SKIVIK HOLDINGS CO. LTD.

301 Fairway Dive North Vancouver B.C. V7G 1L4 PH:604.990.4889

SUMMARY REPORT

documenting the

COWLEY, KEEWENAW AND GEM COPPER DEPOSITS

on the

Lobo Property

YUKON TERRITORY, CANADA
NTS Map Sheets 105D 10
Latitude 60º 37' N Longitude 135º 03' W

prepared for

Lobo Del Norte Ltd
49 Redwood Street
Whitehorse Y.T. Y1A 4B2

by

W.A. Wengzynowski, P. Eng.
Skivik Holdings Co. Ltd.
301 Fairway Dr.
North Vancouver, British Columbia V7G 1L4

OCTOBER 20, 2014

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INTRODUCTION

This report has been prepared at the request of Mr. Barry Ernewein, President of Lobo Del Norte Ltd. and current owner of the claims hosting the Cowley, Keewenaw, Black Cub and Gem copper±molybdenum±gold±silver skarn deposits. The report provides a brief overview of the Whitehorse Copper Belt and summarizes the historical and most recent accounts of the mineral resources and exploration primarily at the Cowley deposit located in southern Yukon. All resource figures contained in this report are historical in nature and do not adhere to the requirements of reporting in the form of NI-43 101. The scope of this document is to demonstrate the quality, consistency and level of confidence used to generate the historical resource estimates in addition to showing the exploration potential within and peripheral to existing deposits.

PROJECT SCOPE AND TERMS OF REFERENCE

The Author reviewed historical data available from the turn of the century through the early 1980's primarily with access through the Yukon Mining Recorder's website toggle EMR Mining Assessments. Approximately thirty documents were downloaded and reviewed of which roughly half contained relevant information in association with the Cowley and adjacent deposits. The assignment included establishing a knowledge base of the historical accounts of the Whitehorse Copper Belt with a strong focus on the data used to calculate the resources for deposits scheduled for production. This portion of the review included: accumulating and cross referencing diamond drill hole data (intersection thickness and grade), review of available metallurgical work, obtain data documenting accessory mineralization such as gold and silver; and details of proposed mining plans and schedules. Most of the data reviewed at this level pertained to the Cowley, Keewenaw and Gem deposits. Assessment of the most recent exploration campaign at Cowley was also conducted. Diamond drill results from the Cowley Deposit Main Zone obtained through exploration in 2008 were examined and interpreted in the context of the historical cross section and long section data. It should be noted that the majority of the historical data is not in digital format and that only a very small proportion was converted digitally for the design of the 2008 exploration campaign.

This report is not required to have been written in compliance with disclosure and reporting requirements set forth in the Canadian Securities Administrations' current "Standards of Disclosure for Mineral Projects" under the provisions of National Instrument 43-101, Companion Policy 43-101 CP and Form 43-101F1. That stated, however, the author is a qualified P.Eng in British Columbia and has prepared numerous technical reports under the requirements of the noted instrument.

RELIANCE ON OTHER EXPERTS

The Author is not an expert in legal, tenure, environmental, permitting, resource preparation or metallurgical matters and has relied on the information and opinions of legal advisors and consultants with regard to the following:

> *Claim Information*: Data concerning the location and status of mineral claims was provided by the Whitehorse District Mining Recorder. The Author assumes that independent legal advice has been received by Lobo Del Norte

Ltd. regarding the validity and current status of the claims.
Historical and Recent Reports: Assessment reports and Open File documents are heavily relied upon for technical information in this report.

Mineral Resource Estimates: The Author has relied on Feasibility Studies prepared by reputable entities that conducted the necessary industry-standard assessment prior to commencement of production within the belt.

Mineral Processing and Metallurgical Testing: The Author has relied on the work completed by Britton Research Laboratories and Sumitomo Metal Mining Co. Ltd.

PROPERTY LOCATION, CLAIM DATA AND ACCESS

The Lobo Property consists of 27 quartz mineral claims in two separate blocks located in south central Yukon at latitude 60°37´N and longitude 135°03´W on NTS map sheets 105D/10 (Figure 1).

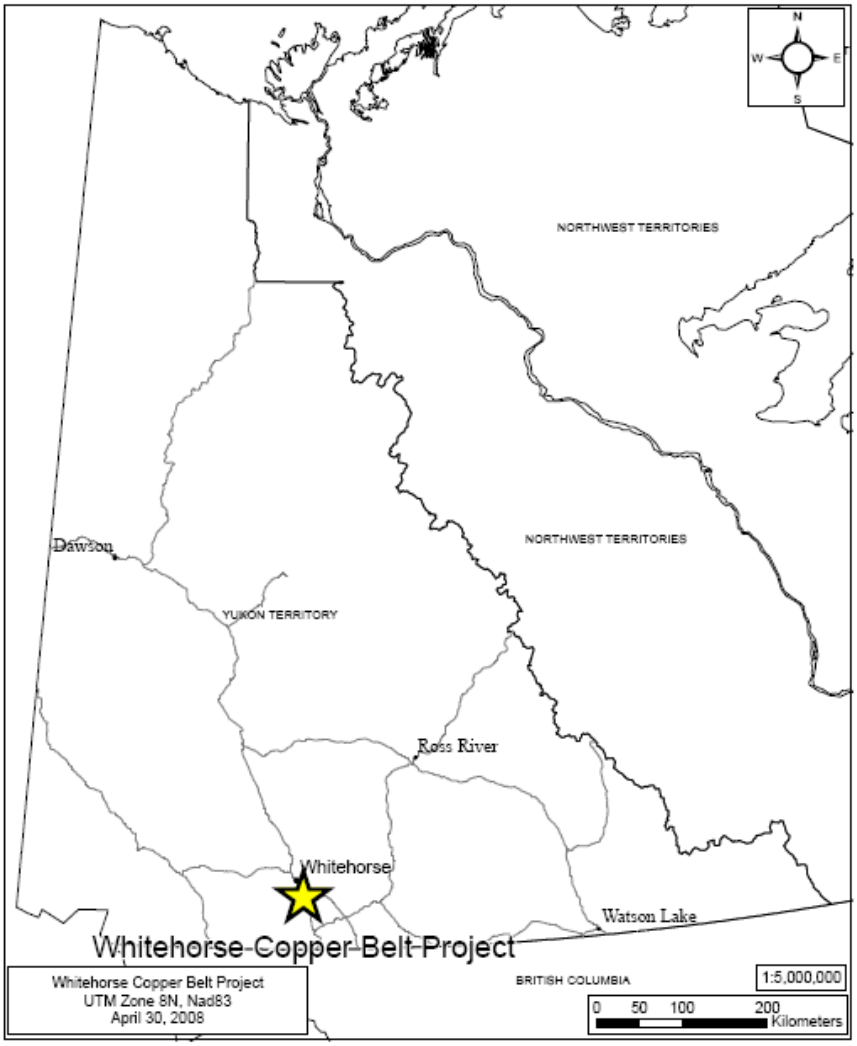


FIGURE 1 – Property Location (after Davis, 2008)

Specifics concerning claim registration are tabulated below, while the locations of individual claims are shown on Figure 2.

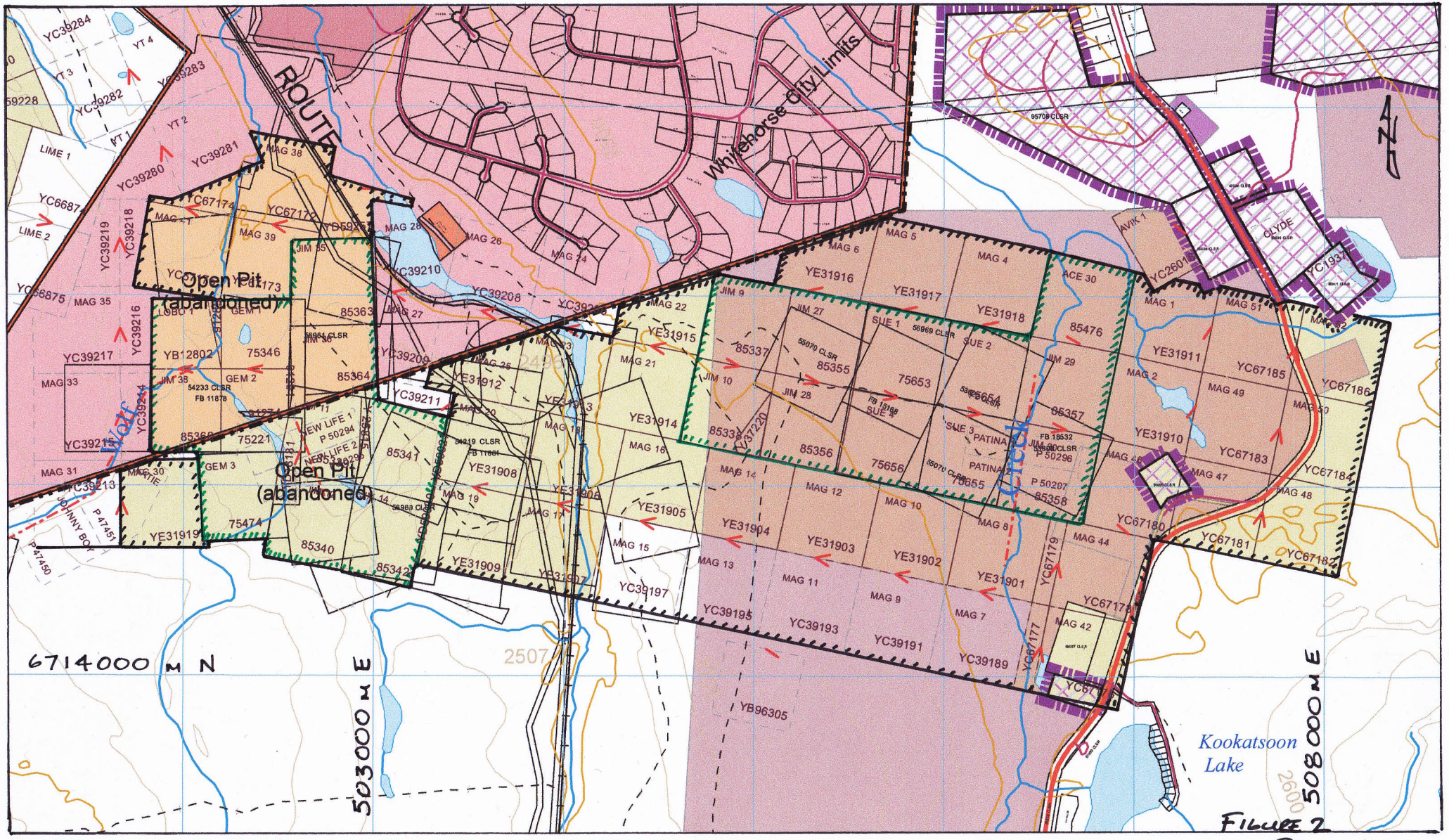
TABLE I – Claim Data

Claim Names	Owner	Grant Number	*Claim Expiry Date
GEM 2	Lobo Del Norte	75221	01/07/2017
GEM 1	Lobo Del Norte	75346	01/07/2017
GEM 3	Lobo Del Norte	75474	01/07/2017
SUE 1-4	Lobo Del Norte	75653-56	01/07/2017
JIM 9	Lobo Del Norte	85337-42	01/07/2017
JIM 10	Lobo Del Norte	85338	01/07/2017
JIM 11	Lobo Del Norte	85339	01/07/2017
JIM 12	Lobo Del Norte	85340	01/07/2017
JIM 13	Lobo Del Norte	85341	01/07/2017
JIM 14	Lobo Del Norte	85342	01/07/2017
JIM 27	Lobo Del Norte	85355	01/07/2017
JIM 28	Lobo Del Norte	85356	01/07/2017
JIM 29	Lobo Del Norte	85357	01/07/2017
JIM 30	Lobo Del Norte	85358	01/07/2017
JIM 35	Lobo Del Norte	85363	01/07/2017
JIM 36	Lobo Del Norte	85364	01/07/2017
JIM 38	Lobo Del Norte	85366	01/07/2017
ACE 30	Lobo Del Norte	85476	01/07/2017
LOBO	Lobo Del Norte	YB12802	01/07/2017
Denis 9 F	Lobo Del Norte	Y25815	01/07/2017
Dennis 1 F	Lobo Del Norte	91274	01/07/2017
Dennis 3 F	Lobo Del Norte	91289	01/07/2017
Dennis 5 F	Lobo Del Norte	91291	01/07/2017
JIM 38 F	Lobo Del Norte	Y 37220	01/07/2017

*Claim expiry dates reflect assessment credits to be filed and but not yet accepted.

Holding costs for quartz claims in Yukon can be maintained by work assessment or payment in-lieu. The cost to keep claims in good standing per year is $105 per claim to a maximum of 5 years per claim in any given year.

The Lobo property covers an area of approximately 400 hectares and the claims are registered with the Whitehorse Mining Recorder in the name of Lobo Del Norte Ltd. (B. Ernewein) which owns the property 100% with no underlying interests.



Claims owned by Lobo del Norte Ltd. 100%

Claims owned by Barry Ernewein 50%
Leonard Doust 50%

FIGURE 2

LOBO PROPERTY
CLAIM LOCATION
105 D10
WHITE HORSE Y.T.
OCT 2014

The project area is situated 16 km southeast of Whitehorse along the paved and/or chip sealed Alaska Highway (Figure 3 - below) and 2 km south of the Alaska Highway-Carcross/Skagway Highway junction.

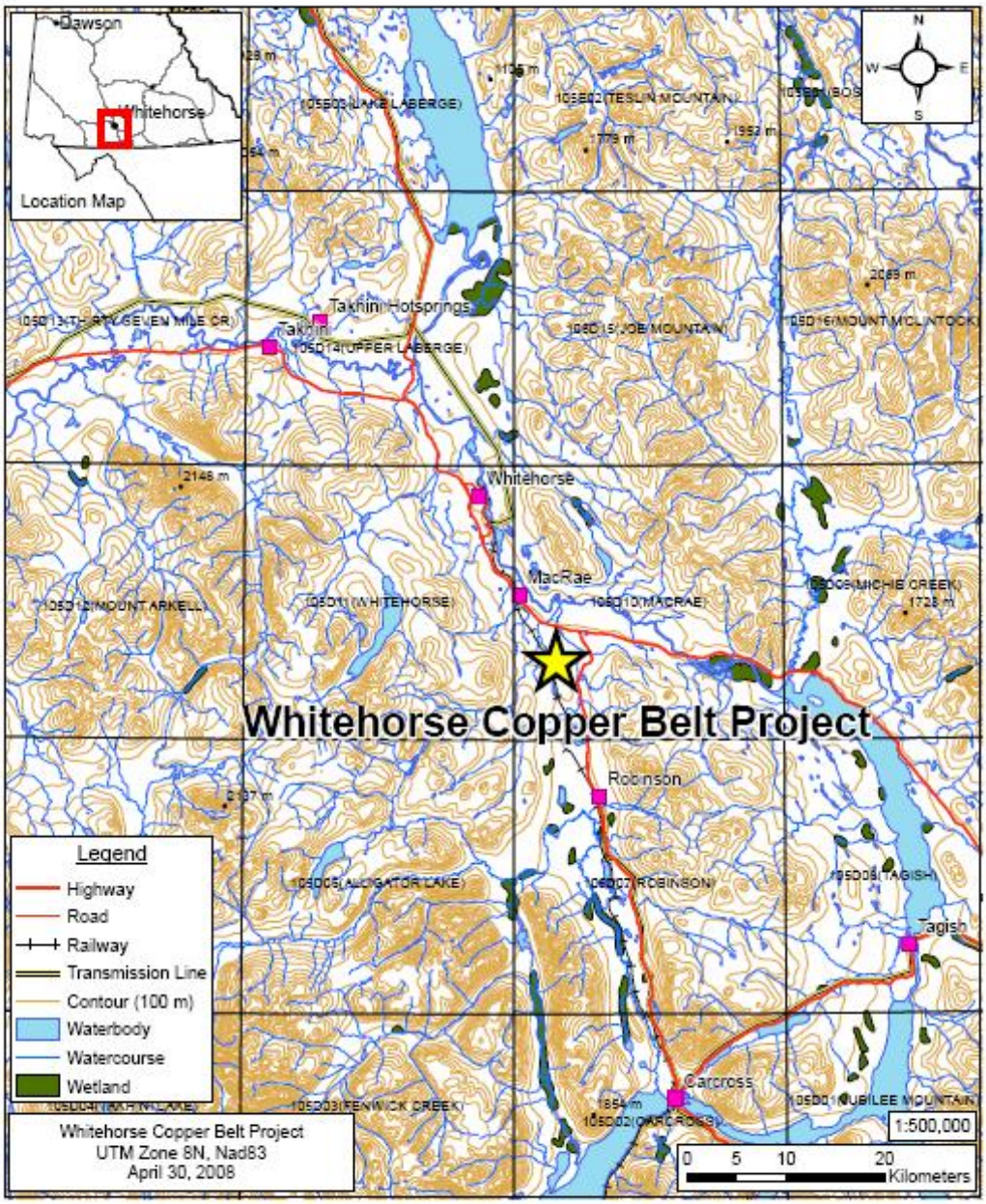


FIGURE 3 – Project Access (after Davis, 2008**)**

Gravel road access exists within 2 km off the Alaska Highway to all of the deposits as part of the historical exploration legacy from the 1960's through 1980's. These roads were utilized during the 2008 exploration program conducted by B. Ernewein primarily at the Cowley Copper Deposit.

The claim blocks are situated at the southern limit of the Whitehorse city limits in Yukon Territory. The eastern portion of the claim block (Sue, Ace, Jim) lays completely outside the city limits and covers the Cowley Copper Deposit, while the

western block (Denis, Jim, Gem, Lobo) is partially within the city limits and covers the Gem, Keewenaw and Black Cub South deposits.

Deep-water port facilities are located roughly 155 km south in Skagway, Alaska. Rail infrastructure (White Pass and Yukon Route railway) passes within 3 km of all the deposits and runs south to Skagway. The rail system was fully operational to Skagway hauling concentrate from the Whitehorse Copper Belt and other operational mines until 1982, which coincided with the last year of production within the Whitehorse Copper Belt.

Topography in the immediate area of the claims comprises gently rolling hills with well developed drainages. Locally, elevations range from 2400 ft to 3300 ft, however, all of the claims are at approximately 2600 ft. Several small lakes and water filled open pits are located nearby. The region is lightly forested having seen significant historical logging and subsequent re-growth of small black spruce, willow and alder. Overburden in the area ranges up to ~10 m but outcrop is visible in a number of areas.

HISTORY AND PREVIOUS WORK

Historically, in Yukon, the most significant production of copper was from the Whitehorse Copper Belt from which production is reported at >10 Mt grading approximately 1.5% Cu with significant gold and silver credits in several of the deposits (Watson, 1984). The Whitehorse Copper Belt comprises over 30 copper-related primarily skarn occurrences along a 35 by 5 km north westerly trending arc (Heon 2004, Tenney 1981). The belt of mineralization sits both within and outside the current city limits of Whitehorse to the west. Much of the historical technical information presented in the summary is derived from Tenney (1981), Watson (1984) and Deklerk & Traynor (2004). The first two mentioned references, Watson and Tenney, are contained in the report as Appendices I and II.

First discovered in 1897, the belt of rocks had seen considerable attention in two distinct phases. The first phase of exploration and mining occurred in the early 1900's in which time nine of the higher grade showings were mined by small-scale underground methods (such as the Copper King and War Eagle sites). This mining activity declined into the late 1920's.

Starting in the early 1960's, the second phase of exploration began focusing on the larger lower grade deposits by open pit mining methods at the Arctic Chief, Little Chief and Keewenaw deposits. The last mining activity in the region ceased in 1982 with the closer of the Little Chief mine. A number of technical and non-technical publications have been produced on the history of the Whitehorse Copper Belt (e.g., Dobrowolsky & Ingram 1993, Hart & Radloff 1990, Heon 2004, MacKay *et al.* 1993, Tenney 1981, Turner *et al.* 2003, and Watson 1984)

The Cowley Copper Deposit (also known as the Dugdale Showing, MINFILE 105D059) sits at the southern end of the Whitehorse Copper Belt and had reached feasibility stage before operations in the belt were shut down in 1982 due to a declining economy and copper prices. Cowley was discovered earlier in the century but received little work until the mid 1960's when the exploration resurgence in the

belt was rekindled. Ground based magnetic and Induced Polarization (IP) geophysical techniques were conducted in the area defining cohesive exploration targets in conjunction with the historical occurrences. Diamond drilling was carried out in the 1960's loosely defining the Main Zone mineralization and more thorough drilling was conducted in the early 1970's culminating in a total of ~125 holes and ~11,500 meters of core (e.g., Hureau, 1981). Approximately 884,000 tonnes of potential ore at the Main Zone (at 1.04% Cu, 3.77 g Ag/t, 0.21 g Au/t, and 0.066% MoS2) and 668,000 tonnes at the South Zone (at 0.90% Cu) were calculated (Tenney, 1981; Watson, 1984). MacKay et al. (1993) stated that "except for the minor underground work in the early 1900's and a 5,000 tonne bulk sample in the 1970's, Cowley Park was never mined and remains the largest open-pittable reserve in the Copper Belt today."

Three other deposits with reported production and reserves are located 4 km west of the Cowley Copper Deposit and are also currently owned by Lobo Del Norte. These deposits, however, are situated with the current boundary defining the City Limits of Whitehorse and are subject to discussions regarding further work within this boundary.

The Gem deposit is the northern most of the three and was the only deposit with no reported production. It was discovered in 1967 by geophysical methods which guided the initial diamond drilling. Subsequent exploration defined an historical reserve estimate of 625,000 tonnes grading 1.01 % Cu (Watson, 1984). The Black Cub South open pit deposit saw advanced exploration by geophysical surveys and diamond drilling and produced 180,000 tonnes of copper ore grading 1.33 % Cu with gold, silver, and molybdenum credits. There is an estimated 20,000 tonnes of ore at 1.25 % Cu remaining. The Keewenaw Deposit is the westernmost outlier of this southern group and received enough geophysical surveying and diamond drilling to advance this area to a production stage. It saw production of 159,000 tonnes of ore at 0.95 % Cu with an estimated 202,000 tonnes of ore at 1.06 % Cu remaining with additional gold, silver, and molybdenum credits. All production and resource data has been taken from the most recent compilation presented by Watson, 1984. The Gem and Keewenaw deposits will be discussed in more detail in the body of this report.

WHITEHORSE COPPER BELT SKARN GEOLOGY

Skarn deposits within the Whitehorse Copper Belt are considered exoskarns that formed within 150 m of the mid Cretaceous calc-alkaline Whitehorse Batholith contact (Figure 4); however, a number of endoskarns are documented within the intrusion as well. Intrusive rocks are largely dorite to granodorite in composition and have locally intruded upper Triassic limestone and calcareous metasediments of the Lewes River Group. As noted by the Yukon Geological Survey, most skarn development occurs as lensoid or tabular bodies coincident with bedding and range in geometry from steeply dipping to flat lying bodies. Two main types of skarn deposits are present:

1) Iron-rich, in which copper occurs with magnetite, serpentine, specularite, talc, chlorite and occasional pyrrhotite and pyrite.
2) Iron-poor, which consist of a wide variety of contact minerals.

Bornite and chalcopyrite are the main copper minerals although chrysocolla, native copper, chalcocite and secondary minerals are present in small amounts. The copper minerals occur as disseminations, blebs, pods and stringers that clearly postdate the skarn minerals. Bornite predominates in the iron-rich skarns and is slightly more abundant than chalcopyrite in the iron-poor skarns. Silver grades are generally proportional to the copper grades while gold tends to be more erratically distributed in the system.

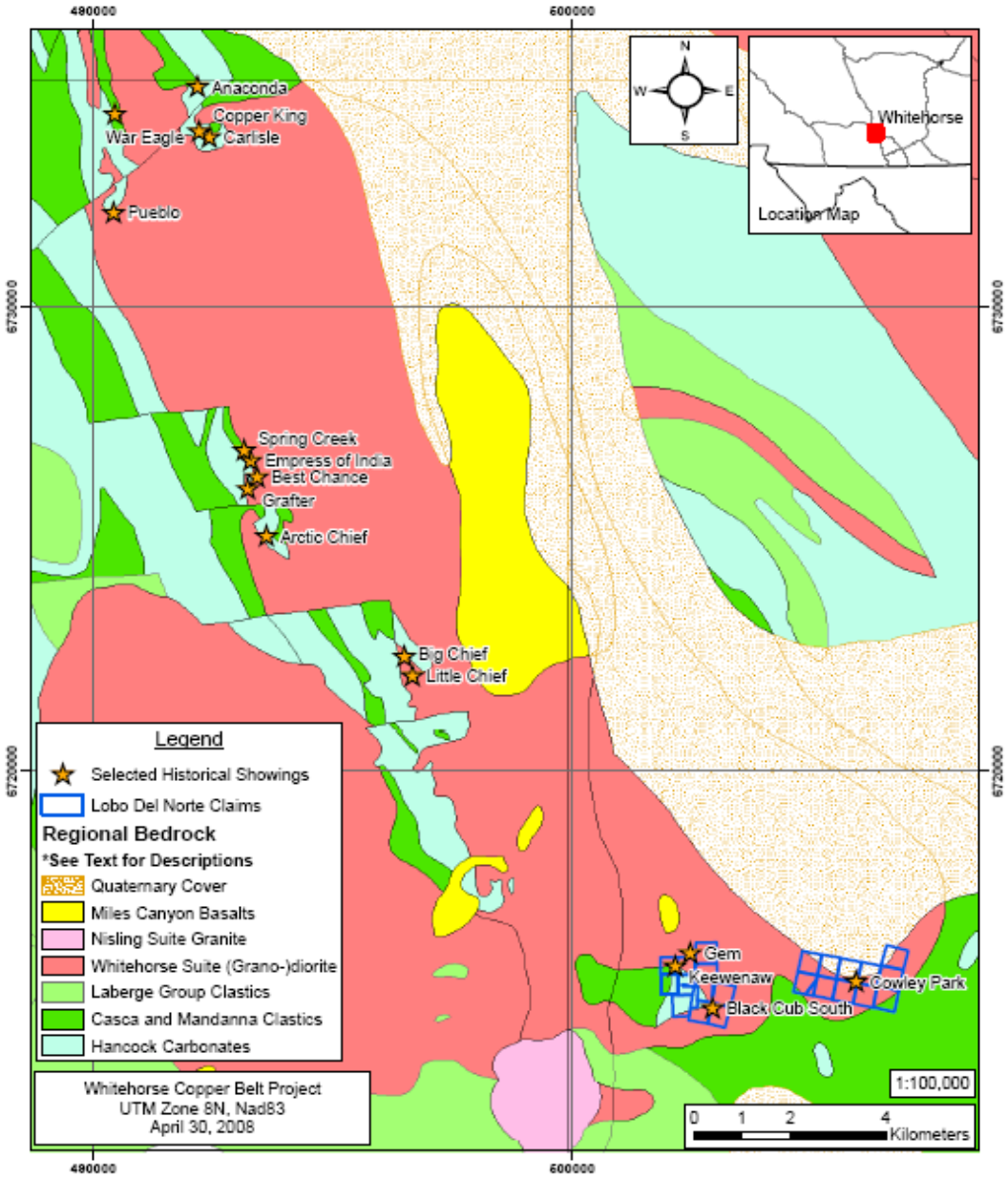


FIGURE 4 – Property Geology (after Davis, 2008)

Table II highlights the production and reserve data for the deposits within the Whitehorse Copper Belt as reported by Watson, 1984 as part of the Open File publication:

Table II - Whitehorse Copper Belt Production and Reserves

Deposit	**Production (tonnes)**	**Reserves (tonnes)**	**Cu (%)**	**Ag (g/t)**	**Au (g/t)**
Little Chief	8,536,400	-	1.50	9.10	0.75
Arctic Chief	201,800	-	1.44	17.10	1.03
Best Chance	-	447,000	0.71	NA	0.03
War Eagle	899,000	-	1.25	8.57	0.22
Pueblo	127,000	-	3.5	NA	0.17
Copper Deposits Currently owned by B. Ernewein					
Cowley Park	-	884,000	1.04	3.77	0.21
		668,000	0.90	NA	NA
Gem	-	625,000	1.01	NA	NA
Keewenaw	159,000	202,000	1.01	NA	NA
Black Cub N.	-	156,000	0.82	NA	NA
Kodiak Cub	-	57,000	1.18	NA	NA
Black Cub S.	187,000		1.33	12.34	0.24
Black Cub S.		20,000	1.25	NA	NA

Note: Grades reported for combined production and reserves are weighted averages. Production and reserve figures reported are not NI 43-101 compliant.

All production was from six open pits at the above-mentioned deposits plus underground development at Little Chief.

COWLEY COPPER DEPOSIT

Geology and Resource

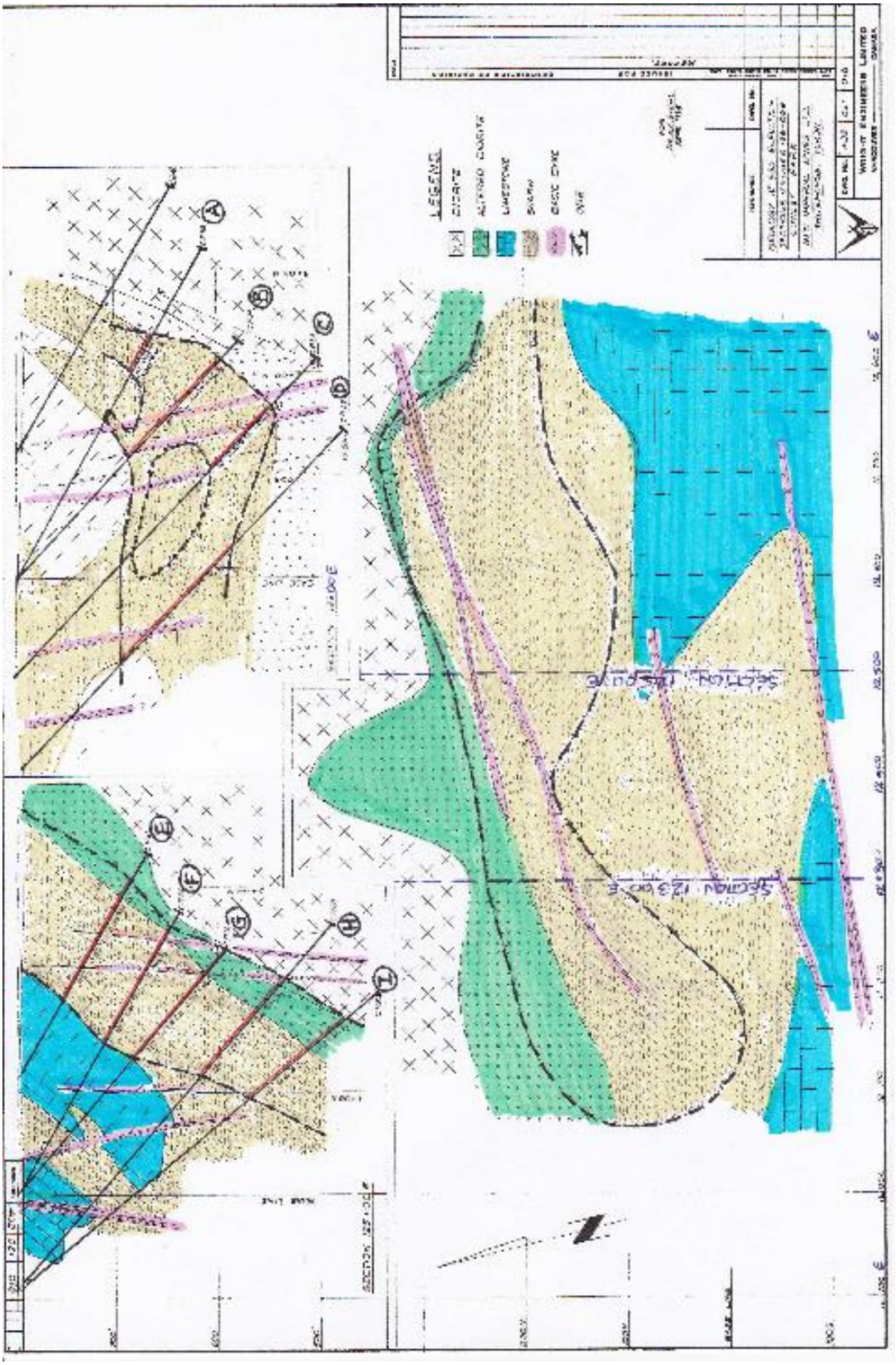
The largest undeveloped resource of the Whitehorse Copper deposits is the Cowley Copper Deposit, situated at the south end of the belt. The Main Zone is hosted by a steeply dipping 300 m long tabular skarn lens comprising typical silicate skarn assemblages of garnet, diopside, actinolite, tremolite/wollatonite with disseminated chalcopyrite and bornite plus lesser magnetite and serpentine (Tenney, 1981). The affects of metasomatism moving progressively distal from the intrusion are denoted by unmineralized garnet skarn and recrystallized limestone. The mineralized horizon outcrops along the northern edge of the east southeast trending Main Zone and is interpreted to flatten at depth.

An historical interpretation of the 200 m (650 foot) level plan in 1965 is shown on Figure 5 on the following page with two drill sections through the mineralized Main Zone. The sections illustrate both the complexity in portions of the skarn as well as the consistent tabular nature. Table III lists the copper and molybdenite assay intervals for sections 123+00 E and 125+00 E as shown on the interpreted level plan.

Table III – Assay Data for 123+00 E and 125+00 E Main Zone

Section 123+00 E		**Interval (m)**	**Cu (%)**	**MoS2 (%)**
A	DDH CP 33	13.72	0.86	0.045
B	DDH CP 34	41.45	0.87	0.089
C	DDH CP 36	26.52	0.53	0.056
D	DDH CP 39	48.46	0.58	0.071
Section 125+00 E				
E	DDH CP 21	41.15	0.50	0.116
F	DDH CP 24	39.62	1.40	TR
G	DDH CP 26	41.76	0.83	0.125
H	DDH CP 28	38.40	1.27	0.096
I	DDH CP 30	36.88	0.61	0.092

(Wright Engineers et al., 1965)



(After Wright Engineers et al., 1965)

FIGURE 5 – 650 Ft Level Geological Plan with Drill Sections along 123+00 E and 125+00 E

Feasibility

In 1965 a feasibility study was prepared by Wright Engineers Limited and J.A.C. Ross and Associates Ltd. Mining Consultants for a 2000 ton per day concentrating plant and related services in consideration for copper production from six deposits within the belt. The feasibility study is contained in Appendix III. Economic analysis was calculated using four copper prices ranging from $0.28 to $0.45 per pound.

Pit design and ore reserve calculations were conducted first in 1965 and again in April 1971 using cut-off grades of 0.50% and 1.00%. Of particular interest was the molybdenite content of the ore body reported at 0.076% MoS2. At the time, however, molybdenum was not considered of economic interest and this commodity was not configured into the economic assessment of the deposit.

Infill diamond drilling was implemented at 17 m intervals within the Main Zone in 1971 between the 30 m spaced sections defining the near surface portion of the deposit. The results of the infill drilling were purportedly disappointing having showed higher discontinuity in the mineralized zone at the cut-off grades originally utilized. Subsequently, the pit design was reconfigured for a lower tonnage than previously calculated. Table IV illustrates the progression of calculations conducted toward the most current figures reported for the Cowley Copper Deposit.

Table IV – Cowley Copper Open Pit Reserve Figures

Date	Tonnes*	Cu %	MoS2 %	Ag g/t	Waste/Ore Ratio	Cut-off Grade (%Cu)
Nov 1965	1,147,400	0.86	0.091			
Jan 1971	438,331	1.20			5.22 : 1	1.00
Apr 1971	1,110,945	0.87			1.51 : 1	0.50
May 1971	885,298	0.89			2.31 : 1	0.50
May 1979**	836,622	0.93	0.066	3.77	1.995 : 1	0.50

* 15% dilution included in tonnage estimate
** Pit optimization figures reported by Tenney, 1981

Ground conditions at Cowley are reported to be generally quite good and diamond drilling was considerably easier at this particular locale due to the lack of pervasive fracturing observed elsewhere in the district. A total of 125 drill holes were located to define the current resources utilizing approximately 11,500 m of drilling.

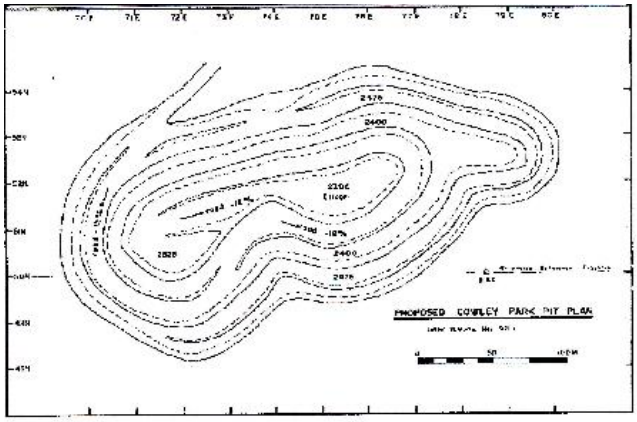
Table V – Cowley Copper Total Reserves (Watson, 1984)

Category	Tonnes	Cu (%)	MoS2 (%)	Ag (g/t)
Open Pit	836,622	0.93	0.066	3.77
Underground	668,000	0.90	unknown	unknown
Total	1,504,622	0.916	unknown	unknown

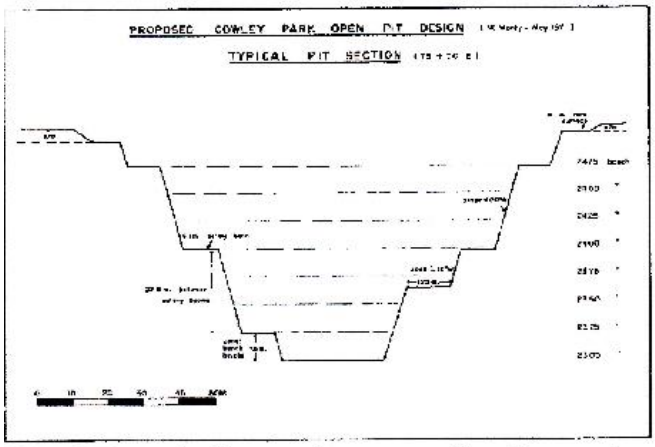
The open pit resource figure reported by Tenney in the 1981 Bulletin is lower than the resource figure reported by Watson in the 1984 Open File by roughly 50,000 tonnes. The reason for this is unexplained.

The initial open pit design for the Main Zone of the Cowley deposit prepared by Ross & Associates in 1965 was reconfigured in 1971 to account for the infill drilling and pit optimization assessment conducted by Tenny. Figure 6 shows a reconfigured pit design prepared in 1971 using the following specifications:

1. benches 7.5 m high
2. 23 m between safety berms and berms 4.5 m to 6 m wide
3. gradient of pit walls 400% to 500%
4. roadways 10 to 12 m wide with gradient 8 to 12%
5. waste/stripping ratio as low as possible and generally not greater than 3:1 waste:ore
6. overburden is stripped to form bare rock surface 6 m wide around the perimeter of the pits
7. slope of the overburden variable but approximately 30 degrees.



Proposed 1971 Cowley Park pit by W. Marty. Road gradients are not normally as steep as 12%.



Typical open pit cross-section (proposed 1971 Cowley Park pit by W. Marty) showing width of safety berms and roads, and slope of walls.

FIGURE 6 – Cowley Open Pit Design (after Tenney, 1981)

Metallurgy was carried out on drill composites by Britton Research Laboratories in Vancouver BC and by Sumitomo Metal Mining Co. Ltd., Tokyo. The results of this work stated that the ore from all six deposits has similar grinding and metallurgical characteristics and the copper, gold and silver could readily be concentrated by conventional flotation of copper sulphides. The grade of the copper concentrate is said to be largely dependant on the bornite content of the mineralization being treated. At the Cowley Deposit the copper concentrate averaged 36.5% at an average recovery of 87%. Feed grades for copper varied between the tests from 0.92% to 0.80%. Preliminary molybdenite flotation tests at Cowley indicate that 65% of the MoS2 can be recovered in a concentrate grading 90% MoS2 (Wright Engineers et al., 1965).

Recent Exploration

B. Ernewein (Lobo Del Norte) purchased the claim blocks hosting the Cowley, Keewenaw, Gem and Black Cub deposits from Hudson Bay Mining and Smelting Company Ltd in 1998. These particular claims have a long standing legacy as they have been actively held and maintained in good standing for nearly a century.

