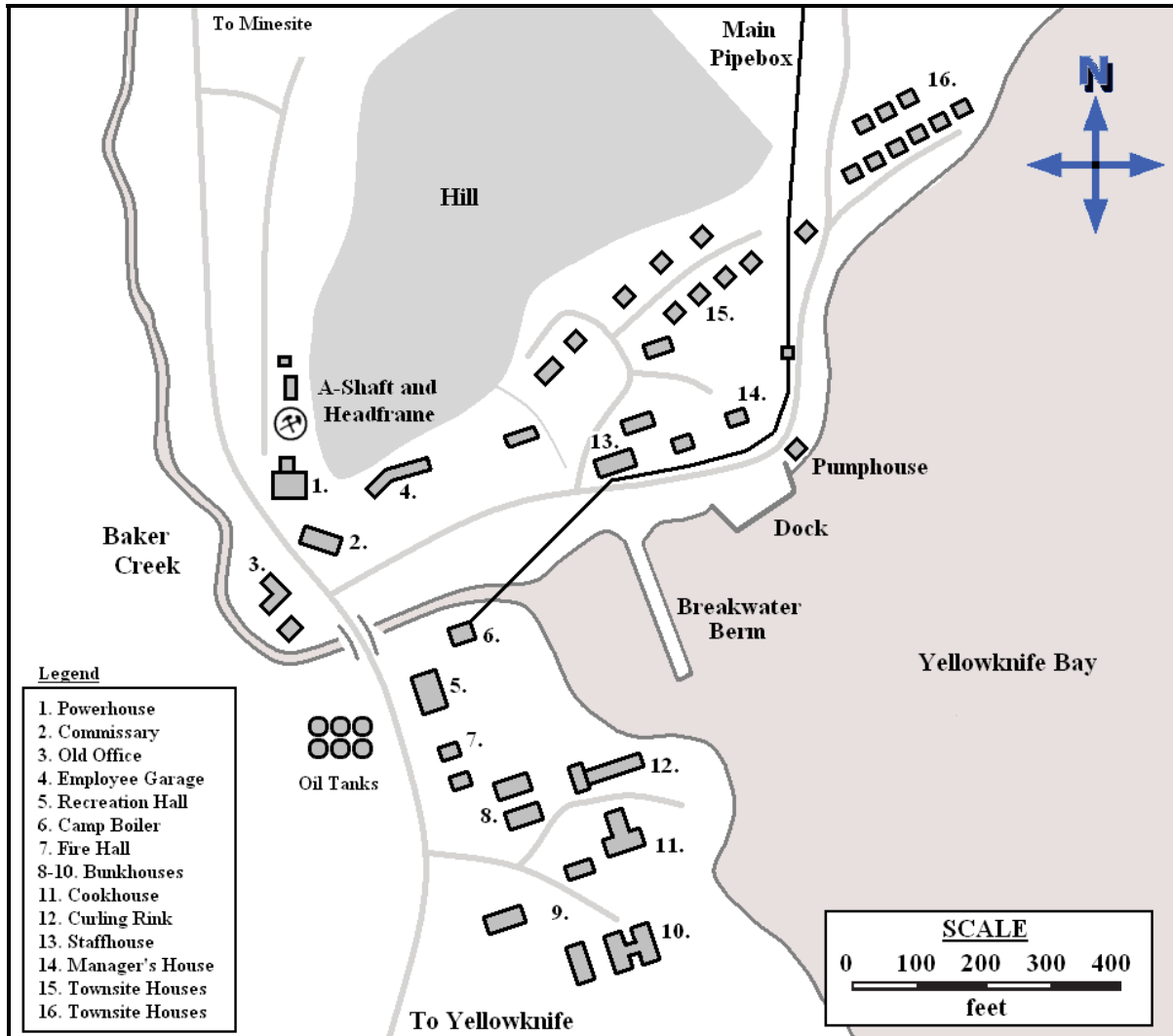


Figure 8. Giant Mine camp site, 1960. At the height of its development.

Milling Operations 1949-1953

With the introduction of the roasting plant, mill tonnage was increased to about 400 tons per day during 1949-1950. The mine now faced the problem of arsenic dust gases, which were being expelled from the roaster stack. To collect these dusts from the stack fumes, an electric precipitation (cold Cottrell) plant was installed during the summer of 1951. This expansion coincided with the increase in milling and roasting capacity through the installation of a new Dorcco roaster and other equipment in early 1952. The new roaster was in operation on May 23rd 1952 (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1950-1952).

The following is a description of the milling process as it was in 1953 (at 700-tons per day):

Ore was hoisted up C-shaft (or trucked on surface from A or B-shaft operations). Crushing on surface was two-stage, with a 30 inch x 42 inch Dominion jaw crusher and a 4-¼ foot Symons standard cone crusher as the primary crushers. Product was screened by two Dillon double-deck screens with oversize being sent to secondary crushing using a 4 foot Symons short-head cone crusher. This product was sent back to the Dillon screens, and -5/16 inch undersize material was conveyed to the mill building and split into three 500 ton fine orebins. These orebins fed an 8 foot x 10

foot Dominion ball mill in closed circuit with a 72 inch Akins spiral classifier, overflow being fed into a 12 inch x 18 inch Denver mineral jig. Some rough gold was recovered by jigs and sent for amalgamation. Jig tailings were classified through a 54 inch Akins duplex classifier, with underflows being re-ground in a secondary 6 foot x 12 foot Dominion ball mill, and overflows being sent to flotation. The flotation circuit consisted of two primary banks of twelve #24 Denver cells each, and two secondary banks of ten #24 Denver flotation cells each. Tailings from the primary cells were sent to the secondary cells, with all flotation concentrates being sent for roasting, and secondary flotation tails being rejected as mill tailings.

Flotation concentrate was split between the original Canadian Allis-Chalmers Edwards-type roaster (50 tons per day capacity) and the new Dorrco Fluosolids roaster (70 tons per day capacity). In each separate circuit, flotation concentrates were put through a 30 foot x 10 foot tray thickener, with overflow being rejected as mill tailings, and underflow being filtered in American disc filters. Filter cake was then roasted. The Allis-Chalmers plant consisted of two flat heart roasters with 16 spindles each. The Dorrco plant was a two-stage unit. Calcine was forwarded to the cyanidation circuit, and roaster gas processed through the cold Cottrell plant, which separated the hazardous arsenic dusts from the gas. Arsenic dust was pumped underground for storage in special vault, while gas was ejected from a 150' brick stack. Calcine, the gold bearing solution produced from the roasting circuit, was circulated through a cyanidation circuit consisting of thickeners and drum filters. Overflow was sent through the precipitation process and pressed by four Perrin presses to form precipitate, which was refined into gold bullion (The Canadian Mining Manual, 1953; Tait, 1957).

Mining Operations 1950s

Mining operations during 1952-1953 focused on the development of the A-shaft workings to mine the East and West zones, and the 409 and Lower B orebodies at the 750-foot level of B-shaft. Stopping and mining at A-shaft was concentrated within the East zone on the 200- and 325-foot levels, but a crosscut was driven into the West zone on the 200-foot level. B-shaft continued to contribute the largest tonnages of ore while C-shaft operations focused on completing station cutting. In 1954 the C-shaft was sunk an additional 500 feet to 1,530 feet depth, with new levels cut on the 950-, 1,100-, and 1,250-foot levels. Exploration in the C-shaft area identified the G.B. zone as a good orebody east of the shaft area on the 750-foot level. Other exploration through diamond drilling on the 750-foot level helped add to the reserve of ore within the Muir, North Giant, and A.S.D. zones (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1952-1954).

Development on the lower levels of C-shaft during 1955 outlined orebodies of substantial importance within the A.S.D. zone. All work was concentrated to the north of C-shaft, and long headings were advanced 3,500 feet north to explore and develop the northern extent of the A.S.D. zone (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1955-1956). The mining method employed during the 1950s was primarily cut-and-fill stoping, but was augmented with some open and shrinkage stoping methods. Cut-and-fill was best suited for Giant Mine because of irregular shaped and steep dipping orebodies. Prior to 1957, waste rock and gravel was used as fill material for stoping operations. Sand fill and tailings delivered from the milling plant was used starting in December 1956 as a source of cut-and-fill material. A great amount of definition drilling was required at Giant to adequately determine shape, size, and grade of the orebodies so that they would be mined economically and efficiently. This drilling was done at 25 to 50 foot intervals (Smith, 1961).

New underground installations during the mid-1950s included a machine shop on the 750-foot level, an underground crusher placed below the 1,250-foot level (at 1,300 feet depth), and an ore loading station at the 1,400 foot horizon below the crusher. An extensive system of ore passes connected all levels of C-shaft to the crusher location. All ore was reduced to -5 inches before being hoisted through the use of this crusher (Giant Yellowknife Gold Mines Ltd. Annual Report, 1956).

The main haulage levels in late 1950s were the 750- and 1,250-foot levels. On the 750-foot level, 60 cubic foot Granby-style ore cars were loaded at the stope ore passes and transported to the main haulage drive using small three ton battery Mancha locomotives. Here they were hooked up to a train of cars pulled by a three ton electric trolley locomotive which transported the cars to the main orepass at the C-shaft station. Other levels used 38 cubic foot Wabi side-dump ore cars or Granby cars and 1-½ or three ton battery locomotives. Mechanized mining machinery for drift development included Eimco 21 mucking machines in each headings to clean up waste and Wabi side-dump ore cars and 1-½ or three ton battery locomotives for hauling. Mining costs were reported to have been significantly reduced from 1955-1960. This was due to a combination of factors including increase tonnages treated per day, the closure of A-shaft mining operations, the mining of wider orebodies, improved mining techniques, the introduction of more suitable equipment, and the stabilization and training of a better workforce (Smith, 1961).

Lolor Property

Development of the G.B. zone indicated that a portion of the orebody was located within adjacent property owned by Conwest Exploration Limited. An agreement to purchase a large share in control of the 'Lolor' claim group was made in 1953 and Lolor Mines Limited was incorporated, of which 87% was held by Giant Yellowknife Gold Mines Limited. In 1954-1955, a small amount of development ore from G.B. workings within the Lolor property were mined and milled by Giant, but full production of the Lolor orebodies would not begin until October 1967 (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1953-1955).

Drilling during 1957-1958 failed to contribute to ore reserves. Continued depth development was recommended to serve as platforms for diamond drilling programs (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1957-1958). Sinking of C-shaft began in January 1959 and by June it had been completed to 2,124 feet depth. Stations were established at the 1,500-, 1,650-, 1,800-, and 2,000-foot levels. By the end of 1960, over 1,700 feet of strike length of the main shear deposit had been explored by drifting and diamond drilling on the 2,000-foot level. Only small intersections of ore had been made on the lower levels, but more work was required. Meanwhile, mining and development of the levels above 1,250 feet was quickly advancing and exploration on the 1,500-foot level had indicated substantial tonnage of ore of excellent grade. By 1958 all mining was focused on the northern section of the property around B and C-shafts as mining operations ceased at A-shaft in 1957 (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1958-1959; Giant Yellowknife Mines Ltd. Annual Report, 1960).

Metallurgical Difficulties

During the mid-1950s the operation suffered from problems with the roaster plant. Excessive dust losses and the mining of more refractory type ores caused a significant drop in gold recoveries. A new Hot Cottrell plant was ordered and new additions to the roaster and cyanidation plant were to be enacted to improve operations (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1954-1959).

Milling Operations 1960s

Additions and alterations made to the mill between 1954 and 1960 form the basis of milling operations at Giant Mine for the remainder of the mine's life (see **Figure 9**). Additions included the following: underground jaw-crusher in 1956, a new grinding circuit in 1954, new cyanidation section in 1956 to treat flotation tailings, and roaster additions including the Hot Cottrell in 1955, the kiln plant in 1957, the baghouse in November 1958, and the new Dorcco roaster on November 18th 1958. As refractory ore feed increased, free-gold in the ores became practically non-existent, and recovery by jig and amalgamation ceased in December 1958. The Hot Cottrell operated unsatisfactory as recovery of gold-bearing dust dropped to 60%, making it necessary to operate both Cottrell units at low-temperatures (Giant Yellowknife Gold Mines Ltd. Annual Reports, 1954-1959).

Ore was crushed underground below the 1,250-foot level by a 36 inch x 48 inch Buchanan jaw crusher set to 5 inch or 7 inch and was hoisted up C-shaft. Crushing on surface was two-stage, with a 30 inch x 42 inch Dominion jaw crusher and a 4-¼ foot Symons standard cone crusher as the primary crushers. Product was screened by two Dillon double-deck screens with oversize being sent to secondary crushing using a 4 foot Symons short-head cone crusher. This product was sent back to the Dillon screens, and -5/16 inch undersize material was conveyed to the mill building and split into four 500 ton fine orebins. Two 8 foot x 10 foot Dominion ball mills in closed circuit with 72 inch Akins spiral classifiers made up parallel primary grinding units. These two mills were satisfactory for operations, and the secondary 6 foot x 12 foot Dominion ball mill was maintained as a backup for when one of the larger units was out of service. Classifier underflow proceeded over a vibrating chip-screen before going to the flotation circuit. The flotation circuit consisted of two primary banks of twelve #24 Denver cells each, and two secondary banks of ten #24 Denver flotation cells each. Tailings from the primary cells were sent to the secondary cells, with all flotation concentrates being sent for roasting, and secondary flotation tails being sent for cyanidation recovery. This circuit consisted of two 40 foot tray thickeners, with overflow being sent as tailing waste, and underflow being filtered in two Oliver drum filters (12 foot x 16 foot size). Filter pulp was processed through two 26 foot x 18 foot Dorr agitators, another two 40 foot tray thickeners and two additional Oliver drum filters, where overflow was sent into the calcine circuit. Final tailings from the flotation-cyanide circuit was rejected as mill tailings or processed into mine backfill.

Flotation concentrates were sent for roasting. All roasting was now performed using the new two-stage Dorcco fluidsolids roaster. Its basic process was to chemically remove arsenic and antimony minerals and make the concentrates amenable to cyanidation. Arsenic and antimony were converted into a gaseous trioxide state; pyrite was altered to hematite; and sulphur in the concentrate acted as the fuel.

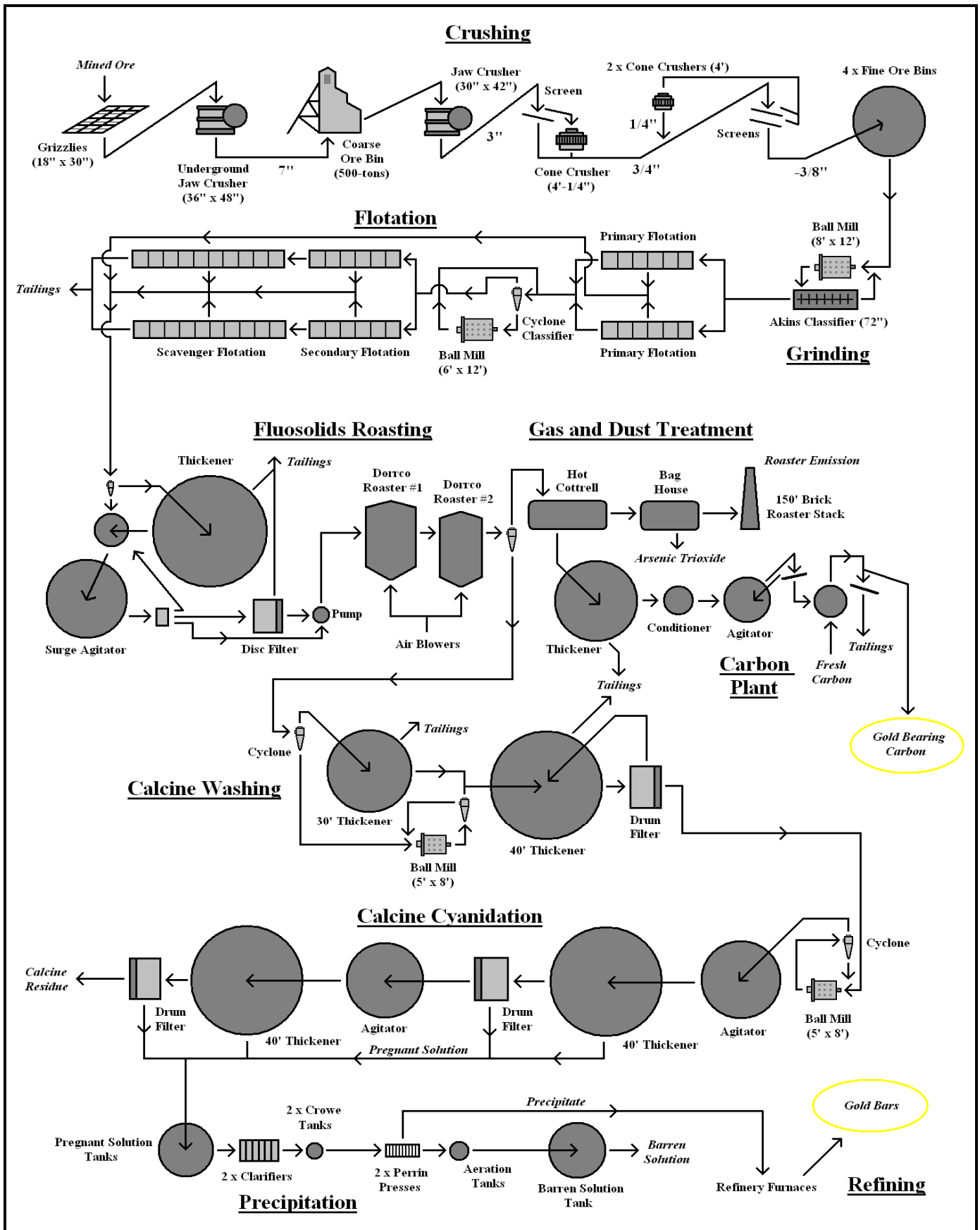


Figure 9. Simplified milling and roasting flowsheet for Giant Mine, c.1969.

On arrival at the roasting section of the milling plant, concentrate was screened to remove wood chips. It was then thickened in a 40 foot thickener, agitated in a 10 foot x 10 foot surge tank, filtered in a 6 foot three-disc filter, with pulp being further agitated to 78% solids. Material was then pumped into the roaster unit. The Dorcco roaster consisted of 1st and 2nd stage reactors, 16 feet and 14 feet in diameter, respectively. Both vessels operated at temperatures of nearly 500° Celsius. Product is processed through a cyclone, which separated dust-laden gases and quenched calcines. Gasses were sent to the Cottrell plant for cleaning, and calcine entered the calcine cyanidation circuit.

Quenched calcine was pumped back to the main mill building for cyanidation. First it was cycloned where underflow was re-ground in a 5 foot x 8 foot Dominion ball mill, and overflow was processed through a 40 foot thickener for washing. Thickener underflow was re-ground again in a 4 foot x 8 foot Allis-Chalmers ball mill in closed circuit with a cyclone where lime and cyanide were added. One stage (and two stages after 1967) of cyanidation followed using a 26 foot x 18 foot agitator, a 40 foot thickener, and a 12 foot x 16 foot Oliver drum filter. Filtrates, together with thickener overflows, were sent for precipitation to recover the final gold solution. Two parallel Merrill-Crowe units and Perrin presses (36 inch x 36 inch) provided precipitation. Gold bullion was poured using Rockwell furnaces.

After being cleaned through the Cottrell plant, arsenic dusts were sent through the baghouse plant (eight compartment Dracco-type) for separation of arsenic dusts and clean gas emissions. Arsenic, in its toxic oxide form, was pumped underground for safe storage in special concrete vaults. During operations about 15 tons of arsenic dust were recovered from the baghouse each operating day. The content was 95% arsenic oxide, with a very small gold content of about 0.15 ounces per ton. Capacity of the milling and roasting plant in 1958 was 1,000 tons per day (Tait, 1957; Pawson, 1973; Moore and Pawson, 1977).

Carbon Plant

Starting in 1960, the mine introduced a separate cyanide-leaching circuit with activated carbon absorption to recover gold-bearing dust from the Cottrell plant. This material was thickened, conditioned, and sent to a 16 foot x 16 foot leaching agitator for 24 hours. A vibrating screen caught carbon granules and recycled them back into the agitator. Pulp passing through the screen reported to a 12 foot x 12 foot stripping agitator, where fresh carbon was added as the stripping agent to which gold would adhere. Pulp was again screened to remove carbon granules, and was constantly recycled back into the agitation cycle. Underflow from the stripping agitator was discharged as waste. On a bi-monthly schedule, carbon was filtered from the leaching agitator, dried, and shipped for refining (mine records).

Other Modifications 1960s

The new 1958 roaster helped to improve operations in the electro-precipitation circuit, and in 1962 both Cottrell's were being run as high-temperature units. As flotation recovery improved during the 1960s, the cyanidation of flotation tailings became uneconomic and, in 1967, this circuit was shutdown and modified to treat the increased calcine tonnage from the roaster. On September 17th 1964, the mine poured its 5,000th gold bar (mine records).

Giant Yellowknife Mines Limited [Falconbridge Nickel Mines Ltd.] (1960-1988)

On June 30th 1960, Giant Yellowknife Gold Mines Limited and Consolidated Sudbury Basin Mines Limited merged to form a new company, Giant Yellowknife Mines Limited, and in May 1962 Ventures Limited, Giant's parent company, became a part of Falconbridge Nickel Mines Limited, who then acquired control of Giant Yellowknife.

In the early 1960s, underground work focused on the North Giant, Trough, A.S.D., and L.A.W. zones to the north of C-shaft from the 750-foot level to surface. The L.A.W. zone was discovered in 1962 by geologist Lorne A. Wrigglesworth by drilling from the northern extension of the 750-foot level. New ore developed in those areas and additional ore found in older stoping blocks replaced ore mined during the year. Drifting extending the underground workings north on the 750- and 2,000-foot levels. On the 2,000-foot level, an extensive program of drifting, crosscutting, and diamond drilling had explored the main ore zone structure for a strike length of 5,500 feet to the end of 1962. A few isolated intersections of near ore grade material was obtained in drill holes, but no continuity was established. Drilling to depth below the main structure disclosed nothing of importance, and in view of these disappointing results no further work on or below the 2,000-foot level was planned after 1962 (Giant Yellowknife Mines Ltd. Annual Report, 1962).

Development and exploration in 1963 centered on the North Giant and the L.A.W. zones where important ore indications were made between surface and the 750-foot level, helping to replace the ore mined during the year. Surface diamond drilling in the area of A-shaft disclosed nothing of importance; similar drilling operations at B-shaft area indicated a 600 foot long structure carrying narrow and erratic lenses of mineralization (Giant Yellowknife

Mines Ltd. Annual Report, 1963). The average grade of mill feed was low in 1963 as a result of erratic and irregular orebodies which increased dilution. In 1964 the mine implemented more control over dilution and brought on stream newer, more productive stopes. By this time, it was recognized that the Giant Mine ore body had been basically delimited and that aside from further development of the North Giant, L.A.W. and G.B. (Lolor) orebodies, there were no indications that any additional large tonnage deposits occurred on the Giant Mine property (Giant Yellowknife Mines Ltd. Annual Report, 1964).

The G.B. zone within Lolor Mines Limited property contained a sizeable reserve with which to augment Giant Mine production. Crosscuts were driven on the 575-foot level into the Lolor property during 1964-1965 and Lolor ore reserves were increased to 162,000 tons. Processing of development ore from the Lolor property began in 1965 and full production was achieved during October 1967 (Giant Yellowknife Mines Ltd. Annual Reports, 1965-1967).

Giant began experimenting with aluminum-alloy bodied Granby-style ore cars (the older models were all-steel) in 1963. They responded very favorably and 27 of these units were in use by the end of 1965. The aluminum-alloy Granby car was also 60 cubic foot capacity but because of its decreased weight size (33% weight savings), ore trains could be longer without increasing the size of locomotives. As a result, underground operations experienced increased tramming productivity between 1964 and 1965. The aluminum Granby cars were manufactured by Dorr-Oliver-Long Limited (mine records).

Exploration and development during 1965 within the Giant claim group concentrated on the L.A.W. and North Giant zone, and within the Upper C and Lower B areas where most of the reserves were added. Surface diamond drilling was carried out on the U.B.C zone, the West B zone, and on the westerly extensions of the North Giant zone (Giant Yellowknife Mines Ltd. Annual Report, 1965). Mining operations suffered in 1966-1967 due to lower grades because of the exhaustion of large, highly productive, high-grade stopes. Instead, Giant began mining a greater number of small, irregular stopes with greater risks of dilution. In addition, a large portion of production was being derived from lower-grade ore blocks. A shortage of skilled miners resulted in the completion of less exploration development, however diamond drilling continued at a high rate. No new ore zones were discovered and only small amounts of ore were added to the reserve, not enough to replace the ore mined during the year. Some success was made in improving mining techniques with new equipment (Giant Yellowknife Mines Ltd. Annual Reports, 1966-1967).

Supercrest Property

In 1965, development also began on accessing the Supercrest property, located north of Giant Mine and previously owned by Akaitcho Yellowknife Gold Mines Limited. Considerable exploration had been carried out after World War II but development of a mine was halted in 1950 after collaring a short shaft, erecting a 90 foot steel headframe, installing equipment, and constructing complete plant and camp facilities, due primarily to the high-costs involved in putting gold mines into production at that time (Lord, 1951).



Giant Yellowknife Mines Ltd.

Figure 10. Mining operations at Giant, 1970s.

In October of 1964, Giant Yellowknife Mines Limited and Akaitcho formed a new joint-venture company to mine and produce from the Akaitcho property - Supercrest Mines Limited. Ownership was split on a 50-50 basis. A long drive on the 750-foot level of Giant Mine was advanced north in the Supercrest property and diamond drilling commenced to confirm the existence of ore within the zone at depth (Giant Yellowknife Mines Ltd. Annual Report, 1964).

Work carried out during 1965 included 1,651 feet of drifting and crosscutting, 265 feet of sub-drifting, 1,244 feet of raising, and over 15,800 feet of diamond drilling. A further 1,636 feet of lateral development was conducted in 1966 on the 575- and 750-foot levels, together with 732 feet of raising. An Alimak raise was driven through to the surface and broke-through at the old Akaitcho shaft. This raise was used for ventilation and an emergency escapeway. Commercial production of Supercrest ores began during October 1967 (Giant Yellowknife Mines Ltd. Annual Reports, 1965-1967).

The 1,100-foot level at Giant was extended into the Supercrest zone during 1969 and development of the lower levels began. Reserves within Supercrest were thus dramatically increased during the year. However, despite an active exploration program in the Giant Mine ore areas, no significant tonnages of ore were discovered in 1969 (Giant Yellowknife Mines Ltd. Annual Report, 1969).

Increasing Costs

Shortage of skilled miners and a general rise in the cost of operations plagued the Giant Mine during the late 1960s and early 1970s. These increased costs forced an upward revision of the mine cut-off grades to 0.46 ounces per ton gold for the Giant and Lolor properties, and to 0.50 ounces per ton gold for Supercrest. As a result, certain low-grade blocks, previously carried as reserves and totaling 275,000 tons, were dropped from the mine plan (Giant Yellowknife Mines Ltd. Annual Report, 1967). Compounding the situation was the deterioration in the free market for gold during 1969 and the freeze of its price. All gold was being sold to the Royal Canadian Mint by this time. The company applied for government assistance through the Emergency Gold Mining Assistance Act during 1969 to aid in operating costs. As a result, all gold produced at the Giant Mine was sold to the Royal Canadian Mint at a fixed rate slightly higher than market value (Giant Yellowknife Mines Ltd. Annual Report, 1969). The destruction by fire of Con Mine's bluefish hydro power plant in January 1971 resulted in large stresses placed on the Snare River hydro power plant, causing extensive blackouts at Giant Mine and increasing costs. A new labour agreement, signed in 1970, also affected operating costs for 1971 (Giant Yellowknife Mines Ltd. Annual Reports, 1970-1971).

Although operating costs were very high in 1970, the costs per ton milled decreased as a result of increased production (combined Giant, Supercrest, and Lolor) and a reduction in mining development. Revenue, however, declined as a result of lowering ore grades. Giant production was lower than 1969 due to reliance on narrow stopes. Supercrest production was derived primarily from the 750-foot level, but two blocks of ore on the 110-foot level also contributed. Tonnage and grade at Supercrest improved and as a result gold production was higher compared to 1969. Development of the lower Supercrest orebodies on the 950- and 1,100-foot levels was on schedule with plans to begin production from these areas in 1971 (Giant Yellowknife Mines Ltd. Annual Reports, 1970-1971).

Declining Ore Reserves

At year-end 1970, Giant management estimated ore reserves would last for another three years of production unless additional economic ore was developed. Exploration during the year again failed to locate significant tonnages of new ore. Some ore was found below the 750-foot level in the Supercrest zones, but overall the outlook was bleak (Giant Yellowknife Mines Ltd. Annual Report, 1970).

1970s Turnaround

In 1971, gold began to rise in value as the marketplace turned away from the U.S. dollar and world governments allowed the metal to find its own level on the markets, and by 1972 the average price received for gold was CDN \$59.87 per ounce. In the summer of 1971, Giant Mine began selling its gold product on the world markets at a higher value than in the past. Operations at the mine continued to decline however, due to increased costs and lack of ore. Considerable expenditure and effort was made in exploration during 1971 but again no new tonnages were added to the reserves. Definition drilling of ore areas previously designated as 'probable' indicated mining possibilities within the Giant and Supercrest sections of the mine, including a block of ore in the old A-shaft area. Production in 1971 was primarily from the Supercrest and Lolor properties, and as a result the Giant orebodies received lower tonnage treatment. Development of all known Lolor orebodies within the G.B. zone were essentially completed during the year (Giant Yellowknife Mines Ltd. Annual Report, 1971).

A-Shaft Reopens

The price of gold continued to rise favorably during 1972-1973, permitting the development and mining of lower grade ore bodies within the mine. Mining of the A-shaft area (West, Creek, and East zones) re-commenced in the summer of 1972 following a 15 year shutdown (Giant Yellowknife Mines Ltd. Annual Report, 1972). The old shaft and headframe were reconditioned and placed back into operation as a service shaft. Mining operations at A-shaft continued normally except for a brief two-week shutdown in September 1976 following an underground explosion at the 1st level of A-shaft, set off by a disgruntled employee. No one was hurt and the shaft was put back into operation following some minor repairs (mine records).

The price of gold continued to rise during 1973 and the cut-off grade was lowered to correspond to economic conditions. Long-hole stoping mining methods were introduced on a large low-grade stope in the Upper C area of the A.S.D. zone. Diesel driven Load, Haul, Dump (L.H.D.) machinery (scooptrams) were put to use to handle surface ore in the B3 section and low-grade ores in the Upper C area. Exploration programs picked up quickly in areas previously

considered sub-marginal. Mining operations continued normally at the Lolor and Supercrest properties, although shortages of skilled miners did have a negative effect (Giant Yellowknife Mines Ltd. Annual Report, 1973).

The ore reserve statement at the end of 1974 shows the dramatic improvement in conditions at Giant Mine. After mining 328,099 tons of ore in 1974, the ore reserves at year-end were 2,400,000 tons grading 0.33 ounces per ton gold, an increase of 1,227,000 tons over year-end 1973. About 400,000 tons of this was located by diamond drilling in the B-shaft area below the 750-foot level in the A.S.D. and G.B. zones. The remaining increase was obtained throughout the mine by applying the lower cut-off grades to ore within the B-shaft area, Upper C, Trough, and A-shaft West zone (Giant Yellowknife Mines Ltd. Annual Report, 1974).

Open Pit Mining

As a result of higher gold prices, ore zones closer to the surface, previously too low-grade, were now economic to mine by open pit methods. The first open pit was started in 1974 on the West zone across from A-shaft (A-1 open pit) (Giant Yellowknife Mines Ltd. Annual Report, 1974). Full production was achieved in 1975.

Other new developments during 1974 included the introduced of L.H.D mechanized scooptrams in the A-shaft area, and a diesel Jumbo drill for high-speed drifting. Diamond drilling in the Lolor area did not add to reserves, however exploration of previous sub-marginal ores at Supercrest added some reserves in the upper ore bodies. Milling operations were hampered by shortages of ore, lower-grades, and the resulting lack of concentrates for the roaster. (Giant Yellowknife Mines Ltd. Annual Report, 1974)

Management and Employees

Operations were still seriously affected by shortage of labour during the early 1970s. At year-end 1974, the work force was down to 327 as compared to 342 at the end of 1973 and 371 at the end of 1972. Labour turnover increased from 143% to 190%. The company made attempts to promote long-term employment by increasing recreational services and providing better housing at Yellowknife. In 1974, the following management personnel were at Giant Mine: Dave J. Emery, mine manager; A.K. Campbell, general superintendent; R.S. Brown, mine superintendent; Hal E. Pawson, mill superintendent; Chet M. Wilkinson, electrical superintendent; Robert W. Spence, exploration manager; H.B. Bye, master mechanic; J.A. Crossfield, construction foreman; C.S. Sra, mine engineer; B.F. Watson, chief geologist; L.F.G. Borden, accountant; Jim W. McKay, purchaser; and A.T. Rivett, personnel manager (Giant Yellowknife Mines Ltd. Annual Report, 1974). In 1975-1976, a new union was organized to represent Giant Mine workers: The Canadian Association of Smelter and Allied Workers (C.A.S.A.W.) Local No. 4. This new union replaced the former United Steelworkers of America (Giant Yellowknife Mines Ltd. Annual Report, 1976).

Environmental Concerns

During 1974, the mine began to investigate better methods of pollution control to bring the operation up to the standards of the day, especially in terms of arsenic management. New, engineered arsenic chambers were excavated underground, and the tailing pond dams were improved during 1975. The company reported that it was keeping up-to-date on all technological advances towards the elimination of hazards from effluent and emissions from the milling and roasting plants (Giant Yellowknife Mines Ltd. Annual Reports, 1975-1976).

Mining highlights in 1975 included full production of the A-1 open pit (29% of mill feed during the year), the start of pillar recovery in high-grade portions of the Upper B area, and an expansion of long-hole stoping in the Upper C area. The price of gold leveled out during the year against rising operational costs. As a result, cut-off grades were raised to compensate, deleting a large amount of prior reserve material. The 72,000 tons of ore added to reserves during the year did not compensate for the loss of the lower-grade material. Ore reserves at year-end 1975 were 1,950,000 tons of ore grading 0.33 ounces per ton gold (Giant Yellowknife Mines Ltd. Annual Report, 1975).

The initial mining of the A-1 open pit was completed in September 1976, and a start was made on stripping three new, smaller pits. The mine purchased the required heavy equipment and operated the open pits itself rather than using a contractor. This lowered costs and improved productivity. Production of Lolor and Supercrest ores was hampered by labour shortages, but exploration revealed some new ore above the 1,100-foot level of Supercrest (Giant Yellowknife Mines Ltd. Annual Report, 1976).

Diamond drilling was conducted in 1977 below the 2000-foot level to test unexplored ground. One hole penetrated to 2,700 feet below the 2,000-foot level, and while some short sections of favourable schist were encountered, no gold values were found (Giant Yellowknife Mines Ltd. Annual Report, 1977). Higher gold prices in 1978-1979 resulted in increased diamond drilling in previously abandoned places of the mine.

Mining Operations 1970s

In 1977, the mine switched from cut and fill mining methods to long-hole and shrinkage stoping methods in order to reduce operational costs. More L.H.D equipment and larger mechanized machinery, including drilling Jumbos, were also being introduced during the late 1970s (Giant Yellowknife Mines Ltd. Annual Report, 1977).

Higher Gold Prices

Substantial increases in the price of gold created record profits for the company during 1978-1979. In 1978, the mine saw improved ore grades and recovery, resulting in higher production. There was lower tonnage produced however, due to breakdowns in the mill's classification circuit and a one-week maintenance shutdown of the roaster. Power outages, and other mechanical breakdowns plagued the operation during the year. There was a reduction in open pit mill feed, due to a planned balanced utilization of open pit and underground ore reserves. Exploration drilling below the 575-foot level at A-shaft was conducted in 1978, but this work ceased in 1979 in favour of drilling in higher potential areas of the mine (Giant Yellowknife Mines Ltd. Annual Report, 1978). In 1979, the mine suffered from several setbacks yet still managed to pull in record profits due to the higher gold prices. The open pit was shutdown for two-weeks in February 1979 because of extreme cold weather, and a ten day shutdown of the roaster occurred in September for maintenance. Mining switched to lower-grade areas of the underground due to poorer than expected grades in other areas of the mine. There was some difficulty in treating this lower-grade ore because of the lack of sulfur content, which was necessary for proper function of the roasting plant. In an effort to improve recoveries, an expansion of the carbon plant was undertaken to ensure better recovery of gold from roaster dust (Giant Yellowknife Mines Ltd. Annual Report, 1979). Modifications to the roaster in 1980-1981 helped to boost recoveries from the lower-grade ores. Mining operations at the Lolor property continued during 1979. Mining at Supercrest was reduced as part of the production plan. A drift drive on the 1,500-foot level was advanced from Giant into Supercrest property during 1978-1979 to reach new ore reserves on that level (Giant Yellowknife Mines Ltd. Annual Report, 1979).

The 1980s

The 1980s began at Giant Mine with two strikes in 1980, the first for a two-week period in April and the second lasting from July 10th to October 24th. The strikes were generally in response to all-time high gold prices; union members felt they should be given higher wages to match the increase value of gold. The strike occurred at a bad time for the company, because four months of what would have been very profitable production had been lost. The strikers held all the cards in the strike, and workers came back to the job with increase wage rates and other benefits. Lower gold prices followed and management considered closing down Giant Mine. Underground production was not achieving anticipated rates and open pit production was relied upon more and more to feed the mill. As a result of the lower-grade and the difficulties in roasting those ores, production declined in 1980 and 1981. The result was selective mining of more high-grade areas of the mine, and modifications to the milling and roaster plants including an expanded carbon plant and new flotation cells (Giant Yellowknife Mines Ltd. Annual Report, 1980). Deep diamond drill holes encountered very low-grade ore beyond the 4,000-foot level but drilling for parallel vein structures on the 1,100-, 1500-, and 2000-foot levels did not reveal anything of interest. By this time it was obvious that no economic ore deposits at Giant Mine existed below the 1,500-foot level (Giant Yellowknife Mines Ltd. Annual Report, 1981).

Environmental Regulations

New environmental regulations in the 1980s demanded that Giant Mine re-treat its wastewater to destroy residual cyanide before it entered the watershed. The new water treatment plant using a chlorine process to destroy the cyanide entered operations in 1982. Other work towards environmental protection included a new tailings dam at the South Pond in 1983. (Giant Yellowknife Mines Ltd. Annual Reports, 1982-1983)

Year:	<i>1981</i>	<i>1982</i>	<i>1983</i>	<i>1984</i>	<i>1985</i>	<i>1986</i>	<i>1987</i>
Marketable Arsenic Produced:	1,205 tons	1,500 tons	800 tons	1,234 tons	2,265 tons	447 tons	none

Table 2. *Arsenic trioxide sales from Giant Mine, 1981-1986. (source: Giant Yellowknife Mines Ltd. Annual Reports, 1981-1987)*

Arsenic Trioxide Shipments

Partly due to new markets and partly due to the environmental liability of arsenic dusts, Giant Mine built an arsenic trioxide handling system in 1981 to store and ship arsenic trioxide dusts for treatment. The arsenic trioxide was converted into copper chrome arsenate and used in the wood preservative industry during the 1980s (Giant

Yellowknife Mines Ltd. Annual Report, 1981). The market later collapsed, but by 1986 the mine had shipped 7,451 tons of arsenic trioxide dust to United States customers (see **Table 2**).

Production of Supercrest ores was affected by the 1980 strike, and operations did not resume until March 1981. Supercrest mining ceased at the end of 1981 due to uneconomic ore reserves, and pending a decision if further work was warranted in light of low gold prices. The drift drive on the 1,500-foot level was not completed (Giant Yellowknife Mines Ltd. Annual Report, 1981). Ore reserves at Supercrest in 1989, calculated after a 41,000 feet diamond drilling campaign, were reported as 224,000 tonnes of ore grading 0.39 ounces per ton gold (Akaitcho Yellowknife Gold Mines Ltd. Annual Report, 1989). Lolor production continued until 1985. The entire operation was plagued with similar problems. Falling gold prices and higher costs of operations (including labour, power, maintenance, and supplies) did not make the mine very profitable. Meanwhile, drilling exploration in 1981 encountered new deposits, including mineable ore south of B-shaft. This would be open pitted through the B-2 pit. Deep diamond drilling encountered very low-grade ore beyond the 4,000-foot level in 1981 and an attempt to locate parallel ore structures on the 1,110-, 1,500-, and 2,000-foot levels failed. Development in 1982 focused on mining higher-grade areas, but it was found that much lateral work was required to reach the higher-grade deposits. These higher gold values, plus grade control and improvement in mill recoveries helped to offset lower gold prices (Giant Yellowknife Mines Ltd. Annual Reports, 1981-1982).

Cessation of A-Shaft Mining

Underground mining of the A-shaft ore bodies ceased in December 1982 following the depletion of economic reserves. It was reported that 200 tons of broken ore remained to be recovered, but the freezing of water lines in the shaft prevented the completion of mining at A-shaft. Soon after abandonment of the workings, concrete and steel bulkheads were installed on the 750- and 575-foot levels to close off access to the area from the C-shaft workings. The hoist and cage system was mothballed in early 1983 (mine records).

It was recognized by company engineers that the only way to make mining at Giant profitable would be to develop new trackless mining methods within the old workings. Although trackless equipment had been in use since the late 1970s, all equipment had to be brought into the mine via C-shaft. A decline from surface to deeper levels of the mine would result in expedient and profitable mining of areas previously inaccessible. Mechanized machinery would also lower labour requirements and generally lower operating costs. In 1983, the mining rate was reduced by 25% to obtain a more complete recovery of the mineral reserves. This new mine plan also called for a five-day milling week, cutting back underground tonnage, and reducing workforce through attrition, early retirement, and layoff. Employees were reduced from 315 to 280 (Giant Yellowknife Mines Ltd. Annual Report, 1983).

D.W.C. Zone

Work began during the summer of 1983 to put the D.W.C. zone into production. This was the showing uncovered by Don W. Cameron in 1938 which sparked renewed exploration at Giant Mine in 1943. It is located on the very south end of the property. The zone was estimated to contain 32,500 tons of ore grading 0.31 ounces per ton gold. A decline was collared on the bottom of the A-2 open pit and advanced into the zone during 1983-1984. Mining was then conducted by raising and stoping to the surface (Giant Yellowknife Mines Ltd. Annual Reports, 1983-1984).

U.B.C. Zone

As part of Giant's modernization program was the development of new vein structures within the U.B.C. zone, a narrow, high-grade vein with strike and dip potential northwest of B-shaft. Work in 1983 and 1984 indicated 18,000 tons grading 0.56 ounces per ton gold. A ramp collared on the bottom of the B-2 open pit to access the U.B.C. zone was planned to become the main decline entrance to the minesite for future trackless mining operations. Mining commenced in 1985 and the ramp was driven 1,894 feet into the zone. By year-end, about 100 feet of ore grading 0.50 ounces per ton gold was opened up by drifting at the bottom of the zone, 235 feet below surface. Higher gold production in 1987 was credited to the U.B.C. stope (Giant Yellowknife Mines Ltd. Annual Reports, 1983-1987).

Satellite Mine at Gold Lake

Giant had other operations from which to extract profits during these troubling economic times. First and foremost was the Salmita Mine, but in 1985 the company also began the development of the Gold Lake Mine deposit north of the property (**Gold Lake Mine** has a separate description in this publication). It was a small high-grade gold deposit accessible by decline development. Production of Gold Lake ores was conducted between 1986 and 1988 using the Giant plant for milling. The company also made an agreement with Treminco Resources Limited for the custom milling of ores from the Ptarmigan and Tom Mine in 1986. The increased tonnage of trucked ore from outside properties necessitated in the construction of a surface crushing plant in 1986. Previously, open pit ores from Giant

Mine were dumped down ore passes to be crushed underground in the normal underground production circuit. The surface crusher also allowed for the introduction of custom ores into the milling process without mixing it with Giant Mine ores (Giant Yellowknife Mines Ltd. Annual Reports, 1985-1987).

Milling Problems

Mill recovery during most of the 1980s was poor due to lower grade ores entering the circuit and higher antimony concentrations within the ore, which impacted roaster operations. The roaster was shutdown almost regularly due to these problems. It was often necessary to operate the mill on the weekends to make up lost recovery. Other modifications were made to increase gold recovery including a carbon column recovery unit installed at the final decant point from the tailings discharge, to capture trace amounts of gold in the effluents. This circuit was fully operational in 1984, with carbon material sent to the United States for gold recovery. The antimony issue in the roaster circuit was also solved by 1984 as levels in the ore dropped. A new carbon circuit was installed in 1986 to permit the direct recovery of gold rather than shipping gold loaded carbon out, also permitting the re-use of carbon (Giant Yellowknife Mines Ltd. Annual Reports, 1984-1986).

Giant Yellowknife Mines Limited [Pamour Incorporated] (1986-1990)

On August 12th 1986, Pamour Incorporated of Australia purchased Falconbridge's 19% interest in Giant Yellowknife Mines Limited and 36.7% interest in Akaitcho Yellowknife Gold Mines Limited (a shareholder of the Supercrest property). On March 10th 1987, Pamour completed the purchase of 50.2% of Giant shares, obtaining a controlling interest in Giant Mine (Giant Yellowknife Mines Ltd. Annual Reports, 1986-1987).

Tailings Retreatment Plant

Serious consideration to the re-treatment of old Giant Mine tailings was made during 1986. In earlier operations, considerable gold values were lost in the tailings, which were estimated to contain 7 million tons of material grading 0-067 ounces per ton gold. Construction of a retreatment plant began in 1987 and was completed in May 1988 at a total cost of \$29 million. The plant was a seasonal operation, usually running from May to October. In 1989, the plant was expanded to handle 10,000 tons per day with an addition of thickener, wood screens, and carbon columns to handle thickener overflows independently from the leach circuit.

There was an improvement in gold production during 1987 as a result of a higher-grade stope in the U.B.C. zone and better than expected grades in the B-3 open pit. Development work was focused on establishing decline ramps from surface (branching off the U.B.C. ramp) to the 1,250-foot level to enable more efficient servicing of underground mobile trackless equipment. Open pit mining of the C-1 pit was completed, mining of the B-3 open pit commenced in February 1987, and in the A-2 open pit during July 1987 (Giant Yellowknife Mines Ltd. Annual Report, 1987).

Mining Operations 1980s

Underground operations accelerated during 1988 as decline development reached the 1,200-foot level and more trackless and mechanized machinery was introduced at Giant Mine. The 1,500-foot level was reached by decline in 1989. (Giant Yellowknife Mines Ltd. Annual Report, 1989) Three underground mining methods were used at Giant during the 1980s: longhole, shrinkage, and cut and fill. These respectively produced about 50%, 10% and 10% of the underground requirements. The remaining 30% of underground ore came from development work. Longhole stoping was prominent due to lower costs and compatibility with the nature of ore deposits at Giant. Mechanized machinery included 2 and 3 yard Jarvis-Clark scooptrams, Jarvis-Clark Jumbo drills, and 13 ton Eimco haul trucks (mine records).

As part of the modernization program, the C-shaft headframe was upgraded (with new steel stiff legs) and new air compressors were installed. A new ventilation plant at both the B-shaft and Akaitcho shaft were commissioned in 1989. The new air compressors and the ventilation plant were propane-fired, an economical alternative to diesel generation. The water treatment plant was upgraded to convert the plant from chlorine treatment to a hydrogen peroxide process in 1989-1990, a more advanced and clean method of breaking down cyanide in the mill effluent (Giant Yellowknife Mines Ltd. Annual Reports, 1989-1990).

Warox Project

In 1989, a new arsenic trioxide re-treatment project was proposed known as Warox. It involved re-treating the arsenic trioxides into a high quality industrial product for the wood preservative industry. Gold could be recovered from the crude feedstock during the purification process. Testwork was to be completed and a production decision was to be made in 1990. The project was cancelled during cutbacks in 1990.

End of Open Pit Mining

Open Pit mining ceased in June 1990 upon depletion of reserves. The A-2 pit was mined out in 1988, and the last pits to be mined were the B-1 and the A-1 pit, the later of which was reactivated briefly in 1989. Throughout the 1980s, the open pits had provided about 1/3 of mine production. To match the output of previous years, it was necessary to accelerate production by introducing more mechanized machinery and adding to reserves (Giant Yellowknife Mines Ltd. Annual Reports, 1988-1990).

Mining costs at Giant continued to increase during the late 1980s, and the company began to suffer financial loss. Operating costs in the first quarter of 1990 were US\$425 per ounce. The Pamour group of companies was suffering from other financial problems at the same time, and the decision was made to sell off the Canadian gold mining assets of the organization in late 1989.

Year:	<i>1988</i>	<i>1989</i>	<i>1990</i>	<i>Total:</i>
Tailings Treated:	944,832 tons	1,094,204 tons	765,180 tons	<u>2,804,216 tons</u>
Gold Recovered:	6,780 oz	22,071 oz	20,486 oz	<u>49,337 oz</u>

Table 3. *Tailings Retreatment Plant (T.R.P.) production, 1988-1990. (source: Giant Yellowknife Mines Ltd. Annual Reports, 1988-1990)*

Royal Oak Mines Incorporated (1990-1999)

In the summer of 1990, junior Canadian mining company Royal Oak Resources Limited, headed by Peggy Witte, began negotiations for the purchase of the Canadian gold mining assets of Pamour Incorporated, including Giant Yellowknife Mines Limited and the Giant Mine itself in Yellowknife. The sale was due to financing problems in Australia which forced Giant Resources Limited, a major shareholder in Pamour Inc., to rid its debt by selling the Canadian assets. Royal Oak would also gain control of the subsidiary companies Akaitcho Yellowknife Gold Mines Limited and Supercrest Mines Limited. The \$33 million deal was closed in November 1990. The companies Giant Yellowknife, Akaitcho Yellowknife, Supercrest, and other Pamour subsidiaries, were amalgamated to form a single new company – Royal Oak Mines Incorporated, which was officially created in July 1991 (The Globe and Mail, June 12th 1991).

TRP Closure

At the end of the summer season of 1990, the Tailings Retreatment Plant shutdown as scheduled. When Royal Oak took control of the mine, they reviewed the economics of the project and decided that the plant should not be re-commissioned the following year. Twenty-eight positions were permanently terminated. The plant has been abandoned since that time. Between 1988 and 1990, the plant processed 2,804,216 tons of tailings to produce 49,337 ounces of gold (see **Table 3**). Royal Oak made major cutbacks at the Giant Mine to make it a more profitable operation. Operating costs in late 1990 were estimated at CDN\$340 per ounce. Royal Oak's objective was to lower costs to CDN\$250 per ounce. Employees were cut to 312 versus the 351 positions in place at the beginning of 1990. New mine management positions were filled and most senior staff was replaced (Giant Yellowknife Mines Ltd. Annual Report, 1990).

Giant Mine Strike 1992-1993

Cost cutting measures implemented by aggressive Royal Oak management became a sore point for long-time Giant Mine workers and the union that represented them. The company offered to maintain hourly wage rates, but trim benefits and tie future wage increases to the price of gold. The union felt this deal was humiliating and on May 23rd 1992, the union voted 80% in favour of rejecting the new tentative agreement and went on strike with 230 members. The union blamed the unethical 'American-style' management that the company was enforcing on the workers. The company claimed the agreement was fair considering the low price of gold and high-operating costs, emphasizing the fact that Giant miners were among the best paid in the country with many good benefits (The Financial Post, May 24th 1992).

The company decided to continue mining operations on a reduced scale by bringing in replacement workers. This action upset the striking union members, resulting in much violence on the picket line. Certain strikers began guerilla

tactics to vandalize and bomb mine facilities, including the vent plant at B-shaft, telephone and power lines, and satellite dishes. Many union members also decided to cross the picket line. Replacement workers were threatened and violence carried over into Yellowknife where emotions ran high between union and company factions. The mine employed 125 contract miners in June 1992 and was producing at a rate of 1,250 tons per day. Because of the shortage of workers, mining operations were focused on areas that could be mined with less manpower and equipment. Sixty working stopes were active as opposed to the 100 working places normally in operation. Mining and milling operations were increased from 5 days per week to 7 days per week to help catch up.

The strike violence soon got out of hand, and on September 18th 1992 striker Roger Warren snuck into the underground workings and set an explosion on the 750-foot level tramway line, killing nine miners. The mine was closed temporarily while the authorities investigated the incident. Roger Warren would later be tried and convicted for the murders. Mining operations resumed on September 25th 1992; however, the strike continued. In October 1993, the Canada Labour Board became involved and required Royal Oak to submit a new labour agreement. The company responded by tabling the original offer, but this time the union voted 96% to accept the deal. The strike was finally settled in late November, and the remaining strikers came back onto the job in early December 1993.

Mining Operations 1990s

In 1994, production at Giant was at a rate of 1,300 tons per day through the C-shaft and ramps. All production was entirely from underground, and mining methods were cut and fill (75%), longhole (15%), and development work (10%). Mechanized mining was an important part of mining, but conventional mining was still used to deal with smaller working areas. Mining methods had to be flexible enough to deal with the small pods of ore common at Giant Mine. Stopes were backfilled using development waste. Mining was concentrated largely on pillar recovery and haloes of previously mined orebodies (mine records).

Supercrest Back in Production

Work in 1994 focused on preliminary development of the Supercrest orebodies, idle since 1981. The plan was to put the orebody into accelerated mining so that Giant Mine was not dependent on Supercrest nearing the end of its life. A new fleet of Jarvis-Clark three yard scooptrams were purchased. A new tramline on the 1,500-foot level was commissioned during 1995 and brought up to targeted capacity in 1996, allowing higher grade Supercrest ores on a productive schedule (Royal Oak Mines Inc. Annual Reports, 1994-1995). Even still, production declined during 1996 as a result of technical difficulties underground, and complications in accessing some higher-grade areas north of C-shaft. Most ore feed was being derived from small, hard to develop stopes. Increased power rates were a factor in the higher operation costs during 1996. A third of Giant Mine's costs were attributed to the purchase of regional hydro power. The company considered the installation of high-speed diesel generators to produce cheaper power at the mine site. A three-week production shutdown was experienced in November 1996 due to a failure of the primary hoist. Development was also underway during 1996 to tap into potential new resources within the L.A.W., Lower B, and Upper C zones (Royal Oak Mines Inc. Annual Report, 1996).

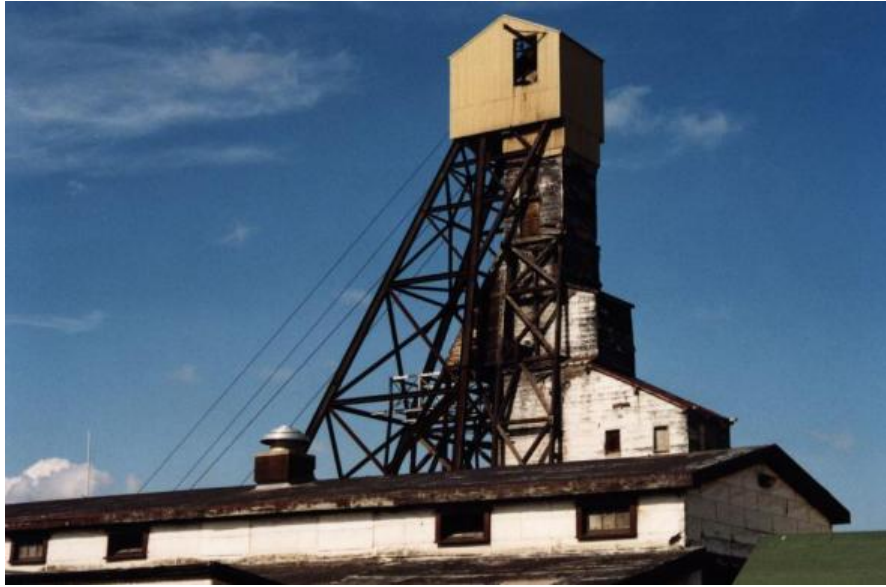
Mining Fleet 1990s

Giant Mine had a large mobile mining fleet during the 1990s. The following is partial list of the equipment in use: six EJC MJM-20B Jumbo drills, six EJC JDT-413 underground haul trucks, nineteen Mancha locomotives, twelve Titan locomotives, two Gator trucks 6x4 (1998 models), five Tamrock HEL-550 pneumatic drills, five Jarvis Clark 300M scooptrams (3-yard) (1993 model), four Jarvis Clark 250M scooptrams (2-yard) (1983 model), three EJC scooptrams (3-yard) (1996 model), two Eimco 921 scooptrams (1988 model), seven Jarvis Clark scooptrams (3-yard) (1987 model) Electricity was available from the Snare River Hydro Plant. Two backup diesel generators were kept on standby. Compressed air was supplied with Canadian Ingersoll-Rand units of 1000-, 1500-, and 2500 cubic feet per minute at the C-shaft hoist room. Two additional Canadian Ingersoll-Rand air compressors (1500, 1000 cubic feet per minute) were in use in the roaster plant. At the B-shaft vent plant were four Quincy air compressor units. Boiler units consisted of two H.B. Smith units at C-shaft, and a Foster-Wheeler and two Cleaver-Brooks boilers at the camp boiler house (mine records).

The company and mine operations were dramatically affected by the sharp decrease in the price of gold during 1997. New ore from Supercrest helped achieve higher grades during 1997 and 1998, but overall production was not as high as previous years. The ore at Supercrest had an average grade of 0.39 ounces per ton compared to the original Giant workings which contained reserves of 0.30 ounces per ton gold (Royal Oak Mines Inc. Annual Reports, 1997-1998). In October 1998 the Government of the NWT agreed to provide Royal Oak with up to CDN\$1.5 million in funds to continue with exploration and development at Giant Mine in an effort to delineate further gold deposits. Later programs, entitled EX-Tech and funded largely by the Government of the NWT, would focus on finding the

relationship between the Giant and Con ore deposits in hope of finding additional pockets of gold (Royal Oak Mines Inc. Annual Report, 1998).

During 1999, 50% of production was derived from the Supercrest zone. Production in the Giant Mine core area was primarily from the Upper, Mid, and Lower B and C areas, the B3/B4 area, the C-Trough zone, plus small amounts of ore from the 1,110-foot level, the G.B. zone, and the U.B.C. zone. Underground exploration diamond drilling was conducted throughout the mine and totaled 38,639 feet. 26,741 feet of this footage was within the Supercrest zone, and added 26,741 tons of ore grading 0.27 ounces per ton to the Supercrest ore reserve. Drilling from the 1,500-foot station of C-shaft was to test a largely unexplored section of ground between C and A-shafts. This work was a continuation of the G.N.W.T. funded exploration as outlined above. (Miramar Giant Mine Ltd., 1999)



Ryan Silke

Figure 11. *Giant Mine C-shaft, 2000.*

Royal Oak Bankruptcy 1999

In February 1999, Royal Oak Mines Inc. filed for bankruptcy. The company had drained its treasury during the construction of a gold and copper mine in British Columbia. Falling commodity prices played a major role as well. The Giant Mine was placed under the protection of creditors and, under the management of Price-Waterhouse Coopers (interim-receiver), was allowed to continue operating. Ownership of the surface rights reverted back to the Crown and the Department of Indian and Northern Affairs (D.I.A.N.D.) began to formulate closure plans for the mine site. Both D.I.A.N.D. and many other local levels of government preferred keeping Giant Mine opened as a gold mine for as long as possible in order to maintain the underground workings and to provide employment and tax revenue.

Operations wound down in October 1999 with production ceasing October 8th 1999. The next month involved cleanup of the milling circuit to recover any remaining gold values, including blasting the interior of the coarse ore bin to retrieve ‘caked-on’ ore. The last gold bricks were poured November 17th (last brick was #11847, also the last carbon plant bar #313 was poured in October). All crews were laid off and the plant was temporarily mothballed; meanwhile, bids were being placed for a reopening of the mine by various interested parties at the discretion of D.I.A.N.D (mine records).

Miramar Giant Mines Limited [Miramar Mining Corporation Limited] (2000-2004)

Miramar, owners of Yellowknife’s Con Mine, was the logical choice for reopening the Giant Mine. Giant could become a high-grade satellite mine to Con, providing a new source of gold for the company with no capital expense. Ore would be trucked to the Con mine mill on the other side of town. Con Mine, which had drastically cut back production, would use Giant ore to bring its mill back up to capacity. The dual mines were seen as a good thing that would bring life back into Yellowknife’s dying gold industry. The price of gold was also projected to rise.

Year:	Total Ore Milled:	Grade:	Gold:	Recovery:	Notes:
1948/1949	84,886 tons	0.82 opt	42,562 oz	93.3%	<i>Fiscal year-end May 31st</i>
1949/1950	126,214 tons	0.71 opt	93,883 oz	86.4%	“
1950/1951	151,814 tons	0.84 opt	111,497 oz	87.2%	“
1951/1952	165,846 tons	0.76 opt	107,831 oz	86.1%	“
1952/1953	245,559 tons	0.85 opt	176,539 oz	85.0%	<i>Period June 1st '52 to June 30th '53</i>
1953/1954	275,985 tons	0.79 opt	177,421 oz	81.9%	<i>Fiscal year-end June 30th</i>
1954/1955	287,796 tons	0.76 opt	173,436 oz	79.2%	“
1955/1956	297,582 tons	0.77 opt	180,267 oz	79.2%	“
1956/1957	309,673 tons	0.80 opt	190,418 oz	77.4%	“
1957/1958	289,220 tons	0.79 opt	158,451 oz	68.9%	“
1958/1959	321,002 tons	0.78 opt	190,644 oz	75.7%	“
1959/1960	363,496 tons	0.78 opt	230,387 oz	80.9%	“
1960	181,101 tons	0.79 opt	117,992 oz	81.9%	<i>Period July 1st '60 to Dec. 31st '60</i>

Table 4. *Giant Mine gold production 1948-1960. See Tables 5 & 6 for subsequent production. opt = ounces per ton*

The deal between Miramar and D.I.A.N.D. was signed in December 1999 and Giant Mine was sold to the company for \$10. It stipulated, in part, that Miramar hire back between 40 and 50 ex-Royal Oak employees of the former Union, and maintain the site so as to keep the underground from flooding. Miramar would not be liable for any prior environmental issues such as the underground arsenic stopes. John Stard, previously Giant's manager under Royal Oak since 1995, was promoted to General Manager for both Con and Giant operations. Ted Bienias was mine superintendent at Giant. Murray Randall was surface and maintenance superintendent. Milling superintendent at Con Mine was Kent Morton until 2002 when Doug Keating replaced him. These management positions remained to closure of Giant Mine in July 2004. In January 2000 mining operations in the Supercrest zone were resumed, and to the end of February, some 3,000 tons had been hoisted to surface and crushed. 7,000 tons grading 0.33 ounces per ton gold remained broken in stopes underground. Shipments of Giant ores to Con Mine began in March 2000 (Miramar Mining Corporation Ltd. Annual Report, 2000).

Crushing and Milling

Giant ore was hoisted up the C-shaft and crushed in the normal circuit. Instead of being conveyed to the old Giant mill, ore was conveyed half way up the belt and dumped down a chute into the stockpile. It was then loaded up into trucks (Weatherby Trucking Limited had the contract for hauling) and brought to Con Mine, where it was weighed and treated in Con's flotation and autoclave circuit. Gold recovery was calculated based on head grades. It was expected that Giant would contribute 350 tons per day of refractory ore to the Con Mine during the first year of mine-life, but Giant did not perform at this rate due to mining problems. This rate of production was required to keep the autoclave in operation while Con Mine developed its refractory reserves at deeper levels. As a result of these operating difficulties, Miramar implemented a new mine plan late in the year. It focused on mining ore zones that required a minimum of development, resulting in a 20% reduction in the workforce. Emphasis was placed on pillar extraction of the older stopes (Miramar Mining Corporation Ltd. Annual Reports, 2000-2001).

Tailings Retreatment

During 2000, there was interest in re-processing the old Giant tailing ponds to recover additional gold and bring Con Mine's autoclave plant up to capacity. A total resource of 11.6 million tons of tailings grading 0.055 ounces per ton gold had been calculated. Previous attempts at Giant Mine had failed due to the refractory nature of the ores. Miramar later abandoned these plans, possibly because of environmental concerns.

Year:	Ore Milled:	Grade:	Gold:	Recovery:
<i>Production from 1948-1960 is listed in Table 4</i>				
1961	366,515 tons	0.78 opt	238,966 oz	82.2%
1962	375,820 tons	0.76 opt	253,189 oz	86.5%
1963	388,190 tons	0.71 opt	241,689 oz	87.3%
1964	400,606 tons	0.75 opt	266,752 oz	89.4%
1965	395,001 tons	0.72 opt	255,024 oz	89.5%
1966	386,800 tons	0.65 opt	226,696 oz	89.9%
1967	331,922 tons	0.67 opt	196,979 oz	88.6%
1968	374,717 tons	0.63 opt	210,358 oz	88.5%
1969	399,647 tons	0.64 opt	230,304 oz	88.9%
1970	424,774 tons	0.61 opt	228,732 oz	87.8%
1971	403,819 tons	0.62 opt	217,702 oz	86.9%
1972	401,272 tons	0.56 opt	201,186 oz	89.3%
1973	389,460 tons	0.46 opt	158,293 oz	88.5%
1974	328,099 tons	0.36 opt	101,514 oz	87.2%
1975	391,969 tons	0.29 opt	98,437 oz	87.8%
1976	428,154 tons	0.28 opt	106,714 oz	88.2%
1977	445,692 tons	0.27 opt	106,714 oz	87.7%
1978	396,657 tons	0.27 opt	95,413 oz	88.8%
1979	416,256 tons	0.21 opt	75,110 oz	87.8%

Year:	Ore Milled:	Grade:	Gold:	Recovery:
1980	227,240 tons	0.19 opt	38,214 oz	85.7%
1981	394,887 tons	0.18 opt	58,678 oz	83.4%
1982	403,500 tons	0.21 opt	72,596 oz	85.0%
1983	327,374 tons	0.23 opt	62,471 oz	85.0%
1984	325,549 tons	0.29 opt	64,066 oz	86.0%
1985	333,283 tons	0.23 opt	65,334 oz	84.8%
1986	316,654 tons	0.23 opt	62,253 oz	87.3%
1987	373,380 tons	0.24 opt	76,517 oz	86.6%
1988	355,484 tons	0.21 opt	62,822 oz	83.8%
1989	393,965 tons	0.24 opt	83,075 oz	87.2%
1990	382,591 tons	0.28 opt	96,766 oz	87.0%
1991	429,973 tons	0.27 opt	102,335 oz	86.5%
1992	395,018 tons	0.29 opt	95,878 oz	84.1%
1993	413,098 tons	0.26 opt	92,948 oz	85.9%
1994	430,238 tons	0.26 opt	101,176 oz	87.0%
1995	410,966 tons	0.25 opt	91,423 oz	86.7%
1996	367,421 tons	0.26 opt	83,385 oz	86.5%
1997	389,443 tons	0.27 opt	91,805 oz	87.4%
1998	346,918 tons	0.29 opt	88,038 oz	86.3%
1999	239,527 tons	0.29 opt	60,504 oz	86.1%

Table 5. Total gold production at Giant Mine 1961-1999 including Supercrest and Lolor, but not including Gold Lake Mine ores or custom ores. Gold recovery does not include Tailings Retreatment Plant production between 1988 and 1990 (see Table 3) (sources: Giant Yellowknife Mines Ltd. Annual Reports; Royal Oak Mines Inc. Annual Reports) *opt = ounces per ton*

Year:	Ore Milled:	Grade:	Gold Produced:	Recovery:
2000	78,957 tons	0.33 opt	21,510 oz	82.9%
2001	73,247 tons	0.39 opt	25,361 oz	88.2%
2002	71,536 tons	0.38 opt	23,899 oz	88.1%
2003	73,508 tons	0.35 opt	22,103 oz	85.3%
2004	50,277 tons	0.28 opt	12,088 oz	85.1%

Table 6. Gold production at Giant Mine between 2000-2004. (source: Miramar Mining Corp. Ltd. Annual Reports) *opt = ounces per ton*

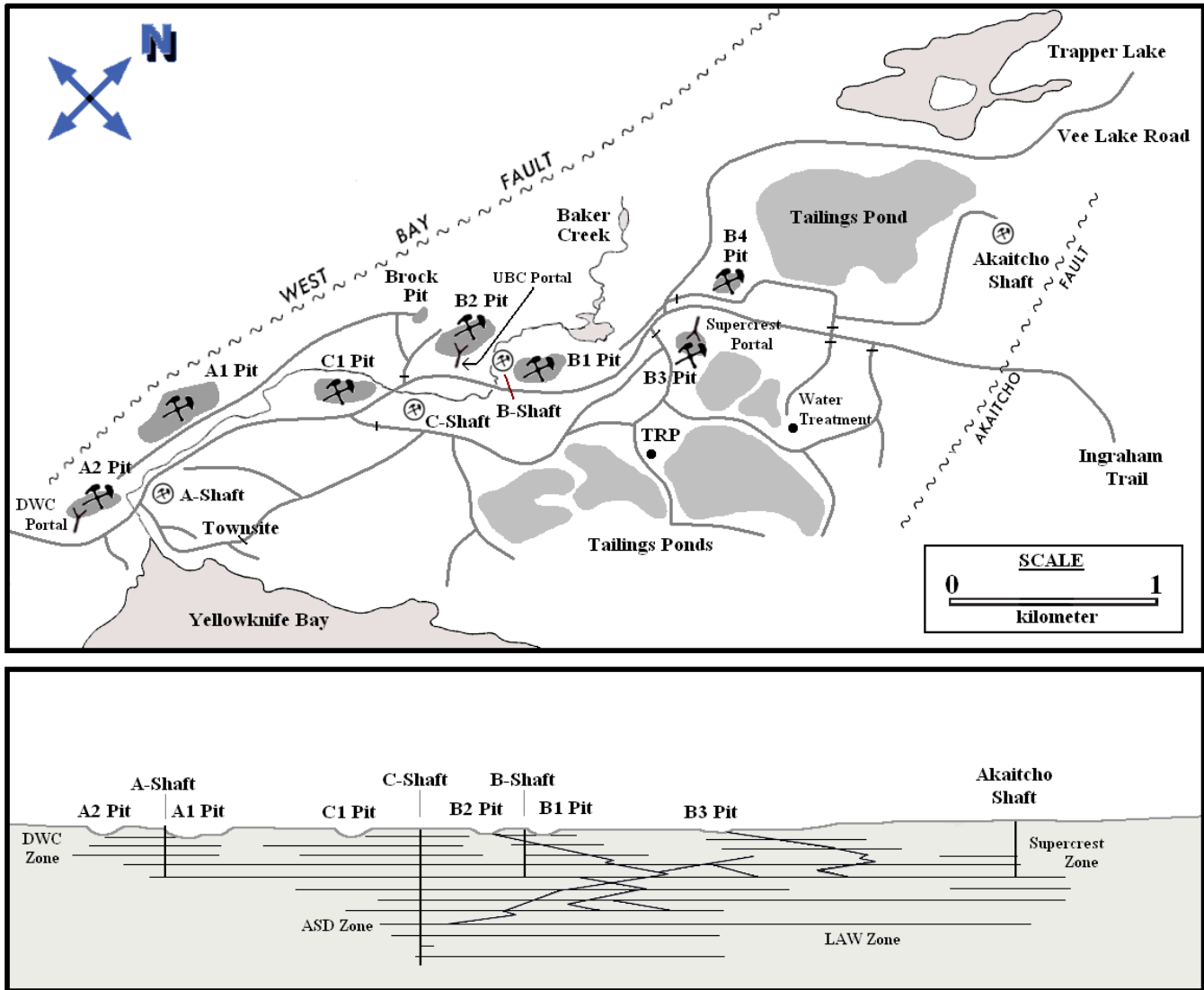


Figure 12. Surface plan of mine working locations, and longitudinal section along the orebody.

Extended Life

In June 2001, it was announced that Miramar planned to cede their mining rights to Giant Mine and return the property to D.I.A.N.D. The mine was viewed as too costly to keep operating. Also, Miramar was operating Giant as a temporary source of refractory ore, so when new refractory reserves were discovered at Con Mine the decision to cease operations at Giant was made. This decision was quickly turned around when in August 2001 it was announced that Miramar geologists had uncovered additional mineralization within the Supercrest zone and that an agreement had been signed with D.I.A.N.D. to extend the life of the mine for another year. In addition, D.I.A.N.D. agreed to pay Miramar a monthly fee, and would bear all environmental costs at the mine. This helped to improve operating costs. Also a factor in the extended life was a new collective agreement with the worker's Union at Con Mine that was signed in July 2002, good for a three-year period. A strike at Con Mine would have shut down both operations, probably indefinitely. (Miramar Mining Corporation Ltd. Annual Reports, 2001-2002) Some production shortfalls were seen at Giant in 2003 as a result of shifting resources to the development of additional ore resources within the Supercrest zone. An increase in manpower was seen at the end of 2003 when Con Mine closed and focus could be put on mining at Giant. Employment was increased to approximately 75 people. Production was increased from 7,000 tons per month to 10,000 tons per month (Miramar Mining Corporation Ltd. Annual Report, 2003).

Final Year of Operations

Operations at Giant were anticipated to extend into 2005. New resources were outlined in the C-shaft area on the 425-foot (3rd) level of the mine in 2004, but poor ground quality and lack of ore continuity prevented mining in this area.

The mine saw lower gold production and higher operational costs during the first few months of 2004. Mining activity was terminated at Giant Mine on July 10th 2004 as a result of the poor performance of the anticipated orebodies at 3rd level C-shaft. 49 employees were laid off in July, followed by 20 employees in August. Up to 10 workers were left on to provide caretaker and maintenance duties. Milling of stockpiled ores continued throughout the remainder of the summer (Miramar Mining Corporation Ltd. Annual Report, 2004).

Miramar Giant Mines Limited relinquished the property on June 30th 2005 to D.I.A.N.D. Permitting to allow for the complete decommissioning and remediation of the Giant Mine property then commenced, however in 2007 the project was referred to an Environmental Assessment, a process that could take up to three years to complete. Until approval is granted, remediation of the site is on hold.

Exploration Since Mine Closure

No exploration work has been conducted since closure in 2004. It is understood that economic ore has been depleted.

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	<h2 style="text-align: center;">GILMOUR LAKE</h2> <p style="text-align: center;"><i>Minor Producer (Abandoned)</i></p>	Tungsten
		62° 29' 15" 112° 56' 30"

Years of Primary Development: 1940-1941

Mine Development: 1 primary trench

Years of Production: 1952

Mine Production: 10 tons milled = 1,150 lbs tungsten concentrates

Introduction

This small operation was located on the eastern shore of Gilmour Lake, 73 kilometers east of Yellowknife, N.W.T. It milled a few tons of ore in a test mill in 1952. The author has not visited the site, and although some aerial reconnaissance of the Gilmour Lake area was performed in July 2003, no ruins were spotted.

History in Brief

Alex Mitchell staked the 'Dot' and 'Eva' group of claims in June 1940 and formed the Alex Mitchell Mining Syndicate. During the year deposits of tungsten were discovered, and prompted an intensive search and investigation of the deposits by the Geological Survey of Canada. Mitchell turned down a deal in which he could have sold his property and retained shares, feeling the deal was insincere. No work was being done after September 1941, by which time 30 tons of ore were stockpiled. In 1950, Boreas Yellowknife Gold Mines Limited acquired control of the claims at Gilmour and Gordon Lakes. A test mill was installed at Gilmour Lake in 1952 and operated during the summer to recover a tungsten concentrate.

Geology and Ore Deposits

The veins of the area occur in folded sedimentary rocks of the Yellowknife Group and are mostly parallel with the beds. Veins are high in gold and scheelite content (Lord, 1951).

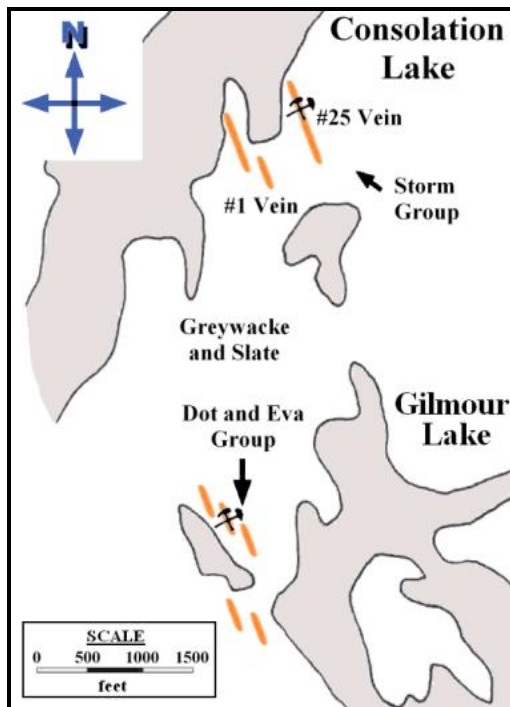


Figure 1. *Gilmour Lake area.*

Boreas Yellowknife Gold Mines Limited (1952)

During 1951, Canadian interest in tungsten production was revitalized for a short time. Boreas Yellowknife Gold Mines Limited acquired the 'Dot' and 'Eva' claims from Alex Mitchell early in 1952, with an interest in installing Mitchell's patented milling machinery to process the 30 tons of stockpiled ore. The grade of this ore was reported as 4-3% tungsten oxides (The News of the North, Apr. 25th 1952).

In 1952, the Alex Mitchell rocker-mill was flown to Gilmour Lake and installed at the claims. During the testing period in May 1952, the mill processed 2,500 pounds of ore to recover 183 pounds of concentrate containing 7.2 units of tungsten oxide concentrate (1 unit = 20 pounds, therefore about 144 pounds of concentrate). Mechanical problems forced an early cessation of work, but repairs were made and production continued during the remainder of the summer (The News of the North, May 30th 1952).

By August 1952, when the operation finally closed, 10 tons of hand-cobbed ore from a surface pit/trench (over 50 feet long) on the 'Dot #5' claim was put through this mill to recover 55 units (1,150 pounds) of tungsten concentrate. Tungsten oxide (WO₃) content of this concentrate is unknown (The News of the North, Aug. 15th 1952).

The Mitchell Mill

This invention was designed and patented by Alex Mitchell in 1950. Its primary function was to provide small high-grade mining operations with a cost-effective milling plant. The uniqueness to the design came from the method in which ore was crushed, by using a rocking manganese plate to grind ore down to size. The rocking motion of the

plant would also allow the ore to pass through the grinding trough, aided by a flow of water. The slurry was then passed down two 8-foot sluice boxes where a concentrate was recovered. The entire mill was powered by a small five horsepower gas engine and had a capacity to mill 4 to 5 tons per day (Canadian Patents Office).

The first trial run of the rocker mill was a great success. After a few mechanical adjustments, the mill proved to recover the metals very efficiently. It was decided to take the plant to Gordon Lake for use at the Treacy Mine in 1953 (*The News of the North*, Aug. 15th 1952).

Exploration Since Mine Closure

Unknown.

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Canadian Patents Office: claims #466271 and #520978.



GLACIER BAY

Minor Exploration (Abandoned)

Silver

66° 05' 00" 117° 59' 30"

Years of Primary Development: 1932-1933

Mine Development: 115' vertical shaft, 1 level (100' dev.)

Introduction

The site is located in the Echo Bay area of Great Bear Lake, three kilometers east of LaBine Point (Port Radium) on the north side of Glacier Bay, Great Bear Lake. It is 440 kilometers northwest of Yellowknife, NWT. The claims were known as the 'Rad' in the 1930s. The site has not been visited by the author and its exact location is unknown.

Brief History

The 'Rad' claims were staked by Charles E. Sloan in 1931 for the Great Bear Syndicate, headed by J.J. Byrne. The property adjoins the Eldorado and Echo Bay Mines to the south. Pitchblende stains and some silver veins were located almost immediately after staking, and J.J. Byrne decided to develop this and other properties in the Great Bear Lake area through his new Great Bear Lake Mines Limited venture. A shaft was sunk to 100 feet in 1933 but no other work is reported.

Geology and Ore Deposits¹

The area is situated within the northern half of the Great Bear Magmatic zone, near the western margin of the Bear Structural Province. The supracrustal component of this Aphebian complex is the McTavish Supergroup, mainly composed of volcanic and minor sedimentary rocks, which has been subdivided into three groups separated by unconformities. All the silver deposits in the area are hosted within the LaBine Group, an early Proterozoic volcanic arc developed on continental crust around 1,800 Ma ago. Most of the area's mineralization occurs within northeast-trending, north dipping fault system, principally where it cuts the tuffs of the Cliff Formation. Mineralization is more commonly noted in the sedimentary rocks than the volcanics.

The #1 vein and the #2 veins, both highly mineralized with quartz, calcite, bornite, chalcopryrite, manganese-oxide, and native silver, were the focus of early exploration. The vein was traced for widths of 15 to 20 inches underground. The #2 vein is located 380 feet southwest of the shaft on the #1 vein, and runs perpendicular to the #1 vein. The #2 vein has well-defined walls with mineralization about 2 feet in width (Dolan, 1932).

Great Bear Lake Mines Limited (1932-1933)

Although the company held numerous claims in the Great Bear Lake area, the 'Rad' claims were the most interesting. Early surface exploration had showed indications of native silver and uranium stain in geology that was very similar to the adjoining high-grade Eldorado Mine property. A crew of 15 men was mobilized in March 1932 to construct camps and conduct some claim surveying and surface explorations at the 'Rad' claims. This crew included J.J. Byrne as company president, Charles E. Sloan as field manager, Ole Hagen as foreman, J.P. Dolan as assistant, Henry Lund, Carl Wicklund, and Thor Nielson. Other work included field reconnaissance of the Great Bear Lake area to locate additional deposits, and some minor development at the 'Bear' claims (see Bear Portal Mine).

Company operations focused solely on the 'Rad' property after 1932. The #1 vein was the focus of most interest, but the #2 was also identified as a good prospect. The following work was reported to December 6th 1932, when work ceased for the winter. Six test pits were sunk on the #1 and #2 veins totaling 76 feet, one of which, on the #1 vein, was continued for a depth of 35 feet as a prospect shaft. Dimensions of the shaft were 5 feet x 7 feet with a cribbed collar to a 10 feet depth. Hoisting was performed by means of a windlass, and work on the shaft was stopped to await the arrival of more advanced hoisting machinery. The first native silver was encountered in #3 pit (#1 vein) at a depth of 12 feet. The break of the vein at this point was 5 feet wide with a seam in the center showing native silver in widths of 15 to 18 inches. Native silver was also encountered in the shaft at a depth of 10 feet, originally sunk as #1 pit. Sinking continued as native silver was found in each round. Good log cabin camps on Glacier Bay were also erected during the year (Dolan, 1932).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office and Dolan (1932)

A custom-made hoisting power plant was brought to the site in September 1933, comprising a 10 horsepower Tangee diesel engine directly connected to a hoist (Byrne, 1971). In October 1933 it was reported that timber had been cut for a headframe, powerhouse, and other buildings. Henry Lund was in charge of the power plant. J.P. Dolan was reported as field manager (The Toronto Star, Oct. 16th 1933). During the fall of 1933, the shaft was sunk to 115 feet and a level was opened up at 100 feet depth. Silver mineralization within the vein persisted at this level, so about 100 feet of drifting was carried out to follow the #1 vein east and west. Over 350 feet of test pitting had also been carried out by this time, over a strike length of 700 to 800 feet (The Toronto Star, Nov. 15th 1933; The Northern Miner, May 24th 1934). The deposit did not improve and as a result all work was suspended in December 1933 (Byrne, 1971).

Exploration Since Mine Closure

In 1954, Athona Mines (1937) Limited, the reorganization of Great Bear Lake Mines Limited, sent prospectors back to the Glacier Bay property to carry out investigations of radioactive anomalies reported by the Geological Survey of Canada. A complete Geiger and scintillometer survey of the claims was planned (The Northern Miner, Aug. 12th 1954). The results of this work are not known, and the claims later lapsed. In 1968, Echo Bay Mines Limited conducted a mapping program and radiometric survey on the 'Echo Bay' #11 and #12 claims, located over a portion of the lapsed 'Rad' group. The radiometric survey identified an anomalous area at the southern part of the property. A small hand trench was dug and sampling over one foot returned 4 ounces per ton silver, trace copper and 0.02% uranium oxides (U₃O₈) (Robinson, 1968).

Alberta Star Development Corporation staked claims in the Glacier Bay area in 2005-2006 and exploration of the region is ongoing.

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GOLD LAKE

Satellite Producer (Remediated)

Gold

62° 32' 30" 114° 22' 00"

Years of Primary Development: 1985-1988

Mine Development: Decline to 275', 3 levels (5,994' dev.) 348' raise

Years of Production: 1986-1988

Mine Production: 33,650 tons milled = 8,632 oz Au

Introduction

Gold Lake was the site of a three-year underground production project by the operators at Giant Mine, and is located just south of Vee Lake along the access road, a few kilometers north of Yellowknife, NWT.

History in Brief

The developed area once comprised two claim groups owned by Lynx Yellowknife Gold Mines Limited and Akaitcho Yellowknife Gold Mines Limited. The 'AES' (Akaitcho) claims were acquired by Giant Yellowknife Mines Limited in 1965, and the 'Gold' and 'Fox' (Lynx) claims in 1977. The acquisition of the Lynx property was based on a 40% production royalty being given to Jimmy Mason. Various programs of exploration were done in this area throughout the 1960s and 1970s. In 1982, the properties were reviewed for their potential, and it was decided to shelve exploration until the price of gold increased. A feasibility study was completed in 1984, and the property was re-named the G.K.P zone after a former Giant Mine geologist G. Ken Polk. In 1985, a decline was started at Gold Lake to access the ore-body, and production began in 1986 with ore trucked to the Giant mill. The 3rd level of the mine was reached in August 1987 at 300 feet depth below the portal entrance and mining of the project was complete near the end of 1988.

Geology and Ore Deposits

The mineralized zone on the south side of Gold Lake strikes northwest and dips 20° southeast. The ore had a tendency to pinch irregularly, changing in width from a few inches to over 15 feet (Mossop, 1988).

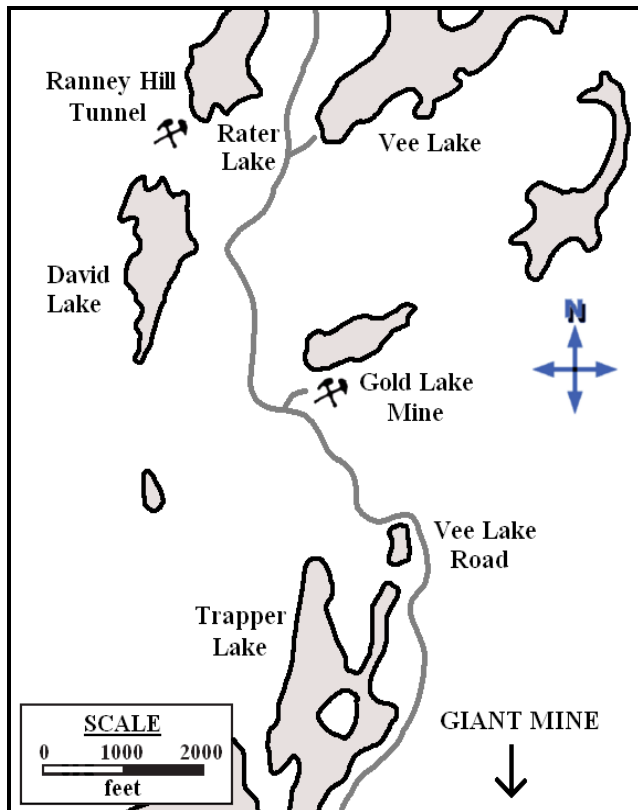


Figure 1. Gold Lake Mine location.

Giant YK Mines Limited (1985-1988)

Site mobilization using spare equipment, crew, and supplies available at Giant Mine began in the spring of 1985. An ore reserve of 31,487 tons with a grade of 0.47 ounce per ton gold was available and considered economic at 1985 gold prices. Previous geological work indicated a structure of ore beneath a large outcropping on the south side of Gold Lake, straddling the claim boundaries of the Lynx property and the Supercrest property. It was planned to drive over 2,600 feet of -15% decline to access the orebody at two elevations, 210 and 310 feet below the portal entrance. The decline was started in June 1985 (Mossop, 1988).

Mining Equipment

The ramp was driven using a MJM 20-B Jumbo driller, and mucking of ore was done with a JS 220 scooptram and a JDT 413 ore-truck. When production began in July 1986, stope mining was accomplished using three-drum electric slusher dumping into the trucks via chutes. On the surface, a Cat 988 loader would load 10-ton haul trucks to bring the ore to the Giant mill a distance of about four kilometers over the Vee Lake road. The operation was powered by two diesel generators to give a supply of 180 kilowatts of electricity.

One portable 850 cubic feet per minute air compressor (diesel driven) was also in use. Diesel fuel for all equipment was stored in a 2,000 gallon fuel area lined with impermeable material. Water was pumped from Gold Lake until sufficient water pockets were encountered underground. Water discharge was via the Trapper Lake system (Mossop, 1988).

Mine Development

Giant's original development plan was changed early in 1986 when a significant water problem was encountered in decline driving. Instead of bringing the ramp around towards the east (hanging-wall), the decline swung west into the footwall and the 1st level was established at a depth of 155 feet from the portal entrance. Production began from the 1st level in July 1986 and driving of the decline to the 2nd level was started. The 210-foot (2nd) level was reached in September 1986, and the 275-foot (3rd) level was reached in August 1987. Production began six-months behind schedule, due principally to the water problems underground. The mine produced 130 gallons of water per minute, and flooding became a major factor in low production rates. The average mining rate of the project was 1,227 tons per month, well below expectations (Mossop, 1988).

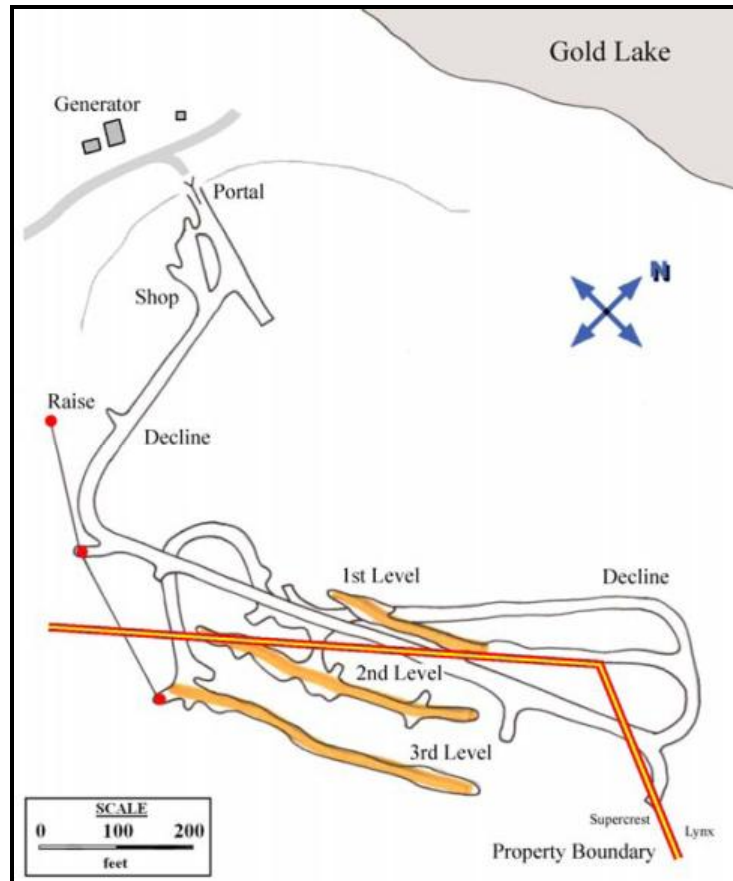


Figure 2. *Gold Lake Mine underground plan.*

The 3rd level was reached in August 1987, two months after the planned shutdown date of the project. An economic analysis was performed at this time by Malcolm Robb, geologist, to determine the viability of further drilling of the deposit. Drilling in later 1987-early 1988 increased the ore reserve to 45,100 tons grading 0.38 ounces per ton gold below and to the north of the 3rd level. The 3rd level drift was extended northward when major water problems and bad ground conditions halted further development. It was decided to mine out the last remaining stopes in the mine, and shut down the project. The last ore was hauled from Gold Lake in October 1988. (Mossop, 1988) The discontinuous nature of mineralization below the 3rd level was also a reason for the cessation of work at Gold Lake. (Akaitcho Yellowknife Gold Mines Ltd. Annual Report, 1989)

Development and Production Summary

Total development performed at Gold Lake consists of decline workings and three levels of development amounting to 5,994 feet. This figure was 600 feet more than the original budget plan due to unscheduled ore development on the 155-foot (1st) level. The development rate was 168 feet per month. Also developed was a 348 foot raise from the bottom of the mine to the surface for ventilation purposes. Operating costs for the Gold Lake project between 1985 and 1988 amounted to \$4 million, plus \$300,000 for capital costs. Mine production is summarized in **Table 1**. The Supercrest property owned 60% of the production from the Gold Lake project both in tonnage milled and gold produced, while the remaining tonnage was owned by Lynx Yellowknife Gold Mines Limited. The Gold Lake Mine milled 33,650 tons of ore to recover 8,632 ounces of gold (Mossop, 1988).

Year:	Ore Milled:	Grade:	Gold:
<i>1986</i>	8,837 tons	0.276 oz/ton	2,435 oz
<i>1987</i>	16,512 tons	0.221 oz/ton	3,656 oz
<i>1988</i>	8,301 tons	0.306 oz/ton	2,541 oz
<i>Total:</i>	<i>33,650 tons</i>	<i>0.268 oz/ton</i>	<i>8,632 oz</i>

Table 1. *Gold Lake Mine production, 1986-1988. (source: Mossop, 1988)*

Exploration Since Mine Closure

No work has been reported. Ore reserves at the GKP zone at December 31st 1999, categorized as unmineable remnants or pillars, consists of 42,045 tons grading 0.29 ounces per ton with a content of 12,324 ounces of gold. (Miramar Giant Mines Ltd., 1999)

References and Recommended Reading

Akaitcho Yellowknife Gold Mines Ltd. Annual Report. 1989.

Giant Yellowknife Mines Ltd. Annual Reports. 1985-1988.

Miramar Giant Mines Ltd., 1999. Ore Reserve and Mineral Inventory Report. December 31st 1999.

Mossop, B., 1988. Postmortem of the Gold Lake Project. Giant Yellowknife Mines Ltd., November 1988.



HIDDEN LAKE

Minor Producer (Abandoned)

Gold

62° 33' 15" 113° 31' 00"

Years of Primary Development: 1940s, 1959-1969

Mine Development: 69' incl. shaft, 1 level (+160' dev.)

Years of Production: 1968

Production: unknown

Introduction

The Hidden Lake Mine, located 45 kilometers northeast of Yellowknife, NWT, was a small high-grade gold mining operation that achieved its most advanced development in the late 1960s. The site was destroyed in the 1998 forest fires, the ruins of which were visited by the author in June 2000 and August 2009. It is also sometimes called the Ragged Ass Mine.

History in Brief

The first claims were staked in the mid 1930s by A. McClure and then staked again as the 'McQueen' group in December 1939 by Ed McQueen, Claude Watt, and Joe Herriman. A mining syndicate was formed by these men and others (Ragged Ass Syndicate), and in the summer of 1940 rich gold was found on the property. In the early 1940s, crews sank an inclined shaft/pit, and shipped high-grade ore to Yellowknife for milling.



Gordon Piro

Figure 1. *Headframe and hoist at Hidden Lake Mine, 1960s.*

The ground was re-staked at least three times between 1940 and 1959. The property was covered by the 'McQueen' and 'HM Shorty' claims in 1945 and was owned by Argonaut Yellowknife Gold Mines Limited. In 1953, the 'Little Giant' claims occupied the mine area, possibly under the ownership of Chuck McAvoy. In 1959, the property was again re-staked as the 'HM' claims by Joe Herriman, and optioned to H. Wist and Associates Limited. Sporadic work was done between 1959 and 1969, including the development of one underground level in a new shaft and the installation of a small milling plant.

Geology and Ore Deposits¹

The showing lies within the Archean age, Burwash Formation of the Yellowknife Supergroup. The area is predominantly underlain by knotted quartz-mica schist (metamorphosed metasedimentary greywacke-argillite turbidite). These rocks are folded by at least two phases of Archean deformation, and metamorphosed to lower amphibolite facies in a 40 kilometer wide, north trending belt. The main gold-bearing quartz vein outcrops 800 feet east of the Hidden Lake shoreline. It is at least 110 feet long. At depth the vein is a series of conformable lenses within thin-bedded greywackes and argillites, striking north and dipping 25° east. A large fold disrupts the vein near the shaft workings, where significant gold mineralization occurs. Spectacular free gold is said to occur near the vein contact with the metasediments.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Ragged Ass Syndicate (1940-1941)

The Northern Miner newspaper reported a gold find by a syndicate composed of Ed McQueen, Claude Watt, Joe Herriman, A.E. Williams, and A. Rudy in 1940 (*The Northern Miner*, Oct. 10th 1940). The claims, known as the ‘McQueen’ group, were staked by this group in December 1939, and the syndicate went by the title Ragged Ass Syndicate (*The Yellowknife Blade*, Dec. 31st 1940; Jan. 11th 1941). Starting in 1940, development consisted of a shaft, inclined at 40° to the east. From this shaft, the Syndicate recovered large amounts of high-grade ore. Some ore was flown to Yellowknife for processing in a milling plant operated by Shorty Holloway (Price, 1967). It is said that a high-grade pocket was encountered at some point in underground exploration, but an excessive water problem resulted in the abandonment of the workings (Barrager, 1962). The Syndicate also excavated two small pits, north and south of the shaft. (Schiller, 1965) There is no record of production, but at least two shipments were made: one in the fall of 1940 before freeze-up (*The Yellowknife Blade*, Nov. 6th 1940), and another in February 1941 consisting of 3,000 pounds of high-grade ore.

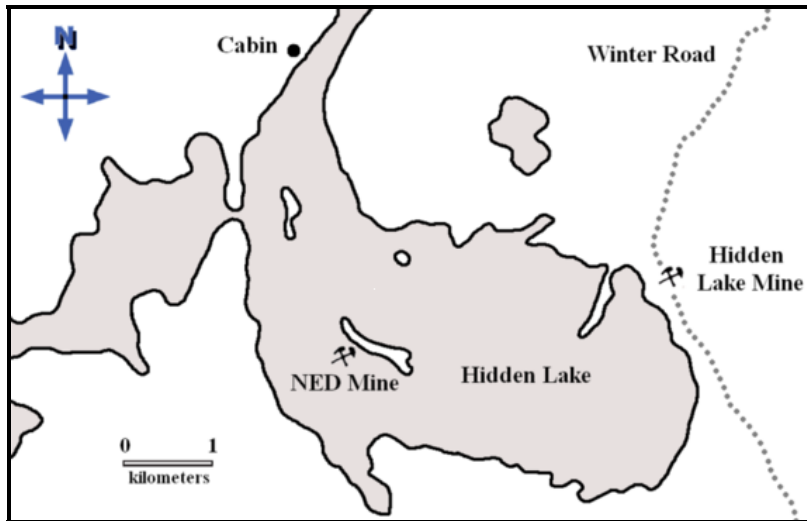


Figure 2. *Hidden Lake Mine location.*

H. Wist and Associates Limited (1959-1963)

Joe Herriman re-staked the property as the ‘HM’ group in 1959 and optioned it to H. Wist & Associates Limited. A new period of development began with the sinking of an 8 foot x 8 foot shaft at an incline of 70° to the north. The objective of this shaft was to re-penetrates the shallow vein encountered by previous exploration and development. Mr. Herriman and C. McChesney worked on the shaft during 1960-1961 when the shaft reached a length of 50 feet, with the last 10 feet within quartz. In the following year, the shaft had reached its targeted 69 feet length, penetrating the quartz with the bottom of the shaft six feet below the vein. In October 1962, drifting south began to intersect a high-grade portion of the vein explored by the original tunnel in the 1940s. To the date of the report, 50 feet of drifting had been accomplished. (Barrager, 1962; Barrager & Hornbrook, 1963) During 1963, it was reported that the drift had advanced 65 feet south through the footwall of the vein to a point on the hanging-wall side. From this point, two sub-drifts were driven roughly parallel to the hanging-wall, 35 feet and 45 feet long respectively. The south drift was expected to intersect high-grade material previously intersected in the 1940s tunnel. The owners planned to continue drifting over the winter of 1963-1964 (Schiller & Hornbrook, 1964). Between 1964 and 1967, some milling equipment was brought to the property and in 1964 it was reported that production was expected to commence in 1965 (Schiller, 1965).

Frank Avery (1968)

Frank Avery became involved in the operation in 1965 when he formed First Northern Exploration Limited. When this company failed to work the ground, Frank Avery obtained a three-year lease on the claims, and then subsequently bought the property in December 1967. Harold Glick earned a 20% interest in the property through financial support (*The Northern Miner*, July 29th 1965; Knutsen, 1968). A small crew including Mr. Avery and his two sons de-iced the shaft early in 1968. De-icing was done using picks and hand chisels, but also included setting fires to melt the ice. The job apparently took three months (Gerry Avery, pers. comm.).

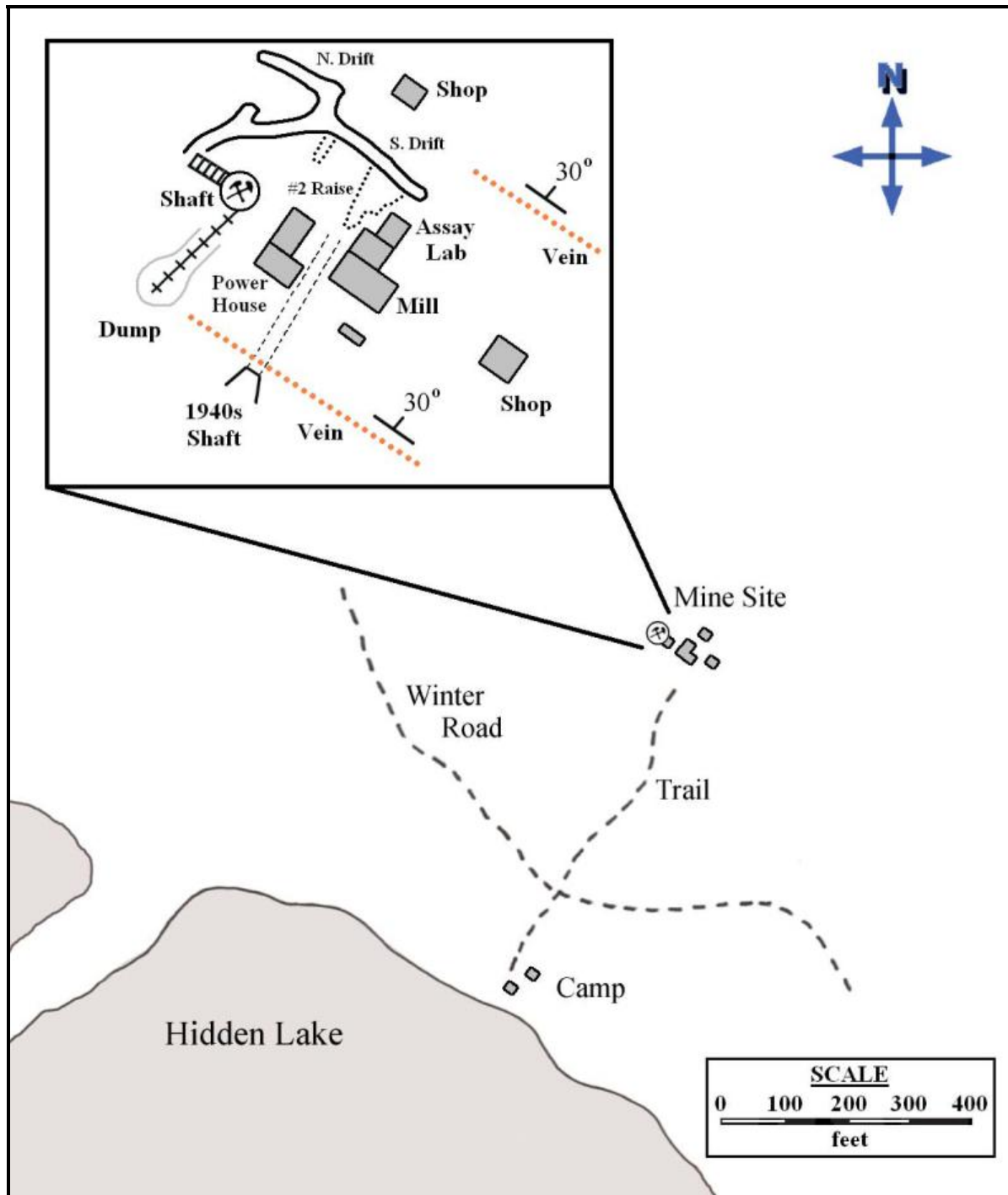


Figure 3. Surface and underground plan of Hidden Lake Mine, c.1969.

Underground development continued at the 69-foot level. On the south drift, two raises were excavated up the dip of the zone to the west, distance of 15 feet and 45 feet respectively, with indicated zone widths of three feet (Knutsen, 1968; Padgham et al., 1978). Frank Avery and his crew removed several hundred tons of ore from the south raise and began the operation of an amalgamation mill. It is believed that this small mill only operated three times during the summer of 1968 (Knutsen, 1968).

Mining Operations

Ore was hoisted up the shaft using a Sullivan tigger air hoist, and underground drills were powered by a Sullivan air compressor driven by an International diesel engine (Gerry Avery, pers. comm.; site evidence).

Milling Plant

Ore was crushed in a jaw crusher powered by gas engine. An amalgamation barrel was utilized as a ball mill to process the ore, and a concentrate was drawn off a Wilfley table. Gold was recovered using mercury methods in a gold furnace. This mill was highly improvised, and did not prove to recover gold adequately. It was powered by a Cat diesel generator of 10 KVA power. Equipment for larger and more efficient milling capabilities was available at the property, and included a ball mill, classifier, jig, and larger jaw crusher. Little gold was produced from the mill, and Mr. Avery was unable to raise the money needed to install the better equipment (Gerry Avery, pers. comm.). The amount of gold recovered is unknown, but it is told that rough gold was sold on the local Yellowknife markets and that jewelry made of gold from the mine could be bought from Yellowknife shops. Harold and Jack Glick, owners of the Gold Range Hotel and YK Radio, were selling such products during 1968 and 1969 (Knutsen, 1968; News of the North, Aug. 8th 1968).

Hidden Lake Gold Mines Limited (1968-1969)

Encouraged by the results of the 1968 mining season, Frank Avery formed a company, Hidden Lake Gold Mines Limited, to develop the property further late in 1968 (The News of the North, Dec. 5th 1968). In July 1969, a crew of five men was on the property rehabilitating the shaft and extending underground drifts 200 feet in total. The #2 raise was driven a length of 125 feet and broke-through into the old 1940s shaft workings. William Knutsen was also hired in the summer of 1968 to sample the underground workings. His report suggested that the south ore zone in the vicinity of the #2 raise averaged 0.75 ounces per ton gold which was economic ore at that time (unknown author, 1969). Overall, most people were not enthused over the potential of the gold deposit. Being hosted in metasedimentary rock, most experts agreed that the deposit lacked size and grade to make a viable producer (Gerry Avery, pers. comm.).

Exploration Since Mine Closure

Hidden Lake Gold Mines Limited continued minor exploration on the property during the 1970s-1980s. Some diamond drilling (6 holes; 277 feet) was conducted in 1973 but only encountered scattered veinlets of quartz (National Mineral Inventory). In 1981, it was reported that consideration was being given to re-mining the deposit. 10,000 tons of ore above the 70-foot level were identified, and detailed geological mapping of the surface and underground workings were conducted. A bulk sample of stockpiled ore was shipped for testing, and it was anticipated that 100 tons of surface ore could be milled in 1982 (Hidden Lake Gold Mines Ltd. Annual Report, 1981). No further work has been reported.

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- geology from NORMIN.DB (<http://www.nwtgeoscience.ca>)
- Personal communication: Gerry Avery



INDORE

Minor Producer (Abandoned)

Uranium

64° 48' 30" 118° 23' 00"

Years of Primary Development: 1950-1953, 1955-1956

Mine Development: 190' vertical shaft + adit, 2 levels (+1,200' dev.)

Years of Production: 1952-1953

Mine Production: unknown

Introduction

The Indore Mine was the first private endeavor to produce a uranium concentrate in the Northwest Territories. The mine is located at the southern end of Hottah Lake, about 330 kilometers northwest of Yellowknife, NWT. It was visited by the author in August 2006. The headframe collapsed many years ago, and the remaining buildings are in poor shape. Most equipment and buildings were dismantled and removed following closure in 1956.

Brief History

The 'Pitch' group of claims was staked by Indore Gold Mines Limited in the summer of 1950 to protect a showing of radioactive material discovered by company prospectors. A program of surface trenching and diamond drilling was followed by underground work in the fall of 1950, through the use of an adit tunnel. Extensive work was completed by 1952 and the decision was made to proceed with production. A small mill went into operation in October 1952 and in 1953 a shaft was sunk to open a second level. Development continued in 1955 on the second level by United Uranium Corporation Limited, but all work ceased in 1956.

Geology and Ore Deposits¹

The geology of the area consists of granites and granodiorites cut by steep-dipping gabbroic or diabase dykes, many striking north. The granitic rocks, and to a lesser extent the mafic dykes, are fractured, and shearing occurs locally along dyke contacts. Quartz, hematite, and pitchblende are found in a narrow fracture zone (up to 25 centimeters wide) along the contact of one mafic dyke (#1 deposit) striking north-east and dipping about 80° east. The dyke is about 7.6 meters wide and has been traced intermittently on the surface and by drilling over a strike length of 183 meters. Pitchblende and hematite are erratically distributed in the fractures, being more concentrated in intensely fractured areas, and at fracture intersections. A few short and very narrow mineralized sections were defined during underground development.

Indore Gold Mines Limited (1950-1953)

The development program of 1950, involving the excavation of numerous trenches along the dyke contact zone, yielded grab samples that assayed over 10% uranium oxides (U₃O₈). In order to get a better look at the deposit, an adit tunnel was started in October 1950 at a depth of 50 feet below the surface exposure (Anderson, 1951a). A crew of 6 to 10 men were employed under the direction of Frank W. Anderson, consulting engineer.

The adit was driven 90 feet westerly to intersect the east contact of the dyke. Drifting then commenced to the south along this contact and the first 30 feet of drifting revealed massive pitchblende with samples assaying as high as 20% U₃O₈. A bulk sample across a full mining width assayed 2.3% U₃O₈. The adit continued westerly for 60 feet to reach the west wall contact of the dyke. Drifting to the north on the east contact was then initiated, as was a drift to the south on the west contact (The Northern Miner, Jan. 11th 1951; Apr. 19th 1951; Apr. 26th 1951).

In the spring of 1951, the Indore company purchased a small milling plant from Eldorado Mining and Refining Limited, previously used at the Eldorado Mine on Great Bear Lake during leach plant tests in 1950. Other equipment, including crusher and power plant machinery, were also acquired second-hand from mining companies in the Yellowknife area (The Northern Miner, Feb. 15th 1951).

Pitch #27-28 Deposit

The 'Pitch' #27 and #28 claims were staked in 1950 to protect a highly radioactive quartz vein structure between the contact of quartz feldspar-porphry and granite. The claims were located about one mile inland from the northeastern shore of Hottah Lake, several kilometers north of the Indore Mine. During the summer of 1951, trenching and diamond drilling was performed along the main exposure to test the zone at shallow depths (Anderson, 1951b).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Samples sent to the Department of Mines in Ottawa during 1952 reported grades of 5.29% U_3O_8 . This work suggested a possible satellite mining operation by open pit methods and added to the projected ore reserves of the Indore property (The News of the North, Mar. 22nd 1952).

Underground work stopped in August 1951 upon driving 620 feet into the zone, including a 150 foot adit crosscut. The southwest drift was reported as 165 feet long with visible pitchblende continuous on each face for a length of 115 feet. In the northeast drift, which was 150 feet long, several radioactive zones and good ore widths were encountered. At 125 feet in the northeast drift, the contact zone split and considerable slashing was required on both sides of the drift to find the orebody. Massive pitchblende mineralization was encountered in the western fork of this division. In the southeast drift, driven a length of 155 feet, visible pitchblende was encountered, as were good ore values contained within lenses and stringers up to 2 inches wide (Anderson, 1951a).

Sampling Results

In the summer of 1951, bulk sampling was completed and 1,500 pounds of material was sent for assaying to the government assay labs at nearby Eldorado Mine (The Northern Miner, July 26th 1951). An average of 0.25% U_3O_8 was obtained across 30 inches along the total exposed length of 470 feet of contact zone on both sides of the dyke. Diamond drilling also returned good values. One intersection of the dyke was made 120 feet below the adit level and cut the dyke (width of 22 feet to 25 feet) with strongly sheared contact walls on both sides. This intersection cut a section of massive pitchblende that assayed 22.55% U_3O_8 across four inches. Drilling also suggested that the dyke could continue for an additional 500 feet beneath the muskeg and overburden, giving an indicated dyke length of 1,500 feet (Anderson, 1951a).



Henry Busse - NWT Archives – N-1979-052-1897

Figure 1. *Indore Mine looking west, 1954. Camp buildings in the foreground.*

The construction of a 20 ton per day milling plant and other buildings were started in 1952. A tractor and sawmill were flown to the mine to assist in surface preparations and construction of buildings. The tractor cleared a 4,000 foot airstrip on the ice of Hottah Lake capable of handling Bristol aircraft, required for the transport of equipment for the proposed shaft sinking program to be undertaken. Over 10,000 board feet of lumber were milled on a nearby timber concession (The Northern Miner, Feb. 14th 1952; The News of the North, Mar. 22nd 1952; Anderson, 1952).

Discovery of #2 Dyke

During the summer of 1952, a geological investigation of the mine area under the direction of F.G. Smith uncovered the #2 dyke, only 150 feet west of the #1 dyke. The two dykes showed similarity in texture, structure, and radioactive counts in Geiger tests, and the new discovery was reported to double the original ore reserve estimates. This discovery motivated the company to consider upgrading the capacity of the future milling plant from 20 to 50 tons per day (The News of the North, June 15th 1952; The Northern Miner, July 17th 1952).

Production Starts

By August 1952, all of the major milling units were installed in the new mill building, but the short supply of several smaller components delayed the start-up of production by a few months. Production began on October 6th 1952 (The News of the North, Oct. 10th 1952). Initial mill feed was from a surface stockpile of ore, but in November 1952 selectively mined, fresh underground material was put through the plant (The Northern Miner, Nov. 20th 1952).

Milling Operations

The purchased milling plant was reported to have a rating of 10 tons per day using a Telsmith jaw crusher, ball mill, mineral jig, spiral classifier, and Diester tables. Capacity of the plant was increased in March 1953 through the installation of eight Lapointe electronic picking plant units. These picking units were designed to reject as much as 80% of the mine ore as waste and retain the radioactive ores, thereby decreasing dilution and increasing capacity (LaPointe, 1953; site evidence). The plant was housed in a building 40 feet x 85 feet. The mill equipment was driven by individual electric motors.

Power Plant and Other Facilities

Indore's power supply was through the use of a 128 horsepower D-13,000 Cat diesel engine driving a 125 KVA Crocker-Wheeler generator, allowing for sufficient power to the mill and camp buildings (The News of the North, Oct. 10th 1952). Compressed air was acquired using a 365 cubic feet per minute Gardner-Denver air compressor driven by a second D-13,000 Cat diesel engine (Anderson, 1952). This unit was augmented by an additional 300 cubic feet per minute air compressor in May 1953 (Anderson, 1953b). The plant was housed in a 20 foot x 50 foot building, which also doubled as a garage (site evidence).

Other facilities consisted of assay office, blacksmith shop, warehouse, and later in 1953 a hoist room and headframe. The hoist was a Canadian Ingersoll-Rand 11x8 PSR 2-drum air operated unit (site evidence). A permanent camp was erected during the early summer of 1952 and winterized by the start of production. Buildings consisted of a cookery, two bunkhouses (14 feet x 24 feet), and an office/staffhouse, capable of housing 20 to 30 men (McGlynn, 1971; site evidence).

Mine Crew

Some of the crew included Paul Gliddon, mine manager; Frank Anderson, consulting engineer; Alf Scott, master mechanic; Tommy Forest, cook; R. Dubois, mine superintendent, Bill LeBlanc, powerhouse operator; Don McDonald and Hilding Alfridith, miners; and Jim Sullivan, mill operator (The News of the North, Oct. 10th 1952).

Mild weather during November and December 1952 and the resulting thin ice prevented the transport of important supplies, fuel, and equipment for the proposed shaft sinking operation. Instead, a short prospect winze was sunk below the adit level to test the depth potential of the deposit (The News of the North, Dec. 12th 1952). The winze was driven at an angle to follow the dip of the pitchblende seam, and high values were encountered for a distance of 35 feet in the winze, which was by May 1953 completed to a depth of 60 feet (The Northern Miner, May 21st 1953). It was also planned to extend the northeast drift and drive a new northwest drift on the adit level (The Northern Miner, Jan. 29th 1953). Normal transport to the property was not resumed until February 1953, when Hottah Lake ice was considered thick enough to land a Bristol freighter (The News of the North, Feb. 16th 1953).

In March 1953, it was reported that a short seam of high-grade ore was exposed in the southwest drift, and that the ore was bagged for direct shipment since it assayed well above 10% U₃O₈ grade. It was also reported ore from the 'to-be-milled' stockpile assayed an average 0.25% U₃O₈. This was higher than the company expected because the dump contained both ore and waste material from early development. Stockpiling of mill concentrates continued and no shipments had been made by the date of the report, because negotiations with Government authorities for the sale of uranium concentrates had not been finalized (Anderson, 1953b).

Fraudulent Campaign

It was about this time when skeptics challenged the Indore company to prove the uranium deposit was worth as much as consistently reported in the press. The Ontario Securities Commission wrote an investigative report on the activities of the company, in which it was stated that the erection of the mill was done not on honest and competent advice, but as part of a deliberately fraudulent sales campaign. This resulted in the company losing its license to issue shares in 1953, and a re-organization later in the year resulted in a name change from Indore Gold Mines Limited to Consolidated Indore Uranium Mines Limited. A large amount of funds remained in the company treasury to finance continued developments (The Toronto Star, May 13th, June 18th 1953; The Northern Miner, May 21st, July 9th, July 23rd 1953).

Consolidated Indore Uranium Mines Limited (1953)

Work continued under the banner of the re-organized company in the summer of 1953. Starting late in May 1953, a vertical three-compartment shaft was sunk from the surface to intersect the adit level. The shaft continued to a depth of 190 feet and a 2nd level station was established at 150 feet depth by August 1953 (The Northern Miner, Oct. 8th 1953).

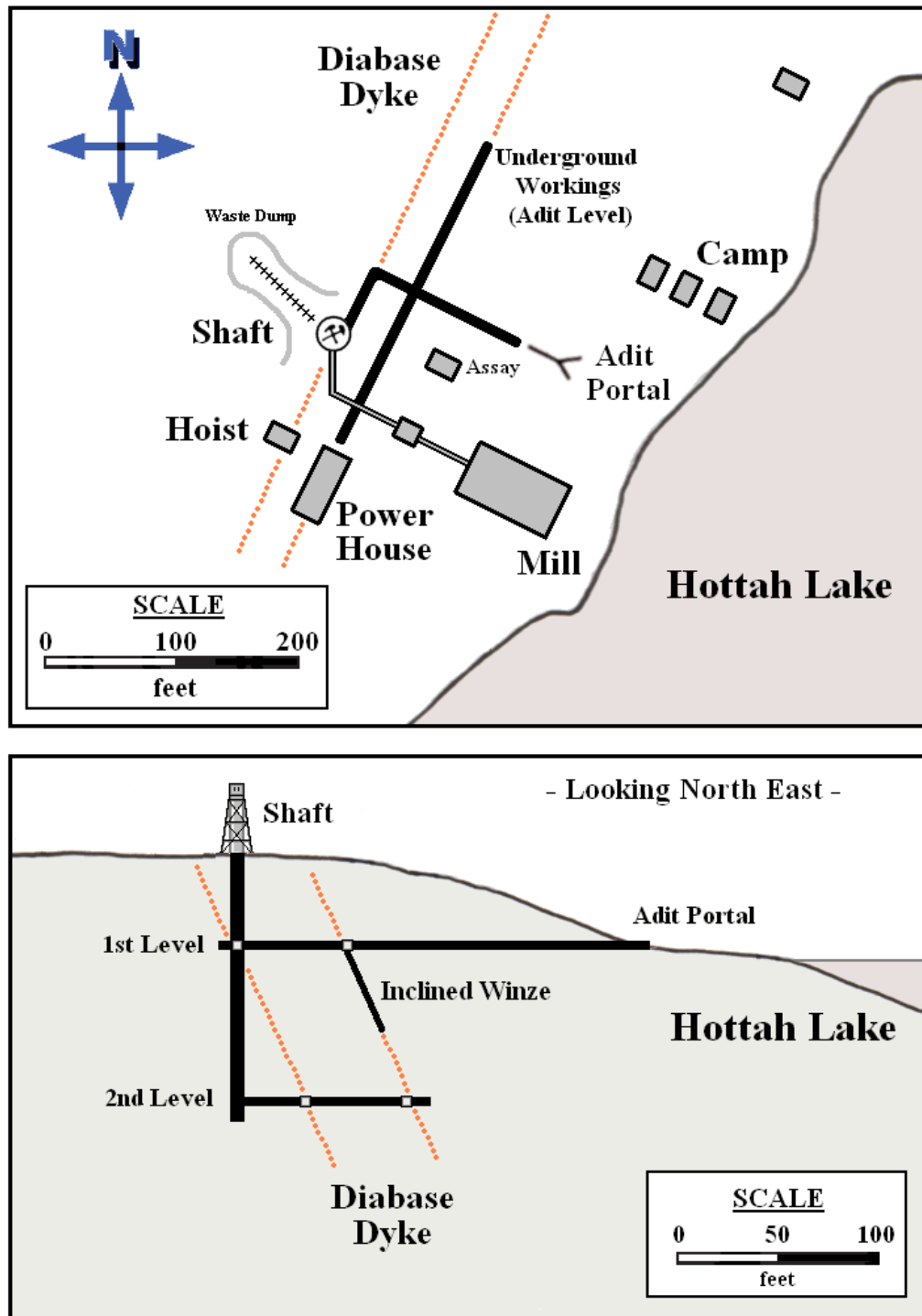


Figure 2. Indore Mine underground and surface plan, c.1955.

At the initiation of shaft sinking, milling operations stopped so that full capacity of mine facilities, crews, and power could be available for development (Anderson, 1953a). No record of production is known to exist as the company kept this information confidential at the time. It is known that a substantial amount of concentrate (with reported grades of 10% U_3O_8) was stored at the property in March 1953. Mill recovery was reported to be averaging 80% (Anderson, 1953b). It should be noted that at some point in the spring of 1953, there were 1,500 tons of broken ore at the ore dump (LaPointe, 1953) Mill tailings in January 1953 were reported running 0.25% U_3O_8 , which was also, oddly, the reported average grade of the deposit before mining commenced (The Northern Miner, Jan. 29th 1953).

Work was suspended in October 1953 due to a lack of funds and an inability to refinance. The Indore company did little work during 1954 other than equipment and supply acquisition, along with some surface exploration of the 'Pitch' claims. Underground development to this point consisted of 622 feet of lateral development on the adit (1st) level (located 60 feet below the surface), a 60 foot inclined winze below the adit level, a 190 foot vertical shaft from the surface (connecting to the adit level), and a station for a 2nd level at 150 feet depth. No lateral development was undertaken on the 2nd level in 1953 (The Northern Miner, Nov. 17th 1955).

Following a meeting of Indore shareholders in August 1954, an appeal was made to the Ontario Securities Commission to allow the company to raise additional finances for the Hottah Lake property. The Commission agreed to allow this activity provided a change of the board of directors and management. Another re-organization of the company was undertaken (The Northern Miner, Aug. 26th 1954).

United Uranium Corporation Limited (1955-1956)

In May 1955, all assets of Indore were sold to United Uranium Corporation Limited in a share exchange (The Northern Miner, May 12th 1955). Underground development resumed under the direction of Norman Byrne and John H. Parker, consulting engineers for the company, in July 1955.

The 150-foot (2nd) level was under development to explore the #1 dyke contacts at this depth, where it was previously indicated to remain strong and high-grade. Six hundred feet of drifting was completed by November 1955, two drifts on the north side and two on the south side of the dyke at both walls. The south drift was directed towards the bottom of the 1953 winze, which was then reached by driving a 92 foot inclined raise. Assays along this raise returned 0.62% U₃O₈ across an average width of 1½ feet. A second raise 18 feet long graded 0.55% U₃O₈ over similar widths. Both raises were driven in the same zone (The Northern Miner, Nov. 17th 1955).

A few short and narrow sections of the dyke were identified during developments as interesting, but overall the erratic distribution of the deposit required selective mining operations that would not be profitable. The total length of these ore zones measured 300 feet with a grade of 0.35% U₃O₈ over an average width of 1½ feet. Diamond drill stations were established on the 2nd level to test the structure at 300 feet and 450 feet depths, with plans to deepen the shaft to 500 feet if warranted. 26 holes were completed in 1956, six of which encountered high values. Operations ceased in the summer of 1956 pending acquisition of more funds to sink the shaft to 500 feet (The Northern Miner, Jan. 26th, Oct. 4th 1956). The company was unable to raise this money, and no further work was being done after 1956. Most equipment and some buildings were dismantled and sold to the Byrne family group of mining companies.

Total Development Summary

The Indore Mine was developed with two levels; the 1st level was the adit level (driven 60 feet below the exposure) from which 622 feet of lateral work was accomplished in 1950-1951. From the adit level a short winze was sunk. This was followed by the sinking of a shaft to a depth of 190 feet with a 2nd level established at 150 feet depth in 1953. Lateral work on the 2nd level commenced in 1955. Total drifting and crosscutting on all levels of the mine is approximately 1,200 feet. A raise from the 2nd level connects to the adit level via the old winze.

The #1 dyke deposit has erratic pitchblende distribution, however some very narrow sections of ore were defined during underground exploration. The total length of these zones is 300 feet with a grade of 0.35% U₃O₈ over an average width of 1½ feet (National Mineral Inventory).

Exploration Since Mine Closure

No work has been reported.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>)



JOHNSTON LAKE

High-Graded (Abandoned)

Gold

63° 01' 30" 114° 10' 00"

Years of Primary Development: 1941-1943, 1981

Mine Development: 1 primary trench; 49m decline, 59m dev.

Years of High-Grading: 1943

High-Grading: 18 tons shipped = 111 oz Au

Introduction

This small gold deposit is located on the northeast shore of Camp Lake, which is just north of Johnston Lake, about 62 kilometers north of Yellowknife, NWT. It has not been visited by the author of this report.

History in Brief

The 'JES' claims were staked in May and October 1941 by James E. Stephens and Stan Hooker. Much prospecting was done over the summer of 1941, and the owners contemplated the installation of a mill in partnership with Robert M. Wynn. Ore was stockpiled from the Cross vein on the north side of Camp Lake, which was mined through a 50 foot long open cut. In January 1943, the claims were optioned to Cominco Limited. A small amount of high-grade ore from the trenches was shipped to Con Mine early in 1943 and processed to produce a small amount of gold. Giant Yellowknife Mines Limited optioned the claims in 1965 together with adjacent groups and performed geological mapping and diamond drilling (5 holes) on a number of quartz veins in a northerly trending depression in the area of the Cross vein. In 1980, Ashnola Mining Company Limited acquired the claims from the estate of James E. Stephens, and in 1981 they drove a short decline on the Cross vein.

Geology and Ore Deposits¹

The region is predominantly underlain by massive greywackes and minor argillites of the Archean Yellowknife Supergroup, which strike north-northeast and dip nearly vertically. A band of mafic to intermediate volcanics several hundred feet wide extends along the east side of Goodwin Lake, while chloritic tuffs, mixed with greywackes occur near Barker Lake. Conformable with the stratigraphy are a number of medium grained quartz diorite sills. Contacts often do not appear sharp because of the fine-grained chill margins and alteration of the enclosing meta-sediments. Many showings are located proximal to these sills. Diabase dykes occur on the property as a late stage post-mineralization event.

Two stages of deformation have affected the stratigraphic sequence, resulting in tight isoclinal folds with very steep plunges and long limbs that are nearly vertical. Only minor flexures such as small drag folds and minor strike slip faults are observed, usually near the fold noses. Mineralization in shear-hosted quartz veins consists of pyrite, arsenopyrite, galena and gold.

Cominco Limited (1943)

The claims were optioned to Cominco Limited in January 1943. Two hundred tons of ore from the Cross vein had been stockpiled by this time. In February 1943, 18 tons of cobbled ore from the stockpiles at the Cross vein was shipped, possibly by tractor, to Con Mine in Yellowknife. This operation produced 111 ounces of gold (Cominco Ltd., 1943). Cominco Limited performed diamond drilling (13 holes), trenching, mapping, and sampling during 1944-1945 for assessment purposes and reported a reserve of 10,150 tons grading 0.48 ounces per ton gold, but the option was dropped in 1948 (National Mineral Inventory).

Ashnola Mining Company Limited (1981)

In June 1980, Ashnola Mining Company Limited optioned the claims from the estate of James E. Stephens and in 1981 they drove a 49 meter long decline south-southeast on the Cross vein, beneath the location of the 1940s open cut. Lateral development consisted of 59 meters of crosscutting at an unknown depth. A trailer camp was erected on the northwest side of Camp Lake and a 2,000 foot trail connected to the decline portal northwest of the Cross vein. This work indicated a quartz vein cutting greywacke turbidites whose northeasterly strike is deflected within 200 feet of the vein's southwestern wall, suggesting faulting along the vein. No other development is reported (Brophy et al., 1984).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

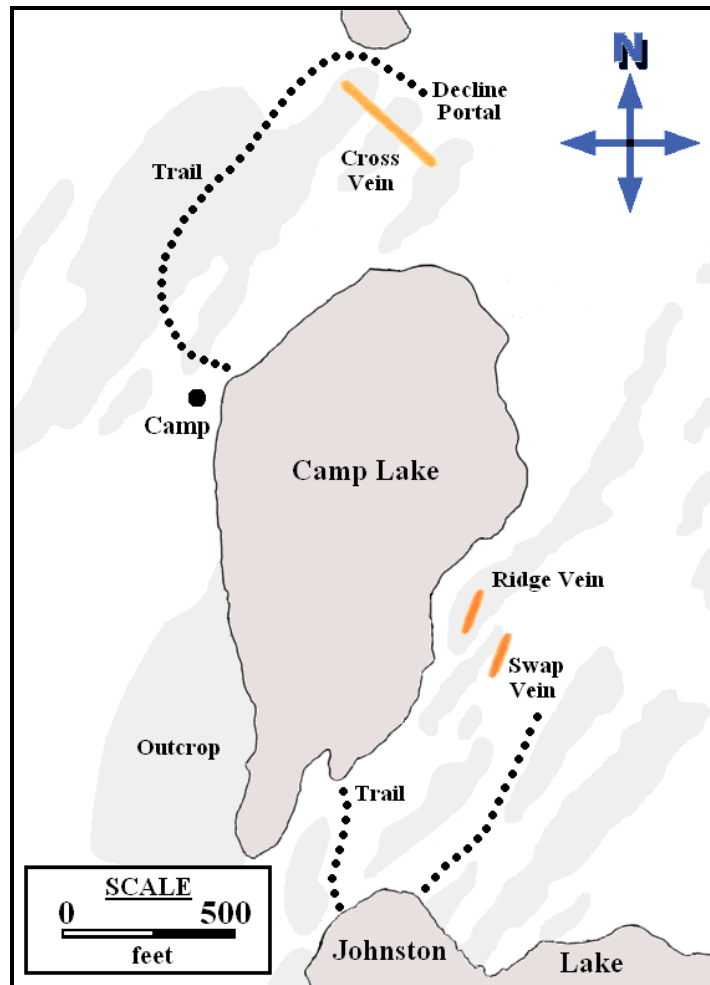


Figure 1. Johnston Lake Mine area.

Exploration Since Mine Closure

No known work.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085OSE0011



MACTUNG

Bulk Sampled (Non-Operational)

Tungsten

63° 17' 00" 130° 10' 00"

Years of Primary Development: 1973

Mine Development: 1,300' adit, 1,000' of crosscutting, 90' raise

Years of Bulk Sampling: 1973, 1984

Bulk Sample: 300 tons shipped = 1.66% WO₃ (1973)

200 tons shipped = ? WO₃ (1984)

Introduction

The Mactung tungsten deposit is the largest known tungsten resource in North America. Mactung is located one kilometer northwest of the Canol Heritage Trail, on the border of the Yukon and Northwest Territories border. It was discovered in 1962 but has not been placed into production. Development occurred during a brief underground exploration program in 1973. It was scheduled for production in the mid 1980s, but a drop in the price of tungsten postponed these plans. The claims are now owned by North American Tungsten Corporation Limited and are scheduled for production once the nearby Cantung Mine has been mined out.

Brief History

Tungsten was discovered here in 1962 by J.F. Allan, and claims were staked by Southwest Potash Corporation Limited, a subsidiary of American Metal Climax Incorporated. The claims were transferred to Amax Exploration Inc. in March 1967, and then transferred to Amax Northwest Mining Company Limited in 1972. A brief underground exploration program followed in 1973. The claims were acquired by North American Tungsten Corp. in 1997.

Geology and Ore Deposits¹

Mactung is hosted by a Late Proterozoic to Cambrian clastic-carbonate sequence at the facies boundary between the Mackenzie Platform and the Selwyn Basin. At the Mactung showing, this zone of transition is characterized by a succession of interlayered beds of pelite, limestone, and limestone breccia. Regional folding and metamorphism in the Selwyn Mountains was developed during Cretaceous orogeny; Mactung is sandwiched between an imbricate thrust zone to the west and a strongly folded zone to the south. The sedimentary sequence around Mactung is intruded by a series of late to post-tectonic Cretaceous biotite, muscovite, and tourmaline-bearing quartz monzonite stocks that have contact metamorphosed aureoles.

The Lower Ore Zone consists of a single massive tungsten-bearing horizon and the Upper Ore Zone consists of three lenses. The two zones are separated by 78 meters of barren hornfelsed shale. The Lower Ore Zone lies entirely within a brecciated limestone unit, which has been metasomatized to pyroxene-marble, pyroxene and pyrrhotite skarn, and minor cherty/chloritic skarn. The highest-grade ore is found in the pyrrhotite skarn. The limestone host to the Lower Ore Zone is folded into an S-shape defined by three limbs, treated separately for mining purposes: the south limb, 150 x 600 x 20 meters, dipping 20° to the south; the north limb, 275 x 75 x 23 meters, also dipping 20° to the south; and the central area, a narrow zone of mineralized skarn connecting the north and south limbs. The Upper Ore Zone consists of three mineralized lenses separated by 18 meters of barren argillite and hornfels. The lower horizon consists of limestone breccia and conglomerate beds up to 1 meter thick, and skarn beds up to 25 centimeters thick, all containing disseminated scheelite. The upper two lenses are 25 centimeters thick layers of scheelite-bearing pyroxene skarn, interlayered with barren hornfels or low-grade cherty skarn.

Amax Northwest Mining Company Limited (1973, 1984)

In 1973, Amax drove an adit tunnel into the deposit to extract a bulk sample. Dimensions of the adit were 7 feet x 10 feet. Total development conducted during that year consisted of 1,300 feet of adit tunnel, 1,000 feet of crosscutting, and a 90 foot raise. A test stope with dimensions of 30 feet x 50 feet was excavated, and 18,000 cubic feet of rock was slashed to make underground diamond drill sites. 5,400 feet of underground diamond drilling was performed and 300 tons of ore were shipped for test milling, with a reported grade of 1.66% tungsten oxides (WO₃). An ore reserve of 30 million tons grading 0.90% WO₃ was announced at the end of the 1973 program (Padgham et al., 1976). In 1982, after extensive reevaluation of exploration results, MacTung ore reserves were upgraded to over 60 million tons of ore grading 0.90% WO₃. In 1984, Amax reported the mining of a 200 tonne bulk sample of ore from the underground workings (Mines and Mineral Activities, 1984).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Exploration Since Mine Closure

Significant exploration has been conducted since the 1970s. By 1980, total diamond drilling footage was reported as 2,321 meters in 51 holes drilled from underground, and 14,190 meters in 90 holes drilled from surface. Canada Tungsten Mining Corporation Limited acquired the property in the mid 1980s and conducted some feasibility studies, including a valuation report in 1986. Aur Resources Limited acquired the property in 1994, and in 1997 North American Tungsten Corporation Limited, the current owners, purchased the claims. The most recent ore reserve suggests 13,669,000 tonnes of ore grading 0.95% WO₃ within the measured and indicated category, plus 13,785,000 tonnes of ore grading 0.84% WO₃ in the inferred category (Roscoe Postle Associates Inc. 2001).

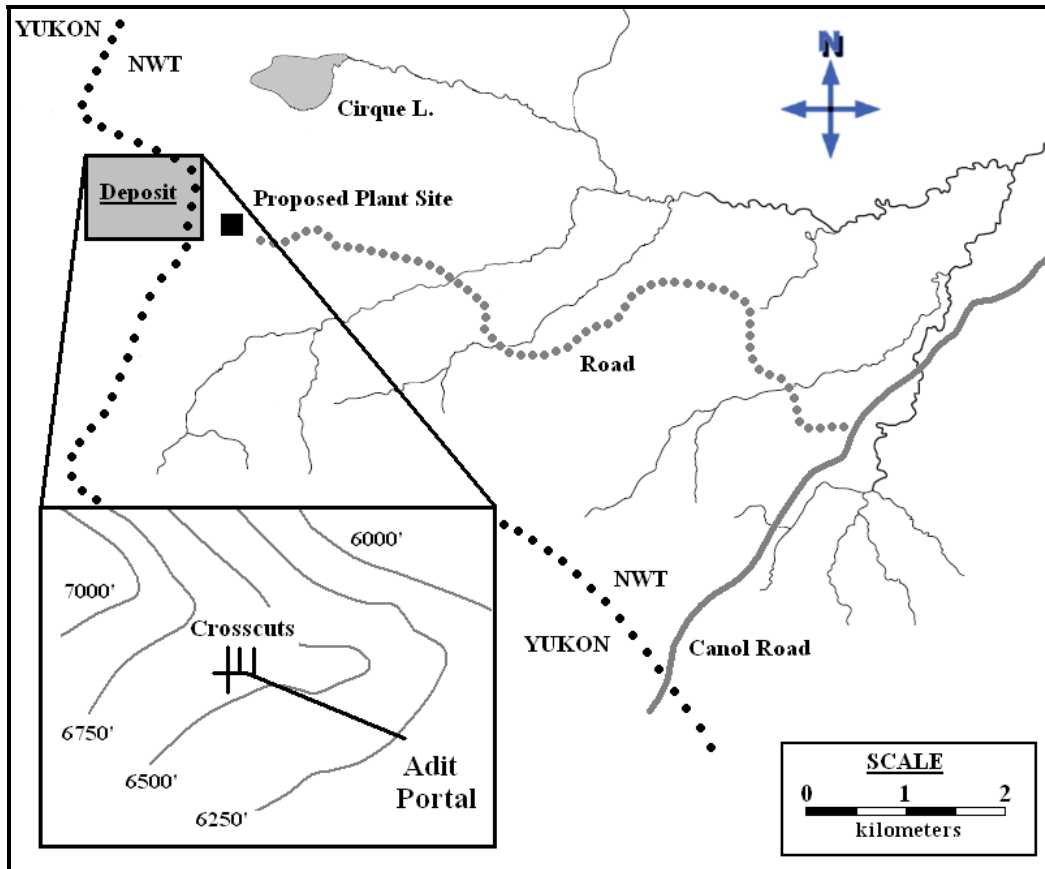


Figure 1. Mactung Mine location map. Inset shows the underground plan.

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**MAHE***Advanced Exploration (Non-Operational)***Gold**

62° 57' 15" 113° 20' 15"

*Years of Primary Development: 1986-1987**Mine Development: 1,600' decline to 200' level***Introduction**

The Mahe property is located on the north side of Knight Bay on the southwest side of Gordon Lake, 76 kilometers northeast of Yellowknife, NWT. The author visited the site in September 2001. The current owners, Bishop Gold Incorporated, are interested in reopening the deposit for exploration.

Brief History

The 'Viv' claims were staked by Jake Woolgar in June 1937 and were acquired by the Borealis Syndicate soon after. Through funding provided by Oro Plata Mining Limited and Borealis, Sentinel Mines Limited was formed to explore the claim group in 1938. The claims later lapsed, and the most interesting portions were re-staked as the 'Mahe' claims in 1944 by Jimmy Mason for Lynx Yellowknife Gold Mines Limited. New claims were staked surrounding the original 'Mahe' claim in 1978 and 1981, and in 1983 Giant Bay Resources Limited took an option on the property. The old ore zones were diamond drilled, suggesting a possible bulk tonnage open pit deposit. An underground exploration program was conducted in 1986-1987, but the results of this were disappointing. The property and surrounding area was re-staked as the 'Knight' group of claims in 2003 by Bishop Resources Incorporated and exploration is ongoing.

Geology and Ore Deposits¹

The showing is underlain by greywackes with interbedded carbonaceous siltstone or argillite of the Archean Yellowknife Supergroup. This package of rocks is isoclinally folded about northwest to west-northwest, with steeply plunging axes. The Kidney Pond zone is auriferous quartz breccia stratabound within thin-bedded alternating meta-sediments. The mineralized zone has been traced along strike for 305 meters, is open at depth and to the northwest. The zone varies in thickness from 6 to 30 meters both along strike and at depth. The sulphide content is generally 2 to 3% and includes arsenopyrite, pyrite, pyrrhotite, chalcopyrite, and galena.

Giant Bay Resources Limited (1986-1987)

Exploration of the property in the early 1980s revealed three main zones plus a number of other interesting surface showings. The largest and most promising of these zones was the Kidney Pond zone (also known as the #1 zone), which was traced for a strike length of 700 feet and drilled to a depth of 550 feet by 1984. More than 50% of the diamond drilling to 1984 had encountered free gold mineralization, which was making it difficult for the company to determine grade of the deposit due to the 'nugget effect'. The only way to accurately ascertain the grade and tonnage of the deposit was to conduct an underground exploration program with bulk sampling. A preliminary reserve calculation in 1984 was 500,000 tons grading 0.15 ounce per ton gold. Metallurgical testing indicated that gold recoveries would be adequate with simple cyanidation methods (The Northern Miner, Aug. 23rd 1984; Giant Bay Resources Ltd. Annual Report, 1984).

By August 1985, over 30,000 feet of diamond drilling had been accomplished basically outlining the deposits. The decision to proceed on an underground exploration program on the Kidney Pond zone was made late in the year, and equipment was mobilized on the 1986 winter road. J.C. Caelles and Jimmy Mason were project managers. The general purpose of the program was to verify the tonnage and grade, with the idea of mining the deposit through open pit methods (Giant Bay Resources Ltd. Annual Report, 1986).

Operations began in April 1986. The decline to the 200-foot level was completed in July 1986, and drifting and raising into the ore zone was underway. The program of development was completed in September 1986. To year end 1986, the following was completed: 1,600 feet of decline (10 feet x 14 feet wide) drive to the 200-foot level, two 10 foot x 14 foot drifts at the 200-foot level for a total length of 540 feet, two 5 foot x 7 foot raises totaling 540 feet, 2,598 feet of surface diamond drilling, and 6,042 feet of underground diamond drilling. Drilling has probed to the 800-foot level. 3,500 individual bulk samples were extracted for testing, taken by sampling each blasted round. Drilling from the drift faces confirmed the continuation of the deposit in either direction, and additional lateral work

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

was planned for 1987 (Burson and Caelles, 1986; Atkinson et al., 1990). Work continued in early 1987 when 700 feet of additional drifting was accomplished on the 200-foot level to follow the deposit, plus additional raising. Operations ceased sometime during the year (Glatiotis, 1989).

Ore reserve statements in 1987 suggested 110,000 tons uncut ore grading 0.62 ounces per ton gold in the Kidney Pond zone, accessible by underground mining methods (Caelles, 1987). This is compared to the over 300,000 tons of ore grading 0.24 ounces per ton gold outlined in earlier diamond drilling, changing the nature of the mine from a high tonnage low grade deposit to the low tonnage erratic grade deposit. It was recognized that open pit mining would no longer be feasible, and underground mining would be the only economical form of mining the Kidney Pond zone.

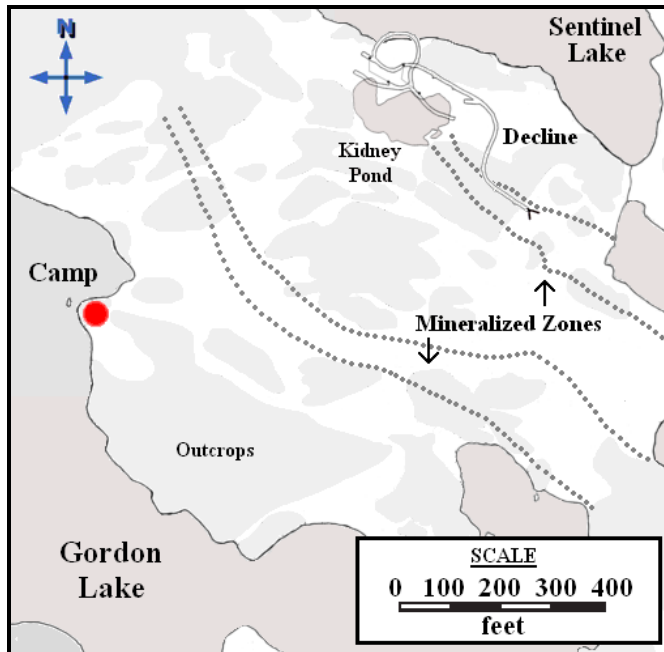


Figure 1. Mahe Mine surface and underground plan.

The results of the underground program fell short of expectations. Bulk samples failed to reproduce drill indicated results (92 muck samples averaged 0.11 ounces per ton gold, and drift and raise wall samples averaged 0.12 ounces per ton gold), and the 'nugget effect' did not assist in tabulating an accurate ore reserve estimate. Economic mineralization occurred in high-grade lenses, but these were inconsistent in nature (Burson and Caelles, 1986). An economic evaluation was performed on the reserve calculations in 1988. This returned negative results, wherein either scenario (open pit or underground mining) could not be mined profitably without significant increases in grades and/or tonnage.

Project managers claimed that the underground work did not test the highest-grade portions of the Kidney Pond zone, but conceded that the results were far below expectations based on target objectives. The project was put on hold in 1988 after some CDN \$5 million spent in exploration since 1983. The company hoped to attract a joint-venture partner to continue with development at the property.

Exploration Since Mine Closure

Most recent ore reserves were published in 1995 and suggest an underground reserve of 10,040 tons grading 0.74 ounces per ton gold and open pit reserves of 3,300 tons grading 0.45 ounces per ton gold (Jarvis, 1995). The property and surrounding area was re-staked as the 'Knight' group of claims in 2003 by Bishop Resources Incorporated. Apogee Minerals Limited had entered into an option agreement in 2004, but no work has been reported.

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National Mineral Inventory (MAHE). NTS 85 I/14 Au 8.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085INW0104

	MITCHELL LAKE <i>Minor Producer (Abandoned)</i>	Gold
		62° 46' 30" 113° 25' 15"

Years of Primary Development: 1957

Mine Development: 15' deep pit/shaft

Years of Production: 1957

Mine Production: 15 tons milled @ 3.0 oz/ton Au

Introduction

This small gold property is located at Mitchell Lake, 60 kilometers northeast of Yellowknife, NWT. A small amount of gold was recovered here in 1957. The site was viewed from the air in August 2004 and a few buildings appear intact.

History in Brief

The original claim was known as the 'Chick' group and was staked in July 1948 by W. Brink. Beneventum Mining Company Limited acquired the claims in August 1956 and some minor development including processing a small tonnage of gold ore was completed in 1957. No mining development has been conducted since 1957.

Geology and Ore Deposits

The property is underlain by Precambrian metasediments comprising greywacke, slate, impure arkose, and quartzite. Gold occurs as coarse grains in a number of quartz veins. Two zones (#1 and 2) of mineralization have been the focus of exploration and mining development (National Mineral Inventory).

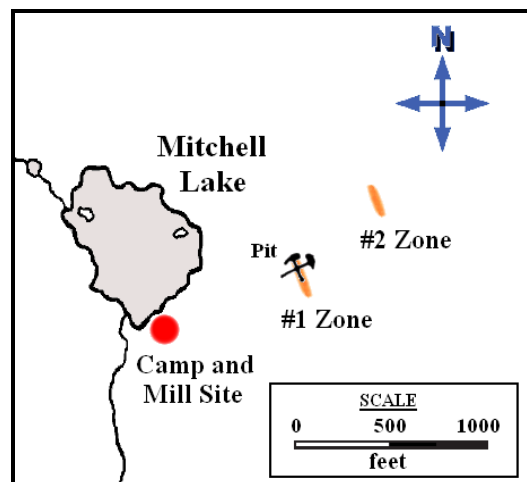


Figure 1. Mitchell Lake Mine location.

Beneventum Mining Company Limited (1957)

During 1957 the company began exploration and development of this property. Two high-grade gold veins, in areas known as the #1 and #2 zones, were the targets of work. Some diamond drilling was done, and gold-bearing intersections of the vein in the #1 zone were reported to a depth of 125 feet.

Some 300 tons of ore grading 3.0 ounces per ton gold were stockpiled from a 12 to 15 foot deep pit on the #1 zone in preparation for the installation of a small milling plant. Only 15 tons of this material was actually processed owing to problems with the crushing plant. Gold recovery is not known, but the assay of 3.0 ounce per ton gold (mentioned above) was reported to be the content of the 15 tons milled as well. The small pit was fitted with a timber-pole A-frame and may have actually been the start of a shaft. A small frame cabin was built on the south-shore of Mitchell Lake, adjacent to the milling plant (The Northern Miner, Jan. 30th 1958; Oct. 23rd 1958).

Milling

The test crusher-mill consisted of a small crusher and tables for the recovery of gold. The plant was designed by Saxum Mill Company Limited, a subsidiary company of Beneventum Mining Company Limited. It was reported that the crusher-mill encountered structural failure during the operation. Alterations were to be made and a larger plant was to be installed (The Northern Miner, Jan. 30th 1958). Shaft sinking on the #1 zone was recommended by Bill McDonald in his 1958 report, but by 1959 the company was developing another gold prospect at Campbell Lake and no further work was completed at Mitchell Lake (McDonald, 1958).

Exploration Since Mine Closure

The property was re-staked by Archie Mandeville as the 'RIBB' claims in 1972 and transferred to Duke Mining Limited in 1974. Five holes were diamond drilled on the #1 vein and four holes on the #2 vein in 1976, outlining 3,000 tons of ore grading 1.50 ounces per ton gold to 120 feet depth, in a 60 foot long ore shoot in the #1 vein (Lorimer, 1974; Von Rosen, 1974).

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National Mineral Inventory (RIBB Claims – CHICK group). NTS 85 I/14 Au 11.

**MON***Minor Producer (Remediated)***Gold**

62° 54' 30" 114° 21' 45"

Years of Primary Development: 1938, 1989-1997*Mine Development:* 2 declines; 64' vertical shaft; open cuts*Years of Production:* 1992-1997*Mine Production:* 11,097 tons milled = ~100 kg Au*Years of Bulk Sampling:* 1990*Bulk Sample:* 2,206 tons milled = 268 oz Au

Introduction

The Mon Mine is located at Discovery Lake, 50 kilometers north of Yellowknife, NWT. The property is accessible by float planes, and in some years, by winter road along the Yellowknife River valley. The current owner (Albert Eggenberger of Yellowknife) has cleaned up the property and has no intentions of reopening the mine.

Brief History

The 'Mon' claims were staked in September 1937 by George Moberly and L. W. Nelson on behalf of Cominco Limited. A short shaft was sunk to 64 feet, but lateral work on this level did not intersect any interesting gold values. In the early 1970s, Jack Stevens operated the claims and put 200 tons of stockpiled ore through a small improvised mill. A second development program got underway in 1989 when Can-Mac Explorations Limited drove a decline and extracted a bulk sample. Ger-Mac Contracting Limited acquired the property in 1991 and installed a milling plant. Gold production continued on a seasonal basis from 1992 until 1997.

Geology and Ore Deposits¹

The Mon Mine is hosted by the Sito Lake Volcanic Complex, part of the Archean Age Yellowknife Supergroup. The quartz bearing ore veins lie near the contacts of a mixed sedimentary - volcanic sequence and thick gabbro sills. The vein system mostly follows the north-northwest striking contact between sills and sedimentary-volcanic rocks, but locally splays out into one or the other rock type. Individual quartz veins are typically lens-shaped, glassy in texture and vary in colour from white to gray. The vein system has been traced approximately 210 meters along strike and to depths generally less than 30 meters. Gold grades are erratic but appear to be correlatable with the sulphide content of the veins which averages less than 1%. Interest has been focused primarily on an S-shaped quartz lens known as the A-zone, a 1 to 6 foot wide continuous vein with a strike length of 320 feet. Gold concentrations are greatest within the hinge of the folded vein (Lord, 1951).

Cominco Limited (1938)

Free gold was located in surface exposures of the A-zone veins in 1937, and immediately it was decided to explore the gold deposit from the underground. Early in 1938 a small prospect shaft (6 feet x 10 feet) was sunk the A-zone to a vertical depth of 64 feet, and a level was started at 62 feet depth. Development included a 30 foot crosscut heading west, followed by a north-westerly 50 foot drift, then a 20 foot crosscut and a 55 foot drift heading directly north. Lateral work was carried out in dimensions of 6 feet x 4 feet. This limited underground investigation encountered only a few quartz stringers carrying low gold values. Work ceased at the end of the 1938 season. P. M. McLaughlin was in charge of a 16-man crew in September 1938 (McDougall and Goad, 1989).

Jack Stevens (1960s-1970s)

In a 1963 report, it was indicated that probably no more than 1000 tons of ore were within the A-zone between the underground workings and the surface, and that up to 500 tons could be extracted by high-grading surface methods. Average grade was reported as 3-53 ounces per ton gold (Rupert, 1963). Jack Stevens purchased a 12.5% equity interest in the claims from G. Moberly, and under a lease agreement with Cominco Limited began to stockpile ore from surface pits starting in 1966. 200 tons of material was stockpiled by 1971. This was essentially a high-grade operation focused on mining selective portions of the A-zone (McDougall and Goad, 1989).

In his 1971 report on operations (submitted to Cominco), Jack Stevens reported milling 48 tons of ore and shipping a 7-pound 7-ounce gold bar to the Royal Canadian Mint. Actual gold content based on Mint returns is unknown (Stevens, 1971). Stevens' mill consisted of a small jaw crusher, cement mixer (used as a ball mill), and a Wilfrey table or jig. The use of a cement mixer as a mill proved unreliable, so Mr. Stevens bought a small ball mill. The mill

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

was powered by a six horsepower Lister diesel generator. Air for pit work was supplied by a 75 cubic feet per minute Canadian Ingersoll-Rand air compressor powered by a Wisconsin VF4 gas engine (Knud Rasmussen, pers. comm.).

Jack Stevens continued milling operations in 1972 and it is believed that the remainder of ore was treated. Total ore milled was probably 200 tons, with grades of 0-10 to 0-20 ounces per ton gold. There is no record of work done after 1972 (Dave Webb, pers. comm.).

Can-Mac Explorations Limited (1989-1990)

The property was optioned to Dave Webb by Cominco Limited in 1988; Webb then assigned the agreement to Can-Mac Explorations Limited. Exploration that began in 1987 resulted in a new ore reserve figure for the A-zone at 5,180 tons with a grade of 0-85 ounces per ton gold. Larger geological reserves of over 7,000 tons were assuming that the A-zone continued in grade to a depth of 230 feet or more. It was recommended by Robin E. Goad, consultant, that a 2,000 ton bulk sample be extracted in order to confirm the continuity of the deposit (McDougall and Goad, 1989).

The program to collect this sample via an underground decline drive began July 25th 1989 by crews from Extender Minerals Limited, contract miners from Ontario. A crew of five men were employed, including one camp cook. Camp facilities were provided by Can-Mac Explorations Limited and consisted of a number of small tents on wood frames. The camp was powered by a 125 kilowatts Cat diesel generator (McDougall and Goad, 1989).

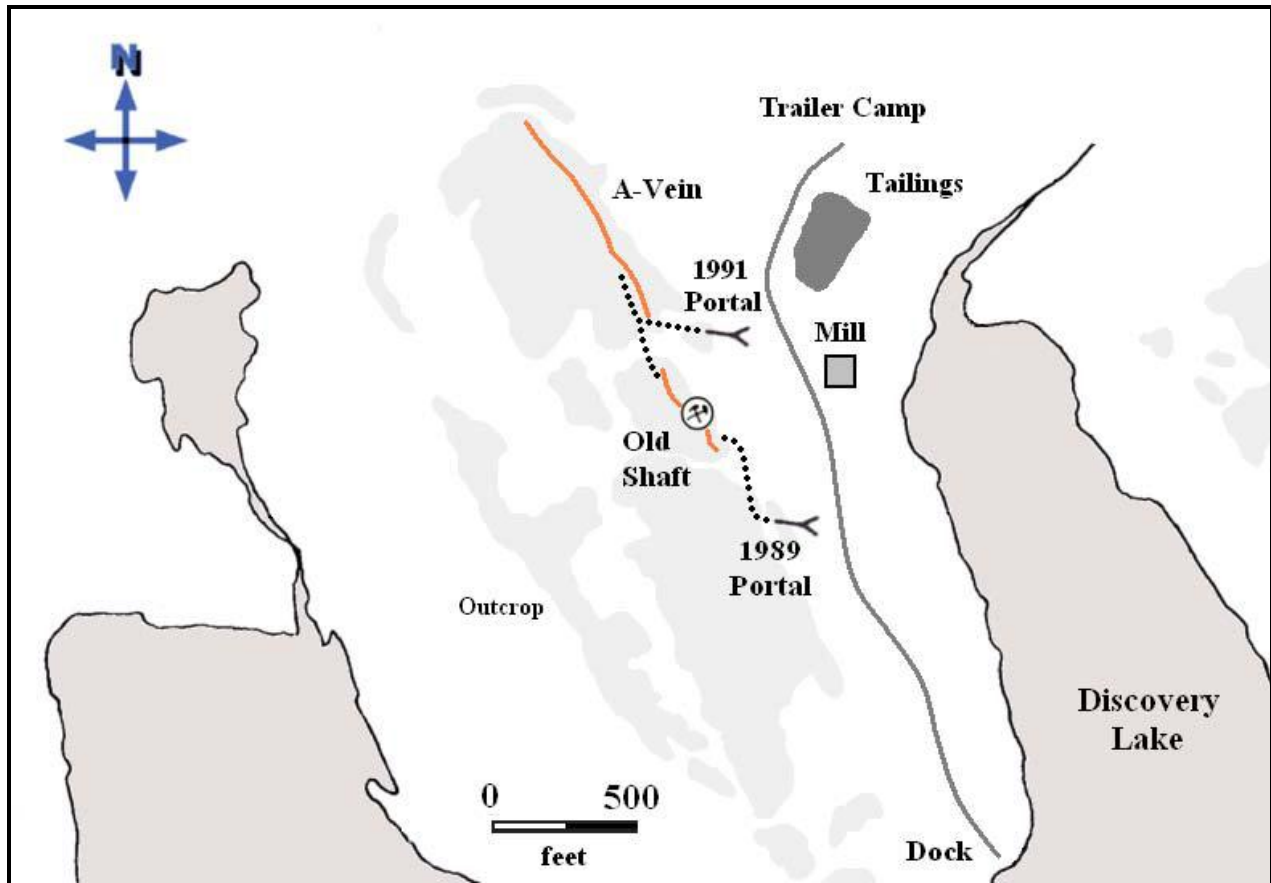


Figure 1. *Mon Mine surface plan and underground workings, c.1992.*

Underground development equipment consisted of a Wagner ST-213 2 yard scooptram, three jackleg drills, and three stoper drills. Air was supplied by a portable 850 cubic feet per minute Joy compressor (McDougall and Goad, 1989).

Development

Access to the vein at a depth of about 80 feet was through a 200 foot decline ramp, 10 feet x 10 feet in dimensions and driven at a grade of -15%. The decline was collared at 630 feet Above Mean Sea Level (AMSL) elevation. Crosscuts totaling 57 feet were driven from the decline to intersect the vein and were used as drawpoints to extract

the ore. The vein was then followed by an 80 foot drift. A 78 foot raise, 5 feet x 5 feet in dimensions, was driven to the surface through the vein, surfacing near the old shaft. Stopping operations were based from the raise area, and the bulk sample was mined to within 15 feet of the surface through shrinkage stopping methods. Total rock extraction amounted to 4,450 tons. Cost of the 1989 program, ending on October 2nd 1989, amounted to \$595,000. About 2,300 tons of ore grading 0.74 ounces per ton gold were stockpiled to await shipment for custom milling (McDougall and Goad, 1989).

Bulk Sample

Early in 1990, negotiations to truck this ore to the Ptarmigan Mine for milling were completed. In March and May of 1990, a total of 2,206 tons were milled through flotation process to recover 268 ounces of gold (Tremenco Resources Ltd., 1990).

Ger-Mac Contracting Limited (1991-1997)

An agreement resulted in the total acquisition of the Mon property by Ger-Mac Contracting Limited, operated by Gerry Hess and Dave Webb. From June to September 15th 1991, a new adit was driven 150 feet west, north of the 1989 portal, to intersect the A-zone at 630 feet elevation, approximately 50 feet below the surface exposures of the A-zone. Other development consisted of a 190 foot drift and two raises totaling 48 feet within the ore zone. A third raise, started at a distance of 100 feet in from the adit portal, was driven 35 feet to break through to the surface for ventilation purposes. 563 tons of ore were removed and stockpiled. Total amount of development waste removed was 2,133 tons. Mining method was conventional drilling with rock removal performed by a 2 yard scooptram. Other equipment consisted of 850 cubic feet per minute air compressor and a 125 kilowatt generator. Gerry Hess was in charge of operations in 1991-1992 (Webb, 1991).

Proven ore reserves above 630 feet elevation were calculated in October 1991 as 5,550 tons grading 0.39 ounces per ton gold (cut). There was also a section of probably reserves along the strike of the vein to the north of the developed workings, totaling 1,320 tons grading 0.39 ounces per ton gold (cut) (Webb, 1991).

An application for the use of a small gravity mill was submitted to the regulatory officials in August 1991 with the hope to install this plant for the following season of work. Approval for this project was granted by year-end. The plant was trucked to the site in the winter of 1991-1992 and production operations commenced in June 1992, lasting throughout the summer months until September 14th 1992 (Mackenzie Land and Valley Water Board - Water License N1L2-1598).

Milling Operations

Ore from a 6 ton capacity steel bin was delivered into a 10 inch x 24 inch Ross Kinetic jaw crusher via a 25 foot feed conveyor. Discharge product from the crusher was ¾ inch size. A fine ore-bin received crushed feed prior to grinding in a 6 foot x 6 foot Marcy ball mill. Gold was then caught by jigs, cyclones, and a concentrator. Gold was refined in Yellowknife, NWT. Tailings were pumped into the stoped-out section of the old Can-Mac decline workings (Mackenzie Land and Valley Water Board - Water License N1L2-1598).

In 1993, the operation was purchased by Albert Eggenberger of Yellowknife and operations continued under the direction of Ger-Mac Contracting Limited. Tailings were deposited into the North stope, accessed from the North portal. Operations ceased for the season in September 1993 (Mackenzie Land and Valley Water Board - Water License N1L2-1598).

In 1994, the period of operation was extended to 6 months between May and October using winterized camps and increased fuel storage. The mill operated at a capacity of 100 tons per day with 87% recovery. This recovery rate was achieved through the installation of a new Knelson concentrator unit in 1995. A new 10,000 ton capacity surface tailings pond was also cleared in early 1994 (Mackenzie Land and Valley Water Board - Water License N1L2-1598). About 10 people were employed onsite, most of whom were friends or family of Albert Eggenberger. Son Ed Eggenberger was in charge of onsite operations, and Garth Eggenberger did the refining operations in Yellowknife and also hauled the winter freight. Don Helfrick worked as Eggenberger's agent (Don Helfrick, pers. comm.).

During 1994-1995, the north decline was extended through about 340 feet of advance. The south portal was re-mined to recover the crown-pillar in the old stoped sections. Underground drilling extended the known vein structure of the A-zone, and mining operations worked upwards from the bottom level towards the old stopes. There was very little milling accomplished in 1995; the mill operated for nine days in October 1995 only (Mackenzie Land and Valley Water Board - Water License N1L2-1598).

In 1996, the mine operated between July 24th and September 24th except for 15 days downtime. In 1997, the last summer of operations, the mine operated for 44 days between July and September. No underground mining or development was conducted in 1997. Mill feed was drawn from a small open cut on the hill near the old surface workings. Some tailings were also re-processed in 1996 and 1997 (Mackenzie Land and Valley Water Board - Water License NIL2-1598).

Year:	Ore Milled:	Gold Produced:
1992	2,072 tons	6.2 kg
1993	2,912 tons	43 kg
1994	1,598 tons	20 kg
1995	465 tons	6.2 kg
1996	2,242 tons	17.5 kg
1997	1,808 tons	?
<u>Total:</u>	<u>11,097 tons</u>	<u>~100 kg</u>

Table 1. Mon Mine gold production, 1992-1997. (source: Mackenzie Valley Land & Water Board - Water License NIL2-1598)

At the end of the operating season of 1997, no work was contemplated for the following year. No known underground reserves existed. The price of gold steadily dropped in the winter of 1997-1998 and continued operations at the Mon Mine were not seen as feasible. No work has been done since. Total production is listed in **Table 1**. A 1997 reserve estimate is 2,800 tons grading 0.357 ounces per ton gold (Mackenzie Land and Valley Water Board - Water License NIL2-1598).

Exploration Since Mine Closure

No exploration has been conducted since 1997 closure.

References and Recommended Reading

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

Mackenzie Land and Valley Water Board Files – Water License NIL2-1598 (Ger-Mac Contracting Limited) *Records used in this file include a 1991 Mining Industry Questionnaire for Water Use Application, 1995 Abandonment and Restoration report, and Annual Reports for the Water License.*

McDougall, J.H. and Goad, R.E., 1989. A Report on the Geology, VLF-EM, and Bulk Sampling Program on the Mon Gold Property. Can-Mac Explorations Limited, December 1989. (*N.W.T Geoscience Office Assessment Report #082922*)

Rupert, R.J., 1963. Exploration Report #6 – Project Termination Report – Mon Property. Consolidated Mining & Smelting Company of Canada, November 1963.

Stevens, J., 1971. Report (MON) Rental 1971. Submitted to Cominco Limited, November 1971.

Tremingo Resources Limited., 1990. Ptarmigan Mine production records, 1990.

Webb, D.R., 1991. Geology and Ore Reserves – Mon Property, Discovery Lake, NWT. Ger-Mac Contracting Limited, October 1991.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085JNE0070

Personal communication: Albert Eggenberger; Knut Rasmussen; Dave Webb; Don Helfrick



MURRAY LAKE

Minor Exploration (Abandoned)

Gold

63° 00' 45" 113° 24' 30"

Years of Primary Development: 1940

Mine Development: 13' vertical shaft, 30' vertical shaft

Introduction

The Murray Lake property is located 78 kilometers northeast of Yellowknife, NWT on the southeast side of Murray Lake. It has not been visited by the author of this report.

Brief History

The 'Pan' claims were staked in September 1937 by Gordon Murray, for Cominco Limited, following the discovery of a high-grade gold vein. Extensive surface prospecting was completed during 1937-1939 by Cominco, and shafts were sunk in the early 1940s. No major work has been done since that time. Alex Mitchell restaked the property as the 'Bairn' group in 1957; the claims later lapsed, and came part of the 'Chris' group. In the 1980s, some surface exploration and sampling was conducted. The 'Murray' claim was staked in January 2003 by Aurora Geosciences Limited, and in July 2003, Evolving Gold Corporation Limited acquired the property through option agreement.

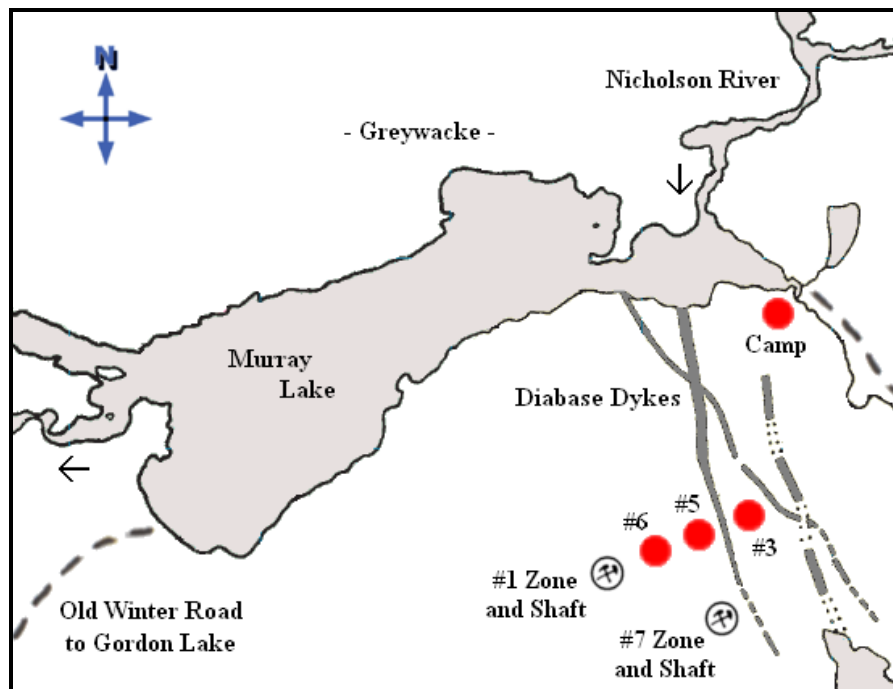


Figure 1. Murray Lake property plan.

Geology and Ore Deposits¹

The claims are underlain by greywacke and slate of the Yellowknife Group, tightly folded across northwest trending axial planes. The beds generally strike northwest and dip northeast at approximately 85°. A few diabase dykes cut the greywacke and slate. Quartz occurs commonly in the meta-sediments as highly irregular veins, masses, and stringers. Most quartz contains abundant white-weathered feldspar, and some contains a little pyrrhotite, sphalerite, galena, pyrite, chalcopyrite, and gold. The better gold showings are southwest of the east end of Murray Lake. Some of the quartz exposures are quite thick here, suggesting a distortion in the axial plane of an anticlinal fold. Here, over a strike distance of 250 feet, numerous quartz showings contain visible gold. One reported channel sample across one of these showings was said to contain up to 10 ounce per ton gold over 3 feet and on another showing, 3 foot channel samples were reported to contain 0-50 ounces per ton gold.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Cominco Limited (1940-1941)

Crews with Cominco completed extensive surface exploration on the 'Pan' claims between 1937-1940. Gordon Murray was in charge of the work (with ten men) during the fall of 1937 and between April and December 1938. About eight men were employed under the direction of J. Kilburn and Gerry Clayton between March and August 1939, when the camp was closed and the property vacated. It is reported that over 110 trenches were excavated, but no diamond drilling or underground work was accomplished (Lord, 1951).

Shaft work was probably undertaken in 1940 or 1941. Two shafts were sunk. On the #1 vein, an 8 foot x 6 foot shaft was sunk to 13 feet depth. On the #7 vein, a 6 foot x 5 foot shaft was sunk to 30 feet depth (Vivian, 2003). Work then ceased, presumably because it was not feasible to place a gold mine into production, especially under war-time conditions.

Exploration Since Mine Closure

Alex Mitchell re-staked the property as the 'Bairn' group in 1957 and conducted some trenching (Barrager and Hornbrook, 1963). The claims later lapsed and became part of the 'Chris' and 'MSSL' groups. In the 1980s, some surface exploration and sampling was conducted (Magrum, 1982). The 'Murray' claim was staked in January 2003 by Aurora Geosciences Limited, and in July 2003, Evolving Gold Corporation Limited acquired the property through option agreement (Vivian, 2003).

References and Recommended Reading

Barrager, W.R.A. and Hornbrook, E.H., 1963. Mineral Industry of the District of Mackenzie. Geological Survey of Canada, Paper 63-9, 1963.

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

Magrum, M., 1982. Geological Report on Chris 1 & 2 Claims Murray Lake Area Northwest Territories. Hidden Lake Gold Mines Limited, 1982 (*Northwest Territories Geology Office Assessment Report #081543*).

Vivian, G., 2003. Summary Report on the Murray Property. 6109527 Canada Ltd., September 2003.

National Mineral Inventory (Bairn - PAN). NTS 85 P/3 Au 2.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085PSW0093



MYRT LAKE

Bulk Sampled (Abandoned)

Gold

62° 47' 15" 113° 14' 30"

Years of Primary Development: 1974

Mine Development: 466' decline ramp, 405' dev.

Years of Bulk Sampling: 1992

Bulk Sample: 2,152 tons shipped = 730 oz Au

Introduction

This mine is located south of Myrt Lake, 68 kilometers northeast of Yellowknife, NWT near the Cameron River. It has not achieved production although underground work was undertaken and a large bulk sample was processed. The site has not been visited by the author.

History in Brief

Gold was discovered in this area in 1938 and the 'SDC' group, covering most of the ground between Dome and Myrt Lakes, was staked. Dome Mines Limited did extensive work including trenching and diamond drilling on many of the properties gold showings, and publicized their plans to put a mine into production. The advent of war in 1939 put these plans to rest. A period of inactivity followed until the claims were re-staked by Sam Otto in 1959 as the 'Myrt' claims. A portion of those claims lapsed in 1960 and the area covering the main deposit was staked by Walter Ternawski as the 'WT' group. Another period of gold exploration commenced, culminating in a 1974 underground development program on the main deposit. In 1992, a bulk sample of ore was trucked to Yellowknife for processing, and no work has been done since.

Geology and Ore Deposits¹

The showing is underlain by sediments of the Archean Yellowknife Supergroup. The mineralized zone is spatially related to an S-shaped drag-fold in greywacke and slates cut by a fault. The main showing (#1 zone) strikes northwest, is about 200 feet long, and reaches a maximum width of 50 feet. Gold is associated with arsenopyrite, pyrrhotite, galena, pyrite, chalcopyrite, and sphalerite in elongated quartz masses and stockwork.

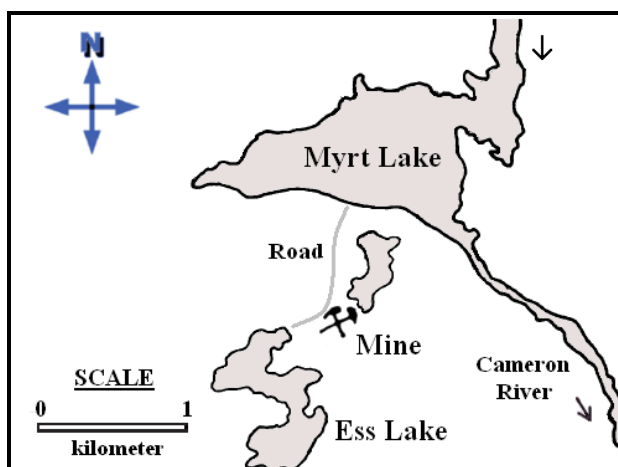


Figure 1. Myrt Lake location.

Precambrian Shield Resources Ltd. (1974)

The property was acquired by Cameron Holdings Limited in 1973, and in 1974 it was bought by Precambrian Shield Resources Limited. During 1974, 466 feet of 15% grade decline was driven easterly under the #1 zone. Lateral development consisted of 405 feet of drifting and crosscutting in the zone. At 125 feet in from the portal, a section of vein 24 feet long was intersected with grades of 0.15 ounces per ton gold. The #1 zone, intersected near the bottom of the decline at a depth of about 125 feet assayed 0.50 opt gold over 28 feet. Chip samples were taken from the walls, and muck samples were taken during the course of drifting and crosscutting. Underground diamond drilling totaled 3,650 feet (Gibbins et al., 1977). About 3,000 tons of ore (grading 0.11 ounces per ton gold) was mined and stockpiled during the 1974 program (Clarke, 1985).

Four gold-bearing ore zones were outlined during the program. The #1 zone was the primary target, and a 175 foot long mineralized ore shoot averaging approximately 35 feet wide was outlined by diamond drilling and underground development (Clarke, 1985). Ore reserves were estimated in 1974 as 25,275 tons grading 0.22 ounces per ton gold to a depth of 125 feet. In 1984, a deep drilling program established ore-grade mineralization to a minimum depth of 500 feet below surface. In 1987, William Knutsen calculated a minable resource to a depth of 110 feet below the surface

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

of 17,510 tons grading 0.35 ounces per ton gold, containing 6,132 ounces of gold. It was believed that this tonnage could be mined using a combination of open pit, long-hole, and underground methods (Knutsen, 1989).

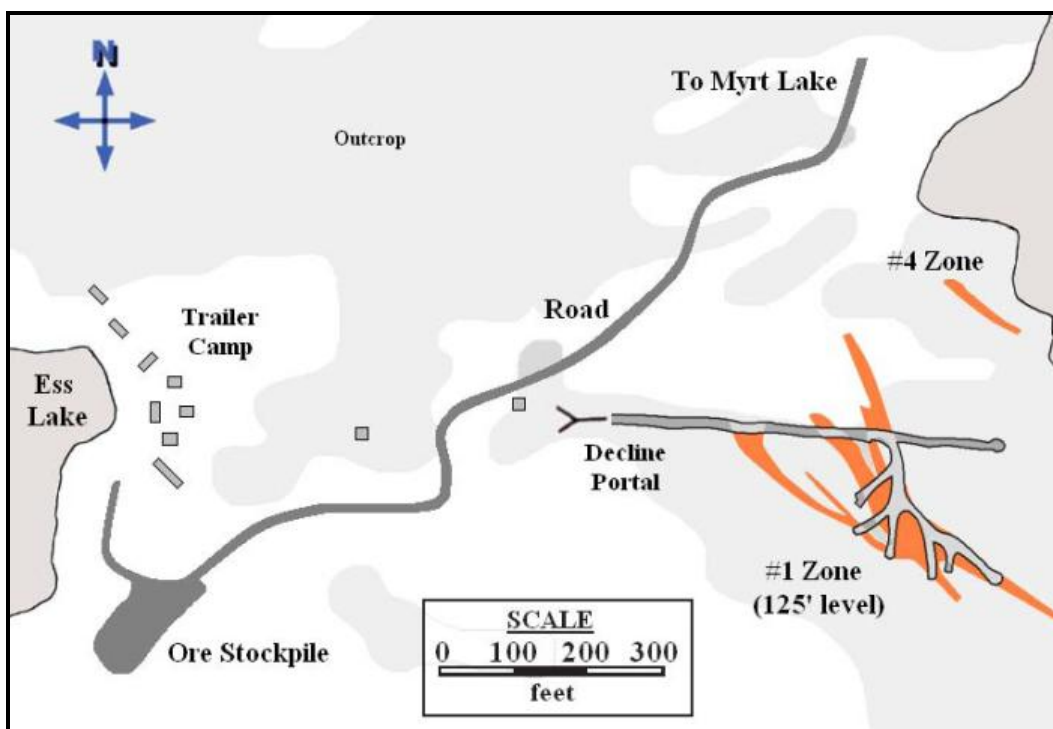


Figure 2. *Underground and surface plan, Myrt Lake.*

Cameron Mining Limited (1987-1992)

William Knutsen, who had been involved in the exploration and development program at the Myrt Lake property in the 1970s, acquired the claims through Cameron Mining Limited in 1987. Decline rehabilitation was reported during that year (Knutsen, 1989). In 1992, stockpiled ore from the decline development was trucked to Yellowknife's Giant Mine for bulk processing, and 2,152 tons were processed in March-April 1992 to produce 730 ounces of gold. Calculated head grade was 0.36 ounces per ton gold with a recovery of 86%. William Knutsen was paid 675 ounces of gold after deduction of treatment fees (Royal Oak Mines Inc., 1992).

Exploration Since Mine Closure

The original mineral leases lapsed in 1998, and in 2003 Trevor Teed staked the new 'WT' claim. No recorded work has been completed since then.

References and Recommended Reading

Clarke, W.E., 1985. Summary Report Diamond Drill Program 1984 - WT Property. Ryan Energy Corporation Limited, January 1985. (*N.W.T. Geoscience Office Assessment Report #15272*)

Gibbins, W.A., Seaton, J.B., Laporte, P.J., Murphy, J.D., Hurdle, E.J., and Padgham, W.A., 1977. Mineral Industry Report 1974 – Northwest Territories. EGS 1977-5. Indian and Northern Affairs Canada, Minister of Supply and Services Canada, 1977.

Knutsen, W.G., 1989. Proposed Plan to Explore and Develop Three Gold Properties in the Gordon Lake-Myrt Lake Area of the Northwest Territories. Cameron Mining Limited, February 1989.

Royal Oak Mines Inc., 1992. Giant Mine custom milling records, 1992.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085INW0105

Personal communication: William Knutsen



MYSTERY ISLAND

Minor Exploration (Abandoned)

Uranium, Radium

66° 02' 40" 118° 02' 00"

Years of Primary Development: 1933-1934

Mine Development: adit tunnel

Introduction

This is the site of a small uranium prospect dating to 1934, located on Mystery Island at the mouth of Echo Bay on Great Bear Lake, 435 kilometers north of Yellowknife, NWT. The island was viewed from the air in July 2005, but no ruins or mine openings could be spotted. It is known that the adit was located on the south side of the island.

Brief History

The claims on Mystery Island were staked by T.G. Donovan of Detroit in the summer of 1931 on what was reported to be an important uranium and silver showing. Donovan publicized very high uranium and silver values at the time of discovery (+58% uranium oxide), but the merits of those assays are unknown ⁽¹⁾. No further work was reported here until 1933 when the uranium deposit was reinvestigated. A small crew was employed driving an adit tunnel. No economic deposits of radium or uranium were uncovered.

Geology and Ore Deposits ²

The area is situated within the Great Bear Magmatic zone, a part of the Bear Structural Province. Most of the island is underlain granodiorite to quartz-monzonite of the Bertrand Lake pluton, part of the Mystery Island Intrusive Suite. These rocks intrude the Port Radium Formation, part of the Early Proterozoic (1.87 Ga) LaBine Group. The LaBine Group, mainly composed of volcanic rocks, outcrops at the western margin of the Wopmay Orogen and rests on a deformed and metamorphosed 1.92 Ga sialic basement. The Port Radium Formation is the oldest unit of the LaBine Group and consists mainly of fine-grained, laminated to thinly bedded, often siliceous argillites, siltstones, and quartzites. The lower sedimentary members are often ripple-marked, crossbedded, and show graded bedding. Cherty argillites often show cavernous structures due to minor amounts of calcareous material.

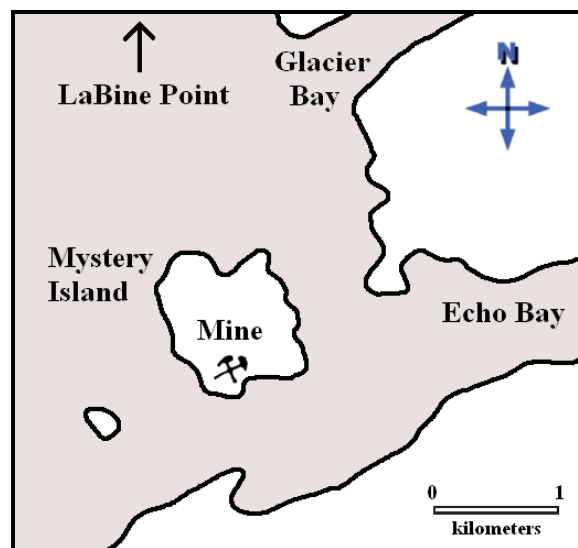


Figure 1. *Mystery Island Mine location.*

The rocks of the LaBine Group are extensively intruded by sill-like, semi-concordant, epizonal plutons of granodioritic to quartz-monzonite composition of the Mystery Island Intrusive Suite. The Bertrand Lake pluton is compositionally heterogeneous, consisting of fine- to medium-grained diorite to granodiorite to the west and monzonite to the east. It generally comprises about 50% plagioclase phenocrysts, 15 to 35% anhedral alkali feldspar, 5 to 12% quartz, and 15 to 20% hornblende and biotite. The plutons and their host rocks were folded at about 1.83 Ga and are cut by a system of northeasterly striking transcurrent faults. The fault zones are filled with hydrothermal quartz veins and stockworks that were deposited between 1,450 to 1,400 Ma ago, in at least five stages, at temperatures of 90 to 450° C. The Mystery Island occurrence is reportedly located near the south end of Mystery Island in granodiorite, intruding the quartzites of the Port Radium Formation, which are preserved in a small area on the east side of the island.

A near vertical, north trending diabase dyke of Middle Proterozoic age cuts the granodiorite near the center of Mystery Island. Mystery Island is displaced by the northeast trending Cameron Bay fault system. Mineralization was reported in 18 veins ranging in width from one to several feet and carrying values in silver and uranium. On the north side of the island a quartz vein was reported to carry values in gold and silver.

¹ The Toronto Star newspaper, July 16th 1931

² Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Radium Corporation of Canada Limited (1933)

A small crew of men were on site in the summer of 1933 under the direction of J.C. Houston developing the silver and pitchblende deposit (The Northern Miner newspaper, Aug. 3rd 1933). Apparently a short adit tunnel was driven into the deposit, although results of this work are unknown. *The Northern Miner* newspaper later reported (Sept. 6th 1934) that no major development was undertaken on the property in 1934, although a small crew under the direction of John Field was employed on the island.

Exploration Since Mine Closure

No known work.

References and Recommended Reading

The Northern Miner newspaper articles, 1931-1934.

The Toronto Star newspaper articles, 1931-1934.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 086LSE0003

**NED***High-Graded (Abandoned)***Gold**

62° 32' 45" 113° 34' 30"

Years of Primary Development: 1940-1942*Mine Development:* trenches*Years of High-Grading:* 1943*High-Grading:* 8 tons ore shipped = 12 oz Au

Introduction

This small gold deposit is located on a tiny island near the southeast end of Hidden Lake and 42 kilometers east of Yellowknife, NWT. It was high-graded with several pits in the early 1940s. The area was visited by the author in August 2009.

History in Brief

The 'Ned' claims were staked in 1938 for NWT Gold Mines Limited. Surface exploration and pitting on the veins was performed 1941-1942, and in 1943 a small shipment of ore was sent out to be milled at Yellowknife's Con Mine.

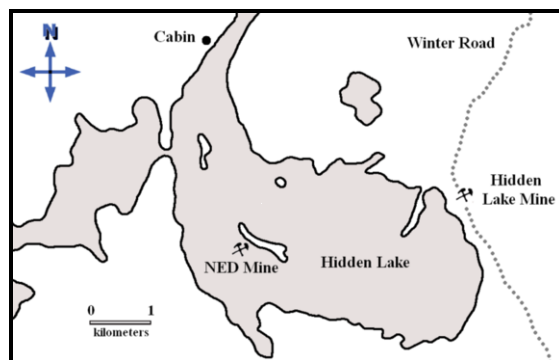


Figure 1. Hidden Lake area, showing location of Ned Mine.

Geology and Ore Deposits¹

The showing lies within the Archean Age Burwash Formation of the Yellowknife Supergroup. The area is predominantly underlain by knotted quartz-mica schist (metamorphosed greywacke-argillite turbidite). These strata are folded by at least two phases of Archean Age deformation, and metamorphosed to lower amphibolite facies in a 40 kilometer wide, north trending belt. Seven veins were discovered on the property.

NWT Gold Mines Limited (1940-1943)

Three men were employed in 1940-1941 with trenching on the vein. A rich shoot 35 feet long by 8 inches wide was identified, and a total of seven veins were discovered by July 1941. Bill McDonald, Sam Otto, George Russell, Norman Easton, and Gil Hagen were some of the men involved in this operation. In 1942, J.A. Buchanan, another member of the outfit, reported that high-grade material was stockpiled awaiting shipment (The Yellowknife Blade newspaper, June 19th 1942). In February 1943, Con mine reported the processing of eight tons of ore from NWT Gold Mines Limited grading 1.47 ounce per ton gold, containing 12 ounces of gold (Cominco Ltd., 1943).

Exploration Since Mine Closure

No known work.

References and Recommended Reading

Cominco Ltd., 1943. Con Mine production records, 1943.

The Yellowknife Blade newspaper articles, 1941-1942.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

**NEGUS***Producer (Remediated)***Gold**

62° 26' 15" 114° 21' 30"

Years of Primary Development: 1938-1952*Mine Development:* vertical shaft + winze to 2,150', 21 levels*Years of Production:* 1939-1944, 1945-1952*Mine Production:* 490,808 tons milled = 255,807 oz Au

Introduction

Negus Mine is located on the Con Mine property, located on the south side of Yellowknife, NWT. It was a substantial gold producing operation of the early years at the Yellowknife gold camp until a depletion of ore reserves in 1952. The site was cleaned up in the following years and the mineral claims were purchased by the owners of the adjacent Con Mine. The area has been re-developed and very little remains of the original operation.

Brief History

The 'Negus' claims were staked by Ole Hagen in early 1936 for Yellowknife Gold Mines Limited. He was one of the first to tie unto the 'Con' group of claims, staked the previous fall by Cominco Limited. Apparently no gold showings were discovered until August 1937 when the Rycon shear zone was traced south into the 'Negus' claims. Negus Mines Limited was formed to develop the property in 1938.



NWT Mining Heritage Society

Figure 1. *Negus Mine, 1942.*

By the summer of 1938, enough work had been done to start major development, and production began in February 1939. Negus was fairly successful during World War II and maintained production well into the conflict, only closing in mid-1944 once conditions became too unbearable. The mine reopened in 1945 and soon thereafter discovered a rich ore zones within its claim group (Campbell shear zone). A program of expansion and development proceeded in 1947-1948, and by 1949 the mine was producing at double the capacity compared to pre-war times. The bulk of the gold resource was tied up in refractory ores, and as a result the mine was not able to pay for its extensive development, suffering massive financial losses. These factors resulted in a 1952 closure of the mine. Some gold recovery continued into 1953, but no work was being done after 1955, when the plant was sold to Rayrock Mines Limited. The mineral claims were purchased by the adjacent Con Mine in 1953.

Mine developments have continued beneath the original Negus property where Con Mine has been mining the far reaches of the Campbell shear zone since the 1950s. Recovery of old stockpiled concentrates also occurred in the 1970s.

Geology and Ore Deposits

The property is underlain by meta-andesite of the Archean Yellowknife Group. Two shear zone systems, the Negus-Rycon, and the Campbell, pass north through the property. The original gold discovery was upon the Negus-Rycon shear, which contains a number of thin, high-grade, west-dipping ore shoots. It is the southern extension of the Rycon

deposit (part of Con Mine) and consists of gold veins and lenses up to 400 feet in length and an average width of two feet. The Campbell shear zone does not outcrop and was discovered by drilling based on the theory that there was a faulted extension of the Giant Mine orebody. It dips to the west and strikes north. Gold within this deposit was found to be somewhat associated with arsenopyrite (Lord, 1951).

Negus Mines Limited (1938-1953)

The first exploration at Negus was financed by Joe Errington of Toronto during 1937, when several drill holes were put down on the narrow and rich veins. It was clear from the start of exploration that the vein system was a southerly continuation of the Rycon shear zones (part of Con Mine), and this fact alone was enough to entice several investors to sink money into a larger development project (The Northern Miner, Nov. 18th 1937). The activity at the adjacent Con Mine speeded work at Negus, and in early 1938, Negus Mines Limited was formed by Charles McCrea, a former Ontario Minister of Mines. Fourteen men were employed on the claims in February 1938 with the construction of camp facilities for 20 men and preliminary mining preparations. Work after August 1937 was under the direction of A.E. Kipps, consulting engineer. Diamond drilling over 20 holes (5,000 feet) through the winter of 1937-1938 tested the veins to a depth of 420 feet and trenching identified numerous parallel vein systems with high-grade widths (The Northern Miner, Feb. 3rd 1938).

Prospect Shaft

First mining development was the sinking of an inclined shaft (#1 shaft) on the #1 vein to a depth of 100 feet. A temporary mining plant was rented from the Giant Mine. This shaft was the only underground opening at Negus when a full mining and milling plant was ordered in March 1938, which speaks of the good values found early in development. W.G. "Bill" Stuart commenced supervision of the project during April 1938 as resident engineer (Lord, 1941).

Surface development and plant preparations were undertaken during the summer of 1938. A log cabin and tent camp, erected in 1937, was replaced with more permanent frame structures during the year, including a two-story bunkhouse and an office building. A small log cookery to house the construction crews was in use.

Sinking of the #2 shaft, designed for production, was started in September 1938. The pouring of the first gold brick at Con Mine during that month further spurred progress at Negus Mine, and the laying of the mill foundation began. It is reported that gravel for the concrete foundations of the powerhouse, mill, and other buildings was acquired from a gravel quarry at Tartan Rapids up the Yellowknife River, where it was barged to Negus dock and hauled by Cat tractor for mixing to form concrete. This was all mixed using "sweat boards".



NWT Mining Heritage Society

Figure 2. *Negus Mine brick pour, 1939. Mine manager Jock McNiven on the left.*

Start of Production

Plant installations were completed by the end of the year, and milling started on February 5th 1939. The first gold brick pour on February 21st 1939 put Negus Mine on the books as the Northwest Territories' second gold producer with production of 50 tons per day. Early production was derived from development ore. The original mine manager, W.G. Stuart, was replaced by Jock McNiven in March 1939. At the start of production no development, other than diamond drilling, had been done below the 100-foot level of the mine. By July 1939, the 200-foot level was being developed with a crosscut driven east to intersect the vein systems (Lord, 1941).

Milling Plant

The 1939 production plant was based on simple amalgamation and cyanidation of gold-bearing ores. Ore from a 50 ton coarse ore-bin was crushed in an Allis-Chalmers jaw crusher set to $\frac{3}{4}$ inches with a rating of 90 tons per day. A long conveyor gallery passed crushed product up into the fine ore-bin of the milling plant, originally housed in a 112 foot x 44 foot timber building. From this bin, ore was feed into a $5\frac{1}{2}$ foot x 6 foot Allis-Chalmers ball mill, grinding to 70% minus 200-mesh, in closed circuit with an Akins 30 inch classifier. Coarse gold was caught in a Gardner-Denver mineral jig and on three blanket tables, from where concentrates were sent for amalgamation. Tailings from the tables entered a standard cyanidation circuit, consisting of two 22 foot x 10 foot thickeners, three 12 foot x 12 foot agitators, and an Oliver 8 foot x 8 foot filter. Final gold was recovered using a Merrill-Crowe precipitation unit. Precipitate and amalgam material were combined and sent for refining (Bruder, 1939).

Power Plant

The milling plant was completely electrified. A Ruston-Hornsby diesel engine was connected to a 125 KVA General-Electric generator, which supplied the entire camp and plant with sufficient power. This unit was installed in the powerhouse, a 40 foot x 80 foot structure that also housed two 750 cubic feet per minute Bellis-Morcom air compressors, originally driven by 156 horsepower Ruston diesel engines (Bruder, 1939). In 1940, a Dominion-Crossley 200 horsepower engine was installed to operate a Crocker-Wheeler electric generator and an 840 cubic feet per minute Canadian Ingersoll-Rand air compressor (National Archives of Canada).

Hoisting Plant

In 1939, the #2 shaft (three-compartment) was fitted with a 50 foot timber headframe and a 30 inch x 18 inch two-drum hoist manufactured by Manitoba Bridge and Iron Works Limited. In 1941, the headframe was enlarged and a new electric two-drum hoist 48 inch x 36 inch size was installed for a deepened shaft (National Archives of Canada).

NegusVille

By 1940, a company town site had been established a short distance south of the Negus campsite, fondly known as NegusVille. A large number of residences were built here over the years, some of which were small log bungalows.

The #2 shaft was opened up at the 300-foot level in August 1939, where high-grade ore from the #1, 2, 3, and 15 vein systems continued to supply the mill with ore (The Globe and Mail, Aug. 17th 1939). As vein systems ran in a north and south direction east of the shaft, and dipped towards the west, crosscuts were run from the shaft easterly to intersect the strike. The #15 vein (the most easterly deposit) intersected the shaft at a depth of about 650 feet, therefore after that depth, all work was being done west of the shaft. The #15 vein was the richest and the focus of most development (National Archives of Canada).

Other expansions during the 1940's included the erection of a larger cookery to replace the old log building. A two-story recreation hall was built in time for New Years celebrations of 1943 organized by the Negus Recreation Association. It contained, on the main floor, a commissary, poolroom, barbershop, reading room/library, and bowling lane. A dance hall and gymnasium was on the top floor. The building was constructed with volunteer labor and for a time was the most advanced recreational building of its type in Yellowknife, until Con Mine built the Jewitt Hall in 1947. Recreational programs included an organized hockey team to compete with other local teams. Baseball teams were also organized, requiring the need for a ball field that was cleared at the camp later in the 1940's. Expansions at the mining operation included the installation of a new boiler plant adjacent the mill building in 1940. Previously, the operation was supplied with heat from a 30 horsepower wood-fired boiler located at the campsite, but the need for additional capacity resulted in the installation of two Foster-Wheeler oil boilers (National Archives of Canada).

Hydro Power

The advent of hydro power from the Bluefish Hydro Plant in 1941 was an important event for Negus, bringing luxuries to the operation that couldn't be economically provided before, such as the electrification of the NegusVille town site. Hydropower also made Negus a more efficient operation, and during 1942 production was increased to 70-tons per day. By this time, the #2 shaft was 950 feet deep and mining was focused on the #3 and #15 veins west of the shaft. This all sparked a building frenzy to provide accommodations for a growing crew. Three single-story bunkhouses were built during 1941-1942 (later converted into duplex residences), along with several other houses including the large mine manager's residence. The Negus Mine employed about 50 people during this time (National Archives of Canada).

Temporary Mine Closure 1945

The war did not hamper operations to a great extent and many believed that Negus would continue to produce during the conflict. Eventually, however, war pressures became too great, and milling ceased on October 18th 1944 due to manpower shortages. The mine became the last gold operation to close in Yellowknife at that time, but development operations underground continued with a skeleton crew. A sinking crew under the direction of Lanky Muyres deepened the #2 shaft to 1,330-foot depth with two new levels at 1,100- and 1,250-foot depth. Ore reserves at December 31st 1944 were 23,800 tons of an unknown grade (The Toronto Star, July 18th 1945). During closure, mine exploration and development proceeded. A new vein was discovered 700 feet east of the shaft on the 200-foot level in early 1945. This may have been the #26 shear zone (The Toronto Star, Apr. 12th 1945).

Post War Operations

Negus resumed milling operations on July 16th 1945 with a crew of 60 men, pouring the first post-war gold bar on August 23rd. Production was reported back to normal rates in October 1945 (Lord, 1951). A fire that destroyed the original warehouse and dry buildings in October 1945 did not delay a resumption of work, and these buildings were quickly replaced with temporary structures. In the 5½ months of operation in 1945, the mine produced at a rate of 60-tons per day with gold recoveries upwards of 94%. Ore reserves in the old vein systems at December 31st 1945 were 36,300 tons averaging 0.61 ounces per ton gold, plus 11,000 tons of probable ore grading 0.57 ounces per ton gold (The Toronto Star, July 19th 1946).

Discovery of Campbell Shear Zone

During 1945, geologists at the adjacent Con Mine were formulating theories about the existence of a massive gold-bearing shear zone underlying the Con and Negus properties. This massive gold bearing zone was predicted to be the southern faulted extension of the Giant Mine ore-bodies then under development. The new ore body was predicted to occur immediately to the east of the shaft, beneath the Negus mine camp. This was confirmed in January 1946 when a diamond drill hole from the 1,250-foot level intersected gold bearing material in the Campbell shear zone. This material was within a mineralized quartz vein 1.5 feet wide with grades of 1.94 ounces per ton gold. The shear zones themselves were in widths of 12 to 30 feet. Two other drill holes from the surface defined its upper boundaries and its western dip towards the Con Mine. Other development in early 1946 consisted of driving a crosscut on the 10th level to follow the main veins to the west of the shaft. A small vein was intersected on the 10th level and was found in widths of 1 foot with grades of up to 3 ounces per ton gold (Lord, 1951).

In preparation for shaft sinking, some new facilities were planned. This included the installation of a new 'central' heating plant (100 horsepower Inglis boiler and 78 horsepower Oilwell locomotive boiler), upgraded steel headframe (90 feet high), and the erection of several new plant and camp buildings. These included a new dry, mine shop, refinery, warehousing, several new bunkhouse units (to replace the two-storey structure destroyed by fire in December 1946), and townhouses. The #2 shaft was deepened to the 1,940-foot level starting in December 1946 and finishing a year later. Two crosscuts from the 1,775- and 1,425-foot levels were driven west from the shaft towards the new shear zone starting August 29th 1947, encountering gold ore in February and June 1948, respectively (Lord, 1951). The average mine payroll in 1947 was 120, including 16 staff, 60 on surface, and 44 underground. Mine staff included Jock G. McNiven, mine manager; E.C. Rudd, assistant mine manager; T.W. Dawson, mill superintendent; A.W. Jolliffe, consulting geologist; Carl W. Wicklund, mine captain; Jack Tibbitt, engineer; Lloyd Roback, assistant mine engineer; D.C. Collin, accountant; James P. Dunn, master mechanic; W.A. McKeown, surface foreman; and H.J. Griffith, warehouseman (The Western Miner, Nov. 1947).

Milling Expansion

Freight and equipment brought to the property during the summer of 1947 allowed for the construction of a mill addition. This was necessary to help treat the refractory ores of the Campbell shear zone. The new addition made it possible for the milling of 180 tons per day with a new Denver flotation circuit, 18 inch x 32 inch Telsmith crushing plant, and secondary ball mills. Although some sections of the expanded mill were in operation in December 1947, the new plant was fully functional by June 1948. The company also made plans for a roaster plant at this time.

Production and Development of Negus Shear

The massive expansion and development program resulted in financial losses during 1946 and 1947, because the original gold veins were not large enough to help pay for the costs of operations. The old veins were narrow and expensive to mine, but luckily they were high-grade. The new Campbell shear zone was expected to reverse those losses once development had made it ready for production. The #15 vein was being explored on the 11th level west of the shaft in March 1948, and was reported to have opened up good ore (The Toronto Star, Mar. 18th 1948). All production from the old Negus vein systems stopped during 1948, and by February 1st 1949 all production was from

the Campbell shear zone (The Western Miner, Oct. 1949). A large exploration program during the fall of 1947 on the 2nd and 3rd levels failed to locate ore in the #26 shear zone, located southwest of the mine camp, and these old workings were abandoned in October 1947 (Lord, 1951).

Campbell Shear Zone Development

Mining and development were focused primarily on the 11th and 13th levels during 1948-1949, within the Campbell shear zone. These two levels by themselves were predicted to contain enough ore to last for many years. Higher-grade ores of greater tonnage were introduced to the mill in the first five months of 1949 compared to the same period in 1948. However, gold recovery decreased due to the inability to process the refractory ore pockets from the new ore zones. Flotation concentrates were stockpiled to await future treatment. Ore production was maintained at over 175 tons per day. Between 1948 and 1950, two sub-levels between the 11th and 13th levels were developed to mine high-grade ore from the Campbell shear zone. Detailed investigations showed, however, that although large, the entire zone did not contain gold-bearing ore. Most mining and stoping development focused on two well-defined ore shoots.

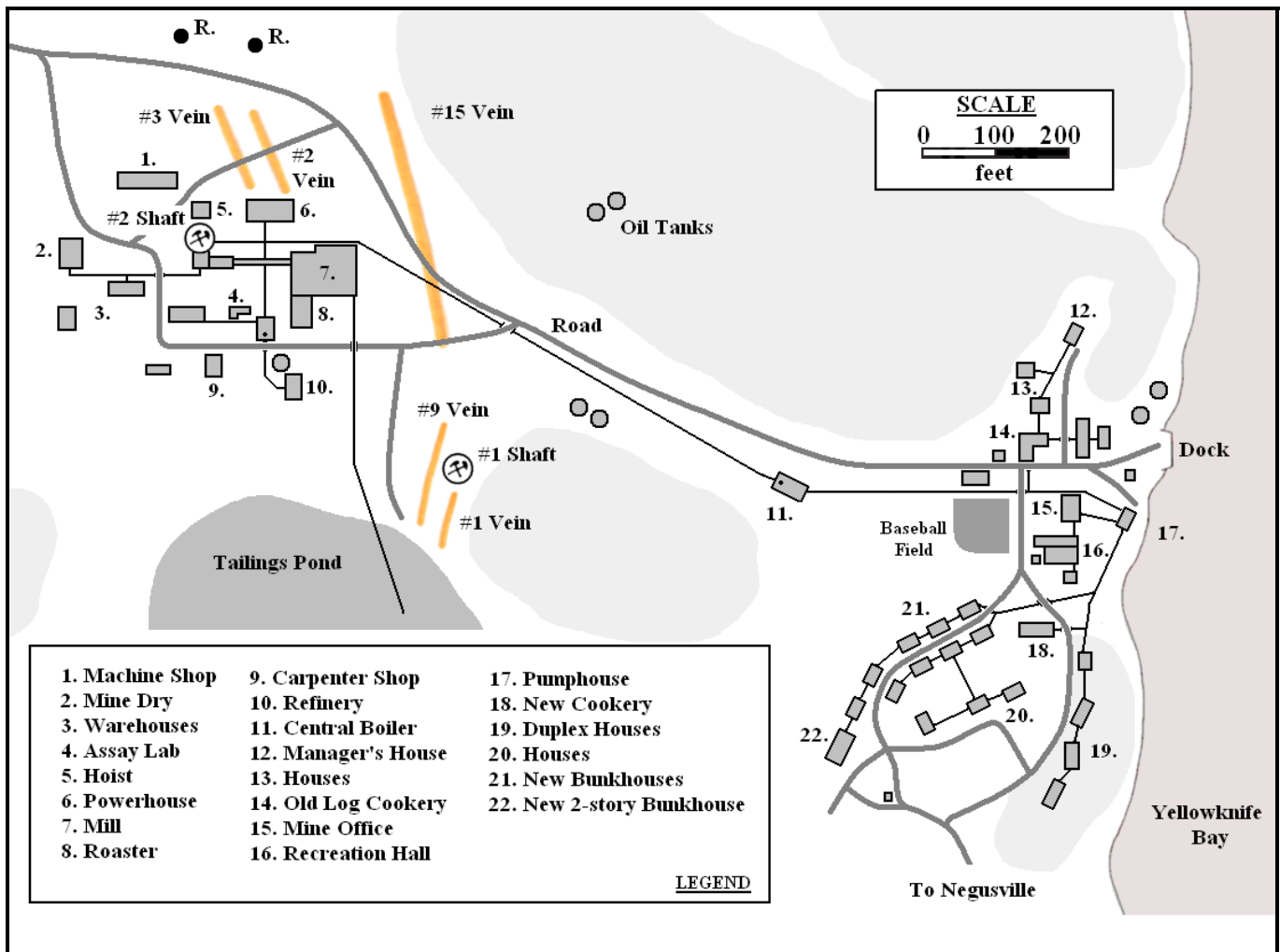


Figure 3. Negus Mine surface plan showing strike of gold veins, c.1952.

Refractory Ore

Since the startup of the new mill in 1948, a flotation concentrate containing gold products was stockpiled. Research indicated that a recovery of 83% could be achieved using a low-temperature roasting plant. Refractory gold only accounted for 15% of production; therefore gold was still being produced in great amounts despite the stockpiling (The News of the North, Dec. 2nd 1949). In June 1950, the 500th gold brick was poured at Negus (Coulson, 1950). The mill was brought up to 225 tons per day capacity in 1950. Underground development took priority during 1950-1951, but equipment for the roasting plant was ordered early in 1951 (The Northern Miner, Feb. 22nd 1951). The Negus company opened up its un-issued shares for public sale in 1949 in order to raise funds for the installation of

the roasting plant, to repay certain loans, and for continued development of the new ore zones (The News of the North, Dec. 2nd 1949).

Winze Development

In order to open up reserves below the workings, preparations were made to sink a winze from the 13th level to a depth of 375 feet to the 2,150-foot level in late 1949. During 1950, the winze opened up three new levels (19th, 20th, 21st), the deepest of which was at 2,150 feet depth. Crosscuts extended from the winze stations to intersect the westerly dipping shear zone, upon which drifts around 900 feet in length were completed for stoping operations both north and south. Mining was concentrated in the numerous shoots and ore lenses on each level (The Northern Miner, Oct. 19th 1950; Feb. 22nd 1951; July 19th 1951).

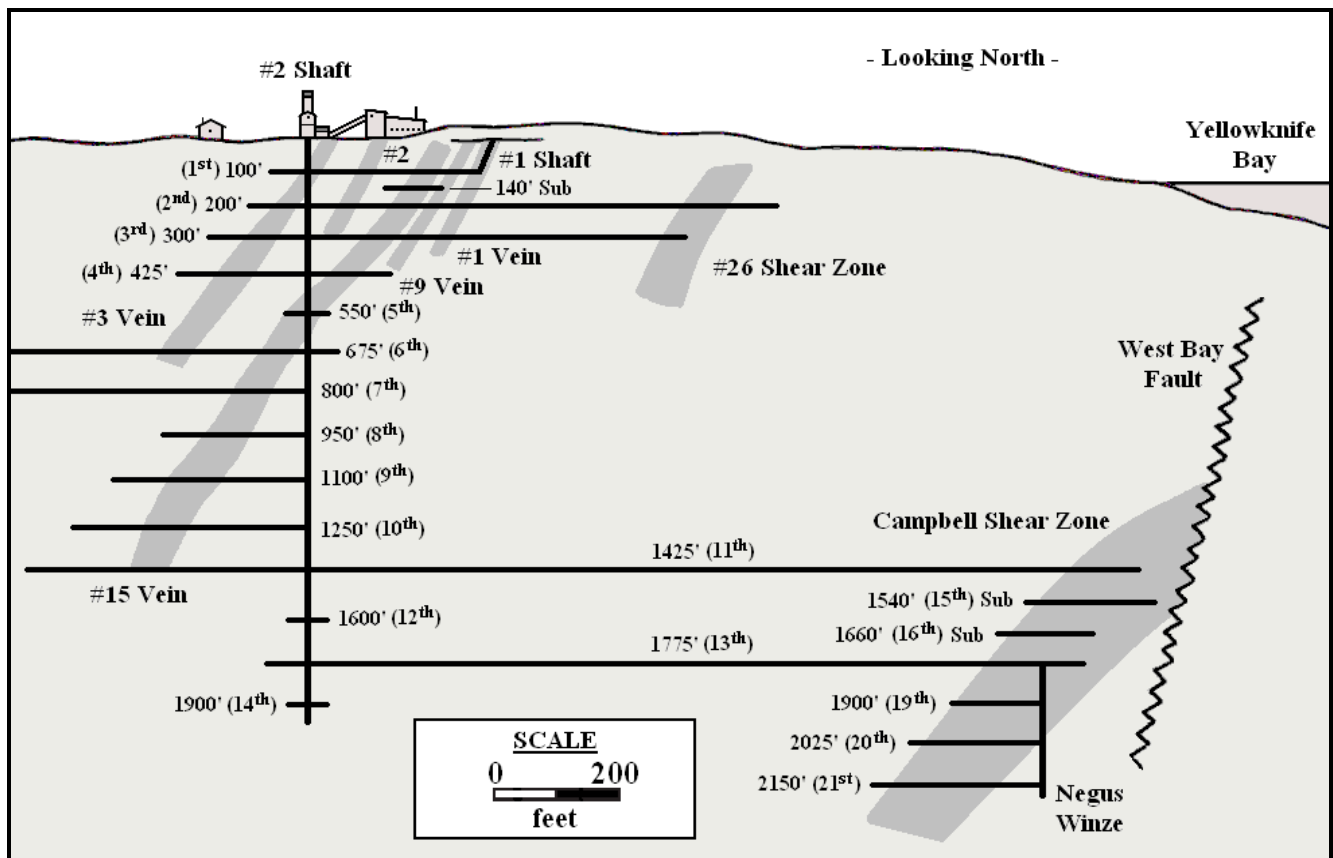


Figure 4. Negus Mine underground longitudinal plan, c.1952.

Mining Equipment

By 1952, the Negus Mine owned a small fleet of underground mining equipment, including twenty-four 40 cubic foot V-dump mine cars, four end-dump cars of 20 cubic feet capacity, two Mancha "Little Trammer" locomotives, and two Eimco 12-B mucking machines. A large amount of Copco and Ingersoll-Rand rock and stoper drills were in service. Oil storage for diesel engines and boilers was supplied by seven 35,000 gallon storage tanks located at the mine and camp (McDonald, 1955).

Yellorex Exploration

In 1950, a diamond drill hole on the 13th level near the southern boundary of the Negus property and south of the Negus fault intersected a wide quartz vein with high assays. It was believed this zone was related to the adjacent Yellorex deposits, and unrelated to the Campbell shear zone. Tunnels were driven south from the 13th level to explore this area in the summer of 1951. Unable to allocate crews for the continuation of the project, no further work was done in the Yellorex zones after 1951. Work to that time revealed the presence of strong shearing with some veining, but no ore grade material. This long south drive was to be used as a diamond drill base to explore the stretch of virgin ground that was theorized to contain continuations of the Campbell shear zone as it continued south towards Yellorex (The Northern Miner, Feb. 22nd 1951; July 26th 1951 Apr. 3rd 1952).

The total mine payroll averaged 150 employees during 1951 under mine manager Jock G. McNiven. Other staff included E. C. Rudd, assistant manager; C.J. Coulson, geologist; James McLellan, mill superintendent; Lloyd Robuck, mine engineer; James P. Dunn, master mechanic; and Carl Quitcott, mine captain (*The Northern Miner*, July 26th 1951).



Don Hurd

Figure 5. *Heading towards Negus Mine in 1952.*

Negus Roaster

Installation of the 2-stage Dorrco 'Flou-Solids' roaster began in November of 1951, and the plant started up on February 20th 1952. It was not until March 1952 that normal operations were realized. There was an estimated 11,000 tons of concentrate available for treatment, containing 14,000 ounces of gold. The roaster removed arsenic from the flotation concentrates first at low temperature, followed by removal of sulphur at high temperature. The plant had a capacity of 30 tons per day, and total capacity of the entire Negus operation was now 250 tons per day. The amount of gold in stockpiled concentrates was reported to be enough to pay for the purchase and installation of the plant (*The Northern Miner*, Apr. 3rd 1952).

Unexpected Closure

While the future of Negus seemed to be guaranteed, problems arose with the new roasting plant. Poor operation of the plant and lower grade ores from the 20th and 21st levels resulted in gold recoveries of 60% against a projected 85%. The decision to temporarily stop production was made during the fall of 1952, and the last gold brick was poured in September after having laid off almost the entire crew. Originally the shutdown was planned as a temporary measure in order to conserve the cost of winter operations. But in March 1953, it was announced that the mining claims would be sold to Cominco Limited, owners of the adjacent Con Mine. This included all underground workings, hoisting plant, and some corresponding surface buildings and machinery. The chances of reopening Negus Mine were gone (*The Northern Miner*, Oct. 9th 1952; Mar. 19th 1953; Apr. 2nd 1953).

Why Negus Closed

Negus Mine had a good chance as they had just started to tap the potential of the Campbell shear zone. Its westerly dip was still located within the 'Negus' claim group, and it was open on either end. The new zones near the Yellorex boundary also had potential. But capital costs associated with opening up this ore would not have made continued operations at the mine profitable. Increasing operational costs were seen during these years. Development during 1947-1950 had cost the company a large fortune, with the only hope of salvation coming from the stockpiles of gold only recoverable with a roasting plant. When this roaster failed to function properly, the company could no longer settle its debts through mining operations and could only save face by selling off its assets.