Very little work was done within the belt after the closure and decommissioning of the Whitehorse Copper mine site with the exception of some cursory ground work by private individuals who had acquired claims as they became available. During the winter of 2008, Lobo Del Norte optioned the property to Yankee Hat Minerals Ltd., a listed company on the TSX Venture exchange. The company worked on historical database digitization and integration in order to identify strategic targets for ground

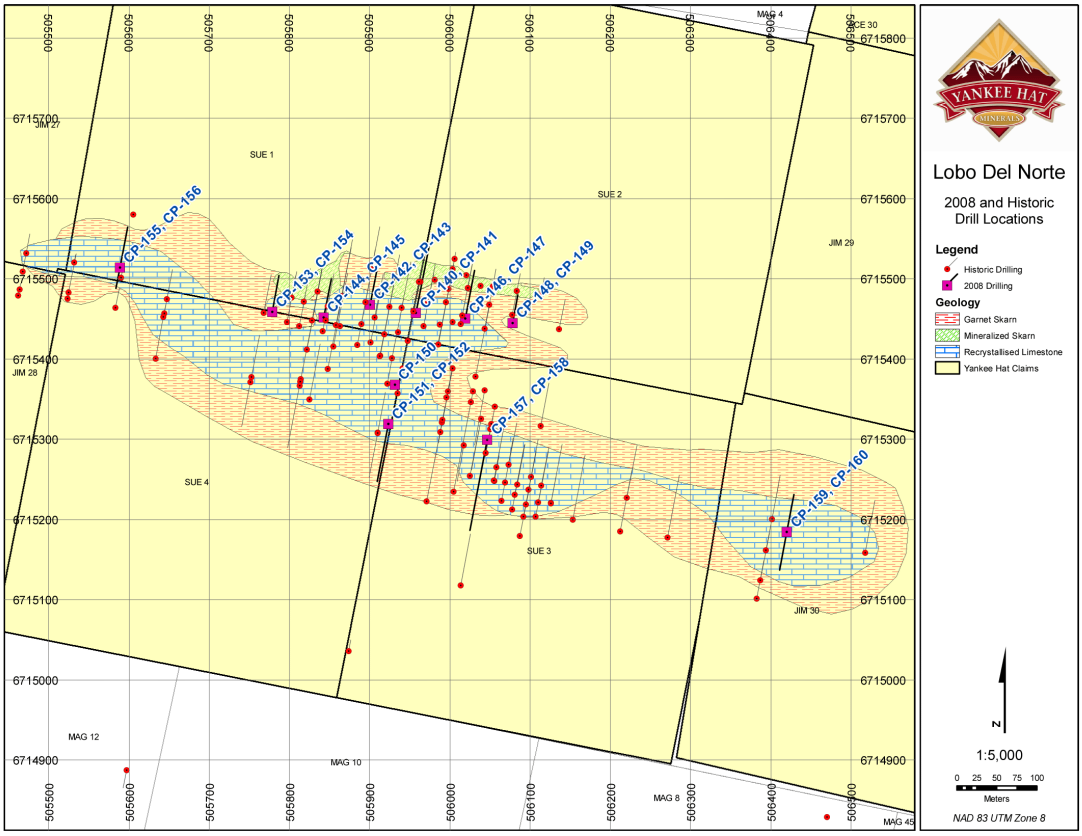


FIGURE 7 – Cowley Drill Plan (after Davis, 2008)

geophysical surveys and diamond drilling within the Main Zone of the deposit. A total of 2134 m of diamond drilling was completed in 21 drill holes as shown on Figure 7 on the previous page. Most of the drill holes were located along the 300 m core portion of the Main Zone on five section lines (Davis, 2008).

Only the core from the first five holes was processed and analysed due to time and monetary constraints. In August 2008, global economic conditions were such that Yankee Hat Minerals Ltd. did not fulfil the required obligations of the agreement and subsequently returned the property to Lobo Del Norte. The remaining sixteen holes are reported to contain various amounts of mineralization and remain unprocessed in a storage facility in a Whitehorse. Diamond drill holes DDH CP-140 through 144 were located on three section lines spaced approximately 60 m apart separated by a total distance of 120m. Two drill holes were collared from each site with the first of the two holes inclined at -45 or -50 degrees drilling to the NNE and the second hole drilled vertically. The three drill sections were designed to test in the vicinity of the core portions of the copper (grade) x (thickness) interpretation as per the 1979 longitudinal section (Figure 8).

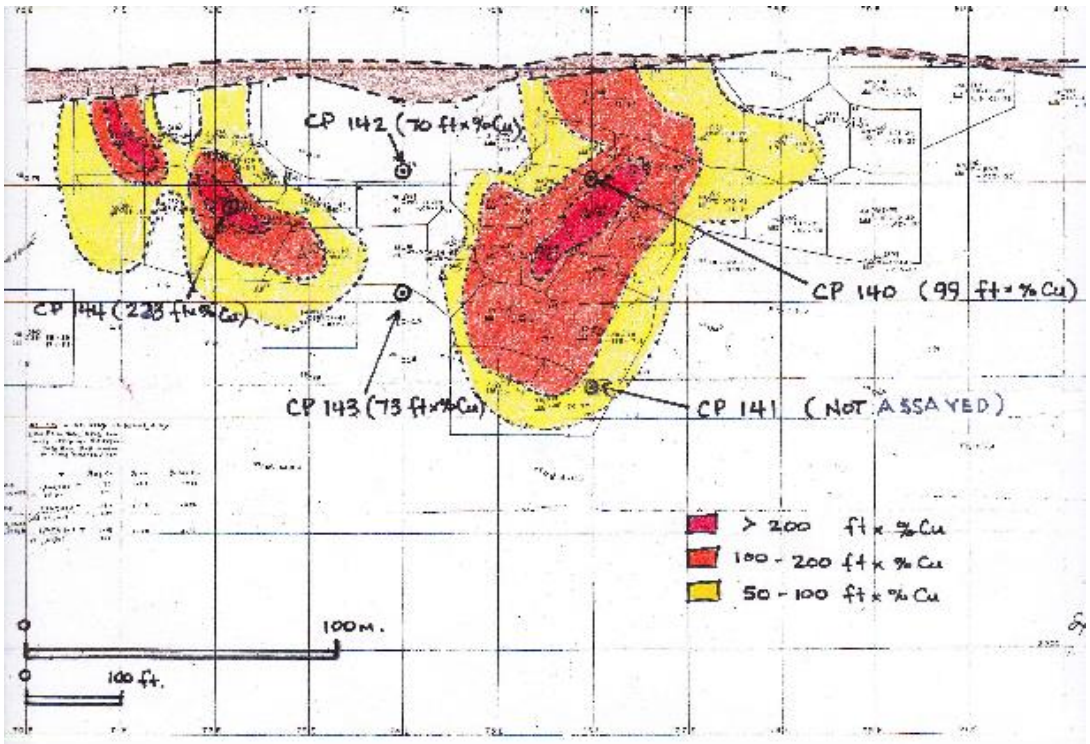


FIGURE 8 – Cowley Longitudinal Section, 1979 (after Hureau, 1979)

A summary of the drill results for each of the sections is documented on Table VI.

Table VI- 2008 Diamond Drill Results

Section	Hole #	From (m)	To (m)	Width (m)	Cu (%)	Au g/t	Ag g/t	MoS2 (%)
CP 140/141	CP-08-140	36.33	44.20	7.87	3.85	-	3.95	0.174
		55.03	64.01	2.34	5.93	-	11.31	0.280
CP 140/141	CP-08-141	41.96	42.96	1.00	2.65	NA	-	0.651
CP 142/143	CP-08-142	48.77	57.43	8.66	2.47	-	14.21	0.066
	including	50.29	50.43	0.14	47.10	-	324	0.038
CP 142/143	CP-08-143	44.74	64.86	20.12	1.12	-	-	-
CP 144/145	CP-08-144	33.98	72.55	38.57	1.76	-	7.15	0.076

Drill results reported in the logs and summaries by Yankee Hat personnel for the five holes processed in 2008 describe two to three skarn horizons comprising the Main Zone mineralization with the lower skarn horizon generally containing the best copper grades. It should be noted that narrow intervals of high grade bornite bearing material were encountered in several holes and where assays purportedly exceeded 47% Cu and 300 g/t Ag.

Concentrations of high grade molybdenum were also encountered in a number of drill holes. Some of these intercepts lie on adjacent sections and have not yet been processed.

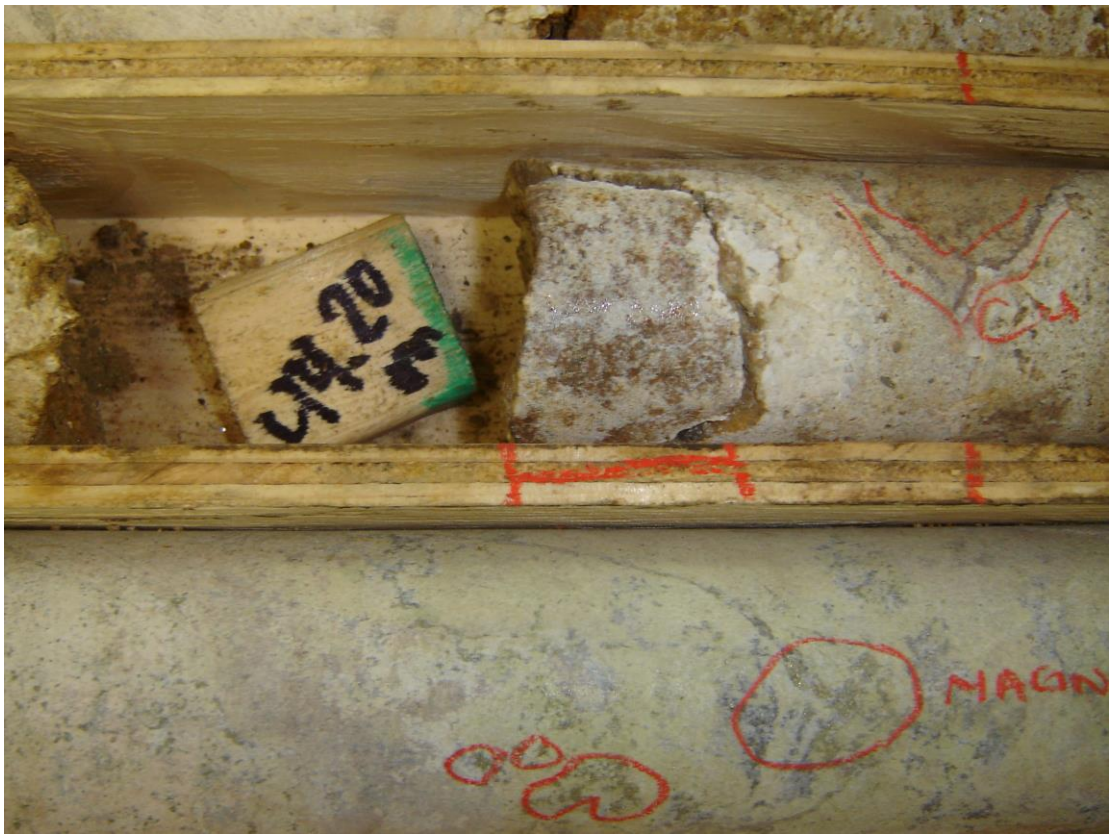
Section CP 140/141 – tested the eastern lobe of the longitudinal copper zone illustrated on Figure 8. Piercement CP 140 plotted near the core portion of the inner grade shell yielding 99 (ft)(% Cu) while the piercement for CP 141 near the base of the grade shell was not analysed. An upper mineralized skarn interval assayed 0.70 % Cu across 1 m sampled but the remainder of the interval down-hole was not sampled. In particular the material between 55 and 82 m was noted to contain visual pyrite, chalcopyrite, molybdenum and bornite as interstitial blebs and veinlets in various amounts. This particular interval should correspond to the base of the grade shell.

Section CP 144/145 – tested the western lobe of the longitudinal copper zone. Only the first hole drilled from this site was processed and its piercement is coincident with the high grade central portion of the lobe yielding 223 (ft)(% Cu).

Section CP 142/143 – tested the low grade gap between the two lobes defining the Main Zone mineralization. Surprisingly both holes encountered weakly and moderately mineralized skarn intervals which yielded 70 and 73 (ft)(%Cu), respectively. Both piercements suggest the lower grade normalization shell should actually connect the lobes into a continuous zone of >50 (ft)(% Cu). This could have implications on the pit design if the gap between the two lobes contains significantly more copper grade than historically interpreted.

The following photos are from holes that were processed within the main part of the zone as well as holes that remain unprocessed and stored in a facility in Whitehorse.

PHOTO 1: CP 144



"Typical pale cream colored calc-silicate magnetite skarn with blebby pyrite and chalcopyrite" "The photo is part of the interval processed that returned ***1.76% Cu, 7.15 g/t Ag and 0.076% MoS2 across 38.57 m".***

PHOTO 2: CP 142



"High grade massive bornite plus bornite and chalcopyrite-pyrite as blebby masses and disseminations. The narrow interval shown returned ***47% Cu, 324 g/t Ag and 0.038% MoS2***".

PHOTO 3: CP 146



"Central portion of the Main Zone – cream colored calc-silicate skarn with massive molybdenum veins and patches. ***This core has not been processed***".

PHOTO 4: CP 149



"Late stage high grade bornite veins associated with quartz and calcite also containing blebby pyrite and chalcopyrite. This interval appears to be localized and ***none of the drill hole has been processed***".

KEEWENAW AND GEM COPPER DEPOSITS

While the primary focus of this report is a review of the Cowley Copper Deposit, the following descriptions of the Keewenaw and Gem deposits have been included to provide a sense of perspective for the mineralization defined along strike to the west and the exploration potential between the series of deposits.

Keewenaw Deposit

In 1965, the current land owners – New Imperial Mines Ltd., contracted Wright Engineers Limited, J.A.C. Ross and Associates Ltd and Dr. A.C. Skerl P.Eng to conduct a feasibility study for the six main deposits in the belt. These deposits included Little Chief, Arctic Chief, War Eagle, Cowley Park, Best Chance and Keewenaw.

As part of the feasibility study, geology and ore reserves were assessed; mining methods proposed including pit designs and underground development; a proposed mining schedule and metallurgical tests were conducted by Britton Research Laboratories of Vancouver, Sumitomo Metal and Mining Company of Tokyo and by the department of Mines and Technical Surveys of Ottawa. Excerpts from the feasibility study have used to describe deposit characteristics in detail.

"The Keewenaw deposit is located on the east side of Wolf Creek in an altered doritic phase of the main intrusion at its contact with the local limestone. It is cut by a swarm of intermediate and basic dykes that make the interpretation of the ore shape difficult. Faulting is also present that adds further complication."

"Most of the mineralization consists of bornite scattered through the dioritic rock. Deep oxidation is present in some parts of the deposit with the copper in the form of chrysocolla." The deposit is defined by 34 diamond drill holes drilled on 50 foot (15.24 m) centres and has a circular surface expression roughly 100 m across tapering at depth. "The estimated sulphide ore resource was calculated at 380,000 tons (344,734 tonnes) grading 1.10 % Cu with a stripping ratio of 0.75 yards of waste to 1 ton of ore."

Ross and Associates found that in their designs for the open pits, a number of the deposits allowed for suitable dilution while maintaining reasonable grade control. The dilution or "as mined" figures reported for the Keewenaw are as follows:

Table VII – Keewenaw Feasibility Study Copper Reserve Data

Deposit	Tonnes	Cu %
Keewenaw	440,152	1.05
Keewenaw	530,871*	1.05

* Oxide component included in reserve figure.

It should be noted that approximately 91,000 tonnes of near surface copper oxide ore was not included in the historical reported reserve as it was deemed unrecoverable at the time. The advent of oxide processing is common place in the modern mining practice, therefore the adjusted deposit reserves are stated in the table above. Much of the oxide ore is believed to be contained in the bench adjacent to the current pit.

A metallurgical flowsheet was developed from test work carried out on drill core composites samples from each of the six ore bodies. The test work for the Keewenaw composites was completed by Britton Research Laboratories of Vancouver B.C. and Sumitomo Metal Mining Co. Ltd. of Tokyo Japan. The conclusions from this testwork are as follows:

"The ore from the six deposits has similar grinding and metallurgical characteristics and the copper, gold and silver can be readily concentrated by flotation of the copper sulphides. The grade of the copper concentrate produced is mainly dependant on the bornite content of the ore being treated but is expected to average about 35% at an average recovery of 85%."

"Gold recoveries will range from 30 to 85% although it is difficult to predict from the testwork due to the small concentrations present. Silver recoveries will range between 69 and 90%."

Table VIII illustrates the average grade, recovery and concentrate values obtained by Britton and Sumitomo from deposit core composite material.

Table VIII – Metallurgical Grade and Recovery Data for the Keewenaw Deposit

Deposit	Cu feed %	Cu Concentrate (%)	Cu Recovery %	Au Recovery g/t*	Ag Recovery g/t*
Keewenaw	1.35	41.5	77.5	0.84	6.4

* Results from tests only performed by Britton Research

The pit designs prepared by J.A.C Ross and Associates Ltd. are described in detail below as per excerpts from the 1965 Feasibility Study:

"Studies have been made of each pit including such pertinent geological features as faults, types of rock, shape of the ore bodies, etc. Such factors have been considered in preparing the various designs. This has resulted in different overall wall slopes, widths of berms and other design features so that all pits are not designed to a uniform set of conditions.

"In general the rock conditions are very competent, which factor, coupled with the relatively small zones should permit mining at pit slopes steeper than normally considered in open pit operations."

"A total of nine pits have been designed in varying degrees of detail. The Little Chief, War Eagle, Cowley Park and Keewenaw have been designed complete with berms, roads, etc. as these operations contain a major portion of the ore reserves." Figure 9 shows the detailed pit design for the Keewenaw Deposit.

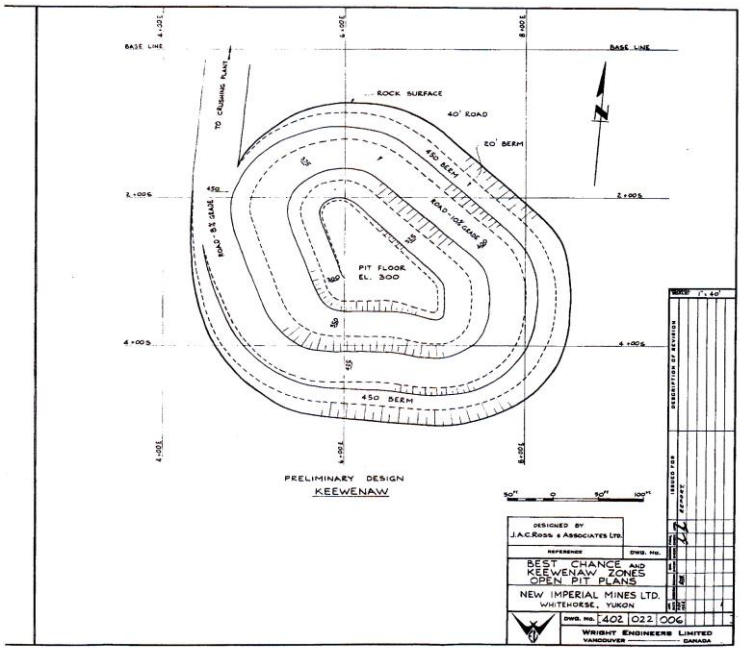


FIGURE 9 – Keewenaw Open Pit Design (after Wright Engineers et al, 1965)

The 1965 feasibility study scheduled the first phase production of the Keewenaw deposit in the 7th year of operations at a rate of 136,000 tonnes for that particular year at an estimated average mine grade of 1.05% Cu. Production commenced in 1971 from the open pit and was reported at 159,000 tonnes at an average grade of 0.95% Cu. The remaining resource at Keewenaw as per the 1965 data is documented at 281,152 tonnes at a grade of 1.05% Cu in the "as mined" category accounting for dilution. The resource figures reported by Watson in the 1984 Open File do account for the "as mined" dilution and are approximately 79,000 tonnes lower.

Gem Deposit

The documentation of the Gem Deposit is far less detailed than that of the Keewenaw Deposit and only a few notes are available for review. However, diamond drill logs collected from the historical New Imperial Mines Limited database representing the Gem mineralization are complete. They indicate the deposit was discovered in 1967 and delineated between 1968 and 1970 with approximately 63 drill holes. This drilling culminated in a resource calculation documenting 625,000 tonnes at a grade of 1.01% Cu. The surface expression of the deposit was stripped in preparation for possible production but no further development was conducted beyond this point.

A status report for the Gem Deposit was prepared by D. Tenney of Whitehorse Copper Mines Ltd. in 1980. The report suggested further work was required for pit optimization to necessitate higher allowances for mine dilution while maintaining grade. It was also suggested that systematic gold and silver analyses be conducted as these particular co-products could add significant value to the deposit economics and potentially provide the buffer required for dilution in the mine phase. Results for these analyses, however, were not located in any of the historical literature.

EXPLORATION POTENTIAL

A Mineral Potential Map was generated by Gartner Lee Limited as part of an initiative conducted by the Yukon Geology Program in 1997 to assess the mineral potential within the Whitehorse City limits boundary. The following excerpts describing the methodology used to evaluate the mineral potential of this specific area has been taken from liaising documents between the Yukon Geology Program and the City of Whitehorse. These documents along with the Mineral Potential Map and the areas ranking is contained in Appendix IV.

"This mineral assessment was undertaken for the City of Whitehorse to provide them with a tool to incorporate mineral resource values in their land use planning. A review of the available geoscientific information included: existing geological maps, government reports and assessment reports. Private archival documents (maps, exploration summaries, etc) were provided by Hudson Bay Exploration and Development (HBED), the "then" current owners of the old Whitehorse Copper Mines (WCM) ground and the largest land holders in the Copper Belt."

"On October 16, 1997 an expert panel was convened to assess the relative mineral potential of tracts within the City of Whitehorse. Members of the panel were: Dave Tenney, former chief geologist for Whitehorse Copper Mines; Gerry Bidwell and Robert Stroshein, both employees of HBED at the time of that company's exploration activity in the Copper Belt; Craig Hart, project geologist with the Yukon Geology Program. All of these participants have extensive expertise with the geology and mineral deposits of the Whitehorse area. Other panellists included Jeff Mitchell, geophysicist contracted through Amerok Geosciences to evaluate the geophysical database and Daniele Heon, mineral assessment geologist with YTG. Andy Hureau, former exploration geologist for WCM, was reached by phone at the end of the session and was asked to comment and to supply additional information."

"The task of the panel was to divide the study area into tracts of similar geology and then rank those tracts on the basis of their relative potential to host metallic mineral deposits. Information relevant to each mineral deposit/area was reviewed. The study area was then divided into tracts which are numbered from 1 to 36. Tract definition was influenced by the types of mineral deposits occurring or expected to occur in each area. Since skarn deposits are the most significant targets within Whitehorse, tracts were drawn overlapping the prospective contact and allowing for the uncertainty of the location of that contact. Through the ranking process, the tracts were grouped in 8 categories of mineral potential. Categories 1 to 3 represent ground of very high mineral potential. Any withdrawal of these tracts from exploration activities is predicted to meet with resistance and criticism from the mineral industry. Tracts belonging in categories 4 through 6 have undergone various levels of exploration; our panel recommends further studies if any of these areas are to be considered for withdrawal. Categories 7 and 8 represent areas of minimal interest to the Industry. Withdrawal of these lands is predicted to be the least contentious."

Category 1 was reported as the highest potential due to the fact that these particular sites contained established mineral resources. The Mineral Potential Map (Appendix IV) prepared as a result of this evaluation illustrates the ranking throughout the belt and clearly shows four areas ranked as "#1" which includes the

Cowley, Keewenaw, Gem and Black Cub deposits in the southern portion of the City of Whitehorse boundary. The claims between the Cowley and Keewenaw deposits are underlain by favourable intrusive/limestone contact architecture and are considered highly prospective for future exploration potential. These claims are held by B. Ernewien and L. Doust and have only seen limited historical exploration.

One of the most effective historical survey methods used in the identification of the skarn zones within the Whitehorse Copper Belt was ground based magnetics. Many of the iron rich skarn zones in the belt contain significant amounts of magnetite relative to the adjacent stratigraphy giving rise to localized anomalous positive magnetic responses. Induced Polarization (IP) surveys have also been implemented at various deposit areas throughout the belt historically but this method has been met with limited success due to the presence of pyrite and graphite with low associated copper values. However, limited historical IP surveys conducted across the Cowley Deposit reportedly responded very well in areas of mineralization.

Soil geochemical sampling was also conducted in areas considered to be prospective targets for skarn development. Copper in soil response showed excellent contrast in areas of known mineralization but many of these anomalies were discounted because of interference attributed to glacial smearing and transport. No modern analysis and investigation of these anomalies has been undertaken.

Figure 10 shows the copper in soil response west of the Cowley Copper Deposit. While the anomalies do show a certain degree of fanning in the direction of glacial retreat, there remains a very high degree of constraint. There does not appear to have been significant follow up to determine the source of these particular anomalies.

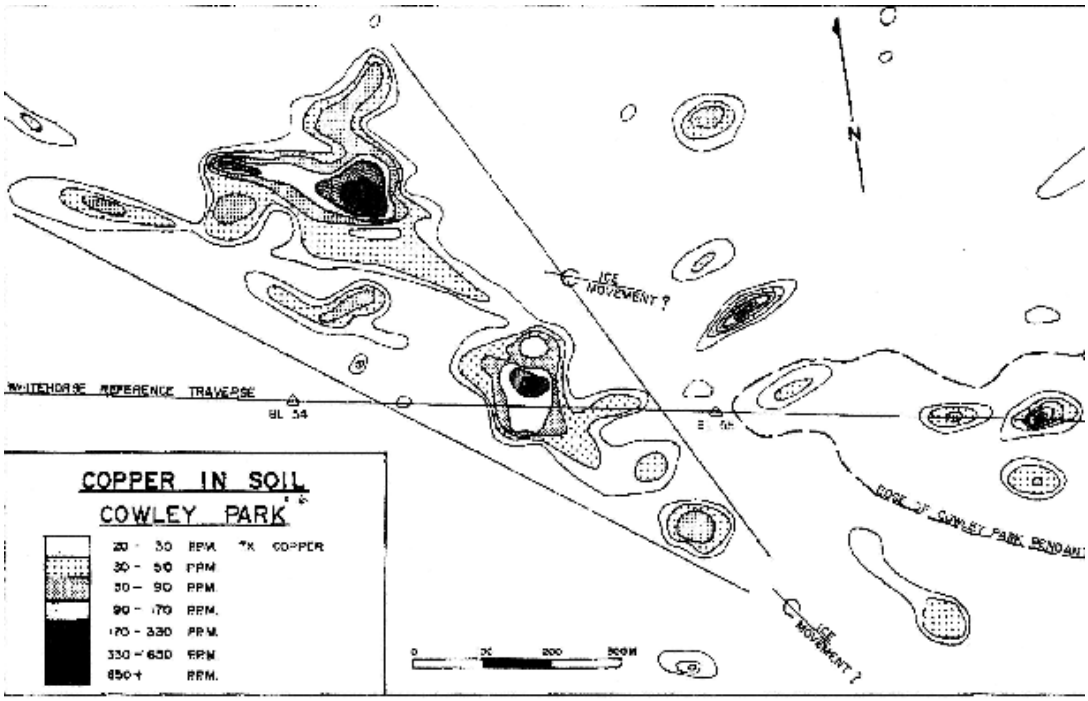


FIGURE 10 – Cowley Copper Soil Geochemistry West of Deposit
(after Tenney, 1981)

During 2008 three broadly spaced lines of IP were located over strategic target areas to evaluate the results of modern survey techniques over areas reported to contain historical IP anomalies. Two lines were located 2 and 3 km west of the Cowley Main Zone mineralization while a third line was located overtop the Gem Deposit which reportedly responded well to historical IP surveys.

The 2008 IP survey utilized a pole-dipole array at 25 m stations. Preliminary pseudo section results were generated interpreting apparent resistivity and chargeability for each of the lines surveyed. A formal interpretation of the results was not provided in the database; however, all sections show good contrast highlighting areas of low apparent resistivity with variable coincident anomalous apparent chargeability.

In 2009 B. Ernewein conducted localized ground based magnetic surveys across the Mag claims adjacent to the Cowley Copper Deposit and along strike to the west. The results of these surveys highlighted a number of well defined magnetic anomalies that have not received modern follow up work. The magnetic anomalies are shown on Figure 11.



FIGURE 11 – Magnetic Anomalies West of Cowley
(Ernewein, 2014 – pers. comm.)

SUMMARY

The Whitehorse Copper Belt hosts a series of significant copper deposits mined historically and that currently contain significant resources. Although the existing deposits are likely to be considered small relative to current producers in many parts of the world, they constitute significant in-situ Net Present Value (NPV) especially considering the ease of accessibility, historical production preparation and wide spread access to local infrastructure.

The Cowley Copper Deposit is the largest undeveloped copper resource within the belt containing a reported global resource of 1,504,622 tonnes grading 0.916 Cu. The global resource is defined by the Main Zone (open pit) and South Zone (underground). The Main Zone constitutes 56% of the global resource (836,622 tonnes) and has also been assigned values of 3.77 g/t Ag and 0.066% MoS2. The deposit was scheduled for production in the latter part of the period between 1971 and 1982 but due to declining copper prices the emphasis for production was shifted toward the higher grade underground development at the Little Chief Deposit.

The Cowley Copper Deposit resource estimate is likely conservative as the cut-off grades used to calculate the resource in 1965, 1971 and 1979 were probably high in order to sustain profitability at the then relatively low copper prices. Although modern production costs would be much greater today, the processing and recovery techniques have advanced considerably giving way to assessing potentially economic deposits with much lower cut-off grade thresholds particularly if by-products such as molybdenum and silver are considered as part of the economic assessment.

Recent exploration by Yankee Hat and Lobo Del Norte has shown that the tenor of the historical mineralization within the Main Zone is reliable specifically within the portions of the historical longitudinal grade shells tested. It should be noted, however, that only five of twenty-one holes were processed through to assay status and many of the unprocessed holes are reported to contain variable amounts of visual mineralization. Drilling within the lower grade region between the historical longitudinal grade shells produced results contradicting the previously interpreted model and suggests the outer zone grade shells of the two historical economic grading lobes are likely to be continuous at a higher grade threshold. This data could have implications potential future production assessment.

Drilling in 2008 also identified narrow zones of high grade copper in the form of semi-massive bornite containing strongly elevated silver values. Visual accounts of this high grade mineralization occur within drill holes that have not yet been processed for assay. Future assessment of the Cowley Deposit should include a review the historical drill data in detail to identify portions of the deposit with high grade bornite bearing copper-silver-molybdenum mineralization. This should be done in conjunction with a detailed review of the South Zone which contains roughly 44% of the currently assigned resource in the underground category.

Exploration potential in the vicinity of the Cowley Copper Deposit is considered good to excellent considering the availability of modern exploration techniques. Because access and infrastructure are pre-existing, the cost of acquiring excellent quality data

is likely to be very cost effective and efficient. Logical starting places for exploration would be in the vicinity of the historical copper-in soil geochemical anomalies immediately west along strike from the Cowley Main Zone and further west where B. Ernewein identified a number of well defined magnetic anomalies between the series of deposits.

Respectfully,

SKIVIK HOLDING CO. LTD.

William A. Wengzynowski, P.Eng.

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STATEMENT OF QUALIFICATIONS

I, William A. Wengzynowski, geological engineer, with business address in Vancouver, British Columbia and residential address at 301 Fairway Drive, North Vancouver, British Columbia, V7G 1L4 do hereby certify that:

1. I am President of Skivik Holding Co. Ltd.

2. I graduated from the University of British Columbia in 1993 with a B.A.Sc in Geological Engineering, Option l, mineral and fuel exploration.

3. I registered as a Professional Engineer in the Province of British Columbia on December 12, 1998 (Licence Number 24119).

4. From 1983 to present, I have been actively engaged in mineral exploration in the Yukon Territory, Northwest Territories, northern British Columbia and Mexico.

5. I have personally collected and reviewed the historical data reported herein.

William A. Wengzynowski, B.A.Sc., P. Eng.

STATEMENT OF EXPENDITURE

Labour	64 hrs @ $125 per hr	$8000.00
GST	5%	400.00
Total		$8400.00

APPENDIX I

OPEN FILE 1984

THE WHITEHORSE COPPER BELT

A COMPILATION

Open File 1984-1
THE WHITEHORSE COPPER BELT - A COMPILATION
Exploration and Geological Services Division, Yukon
Indian and Northern Affairs Canada
1984 - original notes by P.H. Watson
$4.00

Indian and Northern Affairs Canada | Affaires indiennes et du Nord Canada
Northern Affairs Program | Programme des affaires du Nord

1984 Open File:
THE WHITEHORSE COPPER BELT - A COMPILATION
Exploration and Geological Services Division
Whitehorse, Yukon

GENERAL GEOLOGY

The Whitehorse Copper Belt lies within the Whitehorse Trough, a northwest trending island arc complex of upper Paleozoic to Jurassic age. Within a 32 km. long, northwest trending belt, copper-bearing skarns occur mainly in upper Triassic Lewes River Group carbonate and clastic rocks, along the west contact of a Cretaceous quartz diorite batholith.

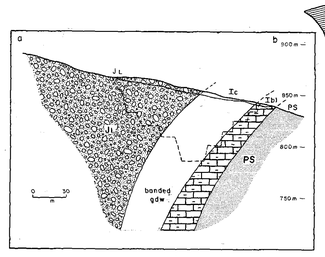
The carbonate units are both dolomitic and calcareous. In general, dolomitic units host magnetite-serpentine skarns, with red to brown garnet, bornite greater than chalcopyrite and valleriite. Calcareous units host magnetite-free wollastonite skarns with beige to yellow garnet, chalcopyrite greater than bornite and molybdenite.

HISTORY

Copper was first discovered in the Whitehorse area in 1897. The first claim, the Copper King, was staked July 6, 1898, by Jack McIntyre. Most of the presently known deposits were staked by 1899 with several properties shipping small ore tonnages between 1900 and 1912. The Pueblo Mine produced 127,000 tonnes between 1912 and 1920.

The next major production period began in 1967, when New Imperial Mines Limited began mining the Little Chief open pit. In 1968, production also came from the Arctic Chief open pits. Both these areas were shut down in 1969, and in 1970 the War Eagle pit came into production. In 1971, mining also took place at the Black Cub South and Keewenaw pits, until operations were suspended in mid-year. The company name was changed to Whitehorse Copper Mines Limited and production began again in December 1972, from the Little Chief-Middle Chief underground mine. In December 1982 the ore body was exhausted and all mining operations in the Copper Belt ceased. Between 1898 and 1982, approximately 10 million tonnes of 1.5% copper, with values in gold and silver, were mined.

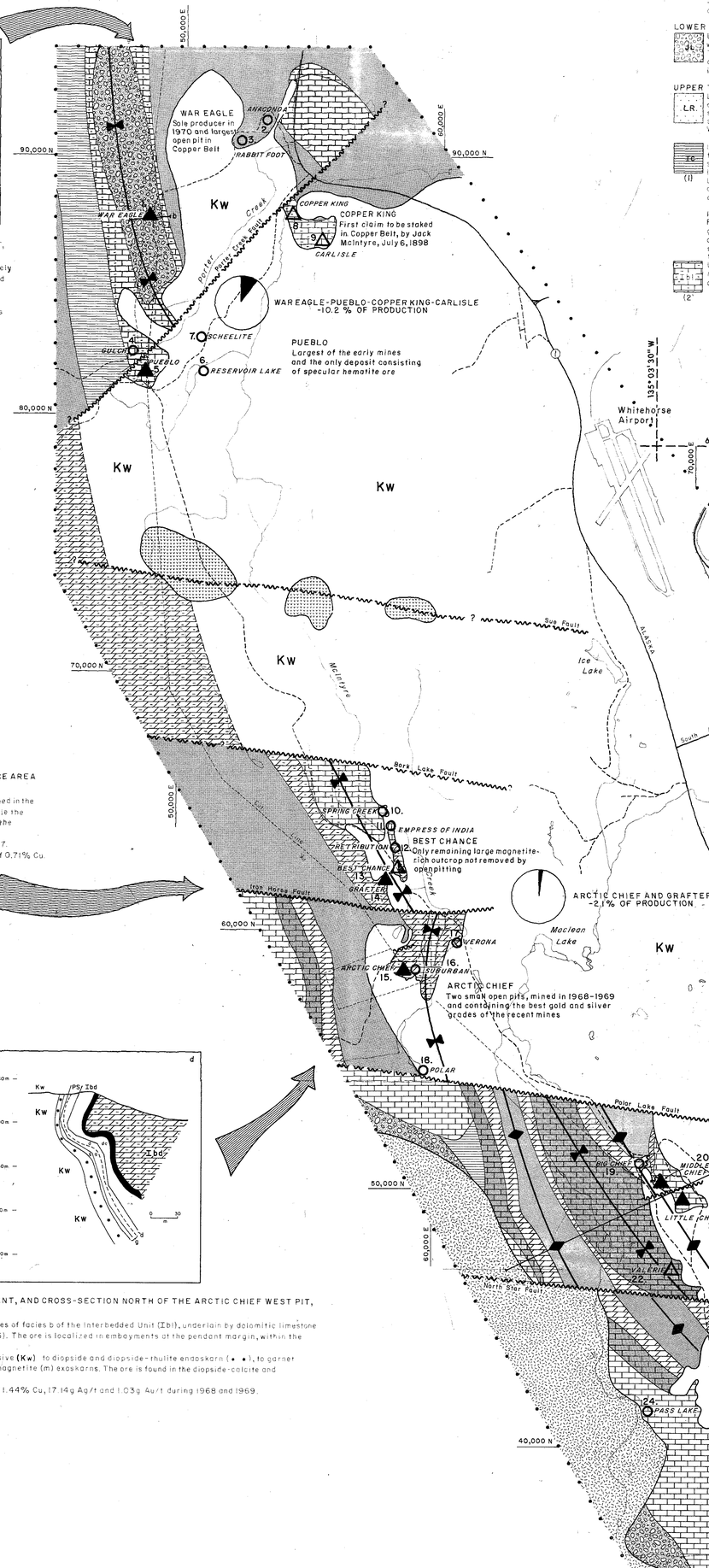
The summary table at the bottom groups 36 occurrences in the Copper Belt by host rock type; it indicates readily visible distinctive ore and skarn mineralogy and lists development, production/reserves and grade information for each occurrence.



GEOLOGICAL CROSS-SECTION OF THE WAR EAGLE DEPOSIT, LOOKING 350° (after Morrison, 1981)

The deposit is located on the east limb of a syncline, approximately 300m west of the intrusive contact. The ore is found in banded garnet-diopside-wollastonite skarn (limestone and siltstone of facies c of the Interbedded Unit (Tc)).

The War Eagle open pit (outlined) produced 894,000 tonnes of 1.25% Cu, 8.57g Ag/t, 0.22g Au/t and 0.005% MoS_2 in 1970.



LEGEND

QUATERNARY

QM — MILES CANYON BASALT - thin, grey to black, vesicular basalt flows, commonly with columnar jointing and in some places, olivine. Scoria, flow tops and intercalated breccia and tuff are commonly reddish in colour.

MID-CRETACEOUS

KM — MOUNT McINTYRE PLUTON - pink to grey, locally granophyric, monzonite to quartz monzonite.

Kw — WHITEHORSE BATHOLITH - grey, equigranular, medium to coarse grained, biotite-hornblende quartz monzonite to granodiorite and hornblende quartz diorite. The interior part of the batholith is a uniform massive quartz monzonite - granodiorite while the margins and apophyses are locally coarser-grained quartz diorite.

LOWER TO MIDDLE JURASSIC

JL — LABERGE GROUP - poorly sorted greywacke and sandstone with interbedded argillite and siltstone (no calcareous units). To the west, coarser material and conglomerates consisting of volcanic and granitic clasts are found. The unit is interpreted as fanglomerate.

UPPER TRIASSIC

LR — LEWES RIVER GROUP - a mixture of calcareous and dolomitic siltstone, sandstone and mudstone; pyritic siltstone, sandstone, argillite, limestone, dolomite and fragmental rocks, subdivided into the following eight units within the Whitehorse Copper Belt:

Tc (1) — INTERBEDDED UNIT, FACIES c - equal portions of non-fossiliferous, pyritic siltstone and calcareous siltstone to sandstone, with occasional thin carbonate beds. Partially laterally equivalent to, and partially overlying Grey Limestone, this unit is interpreted as having been deposited in a lagoon receiving sediments from erosion of a foreland on one side and carbonate reefs on the other.

INTERBEDDED UNIT, FACIES b - Facies b consists of alternating, massive, white to grey carbonate sandstone and laminated, black carbonate mudstone, with local interbeds of pyritic siltstone, sandstone and conglomerate. It includes the two following sub-facies:

Tbl (2) — Limestone Sub-Facies - this limestone-dominant unit makes up the upper part of the facies, interpreted as off-reef and off-bank erosional accumulations.

Tbd (3) — Dolomitic Limestone Sub-Facies - Dolomitic rocks are restricted to the lower sub-facies. The unit represents off-reef and off-bank erosional accumulations.

GL (4) — GREY LIMESTONE - discrete and stacked lenses of light grey weathering, fossiliferous limestone cap many of the low hills in the area. The unit grades down into Black Limestone and interfingers with the overlying Interbedded Unit. Six lithofacies based on fossil types were identified by Morrison (1981). The unit is interpreted as being widely spaced, framework patch reefs, developed along the bank margin.

BL (5) — BLACK LIMESTONE - carbonaceous limy mud with variable fossil content and 2 - 5% carbonaceous material, grades upwards into Grey Limestone and is in sharp contact with the underlying Dolomite. The unit has been divided into three distinct lithofacies based on fossil content and is interpreted as a subtidal, shallow dipping mudstone bank with local patch reefs.

D (6) — DOLOMITE - two light coloured lithofacies make up 90% of this unit. The remaining 10% is made up of 3 lithofacies, all containing black mudstone, which are concentrated as interbeds near the top of the unit. The environment is interpreted as a tidal flat with local euxinic tidal pools and channels.

PS (7) — PYRITIC SILTSTONE AND SANDSTONE - rusty-weathering, dark red-brown to dark grey, massive siltstone with hornfelsic texture is locally laminated and contains 2 - 5% pyrite. Pyritic siltstone is poorly sorted and contains interbeds and lenses of sandstone, grit and tuff. In some areas it forms the upper part of graded sequences within the sandstone unit. The unit was formed in a shallow marine to intertidal area with restricted water movement, such as near shore, back beach lagoon, overlying beach sands.

FR (8) — FRAGMENTAL ROCKS - volcanic breccia and tuff of basalt to basaltic andesite composition, containing intercalated conglomerate, sandstone and minor limestone, outcrops mainly to the west of the area shown. Diamond drilling has indicated that this material underlies the other units in the central part of the Copper Belt.

DYKES - Dykes are common throughout the map area, and can be divided into the following four types:
a) dark grey to black, aphanitic to very fine grained basalt and porphyritic basalt dykes believed to be feeders to Miles Canyon Basalt;
b) grey porphyritic andesite dykes believed to be associated with sub-volcanic portions of Eocene and Late Cretaceous plutonic suites;
c) grey to green, very fine grained andesite - dacite dykes (sometimes porphyritic);
d) pink to grey monzonite to quartz monzonite dykes, believed to be related to the Mount McIntyre pluton.