Operation Summary

Total production at Negus Mine from 1939 to 1952, the end of official operations, amounted to 490,908 tons of ore milled and the recovery of 255,807 ounces of gold (see **Table 1**). Over 500 gold bricks were poured. The mine was

developed with a vertical shaft (#2 shaft) to a depth of 1,900 feet with 13 levels, and a 375 foot winze to open three additional levels to a depth of 2,150 feet, plus the development of two sub-levels (see **Figure 4**)

1953 Recovery Operations

Negus Mines Limited retained ownership of the milling and roasting facilities, and some of the property. An attempt was made to reprocess the flotation concentrates through the roaster during the summer of 1953. Some research was conducted, and the necessary additions to the plant made this an advisable project. An estimated \$330,000 worth of gold could be recovered, paying for the debt incurred by the company (The Northern Miner, Sept. 10th 1953). It is reported that between May and October 1953, about 1,000 tons of material was processed. Metallurgical difficulties of the oxidized concentrate and continued operating deficiencies of the roaster made for a terrible recovery operation. \$16,000 worth of gold (approximately 450 ounces) was recovered versus the anticipated CDN\$40,000 for the amount treated. The project ran at a cost of CDN\$18,000, so company directors decided to write-off the gold concentrates (The Northern Miner, Apr. 4th 1954). Although riddled with debt, Negus Mines Limited was able to reorganize and carry on in the mining industry with other Canadian projects. More debt was written-off by the sale of all remaining buildings, mill equipment, and power plant to Rayrock Mines Limited in 1954-1955 (McGlynn, 1971).

Year:	Ore Milled:	Gold Recovered:	Year:	Ore Milled:	Gold Recovered:
1939	18,996 tons	15,995 oz	1946	24,419 tons	15,772 oz
1940	21,580 tons	21,075 oz	1947	25,356 tons	17,118 oz
1941	22,310 tons	18,349 oz	1948	51,497 tons	23,303 oz
1942	25,458 tons	19,637 oz	1949	64,510 tons	23,762 oz
1943	22,333 tons	19,080 oz	1950	67,954 tons	20,033 oz
1944	18,869 tons	20,723 oz	1951	64,511 tons	20,821 oz
1945	10,039 tons	8,655 oz	1952	53,076 tons	11,484 oz

Table 1. Negus Mine gold production, 1939-1952. (source: Lord, 1951; The Northern Miner newspaper)

Exploration Since Mine Closure

Exploration within the area of the old workings has been conducted by Con Mine since 1953. No detailed information is available. Some 7,300 tons of material grading 0.30 ounces per ton gold was trammed from the Negus workings to Con Mine in 1957. This was ore that had been broken during the mine's previous life and was easily recoverable (Con Mine records).

Rayrock Mines Limited owned the surface rights around the campsite and milling area after 1954. As a result, they became the owners of the old flotation concentrates that had been produced prior to the construction of the roaster in 1952. Considerable gold content remained in this concentrate pile, and Rayrock was interested in recovering this material. In 1959, John H. Parker estimated that there was 11,490 tons of material available for processing (later viewed as an inflated estimate). Twenty tons of concentrate grading 8.93 ounces per ton gold was shipped to the Discovery Mine for treatment during that year, to produce 165 ounces of gold. This was viewed as an excellent recovery program. However, it was not until 1970 that a larger sample was treated. Discovery Mines Limited, who now owned the concentrate pile, arranged for the treatment of 1,721 tons at the Con Mine roasting plant. The concentrate assayed 1.24 ounces per ton gold and the operation, which ran from July to November 1970, produced 1,599 ounces of gold. In 1975, new calculations indicated that only 1,000 tons of material remained. Tundra Gold Mines Limited, the owners of the Negus surface rights at the time, approached Giant Mine regarding the treatment of the old concentrates. After metallurgical testing, Giant concluded that the material was too oxidized and that treatment in their circuit would impact their own gold recovery. The surface stockpile of concentrates was eventually removed, but its fate is unknown.¹

¹ All information from files within the collection of the N.W.T. Mining Heritage Society, originally from corporate records of Tundra Gold Mines Limited found at Giant Mine.

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The News of the North newspaper articles, 1945-1953.



NICHOLAS LAKE

Advanced Exploration (Non-Operational)

Gold

63° 15' 00" 113° 45' 45"

Years of Primary Development: 1994

Mine Development: 2,030' decline to 270' level, 2 levels (680' dev.)

Introduction

The Nicholas Lake property is located at Nicholas Lake, 93 kilometers north of Yellowknife NWT. The claims were developed by underground workings in 1994 in anticipation of an underground bulk sample program. The site was visited in August 1999 by the author. Tyhee Development Corporation Limited, the current owner, is investigating opening the mine for gold production, but these plans are dependent on the success of their Ormsby Mine project.

History in Brief

This area was originally staked as the 'Berch' claims in 1941 by prospectors working for Cominco Limited, but aside from some minor trenching and diamond drilling, no major work was performed and the property was allowed to change hands many times over the next 40 years. In 1986, Dave Webb staked the ground as the 'NIC' group of claims. Chevron Minerals Limited began feasibility studies in 1989 to place the gold deposit into production. Athabasca Gold Resources Limited bought out Chevron Mineral's interest in 1991. Production planning was put on hold until 1993 when a contract for underground development was achieved through funds raised by Royal Oak Mines Incorporated. A decline was sunk to 90 meters depth in 1994. Royal Oak purchased the property from Athabasca Gold Resources Limited in 1995. Studies on the economic viability of putting the mine into production were undertaken, but the bankruptcy of Royal Oak in 1999 and the ensuing court case prevented any further assessment work being done until 2001, when Tyhee Development Corporation Limited gained control of the claims.

Geology and Ore Deposits¹

The Nicholas Lake gold deposit is located near the northern end of the Yellowknife Basin, a supracrustal belt within the Archean Slave structural province. Turbidite metasediments of the Burwash Formation, part of the Yellowknife Supergroup, have been intruded by pink granite and a medium to coarse-grained granodiorite, which is the host for mineralization. Numerous dykes and irregular masses of granodiorite are present in an area extending southwest of the main granodiorite intrusion. The metasediments have been deformed into tight folds with bedding now generally striking northwesterly with a steep dip. A strong, parallel to subparallel axial plane foliation is superimposed on the stratified rocks. Regional metamorphic grade is lower amphibolite facies. The Nicholas Lake deposit comprises a series of essentially vertical quartz-sulphide veins in a subvertical to vertical, altered granodiorite stock about 200 meters in diameter intruding metasediments. Intense shearing, sericitization and silicification are common, particularly along the southern contact. At least three varieties of quartz have been recognized, and multiple episodes of brecciation and rehealing are evident. The best veins and gold values are concentrated along the southern altered border zone and are restricted to the granodiorite. Higher grade gold values are associated with massive, fine-grained pyrrhotite coating angular fragments of arsenopyrite and quartz; massive arsenopyrite and pyrite with lesser sphalerite and galena; vein quartz with visible gold, coarse-grained sphalerite and galena; and quartz veins with pyrite. Some gold is present in association with disseminated sphalerite and minor quartz veinlets within zones of brecciated and fractured granodiorite. Sulphides may also occur as massive lenses up to 65 centimeters wide.

Athabasca Gold Resources Limited (1994)

In early 1994, tender contracts for the underground development of the Nicholas Lake project were given out to Canadian development companies. Procon Mining and Tunneling Limited of Burnaby, B.C. was awarded the contract for the 1994 program. Diamond drilling during the late 1980s indicated a resource to a depth of 1,000 feet. A drill indicated resource of 556,992 tons was reported in 1993. The primary objective of the 1994 work was to extract a small bulk sample of the underground ore and provide access for future bulk sampling operations. If all went well with the program, a production decision could be made by 1995 and hauling ore to Yellowknife over winter road could begin in 1996. Negotiations with Royal Oak Mines Incorporated to custom mill Nicholas Lake ore at the Giant Mine were underway.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

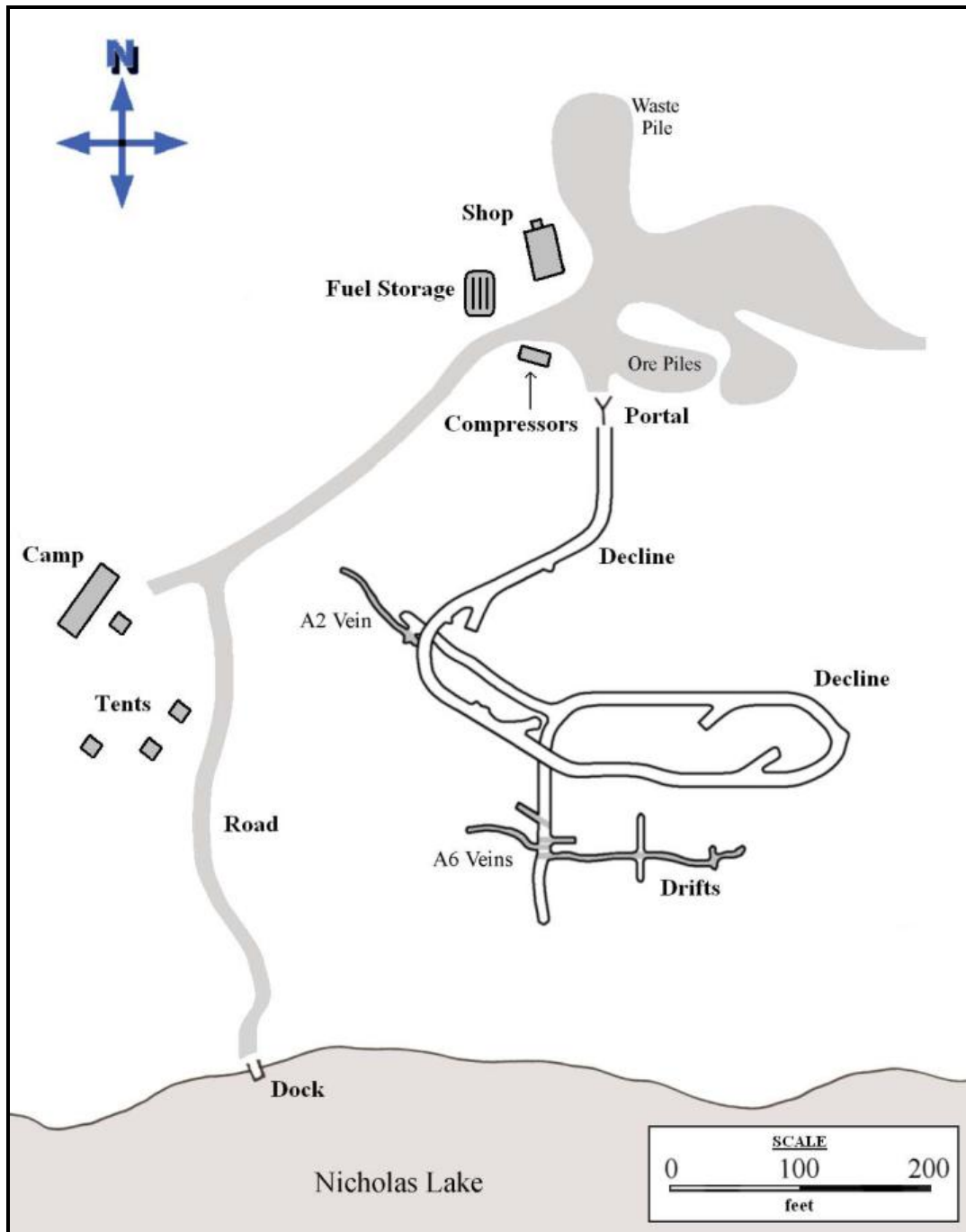


Figure 1. *Nicholas Lake surface and underground plan, 1994.*

A winter access route was cleared west from Gordon Lake, branching off the Echo Bay Lupin ice road for a distance of 62 kilometers. This route was more feasible for the project, since less road clearing would be required between Gordon and Nicholas Lakes. If the project went into production however, it would be more economic to clear the old Discovery ice road through the Yellowknife River valley to reduce ore trucking distances between the Giant mill and Nicholas Lake. Mobilization of supplies and equipment to the proposed decline site was completed in March 1994 by Procon contractors. Underground work began in April and until June.

The decline was driven at a grade of -15% in dimensions of approximately 3-6 meters x 3-6 meters to a depth of 90 meters below the portal. It provides access to the A6 veins at a depth of 90 meters and the A2 vein. About 600 meters of decline ramp were excavated, plus 220 meters of drifting and crosscutting. Two short crosscuts branch off the A6-4 vein east drift (Zerb, 1995; Kermeen et al., 1995).

Mining Equipment

A complete portable plant was setup by Procon for the 1994 project. Mining equipment included three scooptrams, two 15 ton underground haul trucks, a Tamrock 550 drill Jumbo, and a Cat D-6C bulldozer. Compressed air was supplied by two or three Gardner-Denver 800 cubic feet per minute diesel compressors and power was generated by a 225 kilowatt Cat 3306 generator plus some smaller units. The campsite, which consisted of a large trailer complex and several small heated tents, had its own 60 kilowatt generator. On average, the operation saw 20 contract miners at work, plus two staff geologists (N.W.T. Water Board Files - Water License N1L3-1574).

Sampling of the A2 veins was most encouraging and indicated strong mineralization that extended at depth. The A6-4 vein sampling test was disappointing. It was realized that the drift had been driven into a weak section of the vein, and that the higher-grade ore shoot was located below the existing level. An underground drilling program from August to October 1994 totaled 2,974 meters in 36 holes. At the end of the underground program, an ore reserve figure was published, consisting of 338,500 tons of proven and probable ore with a grade of 0.44 ounces per ton gold to a depth of 350 meters below surface. An inferred resource of 169,500 tons grading 0.28 ounces per ton gold was also calculated (Kermeen et al., 1995).

Exploration Since Mine Closure

With an interest to expand its Northwest Territories operations, Royal Oak Mines Incorporated, who had already financed the Nicholas Lake program into development, bought the property from Athabasca Gold Resources Limited in September 1995 for \$3.5 million. It was estimated that the production rate could be 77,000 tons of ore extracted each year for a recovery of 1,130 kilograms of gold per year (Royal Oak Mines Inc. Annual Report, 1995). The dwindling price of gold during the late 1990s and the subsequent bankruptcy of Royal Oak resulted in the abandonment of the project in 1999. Dave Webb, one of the original vendors, entered into legal action to acquire the property back from the bankrupt company. In January 2001, his company, Tyhee Development Corporation Ltd., acquired 100% interest in the Nicholas Lake property. In the summer of 2002, seven diamond drill holes totaling 1,821 meters were drilled as part of a large sampling program to increase knowledge of the resource. Most recent resource inventory suggests: measured ore, 57,000 tonnes grading 13.2 grams per tonne; indicated ore, 605,000 tonnes grading 9.8 grams per tonne; and inferred ore, 220,000 tonnes grading 10.1 grams per tonne. Total reserves in all categories are calculated as 882,000 tonnes grading 10.1 grams per tonne gold (DuPre and Kirkham, 2004).

Tyhee Development applied for the necessary permits to place a gold mine at Nicholas Lake and Ormsby/Discovery into production in 2007, and has now entered an environmental review process which could take two years or more to complete. A mill and camp will be constructed at Ormsby deposit and ore from Nicholas Lake will be trucked to the mill via winter road. An open pit mine is planned at Nicholas Lake.

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**NOREX***Satellite Producer (Abandoned)***Silver**

65° 35' 15" 117° 57' 30"

Years of Primary Development: 1973-1983*Mine Development:* Decline to 600' level, 5 levels*Years of Production:* 1970-1971, 1973, 1977-1983*Mine Production:* 53,697 tons milled = 2,066,744 oz Ag*Bulk Shipments:* 1970-1971*Bulk Shipments:* 55 tons shipped = 109,502 oz Ag

Introduction

The Norex Mine is located along the Camsell River in the Great Bear Lake region, 56 kilometers south of LaBine Point (Port Radium). It is 390 kilometers northwest of Yellowknife, NWT and operated as a satellite silver operation of Terra Mine between 1977-1983. The site has not been visited by the author of this report.

Brief History

The first silver discoveries on the Camsell River occurred during 1932 and a short shaft was sunk. The property was re-staked in 1965 as the 'Italdo' group by H. Wist and Associates, then sold in 1968 to Caesar Silver Mines Limited who undertook a small exploration program confirming the presence of native silver. The claims were subsequently optioned to Norex Resources Limited in 1969. A small mill was constructed in 1970 for the purpose of processing ore from an open cut. A decline ramp was started to gain access to deeper ore shoots.

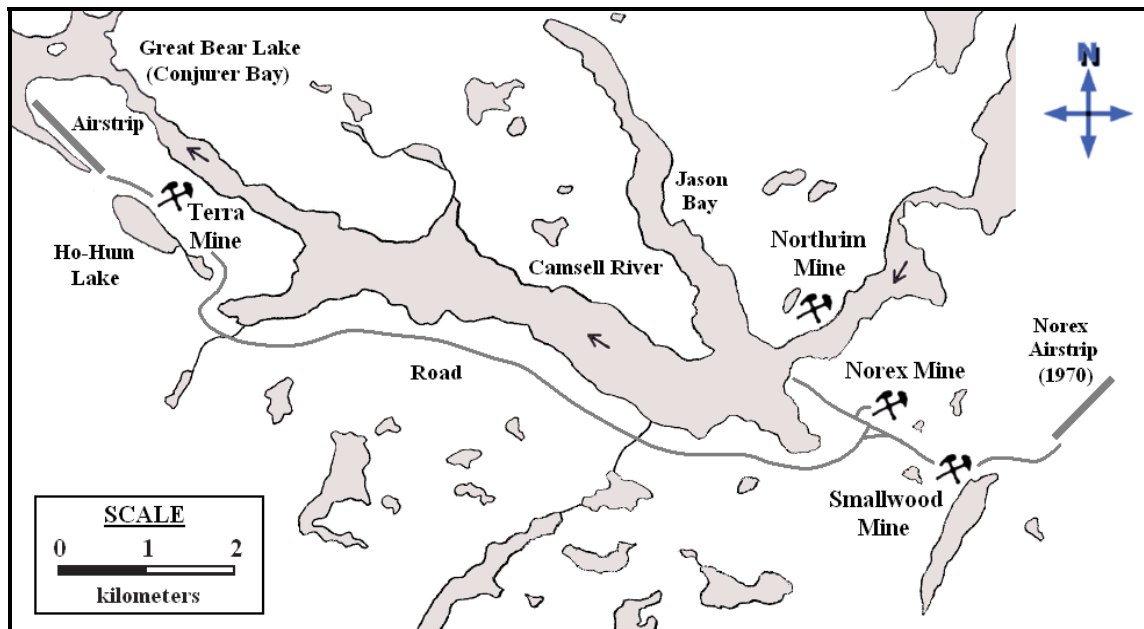


Figure 1. Camsell River area showing location of Norex Mine.

Norex Resources Limited and Terra Mining and Exploration Limited completed a joint-venture agreement for exploring the Norex property in 1971. Due to delayed resolution of certain legal issues, mining and production did not begin until 1977. Ore was milled at Terra Mine after being hauled by truck from the Norex portal. In 1981, Terra became sole operator by buying out Norex Resources Limited. Operations stopped in early 1984 after a depletion of economical ore.

Geology and Ore Deposits¹

The showing is underlain by Aphebian-age andesitic lavas and pyroclastics of the Echo Bay Group, commonly impregnated with metamorphogenic sulphides. Five parallel east-southeast trending veins occupy a 100 meter wide zone of chlorite-epidote alteration and are emplaced in tension fractures between major northeast trending faults. The

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

main mineralized vein, the Graham Vein, is about 200 meters long, dips vertically and consists mainly of quartz and carbonate. Ore lenses in dilatant segments of the vein contain native silver, native bismuth, hematite, and a variety of sulphides and silver and cobalt arsenides.

Norex Resources Limited (1970-1971)

Norex Resources Limited signed an option agreement with Caesar Silver Mines Limited in November 1969. During the winter of 1969-1970, a trailer camp was established on the property and mining equipment was assembled. Seven tons of high-grade ores were mined and shipped to Lumby, B.C. for bulk testing. Results averaged 503 ounces per ton silver. Open-pit excavations began in February 1970 on the Shaft zone where it was planned to mine the ore to a depth of 80 feet and along a strike length of 125 feet. Mining operations were carried out on three benches within the small pit using Atlas-Copco air-leg drills. Ore was mucked out of the pit using an air-operated slusher, and loaded into a truck with a John-Deere front-end loader (Ashton, 1970; Mines and Mineral Activities, 1970).

Records indicate that the open pit was 140 feet long, 12 feet wide, and 41 feet deep. Short adit tunnels were driven at two horizons within the wall of the pit, but stopped far short of any objective. A decline was also collared 1,500 feet northwest of the open pit and in the first 33 feet averaged about 14 ounces per ton silver over a decline width of 15 feet. A small milling plant was ordered and arrived during the spring of 1970. Ore reserves were calculated in April 1970 to be 14,220 tons at a grade of 149 ounces per ton silver (Ashton, 1970). An 80 foot x 2200 foot airstrip, capable of handling DC-3 aircraft, was cleared east of the minesite (northeast of Smallwood Lake) in 1970 (Ashton, 1971).

Bulk Sampling Operations

High-grade silver ore was hand-cobbed for shipment directly to a smelter. The remaining ore was passed through a 50 ton per day 'Impact' mill, consisting of a two-stage crushing circuit (12 inch x 18 inch jaw crusher and small cone crusher to reduce to ¾ inch size), an Impact mill, two jigs, and two Wilfley tables. The mill drum operated at a speed of 8 rates per minute in a clockwise motion, while the interior impellers operated counter clock-wise at a speed of 800 rates per minute. Ore feed was thrown by the impellers against interior anvils and disintegrated (Ashton, 1970; Mines and Mineral Activities, 1970).

Silver Production 1970-1971

Records indicate that 700 tons of ore were milled in the Norex test mill during 1970 to produce 25 tons of concentrate with a grade of 644 ounces per ton silver. Concentrates were shipped to the Cominco smelter at Trail, B.C. and a smelter in Montana, and gross silver content was reported as 16,099 ounces silver. The mill was again put to use in 1971 and 500 tons were milled to produce 16,501 ounces of silver. A large load of ore was also sent for custom milling at the Echo Bay Mine at Port Radium where 975 tons of ore was processed to yield 37 tons of concentrate, containing 57,249 ounces silver. Several bulk shipments of ore to Trail, B.C. and Montana during 1970-1971 totaled 55 tons of high-grade ore containing 109,502 ounces of silver (Ashton, 1971).

Terra Mining & Exploration Limited (1973-1982)

Sensing an opportunity to expand its Camsell River mining operations, Terra Mining and Exploration Limited entered into a deal with Norex for 50% interest in the property in October 1971. Work commitments and payments required by the agreement called for \$200,000 cash and \$300,000 to be spent on the property by October 1974. A fire in 1973 destroyed most of the equipment and buildings at Norex, further delaying underground development. Stockpiled ore from the open pit was hauled on winter road to the Terra Mine for processing in 1973. A total of 1,168 tons with a grade of 32 ounces per ton silver was milled at Terra, yielding 35,900 ounces of silver. A ten kilometer all-weather road was completed between Norex and Terra Mine in 1974, and 300 feet of decline advance was started. No work was done in 1975, pending settlement of a lawsuit against Terra by Norex Resources, concerning some details of the original agreement (Terra Mining and Exploration Ltd. Annual Reports, 1973-1975).

Development and Production Underway

A new deal was reached in January 1976 and work at Norex Mine continued. In 1976, the decline was driven a length of 1,160 feet and in April 1977 the Graham vein was intersected at 300 feet depth. Lateral development began immediately, and trucking of ores to the Terra mill commenced in September 1977. Very high-grade bismuth and cobalt-nickel arsenides were encountered on the lower levels, but only spotty silver was found at that time. An incline was started in 1977 to reach the 160-foot level, above the original workings. The #11 vein was also discovered 80 feet past the Graham vein on the 300-foot level, and a sulphide zone was encountered 40 feet beyond the #11 vein (Terra Mining and Exploration Ltd. Annual Report, 1977).

Surface drilling in 1978 also uncovered a new silver find southeast of the Norex Mine, which later became the Smallwood Mine, also a Terra Mines Limited property. By April 1978, the stopes on the 1st and 2nd levels of Norex were nearly exhausted and production stopped in May 1978. Mining was put on hold pending development of the lower levels. Considerable diamond drilling was undertaken from the 300-foot level to establish depth potential of the mine. Results were good, and in May-June 1979 the decline was advanced 800 feet to the 400-foot (3rd) level. 482 feet of lateral work was completed on this level by year-end 1979, and diamond drilling in 16 holes intersected some silver. **Table 1** lists ore reserves as of 1979 (Terra Mining and Exploration Ltd. Annual Reports, 1978-1979).

Ore Type:	Probable:	Possible:	Silver:	Cobalt:	Bismuth:	Nickel:
<i>Arsenide Ore:</i>	17,500 tons	42,500 tons	2 oz/ton	1.01%	1.01%	0.74%
<i>Silver Ore:</i>	5,000 tons	15,000 tons	80 oz/ton	-	-	-

Table 1. *Norex Mine ore reserves, 1979. (source: Terra Mining & Exploration Ltd. Annual Report, 1979)*

Most interest was in the bismuth and cobalt-rich intersections within the Harter vein. In June 1979, 214 tons of ore was mined and milled from the Harter vein in an effort to test cobalt and bismuth recoveries. The ore graded 1.99 ounces per ton silver, 0.21% copper, 1.01% bismuth, 1.01% cobalt, and 0.74% nickel. Metals produced from this ore were 426 ounces of silver, 811 pounds copper, 3,541 pounds bismuth, 3,297 pounds cobalt, and 2,715 pounds nickel. This test milling of ore was not included in the main production statistics for Norex Mine. Meanwhile, development ore from the mine was fed into the mill during August and September 1979 (mine records).

Development to year-end 1979 consisted of a long decline ramp to a depth of 400 feet to develop the Graham vein on three levels. The 1st level was developed by an incline driven off the main decline. The following statistics on underground development were reported at year-end 1979: decline ramps, 2,890 feet; incline ramp to 1st level, 727 feet; 1st level development, 1,714 feet; 2nd level development, 1,842 feet; 3rd level development, 482 feet; and total raising, 822 feet.

In 1980, Terra Mining and Exploration Limited purchased the 50% interest held by Norex Resources Limited and became the sole owner of the Norex Mine. Production was cut back during 1980-1982 in order to focus on development and exploration. In March 1981, surface diamond drilling was conducted north of the Graham vein, to test possible targets. The drilling together with extensive geological mapping and interpretation led to the discovery of four new vein structures by the end of 1981 (#3, #5, #6, #7 veins) all north of the Graham vein. Drilling was also conducted below the 400-foot level to test the Graham vein at depth. Lateral work on the 2nd level was conducted north of the primary workings in 1981-1982 to access the #6 and #3 veins (Terra Mining and Exploration Ltd. Annual Reports, 1980-1981).

Terra Mines Limited (1982-1983)

Terra Mining & Exploration Limited merged with Duke Mining Limited in 1982 and was renamed Terra Mines Limited. A 500 foot incline was driven from the 2nd level during May-July 1982 to access the upper reaches of the #3 vein, and a decline was driven below the 2nd level to access the lower reaches of the #3 vein. The #3 vein at Norex Mine was brought into production in October 1982. Early indications suggested that the structure was similar to the high grade Graham vein. By this time, the Graham vein was almost mined out except for the crown pillar, which was considered uneconomic to recover at 1982 prices (Terra Mines Limited Ltd. Annual Report, 1982).

Power Plant

Operations at Norex were supplied by portable diesel generators and air compressors. In 1982, a Cat 379 generator was installed at the Norex portal to service both Norex and the Smallwood Lake Mine. Air was supplied by portable Atlas-Copco and Gardner-Denver compressors (mine records).

Mining Methods

All operations of Terra Mines Limited in the Camsell River area were a combination of trackless and conventional mining, with ore mined by shrinkage stoping. Broken material was extracted through drawpoints below the stopes using Wagner scooptrams to fill 20 ton Wagner low-profile ore trucks. The ore was trucked to surface and stockpiled, then loaded into Kenworth haul trucks and brought to the Terra Mine mill, a distance of 10 kilometers by road (mine records).

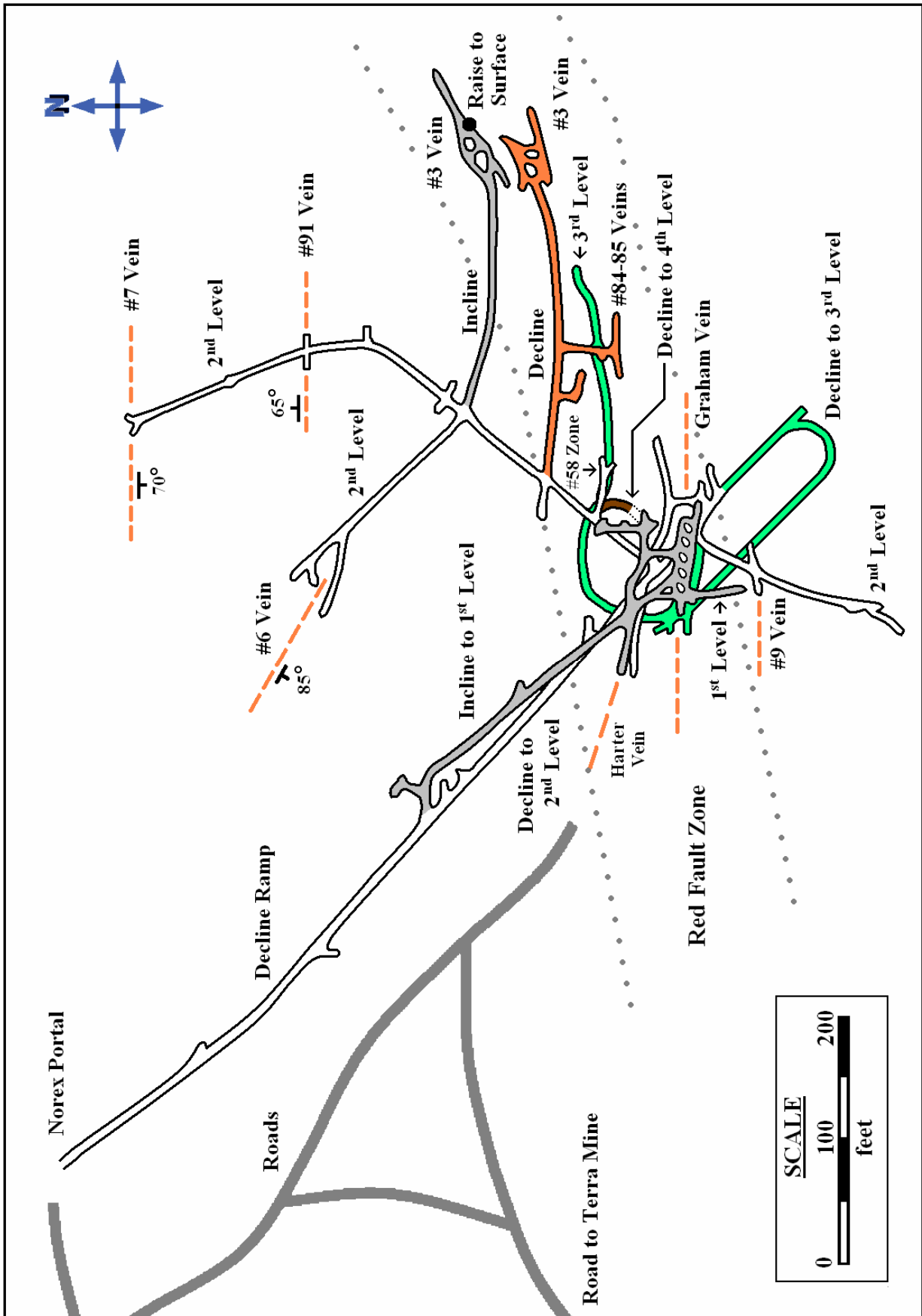


Figure 2. Norex Mine underground composite plan, 1983.

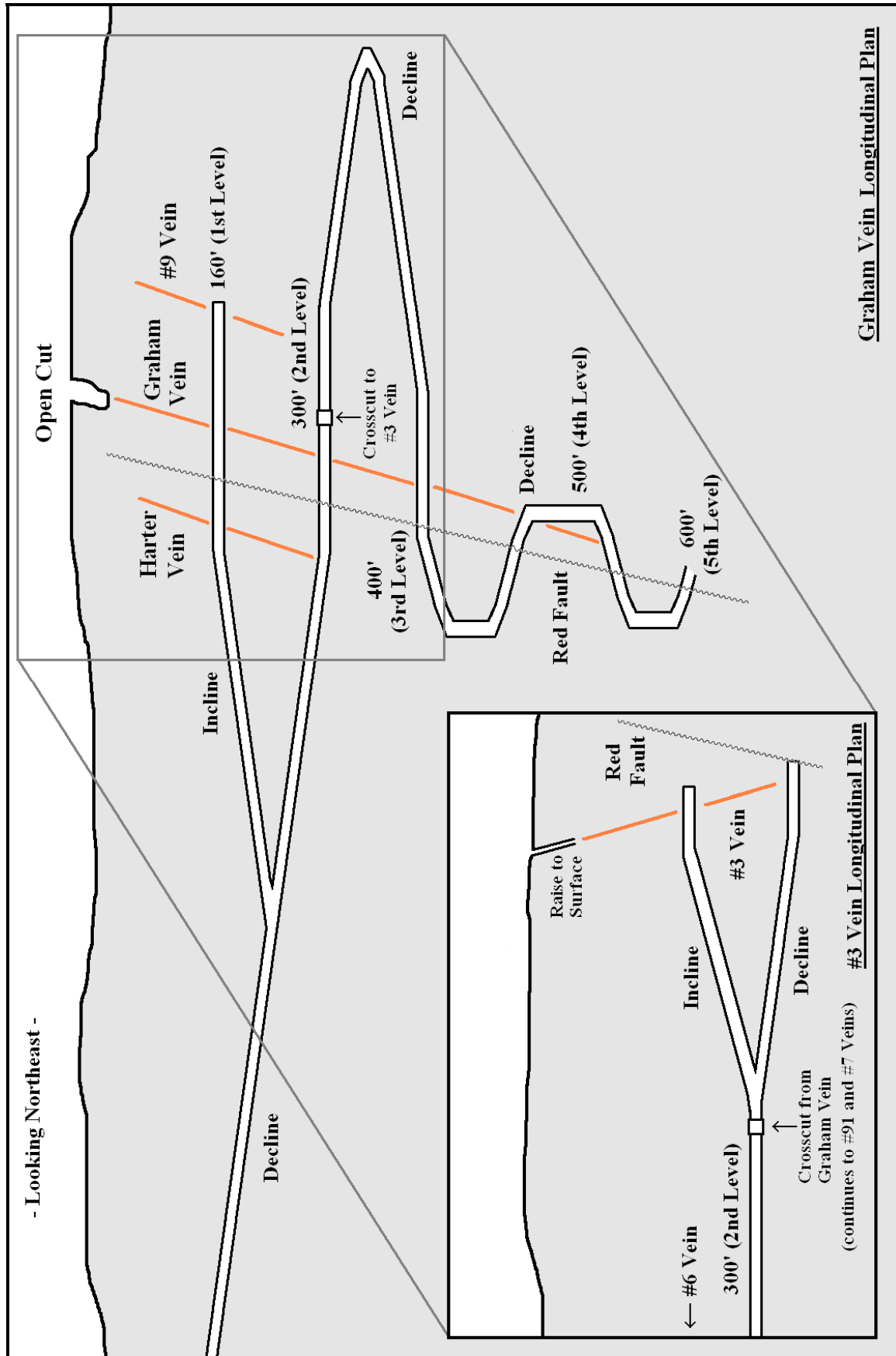


Figure 3. Norex Mine section plan 1983, cut across the Graham Vein and #3 Vein (inset).

Decline:	Incline to 1 st Level:	Incline to #3 Vein:	1 st Level: (160')	2 nd Level: (300')	3 rd Level: (400')	4 th Level: (500')	5 th Level: (600')	Raising:
3,787'	727'	500'	2,069'	3,886'	1,862'	559'	0	2,050'

Table 2. Norex Mine development summary, to December 31st 1983. (source: mine records)

Milling Operations

Please see the mill circuit description for Terra Mine. No unusual problems were encountered with the processing of Norex ores in the Terra mill. Bismuth and cobalt were reported as easily recoverable. No attempt was made to separate and obtain payment for values of bismuth, cobalt, lead, and zinc in the Norex ores, and as such production of these metals was never reported. More research was required to establish if the minerals had any economic value.

The decline was advanced 890 feet to the 500- and 600-foot (5th) levels during 1982-1983. The #84 vein was discovered during the year, and it was accessed from the #3 vein decline. In August 1982, a single deep hole from the surface intersected nine feet of vein averaging 139 ounces per ton silver at 900 feet depth. Subsequent follow-up holes delineated the structure, but failed to locate more silver. An extension of the main decline was being driven late in the year to provide drilling platforms for exploring this area. A 400 foot drift was driven off the 2nd level to access the new #59 vein (also known as the #7 vein) at 300 feet depth, discovered by surface drilling in March 1983. Production during 1983 at Norex was derived from the Graham vein crown pillar, some remnant open pit material, three stopes on the 2nd level (#3 and #84 veins), and two individual stopes on the 3rd and 4th levels (McCormack, 1983; Morris, 1984; Terra Mines Ltd. Annual Report, 1983).

Year:	Ore Milled:	Silver:	Copper:	Notes:
1970	700 tons	16,099 oz	-	Norex Test Mill
1971	500 tons	16,501 oz	-	Norex Test Mill
1971	975 tons	57,249 oz	-	Milled at Echo Bay Mine
1973	1,168 tons	35,900 oz	1,744 lbs	Milled at Terra Mine
1977	7,909 tons	446,503 oz	8,899 lbs	Milled at Terra Mine
1978	7,837 tons	770,946 oz	44,611 lbs	Milled at Terra Mine
1979	2,149 tons	56,002 oz	?	Milled at Terra Mine
1980	520 tons	2,944 oz	?	Milled at Terra Mine
1981	-	-	-	No production
1982	4,088 tons	106,379 oz	?	Milled at Terra Mine
1983	27,851 tons	558,221 oz	30,942 lbs	Milled at Terra Mine
Total:	53,697 tons	2,066,744 oz	Total ?	

Table 3. Norex Mine production, 1970-1971, 1973, 1977-1983. Includes production from the Norex test mill operation in 1970-1971, plus the ores shipped to Echo Bay Mine in 1971. All subsequent production was milled at Terra Mine.

Does not include 55 tons of bulk shipments in 1970-1971, which yielded 109,502 ounces of silver. Also does not include a 214-ton test milling of ore from the Harter vein in 1979, which produced 426 oz of silver and values of copper, bismuth, cobalt, and nickel. (sources: mine records; Terra Mines Ltd. Annual Reports; Ashton, 1971; Lorimer, 1973; Morris, 1984)

As a result of falling silver prices, Terra Mines Limited put all development and production at the Norex and Smallwood Lake Mines on hold early in 1984. So far as is known, no ore from the Norex Mine was milled during the year with the last available information suggesting that by year-end 1983 all production was derived from Terra Mine ores only. The underground workings were allowed to flood in 1984 (Terra Mines Ltd. Annual Report, 1984).

Development Summary

Early work in 1970-1971 consisted of a deep open pit on the Graham vein, from the bottom of which a small adit tunnel or drift was driven. Decline development began in 1976 and five levels were established at 160-, 300-, 400-, 500- and 600-foot depths. No lateral work was done on the 600-foot level. Ore was derived from the Graham and #3 veins. **Table 2** lists development statistics to year-end 1983 and **Table 3** lists total mine production.

Exploration Since Mine Closure

A surface air-track drilling program was performed in the Graham vein area during the summer of 1985 by Terra Mines Limited. 4,800 feet of drilling in 50 holes was performed, but only two holes produced economic assays. The drilling intersected three structures, which were believed to be sub-parallel splays of the main veins (Hitchins, 1985). Ore reserves in 1985 suggested a possible 545,970 ounces of silver above the 300-foot level within 23,280 tons of ore within the Graham, 84, and 58-veins. Deep drilling in the area of the Graham vein and the Red Fault suggested an additional 501,350 ounces within 7,330 tons of ore (Henneberry, 1985). Another source dated August 1985 reported Norex ore reserves as 2,100 tons grading 102 ounces per ton silver, but the methods used to calculate this figure are unknown. (Nicholas, 1985) Terra Mines Limited sold the mine to Octan Resources Incorporated in 1988.

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NORTH INCA

Advanced Exploration (Remediated)

Gold

64° 15' 00" 115° 13' 15"

Years of Primary Development: 1947-1949

Mine Development: 320' vertical shaft, 2 levels (2,186' dev.)

Introduction

North Inca was an underground gold prospect that operated for three years in the late 1940s. It is located at Indin Lake, 206 kilometers north of Yellowknife, NWT. The old site was visited by the author in September 2000 and June 2007. The ruins of the site were cleaned up by the Federal government in the spring of 2009.

History in Brief

The Indin Lake region was formerly known as the Wray Lake region in the late 1930s when gold was first discovered at the North Inca property. The A-1 and 2 (Brown) veins and the Johnson veins were trenched, drilled, and mapped during World War II but the claims later lapsed. In 1944, the property was re-staked as the 'Tartan' claims. North Inca Gold Mines Limited was formed to develop the large group of claims, which occupied a portion of Indin Lake that expanded from Inca Peninsula towards the southern shore. Work outlined two ore zones, and in 1947 shaft work and site preparation began. Shaft sinking got underway in 1948 and by the end of operations in 1949, two levels each explored two deposits of gold ore. No production decision was made and the mine was closed.



Mike Piro

Figure 1. *North Inca Mine, 1980s.*

Geology and Ore Deposits¹

The Indin Lake area is situated in the southwest portion of the Slave Structural Province and is located within the Indin Lake Supracrustal belt of Archean supracrustal metasedimentary and metavolcanic rocks of the Yellowknife Supergroup. The rock types and metamorphic zones form a series of broad, north-northeast trending belts that extend south from the south side of Truce Lake to the southern portion of the Snare River. North Inca Mine's A-zone and Main-zone deposits are located on the Inca Peninsula and are part of a series of gold showings that occur near the north-northeast trending contact between the metavolcanics and the metasediments. The A-zone contains several veins: A-1 (east Brown vein), A-2 (west Brown vein), and A-3. The A-zone has been traced by diamond drilling for a length of about 123 meters and to a depth of 73 meters. It dips easterly at angles varying from 75° to vertical, striking approximately northward. The A-zone and Main-zone are truncated by the sinistral Inca fault 30 meters south of the peninsula, below Indin Lake. The A-zone extension is displaced along the Inca Fault about 450 meters to the southeast of the A-zone veins. The veins walls are sharp and lined by rusty sericite-carbonate (pyrite-pyrrhotite) schist up to 24 inches thick.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

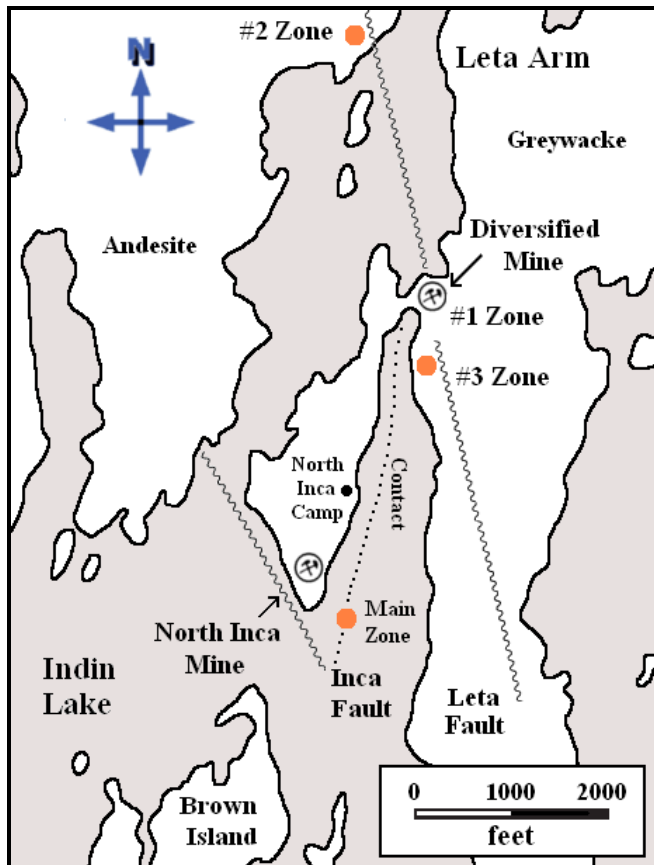


Figure 2. Location of North Inca Mine.

North Inca Gold Mines Limited (1947-1949)

Over 21,000 feet of diamond drilling during 1946 indicated two potential ore zones in the vicinity of the Inca Peninsula: the A-zone and Main-zone. The Main-zone was previously unexplored as it does not outcrop and is in fact completely underwater to the east of the Inca Peninsula. It was discovered through the belief that the Diversified orebody to the north of the property extended south into the claims held by North Inca. Through drilling, the Main zone was found to be located only 500 feet from the shore of the Peninsula (Lord, 1951).

It was decided most logical to position the shaft on the Inca Peninsula where immediate access to both deposits through 320 feet of shaft and 2,000 feet of lateral work on two levels would be made economic for conditions at the time. The shaft decision was made in late March of 1947 at which time it was difficult to transport freight to the property from Yellowknife, as break-up was imminent. Some 30 tons of equipment and supplies were purchased from local mining operations such as Colomac Mine where it was possible to move freight through the use of Cat trains. An ice field was cleared on the lake so that DC-3 aircraft could also deliver supplies, which amounted to 200 tons before break-up hit in April (Lord, 1951).

To aid in the delivery of shaft sinking equipment in time for a fall start-up, it was decided to build a summer airstrip at Indin Lake. Due principally to ignorance of local conditions, the work crews failed to drain a portion of the lake that would have served as a basin for the proposed airstrip. Work during the summer of 1947 was therefore confined to construction and shaft sinking preparations. By year-end, several buildings both at the camp and the shaft site had been built, a timber headframe was erected, and a production shaft was collared to a depth of 41 feet (North Inca Gold Mines Ltd. Annual Report, 1947).

North Inca Mine reopened in February 1948 and airfreighting continued. The remainder of the mine plant and equipment arrived and was installed. During this time, the crews put down five diamond drill holes on the ice to probe the Main-zone (North Inca Gold Mines Ltd. Annual Report, 1947). The company did not authorize shaft sinking until May 1948, and due to financial restraints, development for the time being was to be performed on one level only. Station cutting for a 1st level at 175 feet depth was completed June 21st 1948 (Gilchrist, 1949.) Owing to

The quartz is fractured and mottled dark grey to white, cut by milky white vuggy quartz seams up to one inch wide, and masses that contain an iron-carbonate mineral. White and grey quartz are found in equal abundances. Some seams of schist parallel to vein walls are found in the veins. Metallic minerals (pyrite, arsenopyrite, gold, and very little galena, pyrite, and pyrrhotite) make up less than 1% of the veins and visible gold is abundant in grey quartz.

The Main-zone is composed of a stockwork of quartz veinlets and a few orebodies within a sheared graphitic zone of greywacke-argillite-phyllite-slate metasediments, along the contact with the metavolcanic unit. The Main-zone actually continues into the Diversified Mine property to the north, sinistrally offset by northwest trending faults. The zone is from two to 15 meters wide with well defined borders. Within the zone, veins are hosted by black graphitic schist; with slate and sheared argillaceous greywacke containing pyrite, arsenopyrite, and a little gold where arsenopyrite is especially abundant. Probably less than 1% arsenopyrite, pyrite, pyrrhotite, chalcopyrite, and visible gold are present in the quartz seams. Ore was also found in sections with abundant sheared rock in addition to quartz.

the depth of the lake, it was not possible to explore the Main-zone from this level. The 1st level therefore was driven westward to develop the A-zone deposit. Drifting on the A-1 vein indicated a length of 111 feet with a vein width of 2.5 feet grading (un-cut) 0.62 ounces per ton. The best intersections came from the A-3 vein where a length of 63 feet and width of 2.5 feet gave grades of 1.14 ounces per ton. This development continued until October 18th 1948 when shaft sinking to 320 feet was resumed. During the summer of 1948, further construction was undertaken to complete the program for the proposed exploration program (Gilchrist, 1949).

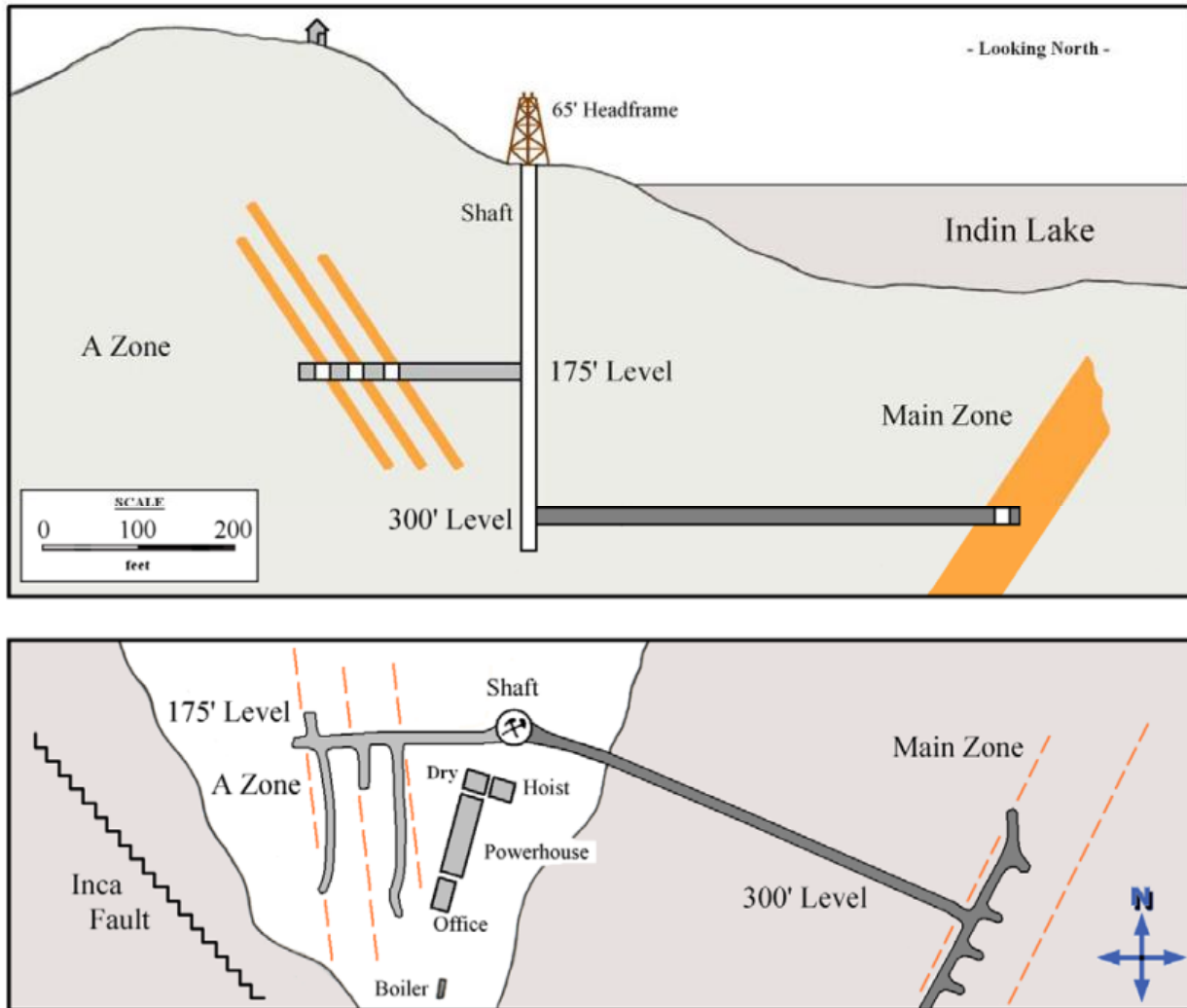


Figure 3. North Inca Mine surface and underground plan, 1949.

Camp and Plant Facilities

The camp facilities were built to accommodate a crew of up to 40 men. Facilities consisted of a 26 foot x 60 foot 2-storey bunkhouse with 18 double rooms, and a 24 foot x 36 foot cookery with a warehouse in the basement, and a large log cabin guesthouse. These buildings were built north of the shaft site, and were heated by individual electric steam generators. Plant facilities included a 20 foot x 75 foot Steelex prefab building for use as a combined powerhouse/shop/garage for D-6 Cat tractor, a 20 foot x 25 foot Steelex prefab hoist room, a frame miner's dry, an American army hut used as an office, and a 65 foot timber headframe. Other buildings consisted of an assay lab, warehouses, and a small log cabin built atop the nearby hill housing a 100 watt radio receiver and transmitter (Gilchrist, 1949).

Power Plant

The camp and plant was powered by a primary 31 KVA single-phase 50 horsepower Cummins diesel generator. Backup power was supplied by a three KVA Briggs & Stratton gasoline generator. The property diesel engines and heat plants consumed over 65,000 gallons of oil during development. Total capacity of the compressed air plant was

over 1,000 cubic feet per minute using three units. Two of these were Canadian Ingersoll-Rand air compressors driven by Waukesha diesel engines, and the third was a Gardner-Denver unit driven by a Cat D-13,000 diesel engine. In 1947, a large 100 horsepower Oilwell wood-fired boiler was partially installed at the shaft site (Lord, 1951). Going into 1948, it was decided to abandon this service and use individual steam heaters in buildings, another cost saving measure (Gilchrist, 1949).

Hoisting Plant

A small Canadian Ingersoll-Rand 10x12 air hoist was purchased used for this operation, a cost and time saving measure. The hoist was a single-drum unit, so in order to use it for counter-weight development operations, the drum was split with a divider to create a double-drum. A large 65 foot timber headframe was built in anticipation for future production operations (Lord, 1951).

Underground Development

Lateral work was performed using four Canadian Ingersoll-Rand DA-35 type drifter drills. Twenty hand-operated mine cars and an Eimco mucking machine were in use. All headings were cut in 6 feet x 7 feet size. Since development was being performed in agglomerate rock, it was believed that water problems would be minor. When developing the 2nd level at a depth of 320 feet in 1948, the precaution of installing a large two-stage 500-gallon per minute pump was made. A separate Cat D-13,000 diesel generator powered this unit. Water inflow was around five gallons per minute. The North Inca operation, on average, employed 26 men. During shaft sinking and construction periods, this number was increased up to 40 men. William Gilchrist was manager in charge of operations from April 1947 to August 1949 (Gilchrist, 1949).

Shaft sinking to 320 feet was completed in December 1948 and crosscutting east towards the Main-zone on the 300-foot level began December 24th. The crosscut reached the zone 500 feet from the shaft on February 11th 1949, and drifting along the zone was completed at the end of April. At this date, the initial plan of development had been completed (Gilchrist, 1949). Both the A-zone and the Main-zone were opened up underground where a clearer indication of the grade potential of the deposits could be measured. In 1948, a small sample of ore from the A-zone suggested that gravity recovery combined with flotation and cyanidation would recover close to 99% of the gold. Ore tested from the Main zone suggested a similar recovery.

Underground development on the Main-zone continued until July 1949. At this time, the A-zone had been explored with a 315 foot crosscut on the 175-foot level and three southern drifts on each vein (A-1, A-2, A-3) totaling 443 feet. The Main-zone was developed by a 497 foot crosscut on the 300-foot level and developed with a southerly drift 543 feet long, and 440 feet of several short crosscuts spaced at 50 foot intervals designed as future draw points. These crosscuts were sampled and averaged 0.23 ounces per ton across nine feet. A small amount of raising (35 feet) was done from the top of one of these draw-points. Additional diamond drilling was performed until final closure in August 1949. Total diamond drilling on surface was 5,788 feet and total drilling underground was 9,176 feet. Total underground work is 320 feet of shaft sinking, 1,200 feet of crosscutting, 985 feet of drifting, and 35 feet of raising (Gilchrist, 1949).

Costs

The total cost of operations from April 1947 to August 1949 amounted to CDN\$499,770, including \$129,683 on labor, \$79,112 on shaft sinking, \$131,416 on freighting expenses, \$108,999 on equipment and supply purchases, and \$38,134 on cookery expenses. In 1948, crosscutting work cost on average \$57 per foot advance over 315 feet progress. Drifting for the same year cost \$66 per foot for an advance of 443 feet. During the early winter of 1947, 200 tons of material was brought in by DC-3 aircraft at a cost of up to \$110 per ton. Air freight cost during 1948 for 241 tons was \$118 per ton. A 120 ton load of material arriving by Cat train in 1947 cost \$45 per ton. Total freight handled and delivered to North Inca from 1947 to 1949 amounted to over 800 tons (Gilchrist, 1949; Lord, 1951).

Ore Reserves

Surface diamond drilling of the A-zone indicated 23,000 tons of ore grading 0.54 ounces per ton gold across a mining width of 2.5 feet, contained in three ore shoots with an aggregate length of 630 feet (Lord, 1951). No development has been undertaken at the North Inca Mine since 1949; the North Inca Gold Mines company dropped the claims in 1955.

Exploration Since Mine Closure

The property was acquired in 1980 by Indin Gold Limited. A report by Kilborn Engineering in that year reported an ore reserve of 90,000 tons grading 0.20 ounces per ton gold within the Main zone. Some diamond drilling was performed

near the deposit in 1988 by Stockmen Minerals Limited (National Mineral Inventory). Globaltex Industries Limited (now Pine Valley Mining Corporation) acquired the property in the 1990s.

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NORTHLAND

Advanced Exploration (Remediated)

Uranium

63° 07' 30" 116° 19' 45"

Years of Primary Development: 1955-1956

Mine Development: 274' vertical shaft, 2 levels (2,618' dev.)

Introduction

This former uranium prospect is located 125 kilometers northwest of Yellowknife, NWT at Chico Lake, which is seven kilometers north of Marian Lake. It was to become a satellite uranium operation to the nearby Rayrock Mine (38 kilometers north), but the mine closed due to a fire in 1956 and did not reach production. A forest fire burned down remaining buildings in the 1970s and the site was remediated in the 1980s.

An old all-weather road connects the site to the north end of Marian Lake, and is now used as a winter road to connect to Snare River Hydro and the old Colomac Mine. The mine has not been visited by the author.

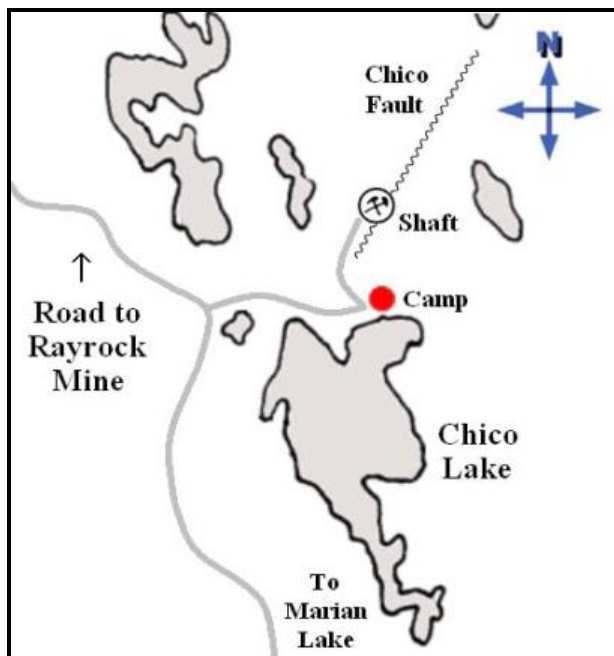


Figure 1. *Chico Lake region showing mine location.*

History in Brief

Interest in uranium deposits in the Northwest Territories took off during the 1950s, resulting in a minor staking rush in the Great Bear, Hottah Lake, and Marian River areas. The 'Sun' group of 16 claims were staked in August 1954 by Hubert and Bruce Giauque following the detection of a highly radioactive zone centered near Chico Lake.

Immediately, the Byrne group of companies became interested in the deposit and formed Consolidated Northland Mines Limited to acquire the claims in December 1954.

Diamond drilling was conducted during 1955, followed by shaft sinking on the zone later in the year. A contract to supply uranium concentrates was negotiated with the Federal Government early in 1956, but a fire that destroyed the mine plant delayed a production decision. All prospects faded following the cancellation of United States purchasing of uranium products from Canada in 1959.

Geology and Ore Deposits¹

The area is located within the Bear province and is underlain by Precambrian rocks consisting of andesite, granites and a quartz stockwork. The stockwork is the principal host for radioactive mineralization. The regional controlling structural feature is the Chico Fault which permitted invasion and replacement of the original rocks by the quartz stockwork. The stockwork, bounded on one side by the fault, is two to four hundred feet wide, finely fractured and mineralized by pitchblende in the form of veinlets, blebs and sheets. Veinlets are irregular and discontinuous. Fine fractures are usually surrounded by red, altered, brecciated zones mineralized with pitchblende forming patches or lenses of ore. In the andesite, on the southeast side of the stockwork, pitchblende occurrences are more regular and are usually accompanied by fine pyrite mineralization.

The Chico fault strikes at 015 degrees azimuth through the property. It splits into a subsidiary fault about two kilometres north of Chico Lake. Between the branches a quartz stockwork 550 meters long by 183 meters wide occurs in a quartz feldspar porphyry. The stockwork strikes at 035 degrees azimuth and dips 60° northwest.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

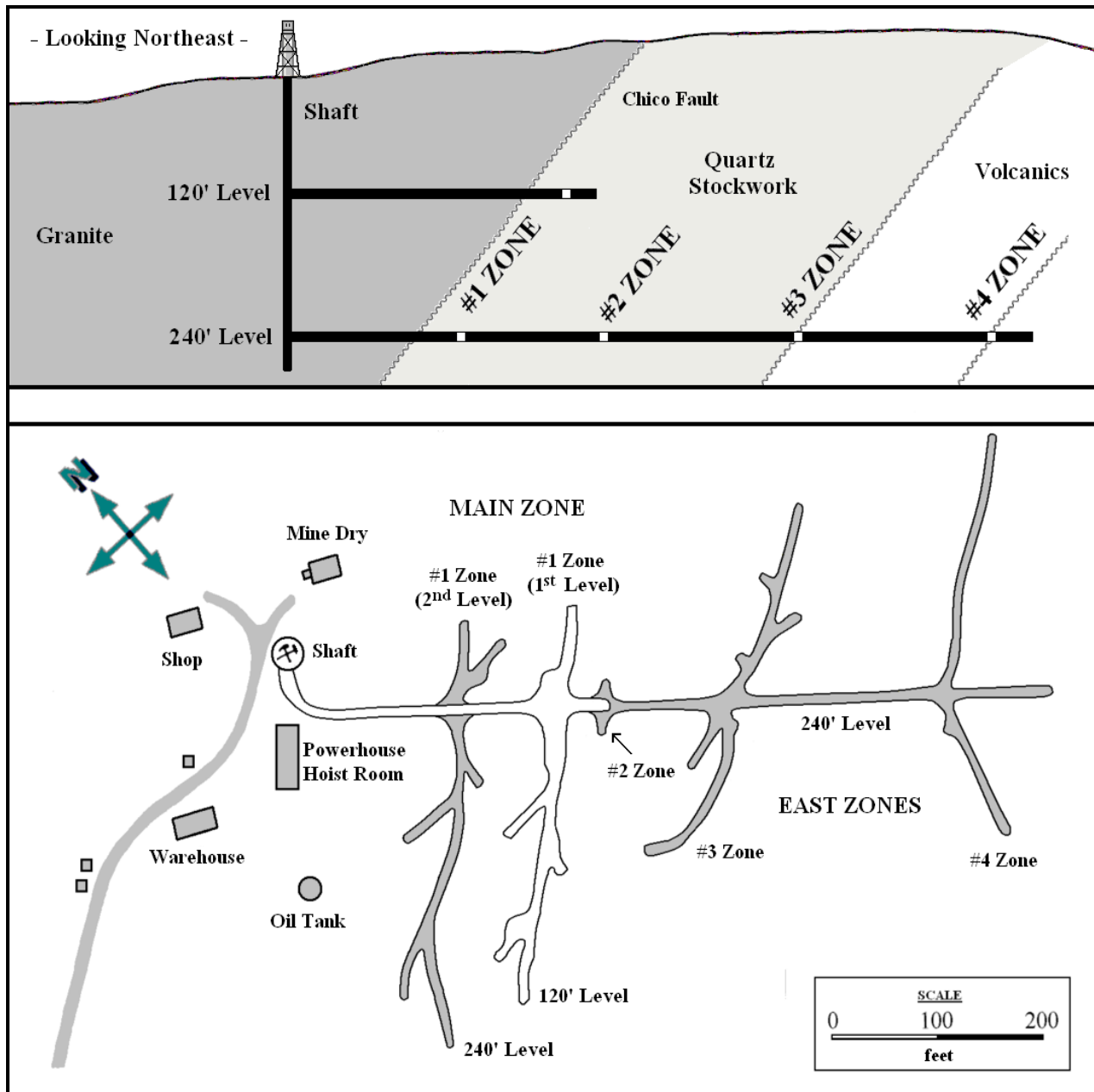


Figure 2. Northland Mine surface and underground plan, 1956.

Consolidated Northland Mines Limited (1955-1956)

A diamond drilling program was undertaken early in 1955 at Chico Lake. Thirty-eight holes were drilled to uncover good widths of pitchblende ore across the 240 foot wide quartz stockwork. In June 1955 this program ceased upon the decision to initiate an underground exploration program, with the hopes of attaining similar results found at the Rayrock Mine to the north. Work was concentrated on the #1 zone or Main zone (Byrne, 1956). Crew and plant equipment were assembled during July and August 1955. The shaft was collared and sinking of the first round began on September 6th 1955. Meanwhile construction of the camp and plant facilities was underway (Consolidated Northland Mines Ltd. Monthly Progress Reports).

Marian Lake Road

An all-weather road was cleared from the property to Marian Lake, where deep-water navigation from Great Slave Lake was possible. Transportation costs would be reduced significantly through the use of this route. As a result of the shared management between the Northland and Rayrock companies, money was easily pooled into the

development of this road that would service both mines. Some government assistance was sought to complete the project (Byrne, 1956).

A complete mining plant was installed at Northland for the sinking and development job in October 1955. This included an 8x6 two-drum air hoist (24 inch x 18 inch size), two Canadian Ingersoll-Rand compressors (diesel driven), and a small Cat D-135 and Onan light plant (Consolidated Northland Mines Ltd. Monthly Progress Reports).

The two-compartment shaft (6-½ feet x 10 feet) was completed to a depth of 274 feet on December 9th 1955 after the blasting of 63-½ rounds, which removed 2,150 tons of waste rock and used over 6,000 pounds of explosives. Crosscutting southeast of the shaft stations at 120 feet and 240 feet depths was started in December 1955, and the #1 zone was intersected on the 2nd level 110 feet east of the shaft by the end of the year (Consolidated Northland Mines Ltd. Monthly Progress Reports). This work opened a width of 26 feet averaging 0.35% uranium oxide (U₃O₈) per ton with a high-grade core averaging 0.66% across 12 feet in wall sampling (Byrne, 1956). Lateral tunnels were driven in 5 feet x 7 feet dimensions. Equipment used for mucking consisted of an Eimco 12-B mucking machine on each level (Consolidated Northland Mines Ltd. Monthly Progress Reports).

During shaft sinking in late 1955, the previous diamond drill results were further studied and it was concluded that the East zone be explored as soon as possible. Crosscutting on the 1st level reached the quartz stockwork of the #1 zone in January 1956, and in February it was decided to continue the 2nd level drive south towards the East-zone. In March, the 2nd level crosscut reached the East-zone and drifting here begun within the #3 and #4 zones. Most development of the 1st level stopped during the month as the 2nd level developments were proving to be more interesting. Also during this time, a loss of mine crews slowed underground work, but did not shut down the operation (The Northern Miner, Apr. 26th 1956).



The Western Miner magazine

Figure 3. Northland Mine, 1955.

Overall, good ore was being found on both levels. On the 1st level, a 100 foot long oreshoot gave an average grade of 0.207% U₃O₈ (or four pounds per ton) over an average width of 6-½ feet. On the 2nd level, a 128 foot oreshoot averaged 0.212% U₃O₈ (or four and a half pounds per ton) over an average width of seven feet. A bulk sample was prepared at the site and sent to Ottawa for analysis (Byrne, 1969). After a period of negotiations, a deal was signed (April 16th 1956) with the nearby Rayrock Mines Limited for the milling of 40 tons per day during the summer months, beginning in 1957. Because Eldorado Mining and Refining Limited, a Canadian Crown Corporation, was the purchaser of Rayrock concentrate, the deal had to be approved by them (Byrne, 1957).

Camp Site

The Chico Lake campsite consisted of two 20 foot x 34 foot bunkhouses, an office/staffhouse, and a cookery. In March 1956, a house was hauled to the site from the DeStaffany Mine for use by the mine manager and his wife. A permanent camp for 40 men was built on the north end of Chico Lake. In 1955 the operation was headed by Jay Murphy, resident engineer. He was later replaced by R.W. Johnston in 1956 (Consolidated Northland Mines Ltd. Monthly Progress Reports).

Lateral work on the 2nd level during the spring of 1956 gave very interesting results with the East zones, which consists of three zones (#2, #3, and #4 zones). Drifting was performed within the #3 zone during March-April 1956 and within the #4 zone during May-June 1956. Two rounds were blasted in either direction along the #2 zone, but no major lateral work was accomplished. A horizontal diamond drill into the #2 zone from this heading showed 36 inches of 0.56% U₃O₈ and 12 inches of 0.98%. No oreshoots were outlined in the East zones when work ceased (Parker, 1957).

1956 Hoist Fire

In May, an underground diamond drilling and raising program was begun. Good drill results were obtained 50 feet below the 2nd level in the Main-zone area. Drifting was continuing within the new ore zones on the 2nd level. Little

progress was made however when in July 1956 a fire destroyed the powerhouse building, which housed the hoist and compressor plants. This was a severe blow to the Northland operation. Logistics suggested that permanent plant replacements would not be available until the following summer of 1957, and that ore shipments to Rayrock wouldn't occur until 1958. The future of the Canadian uranium mining industry was also in doubt, as it was feared that the government would not continue the purchase of private stockpiles of the material (Byrne, 1969; Byrne, 1957).

In 1959, the United States government announced that it would cancel the purchasing of uranium products from Canada. Eldorado Mining and Refining Limited therefore did not agree to any further purchasing contracts beyond that point. Uranium exploration in the north was over and the Northland Mine was never placed into production.

Total Development Summary

The Northland deposit is reported to be under-developed, and the full potential is not examined. The company estimated that the ore opened to date averaged 0.25% U₃O₈. Four uranium-bearing zones were investigated by surface drilling and underground work. Two of these zones (Main and East) contained ore shoots that were probed by the underground development; the other zones required more work (The Northern Miner, May 9th 1957).

Development consists of a 274 foot shaft with two levels at 125- and 250-foot depths from the shaft collar. Drifting and crosscutting on the 1st level consisted of 869 feet of advance on the Main-zone, in which two ore shoots were opened up with a combined length of 100 feet, and 6 feet width. On the 2nd level, 1,749 feet of lateral work was completed on both the Main and East-zones, outlining two ore shoots with a combined length of 127 feet and average width of 7 feet. A third section in the Main-zone on the 2nd level was not fully investigated but some good uranium values were encountered. 132 feet of raising and 1,456 feet of underground diamond drilling is also reported (Parker, 1957; Byrne, 1957).

Exploration Since Mine Closure

Consolidated Northland Mines Limited was amalgamated with North Goldcrest Mines Limited in 1965 to form Crestland Mines Limited. Additional claims were staked ('Rose' group) and some surface prospecting, trenching, and sampling was conducted in 1969. Also in that year, a scintillometer survey was conducted over the Main and East zones to outline the extent of radioactivity. An area 235 feet x 64 feet with an estimated grade of three pounds uranium oxide per ton (1,250 tons per vertical foot) was calculated within the Main and East zones, above the underground workings. In 1976, Crestland Mines Limited merged with Radiore Uranium Mines Limited to form Pyx Explorations Limited. Further exploration was conducted in 1977-1978 under option to Rayrock Mines Limited, but diamond drilling failed to locate any new areas of mineralization. The tonnage potential was considered too small to justify production (National Mineral Inventory).

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NORTHTRIM

Minor Producer (Abandoned)

Silver

65° 35' 45" 117° 58' 45"

Years of Primary Development: 1933-1935, 1968, 1970-1971, 1975-1978 *Mine Development:* Decline to 340' level, 4 levels

Years of Production: 1971-1972, 1977-1978 *Mine Production:* ~12,000 tons milled = +30,000 oz Ag

Introduction

The Norex Mine is located along the Camsell River in the Great Bear Lake region, 54 kilometers south of LaBine Point (Port Radium). It is 392 kilometers northwest of Yellowknife, NWT. The author visited the site in August 2006.

Brief History

The first claims were staked west of White Eagle Falls on the Camsell River in 1932 by members of the A.X. Syndicate. These claims were called the 'Otter' group. White Eagle Silver Mines Limited was formed and in 1933-1935 underground development consisting of an adit and a winze was completed. In the late 1960s, Frank Lypka shipped a few tons of high-grade cobbled silver ore from the reactivated property. This work indicated that production would likely be feasible. Silver Bay Mines Limited acquired the property in 1968, driving a new adit to intersect the old workings. Work continued under the direction of Federated Mining Corporation Limited in 1970 and in 1971 production began. Operations stopped in 1972, but the mine was reactivated in 1975 under the operating name of Northrim Mines Limited. The mine suspended operations in 1978 and shortly thereafter the company went bankrupt.

Geology and Ore Deposits¹

The area lies within the Great Bear Magmatic Zone and is underlain by Apebian volcanics including hornblende meta-basalt, andesite, and tuffs intruded by a syenite-monzonite complex to the north. The volcanics contain varying amounts of pyrite and chalcopyrite. The #1 vein consists of quartz and quartz-carbonate along a shear-breccia zone striking northwest within the andesitic volcanic rocks. The vein has been traced on surface for a length of 1,000 feet and mineralization includes silver, argentite, pyrite, arsenopyrite, chalcopyrite, galena, sphalerite, niccolite, bismuth and bismuthinite. Sampling from the #1 vein indicated an average grade of 42 ounces per ton silver across a width of 10 inches for a length of 100 feet. Native silver is disseminated throughout granular translucent quartz in veins a few inches wide. Niccolite and galena are fairly common. A shipment consisting of most of the bagged ore was reported to have averaged 438 ounces per ton silver. The main zone on the property apparently contains some pitchblende locally and a sample from where the adit intersected the vein is reported to have assayed 7.4 ounces per ton silver and 0.226% uranium oxides (U₃O₈).

White Eagle Silver Mines Limited (1933-1935)

Rich silver veins were found on the 'Otter' and 'Elite' claim groups in the summer of 1932 and White Eagles Silver Mines Limited was incorporated in the summer of 1933 to develop the property. A crew was assembled and put to work on the claims in August 1933, building camp and trenching the silver veins. Early surface work on the #1 vein followed ore for a length of 130 feet, with widths of 18 inches and averaging 800 ounces per ton silver. High-grade ore, rich enough to bag and ship for smelting, was reported in surface showings. The vein at that point had been traced for over 1,100 feet on surface (The Toronto Star, Sept. 14th 1933). A mining plant was shipped to the property in September 1933 and underground work was initiated on the 'Otter' claims by November 1933. A tunnel had advanced 100 feet by the end of the year (The Toronto Star, Dec. 22nd 1933).

A mining plant consisting of an Ingersoll-Rand engine, 370 cubic feet per minute compressor, 20 horsepower boiler, and tugger hoist was in use in September 1934. A small log cabin camp was established on the point of the entrance to Jason Bay, ½ kilometer west of the mine workings. Len G. Smith was mine manager in charge of 24 men. Development consisted of a 60 foot adit, extending north from the shore of the river, turning northwest 300 feet to follow the #1 vein. A crosscut was driven northeast for a distance of 100 feet at a distance 200 feet along the previous drift. A small winze was then sunk 50 feet in from this crosscut to a depth of 125 feet and about 425 feet of development was reported on the 125-foot level. It was planned to crosscut towards the #2 vein and open both veins at depth with a production decision planned for 1935 (The Northern Miner, Sept. 6th 1934). Silver values on the lower level were not as rich as those obtained through sampling on the adit level, and as a result the company ceased

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

operations at the property in January 1935. High-operating costs in the district, a low price of silver, together with the capital costs involved in fully exploring the silver property were also factors in the cessation of work (The Toronto Star, Jan. 11th 1935; May 23rd 1935).

Work to date suggested that the high-grade concentrations of silver ore were irregular and erratic in nature. There were no indications that conditions would improve at depth. Surface showings of high-grade silver, while interesting and encouraging, were not conclusive enough to promote continued expenditure at the mine (The Toronto Star, May 23rd 1935).



Figure 1. Mine camp on Camsell River, 1970

Silver Bay Mines Limited (1968)

Silver Bay Mines Limited was incorporated in January 1968 and entered into an agreement with Frank Lypka for exploration of the property. A new portal was excavated on the adit level to provide easier access to the silver deposit (Dollery-Pardy, 1969a). Silver Bay set up a trailer camp consisting of four large units, de-iced and de-watered the underground workings, rehabilitated the old winze, carried out 280 feet of raising from adit level to surface in three raises, 350 feet of lateral development on the adit level, and an equivalent of 300 feet of slashing to widen the level so that large wheeled vehicles could be used underground. They also excavated a large portal to park vehicles underground (this excavation was later used for the milling plant). Crosscutting intersected the #4 vein north of the previous workings.

This work was carried out by a crew of about 10 to 15 men (not including diamond drillers) under the direction of Jack McBeath. Equipment reported on the property in 1968 consisted of two 250 cubic feet per minute air compressors, one 600 cubic feet per minute air compressor, two older compressor units, one Allis-Chalmers loader, scooptram, one ore carrier truck, and stoper and jackleg drills (Brown, 1968; Oliver, 1968). Development ceased at the end of the 1968 summer season. Exploration continued in 1969 during which time geologist William Dolerdy-Pardy was hired to map the underground workings and surface exposures (Dollery-Pardy, 1969a and b).

Federated Mining Corporation Limited (1970-1972)

The property was acquired by Federated Mining Corporation Limited in August 1970. Exploration that began in 1968 continued during this year in order to establish an exact ore reserve. This work indicated 22,785 tons within the #1 vein with content of 104 ounces per ton silver. Bulk samples of sulphide ore and high-grade pickings were shipped in 1970 to British Columbia for test smelting at the labs of Delta Smelting & Refining Limited, and 3,980 pounds of ore were processed to recover 2,406 ounces of silver (Delta Smelting & Refining Limited, 1970).

By 1971, a 7,000-ton stockpile of ore mined since 1968 was awaiting processing. A portable 50 ton per day Impact-type mill was set up on property and began to process the stockpile in March 1971 (Dollery-Pardy, 1971a). It was housed in a make-shift frame and plywood structure. The flowsheet was conventional crushing in a jaw crusher and a 50 ton per day ore champ-impact, followed by gravity concentration on a Wilfrey table and Denver duplex jig. Concentrate was passed through a small flotation and thickener tank before being dried and packed for shipment. Bill Cherette was mine manager in charge of 1971 operations with a small crew of 12 men. The milling plant operated at a low capacity of 20 tons per day with a recovery of only 75% (Dollery-Pardy, 1971a). According to a report dated May 1971, milling to date had produced 44 tons of concentrate, of which 17 tons had already been flown out. The concentrate was reported to assay 5,000 ounces per ton silver, 8 ounces per ton gold, and 16 pounds per ton bismuth (Murphy, 1971). Sixteen tons of concentrate was delivered to Delta Smelting & Refining Limited's plant in British Columbia in July 1971. Stopping was underway on the adit level, and preparations for driving a decline to the 325-foot level were to start the following year.

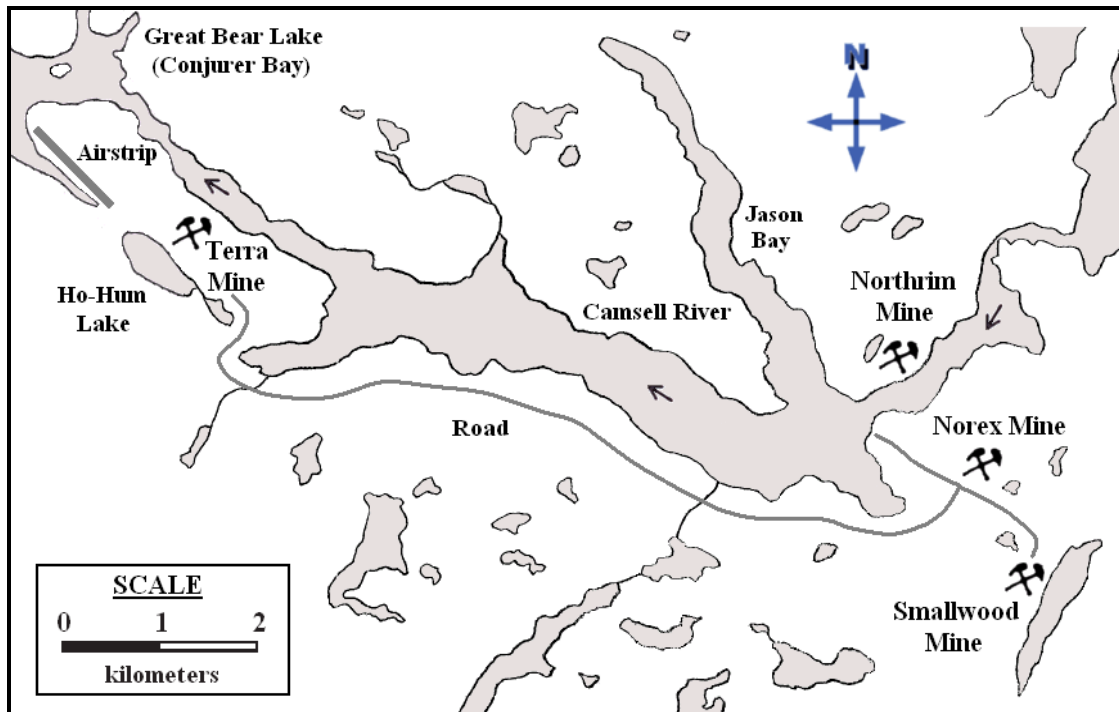


Figure 2. Camsell River area showing location of Northrim Mine.

Underground Mill

In the fall of 1971, a larger 100 ton per day milling plant was ordered. It was decided to install the plant underground for two main reasons: to reduce heating costs and to reduce haulage distance from the mine workings. An excavation previously blasted as use for a shop and garage was enlarged to make room for this mill plant. The underground mill went into operation February 1972 as the first of its type in the Northwest Territories (Badham and Morton, 1972).

Mining Operations

Primary entrance to the underground was via the 1968 portal connecting to the adit level. All mining development was along this heading where three raises broke through to surface and stoping was completed. Shrinkage stoping was the preferred method of extraction, but open stope methods and rill stoping were used in narrow vein areas. Water for drilling was supplied by an underground deposit within the old 1934 winze. Underground equipment consisted of four Atlas-Copco rock drills, four Jackleg drills, one stoper drill, a 1½ ton Getman ore carrier, an Eimco #630 mucking machine, and a Cavo #310 loader (Badham and Morton, 1972; Bullis, 1972).

Mill Operations

Ore from the stope workings was hauled up a 240 foot incline passage connected from the primary adit, and delivered into a 50 ton coarse ore-bin. Coarse ore was fed into the primary Universal 12 inch x 24 inch jaw crusher and reduced to -1½ inch size, followed by fine-ore storage, then secondary crushing in a Kue Ken jaw crusher which reduced the ore to -¾ inch size. A 6 foot x 8 foot Williamson ball mill received crusher product at a nominal rate of 78 tons per day. The flow discharged from the mill and passed over a Denver 16 inch x 24 inch duplex jig, where coarse silver was collected as a jig concentrate. Rejects from a Dorr classifier were sent back for grinding, while all other material was passed over two Wilfley tables where the table concentrate was recovered. Table tailings were pumped to nine Denver flotation cells, where the flotation concentrate was recovered. All concentrates were dried in a filter and separately bagged for shipment. Tailings were jettisoned from the cleaner cell and pumped out the underground and deposited in a small lake north of the mine (Badham and Morton, 1972; Bullis, 1972).

The mill was housed in a 50 foot x 100 foot underground chamber. The height of this chamber was 15 feet on average but compensated for the ore-bin clearance.

Power Plant

Electricity for the mill and camp was supplied by a primary 500 horsepower Cleveland diesel engine connected to a 250 KVA General Motors generator. Back-up power was available with a Ruston or Lister generator. This power

plant was housed underground adjacent the mill chamber and exhaust heat was used to heat the mine during the winter months. Two portable diesel driven Ingersoll-Rand air compressors of 350 cubic feet per minute and an Atlas-Copco 200 cubic feet per minute compressor were also housed in an underground chamber. Diesel fuel for all mine equipment was stored in a 30,000 gallon tank (Badham and Morton, 1972; Bullis, 1972).



Figure 3. Mine Portal, 1970

Camp Facilities

The camp at the Federated property during 1972 consisted of three trailers for crew accommodation, one dry trailer, an office, and a frame building for kitchen-dining hall. These facilities could hold a 24 man work crew, the camp population at the time being about 20. Most work was under the direction of geologist William Dollery-Pardy and Federated Mining Corporation's president, Jim Farrell (Badham and Morton, 1972; Bullis, 1972).

Transport

Freight was brought to the property by barge from Great Bear Lake or floatplane during the summer months. A proposed airstrip was started north of the mine on an esker on the shores of Conjuror Bay (Great Bear Lake), but was only 50% complete at shutdown in 1972. The ice road from Marian Lake to Great Bear Lake to service the Echo Bay and Terra silver mines could also bring in freight to the Camsell River area.

Production

Records for production during this period are incomplete. About 4,000 tons of ore were milled during the two periods of mill operations in 1971 and 1972. Mill feed was mostly low-grade muck containing 40% wall rock, with mill heads of 5 to 32 ounces per ton silver. From February 1972 to about early April, the mill treated 750 tons to produce 2,500 pounds of jig, 2,000 pounds of table, and 14,500 pounds of flotation concentrate (Badham and Morton, 1972). Assays were reported in a wide range of values. Jig concentrates ran from 500 to 4,000 ounces per ton silver; table concentrates ran from 400 to 600 ounces per ton silver; flotation concentrates ran from 200 to 500 ounces per ton silver. Tailings ran about one oz per ton. This assay information indicates that the ore body being mined was erratic. There is no record of actual silver production for this period. The mill only treated about 70 tons per day on average with the operation closing in June 1972. The mine closed before depth development through the use of a proposed decline to the 325-foot level could begin. The cost of operation from 1970 to 1972 totaled about CDN\$800,000 (The Northern Miner, July 6th 1972).

Northrim Mines Limited (1975-1978)

This company acquired the Camsell River property in 1973 and began plans to reopen the deposit. During 1975, a decline was advanced 520 feet to the 125-foot (2nd) level to intersect the workings previously opened up by winze development in 1934. Exploratory drilling intersected additional ore below this level, and 1,000 tons of ore were mined. Ore reserves pre-production were reported as 25,958 tons, with 10,208 tons in the #1 vein and 8,750 tons in the #2 vein (The Northern Miner, July 29th 1976). Production from the mill started in October 1976 treating 50 tons per day. The underground mill was modified by the creation of a more efficient conveyor system from ore-bins to ball mill and by the addition of two Denver flotation cells. Also, two smelter furnaces to melt dore silver bars were installed. It is not known if any bars were poured (Morton, 1976).

New Power Plant

An all-new power generating plant and air compressors were purchased for the increased operation. Two 500 KVA diesel generators, consisting of a Cat D-379 unit and a 12-cylinder Paxman unit, were used for primary power. A 20 kilowatt Lima and a seven kilowatt Stamford generator were used as standby. A new 125 horsepower Gardner-Denver Electrica-Screw compressor was put to use, along with a Twistair-Joy compressor and a 600 cubic feet per minute Ingersoll-Rand engine. All units were portable and were housed underground. Camp facilities were expanded to house a 25 man crew. Six trailer units and a 50 foot x 35 foot frame building used as cookery were at the campsite.

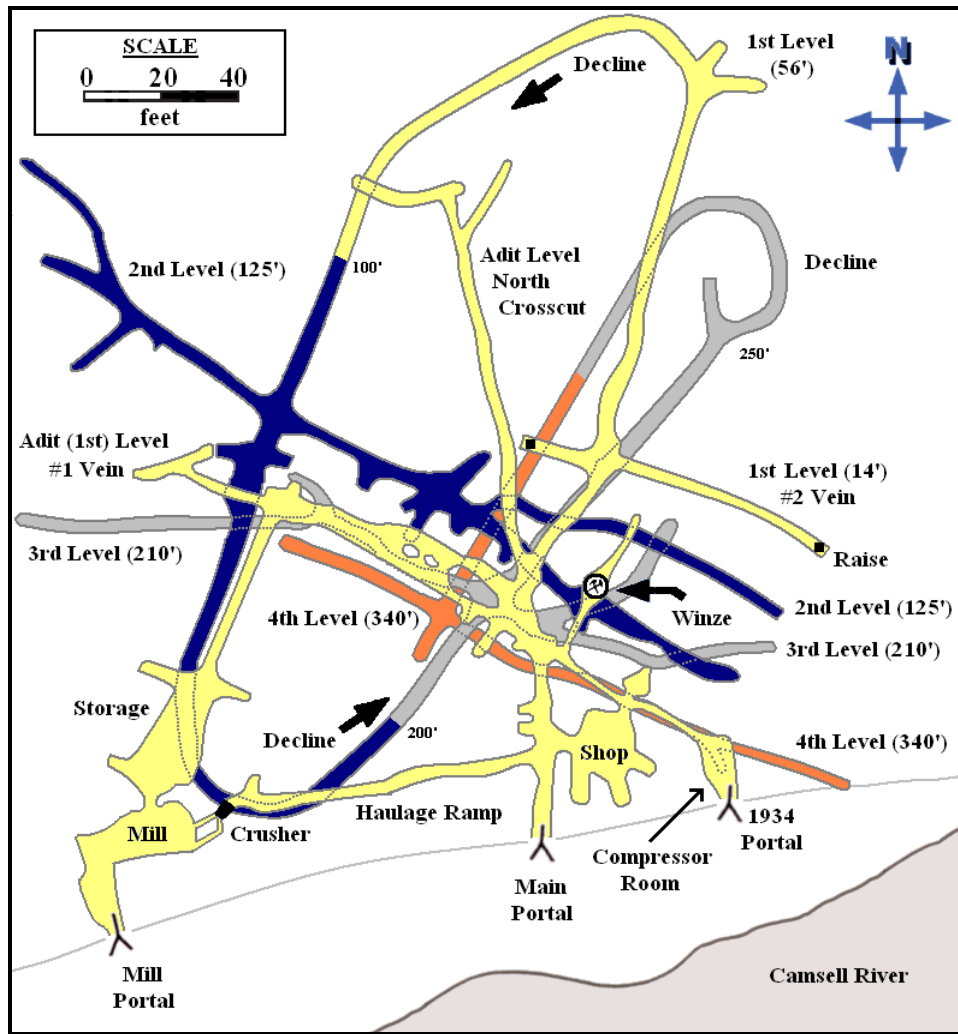


Figure 4. Northrim Mine composite underground plan, c.1978.

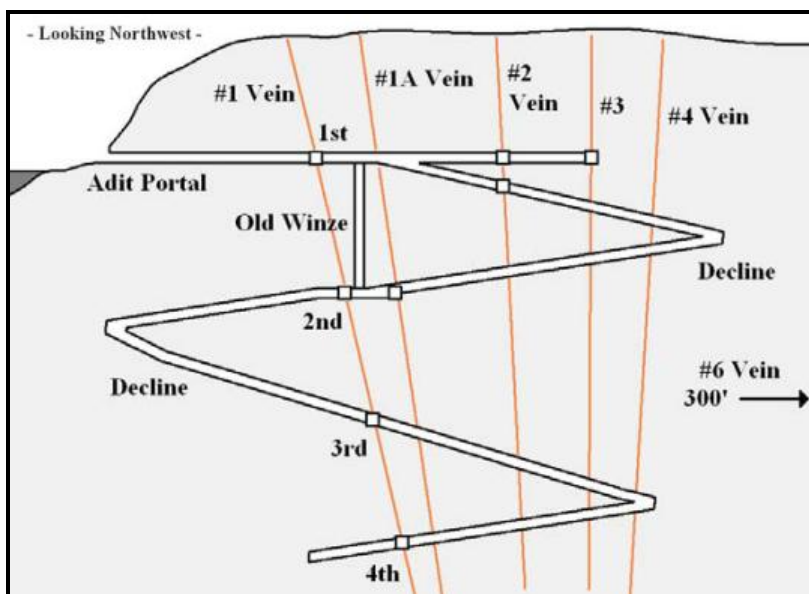


Figure 5. Northrim Mine simplified longitudinal plan.

A 20 foot x 30 foot Butler building was at the portal site used as a warehouse and shop. Oil storage capacity was 135,000 gallons. Twenty-two persons were employed in October 1976.

The former underground production equipment was found to be in bad shape. Three Ingersoll-Rand stopper drills and seven Ingersoll-Rand rock drills were bought for the operation. Heavy equipment consisted of an Eimco scooptram, mucking machine, and Capco and Scoot-Crete ore carriers (Morton, 1976; Clarkson, Gordon & Co., 1979).

In 1977, the decline was advanced to the 210-foot (3rd) level where it intersected a pod of saccharoid quartz with interstitial native silver in the #1 vein. Drifting and stoping within the vein extracted the ore pod but did not reveal any continuation of the ore. The #2 vein, also intersected on the 3rd level in 1977, did not contain any economical silver deposits. Production in 1977 amounted to the treatment of 3,752 tons to produce approximately 15,000 ounces of silver. This ore was primarily derived from stoping a 300 foot strike length of the #1 vein on the 2nd level (Lord et al., 1981).

The decline advanced to the 340-foot (4th) level in 1978 but little development was undertaken. A small ore section within the #1 vein on the 4th level east drift was reported to contain 1,500 tons grading 100 to 150 ounces per ton silver (The Northern Miner, Sept. 1st 1978). Work on the 2nd level early in 1978 encountered a new, high-grade silver zone. It had been opened a length of 50 feet with an average width of four feet. Samples averaged 1,103 ounces per ton silver and 7.9% lead (The Northern Miner, Mar. 2nd 1978). In 1978, 4,189 tons were milled to produce 15,000 ounces of silver. The mine closed in June 1978 due to a lack of working capital, a fire in the mill plant, and the breakdown of underground equipment (The Northern Miner, Sept. 1st 1978; Lord et al, 1983).

Between 1975 and 1978, a total of 2,563 feet of decline was driven and 4,087 feet of lateral development completed on three levels and two sub-levels, along with 1,550 feet of raising. Practically all production was derived from sections of the #1 vein, with minor production also coming from the #2 vein. Exploration and sampling of the #2 vein mostly revealed uneconomic silver values. Underground plans are shown in **Figures 4 and 5**.

Exploration Since Mine Closure

Terra Mining and Exploration Limited acquired the property following the insolvency of Northrim Mines Limited in 1979. They conducted some surface exploration in the mine area and deiced and dewatered the underground workings in 1981 to perform sampling. There is very little information available about this work.

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O'CONNOR LAKE

Bulk Sampled (Abandoned)

Lead, Zinc

61° 19' 30" 111° 47' 30"

Years of Primary Development: 1952

Mine Development: 180' vertical shaft, one level (329' dev.)

Years of Bulk Sampling: 1952

Bulk Sampling: 35 tons ore shipped = 14 tons lead

Introduction

The O'Connor Lake Mine is located on the north end of O'Connor Lake, 146 kilometers north of Fort Smith, NWT and about 50 kilometers east of the old community of Rocher River. Distance from Yellowknife is 185 kilometers. It was a small lead and zinc underground prospect and did not attain production. The site was visited by officials with the federal government in the summer of 2006 and was found to remain largely intact however all remaining buildings are in very poor condition. The old headframe still stands as of that date (see **Figure 1**).

Brief History

The property was staked (49 claims, 'MWK' group) to cover a long gossan showing in 1948 by Frank Morrison, Harold Killins, Claude Watt, and Norman Burgess. These men soon after discovered a lead-zinc-copper-silver vein nearby, and formed the O'Connor Lead Syndicate to develop the claims for its high-grade lead content. Some 300 tons of surface ore were outlined in 1948 with grades of nearly 75% lead, rich enough, it was believed, to warrant direct shipping to a smelter. Surface sampling returned values in manganese, lead, and zinc. In 1949 a new company, O'Connor Lake Lead Mines Limited, was formed and a diamond-drilling program (six holes) was initiated. 200 tons of cobbled ore of 70% lead were reported available for shipment in 1950. Some 27 tons of this material was bagged.

American Yellowknife Mines Limited acquired the property in November 1951, and initiated a new diamond drill program. The deposit was interesting enough that the decision was made to go underground in the fall of 1952, and a shaft was sunk to one level. More extensive work was required to determine if the lead deposit held any economic merit, and work ceased at the end of 1952.

Geology and Ore Deposits¹

The O'Connor Lake area is located near the western margin of the Taltson Magmatic Zone. The oldest rocks are Archean, supracrustal, strongly metamorphosed paragneisses and schists with minor amounts of lesser-deformed greywacke, argillite, shale, and quartzite. Small, granitic lenses and pegmatite bodies and lit-par-lit pegmatite injections occur commonly in the paragneiss. Interbanded with the paragneiss and metasediments are small bodies of metavolcanic amphibolite and amphibole gneiss, with minor banded iron formation and ultramafic rocks. Quartz-carbonate veins of 50 centimeters to two meters widths trend northwesterly and crosscut all lithologies including diabase dykes. These fissure type veins are host to several copper, zinc, gold, silver, and lead sulphide showings. Large, brecciated quartz veins zones of 25 to 30 meters width consist of a stockwork of quartz veinlets with pink feldspar and are usually barren of mineralization.

The #1 vein is located on a peninsula in the eastern part of O'Connor Lake. The vein occurs in a shear zone in banded amphibolites and amphibole-biotite gneiss that contains some granitic material. The vein is partially exposed on surface, trends 315 degrees near the north end and 335 at the south end, the dip varies between 50-75° to the southwest. The width varies between 50 centimeters and two meters, averages about 1-½ meters and is constant at depth. Vein filling consists of white to creamy coloured, fine- to coarse-grained carbonate and fine-grained white quartz. In some places quartz veins occur in carbonate, and are therefore later. Wallrock contacts are relatively sharp with minor quartz veins and stringers in the gneiss that average 15 centimeters in thickness and run parallel to the main vein. A thin seam of graphite occurs at the footwall contact. Sphalerite and galena are mainly disseminated throughout the quartz together with lesser pyrite and pyrrhotite; occasionally massive sulphide pods are up to 12 centimeters wide and several feet long occur. Pyrite and chalcopyrite are abundant locally. Larger masses of sphalerite and galena tend to occur near the vein margins, and most exploration work has concentrated on several shoots of this material, which tend to pitch at 50° to the northwest in the plane of the vein. The vein is hosted by amphibolite interbedded with quartz-biotite-oligoclase gneiss and sillimanite-garnet gneiss. The amphibolites are fine- to medium-grained, dark green or black with minor amounts of garnet, pyrrhotite, and graphite. Both massive

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

and well-banded amphibolites are present. Medium-grained, coarsely banded pink quartz-biotite-oligoclase gneiss occurs to the west. Foliation in the gneisses strikes about 340° and dips steeply to the northeast.



DIAND-CARD photo

Figure 1. *O'Connor Lake Mine headframe, 2006.*

American Yellowknife Mines Limited (1952)

The property was acquired by American Yellowknife Mines Limited in November 1951 from the O'Connor Lead Syndicate for CDN \$20,000 cash plus a 7½% royalty to the syndicate. The property was adjacent to American Yellowknife's 'Frank' base metal property. The #1 vein was the focus of interest. Mineralization was confined, on surface, to an oreshoot 300 feet long and 5 feet wide. In 1949, 27 tons of hand cobbled ore from an open cut on the #1 vein were bagged and stockpiled. Random sampling of this ore showed average of 62.6% lead and 6.9% zinc. Silver content was not assayed (The Northern Miner, Nov. 15th 1951).

The company initiated a new diamond drilling campaign in December 1951 to test the #1 vein along its length and at depth. It was immediately recognized as an impressive showing and company engineers believed it had mine making possibilities. New financing arrangements in January 1952 called for the raising of enough funds to bring the mine to the underground development stage as quickly as possible (The Northern Miner, Jan. 3rd 1952).

A 45-mile winter road was cleared from Rocher River in early 1952. The original road was cleared using a team of horses, but later in the spring an improved road was made using two D-7 Cats under contract with Cinnamon & Vincent Company of Yellowknife. Drilling encountered parallel veins on both sides of the #1 vein in January 1952. Mineralization was intersected in widths of up to 2½ feet (The Northern Miner, Jan. 10th 1952). Diamond drilling ceased in April 1952 after drilling the vein over a 1,000 foot length. A 550 foot long ore shoot within the #1 vein was explored to 200 feet depth with indications that the structure widened at depth and on strike. The #2 vein was also discovered 1,500 feet southeast of the #1 vein in March 1952, which may have been the faulted extension of the main zone break. An ore shoot 225 feet long and 3 feet wide was indicated in this vein (The Northern Miner, Mar. 13th 1952).

Essential mining equipment was placed on order for direct delivery during the summer of 1952. Work during May 1952 included extensive geological mapping of the property. Several new veins were discovered (#3, 4, 5). The Gossan zone area was also to be investigated thoroughly. A total of 16 veins were ultimately discovered, but only the #1 and #2 veins were considered economic. The Gossan zone was not commercial. Permanent camp buildings were completed and a staff was on site. Diamond drilling to date indicated two 600 foot long oreshoots within the #1 vein, estimated to contain 250 tons per vertical foot averaging 18% combined lead and zinc. The #2 vein was traced for

2,700 feet with possible tonnage of 50 tons per vertical foot, grading 18% lead and zinc. This was based on trenching results only, as no diamond drilling was performed on the #2 vein (The Northern Miner, Sept. 4th 1952).

Shaft Sinking

In August 1952, the decision was made to start shaft sinking on the #1 vein. A shaft-sinking contractor (Lanky Muyres) was hired and the shaft site was cleared and collared on August 11th. Work was suspended briefly because of an air compressor malfunction, but resumed on August 26th. The timber headframe was erected and shaft sinking commenced on September 17th 1952. The shaft was completed to 180 feet depth in 23 days. On October 20th 1952, a shaft station was cut on the 150-foot level and crosscutting to reach the vein commenced November 1st using company miners. Over 90 feet of crosscut advance was required to reach the vein, which was intersected on November 20th. Drifting then commenced north and south along the vein (Byrne, 1952b). From August-December 1952, 4,721 tons of waste was hoisted, and from November-December, 995 tons of ore were also hoisted (NWT Mining Inspection Services).

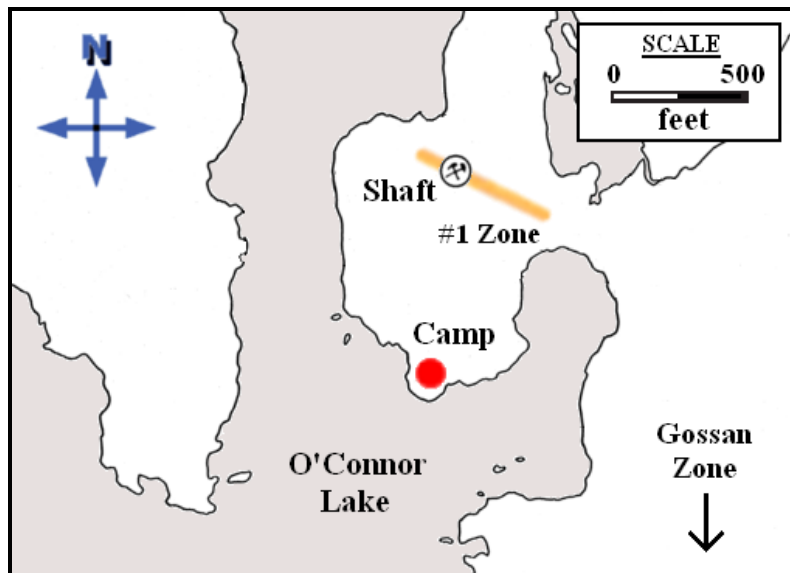


Figure 2. *O'Connor Lake Mine property plan.*

Camp, Plant, and Equipment

A campsite was located on the shore of O'Connor Lake and consisted of a cookery, bunkhouse, warehouse/office, and a log cabin used as the mine manager's house. A short tractor road connected to the shaft site about 1,500 feet north of the camp. Here was located a powerhouse/hoist, boiler house, blacksmith shop, 42 foot timber headframe, and miners dry. Known mining equipment in use included two Cat D-13,000 diesel driven air compressors, a 24 inch x 18 inch two-drum air hoist, six mine cars, and an Eimco mucking machine. Up to thirty men were employed in September-October 1952 (20 contractors, 10 company staff). Staff included Robert Evans, mine manager; Pat McNeil, underground shift boss; and Sid Holden, camp cook. Norman Byrne was consulting engineer. Mine manager Robert Evans' family resided on site in a small log cabin.

In September 1952, 35 tons of hand cobbled ore from the surface pit on #1 vein were shipped to Cominco in Trail, B.C (The Northern Miner, Sept. 4th 1952). Results of assaying this material is unknown, but other samples sent to labs in Vancouver for mill testing assayed 95% to 97% lead and zinc (The Northern Miner, Sept. 25th 1952). According to Stats Canada, 27,445 pounds (14 tons) of lead was recovered from the Northwest Territories in 1952, presumably from the Cominco bulk sample (Statistics Canada, 1957).

Operations were suspended December 16th 1952 to await better metal prices and the future large-scale developments at Pine Point. A preliminary ore reserve was published at this time. Within the #1 vein, two blocks of ore totaling 540 feet length to 150 feet depth indicated 33,160 tons averaging 15% lead and zinc. Calculations to 250 feet depth and a continuous length of 600 feet showed 67,950 tons averaging 12% lead and zinc. Mill test work indicated the ore was free of impurities and milling operations would involve a simple flow sheet requiring coarse grinding (The Northern Miner, Jan. 15th 1953; American Yellowknife Gold Mines Ltd. Annual Report, 1953). No work has been done since 1952.

Underground Development Summary

A 3-compartment shaft was sunk to 180 feet depth and a level was established at the 150-foot level. A 25 foot long station was cut. The crosscut to the #1 vein on the 150-foot level was 127 feet long. The north drift was 76 feet long and the south drift is 126 feet long. Total lateral development on the 150-foot level is therefore 329 feet advance.

Exploration Since Mine Closure

No known work.

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OLD PARR

Minor Producer (Abandoned)

Gold

62° 44' 30" 113° 31' 45"

Years of Primary Development: 1947-1972

Mine Development: 5 primary trenches/pits

Years of Production: 1947-1964

Mine Production: 720 crude oz Au, plus unrecorded high grading

Introduction

The Old Parr property is located between Little Sproule and Parr Lakes, 53 kilometers northeast of Yellowknife, NWT. The site was visited in August 2004 by the author.

Brief History

The first major gold find at the property was in May 1947 by Louis Garskie, staking the 'Old Parr' claims. Garskie, with the financial support of Martin Bode, spent 19 months at the property, and arrived back to Yellowknife late in 1948 to showcase \$5,000 worth of coarse gold in four beer bottles. Apparently the property had been staked at least twice in the past with no gold being found. The discovery created quite a stir around Yellowknife. In 1950 the property was optioned to Garskie Gold Mines Limited, who conducted surface diamond drilling from the ice on Parr Lake during the winter. The operators of this program found little of value underground and let the option drop the following year. Louis Garskie continued his one-man operation into the late 1950s, and bought out Martin Bode's interest in the claims in 1957. In 1964, the Liten Mining Company optioned the claims and installed a small mill. After recovering a bit of gold from surface pits, the operation was closed. Louis Garskie continued work at his claims until 1972 when he retired.



The Western Miner magazine

Figure 1. *Louis Garskie, after his 19 month milling effort in 1948.*

Geology and Ore Deposits

The principal rocks in the area are metamorphosed strata (greywacke and slate) of the Yellowknife Group. The rocks strike northeast and dip from 50° northwest to near vertical. Quartz veins strike parallel to the country rock along small and narrow folds and fissures. Pegmatite dykes cut the sediments. Gold occurrences, though historically spectacular, are sporadic and most commonly found at the margins of the quartz veins (Lord, 1951).

Louis Garskie (1947-1963)

Louis Garskie and Martin Bode churned out 195 crude ounces of gold in their first two years of work in 1947-1948. Ore was extracted from the Galena veins, which were developed by small pits using single jack hand mining. It was then crushed and ground using a large boulder fitted with a handle. Laborious hammering to reduce the rock down to a fine size was followed by roasting of the high-grade gold ore over an open fire. About 120 tons of ore was mined (The Western Miner, Feb. 1950; Jeckell, 1960a).

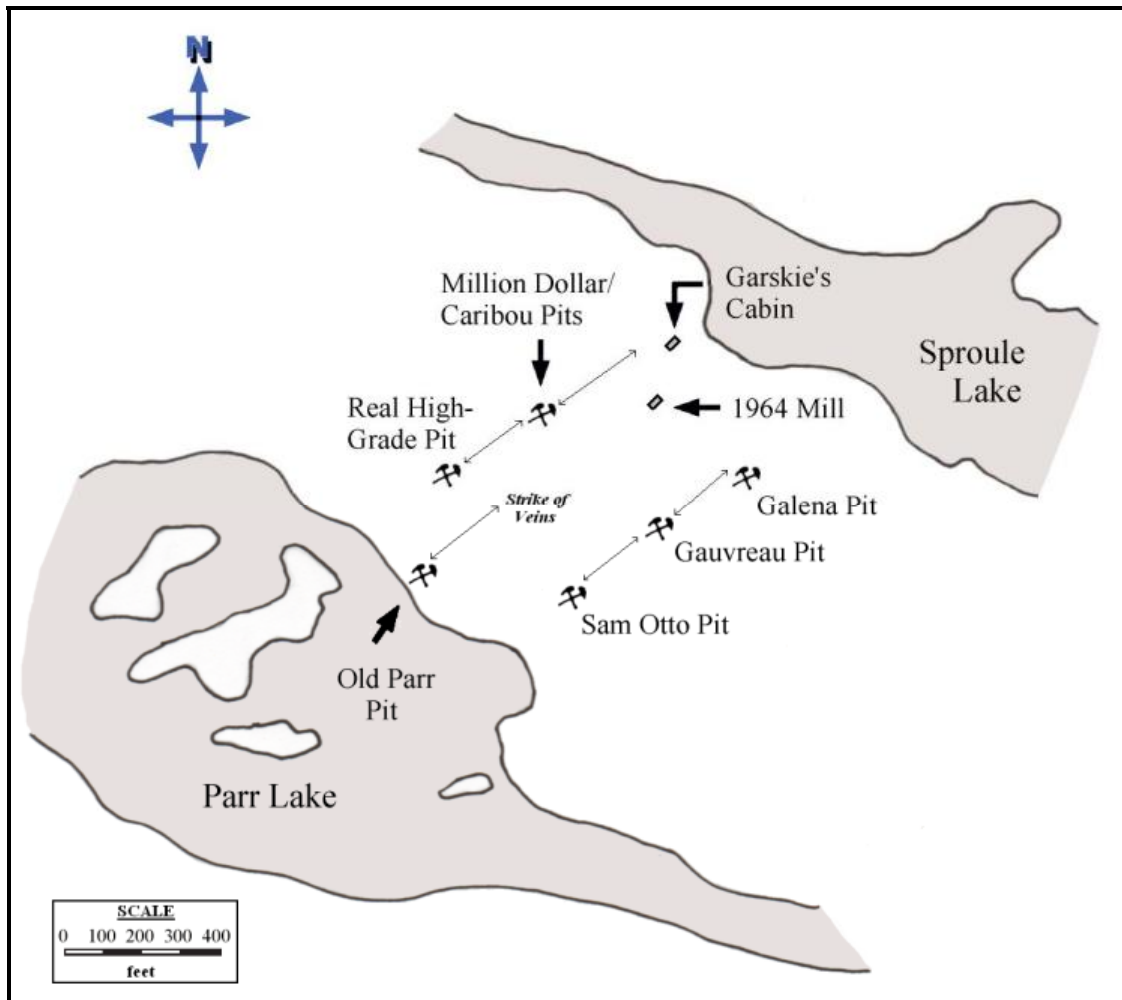


Figure 2. Old Parr Mine area.

The main area of interest at the property was within a southwesterly trending strata belt 500 feet wide between the two bordering lakes. Quartz veins intersected the strata throughout. The most important deposits were the Galena veins, separated by 2 feet to 4 feet of host rock and trending northeast. A pit, known as the Million Dollar pit, was the main source of vein material, although several other pits known as the Caribou, Jewelry Shop, and Real High Grade pits had been blasted in this area (Lord, 1951).

Garskie Gold Mines Limited optioned the claims in 1950 and performed a small amount of diamond drilling, but no gold values higher than 0.10 ounces per ton were encountered (Jeckell, 1960b). The option was dropped and control of the property reverted back to Garskie and Bode in 1951. Louis Garskie continued his endeavor to hand-cob gold ore and he began to sink a small shaft at the bottom of the Million Dollar pit. By 1957, the shaft had been sunk to 26 feet and a short seven foot drift was begun. Also in 1957, Martine Bode sold his 50% interest in the claims to Garskie. Work from 1951 to 1957 had awarded Garskie with 300 crude ounces of gold (Jeckell, 1960a). A shipment of '10 bars' in 1956-1957 to the Royal Canadian Mint indicated a content of 180 fine ounces of gold (National Archives of Canada).

In 1960, Vanguard Explorations Limited optioned the claims to conduct bulk sampling and surface mapping. This work revealed no significant deposits that would make an economic full-scale gold mining operation (Jeckell, 1960b). Development by Louis Garskie to 1960 consisted of several pits. The most extensive working was the Million Dollar pit, a 65 foot long and up to 12 foot wide opening with depths ranging from 10 to 26 feet, including the shaft. **Table 1** gives estimated excavation tonnage from each pit up to 1960. During 1960, Garskie deepened his shaft in the Million Dollar pit to 40 feet (Jeckell, 1960b). Garskie continued his lone, primitive operation into the 1960s intriguing the public with his tenacity. In 1963, 150 ounces of gold were recovered from slashing the pit walls of the Million Dollar pit (Schiller and Hornbrook, 1964).

Million Dollar:	270 tons
Million Dollar Shaft:	75 tons
Caribou:	70 tons
Jewelry:	117 tons
Real High Grade:	78 tons
Galena:	125 tons
Old Parr:	85 tons

Table 1. *Pit excavations to 1960. (source: Jeckell, 1960b)*

Liten Mining Company Limited (1964-1965)

Work began to create a full production mine at Parr Lake during 1964. An Edmonton based private company called Liten Mining Company Limited acquired the claims through option from Garskie early in the year. A winter road from Prelude Lake, up through Bliss Lake, and then through into Parr Lake, was cleared and a two-truck convoy of equipment and supplies arrived in April 1964. A mill was in the process of being erected, and tests of Garskie ore indicated a favorable recovery by gravity methods. First development consisted of enlarging the Galena pit and installing a mechanized boom for easy hoisting of ore. The Galena pit now had dimensions of 30 feet long, 10 feet deep, and 5 feet wide. Other work during the year involved deepening the Million Dollar shaft to the 60-foot level and driving two drifts for 5 feet and 13 feet, one in a northeast direction and the other in a southwest direction. A fault was noted in one of the drifts, cutting off the gold-bearing quartz mass. Pit excavations from the Million Dollar pit amounted to about 150 tons. For the first time ever, power for mining operations was supplied by heavy equipment: a 120 cubic feet per minute Jaeger air compressor with drills (Schiller, 1965).

Milling Plant

Ore was crushed in a small gasoline jaw crusher and conveyed into a log hopper, which fed a 3 foot x 3 foot Allis-Chalmers ball mill. Discharge was put through two cyclones. The first cyclone removed 50% of the original feed, with the remaining 50% going through the second cyclone was transmitted as follows: 82% was returned to the first cyclone, 10% was discharged as waste, and 8% was passed over a jig and blanket table for a gold concentrate recovery. It was then roasted to remove sulphides and delivered to an amalgamation barrel. Two gold shipments in 1964 consisted of 83 ounces of gold and 12 ounces of silver. The mill was powered by a small 100 horsepower Deutz diesel engine and operated during September 1964 (Schiller, 1965; Knud Rasmussen, pers. comm.).

There was no record of gold production in 1965, but some development continued on the Million Dollar, Galena, and Old Parr pits. This development included the excavation of about 300 cubic yards of rock, and the deepening of the Old Parr and Galena pits. The Galena pit was sunk 20 feet in an area 6 feet x 20 feet, to give the Galena pit a final depth of 25 feet. The Old Parr pit was sunk 28 feet in an area 4 feet x 20 feet, and a 7 foot drift (3 feet wide) was driven from this depth (Thorpe, 1966).

Final pit dimensions and development are estimated as follows. Million Dollar Pit: The pit is 46 feet long, 26 feet wide, and averages 12 feet deep. A 66 foot long trench forms the northern expansion to the pit. A shaft was sunk at the northern end of the pit to a reported depth of 60 feet, and limited drifting carried out at two horizons (18 feet total). Galena Pit: Workings consist of three pits, the largest of which is 33 feet long, 10 feet wide, and 25 feet deep. About 46 cubic yards of material was removed in total. Old Parr Pit: This consists of two pits sunk 15 feet and 28 feet each and a 7 foot drift at the bottom. 120 cubic yards of material removed in total. Jewelry Shop Pit: Two pits, the deepest of which is 15 feet (Jeckell, 1960b; Thorpe, 1966).

Louis Garskie (1970s)

Louis Garskie continued his one-man gold mining operation for a few years, but in 1972 he called it quits because of poor health. No further record of gold production during this period exists (News of the North, Aug. 10th 1972).

Total gold production, from both Louis Garskie high-grading and the 1964 mill, is listed in **Table 2**.

Years:	Rough Gold Produced:	Rough Silver Produced:
1947-1948	195 oz	-
1951-1957	300 oz	-
1963	150 oz	-
1964	83 oz	12 oz

Table 2. *Old Parr Mine Production. All gold was recovered by crude hand methods with the exception of 1964 when a mill was placed into operation. Does not include other miscellaneous high-grade samples.*

Exploration Since Mine Closure

No known work.

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Personal communication: Knut Rasmussen


ORMSBY
Advanced Exploration (Operational)
Gold

63° 10' 15" 113° 55' 30"

Years of Primary Development: 1997, 2004-2005
Mine Development: decline to 117m depth; 75m ventilation raise

Introduction

The Ormsby Mine is located south of the old Discovery Mine property, 82 kilometers north of Yellowknife, NWT. It was reactivated for underground exploration in 2004 having been inactive since 1995. Although the author has visited the Discovery Mine area numerous times over the years, he did not visit the Ormsby property during this period of operation. Tyhee Development Corporation Limited, the current operators, hopes to prove the potential for a new Yellowknife region gold producer.

Brief History

The Ormsby gold deposit, although now tied in with the Discovery Mine property, originated as a separate gold property when staked in 1944 as the 'Bruce' and 'Avis' claims by Alfred V. Giauque and sons. LaSalle Yellowknife Gold Mines Limited performed extensive diamond drilling in the late 1940s. Ormsby Mines Limited was formed to acquire the claims in 1954, and Discovery Mines Limited optioned the claims. A 2,400 foot long drive from the 950-foot level of the Discovery Mine southward was completed in 1955-1956 to provide a platform for underground diamond drilling in the Ormsby zone. An ore reserve of 100,000 tons grading 0.30 ounces per ton gold was calculated within the Ormsby zone. This ore was not economic to mine at the time.

The 'GMC' claim was staked in 1992 by Dave Webb and Gerry Hess to cover both the Ormsby deposit and the old abandoned Discovery Mine. G.M.D. Resources Corporation Limited bought the claims and did extensive exploration, including a 1995 program that drove an underground decline into the orebody. In 2001, Tyhee Development Corporation Limited acquired the property. Tyhee reopened the underground workings in 2004 with plans to bring the mine into production. Underground development ceased in September 2005 upon the completion of the development program. In 2008, Tyhee announced the decision to permit an open pit gold mine on the Ormsby zone using the Nicholas Lake ore deposit as a satellite producer. An environmental review of the project is now underway.

Geology and Ore Deposits

The Ormsby zone is hosted entirely within mafic metavolcanic rocks. Gold mineralization in the Ormsby zone occurs within an extensive metavolcanic hydrothermal breccia that hosts numerous steeply dipping north-easterly striking echelon gold-bearing zones. These zones are characterized by altered hydrothermal breccia containing elevated sulphide, carbonate, biotite, and late garnet concentrations. A bulk-mineable lower-grade resource has been identified within the Ormsby zone. The lower-grade gold mineralization is disseminated in a halo surrounding higher-grade auriferous quartz veins. Three vein systems have been noted and are named the #40, #35, and #30 zones (also sometimes referred to as the A, B, and C veins respectively). Other zones include the #55 and #60 zones (D and E-veins) (DuPre and Kirkham, 2004).

G.M.D. Resources Corporation Limited (1995)

G.M.D. Resources Corporation Limited did a limited amount of underground exploration in 1995-1997, driving a short decline using trackless equipment on the Ormsby zone. Camp mobilization and purchase of necessary fuel and equipment was completed over the 1995 winter road season, and the portal was collared during July 1995. The plan was to drive the decline for a length of 240 feet to a depth of 150 feet to extract a bulk sample (G.M.D. Resource Corp. Press Release, July 25th 1995). In addition to exposing high-grade veins, the ramp intersected three previously unrecognized vein systems. One of these veins graded 1.13 ounces per ton gold, 3.4 ounces per ton silver, 4.7% lead, and 19% zinc (G.M.D. Resource Corp. Press Release, Sept. 18th 1995).

Underground work ceased in December 1995 after the completion of a 290 foot long ramp (10 feet x 13 feet dimensions) at a grade of -15%, to unknown depths (G.M.D. Resource Corp. Press Release, Jan. 17th 1996). At 100 feet in from the portal, the decline intersected shear zone material containing rich amounts of gold. The portal was blasted into the side of a 60 foot outcrop hill at approximate elevation +1010 feet above sea level (all depths are recorded from +1050 feet above sea level which is where #40 zone outcrops). The #40 zone was encountered at the 66-foot level as the decline swung southwesterly. Two small drifts followed the vein in workings totaling about 160 feet. To the northeast drifting totaled about 200 feet but no veins were encountered. Total workings were estimated to

be 215 meters in extent. Gerry Hess, representing G.M.D. Resources, was in charge of development during this period. Steadily dropping gold prices put a halt on this program at the end of 1997 (DuPre and Kirkham, 2004).

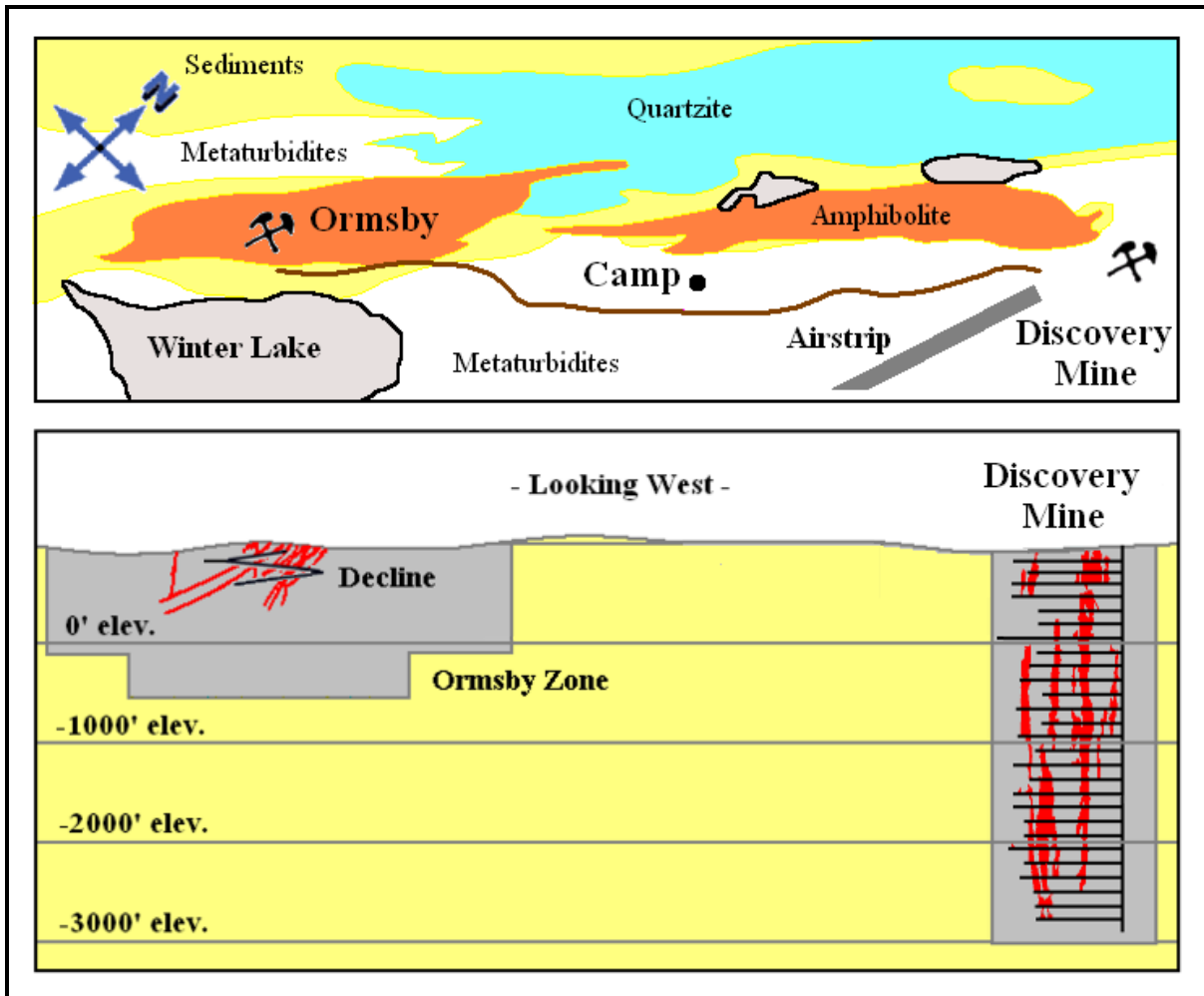


Figure 1. Ormsby Mine regional geology and longitudinal section, also showing the abandoned workings of Discovery Mine.

Tyhee Development Corporation Limited (2004-2005)

Tyhee Development Corporation Limited acquired the claims in January 2001 and proceeded with exploration. Based upon the encouraging diamond drill results of the 2002 and 2003 season, in which significant gold assays were encountered to a depth of 1,650 feet over a 1-½ kilometer strike length on the Ormsby zone, the decision was made to proceed with underground development. A new gold bearing zone, between the Ormsby zone and what was known as the West zone at Discovery, was also encountered during diamond drilling exploration (DuPre and Kirkham, 2004).

Plans in 2003 were to extend the decline by a length of 640 meters at a grade of -15° to a depth of 100 meters, with 420 meters of lateral work and 285 meters raising to allow access to six of the outlined mineralized zones. Drill stations would be established and further diamond drilling from the underground would be undertaken (Tyhee Development Corp. Press Release, Jan. 28th 2004).

Financing was arranged, equipment ordered, and necessary regulatory permits acquired early in 2004. Camlaren Mine Development Limited was awarded the contract for rock and blasting services. The Discovery Mine winter road was reopened and freight was trucked to the site during the winter hauling season in 2004. Forty-six truckloads were estimated to be required. An expanded 25 man camp, consisting of wood framed tents and seven Atco trailers, was set up in March 2004. Six prefab containers for use as storage and shops at the Ormsby portal were also set up (Tyhee Development Corp. Press Release, Mar. 23rd 2004).

Equipment

Machinery brought to the site for the 2004 decline driving program included two 3½ yard scooptrams, a three-boom jumbo air drill, a 15 ton haulage truck, D-6 Cat dozer, three 800 cubic feet per minute diesel compressors, two 220 kilowatt Cat gen-sets, six jackleg drills, and three stoper drills.

Underground work commenced in July 2004. Slashing of the decline and portal walls was undertaken to widen the workings, and portal walls and ceilings were also screened for safety. By the end of August the decline drive had advanced 120 meters. The company reported that mineralized zones were intersected where predicted, based on the resource estimates compiled as a result of previous drilling work. The #35 zone was intersected during the decline drive at 40 meters depth, as was the #30 zone. Also in August 2004, Tyhee hired Doug Levesque to act as project manager at the Ormsby property (Tyhee Development Corp. Press Release, Sept. 2nd 2004).

By October 2004, the decline advanced 287 meters total and excavation of lateral drifts expected to amount to 400m were being started (Tyhee Development Corp. Press Release, Oct. 6th 2004). The downward extension of the #40 zone was encountered at a depth of 30 meters during November. Muck sampling and chip sampling of the decline walls showed significant gold assays in all zones (Tyhee Development Corp. Press Release, Nov. 10th 2004).

Work ceased temporarily on December 15th 2004 for the holiday season. Total development during the 2004 program consisted of 460 meters of decline, 200 meters of drifting, and 200 meters equivalent of slashing to widen the decline. The Ormsby zone was opened up to a depth of 75 meters below surface. Drifting was performed at a depth of 40 meters within the #40 and #35 zones and at a depth of 70 meters in the #60 zone. It was reported that this work outlined ore zones that confirmed the importance of the deposit. Greater widths and greater continuity than expected were encountered (Tyhee Development Corp. Press Release, Jan. 11th 2005).

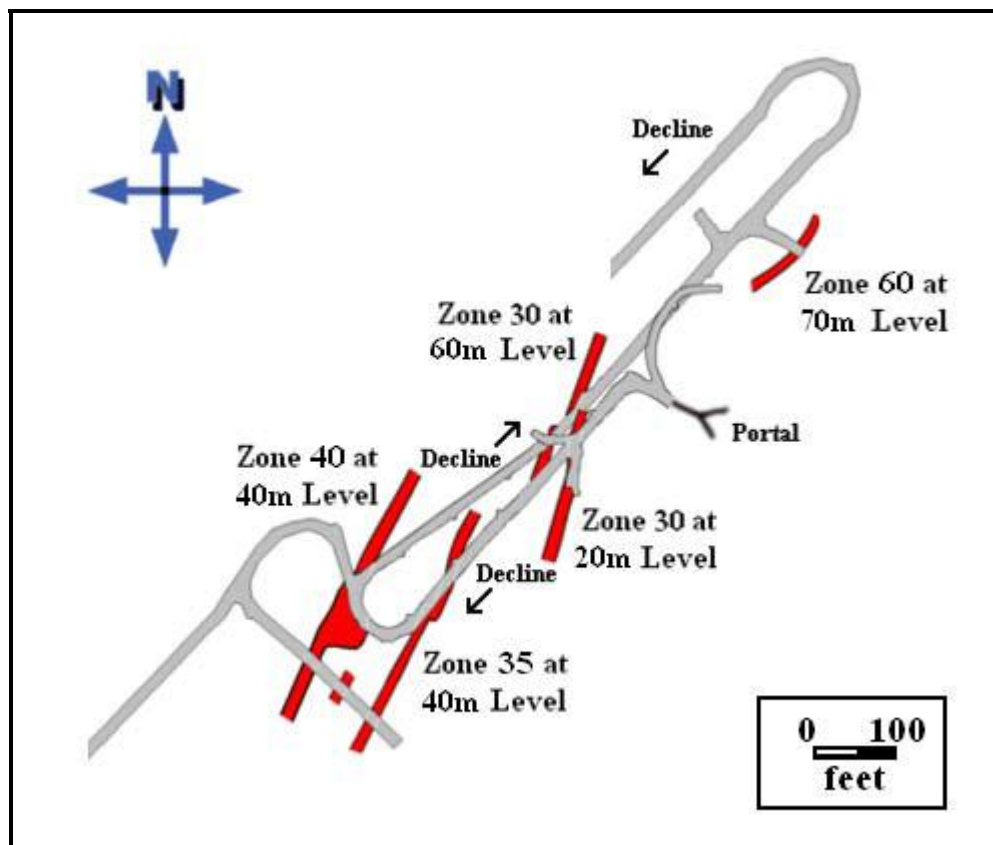


Figure 2. *Plan of underground workings, 2005.*

2005 Development

Work resumed in January 2005 with plans to identify new zones of mineralization and to confirm that the ore zones were amenable to conventional mining techniques. The camp was re-supplied over winter road in February-March 2005, and underground development resumed on April 7th 2005 (Tyhee Development Corp. Press Release, Apr. 18th

2005). A new ore reserve was announced in February 2005 upon the tabulation of diamond drill and development data. The Ormsby zone ore reserve (measured and indicated) was 3,006,678 tonnes grading 8.91 grams per tonne gold with a total content of 862,000 ounces of gold. An additional inferred ore reserve of 1,222,590 tonnes grading 7.57 grams per tonne gold was also announced for the Ormsby zone. The Ormsby West Extension zone had an inferred ore reserve of 221,897 tonnes grading 9.81 grams per tonne gold (Tyhee Development Corp. Press Release, Feb. 21st 2005).

A new diamond drill capable of drilling to depths of 760 meters below the surface was brought to the site. The trailer camp was expanded to 50 man capacity. Underground development completed over the summer of 2005 consisted of extending the 1st level drift (40-meter level) an additional 75 meters to the south. The decline was advanced further north in preparation for establishing a southerly turn to open new horizons below the present workings. 30 personnel were employed in May 2005. A raise was excavated to the surface to provide ventilation and an emergency escapeway. As of June 30th 2005, 33 underground diamond drill holes (totaling 3,095 meters), 9 surface diamond drill holes (totaling 3,797 meters), 550 meters of drifting, crosscutting, and decline ramping, and 40 meters of ventilation raise had been accomplished as part of the 2005 underground program (Tyhee Development Corp. Press Release, July 5th 2005). Metallurgical testwork of Ormsby zone ores suggested 95% gold recovery using a combination of gravity, cyanidation, and flotation methods (Tyhee Development Corp. Press Release, July 19th 2005).



Tyhee Development Corp. Ltd.

Figure 3. Ormsby zone portal and laydown, 2006.

In July 2005, an additional 190 meters of lateral advance was completed, including 65 meters of sub-drifting on the #30 zone and 57 meters on the #60 zone. The 75 meter ventilation raise to the surface was also completed. (Tyhee Development Corp. Press Release, Aug. 3rd 2005) Surface mapping and diamond drilling during the late summer of 2005 uncovered a 360 meter x 150 meter wide brecciated extension of the Ormsby zone. It was intersected by diamond drilling 250 meters below the surface (Tyhee Development Corp. Press Release, Aug. 16th 2005).

Stockpiling of ore began in the summer of 2005. In September, it was reported that 6,800 tonnes of ore from the #30 and #60 zones had been removed and stockpiled. The second-stage of underground development was completed in September. Total development accomplished during the 2005 program was reported as 700 meters of lateral work (drifting, crosscutting, and decline ramp) and 80 meters of raising, giving access to several gold-bearing zones to a depth of 117 meters below surface. The work successfully demonstrated the lateral and vertical continuity of the mineralized sections of the Ormsby zone and established engineering data on rock strengths, mining costs, and potential stope geometry. Diamond drilling, on surface and underground, established a vertical continuity of the ore to a depth of 700 meters. (Tyhee Development Corp. Press Release, Sept. 28th 2005). Mobilization of supplies and equipment over the winter road occurred in 2006 in preparation for a resumption of underground operations at a later date. This program was cancelled as resource and feasibility studies were prepared in 2006-2007.

Tyhee has completed over 130,000 meters of diamond drilling on the Ormsby-Discovery properties to the end of 2007. A resource estimate for the Yellowknife Gold Project (including Nicholas Lake Mine) announced on June 20, 2007 is 9.6 million tonnes (measured and indicated) grading 3.9 grams per tonne containing 1.2 million ounces. An additional 3.22 million tonnes of ore is estimated as an inferred resource, grading 3.41 grams per tonne (353,000 ounces of gold). (Tyhee Development Corp. Annual Report, 2007)

In 2007, Tyhee Development proposed an underground project; in 2008, because of economics, a combination open-pit underground gold mine was promoted. During 2008, an application was made to the Water Board authorities to bring a gold mine into production and the project is now undergoing environmental assessment.

Exploration Since Mine Closure

Not applicable.

References and Recommended Reading

DuPre, D.G. and Kirkham, G., 2004. An Estimate of the Gold Resource of the Yellowknife Gold Project. Tyhee Development Corporation, May 2004.


Mackenzie Valley Land and Water Board Files – Water License MV2002L2-0017 and MV2005L2-0003 (Tyhee Development Corporation)

Nass, C., 2001. Review of the Yellowknife Gold Project. Tyhee Development Corporation, March 2001.

Pratico, V., 2001. Report on the Gold Estimate of the Ormsby Zone at the New Discovery Mine. Tyhee Development Corporation, August 2001.

G.M.D. Resources Corporation Ltd. News Releases. 1994-1996.

Tyhee Development Corporation Ltd. Annual Reports & News Releases. 2003-2007.

	ORO LAKE <i>Minor Producer (Abandoned)</i>	Gold
		62° 38' 15" 114° 20' 00"

Years of Primary Development: 1942-1943

Mine Development: primary trench/pit

Years of Bulk Sampling: 1943

Bulk Sample: 4 tons shipped = 29 oz Au

Years of Production: 1943

Mine Production: ~15 tons milled = 17 oz Au

Introduction

This property was a very minor gold producer during World War II and has seen no development (aside from exploration work) since. It is located at Oro Lake, 19 kilometers north of Yellowknife, NWT. An old log cabin is maintained by snowmobilers as a shelter. The author hiked into the old site in August 2004.

History in Brief

The property was first staked in 1936 as the 'Chan' and 'Oro' groups of claims by men with the Great Slave Prospectors Trust. Minor surface development occurred in 1937-1938 under the direction of Lambert Turcotte for Chan Yellowknife Gold Mines Limited. The claims lapsed in 1940 or 1942.

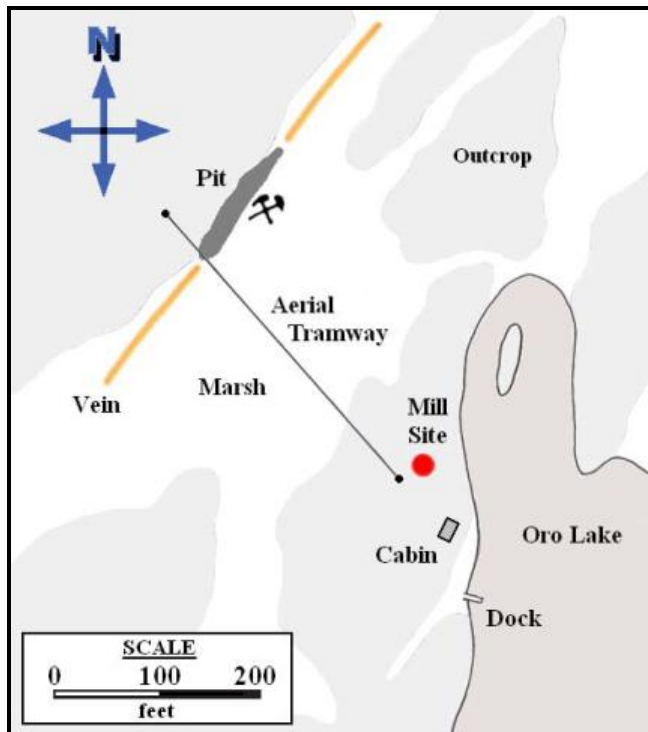


Figure 1. Oro Lake Mine property plan, 1940s.

The 'MMM' group was staked to cover the original gold showings immediately following the lapse. Jimmy Mason, Bill McDonald, and Frank Moyle were the participants in the staking. Some ore was brought by tractor to Con Mine in spring 1943. Bill McDonald erected a milling plant on property, probably in the summer of 1943, and milled some ore. In 1944 the claims were acquired by Lynx Yellowknife Gold Mines Limited and remained a focus of gold exploration in the Yellowknife gold belt for many years. Jimmy Mason later returned to the property in the 1980s to recover some of the old tailings.

Geology and Ore Deposits

Gold values are mainly concentrated in mineralized quartz veins contained in sheared and altered andesites. The andesite appears to be slightly mineralized (pyrite and chalcopyrite) and locally siliceous/silicified. The quartz veins are mineralized with pyrite, chalcopyrite, arsenopyrite, galena, and pyrrhotite. Most if not all the quartz veins with elevated to multi ounce gold appear to be narrow in the few inch to 3 feet range (Mason, 1945).

Bill McDonald (1942-1943)

The #1 vein was mined by trench and pit during the summer of 1942 or earlier by Jimmy Mason, Bill McDonald, and friends. Exact date is unknown but it is assumed that work was done over the summer months but prior to the spring of 1943 (unless the ore was blasted during the winter) (Mason, 1945). A small bulk sample of this ore was brought to Yellowknife by tractor in March 1943 for processing at Con Mine. Four tons were shipped and milled to produce 29 ounces of gold, with a calculated grade of 6.93 ounces per ton gold (Cominco Ltd., 1943).

Results of the bulk sampling suggested that milling of the remainder of the stockpiled ore was warranted. Approximately 15 tons of ore from the #1 vein were milled in a portable milling plant by Bill McDonald and friends

(Mason, 1945 and 1988). This plant was possibly the same as used at the Consolation Lake Mine the previous year, since the owners were the same group of people.

Ore was hand mucked from the surface trench and loaded into ore buckets attached to an improvised aerial tramway system. This tramway, consisting of a small winch and cable and at least two ore buckets, stretched from the south end of the pit across a marshy patch of ground toward Oro Lake to a spot about 75 to 100 feet behind the campsite.

Milling Plant

Before being processed, the ore was burned to prevent sliming during the tabling process. The ore was then broken into three inch pieces and run through a 4½ inch x 5½ inch jaw crusher, then a 24 inch x 12 inch Straub ball mill. This machinery was powered by a seven horsepower Wisconsin gas engine. Two sluice-boxes were built to receive the ball mill product, one of which was floored by 16 feet of rubber matting, and the other with 24 feet of corduroy cloth. These boxes were fitted side by side, and product from the mill was switched into either circuit after a few minutes when a visible concentrate was seen on the tables. This helped to reduce metal losses in the tailings.¹ Tailings were deposited down the hill into the lake.

The pit was dug down 15 feet on the vein from which all mill feed was derived. Dimensions of this pit today are approximately 50 feet long x 5 feet wide, and is filled with water. The campsite consisted of a single log cabin, built in 1937, 16 feet x 24 feet in dimensions. Primary operator during this term was Bill McDonald.

According to records of the Royal Canadian Mint, a shipment of gold from Mr. McDonald in 1943 contained 3-4 fine ounces of gold and 0-40 fine ounces of silver. Another shipment of gold bars from Mr. McDonald in 1944 contained 14 fine ounces of gold. It is unknown if either shipments were from the Oro Lake property, but it is believed to be so (National Archives of Canada).

The property was acquired by Lynx Yellowknife Gold Mines Limited (Jimmy Mason's company) in May 1944, but no further mining work was done. Diamond drilling, trenching, and geological investigations were conducted between 1944 and 1950 by the company (Mason, 1945 and 1988).

Jimmy Mason (1980s)

In 1984 or 1985, Jimmy Mason recovered 15 tons of old tailings from Oro Lake. A suction dredge was used to pump the tailings out. They were shipped to Trail, B.C. and averaged 3-3 ounces per ton gold (Mason, 1988).

Exploration Since Mine Closure

The area was re-staked in....? and optioned to Tyhee Development Corporation in June 2008. An exploration program is planned for the summer of 2008.

References and Recommended Reading

Cominco Ltd., 1943. Con Mine production records, 1943.

Mason, J.D., 1945. Report on the Property of Lynx-Yellowknife Gold Mines Limited. April 1945.

Mason, J.D., 1988. Notes on Oro Lake Property. Memo to Con Mine Geology Department, September 1988.

National Archives of Canada: Royal Canadian Mint Collection (*RG 120*)

¹ It is not officially known whether this layout and equipment was at use at the Oro Lake Mine, but since the owners of this property also owned the Consolation Lake Mine which produced the previous summer, it is assumed that the same equipment would have been utilized.



OUTPOST ISLAND

Producer (Remediated)

Gold, Tungsten, Copper

61° 44' 10" 113° 27' 30"

Years of Primary Development: 1936-1938, 1940-1942, 1951-1952

Mine Development: 525' vertical shaft, 5 levels; 134' inclined shaft

Years of Production: 1941-1942, 1951-1952

Mine Production: 23,580 tons milled = 506 tons concentrate (10,185 oz Au, 35,436 lbs WO₃, 119,763 lbs Cu)

Introduction

Outpost Island is located 94 kilometers southeast of Yellowknife, NWT in Great Slave Lake. The mine was originally a gold showing, but the presence of tungsten ores elevated the property into war-time service between 1941 and 1942. It reopened briefly in the early 1950s. The site suffered a fire in 1955, and the ruins were cleaned up by government crews in 1994-1995. It has not been visited by the author.

History in Brief

The 'Fox' group of claims was staked in July 1935 by prospectors of the Athabasca Syndicate over a showing of copper and gold. A new company was formed by members of the syndicate – Slave Lake Gold Mines Limited, with J.J. Byrne as president. Almost immediately, the N.A. Timmins Corporation Limited optioned the ground under an agreement that a certain amount of development would be undertaken on the claims. After sinking a shaft and erecting mine facilities, the erratic nature of the deposit discouraged the company from spending any further money. With the failure of the Timmins company to perform its option, Slave Lake Gold Mines Limited hired geology professor Dr. Hawley to examine Outpost Island's gold and tungsten potential. It was found that the tungsten ores could be economically extracted and that the potential for large amounts of tungsten on the property was great. In 1939, tungsten's market value was on the rise due to World War II.



The Precambrian magazine

Figure 1. *Outpost Island Mine, 1938.*

Re-construction of the property began in 1940 and production of gold, copper, and tungsten concentrates began in 1941. Financial difficulties resulted in the closure of the mine in the fall of 1942. Following the war, there was still a great need for tungsten products. Although one company failed to reopen the mine, Outpost Island did again become productive in 1951 through the work of the Tungsten Corporation of Canada Limited. This work stopped in 1952 and three years later the old property burned to the ground.

Geology and Ore Deposits¹

The Outpost Islands are underlain by metamorphosed sedimentary rocks of early Precambrian age, belonging to the Wilson Island phase of the Point Lake-Wilson Island Group. Mineralized zones cover an area over 7,000 feet long and 750 feet wide encompassing four islands of the Outpost Island group. The zones dip southerly at angles between 75 and 85°. Most of the ore zones are contained in silicified, sheared, and/or brecciated greywackes, and pelitic schists. The mineralized zones consist of pyrite, chalcopyrite, native gold, scheelite, ferberite, marcasite, and bornite.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Other metallic metals include specularite, magnetite, and covelite. Most of the zones mined were between two and three feet wide, with strike lengths up to 180 feet. Near surface, the main zone was reported to be as much as six feet wide. Mineralization was reported to be confined to the narrow intervals mined, and economic values dropped off considerably outside these zones. Most work was done on the #7-8 shear zone, where the #1 ore shoot was mined to a depth of 525 feet.

N.A. Timmins Corporation Limited (1935-1938)

In September 1935, six veins with high assays were identified. In November 1935 the N.A. Timmins Corporation Limited optioned the claims under an option agreement with Slave Lake Gold Mines Limited, in which the N.A. Timmins company was required to conduct underground exploration and if warranted install a mill (The Northern Miner, Nov. 21st 1935). A crew of 12 men was assembled during the winter of 1935-1936 under the direction of Len G. Smith, W. Coughlin, and Alphonse Pare to begin camp construction and initial exploration. Diamond drilling was conducted in February-March 1936. 14 holes were put down with footage totaling 2,600 feet (The Northern Miner, Mar. 4th 1937).

1936 Shaft Sinking

The first stages of the 2-compartment shaft (#1 shaft) were sunk using handsteel to a depth of 66 feet starting in August 1936. A gas hoist was used. One level was opened up at the 50-foot level, and 60 feet of drifting was conducted east and west (The Northern Miner, Mar. 4th 1937). A bulk sample weighing 1,063 pounds grading 3.15 ounces per ton gold and 1.20% tungsten oxides (WO₃) was removed in 1936 (Jolliffe, 1942).

1937-1938 Development

A complete mining plant was delivered to the property in 1937 and the shaft was sunk to 450 feet depth during 1937-1938. Levels were established at 125-, 200-, 325-, and 425-foot depths and drifting followed the length of the vein east and west (Lord, 1941). Equipment in use during this program included one 80 horsepower Waukesha-Hesselman diesel engine with 63 KVA generator, one 80 horsepower Waukesha-Hesselman diesel engine with Canadian Ingersoll-Rand 315 cubic feet per minute air compressor, one 15 horsepower wood boiler, a 1-drum 6x7 Mead-Morrison air hoist, two N-69 drifter/sinker drills, two Ingersoll-Rand stoper drills, two 20 horsepower electric pumps, one 7 inch x 10 inch Denver jaw crusher, one Prospector diamond drill, a complete sawmill, and a Caterpillar RD-4 tractor (NWT Archives). Work ceased in March 1938 due to poor assay results, erratic ore zones, and an uncommitted work force. Underground work from 1936 to 1938 outlined 20,000 tons of gold ore. Total lateral work amounted to 1,389 feet of drifting and crosscutting and the company spent over \$280,000 during the 18 month program. The agreement with N.A. Timmins Corporation Limited was terminated and the property reverted back to Slave Lake Gold Mines Limited. All equipment was sold (Lord, 1941).

Slave Lake Gold Mines Limited² (1938-1942)

Slave Lake Gold Mines Limited began a re-evaluation of the deposit during 1938. After consulting with geologist Dr. Hawley of Queen's University, it was recognized that tungsten was important in the Outpost Island ore body. An assay plant was shipped to the mine in the fall of 1938 and testing began. The ore contained up to 3% WO₃ with grade increasing at depth (Bruder, 1941).

Company president J.J. Byrne quickly raised the funds to bring the mine back into operation. With the start of war in 1939 the price of tungsten rose to \$25 per unit (1 unit = 20 pounds). The first construction crews arrived in September 5th 1940 under the direction of J.C. Byrne, Norman Byrne, and Art Dion. Priorities were to recondition the old camp and plant buildings, de-water the shaft, and construct a milling plant. Work continued throughout the season despite bad weather and a shortage of building materials. The lumber shortage was overcome by setting up a portable sawmill, and dismantling old buildings for use in the new. The underground was pumped out early in December 1940 and mining operations began. One of the first development drives was a short westerly drift on the 50-foot level, from which a raise was put to surface, to tap the ore dump from previous operations. Drifting on the 125-foot level east of the shaft opened up an 80 foot length of ore grading 0.80 ounces per ton gold across a width of 30 inches and on the 200-foot level east of the shaft, a section of ore 92 feet long was opened up grading 0.90 ounces per ton gold across 30 inches. Raising was also initiated on the 200- and 300-foot levels within the ore body (Bruder, 1941).

Production Begins

All facilities were completed early in January 1941 and production was ready to start. The mine was brought into production during a four-month period, a major accomplishment at a time with limited resources and a small

² Reorganized as International Tungsten Mines Limited in May 1942

workforce. Milling started January 5th 1941 and the first gold was poured in February. Source of ore was from stopes on the 1st and 2nd levels of the mine (Bruder, 1941).

Milling Operations

By May 1941, additions and alterations to the circuit made it possible to treat and recover tungsten from the processed ore. During August 1941, further modifications were performed to ensure a greater recovery of tungsten, and by May 1942, the following circuit was employed:

Ore was hoisted in mine cars and pushed directly into the headframe coarse ore-bin. Crushing to ¾ inch was achieved by a 9 inch x 15 inch Denver jaw crusher. Following a discharge from the 90-ton fine ore-bin, ore was ground to 70% minus 200-mesh in a 4-½ foot x 5 foot Allis-Chalmers ball mill loaded with 5 tons of ball charge, then over a Denver duplex mineral jig, and through an Akins spiral classifier. Overflow from the classifier was passed over three blanket tables where a gold concentrate was taken. Table tailings were conditioned, and then treated in a bank of four Denver flotation cells. A copper concentrate was removed and sent for thickening then to a 3 foot x 4 foot Allis-Chalmers regrind ball mill. Gold and copper concentrates were put through an amalgamation unit separately, with the gold poured in the refinery, and the copper concentrate dried and sacked. A tungsten concentrate was recovered by passing flotation tailings over two Wilfley tables. Slimes from the tables were sent back to a second set of four Denver flotation cells and a final concentrate recovered. Final tailings were jettisoned from the mill into a small bay within Outpost Island and impounded (Bruder, 1942).



Bob Duncan photo

Figure 2. *Outpost Island Mine, 1942. Note headframe being reconstructed for future deepening of shaft.*

Power Plant

The mill required an input of over 200 horsepower, and this was supplied by a Ruston-Lincoln diesel generator of 165 KVA, with backup power available through a 35 horsepower Ruston diesel engine and 25 KVA generator. A 150 horsepower Crossley diesel engine operated a 600 cubic feet per minute Canadian Ingersoll-Rand air compressor for underground drills and the air hoist: a Canadian Ingersoll-Rand 6x8 double-drum unit. A 40 horsepower wood-fired boiler supplied heat to all buildings on Outpost Island, with the exception of the camp buildings that were built on a separate island (Bruder, 1941).

Camp and Crew

Camp facilities in 1941 consisted of one or two small bunkhouses and a cookery, located on East Island. Men would get to work on boat or skiff during the summer and walk over the ice in the winter. A recreation hall was built in 1942

and a larger bunkhouse was also added to this camp. Before the end of 1941, plans were made to erect a new camp on the main island near the mine plant. In September 1941, 60 men were employed at the mine, of which 18 were in the mine and 10 had duties in the mill. There were four married couples and four small children living on the property. Family accommodation was adequate at the camp (Meikle, 1941).

General Operating Data

The location of Outpost Island required three different types of transport depending on the season. Plane service was provided regularly, twice a week. Food supplies and other essentials were delivered at 52 cents per pound in 1942. In the winter, skied planes were used to bring in small freight at a similar cost. Heavy equipment was brought in by Cat train for \$130 per ton. During the summer, heavy equipment and bulk supplies could be brought in by barge from Fort Smith, NWT at a cost of \$34 per ton. No hospital serviced the mine, but the doctor in Yellowknife was available for monthly checkups and emergencies (Bruder, 1942).

1941 Operations

Flotation equipment additions to the mill in April 1941 allowed for the recovery of a tungsten and copper-gold concentrate (The Northern Miner, Apr. 24th 1941). However, problems arose with the amalgamation circuit owing to the complex nature of ore being treated (arsenic, antimony, and some oxidized gold). Only after considerable experimentation were gold recoveries raised to 70%. It was determined that certain sections of the stopes had ore with different chemical properties, reacting unfavourably to amalgamation. Because of the fine grinding required to recover the gold, it was difficult to recover tungsten economically. A number of changes were made to the mill during July-August 1941. The company announced a plan to focus on mining the high-grade gold deposits above the 200-foot level and then switch the flowsheet to focus on the tungsten ores. It was believed that economic gold would not be found below the 200-foot level because of a shear cutting through the vein above the 325-foot level (The Northern Miner, Aug. 7th 1941). Underground development to Sept. 1941 is listed in **Table 1**.

<i>Level:</i>	<i>Depth:</i>	<i>Lateral Development:</i>
1	50'	127'
2	125'	339'
3	200'	1,202'
4	325'	419'
5	425'	361'
<i>Total:</i>		<i>2,448'</i>

Table 1. Total underground development at Sept. 1941, #1 shaft. (source: Meikle, 1941)

Underground work continued early in 1941, with focus on developing ore on the 125- and 200-foot levels in the east and west drives. Over 250 feet of drifting was reported to the end of March 1941, of which 200 feet was within ore averaging 0.80 ounces per ton gold across a width of 30 inches (The Northern Miner, May 22nd 1941). By October 1941, the 325- and 425-foot levels were dewatered and rehabilitated in preparation for mining. It was hoped that tungsten values would improve at depth. Work during September-October 1941 showed that gold values did persist to the lower levels. It was reported that an old drill hole west of the shaft returned an assay of 13 ounces per ton gold at a depth of 625 feet (The Northern Miner, Oct. 9th 1941).

Drifting and raising east of the shaft on the 325-foot level opened up 90 feet of new ore, with a large percentage of this material containing quartz. On the 425-foot level, the east drift was continued 185 feet where the shear zone was well mineralized with widths up to six feet. However, the bottom levels had not been very productive at the end of the year. There was still hope that more ore would be encountered at depth, and starting in December 1941 an inclined winze was sunk from the 425-foot level of the #1 shaft workings. It was designed to access the 525-foot horizon to develop deeper ore and where, if warranted, the shaft could be deepened (Slave Lake Gold Mines Ltd. Annual Report, 1941).

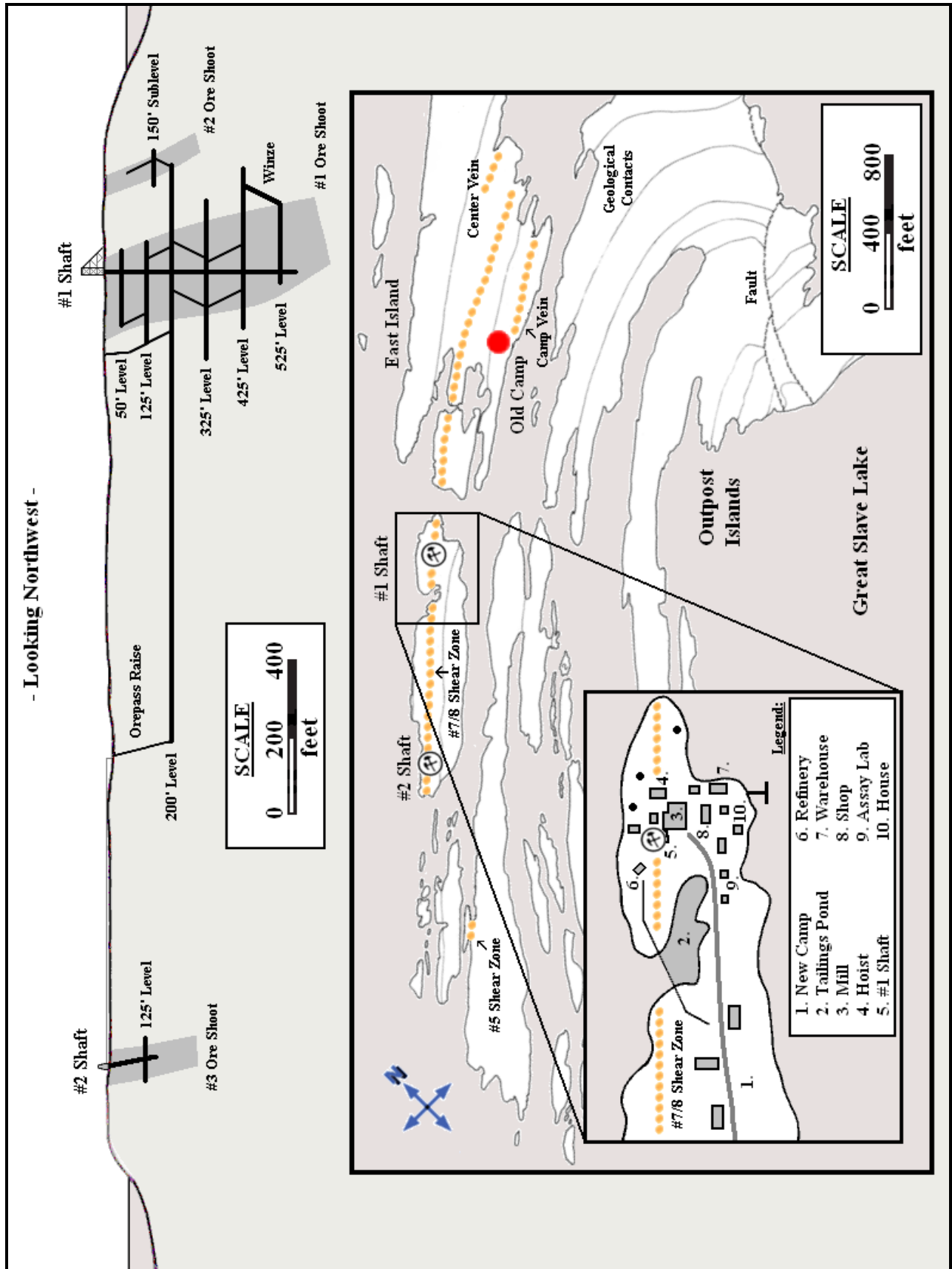


Figure 3. Outpost Island property plan with underground longitudinal section of the #7/8 shear zone, c.1942.

In the summer of 1941, a second shaft (#2 shaft) was begun on the west-end of Outpost Island to explore the extreme western portion of the #7-8 shear zone. During 1941 it was sunk to a depth of 22 feet with handsteel (Slave Lake Gold Mines Ltd. Annual Report, 1941). It was continued to an inclined depth of 134 feet in 1942 with 80 feet of work done on the 125-foot level. A drift on the 200-foot level of the #1 shaft workings extended 900 feet westerly towards the #2 shaft and a raise was driven to the surface. It joined up with a rail-trestle way that transported ore from the #2 shaft to the raise. Minor mill feed was derived from this shaft's workings (Meikle, 1942). A visit in September 1941 by Mackay Meikle, the federal mining inspector, highlighted many problems associated with the operations (Meikle, 1941). One major change that came out of Mr. Meikle's visit was the construction of a new camp on the main island. The older camp on East Island was in bad state of repair and generally unsanitary. A start was made on building new camp buildings near the mine plant in 1942, but was not completed before the mine closed.



NWT Mining Heritage Society photo

Figure 4. *Outpost Island Mine mill, miner's dry, and headframe, 1941.*

1942 Operations

Going into the New Year, the outlook for the tungsten producer did not look good. The company was still in the process of reform and money to keep the operation alive was not being raised. Ore reserves were quickly being depleted and exploration of the 325- and 425-foot levels were not turning up decent gold grades. Sinking of the #1 winze below those levels continued, as did efforts to locate ore in the #2 shaft area. In May 1942, the company reorganized as the International Tungsten Mines Limited company to better reflect its goal as a future war metal producer. The company tried to acquire loans from the government in order to continue development at Outpost Island. All requests were turned down by the Metals Control division in Ottawa. With proper funding, the company believed they could install better milling machinery and recover the high-grade tungsten deposits located on the lower levels more efficiently.

In July 1942, the #1 winze from the 425-foot level reached the 525-foot level. A drift was driven westerly beneath the shaft, and then a raise was driven up to intersect the bottom of the shaft, deepening the shaft itself to 525 feet. By September 8th 1942, all ore above the 425-foot level had been mined out and stoping on the 525-foot level had just commenced. In September 1942, there were 60 persons on the payroll at Outpost Island, of which 25 were in the mine, 10 in the mill, and 25 on surface. There were 8 women and 11 children living on the property. Staff included the following: J.C. Byrne, manager; N.W. Byrne, engineer; C.R. Carlson, mill superintendent; H. Lund, mine captain; D. Hamilton, accountant; J.A. Evans, assayer; Art Dion, surface foreman; and A. Sequin, master mechanic (Meikle, 1942).

On orders from head office, the mine was to keep the mill running to provide for payroll and other expenses, but mine manager J.C. Byrne shut the mill down August 1st 1942 and began plans to sink the #1 shaft beyond the 525-foot level. This work did not get very far before money ran out to continue operations. In September 1942, without company support and lacking supplies, the mine crew decided to abandon Outpost Island using a homemade barge, leaving on September 24th (Price, 1967). Production from 1941-1942 is listed in **Tables 2 and 3**.

Level - # of Stopes:	Hoisted:	Sent to Waste:	Sent to Ore Dump:	Sent to Mill:
125 – 1:	6,743 tons	338 tons	80 tons	6,325 tons
200 – 3:	4,807 tons	269 tons	302 tons	4,236 tons
325 – 5:	3,314 tons	299 tons	3 tons	3,012 tons
425 – 2:	2,264 tons	122 tons	-	2,142 tons
525 – 1:	498 tons	121 tons	-	377 tons
<i>Previous Ore Dumps:</i>	1,691 tons	-	-	1,691 tons
<i>Development:</i>	2,397 tons	-	-	2,397 tons

Table 2. *Source of production ore, 1941-1942. (source: Buffam, 1942)*

Ore Milled:	20,324 tons
Gold in Bullion:	8,809 oz
Gold in Concentrates:	1,096 oz
Tungsten-Gold Concentrates Shipped:	69 tons (22% tungsten, 2.4 oz/ton gold)
Tungsten Oxides in Concentrates:	27,700 pounds
Copper-Gold Concentrates Shipped:	312 tons (17% copper, 3.2 oz/ton gold)
Copper in Concentrates:	112,863 pounds

Table 3. *Outpost Island production, January 1941 to August 1942. (source: Buffam, 1942; Lord, 1951³)*

Philmore Yellowknife Gold Mines Limited (1946-1947)

Following the end of World War II, there was renewed interest in tungsten deposits around Canada. In 1946-1947, Philmore Yellowknife Gold Mines Limited attempted to reactivate the Outpost Island property but with no success. It was planned to re-mill the old tailings pond that held a substantial amount of gold and tungsten values. It was anticipated that exploration would uncover additional underground reserves below the 525-foot level. During 1947 some mill cleanup and recovery of tailings was accomplished, and it would appear as though the company recovered 27 ounces of gold from this work (National Archives of Canada).

A total reserve of about 10,000 tons grading 0.60 ounces per ton gold and 0.60% WO₃ were available within the current underground workings, and another 12,000 tons probable reserves located at #2 shaft and below the 525-foot level (Way, 1946). Despite these encouraging estimates, the company was unable to raise the finances needed for the project and no work was being done in 1948.

³ Lord's production statistics were apparently based on data supplied to Statistics Canada. The data conflicts with production numbers reported by Buffam (1942) which would normally be considered more accurate since he was going by mine records. But Buffam's metal content of concentrates was based on mine assays, and the numbers reported by Lord (1951) would have likely been the correct returns following smelting of the concentrate. Weight and assays of shipped concentrate is from Buffam and is considered correct.



Western Miner magazine photo

Figure 5. Crews at the Outpost Island Mine, 1951.

Tungsten Corporation of Canada Limited (1951-1952)

Tungsten Corporation of Canada Limited was formed in 1950 to renew the plans of Philmore Yellowknife. During the year, a large amount of diamond drilling (at least 5,500 feet) was completed exploring the island, with encouraging results obtained. Two samples were taken from the tailings ponds to test for gold content (The Northern Miner, Nov. 9th 1950). This new period of exploration was spurred by the Korean War and new demand for tungsten. Since the last work done in 1947, the property had been vandalized and it was estimated that \$300,000 would be required to fix up the property. Included in this work were final touches to complete the new camp, abandoned in 1942 when the mine first closed. The new camp consisted of a two-storey bunkhouse and two-storey cookery supplied with heat and water from the mine's boiler house.

Mill Operations

Test milling of tailings pond material began in October 1951 but initial operations were sporadic. Steady production was achieved on January 18th 1952 and from that point on the operation was marred by only minor shutdowns (The Northern Miner, May 1st 1952). Tailings were blasted from the frozen pile and slushed into the milling circuit. Very little change from the original flowsheet was required. Improvements included the addition of a set of Craig-type cleaning tables, designed to obtain very high-grade tungsten concentrate (Irwin, 1952). No gold bullion was produced during these years, as the circuit had been modified to primarily produce tungsten concentrate and a gold-copper concentrate.

In April 1952, the mill was operating at 25 tons per day with mill feed derived from impounded tailings assaying 0.30 ounces per ton gold and from 0.40 to 0.70% WO₃. The gold concentrate produced averaged 6.0 ounces per ton gold and the tungsten concentrates assayed 18 to 20% WO₃. Gold recovery was expected to improve with the installation of a Denver mineral jig (Irwin, 1952). It was later found that the problem was an issue of sizing, so a small ball mill was added to the circuit as well. In May 1952, tungsten recovery was reported to be 92% up from the 85% recoveries attained during test milling, and gold recovery was 60%. The mill was processing about 25 tons per day on average with 40 tons per day the maximum achieved in May 1952 (The Northern Miner, May 1st; May 29th 1952). In March 1952 the tungsten concentrates assayed 16% WO₃, slightly above commercial grade. The company hoped to achieve a 30% grade (The Northern Miner, April 3rd 1952). Tungsten concentrates were to be shipped to California and gold-copper concentrates to Tacoma, Washington (The Northern Miner, May 1st 1952). With one exception, old equipment at the mine held up well despite ten years of inactivity. One major problem with the old power plant was that the old wood-boiler, which was unable to produce enough heat. A new 35 horsepower oil-fired boiler was shipped to the island in March 1952 to supplement heat power (The Northern Miner, Feb. 14th 1952).

Crew

H.W. McKittrick was in charge of operations as mine manager, along with Bernhard Day as consulting engineer (Irwin, 1952). During this time there were about 45 men employed, along with 7 or 8 families living at the mine.

Mining operations began underground on April 24th 1952 and mill-feed from the 200-foot level was processed in May 1952, where a scheelite zone was discovered earlier in the year. Ore grades of 0.75% WO₃ were encountered (The Northern Miner, May 29th 1952; June 12th 1952). In August it was announced that milling and mining from the 200-foot level had stopped in favor of exploration on the 525-foot level of the mine. A 60 ton test run of ore was conducted from the 525-foot level in July, but no information is available on this bulk sample. The operation was running at a loss because of high costs, and unless new high-grade ore was found, the future of Outpost Island was uncertain (The Northern Miner, Aug. 21st 1952). Some diamond drilling (6 holes) was conducted beneath the 525-foot level of the mine in the summer of 1952 and although good gold values were encountered to 665 feet depth, tungsten values were erratic and it was doubtful if there was enough ore to pay for development. The indicated ore shoot was estimated to contain 40 to 50 tons per vertical foot, which was not considered sufficient to warrant

deepening the shaft (The Northern Miner, Feb. 19th 1953). This, plus poor markets following the end of Korean War hostilities, resulted in a suspension of all work.

1951-1952 Production

Mill feed, believed to include all production commencing in October 1951 and ending July 1952, was 3,256 tons of ore (3,196 tons to end of June plus the 60-ton sample in July). 1,549 tons of this was tailings material produced until the end of April 1952, when underground ore was introduced to the mill. Between May and the end of June 1952, 1,647 tons of combined underground ore and surface tailings were treated in the mill. A 60 ton test sample was milled in July 1952 from the bottom level of the mine (The Northern Miner, Aug. 21st 1952).

Year:	Ore Milled:	Gold Produced:	Total Concentrates Shipped:	Tungsten Oxides Produced:	Copper Produced:
<i>1941-1942</i>	20,324 tons	9,905 oz	381 tons	27,700 pounds	112,863 pounds
<i>1951-1952</i>	3,256 tons	280 oz	125 tons	7,736 pounds	6,900 pounds
<i>Total:</i>	<i>23,580 tons</i>	<i>10,185 oz</i>	<i>506 tons</i>	<i>35,436 lbs</i>	<i>119,763 pounds</i>

Table 4. *Total Outpost Island Mine production.*

The following concentrates were on hand in August 1952: 30 tons tungsten concentrates and 95 tons copper-gold concentrates. This material was to be shipped immediately for sale (The Northern Miner, Aug. 21st 1952). According to Statistics Canada, the Northwest Territories produced 6,900 pounds of copper and 7,736 pounds of tungsten oxides in 1952, presumably from Outpost Island since it was the only mine producing these commodities in that year (Statistics Canada, 1957). Gold production, recovered from concentrates, is officially unknown, but an estimate of 280 ounces has been made by deducting known gold production at the Northwest Territories four other gold mines operating in 1952 from the total gold produced in the NWT, according to Statistics Canada records. This number also corresponds very closely to the concentrate to gold recovered ratios of 1941-1942 production (Statistics Canada, 1957). Total production during both periods of operation at Outpost Island Mine is listed in **Table 4**.

Exploration Since Mine Closure

Tungsten Mining Corporation of Canada Limited was reformed into Mount Wright Iron Mines Limited in 1958. In 1982, Etthen Mines Limited and Dave Smith proposed leaching the old mine tailings to recover the gold content but this plan fell through because of environmental concerns. In 1987, the claims were optioned to Rapparee Resources who conducted a small diamond drill program. The results were insignificant and the original claims lapsed about 1990 (Poirier, 1988). The new 'Fox' claim was staked in 1999 by Dave Nickerson to cover the old minesite and tailings pond. Hendrick Falck also recorded the 'Outpost' claim in 1999 to cover the entire Islands, but this claim lapsed in 2001. Nickerson was primarily interested in recovering gold values from the old tailings pile (Nickerson, 1999).

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PEG TANTALUM

Minor Producer (Abandoned)

Tantalum

62° 44' 30" 113° 06' 45"

Years of Primary Development: 1945-1947

Mine Development: one primary open cut on #3 dyke

Years of Production: 1946-1947

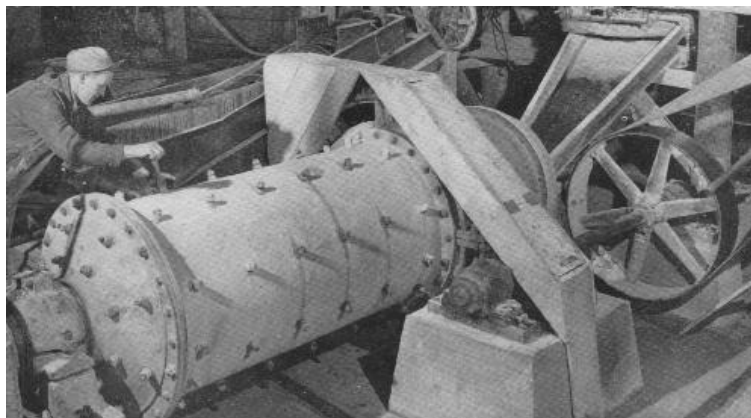
Mine Production: 940 tons milled = 3,750 pounds conc. (4% Ta, 14% Cb)

Introduction

This small tantalum-columbium/nobium mine is located on the northeast side of Upper Ross Lake, 70 kilometers northeast of Yellowknife, NWT. The concentrating plant operated for only a short time during periods in 1946 and 1947. The site was largely destroyed in the 1998 forest fires, although some ruins were spotted on an aerial pass over the site in June 2003. Access is by floatplane. An ice-road passes through Ross Lake, seven kilometers west of the old mine, and snowmachine trails likely head into Upper Ross Lake.

History in Brief

The tantalite-columbite showings were first discovered in the Upper Ross Lake area in 1943 by members of the Geological Survey of Canada during a strategic metal investigation of the Yellowknife region. The 'Peg' claims were subsequently staked by J.R. Saunders. Peg Tantalum Mines Limited acquired the 'Peg' and other claim groups in 1944 and placed a small milling plant into operation in 1946 and 1947. Although more assessment work was done by various companies into the 1950s, no further mining or milling operations have been conducted at the property since 1947.



Nationwide Minerals Ltd.

Figure 1. Rod mill under adjustment at Peg Tantalum Mine, 1940s.

Geology and Ore Deposits

Columbite-tantalite mineralization occurs within pegmatite dykes that cut dioritic dykes which in turn intrude granodiorite and quartz-mica schist. The ore bearing pegmatites are composed mainly of feldspars, quartz, and muscovite, with various concentrations of beryl, tantalite-columbite, tourmaline, spodumene, amblygonite, lithiophilite, and lazulite. The #1 and #3 dykes have been the furthest developed by mining and milling operations. The #1 dyke is reported 110 feet long striking northeast and dipping southeast. At the surface, the dyke has a width of 7 feet. The #3 dyke is nearly 200 feet long and dips 50° east, with a north strike. The dyke is cut by a fault (Lord, 1951).

Peg Tantalum Mines Limited (1945-1947)

Late in 1945, a 15 man crew under the direction of J.C. Finnan was completing the mill foundation and camp construction at the 'Peg' claims, in preparation for equipment installation (The Yellowknife Blade, Nov. 28th 1945). A contract for the cutting of 500 cords of local timber was made, and machinery was on its way to the property by Cat train in 1946. Spring break-up and the resulting transportation difficulties necessitated the airlift of this equipment (by Canso aircraft) from Tibbett Lake to the property (Lord, 1951).

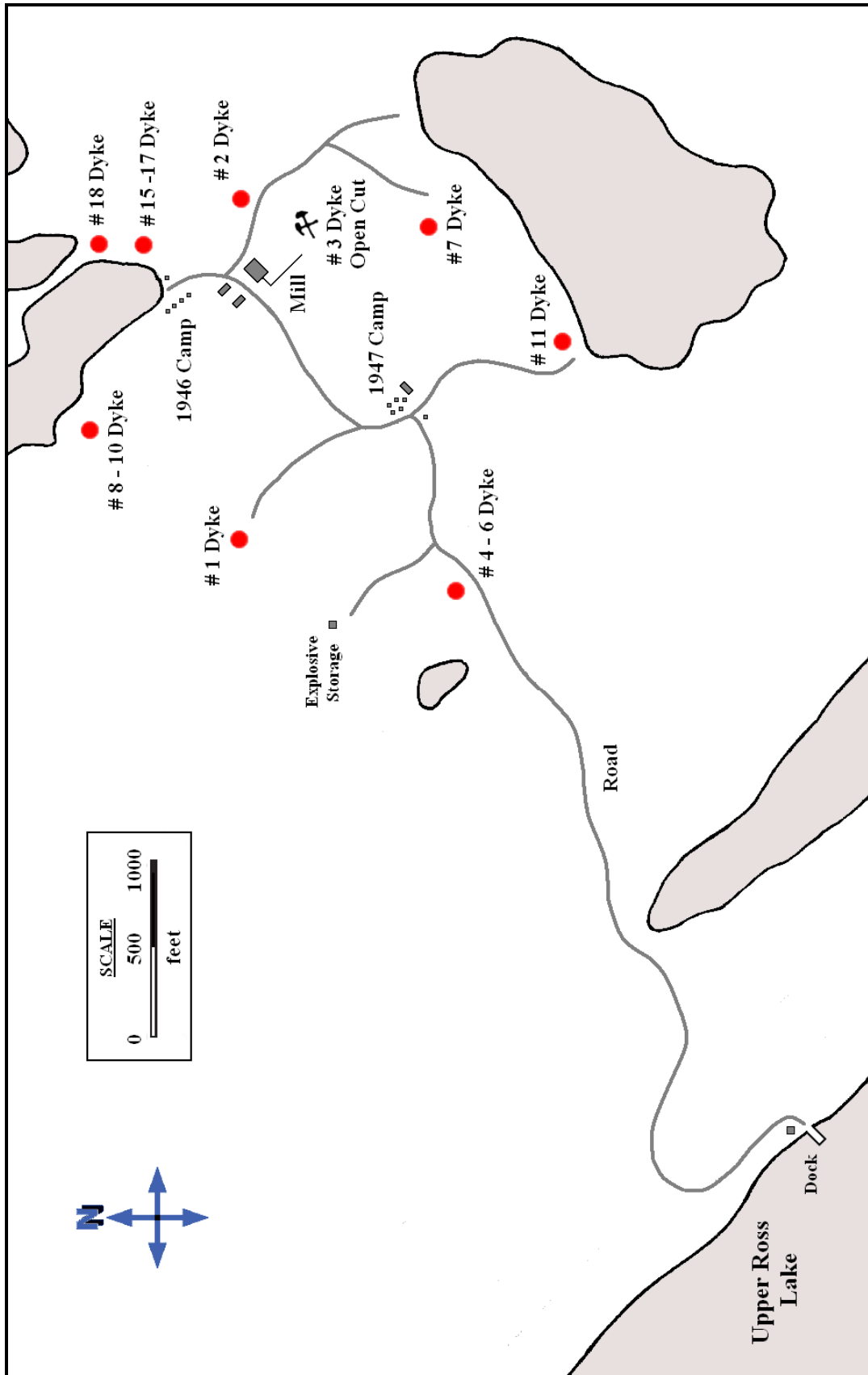


Figure 2. Peg Tantalum Mine property map, 1947.

1946 Operations

In September 1946, a crew of 20 men were at work erecting the mill plant and pit blasting on the #1 and #3 dykes. J.C. Finnan was manager in charge of operations. Five hundred tons of ore were stockpiled. Mill test runs were conducted over a five week period in September and October 1946, during which time about 200 tons of ore were processed from the #3 dyke, and 150 pounds of concentrate were recovered. The company shut down the operation due to mechanical difficulties within the plant and the inability to obtain a clean concentrate. The lack of certain equipment was responsible for poor recoveries, and the plant had difficulty operating in the winter because of freezing water lines. Work during the winter of 1946-1947 was focused on installing new equipment in the mill and building a better campsite. Mining development was not conducted because of winter conditions (Bateman, 1946; Christie, 1946; Lord, 1951).

1947 Operations

Milling was resumed in April 1947 with new equipment including classifier and screens. Milling continued, with interruptions, until July 8th 1947. J.C. Doyle was manager in charge. Mining focused on extracting ore from the #3 dyke (Lord, 1951).

Tantalum Refinery

In 1946, an associated company, Tantalum Mining and Refining Corporation of America Limited, raised funds to construct a high-tech tantalum refinery in Edmonton, Alberta. It is reported that the plant started operations in May of 1947 using Peg Tantalum concentrates as feed (The News of the North, May 16th 1947).

Milling Plant

The tantalum-columbium concentrating plant (50 tons per day) went into operation late in 1946. It was modified from its original design early in 1947, and the following is a description of the plant at shutdown in July 1947. Mill feed was entirely from the #3 dyke, located 400 feet southeast of the mill. An inclined timber trestle with 36 inch gauge track connected the #3 dyke pits with the milling plant, on which a small ore car was employed to transport ore. This car may have been pulled up the track using a winch, because at the head of the trestle there was a 'headframe' type structure that must have been used for this purpose. The trestle-system may have been used briefly in 1946, but it was impractical and by 1947 was no longer being used. Instead, ore was transported from the #3 dyke to the mill by a Cat D-2 tractor and sleds. The ore was crushed to 1-¼ inch size by a 7 inch x 12 inch Wettlaufer jaw crusher, with the product conveyed to a 200 ton fine ore-bin. It was then fed into a Hardinge ore-feeder before being processed in the 3 foot x 6 foot Hardinge rod mill, from which material passed over an 8 inch x 12 inch Denver mineral jig and then through screens, with the resultant 20-mesh passing through a Fahrenwald sizer and over two Wilfley tables, each producing a middling, concentrate and a tailing. Tailings were discharged from the plant, and the middling, combined with plus 20-mesh material from the screens was treated in a 16 inch Denver spiral classifier. Overflow from the classifier was re-tabled, and the final concentrate dried and bagged for shipment. The tailing product was impounded with the possibility of recovering the beryl not recovered during milling. The overall recovery of the metals was said to have been very low due to an ineffective milling plant. Only a middling concentrate, less valuable than a high-grade product, was recovered (Bateman, 1946; Christie, 1946; Lord, 1951).

Equipment

The mill equipment was directly operated by two 43 horsepower Cat diesel engines through V-belt and drive shaft. One engine drove the crusher and conveyor-ways, while the other operated the rod mill, classifier, and other units. Water for the mill was pumped from a small lake north of the site. Air for quarry operations was supplied by a portable Gardner-Denver 365 cubic feet per minute air compressor driven by a Cat D-13,000 diesel engine. Other mining equipment included a Warsop gasoline rock-drill and a Cat D-2 tractor. Storage for 6,600 gallons of fuel was available (Bateman, 1946; Christie, 1946; Lord, 1951).

Camp and Plant

The original 1946 camp consisted of tent frames on the shore of a small lake north of the milling plant. During the winter of 1946-1947, while the mill was shutdown, crews built a new camp along the road to Ross Lake, about 1,000 feet west of the mill site. The new camp consisted of a frame cookery building and several tents on frames. The mine and milling plant consisted of an office building, a combined blacksmith shop and warehouse, and a 50 foot x 70 foot mill building, all of frame construction (Lord, 1951; Christie, 1947).

1947 Production

Between April and July 1947, 740 tons of ore were milled to recover 3,600 pounds of tantalum-columbium concentrate (Lord, 1951). Official assay results of these concentrates are not known, but in July 1947 *The Northern*

Miner reported the middling concentrate to assay 4% tantalite and 14% columbite. A concentrate of 30% was desired to make the mine economical. The Peg Tantalum company reported that more research in ore dressing would be needed before any further recovery of tantalite ores would be done at the Yellowknife property. Metal content of the Peg Tantalum concentrates has never been reported (*The Northern Miner*, July 31st 1947). Production operations ended in July 1947. The work force was reduced to nine men who continued mining operations at the #1 dyke and conducted some test milling, operating the machinery as a sampling plant up until September 1947 (Lord, 1951).

Pit Excavations

Development work at Peg Tantalum has been confined to the #1 and #3 dykes. The #1 dyke was developed by a pit 15 feet x 6 feet and about 2 feet deep. The #3 dyke had a pit 50 feet long, 25 feet wide, and sloping from 2 to 10 feet in depth. Other pit blasting had been completed on the #8 and #11 dykes, but no mill feed was derived from those workings. Mining was focused on the wider sections of the dyke where grades were highest (Lord, 1951).

Exploration Since Mine Closure

In 1950, the property was acquired by Nationwide Minerals Limited. Metallurgical testing of the concentrates and ores was conducted during 1950-1952; the most significant result of this work was in 1952 when a 1,535 pound sample of ore was sent to the Department of Mines and Technical Surveys in Ottawa. The ore was concentrated into a much smaller sample (3.28 pounds), assaying 56-68% tantalum-oxides (Ta_2O_5) and 8.32% columbium oxides (Cb_2O_5). These tests proved that a marketable product (of the day) could be obtained from low-grade material (Godefroy, 1952). Tantalum Mining Corp. Limited later investigated the mineral claims and conducted an extensive surface prospecting and sampling program in the 1960s (Mosher, 1969).

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PENSIVE LAKE

Minor Producer (Abandoned)

Gold

62° 44' 15" 113° 21' 15"

Years of Primary Development: 1939, 1941

Mine Development: 20' deep open cut with 50' vertical shaft

Years of Production: 1939, 1941

Mine Production: 91 oz Au, 10 oz Ag

Introduction

The small mine site is located northeast of Upper Pensive Lake, 60 kilometers northeast of Yellowknife, NWT. It was a very small gold operation of the late 1930s and early 1940s with minor production and development. The site was visited in July 2003 by the author.

Brief History

The Pensive Lake region was the site of a little gold rush during 1938, at which time prospectors working for Harry Ingraham staked a large group of claims between Upper Pensive and Dome Lakes. Early development focused on claims west of Dome Lake, but in early 1939 focus shifted to the 'Rare' and 'Ness' groups on the north side of Upper Pensive Lake. The property was originally owned by Harry Ingraham Trust Limited, but during December 1938 a new company was formed as a kind of subsidiary – Pensive Yellowknife Gold Mines Limited – with an Alberta charter. An open cut was begun in 1939 and a small mill was erected. The open cut was later extended as a short shaft, and some gold was produced in the fall of 1939 and 1941. A new company with the same name was formed in 1944 (Dominion charter) to conduct exploration of the property, but the mine did not reopen, possibly due to a 1947 fire that destroyed the headframe and equipment at the shaft site.

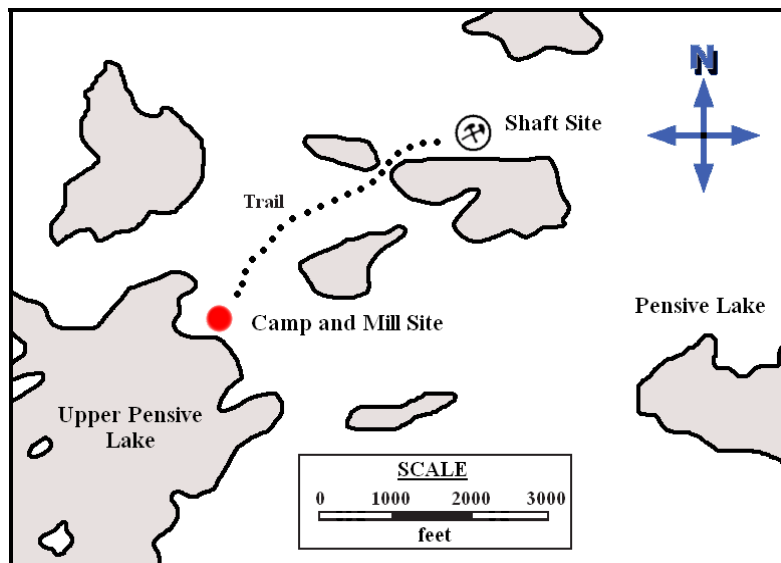


Figure 1. *Pensive Lake Mine location.*

Geology and Ore Deposits

The property is underlain by intricately folded and faulted argillite, slate, and greywacke of the Yellowknife Supergroup. Mine development was focused on a narrow quartz vein in a band of black slate and slaty greywacke that strikes northeast, dips 80° north and ranges in width from 15 to 35 feet in an exposed length of 500 feet. Many beds in the slate band are between 1 and 6 inches thick and the band is bordered by greywacke beds that range up to 11 feet thick. The slate lies on the south limb of an anticline that trends north 75° east and is overturned towards the south. The slate band at the pit is about 100 feet south of the crest of the anticline.

A vein of fine-grained, sugary grey quartz lies along the foliation of the slate and outcrops for 120 feet, passing under drift at the west end where it is one foot wide, and tapering to a rusty crack at the east end. The vein walls are sharp and straight in most places.

In some places the vein branches and in others it is cut by veinlets of coarse-grained white quartz with feldspar. At the east end of the vein, a 20 foot section ranges in width from two inches to hairline, and contains abundant visible gold, a little pyrite and pyrrhotite, and may contain chalcopyrite, galena and native copper. Much of the gold occurs as films on transverse cracks in the quartz, and some gold films extend across the quartz veinlets and as far as ½” into the slate (Lord, 1951).

Pensive Yellowknife Gold Mines Limited (1939-1941)

The company owned a large group of claims between Upper Pensive and Dome Lakes, but the vein deposit on the Rare #15 claim was of most importance. A camp was erected during 1939 on Upper Pensive, and a crew of four men began blasting a pit on the vein, that by year-end was 27 feet long, 3 feet wide, and 13 feet deep. The showing was rich enough in gold that the construction of a small milling plant was authorized. Shaft sinking was also begun at the bottom of the pit. Ore from these workings were transported over land (via foot or dog team in winter) and water (via canoe) to the ‘Gibson’ milling plant, housed in a tent frame on the shore of Upper Pensive Lake west of the camp. Milling capacity was about five tons per day. Details of operation are not known (Lord, 1941; site evidence).

1939 Production

Production during the fall of 1939 was reported to be 20 ounces of crude gold from the milling of over 1,000 tons of hand-picked ore (Lord, 1941). Royal Canadian Mint records indicate that from this shipment, content was smelted down to 18 fine ounces of gold and 2 fine ounces of silver (National Archives of Canada). No development was reported during 1940.

1941 Development and Production

Development and production resumed in 1941, and the shaft was deepened to 50 feet from the bottom of the 20 foot deep open cut. A timber headframe was erected and a hoisting plant was brought to property. Equipment for shaft sinking included a Fordson gas tractor, operating an Allison hoist-winch, an air compressor, and a fan (National Mineral Inventory; *The Yellowknife Blade*, Apr. 28th 1941; July 27th 1941; site evidence). An unknown tonnage was milled, but Royal Canadian Mint records indicate that a shipment of 85 crude ounces of gold from Harry Ingraham Trust Limited produced 73 fine ounces of gold and 8 fine ounces of silver (National Archives of Canada).

Upper Pensive Camp

Built along a picturesque hill on the northeast end of Upper Pensive Lake, the original tent camp of 1939 was expanded to include a log cabin cookery in 1940. The Ingraham family vacationed at the property from time to time, maintaining a vegetable garden (site evidence).

Exploration Since Mine Closure

No other mining developments were reported after 1941, although in 1944 Tom Payne acquired control of the property through a new company (Pensive-Yellowknife Mines Limited) and conducted exploration through surface sampling and possibly diamond-drilling. A fire in 1947 burned down the buildings at the shaft site.

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PINE POINT

Producer (Remediated)

Lead, Zinc

60° 50' 00" 114° 30' 00"

Years of Primary Development: 1928-1929, 1952-1954, 1963-1987 *Mine Development:* 48 open pits, 2 u/g production mines

Years of Production: 1965-1988

Mine Production: 69,416,000 tons milled = 2,250,000 tons lead concentrate, 7,378,000 tons zinc concentrate + 1,156,900 tons of high-grade ore shipments (2,001,268 tons lead, 4,399,031 tons zinc)

Introduction

Pine Point is a former lead and zinc mining operation and townsite located on the southern shore of Great Slave Lake between Hay River and Fort Resolution, NWT. It operated between 1965 and 1988. The associated townsite served a population of up to 1,900 until 1988 when the mine closed and the town was dismantled. The author has not visited the site.

Brief History

The showings of lead and zinc were known to exist in this region long before the first mineral claims were staked in 1898. Lead was first observed by local native groups, but its useful applications for hunting and fishing were not known to them until fur traders ventured into the region. Ed Nagle, a fur trader himself, staked the first claims at Pine Point during the "Klondike Gold Rush". As interest was solely in the silver potential - which the property lacked - the claims were allowed to lapse. Between 1898 and 1920, more claims were staked and allowed to lapse. First major development was done in a joint venture between Cominco Limited, Ventures Limited, and Atlas Exploration Company Limited during 1928 and 1929 when shafts were sunk on numerous high-grade lead and zinc showings.

The Great Depression had adverted work, and interest waned until the mid 1940s when Cominco obtained a concession in the Pine Point area to conduct extensive exploration. Further underground work in 1954 outlined massive and rich ore bodies, and production planning was initiated. To ship out the lead and zinc concentrates to a southern smelting plant, a railroad was constructed between Peace River, Alberta, and Pine Point in 1962-1964. Construction of plant and the laying out of a new N.W.T. townsite began during 1962 and 1963 together with the establishment of a railroad from Alberta. The mine operated between 1964 and 1988 when economic ore reserves were diminished. The town was closed in 1988. Shipment of concentrates continued into 1990, and the property was remediated.

The original claims lapsed during 2000-2001. The 'N', 'M', and 'S' groups of claims were staked by Ross Burns between 2000-2002 to cover the ore trends previously produced by Pine Point and those areas to the west identified as containing additional ore reserves. Rising lead and zinc prices and new ideas on how to economically produce ores at Pine Point are the basis for a new period of development. In September 2004, the property was optioned to Tamerlane Ventures Incorporated and exploration is ongoing. An underground bulk sample program to test one of the deposits is underway in 2008.

Geology and Ore Deposits

Pine Point's lead-zinc deposits are Mississippi Valley type. The ore is found in an extensive middle Devonian barrier complex trending in a southwest direction and plunging gently west into northern Alberta. The area is covered with approximately 40 feet of glacial till and overburden. Barrier sediments, known as the Pine Point Group, occupy a stratigraphic interval of 400 to 500 feet. Fine, sandy textures dolomite occurs in the lower part of the barrier. Above the sandy dolomite lie coarse crystalline Presqu'île and limestone beds, both of which are hosts to the Pine Point lead and zinc orebodies. These orebodies occur as large prismatic lenses of mineralized breccia or as flat-lying sheets, discontinuous lenses, and runs. The former outcrop on surface and the later deposits are found at various depths. The orebodies are usually tabular shaped, varying in thickness from a few feet to 125 feet. Ore minerals consist of sphalerite and galena, while gangue minerals consists of marcasite, grey and white vein dolomite and calcite (Fish, 1981).

Ed Nagle (1898)

Ed Nagle staked the first claims in the Pine Point area in 1898. He was primarily interested in the silver and gold potential of the galena deposits. Nagle's claims centered on what has become known as the P-32 deposit. He was able

to hire the help of two 'Klondike' bound prospectors during the year to sink a shaft on the claims. The men sunk the shaft to a depth of 20 feet before they stopped work. Heavy sulphur was encountered during the sinking. Samples were removed from the shaft and sent for assaying. Rumour of rich silver deposits in the area prompted a staking rush and during the winter of 1898-1899 hundreds of claims were staked by prospectors camped at Fort Resolution. When Nagle's ore samples failed to show silver and gold values, interest in the area ceased and all the original claims lapsed. Although lead was a known and rich occurrence, its value was diminished by the isolation of the region (Nagle and Zinovich, 1989).

Atlas Exploration Co. Ltd (1920-1921)

By the 1920s interest in the Pine Point lead and zinc deposits was renewed. C.B. Dawson, a mining engineer for Dr. James Mackintosh Bell, returned to the property in the winter of 1920-1921 and restaked the claims. A log cabin was built at the old showings and a shaft was sunk 25-feet. Several test pits were excavated on various showings.

Atlas Exploration Co. Ltd and Cominco Ltd. (Joint Venture) (1928-1930)

A joint-venture was formed by three major exploration companies to exploit the lead and zinc claims at Pine Point: Cominco Limited, Ventures Limited, and Atlas Exploration Company Limited. Cominco sent William McDonald and Ted Nagle to Pine Point in 1927 to report on the claim groups. Through a series of deals with Atlas Exploration Company, Cominco was able to finance the work and obtain a certain amount of control in the venture (Nagle and Zinovich, 1989).

This became the first major attempt to develop the property and during 1928 major equipment for sinking a shaft was landed at Dawson's Landing on Great Slave Lake. A log wharf was constructed and a 25-kilometer wagon trail was cleared south to one of the original lead and zinc showings. A larger log cabin camp was erected near the deposit. Sites were chosen for exploratory shafts on several of the showings. Some test pits were dug during 1928. (Nagle and Zinovich, 1989).

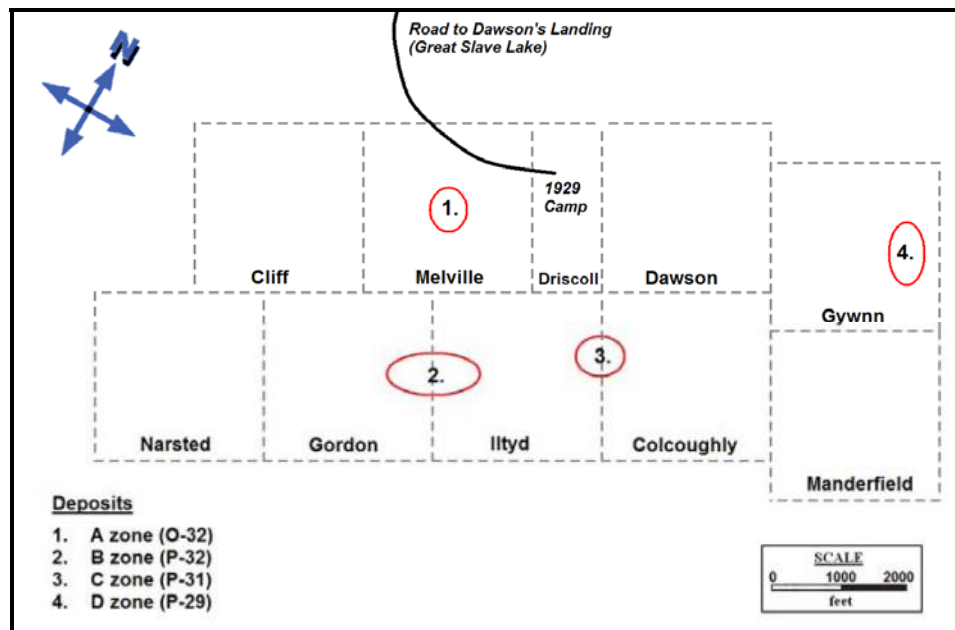


Figure 1. Pine Point claim and Deposit Map, 1929. Claim names are shown in each respective block.

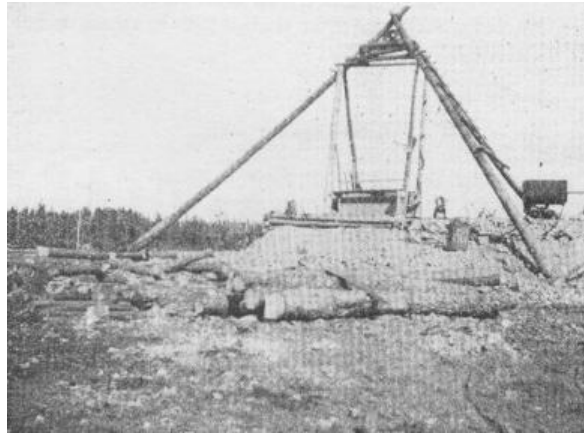


photo from Bell, J.M. The Lead-Zinc Deposits near Pine Point, Great Slave Lake. In Canadian Institute of Mining and Metallurgy Transactions, Volume XXXII, 1929.

Figure 2. Old headframe at the P-31 shaft, 1928-1929.

Shaft Sinking

By the end of 1929, significant surface and underground work had been accomplished four main deposits, known as the O-32, P-32, P-31, P-29 deposits.¹ At least one of these shafts, on the P-32 zone, was sunk by Klondikers in 1898. More shafts were sunk in 1921 by C.B. Dawson's crews. The most significant shaft was sunk on the P-31 zone (see **Figure 1** for location), sunk to a depth of 94 feet. The P-31 deposit was estimated to be 250 feet in diameter, and shaft work showed high values to 55 feet depth. Work stopped at this shaft at the end of 1929 because of inadequate equipment and flooding problems. Equipment used during the program included a portable Sullivan air compressor on steel wheels, originally hauled to the site by an oxen team. A crude timber-pole headframe was erected and equipped with a single-drum Sullivan tugger hoist. A shaft sinking crew was brought in from Cobalt, Ontario. (Meikle, 1930b; NWT Archives - Cominco Collection)

Total development at the four main deposits to the end of 1929 may be summarized as follows: P-32 (shaft to 35 feet, plus 35 test pits up to 14 feet deep), P-31 (shaft to 90 feet, plus 11 test pits up to 15 feet deep), P-29 (shaft to 55 feet, plus 11 test pits up to 15 feet deep), and O-32 (21 foot shaft, and 11 test pits up to 16 feet deep). The P-32 shaft encountered rich ore grading 20% lead and 15% zinc, and diamond drilling from the bottom of the shaft proved the continuation of high-grade galena to 45 feet depth. (Meikle, 1930a+b; NWT Archives - Cominco Collection, N-1980-002) It was found that the deeper ores were not as high-grade as those nearer to the surface. This discouragement, together with the aggravation of constant water problems and the ultimate failure of the pumps, resulted in the cessation of shaft work at the end of 1929.

Work shifted to surface exploration and drilling in the winter of 1929-1930. In 1930, the Northern Lead and Zinc Company Limited was formed by the joint-venture partners, with a controlling interest held by Cominco Limited. Atlas Explorations Limited and Ventures Limited held the remaining interest. Soon thereafter, the holdings of other companies in the area, including General Exploration Company Limited, were merged into the Northern Lead and Zinc Company property at Pine Point. From October 1929 to December 1930, about 21,600 feet of churn drilling had been accomplished, mostly on the P-32 deposit. Diamond drilling in the summer of 1930 totalled 2,900 feet. (National Archives of Canada – Northern Affairs Collection) Expenditures to June 1930 totalled about \$325,000 (Meikle, 1930b)

Conditions attending the Great Depression, and the realization that a mine would not be feasible at this location without an adequate transportation route, led to the temporary cessation of work. By this time, the original showings had been almost fully explored to indicate about 500,000 tons of ore grading 15% lead plus zinc. Widespread surface exploration had failed to find further indications of ore; therefore justification for any additional work did not exist and shareholders of Northern Lead and Zinc Company could not be swayed into advancing further funds for work (Hurdle, 1964; Jewitt, 1966).

¹ In 1952, the Pine Point ore deposits were re-classified using an alpha-numbering system. The O-32, P-32, P-31, P-29 deposits, until then, were known as the A, B, C, and D deposits, respectively. (see **Fig. 1**)

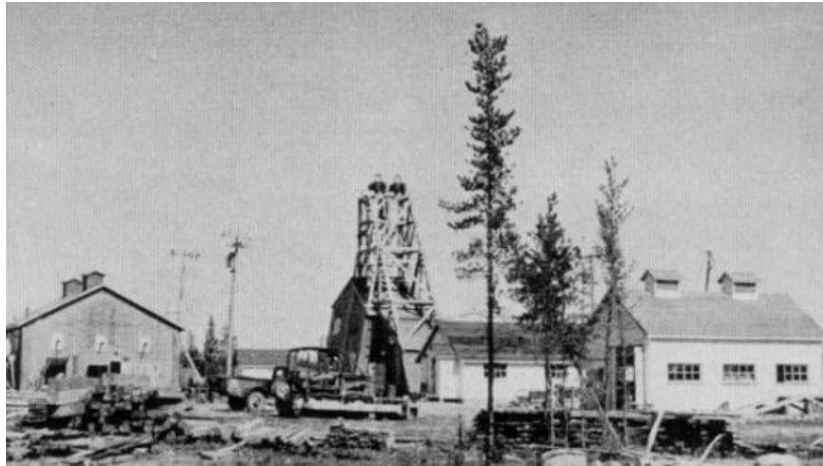


photo from The Western Miner magazine, May 1955.

Figure 3. Pine Point headframe and mining plant at M-40 shaft site, 1954.

Pine Point Mines Limited [Cominco Limited] (1952-1954)

Cominco was responsible for keeping the claims in good standing until a time when lead and zinc markets improved. Exploration resumed in 1947. This revival was based on a theory that existing ore bodies occurred along a projection of the McDonald fault and that other ore bodies might be found along this zone at favourable horizons. Cominco obtained a large concession from the Canadian Government completely surrounding the known ore area. This theory was substantiated by pattern drilling which indicated that the main ore-bearing horizon was the narrow Presqu'île formation overlying this fault zone. Drilling also indicated that while ore continued westerly, it existed at greater depths. About 9,100 feet of drilling was completed in 1948; camps were constructed and an airstrip built. In 1949, 25,500 feet was drilled, and in 1950, 41,570 feet on two concessions (The Western Miner, April 1951). The main purpose of this drilling was to determine the structure and stratigraphy for future exploration.

In May 1951, Cominco formed a new company and began serious developments. The new company was Pine Point Mines Limited, which was controlled by Cominco (Hurdle, 1964). Extensive diamond drilling to enlarge and define known ore bodies were completed between 1951 and 1953 as follows: 43,985 feet in 1951, 41,277 feet in 1952, and 36,199 feet in 1953 (Pine Point Mines Limited Annual Reports, 1951-1953). Open pit ore was calculated at 5,000,000 tons averaging 4.0% lead and 7.4% zinc. Underground ore reserves totalled several million tons but more work was required (The Northern Miner, June 10th 1954).

As part of the extensive exploration program, Cominco wanted to take a closer look at the geology of the deposits and take bulk samples. This could only be accomplished using underground methods. Two zones were chosen for this program - the N-42 and M-40 zones. Between 1952 and 1954, equipment was mobilized for the shaft sinking program. Originally intended to be completed in 1952 or 1953, the project was delayed until the summer of 1954. A small townsite was erected ("Old Town") and a complete mining plant was erected at the M-40 shaft site. A smaller plant was in use at N-42. A rough road, only passable in the winter, was built to connect Pine Point property to the Mackenzie Highway in 1952-1953.

N-42 Shaft Sinking

The N-42 shaft was sunk between July and September 1954 to a depth of 98 feet. Ore was encountered at 35 feet depth and mineralization continued to the shaft bottom. Drilling in this area indicated ore to 120 feet depth. Heavy water inflow halted further work in this shaft. (Pine Point Mines Limited Annual Report, 1955)

M-40 Shaft Sinking

The M-40 shaft, a 3-compartment structure, was sunk to a depth of 162 feet in 1954. A level was cut at the 145-foot level and lateral work was undertaken during the winter of 1954-1955. Total lateral development was 661 feet and 130 feet of raising. (Pine Point Mines Limited Annual Report, 1955) Water problems halted further work at this shaft. Also, ground at the bottom of the shaft was badly fractured resulting in dangerous and unstable mine workings. The M-40 orebody was found to be a flat sheet, 500 feet wide, 2,100 feet long, 16 feet thick, and lying at 150 feet depth.

The shaft penetrated two mineralized zones at 150 feet depth. This orebody would not be suitable for open pit mining, and if a production decision were to be made, the M-40 would be mined by underground methods.

A complete mining plant was erected at the M-40 shaft site, including a 53 foot timber headframe, hoist room, powerhouse, garage, and shops. Equipment included a power plant of three Cat D-17,000 96 kilowatt diesel generators, a 660 cubic feet per minute Canadian Ingersoll-Rand air compressor (driven by a Ruston-Hornsby diesel engine), a 2-drum 42 inch x 30 inch Canadian Ingersoll-Rand electric hoist, and Clarkson boiler.

P.E. Hirst was resident geologist overseeing much of the exploration and development at Pine Point during 1953-1955, under the direction of Cominco's head exploration geologist Neil Campbell (NWT Archives - Cominco Collection, N-1980-002).

Pre-Production Ore Reserve

The extensive exploration program, which began with diamond drilling in 1948, ceased upon completion of the underground work in 1955. A resource calculation was tabulated based on all this work, and suggested an ore reserve of 5,000,000 tons grading 4% lead and 7% zinc. Mine production at Pine Point was dependant on adequate transportation corridors to ensure that lead and zinc ores could be economically shipped to southern markets, and no further development was anticipated pending the completion of a railroad to Great Slave Lake (Hurdle, 1964; Jewitt, 1966).

Pine Point Mines Limited [Cominco Limited] (1963-1988)

Construction of the new railroad began in 1962 and was completed well ahead of schedule in early 1964. The construction of the railroad was jointly funded by the Canadian government, who was eager to sink money in developing new transportation corridors into the North, and Cominco, who assured the railroad company that they would be paying for the transport of large amounts of lead and zinc concentrate at rates of 215,000 tons per year for many years. The rail route followed the recently commissioned Mackenzie Highway from Grimshaw, Alberta north towards Hay River, NWT (Jewitt, 1966).

Construction of the Mine

Work at Pine Point itself during 1963-1964 concentrated on the pre-stripping of open pits, erection of mine and plant buildings, and erecting a townsite. Equipment and supplies was hauled over the Hay River winter road. In 1963, the company built 53 homes, two 50 man bunkhouses, a recreation hall, and water and sewage systems. The company did not want Pine Point to solely be a company town, and invited the NWT government to provide and sell lots to private industry to help promote business. The mining operation was expected to employ about 200 men. In 1964, all phases of construction progressed favourably. The new townsite was opened and crews were assembled in February and March 1964. By year-end, the construction of all plant and service buildings was nearly completed and installation of equipment had started. The airstrip was lengthened to 4,000 feet in July 1964. The highway connecting Pine Point with the Mackenzie highway was completed in October 1964 and daily bus service to Edmonton was established (Pine Point Mines Ltd. Annual Reports, 1963-1964).

Pre-Production Development

Mining equipment was delivered in February 1964 and stripping of two open pits commenced. The first open pits to be mined at Pine Point were the N-42 and O-42 deposits. High-grade ores were encountered and stockpiled to await shipment. These ores graded about 20% lead and 25% zinc and were considered too rich to be treated in the future concentration plant (Pine Point Mines Ltd. Annual Report, 1964). With the completion of the railroad in 1964, shipment of these ores began and by the time high-grade shipments ceased in 1970, 1,156,900 tons of ore had been shipped grading 18% lead and 26% zinc (see **Table 1**).

Also during 1963-1964 a massive staking rush occurred as prospectors and companies sought claims adjoining the Pine Point property. Of most importance was the property of Pyramid Mining Company Limited who secured claims to the south and east of the mine. By 1965, five anomalies were indicated by geophysical exploration.

Taltson River Hydro

The Northern Canada Power Commission began investigations of possible hydropower in the Pine Point area during 1963, and a suitable location was found at the Twin Gorges site on the Taltson River. The hydro project was completed during the fall of 1965 just in time for the start of production at Pine Point. The plant also provided power to Fort Smith, NWT and other nearby communities. In 1963-1964, power at Pine Point was being supplied with three diesel generators (Pine Point Mines Ltd. Annual Report, 1965).

Year:	Ore Shipped:	Grade of Ore Shipped (Lead):	Grade of Ore Shipped (Zinc):
1964	14,100 tons	18.6 %	25.8 %
1965	364,200 tons	22.5 %	29.1 %
1966	282,300 tons	18.8 %	26.3 %
1967	333,000 tons	18.0 %	27.9 %
1968	353,000 tons	18.0 %	25.0 %
1969	--	--	--
1970	92,600 tons	14.5 %	21.5 %
Total:	<u>1,156,900 tons</u>	<u>18.4%</u>	<u>25.9%</u>

Table 1. Pine Point high-grade shipments, 1964-1970.

Start of Production

The concentration plant (mill) went into operation in November 1965, treating 5,000-tons per day. First shipment of ores from the mill was on November 29th 1965. Shipments were made to Trail and Kimberly, B.C., and Idaho (Bunker Hill Company Limited). By 1966, 51% of Pine Point concentrates were being sent to British Columbia smelters, 31% to the United States, 4% to Europe, and others to India. Plans were also made to sell concentrates to Japanese markets. Ore was derived from the N-42 and the O-42 deposits (Thorpe, 1972).

Equipment in use consisted of two 5-cubic yard shovels, one 3-cubic yard shovel, and 30 ton Euclid haul trucks. Joe Scarborough was the original mine manager (1964-1967). John Giovanetto was the original mill superintendent, and Stan Hodgson was the original mine superintendent.

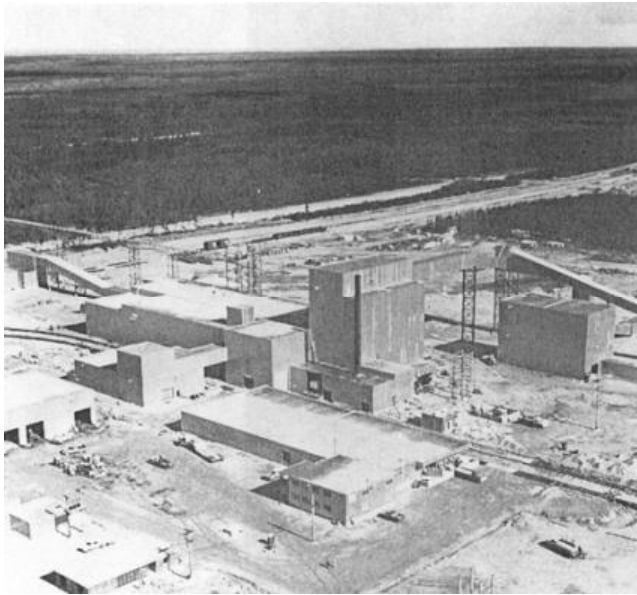
Pyramid Purchase

Pine Point Mines Limited purchased the claims of the Pyramid Mining Company Limited in 1966, adding two additional deposits to its resource inventory. The X-15 orebody was the largest of these (20 million tons reserve) and was quickly prepared for production. A major expansion program was initiated to accommodate the new ores and the mill was increased to 8,000 tons per day capacity through the doubling of the grinding and flotation circuits (Thorpe, 1972). The X-15 was brought into production in January 1969 (Pine Point Mines Ltd. Annual Report, 1969).

Other developments in the late 1960s included the depletion of ores in the N-42 and O-42 pits in August 1967, the bringing into production of J-44 and N-32 in 1967, and the addition of a 7 and 9-cubic yard shovel and new 50 ton Euclid haul trucks (Thorpe, 1972). Mining of the deeper pits such as X-15 showed that water would be a problem, and pumping to keep the pits dry began in 1968 (Pine Point Mines Ltd. Annual Report, 1968).

Milling Operations 1970s

The function of the Pine Point concentrator was to separate the ore minerals galena (lead sulphide) and sphalerite (zinc sulphide) from the iron sulphides and waste rock to produce high quality lead and zinc concentrates. The following milling circuit (10,000 tons per day) was in use at Pine Point by 1974: The ore was received from the primary stockpile and dumped into a 42 inch x 65 inch Allis-Chalmers gyratory crusher where it was reduced to 5 inches. It was then sent for secondary crushing, a circuit that consisted of two 7 foot Symons-Nordberg cone crushers, reducing the ore to $\frac{3}{4}$ inch. The crushing plant had a rated capacity of 1,000 tons per hour. The ore then passed through a Hardinge rotary thawer to thaw the ore before being sent for grinding. Final product from the crushing circuit was conveyed to twelve 750 ton capacity fine-ore bins in the concentrator building. The crushed ore from the ore bins was fed into three grinding circuits by means of an automatically controlled feeder and conveying system, maintaining the feed to each circuit at a predetermined rate.



Cominco Ltd.

Figure 4. Pine Point mill building, 1965.