SKARNS

En — Endoskarn - skarn assemblage developed in the intrusive rocks

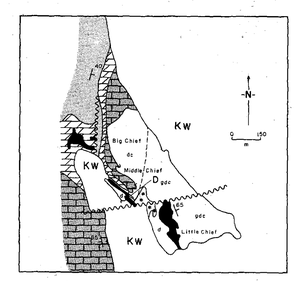
En-Ex — Endoskarn-Exoskarn - skarn assemblage developed in both the intrusive and sedimentary rocks

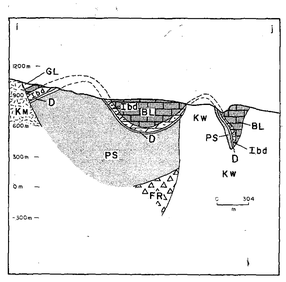
■ m — massive to banded magnetite-rich
g — massive garnet
d — massive diopside
dc — massive diopside-calcite
dcg — banded diopside, calcite, garnet
gdc — massive garnet-diopside-calcite
gwc — massive to banded garnet-wollastonite-calcite
gdw — banded garnet, diopside, wollastonite

▲ Production > 10,000 tonnes
△ Production < 10,000 tonnes
△ Known Reserves > 10,000 tonnes
○ No Known Production

Syncline
Anticline

SCALE 1:25000
Miles 0 1 2 Miles
Metres 1000 500 0 1000 2000 3000 Metres



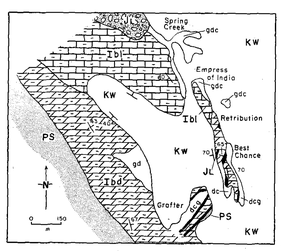


DETAILED GEOLOGICAL MAP AND INTERPRETED CROSS-SECTION, CHIEF MINE, OF THE LITTLE CHIEF PENDANT LOOKING 335° (after Morrison, 1981)

The Little Chief pendant is interpreted as a carbonate 'flap' within the edge of the Whitehorse batholith (Tenney, 1981). The Little and Middle Chief deposits are faulted segments of the same skarn. All three bodies are found in the Dolomite unit (D), which is underlain by Pyritic Siltstone (PS) and overlain by Black Limestone (BL) mixed with the dolomitic sub-facies of facies b of the Interbedded Unit (Ibd).

The schematic cross-section covers 2.75 km from the Mount McIntyre Pluton in the west, to the Whitehorse batholith in the east.

The Little Chief open pit produced 1,128,000 tonnes of 1.29% Cu from 1967 to 1969. The underground operation produced 7,620,000 tonnes of 1.53% Cu, 6.51g Ag/t and 0.58g Au/t from 1972 to 1982.

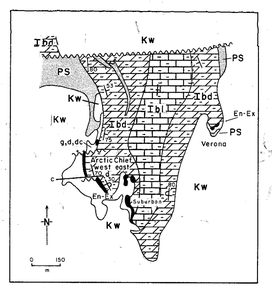


DETAILED GEOLOGICAL MAP OF THE GRAFTER-BEST CHANCE AREA (after Morrison, 1981)

The Grafter, Best Chance and Retribution mineralized skarns formed in the dolomitic sub-facies of facies b of the Interbedded Unit (Ibd), while the Empress of India and Spring Creek mineralized skarns formed in the overlying limestone sub-facies of facies b (Ibl).

The Grafter deposit produced 12,200 tonnes of 6% Cu prior to 1917.

The Best Chance deposit contains an estimated 447,000 tonnes of 0.71% Cu.

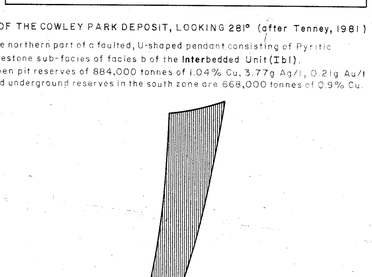


DETAILED GEOLOGICAL MAP OF THE ARCTIC CHIEF PENDANT, AND CROSS-SECTION NORTH OF THE ARCTIC CHIEF WEST PIT, LOOKING 355° (after Morrison, 1981)

The U-shaped pendant consists of a core of the limestone sub-facies of facies b of the Interbedded Unit (Ibl), underlain by dolomitic limestone of facies b of the Interbedded Unit (Ibd) and Pyritic Siltstone (PS). The ore is localized in embayments at the pendant margin, within the dolomitic unit (Ibd).

The cross-section illustrates the zoning from the unaltered intrusive (**Kw**), to diopside and diopside-thulite endoskarn (• •), to garnet endo- and exoskarn (g), to diopside (d), diopside-calcite (dc) and magnetite (m) exoskarns. The ore is found in the diopside-calcite and magnetite exoskarns.

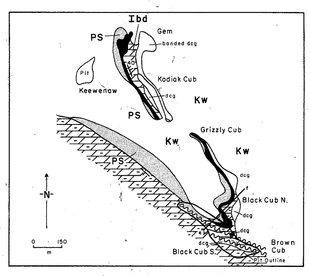
The Arctic Chief East and West pits produced 223,000 tonnes of 1.44% Cu, 17.14g Ag/t and 1.03g Au/t during 1968 and 1969.

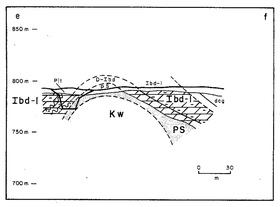


GEOLOGICAL CROSS-SECTION OF THE COWLEY PARK DEPOSIT, LOOKING 281° (after Tenney, 1981)

Mineralization is concentrated in the northern part of a faulted, U-shaped pendant consisting of Pyritic Siltstone (PS) overlain by the limestone sub-facies of facies b of the Interbedded Unit (Ibl).

The main zone has estimated open pit reserves of 884,000 tonnes of 1.04% Cu, 3.77g Ag/t, 0.21g Au/t and 0.066% MoS_2. Estimated underground reserves in the south zone are 668,000 tonnes of 0.9% Cu.





DETAILED GEOLOGICAL MAP OF THE KEEWENAW-GEM-BLACK CUB SOUTH AREA, AND CROSS-SECTION THROUGH THE BLACK CUB SOUTH AND NORTH DEPOSITS, LOOKING 305° (after Morrison, 1981)

The Gem, Kodiak Cub, Grizzly Cub, Black Cub North, Black Cub South and Brown Cub deposits formed in an interbedded limestone and dolomitic limestone unit (Ibd), which is underlain by Pyritic Siltstone (PS) and cut by closely spaced basaltic dykes (not shown). Mineralization at Keewenaw is found in an oval, pipe-like zone of diatreme in the granodiorite.

The Keewenaw open pit, (outlined on map) produced 159,000 tonnes of 0.95% Cu in 1971 and remaining reserves are estimated as 202,000 tonnes.

The Black Cub South open pit (outlined on map and section) produced 180,000 tonnes of 1.33% Cu, 12.34g Ag/t, 0.24g Au/t and 0.008% MoS_2 in 1971. Reserves are estimated as: Gem - 625,000 tonnes of 1.01% Cu; Kodiak Cub - 57,000 tonnes of 1.18% Cu; Black Cub North - 156,000 tonnes of 0.82% Cu.

ACKNOWLEDGEMENTS

This map represents a compilation of both published and unpublished information on the Whitehorse Copper Belt.

The major source of information is an unpublished Ph.D. thesis by Gregg W. Morrison.

Whitehorse Copper Mines Limited provided access to company geology maps and to the property. Special thanks to P. Percival, Whitehorse Mines Ltd. and to G. Bidwell, Hudson Bay Exploration and Development Co. Ltd. for their help in this project.

Larry Meinert, Washington State University, visited several deposits with the author and offered useful advice.

Kathryn Salmon ably assisted in the field and Kate Grapes in the office preparation.

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Compilation by P.H. Watson
Drafting by R. Lewis

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Watson, P.H., 1984. The Whitehorse Copper Belt - A Compilation; Exploration and Geological Services Division - Yukon, Indian and Northern Affairs, Canada, Open File, 1:25,000 scale map with marginal notes.

DEPOSIT NAME	CODE No.	MAJOR HOST ROCK TYPE	DEVELOPMENT YEAR	DEVELOPMENT TYPE	DEVELOPMENT PRODUCTION (tonnes)	DEVELOPMENT RESERVES (tonnes)	GRADE Cu %	GRADE Au (g/t)	GRADE Ag (g/t)	GRADE Mo %	ORE MINERALS OTHER	Bo	Cp	Hm	Mt	Mo	Vl	ORE MINERALS OTHER	SKARN MINERALS serp	phlg	woll	gnt-r	gnt-b	gnt-y	SKARN MINERALS OTHER
Little Chief	21	dolomite	1898-1905 1967-1969 1972-1982	minor open pit u.g.	- 1,128,500 7,407,900		1.28 1.53	pres. 0.75	pres. 9.16		Ga, Pt Pd+	X	X		X	(X)	X	Cv, Cc, Py	X	X		X			
Middle Chief	20	dolomite	1898-1909 1972-1982	minor u.g.	part of Little Chief mine		1.55	0.34			Ga, Pt Pd+	X	X		X		X	Py, Po	X	X		X			trem
Big Chief	19	dolomite	1898-1909	minor			*0.56	0.34			Ga, Pt Pd+	X	X		X		X	Cc	X	X		X			
Valerie	22	dolomite	1899-1907	u.g.	127		*1.65	0.68			Ga, Pt Pd+		X	(X)	X			Po, Py, Ap	X			X			trem, actn
North Star	23	dolomite	1900's	minor			*0.7	0.09	0.05			X	X		(X)				X	X		X			trem
Arctic Chief	15	interbedded (dolomitic)	1899-1907 1968-1969	u.g. open pits	202 201,800		6.53 1.44	10.69 1.03	79.33 17.14		Ga, Pt Pd+	X	X		X		X	Cc, Tt, Py, Cp	X	X	(X)	X			
Grafter	14	interbedded (dolomitic)	1899-1907 1915-1917	u.g. u.g.	1,800 10,400		7.0 6.0	pres. pres.	pres. pres.				(X)	X		X	(X)	Py, Ct	(X)			X			trem
Best Chance	13	interbedded (dolomitic)	1899-1912 1954-1957	u.g. testing		447,000	0.71	0.03			Ga, Pt Pd+	(X)	X		X		(X)	(Cc)	(X)	(X)		X			trem, actn
Suburban	16	interbedded (dolomitic)	1900-1907	minor			*0.89	1.03			Ga+	X	X		X	(X)			X			X			
Verona	17	interbedded (dolomitic)	1900's	minor			*0.20	0.17			Ga+	X	(X)		X	(X)						X			
Gem	27	interbedded (dolomitic)	1967-1970	testing		625,000	1.01					(X)	X		X		X		X	X		X			

DEPOSIT NAME	CODE No.	MAJOR HOST ROCK TYPE	DEVELOPMENT YEAR	DEVELOPMENT TYPE	DEVELOPMENT PRODUCTION (tonnes)	DEVELOPMENT RESERVES (tonnes)	GRADE Cu %	GRADE Au (g/t)	GRADE Ag (g/t)	GRADE Mo %	ORE MINERALS OTHER	Bo	Cp	Hm	Mt	Mo	Vl	ORE MINERALS OTHER	SKARN MINERALS serp	phlg	woll	gnt-r	gnt-b	gnt-y	SKARN MINERALS OTHER
Kodiak Cub	28	interbedded (dolomitic)	1970	testing		57,000	1.18					X	(X)		X				X	X					
Grizzly Cub	29	interbedded (dolomitic)	1970	testing								X	(X)		X				X	X					
Black Cub N.	30	interbedded (dolomitic)	1970	testing		156,000	0.82					X	(X)		X				X	X		(X)			
Black Cub S.	31	interbedded (dolomitic)	1971	open pit	187,000	20,000	*1.33 1.25	0.24	12.34	0.008		X	(X)		X		(X)	Cc, Ct	X	X	(X)	(X)			actn
Brown Cub	32	interbedded (dolomitic)	1970	testing								X	(X)		X				X	X					
Railway	33	interbedded (dolomitic)	1900's	minor			*0.15						(X)		X										actn
War Eagle	1	interbedded (limestone)	1899-1915 1970-1971	u.g. open pit	900 899,000		5.7 1.25	1.03 0.22	68.57 8.57	0.005	Ga+	X	X		(X)	X						X		X	trem, actn
Pueblo	5	interbedded (limestone)	1899-1906 1912-1920	u.g. u.g.	635 127,000		*4.9 3.5	0.17	34.29		Sc, Ga+		(X)	X	(X)			Cy, Ct				X		X	trem
Gulch	4	interbedded (limestone)	1899-1907	minor																					
Anaconda	2	interbedded (limestone)	1898-1907	minor			*0.9	0.17				X	(X)	(X)	(X)				X			X		X	trem
Rabbit-foot	3	interbedded (limestone)	1899-1907	minor			*0.7	0.17				(X)	X			X		Sc	X			(late)	X	X	
Copper King	8	interbedded (limestone)	1898-1907 1915-1920	u.g. u.g.	517 4,800		*1.47	0.17	15.09	0.21		(X)	X		(X)	(X)		Sc.				X	(late)		X	actn, trem
Carlisle	9	interbedded (limestone)	1900-1910	u.g.	907		*3.58	0.34	51.4			X	(X)			X		Sc					X	X	trem
Spring Creek	10	interbedded (limestone)	pre 1907	minor								(X)	(X)		(X)			Cc				(late)	X	X	
Empress of India	11	interbedded (limestone)	1900-1910	u.g.								(X)	X	(X)		(X)		Sc, (Py)				X	(late)		X	

TOTAL PRODUCTION FROM 11 DEPOSITS INTERMITTENTLY MINED DURING THE PERIOD 1898 TO 1982 WAS 10,130,000 TONNES OF APPROX. 1.5% Cu

DEPOSIT NAME	CODE No.	MAJOR HOST ROCK TYPE	DEVELOPMENT YEAR	DEVELOPMENT TYPE	DEVELOPMENT PRODUCTION (tonnes)	DEVELOPMENT RESERVES (tonnes)	GRADE Cu %	GRADE Au (g/t)	GRADE Ag (g/t)	GRADE Mo %	ORE MINERALS OTHER	Bo	Cp	Hm	Mt	Mo	Vl	ORE MINERALS OTHER	SKARN MINERALS serp	phlg	woll	gnt-r	gnt-b	gnt-y	SKARN MINERALS OTHER
Retribution	12	interbedded (limestone)	1900's	minor									X	X	X							(X)		X	actn
Sue	34	interbedded (limestone)		testing			*1.55	trace		0.74			X	X			X						X		actn
Cowley Park	35	interbedded (limestone)	1912 1970's	minor testing		open pit 884,000 668,000 0.9	*1.04 0.9	0.21 pres	3.77 pres	0.066 pres			X	X		(X)	X			(X)		X	(X)		X	
Pass Lake	24	limestone	1900's	minor			*0.4	trace					(X)	X		X								X	
Copper Cliff	25	limestone	1900's	minor									X	X	(X)	(X)									
Keewenaw	26	intrusive	1950's 1971	testing open pit	159,000	202,000	0.95 1.06	pres	pres	pres			X	(X)				Cy, Mi, Cc, Py Cv							
Reservoir Lake	6	intrusive					*1.0						X				X								Bo in sheeted quartz veins in hornblende granite
Scheelite	7	intrusive					*1.7	0.51			0.58% WO_3		X				X	Sc., Mi							quartz-carbonate-Sc vein
Polar	18	fault zone	1900's	minor																				Py, Cy	

NOTE: Ore minerals Bo-bornite, Cp-chalcopyrite, Hm-hematite, Mt-magnetite, Vl-valleriite, Mo-molybdenite, Cv-covellite, Cc-chalcocite, Py-pyrite, Po-pyrrhotite, Sc-scheelite, Mi - malachite, Ap-arsenopyrite, Tt-tetrahedrite, Ct-cuprite, Cy-chrysocolla

Skarn minerals serp-serpentine, phlg-phlogopite, woll-wollastonite, gnt-r-garnet-red, gnt-b-garnet-brown, gnt-y-garnet-yellow, trem-tremolite, actn-actinolite, in addition, diopside, epidote, calcite and thulite are present in most deposits

(limestone) Indicates limestone sub-facies of Interbedded unit facies b.

(dolomitic) Indicates dolomitic limestone of Interbedded unit facies b.

* Next to grade indicates representative assay rather than average grade based on tonnage. Grades indicated for small tonnages mined prior to 1960 often represent hand-sorted high-grade concentrates, rather than true grade of deposit.

+ Gallium, Platinum and Palladium reported by Kindle (1963) as 0.14g Pt/tonne, 0.07g Pd/tonne and 0.03-0.06% Ga.

X Major ore minerals

(X) Minor ore minerals

u.g. Underground

Canada

APPENDIX II

BULLETIN 1

THE WHITEHORSE COPPER BELT: MINING, EXPLORATION AND GEOLOGY

(1967-1980)

Indian and Northern Affairs Canada
Affaires indiennes et du Nord Canada

Northern Affairs Program



THE WHITEHORSE COPPER BELT: MINING, EXPLORATION AND GEOLOGY (1967 - 1980)

by

D. TENNEY

BULLETIN 1

Geology Section,
Yukon Region,
Department of Indian
and Northern Affairs

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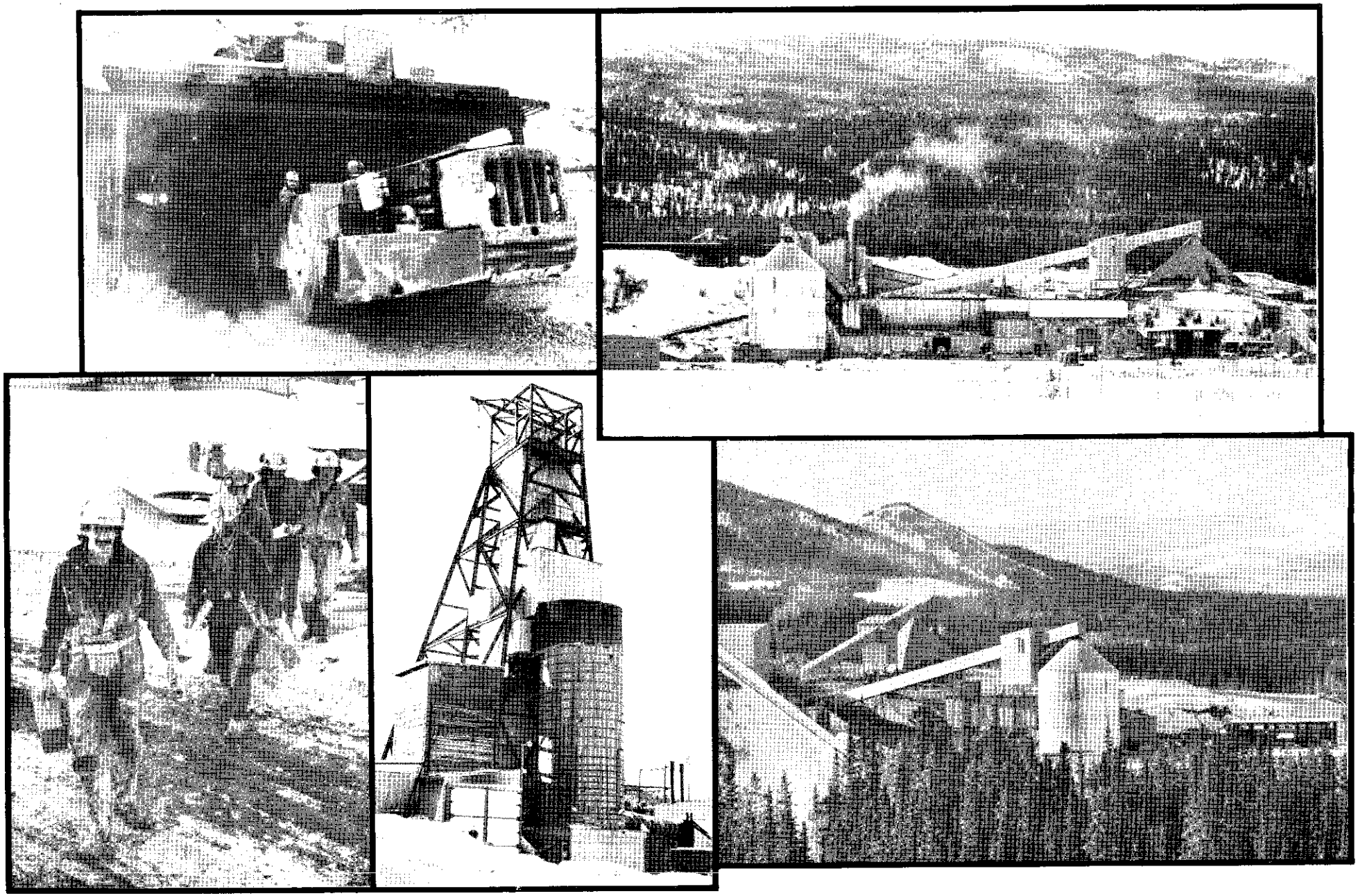
GEOLOGY SECTION, YUKON REGION

BULLETIN 1

THE WHITEHORSE COPPER BELT: MINING, EXPLORATION AND GEOLOGY (1967 - 1980)

BY
D. TENNEY

DEPARTMENT OF INDIAN AND NORTHERN AFFAIRS, CANADA



Frontispiece: Present mining and milling operations on the Little Chief orebody (Whitehorse Copper Mines). Photos by Tim Whillans.

PREFACE

This report on the Whitehorse Copper Belt is written by a geologist who was actively involved in mining and exploration in the district during the decade of the seventies. It includes a review of earlier data and compiles recent work, mining practice, geological concepts and constraints and problems in development and mining. Earlier reports on the district by McConnell and Kindle emphasized geology; this report concentrates on mining and development. The report was written when mining in the district was at its peak; it will be useful to those contemplating future development in the Copper Belt and for comparing the district to others. Mining in the Copper Belt may cease soon and the report was solicited by D.I.A.N.D. from Whitehorse Copper Mines as a record of mining practice and geological concepts.

Whitehorse, March 7, 1981

D. Tempelman-Kluit
Regional Geologist

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THE WHITEHORSE COPPER BELT: MINING, EXPLORATION AND GEOLOGY (1967 to 1980)

INTRODUCTION

The Whitehorse Copper Belt is a northwest trending zone of copper bearing skarns 30 kilometers long about 5 km west of the city of Whitehorse, Yukon Territory. Whitehorse Copper Mines controls a single block of over 700 mining claims effectively covering the belt (Fig. 1). From 1967 to 1971 the company produced from six small open pits, and from 1972 to 1980 from the Little Chief underground mine. Underground ore reserves are enough to keep the mill operating until 1982.

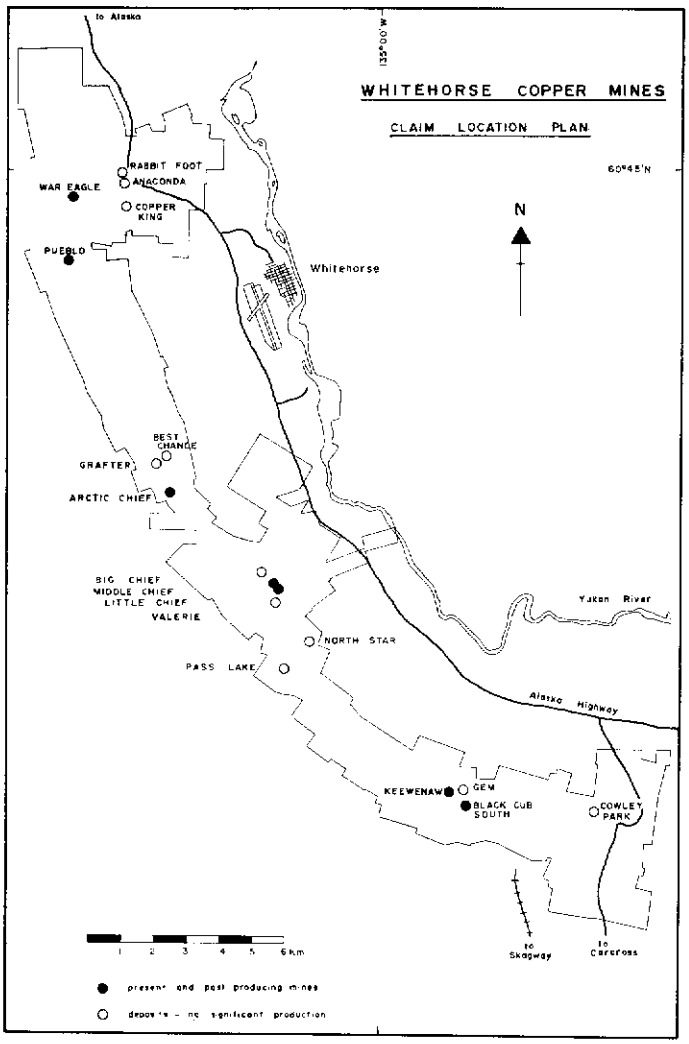


Figure 1. Map of claims held by Whitehorse Copper Mines in 1972 showing the largest copper deposits. The main producers are shown as black circles.

Acknowledgements

This report would not have been produced without the help and cooperation of members of the staff at Whitehorse Copper Mines, and to them, particularly Andy Hureau and Gregg Morrison of the Exploration Department, I give my thanks. Permission to publish the results of the project were given on behalf of Hudson Bay Mining and Smelting by D. Linzey, Vice President, Whitehorse Copper Mines. This work was published through the efforts of Dirk Tempelman-Kluit who critically read the manuscript and made improvements.

History

Discovery of copper in the copper belt is reported to have been made by miners on their way to the Klondike in 1897, but not staking was done until 1898, when Jack McIntyre staked the Copper King claim on July 6th of that year. By 1899 the district had been well prospected and most of the presently known deposits had been staked.

The first shipment of ore was a small one from the Copper King claim in the 1900. McConnell (1909, p. 2) records that this comprised 8 tonnes of hand-picked ore grading 46.40% copper. Other small shipments made at this time included 40 tons from the Valerie, Grafter and Arctic Chief deposits. During the main productive life of the Pueblo Mine between 1912 and 1920 significant ore shipments were made. The Pueblo, by far the biggest of the early mines, produced about 126,000 tonnes of ore grading 3.5% copper before it was closed down in 1920 or 1921. The mine was temporarily shut-down by a fall of rock on March 7th, 1917, which trapped nine men. Three of these men were later rescued.

By 1927 the Richmond Yukon Company had obtained control of the Pueblo, but found no new ore, and interest in the Copper Belt waned. In 1946-7, Noranda and Hudson Bay Mining and Smelting did limited geological and magnetometer surveys and drilled some of the more important showings, notably Cowley Park, Keewenaw, Middle Chief, Big Chief and Pueblo. A few shallow holes were drilled at Little Chief.

In September of 1954, Mr. Aubrey Simmonds directed the formation of a company to be incorporated as Imperial Mines and Metals under an Alberta charter. The company acquired control of a large part of the Whitehorse Copper Belt by option and staking. The Arctic Chief and Best Chance deposits were surveyed using a magnetometer and some drilling was later carried out.

In March 1957 the company's name was changed to New Imperial Mines. The new company was reorganized by Mr. Arnold Pitt and continued to explore and drill the known deposits on their claim holdings. Eventually with financing from Sumitomo Metal Mining of Japan, and the Toronto Dominion Bank, milling of ore from the Little Chief pit started on May 1st, 1967. New

Imperial had a record year in 1969 with a net income over $4 million from the 730,755 tonnes of ore milled. The Arctic Chief Pits and Little Chief Pit were mined out by the end of that year. In 1970 production came from the War Eagle Pit. The Black Cub South Pit was mined in early 1971, and a start had been made on the Keewenaw Pit when falling copper prices forced the company to cease milling on June 30th, 1971. The average price received for copper in 1971 was $1.05/kg compared with an average of $1.54/kg in 1970.

In August of 1971 shareholders approved a re-organization of the company and the new name of Whitehorse Copper Mines was chosen. Hudson Bay Mining and Smelting and Amcan, two associated Toronto based companies, provided funds to continue underground development of the Little Chief ore body under the terms of a mining joint venture agreement. A vertical shaft, started during 1971, was made operational in August 1972, and by late December of that year, development was far enough advanced to allow the mill to be restarted after a closure of nearly 18 months. Ore production from the Little Chief underground workings has been continuous to the present (Nov. 1980). In May 1972 an exploration joint venture agreement with Hudson Bay and Amcan made provision for the spending of up to a half million dollars on that part of the company's property lying north of the Little Chief. This option was terminated in November of 1976, after $450,000 was spent, most of it on surface diamond drilling. Although numerous short copper intersections were obtained during this work, none appeared commercial. Extensive induced polarization and magnetometer surveys were also carried out along with limited geological mapping.

In October of 1976, Whitehorse Copper Mines repaid all money loaned to it and became debt free. The Company then started receiving the first payments from its share of the Little Chief Joint Mining Venture profits. Under the terms of the joint mining venture Hudson Bay Mining and Smelting is entitled to one third of the profits and Whitehorse Copper Mines the remaining two thirds. In May of 1978 Hudson Bay Mining and Smelting made an offer for the outstanding shares of Whitehorse Copper Mines which was accepted by the required majority of shareholders, and the company is now a wholly owned subsidiary of Hudson Bay Mining and Smelting.

GEOLOGICAL SETTING

The Whitehorse Copper Belt is within the Whitehorse Trough, a subdivision of the Intermontane Belt. The trough trends northwestwards through south central Yukon and represents an Island Arc Complex that ranges from upper Paleozoic through Jurassic in age. Within the Copper Belt, clastic and carbonate rocks of the Upper Triassic Lewes River Group and clastic rocks of the Lower Jurassic Laberge Group are the dominant rock types. The copper bearing skarns occur over a length of about 32 km along the western side of a Cretaceous diorite batholith of the Coast Plutonic Complex.

The geology of the Copper Belt, was studied and reported by Kindle (1964) and the reader is referred to this report and to Wheeler (1961) for a description of the geology of Whitehorse map-area (N.T.S. 105 D).

Geology of Deposits

The characteristics of deposits within the Whitehorse Copper Belt are as follows:

1. Ore bodies occur mainly within limestone of the Lewes River Group, adjacent to or within a few hundred feet of diorite contacts.

2. Limestone within the Lewes River Group varies from a fine-grained graphitic type to a pure, massive, white, coarsely crystalline variety. With the exception of Black Cub South, ore is not associated with strongly graphitic limestone.

3. Ore is associated with irregularities in the diorite contact and the largest deposits occur within roof or flank pendants.

4. Most ore zones have irregular boundaries and vary in width and grade over short distances, but they are generally tabular and oriented parallel to bedding.

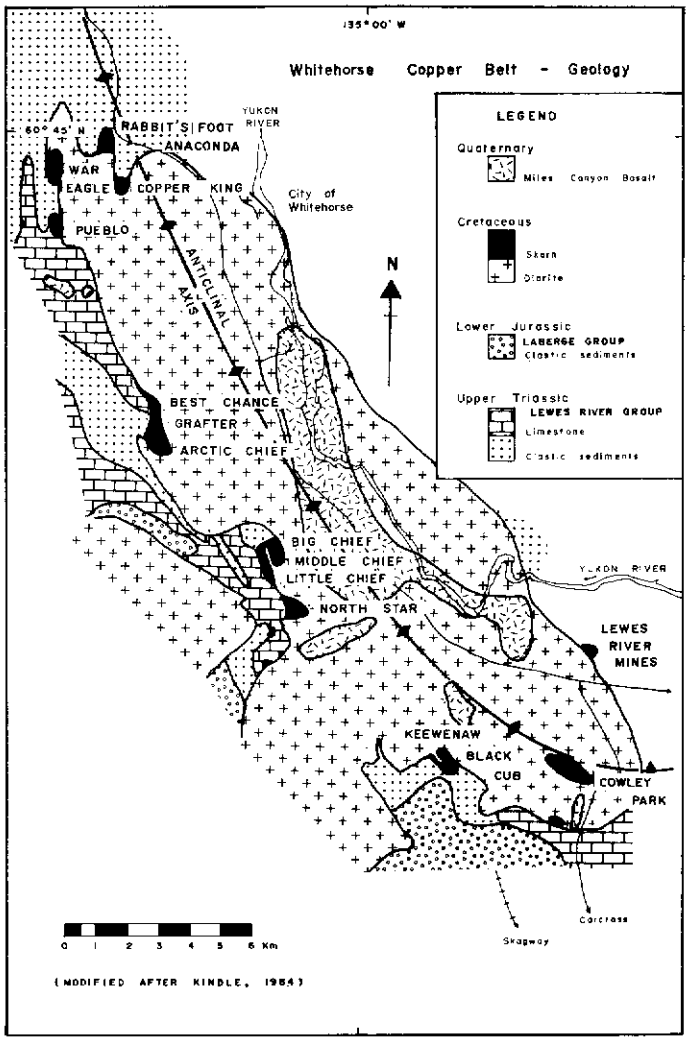


Figure 2. Regional geology of the Whitehorse Copper Belt modified from Kindle (1964).

5. Limestone is generally present on the hanging wall side of the ore and "quartzite" or silicate skarn on the foot wall.

6. The most extensive ore zones are developed where a limestone/quartzite contact is parallel or nearly parallel to the intrusive contact.

7. Calcsilicate skarn ore bodies are associated with a relatively magnetic diorite, whilst serpentine-magnetite skarns are associated with relatively non-magnetic diorite.

8. There are two distinct types of skarn; iron rich and calcsilicate rich. Garnet, epidote tremolite, calcite, actinolite, diopside, feldspar, quartz, talc, and thulite are common to both. Texture and composition ranges from medium and coarse-grained to a fine-grained mixture.

Iron rich skarns are characterized by high content of serpentine/talc/chlorite and associated magnetite and the copper mineral valleriite. They are generally strongly sheared and faulted and may be intruded by numerous dykes.

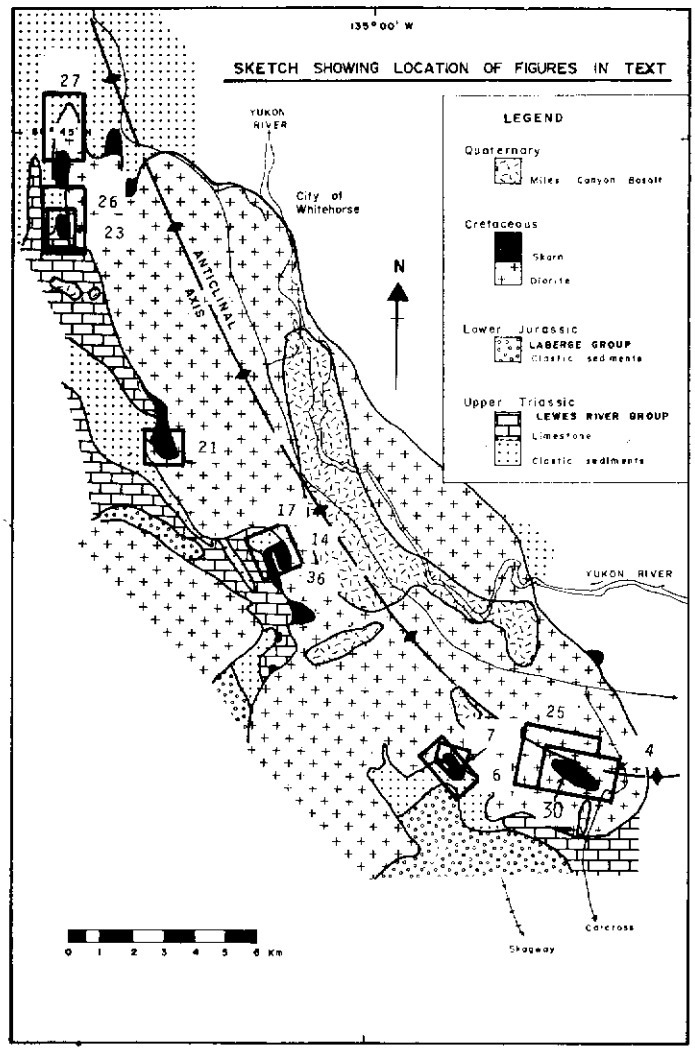


Figure 3. Regional geology of the Whitehorse Copper Belt showing the location of text figures.

Calc-silicate skarns have little or no serpentine or magnetite, and no valleriite. They are rich in the silicate minerals garnet (andradite), tremolite, wollastonite, actinolite, diopside, quartz and feldspar. The calcsilicate skarns are generally less sheared and faulted than the serpentine skarns, giving more competent ground for mining.

Near skarns, siliceous and feldspathic sedimentary rocks are commonly altered, recrystallized and grade to skarn and diorite.

The following are descriptions of the main deposits within the copper belt. Figure 2 shows the location of the deposits and Figure 3, the location of the geological sketch maps.

Cowley Park

The Cowley Park deposit at the south end of the Copper Belt is a steeply dipping 300 metre long tabular lens with a swelling at the west end enclosed in skarn on a limestone-diorite contact (fig. 4). The main zone is a typical silicate skarn and comprises brown garnet, diopside, actinolite, tremolite/ wollastonite, with disseminated chalcopyrite and bornite and minor magnetite and serpentine. Deep drilling indicates that the diorite flattens with depth and that the pendant has a boat shaped cross section (fig. 5). Although skarn is developed at the base of the pendant it thins to the south and is weakly mineralized. Barren skarn also continues east and west beyond mineralization. The deposit contains significant molybdenite, silver and gold. There is no direct correlation between copper grade and that of molybdenite in drill core samples.

The diorite adjacent to the skarn is skarnified and "altered" and contains minor amounts of copper. The sulphides are very fine-grained and commonly occur in altered amphibole. This has prompted comparison with porphyry copper deposits.

An induced polarization survey over the zone gave a strong chargeability anomaly over the mineralized area, and this was used to plan diamond drilling before 1970. A magnetometer survey on lines at 30 m intervals defines three "en echelon" magnetic anomalies above the projected position of the ore zone, and these were used to plan drilling after 1970. Gaps between the three magnetic peaks correlate with weak, or no mineralization. The magnetic anomalies are weak and reflect small quantities of magnetite in the skarn.

Ground conditions are generally good, and diamond drilling has been easier at Cowley than in most other places in spite of the hardness of the rocks: the drill core is relatively unfractured. 125 holes totalling 11,500 m have been drilled. Pit designs and ore reserve calculations were done in 1971 using cut-off grades of 0.5 and 1.0% copper. The content of molybdenite in the ore was calculated as 0.076%. According to P. Conder, Mill Superintendent at the time, the economics of recovering this molybdenite are marginal because new mill circuits must be built.

On the basis of the calculation done during April 1971 further surface diamond drilling was carried out on 17 m sections to determine the continuity between 30 m interval sections previously drilled. The results were disappointing and the May 28th pit design shows a substantially lower tonnage and a higher stripping ratio. The calculations are compared below.

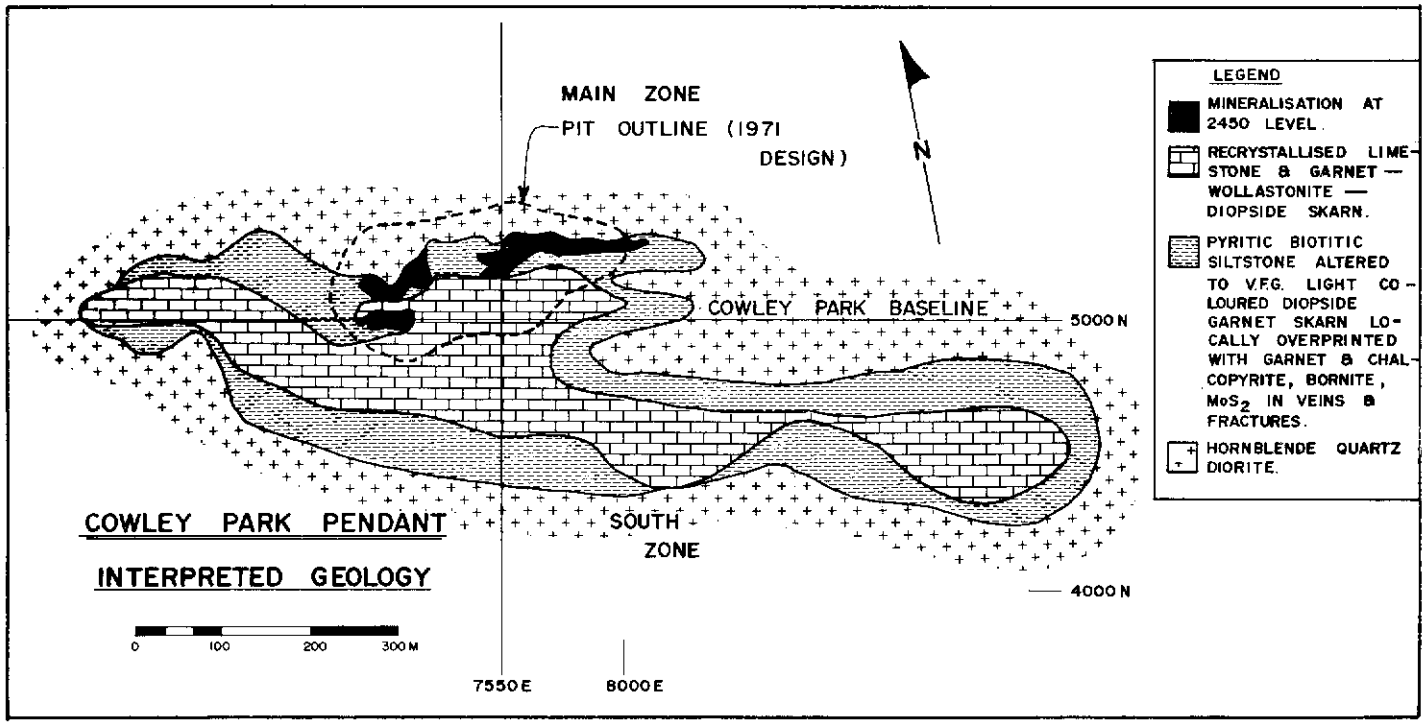


Figure 4. Geological plan of the Cowley Park roof pendant from company plans as reinterpreted by G. Morrison. Note that the Cowley Park South zone, which is the site of a moderately strong induced polarization anomaly, does not crop out at surface (i.e. 2450-2500' elevation).