Each circuit consisted of a ball mill (Dominion 10 foot x 16 foot) and rod mill (Allis-Chalmers 9 foot x 12 foot) and four cyclones. The first stage of grinding took place in the rod mill, where water was added to make a slurry of 80% solids. The ground product was pumped to cyclones and the fines were separated from the coarse particles. The coarse particles through the underflow of the cyclones was sent for secondary grinding in the ball mill, with the discharge from this grinding being fed back into the cyclones. The fine particles left the cyclone through the overflow and was pumped to the flotation circuit. A large number of flotation cell banks extract the lead and zinc minerals from the pulp. The lead mineral was floated off first, followed by the zinc mineral. The concentrates contained 50% water, so they entered a stage of dewatering using large diameter thickeners and rotating drum filters. The lead concentrates containing 5 to 6% moisture was ready for shipment. The zinc concentrates with 8 to 10% moisture was treated in a large dryer to reduce moisture content to 4%. The dewatered and dried concentrates were then conveyed into a loading shed for loading onto railcars for shipment (mine records).

Pine Point Townsite 1970s

Pine Point became a distinct northern community by the early 1970s. By 1970 the townsite boasted 149 houses, 24 apartment units, 96 trailer lots, and many private businesses. A modern arena and curling rink was built in 1971. 20 new houses were built in 1972, together with new trailers, movie theatre, baseball park, Legion hall, and many new businesses. A grade school was also built and was named Matonabee School. Twenty-three additional houses were built in 1973, plus a shopping mall addition. In 1974, 20 new houses and 14 apartments were built, and most of the streets were paved. Town status was achieved in April 1974. The following was the population of Pine Point at the dates given: 700 (1968), 1,200 (1971), 1,500 (1973), 1,800 (1974), 1,700 (1979) (Pine Point Mines Ltd. Annual Reports).

Employees

The following was the amount of people that the mine operation employed at the dates given: 200 (1965), 500 (1973), 650 (1976), 570 (1979), 610 (1980), 650 (1981), 540 (1983). In the late 1970s turnover rates were between 75 and 100% as a labour shortage crisis hit Pine Point. Also during this period there were about 50 natives employed, who represented about 10% of the workforce at Pine Point (Pine Point Mines Ltd. Annual Reports). There were also 41 women were employed in 1976. Senior staff employed at Pine Point in 1976 included W.H. "Bill" Gibney, mine manager; Merlyn J. Royea, general superintendent; D.C. Parker, production superintendent; W.R. Hargrave, dragline project superintendent; D.A. Cormode, mill superintendent; Z. Nikic, chief geologist; G.R. Larouche, shop superintendent; and J.R. Barr, controller. J. H. Salter was chief executive officer of Pine Point Mines Limited, and R.P. Douglas was president (Pine Point Mines Ltd. Annual Report, 1976).

M-40 Underground Mine

Re-development of the M-40 underground mine began in 1970 for the purpose of evaluating the feasibility of underground mining at Pine Point. The deposit was deep, thin, and was composed of very soft rock. A decline was driven into the deposit beneath the old (1954) shaft workings and test-stoping was conducted in 1970-1971 to test ground conditions. They were found to be favourable. 360 feet of drifting was reported in 1972 in preparation for testing a new type of mining machine - the Dosco Miner, a boring machine that scrapped the ore from the face of the stope using tungsten-carbide bits. This work was continued during 1973-1974, and production was achieved in 1975 at a rate of 500 tons per day, eventually increased to 1,000 tons per day (Pine Point Mines Ltd Annual Reports, 1970-1975). Jim Greenhalgh was superintendent in charge with the underground mine during 1976-1977 and 44 men were working on the project, with 33 men underground. Mining operations were conducted at 150 to 200 feet depth using trackless equipment (scooptrams and trucks). Water problems were not sufficient enough to hamper operations

(Northern News Report, Jan. 19th 1977). The M-40 underground operation shutdown in 1977 when zinc prices dropped. Total production at M-40 has been reported as 386,770 tons of ore grading 2.2% lead and 5.5% zinc (Giroux and McCartney, 2004).

New deposits were being located further to the west during the 1970s, and as new open pits were developed it was becoming more expensive to truck ores from greater distances. New testing of alternative and replacement haul trucks was ongoing during the 1970s. In 1972, a 150 ton Terex haul truck was purchased (later named "Fat Albert") and put on trial operation. This type of truck was found incompatible with operations at Pine Point. The general practice at Pine Point in the 1970s was the use of 50 ton and 85 ton units (introduced in 1975) and 40 ton trailer attachments (Pine Point Mines Ltd. Annual Report, 1972).

In 1972, Pine Point purchased the Coronet Mines Limited property adjacent to the south of Pine Point, adding 1 million tons to the reserve. These were the S-65 and R-61 zones. Nine open pits were in production during 1974: the W-17, X-15, N-42, K-62, O-28, K-57, P-31, P-29, and N-38. Tonnage milled was a record high during the year (4.1 million tons) but grade of the ore was lower. Higher metal prices permitted the processing of some lower-grade material. In November 1974, Pine Point purchased an orebody located on the claims of Conwest Exploration Limited. This became known as the A-55 orebody, and contained a reserve of 1.4 million tons grading 3.4% lead and 9.6% zinc (Gibbons et al., 1977). Starting in 1974, Pine Point began an extensive drilling campaign to locate additional ore deposits. The \$1.5 million program was expected to take three years and add many years of life to the mine. The T-58 deposit was one of the zones located by this work (Laporte et al., 1978).

A general trend in the lead and zinc markets at this time showed rising value in lead and zinc metals. But even with this up-turn in the metals market, Pine Point mine was hampered by high cost of operation, specifically in relation to increased stripping costs, distance from new pits to the mill, and increased pumping requirements as orebodies grew deeper. New pumping methods were designed during this period. Deep wells were drilled around the perimeter of the open pits to depths of 250 to 350 feet and pumps were installed. These units pumped up to 1,000 gallons per minute from the wells.

Open Pits

Open pits at Pine Point were in production for various lengths of times, depending on the size of the deposit. A 2 million-ton deposit could have lasted 2 to 3 years, while a larger 10 million-ton deposit would last up to 12 years. The pits were from 500 to 2,800 feet wide, 100 to 300 feet deep, with 25 foot benches and a 45° slope. Pits were developed in stages. First stage was the removal of overburden and any rock overlying the ore deposit. This work was originally done using excavator shovels. Ore on the pit floor was blasted using large drills, collected using shovels, and hauled for processing.

The late 1970s showed a good improvement in productivity at Pine Point mine primarily because of new equipment and better management. 100 people were laid off in 1977 as a result of the productivity and the reduced activities in all parts of the mine, including cessation of work at M-40 underground mine (Pine Point Mines Ltd. Annual Report, 1977). Modifications were being performed in the mill to further reduce magnesium and calcium levels in zinc concentrates - contaminants that lowered the value of the product. These modifications were fully completed in March 1981.

Seven open pits, the X-15, W-17, N-38, R-61, T-58, J-69 and A-70 were in production during 1977. The W-17 pit was closed during the year, and operations at the A-70 pit were temporarily suspended, as were underground operations at the M-40. Two new pits, the R-61 and T-58 (Coronet orebodies), were brought into production in 1977 (Lord et al., 1981).

In 1978, production was from 8 pits: the A-70, N-38, I-46, K-53, J-69, R-61, T-58, and X-15, with the large X-15 open pit supplying 37% of production. The K-53 and I-46 open pits began production during the year. These deposits contained higher than average lead-to-zinc ratios and were located closer to the concentrator, resulting in higher lead production and lower ore haulage costs. The N-38 open pit was mined out in 1978 (Pine Point Mines Ltd. Annual Report, 1978; Lord et al., 1983).

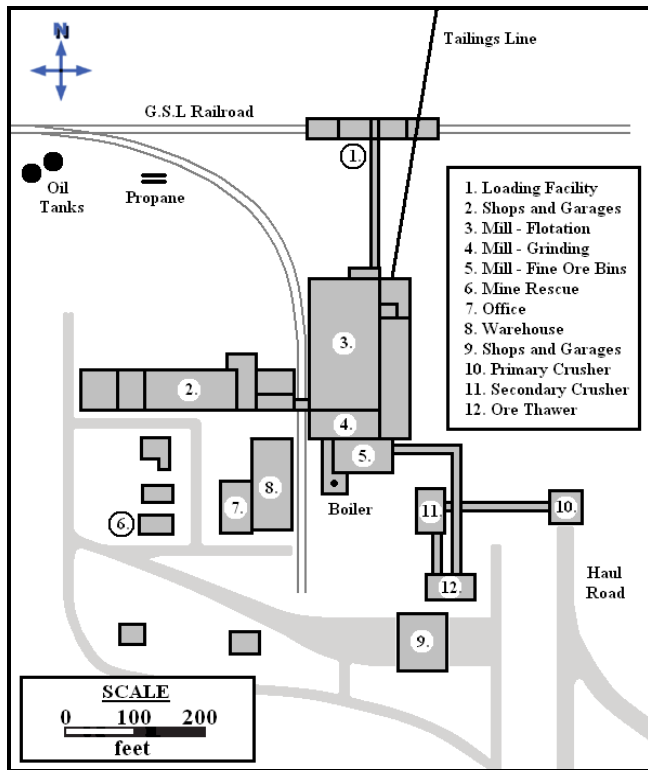


Figure 5. Layout of Pine Point plantsite, 1980s.

Production in 1979 was from the X-15, W-17, L-37, I-46, K-53, T-58, R-61, S-65, J-69, and A-70 deposits. (Brophy et al, 1983) A \$6 million modernization program in the concentrator plant was initiated during 1979. This involved the replacement of flotation machinery to improve recovery and reduce energy and maintenance costs. An expansion and renovation of the acid leach plant was also undertaken in 1979 (Pine Point Mines Ltd. Annual Report, 1979).

Dragline Installation

The problems associated with the expense of stripping operations became detrimental to operations in 1978 as orebodies were separated by greater depths of overburden and waste rock. The strip vs. ore ratio was very high, meaning that significant amounts of waste had to be moved to reach smaller amounts of ore. The solution was the installation of a dragline, a huge machine that scrapped tons of overburden from the surface. Project planning began in 1976, with the erection of the machine commencing in May 1978 and completed May 1979. It was a Page 752-unit, with a 30 cubic yard capacity bucket. The power-supply required for this massive piece of machinery resulted in an expansion at the Talston River hydro plant (Pine Point Mines Ltd. Annual Reports, 1978-1979).

In 1980-1981, production was focused on a number of pits at the North Trend of the property, known as the X-Y-Z deposits. Production was also derived from the A-55 orebody, and the L-37 orebody, which was located near the mill site. A total of eight pits were in production during 1981. The new dragline was intended for use in production operations at the L-37 orebody, but this proved unfeasible and the dragline was used primarily for stripping at the North Trend orebodies afterwards.

Mining Equipment 1980s

The following mining equipment was in use in 1980: one Bucyrus-Erie 40R drill, two Bucyrus-Erie 45R drills, one Page 30 cubic yard dragline, one Bucyrus-Erie 150B 7 cubic yard shovel, one Bucyrus-Erie 190B 9 cubic yard shovel, four Dart 600C 13 cubic yard loaders, one Michigan 675 22 cubic yard loader, one Cat 992B 9 cubic yard loader, eleven GM Terex 33-11 85 ton haul trucks, six Cat 776 haul trucks with 150 ton trailers, five Terex 33-15 150 ton haul trucks, two Cat D9 bulldozers, five Cat D8 bulldozers, two Cat rubber-tired bulldozers, and two Euclid R50 sand trucks. (Fish, 1981) Seven new 85 ton haul trucks were introduced in 1981 and a P+H 15 cubic yard shovel was bought in 1982. In 1984, the mine bought three 140 ton Wabco haul trucks, three more 85 ton haul trucks, a 120 ton haul truck, a bulldozer, and a front-end loader (Pine Point Mines Ltd. Annual Reports, 1981-1984).

Pine Point Townsite 1980s

In 1980, 40 new houses were built, forming a new subdivision at Pine Point. The old bunkhouses were replaced by modern bachelor apartments for 120 men in 1981. The Matonabee school was rebuilt after a fire destroyed the original, and also a new separate school was built for high-school kids (Galena Heights School) (Pine Point Mines Ltd. Annual Reports, 1980-1981).

Discovery of N-81

The life of Pine Point was extended due to the discovery of the N-81 orebody at the west end of the property in 1981. This deposit added 3 million tons grading 7% lead and 12% zinc to Pine Point's ore reserve. (Pine Point Mines Ltd. Annual Report, 1981) N-81 was brought into production in June 1984 and extended the life of the mine by three years (Pine Point Mines Ltd. Annual Report, 1984).

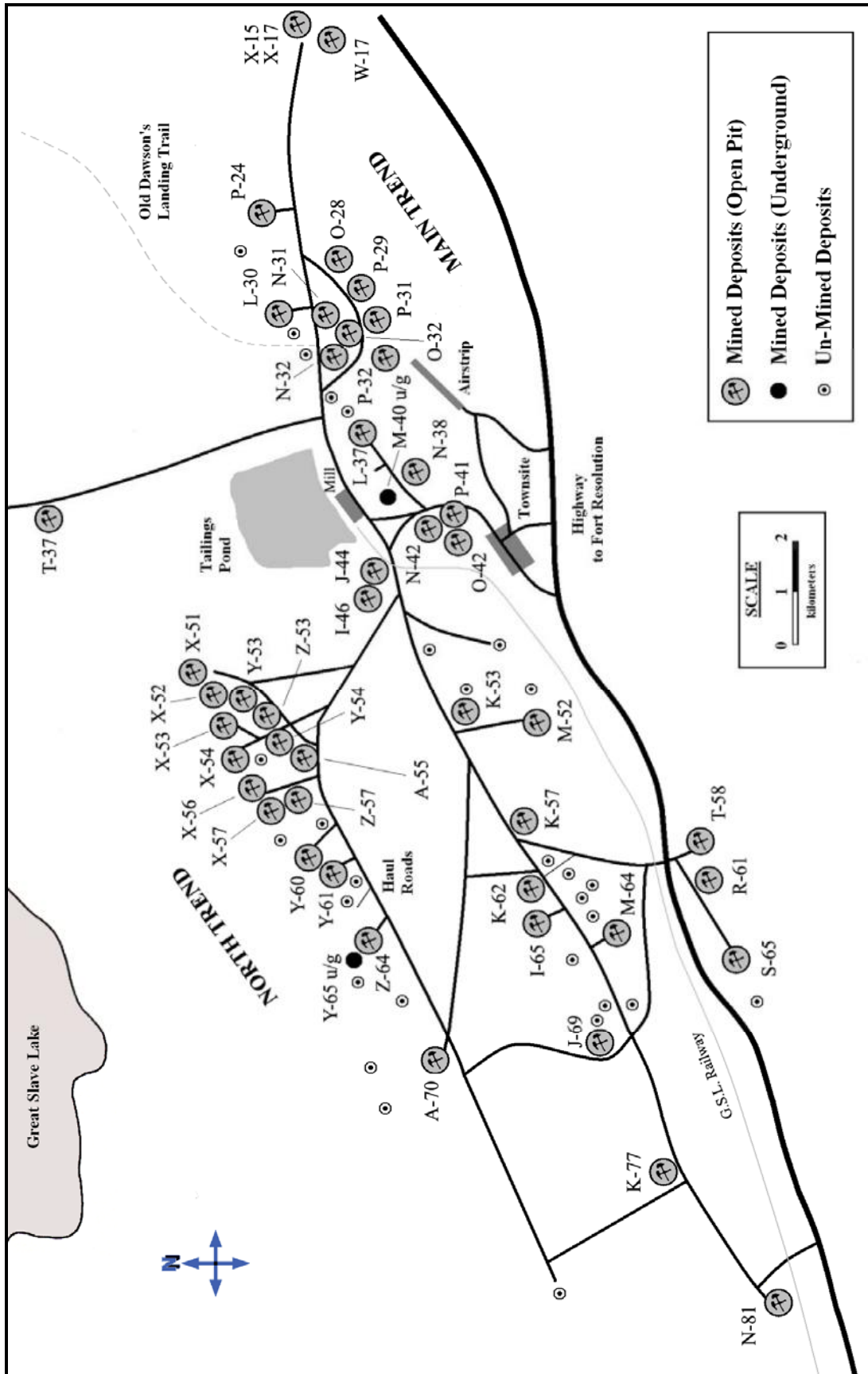


Figure 6. Pine Point Mine property plan showing location of mined and un-mined ore deposits.

Mining operations encountered more tabular orebodies in the early 1980s. These deposits had very erratic outlines and caused greater dilution in mill feed. The mine also faced escalating operating costs as haulage distances increased further and cost of power and labour skyrocketed. As mining operations moved further west the pits had to be sunk deeper to reach the orebodies, due to the westerly dip of the formations. Mill feed of zinc also dropped during this time (Pine Point Mines Ltd. Annual Report, 1984).



Cominco Ltd.

Figure 7. *Open pit mining operations at Pine Point Mine, 1965.*

1983 Temporary Shutdown

Pine Point operations were shutdown between January 2nd and June 15th 1983 due to depressed lead and zinc prices and high operating costs. The company incurred many financial losses as a result of the shutdown period, to the extent that the closed mine was more expensive than having an operating one. Cost restraint programs were initiated by negotiating with the employee Union for a 10% cut in pay, a reduction in transportation and smelting charges, and government assistance for mine development costs. The mine became profitable during the later half of 1983 as a result of the new cost-saving initiatives and an improved market for lead and zinc concentrates (Pine Point Mines Ltd. Annual Report, 1983). Stripping and pre-development of the N-81 orebody began during the summer of 1983 and first mining and milling of ores began in June 1984. Seven pits were in production during 1983-1984 (Pine Point Mines Ltd. Annual Report, 1984).

Y-65 Underground Test Mine

Approval was given early in 1984 to begin developments at the Y-65 deposit in preparation for test mining. Equipment and knowledge gained through the development of the M-40 in the 1970s aided in the project. Driving of a decline from the bottom of the Z-64 pit began in July 1984 north towards the Y-65 deposit. By year-end 1984 the decline had been driven a length of 1,050 feet (Pine Point Mines Ltd. Annual Report, 1984; North of Sixty, Vol. 5 No. 1, Homecoming 1984). 10 men were employed on the project with George Kalmakac in charge. In March 1985, test mining of the deposit produced 3,600 tons of ore grading 4.9% lead and 9.6% zinc. 200,000 tons of ore had been outlined in a zone 100 feet wide, 1,200 feet long, and 20 feet thick. By May 1985, the Y-65 decline had advanced 1,500 feet using a 2-boom Jumbo, two Wagner scooptrams, four underground haul trucks, and the Dosco Miner.

Lateral headings to assess the mineability of the orebody were completed and a ventilation raise was bored. Operations were accelerated in the summer of 1985 to maximize the output of high-grade ores, but it closed in 1985 because of economic issues and the need to supply N-81 operations with steady power (Pine Point Mines Ltd. Annual Report, 1985). Total production at Y-65 has been reported as 165,100 tons of ore grading 7.0% lead and 12.9% zinc (Giroux and McCartney, 2004).

Grouting Curtain

By 1984, the mine had 55 de-watering pumps in service averaging 60,000 gallons per minute with a capital cost of \$5 million per year to keep them running. The mine began testing a new grouting method to keep the open pits from flooding. The N-81 open pit was used in the experiment. They drilled three lines of vertical holes at 50-foot intervals around the perimeter of the pit, filling the holes with cement to a depth of 500 feet. The cement would leak through the cracks in the rock and help seal the inflow of water in the pit area. Ground water inflow was to be reduced by 5,000 gallons per minute through this method. But at N-81, the host rock was found highly permeable and water-saturated, and the experiment failed in 1985 (Pine Point Mines Ltd. Annual Reports, 1984-1985).

Year:	Ore Milled:	Grade of Ore Milled (Lead):	Grade of Ore Milled (Zinc):	Lead Concentrate:	Grade:	Zinc Concentrate:	Grade:
1965	75,000 tons	4.3 %	7.6 %	4,000 tons	?	8,000 tons	?
1966	1,458,000 tons	4.9 %	10.5 %	79,000 tons	?	241,000 tons	?
1967	1,521,000 tons	4.7 %	9.7 %	83,000 tons	?	233,000 tons	?
1968	2,138,000 tons	3.5 %	6.6 %	87,000 tons	?	223,000 tons	?
1969	3,605,000 tons	3.2 %	7.4 %	137,000 tons	75.0 %	431,000 tons	57.0 %
1970	3,860,000 tons	3.0 %	7.1 %	135,000 tons	?	451,000 tons	?
1971	3,892,000 tons	2.6 %	6.5 %	118,000 tons	?	416,000 tons	?
1972	3,810,000 tons	2.7 %	6.2 %	119,000 tons	75.5 %	391,000 tons	55.6 %
1973	3,896,000 tons	2.9 %	6.0 %	130,000 tons	?	371,000 tons	57.6 %
1974	4,135,000 tons	2.5 %	5.3 %	123,000 tons	76.8 %	357,000 tons	56.7 %
1975	3,905,000 tons	2.4 %	4.9 %	104,000 tons	78.2 %	301,000 tons	57.9 %
1976	3,773,000 tons	1.7 %	5.3 %	72,000 tons	74.4 %	323,000 tons	57.4 %
1977	3,443,000 tons	2.1 %	5.3 %	85,000 tons	73.5 %	290,000 tons	56.6 %
1978	3,290,000 tons	2.6 %	5.9 %	100,000 tons	76.5 %	302,000 tons	58.5 %
1979	3,291,000 tons	1.9 %	5.5 %	74,000 tons	73.7 %	288,000 tons	57.3 %
1980	3,626,000 tons	1.9 %	5.5 %	82,000 tons	76.0 %	315,000 tons	57.7 %
1981	3,636,000 tons	2.0 %	4.8 %	86,000 tons	77.1 %	274,000 tons	58.4 %
1982	2,445,000 tons	2.9 %	7.3 %	85,000 tons	76.5 %	287,000 tons	57.3 %
1983	985,000 tons	2.7 %	8.2 %	32,000 tons	73.8 %	130,000 tons	56.9 %
1984	2,512,000 tons	2.3 %	7.6 %	68,000 tons	75.2 %	303,000 tons	58.7 %
1985	2,356,000 tons	3.0 %	8.2 %	83,000 tons	74.7 %	300,000 tons	59.2 %
1986	3,271,000 tons	4.1 %	8.7 %	164,000 tons	73.9 %	458,000 tons	57.5 %
1987	3,514,000 tons	3.9 %	9.6 %	163,000 tons	77.1 %	533,000 tons	59.5 %
1988	979,000 tons	3.3 %	9.7 %	37,000 tons	78.4 %	152,000 tons	59.3 %
Total:	<u>69,416,000 tons</u>	<u>2.9 %</u>	<u>7.1 %</u>	<u>2,250,000 tons</u>		<u>7,378,000 tons</u>	

Table 2. Pine Point concentrator production, 1965-1988.

(source: Pine Point Mines Ltd. Annual Reports)

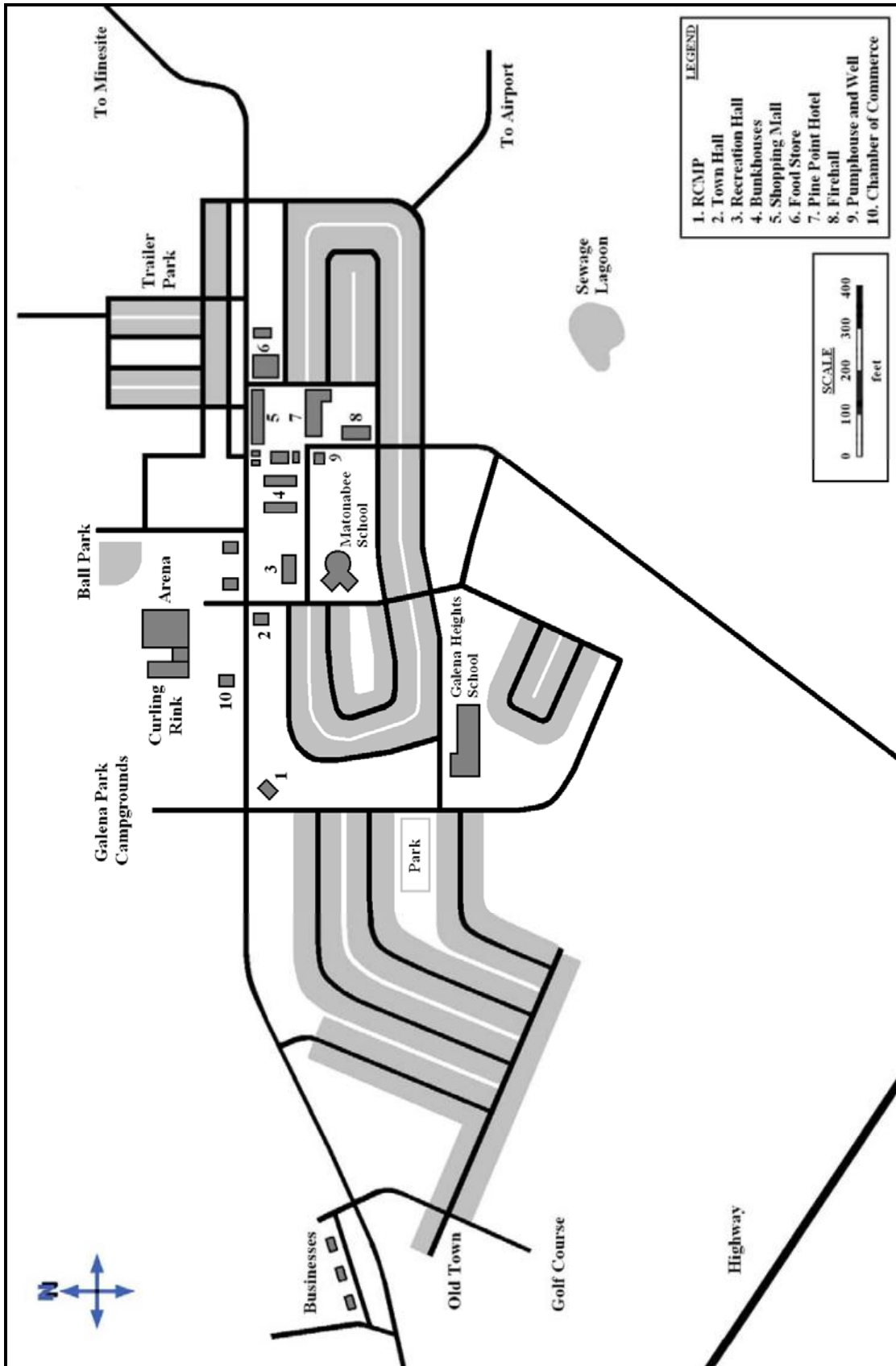


Figure 8. Pine Point townsite, 1980s.

Year:	Lead Recovered from Concentrates:	Zinc Recovered from Concentrates:	Year:	Lead Recovered from Concentrates:	Zinc Recovered from Concentrates:
1965	3,524 tons	8,377 tons	1977	69,111 tons	168,184 tons
1966	121,023 tons	144,613 tons	1978	81,832 tons	178,517 tons
1967	125,640 tons	138,650 tons	1979	59,486 tons	166,481 tons
1968	137,469 tons	214,500 tons	1980	67,705 tons	183,487 tons
1969	102,810 tons	246,001 tons	1981	71,345 tons	162,327 tons
1970	101,780 tons	252,051 tons	1982	70,874 tons	166,815 tons
1971	95,849 tons	239,369 tons	1983	23,616 tons	73,969 tons
1972	96,025 tons	220,045 tons	1984	56,316 tons	180,211 tons
1973	105,706 tons	216,589 tons	1985	67,918 tons	182,445 tons
1974	100,883 tons	205,484 tons	1986	129,209 tons	269,451 tons
1975	87,286 tons	177,694 tons	1987	133,899 tons	323,667 tons
1976	60,995 tons	188,801 tons	1988	30,967 tons	91,303 tons

Table 3. Lead and zinc content of shipped concentrates, 1965-1988. (source: Canadian Minerals Yearbook)

Mining operations were dependent on keeping the pits dry. Pumps were monitored regularly to ensure they remained operational. Water levels were observed very closely during power failure periods and any prolonged failure period would require the evacuation of equipment from the pit. During the winter months, conditions imposed by flooding worsened because new layers of ice would need to be mined themselves before mining of the pit bench could continue. The conditions the mine faced during the mid 1980s included increased thickness of overburden and bedrock that had to be penetrated to reach the deeper ores, increased de-watering costs for deeper pits, more stringent environmental regulations, and low metal prices. The quality of Pine Point's lead and zinc deposits was also declining. The mine reviewed its mine plan and significantly reduced its reserve based on which deposits could be economically mined. The dragline was shutdown in July 1985 as part of production cutbacks (Pine Point Mines Ltd. Annual Report, 1985).

Final Mine Plan

A new mine plan was announced in December 1985. It called for an accelerated production from economic ore sources to produce a large stockpile of cheaply produced concentrates. This meant for an end of mining at Pine Point in 1987, an end of milling in 1988, and the cessation of concentrate shipments by 1990 at the earliest. This plan was successful, in part, because the company benefited from higher metal prices in 1986, from which a greater profit could be made on the high-grade and cheaply produced material (Pine Point Mines Ltd. Annual Reports, 1984-1986).

Pine Point Closes

Mining of the N-81 deposit ceased in January 1987, and all mining operations in the remaining 8 open pits ceased in June 1987 leaving a stockpile of 2.2 million tons to be processed. Crews were laid off on July 1st 1987. Milling of the stockpiles continued until April 6th 1988, and a cleanup of the milling circuits was completed on May 15th 1988. The mill was sold for \$5.1 million and much of the mining machinery was liquidated in the following years. The final shipment of concentrates was made to Japan in October 1991. The Pine Point townsite was officially closed on August 15th 1988 and the last families vacated the property. Many houses and buildings were sold. In 1989, the entire townsite was dismantled with buildings either moved elsewhere or demolished (Pine Point Mines Ltd. Annual Reports, 1987-1989). The mine closed for many reasons: deeper orebodies due to a steep westerly dip of the geology, resulting in increased stripping ratios and enormous dewatering costs, low metal prices, increased haulage distances, and the capital costs associated with running the townsite.

Deposit:	Total Ore Feed:	Type of Deposit:	Deposit:	Total Ore Feed:	Type of Deposit:	Deposit:	Total Ore Feed:	Type of Deposit:
A-55	2%	Prismatic	N-32	3%	Prismatic	W-17	5%	Prismatic
A-70	4%	Prismatic	N-38	2%	Prismatic	X-15/17	27%	Prismatic
I-46	> 1%	Prismatic	N-42	5%	Prismatic	X-51	2%	Tabular
I-65	> 1%	Prismatic	N-81	4%	Prismatic	X-52	2%	Tabular
J-44	2%	Prismatic	O-28	2%	Prismatic	X-53	2%	Tabular
J-69	1%	Prismatic	O-32	> 1%	Prismatic	X-54	> 1%	Tabular
K-53	> 1%	Prismatic	O-42	5%	Prismatic	X-55	> 1%	Tabular
K-57	2%	Prismatic	P-24	> 1%	Prismatic	X-56/57	2%	Tabular
K-62	2%	Prismatic	P-29	> 1%	Prismatic	Y-53	2%	Tabular
K-77	> 1%	Prismatic	P-31	1%	Prismatic	Y-54	> 1%	Tabular
L-30	> 1%	Tabular	P-32	1%	Prismatic	Y-60	> 1%	Tabular
L-37	5%	Tabular	P-41	> 1%	Prismatic	Y-61	1%	Prismatic
M-40	> 1%	Tabular	R-61	2%	Prismatic	Y-65	> 1%	Tabular
M-52	> 1%	Prismatic	S-65	1%	Prismatic	Z-53	> 1%	Tabular
M-64	> 1%	Prismatic	T-37	> 1%	Tabular	Z-57	1%	Tabular
N-31	> 1%	Prismatic	T-58	1%	Prismatic	Z-64	1%	Tabular

Table 4. Source of Pine Point Production, listing the productive ore bodies and the percent of total production that they contributed to mill input. Open pit production accounted for over 99% of mill feed, with underground operations at M-40 and Y-65 producing comparatively little (less than 1%). The highest percentage of ore (27%) was mined from the X-15 / X-17 orebodies. (source: Giroux and McCartney, 2004)

Exploration Since Mine Closure

Additional ore deposits were outlined during exploration at Pine Point and adjoining properties, but these were rendered uneconomic in the 1980s. The original claims lapsed during 2000-2001. The 'N', 'M', and 'S' groups of claims were staked by Ross Burns between 2000-2002 to cover the ore trends previously produced by Pine Point and those areas to the west identified as containing additional ore reserves. Rising lead and zinc prices and new ideas on how to economically produce ores at Pine Point are the basis for a new period of development. In September 2004, the property was optioned to Tamerlane Ventures Incorporated as a 60% interest.

Tamerlane conducted drill testing of three un-mined deposits between February-September 2005, the W-85, GO-3, and R-190 zones. The R-190 deposit was on property originally owned and explored by Westmin Resources Limited in the 1980s and was known as the Great Slave Reef property. Eighteen holes were completed in 2005, seven of which on the R-190 zone. The drilling confirmed the grades of the historic deposits with combined lead and zinc grades above 2%. The highest grade intersections were obtained from the high grade R-190 deposit and included 105 feet of 31% combined lead-zinc, 32.5 feet of 52% combined lead-zinc and 65 feet of 37% combined lead-zinc. The longest intersection, obtained from the GO-3 deposit, was 212 feet of 15% combined lead-zinc while the W-85 deposit returned intersections of 135 feet of 14.4% combined lead-zinc and 90 feet of 19.8%. (Tamerlane Ventures Inc. Annual Report, 2005)

In 2006, Tamerlane acquired the remaining 40% interest in the Pine Point property. Results of the drilling suggested the R-190 deposit should be the focus of exploration. Feasibility studies to develop a mineable resource at the R-190 began in 2006 through underground mining, using a freeze-perimeter to control water seepage in the workings. A dense media separation flowsheet has been proposed. In early 2008, Tamerlane Ventures acquired the necessary permits to begin a bulk sample underground mining program at the R-190. Construction is underway during the summer of 2008. Diamond drilling continues to better define the ore deposits on the entire property.

Drill defined historic deposits on the entire Pine Point property amount to over 70 million tonnes of mineralization grading 1.59% lead and 4.19% zinc in 34 deposits. The R-190 is the highest grade deposit and in 2005 was estimated to contain 1,014,000 tonnes at a grade of 6.3% lead and 12.1% zinc. (Tamerlane Ventures Inc. Annual Report, 2005)

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PRAIRIE CREEK

Advanced Exploration (Non-Operational)

Lead, Zinc

61° 33' 00" 124° 48' 00"

Years of Primary Development: 1968-1970, 1979-1982, 2004-2007

Mine Development: three adits on #3 zone, two adits on #7-8 zone

Introduction

The property is located 555 kilometers west of Yellowknife, NWT in the South Mackenzie Mountains, about 43 kilometers up the Prairie Creek from its confluence with the South Nahanni River. The mine has yet to achieve production, although substantial development was conducted and a mill and camp built in the 1970s-1980s.

Brief History

The original discovery was made in 1928 by aerial explorers with Dominion Explorers Limited on the south side of Prairie Creek, what is now known as the #5 zone. Little work other than cursory examinations was performed until 1965 when Fred Nelson staked the 'Silver' claims. Nelson found evidence of other minerals on the north side of Prairie Creek consisting of float material in talus slides. In 1966, Cadillac Explorations Limited acquired the claims and sent a group of prospectors into the area to dig through the talus material and find the source of the mineralization. Additional claims were staked, and underground development on the main mineralized zone began in 1968. Extensive exploration on both surface and underground continued into the 1970s.

In 1980, a private group led by Nelson Bunker Hunt and William Hunt of Texas agreed to finance the mine into production based on a feasibility study funded by Cadillac in 1979. A financing agreement was arranged with Procon Exploration Company Limited, a Hunt brother outfit, and mobilization of equipment began in 1980. By May 1982 the mine was almost completely ready for production with a 1,200 tons per day mill. A collapse in the price of silver forced Cadillac Explorations Limited into bankruptcy, and the project folded before production was able to begin. No further work was done at this time as the property was tied up in a court battle. Canadian Zinc Corporation Limited (then known as San Andreas Resources Corporation) acquired the Prairie Creek property in 1991 and spent the next ten years preparing for a possible startup of production operations at the mine. Water use permits are required before work can start. There is major opposition to the opening of the mine because it is seen as an upstream threat to the Nahanni National Park Reserve. Current resource calculations suggest a reserve of 11,900,000 tonnes of ore grading 12.5% zinc, 10.1% lead, 0.40% copper, and 161 grams per tonne silver. Canadian Zinc Corporation drove a new decline in 2006-2007 and has advanced the project through the feasibility stages, and in June 2008 applied for the necessary permits to place the mine into production.

Geology and Ore Deposits¹

The Prairie Creek Mine is located on the eastern margin of the Prairie Creek Embayment in the southern Mackenzie Mountains. The area is underlain by Lower Paleozoic deep water basin rocks and platformal rocks of the Mackenzie shelf consisting, from bottom to top, of: Lower Ordovician to Silurian Whittaker Formation dolostone; Silurian Road River Formation cherty shale; Silurian to Devonian Cadillac Formation thinly bedded dolostone; Silurian to Devonian Root River Formation; and Lower to Middle Devonian Arnica Formation and Funeral Formation dolostone and limestone. North-south trending faults and fold axes dominate the structural geology of the area. Faults and fold axial planes dip both east and west. The west dipping Gate Fault and east dipping Tundra Thrust Fault are two of the larger tectonic structural features close to Prairie Creek. The structural pattern exposes windows of older Road River and Whittaker Formation rocks along the core of the main anticline. Most of the mineralized zones at Prairie Creek occur within the Road River Formation shale and Whittaker Formation Dolostone.

Three styles of mineralization are recognized on the Prairie Creek Property: lead-zinc-silver sulphides in structurally controlled quartz veins; stratiform lead-zinc-silver sulphides; and lead-zinc-copper-silver sulphides as solution cavity filling, Mississippi Valley Type mineralization. Twelve separate mineralized vein showings have been identified along the Prairie Creek vein system over a 26 kilometer strike length. The mineralization occurs in quartz veins as zinc-lead-copper sulphides, with significant silver content. The veins occur in the Whittaker Formation and Road River Formation in a crosscutting, steeply east dipping, fault with a northerly strike. The bulk of exploration and development has occurred at #3 zone on one of these vein occurrences. Stratiform massive sulphide mineralization has been intersected in drill core in #3, 4, 5 and 6 zones over a strike length of more than 3 kilometres. The bedded

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

sulphides occur in the Upper Whittaker Formation. The mineralization occurs as laminated, bedding parallel zinc-lead-iron sulphides that contain moderate amounts of silver and minor amounts of copper. The stratiform mineralization is up to 28 meters thick.

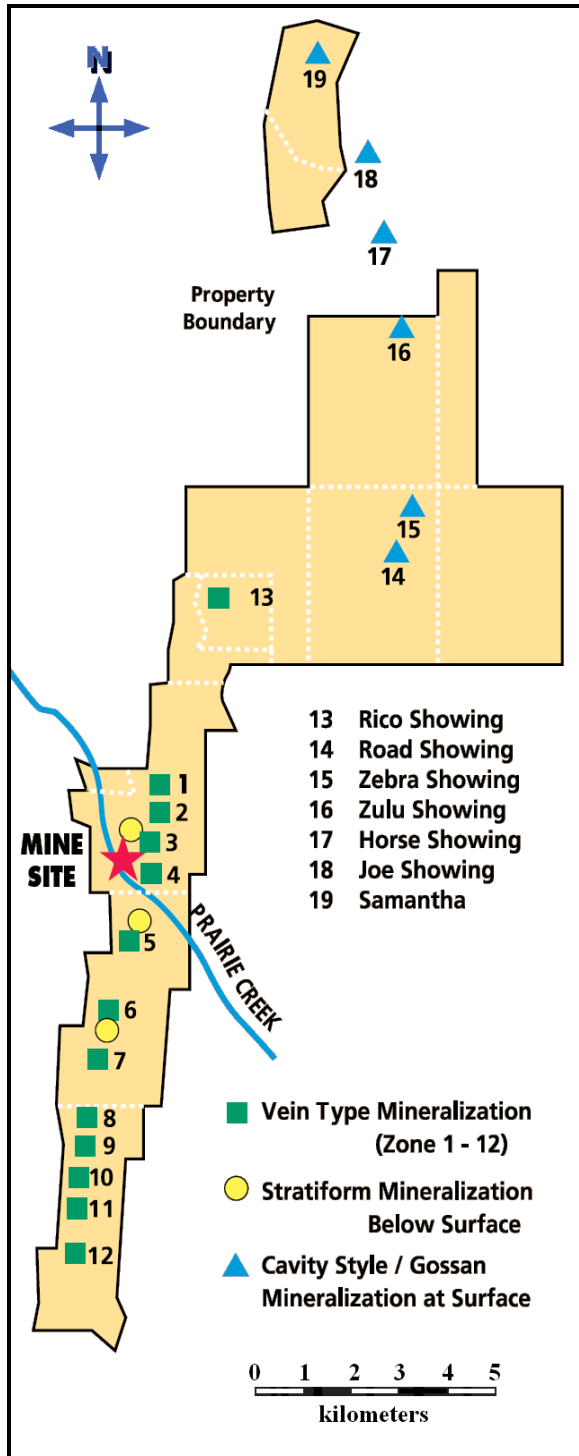


Figure 1. Prairie Creek Mine ore deposits.

The #1 zone consists of two parallel galena-sphalerite veins and lies 750 meters north-easterly from #2 zone. The two zones are not connected. In 1966, a channel sample was collected from #1 zone and returned assay values of 49.65 ounces per ton silver, 61.75% lead, 1.43% zinc and 2.77% copper. Stripping of the zone revealed that it narrows from 18 to 4 inches. #2 zone consists of two parallel galena-rich veins, exposed over 50 meters. The hanging-wall vein is up to 1 meter wide. Assay results on the vein are up to 13.25% zinc, 52.5% lead, 35.8 ounces per ton gold and 0.062% cadmium.

Mississippi Valley Type zinc-lead mineralization occurs in the northern part of the property in a marginal carbonate sequence termed the Mottled Dolomite, which occurs in the Whittaker Formation. The mineralization occurs as cavity filling, stratabound massive sphalerite-galena-pyrite, and is generally proximal to the mineralized veins. The close proximity of the two styles of mineralization may indicate a similar genetic origin. The Mackenzie Valley Type mineralization has been variably exposed on surface over a 10 kilometre long trend. Drilling has intersected up to 28 meters of this style of mineralization with grades up to 4.4% lead and 9.3% zinc.

The mineral resource at the Prairie Creek deposit was calculated in 1998 at 11.8 million tonnes grading 12.5% zinc, 10.1% lead, 161 grams per tonne silver and 0.40% copper. The majority of this resource is from the #3 zone. Mississippi Valley Type mineralization accounts for 1.4 million tonnes of the resource at an average grade of 10.3% zinc, 5.0% lead and 53 grams per tonne silver. In addition to the resource delineated in #3 zone, an additional 300,000 tonnes of similar high-grade zinc, lead and silver was estimated in #7 and #8 zones.

Cadillac Explorations Limited (1968-1982)

Work preparatory to major developments between 1966 and 1968 consisted of setting up a small exploration camp for crews (tents), stripping mineralized zones, and constructing two airstrips (one 1,800 feet in length and the other 3,800 feet). Geological work revealed up to 12 mineralized zones in the Prairie Creek area along a strike length of 8 miles. The most interesting was the #3 zone, discovered when building access roads up the mountain. Stripping revealed a four foot wide vein at the top of the slope, which then separated into numerous veinlets over a width of 23 feet as stripping progressed downhill. The #3 zone was chosen as a good place to initiate underground exploration (Christie, 1969).

Supplies were forwarded to the site via Bristol aircraft in early 1968, and Gremac Limited was contracted to perform the underground development. Equipment included scooptrams, ore carriers, and compressors. The driving of the 6 foot x 8 foot wide adit commenced on July 18th 1968 at an elevation of 3,050 feet. When work ceased in September

1968, 1,003 feet of drifting and crosscutting had been completed. Three veins were encountered in this work. The #3 vein assayed 9 oz/ ton silver, 31.4 % lead, and 1.7% zinc. The #2 vein assayed 3 oz/ton silver, 6.9% lead, and 3.2% zinc. The vein was found to be well mineralized with high oxidation, with a width of about 6 feet. K.J. Christie was in charge of this work (Christie, 1969).



Canadian Zinc Corporation Ltd.



Canadian Zinc Corporation Ltd.

Figure 2. *Exploration camp, 1960s.*

Work continued in 1969 on the #3 zone. A winter road from Fort Simpson was started in October 1968 and new mining machinery was brought to the site early in 1969. Major items included two surface diamond drill rigs, one D-8 and two D-6 Cat bulldozers, a Cat front-end loader, one scooptram, one pneumatic drill, three 5-ton underground ore trucks, and two 30-ton haul trucks. A new trailer camp for 60 men was also constructed. Two mining crews were at work. One crew was drifting in the footwall of the #3 zone on the 3,050-foot level, continuing the 1968 advance. Crosscuts were cut at 100 foot intervals into the vein and by October 1969, 14 crosscuts had been driven. The vein was well mineralized, and oxidation was found to decrease as work continued north. A second crew was at work developing a second adit level at 3,170-foot elevation. This development was initiated in July 1969. Work was to continue through the winter of 1969-1970 (The Northern Miner, Aug. 21st 1969; Oct. 30th 1969).

A start was made on driving a new lower level adit at 2,850-foot elevation to provide future production access into the #3 zone, but only 800 feet of advance was made before work shifted to concentrate on the 3,050-foot elevation workings (The Northern Miner, Oct. 30th 1969).

Adits on #7 and #8 Zones

A brief underground exploration program was conducted on the #7 and #8 zones, south of the #3 zone in 1969. It was estimated that 2,000 feet of drifting would be required on each level to reach the points below the surface mineralization, 1,100 feet above (The Northern Miner, Aug. 21st 1969). On the #7 zone, 912 feet of adit drifting was performed. On the #8 zone, 794 feet of adit drifting was performed. Work ceased in this area at the end of the summer, and results of this development are unknown (Singhai, 1975).

Penarroya Agreement

Under a February 1970 agreement, Penarroya Canada Limited committed to spend \$3 million on the Prairie Creek property to advance the mine to a pre-feasibility stage. Penarroya would then commit the funds to place the mine into production to earn a 50% interest from Cadillac (The Northern Miner, Mar. 12th 1970). Mike Stoner was in charge of work in July 1970 with about 40 to 50 employees. To November 30th 1970, Penarroya completed a diamond-drilling program of 26 holes totaling over 16,000 feet. Small bulk samples of ore were shipped for France for metallurgical testing. The 2,850-foot level adit was also being continued with crosscuts driven at 100 foot intervals. An ore reserve of 1,865,000 tons grading 12.5% lead, 15.5% zinc, 0.60% copper, and 6.4 oz/ton silver was calculated as of December 20th 1971 (Singhai, 1975).

In 1970, 4,633 feet of drifting and crosscutting was accomplished on the #3 zone. Additional development was done in 1971 on the #3 zone: 1,418 feet of crosscutting and drifting advanced was accomplished by Penarroya. Total diamond drilling of 32,658 feet was completed between 1967-1970 on zones #1-9 (Singhai, 1975). Exploration by this point had opened up ore in 12 zones over a distance of six miles (The Northern Miner, May 8th 1980). Underground development to 1971 is listed in **Table 1**.

Zone:	1968:	1969:	1970:	1971:	Zone Total:
#3	1,003'	3,603'	4,633'	1,418'	<u>10,656'</u>
#7	-	912'	-	-	<u>912'</u>
#8	-	794'	-	-	<u>794'</u>
<i>Yearly Total:</i>	<u>1,003'</u>	<u>5,308'</u>	<u>4,633'</u>	<u>1,418'</u>	<u>12,361'</u>

Table 1. Summary of underground lateral development to 1971 (source: Singhai, 1975)

Cadillac Explorations took Penarroya to court for a breach of the option agreement in 1971. Settlement was made in early 1974 in favour of Cadillac, and Cadillac resumed 100% ownership of the claims. Work during 1974 was focused on refurbishing the camp, clearing and maintaining roads to the ore zones, claim surveys, and some trenching (Singhai, 1975).

Following a favourable feasibility report by Kilburn Engineering in 1979, crews were mobilized on the property to recommence development in preparation for production. A low-level adit at 2,850-foot elevation was driven. This was planned to be the primary production entrance, and the portal was just behind where the mill was to be built. The adit tunnel was driven 700 feet during the summer of 1979. Underground work resumed in May 1980 and the adit tunnel was driven further, underneath the #3 zone (The Northern Miner, July 24th 1980).

Ore reserves in the #3 zone, calculated with a dilution factor of 15%, were reported as 1,580,000 tons grading 5.6 oz/ton silver, 10.9% lead, 13.5% zinc, 0.52% copper, and 0.09% cadmium. Additional partially developed zones blocked out another 365,000 tons of probable ore grading 4.3 oz/ton silver, 9.2% lead, 9.9% zinc, and 0.35% copper (The Northern Miner, Mar. 6th 1980).

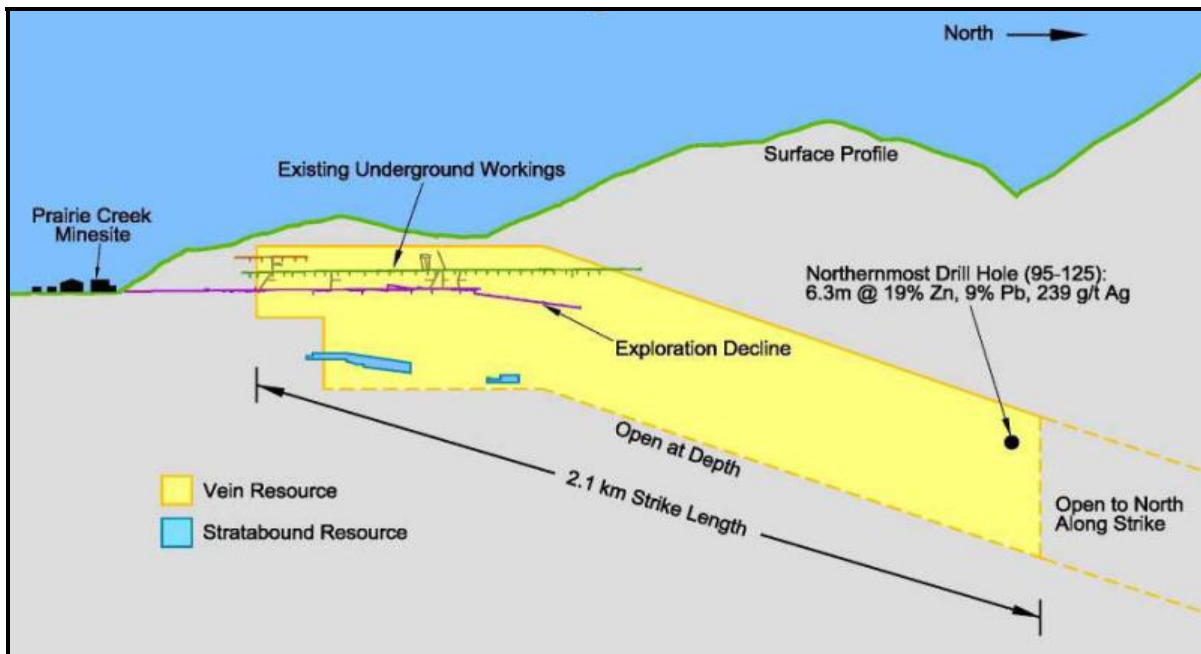


Figure 3. Longitudinal section of Main Zone, Prairie Creek Mine. (courtesy Canadian Zinc Corp.)

Capital costs associated with bringing a mine of this size into production were estimated to total over \$30 million. Lacking the funds to proceed, the project appeared to have stalled. But financing for the project was acquired by Procon Explorations Company Limited, an outfit run by the famous Hunt brothers (Nelson, William, and Lamar) of Texas. The deal stipulated that Cadillac sell a 40% working interest in the project for the sum of \$55 million, which would be used to finance the construction of the mine (The Northern Miner, Apr. 2nd 1981).

Construction of the Mine

The Cadillac company purchased a milling plant previously used at a copper mine near Fort Nelson, British Columbia (The Northern Miner, Mar. 6th 1980). The plant was dismantled and trucked to the site over winter road in early 1981. Work during 1981 included the preparation of tailing ponds, driving ore and ventilation raises, and stockpiling ore for the mill (The Northern Miner, Sept. 3rd 1981). 75,000 tons of ore were stockpiled by April 1982, and 50,000 tons of ore was broken in underground stopes (The Northern Miner, Apr. 8th 1982).

Camp

The camp as of 1980 consisted of twelve 8-man bunkhouse trailers, a minerfeets dry, and kitchen. There was also a recreation hall, small office, and three trailer residences for upper management personnel. More bunkhouse units and an expanded kitchen and recreation complex were planned to accommodate the crews necessary for production operations. These units were presumably installed in 1981.

The Mill

The mill was designed to treat 1,200 tons per day, but was never placed into operation when in May 1982 the company went bankrupt. It was designed to produce three concentrates: a lead concentrate containing 30% of the total silver resource, a zinc concentrate containing minor silver, and a copper concentrate containing 40% of the total silver resource. The flowsheet was to be conventional, consisting of two-stage crushing, ball mill grinding, flotation, thickening, and filtration. Copper concentrates were to be flown out by aircraft. Lead and zinc concentrates were of larger quantities and would have to be trucked out on the proposed all-weather road. Power for the mill and camp was to be supplied by three 1,150 kilowatt Cooper-Bessemer diesel generators. Heat was to be propane fired individual space heaters and also by using waste heat from the diesel engines.



Canadian Zinc Corporation Ltd.

Figure 4. Aerial view of Prairie Creek Mine, 2006.

All plant and camp facilities were constructed during the summer of 1981, and in May 1982 it was reported that facilities were 95% complete and production was scheduled for summer 1982. Cost overruns on construction and installation of the mill were experienced at about this time. During 1982, silver markets began a steady decline causing concern over the feasibility of placing the mine at Prairie Creek into operation. In July 1982, the price of silver dropped significantly and Cadillac Explorations Limited announced the cessation of all work until metal prices became more stable. Facilities were 97% complete at the time. Cadillac was also seeking additional financing of over \$15 million related to the project's higher-than-expected pre-production costs. \$10 million of this was for mill construction and \$5 million was for the completion of an all-weather road. Cadillac was broke and without the funds to proceed with production. Depressed silver markets made it impossible for Cadillac to raise funds on the stock market, and soon the creditors called in their loans. The company filed for bankruptcy in early 1983 (The Northern Miner, July 22nd 1982; June 30th 1983).

Zone:	Classification:	Ore: (tonnes)	Silver: (g/t)	Copper: (%)	Lead: (%)	Zinc: (%)
Main Quartz Vein	Measured	938,624	211.89	0.465	11.63	13.11
	Indicated	2,944,862	212.39	0.472	12.67	11.16
	<i>Measured + Indicated</i>	<i>3,883,486</i>	<i>212.27</i>	<i>0.470</i>	<i>12.41</i>	<i>11.63</i>
	Inferred	5,516,297	215.53	0.516	11.46	13.55
Stockwork	Indicated	682,165	50.15	0.112	2.68	5.85
	Inferred	4,045	51.31	0.126	2.51	5.54
Stratabound	Measured	611,417	67.6	-	6.68	10.85
	Indicated	663,261	62.0	-	5.53	10.15
	<i>Measured + Indicated</i>	<i>1,274,678</i>	<i>64.7</i>	<i>-</i>	<i>6.08</i>	<i>10.49</i>
	Inferred	21,234	55.7	-	5.65	10.49
Combined	Measured	1,550,041	154.9	0.282	9.67	12.22
	Indicated	4,290,288	163.3	0.342	9.98	10.16
	<i>Measured + Indicated</i>	<i>5,840,329</i>	<i>161.1</i>	<i>0.326</i>	<i>9.89</i>	<i>10.71</i>
	Inferred	5,541,576	214.8	0.514	11.43	13.54

Table 2. Resource Calculations, Prairie Creek deposit. (source: Canadian Zinc Corp. Annual Report, 2007)

Exploration Since Mine Closure

No further work was done until 1991 because the property was tied up in court following the bankruptcy of Cadillac Explorations Limited. San Andreas Resources Corporation Limited optioned the Prairie Creek property in August 1991 and in 1992 company geologists discovered stratabound-type mineralization within the ore deposit, which had the potential to exponentially increase ore reserves throughout the claim property. By 1993, the company had exercised their option to gain a 100% interest in the mine. In 1999, San Andreas was renamed Canadian Zinc Corporation Limited and since 2000 the company has initiated plans to bring the mine back into production. During 1994-1995, extensive diamond drilling was performed to prove the northern extension of the main zone and stratabound mineralization at depth. From 1991 to 2001, the company completed 129 holes (Canadian Zinc. Corp. Ltd., 2001).

The project received a land and water use permit for the development of a new decline and diamond drilling in 2003. In 2004, the company drilled 27 holes (5,963 meters) on three targets to follow the down-dip north of the main deposits, rehabilitated the underground workings in preparation for a resumption of development, and extracted a bulk sample from the vein for metallurgical testing. During 2005 the company continued its program of site rehabilitation. The underground development program was initiated in July 2006 and a new decline was constructed to access the lower horizons of the stratiform. Underground development during 2006-2007 totaled 550 meters of decline collared from the lower adit level, 75 meters of crosscutting, 25 meters of sumps and safety bays, and 10 drill stations. Fifty-one diamond drill holes totaling 10,624 meters were drill from the new workings into the stratiform zone. (Canadian Zinc Corp. Ltd. Annual Reports, 2003-2007).

Surface drilling in 2007 totaled 1,671 meters in 12 holes on the 'Gate' mineral claims, five kilometers west of the mine, and on the #8, 9, and 11 zones south of the mine site. No significant mineralization was encountered during the surface exploration program. (Canadian Zinc Corp. Ltd. Annual Report, 2007)

In June 2008, the company applied for the necessary permits to open the mine for production. The project was referred to an environmental assessment in August 2008 because of significant public concern. There is some opposition to the opening of the mine because it is seen as an upstream threat to the Nahanni National Park.

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	PTARMIGAN / TOM <i>Producer (Abandoned)</i>	Gold
		62° 31' 15" 114° 11' 45"

Years of Primary Development: 1938-1942, 1982-1983, 1985-1997

Mine Development: 924' vertical shaft, decline to 300', 6 levels (Ptarmigan); decline to 400', 3 levels (Tom)

Years of Production: 1941-1942, 1986-1997

Mine Production: 400,180 tons milled = 111,200 oz Au

Years of Bulk Sampling: 1983

Bulk Sample: 1,844 tons milled = 1,013 oz Au

Introduction

The Ptarmigan and Tom Mines are located 10 kilometers northeast of Yellowknife, NWT and can be reached by traveling up the Ingraham Trail towards Prosperous Lake. The mine has operated in two occasions, having last closed in 1997. The author has visited it numerous times over the years. The Ptarmigan and Tom started out as separate properties, but during the most recent term of operations they were owned by the same company and were basically run as a single-unit. The mines are separated by 1½ kilometers and are accessed by different workings, mining different veins, but production was combined and not reported separately.

When Treminco Resources Limited (the most recent owner) when bankrupt in 2000, the surface assets were seized by the Government of the Northwest Territories. Buildings and equipment were sold at auction in the fall of 2005, but as of May 2006 most of this material is still on site. Remediation plans for the mine are unknown.

Brief History

The 'Lily' and 'Jack' claims were staked in 1936 by Jack Stevens and Archie Mandeville, respectively. Cominco Limited purchased the two groups in 1938 in a share offering for a new company to be formed – Ptarmigan Mines Limited. Barely any exploration work had been done before it was decided to begin major developments. Shaft sinking was completed in 1941 and production began late in the year. The life of the project was short due to labor and supply shortages of World War II, and the mine closed in 1942. The old buildings was later scraped and demolished in 1969-1970.

The 'Tom' claims were staked by Tom Cassidy in 1938 and received some minor exploration before and during World War II. Cominco optioned the claims in 1941-1942 and did much exploration, including the sinking of a short 55' shaft. No other work was done by Cominco following the closure of the Ptarmigan Mine, although other owners would conduct exploration after the war.

Cominco became interested in reopening the Ptarmigan mine in the 1980s. Some underground work was done in 1982-1983 when they proposed to reopen the mine through decline development and trucking ores to Con Mine. When Cominco pulled out of the Yellowknife area in 1986-1987, they sold the Ptarmigan Mine to Treminco Resources Limited, who had acquired the 'Tom' claims in 1985 and were seeking for additional ore reserves. Production of Tom ores through custom milling at Giant Mine began in 1986, and Ptarmigan ores were being milled full-scale in 1988.

Ptarmigan and Tom operations were in full swing after 1989 when Treminco installed its own milling plant. The mine began to loose profit in 1992 and reserves continued to drop throughout the 1990s. A depletion of economic orebodies forced a final closure in 1997.

Geology and Ore Deposits¹

The Ptarmigan minesite lies near the western side of the Archean Yellowknife Metasedimentary Basin. The Ptarmigan vein is an easterly striking, steeply dipping quartz vein 1,500 feet west of a regional, sinistral, northwest-trending Early Proterozoic fault known as the Ptarmigan Fault. The vein can be traced 1,300 feet and averages 12 feet wide (ranging from 1 to 24 feet and possibly up to 46 feet). Its average width increases from northwest to southeast.

The mineralogy of both the Tom and Ptarmigan veins is more or less the same. They are irregularly shaped bodies of light to dark grey, generally massive quartz. Textures range from coarse-grained and glassy to fine-grained and

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

sugary. Contacts with country rocks are sharp and alteration is not evident. There is a stockwork zone of deformed quartz veinlets in the wallrock north of the Ptarmigan vein. Minerals other than quartz generally constitute less than 1% of the vein, although local concentrations of sulphides are present. Pyrite, sphalerite and galena are the most abundant sulphides; other minerals include arsenopyrite, chalcopyrite, pyrrhotite, native copper, gold, tourmaline, feldspar, carbonate and scheelite. Locally, ribbons of chlorite and micaceous material parallel the Ptarmigan vein walls. Concentrations of sulphides are commonly associated with elevated gold values; and it has also been reported that gold is concentrated where grey, ribboned quartz mineralized with sulphides including galena, occurs along straight, slightly sheared sections of the south wall.

Ptarmigan Mines Limited [Cominco Limited] (1938-1942)

Cominco Limited optioned the claims from the original prospectors in 1938 and begun a program of diamond drilling and trenching on the #1 vein. Crews were based from a campsite located on the Yellowknife River just east of Cinnamon Island, where a rough road connected inland to the claims. A three-compartment vertical shaft was started in October 1938 on the west side of Lilyjack Lake, where the #1 vein is located. The exploration camp was relocated to the shaft site in the fall of 1938. A small gasoline-powered hoist and two Ingersoll-Rand air compressors totaling 500 cubic feet per minute were in use (Lord, 1941).

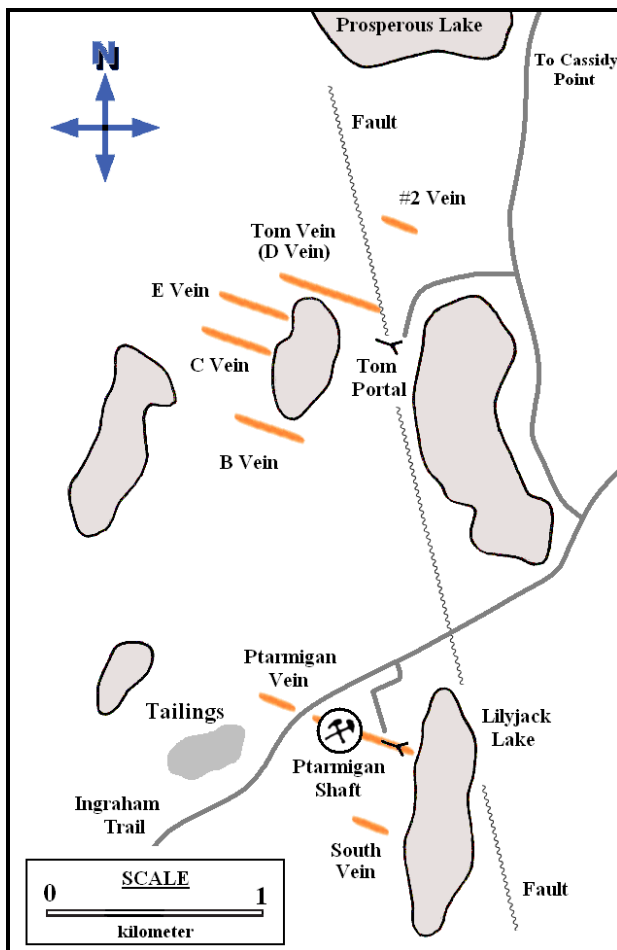


Figure 1. Ptarmigan and Tom Mine area.

At August 1st 1939, the shaft was 336 feet deep with two lateral horizons on the 150- and 300-foot levels, with drifting northwest and southeast totaling 1,230 feet on both levels. Hal M. Powell was brought to the site to supervise the work in 1939, when 50 men were employed. The camp consisted of two bunkhouses, cookery, warehousing, and office and staff quarters (Lord, 1941).

Original plans were to transport ore to Yellowknife's Con Mine for processing. But in late 1939 it was considered that the property held enough gold reserves to warrant the installation of its own milling plant. Although more work was required to establish an ore reserve, Cominco set about plans to placing the Ptarmigan Mine into production by 1940.

Milling equipment was freighted to Yellowknife and stored pending a decision. This was delayed until 1941 because Cominco felt it was better to wait for the completion of the Bluefish hydropower plant, plus they wanted to prove up additional reserves on the adjacent 'Tom' claims, which were acquired by Cominco through option agreement from Tom Cassidy in early 1940 (The Miner, May 1940).

The Ptarmigan Mine shaft was sunk to 620 feet depth in the winter of 1939-1940, with new levels established at 450 feet and 600 feet. The #1 vein continued to persist with good widths and good grades on these bottom levels. Mine development proceeded rapidly during 1940 in preparation for a production decision (The Miner, Jan. 1941).

Shaft sinking to 924 feet depth was continued early in 1941, with a new level at 750 feet depth. By April 1941, some 3,500 feet of drifting and 1,500 feet of raising had been accomplished on four levels above 600 feet (The Miner, April 1941). Ore reserve figures at the end of 1940 were approximately 80,000 tons ore carrying 0.56 ounces per ton gold (The Yellowknife Blade, Apr. 28th 1941). In September 1941, the first stope was being prepared on the 450-foot level with other stopes planned on the 300- and 600-foot levels. The vein was reported to have a mining width of five feet and the walls were strong suggesting that ground conditions would be stable for mining (Meikle, 1941).

Year:	Ore Milled:	Grade:	Gold Produced:	Recovery:
1941	3,096 tons	0.29 oz/ton	883 oz	97.8%
1942	31,333 tons	0.36 oz/ton	11,038 oz	97.8%
<i>Total:</i>	<i>34,429 tons</i>	<i>0.34 oz/ton</i>	<i>11,921 oz</i>	<i>97.8%</i>

Table 1. Ptarmigan Mine production, 1941-1942. (source: Cominco Limited (C.M. & S.), 1953)

A seven kilometer all-weather road was constructed from the northeast end of Yellowknife Bay to the Ptarmigan Mine site during 1940. A dock, warehouse, and oil storage tanks were erected at the Yellowknife Bay end of the route. Hydro power was connected to the site in the summer of 1941, and new power plant equipment was installed.

Start of Production

Equipment for the mill arrived during the summer of 1941 but lack of construction material delayed completion of facilities until late in the fall. Milling operations commenced on November 27th 1941 and the first gold bar was poured January 3rd 1942 (Lord, 1951; The Miner, April 1942).

Power Plant

Although primary power was obtained from the Bluefish Hydro plant, two standby Ruston and Crossley diesel generator plants were in use. Compressed air was supplied by two Canadian Ingersoll Rand compressor units driven by electric motor (Vic Waugh). Water for domestic and plant use was pumped from Prosperous Lake over a three kilometer wood-stave pipeline, starting in 1942. Previously, water was pumped from the underground workings. Water was stored in two water towers, one of 10,000 gallon and the second of 39,000 gallon capacity, and was heated in a small boiler plant (Meikle, 1941).

Camp Site

Ptarmigan Mine became the site of a small but active village during its short time in operation. Single men were housed in two single-story bunkhouses from 1938 to 1941. Additional bunkhousing was erected in 1942 to provide safer and roomier accommodation. Tommy Forest was meal cook at the camp. A small game hall, coffee room, and a ball field/hockey rink provided recreation. One hundred men were employed with construction work in September 1941 under the direction of mine manager Hal M. Powell. It was estimated that normal operations would employ 60 men. Other staff at that date included J.W. Stewart, engineer; G.L. Lauder, accountant; A. Waters, master mechanic; J. Anderberg, mine captain; N. Kerr, surface foreman; and W. Davis, electrician. There were 15 married employees and several families living on site in small townhouses (Meikle, 1941).



The Miner magazine

Figure 2. Ptarmigan Mine, 1941.

Mining Operations

Ore was mined using shrinkage stoping methods, and about six active working places were in operation during 1942 (Timmins, 1986). All haulage was labour intensive using hand-trammed ore cars. Ore was dumped down chutes to the bottom of the shaft where it was loaded into a skip and hoisted to surface.

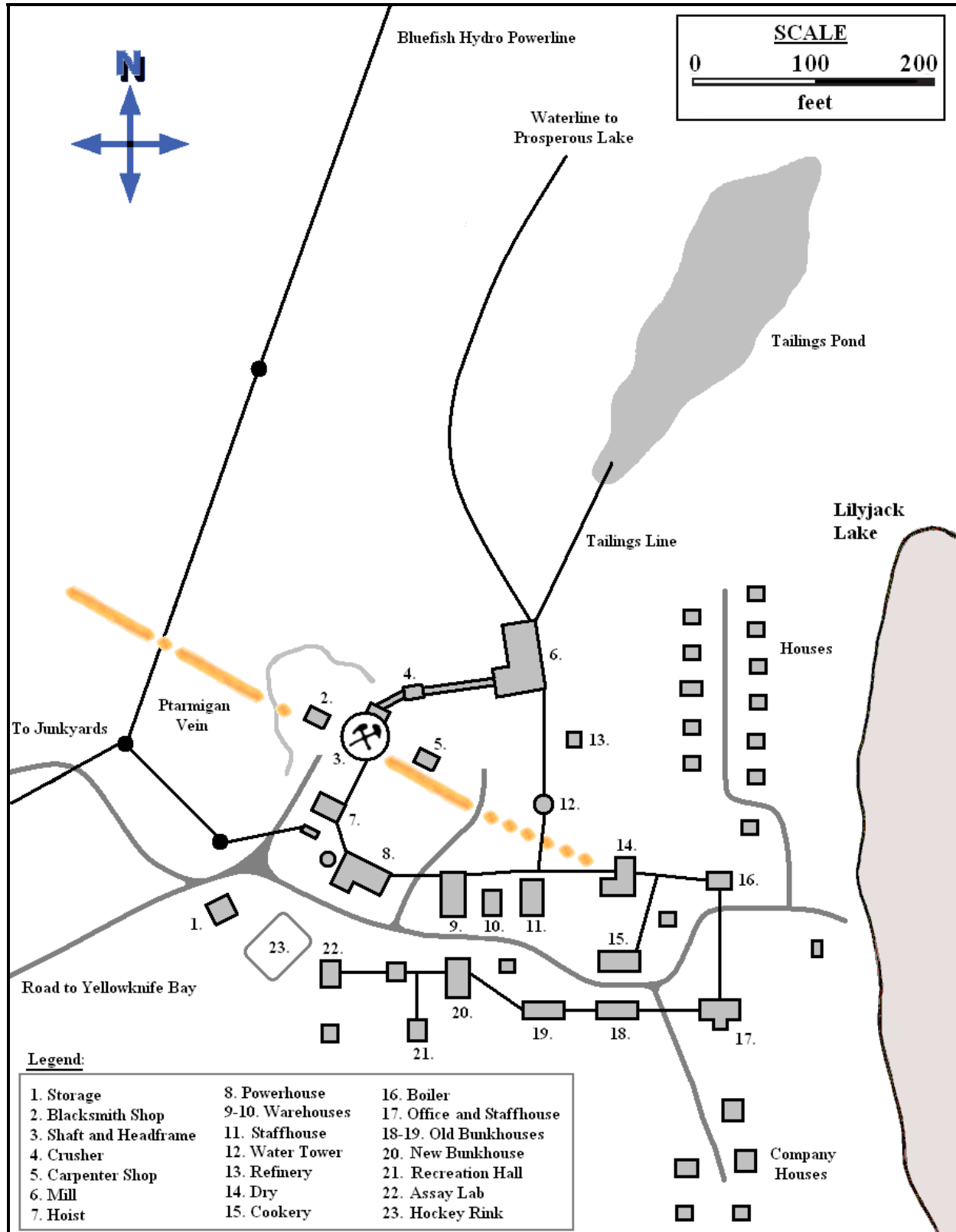


Figure 3. Ptarmigan Mine surface plan, 1942.

Milling Plant

The Ptarmigan mill plant was an amalgamation, cyanidation, and precipitation unit. Capacity of the plant was designed for 100 tons per day, but it was reported that a daily rate of 160 tons was attained during operations (Timmins, 1986). A complete description of equipment used is not available. Crushing was two-stage, and grinding was accomplished in a single ball mill. Gold was then recovered through a standard system of amalgamation, cyanidation, and precipitation, with gold poured in the form of bullion bars. Tailings were jettisoned north of the plant.

Tom Claims

In the summer of 1942, a brief underground exploration program on the adjacent 'Tom' claims was conducted. As already mentioned, Cominco had taken an option on this claim group in 1940 and in 1941 they performed some surface exploration. In 1942, a 55 foot vertical shaft was sunk on the Tom-vein. Because of operational constraints faced by Cominco, work ceased in favour of continued operations at Ptarmigan Mine. The war basically prevented any further work from being considered. Cominco later dropped the Tom option and the property reverted back to the original owners.

1942 Closure

Wartime conditions put a strain on operations at the Ptarmigan Mine in the summer of 1942. In late August the decision was made to close the property and transfer the mine crews to mercury and base metal mines in British Columbia. Mining ceased August 31st, and milling on September 2nd. The last gold bar, #26, was poured on September 10th. In a summary of operations report, it was said that the grade of ore in the mine was lower than originally estimated. A total of 34,429 tons of ore was milled at a grade of 0.34 ounces per ton gold. The mill operated very well with a high gold recovery of 97.8%. Shrinkage mining method also responded well within the deposit (Jewitt, 1943).

Considerable ore reserves remained at the mine (110,400 tons grading 0.43 ounces per ton gold, enough for two-years with good possibilities of adding to reserves) and Cominco planned to reopen after the war was over (Jewitt and Tefler, 1943). Total development at the end of this period has been reported as 4,828 feet of drifting, 970 feet of crosscutting, 924 feet of shaft sinking, 1,641 feet of raising, 9,076 feet of diamond drilling, and 739 feet of surface trenching (Timmins, 1986).

Year:	Ore Milled:	Grade:	Gold Produced:
1983	1,844 tons	0.55 oz/ton	1,013 oz

Table 2. Bulk sample of ore from Ptarmigan Mine 1983, processed at Con Mine.

Ptarmigan Mines Limited [Cominco Limited] (1982-1983)

The property was idle until 1980 when 1,143 feet of diamond drilling (9 holes) were drilled from the surface to upgrade the near-surface resource. A revised ore reserve of 53,000 tons of ore grading 0.26 ounces per ton gold was reported within the 150 BCD stoping block (McMurdo, 1980). Further drilling was performed in 1982 when 16 percussion holes were drilled on the vein from surface. The intent of this drilling was to obtain samples every five feet of core to ascertain grade information within the ore block below the crown pillar. The results of this drilling gave gold values higher than those obtained in diamond drilling, and it was decided to drive a ramp into the ore block and mine a small portion to test it economically (Winters, 1983).

Starting in October 1982 a small crew in charge of Knut Rasmussen began the driving of a short decline to recover ore on the 1st level west of the shaft. This work was done under contract for Cominco Limited, who was hoping to prove the feasibility of reopening the mine and trucking ores to Con Mine for processing. The decline was driven from a point north of the old Ptarmigan shaft a distance of 200 feet to reach the 1st level. Underground development ceased in April 1983 and mining of the pillars and other broken material on the 1st level began (National Mineral Inventory). Between March to September 1983, Cominco processed 1,844 tons of Ptarmigan ore grading 0.55 ounces per ton to produce 1,013 ounces gold. A production decision was not made due to the fact that the Con Mine mill was operating at full capacity on Con Mine ore with no allowance for custom ores (Con Mine, 1983).

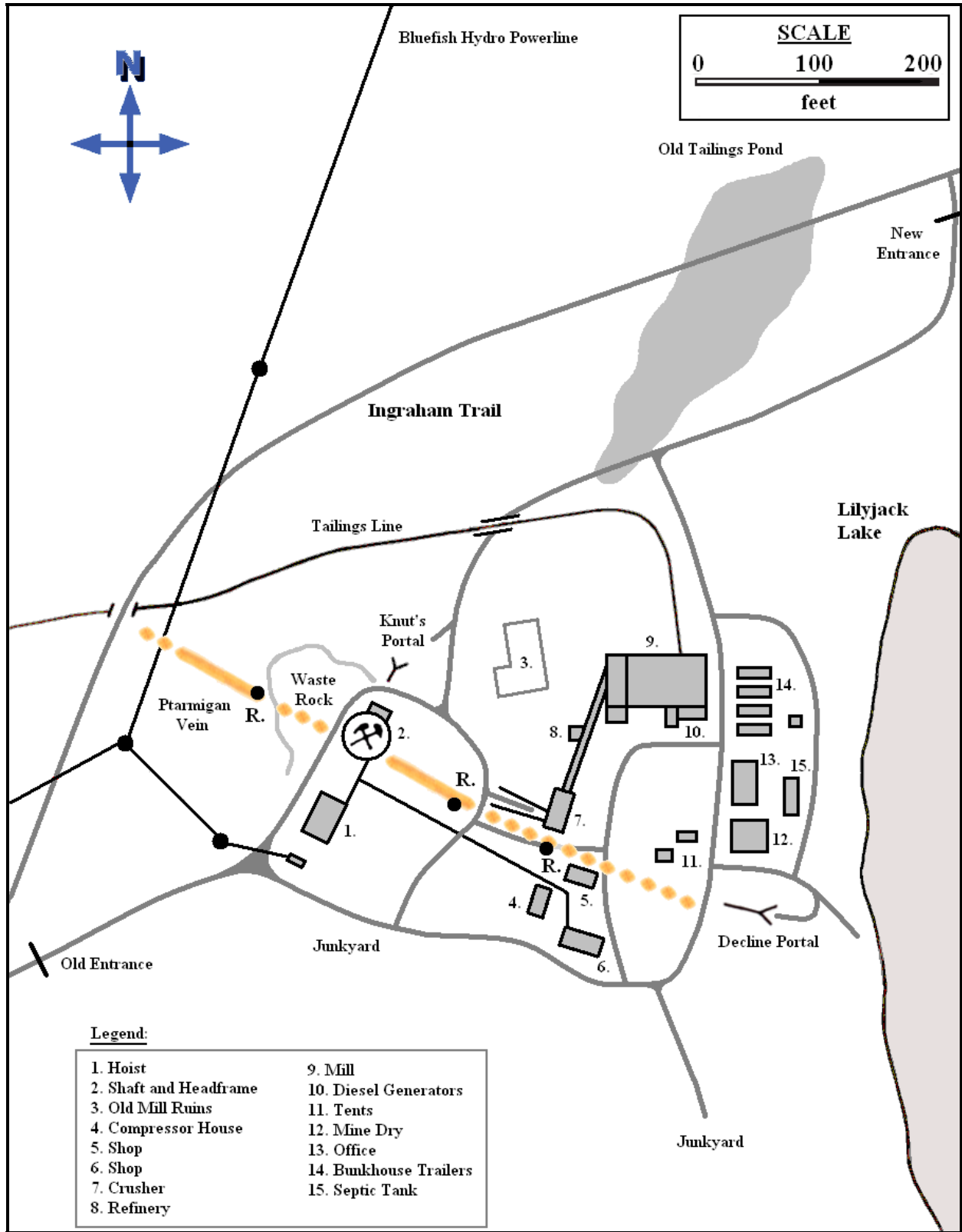


Figure 4. Ptarmigan Mine surface plan, 1990s.

Treminco Resources Limited (1985-1997)

Sensing an opportunity to enter the prospective Yellowknife gold mining field, Roland Treneman, president of Treminco Resources Limited, purchased the 'Tom' claims from Samuel Ciglen in 1985. The property had proven reserves and with gold prices on the rise, the company considered the property very exploitable at a great profit. Reserves at the Tom property's #3 vein, calculated by Treminco geologists in 1985, were 15,000 tons of probable ore grading 0.38 ounces per ton gold and 17,000 tons of possible ore grading 0.24 ounces per ton gold, contained entirely above the 150-foot level. At that time the Tom veins had only been 10% explored although diamond drilling had indicated mineralization of the vein at further depth (Spencer, 1985).

Joint-Venture

It was decided to drive a decline ramp into the known ore to block out and enhance knowledge of the reserves. A joint-venture project was commissioned involving Treminco Resources Limited and Goldrich Resources Incorporated, whereby Goldrich was given the opportunity to earn a 50% interest in the Tom Mine property for funds and logistics supplied to the decline development program. These funds consisted of \$300,000 to be used to drive a decline 1000 feet length to the 150-foot level, 100 feet of raising, and 1,000 feet of lateral development to explore and develop known reserves. Mining methods would primarily be shrinkage stoping. Goldrich was also to supply \$125,000 worth of mining equipment. A deal was then made with Giant Yellowknife Mines Limited for the custom milling of ores from the Tom Mine at their Giant Mine plant at a cost of \$35 per ton shipped. Recoveries of 85% or higher were anticipated using the Giant Mine refractory circuit. Development of the Tom decline began in September 1985. Development ore was sent to Giant starting in February 1986, but commercial production was not viewed as having commenced until September 1986 (Treminco Resources Ltd. Annual Report, 1986).

Ptarmigan Purchase

Treminco's real goal was to reopen and produce from the adjacent Ptarmigan Mine. Profits gained from operations at the Tom Mine were used to purchase Cominco Limited's control of Ptarmigan Mines Limited in March 1987. Ptarmigan Mine ore reserves in 1986 were 112,720 tons grading 0.34 ounces per ton, plus 20,000 tons of broken material in old stopes (Timmins, 1986). These were upgraded during 1987 to 139,000 tons containing 50,000 ounces of gold. A new decline was started at Ptarmigan Mine in July 1987. It reached the 1st level in August, intersecting a new ore shoot along the way. Rehabilitation of the 1st level and cleaning out of the old Ptarmigan shaft to provide an emergency man-way was undertaken. James Almaas was mine manager of Ptarmigan/Tom operations during 1987-1988 (Treminco Resources Ltd. Annual Report, 1987).

Treminco began trucking ores from the Ptarmigan Mine to Giant Mine for milling on a trial basis in September 1987, with full production commencing in April 1988. Also in 1988, the Ptarmigan decline was extended to the 2nd level. Most ore was being derived from the 150 stope on the 1st level, with the 300c stope on the 2nd level being mined out in July 1988 (Treminco Resources Ltd. Annual Report, 1988). Ore was mined by shrinkage stoping methods. On Ptarmigan's upper levels, ore was trammed by Atlas battery locomotives and two ton mine cars, and dumped into truck loading pockets of 100 ton capacity. Scooptrams and low-profile haul trucks hauled the ore out of the mine through the decline portal. It was not considered economical to extend the decline beyond the 2nd level so plans were made to fully rehabilitate the old Ptarmigan shaft for hoisting operations.

Tom Operations

In 1988, the Tom decline was advanced to the 2nd level (230 feet depth) where a new gold vein, parallel to the #3 vein, was intersected. A raise was driven up to intersect the old Tom prospect shaft, which was used for ventilation (Treminco Resources Ltd. Annual Reports, 1988-1989). In 1985-1986, two portable air compressors (a Gardner-Denver 900 cubic feet per minute and an Ingersoll-Rand 850 cubic feet per minute) were in use driving the Tom decline. A 125 kilowatt Cat genset powered the facility. The Ptarmigan and Tom mines shared the same equipment and a listing below lists most of the known units in use during the 1990s. Facilities at the Tom Mine, in 1992, consisted of a 14 foot x 50 foot shop, a trailer unit for dry and shifter's office, and a smaller building for housing power plant units (mine records).

Ptarmigan Mine ore reserves at July 31st 1988 were 151,000 tons measured and indicated ore grading 0.32 ounces per ton gold and 125,000 tons inferred ore at 0.32 ounces per ton gold. Tom Mine ore reserves were 5,500 tons measured and indicated ore grading 0.32 ounces per ton gold and 10,000 tons of inferred ore grading 0.32 ounces per ton gold (Treminco Resources Ltd. Annual Report, 1988).

New Milling Plant

Treminco raised the funds to place Ptarmigan and Tom mines into self-sufficient production and constructed a new milling plant in 1988-1989. The contract for custom milling with Giant Mine ended on July 31st 1989 by which time the new milling plant at Ptarmigan Mine was ready for operation. A tune-up period commenced June 19th 1989, with commercial production attained July 16th 1989. A grand opening was held July 21st 1989. The plant was a gravity and flotation unit rated at 250 tons per day, a capacity that was reached in October 1989. Equipment had been purchased used from an old mine in Washington state and represented many thousands of dollars worth in savings. Costs associated with mill construction and shaft rehabilitation totaled CDN \$1.7 million (Treminco Resources Ltd. Annual Report, 1989).

Early production in 1989-1990 was below the anticipated rates because of lower-grade material and problems with the start-up of the milling plant. There were also mining problems associated with narrow gold veins. The new headframe and hoisting plant was reported to be operating well. The Ptarmigan shaft was de-watered to bottom level (900 feet) and the 3rd and 5th levels were being mined. Development added to reserves on the 4th and 5th levels to the east of the shaft, and other exploration identified reserves on the west end of the Ptarmigan vein (West zone) above the 2nd level where Cominco had not fully explored. Drifting east on the 2nd level beneath Lilyjack Lake failed to confirm ore material within the Lake zone (Treminco Resources Ltd. Annual Report, 1990).

Hoisting Plant

The Ptarmigan shaft (7 feet x 16 feet, three-compartment) was fitted with a 75 foot tall steel headframe (**Figure 5**). Hoisting was performed with a Dominion Engineering 72 inch x 49 inch 2-drum electric hoist, housed in a 36 foot x 36 foot building. The shaft was fitted with a three ton Kimberly aluminum skip and nine-man capacity aluminum cage (Elkhorn Gold Mining Corp., 1999). The plant was designed to service a shaft of 2,300 feet depth. The hoist was connected to hydropower in December 1990 (mine records).



Ryan Silke

Figure 5. Ptarmigan Mine headframe, 2001.

Mill Operations

The Ptarmigan milling plant was a 250 tons per day gravity-flotation plant, housed in a 50 foot x 130 foot steel building. Ore was transferred to the crushing plant by haul truck from the headframe dumping bins or from the Tom Mine decline portal and processed through a two-stage crushing circuit. First stage crushing was through a 10 inch x 36 inch Cedar Rapids jaw crusher. Output material passed over a 42 inch x 7 foot Kolman single-deck vibrating screen which directed the underflow to the conveyor gallery for further processing and the oversize rock into a secondary 3 foot standard Telsmith cone crusher. A 42 inch x 42 inch Eriez belt magnet removed any metal prior to entering the cone crusher. Ore was crushed to 3/8 inch prior to conveyance to the mill building, and stored in a 600 ton fine ore bin. The material was conveyed and weighed by belt scale prior to grinding in the 7 foot x 10 foot Traylor rubber-lined ball mill. A 12 inch x 18 inch Denver duplex mineral jig received ball-mill product and recovered coarse gold. Jig underflow was recovered and upgraded on a 2 foot x 4 foot Diester 15-S concentrating table. Gold bars were produced from this gravity concentrate. Other gold was produced from flotation concentrates. Overflow from the mineral jig was pumped to a Krebs D-10B cyclone, which recycled coarse material back into the ball mill and sent fine material (50% minus 200 mesh) to a bank of eight Denver flotation cells. Through several stages of upgrading the flotation concentrate was pumped to a 6 foot x 6 foot conditioning tank and then through a 6 inch x 4 foot Dorr-Oliver disc filter for moisture extraction. The dried filter cake was loaded into three ton fiberglass

bags and trucked to be smelted in Japan, America, and British Columbia. The majority of production was derived from the gravity concentrates produced from the smelting and pouring of small 4 inch x 6 inch gold bars, which on average contained 70% gold and 15% silver. Final treatment of these bars to produce fine ounces was performed by Engelhard of Canada, a refining company based in B.C. (Elkhorn Gold Mining Corp., 1999). Between July 1989 and August 1997, the mine produced 431 of these small gold bars (mine records).

Assay Office

The new assay lab was completed in July 1989 and was housed in a small extension to the Ptarmigan mill building. It was a modern assay lab, capable of crushing and pulverizing any standard muck, chip, or core sample. It contained laboratory cone and jaw crushers, a pulverizer, and a ceramic assay furnace capable of completing 60 gold/silver fire assays per 8-hour shift (Elkhorn Gold Mining Corp., 1999).

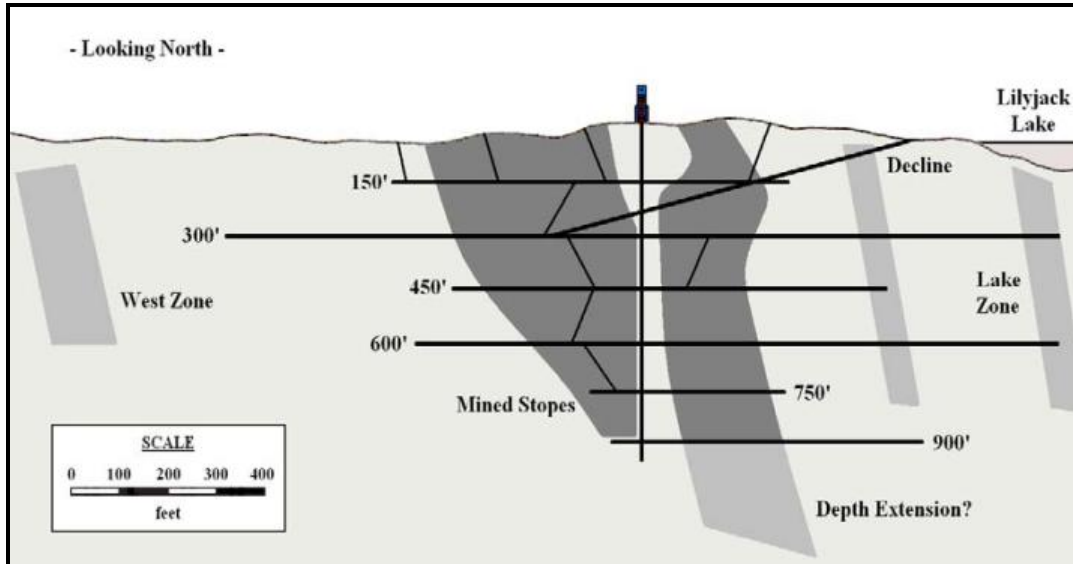


Figure 6. *Ptarmigan Mine underground longitudinal plan, 1990s.*

Power Plant

The Ptarmigan and Tom Mines relied on diesel power to run the mine facilities. Although the entire operation was tied into the Bluefish hydro grid starting in June 1991, it was found more economical to self-generate diesel electricity. In 1988-1989, two 350 kilowatts Cat D-353 diesel generators were the primary power units, installed in a secan addition to the mill building. In 1990 these were augmented by a 500 kilowatts Cat D398 engine. In 1993 this unit broke down and was replaced by a similar model. Heat produced by these engines was used to heat the entire mine complex. A number of portable air compressors, including an 850 cubic feet per minute Ingersoll-Rand unit and a 750 cubic feet per minute Gardner-Denver unit, were used between both the Ptarmigan and Tom operations between 1986 and 1997. A 900 cubic feet per minute Gardner-Denver air compressor used at Tom Mine starting in 1985 was taken out of service in 1990 and replaced by a new 800 cubic feet per minute Gardner-Denver air compressor. Fuel was stored in tanks aggregating 5,700 gallons (mine records; Elkhorn Gold Mining Corp., 1999).

Mining Equipment

During the 1990s the following mining machinery was in use at the Ptarmigan and Tom Mines: three Eimco-Jarvis 912D scooptrams with 2.5 yard buckets, two 10 ton Eimco-Jarvis haulage trucks, one Wagner ST-4A scooptram with four yard buckets, and one Tamrock DH-107M jumbo drill. Track mining operations on Ptarmigan Mine's lower level workings used the following equipment: two Eimco 12-B mucking machines, two Atlas-Copco LM-56 mucking machines, two Mancha locomotives, one Atlas locomotive, and a number of two ton rocker-dump ore cars. Other mobile equipment included a Michigan-Clark 125B front-end loader, Cat D6 bulldozer, Ford dump truck, JCB 3D backhoe, and Cat 941 traxcavator. Buildings at the Ptarmigan Mine included a compressor house, a 16 foot x 45 foot garage, a 24 foot x 50 foot mechanic shop and warehouse, a 40 foot x 60 foot dry complex, a six room mine office 12 foot x 54 foot in size, a building which covered a septic tank, and a group of four portable trailers which originally were used as bunkhouses, but was later partly converted into the new dry when the old one burned down in June 1994 (mine records).

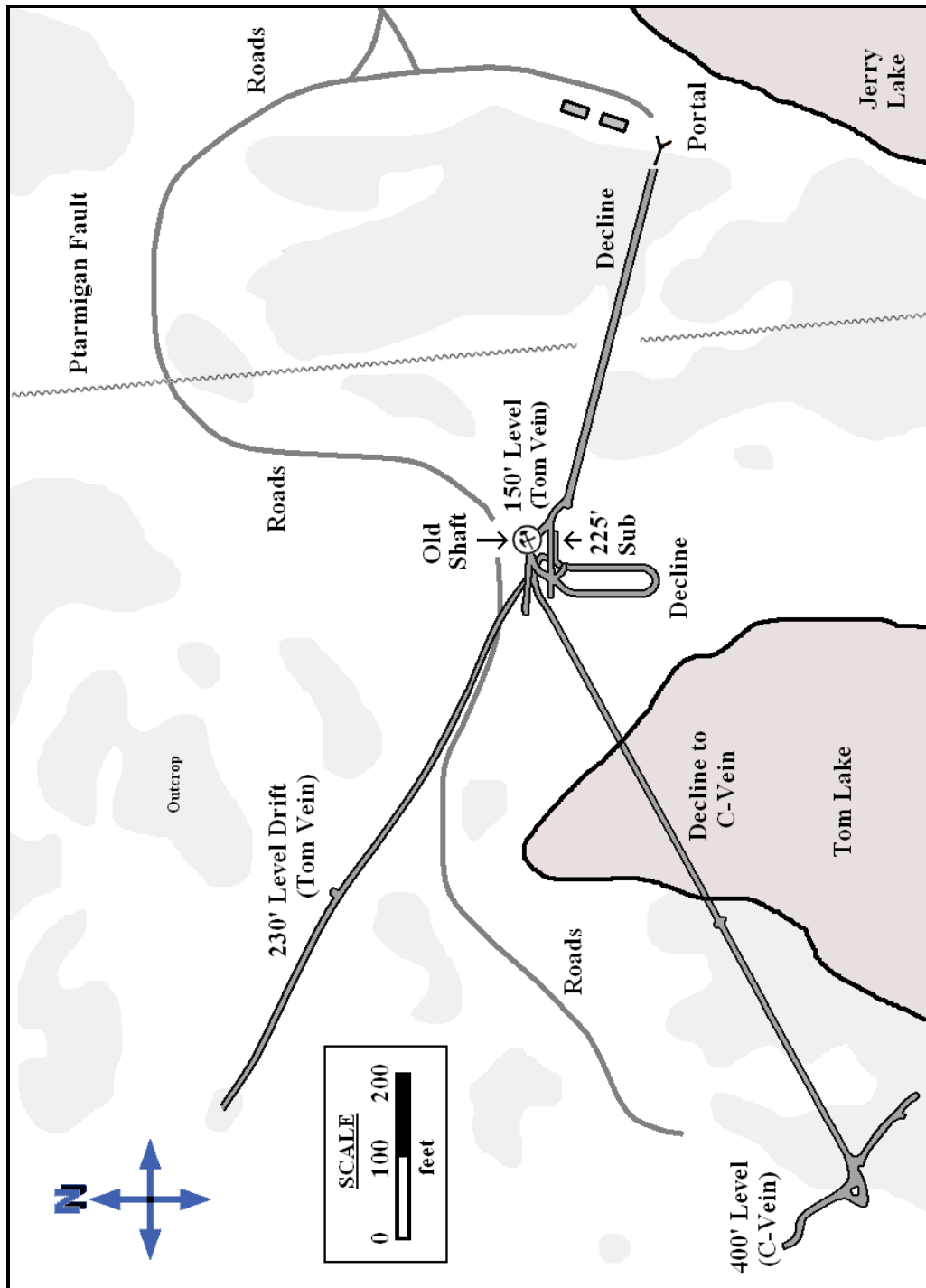


Figure 7. Tom Mine surface and underground plan. Information is believed to be up-to-date at 1997 closure.

Tom Mine C-Vein

Development work in the early 1990s focused on Tom Mine, which Treminco viewed as the most prospective venture of both operations. Diamond drilling down at Tom during 1989-1990 highlighted good gold values within the C-vein, located to the west of the Tom vein. Reserve was estimated at 70,000 tons grading 0.35 ounces per ton gold to a depth of 400 feet. The C-vein was the best source of ore in 1990-1991 as it was clear that reserves at Ptarmigan Mine were running out. The C-vein decline commenced August 1990 and was completed in December 1990. The decline (-15%, 10 feet x 13 feet in dimensions) was advanced 1,184 feet to the 400-foot level. Driving of an Alimak ventilation raise to surface took place during 1991 (Treminco Resources Ltd. Annual Reports, 1990-1991). (see Figures 7 and 8)

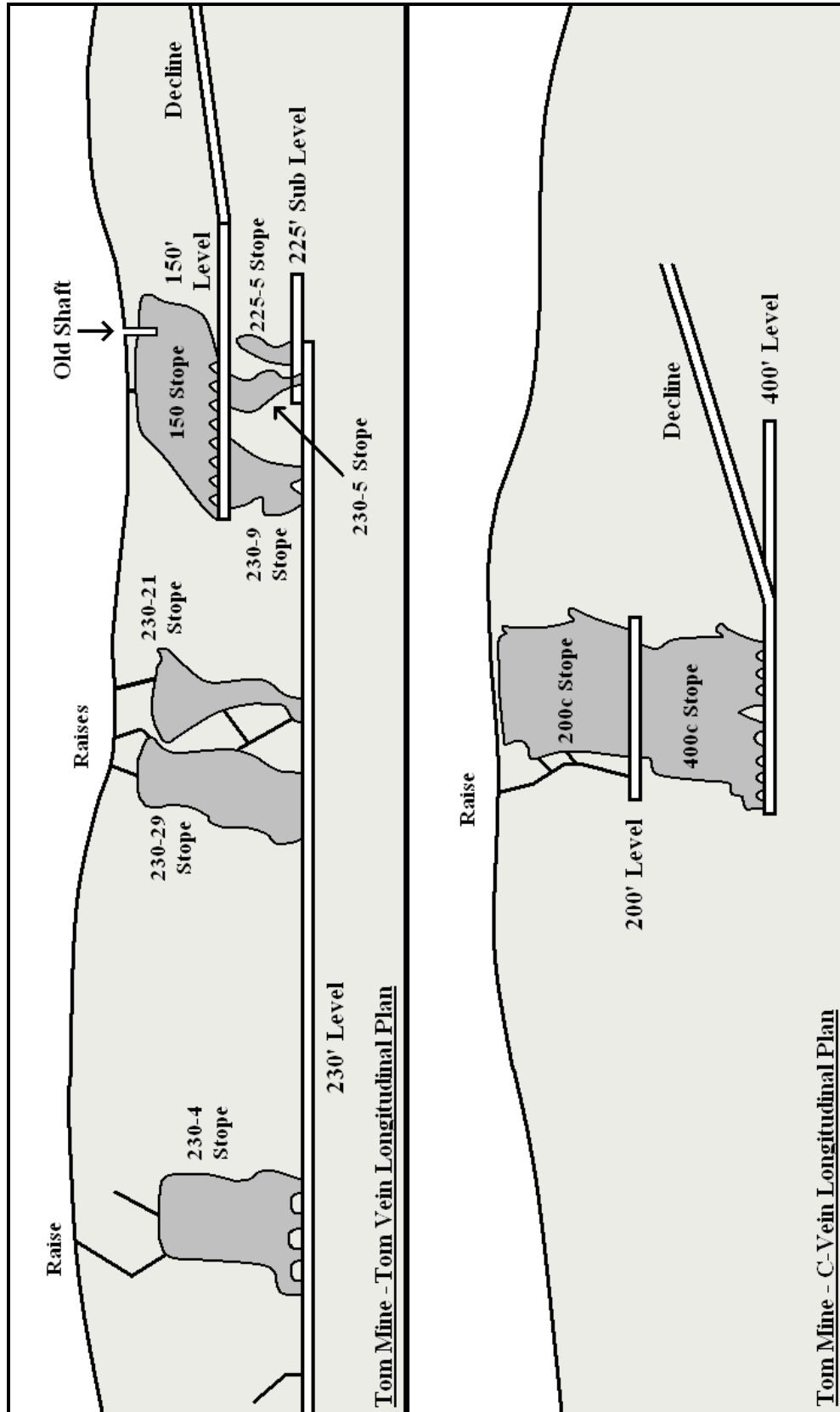


Figure 8. Tom Mine sections. Tom vein and C-Vein. Information is believed to be up-to-date at 1997 closure.

Crestaurum Option

In 1990 it was calculated that known reserves at Ptarmigan Mine would be mined out by June 1991. To increase ore reserves, Treminco investigated the purchase of other properties in the Yellowknife area. One of these was the old Crestaurum Mine which had a large reserve of 320,000 tons grading 0.22 ounces per ton gold. The agreement would have been a joint-venture between Treminco and Royal Oak Mines Incorporated (Treminco Resources Ltd. Annual Reports, 1989-1990).

The company eventually had to opt out of the joint venture at Crestaurum because of a difficult financial period that occurred in 1991-1992. Treminco was losing money on the Ptarmigan and Tom Mine and slowly entering into debt due to the high valuation of the Canadian dollar on world markets. Capital expenditures were halted and no exploration or major development was undertaken during 1992. 19 of 30 employees were laid off and combined reserves from both properties at July 31st 1992 was 55,100 tons grading 0.25 ounces per ton gold. Mining shifted from large-scale stope mining to selective extraction of high-grade portions of the orebodies using reduced crews (Treminco Resources Ltd. Annual Reports, 1991-1992). In June 1992, underground diamond drilling consisting of 6 short holes below 750-foot level failed to locate ore-grade mineralization in the depth extensions of the Ptarmigan vein, although the company reported that more work was required. Additional drilling conducted later below the 900-foot level intersected the vein, but no economic ore was found (Treminco Resources Ltd. Annual Report, 1992).

Mine Managers

Mine manager included James Almaas (1987-1989), Paul O'Hara (1989), Tom McGrail (1989-1990), Maurice Meagher (1990-1991), Len Palmer (1991-1997). The mine employed on average 20 to 30 persons (Robert Treneman, pers. comm.).

The C-vein at Tom Mine was major source of ore during 1991-1992. 15,000 tons of ore grading 0.40 ounces per ton were mined and milled during the fiscal year-end July 31st 1992. A decision was made to extend the ramp down to the next level and develop the western ends of the C-vein, although it was recognized that this material was much lower grade. The original Tom vein was only a minor source of ore during this period, with some material being mined from remnant stopes on the 230-foot level. Surface exploration of the #2 vein northeast of the Tom Mine showed some interesting results (Treminco Resources Ltd. Annual Report, 1992). In 1990, four holes were drilled on the #2 vein over a 400 foot length. Further drilling in 1992 outlined the #2 vein 'East Extension', which was 60 feet long and four feet wide. In October-November 1992, a small bulk sampling program by open cut extracted 263 tons of vein material for processing in the mill. Grade of the material was reported to average 0.35 ounces per ton gold (mine records).

In 1993, the mine conducted two diamond-drilling programs. A two-hole program totaling 1,700 feet tested the Ptarmigan vein and a four hole program totaling 1,000 feet tested Tom Mine. Production operations at the mine suffered from extremely cold weather in the winter of 1993-1994. Mechanical problems forced temporary closures and layoffs in early 1994, including a ball mill failure in April and a crusher failure in May. Crews were returned to work following repairs in both cases (mine records). Operations during 1993-1994 were constrained by lack of minable ore above the mining levels. The drill program of 1993, which tested both mines, resulted in a large increase in reserves within the Tom and C-veins. Drilling and development on the 900-foot level of the Ptarmigan Mine exposed ore-grade mineralization continuing below the shaft bottom (Treminco Resources Ltd. Annual Report, 1994).

Lack of Reserves

During 1994-1995, production was constrained by lack of development ore. In December 1994 production operations at Tom Mine ceased and did not resume until the following summer. In June 1995 an extensive review of ore reserves at Ptarmigan and Tom Mines was completed. It demanded for the start of further exploration to add to a dwindling stock of ore supply. The resultant diamond-drilling program of 1995-1996, both underground and on surface, failed to improve reserves at the properties. Probable reserves were identified below the 900-foot level, but they were not considered economic (Treminco Resources Ltd. Annual Reports, 1995-1996; mine records). It was estimated, in 1994, that CDN \$2 million in funds would be required to deepen the mine, plus another \$500,000 to increase the capacity of the tailings pond. Treminco was never able to raise this kind of cash (Treminco Resources Ltd. Annual Report, 1994).

Final Closure

Lack of ore reserves, skyrocketing costs of production, and finally, during 1997, the drop in the price of gold, sealed the fate of the Ptarmigan and Tom Mines. Mining operations ceased at Ptarmigan Mine in May 1997 and at Tom

Mine in July 1997; milling operations concluded in August 1997 and gold bar #431 was poured (Tremenco Resources Ltd. Annual Report, 1997; mine records). One man was left to act as caretaker. The company tried to sell the property in 2000, but the N.W.T. government overturned the sale due to the owing of back-taxes at the Ptarmigan property. The property was abandoned by Tremenco following its insolvency. Buildings and equipment not destroyed or stolen since then were sold at public auction in 2005.

Year: (*)	<u>Combined (Ptarmigan and Tom)</u>			<u>Ptarmigan Mine</u>		<u>Tom Mine</u>	
	Ore Milled:	Grade:	Gold:	Ore Milled:	Gold:	Ore Milled:	Gold:
1987	19,285 tons	0.24 oz/ton	3,667 oz	-	-	-	-
1988	22,668 tons	0.35 oz/ton	6,806 oz	-	-	-	-
1989	44,450 tons	0.46 oz/ton	16,433 oz	-	-	-	-
1990	62,828 tons	0.30 oz/ton	17,955 oz	-	-	-	-
1991	57,877 tons	0.28 oz/ton	14,866 oz	-	-	-	-
1992	46,652 tons	0.34 oz/ton	15,280 oz	-	-	-	-
1993	35,781 tons	0.22 oz/ton	7,248 oz	-	-	-	-
1994	28,902 tons	0.24 oz/ton	6,700 oz	-	-	-	-
1995	20,514 tons	0.22 oz/ton	4,250 oz	20,280 tons	4,150 oz	234 tons	54 oz
1996	15,461 tons	0.22 oz/ton	3,485 oz	12,404 tons	2,830 oz	3,057 tons	651 oz
1997	11,333 tons	0.23 oz/ton	2,589 oz	5,497 tons	1,222 oz	5,836 tons	1,367 oz
Total:	<u>365,751 tons</u>	<u>0.27 oz/ton</u>	<u>99,279 oz</u>	-	-	-	-

Table 4. Ptarmigan/Tom Mine production, 1986-1997. Ore was milled on a custom basis at Giant Mine from 1986 to June 30th 1989. In July 1989, Tremenco's mill went into operation at the Ptarmigan Mine. Production records for both mines separately are not available for all years. (source: Tremenco Resources Ltd. Annual Reports; mine records)

(*) Tremenco Resources Ltd. fiscal year end July 31st

Remaining ore reserves at Ptarmigan Mine would appear to be small, with an estimated 5,000 to 10,000 tons of ore remaining as stope and shaft pillars. Although drilling on other veins and on depth extensions of the Ptarmigan vein has been completed, they have not been confirmed economically (Robert Treneman, pers. comm.). Total production during Tremenco Resources Limited's period of operation at the Ptarmigan and Tom Mines is listed in **Table 4**.