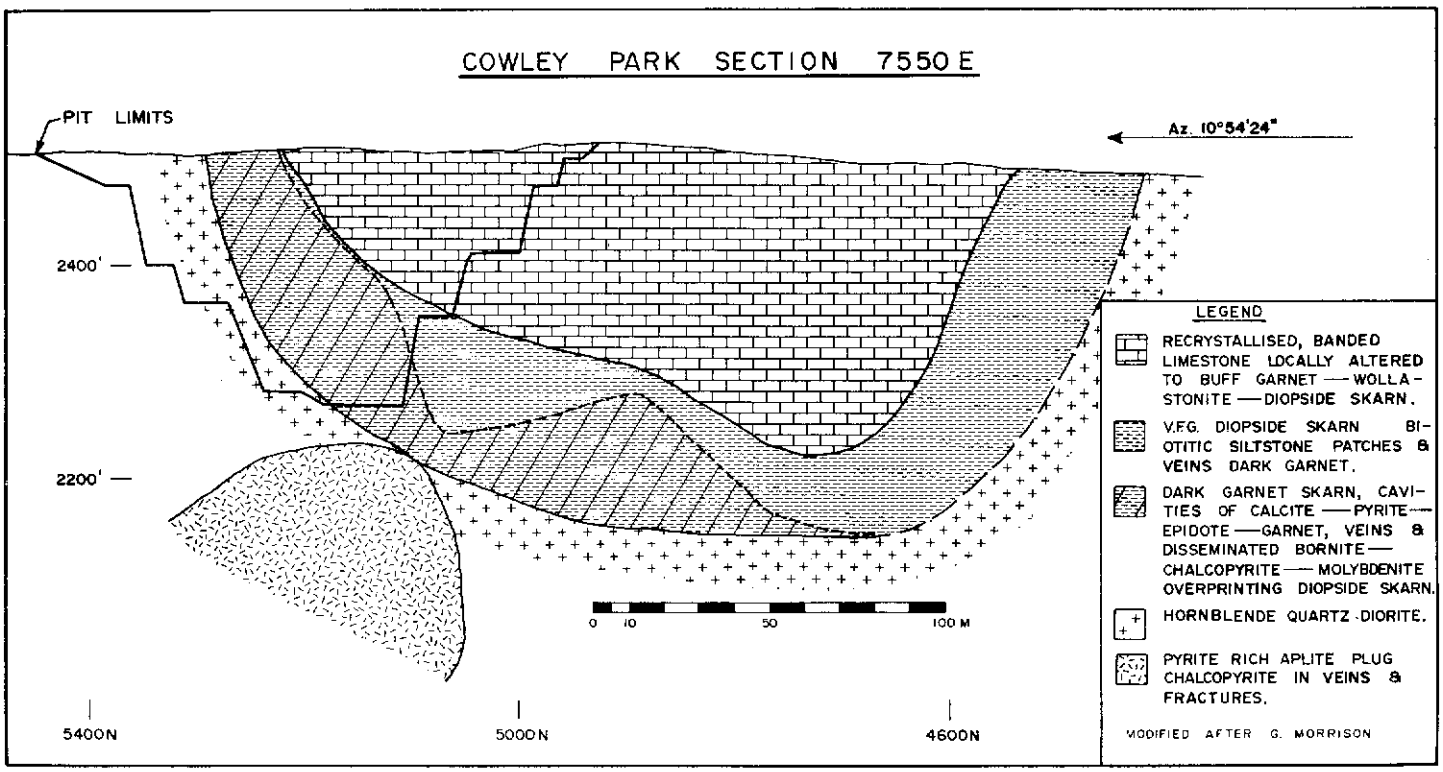


Figure 5. Geological cross-section of the Cowley Park main zone with limits of proposed (1979) pit design. The geology shown is as reinterpreted by G. Morrison.

	Tonnes	Grade	Waste/ Ore Ratio	Cu off Grade
Jan. 28, 71	381,158	1.42% Cu		
(15% dil.)	438,331	(1.2% Cu)	5.22/1	1% Cu
Apr. 16, 71	966,040	1.00% Cu		
(15% dil.)	1,110,945	(.87% Cu)	1.51/1	0.50% Cu
May 28, 71	769,824	1.02% Cu		
(15% dil.)	885,298	(.89% Cu)	2.31/1	0.50% Cu

In the May '71 proposed pit design the proportions of rock types to be mined was determined from diamond drill core.

WASTE	(limestone	26%
	(silicate skarn	32%
	(diorite	6%
ORE	(magnetite skarn	2%
	(silicate skarn	30%
	(diorite	4%

During 1979 a revaluation of the Cowley Park Pit was done by the author using pit optimization techniques similar to those suggested by Koskiniemi. The results, tabulated below, show similar undiluted ore reserve tonnage to the 1971 pit, but a marked reduction in the stripping ratio. Undiluted ore reserve grade, although not directly comparable, is higher, (1.25% -v- 1.02% copper), as expected from applying pit optimization techniques.

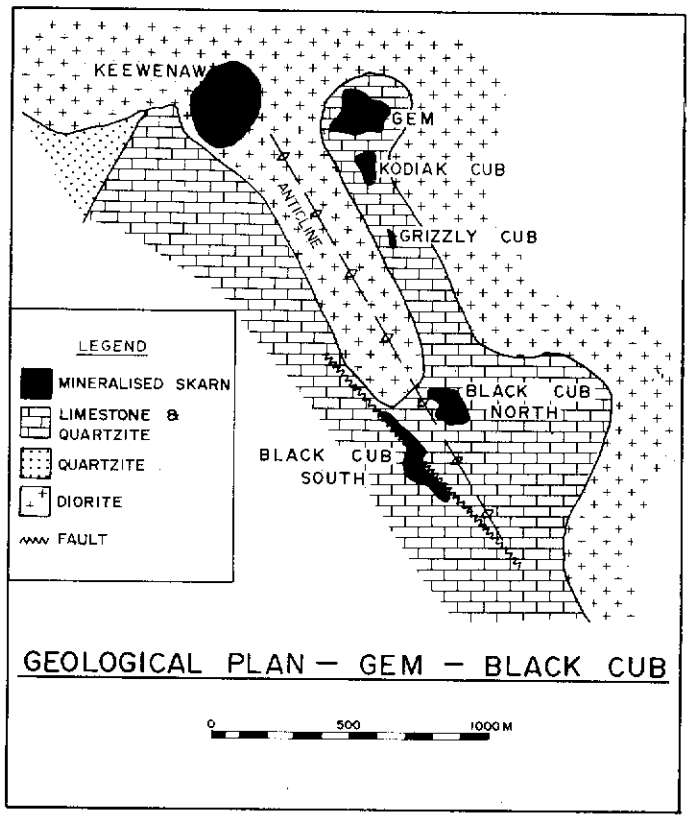
Cowley Park 1979 Pit Design - Reserves

Bench	Tonnes	Ore Grade	Waste Tonnes	Waste/ore Ratio
2268	35,098	1.09	4,193	0.12/1.0
2300	53,700	1.05	15,482	0.29/1.0
2325	94,747	1.25	20,699	0.22/1.0
2350	114,143	1.29	65,781	0.58/1.0
2375	118,652	1.27	112,053	0.94/1.0
2400	123,048	1.31	223,355	1.82/1.0
2525	114,289	1.14	284,941	2.49/1.0
2450	67,508	1.23	390,540	5.79/1.0
2475	34,955	1.60	340,224	9.73/1.0
2500	3,043	2.94	57,396	18.85/1.0
TOTAL	759,185	1.25	1,514,641	1.995/1.0
DILUTED	836,622	0.93	1,669,134	1.995/1.0

1) Grade Molybdenite (MoS2) 0.076% (0.056% dil)
2) Grade Silver 5.44 gm/t (4.003 gm/t dil)
3) Grade Gold 0.1950 gm/t (0.1437 gm/t dil)
4) Diluted grades have been calculated using a grade factor of 0.74 in all cases, which is equivalent to dilution of 35%. Historically open pits have shown an increase in tonnage of about 12% when mined: the diluted tonnage estimate shown above is conservative.

Bear Cub and Gem Deposits

The Cub and Gem deposits (fig. 6) are 5 magnetite serpentine skarns within a northwest trending finger of limestone about 1000 m long. Except for the southwest end, the limestone is surrounded by diorite.



GEOLOGICAL PLAN — GEM — BLACK CUB

Figure 6. Geological plan of the Keewenaw and Gem-Bear Cub zones. Note the similarity in structure to Little Chief-Big Chief (see figure 17).

The chance of finding new ore bodies is poor owing to the proximity of the diorite contact to the known mineralization.

Black Cub South

The ore within the Black Cub South is massive magnetite-serpentine skarn with disseminated bornite/chalcocite and chalcopyrite. Native copper dendrites oxidized to cuprite are common on shears in the serpentine near the middle of the orebody. A wide basic dyke, quartzite and diorite are seen on the footwall side of the ore, and faulted graphitic limestone, feldspathic sedimentary rocks and altered diorite on the hangingwall. Diopside, actinolite, talc, serpentine, chlorite and garnet occur in and around the ore zone. The skarn at Black Cub South was not developed over such a large area as that at Cowley Park, Little Chief and War Eagle.

The Black Cub South orebody has been explored using a Sharpe M.F.I. fluxgate magnetometer on lines 8 meters apart with readings at 8 meter intervals. The survey delineated a strong magnetic anomaly 100 meters long and up to 45 meters wide. The anomaly coincides with the known subcrop of the ore and was used as a guide for diamond drilling. Several smaller adjacent anomalies also indicated mineralization which was included in the ultimate pit design. Glaciofluvial overburden was locally about 9 meters deep. Induced polarization chargeability results are high but the pattern of anomalous readings is distorted by an adjacent zone of graphitic limestone

several km long and does not reflect the presence of the orebody.

Two deep drill holes on the footwall side of the Black Cub South Pit encountered minor sporadic low grade mineralization directly below the pit. A buried limestone-quartzite contact between Black Cub and Keewenaw on an "open" diorite contact, was tested by surface drilling without success.

Black Cub North

The Black Cub North Zone is 150 meters north of the Black Cub South Pit. The area is marked by a weak magnetic anomaly, and lacks outcrop. Reserves are based largely on one drill hole (Bl.C-15); other drilling carried out on 30 m interval lines obtained poorer results. The dip of the mineralization is northeast at 30°. The skarn is an iron variety with magnetite and serpentine and ground conditions are poor like those in the Black Cub South Pit. Because the diorite is nearby, significant ore down dip of the presently known mineralization is unlikely. The depth of overburden and amount of waste rock will discourage open pitting.

Gem

The Gem is the northernmost deposit in the skarn that contains the Bear Cub deposits (fig. 6). It is surrounded on three sides by diorite, and connected by skarn with the smaller Kodiak Cub on the fourth side (figs. 7 and 8). The ore occurs in a narrow shallow dipping lens mainly on the footwall side of a limestone and is cut by a basic dyke between 19 and 25 m thick with nearly the same orientation as the mineralization (fig. 8).

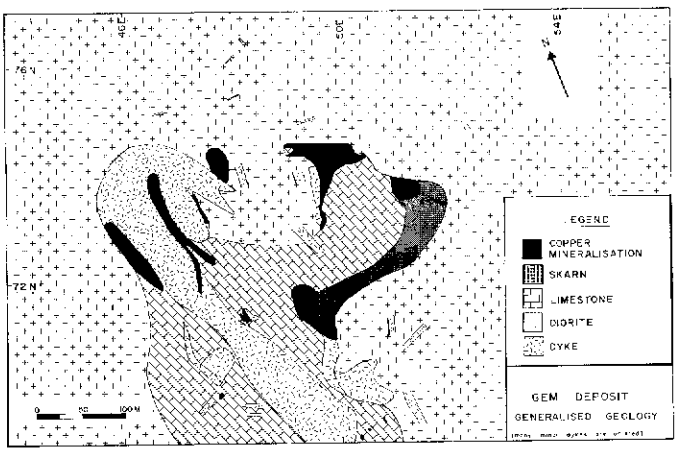


Figure 7. Schematic geological plan of the Gem deposit.

The copper sulphides are partly oxidized and some valleriite is probably present. Valleriite is not recoverable by flotation.

The Gem was discovered in 1967 by a reconnaissance magnetometer survey carried out over the Keewenaw ore body. It is the site of a low order induced polarization chargeability peak. The first drilling on lines at 30 meter intervals was probably guided by this I.P anomaly. A detailed magnetometer survey on lines 7.5 m apart with readings at 7.5 m intervals however indicated that the mineralization was not continuous between 30 m interval drill section lines and three short holes drilled on 15 m interval lines where breaks in the magnetic highs occur encountered little mineralization.

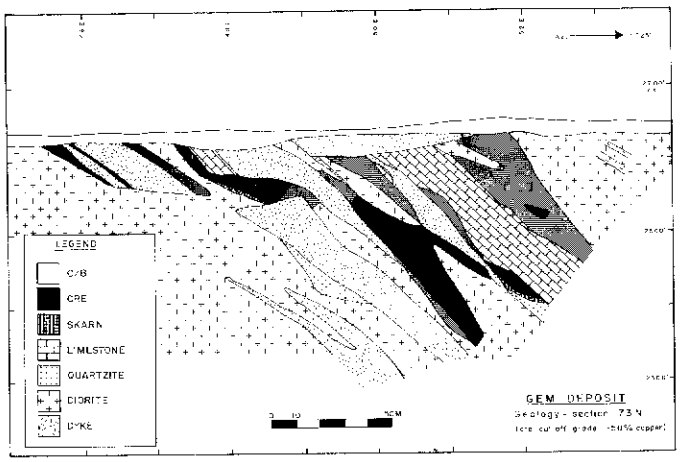


Figure 8. Geological cross-section of the Gem deposit interpreted from surface diamond drilling using a 1.50% copper cut-off grade.

In calculating ore reserves at Gem a dilution factor of 25% with barren material has been used because:

1). ore dips gently (but not flat).
2). ore zones are narrow and locally irregular.
3). ore is commonly discontinuous and cut by thick basic dykes.

In the 10 m of mineralization nearest surface metallurgical test work indicated a maximum copper recovery of 60% using standard flotation procedures.

The history of past ore reserve calculation for the Gem is as follows:

Date	Tonnes Ore	Grade Ore % Cu	Tonnes Waste	Total Tonnes	Stripping Ratio
5/29/69	916,362	1.20	6,341,125	7,257,487	6.92/1
10/7/69	810,743	1.02	2,369,275	3,180,018	2.92/1
9/19/70	686,904	1.05	2,434,252	3,121,156	3.54/1
10/19/70	385,249	1.06	1,508,014	1,893,263	3.91/1
11/19/70	648,764	1.06	1,992,240	2,641,004	3.07/1
1/7/71	625,046	1.01	2,095,281	2,998,643	3.35/1

No diamond drill core assays have been cut in calculating intersection grades and the dilution shown is 25% of tonnage in all cases. The October 19th calculation used an ultimate pit depth to 2475 elevation, and the May 29th calculation 2425 elevation: in all other cases 2400 elevation was used.

Part of the November 19th ore tonnage represents material which cannot be extracted and this tonnage and corresponding stripping ratio are misleading. The general trend of decreasing ore reserves with increased density of diamond drilling is apparent in this table as for the Cowley Park deposit.

Keewenaw

The Keewenaw is a remnant of skarnified diorite close to the contact of the intrusive with the limestone and quartzite immediately west of the Gem deposit. Contoured assay plans for the benches mined to date indicate that the mineralization is an irregular pipe-like mass, surrounded by unmineralized skarn and diorite. The structure of the ore zone is unknown as there is no indication of dip in either the surface diamond drilling or in the pit.

The primary sulphides are bornite, chalcopyrite, chalcocite and covellite and one flake of native gold was reported. Abundant malachite and chrysocolla (copper carbonates and silicates) come from the upper benches in the pit between 2550 and 2650 feet. Several basic dykes, up to 15 m thick, cut the orebody.

Ore reserves remaining in the pit, after 15% dilution, have been calculated as 202,652 tonnes @ 1.06% copper with a stripping ratio of 1.61/1.0 (waste/ore). A 0.5% copper cut-off was used, and any diamond drill core assay above 3.0% copper was cut to 3.0% in these calculations. Cutting assays to this level may cause a slight underestimate of grade of ore remaining in the pit. The lower content of secondary copper minerals and the higher grade ore left in the deposit favour continued mining.

Mill test recovery figures on representative bench samples from diamond drill core rejects (table below) reflect the oxidised nature of the mineralization in the first three or four benches.

Bench Elevation	Bench Tonnes	Reserve Grade	Predicted Mill Recovery
2650-2625	14,250	0.65% Cu	33.0%
2625-2600	45,237	0.69% Cu	33.0%
2600-2575	64,243	0.88% Cu	33.0%
2575-2550	64,246	1.05% Cu	58.0%
2550-2525	71,093	1.07% Cu	85.0%
2525-2500	65,498	1.13% Cu	83.0%
2500-2475	55,438	1.10% Cu	85.0%
2475-2450	41,085	1.33% Cu	85.0%
2450-2425	33,397	1.16% Cu	92.5%
Total	454,489	1.03% Cu	

As these secondary minerals are bright greens and blues, the sorting of the ore, even below 2550 level, should occasion no difficulty. Much malachite is present on the waste dumps on the east side of the mine haulage road at the Wolf Creek Valley. The grade of this material is not high enough to warrant transportation and milling.

Keewenaw is the site of a small induced polarization peak, (+3 milliseconds) hardly recognizable as an anomaly, detected during the 1964 survey by Seigel Associates. The area has low magnetic readings as there is no magnetite associated with the mineralization.

Valerie

The Valerie showing is 3 km east of McRae (fig. 1). The orebody is narrow, probably 3-5 meters wide, and possibly 60 m long with a near vertical dip. The ore is at the contact of a serpentinized limestone, and diorite. The main copper mineral is chalcopyrite (there is no bornite) and arsenopyrite, pyrite, pyrrhotite magnetite, augite, garnet and calcite are also present. A strong magnetic anomaly exists near the old workings, with coincident induced polarization chargeability high. Three drill holes, two under and one into the old workings, produced disappointing results. North of the Valerie shaft a westerly dipping limestone, (fig. 12) underlain by quartzite, contains patchy serpentine skarn and traces of copper.

Little Chief

The Little Chief deposit is the largest ore body so far found in the Whitehorse Copper Belt with total mined reserves of about 7.25 million tonnes grading 1.5% copper (November, 1980). The ore body and two smaller bodies, Middle Chief and Big Chief occur within a flank pendant bounded on the northeast, southeast and southwest by diorite.

Magnetite serpentine skarn developed along the original contact between limestone and an underlying "quartzite" is the main host for copper mineralization at Little Chief. Both the intrusive contact and bedding dip northeast at about 70 degrees. (fig. 11). Most of the "quartzite", locally up to 120 m wide, is a clastic feldspathic sedimentary rock which grades into dioritized quartzite nearer the contact

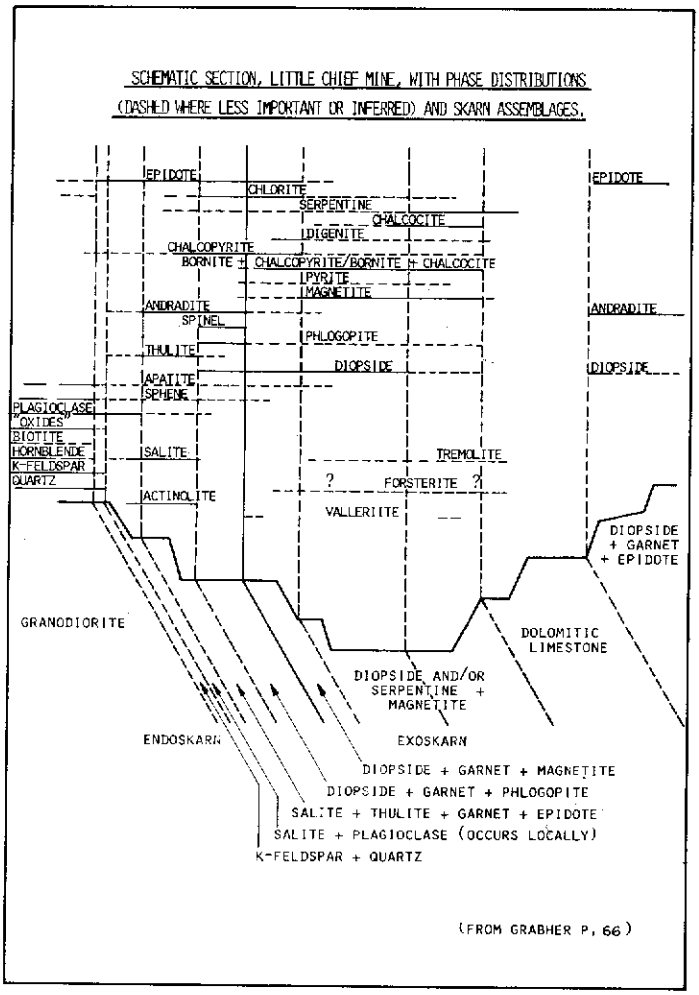


Figure 9. A zoned skarn sequence at Little Chief also showing the pit profile (from Grabher p.66).

with diorite. The "quartzite", dioritized quartzite and diorite may be partially or completely skarnified.

The quartzite nearest the ore is commonly strongly skarnified with apple green diopside, and locally mineralized with minor chalcopyrite. Pyrite makes up one or two percent of the "quartzite". This pyritic "quartzite" contains "trace" gold. Grabher (P. iii) has documented a zoned skarn sequence outward from the granodiorite as follows (fig. 9):

1) K-feldspar+quartz (+remnant plagioclase,magnetite, epidote)
2) salite+plagioclase+zoisite+epidote (+remnant plagioclase, sphene, bornite, chalcopyrite, actinolite, chlorite)
3) diopside+andradite+chlorite+phlogopite+serpentine (+talc, chalcopyrite, bornite, valleriite, pyrite, hematite)
4) diopside+andradite+magnetite-chlorite+serpentine (+bornite, chalcopyrite, degenite, chalcocite, epidote, talc, phlogopite, valleriite)
5) diopside+magnetite+serpentine (+bornite, chalcocite, chlorite, digenite, phlogopite, chlorite, chalcopyrite, valleriite).

A mineral paragenesis, also from Grabher, (p. 67) is shown in (fig. 10). The skarn contains bornite, chalcocite, covellite, chalcopyrite, valleriite, cuprite and native copper: traces of gold, silver, and molybdenite are present although molybdenite is more common in silicate skarns than those rich in iron. Native gold in one section of diamond drill core assayed in excess of 170 gm/tonne over 1.52 m. Surrounding drill holes, also assayed for gold, show no distinct zone of high gold values.

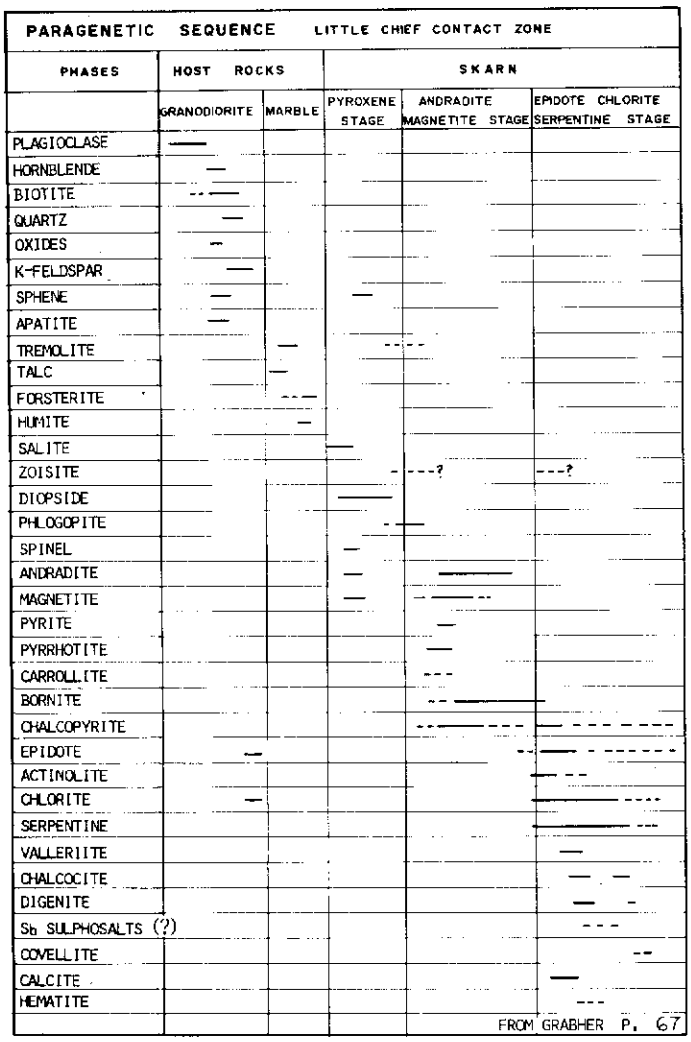


Figure 10. Paragenetic sequence at Little Chief (from Grabber p. 67).

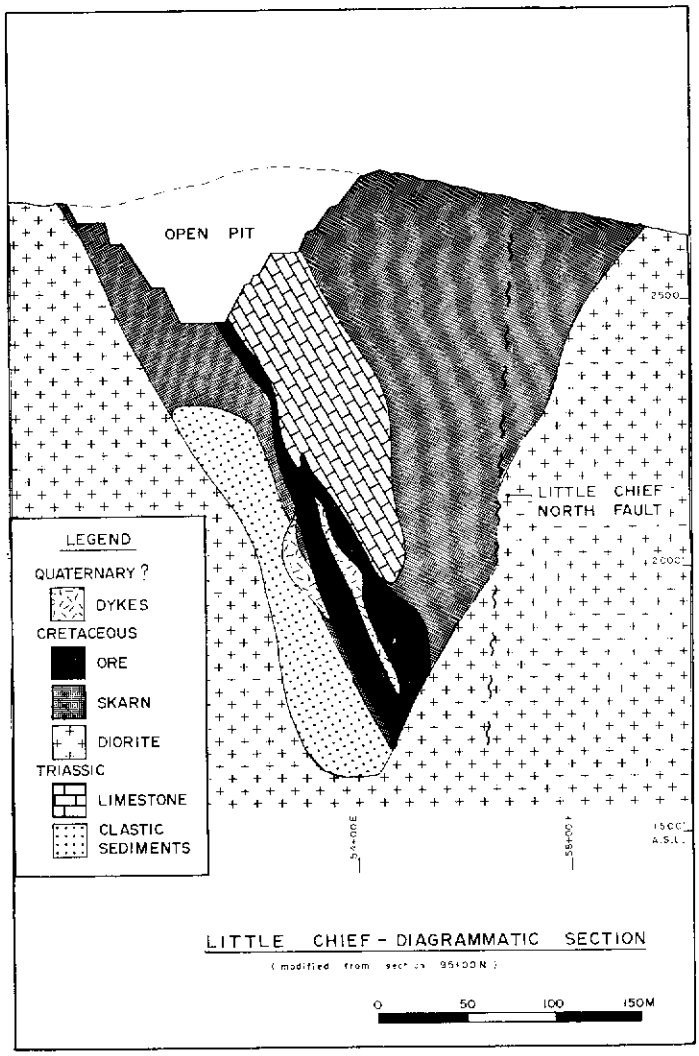


Figure 11. Schematic geological cross-section of the Little Chief flank pendant showing location of the open pit. The footwall side of the ore coincides with the limestone/clastic rocks contact. On the footwall this contact parallels the intrusive contact.

On the northwest, the Little Chief North fault, with 200 m of left handed strike-slip, bounds the deposits (fig. 14,17). This fault is vertical or dips steeply south and contains about 400,000 tonnes of ore (fig. 14). The richest parts of the Little Chief lie adjacent to this fault, (fig. 13), suggesting that the fault may have localized hydrothermal fluids.

Copper mineralization in the Middle Chief and Big Chief becomes weaker to the north, and the depth to which the magnetite skarn can be traced decreases. Whilst the Middle Chief dips to the east like the Little Chief, the Big Chief straddles the small Valerie Anticline (fig. 17).

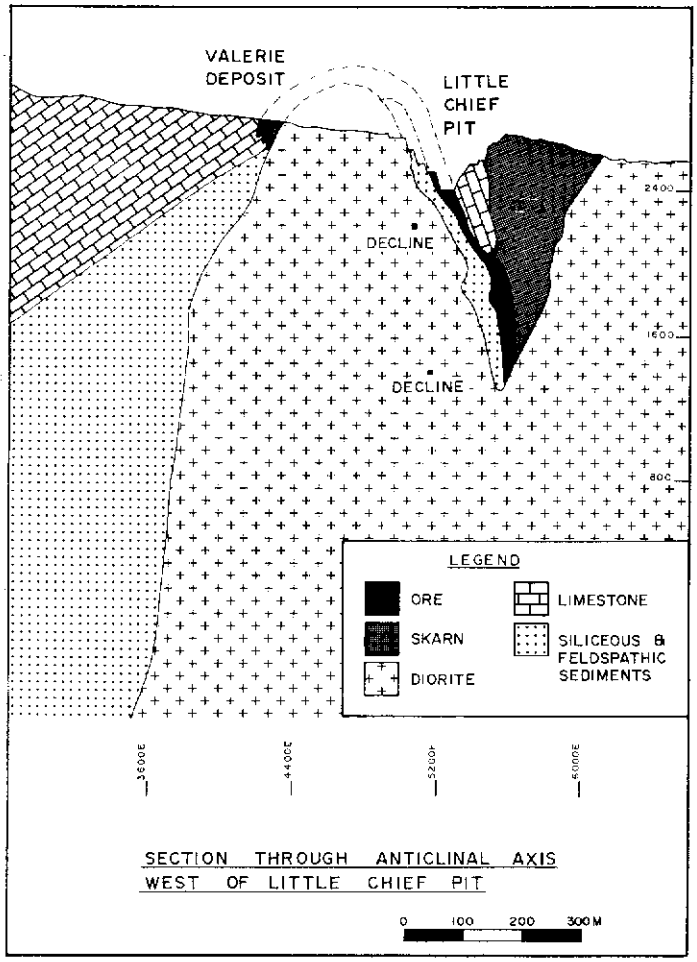


Figure 12. Geological cross-section through the anticline between Little Chief and Valerie. The diorite contact beneath the Valerie showings is vertical which places the Little Chief pendant on the flank of the Whitehorse Batholith. (c.f. Bear Cub deposits Fig. 6).

Magnetometer, EM-16, and induced polarization surveys over the Little Chief show anomalous responses (fig. 16). A magnetometer survey at 7 meter centres over Middle and Big Chiefs in 1970 gave readings 10,000 gammas higher than background, which is common over magnetite skarns. The EM-16 and I.P. anomalies over Little Chief are also well defined, though not particularly large. A magnetometer survey was not completed over the main open pit orebody before mining commenced, but high magnetic readings were obtained over the ore zone.

LITTLE CHIEF UNDERGROUND ORE RESERVES (undiluted)

Date	Source	Tonnes	Grade	Cut-off
1/5/67	New Imperial	3,131,609	@2.14% Cu	0.5% Cu
1967	Annual Rept.	4,535,929	@2.00% Cu	0.5% Cu
1968	Annual Rept.	4,535,929	@2.14% Cu	0.5% Cu
1969	Annual Rept.	4,535,929	@2.14% Cu	0.5% Cu
1970	Annual Rept.	2,580,944	@2.39% Cu	1.4% Cu
Jan'71	J.B. Howkins	1,947,111	@2.91% Cu	1.4% Cu
1971	Annual Rept.	2,048,607	@2.42% Cu	1.4% Cu
1972	Annual Rept.	2,432,041	@2.53% Cu	1.4% Cu
1973	Annual Rept.	2,345,656	@2.52% Cu	1.4% Cu

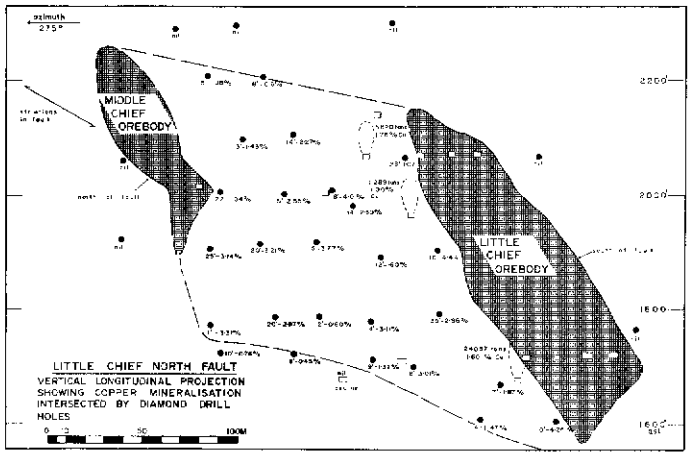


Figure 13. Vertical longitudinal prjection along the Little Chief North Fault (see Fig. 14 & 15) showing the location of ore dragged along the fault between Little Chief and Middle Chief ("B" zone). Some of this ore has been mined by block caving and the grade and tonnage extracted is shown. The direction of movement along the fault is indicated by striations, the distribution of ore in the fault and the relative locations of the two offset portions of the orebody.

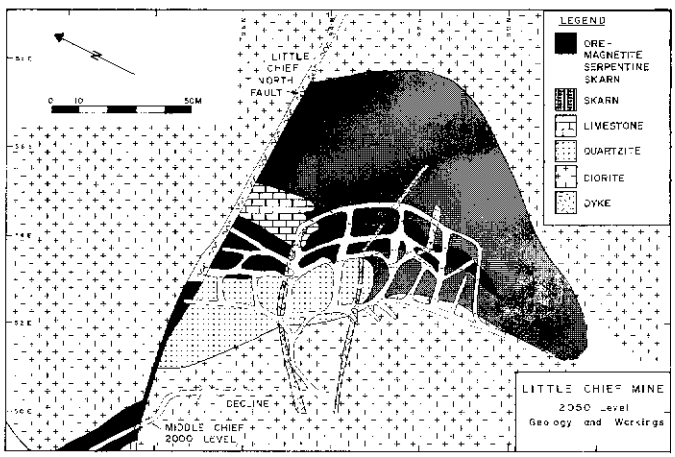


Figure 14. Geological plan of the Little Chief 2050 level showing mine workings and the relationship of "quartzite", ore, skarn and hangingwall limestone in the Little Chief flank pendant. The Middle Chief "B" zone lies north of the Little Chief North Fault. Ore dragged along the fault is also shown.

Deep reserves under the Little Chief Pit were first determined from twenty eight surface drill holes which ranged in depth up to 520 m. Cut-off grades have varied between 0.5% copper in the earlier calculations to 1.4% in later calculations. Tonnage factors used to convert ore volumes, measured on sections, to ore tonnage, also have varied between 244,112 and 256,960 cc/tonne and the higher figure is used now. Density of the ore is less than expected and this is one of the reasons that open pit tonnages are lower than projected. Figure 18 shows that density is related to copper grade, but the relationship is not considered strong enough to warrant using variable tonnage factors.

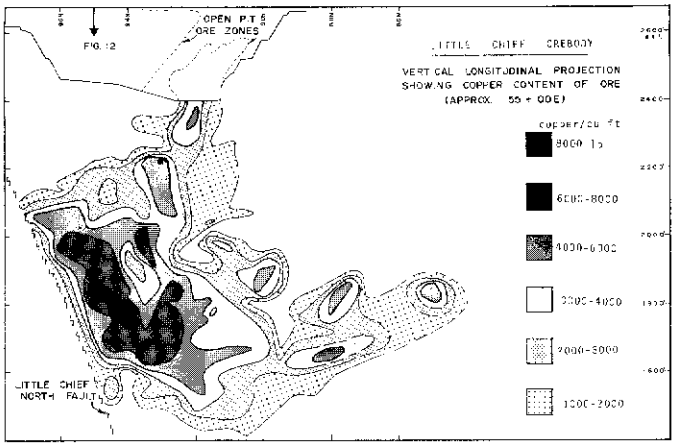


Figure 15. Vertical longitudinal projection of the Little Chief orebody showing the copper content of the ore. This is obtained at the location of each diamond drill hole by multiplying estimated true width of the ore by the grade of the intersection.

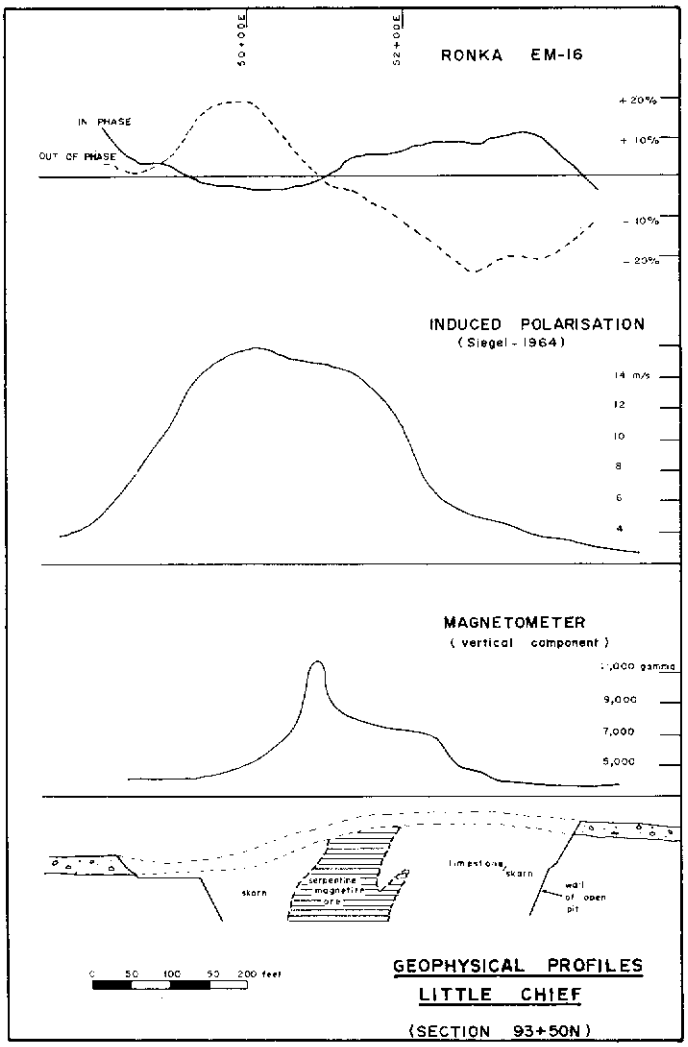


Figure 16. Geophysical profiles across the Little Chief open pit. The magnetometer and IP responses are characteristic. The E.M. - 16 profile is weakly anomalous.

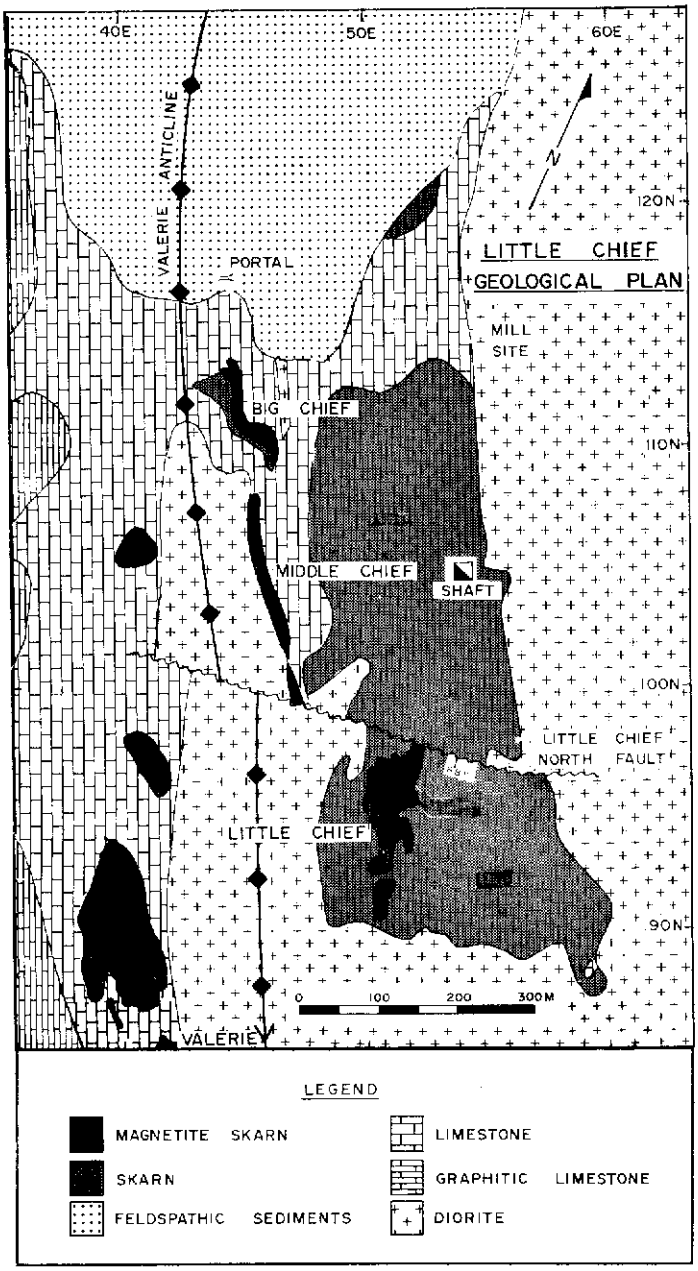


Figure 17. Geological plan of the Little Chief flank pendant, showing the Valerie Anticline. The Valerie serpentine magnetite skarns: they contain less magnetite than Little Chief and more serpentine. Note that the hangingwall limestone at Little Chief does not crop out at surface.

Copper and silver grades in composite diamond drill core samples from Little Chief show a strong correlation (fig. 19), which indicates silver occurs in solid solution with the copper sulphides: no significant quantity of native silver or silver bearing minerals has been identified. Gold shows poor correlation with copper (fig. 20).

Little Chief underground grade control is carried out by one geologist, who also keeps records. A reconciliation of underground ore reserves against production is given in Appendix IV.

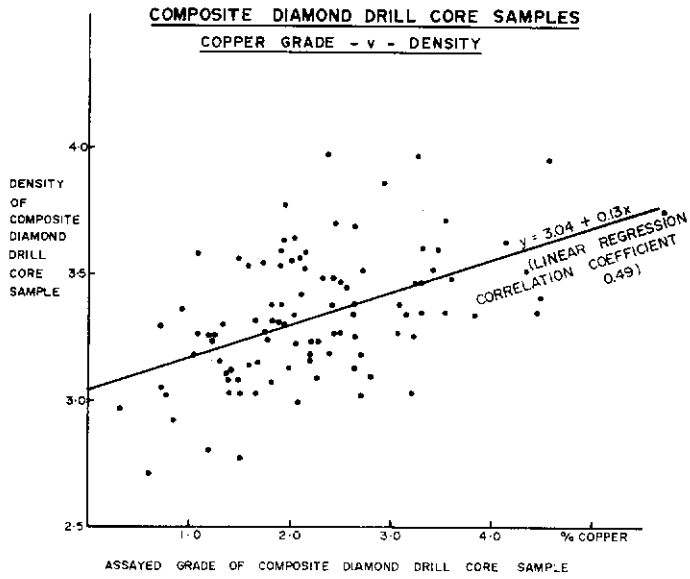


Figure 18. Pot of copper grade against density for composite underground diamond drill core samples. Note the regression line is almost the same as that in figure 32, suggesting that dilution has little influence on density of ore delivered to the mill.

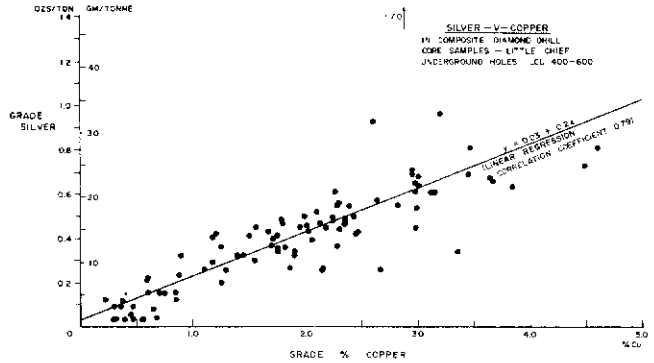


Figure 19. Plot of silver grade against copper grade in composite diamond drill core samples from the Little Chief underground mine. The strong linear correlation (factor 0.79) and the lack of identifiable silver minerals in the ore suggests that silver is present in solid solution in the copper sulphides and that the proportion of silver to copper is constant.

Arctic Chief

The two Arctic Chief pits mined by New Imperial Mines in 1968-1969 lie 4 km north of Little Chief on the west side of a roof pendant somewhat larger than that at Little Chief (figs. 21, 22). These skarns are small but contain massive magnetite and diopside with good grade copper, gold, and silver. The west pit contained a lens of magnetite skarn on the contact between limestone and a diorite. The east pit contained a steeply west dipping lens of magnetite skarn partially enveloped by limestone cut off to the south by a 12 m wide dyke. The dyke dips northward, and limits the southward extent of the small wedge of mineralization which remains under the pit floor. Deep drilling indicates that significant additional ore is unlikely. Magnetometer surveys over Arctic Chif define three anomalous zones, two of which are above ore bodies that have been mined: the third zone, Arctic Chief South, was too low grade to be economic.

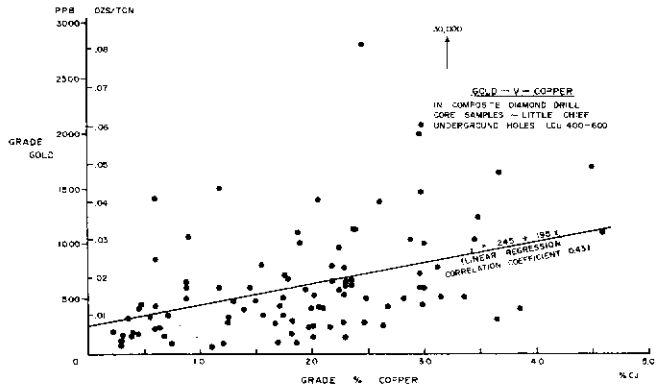


Figure 20. Plot of gold against copper for the same samples plotted in figure 19. The relationship is not as strong as for silver (linear regression correlation coefficient 0.43) which confirms limited visual observations of native gold in diamond drill core samples not being associated with unusual quantities of copper. The gold is generally free.

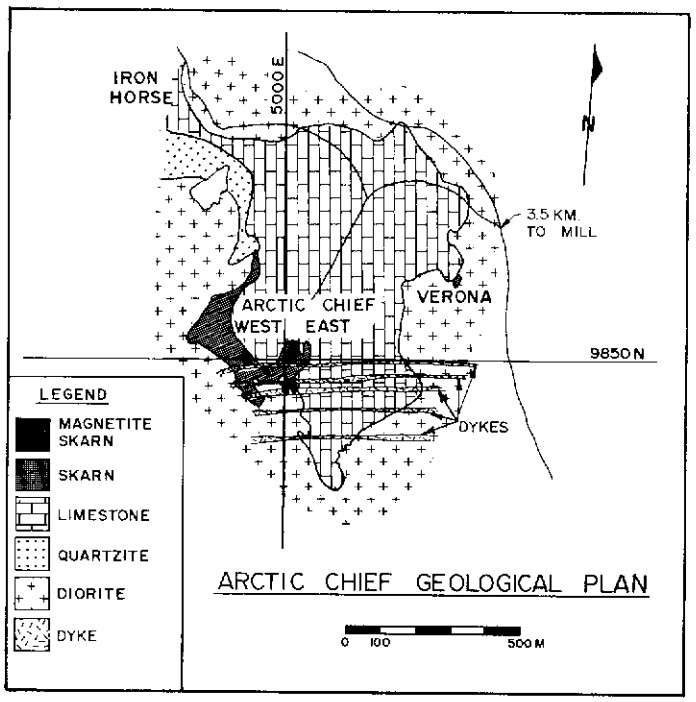


Figure 21. Geological plan of the Arctic Chief pendant.

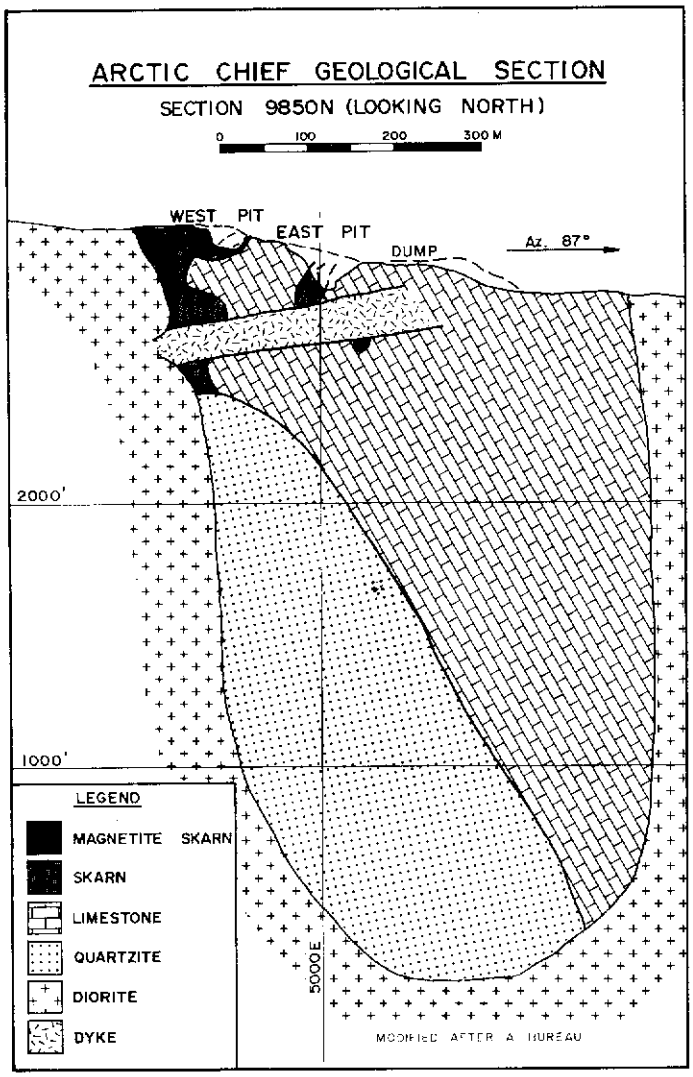


Figure 22. Geological cross-section of the Arctic Chief pendant. The East and West Arctic orebodies are further above the roots of the pendant than the ore at Little Chief or Gem/Bear Cub. The limestone/sediment contact is not parallel to surrounding intrusive contacts and no large orebodies are expected lower in the pendant.

Grafter - Best Chance

Several small deposits including the Grafter and Best Chance lie along the diorite contact north of Arctic Chief. The Grafter was developed by a shaft and some high grade was produced early this century. At Best Chance some high grade was produced from near surface. The area is noted for the complexity of the diorite contact and for the widespread distribution of copper. Surface drilling by Hudson Bay on the Grafter in 1974 intersected some good copper mineralization but left the geological structure largely unresolved. Two deep holes under the Best Chance, drilled the same year, were discouraging.

Early surface drilling by New Imperial located more than 270,000 tonnes of low grade mineralization in the Best Chance deposit, but no significant reserves are known with other showings. The mineralization comprises chalcopyrite and bornite in a massive magnetite serpentine skarn. There are few dykes at Best Chance and drill core indicates better than normal ground conditions. In the original Best Chance ore reserve calculations, some averaging grades relied on the inclusion of one high grade assay in an otherwise low grade intersection to give mineable grade over a considerable width. Experience shows that this leads to ore reserve blocks with excessive grades. In recent ore reserve calculations high assays were cut to 3.5% copper before determining the average grade of the intersection. In low grade blocks, assays were cut to 1.0% copper before determining average grades.

The Best Chance has a strong magnetic response divided into northern and southern parts - the northern part of the anomaly indicates a dip to the west. An induced polarization chargeability high, stretching northwards through Empress of India to Spring Creek correlates with the magnetics.

Pueblo

The Pueblo was the largest of the old showings mined before World War I. The mineralization is unusual because iron oxide occurs as specularite, and the only copper sulphide is chalcopyrite - there are abundant "oxides" mainly malachite and azurite. The ore is in a steeply (easterly?) dipping series of lenses on the contact between limestone and diorite. Plunge is to the north from the surface open pit (figs. 23,24). The area is faulted, and the faults caused the disastrous cave-in in 1917 which shut the mine down. Deep drilling on the showing by Noranda in 1947 and Hudson Bay in 1973-74 failed to substantiate claims of ore in veins north of the underground workings on 525' level. Old blueprints of surface installations and underground level plans were recovered by New Imperial in 1970. The deposit has no magnetic response and produces a weak induced polarization anomaly.

War Eagle

War Eagle is a typical calc-silicate skarn containing no iron oxide, or serpentine. Its mineralogy is like that of Cowley Park. Garnet, tremolite, actinolite, wollastonite skarn have developed in layered sedimentary rocks comprising quartzite, arkose and limestone. Limestone occurs in the footwall of the ore in the east wall of the War Eagle Pit unlike most other deposits. War Eagle is noted for the relative abundance of molybdenite and chalcopyrite. F. M. Smith has calculated the molybdenite grade as 0.038%.

Bedding is well developed with interbedding slips dipping 65o west on the hangingwall (west) wall of the Pit, and the skarn is banded. The ore is abrasive and caused high wear in the mill and the elasticity of the tremolite/wollastonite crystals in the skarn slowed percussion drilling. The loss of ore reserves in the War Eagle South pit was caused by averaging narrow mineralized intersections together to produce ore blocks of mineable width. The mineralization is in discontinous lenses and grade was consequently lower than expected. Some unblasted mineralization inaccessible to open pit mining remains in the bottom of the now flooded War Eagle

North Pit. Deep drilling by Hudson Bay in 1972 failed to indicate significant downward continuation of this mineralization.

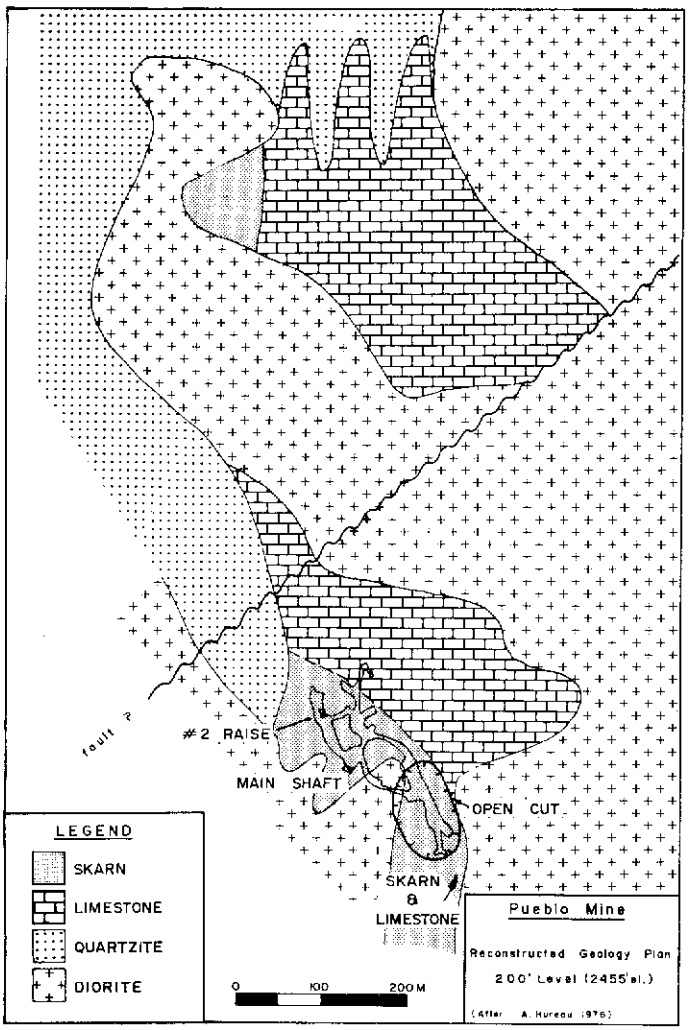


Figure 23. Geological plan of the Pueblo - Gulch area showing the Pueblo pendant, open pit and underground workings. The fault shown is speculative with unknown movement.

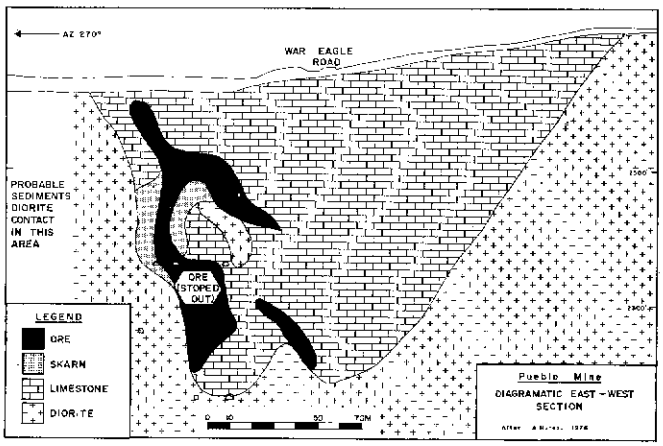


Figure 24. Schematic geological cross-section of the Pueblo pendant. Down dip potential for more ore is limited as the diorite contact cuts off the sedimentary rocks (after A. Hureau, 1976).

EXPLORATION METHODS

Summary of Exploration Methods

Magnetic methods of exploration were successfully applied to finding the magnetite serpentine skarns of the Whitehorse Copper Belt. Early dip needle surveys located Arctic Chief East and the northward extension of the Valerie and more recently the fluxgate magnetometer was responsible for discovering the Gem-Black Cub South Zone. Many of the larger airborne magnetic peaks reflect the Miles Canyon Basalt. Because the ore bodies are weakly conductive, effective electromagnetic methods are limited to the higher frequency instruments such as the Ronka EM-16. This instrument indicated anomalous conditions over the Little Chief. Induced polarization has been successful, and gives high chargeability readings over the ore bodies. However, many other rocks also give high readings. Resistivities are variable, the serpentinous skarns being relatively lower than the silicate skarns. An airborne magnetic/electromagnetic survey flown by Lockwood identified the magnetite but not the silicate skarns. A relatively high frequency was employed and some electromagnetic anomalies detected by this survey may be caused by conductive overburden.

The magnetite in the iron rich skarns leads to their detection by magnetic methods of exploration. Disseminated copper sulphides in the iron-poor skarns do not constitute an electromagnetic conductor and this type of deposit is detected by induced polarization. The small size of the ore bodies prevents detection at depth by magnetometer or induced polarization.

Geochemical exploration was introduced on the copper belt in 1969. Sampling was mainly confined to lines cut at 122 m intervals, with soil samples taken every 30 m. Numerous anomalous zones have been detected in the "B" soil horizon with total copper concentrations ranging from 20 to over 10,000 ppm. Common background concentrations are up to 15 ppm. Normally high values lie in "train shaped" dispersion patterns commonly related to topography. These are thought to be mainly syngenetic patterns formed by glacial transport of the overburden. Hydromorphic anomalies are rare and limited in extent.

Exploration in areas of transported glacial overburden has been limited to tracing anomalous soil values in the direction of the last ice retreat to a cut-off where the source of the anomalaous metal is sought in the bedrock. Although smeared glacial soil anomalies are present in till down ice from several deposits including Cowley Park (fig. 25), Pueblo (fig. 26) and War Eagle (fig. 27) no new deposits have been found using this technique. Geological guides to ore have been used with success at North Star where surface diamond drilling discovered low grade copper mineralization on a favourable limestone quartzite contact 400 m below surface.

Magnetic Methods

The Whitehorse Copper Mines property has been surveyed using a fluxgate magnetometer and a G-816 proton magnetometer. Lines were 122 metres apart at the most. Results have been plotted on scales of 1:480 and 1:4800. The more detailed scale was used in zones of actual and prospective mineralization where detail was required: instrument readings were taken at 7.5 m centres on lines 7.5 m apart. This

technique was useful in defining the structure of the complicated Gem deposit. All surveys were corrected for local magnetic variation using:

1) a second stationary magnetometer recording diurnal variation
   or
2) tie in to a previously surveyed base line or other local magnetic station.

The second technique is the simpler and consequently more popular method, and has been used since 1970.

Because magnetite serpentine skarns commonly give peak magnetic readings of 10,000 gammas above background, variations of less than 100 gammas are not considered important except for the interpretations of structure. Surprisingly, however, the Cowley Park deposit, which is an iron poor silicate skarn, does give a weak but sufficiently well formed magnetic anomaly to define three "en echelon" zones of mineralization. Peak values on the three zones ranged from 1000 to 2000 gammas above background.

The anomaly is caused by magnetite in the skarn. Miles Canyon Basalt has also been responsible for high magnetic readings, but usually these readings are uniformly high and co-extensive with the volcanics. In most cases peak values range up to 4000 gammas above background depending upon the amount and concentration of magnetite. Some Miles Canyon volcanics do not produce any magnetic anomaly either on ground or airborne magnetic surveys. Over Miles Canyon itself, no airborne magnetic anomaly exists on the 1" = 1 mile government aeromagnetic sheets. (No flight lines actually cross the canyon). The basalt near Little Chief is not notably anomalous.

TABLE I

	Magnetic Readings Peak	Average	Comments
Cowley Park	+2000	+500	0 - 7 m O/B*
Black Cub S.	+10,000	+8,000	9 m O/B
Black Cub N.	+5,000	+3,000	9 m O/B
Kodiak Cub	+9,000	+2,000	9 m O/B
Gem	+10,000	+2,000	Patchy ore
Keewenaw	0	0	Weak gradient
Wolf Creek Basalt Plug	+3,000	-	
Valerie Shaft	+3,000	+1,000	Near dump
Valerie N. Zone	+2,000	+1,000	25 m O/B
Little Chief	+9,000	+3,000	Survey incomplete
Middle Chief	+20,000	+5,000	Exposed at surf.
Big Chief	+15,000	+3,500	Exposed at surf.
Arctic Chief E.	+23,000	+10,000	6 m O/B
Arctic Chief W.	+30,000	+6,000	Minor O/B
Best Chance	+27,000	+7,000	Massive magnetite
Pueblo	0	0	No anomaly near pit
Copper King	0	0	
War Eagle	-200	-200	Relative low near magnetic diorite
Diorite Magnetic	+1,000	0-1,000	
Basalt (Miles Canyon)	+4,000	0-4,000	
Limestone	0	0	Uniform low rdgs
Quartzite (etc)	+500?	0-500	Weakly variable

* O/B - overburden

Lewes River strata generally have a lower magnetic response than the surrounding rocks. The airborne survey flown by Lockwood for New Imperial Mines in 1966 defines a magnetic low in the sedimentary rocks west of Little Chief, where limestone and some quartzite crop out at surface. Intrusive rocks of monzonitic/dioritic composition give a variable magnetic response depending upon the amount and concentration of magnetite. The intrusive at War Eagle and at Cowley Park are magnetic whilst those at Little Chief are relatively non-magnetic. Table I gives peak and "average" magnetic readings for the main deposits and some of the rocks on the Whitehorse Copper Belt. The magnetic skarns are indicated by their obviously high magnetic effect. Readings are in gammas.

Induced Polarization

The first induced polarization survey on the mine property, done for New Imperial Mines by H.O. Seigel Associates in 1964 detected chargeability peaks over the then known deposits and the technique has since become standard procedure on the Copper Belt. Both pyrite and graphite have caused problems in interpreting induced polarization results. In general it is hoped to distinguish anomalies caused by sedimentary rocks from those over skarn deposits by using:

1) size - ore deposits are rarely more than 200 m long and 30 m wide (zones containing ore deposits may, however, be up to 800 m long).

2) intensity - ore zones tend to exhibit moderate peak chargeability values over a limited area. Sediments may produce high values over large areas.

3) geological location - is important: diorite and and limestone must be present.

4) shape - because ore zones are found in roof pendants, isolated areas of anomalous chargeability readings constitute prime exploration targets.

5) magnetometer readings more than 5000 gammas above background almost certainly indicate a magnetite serpentine skarn. Such skarns surveyed by I.P. have given high chargeability readings.

Table II gives peak and "average" chargeability readings over the main deposits of the Whitehorse Copper Belt (figures are in milli-seconds).

Resistivity readings calculated from the same survey have no general use in locating orebodies, but are sometimes useful to determine the position of geological contacts. Anomaly contrast is normally good except where host rocks contain graphite or pyrite as at Black Cub South: here contrast is poor. A similar situation exists at War Eagle.

V.L.F. Electromagnetic Methods

The Ronka EM-16 is a one-man electromagnetic instrument that relies on long wave broadcasts to submarines from radio transmitting stations stratigically located in the western hemisphere. Frequencies in use, however, are high by geophysical stan-

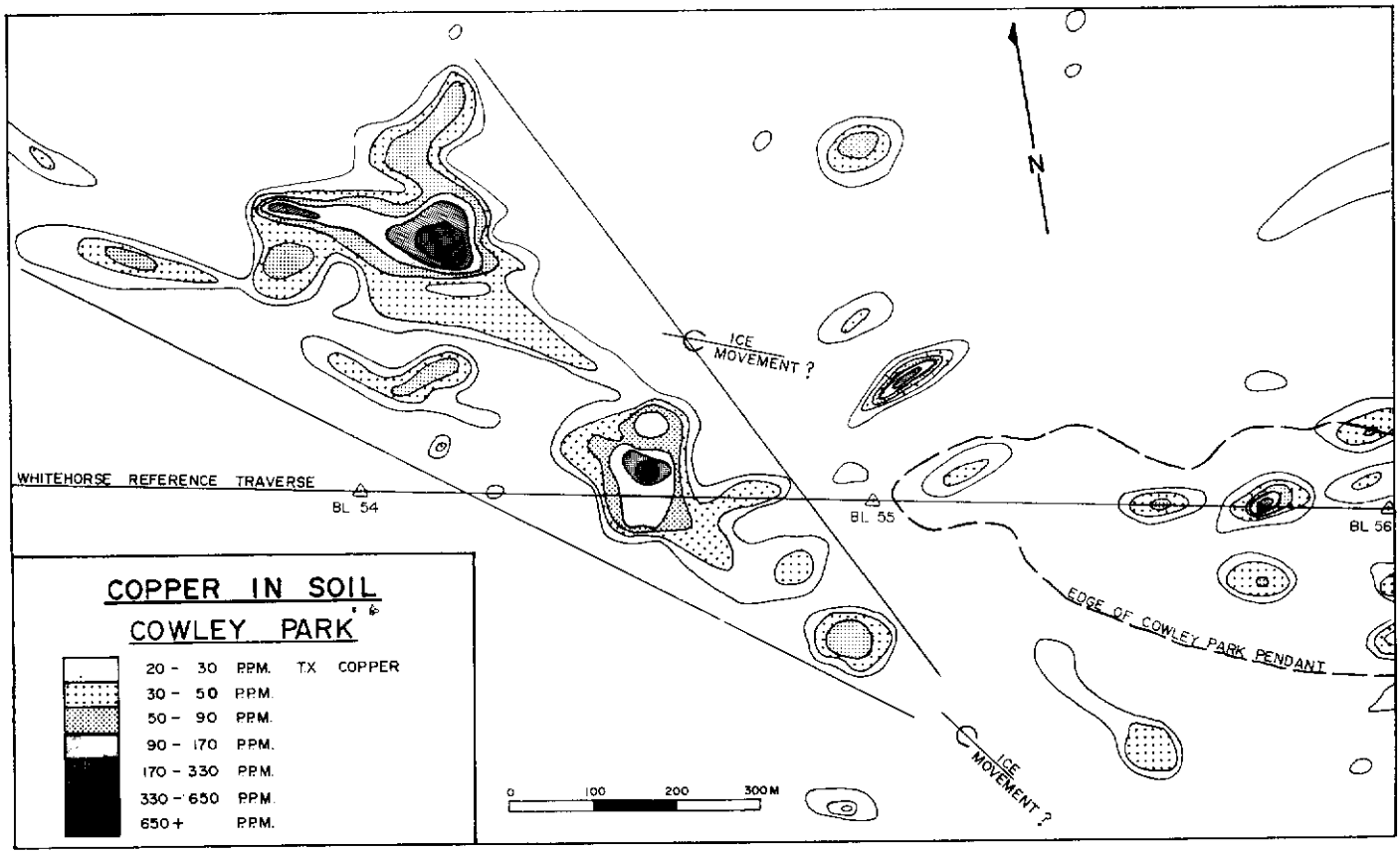


Figure 25. Anomalous fan shaped copper dispersion pattern in overburden at Cowley Park. Two possible directions of ice movement based on this data are indicated. The edge of the Cowley Park roof pendant is shown as it is the likely source of the anomalous copper. These copper anomalies would have led to discovery of Cowley Park were the deposit not already known.

TABLE II

INDUCED POLARIZATION SUMMARY

	Anomaly Contrast	Chargeability Peak	Avge	Bg.*	Survey
Cowley Park	v.good	11	8	2-3	Seigel'64
		13.4	7	2-3	Can. Aero
Black Cub S.	n.d *	11	-	2-3	Seigel'64
		26	20	5-8	Can. Aero'68
Black Cub N.	fair	24	18	5-8	Can. Aero'68
Kodiak Cub	good	20	-	5-5	
Gem	good	10	6	2-3	Seigel'64
	good	8	6	1-2	Can. Aero'68
Keewenaw	poor	6	-	2-3	Seigel'64
Valerie	good	7	-	2-3	Seigel'64
Valerie N.	good	12	7	2-3	Seigel'64
Little Chief	v.good	20	10	2-3	Seigel'64
Middle Chief	n.s. *	-	-	-	
Big Chief	n.s. *	-	-	-	
Arctic Chief E.	n.s. *	-	-	-	
Arctic Chief W.	n.s. *	-	-	-	
Best Chance	v.good	12	7	2-3	Seigel'64
Pueblo	weak	4	-	.1-.2	Can. Aero'68
Copper King	fair	8	6	2-3	Seigel'64
War Eagle	fair	22	13	2-3	Seigel'64
War Eagle	fair	18	12	0-3	Can. Aero'68
Diorite		-	-	2-3	Can. Aero'68
Diorite		-	-	2-3	Seigel'64
Limestone		23	13	2-3	Seigel'64

* n.d.not determined/n.s.not surveyed/bg.background

dards, and overburden and topography may indicate anomalous conditions. The instrument is not favoured for use on the copper belt as better results can be obtained using induced polarization and the magnetometer. The EM-16 has been used to advantage in locating airborne electromagnetic anomalies on the ground where heavier, more reliable EM equipment would have been slower. A test survey over the Little Chief Pit area did give anomalous readings of moderate magnitude (fig. 16).

Geochemical Methods

Geochemical exploration on the copper belt has not been successful in locating new copper ore bodies. One reason is the transported nature of the overburden and of the geochemical anomalies preserved in it. The nature of the overburden is largely controlled by recent glacial history, and this history explains the glacially transported anomalies. Reconnaissance geochemical soil sampling has been done on most of the property on picket lines at 122 m intervals, with sample points 30 m apart along the lines. In areas of detailed sampling the lines are spaced at 61 m intervals. Soil samples were taken from the rusty red "B" soil horizon or the "C" horizon which is generally light yellowish brown. In swampy areas where these were not available organic material of the "A" horizon was taken. The "B" horizon is generally within 30 cm of surface. Permafrost exists on small areas on the

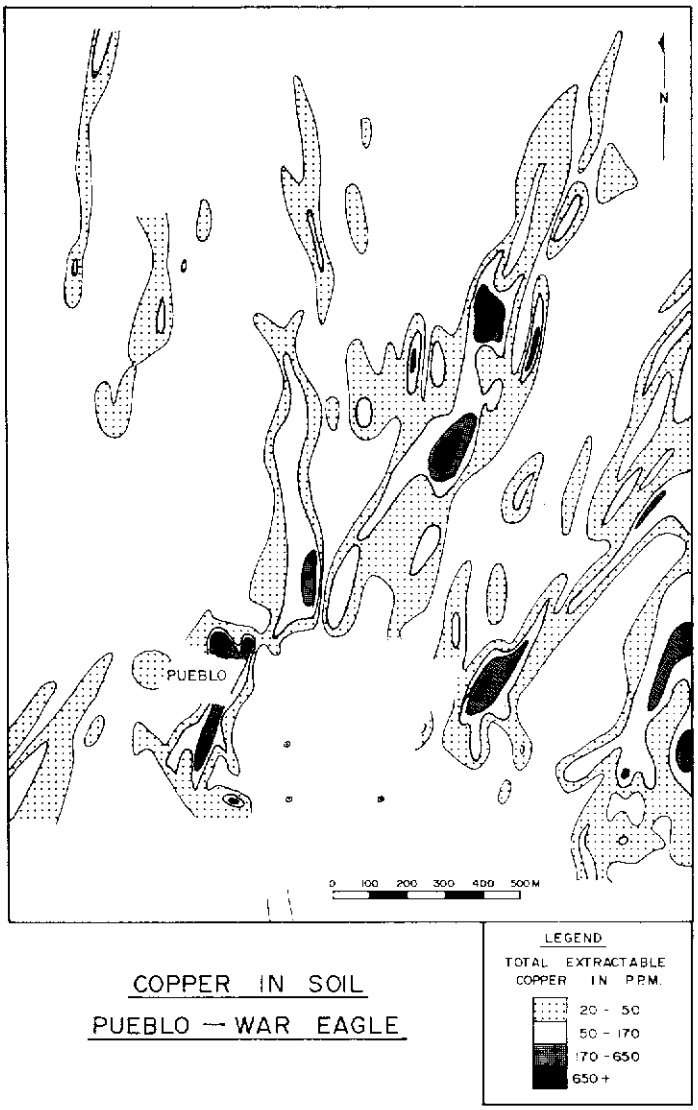


Figure 26. Anomalous train shaped copper dispersion patterns in overburden at Pueblo. These patterns, which may be modified by glaciofluvial action, indicate the direction of ice movement during the last retreat of Wisconsin ice.

east side of Mt. McIntyre, but did not hamper sampling. Sample locations missed in first attempts were resampled successfully later in the summer. Samples were taken using a grub hoe, but a 1.2 m long soil auger was useful especially when resampling the permafrost areas. Samples were dried and sieved through #80 mesh sieve and the fine fraction was analysed for total copper by atomic absorption. Standard samples were analysed, once in every 50 samples, with good repeatability of results: the variation of the 50 ppm standard was over a range of about 4 ppm copper. Background values could be determined from the results by inspection of the plans. Areas with higher copper concentrations which could be contoured were regarded as anomalous. Threshold values are generally in the 15-20 ppm copper range, with "background" lying below and "anomaly" above this range. The mathematical approach using "mean" and "standard deviation" produced threshold and anomalous limits which were too high. Areas of significantly anomalous copper concentration in the soils have been located from Cowley Park to War Eagle. (figs. 25, 26, 27).

Anomalous values in the 200-500 ppm copper range are common, and several samples analyzed in excess of 1000 ppm copper. The shape of anomalous areas commonly shows good correlation with topography: west of the Cowley Park Deposit and north of Reservoir Lake on the Fish Lake Road, are strong train shaped anomalous zones defined, and partly confined by, the small valleys in which they occur. They may represent "fossil" stream sediment anomalies; changes in ph or eh of the soil which has caused precipitation of copper; an erosionally uncovered layer of anomalous glacial overburden not otherwise exposed at surface or glacial smear of material derived directly from copper deposits.

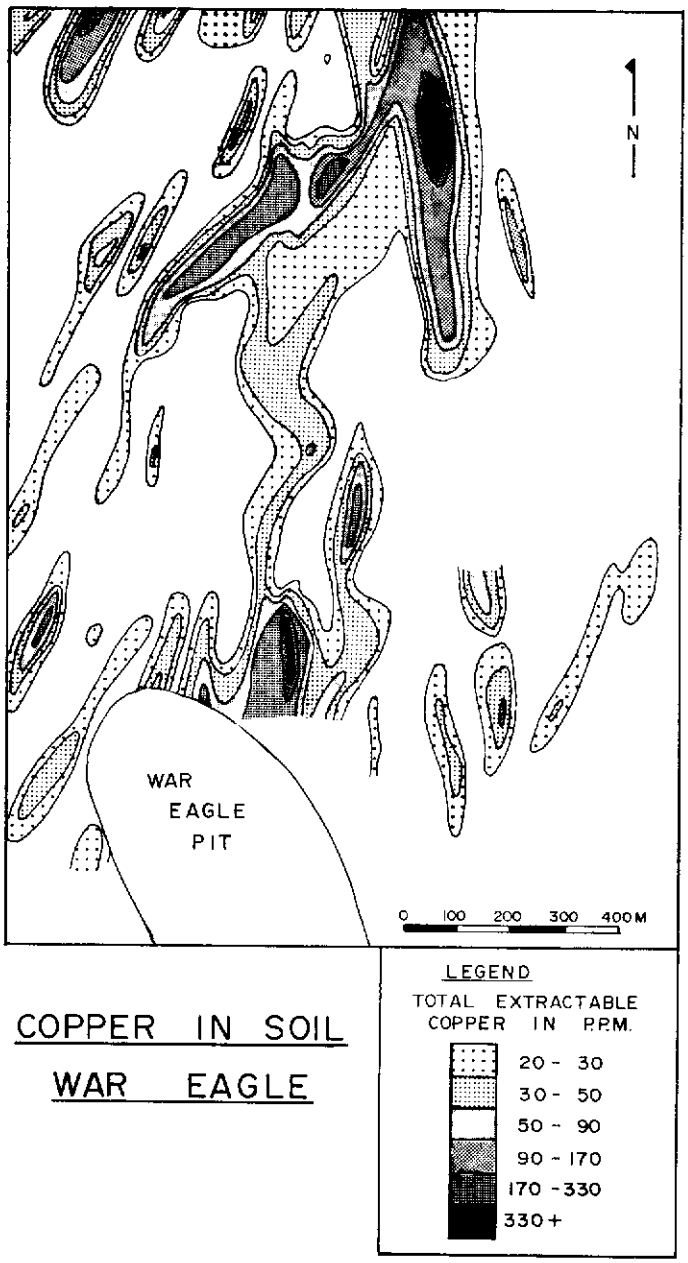


Figure 27. Anomalous train shaped copper dispersion patterns in overburden north of the War Eagle pit indicating the direction of retreat of the last ice to cover this area. The anomalies are interpreted as glacial smear from the War Eagle deposit.

The origin of the overburden is important as it controls the size and shape of syngenetic soil anomalies. Sandy glaciofluvial deposits do not contain recognisable syngenetic geochemical dispersion patterns whereas ground moraine produces many such patterns. The further anomalous material is transported from its source, the more it is diluted and the smaller the chance of geochemical detection. The best geochemical prospecting medium in glaciated terrain is mechanically transported morainic overburden or till. It shows least dilution of the source material and the best contrast with the surrounding non-anomalous areas. Chemical transportation of anomalous concentrations of copper results in epigenetic anomalies. Since their magnitude and extent is limited largely by the nature of the transporting agent, mainly groundwater or vegetation, and they have generally not proved as useful in soils as syngenetic patterns.

OPEN PIT MINING

Table III lists expenditures by New Imperial Mines for the period May 1, 1967, the start of mill production, to the end of 1971, with an extraordinary write-off in 1972. When open pit milling ceased on June 30, 1971, only part of plant and equipment costs had been depreciated as it was planned to charge remaining depreciation against the Little Chief underground orebody on a tonnage mined basis. As the underground Little Chief orebody is higher grade than those mined by open pitting, higher earnings would have been reported for open pit operation if depreciation had been charged on the basis of the quantity of copper produced. This is not standard accounting practice.

Open Pits - Income, Expenditures and Earnings

Table III

	Income $ (Net Smelter)	Expenses* $	Earnings $
1967	3,220,461	2,052,079	1,168,382
1968	5,679,852	4,421,117	1,258,735
1969	10,096,236	5,720,784	4,375,452
1970	8,329,003	8,290,130	38,873
1971	2,241,061	6,451,720	(4,300,659)
1972	-	1,812,649	(1,812,649)
TOTAL 1967-1971	29,566,613	28,748,479	818,134

* production and administration costs, royalty, depreciation, interest payments.
N.B. Underground operations charged to the joint mining venture do not appear in this table (see Appendix I for breakdown of costs).

Open Pits - Production

Ore Milled	Tonnes	Grade	Kg. Cu Produced	Mill Recovery
1967	410,996	1.16% Cu	3,317,667	70.6%
1968	664,146	1.03% Cu	5,515,520	80.6%
1969	730,755	1.09% Cu	6,880,754	86.5%
1970	773,340	1.04% Cu	7,295,912	91.0%
1971	306,409	1.02% Cu	2,375,345	75.8%
TOTAL	2,885,648	1.07% Cu	25,385,198	82.6%

The above figures, which are incorporated into appendix II, show a low mill recovery in 1967 obtained from oxidized ore in the top benches of the Little Chief Pit. The higher copper recovery in 1970 reflects the unoxidized and freer milling nature of mineralization at War Eagle.

Open Pits - Ore Reconciliation

Ore reserve and production tonnages in Table IV agree reasonably well and are within the expected accuracy limits, and production tonnage and milled tonnage agree well. The grade loss in production may result from ore lost in blasting being replaced by waste. Assuming ore reserve grades are correct about 1/3 of the ore would have to be replaced by waste to obtain the observed mill head grade. This is unlikely, and either the original reserve grade is too high or incorrect interpretations of ore boundaries were used. The average grade of mill feed from open pits was 1.07% copper, or 75% of the undiluted ore reserve grade of 1.43% copper (Table IV).

Table IV

Undiluted Ore Reserves			Undiluted Blasthole Prod.		
Tonnes	% Cu	Copper million Kg.	Tonnes	% Cu	Copper million Kg.
Little Chief Pit					
1,177,697	1.28	15.075	1,436,121	1.29	18.586
Arctic Chief E.					
99,588	1.84	1.833	84,962	1.58	1.343
Arctic Chief W.					
120,010	1.73	2.076	137,652	1.36	1.872
War Eagle S. *					
93,607	1.42	1.329	80,010	1.12	0.896
War Eagle N.					
746,633	1.60	11.946	814,725	1.27	10.347
Black Cub South *					
167,289	1.65	2.760	180,274	1.33	2.398
Keewenaw *					
163,459	1.03	1.679	159,086	0.95	1.511
TOTAL					
2,568,278	1.43	36.698	2,892,832	1.28	36.952
Milled: 1967 - 71					
2,885,648	1.07	30.742			

grade factor = 1.07/1.43 x 100 = 74.83

* Mining incomplete - only those benches mined are included in reserves.

Gains and losses in grade and tonnage are plotted for all pit benches mined (fig. 28). There is a tendency for benches to show a loss in tonnage and grade on mining which is partly offset by some benches which show very large tonnage gain. A few benches show a gain in grade and tonnage.

Agreement between the quantity of copper mined and that predicted in reserves improves with the size

ORE RECONCILIATION

BENCHES MINED FROM OPEN PITS 1967 - 1971

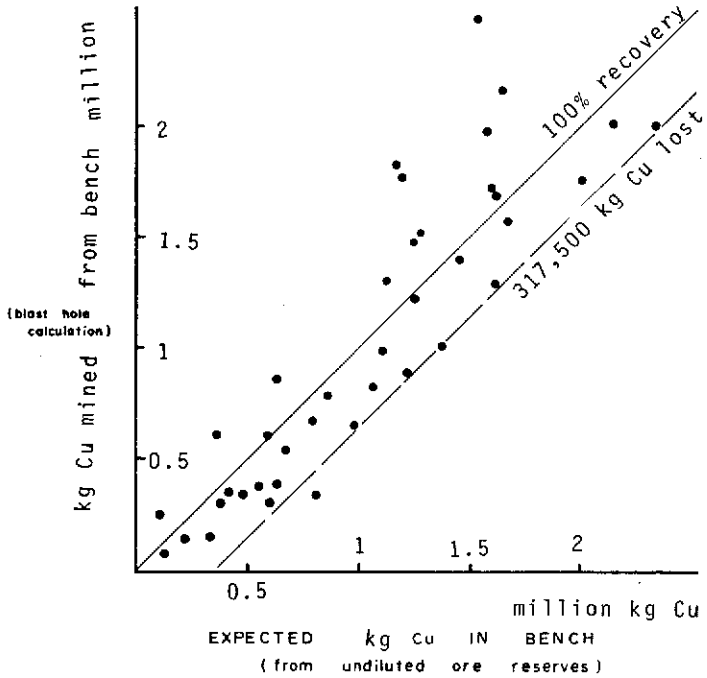


Figure 28. Open Pit ore reconciliation by benches showing the quantity of copper in undiluted ore reserves plotted against the quantity of copper mined according to uncorrected blasthole production calculations.