Exploration Since Mine Closure

No exploration work has been conducted since closure in 1997. Robert Carroll of Yellowknife staked the 'Ely' claim overtop of the Ptarmigan mine site in August 2000. Pat Hungle staked the 'Tom' claim overtop of the Tom mine site in February 2003. Surface prospecting continues.

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RANNEY HILL

High-Graded (Abandoned)

Gold

62° 33' 00" 114° 22' 30"

Years of Primary Development: 1930s

Mine Development: 100' inclined tunnel

Years of High-Grading: c.1939

High-Grading: unknown

Introduction

The Ranney Hill property is located on the south end of Rater Lake, northeast of Ranney Hill and a short jaunt north of the Martin Lake hiking trail. These areas are accessible from the Vee Lake Road, nine kilometers north of Yellowknife, NWT. The author visited the property numerous times during 2002 and 2003. Only the ruins of a cabin and the collapsed mine tunnel can be found today.

Brief History

Ranney Hill is named after Winslow C. Ranney, a prospector and trapper who first came into the Yellowknife region in 1934. Numerous claims were staked in this area between 1935 and 1937 by Mr. Ranney. Development was undertaken by a small crew under Ranney's supervision during 1937 and 1938, and apparently Ranney utilized a small milling unit to recover some gold from a short mine tunnel around 1939. Further exploration was conducted at this site during the 1940s by Ranney Gold Mines Limited, but no further mining development was undertaken. Robert Carroll staked the 'R+R' claims in 1998 and performed some minor diamond drilling on the #1 vein, but the claims lapsed in 2003.

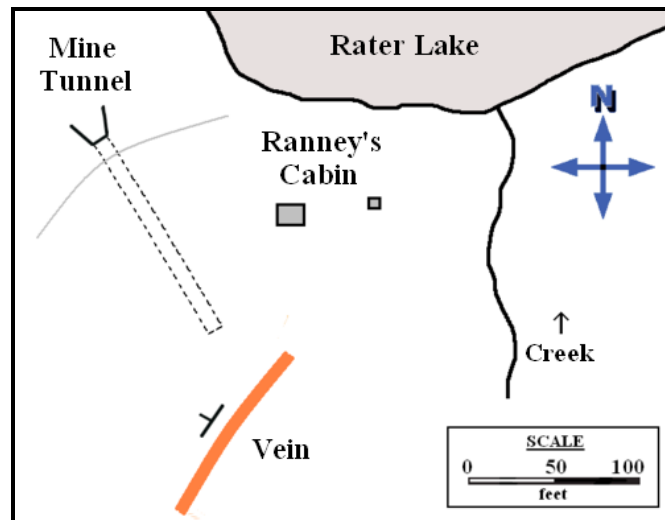


Figure 1. Ranney Hill property map.

Geology and Ore Deposits

The property is underlain by volcanic rocks, consisting mostly of pillow lavas with some spherulitic phases. The rocks have a general strike of north 10° east, with local deviations. The structure is tightly folded and intensely faulted. The chief structural feature of the property is the West Bay fault that cuts north-south. For the most part, it separates granitic rocks from the volcanics. The veins strike north and south and lie in shear zones. Three veins were identified in the 1930s, known as the High Grade, #1, and #2 veins. The High Grade vein is located on the southwest side of David Lake and is 50 feet long and one inch wide, with quartz occurring as narrow stringers. While high-grade, it was considered too small to be mined commercially.

The #1 vein strikes north-south and dips 50° to the east between David and Rater Lake. It occurs in a four foot wide shear zone which contains irregular quartz lenses. The vein is divided by about 300 feet of overburden and swamp. The north part is exposed for 125 feet and the south part for 290 feet. It was the focus of exploration. The #2 vein lies further north of Ranney Hill and was traced for a length of 840 feet (Riley, 1937).

Winslow C. Ranney (1930s)

Work was focused on the #1 vein between David and Rater lakes. According to a brief mention in a 1940 government report, Winslow C. Ranney had a small “hand-mill” on the property and had plans to process some gold ores (Meikle, 1940). The hand-mill was most likely a homemade contraption, probably an oil drum converted into a type of tumbler. No mention is made of production.

Ore was likely derived from the surface pits. A short tunnel was also blasted to follow the strike of the #1 vein as its strike changed to the northwest. The tunnel was apparently 100 feet long and 100 feet below from the surface of the vein. As the tunnel begins at the base of a 30 foot hill, the tunnel drives down 70 feet vertically below the portal elevation at approximately a 30 to 40° angle. The surface pits suggested that the #1 vein dipped to the east. In the tunnel, it was found that the dip of the vein change to the west. No other information is available concerning the mine workings or production (Ranney Gold Mines Limited, 1940s).

Exploration Since Mine Closure

A company called Ranney Gold Mines Limited owned the claims in the 1940s and conducted diamond drilling on the large property originally staked by W.C. Ranney. Robert Carroll staked the ‘R+R’ claims in 1998. He conducted some shallow x-ray diamond drilling on the main vein, but results of this work are unknown (Robert Carroll, pers. comm.). The claims lapsed in 2003.

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Personal communication: Robert Carroll.

	RAYROCK <i>Producer (Remediated)</i>	Uranium
		63° 27' 00" 116° 32' 45"

Years of Primary Development: 1955-1959

Mine Development: adit tunnel, winze to 1,000', 9 levels (15,579' dev.)

Years of Production: 1957-1959

Mine Production: 78,781 tons milled = 458,020 pounds U₃O₈ concentrate

Introduction

The former uranium producer, Rayrock Mine, is located 156 kilometers northwest of Yellowknife, NWT on the northwest side of Sherman Lake, east of the Marian River. It operated between 1957-1959 when a depletion of economic reserves forced a closure. Most equipment and some buildings were removed in 1960-1961. Any remaining buildings were demolished in 1987. In 1996, radioactive waste remediation began and monitoring is ongoing today. The site was visited by the author in September 2009. An old all-weather road connects to the site from the north-end of Marian Lake, now used as a winter road to connect to the Snare River hydro plant and the old Colomac Mine.



Figure 1. *Rayrock Mine area.*

History in Brief

The Rayrock deposit was originally staked in 1948 as the 'Bob' claims by the Sandy and Hubert Giauque. In 1950, the property was re-staked as the 'MM' claims by M. Martin, and again in December 1951 as the 'Beta' group by Bert Bolduc and Jack Stevens. American Yellowknife Mines Limited acquired the property in 1953. This company, renamed Rayrock Mines Limited in 1954, continued work into 1955 when a diamond drilling program was initiated to test the radioactive zones. A decision was made to go underground, and a small mining plant was shipped to Marian Lake.

An underground adit tunnel was driven into the radioactive zone and sufficient ore of high-grade tonnage was uncovered to warrant a production decision. The mine achieved production in June 1957, but the company was unable to find economic ore at greater depths and the mine closed in July 1959. No work has been done since.

Geology and Ore Deposits

The minesite is underlain by granodiorite and quartz monzonite. The granodiorite is traversed by quartz stockworks which strike parallel with the Marian River Fault. Subsidiary tension fractures branching from the main fault appear as zones of intense fracturing within the quartz veins. It was reported to have appeared that late in the formation of the stockwork, pitchblende and intimately associated hematite was introduced in open fractures and crushed zones in the quartz and in altered granitic rock. The principle pitchblende bearing veins and breccia zones on the property were found in zones of subsidiary tension fractures branching from the main fault and appear as zones of intense fracturing within the quartz veins. The #6 zone, the primary deposit at Rayrock, outcrops on surface 900 feet west of the fault, somewhere between 225 feet and 300 feet high above the old portal entrance. The ore shoots have been found to be erratic both in distribution and in uranium grades (Byrne, 1957).

Rayrock Mines Limited (1955-1959)

Extensive surface diamond drilling at the property during the winter of 1954-1955 outlined a highly mineralized fracture zone over a length of 375 feet and to a depth of 300 feet. Underground development to test the #6 zone and other deposits was authorized late in 1954. A small mining plant was shipped from the defunct O'Connor Lake Mine north of Fort Smith, NWT. 250 tons of equipment and supplies were brought to the site by tractor train and airplane in March 1955 (Byrne, 1957).

Adit Driving 1955

The portal was collared in April 1955 and the adit level crosscut was driven for a length of 890 feet in a northward direction to reach the #6 zone at 225 feet depth, from which point a total of 930 feet of other lateral work was completed within the zone, along with 531 feet of raising to October 31st 1955. Also encountered during the adit driving was the #1 zone about 750 feet from the portal. In both zones, it was found that the deposit remained in similar conditions as that uncovered on the surface, with pitchblende occurring in pods and lenses within and beyond the main fracture zone in the quartz stockwork (Rayrock Mines Ltd. Annual Report, 1955).

Late in 1955, a sub-level was established 100 feet above the adit level, or 125 feet below surface, to develop the upper extension of the #6 zone. During 1956, a short winze was sunk 75 feet to open up a sub-level at a depth of 300 feet. Three levels were under development. The goal was to increase ore reserves as quick as possible before March 31st 1956, the deadline set by the Federal government for special uranium pricing contracts. Samples of Rayrock ore were shipped to Ottawa for mineralogical testing in the winter of 1955-1956 to find a suitable recovery method. Small and large scale test work indicated favorable uranium recovery using a standard leaching process.

Delivery of 2,500 tons of material and equipment by tractor train during January-April 1956 allowed for the commencement of construction of mine and camp facilities during the summer of 1956. Most buildings were closed in by November, and installation of mill equipment was initiated during the winter of 1956-1957. Construction of a powerline from the Snare River hydro plant to Rayrock Mine commenced late in 1956 (Rayrock Mines Ltd. Annual Report, 1956).



Rayrock Mines Limited

Figure 2. *Aerial view of Rayrock Mine, 1957.*

Winze Sinking

A 435 foot, 3-compartment production winze (7 feet x 18 feet dimensions) begun in May 1956 from the 225-foot adit level was completed early in 1957. This winze was sunk in its first stage to a depth of 660 feet below surface and the 375-, 500-, and 625-foot levels were established (Rayrock Mines Ltd. Annual Report, 1956 and 1957). An underground hoist-room was fitted with a Canadian Ingersoll-Rand 48 inch x 36 inch 2-drum electric hoist (Byrne, 1957).

Road Cleared

A 56 kilometer road was started in May 1956 from the north-end of Marian Lake, a joint venture project between Rayrock, Consolidated Northland Mines Limited, and the Government of Canada. It was completed in November 1956 and allowed for easy freight haul to the property. Major milling and mining equipment, dismantled at the Negus Mine in Yellowknife, was barged to the end of Marian Lake and trucked up this road during the fall of 1956. At October 31st 1956, 1,702 tons of freight had been delivered by barge to the Marian Lake wharf. Only 125 tons of this material had been trucked to the mine at that date. Aircraft were responsible for the delivery of 442 tons of freight,

and together with the material brought in by tractor train early in 1956 (2,500 tons), total freight delivered to Rayrock Mine during the fiscal year ending October 31st 1956 was 4,769 tons (Rayrock Mines Ltd. Annual Report, 1956).

An ore reserve of 111,250 tons grading 0-408 % uranium oxides (U₃O₈) above the 325-foot level and within the #6 zone was calculated on October 31st 1956. Mining development during 1956-1957 was focused on sinking of the winze and development of the new levels within the #6 zone. 2,075 feet of drifting and 1,214 feet of raising was completed on the 375-, 500- and 625-foot levels during the fiscal year-end October 31st 1957. While work on the 375- and 500-foot levels showed good ore, drifting on the 625-foot level indicated only minor zones of radioactivity. Pre-production development included the opening of stopes on the 125-, 225- and 300-foot levels. On the 125-foot level, a total ore length of 395 feet was exposed along the #6 zone. Crosscuts to the #1 and #7 zones gave inconclusive results. On the 225-foot level, a total ore length of 275 feet was exposed within the narrow #3 zone (Rayrock Mines Ltd. Annual Report, 1956 and 1957).

Production Starts

Milling operations began on June 1st 1957 using low-grade surface stockpiles for millfeed. The first uranium concentrate ('yellowcake') was produced in July 1957. Problems relating to the distant location of the mine, together with a delay in activating underground stopes due to a complex ore structure, resulted in limited production operations for the first five months. Commercial production commenced on November 1st 1957, the beginning of Rayrock Mines Limited's fiscal year (Rayrock Mines Ltd. Annual Report, 1957).

Milling Operations

Mine-run ore was received in the coarse ore-bin near the entrance to the adit. From here, the ore was delivered into the crushing plant, a 72 foot x 26 foot building housing screens, and two-stage crushing units. Initial crushing was accomplished using a Telesmith jaw crusher (18 inch x 32 inch), with oversize pieces being re-crushed in a 3 foot Symons cone crusher. Conveyor ways transferred the ore into the mill building, a 144 foot x 106 foot structure. The fine ore was then processed through a 5 foot x 8 foot Denver ball mill, which in turn fed its product into a Dorr rake-type classifier. From the grinding circuit, pulp was pumped into the leaching circuit consisting of a 22 foot x 10 foot Dorr thickener, four agitation tanks, and two 8 foot x 16 foot Oliver filters. Filtrates from the two filters were clarified in a Whitco unit, and pregnant solution purified and concentrated in an ion exchange system. The uranium rich eluate from the process was neutralized by further agitation and the uranium precipitated by the introduction of magnesia. The resulting uranium precipitate was filtered and washed, then dried, with the product sealed in drums for shipment to the Eldorado refinery in Port Hope, Ontario (Byrne, 1957).

Freight Delivery 1957

The bulk of the equipment and freight for the construction of the mill was transported to Rayrock during the winter of 1956-1957 following the completion of the Marian Lake road. During the fiscal year-end October 31st 1957, 1,019 tons was delivered by tractor train, 705 tons by barge to the Marian Lake wharf, 274 tons by aircraft from Yellowknife, and 956 tons by truck from Marian Lake (Rayrock Mines Ltd. Annual Report, 1957).

Power Plant

Electricity was purchased from the Snare River hydro plant, located 30 kilometers east of Sherman Lake. Two backup diesel generators, a Ruston and a Crossley engine, were available to supply 400 horsepower to the camp and plant if needed. Air power was supplied by two electrified Bellis-Morcom air compressors totaling 2,500 cubic feet per minute. There were also two backup diesel-powered air compressors. A 100 horsepower Inglis electric boiler and a 100 horsepower Spencer boiler supplied heat to buildings. Water supply of 35,000 gallons per day was supplied from Sherman Lake (Byrne, 1957).

Rayrock Camp and Crew

In 1957, the operation employed 145 and in 1958, with steady operations, it was reported that 135 employees were working at Rayrock Mine. In March 1959, the last date for which information is available, the mine employed a total of 136, including 10 natives who worked manual labour jobs. By department, this workforce was reduced to: mining, 54; shops, 17; general surface, 13; contractors, 19; mill, 19; laboratory, 8; cookery, 12; and office, 13 (mine records). Most of these employees were single men housed in four bunkhouses with total capacity for 140 persons, and a staffhouse for 14 persons. The cookery was rated to handle 125 men at one sitting, and the company townsite contained almost twenty residences including four duplex houses. In 1957, it was reported that the Rayrock townsite contained nine residences including three duplex units, accommodating 12 families. There were also eight school pupils in that year, housed in a small school house on the property. Recreation was provided by an indoor curling rink and recreation room with café, games, and library, built in 1958. Additional family housing was erected in 1958.

Population of the townsite and mine camp in 1958 was reported as 17 families and 135 employees (Byrne, 1957; Rayrock Mines Ltd. Annual Report, 1958). The mine manager at Rayrock during years of production was Walter E. Clarke. Other staff included Jack Boulding, mill superintendent; John Nuss, chief engineer; Dave Hempfill, geologist; and Don Salo, master mechanic. Consultants involved in the mine building and operational process included Norman Byrne, consulting engineer; and Fred Brien, metallurgical engineer (Byrne, 1957).

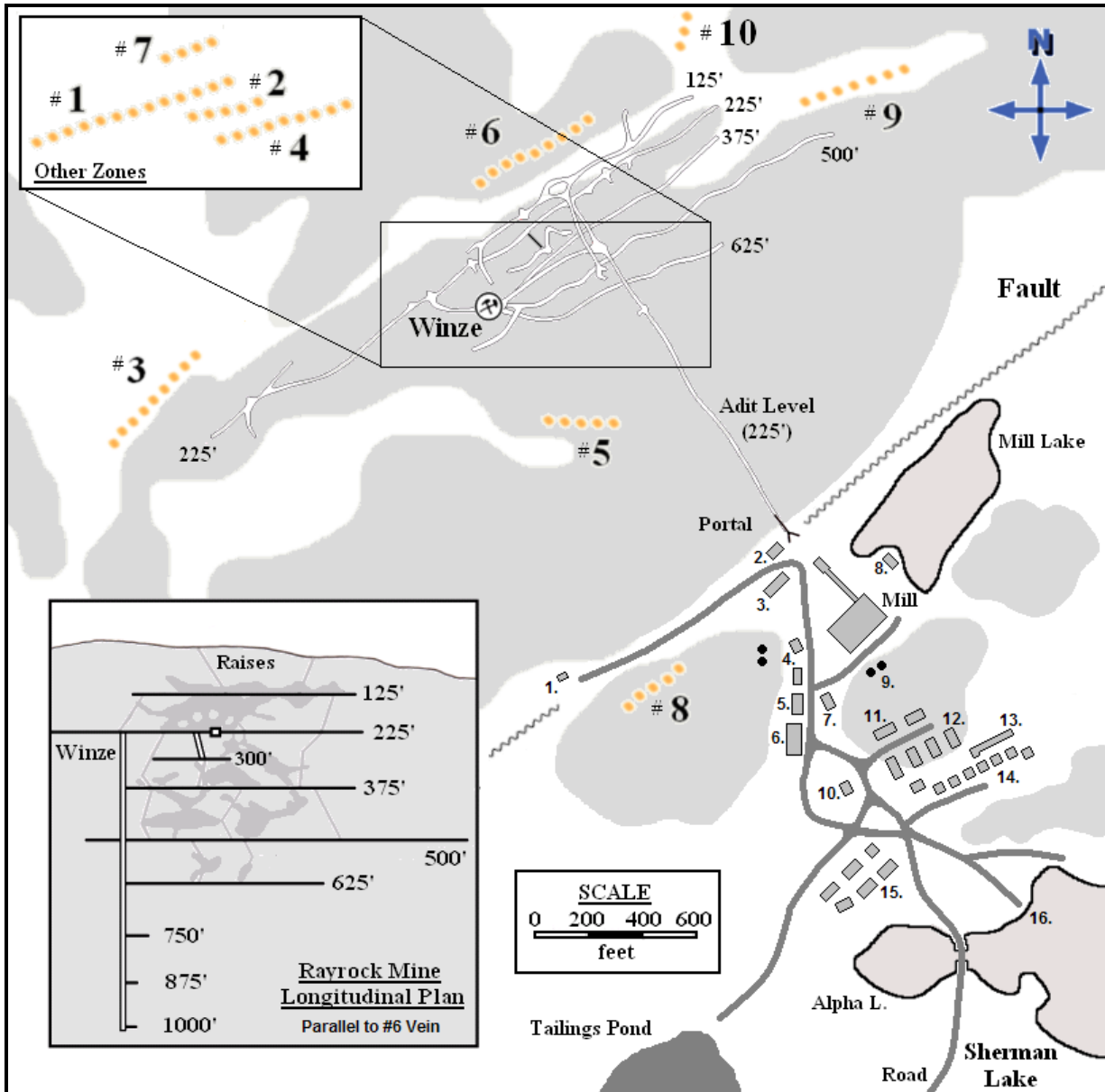


Figure 3. Rayrock Mine surface and underground plan, 1959.

(Building Legend: 1, power magazine. 2, mine dry. 3, machine shop. 4, shops. 5, office/warehouse. 6, cold warehouse. 7, powerhouse. 8, pumphouse and compressors. 9, oil tanks. 10, boiler. 11, cookery. 12, bunkhouses. 13, recreation hall and curling rink. 14, houses. 15, duplex houses. 16, dock.)

Continued Mining Developments

During 1957, development of the three levels of the winze were underway to explore the #6 zone. Lateral exploration of these horizons were completed by year end as well as some further sub-level development and raising between all levels. Diamond drill exploration was conducted in the upper levels to seek out any remaining deposits between stopes. Drilling later focused on exploring the downward extension of the zone beyond the present depth of the winze. As development of these orebodies continued during 1957, it became evident that the ore characteristics were

far more erratic than anticipated. As a result, the operation was plagued with sections that pinched-out with no advanced warning. Dedicated assaying procedures were enacted to maintain an accurate grade in mining operations and close supervision of ore grades and stoping procedures became necessary to ensure steady production (Rayrock Mines Ltd. Annual Reports, 1957-1958).

Company Profits

During 1958, it was reported the Rayrock company had a profitable fiscal year (Nov 1st 1957 to October 31st 1958). The sale of uranium concentrates provided a revenue of CDN \$2,620,000. Cost of operations for the fiscal year was \$1,752,000 or \$43 per ton milled. Profits before write-offs for the year were about \$868,000. The 1958 fiscal year was also the first full year of production, when mill recoveries averaged 97% with a concentrate grade of 75.8% U₃O₈, the best recorded in the uranium industry (Rayrock Mines Ltd. Annual Report, 1958).

During 1958 it was anticipated that the #6 zone would continue as a strong ore structure. An extensive development program was underway during 1958-1959 to explore the zone north of the current workings and milling capacity was scaled back to allow resources for this work. The winze was deepened 380 feet to a depth of 1,000 feet below surface to give this shaft a final length of 815 feet. Three new levels were opened up to explore the zone, at 750-, 875-, and 1,000-foot depths. Development of these levels was conducted to reach the drill-indicated ore zones but no stoping was performed during the year (Rayrock Mines Ltd. Annual Reports, 1958-1959).

Mining problems continued during 1958 due in part to high costs. Almost all developed stopes required sub-level development to because of the irregular nature of the ore. Close supervision by the geology and engineering staff was required to maintain steady production and grade control. During 1958, all levels above the 625-foot level were being mined. A major construction program was completed in 1958 with the addition of several new houses to the townsite, the erection of a recreation hall/curling rink, and an addition to the warehouse. A 2,400 foot airstrip was also cleared near the mine in October 1958 (Rayrock Mines Ltd. Annual Report, 1958).

Exploration took almost full attention during 1959 and absorbed 30% of the costs of the fiscal year ended October 31st. Over 80% of development was carried out on the 375-, 875-, and 1,000-foot levels north from the shaft. A large portion of lateral advance was conducted on the 375-foot level, from which exploration of the #6 zone was completed to within 1,250 feet of the north boundary of the property. The exploration program was extensive and very costly. It was conducted in a block of ground between the 500- and 1,000-foot levels within the #6 zone. It was found that the #6 zone structures continued very strong with many ore grade drill intersections. Subsequent development, however, showed the ore to occur in a number of small isolated lenses with no tonnage potential. Drilling to a depth of 300 feet below the bottom level (1,000 feet) resulted in disappointing results, and lateral drilling east and west of the #6 zone did not locate parallel ore structures. Geological mapping was performed in the summer of 1959 to identify additional ore zones that might become productive, but this work was futile (Rayrock Mines Ltd. Annual Report, 1959).

Year: *	Ore Milled:	Uranium Precipitate:	U ₃ O ₈ Content:
1957	14,859 tons	62,212 pounds	70.4 %
1958	40,088 tons	243,407 pounds	75.8%
1959	23,834 tons	152,401 pounds	73.0%
<i>Total:</i>	<i>78,781 tons</i>	<i>458,020 pounds</i>	<i>73.1%</i>

Table 1. Rayrock Mine production, 1957-1959. (source: Rayrock Mines Ltd. Annual Reports)

* Rayrock Mines Ltd. fiscal year-end October 31st

1959 Closure

The poor results of exploration and dwindling of available ores resulted in the phasing out of production operations, and the mine closed at the end of July 1959 (Rayrock Mines Ltd. Annual Report, 1959).

Mine Development Summary

The Rayrock Mine, in summary, has been developed by an adit entrance 225 feet below the surface outcropping of the #6 zone. About 850 feet in from this northwesterly directional crosscut, drifting occurred along the strike of the

deposit in a northeast-southwest direction. An 815 foot production winze was sunk to a vertical depth of 1,000 feet below the surface outcrop of the zone and nine levels were developed, all focusing primarily on the #6 zone. The bottom levels of the mine were not fully developed with only limited exploration carried out. The levels followed the southeasterly dip of the zone and the deposit has been explored underground over a length of 2,000 feet or more. Development consists of 15,579 feet of drifts, crosscuts, and slashing equivalent, 6,399 feet of raising (with surface breakthroughs), an 815 foot vertical production winze from the adit level, and a 75 foot vertical exploratory winze from the adit level. (see **Table 2**)

Mine Production

The Rayrock Mine was in operation between 1957 and 1959, milling 78,781 tons of ore to produce 458,020 pounds of uranium precipitate. All uranium materials were shipped for refining to Port Hope, Ontario. (see **Table 1**)

Year: *	Lateral Development:	Raising:	Underground Drilling:	Surface Drilling:
<i>Prior to 1955</i>	-	-	-	4,798'
<i>1955</i>	2,112'	531'	1,108'	7,811'
<i>1956</i>	4,010'	230'	7,535'	-
<i>1957</i>	3,186'	1,938'	4,998'	-
<i>1958</i>	2,185'	2,133'	11,956'	-
<i>1959</i>	4,086'	1,567'	19,246'	-

Table 2. *Rayrock Mine development, 1955-1959. (source: Rayrock Mines Ltd. Annual Reports)*

* *Company fiscal year-end October 31st*

Exploration Since Mine Closure

New claims were staked in 1995 and acquired by G.M.D. Resources Corp. Limited in 1996. The target of exploration was towards finding an Olympic Dam style deposits similar to the nearby NICO deposit (gold-copper-bismuth deposit) but nothing important was located in the area.

References and Recommended Reading

Byrne, J.C., 1957. Rayrock in Production. *In* The Western Miner magazine, July 1957.

Byrne, N.W., 1957. The Rayrock Story. *In* The Western Miner magazine, April 1957.

Rayrock Mines Ltd. Annual Reports. 1954-1959 (fiscal year-end October 31st).

Rayrock Mine operational records. 1957-1959.

McGlynn, J.C., 1971. Metallic Mineral Industry, District of Mackenzie, Northwest Territories. Geological Survey of Canada, Paper 70-17.



RED 24

Satellite Producer (Abandoned)

Gold

64° 06' 30" 111° 16' 15"

Years of Primary Development: 1986-1987

Mine Development: open pit, ramp portal (not completed)

Years of Production: 1986-1987

Mine Production: 7,042 tons milled = 3,053 oz Au

Introduction

The Red 24 Mine is located in the Courageous Lake region at the northeast end of Matthews Lake, 240 kilometers northeast of Yellowknife, NWT. It was a small producing operation which operated as part of the Salmita Mine project in the 1980s. The author visited the site in August 2001.

Brief History

The area was staked as the 'MC' claims before 1964. Only the 'MC #5' claim was still in good standing in 1964 as it contained the mineral showing. The 'Red #1-23' claims were staked to surround the 'MC #5' in 1964 by Giant Yellowknife Mines Limited. Copper was the target in these years. The 'MC #5' claim lapsed in 1973 and was re-staked by Knud Rasmussen as the 'Red #24' claim. Giant conducted geophysical work and diamond drilling through the 1970s-1980s, and by 1985 a sizable gold deposit had been outlined. The Red 24 Mine acted as a satellite operation to the adjacent Salmita Mine, also owned by Giant Yellowknife Mines Limited. Initial production was derived from four benches of an open pit in 1986. Ore was trucked to the Salmita mill. An underground ramp was planned to tap the lower section of the deposit, but no work was done. The material proved difficult to treat and the gold recoveries were poor. A fifth bench was mined in 1987 when Giant entered into an agreement with Giant Bay Resources Limited for the installation of a pilot plant to test new bioleach technology, using Red 24 ore. No work was being done after 1987 when Salmita Mine closed.

Geology and Ore Deposits¹

The property is underlain by a wedge of mafic volcanics extending from the southeast corner two thirds of the way to the north boundary. This wedge separates sediments to the east from felsic volcanics to the west. Auriferous zones on the property are hosted by the felsic volcanics adjacent to a diabase dyke. Mineralization in the Red 24 open pit consists largely of layered felsic tuffs containing up to 10% acicular arsenopyrite concentrated in layers, with minor pyrite, galena, and sphalerite occurring near the footwall. Gold is associated with arsenopyrite-rich layers. The upper part of the mineralized zone is brecciated.

Giant Yellowknife Mines Limited (1986-1987)

During development at the Salmita Mine, the company realized that a profit may be made in the mining of a bulk sample from the 'Red #24' claim, where a small yet high-grade gold showing existed. Late in 1985 plans were made to mine known ores down to the 40 meters depth and, if warranted, perform additional development to bring Red 24 online as a satellite producer to the Salmita Mine. The Red 24 deposit, although high-grade, differed in metallurgy and style from the Salmita Mine deposits, so company geologists knew that much work was required to recover the gold economically. Metallurgical tests suggested 75% recoveries using the Salmita mill. Ore reserves themselves, calculated in February 1986, were 12,321 tonnes grading 19-62 grams per tonne. The plan was to mine the ore via open pit and underground methods. The open pit was to have a total depth of 15 meters mined in three five-meter benches, with total excavated volume of 5000 meters³. Access to the benches was via 15% ramps. An underground ramp was to be driven at a -15% grade for 149 meters to mine all ore down from 15-meter level, plus 24 meters of lateral work on the 40-meter level to provide drawpoints for stopes. There was also to be a 21 meter drop raise. Mining methods used would be modified long-hole stoping. It was planned to mine over 9,000 tonnes of ore and truck it for processing.

Equipment

Equipment to be used in 1986 included: a Gardner-Denver 3700 airtrac drill and 1,050 cubic feet per minute air compressor (Arctic West contractor equipment), a Jarco Jumbo drill for decline development, a 600 cubic feet per minute electric compressor and 700 cubic feet per minute compressor, a 250 KVA generator, 20 ton Mack haul trucks, a 988 Cat loader, and Jarvis Clark 2-½ yard scooptrams.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

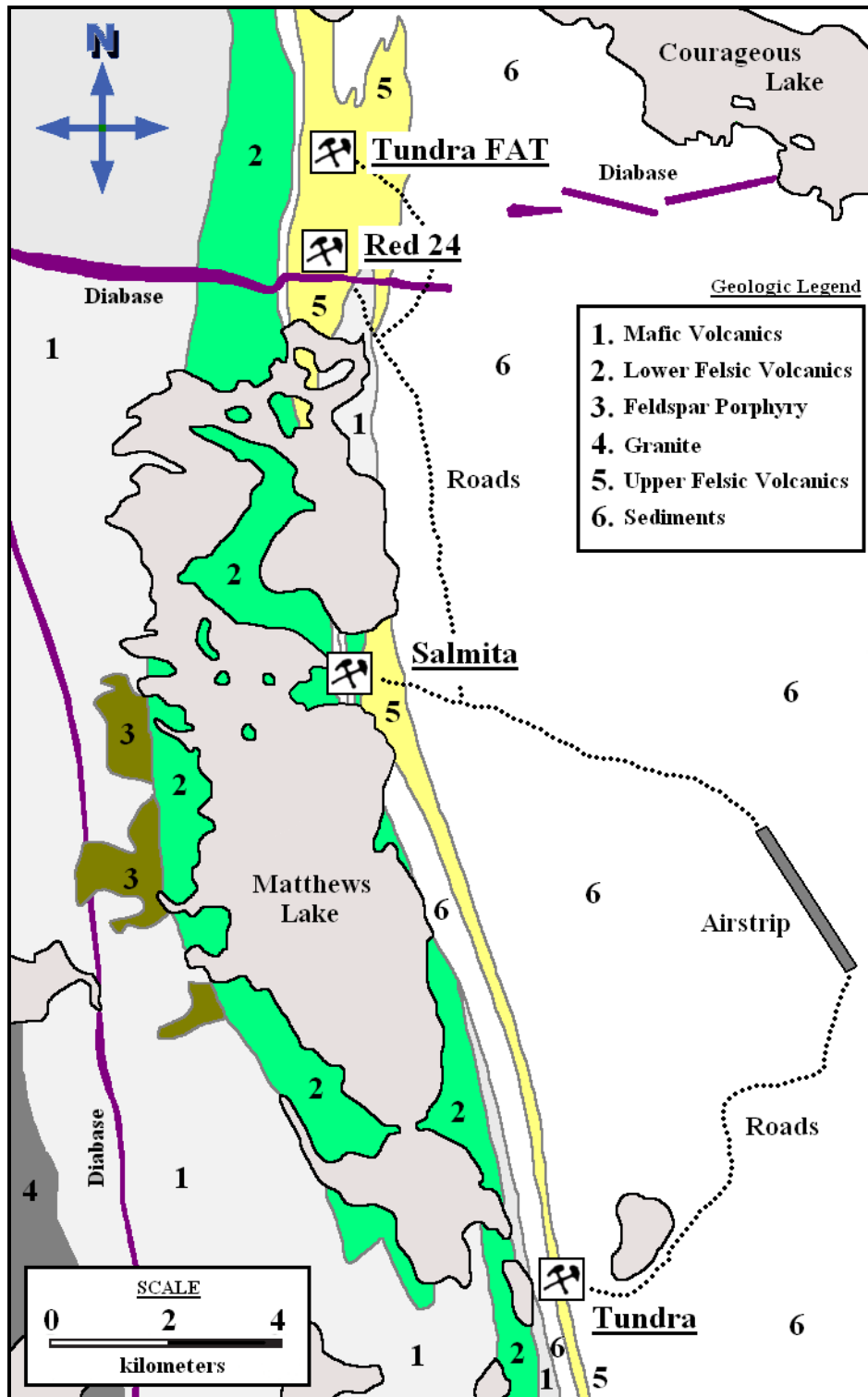


Figure 1. Matthews Lake area geology and location of Red 24 Mine.

1986 Production

Four open pit benches were mined between February 1st 1986 and April 5th 1986, and the Salmita mill treated 6,521 tons of ore to produce 2,711 ounces of gold. The material was found difficult to treat in a cyanidation plant, and gold recovery was very low at around 75%. The open pit attained a depth of 22 meters in April 1986. Underground ramp operations were terminated on February 19th 1986 due to severely cold temperatures. The decline project was not completed and so far as known, no excavation, aside from portal blasting, was accomplished.

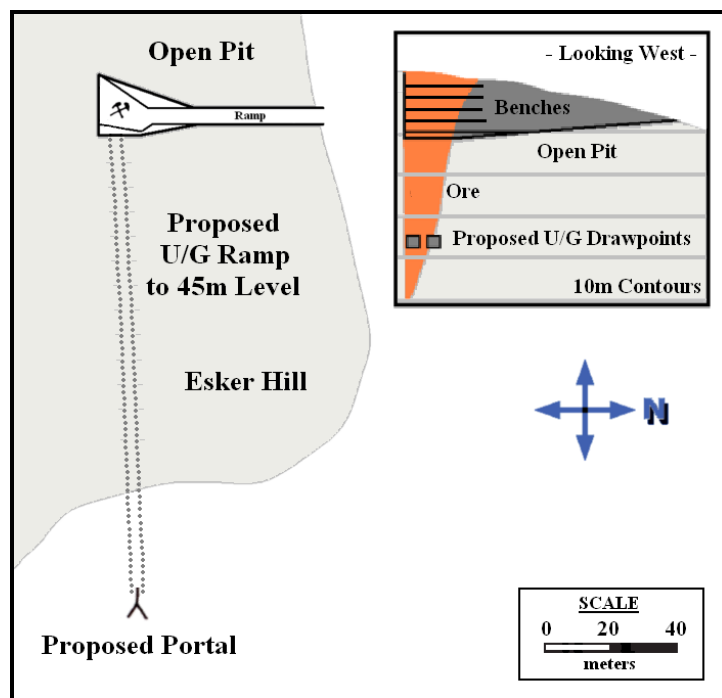


Figure 2. Red 24 Mine property plan, 1986. The proposed underground ramp was never developed.

Buildings and Staff

Onsite buildings in 1986 included a trailer for office and lunchroom, and a generator and compressor shack, all of which were portable structures. Bill Muir was mine manager of the Salmita and Red 24 operations during this period. Nick Majacich was mine superintendent. Malcolm E. Robb and Tony Ransom were mine geologists involved in the project.

1987 Operations

In late 1986, Giant approached Giant Bay Resources Limited, a company developing a bioleaching technology to recover gold from refractory ores. A deal was made in which Giant Bay would install a 10 ton per day pilot plant in the Salmita mill to process a batch of Red 24 Mine ore, thereby providing an effective flowsheet for Red 24 ore and allowing Giant Bay to develop a functional bioleach plant. Tests in January 1987 suggested an improved recovery of 91% based on a 0.31 ounces per ton gold sample of Red 24 ore. Open pit mining recommenced in the summer of 1987 and a fifth bench was mined with ore being excavated by a Drott-50 backhoe. Between July and August 1987, 521 tons were processed at the bioleach plant at Salmita, producing a 9.7-kilogram (342 ounces) gold bar, with a calculated recovery of 92.4%.

Overall, the Red 24 operation was found to be a marginal operation with high operating costs. No underground mining took place because of the impracticalities of the underground project. It was estimated that 8,000 tons of reserve could be recovered by the underground project. More diamond drilling was required to determine the deposit's mineralization at further depth (all information from Salmita Project records).

Exploration Since Mine Closure

The claim lapsed and was re-staked by Glen Warner as the 'Red 25' claim in 2001. The property was optioned to Seabridge Gold Incorporated and exploration of the area is ongoing as of 2008.

References and Recommended Reading

Giant Yellowknife Mines Ltd. Annual Reports. 1985-1987.

Salmita Project/Red 24 Operational Records. 1985-1987.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 076DSW0006



RODSTROM

Bulk Sampled (Abandoned)

Gold

62° 30' 00" 114° 26' 15"

Years of Primary Development: 1970s, 1990s

Mine Development: open cuts

Years of Bulk Sampling: 1979, 1997

Bulk Sample: 14 tons shipped @ 0.95 oz/ton Au

Introduction

The Rodstrom property is located 1½ kilometers north of Long Lake (Fred Henne Territorial Park) and is within Yellowknife city limits. Two bulk samples were obtained from the deposit in 1979 and 1997.

Brief History

The 'R' claims were staked in 1962 to protect the #15 vein and extensive diamond drilling was conducted in 1963-1964 by Rodstrom Yellowknife Mines Limited on what appeared to be a rich gold deposit. The #22 vein was discovered in 1964. The claims subsequently lapsed and the ground was re-staked by Dave Nickerson in 1975 as the 'Rod' group of claims. Two bulk samples were collected; one in 1979 and a second in 1997.

Geology and Ore Deposits¹

The showing occurs in the Western Plutonic Complex near the western edge of the Yellowknife Volcanic Belt. The showing is underlain by massive, fine to coarse-grained, grey and pink biotite granodiorite intruded by pegmatite and aplite, and locally containing volcanic inclusions. Areas containing volcanic inclusions appear to be important to mineralization. Gold is hosted by variously altered, quartz-filled fractures, fracture zones, and shear zones; the main ones are the #15 and #22 veins. The zones are consistent in width but do vary, from narrow cracks to four meter wide shear zones. The wallrock is invariably hematized, for 30 centimeters to one meter outward from the zone.

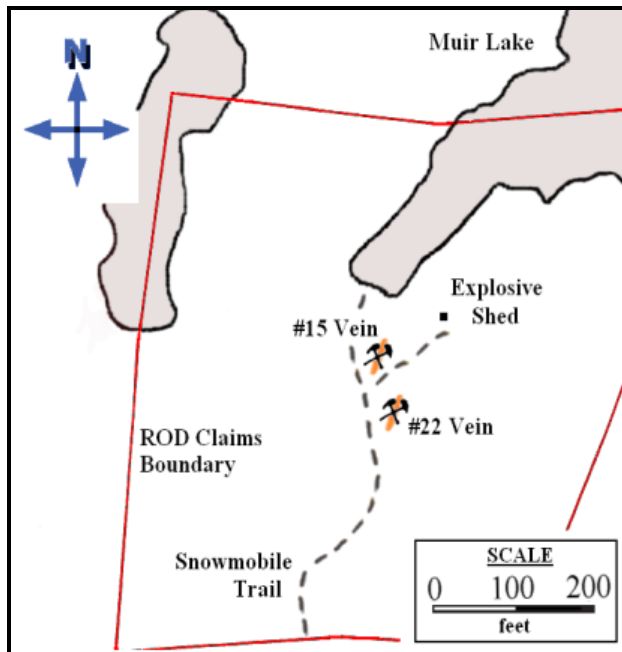


Figure 1. Rodstrom Mine property plan.

The main zones in the area comprise sheared quartz in lenses, stringers, and pods, often ribboned with aplitic material, and surrounded by sheared granitic and aplitic rocks. The quartz is lightly mineralized with pyrite and hematite (less than 1% combined), and locally galena, chalcopyrite, and sphalerite. Gold occurs freely and associated with sulphides.

Dave Nickerson (1979)

Ore reserves calculated in 1975 were as follows: #15 vein, 930 tons grading 1.43 ounces per ton gold over a 3-½ foot width; #22 vein, 156 tons grading 2.74 ounces per ton gold over a 2-½ foot width (National Mineral Inventory). During the May 1979, 12 tons of hand picked material from the #15 vein on the 'Rod' claims were sent to Con Mine for custom milling. Grade was reported to be 1.65 ounces per ton gold with a recovery of 94%, producing 19 ounces of gold (Cominco Ltd., 1979). Ore was derived from a small trench on the #15 vein. A four-wheel drive road was cleared north from Fred Henne Territorial Park and around Fox Lake to connect to the 'Rod' claims.

Equipment used during this program included an air-track drill, Cat 930 front-end loader, dumptruck, E-80 backhoe, and Cat bulldozer. Knud Rasmussen was contracted to perform the mining work (N.W.T Geoscience Office Assessment Report #082318).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Bittern Investments Limited (1994-1997)

Further ore was recovered on the old Rodstrom property in the 1990s by owner Dave Nickerson and his associated company, Bittern Investments Limited. In April 1994, material from #22 vein was mined and stockpiled. Some sampling was conducted but assay results were very erratic and it was decided to bulk sample the entire tonnage mined. A total of 2.20 tons of material was bagged in June 1996 in preparation for shipment. This included newly mined material from the #22 vein plus older spoils from trenching performed on the vein in 1963. 1.40 tons from the old trenches were bagged, plus 0.80 tons from the newly mined trench. (Nickerson, 1996)

Dave Nickerson arranged for the transport of this ore to Yellowknife in January 1997. Ed Eggenberger was hired to use snowmobiles and sleighs to haul the 2.2 tons to Robert Carroll's Kam Lake yard in Yellowknife, where Carroll utilized a small gasoline jaw crusher to provide first stages of crushing (-3/8 inch) on February 6th 1997. A 40 pound sample was obtained from the crushed ore and was further reduced at the Ptarmigan Mine assay lab to -3/16 inch size. The sample was sent south for assaying and the grade averaged about 0.30 ounces per ton gold. Expecting assays of 2 ounces per ton or higher, Dave Nickerson and Bittern Investments Limited decided to sell the remaining ore to Robert Carroll. The cost of the 1997 sampling program was \$600. A proceed of \$150 was made on the sale of the ore. (Nickerson, 1997)

Exploration Since Mine Closure

No known work since 1997.

References and Recommended Reading

Cominco Ltd., 1979. Con Mine production records, 1979.


Nickerson, D., 1996. Information Respecting Part of a Bulk Sampling Program - Rod Mineral Claims. Bittern Investments Ltd., June 1996. (*N.W.T Geoscience Office Assessment Report #083662*)

Nickerson, D., 1997. Results of Bulk Sampling Program - Rod Mineral Claims. Bittern Investments Ltd., March 1997. (*N.W.T Geoscience Office Assessment Report #083815*)

N.W.T Geoscience Office Assessment Report #082318

National Mineral Inventory (ROD Claims). NTS 85 J/8 Au 8.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085JSE0050

	RUTH <i>Minor Producer (Abandoned)</i>	Gold
		62° 27' 45" 112° 34' 30"

Years of Primary Development: 1941-1942

Mine Development: 230' inclined shaft, 2 levels (229' dev.)

Years of Production: 1942, 1959

Mine Production: 776 tons milled = 550 oz Au, 93 oz Ag

Introduction

The Ruth Mine is a former gold producer located 96 kilometres directly east of Yellowknife, NWT. This property is accessible by floatplane only, although a winter road and an airstrip were used in the past. The author visited the property in 2000, 2007, and 2008. The site is being targeted for demolition by the government.

Brief History

The 'Ruth' group of 14 claims were staked in August 1940 by John Michelson, a prospector for Cominco Limited. Following a brief period of trenching and diamond drilling in 20 short holes, a small shaft was started by hand steel under contract by Smokey Heal of Yellowknife. Underground operations continued with a full sized crew in 1942, and gold production began. The mill only operated for 12 days when World War II halted operations. Production resumed under new owners in 1959 for a short time, but to this date the Ruth Mine has been unable to be a steady gold mining operation.

Geology and Ore Deposits¹

Gold occurs in quartz veins hosted by turbiditic metasediments belonging to the Burwash Formation which is part of the Archean Yellowknife Supergroup. These mainly include medium-bedded greywackes rhythmically interbedded with thinner beds of argillite. In the vicinity of mineralization, bedding has been deformed into a tight, almost vertically plunging anticlinal fold with axial plane striking 010 degrees and dipping subvertically. A series of quartz feldspar porphyry dykes up to two meter wide cut the sediments about 50 meters east of the axial plane and strike sub-parallel to it. All stratified rocks in the area have been metamorphosed to greenschist facies.

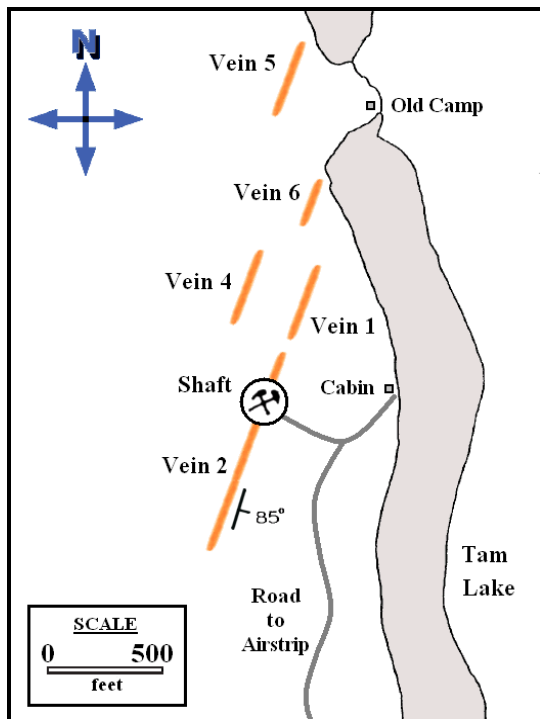


Figure 1. Ruth Mine area showing gold veins.

Three main auriferous quartz veins are present in the immediate vicinity of the old Ruth Mine. A fourth, the #3 vein, is situated about 1.6 kilometers to the north. The main deposit is hosted by the #2 vein, which lies within an argillaceous bed in the turbidites and is nearly parallel to it. It was exposed over a 380 meter strike length by trenching and has an average width of 40 centimeters. Quartz in the vein is generally fine-grained and locally includes thin partings of partially to totally biotitized argillite. Accessory minerals include small amounts of arsenopyrite, pyrite, pale brown scheelite, feldspar, native gold and an unidentified soft, grey, metallic mineral.

The shaft was sunk on an ore shoot reported to have graded 3.69 ounces per ton gold (uncut) over a strike length of 100 meters with average width of 18 centimeters. This portion of the vein graded about 0.10% WO₃. The #1 vein is located immediately east of the #2 vein and is more or less restricted to an argillite bed in the sediments. It was exposed by trenching over a strike length of 100 meters and has an average width of about 70 centimeters in that area. The vein consists mainly of quartz with locally abundant wall rock inclusions, together with small amounts of arsenopyrite, scheelite and native gold.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Arsenopyrite also occurs in the wall rock and in inclusions. The vein is reported to contain a shoot 15 meters long and 50 centimeters wide with an average grade of 0.50 ounces per ton gold. The #4 vein is located about 50 meters west of the #1 vein and was exposed over a strike length of 200 meters by trenching. It is irregular and discontinuous and was reported to contain some gold and scheelite. Numerous other narrow quartz veins, some of which contain native gold and coarse galena and sphalerite, occur to the south of the main veins. All veins generally strike northeast and dip 80° east.

Cominco Limited (1941-1942)

During 1941, a shaft (5 feet x 7 feet) was sunk to a depth of 55 feet by a small crew headed by Smokey Heal. This shaft was sunk at an incline of 80° to the east, parallel to the dip of the #2 vein, the most promising gold showing on the 'Ruth' claims. A camp was located on the north-end of Tam Lake. Work ceased for the winter period. Operations resumed early in 1942 and 12 men were at work getting the site ready for production operations. Construction on the camp was underway in April and shaft sinking using mechanized equipment was well underway. The 100-foot level was reached in the month, and the 200-foot level was reached in May. Stations were cut at both levels, but only the 1st level was developed, with two drives, north and south following the #2 vein.

The development of this gold property was questionable even when plans were laid for the installation of a mill. The war was creating supply problems. Even after ten men quit the job in June 1942, Cominco managers were determined to get the Ruth mine into production. Major equipment arrived via the winter road in April 1942. A dock and warehouse were built at Francois Bay on Great Slave Lake by Cominco specifically for the purpose of getting supplies and equipment to the Ruth operation. A large amount of the needed inventory did not make it to the mine before spring break-up, therefore production was delayed and float planes were required to fly in certain items.



Cominco Ltd.

Figure 2. Ruth Mine during construction, 1942.

Underground work on the #2 vein showed a promising gold deposit, with values becoming greater at larger depths from the surface. However, the size of the vein remained almost constant. The #2 vein was also found to be rich in scheelite content. It was predicted that a low-grade tungsten concentrate might be recovered with modifications to the future mill circuit. By July 1942, camp and plant construction and most essential equipment installations for the mill and power plant had been completed. The mill was only partly complete however and the cyanidation section was not ready. Despite this, it was decided to begin production. The date was August 1st 1942 (Lord, 1951).

Milling Plant

The mill at Ruth was designed to treat free-milling gold-bearing ore using an amalgamation and cyanidation flowsheet. Ore was hoisted from the shaft and fed into an inclined conveyor way-picking belt. In the gallery, scheelite ores were separated through the use of an ultra-violet lamp and stockpiled for future possible treatment. Gold-bearing ore was crushed in a 6 inch x 6 inch Fraser jaw crusher and sent to fine-ore bin, then into a 4 foot x 4 foot Colorado ball mill. After passing through a Denver jig, the pulp entered a Dorr-rake classifier (Edgar, 1958; site evidence).

Gold bearing overflow passed down onto a Diester vibrating-deck concentrating table where a gold concentrate was taken and sent to the amalgamation barrel. Rough gold was recovered in amalgam form and shipped to Yellowknife's Con Mine for refining. Had the mill been completed, tailings from the Diester table would have reported to a standard cyanide flowsheet consisting of thickeners, agitators, and a filter. When the mill closed on August 12th 1942, no progress had been made in completing this circuit. The mill had a capacity of 25 tons per day production and was reported to have made 90% recoveries, even without the cyanide circuit (Edgar, 1958; site evidence).

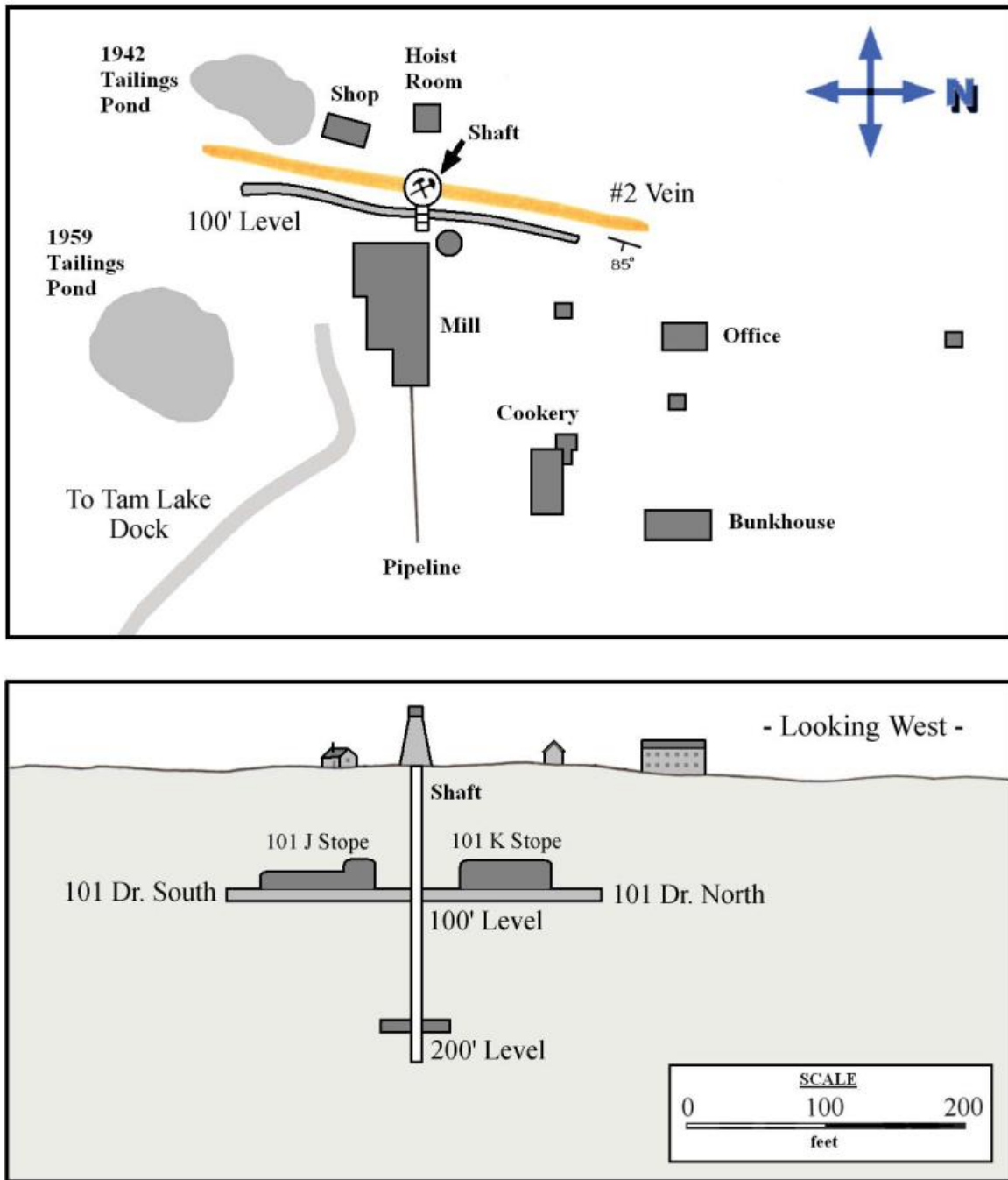


Figure 2. Ruth Mine surface and underground longitudinal plan, 1959.

Power Plant

The main mill equipment (ball mill and classifier) was directly driven by a Ruston-Hornsby diesel engine. This diesel engine was also connected to a Westinghouse generator that supplied about 20 kilowatts of power to the rest of the mill operation, along with the camp. A Cat diesel engine operated a Gardner-Denver air compressor (Edgar, 1958; site evidence).

Hoisting

The shaft was serviced with a large timber headframe and a small single-drum (14 inch x 16 inch) gasoline powered hoist, manufactured by Ratta Transmissions (Edgar, 1958; site evidence).

Other Facilities

The blacksmith shop equipment included a Canadian Ingersoll-Rand steel sharpener and a Gardner-Denver oil furnace. A two-storey bunkhouse provided accommodation for 50 men, the cookery was complete with a modern kitchen and cooler, and a three-room guesthouse was available for Cominco VIP's and other guests. There was also a two-storey warehouse, office, and staff house. Fresh water was pumped from Tam Lake and stored in a 15,000 gallon water tower. No central heating plant was installed at the mine. Buildings may have been heated by individual oil or wood burners (Edgar, 1958; site evidence).

Production and Development

Cominco removed 1,755 tons of development muck and 273 tons of vein material from underground development. This ore was reported to assay 0.36 ounces per ton gold. The mine was closed on August 12th 1942 due to conditions of the war. Of the 273 tons of vein material hoisted, 187 tons of ore were milled to produce 152 ounces of gold and 23 ounces of silver (see **Table 1**). Without the use of the cyanide circuit, it was feared that a large amount of gold values were being lost in the mill tailings. Unable to justify any further capital expenditures, Cominco mothballed the plant in preparation for a period of dormancy that was hoped to only last as long as World War II (Lord, 1951; Knutsen, 1973; Kelly, 1982).

The full extent of the Ruth deposit was untapped when the mine was closed. Development at this time consisted of a shaft to 230 feet with two levels opened up. Only the 100-foot level received development with 178 feet of drifting and 51 feet of crosscutting, plus 2,417 cubic feet of stoping excavation in two stopes. At closure, Cominco calculated an ore reserve of over 7,000 tons grading 0.73 ounces per ton gold across a width of 2.8 feet in the #2 vein (Knutsen, 1973).

Year:	Ore Milled:	Gold Produced:	Silver Produced:
1942	187 tons	152 oz	23 oz
1959	589 tons	398 oz	70 oz
<i>Total:</i>	<i>776 tons</i>	<i>550 oz</i>	<i>93 oz</i>

Table 1. Ruth Mine production, 1942 and 1959. (source: Kelly, 1982)

Ruth Gold Mines Limited (1959)

The Ruth Mine was reopened in 1959 when a private company in Vancouver, B.C. leased the claims from Cominco Ltd., and then subleased the claims to Ken Suitor who formed Ruth Gold Mines Limited. This company intended to mine the underground ore blocked out by the previous operators, and open up the 200-foot level for production (The Northern Miner, Aug. 13th 1959). Preparations for a resumption of operations included the clearing of an airstrip south of the mine, capable of handling Bristol freighters. A mine road connected the airstrip to the shaft area. The shaft was also de-iced and de-watered. During June and July of 1959, stockpiled ore from Cominco's operations was put through the mill. The company then mined the underground stope (101 K), removing 1,245 tons of material, including 84 tons of ore. Production to closure in August 1959 amounted to the treatment of 589 tons of ore to produce 398 ounces of gold and 70 ounces of silver. The company planned to reopen the following summer with the installation of machinery to increase mill capacity, but due to a lack of funding, the Ruth project was canceled (McGlynn, 1971; Knutsen, 1973; Kelly, 1982).

Exploration Since Mine Closure

Ice Station Resources Limited optioned the property from Cominco Limited in 1973 and conducted diamond drilling (17 holes; 3,930 feet). The program showed a modest tonnage of ore in place within the #2 vein and an ore zone that was open at depth. Reserves were calculated as 2,447 tons of ore grading 2.22 ounces per ton gold. In 1980, Roxwell Gold Mines Limited leased the claims from Cominco; this lease was soon abandoned, and a new deal was struck between Cominco and Hidden Lake Gold Mines Limited in 1982. In 1983-1984, the shaft was rehabilitated and some new equipment installed; the drifts and stopes were sampled and 12 holes were diamond drilled. Hidden Lake Gold Mines Limited acquired 100% interest in the claims in 1986 subject to a 2% net profit interest payment to Cominco (National Mineral Inventory). Surface sampling of vein exposures was conducted in 1988 and 1996 (Campbell et al., 2005).

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0020



SALMITA

Producer (Remediated)

Gold

64° 04' 45" 111° 14' 45"

Years of Primary Development: 1951-1952, 1975, 1982-1986

Mine Development: 145' vertical shaft; decline + winze to 1,750' depth, 11 levels

Years of Production: 1983-1987

Mine Production: 238,177 tons milled = 179,906 oz Au

Introduction

The old Salmita mine is located in the Courageous Lake region at the northeast end of Matthews Lake, 238 kilometers northeast of Yellowknife, NWT. It produced gold during 1983-1987. The Salmita and Tundra mine sites were visited in August 2001 and August 2006 by the author.

History in Brief

During the summer of 1945, prospectors Mike Mitto and Frank Salerno descended into the Matthews Lake region northeast of Yellowknife and staked 18 gold claims. Salmita Consolidated Mines Limited was formed almost immediately. Extensive exploration was carried out in the late 1940s and a shaft was sunk in 1951-1952. The mine remained dormant until 1975 when Giant Yellowknife Mines Limited took an option and drove a decline towards the old shaft. Production was achieved in 1983, with ore being milled at the nearby Tundra Mine plant. The orebody was mined out in 1987.



Salmita Consolidated Gold Mines Ltd

Figure 1. *Construction at Salmita Mine, 1951.*

Geology and Ore Deposits¹

The Salmita mine area lies within the Yellowknife Supergroup Archean rocks regionally associated with the contact zone between younger finely laminated argillaceous metasediments to the east and older mafic intermediate to felsic volcanics in the west. The upper felsic cycle in the Matthews Lake area is often characterized by medium to coarse pyroclastics and finely banded tuffaceous units. The contact zone between these volcanics and the underlying rhyodacite flows or overlying sediments exhibits moderate exhalite character and increased sulphide content. Gold occurs at or near this contact with hydrothermal quartz veins.

Four major showings occur on the property; the North and South showings, hosted mainly by felsic volcanics, and the sediment-hosted Olsen and Southwest showings. The Olsen zone, located about 500 meters southeast of the shaft site, hosts erratic gold mineralization across 91 meters. The Salmita North zone includes six veins, the B, T, A, C, E and F. The north striking B-vein, which dips 80 to 85° east, is divided into the South, Main and North shoots, approximately 160 meters long and from 1.5 to 2.5 meters wide, averaging 1.7 meters in width. The vein is primarily stratiform between argillite on the footwall and metabasalt on the hanging wall. The vein consists of massive, white, grey or black quartz with minor feldspar, tourmaline, sericite and epidote. The T-vein, roughly 100 meters southeast of the B-vein, is contiguous over a strike length of 150 meters and averages 10 centimeters in width. The vein

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

consists of lenticular to semi-continuous quartz bodies in a sheared or cleaved schistose felsic agglomerate. Mineralization consists of native gold with minor arsenopyrite and pyrite. The South zone is located about 3 kilometers southeast of the B-vein and about 175 meters east of Matthew Lake. The zone hosts a northwest striking, 80° northeast-dipping quartz vein exposed for 25 to 30 meters which is 30 centimeters to one meter wide. The vein is in fragmental volcanic rocks and slate near the contact with the sedimentary rocks to the east. The vein consists of bluish white to grey, sugary to coarse grained quartz containing visible gold, arsenopyrite and tourmaline.

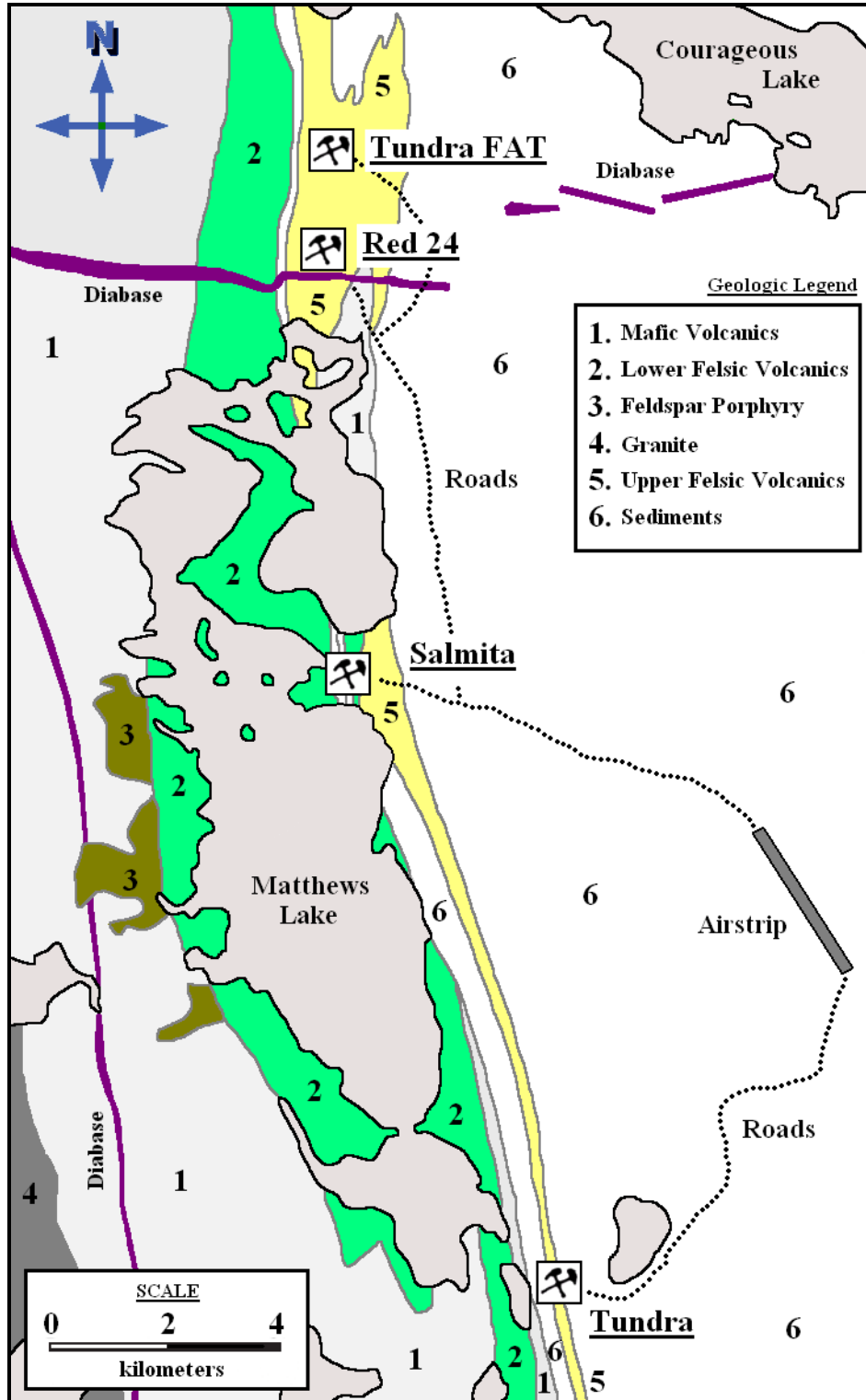


Figure 2. Matthews Lake area geology and location of Salmita Mine.

Salmita Consolidated Mines Limited (1951-1953)

Surface prospecting during 1945 uncovered three mineralized zones on the Salmita property – the North, South, and Eastern zones. In 1946, diamond drilling was initiated and surface work outlined three new veins – The B, T, and C-veins. 5,621 feet of diamond drilling (52 holes) was accomplished in 1946 on the South zone, the T-vein, and the B-vein, suggesting good gold grades. The B-vein showed the most promise as a high-grade gold deposit and also as the best underground exploration target. (Banfield, 1946) During 1947, a camp consisting of three Jamesway huts were erected and some mining equipment, including timbers for a headframe, were onsite (Lord, 1951). In 1947, 4 holes (1,273 feet) were drilled on the B-vein and 4 holes (856 feet) were drilled on the North zone. The North zone contained seven gold-bearing occurrences across 350 feet, including the promising B and T-veins (Burton, 1947). By 1950, plans were laid out for the underground program to explore the B-vein. Diamond drilling totaling 7,307 feet was performed on the North zone and the Olsen zone (discovered in September 1949) during 1950. This work confirmed the depth of the B-vein to 255 feet vertical and a strike length of 500 feet, and also suggested the possibilities for other ore bearing structures within the North zone (Singer, 1951a).

1951 Operations

Early in 1951, a 5,000 foot airstrip was cleared on the ice of Mathews Lake and 110 tons of freight was moved to the site by DC-3 aircraft in May 1951. A 24 man crew was assembled, the camp facilities were increased, plant equipment installed, and a 40 foot timber headframe erected. The shaft, sunk in two-compartments, was completed at a depth of 145 feet by the end of 1951. The 1st level was cut at a depth of 125 feet, where 55 feet of crosscutting was required to reach the B-vein. Drifting 61 feet in both directions from the crosscut was completed within the vein. The shaft was collared on the west side of the B-vein. Development was entirely within sedimentary rocks until crosscutting approached to within a few feet of the B-vein. The vein was found to be extremely high-grade on the north section of the drift, but only slightly above ore-grade in the south section (Singer, 1951b).

During shaft sinking, two ¾-ton ore buckets were used in the shaft. These were replaced with a cage and end-dump ore cars for lateral development operations. All mucking was done by hand. A very small mining plant was set up at the site in 1951. Power for Copco air drills was supplied by a 420 cubic feet per minute Gardner-Denver air compressor driven by a D-13,000 Caterpillar diesel engine. This compressor also supplied power for a Canadian Ingersoll-Rand 36 inch x 24 inch single-drum air hoist (Singer, 1951b). At completion of underground work September 15th 1951, an ultra-violet lamp was taken underground to explore the possibilities of tungsten occurrences at Salmita. Although some interesting occurrences of tungsten were discovered at this level, no further work would be done towards mining this mineral (Singer, 1951b).

1952 Operations

The camp was reopened in 1952 with plans to bring the gold property into production. A milling plant was purchased from Con Mine in Yellowknife, consisting of Traylor jaw and cone crusher, Hardinge ball mill, duplex classifier, blanket tables, and other equipment. Other equipment additions included a new two-drum electrified hoist, two 150 horsepower Paxman-Ricardo diesel generators, an 8,000 gallon oil tank, an International TD-14 tractor, and two Ford trucks. Construction during 1952 consisted of the erection of a large mill building (40 foot x 75 foot), a powerhouse (50 foot x 20 foot), Quonset-hut garage, an office/warehouse, and the clearing of an all-season airstrip 3 kilometers east of the mine. This airstrip was 3,300 feet long and capable of handling Bristol air freighters (Singer, 1952). Underground development during the year was minimal and included the excavation of a raise from the 125-foot level to the surface. This raise was designed to serve as a manway compartment, so that the two-compartment shaft could be put to full use as a production and development shaft (Singer, 1952).

1953 Operations

The year 1953 opened up with the laying out of further construction plans. These called for the erection of a large bunkhouse, and other service buildings. These were not built. Rather, time was spent installing and fine-tuning equipment in the mill and powerhouse. The headframe was upgraded and raised to a height of 75 feet. In September 1953, the company reported that all equipment was installed and that milling would be achieved in October (The Northern Miner, Sept. 10th 1953). A report by consulting engineer E.M. Dillman instructed the company to postpone plans for a mill, as he believed that the ore blocked out by current development was not sufficient to ensure continuous milling operations. In order to mill at a capacity of 150 tons per day, it was recommended that more development be completed to guarantee enough ore reserves. This included the deepening of the shaft to 250 feet and driving 1,000 feet of lateral development upon the B-vein on the 1st and 2nd levels (The Northern Miner, Oct. 22nd 1953). A lack of labour was one of the primary reasons for the temporary cessation of work. In 1954, the Salmita company decided to defer operations in light of less favorable conditions in the gold mining industry (The Northern Miner, Nov. 18th 1954).



Giant Yellowknife Mines Ltd.

Figure 3. *Salmita Mine with old headframe, c.1975. Decline portal in the foreground.*

Giant Yellowknife Mines Limited (1975-1976)

A new period of development began in 1975 when Giant Yellowknife Mines Limited optioned the Salmita property from Bluebell Enterprises Limited and collared a new underground opening on the B-vein showing area. Canadian Mine Services Limited was hired as a contractor for the underground work. A decline was driven a length of 1,050 feet to a vertical depth of approximately 150 feet. During 1975-1976, exploration was completed to formulate a reserve of 135,000 tons grading 0.63 ounces per ton gold to 600 feet depth; too low to support a production decision during these years. Ore reserves would have to be double that to break even at the current gold price. Total money spent on the Salmita project by Giant Yellowknife Mines Limited to the end of 1976 was CDN \$797,000. Facilities and equipment used during the 1975 underground development project included a trailer camp for 16 persons. Following the end of the underground program, all mining equipment and supplies other than the trailer camp were removed to Yellowknife (Giant Yellowknife Mines Ltd. Annual Reports, 1975-1976).

Giant Yellowknife Mines Limited (1981-1987)

Interest in the viability of the Salmita Mine was revived in 1980 with increased gold prices. The royalty interest held by Sam Walker, one of the original developers of the Salmita Mine, was purchased and Giant Yellowknife Mines Limited gained 100% control of the property from Bluebell Enterprises Limited (Giant Yellowknife Mines Ltd. Annual Report, 1980). At this time, it was proposed to sink a new shaft to a depth of 1,000 feet and install a milling plant of 150-tpd or more. The concept of the Salmita project changed the following year when it was feared that the high price of gold would not last. Further delineation of gold at greater depths was necessary before such a production decision could be made (Giant Yellowknife Mines Ltd., 1982).

Therefore, the mining plan was changed so that the decline would be extended beyond the depth achieved in 1975 instead of sinking a production shaft. Consideration was given towards leasing the old nearby Tundra Mine facilities including the mill instead of construction all new facilities at Salmita, but more development was required before a production decision could be made. The old Tundra-Salmita airstrip was extended to 4,500 feet and equipment was flown to the site. A fleet of construction equipment was brought over the winter road and included a D76 Cat bulldozer, a 977L Cat front-end loader, and two Mack haul trucks. The mine camp was completed in 1981 and consisted of 14 trailer units containing 40 rooms, mine dry, offices, cookery, and warehouse (Giant Yellowknife Mines Ltd., 1982).

Mine development recommenced in November 1981 through de-icing and de-watering of the 1975 decline. Until year-end, all mining work involved de-icing 386 feet of decline and 56 feet of the old shaft (Giant Yellowknife Mines Ltd. Annual Report, 1981). This process was slow and expensive, and as a result of cost overruns exceeding the

budget, the extent of underground development was reduced by the deletion of one of the three proposed levels and by blasting a new decline off the 1975 ramp instead of continuing through the frozen tunnel. The decline was driven 10 feet x 12 feet dimension at a grade of -13% in a circular design, designed as a safety guard in the event of runaway vehicles, and also to provide sub-level drifting if necessary. The underground development program concluded June 11th 1982 with three levels established at 165-, 260-, and 310-foot depths. Work during the 1981-1982 program included 2,430 feet of decline, 412 feet of lateral work on the 165-foot (1st) level, 82 feet on the 260-foot (2nd) level, 1,200 feet on the 310-foot (3rd) level, 300 feet of other lateral work, and 317 feet of raising (Giant Yellowknife Mines Ltd., 1982).

Development Operations 1982

Heating and ventilation was supplied through a 40,000 cubic feet per minute propane fired fan located at the top of the old shaft, downcast through raises and up through the decline portal. The old headframe was demolished in August 1981 and the shaft was also used as an emergency escapeway. Compressor air was supplied by two 1200 cubic feet per minute diesel driven air compressor units. Drilling through the permafrost was aided by the addition of calcium chloride to the water used in the equipment. Equipment used included two Ingersoll-Rand 1200 cubic feet per minute air compressors, two 200 KVA gensets, and a 977L front-end loader. Mining equipment was contractor owned and information is not available (Giant Yellowknife Mines Ltd., 1982).

The operation was placed on hold in July 1982 pending acquisition of all necessary government licenses and permits. The camp was put on care and maintenance and a small crew was retained to keep the pumps working underground. Preparations were made for future construction in the event of a production decision (Giant Yellowknife Mines Ltd. Annual Report, 1982; Giant Yellowknife Mines Ltd., 1982). Ore reserves calculated in March 1983 were 154,000 tons grading 0.82 ounces per ton with 30% dilution. Underground work to date exposed the B-vein with an average width of 5.5 feet. The vein was divided into three steeply plunging ore shoots with a total strike length of 530 feet, extending to a depth of at least 650 feet below surface. 18 diamond drill holes totaling 6,850 feet were drilled below the 310-foot level to test the vein for downward extensions to the 650-foot level (Giant Yellowknife Mines Ltd. Annual Report, 1982).

Production Decision is Made

In March 1983, it was announced by company officials that regulatory agencies had approved of the project and that the Salmita Mine would be placed into production during the summer of 1983. Underground work resumed in April 1983 under a new contract with J.S. Redpath Limited. Essential to the decision for production was the acquisition of the old Tundra Mine facilities, located ten kilometers south of Salmita. A deal was signed early in 1983 with Tundra Gold Mines Limited for the purchase of all buildings and plant facilities located at Tundra Mine, plus the mineral rights to the property. The Tundra milling plant, power plant, and accommodation buildings were rehabilitated during the spring of 1983. A 50-man camp was established. Ore would be trucked from Salmita decline workings to the Tundra Mill over the existing roads (Giant Yellowknife Mines Ltd. Annual Report, 1983).

Start of Production

Mining operations got underway on July 14th 1983 with a small crew by opening stopes on the 1st and 2nd levels. The mill was commissioned on August 4th and the first gold was recovered on September 19th 1983. Although startup problems were encountered, tonnage throughput was stabilized and bullion production was increased to desired levels by year-end. Most of the ore treated in 1983 came from lower-grade development headings. Total cost to bring the mine into production was \$11.1 million, under budget by nearly \$3 million. The decline was advanced from the 260-foot (2nd) level to the 4th level and by year-end 1983, mining was being conducted on three levels (Giant Yellowknife Mines Ltd. Annual Report, 1983).

Employees 1983

Employees at the start of mining operations at Salmita included W.H. Muise, project manager; Bill Muir, mill superintendent; Nick Majacich, mine superintendent; and Tony Ransom, exploration manager. Numerous contractors were employed during 1983 on building rehabilitation at Tundra, underground development, and general construction activities. Average workforce during these years was 90 employees split between mining and milling operations. Rotation for the mining department was 6-week in, and 2-weeks out, on 10-hour shifts. The milling department worked on a 2-week rotation schedule with 12-hour shifts (Davies, 1984). Two camps were operational, one at the Salmita mine site and one at the Tundra milling site. The Salmita camp was a large trailer camp with a small cafeteria, bunkhouses, recreation hall, and first aid station. The Tundra camp utilized the old bunkhouse, cookery, and recreation center from the old Tundra Mine.

Mining Operations

The mining method used at Salmita Mine was shrinkage stoping using mechanized equipment. Geological conditions including ore widths of five feet, steep dip, high-grade ore, and competent ground conditions made this method most desired, and it resulted in good productivity and small dilution (30%). Extraction drifts and drawpoints were driven on the hanging wall side of the deposit on each level, and ore was extracted using 2-½ yard Jarvis-Clark and Wagner scooptrams. Ore was hauled to surface via the decline ramp in 9-ton low-profile JDT trucks and then dumped into a stockpile for transportation to the milling plant at Tundra Mine (Davies, 1984). Below the 165-foot (1st) level, joints and fractures within the bedrock allowed water into the workings, but it was not excessive and only 250 gallons per minute were pumped from the workings. Design of the spiral decline was largely an attempt to avoid the major water-bearing conduits. In 1985, ventilation flow was about 80,000 cubic feet per minute downcast through vent raises and exhausted up the decline (Lebel, 1984).

Milling Operations

Ore from the mine was trucked to the mill site a distance of 10 kilometers on all-weather road or 6 kilometers on ice road, and stockpiled. Although much of the old Tundra milling plant was still intact, the original crushers had been removed and new units were required. Primary crushing was accomplished with a 24 inch x 36 inch Pioneer jaw crusher, reducing the ore to -6 inch. Product was screened and oversize material sent for secondary crushing in a Universal 18 inch x 32 inch jaw crusher, where ore was reduced to -1½ inch. A cone crusher of 3 foot size further reduced the ore where the final -3/8 inch product was conveyed to the 200 ton fine ore bin. The grinding circuit until June 1984 consisted of two ball mills, a 5 foot x 8 foot Denver and a 5 foot x 8 foot Marcy ball mill. These two units proved to bottleneck the operation and in 1984 an 8 foot x 8 foot Allis-Chalmers ball mill was installed, replacing the older units for primary grinding. Cyclone classification was used with oversize being returned to the older ball mills for re-grinding, and fine material being pumped to the cyanidation circuit. This circuit consisted of a 28 foot de-watering thickener, three agitator tanks, and Oliver filters. The liquid mill solution was clarified, precipitated, pressed, and refined into gold bars. (Giant Yellowknife Mines Ltd., 1983).



Giant Yellowknife Mines Ltd.

Figure 4. *Salmita Mine complex looking east, 1986.*

Power Plant

At the Tundra milling plant, the old Cooper-Bessemer diesel generator of 750 kilowatt was repaired and re-commissioned as the primary power unit for the operation. Most of the power output is used in the milling plant, but any excess power was transferred to the Salmita mine site via a powerline. Smaller diesel generator sets used at the Tundra plant could be powered up to augment power requirements, although normal site requirements required only the Cooper-Bessemer unit. In 1984, two 400 kilowatts Cummins diesel generators were installed in the Tundra powerhouse to take the load off the Cooper-Bessemer. One of the units remained on standby at all times. The Foster-Wheeler boiler was used to heat the shop/office complex, mill, and the bunkhouse at Tundra. At the Salmita site, backup power was supplied by two additional 400 kilowatts Cummins diesel generators and Ingersoll-Rand 1200 cubic feet per minute portable air compressors (mine records).

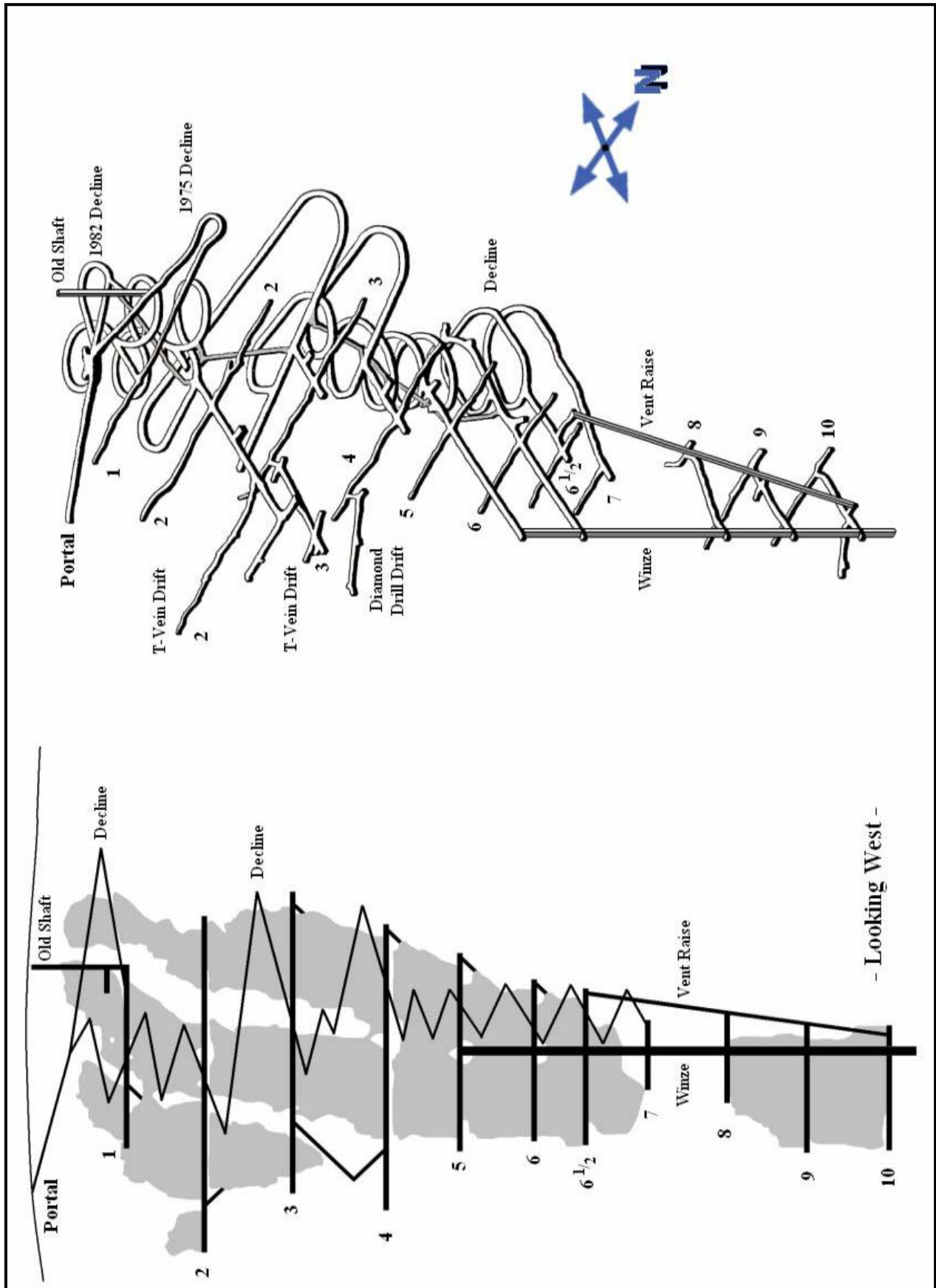


Figure 5. Salmita Mine longitudinal plan and 3-dimensional schematic of B-vein development, 1986.

1984 Operations

During the year, it became apparent that the deposit was pinching at depth. While the first three levels of the mine all had three stopes, the deeper levels were only accommodating two stopes on each. Production grades were an improvement during 1984 as higher-grade areas were encountered at depth. Grade increased from 0.63 ounces per ton early in the year to 0.93 ounces per ton by year-end. A development program below the 4th level was initiated in August 1984 following interesting diamond drill intersections, and by the end of the year the decline had nearly reached the 6th level. Lateral development and extraction drifts had been established on the 5th level (Giant Yellowknife Mines Ltd. Annual Report, 1984).

Early in the year, production was adversely affected by cold conditions that froze and hardened the ore stockpiles, restricting mill throughput. A screen was installed in the ore bin to separate the 'fines' that were causing some of the freezing problems. A larger ball mill, an 8 foot x 8 foot Allis-Chalmers unit, was also installed to enable production to reach targeted levels. The new ball mill replaced the older two Denver and Marcy units in the Tundra mill. The modified milling plant was operational by June 1984 and no further problems were encountered. Lost production was made up for during the second half of the year (Giant Yellowknife Mines Ltd. Annual Report, 1984).

T-Vein Development

Underground exploration was performed on the T-vein during 1984, a parallel ore structure located 200 feet east of the B-vein. Drilling and drift development was disappointing due to the erratic nature of the deposit, but a test stope was started on the 2nd level. (Giant Yellowknife Mines Ltd. Annual Report, 1984)

1985 Operations

At year start, mining operations were focused on the 4th and 5th levels. The decline ramp was completed to the 6th level in February and stoping got underway. New diamond drilling below the 6th level yielded additional tonnage and 23,000 tonnes grading 60 grams per tonne were outlined. The decline was continued to the 7th level by December 1985, and a sub-level, known as the 6-½ level, was developed into the ore block. Meanwhile, approval was given to sinking of a winze to explore the deposit at greater depths because continued decline ramping below the 7th level would not be economic. A sinking contract was awarded to R.M. Drilling Limited and work began in September 1985 to sink the shaft from the 5th level to a planned vertical depth of 1,700 feet below the surface (Giant Yellowknife Mines Ltd. Annual Report, 1985).

The two-stages of further development called for a reassessment of the power requirements at the mine. Three 400 kilowatts diesel generators (Cummins) were installed at the Salmita site in 1984 to ease the demand on the Cooper-Bessemer unit at the Tundra Mine mill plant. Worn out mining equipment was also a concern as maintenance costs were very high in 1985. Plans were made for purchase of new equipment in 1986 (mine records).

Hoist

The hoist used for sinking operations and later production in the Salmita winze was a Canadian Ingersoll-Rand 48 inch x 36 inch two-drum electric hoist (mine records).

Employees

During 1986, the following management personnel were involved in the Salmita Mine: Ken Blower, general manager for Giant Yellowknife Mines Limited; William Muir, mine manager; Tony Ransom, exploration manager; Nick Majacich, mine superintendent; Malcolm Robb, geologist; Ray Gagnon, mill superintendent; and J. Walsh, accountant. There were 90 company employees and an unknown amount of contractors from R.M. Drilling, the winze sinking contractor. Turnover was approximately 40% (mine records).

1986 Operations

Supplying ore to the mill was a principle concern during 1986, and several projects were initiated to extend the operational life of the Salmita Mine. Development of the Red 24 deposit (see Red 24 Mine) was initiated but it only supplied a small amount of ore. Sinking of the winze continued below the 6th level with the 8th, 9th, and 10th levels being reached in February, March, and April 1986, respectively. Sinking of the winze to a vertical depth of 1,750 feet was completed on May 18th 1986, and the changeover from sinking to a production winze was made. Mining operations in 1986 were affected by increased ore dilution, caused when concussion blasting to bring down ore pillars broke off waste rock from stope walls. Lower grades were also encountered at the deeper levels, but a higher than anticipated amount of ore was mined. This raised the costs of operation significantly during 1986. The new levels (8-10) were developed and largely completed by year-end 1986 (Giant Yellowknife Mines Ltd. Annual Report, 1986). New mining equipment bought during the year included two JDT 413 haul trucks, one Wagner ST2B scooptram, one

2-boom MJM jumbo drill, and one 850 cubic feet per minute air compressor. These units were largely introduced because of the Red 24 project, although since the project was not a year-round affair the equipment was also used at Salmita operations (mine records).

Exploration during 1986 was extensive in the B-vein. Ore grade mineralization was defined on the 4th level and the 7th level. Nothing significant was found in lateral exploration north of south of B-vein. T-vein exploration was disappointing. Plans were made to drill below the 10th level to a depth of 3,000 feet, or 1,300 feet below the bottom of the mine. Three holes drilled 660 feet below the 10th level gave interesting gold values, but it was believed that insufficient gold would be encountered to permit deepening of the winze. Surface exploration totaled 40,400 feet of diamond drilling, concentrated on the sedimentary/volcanic contact between the Salmita and Tundra Mines (Giant Yellowknife Mines Ltd. Annual Report, 1986).

Final Months of Operation

By year-end 1986 it was clear that the deposit was pinching out and becoming uneconomical at depth. Practically all mining development and stoping had ceased and ore remaining as broken material in stopes or in the surface stockpile was enough to keep the mine operational until April 1987. Production ceased and operations switched to preparing for summer surface exploration and first stages of remediation. Exploration during the summer of 1987 failed to locate any favorable showings in the mine area, and by 1990 the Salmita mine site had been remediated. The Tundra milling site was left intact (Giant Yellowknife Mines Ltd. Annual Report, 1987). Total Salmita production is listed in **Table 1**. See **Figure 5** for a longitudinal plan and three-dimensional schematic of the underground workings.

Year:	Ore Milled:	Grade:	Gold Produced:	Year-End Ore Reserves:
1983	13,146 tons	0.47 oz/ton	5,116 oz	116,000 tons @ 0.82 oz/ton
1984	60,702 tons	0.76 oz/ton	44,414 oz	88,000 tons @ 0.85 oz/ton
1985	71,145 tons	0.91 oz/ton	63,697 oz	64,000 tons @ 0.97 oz/ton
1986 *	69,865 tons	0.73 oz/ton	49,144 oz	12,000 tons @ 0.75 oz/ton
1987 *	23,319 tons	0.77 oz/ton	17,535 oz	-
<i>Total:</i>	<i>238,177 tons</i>	<i>0.73 oz/ton</i>	<i>179,906 oz</i>	-

Table 1. Salmita Mine production, 1983-1987. (source: Giant Yellowknife Mines Ltd. Annual Reports)

* 1986 and 1987 production probably includes Red 24 Mine ore.

Exploration Since Mine Closure

New claims were staked in the area by Seabridge Gold Incorporated and exploration of the area is ongoing as of 2008.

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SMALLWOOD LAKE

Satellite Producer (Abandoned)

Silver

65° 35' 00" 117° 56' 45"

Years of Primary Development: 1979-1983

Mine Development: 3,400' decline, 4 levels

Years of Production: 1979-1980, 1982-1983

Mine Production: 18,204 tons milled = 169,720 oz Ag

Introduction

The Smallwood Lake Mine is located along the Camsell River in the Great Bear Lake region, 56 kilometers south of LaBine Point (Port Radium). It is 390 kilometers northwest of Yellowknife, NWT and operated as a satellite silver operation of Terra Mine between 1979-1983. The site has not been visited by the author.

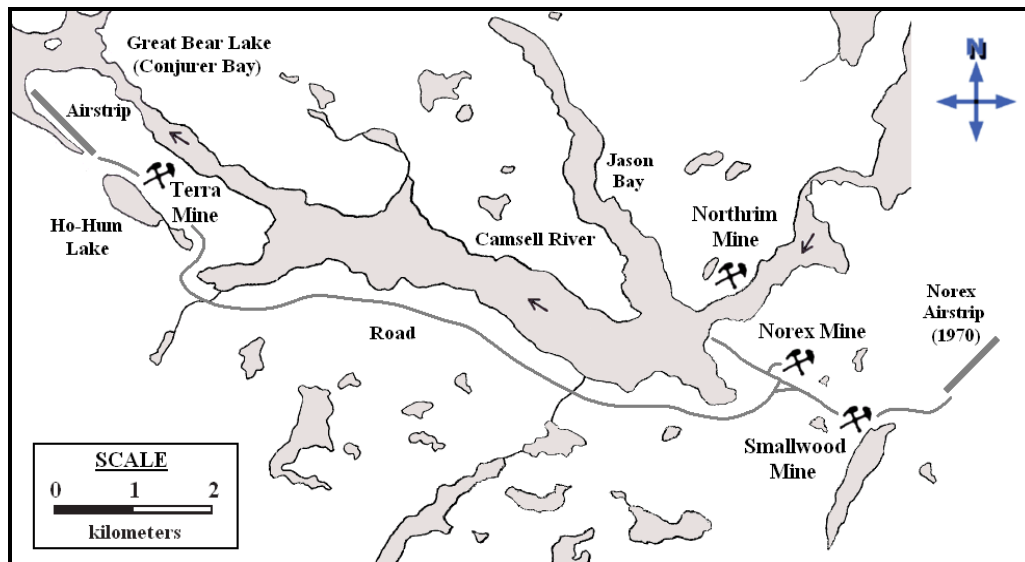


Figure 1. *Smallwood Lake Mine location.*

Brief History

The first silver discoveries on the Camsell River occurred during 1932, but little work was recorded done on this property until 1968 when Caesar Silver Mines Limited performed extensive surface exploration of the area. The claims were optioned to Norex Resources Limited in 1970. Terra Mines Limited entered a joint-venture agreement with Norex for the production of Norex Ores in 1973. Terra continued surface exploration of the immediate area surrounding Norex Mine and in 1978 they intersected good silver values in the vicinity of Smallwood Lake, on the southeast end of the Norex property. In 1979, Terra began underground development through a decline ramp and began trucking ore to the Terra Mine mill. Full production began in 1982 but the life of the mine was short and operations ceased at the end of 1983.

Geology and Ore Deposits¹

The showing is underlain by Aphebian-age andesitic lavas and pyroclastics of the Echo Bay Group, commonly impregnated with metamorphogenic sulphides. Rocks consist of acidic and basic volcanic flows with associated tuffs and agglomerates. The volcanics are siliceous, fine grained and light grey in colour. Diabase dykes appear to be associated with faults and strike east, northeast, and northwest. The major northeast striking Smallwood fault is thought to have provided the fracture pattern which accommodates local silver-bearing veins. The Smallwood Lake Mine has three known veins, one of which has been mined. The #4 vein is unique for the area because it has greater widths and strike lengths and because it occurs within a fracture zone up to 15 feet wide. Veins are associated with areas of sulphide enrichment, occurring in large gossanized patches roughly parallel to the contact with the Balachey Lake igneous intrusive. Additional lead-zinc sulphides occur as halos around silver veins, and at least one stope was mined for its lead content.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

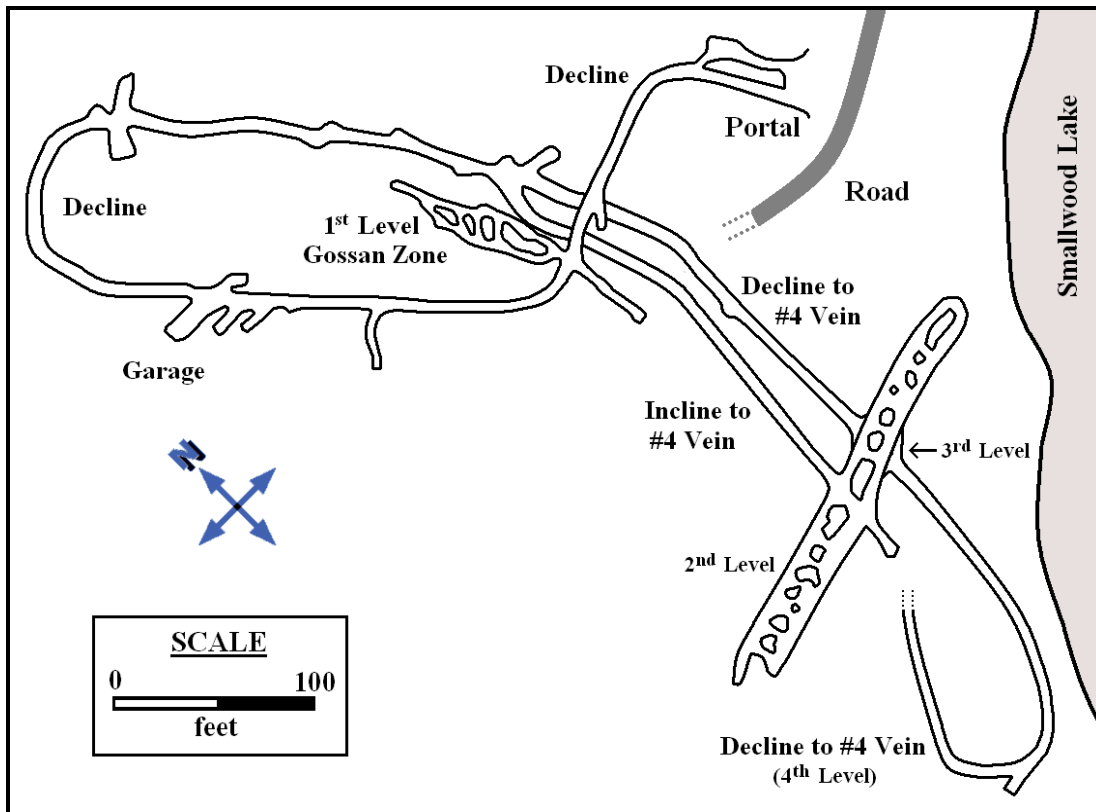


Figure 2. Smallwood Lake underground plan, 1982.

Terra Mining & Exploration Limited (1979-1982)

Original interest at Smallwood Lake was upon a gossan zone with high silver assays. A haul road connecting the Smallwood property with the Terra Mine was completed during 1979. Between July and December 1979, a decline haulage-way was driven 881 feet. A number of silver bearing structures in sulphide-rich rock were opened up within the gossan zone through 792 feet of drifting, crosscutting and raising. Test stoping of the vein in this section began early in 1980. Small shipments of development ore to the Terra Mine milling plant took place in 1979-1980. Work within the gossan zone during 1979-1980 failed to locate significant tonnages of high-grade ore (Terra Mining & Exploration Ltd. Annual Reports, 1979-1980). Surface and underground diamond drilling was conducted during 1980-1981 to delineate the depth potential of the vein systems. The results of this work were very encouraging, identifying possible new vein systems: the #2, #2A, #2B, #3, and #4 systems to depths beyond 450 feet. In 1981, it was decided to extend the decline to follow the #4 vein at depth (Terra Mining & Exploration Ltd. Annual Reports, 1980-1981).

Terra Mines Limited (1982-1983)

Terra Mining & Exploration Limited merged with Duke Mining Limited in 1982 and was renamed Terra Mines Limited. The Norex decline was advanced 1,600 feet during 1982 to open the #4 vein on the 3rd level. The 2nd level at 200 feet depth, also within the #4 vein, was accessed from a short incline. The #4 vein was reported to be very wide and with a great strike length, but of lower grade (17 ounces per ton silver). It was envisioned as a bulk tonnage deposit with the hope that grade would improve with depth. The decline was further extended during 1982-1983 to the 4th level where diamond drilling encountered intersections of 110 ounces per ton silver over 2-8 feet widths. A test stope within the #4 vein was begun on the 4th level during 1982. A stope containing grades of lead and zinc on the 3rd level was also mined during the year. Ore reserves at November 1982 totaled 24,292 tons grading 28.3 ounces per ton silver. The Smallwood Mine was placed back into normal production in October 1982 with the #4 vein as the source of mill feed (Terra Mines Ltd. Annual Report, 1982).

Mining Operations

Ore was mined through shrinkage stoping methods and hauled from the underground via scooptram. Haul trucks were used to transport the ore to the Terra Mine milling plant, 10 kilometers distance by road.

Power Plant

Many different power units were made available at the Smallwood Lake site as Terra Mines Limited had a fleet of portable units that were rotated between Terra, Norex, and Smallwood Lake Mines. In 1979, a power plant of two Atlas-Copco portable air compressors and a Deutz generator was in use. In 1982, a Caterpillar D-379 diesel generator installed nearby at Norex was connected up to power the Smallwood Lake site as well. Two portable Gardner-Denver 1,200 cubic feet per minute air compressors were originally in use, but these were replaced with new stationary 750 cubic feet per minute compressors in 1983. All equipment was housed in the decline portal (mine records).

Year:	Ore Milled:	Silver Produced:
<i>1979</i>	443 tons	3,982 oz
<i>1980</i>	2,307 tons	12,732 oz
<i>1981</i>	-	-
<i>1982</i>	5,552 tons	87,716 oz
<i>1983</i>	9,902 tons	65,290 oz
<i>Total:</i>	<u>18,204 tons</u>	<u>169,720 oz</u>

Table 1. *Smallwood Mine production, 1979-1983. (source: mine records)*

Development of the #4 vein was discontinued in April 1983 upon the reconciliation of stoping grades and the testing of the vein on the 4th level. No commercial grades could be encountered in the #4 vein. Production from Smallwood Lake during 1983 was derived from the 269, 204, and 304 stopes (2nd and 3rd levels) where some medium-grade ore of the #4 vein was available. As a result of falling silver prices and the poor results on the #4 vein, Terra Mines Limited put all development and production at Smallwood Lake Mine on hold late in 1983. The underground workings were allowed to flood in 1984 and no work has been done since (McCormack, 1983; Terra Mines Ltd. Annual Reports, 1983-1984). Ore reserves, reported in August 1985, were 1,725 tons of ore grading 15 ounces per ton silver (containing 26,500 ounces of silver), but the methods and details of this resource calculation are unknown (Nicholas, 1985).

Exploration Since Mine Closure

Terra Mines Limited sold the mine to Octan Resources Incorporated in 1988. The mineral claims are now (2008) owned by Cooper Minerals Inc.

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SNAP LAKE

Producer (Operational)

Diamond

63° 36' 00" 110° 52' 00"

Years of Primary Development: 2000, 2004-current

Mine Development: underground ramp

Years of Bulk Sample: 1998-1999, 2000-2001, 2004-2005

Bulk Sample: 34,181 tonnes milled = total carats unknown

Years of Production: 2007-current

Production: see **Table 2**

Introduction

The Snap Lake diamond mine is located 218 kilometers northeast of Yellowknife, NWT. It commenced production in August 2007 and is the first fully underground diamond mine in Canada. It is owned by De Beers Canada Incorporated.

Brief History

This large property was staked in 1991 and explored during 1992-1993 by Amarado Resources Limited. Aber Resources Limited acquired its interest in 1993 and continued with exploration. Initial results were disappointing. Exploration shifted to other areas of the claims and in 1996 indicators minerals and kimberlite boulders were located at Snap Lake. Winspear Resources Limited joined forces with Aber through a joint-venture agreement. Much work was done in the following years, so that by 1998 a dyke had been outlined that contained what appeared to be a sizable kimberlite orebody. Bulk sampling of the surface exposures began in 1998 and continued in 1999. Underground development started in 2000 but was halted in 2001 when the operation was acquired by De Beers Canada Corporation Limited. Approval for the mine was granted by De Beers in May 2005 and construction proceeded during the following two years, with production achieved in August 2007. The mine ramped up to full production with some difficulties during the winter of 2007-2008. Official grand opening was held July 25, 2008.

Geology and Ore Deposits¹

The Snap Lake Dyke showing occurs within the Archean Slave Supracrustal Province. The Supracrustal rocks are of the Yellowknife Supergroup, dominated by a volcanic-greywacke-mudstone sequence forming various greenstone belts that cover about one third of the Slave province. Granitoid intrusions make up the bulk of the craton and are grouped as pre-, syn-, and post- Yellowknife Supergroup. In addition, several major Proterozoic events of diabase and mafic dyke emplacement have occurred, orientated between 140 and 180°. Also present in the regional area are diamondiferous kimberlite pipes occurring roughly along and proximal to the dyke swarms. The kimberlite dyke is gently dipping towards the east. The average true thickness of the dyke is 2.5 metres.

Winspear Resources Limited (1998-2000)

In 1998, worked shifted from exploratory and sampling drilling to small scale mining operations focused on the selection of larger bulk samples.

The 1998 Program

During early 1998, Windspear mined 200 tonnes of ore from the #1 and #2 pits of the Northwest dyke and processed it to produce 229 carats of diamonds. Average grade was 1.14 carats per tonne. The pits were located about 775 feet apart and both located along the strike of the Northwest Dyke. Total cost of the 1998 program for Winspear Resources Limited was \$6 million. In December 1998, the joint venture partners (Windspear and Aber) agreed to spend CDN \$12.4 million in 1999 to conduct a larger bulk sample operation (Winspear Resources Ltd. Annual Report, 1998).

The 1999 Program

The 1999 program was designed to collect data on the Northwest dyke and to follow-up on the preliminary diamond values obtained from the 1998 mini-bulk sample. Mining of the #3 and #4 pits of the Northwest dyke (850 feet apart) began early in 1999, and ore was trucked via winter road to a bulk sampling plant owned by Tahera Corporation Limited at Lupin Mine, Nunavut starting in May 1999. The plant was rated at 10 tonnes per hour. Cost of renting this

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

facility was \$321,000. Total cost of the 1999 program for Winspear Resources Limited was \$18 million (Winspear Resources Ltd. Annual Report, 1999).

Bulk Sampling

5,985 tonnes of ore was processed in total during the 1999 program, with a recovery of 10,708 carats of diamonds grading 1.79 carats per tonne. Exploration work between 1997 and 1999 suggested a reserve of 12,592,100 tonnes of ore grading 1.75 carats per tonne. The mine plan called for mining to a depth of over 1,000 feet below surface through underground workings and also mining the top portion of the dyke by open pit (Winspear Resources Ltd. Annual Reports, 1998-1999).

The 2000 Program

Work in the year 2000 focused primarily on underground development, with the objective of tapping into the Northwest dyke and extracting a larger bulk sample. Plans called for three 2,000 tonne samples extracted from different parts of the orebody. The company planned a 4,000 foot underground ramp to intersect the kimberlite dyke and extract the bulk samples (Winspear Resources Ltd. Press Release, Feb. 14th 2000).

Year:	Ore Milled:	Grade:	Diamonds:
1998	200 tonnes	1.14 carats/tonne	229 carats
1999	5,985 tonnes	1.79 carats/tonne	10,708 carats
2000	4,607 tonnes	?	?
2001	7,511 tonnes	?	?
2004	8,802 tonnes	?	?
2005	7,076 tonnes	?	?

Table 1. Snap Lake bulk sampling. (source: Winspear Resources Ltd. annual Reports; Mackenzie Valley Land and Water Board Files – NIL3-1735, MV2001L2-0002)

In February 2000, work on the decline ramp began. Tahera's bulk sampling plant of 10 tonne per hour capacity was bought for \$2.3 million and trucked to site over the winter road. A 100 foot x 3,000 foot airstrip was cleared in April to accommodate year round freight transportation. An 84 man camp was built. About 175 men were employed during spring 2000 with underground operations and construction (Winspear Resources Ltd. Press Release, Apr. 3rd 2000).

Diamond Reserve

Winspear estimated that the Snap Lake deposit contained 39.5 million tonnes of mineable ore with a carat resource estimated at 67 million. If a production decision was made, it was estimated that revenue over a 21 year mine would be CDN \$11.5 billion (Winspear Resources Ltd. Press Release, July 11th 2000). By the end of August 2000 the decline had reached the kimberlite deposit at a depth of 80 meters after over 1,000 meters ramp advance, and mining was underway (Winspear Resources Ltd. Press Release, August 10th 2000).