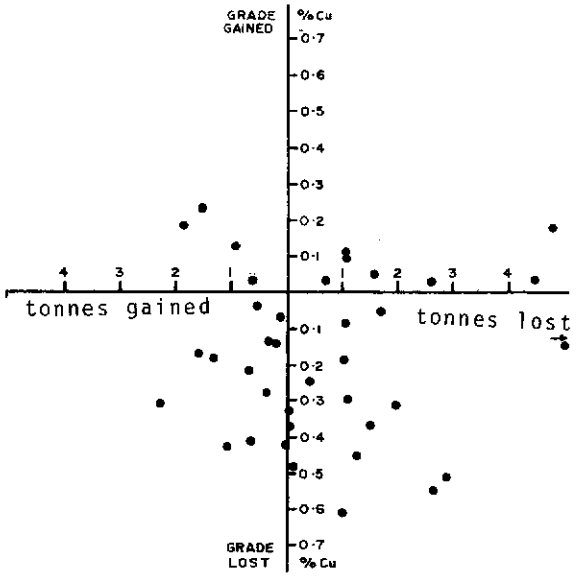


Figure 29. Open pit reconciliation by benches showing blasthole indicated grade and tonnage produced as a gain (upwards and right) or loss (downwards and left) when compared with undiluted diamond drill reserve grade and tonnage.

of the ore block. This suggests that losses in grade result from misinterpretations of the shape of thinner, irregular and discontinuous sections of an orebody. One of the benches showing the largest gains in tonnage and grade mentioned above, cut through bulkiest part of Black Cub South orebody. War Eagle South, a patchy discontinuous deposit, by contrast, shows some of the largest losses although grading was done the same way. Grading of reserve blocks in such instances depends on experience. Some of the losses recorded against the Keewenaw Pit were caused by oxidation of the ore. Figures 28 and 29 show blast hole indicated gain or loss in grade and tonnage separately. The foregoing suggests that the Little Chief #7 stope and #7 pillar, two of the bulkiest parts of the underground orebody, should produce close to the predicted grade and tonnage in reserves. This proved correct for #7 stope, but not for #7 pillar which was extracted by block caving techniques and suffered from excessive dilution.

Density

Specific gravity is quoted by Wright Engineer's feasiblility report November 1965 as:

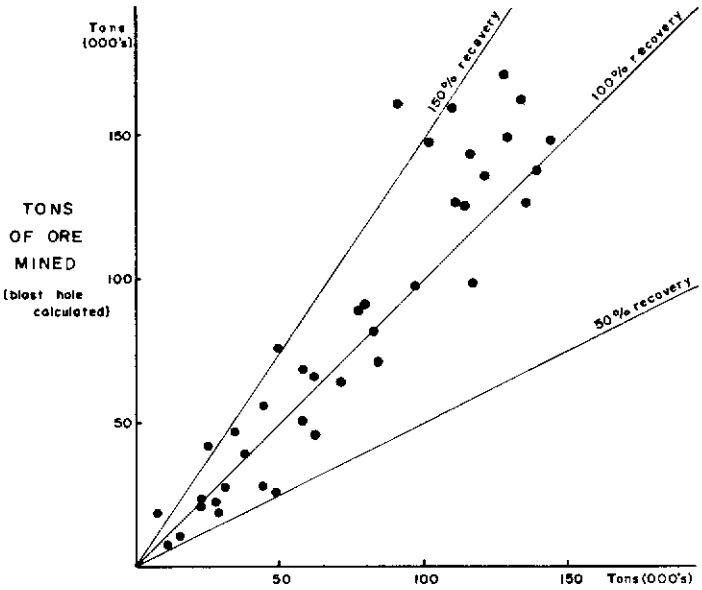
	Specific Gravity	cc/tonne
Little Chief	3.53	245,397
Arctic Chief	3.63	226,382
War Eagle	3.01	273,148
Cowley Park	3.09	266,210
Keewenaw	2.75	299,101
Best Chance	3.59	228,951
Waste Rock	2.91	268,656

These figures probably originate from composite diamond drill core samples, and they have been used in all diamond drill and pit blast hole calculations up to the end of June 1971. Factors used for the Little Chief underground ore body have varied from 244,112 to 256,960 cc/tonne.

The density of mill heads was determined during 1973 on the weekly composite. As expected, density of mill heads is related to copper grade (fig. 32). Specific gravity determinatiosn done on composite pulps from diamond drill intersections show the same relationship (fig. 18).

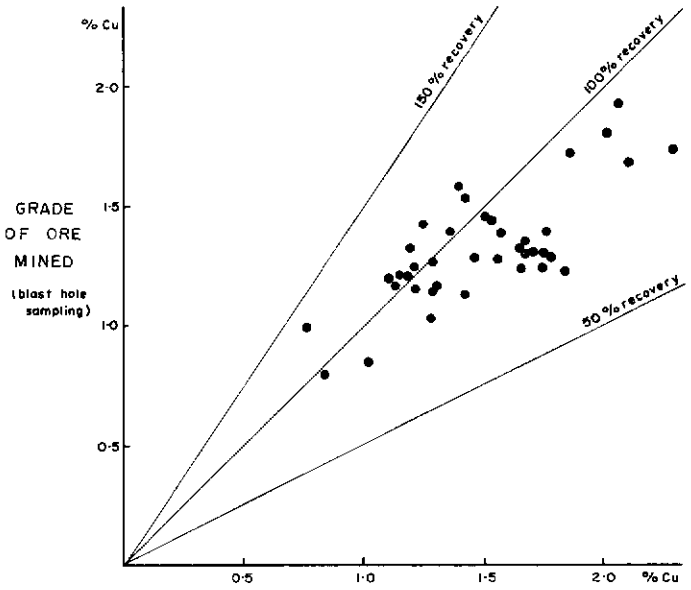
ORE RECONCILIATION

BENCHES MINED FROM OPEN PITS 1967 - 1971



DIAMOND DRILL INDICATED TONS ORE

Figure 30. Open pit ore reconciliation by benches showing tonnage in undiluted diamond drill reserves against tonnage mined. Large tonnage benches produce as much, or more ore than predicted while smaller benches show some of the largest losses. This indicates that the geological interpretation of ore outlines for smaller ore zones is suspect.



DIAMOND DRILL INDICATED ORE GRADE

Figure 31. Open pit ore reconciliation by benches showing undiluted diamond drill reserve grade against grade mined according to blasthole production. Grade is between 70% and 120% of that predicted, but this needs correction (see Table IV).

Ore Reserve Calculations

Since the start of open pitting, the grade of diamond drill core intersections has been calculated by summing the product of width and grade of assay and dividing this total by the length assayed. i.e.:

$$\text{Average grade of intersection} = \frac{\Sigma(w \times a)}{\Sigma(w)}$$

Σ = sum of
w = length of individual assay
a = grade of individual assay

This has produced results that are high compared with results achieved in the mill. Calculated grades suggest that about 25% of the copper theoretically contained in the ore is lost during mining and transportation of the ore to the mill. This loss is unlikely to result from mining or mechanical handling and three reasons are possible:

1. the grade assigned to the ore is mistakenly high.
2. the interpreted limits defined from diamond drilling are incorrect owing to the irregularity of the orebody contacts.
3. errors in calculations, assaying and sampling.

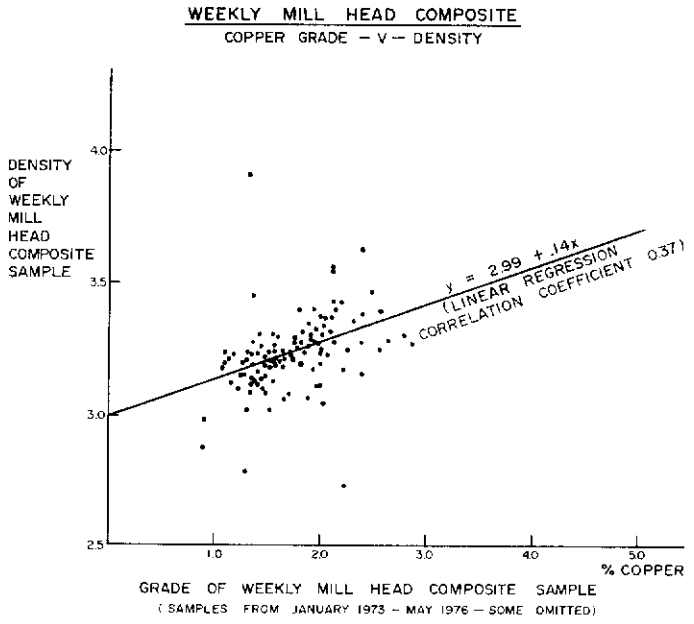


Figure 32. Copper grade plotted against density for weekly mill head composite sample (see also figure 18).

Grade of diamond drill core intersections will vary depending upon the parameters used to weight individual assays. McKinstrey (pp 46-57) shows how width, the assay frequency, and the assay value itself may be used for weighting, either alone or in combination to calculate average grade. The application of frequency weighting to the open pit deposits produced some contradictory results. Grade has consequently been calculated by using the standard method and applying a correction factor to get the diluted grade. This correction can be varied from one ore body to another to reflect dilution. Assays have not normally been cut in calculating the average grade of intersections.

Tonnages have been calculated by measuring the area of ore on section and projecting this area half way to adjacent sections. The resultant volume is divided by the appropriate specific volume. Areas are measured with a planimeter. Tonnages interpreted as ore from diamond drill sections are suspect as skarn ore bodies are notoriously irregular. Inward swings of waste may be mined along with the ore, but outward swings in the ore commonly go into the wall of the pit or stope and are unmineable. Irregular ore contacts lead to a loss of ore during mining.

Although efforts to avoid errors are made the following list indicates sources of discrepancy between calculated and actual grade:

1. Large diamond drill core losses leading to inaccurate grading.
2. Dykes not considered in calculating the grade of ore blocks.
3. Core stretch giving rise to tonnage estimates which are too large.
4. Inaccurate assaying of diamond drill core.
5. Wrong geological interpretations of ore outlines especially adjacent to dykes and faults.
6. Methods of calculating grade of diamond drill intersection inherently wrong.

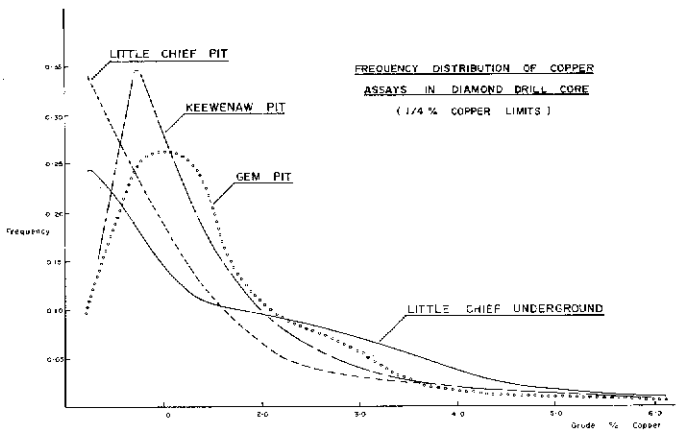


Figure 33. Frequency distributions of diamond drill core assays from selected open pits and the Little Chief underground mine. Curves are skewed positively. The Gem pit looks most attractive on this basis but mining difficulties offset this.

Ore Reserves

For open pits a cut off of 0.5% copper has been used to calculate ore reserves. The corresponding figure used for the Little Chief Underground Reserves has been 1.40% copper during the period 1970-1976. Earlier calculations were done at 0.5% Cu cut-off.

For the pits a dilution rate of 15% has been applied except in the case of Gem where dilution is estimated as 25%. The underground reserves listed below are calculated using estimated rates of dilution that agree with experience.

Low Grade Reserves Pit or Zone	Tonnes	% Cu	Stripping Ratio W/O
Best Chance	447,004	0.71	N/A
Black Cub North	156,035	0.82	N/A
Black Cub South	19,925	1.25	-
Cowley Park	758,971	0.93	2.0/1.0
Gem	625,046	1.01	3.35/1.0
Keewenaw	202,653	1.06	1.6/1.0
Kodiak Cub	57,152	1.18	N/A

Underground Ore Reserves
(Proven and Probable, Diluted, January 1980)

Little Chief	968,874 tonnes	1.55 % Cu
Middle Chief	847,311 tonnes	1.55 % Cu
Little Chief (North Fault)	273,970 tonnes	1.28 % Cu
Total	2,090,156 tonnes	1.51 % Cu

The Cowley Park and Keewenaw zones are the only potential open pits likely to prove profitable. Best Chance is too low grade and the remainder are too small to be likely producers even at higher copper and precious metal prices. All pits contain traces of gold and silver, and molybdenite is present at Cowley Park.

Geostatistics

Kriging has been used by A. Sinclair to calculate grade and tonnage of ore reserves in the Little Chief underground mine. Reserves were similar to those obtained using the standard methods. The technique is not applied routinely, but was of use in determining the reliability of grade estimates, and in planning diamond drill hole layouts.

The frequency distribution of diamond drill core assays from several deposits is shown in figure 18. Data is from drill holes inside reserve blocks. All curves show strong positive skew, a feature exhibited by most other deposits where grade is low relative to the other ore constituents.

Open Pit Mining

The relatively small pits operated by New Imperial Mines were mined at stripping ratios of about 3:1 (waste:ore) was too high to sustain economic operations under conditions during the late 1970, and early 1971. When the pits were shut down in June of 1971 even the lower stripping ratios and better grade in the bottom of the Keewenaw Pit were insufficient for profitable operation.

Pit designs (figs. 34 and 35) were drawn to the following specifications:

1. benches 7.5 m high (shovels must be able to scale nearly up to bench above).
2. 23 m between safety berms - berms 4.5 m to 6 m wide.
3. gradient of walls 400 - 500%.

4. roadways 10 to 12 m wide with gradient about 8%.
5. waste/stripping ratio as low as possible and generally not much greater than 3:1 (waste:ore).
6. overburden is stripped to form a bare rock surface 6 m wide around the perimeter of the pits.
7. slope of the overburden variable but about 30°.

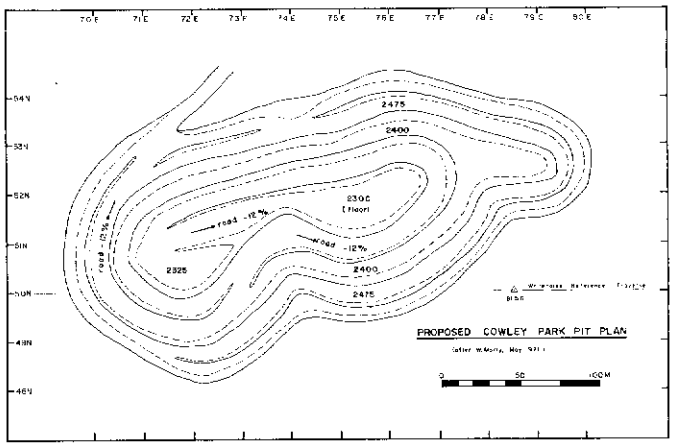


Figure 34. Proposed 1971 Cowley Park pit by W. Marty. Road gradients are not normally as steep as 12%.

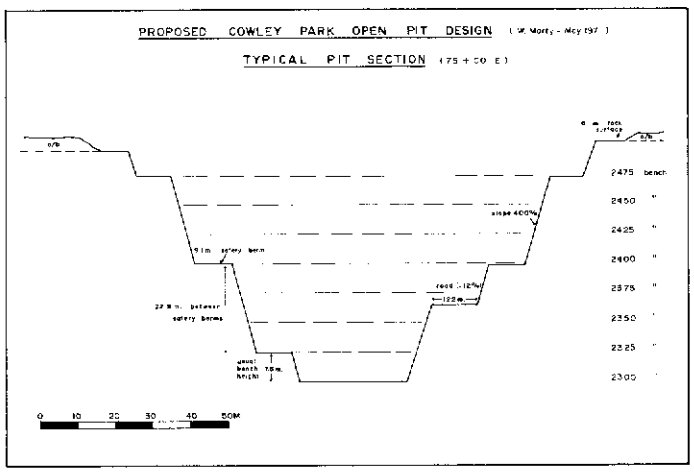


Figure 35. Typical open pit cross-section (proposed 1971 Cowley Park pit by W. Marty) showing width of safety berms and roads, and slope of walls.

These specifications were subject to modification depending on local circumstances. The average slope of the walls of the Little Chief Pit is 52° on the hangingwall side measuring from the crest at surface (2730 elev) to the toe on the bottom of the pit (2400 elev.) The rock is a competent garnet skarn and only minor sloughing occurred up to the time when caving in the underground mine broke into the floor of the pit at 2400 in January 1974.

Blasting in the pit was done using 15.24 cm. diameter blastholes in a triangular pattern, (fig. 36) the distance between adjacent holes being about 5 m. The walls of the pits were presheared, with holes spaced at 1.5 m intervals.

The commonest mining difficulties were caused by undercutting faults and shear zones, and by the presence of groundwater. At the Black Cub South Pit ground conditions were poor and this and abundant groundwater led to the premature closure of the pit by the Mining Inspector before the small bottom 2525 level had been mined. There were a few places on the footwall side of the Little Chief Pit where wedge failure occurred in serpentine skarn. Elsewhere, as at War Eagle, Arctic Chief and Keewenaw ground conditions were better, and these pits are still accessible. Black Cub South, Keewenaw and War Eagle pits are now partly flooded.

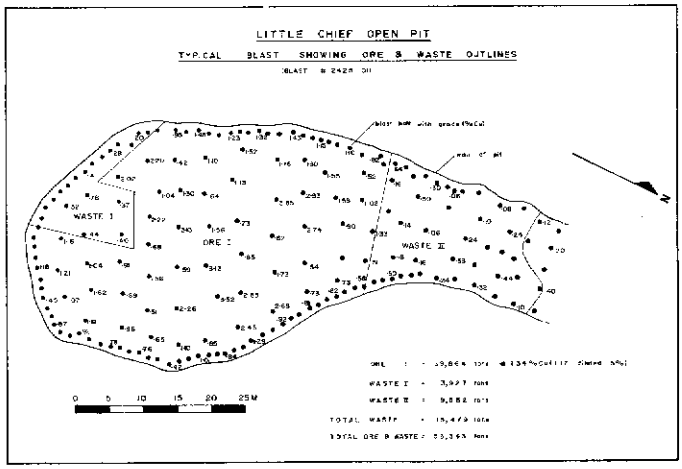


Figure 36. Typical open pit blasthole layout (the Little Chief 2425 - 01 blast) showing the spacing of drill holes, and preshear holes around the pit wall. Areas of waste and ore planimetered converted to waste and ore tonnage blasted are also shown.

Blasthole sampling in the pits was carried out as soon as practical, after the blasthole was complete. The rock chips from the hole were mixed thoroughly with a shovel and coned and quartered to produce 3 kg. of chippings for assay. The reconciliation (Table IV) shows that blast hole grade agrees more closely with diamond drill reserve grade than mill heads. Some very high blasthole assays were recorded from holes which were wet: this probably accounts for the blasthole grade being unexpectedly higher than the mill head grade. (The ring drill samples taken underground at the Little Chief, which were also wet, indicated higher grades than reported at the mill head). As pit blasts contained both ore and waste, an ore sorter was responsible for on the spot direction of mucking, which was generally done by visual inspection. Ore boundaries were determined on the blasthole plans (fig. 36) using a cut-off of 0.5% copper, and the area of the ore block measured with a planimeter. Volume and tonnage could be calculated by knowing the average depth of the holes, and the specific volume (density) for the ore.

UNDERGROUND MINING

Underground Mining at Little Chief has been described by Bent (ed.), and Janssens and Percival. Mining techniques evolved during the years 1972 to 1980 are interesting from a geological point of view as they illustrate adaptation to poor ground conditions in magnetite serpentine skarns. The ore reconciliation in Appendix IV should be studied with reference to figure 36, a vertical longitudinal projection of stopes and pillars, and used to judge the effectiveness and suitability of the mining methods. Mining costs are limited by the ability of the ore grade to support them, and as grades at Little Chief were considered low, inexpensive mining methods were adopted. Sub-level open stoping with 12 m stopes and 12 m pillars was used to mine between the 2050 level and the floor of the Little Chief open pit (fig. 38). Because the ore was jointed (fig. 39) pillars caved before extraction of broken ore in the stopes was complete, and could not be blasted effectively. The reconciliation (Appendix IV) reflects these difficulties in the poor recovery of ore above 2050 level. The ore from 1750 level to 2050 level was also mined by open stoping similar to that used above 2050 level excepting that stopes were 58 m long and pillars 27 m long. This gave good recovery of ore from 7 stope, but the sill left over the stope failed before it could be blasted. 7 pillar was "choke blasted" against the broken ore and waste which filled 7 stope, and was virtually mined by block caving. Strong dilution resulted. (see Appendix IV). Similar problems arose in 8 stope and in stopes, pillars and sills between 1450 and 1730 levels. The high recovery of ore from 1730 level to 2050 level reflects the mucking of ore "lost" above 2050 level.

In 1979 vertical crater retreat mining described by Janssens and Percival was started in 14 and 16 stopes with the hope this would improve the fragmentation of the ore and control dilution. To the present, (Oct. 1980), this technique has been successful in doing both (see appendix IV) and will be applied to mining the Middle Chief Orebodies.

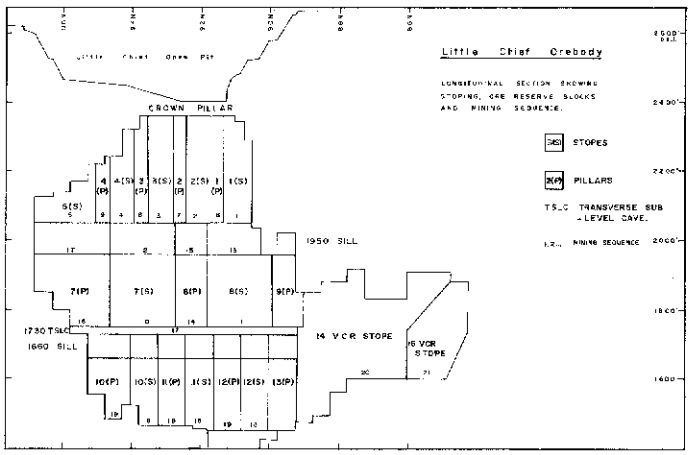


Figure 37. Vertical longitudinal projection of the Little Chief underground mine showing stopes and pillars with mining sequence. This section should be viewed in conjuction with the Little Chief underground ore recondiliation (Appendix IV) to judge the effectiveness of the mining methods used.

Transverse sub level cave mining with cross-cuts at 12 m intervals was established on one level, 1730, to recover large amounts of ore not accessible to open stoping on the 1750 level above.

The level recovered much of what had been left behind, and a similar sub-level caving operation is planned for 1400 level.

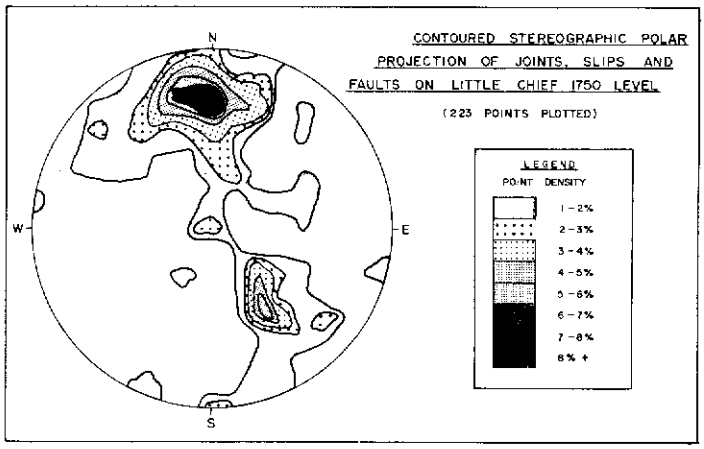


Figure 38. Contoured stereographic polar projection of joints, slips and faults Little Chief underground 1750 level. Two sets of joints dipping approximately 70^{0} south and 50^{0} north are apparent. The intersection of these two joint sets on the hangingwall side of the stopes induced caving out into the hangingwall rather than slabbing of hangingwall rocks into the stope. The shear zone along the footwall side of the Little Chief underground orebody is not represented in this diagram.

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APPENDIX I
New Imperial Mines/Whitehorse Copper Mines
Departmental Costs

	1967		1968		1969		1970	
	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $
MINING								
Staff								
Crew								
Explosives								
Drilling Supplies								
LHD Equipment								
Tramming								
Miscellaneous Supp.								
Diamond Drilling								
Total	1.54	635,082	2.29	1,527,627	2.45	1,789,396	4.62	3;569,028
MILLING								
Staff								
Crew								
Crushing								
Grinding								
Flot. & Tail. Disp.								
Dewatering								
Assaying								
Svce. & Repair Supp.								
Total	1.81	742,532	1.66	1,107,873	1.87	1,377,586	2.05	1,588,020
PLANT								
Staff								
Crew								
Power								
Heating								
Supplies								
Total	.29	117,761	.37	247,959	.47	347,400	.60	466,306
ADMINISTRATION								
Staff								
Office Exp. & Supp.								
Warehouse Supply								
Personnel								
Taxes								
Housing								
Miscellaneous								
Total	.65	266,594	.30	195,599	.29	207,637	.44	339,919
HEAD OFFICE								
Staff								
Supplies & Services								
Total								
TOTAL OPERATIONS	5.27	1,761,969	4.64	3,079,058	5.09	3,722,019	7.71	5,963,273
OTHER COSTS								
Interest Expense	-	-	-	-	-	-	-	-
Deprec. & Amortiz.	.67	275,273	.94	624,763	1.00	732,000	1.25	960,000
Exploration	.27	113,670	.21	140,250	.12	87,082	.31	242,143
Development	.09	36,409	.22	144,316	.25	183,800	-	-
Total								
TOTAL PRODUCTION COSTS	6.55	2,187,321	6.01	3,988,387	6.47	4,724,891	9.27	7,165,416
Marketing	.73	299,204	.70	467,394	.79	577,316	.74	572,963
Smelting	1.25	511,283	1.27	839,471	.82	593,645	.78	604,568

APPENDIX I
New Imperial Mines/Whitehorse Copper Mines
Departmental Costs

1971			1972	1973		1974		1975		1976	
$/Tonne Milled	Costs $			$/Tonne Milled	Costs $	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $
		Re-opened December 1972		.46	293,074	.59	340,611	.57	376,614	.55	402,624
				2.05	1,304,626	2.55	1,446,444	2.69	1,800,729	2.42	1,758,632
				.54	345,672	1.07	608,003	.68	461,233	.36	265,905
				.31	192,791	.30	190,034	.19	124,358	.29	208,881
				.67	430,110	1.11	630,321	.74	497,461	.93	673,920
				.19	120,223	.19	109,195	.24	165,677	.22	160,393
				.68	433,876	.94	535,295	1.11	745,881	.83	597,253
				.16	108,378	.13	75,827	.14	96,150	.14	107,550
5.54	1,699,298			5.07	3,228,750	6.91	3,935,730	6.37	4,268,103	5.74	4,175,158
				.24	148,413	.32	183,348	.30	198,334	.29	206,202
				.52	328,241	.57	325,621	.89	587,471	.70	516,045
				.11	70,346	.26	149,522	.10	63,946	.10	74,940
				.20	127,200	.29	165,597	.30	200,272	.24	179,471
				.22	142,336	.34	195,481	.19	126,660	.22	160,117
				.10	64,032	.16	95,594	.08	55,107	.08	55,451
				.01	9,755	.04	19,322	.02	17,829	.04	29,958
				.11	66,903	.16	90,708	.09	57,129	.10	69,299
2.29	703,896			1.51	957,226	2.16	1,225,193	1.95	1,306,748	1.78	1,291,483
		Pits shut down June 1971		.08	54,928	.08	44,530	.14	94,674	.15	113,738
				1.15	728,086	1.33	760,384	1.39	929,894	1.20	876,424
				.55	351,452	.65	367,719	.76	505,929	1.04	756,516
				.33	208,608	.29	154,070	.28	181,641	.16	122,785
				.24	151,096	.47	272,393	.46	310,923	.36	251,561
.80	246,710			2.35	1,404,170	2.82	1,599,096	3.03	2,023,061	2.92	2,121,024
				.24	150,294	.34	192,352	.35	227,860	.34	247,437
				.10	65,427	.12	71,144	.12	82,047	.20	134,731
				.01	4,705	-	1,232	-	-	-	-
				.03	19,124	.05	28,759	.03	25,763	.07	44,666
				.08	46,641	.16	96,918	.16	112,958	.18	127,877
				.07	40,164	.09	51,369	.08	52,735	.07	49,268
				.002	1,409	.01	4,336	-	-	-	-
.69	212,658			.53	327,764	.78	446,110	.75	501,363	.84	603,979
				.36	232,014	.46	261,706	.47	315,390	.46	335,244
				.14	88,985	.21	121,384	.18	117,785	.19	135,560
.55	169,399			.51	320,999	.67	383,090	.65	433,175	.65	470,804
8.98	3,031,961			9.96	6,328,909	13.35	7,589,219	12.75	8,532,450	11.93	8,662,448
.34	106,039	No Production		.13	85,538	.21	118,306	.14	98,891	.23	164,556
3.22	988,632			2.80	1,778,219	2.27	1,292,407	2.01	1,273,402	1.88	1,369,154
.31	94,827			.05	34,728	.03	18,472	-	294	-	-
-	-			-	-	-	-	-	-	-	-
				2.99	1,898,485	2.09	1,192,573	1.86	1,174,805	1.65	1,204,598
13.78	4,221,459			12.95	8,227,394	15.44	8,781,792	14.62	9,707,255	13.58	9,867,046
.61	185,000										
.68	210,000										

APPENDIX I
New Imperial Mines/Whitehorse Copper Mines
Departmental Costs

	1977		1978		1979	
	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $	$/Tonne Milled	Costs $
MINING						
Miscellaneous	2.30	1,880,985	2.46	1,927,073	2.56	2,119,316
U/G Supplementary	.32	264,033	.47	370,997	.68	570,978
Drilling, Dev. etc.	1.21	994,703	1.36	1,064,473	1.06	874,698
Mucking	.91	745,202	.99	776,904	1.56	1,298,303
Crushers	.21	170,909	.25	202,177	.36	303,101
U/G Water & Dev.	.14	114,200	.21	161,297	.21	175,321
Hoisting & Tramming	.50	402,274	.43	334,263	.51	424,097
Ground Support	.70	579,479	.98	765,125	.91	755,294
Explosives	.69	567,685	.58	455,227	.57	474,871
Geology & Eng.	.47	389,813	.26	210,368	.34	278,252
Total	7.47	6,109,283	8.00	6,267,904	8.77	7,274,231
MILLING						
General	1.20	980,474	1.26	984,604	1.42	1,174,347
Crushing	.35	288,810	.44	347,928	.48	399,135
Grinding	.35	290,882	.37	291,628	.53	439,164
Flotation	.34	275,759	.45	355,508	.45	374,679
Dewatering	.21	173,657	.22	169,040	.20	165,503
Tailings Disposal	.19	148,806	.19	150,248	.14	121,499
Assaying	.12	100,726	.13	106,003	.08	63,964
Mill Services	.19	155,892	.19	142,960	.18	150,488
Gold Recovery	.02	14,569	.01	11,167	.02	19,137
Total	2.98	2,429,575	3.26	2,559,086	3.50	2,907,916
PLANT						
General	.29	236,414	.15	123,063	.24	197,387
Mobile & Surf. Equip.	.22	179,377	.19	145,003	.25	212,414
Builings, Roads	.10	82,303	.09	64,992	.10	81,265
Services	.76	618,625	.67	523,708	.72	592,495
Total	1.37	1,116,719	1.10	856,766	1.31	1,083,561
ADMINISTRATION						
Staff	.37	303,536	.41	319,675	.47	396,521
Office Exp. & Supp.	.16	134,168	.20	159,083	.18	148,994
Personnel	.02	15,593	.01	11,035	.08	60,993
Taxes	.16	136,708	.15	122,768	.15	125,593
Housing	.07	58,111	.10	77,025	.11	88,307
Rent	.02	19,928	.03	25,002	.01	6,646
Total	.82	668,044	.90	714,588	1.00	827,054
HEAD OFFICE						
Staff	.44	361,598	.40	313,740		
Supplies & Services	.19	149,529	.15	120,284		
Total	.63	511,127	.55	434,024	.30	244,127
TOTAL OPERATING	13.26	10,834,748	13.83	10,832,368	14.89	12,336,889
OTHER COSTS						
Interest Expense	.10	78,287	.14	110,727		
Deprec. & Amortiz.	1.39	1,139,908	1.31	1,023,364	1.38	1,142,575
WCM Dep. Charge			1.13	892,199	1.43	1,190,748
Total	1.29	1,061,621	2.30	1,804,836	2.81	2,333,323
TOTAL PRODUCTION COSTS	14.55	11,896,369	16.14	12,637,204	17.70	14,670,212

APPENDIX II

NEW IMPERIAL MINES - WHITEHORSE COPPER MINES

PRODUCTION STATISTICS 1967 - 1979

	1967	1968	1969	1970	1971	1972	1973
Tonnes Ore Milled	410,997	664,146	730,755	773,341	306,410	9,713	635,079
Grade Ore Milled (% Cu)	1.17	1.03	1.09	1.04	1.02	1.92	1.83
Grade Tailings (% Cu)	.37	.20	.15	.10	.25	.32	.30
Copper Produced (kg)	3,317,671	5,515,521	6,880,754	7,295,911	2,375,344	157,118	9,780,842
Mill Recovery (%)	70.6	80.6	86.5	91.00	75.83	84.27	84.33
Gold Produced (gm)	272,213	516,225	452,867	173,562	113,403	6,936 (est)	431,965
Silver Produced (gm)	2,792,614	4,609,197	6,110,874	6,581,936	1,656,977	125,227 (est)	7,795,377
Tonnes of Concentrate	11,369	17,532	22,481	23,820	8,426	475	24,934
Grade of Concentrate (% Cu)	29.42	31.28	30.61	30.63	28.20	32.92	39.23
Waste from pits (tonnes)	1,918,232	2,259,486	1,114,041	1,613,562	1,433,210	-	-
Underground waste (tonnes)	-	-	N/A	N/A	N/A	N/A	60,688
Development (metres)	-	-	N/A	N/A	N/A	N/A	3,234

	1974	1975	1976	1977	1978	1979	TOTAL
Tonnes Ore Milled	568,389	669,559	726,507	871,791	782,985	829,222	7,924,894
Grade Ore Milled (% Cu)	1.84	1.52	1.69	1.65	1.40	1.12	1.38
Grade Tailings (% Cu)	.19	.16	.18	.20	.20	.16	0.20
Copper Produced (kg)	9,439,606	9,100,044	11,051,444	11,947,933	9,490,536	7,935,161	94,287,986
Mill Recovery (%)	90.40	89.36	89.74	88.35	86.34	85.66	86.3
Gold Produced (gm)	551,496	571,060	576,970	748,288	604,745	493,550	5,521,560
Silver Produced (gm)	6,516,556	6,761,808	7,500,889	7,765,673	6,088,977	5,255,621	69,560,646
Tonnes of Concentrate	25,263	25,152	26,937	28,259	21,272	17,817	253,917
Grade of Concentrate (% Cu)	36.89	36.16	41.05	42.28	44.62	44.54	37.11
Tonnes waste from pits	-	-	-	-	-	-	9,419,722*
Underground waste (tonnes)	70,989	81,620	67,966	30,675	50,332	12,272	374,543
Development (metres)	2,854	2,655	3,005	2,555	2,583	1,742	18627.5

* Includes waste mined in 1966

N/A Not available

APPENDIX III

WHITEHORSE COPPER MINES LTD.

Detailed Statement of Mine Revenue and Expenses 1967 - 1976 (in 000's $C)

	1967	1968	1969	1970	1971	1972	1973**	1974**	1975**	1976**	Total
Revenue											
Concentrate sales	3,900	6,976	11,755	9,905	2,669	--	17,587	19,455	14,936	18,438	105,623
Less: treatment cost	679	1,296	1,659	1,576	428	--	3,193	3,511	4,392	5,447	22,185
Net Smelter Return	3,220	5,679	10,096	8,328	2,241	--	14,393	15,944	10,543	12,990	83,438
Interest Income	--	--	--	--	--	--	--	113	98	155	368
Other	--	--	--	--	--	--	--				
Net Income	3,220	5,679	10,096	8,329	2,241	--	14,393	16,057	10,642	13,146	83,806
Expenses											
Conc. Production Cost	1,212	2,730	3,752	5,588	3,039	--	5,728	6,711	7,505	7,727	43,996
Administration	228	514	547	506	343	37	665	833	899	1,080	5,656
Yukon Royalty Provision	20	30	175	--	--	--	--	--	--	--	225
Exploration	46	29	--	--	--	--	34	27	--	--	137
Sub Total	1,507	3,304	4,474	6,094	3,382	37	6,428	7,572	8,404	8,808	50,015
Operating Income (Loss)	1,713	2,375	5,621	2,234	(1,141)	(37)	7,965	8,484	2,237	4,338	33,791
Interest on Debt	106	133	426	231	27	--	66	--	--	--	993
Earnings (Loss)	1,606	2,241	5,195	2,002	(1,169)	(37)	7,898	8,484	2,237	4,338	32,797
Other Expenses											
Depreciation	274	624	806	1,232	849	--	1,435	323	510	584	6,642
Amortization	--	--	480	725	299	--		728	763	784	3,780
Interest on Debt	162	354	--	--	116	--	--	--	--	--	633
Other	--	4	13	6	273	74	--	--	--	--	371
Total Other	437	982	1,299	1,963	1,538*	74	1,435	1,052	1,273	1,369	11,427
NET EARNINGS (Loss)	1,168	1,258	3,895	38	(2,707)	(112)	6,463	7,432	964	2,969	21,369

Notes:
i) Open pit ore was milled from May 1st 1967 to June 30th 1971.
ii) Underground Little Chief ore was milled from December 18th 1972.
* Excludes extraordinary write off.
** Little Chief joint mining venture.

APPENDIX IV

LITTLE CHIEF UNDERGROUND ORE RECONCILIATION DECEMBER 1972 to AUGUST 1980

Ore Block	Mining * Method	Undiluted Ore Reserves Tonnes	% Copper	Recovered Ore Tonnes	% Copper
#1 Stope and Pillar	SLOS	105,219	1.90	88,834	1.38
#2 Stope and Pillar	SLOS	226,317	1.81	251,603	1.69
#3 Stope and Pillar	SLOS	161,131	2.67	141,039	1.75
#4 Stope and Pillar	SLOS	88,474	2.52	89,898	2.10
#5 Stope	SLOS	75,152	2.24	86,927	1.76
2050 H.W. Wedge	LSLC	130,018	2.39	45,128	1.82
2385 Sub Level	C	18,144	2.00	Not Identified	
2360 Crown Pillar	C	45,359	2.13	Not Identified	
Total Above 2050 Level		849,815	2.21	703,448	1.73
#7 Pillar + Sill	C	386,788	2.90	514,948	1.84
#7 Stope	SLOS	334,287	2.57	322,991	2.27
#8 Pillar	D	142,784	2.12	5,782	2.12
#8 Stope	SLOS	275,494	1.82	300,276	1.58
#9 Pillar + Sill	D	84,564	2.17	5,828	1.07
#7 Stope Sill	C	199,741	2.19	558,183	1.50
#8 Pillar Sill	C	49,053	2.19	102,899	1.26
#8 Stope Sill	C	58,510	2.11	178,375	.78
1730 Level	T.S.L.C.	269,100	2.70	576,942	1.71
1730 - 2050		1,800,321	2.42	2,566,225	1.66
Above 1730		2,650,136	2.35	3,269,673	1.67
#10 Pillar & Sill	SLOS	106,286	3.19	206,817	1.50
#10 Stope & Sill	SLOS	129,198	2.71	127,890	1.66
#11 Pillar & Sill	SLOS	147,262	2.64	279,332	1.40
#11 Stope & Sill	SLOS	194,203	2.38	482,810	1.33
#12 Pillar & Sill	SLOS	171,661	1.93	189,251	1.09
#12 Stope & Sill	SLOS	165,398	1.65	200,619	1.14
1400 Level	TSLC	26,054	1.70	88,149	1.10
#14 V.C.R. Stope					
1450 to 1870		199,652	1.98	105,432	1.44
1530 to 1870		237,007	2.07	147,704	1.77
1610 to 1870		71,886	2.26	57,418	1.95
1680 to 1870		85,678	1.98	96,964	1.75
SUB TOTAL 14 Stope		594,224	2.05	407,518	1.71
#16 V.C.R. Stope		196,097	2.46	28,083	1.74
TOTAL 1400 - 1730		1,730,384	2.25	2,010,471	1.41
TOTAL LITTLE CHIEF		4,380,520	2.31	5,280,144	1.57

* SLOS Sublevel open stoping
LSLC Longitudinal sublevel caving
TSLC Transverse sublevel caving
C Block caving
D Combined TSLC and C.
VCR Vertical crater retreat

APPENDIX III

FEASIBILITY STUDY FOR 2000 TPD COPPER ORE MINE CONCENTRATE PLANTAND RELATED SERVICES

FEASIBILITY STUDY

FOR

2000 TPD COPPER ORE MINE
CONCENTRATOR PLANT AND RELATED SERVICES

NEW IMPERIAL MINES LTD.
WHITEHORSE, Y.T.

BY

WRIGHT ENGINEERS LIMITED

J.A.C. ROSS AND ASSOCIATES, LTD.
Mining Consultants

DR. A.C. SKERL, P.ENG.
Consulting Mining Geologist

PROJECT NO. 402

NOVEMBER, 1965.

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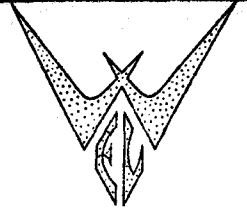
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Authors

The feasibility study and ensuing report has been written by:-

Dr. A.C. Skerl, P.Eng. as author of Section 1B.
J.A.C. Ross, P.Eng) of J.A.C. Ross and Associates Limited,
T. Swanson, P. Eng) Section II Mining.

Wright Engineers Limited personnel were responsible for the remainder of the report and the overall co-ordination.

BIBLIOGRAPHY

Metallurgical Test Reports by Britton Research Laboratories, Vancouver
by Sumitomo Metal Mining Company, Tokyo
by Dept. of Mines & Technical Surveys, Ottawa.
as listed in Section III.

Wright Engineers Limited Reports

1. Feasibility Study for 1000 TPD Iron-Copper Ore Mining and Concentrating Plant and Related Services with Expansion to 2000 tons per day - June 16th, 1964.

2. Memorandum on Preliminary Estimates of Cash Flow on Yukon Orebodies when Mined at Rate 2000 TPD - March 31st, 1965.

TERMS OF REFERENCE

The terms of reference for this study were developed through a series of meetings and correspondence between New Imperial Mines and Wright Engineers Limited starting in January, 1965 and ending in August, 1965.

In a letter dated January 22nd, 1965 from Mr. A. Pitt, President of New Imperial Mines, Wright Engineers were asked to update the previous report of June 16th, 1964.

On request Wright Engineers submitted on March 31st, 1965, a report entitled "Memorandum on Preliminary Estimates of Cash Flow on Yukon Orebodies when mined at the rate of 2000 TPD."

On May 21st, 1965 Wright Engineers Limited were instructed to commence gathering material for a new feasibility report and the terms of reference were discussed on June 25th, 1965, in a meeting in Vancouver between L.F. Wright and R. Kenway and a tentative target date was set for the report of September 30th, 1965. At this meeting the format of the report was outlined and the arrangements for splitting the responsibility for the report between Dr. A.C. Skerl, J.A.C. Ross and Associates and Wright Engineers Limited were outlined. At a further meeting in Vancouver on July 13th with J.A.C. Ross, T. Swanson, Dr. Gillanders, Mr. D. Maxey, Mr. R. Kenway and L.F. Wright the details of these arrangements were confirmed.

After preliminary estimates by J.A.C. Ross and Associates and Wright Engineers Limited, it was established that 2000 tons per day was a practical mining tonnage rate and would produce the best economic return. The tonnage rate was confirmed in a letter from Mr. R. Kenway dated August 5th, 1965.

SECTION 0
SUMMARY AND CONCLUSIONS

Summary and Conclusions

The total capital investment required to construct the surface plant and related facilities and to provide the necessary equipment to mine and mill at the rate of 2,000 tons per day from the Little Chief ore body is estimated to be about $7,250,000. An additional $2,160,000 will be required over a six year period to bring the remaining pits into operation and to expand the concentrator for molybdenite recovery from the Cowley Park ore. It is assumed that this expenditure will be paid from operating revenue.

The following table summarizes the value, operating cost, and revenue per ton of ore based on a copper price of 38.5¢ Canadian per pound.

	Value $/Ton	Operating Cost $/Ton	Revenue $/Ton
Initial (Little Chief)	8.09	4.05	4.04
Maximum	9.37	4.13	5.34
Minimum	6.20	3.64	2.53
Weighted average	7.54	3.88	3.66

The financial summary of the operation at a copper price of 38.5¢ Can./lb. is therefore estimated as follows:

Total value of 5,022,730 ton open pit ore reserve	$ 37,871,400	
Less total operating cost		$ 19,488,200
Less initial capital investment		7,253,900
Less additional expenditure		2,156,100
Plus return of working capital	721,800	
Plus return of housing loan	36,000	
	$ 38,629,200	$ 28,898,200

. . Total profit before financing charges, head office expenses, interest payments and taxation = $ 9,731,000

A financial analysis has been prepared by McDonald Currie and Co., based on the estimates in this report and their estimate of $212,800 for interest during the construction period is included in the capital cost summary.

Summary and Conclusions - Cont'd.

The concentrator is designed to treat 2,000 tons per day of Little Chief ore, but the capacity for treatment of other ores will range from 1,900 to 2,200 tons per day because of differences in grinding characteristics of the other ores. On testwork to date a detailed analysis of the exact tonnage rates for each ore body and mixtures of ores from more than one pit is not justified.

A preliminary analysis to determine the optimum tonnage rate showed that the total profit from a 3,000 tons per day operation is about the same as for 2,000 tons per day. The increase in capital investment of about 2 to 2-1/2 million dollars cannot therefore be justified unless taxation considerations permit a faster return of this capital. The previous feasibility report in June 1964 showed that total profit from a 2,000 tons per day operation was higher than a 1,000 tons per day operation, and since then there has been a substantial increase in proven ore reserves.

The mining schedule shows the sequence in which the ore bodies must be mined to return the greatest profit in the shortest time. The mining estimates are based on the company carrying out its own mining operations.

Winter conditions are not expected to affect output, but provision has been made for insulating the water pipeline from the Yukon River, screening fines from the ore before stockpiling, and drying the concentrate to 3 to 4% moisture during winter months to prevent freezing.

It is assumed that housing for company personnel will be available in Whitehorse and that company assistance will be confined to a loan for initial equity payment for married personnel and relocation expenses for single personnel.



Scope of the Report

The summarized capital costs cover all the direct investment needed to develop the mine and construct the various sections of the plant as described in the report.

The report does not include the cost of financing, interest charges, taxation and head office expenses. A financial analysis by McDonald Currie and Co., has been prepared to cover these items.

The construction and installation costs are based on an orderly progression of work by small contractors at expected 1966 labour rates and at current costs for equipment and materials. No allowance has been made for further labour rate escalation or price increases. Should the construction rate be accelerated then an increase in labour costs by up to 40% could be expected.

All equipment costs are based on new equipment, F.O.B. Vancouver. The freight cost to site is included at $80/ton.

The Yukon Territory has no Provincial Tax, but Federal Tax at a rate of 11% is included.

All operating costs are based on wage rates as we believe they will be in 1966 without taking further escalation into account. The operating costs shown in the individual sections are calculated on basic labour rates without including cost of fringe benefits. Fringe benefits are summarized in the administration section and have not been pro-rated to other sections. Operating costs are estimated as straight costs with no allowance for depreciation or replacement of equipment.

A contingency of 10% has been included in all mining costs to allow for remote location and winter conditions. An allowance of $0.10 per ton has been included for molybdenite recovery when treating the Cowley Park ore.

It has been assumed that the concentrates are dried to an average of 6% moisture for freight rate calculations.

The economic analysis has been calculated based on four copper prices: 28.5 and 30.0 U.S. ¢/lb. as requested by the Sumitomo Metal Mining Co. Ltd., and 38.5 and 45.0 ¢ Can./lb. as requested by New Imperial Mines Ltd.

Short tons (2,000 lbs.) Imperial Gallons and Canadian Dollars are used throughout the report except where otherwise stated.

Ore Reserves

	PROVED			AS MINED		
Ore Body	Tons	Grade %Cu	Ass.Min. $/Ton *	Tons	Grade %Cu	Ass.Min. $/Ton *
Little Chief	1,780,000	1.40	1.07	1,518,610	1.21	1.07
Arctic Chief	400,000	1.32	1.67	420,750	1.27	1.67
War Eagle)	1,080,000	1.33	0.73	926,530	1.24	0.73
War Eagle S)				215,330	0.96	0.73
Cowley Park	780,000	0.84	1.47	1,265,050	0.86	1.47
Keewenaw	380,000	1.10	0.96	485,180	1.05	0.96
Best Chance	200,000	1.00	0.26	191,280	0.98	0.26
	4,620,000	1.23	1.07	5,022,730	1.10	1.13

* Recoverable gold, silver and molybdenite.
Probable reserves are listed under Section IB.

Mining Schedule

Year	Ore Body	Tons	
1.	Little Chief	650,000	
			650,000
2.	Little Chief	599,010	
	Arctic Chief	120,990	
			720,000
3.	Little Chief	269,600	
	Arctic Chief	299,760	
	War Eagle	150,640	
			720,000
4.	War Eagle	613,650	
	War Eagle South	106,350	
			720,000
5.	War Eagle	162,240	
	War Eagle South	108,980	
	Cowley Park	448,780	
			720,000
6.	Cowley Park	570,000	
	Keewenaw	150,000	
			720,000
7.	Cowley Park	246,270	
	Keewenaw	335,180	
	Best Chance	138,550	
			720,000
8.	Best Chance	52,730	
			52,730
	Total		5,022,730 Tons

Capital Cost Summary

Section		Equipment	Construction & Installation	Totals
II	Mining	$ 1,283,240	$ -	$ 1,283,240
	Preproduction Mining		464,480	464,480
IV	Access Road		54,000	54,000
V	Ancillary Buildings	38,500	188,900	227,400
VI	Power	50,390	35,300	85,690
VII	Crushing Plant & Coarse Ore Storage	382,100	470,600	852,700
	Concentrator & Fine Ore Storage	583,100	1,118,900	1,702,000
VIII	Water Supply & Fire Protection	93,200	198,300	291,500
	Tailings Disposal	9,190	77,600	86,790
IX	Administration & General Costs	432,100	-	432,100
	Sub Total -	$ 2,871,820	$ 2,608,080	$ 5,479,900
	Contingencies @ 10% on $4,196,660			419,700
	Engineering, Construction Supervision and Company Administration @ 10% on $4,196,660			419,700
	Interest from beginning of construction to first revenue payment			212,800
	Working Capital for first 100 days operation			721,800
	Total Capital Investment Required -			$ 7,253,900

SUMMARY OF OPERATING COSTS

Yr.	Pit	Mining $/Ton	Milling $/Ton	Freight $/Ton	Ad. & Serv. $/Ton	Explor. $/Ton	Total Operating Cost $/Ton
1	Little Chief	1.516	1.419	0.352	0.616	0.150	4.053
2	Little Chief	1.464	1.281	0.305	0.556	"	3.756
	Arctic Chief	"	"	0.369	"	"	3.820
3	Little Chief	1.373	"	0.403	"	"	3.763
	Arctic Chief	"	"	0.369	"	"	3.729
	War Eagle	"	"	0.359	"	"	3.719
4	War Eagle	1.637	"	0.396	"	"	4.020
	War Eagle S	"	"	0.305	"	"	3.929
5	War Eagle	1.715	"	0.433	"	"	4.135
	War Eagle S	"	"	0.305	"	"	4.007
	Cowley Park	"	1.381	0.270	"	"	4.072
6	Cowley Park	1.521	1.381	0.270	"	"	3.878
	Keewenaw	"	1.281	0.271	"	"	3.779
7	Cowley Park	1.378	1.381	0.270	"	"	3.735
	Keewenaw	"	1.281	0.271	"	"	3.636
	Best Chance	"	"	0.298	"	"	3.663
8	Best Chance	1.378	"	0.298	"	"	3.663

Copper Valuation

Let copper price be X U.S. ¢/lb.

Assume concentrate grade of 35% Cu.

Then value per lb. Cu contained in concentrates

$$= \frac{2204.6 \times \frac{(35-1)}{100} \times \left(\frac{X}{100} - 0.005\right)}{\frac{2204.6 \times 35}{100}}$$

$$= \frac{34}{35}\left(\frac{X}{100} - 0.005\right)$$

U.S. $ = 0.009714X - 0.004857
U.S. ¢ = 0.9714X - 0.4857

Smelter charge and sea freight

= U.S. $22.50/Metric Ton Dry Concs.

∴ Cost/Lb. Cu Metal $\frac{22.50 \times 100}{35 \quad 2204.6}$ = U.S. $ 0.0292 = U.S. ¢ 2.92

∴ Net value U.S. ¢/Lb = 0.9714X - 0.4857 - 2.92
= 0.9714X - 3.406

Copper Price U.S. ¢/Lb.	28.50	30.0	35.81	41.86
Copper in Conc. U.S. ¢/Lb.	24.28	25.74	31.38	37.26
Copper in Conc. Can. ¢/Lb. (x1.075)	26.10	27.67	33.74	40.05

Au, Ag, & MoS_2 Valuation

Deposit	Arctic Chief		Best Chance		Little Chief		War Eagle		Keewenaw		Cowley Park	
Brittons Sample assay % Cu.	0.88		0.85		1.20		1.25		1.31		0.92	
Pit Design assay % Cu.	1.27		0.98		1.21		1.24		1.05		0.86	
	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.	Au.	Ag.
Brittons sample oz./ton	0.03	0.22	0.003	0.17	0.023	0.30	0.01	0.46	0.03	0.27	0.004	0.15
Pit Design reduced value oz./ton	0.043	0.317	0.003	0.196	0.023	0.302	0.010	0.456	0.024	0.216	0.004	0.140
Recovery	78	76	30	90	84	80	65	88	82	69	55	70
Recoverable value oz./ton	0.034	0.241	0.001	0.176	0.019	0.241	0.006	0.401	0.019	0.149	0.002	0.098
Au. 95% price basis oz./ton	0.032	-	0.00.	-	0.018	-	0.006	-	0.018	-	0.002	-
Price ($5 bonus above 10 gms/ton) (1) 34.9125 (2) 39.9125	(2)	-	(1)	-	(2)	-	(1)	-	(2)	-	(1)	-
Ag. 90% price basis oz./ton		0.217		0.158		0.217		0.361		0.134		0.088
Price U.S. $1.293												
Value U.S. $/ton	1.277	0.281	0.035	0.204	0.718	0.281	0.209	0.467	0.718	0.173	0.069	0.114
Value Can. $/ton	1.373	0.302	0.038	0.219	0.772	0.302	0.225	0.502	0.772	0.186	0.074	0.123
Total Value Can. $/ton	1.67		0.26		1.07		0.73		0.96		0.20	
% MoS_2	-		-		-		-		-		0.091	
Estimated recovery 65%											0.059	
Lbs/Sh Ton MoS_2											1.18	
U.S.$/ton MoS_2 ($ U.S. 1.00/lb)											1.18	
Can.$/ton MoS_2											1.27	
Total Associated Minerals	1.67		0.26		1.07		0.73		0.96		1.47	

VALUATION OF ORE BODIES

Yr.	Pit	Cu Price U.S. ¢/LB	Cu Price Can. ¢/LB	Cu Value In Conc. Can ¢/LB	% Cu	Rec %	Rec % Cu	Cu LB/Ton	Cu $/Ton Ore	* Ass. Min. $/Ton Ore	Total $/Ton Ore	Oper. Cost $/Ton	Revenue $/Ton
1	Little Chief	28.5	30.64	26.10	1.25	83.3	1.04	20.8	5.43	1.07	6.50	4.05	2.45
		30.0	32.25	27.67	"	"	"	"	5.76	"	6.83	"	2.78
		35.81	38.5	33.74	"	"	"	"	7.02	"	8.09	"	4.04
		41.86	45.0	40.05	"	"	"	"	8.33	"	9.40	"	5.35
2	Little Chief	28.5	30.64	26.10	1.08	83.3	0.90	18.0	4.70	"	5.77	3.76	2.01
		30.0	32.25	27.67	"	"	"	"	4.98	"	6.05	"	2.29
		35.81	38.5	33.74	"	"	"	"	6.07	"	7.14	"	3.38
		41.86	45.0	40.05	"	"	"	"	7.21	"	8.28	"	4.52
	Arctic Chief	28.5	30.64	26.10	1.27	86	1.09	21.8	5.69	1.67	7.36	3.82	3.54
		30.0	32.25	27.67	"	"	"	"	6.03	"	7.70	"	3.88
		35.81	38.5	33.74	"	"	"	"	7.36	"	9.03	"	5.21
		41.86	45.0	40.05	"	"	"	"	8.73	"	10.40	"	6.58
3	Little Chief	28.5	30.64	26.10	1.43	83.3	1.19	23.8	6.21	1.07	7.28	3.76	3.52
		30.0	32.25	27.67	"	"	"	"	6.59	"	7.66	"	3.90
		35.81	38.5	33.74	"	"	"	"	8.03	"	9.10	"	5.34
		41.86	45.0	40.05	"	"	"	"	9.53	"	10.60	"	6.84
	Arctic Chief	28.5	30.64	26.10	1.27	86	1.09	21.8	5.69	1.67	7.36	3.73	3.63
		30.0	32.25	27.67	"	"	"	"	6.03	"	7.70	"	3.97
		35.81	38.5	33.74	"	"	"	"	7.36	"	9.03	"	5.30
		41.86	45.0	40.05	"	"	"	"	8.73	"	10.40	"	6.67
	War Eagle	28.5	30.64	26.10	1.13	94	1.06	21.2	5.53	0.73	6.26	3.72	2.54
		30.0	32.25	27.67	"	"	"	"	5.87	"	6.60	"	2.88
		35.81	38.5	33.74	"	"	"	"	7.15	"	7.88	"	4.16
		41.86	45.0	40.05	"	"	"	"	8.49	"	9.22	"	5.50

* Au. Ag. and MoS_2

VALUATION OF ORE BODIES (Cont'd)

Yr.	Pit	Cu Price U.S. ¢/LB	Cu Price Can. ¢/LB	Cu Value In Conc. Can. ¢/LB	% Cu	Rec %	Rec % Cu	Cu LB/Ton	Cu $/Ton Ore	* Ass. Min. $/Ton Ore	Total $/Ton Ore	Oper. Cost $/Ton	Revenue $/Ton
4	War Eagle	28.5	30.64	26.10	1.25	94	1.17	23.4	6.11	0.73	6.84	4.02	2.82
		30.0	32.25	27.67	"	"	"	"	6.47	"	7.20	"	3.18
		35.81	38.5	33.74	"	"	"	"	7.90	"	8.63	"	4.61
		41.86	45.0	40.05	"	"	"	"	9.37	"	10.10	"	6.08
	War Eagle S	28.5	30.64	26.10	0.96	94	0.90	18.0	4.70	"	5.43	3.93	1.50
		30.0	32.25	27.67	"	"	"	"	4.98	"	5.71	"	1.78
		35.81	38.5	33.74	"	"	"	"	6.07	"	6.80	"	2.87
		41.86	45.0	40.05	"	"	"	"	7.21	"	7.94	"	4.01
5	War Eagle	28.5	30.64	26.10	1.35	94	1.27	25.4	6.63	"	7.36	4.13	3.23
		30.0	32.25	27.67	"	"	"	"	7.03	"	7.76	"	3.63
		35.81	38.5	33.74	"	"	"	"	8.57	"	9.30	"	5.17
		41.86	45.0	40.05	"	"	"	"	10.17	"	10.90	"	6.77
	War Eagle S	28.5	30.64	26.10	0.96	94	0.90	18.0	4.70	"	5.43	4.01	1.42
		30.0	32.25	27.67	"	"	"	"	4.98	"	5.71	"	1.70
		35.81	38.5	33.74	"	"	"	"	6.07	"	6.80	"	2.79
		41.86	45.0	40.05	"	"	"	"	7.21	"	7.94	"	3.93
	Cowley Park	28.5	30.64	26.10	0.86	88	0.76	15.2	3.97	1.47	5.44	4.07	1.37
		30.0	32.25	27.67	"	"	"	"	4.21	"	5.68	"	1.61
		35.81	38.5	33.74	"	"	"	"	5.13	"	6.60	"	2.53
		41.86	45.0	40.05	"	"	"	"	6.09	"	7.56	"	3.49
6	Cowley Park	28.5	30.64	26.10	0.86	88	0.76	15.2	3.97	"	5.44	3.88	1.56
		30.0	32.25	27.67	"	"	"	"	4.21	"	5.68	"	1.80
		35.81	38.5	33.74	"	"	"	"	5.13	"	6.60	"	2.72
		41.86	45.0	40.05	"	"	"	"	6.09	"	7.56	"	3.68

* Au. Ag. and MoS_2

VALUATION OF ORE BODIES (Cont'd)

Yr.	Pit	Cu Price U.S. ¢/LB	Cu Price Can. ¢/LB	Cu Value In Conc. Can. ¢/LB	% Cu	Rec %	Rec % Cu	Cu LB/Ton	Cu $/Ton Ore	* Ass. Min. $/Ton Ore	Total $/Ton Ore	Oper. Cost $/Ton	Revenue $/Ton
6	Keewenaw	28.5	30.64	26.10	1.05	76	0.80	16.0	4.18	0.96	5.14	3.78	1.36
		30.0	32.25	27.67	"	"	"	"	4.42	"	5.38	"	1.60
		35.81	38.5	33.74	"	"	"	"	5.40	"	6.36	"	2.58
		41.86	45.0	40.05	"	"	"	"	6.41	"	7.37	"	3.59
7	Cowley Park	28.5	30.64	26.10	0.86	88	0.76	15.2	3.97	1.47	5.44	3.73	1.71
		30.0	32.25	27.67	"	"	"	"	4.21	"	5.68	"	1.95
		35.81	38.5	33.74	"	"	"	"	5.13	"	6.60	"	2.87
		41.86	45.0	40.05	"	"	"	"	6.09	"	7.56	"	3.83
	Keewenaw	28.5	30.64	26.10	1.05	76	0.80	16.0	4.18	0.96	5.14	3.64	1.50
		30.0	32.25	27.67	"	"	"	"	4.42	"	5.38	"	1.74
		35.81	38.5	33.74	"	"	"	"	5.40	"	6.36	"	2.72
		41.86	45.0	40.05	"	"	"	"	6.41	"	7.37	"	3.73
	Best Chance	28.5	30.64	26.10	0.98	90	0.88	17.6	4.59	0.26	4.85	3.66	1.19
		30.0	32.25	27.67	"	"	"	"	4.87	"	5.13	"	1.47
		35.81	38.5	33.74	"	"	"	"	5.94	"	6.20	"	2.54
		41.86	45.0	40.05	"	"	"	"	7.05	"	7.31	"	3.65
8	Best Chance	28.5	30.64	26.10	0.98	90	0.88	17.6	4.59	0.26	4.85	3.66	1.19
		30.0	32.25	27.67	"	"	"	"	4.87	"	5.13	"	1.47
		35.81	38.5	33.74	"	"	"	"	5.94	"	6.20	"	2.54
		41.86	45.0	40.05	"	"	"	"	7.05	"	7.31	"	3.65

* Au. Ag. & MoS_2

OPERATING INCOME

Yr.	Pit	Tons	COPPER PRICE U.S. ¢/LB. 28.5	30.0	35.81	41.86
			COPPER PRICE CAN. ¢/LB. 30.64	32.25	38.5	45.0
1	Little Chief	650,000	1,592,500	1,807,000	2,626,000	3,477,500
	Total	650,000	$ 1,592,500	1,807,000	2,626,000	3,477,500
2	Little Chief	599,010	1,204,000	1,371,700	2,024,700	2,707,500
	Arctic Chief	120,990	428,300	469,400	630,400	796,100
		720,000	1,632,300	1,841,100	2,655,100	3,503,600
	Less Addn. Mine Equip.		22,200	22,200	22,200	22,200
	Less Arctic Chief Preprod.		179,700	179,700	179,700	179,700
			$ 1,430,400	1,639,200	2,453,200	3,301,700
3	Little Chief	269,600	949,000	1,051,400	1,439,700	1,844,100
	Arctic Chief	299,760	1,088,100	1,190,000	1,588,700	1,999,400
	War Eagle	150,640	382,600	433,800	626,700	828,500
		720,000	2,419,700	2,675,200	3,655,100	4,672,000
	Less Addn. Mine Equip.		100,800	100,800	100,800	100,800
	Less War Eagle Preprod.		590,600	590,600	590,600	590,600
			$ 1,728,300	1,983,800	2,963,700	3,980,600
4	War Eagle	613,650	1,730,500	1,951,400	2,828,900	3,731,000
	War Eagle South	106,350	159,500	189,300	305,200	426,500
		720,000	1,890,000	2,140,700	3,134,100	4,157,500
	Less Addn. Mine Equip.		283,200	283,200	283,200	283,200
	Less Cowley Pk. Preprod.		593,000	593,000	593,000	593,000
	Less Cost MoS_2 Circuit		200,000	200,000	200,000	200,000
			$ 813,800	1,064,500	2,057,900	2,981,300
5	War Eagle	162,240	524,000	588,900	838,800	1,098,400
	War Eagle South	108,980	154,800	185,300	304,100	428,300
	Cowley Park	448,780	614,800	722,500	1,135,400	1,566,200
		720,000	1,293,600	1,496,700	2,278,300	3,092,900
	Less Keewenaw Preprod.		102,700	102,700	102,700	102,700
			$ 1,190,900	1,394,000	2,175,600	2,990,200
6	Cowley Park	570,000	889,200	1,026,000	1,550,400	2,097,600
	Keewenaw	150,000	204,000	240,000	387,000	538,500
		720,000	1,093,200	1,266,000	1,937,400	2,636,100
	Less Best Chance Preprod.		83,900	83,900	83,900	83,900
			$ 1,009,300	1,182,100	1,853,500	2,552,200
7	Cowley Park	246,270	421,100	480,200	706,800	943,200
	Keewenaw	335,180	502,800	583,200	911,700	1,250,200
	Best Chance	138,550	164,900	203,700	351,900	505,700
		720,000	$ 1,088,800	1,267,100	1,970,400	2,699,100
8	Best Chance	52,730	62,700	77,500	133,900	192,500
		52,730	$ 62,700	77,500	133,900	192,500

SECTION 1A
GENERAL INFORMATION

GENERAL INFORMATION

Location

The claims of New Imperial Mines Limited are situated along the Whitehorse "copper belt" shown on map 49-1962 "Geology Whitehorse Copper Belt" published by the Geological Survey of Canada. The company holds 286 staked claims and one crown granted claim and has options on 88 staked claims and 21 crown granted claims.

The six orebodies planned for mining extend over 16 miles in a N.W.-S.E. line approximately parallel and 1-3/4 miles to the west of the Alaska Highway. The mill is situated midway along of this line, 5-1/2 miles due south of Whitehorse and 1,200 ft. from the Little Chief pit. The railhead at MacRae is 2-1/2 miles east of the mill.

History

The important showings along the Whitehorse Copper Belt were all staked in 1898 and 1899. Until 1909 not more than 4000 tons of ore had been shipped. Then the construction of a spur of the Whitepass and Yukon Railway for ten miles along the belt enabled a number of the properties to ship crude ore to the smelter at Anyox, British Columbia.

The main production was during the first world war after which all activity ceased apart from drilling in the boom years of 1926 to 1929.

In 1947 and 1948 Noranda Mines did extensive geological mapping, geophysical surveying, trenching and diamond drilling.

In 1955 Imperial Mines and Metals acquired by staking, purchase, or option numerous claims along the copper belt. In the following year the company did 2191 feet of diamond drilling on the Best Chance ore-body as well as geological and magnetometer surveys both around this deposit and in the Arctic Chief areas.

In 1963 the company was reorganized as New Imperial Mines and immediately proceeded on a development plant to investigate the other important properties by means of diamond drilling and geophysical surveys.

It is now considered that sufficient ore has been found to warrant planning for production.

Topography

Whitehorse lies only 80 miles north of the Lynn Canal, the nearest area of the Pacific Ocean. South of Whitehorse lies the Coast Range with peaks to 8,000 feet. The main pass to the south is White Pass of elevation 2,915 feet. The St. Elias Range with elevation up to near 20,000 feet lies to the southwest and west. The ground to the east and north of Whitehorse is of lower rough highland with progressively lower elevations to the north. The valley configuration throughout is northwesterly.

The main physical feature of the Whitehorse district is the large valley of the Lewes River which is approximately four miles wide opposite the town of Whitehorse. The central part of what was a preglacial valley is now filled with silts and boulder clays through which the present Lewes River has cut a narrow, winding secondary valley about 200 feet deep. Many canyon-like tributary valleys enter it from both sides.

Away from the valleys, the terrain is quite rough with rock ridges and hummocks of glacial material.

Climate

Although MacRae lies only 75 miles from the Pacific Ocean, the climate must be classed as continental. The two main mountain ranges act as a rain barrier. While cloud shields from the frequent storms in the Gulf of Alaska will spread rapidly into the area, generally only light precipitation occurs. Heaviest precipitation occurs with a northerly circulation which with the higher elevations to the south give an upslope circulation. Summer precipitation is mainly of a shower nature. Thunderstorms are infrequent in the valley but are more common along the adjacent mountains. Summer precipitation averages 5.49 inches of rain and the average yearly snowfall is 45.6 inches.

The temperature range is also continental with a mean temperature of 30.8 degrees F. A summer maximum of 80 can be expected most years although the maximums are generally near 65° to 70°. The highest temperature reported is 91.1°F. Winters are cold but the proximity of the ocean allows frequent mild spells uncommon in other parts of the Yukon. Thawing temperatures have occurred in every winter month and at some time in every winter. On the other hand, 40° below can be expected most winters and a reading of 50° below, one out of every three winters. The coldest temperature reported is -61.6°F.

The prevailing wind direction is southwest and to a lesser degree north-west. Wind speeds of 20-40 MPH are common from the southeast particularly during the winter months.

INSURANCE CONSIDERATIONS

Apart from the insurance coverage needed during the construction period we are here primarily concerned with fire-insurance, coupled with clauses regarding business interruption.

The rates quoted can vary considerably depending on the provisions made for fighting fire and it has been our experience that the insurance underwriters should be given the opportunity to examine the plans and make suggestions for improvement of the system.

Because of the rate structure the chosen system will depend on the life expectancy of the mine for instance in the past the cost of sprinkler systems can not be balanced off against reduced insurance costs in less than five to seven years.

Each plant must be evaluated through co-operation between engineers and underwriters at an early stage of the design to obtain an optimum economic solution.

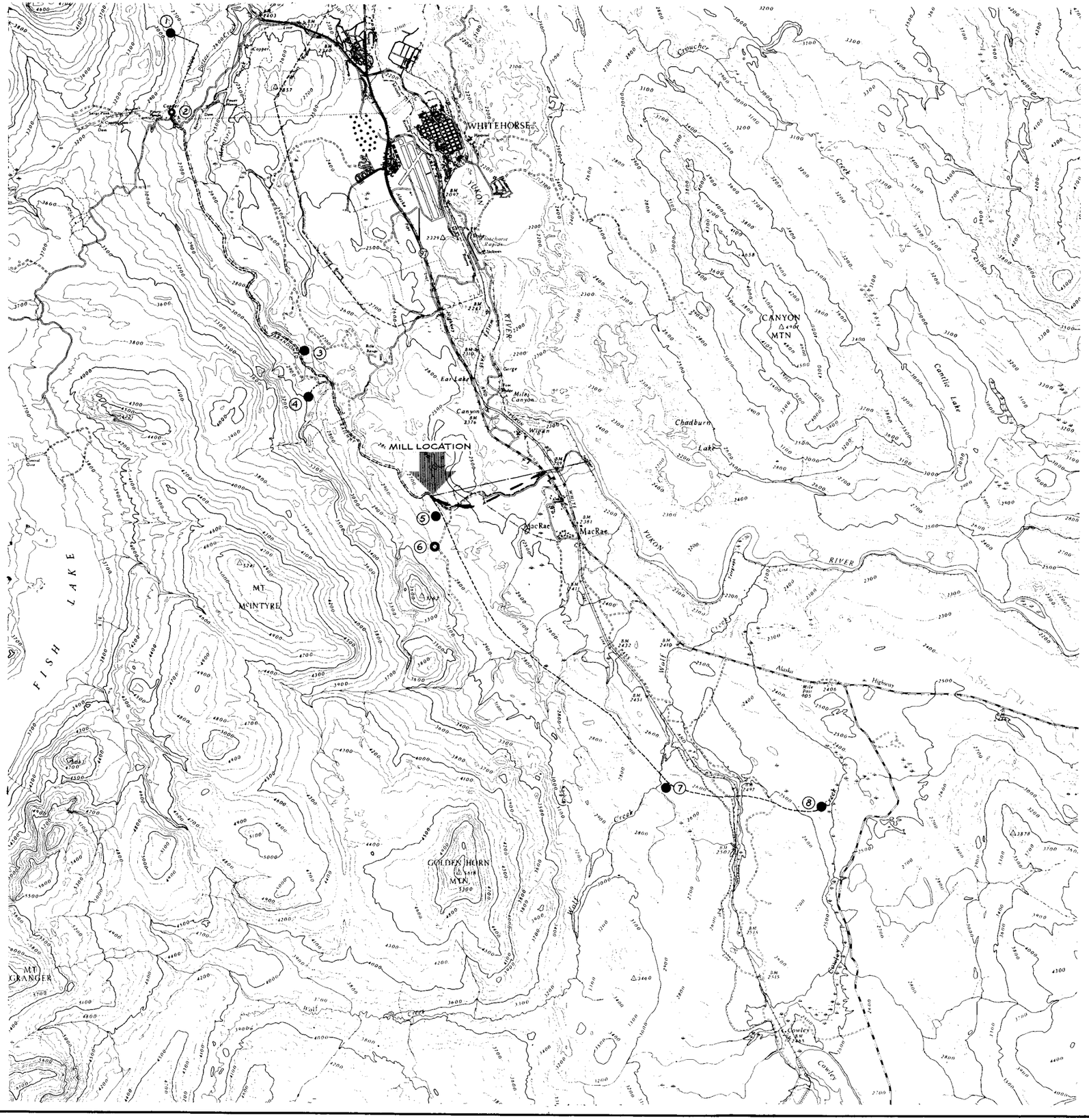
The estimate include allowances for both chemical fire extinguishers in the electrical control rooms and sprinkler systems in the main building areas. Buildings will have an outer cladding of metal to reduce hazards from brush fires and independent fire pumps will be installed in the Little Chief reservoir, one of which will be diesel powered.



LOCATION MAP.

NEW IMPERIAL MINES, LIMITED · WHITEHORSE, YUKON

WRIGHT ENGINEERS LIMITED VANCOUVER ——— CANADA	DRAWN BY NH DATE NOV 65	DRAWING No. 402 | 021 | 001



LEGEND

- ● ORE DEPOSITS
- ○ TENTATIVE ORE DEPOSITS
- (1) WAR EAGLE
- (2) PUEBLO
- (3) BEST CHANCE
- (4) ARCTIC CHIEF
- (5) LITTLE CHIEF
- (6) VALERIE
- (7) KEEWENAW
- (8) COWLEY PARK
- PROPOSED MINE ROAD ----------
- PROPOSED WATER PIPE LINE —·—·—

1 ½ 0 1 2 Miles

SCALE:

DESCRIPTION OF REVISION	ISSUED FOR	APPR.	FINAL CHECK	DESIGN CHECK	DIM. CHECK	DRAWN	DESIGN	APP. DATE
	REPORT					LM		1965

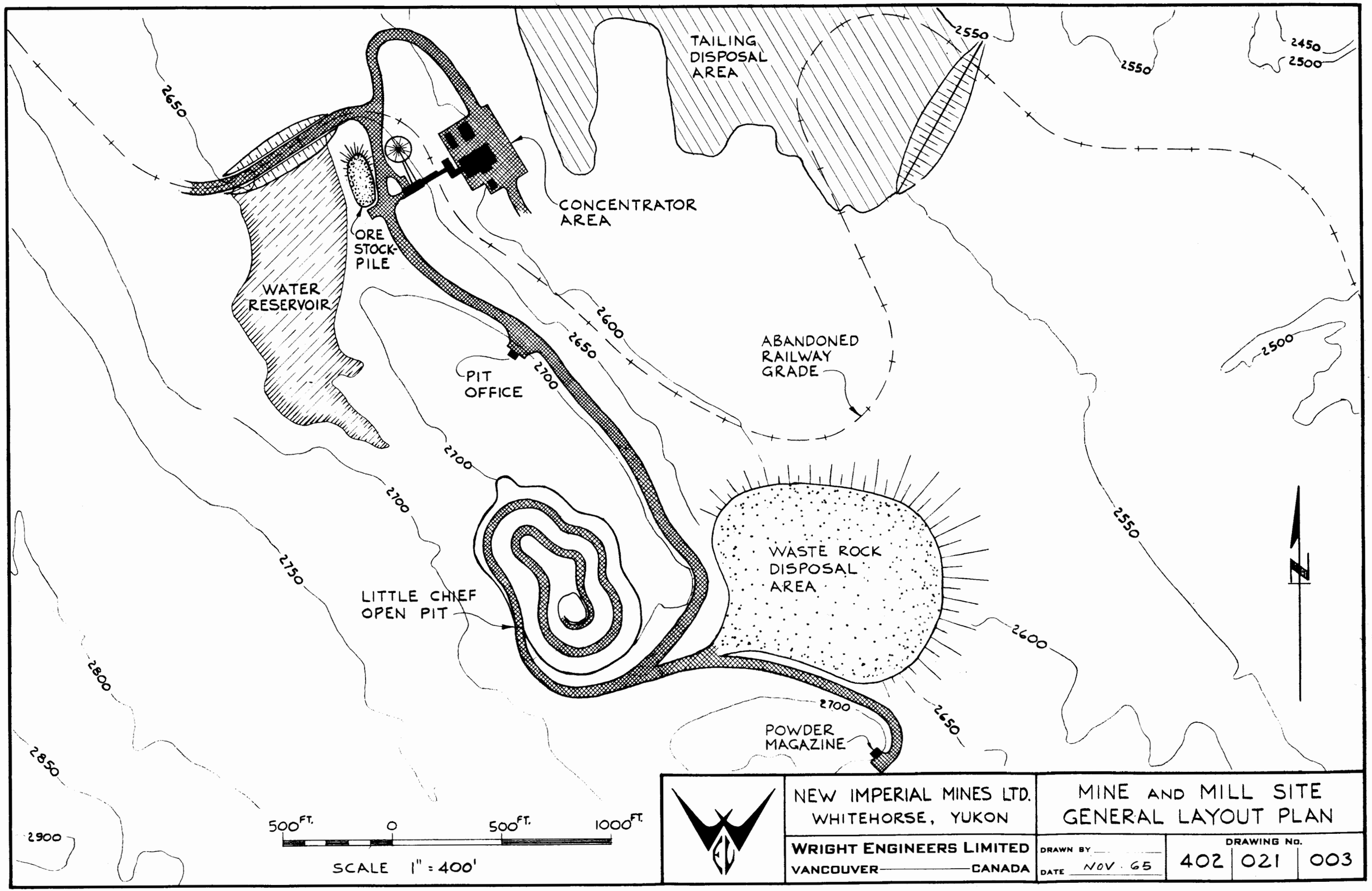
REFERENCE | DWG. No.

LOCATION MAP
WHITEHORSE AREA

NEW IMPERIAL MINES LTD.
WHITEHORSE YUKON

DWG. No. 402 021 002

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA



NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

MINE AND MILL SITE GENERAL LAYOUT PLAN

WRIGHT ENGINEERS LIMITED
VANCOUVER——CANADA

DRAWN BY

DATE NOV. 65

DRAWING No. 402 021 003

LEGEND

1. PIT STOCKPILE AREA
2. CRUSHING & SCREENING PLANT
3. COARSE ORE STOCKPILE
4. FINE ORE BINS.
5. CONCENTRATOR.
6. MINE OFFICE & DRY
7. SERVICES BLDG. & COLD WAREHOUSE
8. ASSAY OFFICE
9. ADMINISTRATION OFFICES & WARM WAREHOUSE.

2600
2575
2625
2650
TO TAILING DISPOSAL AREA
TO WHITEHORSE
N
PIT ROAD
PIT ROAD
ABANDONED RLWY.
2600
2625
2625

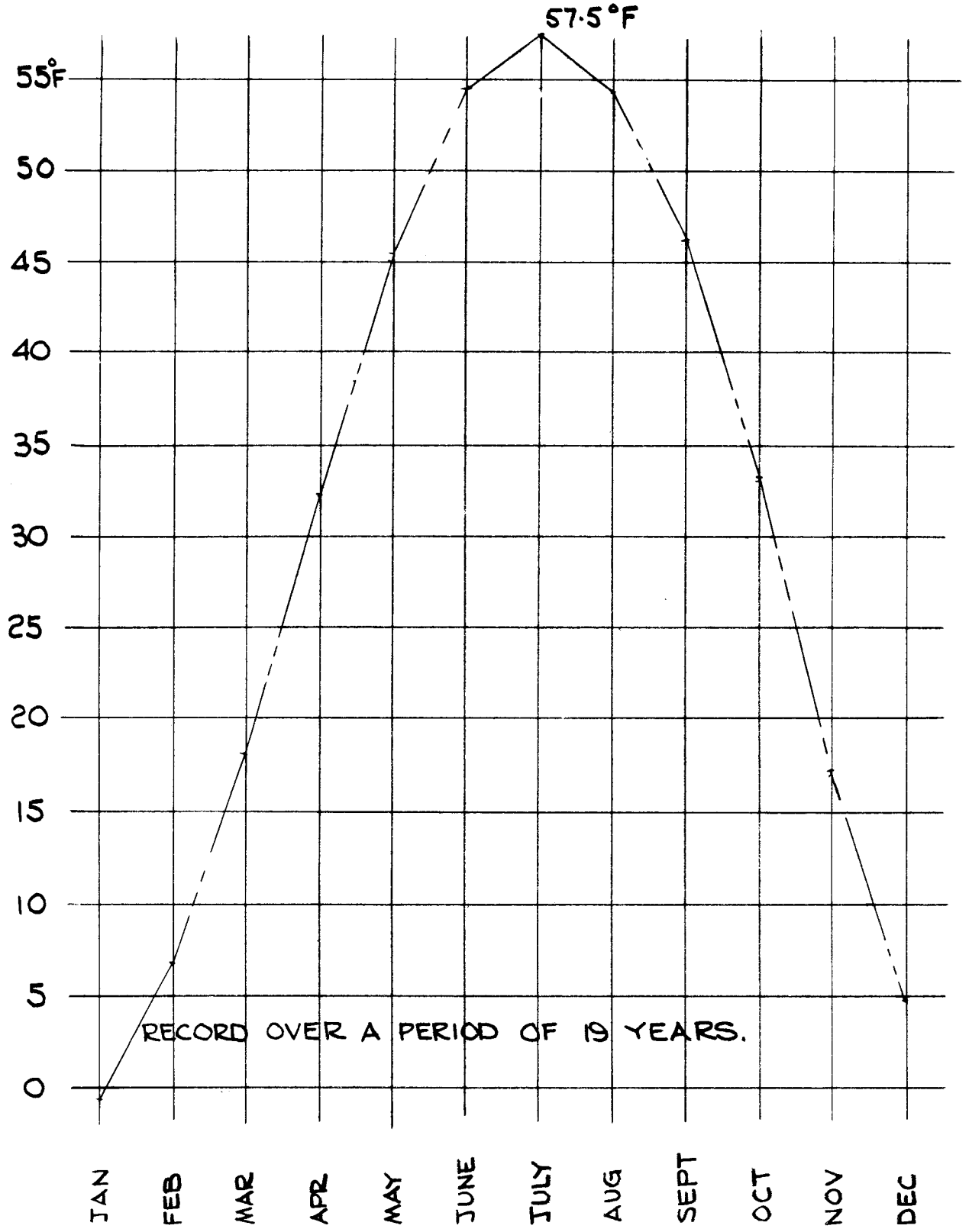
NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

WRIGHT ENGINEERS LIMITED
VANCOUVER——CANADA

MILL SITE

DRAWN BY J. E.
DATE NOV 1965

DRAWING No. 402 021 004



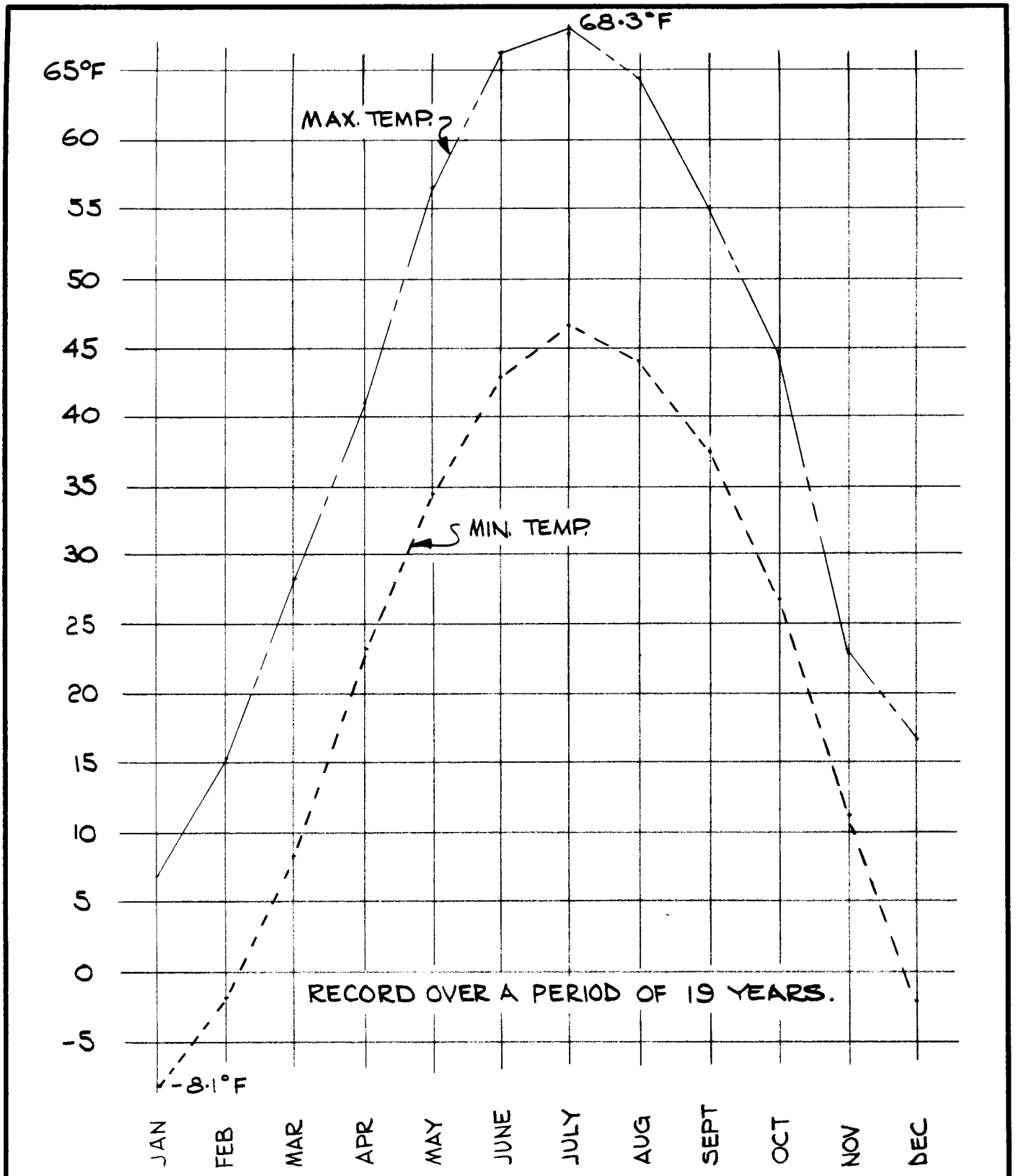
AVERAGE MONTHLY TEMPERATURE FOR WHITEHORSE AIRPORT

NEW IMPERIAL MINES LIMITED - WHITEHORSE, YUKON.