A 20,000 tonne bulk sample was mined and stockpiled for processing. Three batches of 2,000 tonnes were to be processed initially. On August 14th 2000 it was reported that bulk sampling began (News/North, Aug. 16th 2000). Processing of these ores continued to March 2001 when the plant was temporarily shutdown. Processing resumed from June to September 2001. Although diamond recoveries and grades have not been published, it is known that 12,118 tonnes of kimberlite material were processed during 2000-2001. John Goyman was mine manager in 2000-2001 (Mackenzie Land and Valley Water Board - Water License NIL3-1735). In August 2000, De Beers Canada Incorporated made an offer to acquire Winspear's share interest in the property. In November the deal went through and controlling interest in the Snap Lake Mine was acquired by De Beers (News/North, Nov. 29th 2000). Shortly thereafter, in early 2001, Aber Diamond Corporation Limited (previously Aber Resources Limited), seeking additional funds to bring the Diavik Mine into production, sold the remaining interest to De Beers (News/North, Feb. 5th 2001). Initially the company made optimistic plans for a 2004 production date, but it was soon realized that the regulatory process involved in getting permission to build the mine would delay those aspirations. De Beers put the site on care and maintenance in September 2001 (News/North, Aug. 29th 2001).

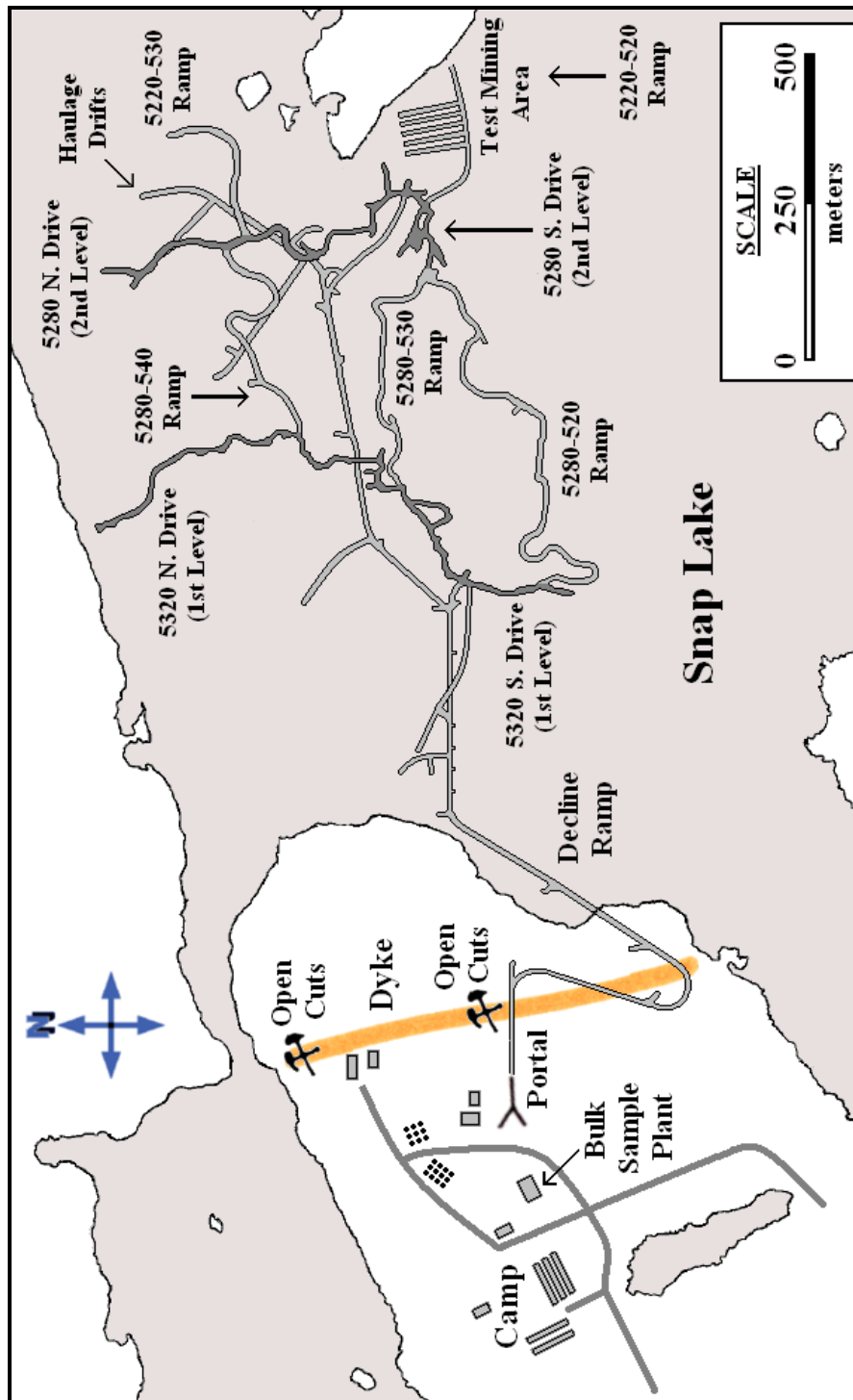


Figure 1. Snap Lake Mine underground plan, as of December 31st 2005.

De Beers Canada Incorporated (2004-current)

On May 31st 2004, De Beers received final regulatory approval to place the Snap Lake Mine into production. Transportation of supplies and freight to the mine was conducted during the 2004 winter road season, during which period 249 truckloads of material, including new fuel tanks, equipment for a water treatment plant, additional machinery for the bulk sampling plant, and 3.8 million litres of fuel, were brought to the property. Pre-development operations began through the hiring of mine staff, de-watering of the underground ramp and workings, construction of a water treatment plant, equipment acquisition, and drafting of final engineering plans for mine operation.

Four additional fuel tanks, each of 333,000 litre capacity, have increased total capacity at Snap Lake Mine to 4.2 million litres. The trailer camp was expanded to allow accommodations for 111 persons and new on-site offices were set-up for company crews previously based in Yellowknife, N.W.T. Chantal Lavoie was appointed mine manager. About 110 people were employed during the summer of 2004. Starting in March 2004, modification to the bulk sampling plant was started. This included the addition of an optical waste rock sorter, a high-pressure roll crusher, and a new x-ray sorter. These changes are designed to improve the efficiency of the plant and will be used during continued bulk sampling operations as the company refines its designs for a final production flowsheet. Processing of bulk samples was begun on July 29th 2004. From July 2004 to June 2005, 15,878 tonnes of kimberlite material was processed but diamond recovery has not been published (De Beers Canada Inc. Snap Lake Project, 2005).



De Beers Canada Inc.

Figure 2. Snap Lake Mine aerial photo, 2008.

Underground development resumed, with de-watering of the main ramp beginning on June 8th 2004. The 1st level was reached and de-watered by the end of June and the 2nd level was de-watered by early August. Mining development began on the 1st level in late July and on the 2nd level in early August. Work consisted of driving three new ramps between the 1st and 2nd levels, drifting north and south on the 2nd level, continuing two branches of ramps below the 2nd level, and test-mining of the ore deposit beyond the 2nd level (De Beers Canada Inc. Snap Lake Project, 2005). Work continued during 2005 with the completion of the 5280-520 and 5280-530 ramps on the 1st level, the completion of the 5280-520 ramp on the 2nd level (test mining area), the partial completion of the 5280-530 ramp on the 2nd level, and drifting on the 2nd level (5520 N. and S. drifts). Phase II mining development was also started during 2005, aimed at providing access to areas where major infrastructure such as the crusher and ore conveying system to surface will be located. Haulage levels were driven below the ore drifts to accommodate ore flow from the ore horizon to the haulage level and crusher. At the end of the year, all accesses to the conveyor ramp location were essentially complete, the South Ramp had been advanced approximately 250 meters and the 5250 N. and S. haulage drifts had started. Underground development to December 31st 2005 is shown in **Figure 1** (De Beers Canada Inc. Snap Lake Project, 2005).

Surface construction during 2005 included the installation of expanded camp facilities (260 man capacity), laying down of waste rock for construction pads, erection of the first (of three) 12 million litre fuel tank, construction of the tailings ponds, extension of the airstrip to allow for Boeing 737 jets and Hercules freighters (now 150 feet x 5,300 feet), and foundations for the mine services and processing plant buildings (De Beers Canada Inc. Snap Lake Project, 2006).

Construction and Development in 2006-2007

Construction proceeded in 2006 to erect the services, power plant, and processing plant facilities, two additional 12.5 million litre fuel tanks, expanding the camp facilities through the addition of 350 more beds, and completion of the water treatment facility and associated pumping services. Underground work in 2006 included the excavation of the 1.6 kilometer conveyor ramp, underground crusher chamber, ventilation facilities, waste access drifts, and preliminary ore drifting. By year end 2006, a total of 255,000 tonnes of rock were removed from the underground during this development program. Construction continued into 2007 and the process plant was commissioned in

August 2007 when the first diamonds were recovered. Construction of the underground conveyor to allow for full-production at a rate of 3,150 tonnes per day continued during the winter of 2007-2008.

Year:	Ore Milled:	Diamonds:
2007	113,000 tons	81,000 carats
2008	903,000 tonnes	926,000 carats

Table 2. *Snap Lake production to date (2008).*

(source: *De Beers Group – Operating and Financial Review 2007 and 2008*)

Operations 2008-2009

Official grand opening of the Snap Lake diamond mine was held with ceremony on July 25, 2008. The mining method used at Snap Lake is modified ‘room and pillar’ with paste backfill. Kimberlite is conveyed out of the mine and processed in a standard diamond recovery circuit; ore is crushed, washed, screened, pumped, classified, and concentrated using a DMS (dense-media separation) process. Fuel is stored in three tanks of 12.5 million litre capacity and an older 4.3 million litre fuel tank farm. Power generation is four 4.4-megawatt diesel generators and a standby plant of three 1.25-megawatt generators. Other facilities include a camp for 500 full-time workers, an Ammonium Nitrate/Fuel Oil plant for the production of explosives, water treatment facility, and a 1,600 meter gravel airstrip.

Because of economic conditions and the gradual diminishing of diamond prices worldwide, De Beers temporarily ceased mining and production operations at Snap Lake between July 22 and August 26, 2009. While a second shutdown was planned for December 2009, it was announced in October that market conditions had improved enough that further shutdowns would not be necessary at this time.

Exploration Since Mine Closure

Not applicable.

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De Beers Group – Operating and Financial Review 2007 and 2008.

Mackenzie Land and Valley Water Board Files – Water License N1L3-1735 (Winspear Resources Limited)

Winspear Resources Ltd. Annual Reports & Press Releases. 1998-1999.

News/North newspaper articles, 2000-2001.

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SPECTRUM LAKE

Minor Producer (Abandoned)

Gold

62° 22' 30" 112° 48' 15"

Years of Primary Development: 1959

Mine Development: 128' incline shaft, one level, 72' raise

Years of Bulk Sampling: 1959

Bulk Sample: 1,853 pounds ore shipped @ 3.04 oz/ton Au

Years of Production: 1983

Mine Production: ~500 tons milled @ 0.25 oz/ton Au

Introduction

This mine was a small gold property during 1959. It is located 81 kilometers east of Yellowknife, NWT and about five kilometers east of Campbell Lake. A small mill was installed in 1983. The site was viewed by the author from the air in July 2003.

History in Brief

The 'AL' and 'BB' claims were staked in 1945 by James E. Stevens and G. Bleiler. Three parallel gold veins were discovered and trenching indicated high values. The property was acquired by the Beneventum Mining Company Limited in 1958, and in the following year an inclined shaft was sunk into the vein deposit. Lack of funding ended the program at the end of 1959. The property reverted back to James Stevens and was later acquired by Gateway Ventures Limited in 1968. They conducted some diamond drilling in 1968-1969 but recommended that no further work be done. Dave Smith staked the 'Gary' claims in 1980-1981, and in 1983 Etthen Mines Limited erected a mill and treated a small tonnage of ore.

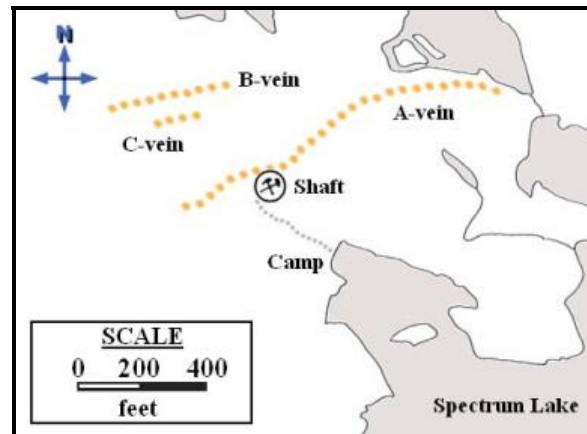


Figure 1. *Spectrum Lake Mine surface plan.*

Geology and Ore Deposits

The property is underlain entirely by strongly folded greywacke of the Yellowknife Super Group. Quartz veins occur throughout the area in a sporadic, but occasionally concentrated manner. The veins strike west-east and are parallel to the schist bedding of the greywacke. Three parallel veins, the A, B, and C-veins, were identified as gold bearing. The A-vein, the target of most work, dips steeply (60° to vertical) and has been traced for over 3,000 feet. More recent work, in 1981, exposed the vein for 100 feet, showing vein widths of 8 to 24 inches. Scheelite has been noted in the A-vein, although no tungsten assays have ever been reported. The B and C-veins have been traced for 700 feet and 500 feet respectively and are structurally similar to the A-vein, although gold values are not as high (McDonald, 1958; McGlynn, 1971).

Beneventum Mining Company Limited (1958-1959)

In early 1958, the 'AL' and 'BB' claims were optioned by the Beneventum Mining Company Limited following a favorable report by Bill McDonald, in which he recommended diamond drilling of the A-vein to outline the depth of mineralization (McDonald, 1958). A few short x-ray holes were diamond drilled on the A and C-veins. On the A-vein, drilling was conducted at 10 foot intervals and some visible gold was reported to depths of 25 feet. Six holes were drilled on the C-vein (The Northern Miner, Nov. 27th 1958). A total of 14 holes were drilled before the winter season. Five holes at the east end of the A-vein returned only low values, but the next 90 feet of vein averaged 2.75

ounces per ton gold across a width of 14 inches. A few holes reached a depth of 50 feet and 100 feet where the vein widened to 20 inches, but the material was lower grade. Two holes intersected visible gold. Exploration work by this point had indicated a vein 230 feet long with an average grade of 3.75 ounces per ton gold cross a 12 inch width (The Northern Miner, Jan. 22nd 1959). Shaft sinking on the A-vein was viewed as the best option for preliminary underground exploration, from which the B and C-veins could also be easily tapped (The Northern Miner, Nov. 27th 1958). Equipment was brought to the property by tractor train and snowmobile during April 1959, and the shaft was collared. It was sunk about midway up the A-vein and a short distance south. J.F. Nisco was manager of the operation (The Northern Miner, Apr. 30th 1959).

Bulk Sample

A small bulk sample of 1,853 pounds (from a series of 40 individual samples) of ore mined from trenches of the A-vein was shipped in late 1958, with an average assay of 3.04 ounces per ton gold. A length of 230 feet was sampled on the vein. (The Northern Miner, Dec. 11th 1958)

The Shaft

The inclined shaft (60°) was completed a length of 128 feet during July 1959 and a level was started at a depth of 110 feet. Lateral development began with drifting along the A-vein where an 18 inch wide vein was discovered (McGlynn, 1971; The Northern Miner, July 30th 1959). A raise was started in August and had advanced 35 feet at an incline of 70° through the vein. Completion of the raise to the surface was considered an important part of the development program, and lateral work on the 110-foot level was halted (The Northern Miner, Sept. 17th 1959). The raise had advanced 72 feet when work stopped in October 1959 in time for the freeze-up period (The Northern Miner, Nov. 5th 1959).

Negotiations for further funding were required before the company could proceed with completing the underground evaluation. A program of winter work was outlined which included a completion of the raise, additional drifting, and deep diamond drilling (The Northern Miner, Nov. 5th 1959). However the company did not continue work.

Etthen Mines Limited

The 'Gary' claims were staked over the old site in 1980 or 1981 by Dave Smith. Some rehabilitation of the old shaft workings was conducted. In 1983, Etthen Mines Limited optioned the claims and proposed erecting a small gravity mill, employing four men for 14 weeks in the summer months to process a 350 ton stockpile of ore. A simple flowsheet using jaw crusher, ball mill, jig, cyclone, and blanket tables was proposed, processing an average five tons of ore per day, one day a week, for 14 weeks. Tailings were to be disposed into a natural depression near the site. Maureen Crowe was agent of the project on behalf of Etthen Mines Limited. (NWT Water Board Files – Water License N1A3-1246) There is no public information about the operation beyond the company's application for a Water License. There is no record of ore treated, gold recovered, nor the longevity of the project. It is understood however that the operation was run for a number of consecutive summers, some ore was mined from the underground, and run through the gravity mill, treating about 500-600 tons of ore grading 0.25 oz/ton gold.

Exploration Since Mine Closure

In 1993, Fran Hurcomb staked the 'Frip' claim. Sampling of the veins was conducted. (Beauregard & Smith, 1994)

References and Recommended Reading

Beauregard, M. and Smith, D., 1994. Report of Prospecting Activities - Frip #1 Claim. July 1994. (*N.W.T Geoscience Office Assessment Report #083469*)

McDonald, W.L., 1958. Mining Engineer's Report – AL Mineral Claims. Beneventum Mining Company Ltd., July 1958. (*N.W.T Geoscience Office Assessment Report #062074*)

McGlynn, J.C., 1971. Metallic Mineral Industry, District of Mackenzie, Northwest Territories. Geological Survey of Canada, Paper 70-17, 1971.

Saunders, C.R., 1969. Summary Report – AL Property, Northwest Territories. Gateway Ventures Ltd., October 1969. (*N.W.T Geoscience Office Assessment Report #060392*)

NWT Water Board Files – Water License N1A3-1246 (Etthen Mines Limited)

The Northern Miner newspaper articles, 1958-1959.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0014



STARK LAKE

Advanced Exploration (Abandoned)

Uranium

62° 26' 45" 110° 18' 30"

Years of Primary Development: 1952

Mine Development: adit tunnel (563' dev.)

Introduction

The Stark Lake Mine is located on Regina Bay (Stark Lake), 22 kilometers east of Lutsel'ke, NWT in the East Arm region of Great Slave Lake. It was the site of a brief uranium development program in the early 1950s. The site has not been visited by the author.

Brief History

The 'Rex' claims were staked in 1949 by Andy Krysz and H.R. Wilson and subsequently optioned to Charles E. Ridley. During the 1950 field season, trenching and bulk sampling were undertaken on several of the showings, and diamond drilling was done on the C-vein, the most promising on the property. In 1952, underground work commenced on the C-vein with the purpose of obtaining a sample of the deposit. No further development was completed.

Geology and Ore Deposits¹

The property is underlain by a large laccolithic intrusion which has formed a large elongate topographic node above the surrounding ferruginous and limonitic sediments. All rocks are considered to be of Precambrian Age. Zones of interest are located at the contacts of the intrusive-sedimentary contacts. The injection of most uranium-bearing actinolite vein material occurred with the fracturing and faulting of the granite intrusions, but there is evidence within the B-vein that suggests some of the ore occurred pre-faulting. There is also one small vein that appears to have been formed in the post-faulting period within the limonitic breccia.

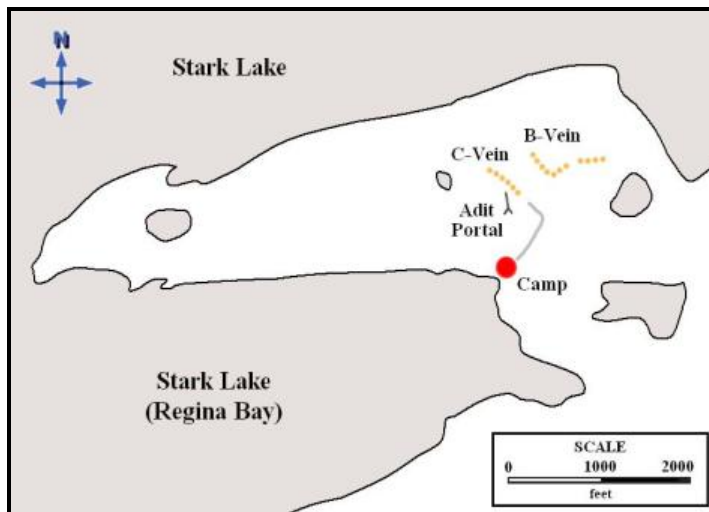


Figure 1. *Stark Lake Mine location.*

Ridley Mines Holding Company Limited (1952):

Development started in August 1952, driving a 140 foot adit at a depth of about 50 feet from the surface exposure of the C-vein. Lateral work began at the end of the month. Radioactivity was spotty during the first advances of this drift, but uranium values soon improved (The Northern Miner, Nov. 6th 1952). Drifting within the C-vein totaled 423 feet. A 50 foot raise was also excavated about 290 feet northwest of the adit. The raise did not break-through to surface (Fraser, 1955). During November and December 1952, 751 tons of ore were mined and stockpiled. Underground operations were suspended December 6th 1952. Average number of employees was 13 (NWT Mining Inspection Services).

There does not appear to have been any stoping development, and any ore removed from the mine was from development (Bert Varkonyi, pers. comm.). Sampling in the drift gave erratic values, however, some high-grade uranium oxides (U_3O_8) were encountered. Face sampling indicated an ore length of 320 feet along the drift averaging 0.22% U_3O_8 across 44 inches. This underground development indicated 7,529 tons within the C-vein above the adit level, averaging 0.22% U_3O_8 . The full length of the ore zone had not yet been explored, with an additional 100 feet expected based on earlier diamond drill results from the surface. It was also believed that other ore zones on the property held considerable value. Cost of the 1952 underground exploration program was CDN \$116,000. A 25 ton

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

per day mill was considered in 1952-1953, but this was never installed and no further mining development has been accomplished here. It was considered that the property's ore tonnage was too small to allow for an economic mining operation at Stark Lake (*The Northern Miner*, Apr. 16th 1953).

Exploration Since Mine Closure

Ridley Uranium Mines (Canada) Limited was formed in 1954 to acquire the property. They conducted some surface prospecting and sampling of the underground workings. Although prospecting revealed a number of new radioactive zones, none were important enough to merit further work. Sampling returned low values or tonnages that were too small to be of economic importance. Diamond drilling (32 holes) was conducted on the numerous zones, but failed to intersect any interesting grades and only proved that the deposits were shallow (Fraser, 1955). In 1969, Hudson's Bay Oil & Gas Co. Limited optioned the claims and conducted a radiometric survey, but failed to locate any additional anomalies in the area (National Mineral Inventory).

References and Recommended Reading

Fraser, N.H.C., 1955. Report on Property of Ridley Uranium Mines (Canada) Limited. April 1955.

Lassila, P., 1970. Geophysical Report Stark Lake Project. Ridley Uranium Mines (Canada) Ltd. February 1970.

N.W.T. Mining Inspection Services. Monthly Progress Reports – Ridley Mines Co. Ltd. (1952) Submitted to NWT Mining Inspector.

Ridley Uranium Mines (Canada) Ltd. Informational Bulletin. August 1954.

The Northern Miner newspaper articles, 1952-1953.

National Mineral Inventory (REX). NTS 75 L/8 U 1.

Personal Communication: Bert Varkonyi



STORM

Minor Exploration (Abandoned)

Tungsten

63° 00' 15" 113° 07' 15"

Years of Primary Development: 1942

Mine Development: 26' shaft, 45' shaft

Introduction

The Storm tungsten property is located 83 kilometers northeast of Yellowknife, NWT on the east side of Gordon Lake. Ruins of this site were destroyed by forest fires in 1998. The author of this report has not visited the property.

Brief History

The 'Storm' claims were staked in 1941-1942 by Cliff Brock, Gordon McLeod, and H. Campbell, prospectors with Cominco Limited. Exploration work completed during the summer of 1942 included stripping, trenching, and sampling the #2 and #3 veins, and sinking of two prospect shafts on the #2 vein. Work ceased because the gold and tungsten deposit was not economic.

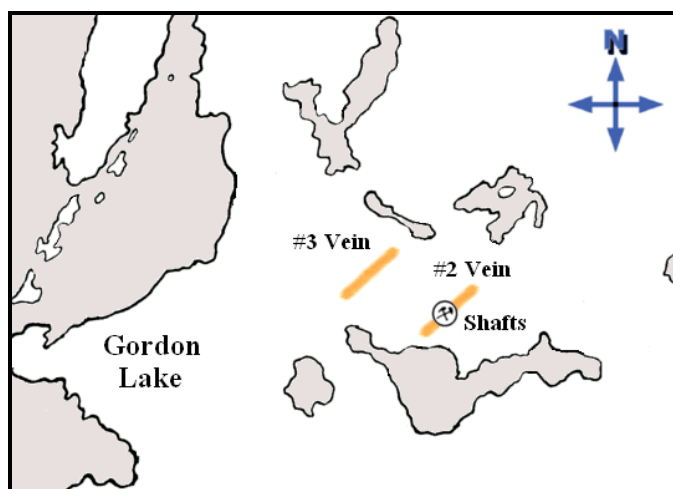


Figure 1. Storm Mine property plan.

Geology and Ore Deposits¹

Gordon Lake is underlain by greywackes and slates of the Yellowknife Supergroup. These thin interbeds of dark slate and greywacke have been deposited as a turbidite sequence. In places, the greywacke beds become thick and massive and any bedding has been obliterated by metamorphism. Quartz veins, which are abundant throughout the sedimentary rocks, consist of high-temperature glassy quartz with tourmaline and a few feldspar crystals in schists and hornfels. A granite batholith intrudes the rock group about 14 kilometers east of the property, and a few post-dyabase dykes have been noted in the area. Several quartz veins with gold and scheelite mineralization have been found, but the veins of importance are the #2 and #3 veins. The #2 vein strikes northeasterly and dips about 65° southeast and is intersected by several faults and shear zones. Scheelite occurs in the quartz as aggregates, ranging in size from small grains to several inches. It is commonly associated with carbonate material in the southeast (hanging) wall of the vein. The # vein is located 3,000 feet north of the #2 vein, and coarse scheelite has been observed in many places.

Cominco Limited (1942)

Work was reported between August and December 1942 by crews with Cominco Limited. The #2 and #3 veins were the focus of most exploration, with the #2 vein the focus of underground development through two shafts at either end of the vein. The #1 shaft was sunk on the northeast end of the vein; its dimensions are 8 feet x 6 feet and it was sunk to 26 feet depth. The #2 shaft was sunk on the southwest end of the vein; its dimensions are 6 feet x 5 feet and it

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office, and Lord (1951)

was sunk to 45 feet depth. Two mineralized oreshoots were identified; one at #2 shaft , 61 feet long and 7 feet wide with grades of 0.38% tungsten oxides (WO₃); the second at #1 shaft, 69 feet long and 6 feet wide with grades of 0.51% WO₃. A sample consisting of 215 pounds of ore from the #1 shaft to a depth of 12 feet assayed 0.25% WO₃, and trace of gold. Work ceased at the finale of the 1942 exploration season. The tungsten deposit was not considered economic enough to place a mine into production (Lord, 1951).

Exploration Since Mine Closure

The original 'Storm' claims lapsed and were re-staked by the same name in 1979 by L. Morrisroe. In 1981, Cadillac Exploration Ltd. optioned the claim group and diamond drilled 22 holes (1,269 meters) to test the various zones of the #2 and #3 veins (Brophy et al., 1984).

References and Recommended Reading

Brophy, J.A., Gibbins, W.A., Laporte, P.J., Lord, C.C., Padgham, W.A., Seaton, J.B., 1984. Mineral Industry Report 1980/1981 Northwest Territories. EGS 1984-5. Indian and Northern Affairs Canada, Minister of Supply and Services Canada, 1984.

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085PSW0026



STRIKE LAKE

Minor Producer (Abandoned)

Gold

62° 25' 30" 112° 52' 00"

Years of Primary Development: 1977-1978

Mine Development: several open cuts

Years of Production: 1977-1978

Mine Production: ~30 tons milled = 100 oz Au

Introduction

The property was a small low-grade gold operation located 77 kilometers east of Yellowknife, NWT at Strike Lake. A small amount of gold concentrate was recovered in 1977-1978. The site has not been visited by the author.

History in Brief

The property was originally staked as the 'June' group in 1939 by Cominco Limited. Work done during the following years through diamond drilling was to warrant a 20 ton per day mill which was to be installed in 1941. Conditions of World War II seemed to destroy the idea of putting the small tonnage deposit into production at this time. In 1973, the property was re-staked as the 'Joon' group by Dave Nickerson and in 1977-1978 a small power plant and mill was brought to the property.

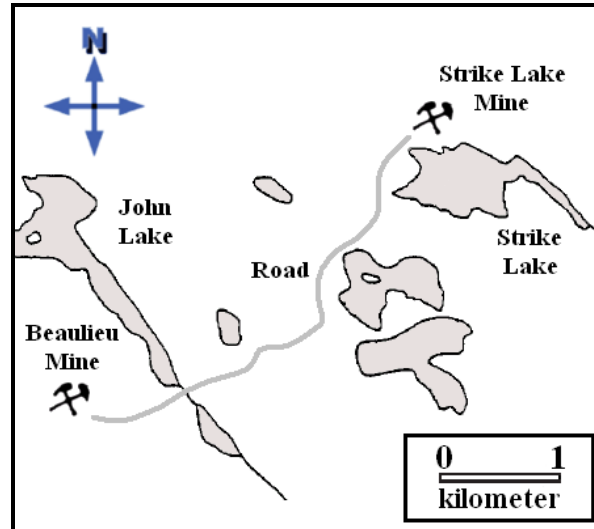


Figure 1. Strike Lake Mine location.

Geology and Ore Deposits¹

The Joon gold prospect is situated in the Yellowknife Basin, a supracrustal belt within the Archean Slave structural province. Gold occurs in quartz veins hosted by turbiditic metasediments belonging to the Burwash Formation which is part of the Archean Yellowknife Supergroup. The latter include medium-bedded greywackes rhythmically interbedded with argillite and occasional layers of grey to black, thin-bedded phyllite. Regional metamorphic grade is lower greenschist facies. Numerous stringers and small veins of quartz cut metasediments in the main mineralized zone, many of them parallel with foliation. The largest vein is about 7 meters long and 30 centimeters wide. Most of the quartz is grey and fine-grained but a small proportion is coarser grained, glassy to white and is accompanied by up to 10% feldspar. The coarse-grained variety can be seen to cut the fine-grained quartz in places. Very small amounts of native gold, arsenopyrite and pyrite occur in the fine-grained quartz.

Strike Lake Resources Limited (1977-1978)

Organized for the acquisition of the Yellowknife property, Strike Lake Resources Limited, a Vancouver outfit of 'bikers-turned-gold miners', trucked equipment to the property early in 1977 and began to mine surface pits and high-grade gold specimens. In June 1977, Strike Lake Resources was granted approval through the NWT Water Board to

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

carry out a mining and milling operation on its property. Allan Tipman, Bill Groves, and Robert Rollins were some of the men in charge of operations for Strike Lake Resources. It was decided to use existing structures at the old Beaulieu Mine, five kilometers to the west of the Strike Lake deposit, for milling operations. The old mill building was rehabilitated and new equipment installed. Some of the old camp buildings were also rehabilitated as living quarters for crews. Tailings were originally to be jettisoned into a small plywood and polyethylene containment area adjacent to the mill. As part of its water license, the tailings could not be jettisoned into the existing 1940s tailings from the Beaulieu Mine, nor could those tailings be disturbed. Water at a rate of 300 gallons per day was pumped from John Lake. (NWT Water Board Files – Water License N1L3-0504)

Mining

A series of high-grade gold veins located at Strike Lake were mined using small drills, powered by a Jaeger air compressor. In September 1977, three pits had been blasted on the vein systems; one 80 feet long, 5 feet wide, 10 feet deep; one 40 feet long, 6 feet wide, 10 feet deep; and a cross-trench about 20 feet long, 6 feet wide, and 8 feet deep. The ore was transported by truck or loader over a rough five kilometer road to the Beaulieu milling facility. (NWT Water Board Files – Water License N1L3-0504)

Milling in 1977

A simple flowsheet consisting of a crusher, a 3x3-foot Allis-Chalmers rod mill, mineral jig, riffle tables, thickener, and flotation cell was set-up to recover a high-grade flotation concentrate. Tailings were to be disposed into a small plywood containment area. Mechanical problems with the mill, and the loss of considerable gold into the tailings, required a change in original plans. The flotation cell was not used in 1977 and tailings were loaded into barrels for future re-treatment to recover all the gold. In September 1977, the mill was not operating because of these problems and crews were instead separating the gold from high-grade material by hand. (NWT Water Board Files – Water License N1L3-0504) The mill was powered by a Deutz generator, and perhaps some older units available at the old Beaulieu Mine (Knud Rasmussen and Dave Nickerson, pers. comm.) Mill heads averaged in excess of 5 ounces per ton gold (Aussant, 1985).

Operations in 1978 changed, as the crews abandoned the milling operation at Beaulieu and decided to process the high-grade material at the vein deposit itself. Part of the decision rested on the work necessary to transport the ore to the distant mill. A series of crushers and a 54-foot sluice box were erected at Strike Lake. (NWT Water Board Files – Water License N1L3-0504) The group planned to open pit the Strike Lake deposit to 80 feet depth and truck ore to Yellowknife's Con or Giant Mines for custom milling. Neither Con nor Giant Mine could support the idea, and the operation folded (Emery, 1978).

Production

There is no good record of production in the two season of work. At least 100 ounces of gold were recovered in total. In the month of July 1977, the mill at Beaulieu processed 15 tons of ore. This is the only month where there is a record of the mill in operation. Subsequent operations were apparently sporadic because of mechanical and recovery issues. In the summer of 1978, operations focused on high-grade mining and sluicing of ores. In October of 1978, 70 barrels of tailings (45-gallon drums) were stockpiled ready for future treatment. (NWT Water Board Files – Water License N1L3-0504) All together, 95 barrels of mine tailings were produced, running about two ounces per ton, each with 680 to 700 pounds of material in each barrel. This suggests approximately 30 to 33 tons of ore were all together mined and processed (Emery, 1978).

Rivalry amongst the members of the Strike Lake Resources outfit and other internal management issues resulted in the abandonment of the operation, and the men retreated back to Vancouver. Later, in 1980, 20 tons of the old tailings were sent to Vancouver for processing by Robinson Investments Limited. Results showed the tailings assayed 1.5 ounces per ton gold (Aussant, 1985).

Exploration Since Mine Closure

In 1985, Genesis Resources Corporation Ltd. conducted a reconnaissance prospecting program on their claim group, which included the Strike Lake property, under option from Dave Nickerson. Detailed prospecting was completed, and old trenches were chip-sampled. VLF-EM, magnetometer, and soil geochemical surveys were also conducted. The prospecting uncovered old trenches and also located new veins, but none of these were of any interest. Sampling of the old veins and trenches confirmed the high-grade nature of the deposits, but unless they widened at depth the veins would not be economical to mine. More diamond drilling was recommended. The VLF-EM survey uncovered four conductive zones that required more investigation (Aussant, 1985).

References and Recommended Reading

Aussant, C.H., 1985. Geological, Geochemical, and Geophysical Report on the Strike Lake Property. Genesis Resources Corporation Ltd., August 1985. (*N.W.T Geoscience Office Assessment Report #082086*).

Emery, D.J., 1978. Notes of a Meeting With Mr. Rollins, Strike Lake Resources. Giant Yellowknife Mines Ltd., December 11th 1978.

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

Nickerson, D., 1973. Preliminary Report on the Joon Mineral Claims. June 1973. (*N.W.T Geoscience Office Assessment Report #060059*).

NWT Water Board Files – Water License NIL3-0504 (Strike Lake Resources Limited)

National Mineral Inventory (June Group). NTS 85 I/7 Au 3.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0018

Personal communication: Knud Rasmussen; Dave Nickerson



SUNSET LAKE

Advanced Exploration (Remediated)

Gold

62° 51' 15" 112° 20' 00"

Years of Primary Development: 1946-1947

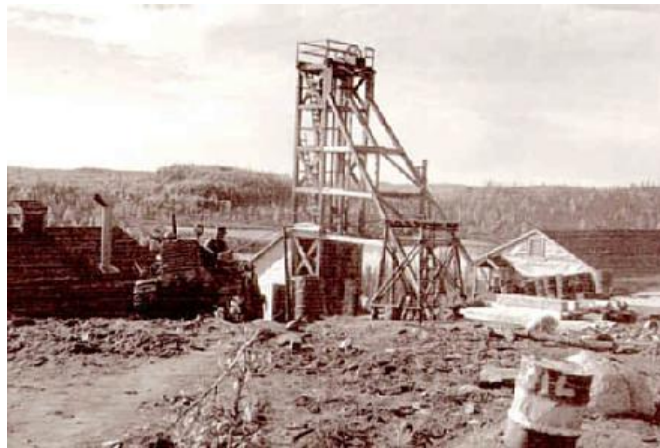
Mine Development: 145' vertical shaft, 1 level (200' dev.)

Introduction

The Sunset Lake Mine is located to the east of Sunset Lake, 112 kilometers northeast of Yellowknife, NWT. The property is developed with underground workings consisting of a shaft that was sunk in 1947. A government cleanup of this abandoned mine was completed in 1996.

History in Brief

The area around Sunset Lake was considered favorable prospecting ground during the original gold boom in Yellowknife, and in 1938 a two-man party of Fred and Bob Thompson landed on the shores of the lake, made a gold strike, and staked the 'Alice' claims. A little drilling exploration was completed in 1939 but interest petered out during World War II. In 1945, Sunset Yellowknife Mines Limited was formed to develop these claims. Diamond drilling during the year suggested enough gold to sink an exploration shaft, which was completed in 1947. No further mining development has been conducted.



Geddes Webster - NWT Archives – N-1991-009-0113

Figure 1. *Sunset Lake Mine, 1947.*

Geology and Ore Deposits¹

The deposit is located in the southeastern extension of the Beaulieu River greenstone belt in the Slave structural province. This portion of the belt comprises mainly mafic flows with lesser fragmentals, felsic volcanics and sediments. Mineralization occurs in the Alice shear zone which is approximately 900 feet long and trends about northwest and has a dip of 85° west. The shear zone follows a band of chert like interflow tuff that varies from 12 inches to approximately 20 feet wide. This tuff is hosted in fine to coarse grained greenstones derived from andesitic and dacitic lavas. These flows are massive, pillowed and schistose and range from 75 to 150 feet wide. Tuff horizons and cherty zones occur within this greenstone package and range from a few inches to 7 feet thick. The tuff is a greenish grey, thin-bedded, fine grained to cherty textured rock that weathers light grey or white and breaks with a conchoidal fracture.

The rocks within the shear zone are sericite and chlorite schist derived from the tuff and lava, and scattered veinlets and lenses of quartz a few of which attain local widths of a foot or more. Pyrite is abundant and zones can range in thickness from a few inches to several feet wide. The average width of the mineralized part of the exposed shear zone is probably between 2 and 3 feet, and its exposed surface length is around 500 feet. Gold mineralization is erratic in the shear and abundant pyrite was encountered within the deposit.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

Sunset Yellowknife Mines Limited (1946-1947)

During 1945, a diamond drill was set up on four claims which covered the most interesting deposit – the Alice shear zone. Drilling was also performed on other sections of the property, and in total 3,250 feet of drilling was accomplished that year. Operations ceased for the coldest winter months of 1945-1946, but resumed in February 1946. It was announced early in the year that the company intended to sink a shaft to at least 200 feet on the Alice zone (Lord, 1951).

1946 Work

Equipment failed to arrive on time during the early 1946 winter transport season. A Cat Train full of supplies and equipment for the company was lost on Great Slave Lake, so a start at shaft work did not get underway until the summer of 1946. The shaft was collared by hand steel and brought down 28 feet before freeze-up (Lord, 1951).

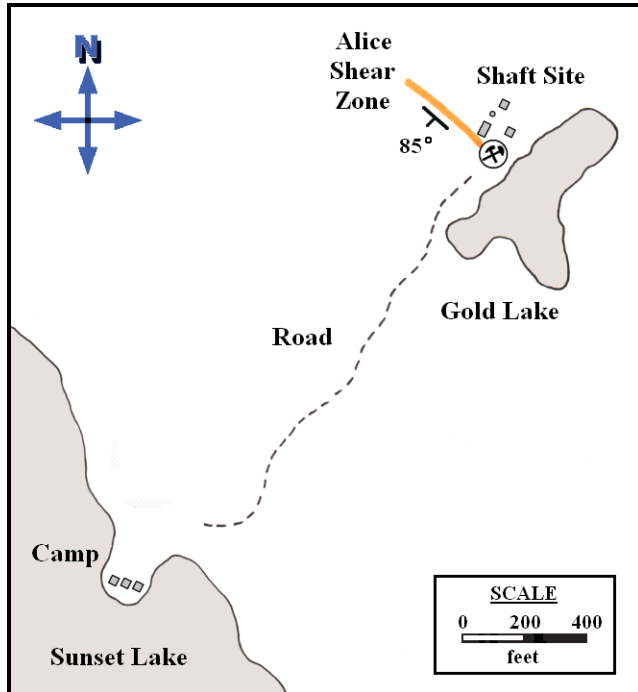


Figure 2. *Sunset Lake Mine property plan.*

A small camp was erected on a sandy beach on the east side of Sunset Lake. A tractor road connected to the shaft site, which was located 1½ kilometers northeast of the camp on the north side of Gold Lake. Camp buildings consisted of three 16 foot x 32 foot frame buildings used as bunkhouse, cookery, warehouse, office, and staffhouse. They were not designed for winter use, as it was not planned to continue work over the winter (Lord, 1951).

1947 Work

Activity at Sunset Lake resumed in March 1947 when a Cat train convoy arrived at the property with a full mining plant. A shaft crew was hired from Cobalt, Ontario, and put under charge of Walter Hylands, mine manager. About 20 men were employed during the summer of 1947, including a staff of three (Lord, 1951).

Equipment installed at the shaft site included a WL 60 Holman compressed air plant operated by an International diesel engine. A small single-drum 6x8 Canadian Ingersoll-Rand air hoist, consumed the majority of this air output. Other machinery included a Gardner-Denver steel sharpener, and a Cat D-4 tractor (Lord, 1951).

Buildings at the shaft site consisted of hoist/compressor house, shop, dry, and assay lab. A 42 foot timber headframe was erected over the shaft opening (Lord, 1951).

By September 10th 1947, when the property was closed, the two-compartment shaft (6 feet x 12 feet) had been sunk to a depth of 145 feet. A level was established at 125 feet and two drifts each of a length of about 100 feet extended north and south from the shaft. Many samples were taken from the underground workings and apparently assayed onsite. Some of these ranged as low as 0.22 ounces per ton to as high as 5.56 oz/ton over a width of 1.6 feet. While sinking the shaft, a section was encountered assaying 2.6 oz/ton over a width of 1.4 feet. Despite these interesting grades, the deposit was considered too isolated and erratic to be mined economically. Underground work stopped in September 1947 and the camp was later closed (Lord, 1951). No further mining development has been undertaken.

Exploration Since Mine Closure

Giant Yellowknife Mines Limited optioned the claims in 1966 to perform geological mapping, a small EM survey, and 469' of diamond drilling west of the old shaft. The property was acquired by United Cambridge Mines Limited in 1976 and conducted more geophysical surveys. Part of the property was re-staked as the 'Lucky' claims by J. Arden in 1981 and was then subsequently optioned to Aber Resources Limited. The shear zone was sampled, but low assays were reported (National Mineral Inventory). In 1998, Aber returned to perform additional surveying through a joint-venture agreement with Noranda Explorations and Hemisphere Development Corporation. Hemisphere Development

conducted surveys and sampling in 1999, but the claims have since been allowed to lapse (Vivian, 1998; Hopkins et al, 2000).

References and Recommended Reading

Hopkins R., Stubley, M.P., 2000. Compilation and Evaluation of Previous Exploration Activities; Geology of the Southern Sunset Lake Area. Aber Resources Ltd. and Hemisphere Development Corp. January 2000. (*N.W.T. Geoscience Center Assessment Report #084254*)

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

N.W.T. Geoscience Center Assessment Report #061716 (underground plan)

Vivian, G., 1998. Report on the 1998 Summer Reconnaissance Prospecting, Mapping and Sampling Program at Sunset Lake. Hemisphere Development Corp. Sept. 1998. (*N.W.T. Geoscience Center Assessment Report #084095*)

National Mineral Inventory (Lucky - Alice). NTS 85 I/16 Au 1.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085INE0157



TERRA (aka SILVER BEAR)

Producer (Abandoned)

Silver, Copper

65° 36' 15" 118° 07' 00"

Years of Primary Development: 1968-1985

Mine Development: main decline to 1,300' depth, 13 levels; 600' vertical shaft; two small decline/adit tunnels

Years of Production: 1969-1985

Mine Production: 507,629 tons milled = 14,236,325 oz Ag, 2,284 tons Cu

Introduction

The Terra Mine is located along the Camsell River in the Great Bear Lake region, 53 kilometers south of LaBine Point (Port Radium). It is 395 kilometers northwest of Yellowknife, NWT and was a silver and copper producer between 1969 and 1985. The site was visited by the author in August 2006. The mine is often also called the Silver Bear Mine.

Brief History

The property was originally staked in the 1940s by Eldorado Mining and Refining Limited during their exploration of uranium in the Great Bear Lake area. Diamond drilling in the 1960s showed significant silver values, but the claims were allowed to lapse. They were re-staked in 1966 by Charlie Sutton as the 'A' claims and acquired by Silver Bear Mines Limited, a company which became a wholly owned subsidiary of Terra Mining and Exploration Limited in 1967. An exploration program during 1967-1968 uncovered interesting and high-grade silver deposits, and quickly the decision was made to place the property into production.

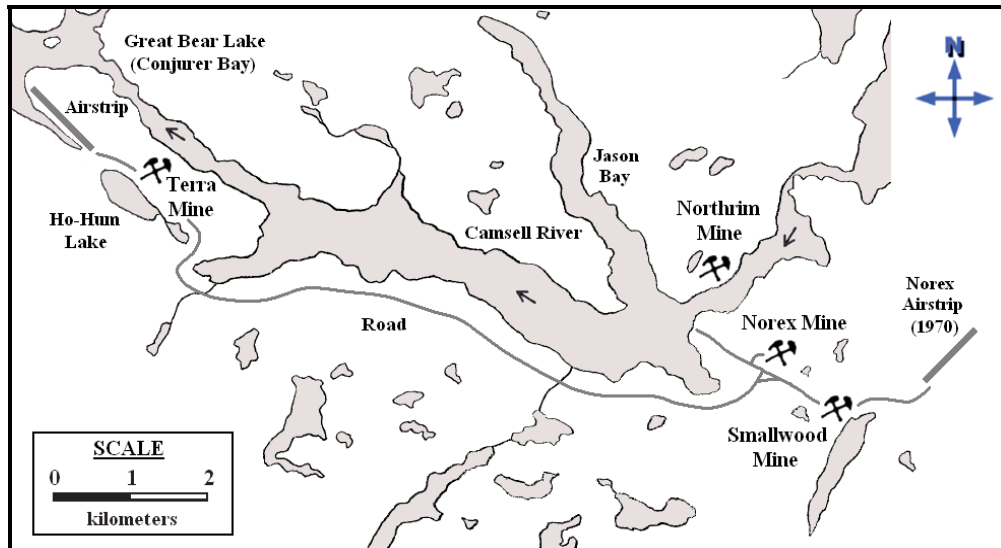


Figure 1. Terra Mine location.

Production started in the fall of 1969 with practically unknown ore reserves. Management was optimistic that continued development would prove the mine had merit, although in the early 1970s the company was not always financially well off. Conditions improved in the 1970s as depth development uncovered new ore reserves, and Terra joint ventured the rich, adjacent Norex Mine into production in 1977. New ore was also uncovered at the northeast end of the property in what became known simply as the North zone, and by the early 1980s the company was processing ores from four properties: the original Silver Bear zone, the North zone, and the Norex and Smallwood Lake Mines. Declining ore reserves and a lower commodity price of silver resulted in the closure of the mine in 1985.

Geology and Ore Deposits¹

The Terra Mine is located on the southwest limb of the Norex syncline near the north boundary of the Rainy Lake intrusion. The mine is hosted in the Terra Formation, consisting of an upper member comprising lithic arkose, calc-

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

argillite, conglomerate, siltstone and mudstone, and a lower member comprising mudstone, breccia, cherty tuff, lapilli tuff, siltstone and limey mudstone. Most of the ore is upper member, which in the mine area contains more than 10% banded and disseminated sulphides: mainly consisting of pyrite, pyrrhotite, and chalcopyrite, but also argentite, cobalt-bismuth arsenides, native silver, and native bismuth. The sulphides are used as a guide to the silver-bearing veins which are commonly in the sulphide-rich zones. The ore is concentrated in steeply to moderately south-dipping, northeast-striking quartz-carbonate-hematite vein systems that cut the banded sulphides. The veins, which are usually less than a meter wide, are related to an east-northeast-trending splay from a major northeast-trending fault to the west. The veins are composite and complex, containing silver-bearing pods surrounded by a thin halo of silicic, hematitic, chloritic, and carbonate alteration. Associated minor minerals include skutterudite, safflorite, rammelsbergite, parammelsbergite, matildite, and sphalerite. Ore is also found in thin gash veins that trend north to northwest and dip steeply west, as well as, in fractures within syenite to the south.

The North zone lies immediately north of the Terra veins and is in the Camsell River Formation, a series of andesitic to basaltic flows, tuffs and breccias that overlie the Terra Formation. Silver bearing veins (#52-55) are associated with a fault that underlies the Camsell River. The veins strike easterly and dip 70 to 80° south.

Terra Mining and Exploration Limited (1968-1982)

Upon review of Terra's northern properties, the company's consultants agreed that the silver property at Camsell River, Great Bear Lake had the best chance of becoming a mine. A diamond drill program was conducted during 1967-1968 and intersected rich sections of silver and copper minerals within a shear zone at least 150 feet wide and 3,000 feet long. Bismuth and some cobalt were also encountered. In the early spring of 1968 the decision was made to go underground (Terra Mining & Exploration Ltd. Annual Report, 1969). A major supply program by truck over winter road was initiated in March 1968. In April, the exploration camp was expanded and underground work on the #8 vein gossan was started. This short tunnel, from which about 300 feet of lateral work was performed, is known as the 'A' Mine. Work at the 'A' Mine then ceased and a new decline portal was started further south on the north shore of Ho-Hum Lake. In July 1968, underground diamond drilling started and was finished by October. In November 1968, the decision was made to place the property into production and major re-supply was started in January 1969 (Spence, 1970).

Decline Drive

The driving of declines was a new method in Canadian mines at the time, and Terra was apparently one of the first properties to conduct this type of underground excavation. The decline was driven during 1968-1969 at an incline of -17% for a length of 2,000 feet to a depth of 560 feet below the surface. A raise 5 feet x 7 feet was driven 410 feet from the 300-foot level to the surface. The dimensions of the decline are 12 feet high and 15 feet wide, allowing the entry of trackless scooptrams and later ore trucks (Terra Mining & Exploration Ltd. Annual Report, 1969).

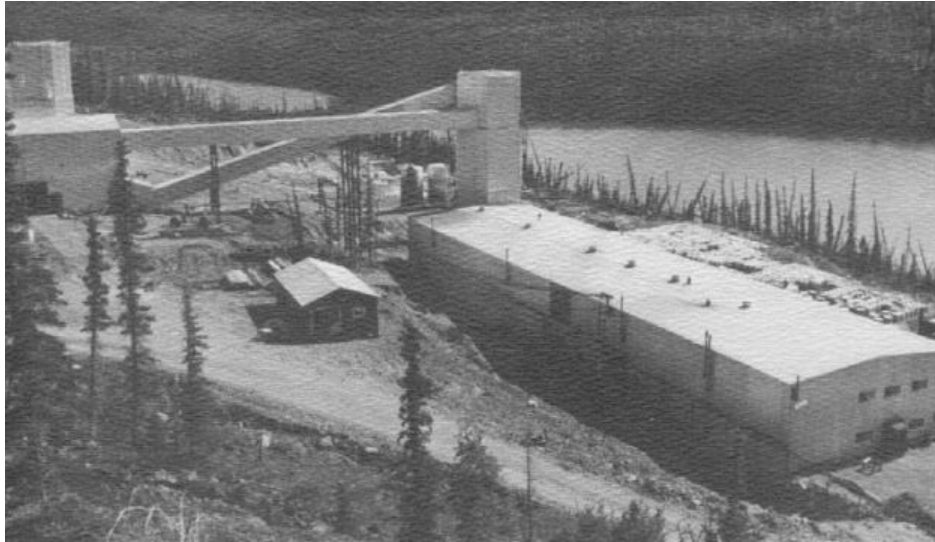
Two levels were established at 150- and 300-foot depths. The 1st level was only accessible through a service manway from the main decline and also through the original decline portal that connected to the 1st level through a short raise. A small EMCO L.H.D. scooptram unit was in use on the 1st level, which was dismantled and hoisted down the raise in pieces from the old decline. From the 1st level, crosscuts were driven on the high-grade veins. Only three of the seven veins were explored from the underground at this point, each disclosing high-grade sections of silver. The zone on the 2nd level was reported in 1970 to be much wider and had potential for bulk tonnage (Spence, 1970; McDonald, 1970; Sanche, 1970).

Construction of a 300 tons per day capacity mill and camp facilities for 80 persons was initiated in the spring of 1969. The milling plant was a newly purchased unit and was described as a first rate facility, bought and installed at a cost of over CDN \$2 million (1969 figures). Underground development recommenced in May 1969, and by October a full crew of miners was at work. During early advanced, nine additional silver-bismuth veins were intersected, four of which contained high-grade silver. The company announced that reserves existed to permit a 4 to 5-year mine life, although no actual reserve figures were released at this time (Terra Mining & Exploration Ltd. Annual Report, 1969).

Production Begins

The plant start-up commenced in early September but due to a faulty crusher, production of concentrates did not get underway until late in October 1969. Production was at a rate of 150 tons per day. A 450 ton shipment of flotation concentrate was made in May 1970 to customers in Sweden. The silver and copper content was quite low since the ore mainly came from development muck and lower grade stopes. Because the mill and camp cost about 100% more than anticipated, no stopes had been developed ready for production, and it wasn't until January 1970 that stopes were opened up for steady production (Terra Mining & Exploration Ltd. Annual Report, 1970).

Production was interrupted in February 1970 by a fire in the crushing plant, conveyors, and ore bins. Repairs were not completed until May 9th 1970, when production resumed (The Northern Miner, Sept. 24th 1970). Other problems faced during the first year of operations were the freezing of ore in the ore bins during the winter of 1969-1970. A steam heating facility was installed and the ore bins were insulated. The cone crusher was faulted and required replacement, and the ball mill was incapable at running at capacity due to a misalignment of the concrete pedestal foundation (Terra Mining & Exploration Ltd. Annual Report, 1970).



Terra Mining & Exploration Ltd.

Figure 2. Terra Mine milling plant, c.1971.

Although the fire damage was fully covered by insurance, the delay encountered during the three-month shutdown period resulted in a squeeze in available funds. The first year of operations at Terra Mine was effected by many problems: shortage of working capital, lack of ore grade mill feed, an unreliable crushing and grinding plant, lack of data on copper and silver grades, over-evaluation of the bismuth values in the cash-flow scheme, and miscalculations of the mill grinding capacity due to the use of higher-grade ore samples when conducting the original milling designs.

The primary problems with the mill-feed grade was the lack of mining experience. The main mineralized zone was a shear with high chalcopyrite in sub-commercial quantities, known as the Copper zone. Three silver veins, the #7 to #9, occurred within this zone. The relationship between the Copper zone and the silver veins had not been clearly established, resulting in incorrect estimates of potential mill head grade from the Copper zone. For the period 1970-1971, production was maintained from the Copper zone stopes while efforts were made to explore the veins and develop silver stopes. The decline was extended to the 400-foot (3rd) level and a new stope was developed. This stope, the 309, undercut the best grades yet encountered: 153 ounces per ton silver over an area of 46 tons per vertical foot with a mining width of four feet. By this time, the original #8 vein was practically mined out above the 2nd level, and production was available from the #7 and #9 veins. Depth extension of the #8 vein on the 3rd level had not yet been established (Terra Mining & Exploration Ltd. Annual Report, 1971).

Milling Operations

The following is based on a 1977 description of the plant flowsheet. Ore was fed into the crushing circuit via scooptram and later low-profile haul trucks and loaders. Crushing was accomplished two-stage with oversize material from the first crusher (24 inch x 36 inch Robertson jaw crusher) passing over the screen and reporting for secondary crushing in a 4 foot Symons cone crusher. Undersize material (-3/8 inch) was sent for grinding in a 8 foot x 6 foot Denver ball mill with manganese liners operating in closed circuit with a duplex 16 inch x 24 inch Denver mineral jig. A silver-bismuth gravity concentrate was recovered from the jig, dewatered, dried, and bagged in 100 pound sacks for shipment. Jig tailings passed through a 6 inch cyclone classifier, with underflow material being sent back to the ball mill. Cyclone overflow, at 40% solids, was pumped directly to the flotation circuit.

Three stages of flotation were performed. The first stage was rougher flotation (using polypropylene glycol as a frothing agent) in a Denver four-bank unit. Tailings from the rougher flotation unit were sent to the scavenger

flotation tanks (Denver 12-bank unit) where sodium isopropyl xanthate and potassium amyl xanthate were added. A final waste tailing was produced from the scavenger cells (pumped into Ho-Hum Lake), while a scavenger concentrate was pumped to a conditioning tank together with the rougher concentrates from the first stage of flotation. Hydrated lime was added to the conditioning process. The conditioned material was then sent to the third stage of flotation (cleaner flotation, consisting of a Denver 6-bank unit). The cleaner flotation tailing was recycled back into the first stage of flotation, while the final clean concentrate (silver-copper) was thickened and vacuum filtered, then dried prior to being packaged in one ton cardboard crates.

Jig concentrates were shipped from the mine via airplane (DC-3) to Yellowknife. The bulkier flotation concentrates were shipped out in one ton containers on Hercules airplane or on the winter ice-roads (much cheaper). All concentrates were then trucked south to the Cominco smelter at Trail, B.C. for refining. Jig (silver-bismuth) concentrates had much higher silver values (5,000 to 10,000 ounces per ton) while flotation (silver-copper) had much lower silver values (250 to 300 ounces per ton, and 25% copper). Bismuth, cobalt, and nickel were available in these concentrates, but the company was never able to recover enough of these metals to make their refining profitable. Therefore, the production of those metals was never published and any data regarding its recovery was based on mine assays of concentrates (Shaede, 1977).

The flowsheet and operation of the mill generally remained consistent throughout the life of the mine, the only major change being in 1983 with the installation of a larger ball mill. In 1982, silver recovery was reported as between 95% to 99% (Bacon, 1982).

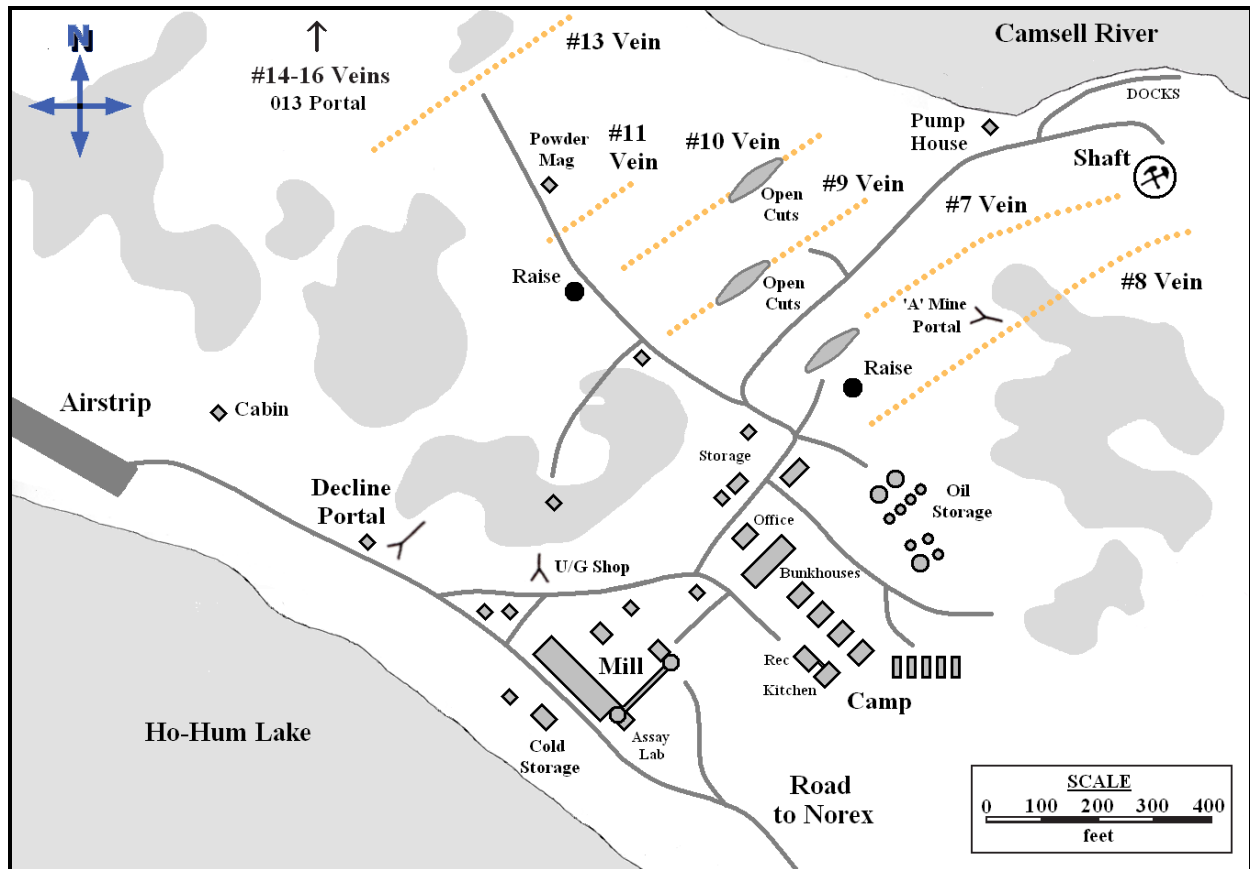


Figure 3. Terra Mine camp and plant layout, also showing the location of veins. The North zone deposit is located to the northeast of the map, under the Camsell River.

Power and Heating Plant 1970s

Power was originally (1970) supplied to the operation by two 500 KVA Cat D348 diesel generators, a 125 KVA Volvo standby generator, three stationary Atlas-Copco air compressors of 600 cubic feet per minute capacity, and one portable 500 cubic feet per minute Holman air compressor. An oil-fired boiler supplied heat to the camp and service buildings (Spence, 1970).

Mining Operations

A modified shrinkage stoping mining method was found to be the most economical at Terra. A series of chutes and boxholes emptied to loading pockets on the 2nd level, from which ore was collected by scooptrams. Mining on the 1st and 2nd levels during 1970-1971 was based on the L.H.D. system (load, haul, dump) using Wagner scooptrams for all material handling. It was realized that such a mining setup would not be economical beyond the 2nd level, and in 1975 low-profile haul trucks were introduced to truck ore out of the mine (L.M. Manning & Associates Ltd., 1975).

Camp Facilities and Employees

The camp consisted of three 20 man bunkhouses, cookery and kitchen, and a recreation hall. Total capacity was 80 men. In 1971, it was reported that the property employed 65 persons, working on an 8-hour, 5-week rotation.

The mine and company operated at a financial loss for two years until October 1971, when conditions improved because of better mill-feed grade from the silver vein stopes, operating improvements, and a refined understanding of the mineral reserve. The failure of one of the diesel generators in November 1971 resulted in a temporary shutdown of the mill until May 1972. During the shutdown period, the company was able to reach new financing agreements. They were based heavily on the sale of concentrates, which were sold at 60% of their gross value at the mine site, the remaining 40% (less freight and smelter charges) returned to Terra Mining & Exploration Limited after sale to the smelter. The advanced payments on the concentrates provided excellent capital to permit the company to carry on normal operations and conduct exploration (Terra Mining & Exploration Ltd. Annual Reports, 1971-1972).

About two thirds of the total crew was returned to work for recommencement of mining on May 5th 1972. Exploration through geophysical and diamond drill work showed good indications for silver-bearing veins to the west of and parallel to the existing veins. Drifting commenced towards this new area from the 1st and 2nd levels, uncovering the #11 vein. Extensive development was completed on these veins during 1973-1974. Good silver-bearing vein and sulphide ore was established in a strong zone of #11 vein, with assays of up to 70 ounces per ton silver. The decline was driven to the 4th and 5th levels in 1972-1973, 600 feet below the surface. Production operations during 1972 was derived from the 107, 208, 307, and 309 stopes (Terra Mining & Exploration Ltd. Annual Reports, 1972-1973).



Terra Mines Ltd. photo

Figure 4. Scooptram transporting ore to the crusher, 1970.

In late 1973, a flat exploration drill hole from the 4th level intersected native silver between the #9 and #11 veins. The #10 vein was subsequently developed on three levels (2nd, 4th, and 5th), and was reported as the strongest mineralized structure encountered in the mine to date, in terms of size and quality. The deepest intersection on the #10 vein was at a depth of 650 feet, averaging over 3,000 ounces per ton silver across 3.5 feet of diamond drill core length. Exploration drifts were also driven towards the #7 and #9 veins on the 4th level during 1973. The #10 and #11 deposits persisted at depth (Terra Mining & Exploration Ltd. Annual Report, 1973).

In 1974, production was primarily from the #10 vein above the 5th level. The 210 stope was started in March 1974. Meanwhile, the decline was advanced towards the 6th level during the year; it continued to the 8th level in 1975. Production in 1975 was from three veins structures on practically all levels: #10 vein from surface to 6th level, #11 vein from 4th to 7th level, and #13 vein on the 2nd, 5th, and 7th levels which was just starting to be developed. In 1976, the decline was extended to the 11th level. The main source of mill feed in 1976 was the #10 and #11 veins, and related structures between the 7th and 9th levels. In 1977, the 12th level was opened up and mining commenced on the #11 and #12 vein structures at that level by year-end. Production was derived mainly from the 911 stopes (Terra Mining & Exploration Ltd. Annual Reports, 1974-1977).

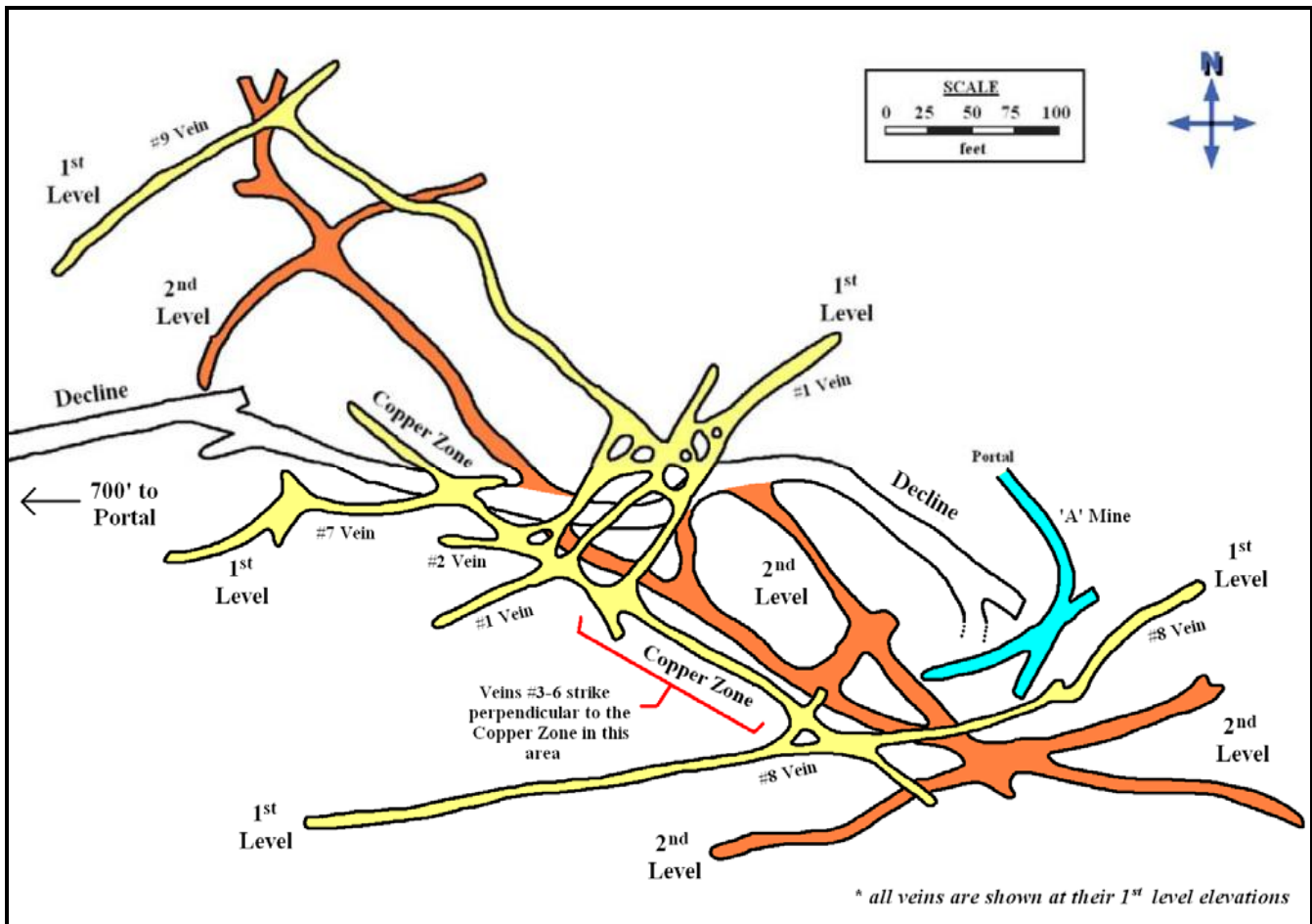


Figure 5. Terra Mine underground plan, July 1972, showing the first two levels and their extent at that date. The decline continued to the 3rd level at this period, but the 3rd level is not shown.

North Deposit Discovered

Terra brought a large diamond drill to the property in 1976 to begin an extensive drilling program to test the regional geology to a depth of 3,000 feet. Late in the year, a new ore deposit was discovered: the North zone, located beneath the Camsell River. Crosscuts were driven north to intersect the zone on the 6th level in 1977 and on the 7th level in 1978. A raise was driven from the 6th level to surface in the North deposit area for ventilation purposes (Terra Mining & Exploration Ltd. Annual Reports, 1977-1978).

Mining commenced on the #54, #55, and #56 veins during 1978-1979. The #54 vein was told to be the most interesting, together with early indications of ore-grade material in the #52 and #53 veins parallel to the west. Production of ores from the North deposit were only performed in 1979, when 4,892 tons of ore was milled to recover 65,000 ounces of silver. The North deposit also contained high uranium oxide values, and it was considered that more work was needed in separating the uranium from the silver values in order to derive economic benefit from the ore material (Terra Mining & Exploration Ltd. Annual Reports, 1978-1979).

Shaft Project

In 1978, the mine decline was completed to a depth of 1,300 feet on the 13th level. Because of the possibilities of silver veins persisting at greater depths, Terra made plans for sinking of a shaft from the surface to the 1,300-foot level. It would allow for an increased development rate with present equipment and significantly reduce the cost of ore transportation. This shaft was sunk by J.S. Redpath Limited in 1978-1979, but only reached the 6th level by the time work ceased. A foundation for a large and expensive hoist was prepared, but the hoist machinery itself never left the manufacturer's yard (Terra Mining & Exploration Ltd. Annual Report, 1978).

Terra Mine production for the first half of 1978 was derived from stopes on the 12th and 13th levels; production ceased during the second half of the year in order to focus on milling of Norex Mine ores. The shaft project was curtailed in 1979 with production coming chiefly from the 2nd level stopes (the old Copper zone) and the North deposit area. The #52 and #53 veins in the North zone were found to lack significant ore reserves, while diamond drilling of the #54 to #56 structures indicated significant potential. A 1,500 foot crosscut was driven westerly to explore the #15 vein area in 1979 (Terra Mining & Exploration Ltd. Annual Reports, 1978-1979). Production in 1980 was focused on Terra Mine and consisted of mining out ore in the #7, #13, and #15 veins. The #7 vein was a small tonnage producer, and the #13 and #15 veins did not produce commercial ore. A small amount of production was derived from the erratic yet high-grade #14 vein. Development ceased in the North zone area during the year (Terra Mining & Exploration Ltd. Annual Report, 1980).

The decline in production and ore potential since 1978 was the result of two main factors: exhaustion of the vein systems in the upper levels that previously sustained the mine, and the physical limits imposed on the existing mine facilities. In dealing with the first issue, diamond drilling in the old areas of the mine failed to locate sizeable extensions to known deposits, and the second issue was the result of longer haulage requirements in the lower levels and the inability to raise the money needed to invest in new mining methods.

Suspension of Milling

Full-scale production and milling operations were suspended in January 1981 in order to focus on exploration of the region, as per an agreement with Procon Explorations Limited signed in December 1980. This agreement called for the transfer of a large block of mineral claims north to and adjacent of the Terra Mine to Procon. Procon funded a joint-venture exploration program in exchange for 50% interest in the Terra Mine. This extensive surface diamond drilling program failed to find any economic silver deposits. Underground exploration of the #5 and #9 veins also failed to increase ore reserves. At the adjacent Norex and Smallwood Mines, new ore reserves were discovered so practically all mine development in the last three years of operations was focused on those sites rather than Terra Mine (Terra Mining & Exploration Ltd. Annual Report, 1981).

Terra Mines Limited (1982-1985)

Production resumed in June 1982 following the abandonment of the December 1980 agreement with Procon Explorations Limited, due to the poor exploration results attained. Procon traded its 50% right to an interest in Terra Mine for 100% ownership of the claims on the north side of the Camsell River. Also during the year, Terra Mining completed its acquisition of Duke Mining Limited and changed its name to Terra Mines Limited. Shares of the new company were authorized for trading on the Toronto Stock Exchange in January 1983 (Terra Mines Ltd. Annual Reports, 1982-1983).

New management was hired early in 1982 to help revitalize the silver mines in the Camsell River operation. Production at Terra Mine from 1982 onward consisted of mining out remnant stopes. During 1982, ore was mined from the 105, 107, 102, 209, 309, 511, 610, 711, and 1211 stopes in the Terra Mine. Broken reserves left previously in the old stopes of the #10 and #11 veins were recovered. A scaled-back exploration program was undertaken in the last ditch effort to locate remaining pockets of silver mineralization. This included lateral development of a diamond drill intersection within the #10 vein on the 4th and 5th level, exploration development in the 111 and 211 stope areas, diamond-drilling of the #10 vein between the 3rd and 6th levels, geological surface mapping, and diamond drilling below the bottom level of the mine (13th level) to test for sulphides (McCormack, 1983).

Other work in 1982 included the expansion of camp facilities through the construction of a two-story bunkhouse. More expansion was planned in order to maximize the efficiency of the mill. In 1983, a new ball mill was added to the circuit to bring capacity of the plant to 400 tons per day. This daily tonnage figure was never actually reached, and the mill produced ores from the Terra, Norex, and Smallwood Lake Mines at a steady rate of 200 to 250 tons per day during this period. With the increased milling capacity, the company planned to only operate the plant at only 50% of the available hours in order to save on fuel consumption. Other new equipment helped to modernize the mining operations. An underground shop was excavated near the main decline portal during this period. The Camsell River operations employed 100 people in the summer of 1983 (Terra Mines Ltd. Annual Report, 1982-1983).

Mining Equipment 1980s

In 1982, the following equipment was listed in use at the Camsell River operations of Terra Mines Limited (also used at the Norex and Smallwood Lake Mines): three Wagner ST2B scooptrams, four Wagner ST2D scooptrams, one Eimco 912 scooptram, two Wagner ST5A scooptrams, three Jarvis-Clark JDT420 low-profile haul trucks, one Jarvis-Clark JDT413 low-profile haul truck, one Jarvis-Clark MJM2DB two-boom jumbo drill, two single-boom hydraulic

jumbo drill, four Deutz tractor-mancarrier trucks, one Jarvis-Clark ranger underground truck, three Atlas-Copco 600 cubic feet per minute air compressors (stationary), one Gardner-Denver 600 cubic feet per minute electric air compressor, two Gardner-Denver 750 cubic feet per minute portable air compressors, three Copco 600 cubic feet per minute diesel portable air compressors, one Copco 700 cubic feet per minute diesel portable air compressor, one Holman 600 cubic feet per minute diesel portable air compressor, one Ingersoll-Rand 600 cubic feet per minute diesel air compressor, and one Gardner-Denver 1200 cubic feet per minute air compressor. Surface equipment included one Clark forklift, one Cat 966C loader, one Cat D-3 dozer, one Cat D-6 dozer, one International truck, one John-Deere 544 forklift, two Kenworth dump trucks, one GMC dump truck, one International dump truck, one Ford tractor, one Champion grader, and 11 pickup trucks. Underground drills consisted of a large collection of Secan S250 stoper and jackleg drills. Power at the Terra Mine was supplied by two Cat D379 diesel generators producing 420 kilowatts each and two Cat D348 diesel generators producing 530 kilowatts each. Norex and Smallwood Lake mines had their own sources of diesel power (Bacon, 1982).

Production in 1983 was derived again from remnant stopes, crown pillars, and cleanup work on all levels of the mine. Stope production included the 107, 137, 211, and 510 stopes. New development consisted of a short decline from surface to access and mine-out a small pocket of silver mineralization in an area known as the 013, about 50 feet below surface. Open pit stripping also commenced in the #7 to #10 vein areas. Surface exploration of the #13 vein showed some ore-grade material above the 2nd level, resulting in some development and production of this area during the 3rd quarter of 1983. Ore reserves at September 20th 1983 for the Terra Mine were 2,120 tons broken ore grading 13.6 ounces per ton silver, 380 tons proven ore grading 20 ounces per ton silver, 2,275 tons probable ore grading 26.8 ounces per ton silver, 2,300 tons drill-indicated ore grading 16 ounces per ton silver, 9,245 tons of sub-grade ore grading 3.9 ounces per ton silver, and 2,700 tons of stockpiled ore grading 11.1 ounces per ton silver (McCormack, 1983; Terra Mines Ltd. Annual Report, 1983).

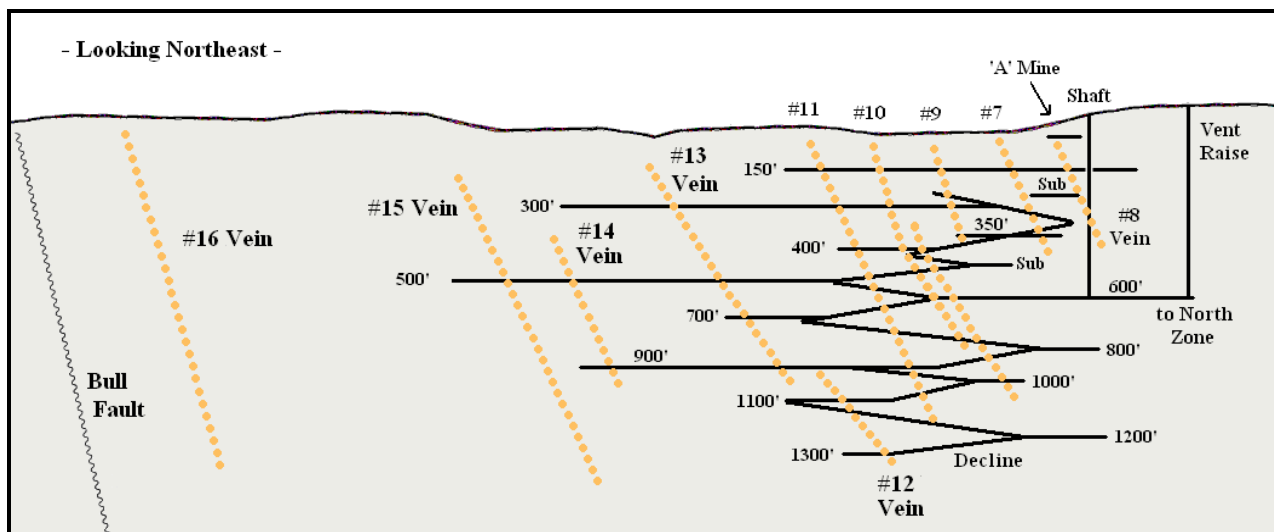


Figure 6. Terra Mine longitudinal section, c.1980.

Terra Mine Staff

Staff turnover was high, and it seems a new manager was in place every year. The following has been tabulated from mine records: Len C. Dixon (1969), David Williams (1969-1970), E. Bengst (1970) J.J. McKay (1970), Douglas Parent (1970-1971), C.A. McLeish (1971-1972), Michael Stoner (1972), D.M. Hogarth (1972-1973), D. Maciejowski (1973), J.F. Irwin (1974), M.V. Maki (1974-1975), G. Crompton (1975-1976), R.L. Roscoe (1976), C.P. Moore (1976-1978), C.A. McLeish (1978-1979), E.A. Shaede (1979-1980), Steve M. Hodgson (1980-1982), Fred J. Sveinson (1982-1983), Jim A. McCormack (1983-1984), Mike F. Horne (1984-1985), and G. Zander (1985). Hank Sanche was mine geologist for practically the entire life of the mine. Alvin Harter, Terra Mining and Exploration Limited president from 1969-1982, was instrumental in operations and development at Terra. In 1982, Barry Way was mine geologist and Dave Chapman was mill superintendent (Terra Mines Ltd. Annual Reports; mine records).

Final Year of Operations

Operations at the mine suffered in 1984 due to falling grades and falling silver prices. The mill was temporarily shut down in January 1984 to save costs on running the plant during the cold winter months. Exploration and mine

development continued during this period, and milling resumed in April 1984. The price of silver on international markets then plummeted during the summer of 1984, forcing the company to scale back operations rather than expand them as planned. No production was derived from the satellite mines (Norex and Smallwood Lake) with all production from Terra Mine. Ore was obtained from small vein structures in previously mined areas, thereby reducing development costs. As a result of the depressed state of silver markets, high costs of operation, and the lack of ore reserves, Terra Mine ceased operations in April 1985. Crews and equipment, wherever possible, were transferred to Terra's Bullmoose Lake Mine near Yellowknife. The mine was allowed to flood but the site was maintained in the event that silver prices recovered and the mine could be placed back into operation (Terra Mines Ltd. Annual Reports, 1984-1985).

Year:	Ore Milled:	Silver:	Copper:
1969	9,686 tons	74,096 oz	17 tons
1970	32,867 tons	393,054 oz	275 tons
1971	48,714 tons	1,193,396 oz	392 tons
1972	24,722 tons	1,917,922 oz	95 tons
1973	38,787 tons	1,394,180 oz	250 tons
1974	45,684 tons	1,093,919 oz	238 tons
1975	42,881 tons	1,320,724 oz	55 tons
1976	46,090 tons	1,900,991 oz	71 tons
1977	29,028 tons	1,040,984 oz	79 tons
1978	29,072 tons	663,697 oz	23 tons
1979	31,042 tons	346,680 oz	327 tons
1980	26,948 tons	294,846 oz	240 tons
1981	1,725 tons	56,321 oz	11 tons
1982	30,734 tons	677,033 oz	110 tons
1983	41,429 tons	866,635 oz	101 tons
1984	22,340 tons	785,098 oz	-
1985	5,880 tons	216,749 oz	-
<u>Total:</u>	<u>507,629 tons</u>	<u>14,236,325 oz</u>	<u>2,284 tons</u>

Table 1. Terra Mine production, 1969-1985. Does not include ores from the satellite mines of Norex and Smallwood Lakes. (sources: Terra Mining and Exploration Limited Annual Reports; Terra Mines Ltd. Annual Reports; Terra Mine production records; L.M. Manning & Associates Ltd., 1975)

Mine Production and Development Summary

The Terra Mine has been developed through the use of a decline to the 1,300-foot level with 13 primary levels and several sub levels (see **Figure 5**). Workings are focused in the central zone where the original veins and the Copper zone were developed. Crosscut drives extend to the west to explore other vein systems. The North deposit is accessed from the 6th and 7th levels through northerly crosscuts under the Camsell River; a raise apparently was driven to the surface somewhere in this area. A shaft was completed to the 600-foot level but was not put into use. Two other short

decline or adit portals have been blasted to recover small tonnages of ore near the surface. There has also been some open pit or stripping development to recover crown-pillar ore.

Terra Mine production (Silver Bear and North Mine) from 1969 to 1985 amounted to the milling of 507,560 tons of ore to produce 14,246,690 ounces of silver and 2,283 tons of copper (see **Table 1**). This figure does not include the milling of ores from the Norex or Smallwood Lake Mines.

Exploration Since Mine Closure

Terra Mines Limited conducted extensive geological studies after closure in an effort to assess the economics of the remaining reserves and future ore potential in the area. One report suggested that the potential for additional ore in the North zone was good, and more work was recommended to outline a tonnage (Hitchins, 1987). Ore reserves in all categories within the developed portions of Terra Mine in August 1985 were calculated as 6,043 tons grading 28 ounces per ton silver (total 169,200 ounces of silver), not including Norex or Smallwood Lake Mines (Nicholas, 1985). Terra Mines Limited sold the mine to Octan Resources Incorporated in 1988. Octan planned major developments and a restart of production through open pit methods, but no work was actually done. Tyhee Development Corporation Limited acquired the property in 2001, but let the claims lapse in 2005.

New claims have since been staked and in 2007 Cooper Minerals Incorporated acquired claims covering the Terra, Norex, Smallwood, and Northrim mines. Surface sampling and diamond drilling was conducted in 2007-2008. Cooper Minerals is hoping to identify a significant IOCG-type deposit in the Great Bear Lake region.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 086ENE0012



THOMPSON-LUNDMARK

Producer (Abandoned)

Gold

62° 36' 45" 113° 28' 30"

Years of Primary Development: 1938-1949

Mine Development: 2 inclined shafts, 1 vertical shaft, inclined winze; 650' vertical depth total

Years of Production: 1941-1943, 1947-1949

Mine Production: 133,989 tons milled = 70,339 oz Au, 13,782 oz Ag

Introduction

The Thompson-Lundmark Mine is located at Thompson Lake, 48 kilometers northeast of Yellowknife, NWT. This was one of the more famous gold mining operations of the Yellowknife area. The site was destroyed in the 1998 forest fires, the ruins of which were visited by the author in June 2000.

Brief History

The property was discovered at the tail end of the original gold rush at Yellowknife. Fred W. Thompson and Roy Lundmark, prospecting for Glyn Burge of the Thompson Prospecting Syndicate, touched down near Hidden Lake in July 1938 and staked 46 claims. The discovery was based on a hunch by Fred Thompson that gold could be found in the rock sediments of the area. Shaft sinking and underground lateral work commenced in 1939. The Thompson-Lundmark company was handicapped by a lack of funds, but a deal with Cominco Limited guaranteed the needed money to put the gold mine into production. The mine produced gold between 1941 and 1943. After a short shutdown period due to conditions of World War II, the mine reopened in 1947. Economic reserves were depleted in 1949 and the mine closed.

Geology and Ore Deposits¹

The deposit lies within the Archean Age, Burwash Formation of the Yellowknife Supergroup. The mine property is predominantly underlain by knotted quartz-mica schist of metasedimentary origin (greywacke-argillite turbidite). Some ore veins however are associated with unknotted schist. These strata are folded by at least two phases of Archean Age deformation, and metamorphosed to lower amphibolite facies in a 40 kilometer wide, north trending zone. Many plugs, stocks and plutons of the Prosperous Lake granite suite intrude the Burwash Formation west, northwest and southwest of the Thompson-Lundmark Mine. The deposit area is underlain by northwest-striking nodular (cordierite+/-andalusite) quartz-mica schists that dip 45 to 65° to the northeast. Most ore veins are interpreted to lie on the east limb of a broad anticline, the axial plane of which lies about 1000 feet southwest of the Kim shaft. The anticlinal axis trends northwest and dips steeply to the northeast. Two veins, the Trail and Lahti appear to lie near the crest of this anticline. The dominant schistosity sub-parallel bedding in the sediments and most of the ore veins are roughly conformable. Gold bearing quartz veins are grey to white in colour and contain very little other metallic minerals. Gold-barren pegmatite sills and dykes crosscut gold bearing veins. Both the gold bearing veins and the pegmatite dykes are considered to be related to the intrusion of the Prosperous Lake granite suite. A younger set of milky white coloured but gold barren quartz veins also cut the ore bearing veins. Some veins are displaced by faulting although the maximum displacement is only two feet.

Seven veins have been identified, three of which have been the focus of mining development. The Fraser vein strikes north and dips 45° east. It ranges from 6 inches to 5 feet wide on surface, averaging 2.5 feet. In underground workings, the best shoot 0.66 ounces per ton gold and measured 560 feet long by an average 1.5 feet wide, to a depth of 750 feet. Diamond drilling traced the vein an additional 300 feet down-dip, encountering some visible gold locally. The hanging wall rock of the Fraser vein is crushed and fractured, parallel to the vein wall in a sheeted zone one foot thick. The footwall is less intensely fractured in a narrower zone. The ore quartz is banded and streaky, and contains a little tourmaline, pyrite, galena and visible gold. The Kim vein is a zone of at least three parallel quartz veins. The zone strikes north and dips 45° east, and is 1800 feet long and 4 feet to 13 feet wide, averaging 6.6 feet wide. Each vein within the zone is up to 6 feet wide but usually less than one foot. The best ore shoot graded 0.38 ounces per ton gold and was 300 feet long throughout a 450 feet depth, in which the average aggregate width of veins was two feet. The Kim ore-bearing quartz is banded and carries tourmaline, biotite and sulphides. A two inch thickness of wallrock has been altered to a mixture of minerals including tourmaline and white mica.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

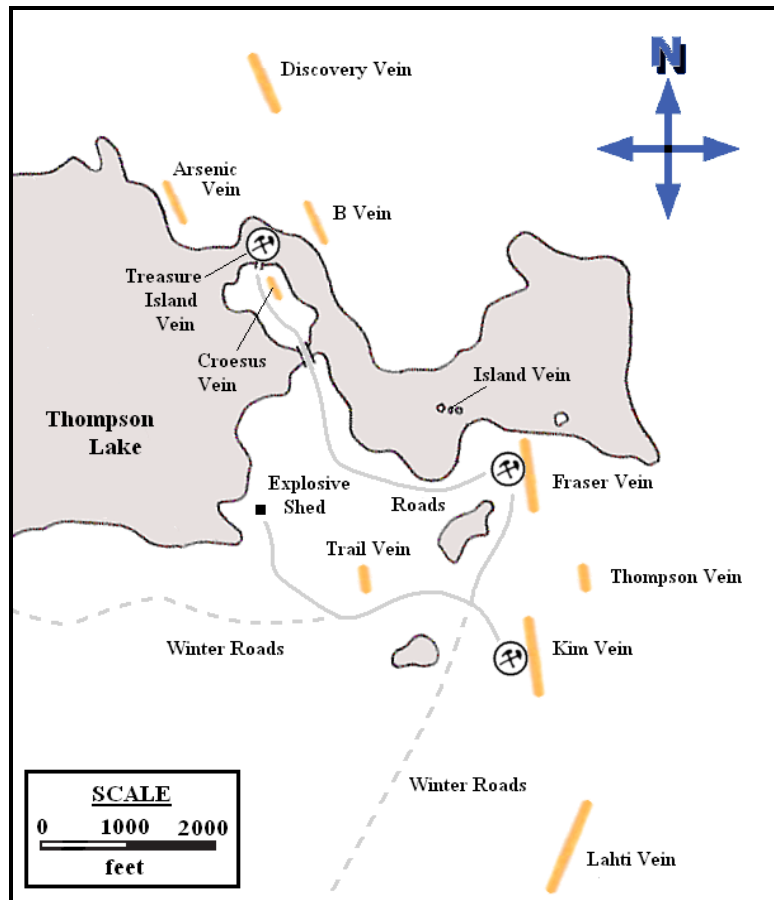


Figure 1. Thompson-Lundmark Mine area showing important gold veins.

The Treasure vein is a mass of quartz striking northwest and dipping moderately northeast, with offshoots into the enclosing schists. Some spectacular pockets of coarse gold at surface some surface mining, but subsequent exploration failed to extend the ore zone at depth.

The Island vein strikes north and dips 50° east, parallel to bedding at this locality. It has been traced for 300 feet along strike, and an ore shoot 152 feet long and 9 inches wide grades 1.46 ounces per ton gold. The shoot was not encountered at 200 feet depth in a drill hole undercutting the outcropping ore. The Island Vein carries about 2% sulphides, abundant tourmaline and visible gold, and some biotite. Wallrock is locally altered up to two centimetres from the vein, to a tourmaline-white mica assemblage.

The remaining veins are lower grade and/or tonnage showings. The B-vein and Lahti vein contain small amounts of gold. The Trail vein is said to contain three ore-grade shoots as indicated by trenching and one drill hole. An additional gold bearing quartz vein known as the Thompson vein was discovered in 1948 within 700 feet of the Kim shaft. It was traced for 150 feet on surface with visible gold over a 100 feet interval. It was not developed. The Arsenic and Croesus veins were trenched but contains little gold. A pegmatite sill named the Waco was found to contain considerable tantalum-columbium minerals.

Thompson-Lundmark Gold Mines Limited (1939-1940)

During the summer and fall of 1938, a massive program of prospecting, diamond drilling, and trenching was conducted on the Thompson-Lundmark property. This work uncovered a number of gold-bearing quartz outcrops, of which the Kim vein and the Treasure Island deposit were regarded as the most important. The original syndicate that staked the claims formed a new company to develop the new gold property – Thompson-Lundmark Gold Mines Limited. A winter tractor road was cleared from Yellowknife via the Jennejohn Lake route in the winter of 1938-1939, and a mining plant was purchased from the closed Camlaren Mine (Lord, 1941). The Kim vein was the target of first underground exploration. By year-end 1939, an inclined (50° east) shaft was put down to an inclined depth of 325 feet with two levels at 150- and 300-foot inclined depths. Levels at Thompson-Lundmark Mine were recorded as inclined

depths from the shaft collar. Vertical depths below surface for the underground workings at the mine are listed in **Table 2** (Thompson-Lundmark Gold Mines Ltd. Annual Report, 1939).

Kim Mining Plant

Equipment used to sink the Kim shaft included a Gardner-Denver air compressor of 325 cubic feet per minute driven by Cat diesel engine and a small steam hoist. Ken Muir, formerly mine manager at Camlaren, was in charge of operations. E.V. Neelands was consulting engineer. A tent camp was erected on the shores of Thompson Lake, about a kilometer west of the Kim vein. A rough road connected this camp with the Kim shaft site. Buildings at the shaft site included a large timber-pole headframe, hoist-room/powerhouse, and a mine office/dry (Lord, 1941).

Discovery of Fraser Vein

The Kim shaft was the focus of development until the summer of 1939 when geologist Hugh Fraser found another interesting gold showing on the property - the so-named Fraser vein. By now, nine separate gold veins had been found on the property and the company hired more geologists to work on the mapping and development of the mine (Thompson-Lundmark Gold Mines Ltd. Annual Report, 1939). Work was affected by a lack of investment in the company, due primarily to uncertainty in international affairs. The outbreak of war in September 1939 further hampered development, but adequate funding was acquired from Ventures Limited (Hoffman, 1947).

Fraser Shaft

The mine plant was moved from the Kim shaft site to its new location at the Fraser vein during the fall of 1939 and shaft sinking begun. The inclined Fraser shaft followed the dip of the vein lying on the footwall side. At year-end 1939, the shaft was completed to an inclined (48° east) depth of 322 feet with two levels at 150- and 300-foot inclined depths. No raising was accomplished (Thompson-Lundmark Gold Mines Ltd. Annual Report, 1939).

	Kim Vein:	Fraser Vein:	Other:
<i>Shafts:</i>	325'	322'	-
<i>Drifts:</i>	1,202'	712'	-
<i>Crosscuts:</i>	58'	49'	-
<i>Shaft Stations:</i>	41'	43'	-
<i>Diamond Drilling:</i>	3,339'	1,557'	2,992'
<i>Trenching:</i>	359 cu yards	61 cu yards	569 cu yards

Table 1. Mine development at December 31st 1939. (source: Thompson-Lundmark Gold Mines Ltd. Annual Report, 1939)

Cominco Limited (1940-1943)

Going into 1940, Thompson-Lundmark Gold Mines was unable to raise additional capital. The operation was saved when Cominco Limited agreed to assume management of the mine and provide the funds necessary to bring it into production. Under this deal, Cominco supplied the first loan of CDN \$350,000. If more funds were to be required, Cominco and Ventures Limited would equally loan \$100,000, and if further loans were again required Cominco would supply them and Ventures would have the option to do so. Thompson-Lundmark Gold Mines Limited would work towards repaying these loans, and management of the gold mine would remain under Cominco control until three years after repayment of the loans. Cominco assumed management as of July 1st 1940. Bob Armstrong was brought to the site to assume management of the operation for Cominco. 30 men were reported employed in October 1940 under Armstrong's direction, including Ed Jewell, mine captain; Howard Barker, engineer; and Eric Caldicott, accountant (The Northern Miner, Oct. 10th 1940; Thompson-Lundmark Gold Mines Ltd. Annual Report, 1941).

The following work was accomplished during 1940-1941: a transmission line was erected from the mine to the Bluefish Lake hydropower-plant, west of the property; a new mining plant was erected; a townsite was built at the Fraser shaft; a Hadsell milling unit was installed; and the Fraser shaft was completed to 834 feet slope depth. New levels were established at 450-, 600-, and 750-foot inclined depths. A considerable amount of development was completed on the Fraser vein in preparation for production. From July 1st 1940, when Cominco assumed management,

to May 1st 1941, 419 feet of shaft sinking, 1,028 feet of drifting, 100 feet of crosscutting, and 213 feet of raising was accomplished. No work was done on the Kim vein. Ore reserves at January 1st 1941 were 67,000 tons averaging 0.56 ounces per ton gold, presumably in both the Kim and Fraser veins (The Toronto Star, July 16th 1941).

Production Begins

Gold production began in August 10th 1941 treating ore from the Fraser workings, and the first gold was poured on September 20th 1941. Early in 1942, the grade of ore being milled was 0.64 ounces per ton gold with mill recoveries of 98.1%.

Power and Hoisting Plant

The Fraser shaft was equipped with a 54 foot timber headframe to support the inclined shaft. The hoist was a Canadian Ingersoll-Rand 8x6 air hoist. Air was supplied by two 670 cubic feet per minute electrically driven Canadian Ingersoll-Rand air compressors. The entire camp and plant was supplied with hydropower, but a 75 KVA Crocker-Wheeler generator and Cat D-13,000 diesel engine set provided back-up power. Heat was supplied by two wood-fired boilers to provide a total output of 75 horsepower (Feniak, 1948; Wilson, 1949; Lord, 1951).



NWT Mining Heritage Society – Ted Hunter collection

Figure 2. *Thompson-Lundmark Mine, 1948. Looking southwest from Thompson Lake.*

Mining Operations

Stopes were excavated using shrinkage methods with ore being hand collected and pushed by mine cars to the loading pockets. All production was focused on the Fraser vein workings during this period.

Milling Operations

Mine ore was worked through a 10 inch grizzly underground before being hoisted. No separate crushing plant was employed in the mill circuit, with ore being reduced to 57% minus 200 mesh in the primary 4 foot x 12 foot Hadsell dry mill unit. The mill operated by using the ore itself as a crushing medium. To help reduce the natural moisture content of the ore, air in the mill was heated by oil burner at 2,000 cubic feet per minute. The ground product was exhausted from the mill and run through two wet-cyclone dust collectors, where gold-bearing dust was washed in a Wallis agitator, and fine dust exhausted from the milling plant through a stack. Gold was then recovered in a standard circuit of cyanidation and amalgamation, consisting of three blanket tables, amalgamation, two 18 foot x 12 foot Dorr thickeners, two 22 foot x 18 foot agitators, an 8 foot x 8 foot Oliver filter, and a Merrill-Crowe sock precipitation unit. Mill tailings were deposited south of the mine into a small pond known as Chum Lake. Approximately 45% of the gold and silver were caught on the blanket tables and amalgamation circuit. Every second week, the amalgam sponge was retorted and refined and bricks were produced grading 82% gold and 15% silver average. Bricks poured from the cyanidation and precipitation circuit graded 78% gold and 15% silver. During the operational period of 1941-1943, mill heads were 0.66 ounces per ton gold with recoveries of 98.3%. Average daily tonnage was 96 tons (Wilson, 1949; Lord, 1951).

Kim Shaft Sinking

The Kim shaft was completed to 652 feet incline depth in March 1942. New levels were established at 450- and 600-foot inclined depths (The Toronto Star, Apr. 25th 1942). Drifting to the north on the 450-foot level opened up two new ore shoots, the first of which with a length of 174 feet averaging one foot wide and the second 44 feet long with widths of 1½ feet (The Toronto Star, Aug. 29th 1942). An ore stockpile of over 7,000 tons was built up on surface at the Kim shaft but no production came from these workings at this time. A drive from to 750-foot level of the Fraser area south towards the Kim shaft was begun early in 1943, but it had to be abandoned due to labour shortages (The Toronto Star, May 17th 1943). The Kim drive had advanced 350 feet when work stopped, leaving 2,150 feet to be completed (Wilson, 1949). Ore reserves at January 1st 1942 were 63,630 tons averaging 0.50 ounces per ton gold (The Toronto Star, Feb. 5th 1942). Ore reserves at June 30th 1942 were 45,000 tons of unknown grade. At that date it was recognized that considerable exploration and development would be required to add to this tonnage (Thompson-Lundmark Gold Mines Ltd. Annual Report, 1942).

Treasure Island

A small amount of ore (35 tons) was mined by open cut on the Treasure vein in 1942 and processed, showing a grade of 0.72 ounces per ton gold. Further development was planned (The Northern Miner, Sept. 11th 1947).

The Thompson-Lundmark operation was greatly hampered by the war during 1943, causing labour and supply shortages. To add to the problems a night fire destroyed the only bunkhouse in July 1943, causing concern of whether it was economic to continue operations. Supplies required to rebuild could not be guaranteed before winter arrived. Meanwhile, temporary tents and prefabricated huts provided short-term accommodation solutions.

Kim Vein:	Inclined Depth below Shaft Collar:	Vertical Depth below Surface:	Fraser Vein:	Inclined Depth below Shaft Collar:	Vertical Depth below Surface:
<i>Shaft bottom</i>	652'	500'	<i>Shaft bottom</i>	834'	600'
<i>1st Level</i>	150'	110'	<i>1st Level</i>	150'	110'
<i>2nd Level</i>	300'	230'	<i>2nd Level</i>	300'	220'
<i>3rd Level</i>	450'	345'	<i>3rd Level</i>	450'	320'
<i>4th Level</i>	600'	455'	<i>4th Level</i>	600'	430'
<i>Kim Haul</i>	750'	540'	<i>5th Level</i>	750'	540'
			<i>Winze bottom</i>	950'	650'

Table 2. Summary of elevations of the underground workings.

In August 1943, Cominco recommended to the directors of Thompson-Lundmark Gold Mines Limited that the mine be shutdown and put on caretaker status. Hoisting of ore was completed on September 18th 1943 and milling stopped September 20th. Lateral work underground at both shafts consisted of 5,997 feet of development in September 1943 (Meikle, 1943). Accumulative mine production at this time (1941-1943) was 73,235 tons milled to produce 47,632 ounces of gold and 9,258 ounces of silver, poured in the form of 83 gold bricks (Lord, 1951). Total reserves at 1943 shutdown were estimated as 62,586 tons grading 0.40 ounces per ton gold. The Fraser vein was credited with 9,560 tons grading 0.51 ounces per ton gold: 5,000 tons (0.49 ounces per ton) remaining to be stoped on the 750-foot level and 4,500 tons (0.53 ounces per ton) of upper level pillars. The Kim vein was credited with the remaining 53,000 tons grading 0.38 ounces per ton gold (The Northern Miner, March 20th 1947; Sept. 11th 1947; Lord, 1951).

Thompson-Lundmark Gold Mines Limited (1946-1949)

A reported operating profit of \$786,000 allowed the Thompson-Lundmark company to repay the loans supplied by Cominco in July 1940; therefore, operating control of the gold property reverted back to the original owners in July 1946. With the end of the war, conditions were perfect to reopen the mine (Hoffman, 1947). Operations begun in July 1946 to place the site back into production. Diamond drilling during the fall of 1946 totaled 1,604 feet, some of which was conducted on the Island and Trail veins but most of which was conducted on the Fraser and Kim workings (The

Northern Miner, Jan. 9th 1947). Additional drilling early in 1947 disclosed interesting values in the northern sections of the Fraser vein extending under Thompson Lake, and it was planned to extend the drifts on the 2nd and 3rd levels into this area. Underground development resumed in March 1947 (The Northern Miner, Mar. 20th 1947). At this point, it was realized that a new source of ore would be needed to re-start production. Known reserves in the Fraser vein were basically exhausted, so attention was focused on the Kim vein. To mine the Kim vein, the haulage-way was completed a distance of 2,500 feet between the two shafts on the 750-foot level. Raises connected the haulage-way to the 650-foot level of the Kim shaft. With this mine plan, all ore could be hoisted via the operational Fraser shaft with the Kim shaft being used for ventilation. The Kim haulage-way was developed using a Sullivan Jumbo drill and Eimco 12-B mucking machine. The working crew consisted of four men, with an advance of six feet per shift with a monthly average of 400 feet per month. It had a width of seven feet (Wilson, 1949).

Due to the stress on the air compressors by the Fraser hoist, most development in 1947 was on the Kim haulage-way drive. A new Canadian Ingersoll-Rand 42 inch x 30 inch 2-drum electric hoist arrived at the mine in 1947 and the transmission line to the Bluefish hydropower plant was reactivated. Additional work completed during 1946 included the erection of a new 55 man bunkhouse, replacing the one destroyed by fire in 1943. Other accommodation consisted of a 16-man Quonset Hut, a 100-man cookery, staff quarters for 10 single men, three houses, one duplex apartment, as well as eight winterized tent frames. Recreation was provided by an indoor curling rink and a small game room. Roads totaling three kilometers were cleared between the Fraser, Kim, and Treasure veins (Wilson, 1949; Lord, 1951).

Production Resumes

Milling resumed at Thompson-Lundmark on August 28th 1947 and the first gold was poured on September 20th 1947. Initial mill feed was from the 750-foot level in the Fraser workings and a small 7,359 ton surface stockpile of ore from earlier Kim development. Pillar ore (4,500 tons) in the upper levels of the Fraser vein were not viewed as recoverable at this time (The Northern Miner, Sept. 11th 1947; Nov. 30th 1947). On October 10th 1947, the Kim haulage-way was completed and two short raises were blasted to reach the 600-foot level of the Kim workings. Owing to a large build-up of ice in the Kim shaft, un-watering was not completed until November 24th 1947 (The Northern Miner, Dec. 11th 1947). Known ore in the Fraser vein was depleted in April 1948 and the Kim vein became the sole source of ore.

Mining Operations

Ore was mined by shrinkage stoping methods through the use of slusher hoists and box-holes, so that all ore would drop directly into ore cars with no mucking required. Mining and stope blasting was conducted during the day shift. Ore was hauled from the Kim vein to the Fraser shaft with a Mancha "Little Trammer" locomotive and six 30 cubic foot side-dump ore cars. Ore hauling was conducted during the night shift (Wilson, 1949; Lord, 1951).

Production Improvements

Late in 1947 the following changes were made to the operation to allow for efficiency: mill operating crew was reduced from 12 men to 7 men; the hoisting capacity was increased to 300 tons per day; ore pocket loading chutes were increased to 100 tons per day, up from 40 tons per day; and pumping equipment capable of 200 gallons per minute were installed (The Northern Miner, Dec. 11th 1947). In an attempt to increase mill tonnage, steel balls were added to the Hadsell mill unit starting in 1947. Usually, the Hadsell mill operated using the ore itself as a grinding medium, but the use of mill balls did indeed aid in increased milling capacity. The average daily mill production figures during 1947-1949 were as follows: tons hoisted, 108 tons; tons waste discarded, 5½ tons; tons milled, 102½ tons; mill heads, 0.37 ounces per ton gold; tailing heads, 0.01 ounces per ton gold; recovery, 97% (Wilson, 1949).

Employees

The mine was managed by Del R. Wilson during these years. Other staff included (in 1947) J.E. Rae, mine engineer; R.Cecil Evans, mill superintendent; C. Anderson, mine captain; F. Cahill, accountant; Vincent Aural, surface superintendent; and Robert D. Hoffman, consulting engineer. The average amount of employees per day was 84 with 100 persons being the maximum at any time. In 1947 there were 87 names on the payroll; 6 were staff, 48 were employed on the surface and in the mill, and 33 were underground workers (The Western Miner, Nov. 1947).

Approximately 1,600 feet of diamond drilling was completed on the large property during 1947-1948 in an attempt to locate additional ore in the known vein systems. Some of this was done on the 750-foot level of the Fraser vein to target its depth potential. Drilling proved that the Kim and Fraser orebodies continued to a vertical depth of at least 1,200 feet, and also established that the Fraser vein extends to the north under Thompson Lake. A new vein was discovered in the summer of 1948 by Fred W. Thompson, located between the Fraser and Kim veins - the Thompson vein. It was exposed a distance of 150 feet and averages one foot in width. No known work was completed on this exposure during this period (Feniak, 1948).

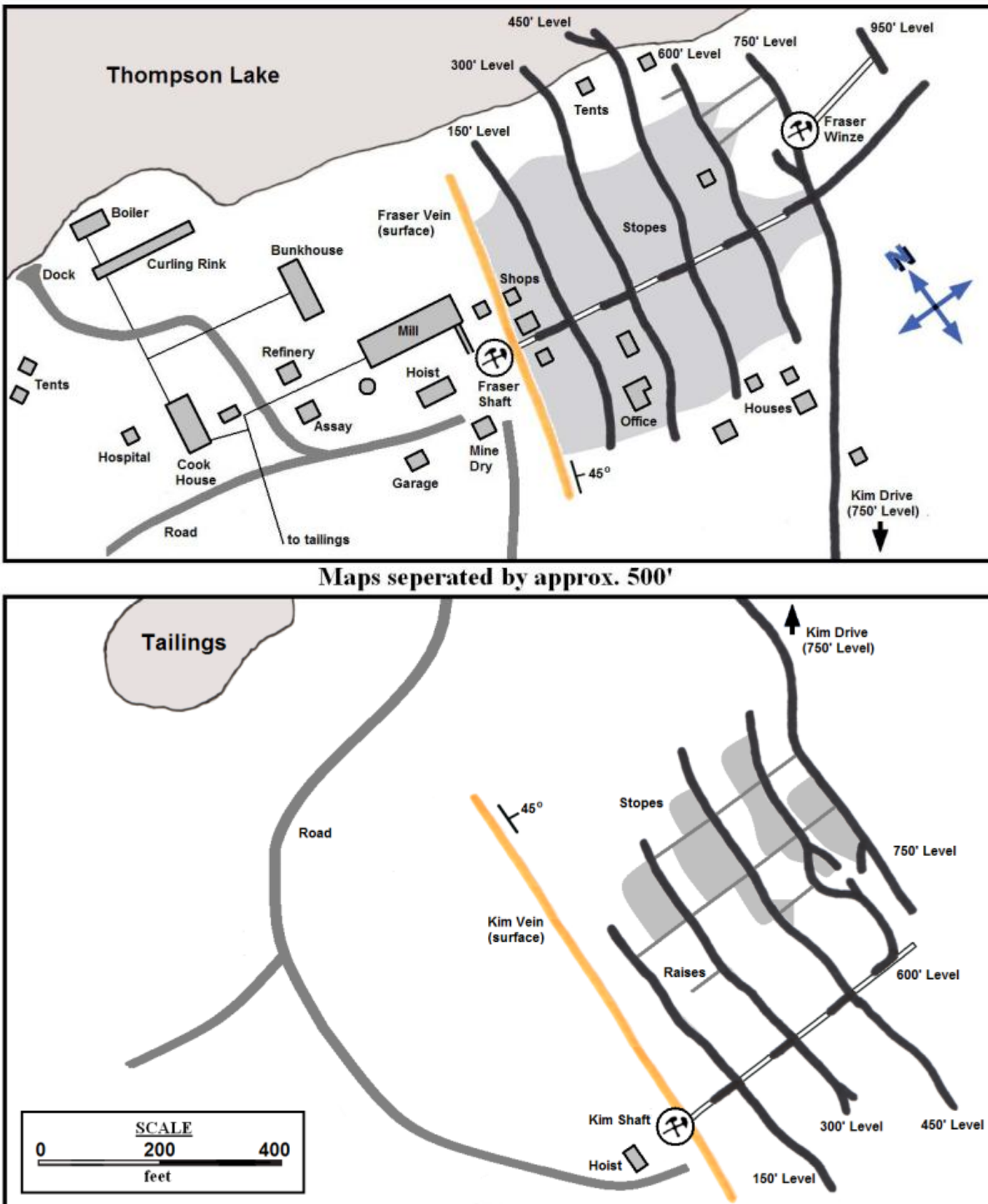


Figure 3. Surface plan and underground workings within the Fraser and Kim veins, c.1949.

Treasure Island Shaft

Two major exploration programs were completed during 1947-1948 at Thompson-Lundmark Mine, both involving underground development. The first was a shaft-sinking program on the Treasure vein, located on Treasure Island near the original gold discovery in 1938. Diamond drilling results warranted only a minimum of expenditure for this vein; therefore only 143 feet of vertical shaft was sunk. This work was completed using makeshift equipment including a Joy-Sullivan 210 cubic feet per minute air compressor modified to be operated by the engine off a Cat D-4 tractor. A single-drum 6x5 air hoist was also rounded up for the job, fitted with 10 cubic foot buckets (Wilson, 1949).

On the 100-foot level of the Treasure Island shaft, 165 feet of drifting was completed. Not enough ore was found to justify production or any further expenditure at this vein. No known mill feed came from the Treasure Island shaft (The News of the North, Oct. 28th 1949).

Fraser Winze Sunk

Elsewhere, at the deepest level of the mine, it was decided to sink an inclined winze to explore the deeper regions of the Fraser vein. A single-drum electric hoist, 24 inch x 18 inch, was installed on the 600-foot level, and a raise connected the 750-foot to the 600-foot levels to serve as a hoisting compartment. Sinking then proceeded for 226 feet length beyond the 750-foot level to open a new level at 950 feet incline depth, the deepest in the mine at a vertical depth of 650 feet (Wilson, 1949).

Kim Hanging Wall Vein

Another exploration target during 1948 was the Kim Hanging Wall vein, a parallel structure to the Kim vein located 80 feet east. Diamond drilling indicated good values. It was planned to start a crosscut on the 450-foot level to develop this structure.

Late in 1948 it appeared as though economic ore reserves would soon be exhausted. The Fraser vein was mined out to the 750-foot level and exploration had failed to locate economic ore below that horizon. The Treasure vein also proved to be less than important. The Kim vein was the sole source of ore, and even here the future looked bleak. Remnant mining of the 150-foot level of the Kim vein was undertaken in 1948-1949. This work had been delayed due to a great build-up of ice from previous flooding that needed be excavated before mining could begin. The company acquired assistance funding under the mandate of the Emergency Gold Mining Assistance Act, the government program that alleviated the financial burden placed on gold mines in the expensive post-war years.

Development of other vein structures on the property would prove to be an expensive proposition. Additional underground development was ruled out due to the huge costs in mobilization and manpower. Power allocations for the mine, although adequate for the present requirements, would not permit expanded operations, as the pumping needed to keep the underground workings dry would be too great a burden on the power situation (Feniak, 1948).

Final Closure

The company was slowly exhausting its treasury, and without the price of gold increasing or the high costs of operation being lowered, the Thompson-Lundmark Mine would not continue to operate. In March, a 10,000 foot diamond-drilling program was started to probe the areas of the Kim, Fraser and Treasure Island veins (The Northern Miner, Mar. 31st 1949). Mining operations ceased with depletion of known ore on April 13th 1949, and milling stopped in May 1949 (The News of the North, Oct. 28th 1949). Gold brick #122 was poured. All equipment except the milling and boiler plant was removed and sold, and the property was put on care and maintenance (The Northern Miner, Oct. 26th 1950).

Year:	Ore Milled:	Gold:	Silver:
1941	11,915 tons	8,231 oz	1,598 oz
1942	37,775 tons	22,587 oz	4,373 oz
1943	23,545 tons	16,814 oz	3,287 oz
1947	11,309 tons	3,062 oz	652 oz
1948	37,757 tons	14,653 oz	2,904 oz
1949	11,688 tons	4,992 oz	968 oz
<i>Total:</i>	<i>133,989 tons</i>	<i>70,339 oz</i>	<i>13,782 oz</i>

Table 3. Thompson-Lundmark Mine production, 1941-1943, 1947-1949. (source: Lord, 1951)

Operations Summary

The Kim inclined shaft, 652 feet long, is sunk 500 feet vertically below the surface. The Fraser inclined shaft, 834 feet long, is sunk 600 feet vertically below the surface; but including the 226 foot long Fraser inclined winze, the mine has a vertical depth of 650 feet. The Treasure Island shaft was sunk vertically 143 feet with one level and limited underground exploration. Production between 1941 and 1949 amounted to 133,969 tons milled to produce 70,339 ounces of gold and 13,782 ounces of silver. (see **Table 3**)

Exploration Since Mine Closure

A report on the property in 1978 by the Thompson-Lundmark company suggested ore reserves of value in stope pillars (Kim vein: 15,000 tons grading 0.60 ounces per ton gold) and a probable surface reserve on the trail vein, 230 feet long, 6 inches wide, and with grades of 0.71 ounces per ton gold. It was recognized that recovery of stope pillars would not yield significant tonnages to make the effort worthwhile. Other veins were considered too low grade to justify further development. (H.E. Neal & Associates Ltd., 1978) A 1980 report states an estimated 45,000 tons of ore down to the 300-foot level of the Kim vein. Another source dated 1982 reports a proven 72,000 tons grading 0.20 ounces per ton gold down to the 300-foot level in the Kim vein, not including pillars. Drilling on the Treasure Island indicated an ore shoot 100 feet long to a depth of 30 feet. (Gates, 1980)

Ardic Exploration and Development Ltd. optioned the claims from Thompson-Lundmark Gold Mines Limited in 1983. An exploration program was conducted consisting of detailed geological mapping, surveys, and diamond drilling. A crown pillar reserve in the Kim vein was calculated consisting of 14,880 tons grading 0.20 ounces per ton gold from surface to the 150-foot level. 6 tons of bulk samples and chip samples were sent to Lakefield Research in 1983. (The Northern Miner, Jan 20th 1983; Nov. 3rd 1983) In 1987, Ardic continued exploration and conducted a 21-hole (9,043 feet) drilling campaign on the Kim and Trail veins. Work on the Kim vein tested the extensions of the deposit over a strike length of 2,800 feet and a depth of 440 feet, delineating at least two potential ore shoots. This new information, plus known reserves from previous exploration, suggested a mineral inventory of 190,000 tons of ore grading 0.30 ounces per ton gold within the Kim vein. Many of the drill holes from both the Kim and Trail veins revealed visible gold. (The Northern Miner, Mar. 28th 1988)

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085INW0037



THOR LAKE

Bulk Sampled (Non-Operational)

Beryllium

62° 07' 00" 112° 36' 00"

Years of Primary Development: 1985-1986

Mine Development: 1,600' decline to 250' depth

Years of Bulk Sampling: 1984-1986

Bulk Sample: 925 tons = 31% BeO

Introduction

Thor Lake is a prospective beryllium mine located 105 kilometers southeast of Yellowknife, NWT and five kilometers north of Hearne Channel on Great Slave Lake. It has been described as one of the richest beryllium deposits in the world, but production cannot begin until the mine is passed through the regulatory stage. A seven kilometer road connects the mine to Great Slave Lake.

History in Brief

First claims were staked in 1970 when the target was uranium. In 1976, Highwood Resources Limited discovered occurrences of uranium, tantalum, and columbium at Thor Lake and staked a large group of claims covering what is known as the Lake zone. Further exploration uncovered high amounts of valuable beryllium deposits within the T-zone. Diamond drilling during 1983 disclosed the significant resources within the T-zone, and underground exploration was conducted in 1985. A bulk sample was stockpiled and some minor testing has been completed. In 1999, plans were announced that called for a 100,000 tonne bulk sample to be processed at a pilot mill in Hay River. This operation was not approved by government regulators and production planning has been put on hold. Exploration and metallurgical testing, together with feasibility studies, continues however, in response to record high beryllium prices in recent years. In 2002, Highwood Resources Limited was reformed into Beta Minerals Incorporated; in 2005, Beta Minerals sold the Thor Lake property to Avalon Ventures Incorporated.

Geology and Ore Deposits¹

The Thor Lake deposits are situated in the alkaline Blatchford Lake Plutonic Complex which intrudes Archean plutonic rocks and Archean metasediments of the Yellowknife Supergroup at the southern margin of the Slave Province. Mineralization is developed in the central core of the complex which is referred to as the Thor Lake Syenite, a 30 square km oval in the centre of the Grace Lake Granite pluton. Six varieties of syenite including a pegmatitic phase have been identified on the basis of textural and compositional differences. In the vicinity of the mineral deposits, the syenite consists of a massive medium to coarse grained assemblage of K-feldspar with interstitial sodic amphibole, magnetite and minor quartz.

An area of late stage veining, alteration and mineralization is centered on Thor Lake in the western part of the Thor Lake syenite. This area contains five zones of niobium, tantalum and rare-earth elements enrichment along with high concentrations of zirconium, gallium, beryllium, fluorine and locally yttrium, thorium and uranium. Only the Lake and T zones are of potential economic interest. The Lake Zone is a 2 square kilometer, triangular-shaped area of dark, altered and brecciated rock beneath and south of Thor Lake. It is the largest of the five mineralized zones and contains niobium, tantalum, rare-earth elements, yttrium and zirconium. Two main alteration types are present: 1) an albitite and K-feldspar-rich syenitic pegmatite resembling a chaotic breccia that has been intensely metasomatized; and 2) a diverse suite of rocks rich in mafic minerals such as aegirine, biotite, iron and titanium oxides, albite, K-feldspar and quartz. Accessory minerals include zircon, fluorite, allanite, ferro-columbite and members of the bastnaesite-group. The mafic rocks are also brecciated but alteration is more intense. A body of nepheline syenite including minerals such as sodalite, pectolite, analcite, catapleite, eudialyte, andradite, willemite, mesolite and natrolite has been identified below the Lake Zone. It is in part altered, but is not enriched in incompatible elements and is believed to be a late stage intrusion. Twenty-nine drill holes in the Lake Zone have outlined a resource of 65 million tonnes grading 0.03% tantalum, 0.4% niobium, 1.7% combined rare-earth and 3.5% zirconium.

The T Zone trends north-northwest away from the Lake Zone for approximately 1 kilometer, is up to 275 meters in width and extends to a depth of 150 meters. It straddles the Grace Lake Granite-Thor Lake Syenite contact and offsets it. Near the Lake Zone, it is dyke-like and is known as the South T Zone. The northern end, known as the North T Zone, is a small circular body. Some 15 lithologies arranged in concentric to sub-horizontal shells have been

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

recognized in the T Zone and have been grouped into four zones. The T Zone is of particular interest for its beryllium content. The outermost Wall zone is a pink to buff weathering, massive unit of variable texture dominated by relict microcline, albite and minor quartz. Columbite is a common accessory mineral and high levels of gallium are associated with feldspars in which it substitutes for trivalent aluminum.

Moving in, the Lower Intermediate zone includes five lithologies consisting of various proportions of quartz, biotite, feldspar, chlorite and magnetite. Beryllium is present in the mineral phenacite. Yttrium and rare-earth elements are in an intimate admixture of a metamict species believed to be thorite. Niobium is in the mineral columbite, which occurs as massive, granular and radiating spicular aggregates. Other minerals include purple fluorite and dolomite. The Upper Intermediate zone is transitional between the Lower Intermediate and central Quartz zones and is similar in gross mineralogy to the former. It differs in that polyolithionite and phenacite are particularly abundant, making this zone the most important host for beryllium. Polyolithionite is of interest for its lithium and rubidium content. Other beryllium minerals include bertrandite, gadolinite and helvite. Yttrium and rare-earth elements are present in bastnaesite-group minerals, thorite and thorium- yttrium silicates.

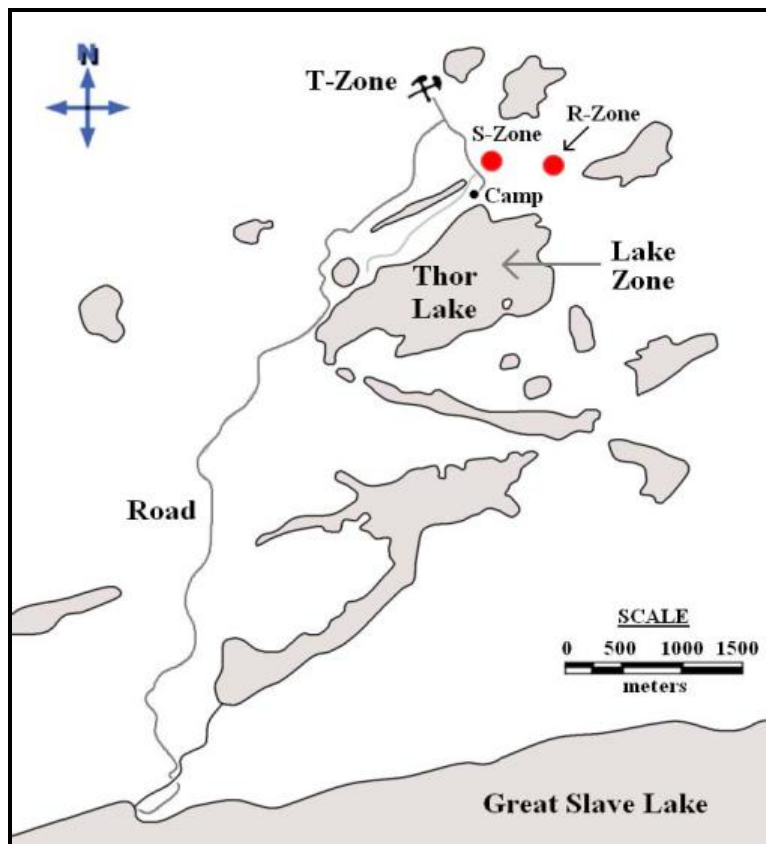


Figure 1. Thor Lake area and ore deposits.

The Quartz zone occupies the core of the North T Zone and is present in patches within the South T Zone. It is essentially monomineralic, consisting of white and greyish translucent quartz. A total of 124 drill holes in the T Zone has outlined 1.6 million tonnes grading 0.85% beryllium oxides (BeO), including 435,000 tonnes of 1.4% beryllium oxides, 0.26% yttrium oxides (Y_2O_3), plus neodymium, samarium, gadolinium, and gallium.

The Fluorite zone is located at the southeastern end of the Lake Zone. It is approximately 150 meters in length, up to 15 meters wide and consists of a dense, dark-brown siliceous rock with occasional pods of fluorite. Accessory minerals include zircon, xenotime, allanite and members of the bastnaesite-group.

The R zone is a series of pegmatitic lenses and patchy zones of albitization in foliated syenite. It is about 4 kilometers long, up to 30 meters wide and is locally enriched in beryllium, thorium, niobium and rare-earth elements. The S zone is subparallel to the R zone and lies about 200 meters north of it. It is 300 meters long, averages 10 meters in width

and consists of syenite altered to albite and albite-polyolithionite. Patchy enrichments of beryllium, uranium, rare-earth elements, yttrium and particularly niobium in ferro-columbite are present.

Highwood Resources Limited (1985)

An exploration program conducted under the engineering management of Strathcona Mineral Services Limited was conducted on the property between 1983 and 1987. This work was concentrated in the T-zone area, where significant deposits of beryllium and other rare-earth minerals had been identified. By the spring of 1984, over 100 drill holes in the T-zone had outlined two beryllium deposits, one of 479,000 tons grading 1.4% beryllium oxide and another of 1,300,000 tons grading 0.66% beryllium oxide. Extensive blasting and assaying of the mineralized zones was also completed during the year, and more high-grade samples were shipped for testing (25 tons). An underground exploration program on the T-zone was authorized late in the year. Early in 1985, Highwood Resources purchased the remaining 30% of the claim rights and become the sole owner of the Thor Lake project. A 120-kilometer ice road on Great Slave Lake from Yellowknife to Thor Lake was built early in 1985, and 500 tons of equipment and supplies were brought to the site in preparation for underground development. A 30 person trailer camp was erected and complete mine services were installed (Highwood Resources Ltd. Annual Report, 1984).

Mine Development 1985

From May to August 1985, a decline was driven a length of 1,600 feet to a depth of 250 feet within the North T-zone. Strathcona Mineral Services Limited provided engineering and management services, while the underground development was sub-contracted to Canadian Mine Development Limited. The purpose of the underground program was to provide a bulk sample of the various sub-zones for metallurgical testing, but also to verify and compare assay results with drilling results. About 10,000 tons of material was mined and stockpiled. During the 1985 season, 150 tons of ore was crushed and split out by passing the rock through a sample-tower to achieve a representative sample for metallurgical testing. This ore was then transported by helicopter to Hay River, thence by truck to Lakeview Research Limited in Ontario (Highwood Resources Ltd. Annual Report, 1985).

1986 Operations

Work during 1986 focused on crushing and splitting the remaining ore into bulk samples. A 12-kilometer road was built from Great Slave Lake to the mine site to accommodate future operations. About 750 tons of ore was shipped by winter road early in the year to Hay River, and then shipped by rail to Lakeview Research Limited. A pilot plant was constructed in Ontario and began treating Thor Lake ores in April 1986. Assays of up to 31% beryllium and 60% rare-earth were reported, with 90-95% recoveries. Highwood Resources finalized an agreement with Hecla Mining Corporation in August 1986 whereby Hecla would conduct and finance exploration of Thor Lake and, pending favorable results, would bring the mine into commercial production. A formal marketing study was initiated in 1986 to study the feasibility of selling Thor Lake ores on the market. Environmental studies were also started to measure the impact of a mine in the Thor Lake area (Highwood Resources Ltd. Annual Report, 1986).

Surface exploration through diamond drilling continued throughout this period, and in 1988 reserves were reported as follows: (see **Table 1**)

Zone:	Tonnage:	Grades:
<i>North T Zone</i>	510,000 tons	1.11% Beryllium Oxides, 0.17% Yttrium Oxides, 0.28% Rare Earth Elements, 0.58% Niobium Oxides
<i>North T Zone</i>	60,000 tons	8.24% Rare Earth Elements
<i>South T Zone</i>	1,250,000 tons	0.62% Beryllium Oxides, 0.10% Yttrium Oxides, 0.20% Rare Earth Elements, 0.46% Niobium Oxides
<i>Lake Zone</i>	63,000,000 tons	0.03% Tantalum Oxides, 1.70% Rare Earth Elements, 0.40% Niobium Oxides

Table 1. Thor Lake ore reserves, 1988. (source: Highwood Resources Ltd. Annual Report, 1988)

The project was put on temporary hold in 1990 following the withdrawal of Hecla Mining Company as a joint-venture partner. Hecla dropped out of the project due to economic concerns and fears that they would be unable to secure commitments from potential customers of the ores (Highwood Resources Ltd. Annual Report, 1989).

Exploration Since Mine Closure

Highwood Resources Limited continued to evaluate and explore the property throughout the 1990s. In 1999, plans were announced that called for a 100,000 tonne bulk sample to be processed at a pilot mill in Hay River. This operation was not approved by government regulators and production planning has been put on hold. Exploration and metallurgical testing, together with feasibility studies, continues however, in response to record high beryllium prices in recent years. In 2002, Highwood Resources Limited was reformed into Beta Minerals Incorporated; in 2005, Beta Minerals sold the Thor Lake property to Avalon Ventures Incorporated.

Avalon has performed considerable economic assessment of the Thor Lake project with most focus on the Lake zone. In 2007, 16 holes totaling 2,551 meters were drilled confirming the presence of at least two mineralized alteration zones within the tabular Lake zone deposit. (Avalon Ventures Ltd. Annual Reports 2005-2007) Further exploration and completion of a feasibility study is planned.

References and Recommended Reading

Avalon Ventures Ltd. Annual Reports. 2005-2007.

Highwood Resources Ltd. Annual Reports. 1978-1989.

geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085ISE0006



TREACY

Minor Producer (Abandoned)

Gold

62° 56' 30" 113° 20' 15"

Years of Primary Development: 1953

Mine Development: primary trench

Years of Production: 1953

Mine Production: ~5 tons milled = 350 pounds gold concentrates @ 23 oz/ton Au

Introduction

This small mine was located on the southern shore of Knight Bay, on the southeast end of Gordon Lake, 75 kilometers northeast of Yellowknife, NWT. A small mill was installed to process a small tonnage of ore in 1953. The site was visited in September 2001 by the author.

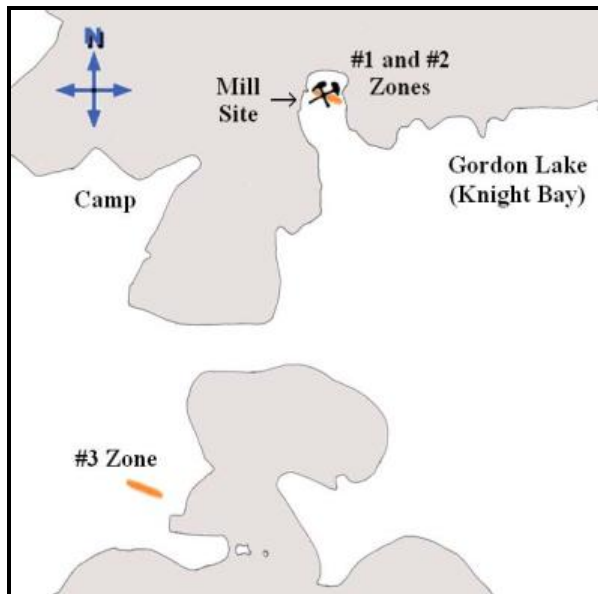


Figure 1. Treacy Mine location.

History in Brief

Alex Mitchell staked the 'Treacy' group of 7 claims in 1946, and later other groups were added to the claim block. The claims were held by Chinook Yellowknife Mining Syndicate until 1950 when Boreas Yellowknife Gold Mines Limited took control. By 1952, sufficient work had been done to warrant the installation of a small test milling facility that operated during the summer of 1953.

Geology and Ore Deposits

The property is underlain by greywacke and slate of the Yellowknife group. Quartz veins and masses occur in several narrow shear zones, and the most important discoveries made have been the #3 vein and the East and West zones. The strike of the #3 vein is northwest and dips steeply west. These zones occur near the axis of a fold that is sheared and fractured. The quartz is blue to grey in color and mineralized with pyrite, arsenopyrite, galena, sphalerite and chalcopyrite (Kelly, 1964).

Boreas Yellowknife Gold Mines Limited (1953)

Starting in 1950, a diamond drill program was initiated totaling 664 feet in eight holes (National Mineral Inventory). Previously, the claims were trenched. Drilling in four holes revealed visible gold in the veins systems (The News of the North, July 21st 1950). During the following year, Alex Mitchell and Capt. J.E. Treacy directed a crew of men in exposing the vein by a massive trenching program, in which over 30 tons of ore were stockpiled by June 1951. The company considered sinking a 50 foot shaft on the vein. By this time, Alex Mitchell had completed work on his patented rocker-mill invention, and planned to put it into use at the 'Treacy' claims (The News of the North, June 20th 1951). At the end of 1951, the Boreas company decided to focus work on the Gilmour Lake tungsten property, and the prototype mill was shipped there instead. In 1953, the rocker-mill was flown into Gordon Lake and installed at the 'Treacy' claims. During the summer, it is told that 350 pounds of gold concentrate grading 23 ounces per ton gold were recovered from 5 tons of stockpiled ore (The Western Miner, April 1954).

The Mitchell Mill

This invention was designed and patented by Alex Mitchell in 1950. Its primary function was to provide small high-grade mining operations with a cost-effective milling plant. The uniqueness to the design came from the method in which ore was crushed, by using a rocking manganese plate to grind ore down to size. The rocking motion of the plant would also allow the ore to pass through the grinding trough, aided by a flow of water. The slurry was then passed down two 8' sluice boxes where a concentrate was recovered. The entire mill was powered by a small five-horsepower gas engine and had a capacity to mill 4 to 5 tons per day (Canadian Patents Office).

Exploration Since Mine Closure

The claims remained held by Alex Mitchell and the Boreas company into the 1960s. Sampling was conducted by numerous companies, including Consolidated Northland Mines Limited in 1958 and Giant Yellowknife Mines Limited in 1964. In 1963-1964, the entire property was geologically mapped and the ore zones were again sampled by Expander Mines and Petroleum Limited. The claims lapsed in 1980 but the property was re-staked as the 'AM' claim by Dave Nickerson. The 'GB' group of two claims were staked in 1983, and subsequently purchased by Salish Resources Limited. In 1984-1985, a VLF-EM survey was conducted but did not reveal any anomalies. Mapping and prospecting confirmed the existence of four gold showings. Two diamond drill zones were explored on the #3 zone and the #1 zone intersected only weak or non-existent mineralization (Brophy et al., 1987).

References and Recommended Reading

Brophy, J.A., Crux, J.C., Gibbins, W.A., Laporte, P.J., Padgham, W.A., Seaton, J.B., 1987. Mineral Industry Report 1984/1985 Northwest Territories. Indian and Northern Affairs Canada, 1987.

Canadian Patents Office: claims #466271 and #520978

Goad, B., 1985. Geology, Geophysics and Drilling on the Salish Property - Gordon Lake, N.W.T. Salish Resources Ltd., October 1985. (*N.W.T. Geoscience Office Assessment Report #081927*)

Kelly, J.A., 1964. Report on the #3 Vein, Treacy Group. Giant Yellowknife Mines Ltd., 1964.

McGlynn, J.C., 1971. Metallic Mineral Industry, District of Mackenzie, Northwest Territories. Geological Survey of Canada, Paper 70-17, 1971.

Nickerson, D., 1982. Evaluation Survey - AM Mineral Claims - Gordon Lake, N.W.T. August 1982. (*N.W.T. Geoscience Office Assessment Report #081527*)

The News of the North newspaper articles, 1950-1955.

The Western Miner magazine, April 1954.

National Mineral Inventory (Treacy Group). NTS 85 I/14 Au 3.



TRY ME

Minor Exploration (Abandoned)

Gold

63° 04' 15" 113° 28' 45"

Years of Primary Development: 1941

Mine Development: 39° incline shaft

Introduction

The Try Me property is located 81 kilometers northeast of Yellowknife, NWT on the southwest side of Mac Lake. It has not been visited by the author of this report.

History in Brief

The 'Try Me' claims were staked in June-July 1938 by Spud Arsenault and Gerald Stewart for Cominco Limited. Surface exploration and 34 diamond drill holes were accomplished during the year. Work ceased in 1939, but resumed in 1941 when a small crew put down an inclined shaft on the gold vein. No major work has been done since.

Geology and Ore Deposits¹

The rock near the vein is sedimentary quartz biotite schist of the Yellowknife group. The quartz vein is exposed at intervals throughout a length of 2,450 feet. At the north end of the outcrop it strikes north 5° east and passes under Mac Lake, gradually changes towards the south and strikes roughly south 30° east near the south end. Most known gold occurs about 1,400 feet south of the north end of the vein outcrop.

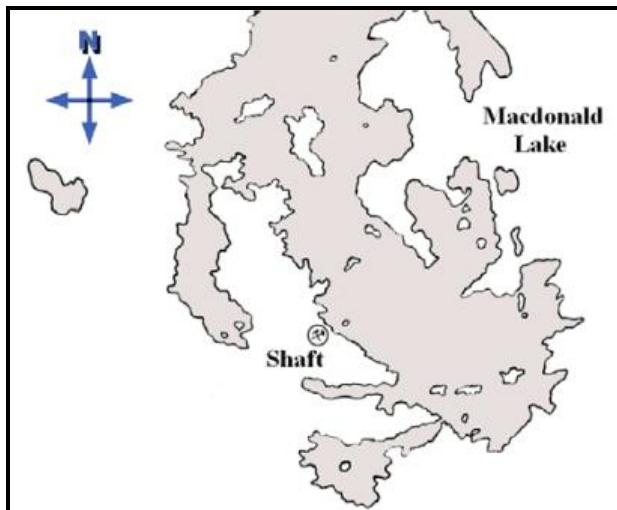


Figure 1. Try Me property plan.

Cominco Limited (1941)

In 1941, Cominco sank a short shaft on the #1 vein at the Try Me property in order to fulfill assessment work requirements. This shaft, with dimensions of 5 feet x 7 feet, was sunk an inclined length of 39 feet (N.W.T. Geoscience Office Assessment Report #015120).

Exploration Since Mine Closure

No other work is reported. New mineral claims were staked by Max Braden surrounding Macdonald Lake in 2003.

References and Recommended Reading

Lord, C.S., 1951. Mineral Industry of District of Mackenzie, Northwest Territories. Geological Survey of Canada, Memoir 261, 1951.

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geology from NORMIN.DB (<http://www.nwtgeoscience.ca>) Showing 085PSW0013

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.



TUNDRA

Producer (Remediated)

Gold

64° 02' 15" 111° 10' 45"

Years of Primary Development: 1952-1953, 1957-1968

Mine Development: 1,250' vertical shaft, 8 levels (20,986' dev.)

Years of Production: 1964-1968

Mine Production: 187,714 tons milled = 104,476 oz Au

Introduction

The Tundra Mine is located in the Courageous Lake region at the southeast end of Matthews Lake, 238 kilometers northeast of Yellowknife, NWT. It was a gold producer between 1964 and 1968. The site was reactivated in 1983 when the old milling plant was used by Salmita Mine, but the mine itself has not been in production since 1968. The author visited the site many times between 2001-2006. The site was completely remediated by a government-funded program in the spring and summer of 2007.

Brief History

The Courageous Lake region was first prospected by the Territories Exploration Limited group in 1938 when several gold showings were noted. After World War II, the area attracted more prospectors including Jack Matthews who staked the 'Jeja' group of claims at Matthews Lake in 1946. Bulldog Yellowknife Gold Mines Limited was formed in 1948 to undertake an extensive surface exploration program. Rich results paved the way for starting a shaft-sinking program in 1952, but no development was accomplished because of depressed gold markets and other economic problems, despite the erection of a camp and installation of equipment.



Ryan Silke

Figure 1. *Tundra Mine headframe, 2005.*

The Bulldog company reformed as Taurcanis Mines Limited in 1956 and the mine was reopened the following year. A six-year development program ensued and gold milling began in 1964 under the banner of Tundra Gold Mines Limited. The mine suffered from some early operational problems. Economic matters relating to the state of gold on the markets, labor shortages, and increased operational costs took a toll on the feasibility of operations, and the mine was forced to close in early 1968. The mine did not reopen, but most of the buildings on site were used by Giant Yellowknife Mines Limited in the 1980s for gold production at the nearby Salmita Mine.

Geology and Ore Deposits¹

The Tundra Mine area is underlain by the north-northwest trending Courageous-Mackay Lake supracrustal belt. Turbidites to the east conformably overlie and are locally interbedded with the volcanics. Granitic rocks intruded the

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

volcanics to the west. In the property area two volcanic cycles have been identified, each consisting of thick massive to pillow basaltic flows overlain by a thinner unit of dacite and intermediate tuff, in turn capped by an upper unit of felsic tuffs and rhyolite flows. Many of the gold bearing quartz veins are stratigraphically associated with an interval of greenschist facies felsic volcanics associated with the upper cycle of felsic volcanics. The gold bearing deposits occur in quartz veins in two settings: 1) on or near the contact between sediments (greywacke-slate) and volcanics (mafic lavas and tuffs) of the Yellowknife group, or 2) entirely within volcanic rock. The principal deposits of the Tundra Mine are the Matthews vein, #2 vein and the South zone. Between the Matthews vein and the South zone the volcanic-sediment contact is thrown about 200 feet to the east by left-hand offsets on several parallel faults striking northeast.

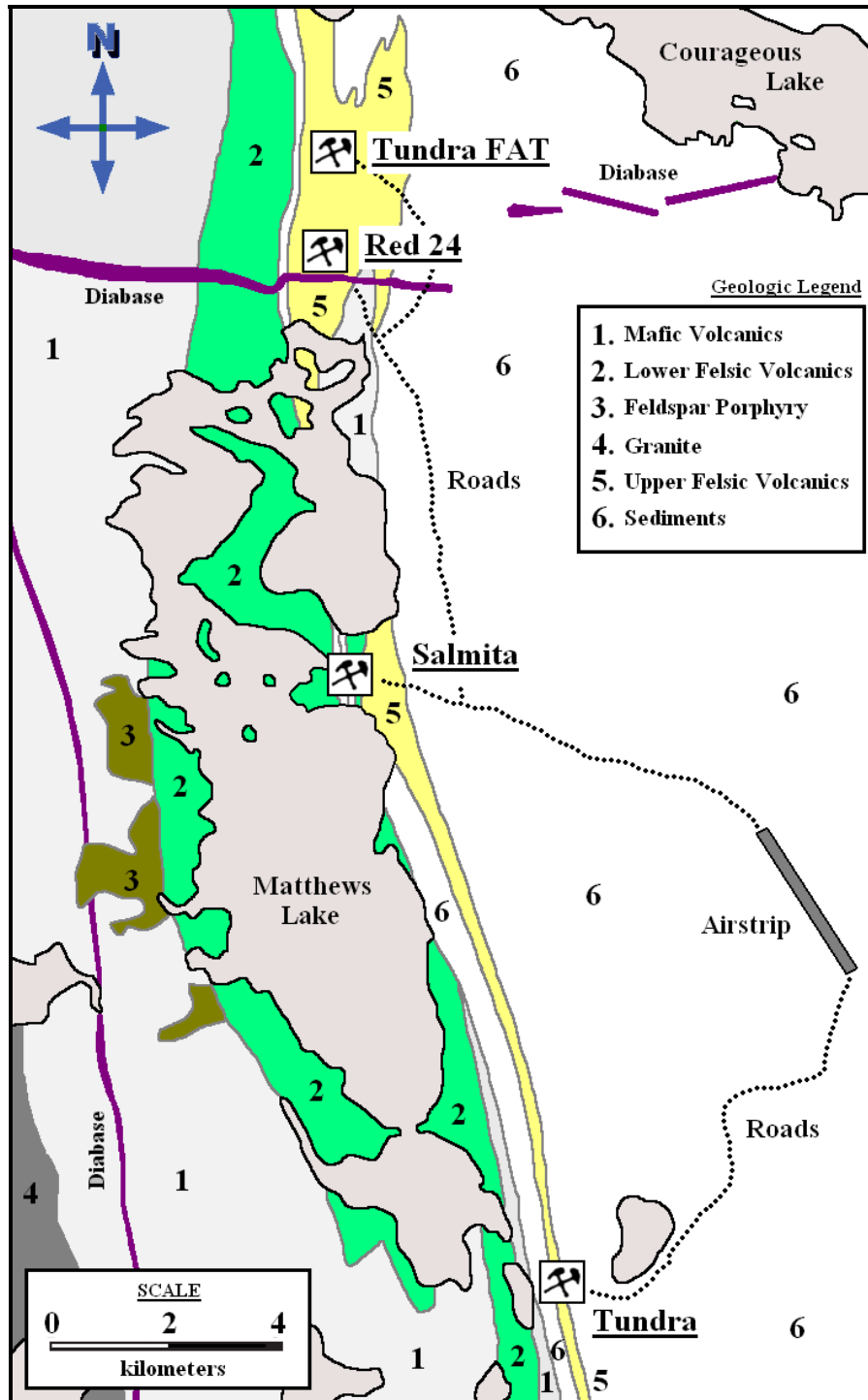


Figure 2. Matthews Lake area geology and location of Tundra Mine.

The Matthews vein strikes northerly and dips northeasterly at 75 to 85°. It lies within the sedimentary formations, near and almost parallel with the volcanic contact. Within the Matthews vein visible gold is common but generally fine-grained and more easily seen where it occurs along sericitic and longitudinal fractures. Galena, pyrite, pyrrhotite and scheelite are common but not abundant. Sphalerite and chalcopyrite are rare. No consistent relationship was discerned between the distribution of minerals and gold.

The South zone deposit occurs within garnetiferous amphibolite. Visible gold lies within quartz veins and lenses and is associated with arsenopyrite in chloritic wallrock near the quartz. Pyrrhotite is widespread and pyrite and chalcopyrite are present. The #2 vein was not exposed at surface but was encountered during mining. The vein contained an ore grade shoot 18 meters long and 2 meters wide, averaging 0.58 ounces per ton gold.

Bulldog Yellowknife Gold Mines Limited (1952-1953)

An extensive gold exploration program was conducted between 1948 and 1950 in which over 20,000 feet of diamond drilling had indicated a sizeable and high-grade deposit at Bulldog's property on Matthews Lake. In 1951 the company decided to go underground to explore the ore zones at depth. A large amount of equipment was acquired and a 70 ton convoy of sleds pulled by Cat train made its way from Yellowknife, east on Great Slave Lake to Thompson Landing, and north via trail to Matthews Lake, the first Cat Train haul into the Barrenlands in the Northwest Territories. It was intended to sink directly on the Matthews vein to 500 feet or more depth, construct mine facilities, and begin production by trucking ore to the nearby Salmita Mine where a mill was being erected. Work began during the spring of 1952 under the direction of Jock McNiven of Trans-American Mining Corporation Limited. The shaft was collared to 33 feet depth and a complete mining plant was installed but no shaft work was completed due to the unavailability of experienced shaft crews in the NWT and the depressed state of gold mining in Canada at the time. The site was mothballed in 1953 with plans to reopen as soon as conditions improved (Byrne et al., 1964).

Plant and Equipment

The following equipment was installed by 1953 at the mine: 246 horsepower National diesel engine with 120 kilowatts Brush generator, 110 volt Fairbanks-Morse light plant, 800 cubic feet per minute Broomwade air compressor, Canadian Ingersoll-Rand 36 inch x 24 inch two-drum electric hoist, plus a large surplus of supplies and tools for the shaft sinking job. Structures built included a 65 foot timber headframe, powerhouse, hoist building, three 'igloo' huts for the camp, and garage (Parkes, 1955).

Taurcanis Mines Limited (1957-1963)

Early in 1956, a reorganization of Bulldog Yellowknife Gold Mines Limited was arranged through the advent of Arthur White as president of the company. The name of the company was changed to Taurcanis Mines Limited, and financing for underground exploration was provided by Consolidated Discovery Yellowknife Mines Limited, New Dickenson Mines Limited, and Brewis & White Limited. In April 1957 a small crew was mobilized to refurbish the camp and plant and prepare an ice airstrip on Matthews Lake. A tractor train was dispatched to the property from Yellowknife carrying 107 tons of equipment and supplies, but due to winter transportation difficulties most of the supplies was air freighted to the mine. Older power units were not considered adequate, and two 200 horsepower Paxman-Ricardo diesel generators were acquired from the adjacent Salmita Mine. Other equipment and even some buildings were brought over from this closed gold project (Byrne et al., 1964).

1957 Development

Shaft sinking under contract with Lanky Muyres was underway on July 23rd 1957. Sinking continued through August and two shaft station levels were cut, at 175 feet and 325 feet depth. The shaft was completed to its first stage on September 2nd 1957 to a depth of 364 feet below surface collar. 4,053 tons of waste was hoisted during the shaft sinking operation using a Cryderman clam bucket. Lateral development began on the 1st and 2nd levels on September 13th and the Matthews vein was intersected on the 2nd level on September 18th 1957. The vein was intersected on the 1st level on September 27th 1957. Drifting progressed on both levels until temporary shutdown at the end of October 1957 due to bad weather (Byrne, 1957).

A total of 1,233 feet of lateral advance (including slashing equivalent) was accomplished in 1957, and the Matthews vein was explored for a strike length of 352 feet. An ore shoot 90 feet in length and assaying 0.78 ounces per ton gold across an average vein width of three feet was encountered on the north drive of the 2nd level. On the 1st level, visible gold and sporadic mineralization was encountered to the north. Results of the southern drives were not as interesting, but more work was required in this area. Mining equipment in use at the mine in 1957 included one Mancha "Little Trammer" battery locomotive, eleven side-dump ore cars, and two Eimco 12-B mucking machines (Byrne, 1957).

Work resumed in the spring of 1958 with extensive lateral development on the 1st and 2nd levels in both directions. Underground diamond drilling below the 2nd level intersected the Matthews vein at the 475- and 625-foot horizons showing strong zones of alteration and mineralization and a solid vein structure. In order to bring in the large amount of supplies and tools needed for the anticipated development program, the old Salmita airstrip (2,300 feet in length) located 5 kilometers east of Matthews Lake was repaired during 1958 using an International-Harvester TD-14 tractor. This strip was lengthened to 4,000 feet in 1959-1960 using a four-yard carryall and 2-12 Cat grader to accommodate large Bristol freighters with three-ton payloads. The strip was constructed at a cost of CDN \$35,000 (Byrne et al., 1964).

The Ice Road

New winter road routes were scouted in 1960 to the mine from Yellowknife. The route, ploughed by ice road engineer John Denison, started at the Discovery Mine and continued north to Matthews Lake. In the first year of operation, Denison was contracted to truck 275 tons of material to Tundra at rates of \$60 per ton from Yellowknife (Byrne et al., 1964).

Work was dormant for the winter season between October 1958 and April 1959. During this period an accelerated, permanent underground work program was planned, designed to outline sufficient tonnage to allow a production decision. During May-June 1959, the shaft was deepened to 665 feet providing two new levels. Drifting north and south of the shaft was undertaken to develop the oreshoots outlined on the upper levels, and a long drive was advanced south on the 2nd level towards the South zone. Meanwhile, surface construction included the erection of a large steel building, known as the Personnel Services Building, designed to house and feed as well as to provide offices, warehousing and heating facilities for the entire operation on a year-round basis. The entire mine operation was commissioned as a year-round affair, whereas before development ceased during the winter months (Taurcanis Mines Ltd. Annual Report, 1959).

This program of development on four levels continued during 1960. Diamond drilling below the 4th level was conducted from three drill stations. Drilling advanced during the fiscal year-end September 30th 1960 amounted to 12,530 feet of standard drilling, plus 5,312 feet of flat Packsack drilling to outline additional ore widths in the Matthews vein. A long raise was advanced towards surface within the South zone to provide a ventilation conduit (Taurcanis Mines Ltd. Annual Report, 1960). Shaft sinking below the 625-foot level was started in December 1960 and completed to a depth of 1,250 feet in March 1961 to provide a total of eight levels. Most work was concentrated on the 2nd and 8th levels where the southern reaches of the ore body were being explored by drifting. The South zone was entered on the 2nd level by an extension of the south drift within the Matthews vein. It was explored by crosscuts, a raise to the surface, and diamond drilling.

Work in 1962 was aimed primarily at exploring the South zone. The 4th and 8th level drifts were driven 3,000 feet south of the shaft beyond the volcanic-sediment contact. A series of faults displaced the ore zone 200 feet east of its projected position, so some crosscutting was required to reach the vein. The delineation of numerous quartz veins along the contact in this zone was the primary achievement in 1962 but the grades encountered thus far were considered too sporadic, and not enough work had been done to label the South zone as an economic ore source for production (Taurcanis Mines Ltd. Annual Reports, 1961-1962).

Underground development ceased in September 1962 pending a production decision. Six years of development at the mine had indicated a very rich orebody within the Matthews vein, although not entirely large. It was hoped that additional exploration over time would disclose a greater ore picture, especially within the South zone. The decision was made to go ahead with production. New financing partners were brought on board to help bring the mine to the production stage. In 1962, these included: Consolidated Discovery Yellowknife Mines Limited, Dickenson Mines Limited, Rayrock Mines Limited, and Radiore Uranium Mines Limited. The inclusion of Rayrock was an important asset, who sold Taurcanis a complete milling and power plant previously used at the Rayrock Mine (Byrne et al., 1964).

Tundra Gold Mines Limited (1963-1968)

The annual general meeting of the Taurcanis company in February 1963 ended with a resolution for a change in company structure, including a new name. The new company was called Tundra Gold Mines Limited, better reflecting the romantic nature of Canada's first Barrenland Gold Producer. Work during 1963 focused on construction of mine facilities, including the erection of the mill and power plant facilities. Underground mining resumed in December 1963 to prepare stopes on eight levels for production. A significant block of ore within a

primary ore shoot 400 feet south of the shaft was to be the primary source of mill feed (Tundra Gold Mines Ltd. Annual Reports, 1963-1964).

Production Starts

Construction was completed in March 1964 and gold production began March 15th. The exciting first brick was poured on April 12th 1964 and an opening ceremony was conducted in July. At the start of production, 17 stopes were ready for mining and represented about 110,000 tons of ore reserves (Tundra Gold Mines Ltd. Annual Report, 1964).



NWT Mining Heritage Society

Figure 3. *Tundra Mine aerial view, 1964.*

Milling Operations

An extensive amount of metallurgical testing of Tundra ores was done during the exploration years. It was found that the ore body was slightly refractory, but at such a small percent that standard cyanidation of the ores would give a gold recovery of 96%. The plant was designed for production of 150 tons per day. Ore was first received into a 110 ton coarse ore-bin after being hoisted from the underground. A two-stage circuit of crushing was in use at Tundra. Large pieces were first crushed in a Telsmith 18 inch x 32 inch jaw crusher to reduce ores to two inch size or less. Rock too large for further treatment were re-crushed in a Symons 3 foot cone crusher to $\frac{3}{4}$ inch size and conveyed into the mill building. Grinding was also two-stage, using Denver and Marcy 5 foot x 8 foot ball mills and a Krebs cyclone classifier to sort undersize and oversize material. A rough gold concentrate was recovered in Denver jigs and sent for amalgamation.

Slurry from the grinding circuit was sent for cyanidation using a 32 foot x 10 foot de-watering thickener and three agitators. This was followed by waste separation using two 8 foot x 10 foot Oliver drum filters, where a tailing was discharged from the plant. The pregnant solution from the thickener and filters was pumped to clarification, and then into precipitation where zinc was added to the solution. A 10-plate Whitco press was in operation, located in the refinery building. Amalgamation material and precipitate material was combined to pour gold bars using a Rockwell oil-fired furnace (Byrne et al., 1964).

Power Plant

Primary power generation was supplied by a large Cooper-Bessemer diesel engine of 1000 horsepower, driving a Canadian General Electric generator of 750 kilowatts. Backup power was available with a National (120 kilowatts), Crossley, General-Motors (250 kilowatts), and two Paxman (125 kilowatts) diesel engines. An emergency light plant for the accommodation complex, a Caterpillar D-3,400 15 kilowatts genset, was also available. Compressed air was supplied by a Canadian Ingersoll-Rand 675 cubic feet per minute capacity unit driven by electric motor. Smaller Gardner-Denver air compressor units supplied back-up air power.

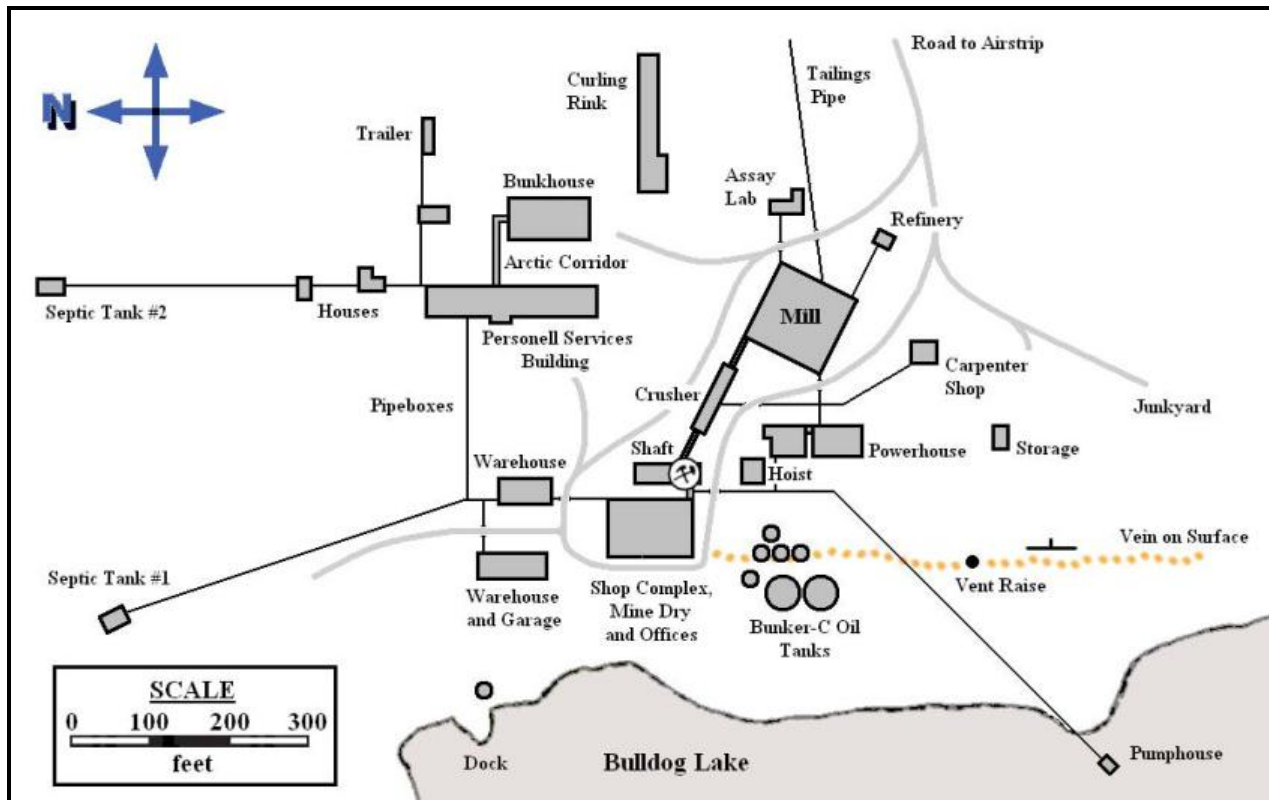


Figure 4. Tundra Mine plant and camp surface plan, 1964.

Heat for the entire operation was supplied by a large Foster-Wheeler 150 horsepower oil-fired boiler, using a system of heat recovery on exhaust gases from the diesel engines. Oil storage was supplied by several large tanks. Capacity for 350,000 gallons of bunker-C oil (for the boiler) was available in two large tanks. Diesel fuel was stored in six tanks to provide capacity for 225,000 gallons (Byrne et al., 1964).

Hoisting Plant

The 1,250 foot shaft at Tundra was serviced with a 62 foot timber headframe using man-cage and skip as counterbalance in the three-compartment structure. The hoisting plant consisted of a Canadian Ingersoll-Rand 48 inch x 36 inch size two-drum electric hoist, installed in 1960 for the deepened shaft (Byrne et al., 1964; mine records).

Mine facilities were built compactly to reduce the costs of heating. Buildings were metal-clad design of Armco and Butler make, with vapor barriers and insulation on all walls and ceilings to keep the driving wind from entering. Mine facilities were mostly housed in the Plant Services Building, in which was the machine and blacksmith shops, the mine offices, and the miner's dry. Other buildings provided for the warehousing, carpenter shop, and assay office (Byrne et al., 1964).

Mining Operations

Shrinkage stoping was the mining method employed at Tundra Mine. Permafrost did not hamper mining operations to any great extent. In 1957, during original development, it was reported that some difficulty was encountered by water freezing in drill holes. This problem was overcome by introducing alcohol into the air supply to lower the freezing point of the water. During production operations after 1964, dilution became a big problem because of a weak hanging wall, requiring extensive rock bolting. Also contributing to dilution was a mud-filled fault which parallels the Matthews vein in the northern section of the mine. Due to the nature of gold occurrences in the Matthews vein, individual stopes were usually very small which meant that several stopes had to be in operation concurrently to provide economic mill tonnage. Mechanized mining machinery included 25 rocker-dump ore cars of one-ton capacity, four Mancha "Little Trammer" battery locomotives, and several Eimco 12-B mucking machines (Byrne et al., 1964; mine records).

Accommodation Complex

The camp consisted of two large buildings making up the accommodation complex. The 2-story bunkhouse had room for 62 single men, plus a small hospital, laundromat, and four staff residences for married couples. The Personal Services Building contained cookery, commissary, recreation hall, gymnasium, staff quarters, and library. The two camp buildings were connected by a short arctic-corridor, and the rest of the buildings onsite were connected by rope (life-lines) so that men would not get lost in blizzards walking to work. The mine manager had his own residence on site, and in 1964-1965 trailer units were brought in on winter road to provide additional family housing. In 1965, there were eight families residing on site. A crew of about 100 to 120 persons worked on the site. Jack Boulding was mine manager at the start of production, but was forced to retire in May 1965 due to illness. Len Dixon replaced him as manager at Tundra Mine (Byrne et al., 1964).

Despite the short summer season, alternatives for outdoor recreation were available. An 18-“barrel” golf course was located on the airstrip using a No. 7 iron, with the objective being to either sink the ball into the oil drums, or ding them, which were used to mark the edges of the strip. The fishing on Matthews Lake was excellent (Byrne et al., 1964).

The first year of production at Tundra Mine (April 1964-March 1965) provided some disappointments. Grade of ore milled was below expectations, partially because of excessive dilution. Dilution was caused because of a weak hanging wall in which waste rock broke into the ore piles. To improve this problem it was decided to boost milling rates from 120 tpd to 160-tpd in an attempt to generate more profit from lower grade ores. To decrease dilution, the mine applied more rock bolts into the hanging walls of the stopes. Operating costs were seen as substantially better than pre-production forecasts, and generally the mine operated very well in its first year of operation.

Mine development resumed in December 1964 with drifting south on the 6th level to provide access to the southerly ore shoots. Stope preparations were carried out on several levels and mining was confined to the Matthews vein within three ore shoots. The primary ore shoot was located 400 feet south of the shaft and was being mined on all eight levels. A second shoot at 1,000 feet south and a third at 1,500 feet south of the shaft was being mined on the 2nd level. The third ore shoot was producing very high-grade ores averaging 1.28 ounces per ton gold over a width of six feet (Tundra Gold Mines Ltd. Annual Report, 1965).

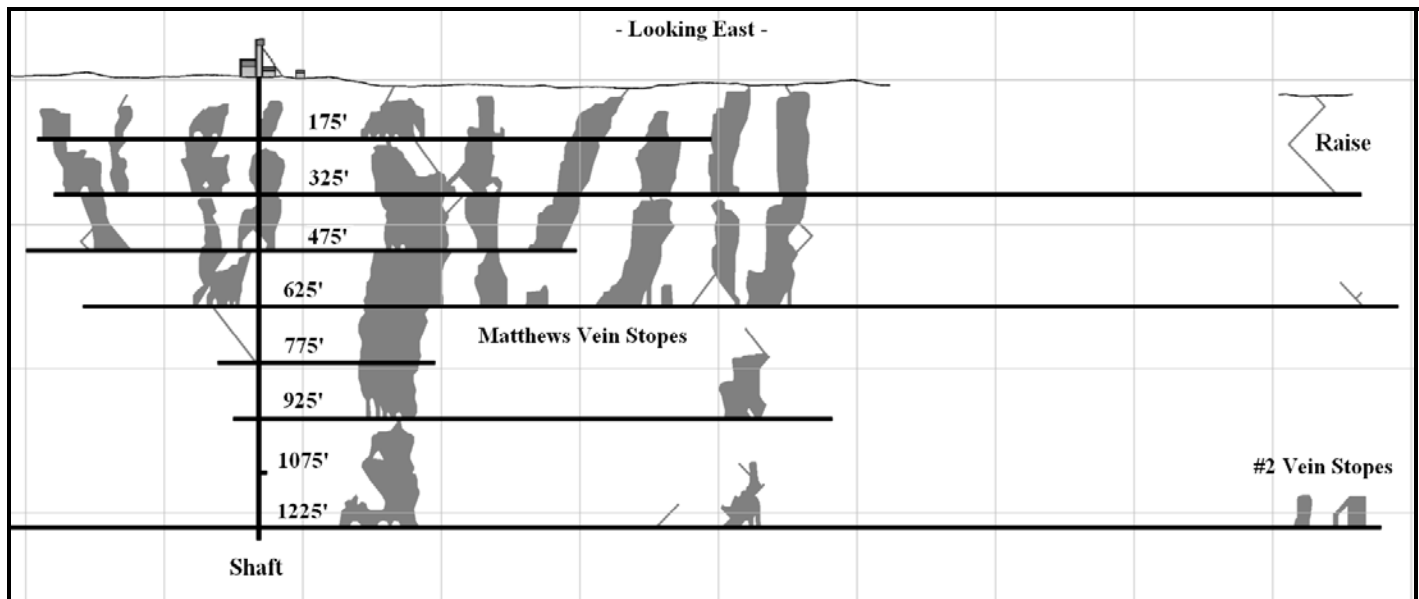


Figure 5. *Tundra Mine underground longitudinal plan, 1968.*

Mining operations in 1965-1966 were again focused on the Matthews vein. Dilution of the stopes in the northern sections of mine was improved through rigid ground control methods including rock bolting. The mine suffered some operational handicaps in 1965 because of increasing costs and labour shortages. Mining operations were affected by serious shortages of manpower. A breakdown of the Cooper-Bessemer diesel power plant in October-November 1965 curtailed production for a brief time. Mine development was conducted on the 6th and 8th levels south of the shaft. On the 8th level, the #2 vein was intersected and drifted upon for a length of 114 feet averaging 0.67 ounces per ton gold

across four feet. This new vein was predicted to be a stable ore occurrence, and raising was conducted to determine the feasibility for stoping. Drifting north of the shaft on the 1st, 2nd, 3rd, and 4th levels in 1965-1966 opened some new ore and extended known ore shoots in depth. These deposits were narrow but high-grade and were expected to become substantial ore sources (Tundra Gold Mines Ltd. Annual Report, 1966).

The third full year of production (April 1966-March 1967) was also difficult because of high costs and labor shortages. Labor turnover with miners was 300% during the year. This shortage of miners resulted in a drop in broken ore reserves and a reduction in the daily milling rates. A full crew of miners was hired in January 1967 and remained stable into the year. In late March 1967 the daily milling rate was increased to nominal rates (140 tons per day). Only 16,000 tons of new ore reserves were proven during the year, resulting in a total at March 31st 1967 of 40,476 tons grading 0.58 ounces per ton gold (Tundra Gold Mines Ltd. Annual Report, 1967).

New developments during the year included a high-grade ore shoot on the 1st and 2nd levels south of the shaft. A drift was driven south on the 3rd level in 1967 to intersect the downward extension of this shoot. In 1966-1967, the #2 vein was test stoped on the 8th level for a vertical distance of 80' but this work ceased because of low-grade ore values. Only 1,612 tons of material grading 0.24 ounces per ton gold was mined and milled from this section. Clean up of stopes in the north section of the mine began during 1967. A drift on the 5th level was driven north to intersect possible downward extension of the high-grade shoots mined in the north section. The primary ore shoots in the southern section of the Matthews vein were mined out during 1967 from the 8th level to the surface. Mining was focused largely on the upper four levels in this last year of operations (Tundra Gold Mines Ltd. Annual Report, 1967).

Year: *	Ore Milled:	Gold:	Year-End Ore Reserves: *
1963 / 1964	-	-	110,000 tons @ 0.93 oz/ton Au
1964 / 1965	44,788 tons	25,115 oz	101,040 tons @ 0.76 oz/ton Au
1965 / 1966	53,462 tons	33,036 oz	73,217 tons @ 0.57 oz/ton Au
1966 / 1967	48,588 tons	23,533 oz	40,476 tons @ 0.58 oz/ton Au
1967 / 1968	40,876 tons	22,792 oz	-
Total:	187,717 tons	104,476 oz	-

Table 1. Tundra Mine production and ore reserves, 1964-1968. (source: Tundra Gold Mines Ltd. Annual Reports)

* company fiscal year-end March 31st

1 st Level: (175')	2 nd Level: (325')	3 rd Level: (475')	4 th Level: (625')	5 th Level: (775')
2,204'	5,218'	1,621'	4,641'	691'
6 th Level: (925')	7 th Level: (1,075')	8 th Level: (1,225')	Raising:	Shaft Sinking:
1,969'	0'	4,642'	3,438'	1,250'

Table 2. Total underground lateral development per level, raising, and shaft sinking, 1957-1968.

(source: mine records)

1968 Closure

Worsening economic conditions in late 1967 convinced company officials that the Tundra Mine could no longer operate profitably. Increased costs, depressed gold market, and operational handicaps were the primary factors. The mine was closed in January 1968. On January 6th 1968 the last ore was hoisted, and the mill circuit was cleaned out to recover any remaining gold values. Most usable equipment was sent to Discovery Mine over the winter road. About 5,000 tons of broken ore remained underground, considered too low-grade for treatment in 1968. A substantial rise in the price of gold was considered necessary to reactivate the property (Tundra Gold Mines Ltd. Annual Report, 1968).

Production and Development Summary

The Tundra Mine milled 187,717 tons of ore to produce 104,476 ounces of gold between March 1964 and January 1968 (see **Table 1**). Silver was also produced but its recovery was not always reported. A total of 202 gold bars were poured with the last three bars poured on January 25th 1968 during the cleanup process. The mine is serviced with a 1,250 foot deep shaft with eight levels (20,986 feet of lateral development and 3,438 feet of raising). (see **Table 2**)

Exploration Since Mine Closure

Increased gold prices in the 1970s resulted in some renewed exploration by Tundra Gold Mines Limited. In 1973, a program of geophysical surveys was conducted in the area north of the shaft. Diamond drilling was proposed, but this work was postponed (Tundra Gold Mines Ltd. Annual Report, 1974). Giant Yellowknife Mines Limited conducted some surface exploration and also reviewed the old records of the mine during the 1980s to find out if any economical gold deposits remained. Recovery of the crown pillar was one of their objectives. They were producing from the Salmita Mine at the time and were leasing the surface plant and mineral claims. New claims were staked in the area by Seabridge Gold Incorporated and exploration of the area is ongoing as of 2008.

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TUNDRA (FAT)

Advanced Exploration (Remediated)

Gold

64° 07' 00" 111° 16' 30"

Years of Primary Development: 1988-1989

Mine Development: 473m vertical shaft, one level (1,756m dev, 131m raising)

Introduction

The Tundra (FAT) project is located between Matthews and Courageous Lakes, 240 kilometers northeast of Yellowknife, NWT. In 1988-1989, a shaft was sunk on the promising Fat zone, but despite the deposits promise as a high tonnage gold producer, work temporarily ceased. Seabridge Gold Incorporated now owns the property and is extensively exploring the claims, with hope of an open pit gold mine in the future.



NWT & Nunavut Chamber of Mines

Figure 1. *Tundra (FAT) Project site, 1989. Looking south to Matthews Lake.*

Brief History

This area was first explored during the late 1940s and led to the development of the Salmita and Tundra Mines. The 'FAT' claims (felsic-ash-tuff) were staked in 1977 by Noranda Exploration Company Limited after having flown a combined EM and magnetometer survey of the Courageous Lake volcanic belt. The target of exploration was base metals at that point. Gold became the focus of exploration in 1980 as its price went up. A joint venture was formed between Noranda Exploration and Getty Resources Limited. The attraction of the Hemlo rush in Ontario also sparked a lot of interest in this region. More than 47,000 meters of diamond drilling was performed on the Main and Carbonate zones to a depth of 500 meters between 1983-1987. Total Resources Limited acquired Getty Resources Limited in September 1987 and amalgamated it with its other subsidiary, Total Erikson Resources Limited, to become Total Energold Corporation Limited in September 1988. Principle interests in the joint venture were Noranda (25.5%), Hemlo Gold Mines Inc. (25.5%) and Total Energold (49%). In 1987, the joint venture partners announced plans for sinking an exploration shaft. The shaft was completed in 1989, but the metallurgy of the deposit suggested expensive gold recovery if the mine was to go into production.

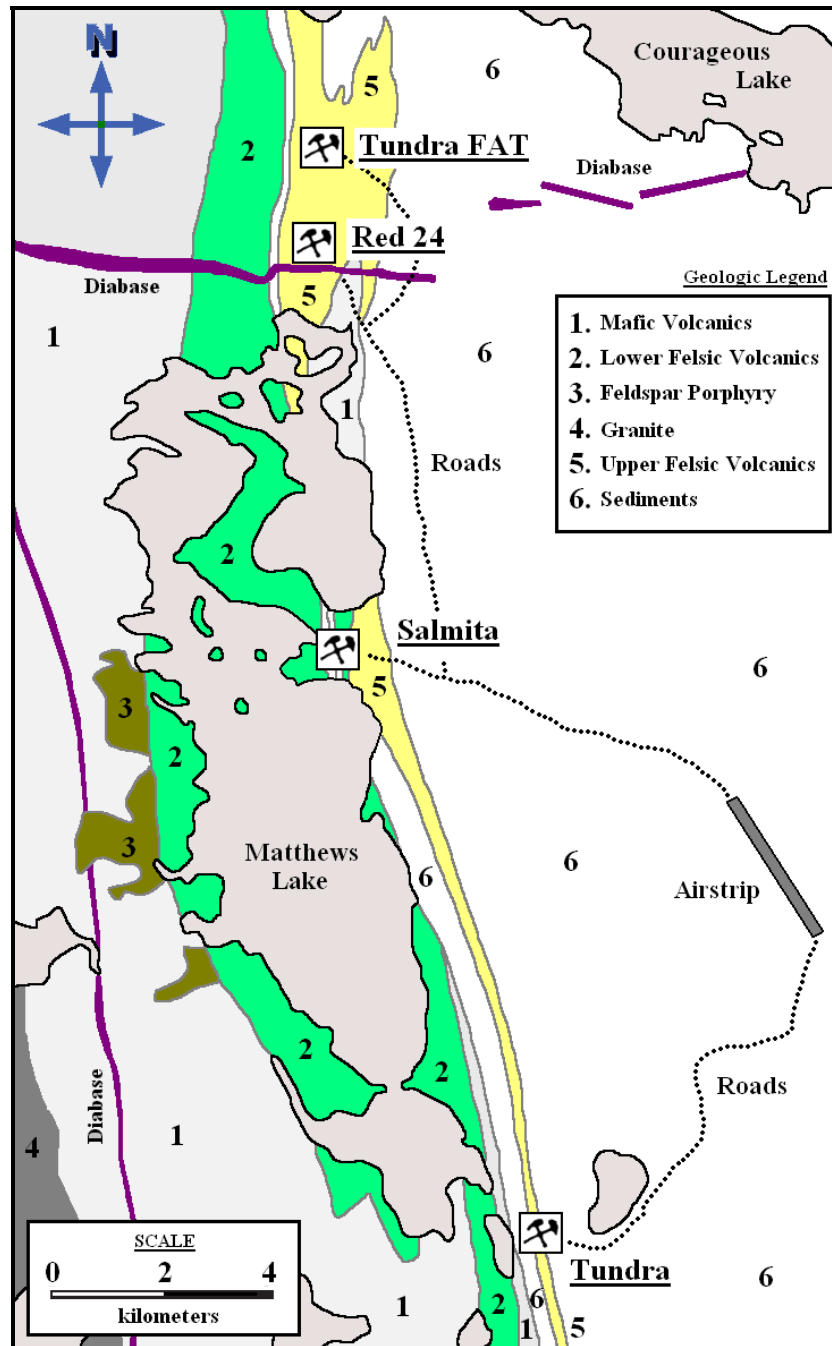


Figure 2. Matthews Lake area geology and location of Tundra (FAT) Mine.

Geology and Ore Deposits¹

The Tundra-FAT deposit is located in the Courageous-Mackay Lake greenstone belt in the Slave structural province. All stratified rocks have been mapped as belonging to the Archean Yellowknife Supergroup and have undergone greenschist facies metamorphism. Two separate gold-bearing zones are present in a volcanic sequence known as the Matthews Lake Volcanic Complex, divided into a lower and upper cycle. The lower cycle comprises a 400 meter thick unit of rhyodacitic flows overlain by 45 to 150 meters of interbedded calcareous sediments and calcareous rhyodacitic lapilli tuff, and capped by a narrow layer of locally graphitic pelite. The base of the upper cycle comprises a 30 meter thick unit of rhyodacitic lapilli tuff to agglomerate, overlain by felsic volcanics transitional to typical Yellowknife Supergroup sediments.

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

The largest auriferous zone, known as the Fat zone, occurs in felsic volcanics of the upper cycle. It consists of at least 17 parallel, sub-vertically dipping shear zones subparallel to strata, within a 200 meter wide alteration zone with disseminated arsenopyrite, pyrite and pyrrhotite. Individual shears range from 30 centimeters to 20 meters in thickness and are characterized by variable but generally strong sericitization, silicification, chloritization and irregular blue-grey to dark grey quartz veins and masses. Many of the latter have diffuse boundaries. Very fine acicular arsenopyrite is the main sulphide mineral, occurring as disseminations throughout the sericitized rock and as felted masses. Lesser pyrrhotite and pyrite occur as disseminations. Chalcopyrite, sphalerite and scheelite have been identified within the zones but are rare. Gold is largely refractory, being tied up within arsenopyrite. However, a significant proportion occurs as extremely fine-grained free gold averaging less than 10 microns in size. Some of this occurs in fractures or inclusions in arsenopyrite but most is interstitial to silicates. Visible gold is rare, seen only in some quartz veins.

The Tundra zone has been traced by drilling over a 1500 meter strike length and locally to a depth of at least 120 meters and is open at depth. Average grade of mineralization within the 190 meter wide alteration zone has been variably reported as 0.03 ounces per ton gold and as 0.043 ounces per ton gold. The Carbonate zone is not well described in available literature. It occurs at the contact between felsic volcanics and overlying pelites in the lower cycle.

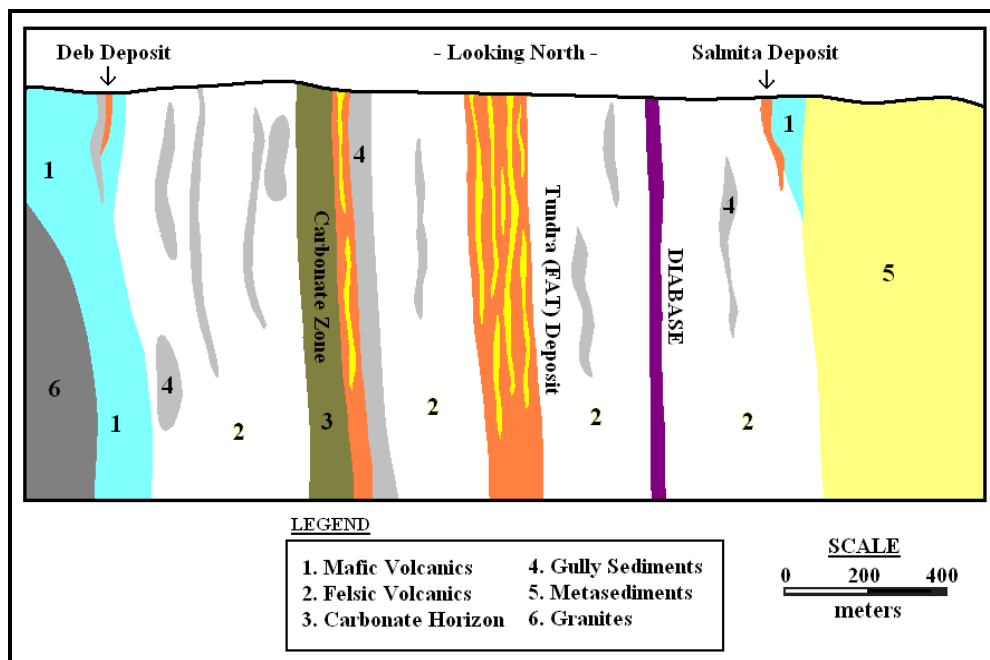


Figure 3. Cross-section geology, Tundra FAT deposit.

Noranda Exploration Co. Limited (1988-1989)

In December 1987, Noranda Exploration Company Limited, Getty Resources Limited, and Hemlo Gold Mines Inc. announced approval of a two-year CDN \$35 million underground exploration program to confirm the FAT deposit's continuity, mineability, and economic viability. Acting as operator of the Tundra project, Noranda Exploration Company Limited hired Thyssen Mining Corporation of Canada Limited to sink the shaft on the FAT property. Mobilization was initiated early in 1988 and the shaft was collared during severe weather in April 1988 (Getty Resources Ltd. Annual Report, 1988). Project manager with Noranda Mines was Eric Seraphim, and project engineer with Thyssen Mining was Andy Fearn. The operation during the height of shaft sinking employed about 20 to 30 persons. A camp consisting of cookery, two bunkhouses, and shops was located at the north end of Matthews Lake. A gravel road connected the camp to the shaft site. Transportation to the property was available by floatplane, or larger aircraft using the old Salmita/Tundra airstrip. Winter roads were cleared from the Lupin Ice Road (Watt, 1988; Werniuk, 1989).

Mining Plant

A portable mine plant had been erected which consisted of a 75 foot prefabricated headframe, and a large building for housing a two-drum Lakeshore-type hoist (84 inch diameter), four 500 kilowatts Caterpillar diesel generators, and

four 700 cubic feet per minute Atlas-Copco air compressors, along with housing garage and shop facilities. Mining equipment included one single-boom Atlas-Copco rail jumbo, Atlas-Copco LM56 mucking machines, and Haglum ore cars (Andy Fearn).

Underground Development

Shaft sinking (3 compartments) was initiated July 3rd 1988 and was completed in April 1989 to a depth of 473 meters below surface. A level was established on the 425-meter level, from which a 767 meter drill drift was excavated north and south for the purpose of testing the gold zone at 50 meter intervals along an 800 meter strike length, 100 meter above and below the level. Total underground diamond drilling was 28,046 meters in 125 drill holes and 36 bazooka holes. Lateral development consisted of a 360 meter crosscut from the 425-meter level station into the ore zone, and 629 meters of drifting within the zone. Six raises totaling 131 meters were driven and a 1,283 tonne bulk samples was prepared (Total Energold Corp. Ltd. Annual Report, 1989).

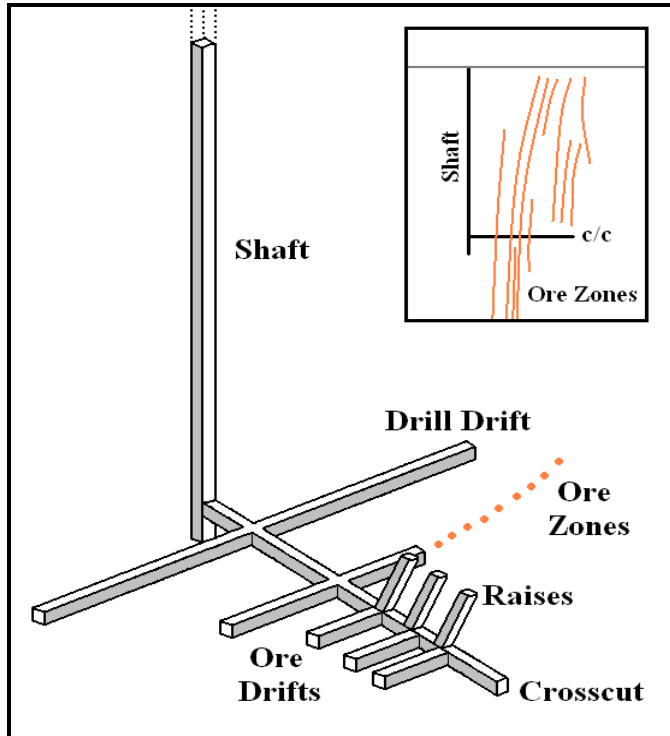


Figure 4. *Simplified underground plan, 425-meter level.*

Extensive metallurgical testwork was completed on the bulk samples during 1989-1990. A pre-feasibility study was completed under the direction of Wright Engineering Limited early in 1990, suggesting that while the deposit was large with a sizeable gold resource, the grade of the ore was too low and discontinuous to be mined economically at current gold prices. The refractory nature of the ore also provided metallurgical difficulties that the company was unwilling to face (Total Energold Corp. Ltd. Annual Report, 1989).

Upon the completion of drilling and development in October 1989, the company decided to postpone any further underground work. During 1990 the plant buildings were dismantled and everything was removed from the site by winter road early in 1991. A report prepared by Noranda in 1990 estimated a total reserve of 14,470,000 tonnes grading 6.5 grams per tonne or a total inventory of 2.6 million ounces of gold. Some limited surface exploration was carried out on the large claim block by Noranda during 1990 (Total Energold Corp. Ltd. Annual Report, 1990).

Exploration Since Mine Closure

In 1992, Noranda merged Hemlo Gold Mines Incorporated with Battle Mountain Gold Limited and reinitiated exploration on the property. In 1996, Battle Mountain Gold merged with Newmont Gold Corporation, and Newmont became the registered owners of the property. Surface drilling began in 1997-1998 under option by Placer Dome Explorations Limited with the view that the deposit might be economically mined by open pit. Placer Dome completed 13,345 meters of diamond drilling. Low gold prices in subsequent years put a halt to exploration (Arik and Lechner, 2004).

Seabridge Gold Corporation Limited purchased the property in 2002 from Newmont, with Newmont retaining a 2% net-smelter royalty. Surface exploration commenced, and by 2003 they reported the finding of 12 gold targets identical to the FAT deposit. A review of previous diamond drilling results and the resampling of over 100,000 meters of drill core confirmed that 9 out of those 12 targets had the potential to host a bulk-tonnage gold deposit. These targets include areas around the old Tundra and Salmita mines (Seabridge Gold Corp. Ltd. Annual Report, 2003). In 2004, a 7,940 meters (23 holes) diamond drill program was undertaken, and a new ore reserve was published as a result of exploration results. Measured and indicated reserves stood at 44.2 million tonnes grading 2.49 grams per tonne (3.5 million ounces) plus an additional 65.5 million tonnes of ore grading 2.32 grams per tonne (5 million ounces) in the inferred resource category (Seabridge Gold Corp. Ltd. Annual Report, 2004; Arik and Lechner, 2004).

During 2005, a new diamond drill program was started. 4,500 meters were drilled with the purpose of investigating the strike potential of the FAT deposit to the north and south. This work showed an 850 meter strike extension potential for the deposit to the south. Two drilling programs were accomplished in 2006. The winter program comprised of 26 holes totaling 7,100 meters, successfully testing higher grade structures of the FAT deposit, two new structures to the west, and a northern extension of the deposit. The summer drill program was focused on testing gold zones to the west of the FAT deposit, totaling 2,900 meters in ten holes. (Seabridge Gold Corp. Ltd. Annual Report, 2005-2006). The deposit's characteristics indicate that a bulk open-pit operation is the best mining method.

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Personal communication: Andy Fearn



VIKING

Advanced Exploration (Abandoned)

Gold

63° 06' 30" 114° 03' 15"

Years of Primary Development: 1947

Mine Development: 160' inclined shaft, 1 level (400' dev.)

Introduction

The Viking Mine is located 74 kilometers north of Yellowknife, NWT on the southwest side of Morris Lake. The property was visited briefly in August 2000.

History in Brief

The original 'Ola', 'Arlene' and 'BBB' claims were staked in 1945 by Bill Rossing for Fred Giauque. Athona Mines Limited acquired the claims in 1946 and conducted surface exploration. A shaft was then sunk in 1947 by Viking Yellowknife Gold Mines Limited, but no further mining development has been undertaken. Exploration is ongoing.

Geology and Ore Deposits

The Morris Lake area is underlain entirely of sedimentary rocks, with the exception of a small lens of volcanic rock northwest of the Viking camp. Development has focused on the Main zone, a sill of altered diorite crossed by many irregular bodies of quartz with showings of gold. The Main zone has been traced for a distance of over 2,000 feet in a northeasterly direction, with widths of 10 to 60 feet and a dip of 80° southeast (Lord, 1951).



Geological Survey of Canada

Figure 1. *Viking Mine, 1940s.*

Athona Mines Limited (1947)

Drilling done and other surface work on the property in 1946 by Athona Mines Limited disclosed a number of vein masses within the underlying sediments on both the Main and East-zones. Over 12,000 feet of diamond drilling in 35 holes was accomplished by January 1947. The gold assays were somewhat erratic and late in 1946 it was decided to sink a shaft on the Main-zone. The purpose of this work was to carry out a thorough program of channel, car, and bulk sampling in order to arrive at a definitive ore grade for the Main-zone. It was decided to sink an inclined two-compartment shaft directly within the Main-zone rather than a full production shaft sunk at the footwall of the deposit. This would result in a minimum of expense, and also allow for full investigation of the deposit while sinking. It was also the more feasible option, since sinking a production shaft at this time would have been logistically difficult. A portable mining plant was shipped to Morris Lake from the nearby Discovery Mine in March 1947 along with 60 tons of other supplies needed to carry out the program. Shaft sinking contractor was Fred Nilsson. The first shaft round was pulled on April 3rd 1947. Three buildings used at the shaft site, the powerhouse, miners dry, and the ore bin, were pre-fabricated in Yellowknife by Ivor Johnson and then hauled to the minesite by tractor in early 1947 (Lord, 1951; Athona Mines (1937) Ltd., 1947).

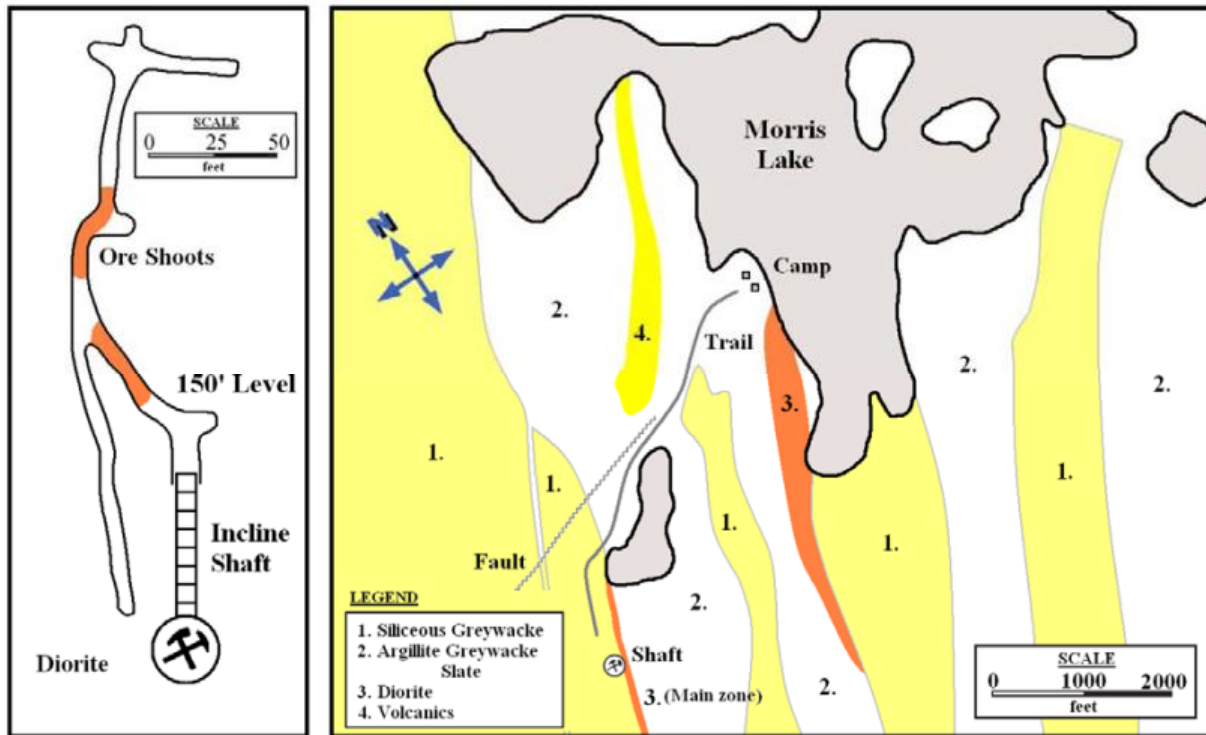


Figure 2. Viking Mine local geology and underground plan.

Viking Yellowknife Gold Mines Limited (1947)

Athona Mines Limited subsequently gave operating control to a new company, Viking Yellowknife Gold Mines Limited in June 1947 (The Precambrian magazine, June 1947). The shaft (7 feet x 10 feet) was driven to an incline depth (65° to the north) of 160 feet. About 400 feet of drifting was done on the Main zone at a depth of 150 feet. Underground diamond drilling totaling 734 feet was also performed. Most exploration focused on the northern extension of the Main zone where the best grades were attained. Shaft sinking operations through the siliceous quartz-diorite sections of the Main zone uncovered very high-grade portions near the surface, which could possibly continue across the property for a length of 12,000 feet (Tremblay, 1952; Lord, 1951).

Power Plant

A gasoline powered 35 horsepower Chrysler motor was connected to a Holman two-stage air compressor supplying about 135 cubic feet per minute for the hoist and drills. The hoist was a single-drum Holman air winch. The plant was housed in a 14 foot x 22 foot frame building. A 32 foot timber headframe and 30 ton ore-bin serviced the small shaft. Other plant buildings consisted of a miners dry/shop building (Lord, 1951).

A camp was erected on Morris Lake and connected to the shaft via a short trail. It consisted of a frame cookery and several small tents. Henry Lepp, K.A. Matheson, and Norman Byrne were in charge of underground operations at Viking during the summer of 1947. About 10 men were employed (Tremblay, 1952; Lord, 1951).

Shaft sinking during 1947 was completed at a total direct cost of \$53/foot. The work was done by a crew of four miners in 28 days, and a total of 35 rounds were blasted. This work did not include lateral development at the first level (Byrne and Lepp, 1948). Early in 1947, it was intended to clear an airstrip on a nearby gravel esker. Such a project would be of benefit to the other possible gold mines opening in the area, but one was never built (The Precambrian magazine, June 1947).

The 1947 operation at Viking was considered as one of the most economical and speedy shaft sinking projects in the Northwest Territories for its day (Byrne and Lepp, 1948). The camp was closed in September 1947 and underground samples were flown to Yellowknife for assaying. These samples did not give an accurate portrayal of the grade of the deposit, and more development would be required. Overall, company officials were not entirely enthused over the potential of the project.

Exploration Since Mine Closure

In the 1950s, it was considered feasible to open the mine and truck ore to Discovery Mine for milling. No work was done. In 1968, the property was optioned to Discovery Mines Limited and it was calculated that the property contained 700 tons per vertical foot grading 0-60 ounces per ton gold. Discovery carried out surface exploration but dropped the option in 1977 (National Mineral Inventory). In 1987, Canamax Resources Inc. staked new claims in the area and performed mapping of the area. 17 holes were diamond drilled during 1988-1989 by Canamax under option from Viking Yellowknife (Hitchins, 1990). In 2003, Copper Hill Corporation Ltd. optioned the property from Lakota Resources Limited (of which Viking Yellowknife Gold Mines Limited is a subsidiary). This company is now known as Viking Gold Exploration Incorporated, and in 2004 they commenced field exploration and sample, followed by diamond drilling and geochemical surveys in 2005. Surface exploration and geochemical and geophysical evaluation continued in 2006. A second phase of diamond drilling was performed in the winter of 2007. Exploration is ongoing. (Viking Gold Exploration Inc.).

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WEST BAY (DAF)

Minor Producer (Abandoned)

Gold

62° 55' 00" 113° 14' 00"

Years of Primary Development: 1947-1948, 1990

Mine Development: open pit 60' depth

Years of Production: 1947-1948, 1991

Mine Production: 1,694 tons milled = 1,362 oz Au

Introduction

The West Bay Mine, better known locally as the DAF property, is located on the southeast shore of Gordon Lake and 76 kilometers northeast of Yellowknife, NWT. The site was visited in September 2001.

Brief History

The 'DAF' claims were staked by Jake Woolgar and Gordon Wonnacott in June 1946 over a high-grade gold showing called the Hump vein. The property was optioned to Zolota Yellowknife Mines Limited in October 1946, and over 6,700 feet of diamond drilling and some trenching was performed. In 1947, the owners, in association with Jim McAvoy, erected a small mill. The following year, this mill was expanded under new management with West Bay Yellowknife Mines Limited. A small amount of gold was produced during these operations. J.F. Doucette re-staked the property as the 'MQ-001' claim in 1977. In 1983, Blackridge Gold Limited conducted diamond drilling (20 holes; 1,400 feet); in 1984, more drilling was done in partnership with Cruiser Minerals Limited (24 holes; 3,968 feet), tracing the vein along 400 feet of strike. In 1990, William Knutsen and Knud Rasmussen of Cameron Mining Limited mined the vein by open pit and trucked the ore to Yellowknife for processing in 1991.

Geology and Ore Deposits¹

Mineralization occurs within gold bearing quartz veins cutting greywacke and slate interbeds of the Archean Yellowknife Supergroup. The veins are along the axis of a syncline and strike north 10 to 40° east and dip 55 to 75° southeast-east. The main vein is exposed for 320 feet, is from 1.5 to 10.5 feet wide and averages about 4.5 feet wide. There is a wide portion of the vein called the 'Hump' which is up to 35 feet wide. This segment of the vein is hosted by the east flank of the fold. The veins mineralogy include pyrite, galena, chalcopyrite, sphalerite and visible gold.

Jake Woolgar (1947)

A mill was shipped from local properties during the summer of 1947 to the 'DAF' claims, and open pit work began on the Hump vein. A tent camp was built down on the shore of Gordon Lake, and a narrow-gauge rail track was laid from the dock up to the mill-site. It was operated by an air-winch. This small, improvised mill operated at a very low daily rate. During a period of September 21st to October 12th 1947, it is told that 18 tons of high-grade ore was milled to recover 50 rough ounces of gold through amalgamation. The operation then closed while the owners contemplated refinancing the operation (Lord, 1951).

West Bay Yellowknife Mines Limited (1948)

In April 1948, a deal was struck between Jake Woolgar and Jim McAvoy for the purchase of the property. The claims were purchased for \$10,000 and 400,000 shares in a new company - West Bay Yellowknife Mines Limited (The Northern Miner, July 15th 1948). The property was put under the direction of Jim McAvoy and Henry W. McKittrick. In 11 weeks between purchase and the start of production in June 1948, a complete milling and power plant capable of producing up to 10 tons per day was erected on the 'DAF' claims (The News of the North, June 25th 1948).

Milling Plant

Ore from the open pit was crushed in a small Woodstock 6 inch x 4 inch jaw crusher, and feed into the ore-bin by bucket elevator. Grinding was accomplished through two ball mills, one approximately 2 feet x 4 feet in size, and the second of a smaller size. Gold was recovered off jigs and perhaps a small blanket table or sluice box. The milling plant was operated by a 10 horsepower Lister diesel engine and a 25 horsepower Jimmy engine through drive shaft and V-belt. Rough gold was sent to a crude onsite refinery plant, where a small gold bar was poured (Knud Rasmussen, pers. comm.; site evidence).

¹ Geology and Ore Deposits section extracted and modified from NORMIN.DB, NWT Geoscience Office.

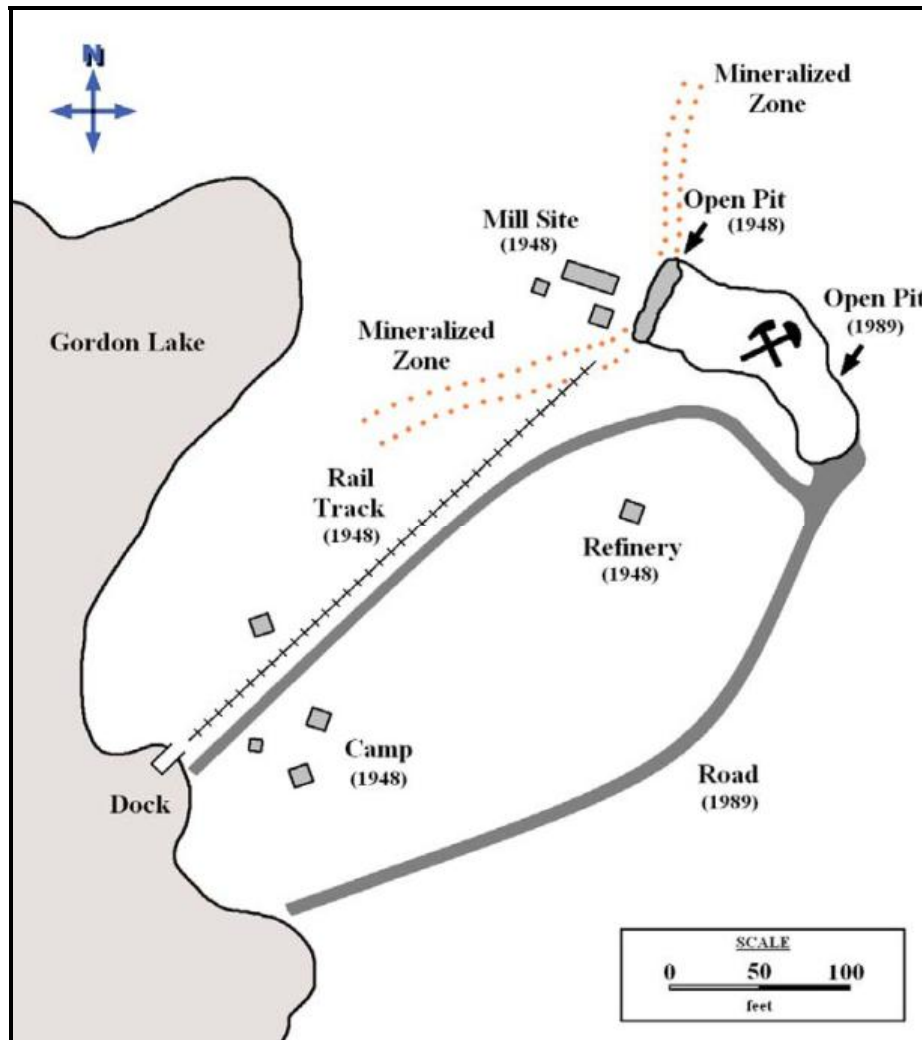


Figure 1. West Bay Mine property plan.

1948 Operations

Operations started in June 1948, with the first gold bar (weighing 70 to 80 ounces gold) poured on June 22nd (The Northern Miner, July 15th 1948). Mining development was concentrated on a single open cut adjacent to the milling plant. On July 17th 1948, the pit had the following dimensions: length 40 feet, width 12 feet, slope depth 22 feet. The pit followed the strike of the vein and was fitted with a small headframe, tugger hoist and bucket (Lord, 1951). Diamond drilling was underway in July 1948 to explore a new vein that was intersected at shallow depths in the open cut. It was reported to extend northeast underneath light overburden. Twenty men were employed during this time with milling and open pitting. A sawmill was also in operation (The News of the North, Aug. 6th 1948). Air power was supplied by a Canadian Ingersoll-Rand or Schram air compressor (Knud Rasmussen, pers. comm.).

It was found that the small mill was unsatisfactory. While the plant had a capacity of 10 tons per day, it operated on average no more than 3 tons per day. It operated well when quartz material was treated, but recovery and efficiency fell quickly because of dilution caused by the mining of greywacke in the wall rocks. The company reported in September 1948 that negotiations were underway for the purchase of a larger mill, two of which were available in the area (The Northern Miner, Sept. 9th 1948). To closure at the end of August 1948, 289 tons of ore were treated producing 221 fine ounces of gold and 29 fine ounces of silver. 10 rough ounces were also recovered (Lord, 1951). Five small gold bars were poured (The Northern Miner, Sept. 9th 1948).

Cameron Mining Limited (1990-1991)

In 1989, ore reserves at the 'DAF' claims were calculated as 3,540 tons grading 1.23 ounces per ton gold containing 4,367 ounces of gold to depth of 110 feet. This reserve was calculated by tabulating the results of diamond drilling by Blackridge Gold Mines Incorporated in the 1980s (Knutson, 1989). Early in 1990, Cameron Mining Limited mined a

large bulk sample from the 'DAF' claims. The original pit was enlarged and blasted to a depth of over 60 feet. Knud Rasmussen was contracted to do the mining development using air-track drills powered by portable Gardner-Denver air compressors. William Knutsen was manager in charge of all work. It was planned to ship the 2,000 ton sample to Burnt Island Mine for milling, but due to the closure of that property, Knutsen sold the ore to Rasmussen (Knud Rasmussen and William Knutsen, pers. comm.). Knud Rasmussen then sent 1,387 tons to the Ptarmigan Mine over the ice road in May 1991. This operation produced 1,081 ounces of gold (Tremenco Resources Ltd., 1991).

Exploration Since Mine Closure

No known work. In 1991, new claims were staked in the area by Raymond Essery. Trevor Teed staked new claims in January 2003.

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Personal communication: Knud Rasmussen; William Knutsen

	WILSON ISLAND <i>Minor Exploration (Abandoned)</i>	Gold
		61° 47' 45" 113° 10' 30"

Years of Primary Development: 1920s

Mine Development: 43' shaft, 10' shaft

Introduction

This former gold prospect is located on the south-western tip of Wilson Island, Great Slave Lake, 96 kilometers southeast of Yellowknife, NWT. The author has not visited the site and is unsure of the exact location of the mine workings.

Brief History

Claims were staked in September 1916 by Robert H. Wilson, a prospector from Tacoma, Washington. Two claims were staked: the 'Big Bear', under the name of Cassia P. McTavish, and the 'Big Moose', under Wilson's name. Mrs. McTavish, a schoolteacher in Fort McMurray, Alberta, had granted Wilson a power of attorney to stake claims on her behalf. Wilson had apparently discovered showings of gold, silver, and lead on Wilson Island. In 1919, a company was promoted into existence to develop the gold claims - Aurous Gold Mining Company Limited, based in Seattle, Washington. By 1922, this company had sunk a shaft and built a log cabin camp. Operations were placed under great pressure during the 1920s following the death of company president Robert Wilson. The company was reformed as a Canadian corporation in 1924-1925 and work resumed. Additional shaft sinking had been done up to 1928, but by this time the company was running out of finances to continue with work, citing an unwilling group of investors as primary cause. The company also faced allegations of fraudulent promotion surrounding claims of having deposits of platinum, iridium, and uranium at the Wilson Island property. No further work was being done after 1928, and the original claims lapsed in November 1934.

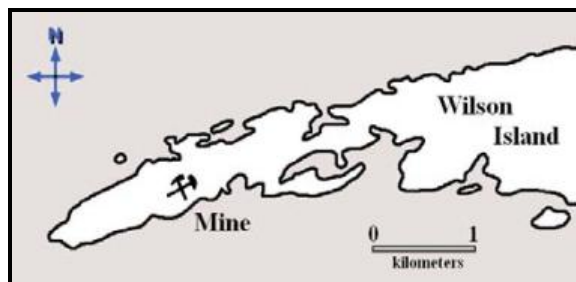


Figure 1. *Wilson Island and approximate location of the mine site.*

Geology and Ore Deposits

The showing occurs in quartzites of the Wilson Island Group. These are highly fractured, which are coated with hematite, and specularite. These iron-oxides occur in small masses up to 4 inches thick penetrating the shattered quartz and turning it red. Mineralized zones occur as quartz lenses and stringers, which strike east and west, and dip 70 to 80° to the north. They have a variable width of 3 inches to 3 feet on the surface. No free gold is visible, and pyrites occur occasionally (Meikle, 1930).

Aurous Gold Mining Company Limited (1919-1923)

In February 1919, the 'Big Moose', 'Big Bear', together with many other claims, were transferred to the Aurous Gold Mining Company Limited, based out of Washington State. During the summers of 1918-1919, Robert H. Wilson was in charge of work. Preliminary development consisted of cutting wood, building four log cabins, and trenching (National Archives of Canada). Plans were drawn up for a 100 ton per day milling plant and equipment was ordered (Meikle, 1930). Wilson died in the July 1920 in an accident on the Athabasca River while transporting freight up north for the 1920 season. He drowned while pushing the barge off a sand bar (Baker, 1976).

Crews continued to the property in 1920 despite the tragedy that befell their leader, and the fact that many of their boats (or scows) were helplessly beached along the Athabasca River or piled up on the Slave River portage. Transportation of equipment and machinery to the property was significantly delayed for the next few years.

There are records that show Ernest Ballman was in charge of some work in the summer of 1920, when a 7x7-foot shaft was sunk to a depth of 12 feet. Crews had a hard time with rock drilling because of inadequate steel-sharpening tools and unusually hard rock. (National Archives of Canada) Wilson's party, minus the deceased leader, arrived at Wilson Island just before freezeup in October 1920 and began to build a log cabin camp. Three men under W.F. Banquist remained on the island in the winter of 1920-1921 surveying the island, collecting rock samples, and cutting timber for a mill. A gold seam was described as five feet in width and was traced for 75 feet before dipping into the lake. (The Edmonton Bulletin, July 8th and July 14th 1921)

During the winter of 1921-1922, under the direction of Andrew Sundquist and Captain N.E. Warner, the shaft was deepened to 43 feet (National Archives of Canada). A short headframe was erected, and hoisting was done via a windlass. Forty-five gallon oil drums, sawed in half, were used as ore buckets. The shaft was timbered throughout and equipped with rung ladders. The shaft is located on the top of a low glaciated ridge about 300 feet from the shore. It was sunk in the hanging wall and was 50 feet north of the outcrop of the quartz vein, which had a dip of 70 to 80° to the north (Meikle, 1930). More buildings were apparently erected by this time, and a complete assay facility was available. The bulk of the mining machinery was still grounded in Fort McMurray and Fort Smith. One of the company's plans involved bringing up an oil drilling rig to test for oil in the Great Slave Lake area, and if they found enough this crude could be used to power their future mine on Wilson Island. (The Edmonton Bulletin, July 19th 1922)

In 1923, drilling began on the surface showings with Leon Bissell in charge (National Archives of Canada). No work was done during 1924 as the company began its reorganization. However, it was reported in the *Financial Post* newspaper that a shipment of ores from the property of Aurous was sent to Trail, B.C. for bulk testing (The Financial Post, July 25th 1924). Results of this work are unknown. At this time, it was reported that a small rotary mill was being shipped north but the greater part of this equipment was still at Fort Smith. Equipment on the property included a boiler, hoist, compressor, and air drill (Burwash, 1923). Positive reports from the property continued to be published by media during 1922 and 1923 and the company appeared ready to open a gold mine on Wilson Island.

Aurous Gold Mining Company of Canada Limited (1925-1928)

A new Canadian company was organized during 1924 and was given control of the Wilson Island property in June 1925. A major benefit of moving the company to Canada was the abolition of a double-tax. During the 1925-1926 summer seasons, Andrew Sundquist was in charge with drilling, stripping, and trenching operations. 1,500 feet of trenching was reported in 1926. In 1925 the company was still trying to bring mining machinery north, but the majority of it remained on the Fort Smith portage. By 1928, this equipment had still not made it to Wilson Island. The company was accused of fraudulent promotional tactics during 1926 because of their reports that claimed they had a 100 ton per day milling plant on the property. They also claimed that the property contained values of platinum, uranium, and iridium. In 1928, the last year that work was recorded, A.G. Bloomquist was in charge of work through the sinking of a 4 foot x 6 foot shaft to a depth of 10 feet (located about 250 feet southeast of the main shaft). Only three feet of lateral drifting was reported. The company reported difficult times, as shareholders were unwilling to support the company (National Archives of Canada). Aurous was on the verge of bankruptcy and could no longer hold onto the mineral claims; they lapsed in 1934.

It is interesting to note that despite the development campaign funded by the Aurous company during the 1920s, prospecting in recent years by other groups has been unable to detect the presence economic of gold on Wilson Island. Some have suggested the whole thing was a scam from the start. Mining inspector Mackay Meikle, who visited the property in 1930, reported that the shaft dump was mostly composed of waste rock with only a small amount of quartz, suggesting that the vein was not cut by underground work (Meikle, 1930). Perhaps all the quartz veining was removed as part of a bulk sample of ore.

Exploration Since Mine Closure

No known work. The property was re-staked as the 'Victory' group in 1934. An old collapsed windlass-headframe over top of the primary shaft was the only structure noted in a site investigation in 1992. This structure and any other small debris were apparently cleaned up in 1994 through a government-funded program.

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I also thank my parents, Alf and Sarah Silke. My father spent 25 years at Con Mine as a geologist. My earliest memories are of the times I would hang around his office looking at the rocks, the old maps on the wall, and admiring the stark beauty of the Robertson shaft headframe. I am proud of my families mining background, a legacy that dates back to my grandfather's time in Kirkland Lake, Ontario. Without this sense of heritage, I probably never would have acknowledged its importance.

Ryan Silke