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DRAWN BY S.A. FOX
DATE NOV 65

DRAWING No. 402 | 021 | 005



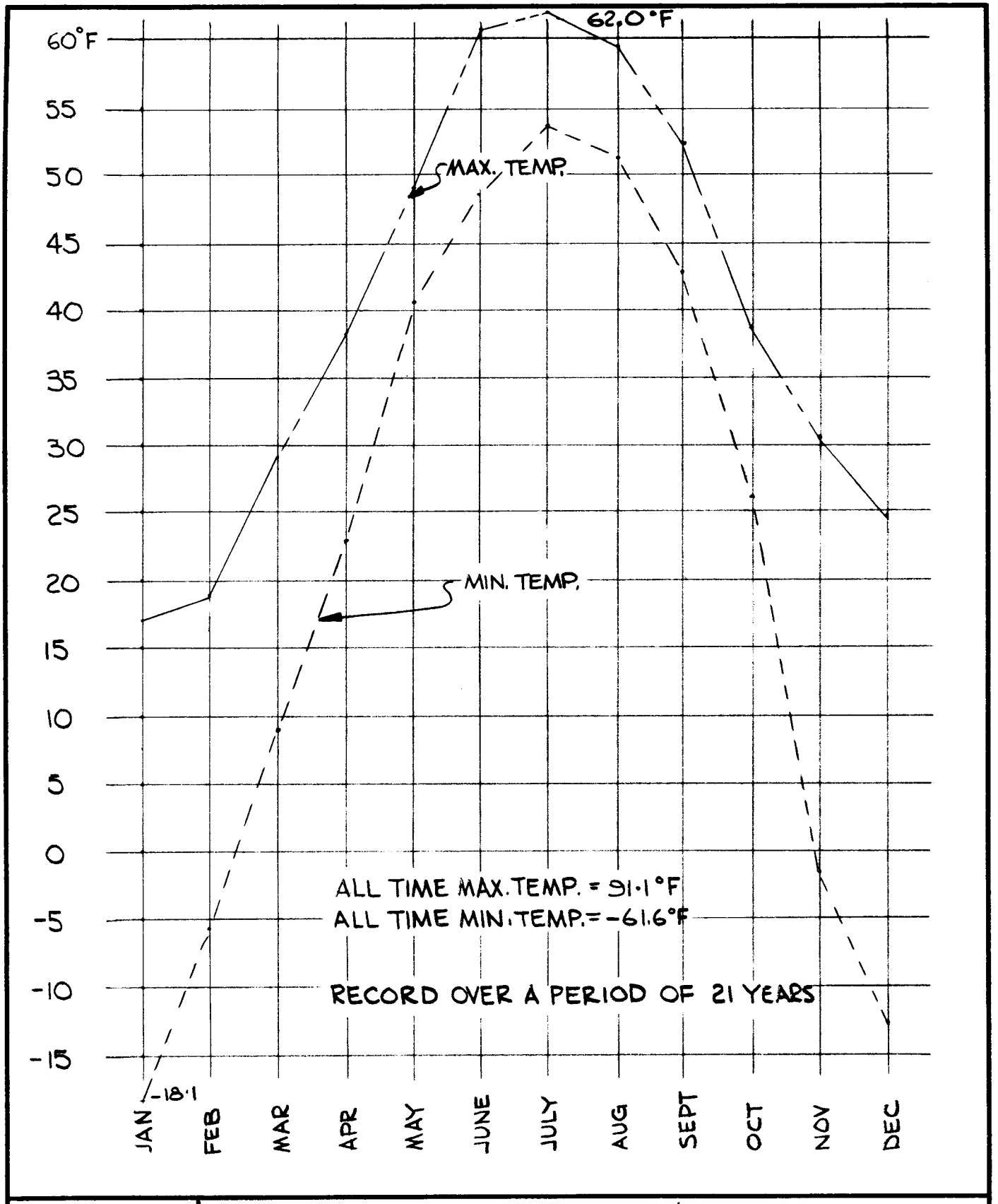
AVERAGE MONTHLY MAX. & MIN. TEMPERATURE FOR WHITEHORSE AIRPORT.

NEW IMPERIAL MINES LIMITED, WHITEHORSE YUKON.

WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

DRAWN BY S.A. Fox
DATE NOV 65

DRAWING NO. 402|021|006



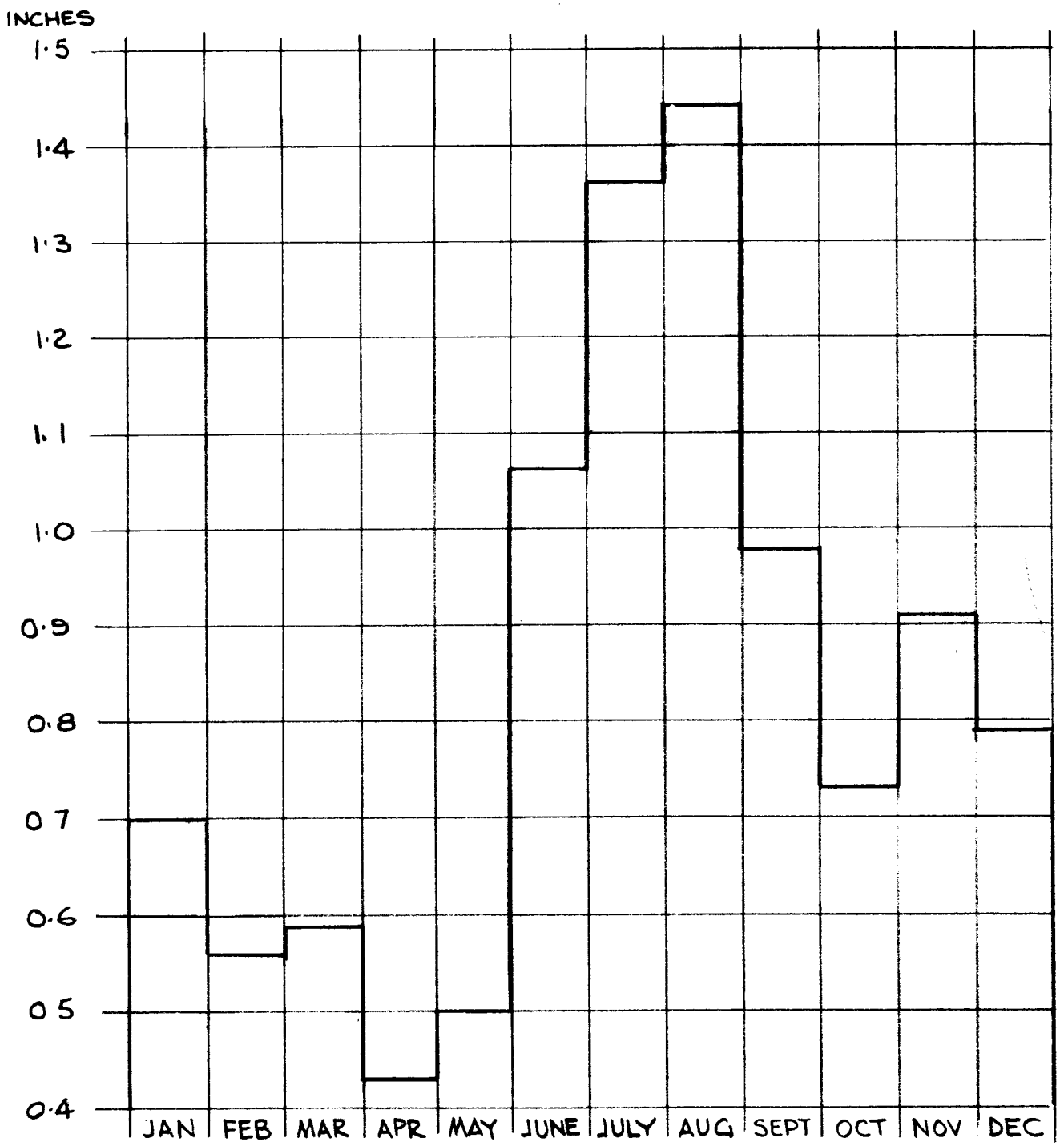
AVERAGE EXTREME MAX. & MIN. TEMPERATURE FOR WHITEHORSE AIRPORT.

NEW IMPERIAL MINES LIMITED, WHITEHORSE YUKON.

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DRAWN BY S.A.Fox
DATE NOV 65

DRAWING No. 402 | 021 | 007



TOTAL ANNUAL PRECIPITATION = 10·05"
RECORD OVER A PERIOD OF 19 YEARS

AVERAGE MONTHLY PRECIPITATION FOR WHITEHORSE AIRPORT.

NEW IMPERIAL MINES LIMITED - WHITEHORSE, YUKON.

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DRAWN BY S.A.FOK
DATE NOV 65

DRAWING No.
E402 | 021 | 008

SECTION 1B
GEOLOGY AND ORE RESERVES

GEOLOGY AND ORE RESERVES

GENERAL GEOLOGY

The recently issued geological map of the Whitehorse Copper Belt by the Canadian Geological Survey shows predominant granitic intrusives containing an elongated zone of limestone and associated sediments with a north-northwesterly strike and a width of up to two miles.

The granitic intrusions gave rise to widespread masses of skarn that have replaced the limestone and consist of varying amounts of garnet, epidote, tremolite, calcite, augite, etc. Magnetite is a common constituent of the skarn and sometimes predominates within the central four miles of the copper belt.

The Pueblo mine was unusual in that hematite was the principal mineral of its large orebody.

All the deposits that have been drilled to date are associated with both major and minor faults which probably were significant during mineralization.

The so-called acid dykes are another invariable associate. They range in composition from granitic to dioritic and frequently are classed as feldspar porphyry. They appear to have been post-skarn formation but to have immediately preceeded mineralization.

Basic dykes of post-ore age are also present in all deposits.

MINERALIZATION

The copper sulphides, bornite and chalcopyrite and their common oxidation products malachite, azurite, cuprite, malaconite, chrysocolla and native copper are found associated with skarn minerals at intervals throughout the contact zones. Late in 1963 the rare copper sulphide valleriite was recognized in the drill core at the Little Chief deposit.

Molybdenite is of economic importance in at least one deposit. Pyrite and pyrrhotite are sometimes present although not in the ore-bodies developed to date.

Both gold and silver are found by assay in appreciable amounts associated with the copper minerals in all the deposits.

ORE DEPOSITS

These usually consist of skarn with a sufficient concentration of copper minerals to form orebodies.

Although small amounts of high grade ore ranging from 10% to 25% Copper have been shipped in the past, large scale mining under present conditions will depend on orebodies ranging from 1% - 2% Copper.

Usually precious metals values are present and so far have averaged as much as 0.05 oz Gold and 0.50 oz Silver per ton in the orebodies tested.

ORE DEPOSITS Cont'd.

New Imperial Mines controls important claims in four separate areas along the copper belt. From south to north they are:

1. Cowley Park - Keevenaw
2. Valerie - Little Chief - Big Chief Group
3. Arctic Chief - Best Chance Group.
4. Pueblo - War Eagle - Copper King Group.

It is roughly 5 miles between the first two areas, 1½ miles between the second and third and 4 miles between the third and fourth.

The second and third areas contain all the deposits in the Copper Belt that are noteworthy for their high content of magnetite. The recently published aeromagnetic map for the Whitehorse area clearly indicated both the Arctic Chief deposits and the Valerie - Little Chief group.

A strong anomaly in the drift-covered area between the Pueblo and Best Chance deposits was found to be due to Quaternary Basalt buried under 200 feet of glacial materials.

In the following pages the salient features of the various mineral occurrences will be described.

Typical plans and sections, which illustrate the geology of the various ore-bodies that have been drilled, accompany this report. They are based on the complete sets of detailed plans and sections to a scale of 1 inch equals 20 feet that have been prepared by the geologists at the mine.

VALERIE - LITTLE CHIEF - BIG CHIEF GROUP

In this area two sets of claims named Oro and Geno have been staked over the old Valerie Mine and the three deposits known as Little, Middle and Big Chiefs that were explored by Noranda Mines in 1947.

The geology of this section is quite significant when studied on the map. A ' finger ' of granite 3000 feet long striking north-northwest has intruded the limestone. On the east side at the base of the finger is the important Little Chief deposit. Immediately north of it is the Middle Chief whilst the tip of the finger is capped by the 'V' shaped Big Chief. On the west side at the base of the finger is the old Valerie Mine. The 3000 feet between it and the Big Chief is covered by overburden, but Noranda Mines detected four anomalies with a dip needle survey that could be due to valuable mineralization. Both magnetometer and self potential surveys conducted in the summer of 1963 confirmed these anomalies. Further confirmation was obtained by an induced polarization survey in 1964.

The drilling in this group of deposits has shown that there is a continuous zone of serpentine between the siliceous skarns and the limestone. This serpentine zone is of great importance since it is usually the host for the mineralization. Originally this serpentine may have been a dolomitic limestone or alternatively a basic lava flow on the sea floor.

VALERIE

The original Valerie Mine is at an embayment in the contact of the granite and the limestone. The outcrops of the ore are now covered by the extensive dumps.

The two shafts are now flooded and no plans exist of the underground workings. The description of the mine by McConnell (1909) applies to the earlier work only.

The old mine itself gives a geophysical anomaly that is 450 feet long and to the north there are four more anomalies totalling 1200 feet in a distance of 2800 feet.

In 1963 and 1964 these five anomalies were tested by 16 diamond drill holes totalling 4,062 feet. Interesting copper-magnetite mineralization was found in each anomaly but nothing of ore grade. A detailed study of the results may justify further drilling.

LITTLE CHIEF

So far this deposit is the largest proven on the Copper Belt as demonstrated by 71 diamond drill holes totalling 28,497 feet, on sections spaced 50 feet apart.

It has a length of 700 feet and horizontal widths up to 180 feet. Until recently the deepest ore known there extended 400 feet below the surface but now the ore at the south end has been found to extend to 750 feet below the surface.

In detail the ore boundaries are highly irregular and are probably the result of the numerous faults that have been noted in the drill core.

Most of the ore is associated with magnetite which in turn is associated with serpentine that it has partly replaced.

The ore minerals are bornite, chalcocite, chalcopyrite and valleriite. Any one of these may predominate in a given section.

The deposit is divided into two by a prominent dyke on which there appears to have been a vertical displacement of the ore body for 80 feet with the south end moved down.

The north end has been cut off by a steep fault with a displacement of 300 feet to the west to the Middle Chief deposit.

The south end of the Little Chief appears to have been dropped on another fault for at least 100 feet vertically.

The south limit is formed by dioritic rocks but these may have been brought into this position by still another fault so detailed investigation is being done in the hope of finding another segment of the deposit.

LITTLE CHIEF Cont'd.

At present the ore reserve at the Little Chief is calculated at:

1,910,000 tons averaging 1.40% Cu.
of which 435,000 tons averaging 1.63% Cu. is in a block below the 600 elevation (about 440 feet below the surface). There are good indications that further drilling could increase this figure to one million tons.

MIDDLE CHIEF

This mineralized zone that is 700 feet long is separated from the Little Chief by an easterly striking fault with a throw of 300 feet and from the Big Chief by a northerly striking fault zone that separates the two by 350 feet. In addition there appears to be another easterly striking fault dividing the Middle Chief into two sections with an offset of 80 feet.

The south segment of the zone appears to be essentially vertical with an average width of 70 feet and a length of 250 feet. For a depth of 100 feet it contains 200,000 tons with an average grade of only 0.54% Cu.

The north segment is almost 400 feet long, 100 feet wide and apparently dips 70^{o} NE. The copper minerals appear to be in four layers of which those on the footwall side which is close to the granite have an appreciable content of magnetite. On the hangingwall side molybdenite is present sporadically. The large amount of included waste results in an average grade of only 0.28% Cu. for this area.

It should be possible to selectively mine on a salvage basis say 100,000 tons of medium grade ore from the Middle Chief. Further drilling of shallow holes in selected areas would be needed to determine the location of selective mining.

BIG CHIEF

Here there is a mineralized contact zone about 50 feet wide in a 'V' shape with the limbs 450 feet long. The principal mineral is magnetite with copper sulphides that have been oxidized on the surface. Noranda's sampling of their trenches gave values of 0.65% Cu to 1.46% Cu over widths of 25 - 70 feet.

The drilling in 1964 showed that the fault zone between the Middle and Big Chiefs was 100 feet wide with a segment of the mineralized zone caught within it.

Two holes in the east limb both gave average assays of 0.50% Cu. A hole in the west limb gave an average of only 0.34% Cu with poor recoveries due to broken ground.

The drilling in the Middle Chief amounted to 14 holes totalling 4,228 feet and in the Big Chief 6 holes totalling 1,480 feet.

ARCTIC CHIEF

In 1954 Mr. P.R. Wilson made geological maps, a magnetometer survey and sampled the underground workings in this area.

The original deposit of the Arctic Chief consists of an irregular lens of massive magnetite that appears to be steep near the surface and then flattens to a dip of 40° to the west beneath a mass of skarn that is up to 400 feet wide and bounded on the west by granite. The footwall of the ore is limestone.

An old tunnel that was driven prior to 1916 follows the ore by means of a drift and three irregularly spaced cross-cuts that indicate a length of 200 feet and an average width of 35 feet. The workings beyond are rendered inaccessible by a bad cave on a fault that dips 30° S.

In 1956 Mr. Wilson obtained an average for his samples in the three crosscuts of 2.9% Cu, 0.10 oz Au, 0.80 oz. Ag.

The 240 feet crosscut adit on the bottom level 60 feet below the upper level has very limited workings on the orebody.

No stoping has been done on the Arctic Chief orebody but there is evidence of hand sorting at the upper portal so that the ore from development was probably shipped. A profit of $4,164 attributed to this mine presumably was obtained in this way.

Diamond drilling in 1963 and recent 'dry drilling', on sections 50 feet apart has shown that there are 155,000 tons in this orebody averaging 1.70% Cu and 28% Fe. Across the dyke that forms the south boundary there is another 18,000 tons averaging 1% Cu. and 9% Fe. The drilling in this area totalled 2,953 feet in 13 holes.

EAST ARCTIC

Here the magnetometer survey carried out by Mr. Wilson confirmed the old survey made in 1947. Further confirmation was obtained in 1962 when lines were run on a different bearing for drilling purposes.

The area was completely masked by overburden where two strong anomalies were obtained separated by a 100 ft. gap in which the projections of an acid and a basic porphry dyke met.

The anomalies appear to indicate a north striking zone of magnetite but the northern section has a prong striking west.

Noranda tested the northern anomaly with a vertical hole and an inclined hole obtaining 1.0% Cu over 99 feet in the first and 1.50% Cu. in the second but recoveries were poor in the first half of each intersection.

EAST ARCTIC Cont'd.

This was the first ore body to be drilled in 1963. An analysis of the results shows that 158,000 tons are present averaging 1.10% Cu and 20% Fe. There is a minor body at 80 feet to the east that contains 17,000 tons averaging 0.7% Cu and 21% Fe. The drilling in this area totalled 2,924 feet in 13 holes on sections 50 feet apart.

The anomaly to the south proved to be due to an ore body amounting to 65,000 tons averaging 1.10% Cu. and 36% Fe, as shown by 9 drill holes totalling 2,390 feet also on sections 50 feet apart.

BEST CHANCE

Here, there is an impressive outcrop of magnetite and skarn over a maximum length of 420 feet in a northerly direction and widths up to 80 feet with patches and disseminations of copper minerals. It is bounded by limestone on the west and granite on the east.

In the past, it was partially explored by two shafts, 35 and 16 feet deep respectively, several pits and at least four vertical shot drill holes for which there are no known records.

The ore consists of chalcopyrite and bornite in scattered patches of the garnet-magnetite skarn with some sections of massive magnetite.

In 1956, Mr. Wilson made a geological map of the immediate area and then magnetometer survey which indicated that the ore extended beneath an area of limestone on the southwest side. This survey also showed that a second ore body was present immediately to the south in an area concealed by overburden and beyond a gap of only 80 feet that the geological mapping indicates is a dyke of granite.

In 1956, a series of -45° holes on 50 feet centres was drilled from the west side of the exposed deposit on six sections of which four had two holes. In 1963 a similar series of holes was drilled on the next six sections to the south but from the east side.

Section 0 shows that the ore body extends under the limestone for a maximum width of 180 feet. Adjacent sections also indicate that granite is present on the west side of the limestone.

Reconstruction of the geology of the sections suggests that the deposit had a highly irregular roof of limestone that is only partially removed. The major part of the copper mineralization is concentrated near the roof or around a large unreplaced lens of limestone within the skarn. Frequently, the copper values are in massive magnetite.

A series of six steep faults that strike northeast are also postulated from the evidence obtained by drilling.

The 21 holes totalled 5,690 feet and the ore reserve calculated from the results obtained amounts to 200,000 tons averaging 1.0% Cu and 20% Fe to a depth of 150 feet.

PUEBLO

McConnell described the ore body exposed by stripping here in 1908 as an irregularly shaped mass 300 feet in length and up to 170 feet wide with an area of 33,000 square feet. It is entirely surrounded by crystalline limestone but is not more than 300 feet from a granite contact.

The ore is essentially cupriferous hematite but for the first 100 feet of depth known in 1909 the copper was in the form of oxide minerals with an average grade of 4% Cu.

In 1926 C.B. White described how the ore had been mined to the 60 feet level by means of open cuts and glory holes. Then a main shaft was sunk to 500 feet and stoping was carried out on levels at 200,300 and 400 feet until March 21st, 1917 when a disastrous cave-in closed the mine. The production to that date had been 140,000 tons.

At the time of this cave-in diamond drilling was being done on the 500 level with considerable success according to the figures of Mr. White. Thus the current flat hole (514) returned 44 feet of 9.6% Cu. at 60 feet depth which would be 240 feet N N W of the shaft. Hole 512 gave 65 feet of 13.8% Cu and 513 gave 11 feet of 3.6% Cu at 40 feet further north showing that the main ore body was still present.

The 514 hole was regarded as discovering a new ore body so that Mr. White recommended drilling from the surface in 1926. As a result three holes cut ore about 200 feet above the 500 feet level intersection and whilst widths of 23 and 24 feet were reported the true width was probably 5 to 6 feet with an average grade of 4%. Mr. White interpreted the results as indicating two ore bodies about 35 feet apart and recommended further drilling.

Mr. E.R. Wilcox in his report on the Arctic Chief dated June 1917 mentions that the'Pueblo mine has been closed down since last March after a bad cave between the second and fourth levels, but is just now commencing operations again to prepare for further production. Previous to the shut-down the mine was producing 200 tons per day.'

Also he wrote ' I have been informed that ore with less than 3.5% copper content was not considered a shipping grade from this mine'.

In 1948 Noranda Mines drilled two inconclusive holes from the surface in an attempt to test the original ore-body at depth on either side of the shaft. They also tried to check an occurrence reported by White on the east side of the creek about 800 feet north of the mine but again obtained poor results.

It is interesting to note that in 1926 Mr. T. Kerruish found from the old records that the Pueblo Mine had made a net profit during the operation of $576,532.

WAR EAGLE

This old mine is about three miles east of Whitehorse and near the Alaska Highway.

WAR EAGLE Cont'd.

McConnell described this property when it had been operating for only two years. The orebody that he saw consisted of an irregularly shaped area, 65 feet across, in which the sulphides were associated with bands of tremolite. He reported an average of 2.71% Cu over a width of 45 feet in an open cut. At 23 feet below the surface a tunnel showed 44 feet of ore consisting of two well mineralized bands 14 and 10 feet wide with lower grade material between. I was able to crawl into this working last year and confirm this description.

A later account was found in the records of the Lowe Estate from which the following was taken:

On the surface there were two lenses of ore each 250 feet long, 6 feet wide and 30 feet apart. They were developed by a shaft 100 feet deep. In the early days production was from open cuts that were 20 feet deep, and amounted to 2000 tons averaging 7% Cu and $2. in gold and silver.

In 1916 and 1917 another 2245 tons were shipped with a recovered value of 5.7% Cu, 0.03 oz Au, and 2.0 oz Ag per ton. This ore came from the shaft and from an open cut 40 feet long and 200 feet to the south.

There is another shaft about 500 feet north of the first which is also 100 feet deep but it produced very little shipping ore. The dump now contains a few tons of massive hematite-magnetite with about 1½ % Cu.

In 1926 and 1929 some drilling was done for which there is only a sketchy record.

In 1939 A.A. McKinnon certified that he had worked on the War Eagle in 1916 when a thirty foot pocket of bornite ore was mined leaving underneath molybdenum ore that was 20 feet long and 10 feet wide.

It is reported that this property produced a net profit of $43,379. for its owners.

During the past twelve months diamond drilling has been successful in outlining an important ore body beneath an area of overburden a few hundred feet north of the old main shaft as described below.

The ore occurs in a metamorphosed succession of limestones, cherts and quartzites that have been explored by diamond drilling over a width of 500 feet and a length of 1600 feet. For half of this drilling the section lines were 50 feet apart and for the rest 100 feet apart.

89 holes were diamond drilled for a total of 25,851 feet and in addition 50 dry percussion holes that totalled 5215 feet.

The sediments strike at N 20° W at the south end of the explored area and at N 30° W toward the north end. They dip from 65° at the south end to about 45° W toward the north end.

Near the north end of the area the sediments have been invaded by a swarm of dykes from 5 to 20 and more feet wide and apparently striking W N W. They are feldspar porphyry dykes with from 2 - 50% hornblende. They appear to pinch out rapidly in the southeast direction apart from one dyke which passes to the east side of the zone and widens from 5 feet at the north to 30 feet at the south end.

WAR EAGLE Cont'd.

There are also a number of fine grained basic dykes, usually less than 5 feet thick, that appear to be interbbeded with the sediments.

The sediments have been replaced by various types of skarn in an apparently irregular manner and then mineralized with bornite, chalcopyrite, chalcocite and sporadically with molybdenite. In some places the feldspar porphyry dykes have been mineralized as well - especially with molybdenite.

In a length of 600 feet and a width from 40 to 120 feet in the centre of the area a body of ore has been demonstrated by the drilling to contain 900,000 tons averaging 1.40% Cu over a vertical range of 250 feet below the surface. On six of the drill sections in this ore body the vertical range has not been fully explored at the bottom so that there is a possibility of developing another 130,000 tons averaging 1.20% Cu.

At its north end this ore body fades out in the swarm of dykes and a possible northeast striking fault. From 250-450 feet further north some preliminary drilling has indicated a body of low grade mineralization that is 100 feet wide and averages about 0.5% Cu. There is also a parallel zone at 80 feet to the east that is 40 feet wide and of a similar grade. Further drilling will be needed here to trace the zone northward in the hope of finding a better grade.

At its south end the main ore body passes into a low grade zone for 300 feet although this includes the old main shaft where good ore was extracted in the past.

Next there is a length of 250 feet indicated by drilling to average about 1% Cu over 35 feet for a depth of 100 feet. Similar parallel bodies were also found at 50 feet and 250 feet respectively to the west and still another at 150 feet to the east.

These four small deposits are together estimated to contain 330,000 tons averaging 1.0% Cu.

This south section of the area is characterized by the lack of limestone which is presumably represented by skarn.

Exploration should be continued along the zone to the south.

COPPER KING

This mine is in an isolated mass of limestone surrounded by granitic rocks. The limestone is largely converted to silicates which are accompanied by copper sulphides especially near the unaltered cores of limestone. The ore sections are characterized by tremolite. In 1900 McConnell reported that the main deposit dipped at 46° towards the granite contact. It had been followed down dip for 130 feet (91 feet vertically). Above the 63 foot level an excavation 15 - 30 feet long and up to 10 feet wide showed the size of an original lens of high grade ore.

About 200 feet north of this is a wide garnet-limestone band with copper mineralization which was followed in a steeply inclined shaft for 65 feet. The first 20 feet followed a small high grade lens of bornite ore but the rest of the workings, including short drifts from the bottom, are in unworkable material.

COPPER KING Cont'd.

At 200 feet south of the main shaft another one was sunk for 40 feet on a lens of ore.

Over 500 tons averaging 15% Cu had been shipped by 1908 and in 1917 Mr. Wilcox stated that the property was shipping 300 tons a month. In 1918 good ore was encountered on the 100 foot level.

Between 1915 and 1920 shipments totalled 3288 tons averaging 10% Cu which netted a profit of $88,073. according to the old records.

A geological map of this area and a later induced polarisation survey suggest the possibility of a medium sized low grade ore-body.

Drilling was started in August 1964 with a light exploration machine. Four holes were drilled of which the most interesting returned 64 feet that averaged 1.30% Cu. Further drilling here is warranted.

COWLEY PARK

This is the most southerly of all the known mineral occurrences on the Copper Belt. It is in an easterly striking metamorphosed remnant of sediments consisting mostly of limestone, at least 3000 feet long and 700 feet wide, at the south end of the main granitic intrusive.

Much of the limestone has been converted to skarn consisting of garnet, tremolite, wollastonite, epidote, etc with some areas mineralized by bornite, chalcopyrite, molybdenite, pyrite and magnetite.

In the main area investigated to date there are two old, shallow shafts and some bulldozed cuts.

A length of 600 feet has been drilled with sections at 100 feet intervals plus another section at 200 feet further to the west.

The granitic rock was found to have a bowl shape with altered hybrid rock between it and the skarn. The ore is restricted to a part of this skarn.

Major faulting is present but its relationship to the ore, if any, has not been determined.

The grades of copper and molybdenite vary widely from hole to hole.

At present the ore reserve amounts to:

1,370,000 tons averaging 0.87% Cu, 0.08% MoS_2.

A total of 13,449 feet was drilled in 45 holes.

KEEWENAW

This deposit is on the east side of Wolfe Creek, 2½ miles west of the Cowley Park deposit, in an altered, dioritic phase of the main intrusion at its contact with limestone.

It is cut by a swarm of intermediate and basic dykes that make the interpretation of the ore shapes difficult. Faulting is also present that adds further complications.

Most of the mineralization consists of bornite scattered through the dioritic rock. Deep oxidation is present in some parts of the deposit with the copper in the form of chrysocolla.

The deposit was trenched, mapped, sampled and diamond drilled with 5 holes by Noranda Mines in 1947.

During the past winter a series of 29 holes on 50 ft. centres totalling 8625 feet were drilled by New Imperial.

A recent interpretation of this drilling shows that near the surface the mineralized ground occupies a roughly circular area about 300 feet across. At 100 feet lower it is 250 feet across and at another 100 feet lower it is not more than 120 feet in diameter.

A core of non-mineralized feldspar porphyry dyke rock strikes northwest across the ore-body for 350 feet with a maximum width of 60 feet at the centre and pinching out to nothing at each end. It is cut off on its northeast side however by a vertical fault striking N 30° W. On the other side of this fault to the north there is a similar mass of the same rock. It could originally have been part of the other mass and have been moved 300 feet horizontally along the fault.

The east side of the second feldspar porphyry is also limited by a fault which dips at 45° S W and cuts the vertical fault.

At depth the southern porphyry dyke rapidly diminishes in length to become pipelike.

I have estimated that the sulphide ore here amounts to 380,000 tons averaging 1.10% Cu. The stripping ratio will be about 0.75 cu yards of waste to 1 ton of ore.

ORE RESERVES

The reasonably assured reserve at each deposit has been calculated under two categories:

1. Proved ore is within 25 feet of a drill hole.
2. Probable ore is from 25 to 50 feet from a drill hole.

In both cases the ore is geologically reasonable.

In most holes the drill core recoveries have been better than 95% but where more than 20% of the core is missing from a sample the missing portion is assumed to be barren in the calculations.

ORE RESERVES Cont'd.

The recoverable values of gold and silver are based on the assays for these metals in the copper concentrates obtained during the mill test work by Mr. J.W. Britton, P. Eng., Consulting Metallurgist.

The value for the molybdenite is based on a 65% recovery from the Cowley Park ore in preliminary work although a better performance is expected with further tests.

The figures for the iron content of the ore-bodies have been deleted because the high cost of transportation and handling the magnetite concentrates would leave no profit. A production of 150,000 tons per year of magnetite concentrates for a period of at least four years was contemplated with a value at Skagway of $1,400,000 per year.

The tabulation of the reserves for the various ore-bodies is as follows:

	PROVED				PROBABLE			
DEPOSIT	Tons	Cu %	Au,Ag $	MoS_2	Tons	Cu %	Au,Ag $	MoS_2
Little Chief	1,780,000	1.40	1.07		130,000	1.60	1.05	
Arctic Chief	400,000	1.32	1.67					
War Eagle	1,080,000	1.33	0.73		150,000	1.00	0.50	
Cowley Park	780,000	0.84	0.20	0.091	590,000	0.92	0.25	0.087
Keewenaw	380,000	1.10	0.96					
Best Chance	200,000	1.00	0.26					
Totals	4,620,000	1.23	0.85	0.015	870,000	1.03	0.41	0.059
Total Reserve	5,490,000	1.20	0.78	0.022 or $1.00 recoverable per ton as Au,Ag & MoS_2.				

Ross and Associates found that in their designs for the open pits the following tonnages and grades would be available from the various ore-bodies after allowing for a suitable dilution in each case. For comparison the corresponding ore reserve figures are given alongside.

	AS MINED				ORE RESERVE			
	Tons	Cu %	Au,Ag	MoS_2	Tons	Cu %	Au,Ag	MoS_2
Little Chief above 600'	1,518,610	1.21	1.07		1,475,000	1.33	1.05	
Arctic Chief	420,750	1.27	1.67		400,000	1.32	1.10	
War Eagle-Main	926,530	1.24	0.73		900,000	1.40	0.75	
-South	215,330	0.96			330,000	1.00	0.50	
Cowley Park	1,265,050	0.86	0.20	0.091	1,370,000	0.88	0.25	0.080
Keewenaw	485,180	1.05	0.96		380,000	1.10	1.05	
Best Chance	191,280	0.98	0.26		200,000	1.09	0.25	
Total	5,022,730	1.10	$ 0.85	$ 0.2	5,055,000	1.13	$ 0.71	$ 0.26

ORE RESERVES Cont'd.

The differences are due to the mining dilution in most cases.

In the War Eagle and Cowley Park some ore was found to be uneconomic to mine.

Because the oxide copper at the Keewenaw is not recoverable, about 100,000 tons was deleted from the ore reserve but since it was present in the test sample it was included in the mining calculations. Further checking and testing will be necessary to clarify the picture here.

FUTURE EXPLORATION

In addition to the possible extensions of the developed ore-bodies there are a number of occurences that should be drilled in future:

1. COPPER KING
   Only a start has been made on the diamond drilling here.

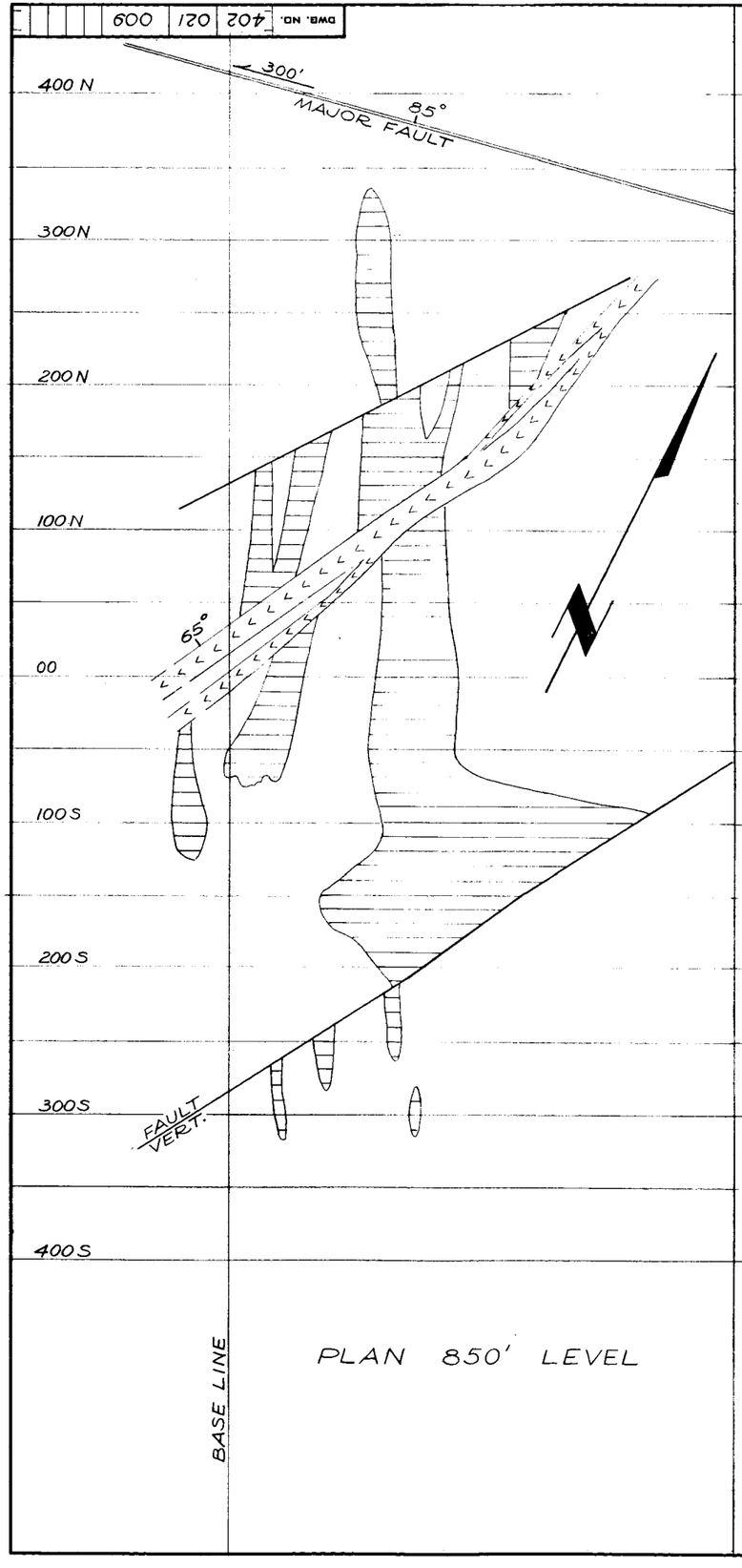
2. PUEBLO
   It should be possible to check the reported high grade ore on the 500 level with a vertical drill hole about 600 feet long from the surface. A possible mineralized area to the northeast of the mine was indicated by the I.P. Survey in 1964. A bulldoze cut failed to penetrate the deep overburden although some mineralized material was found. Dry drilling is proposed to test this ground.

3. ANACONDA AND RABBITT'S FOOT
   These old prospects are near the Alaska Highway on the road to the War Eagle. They should be mapped geologically and then tested with the O. B. machine.

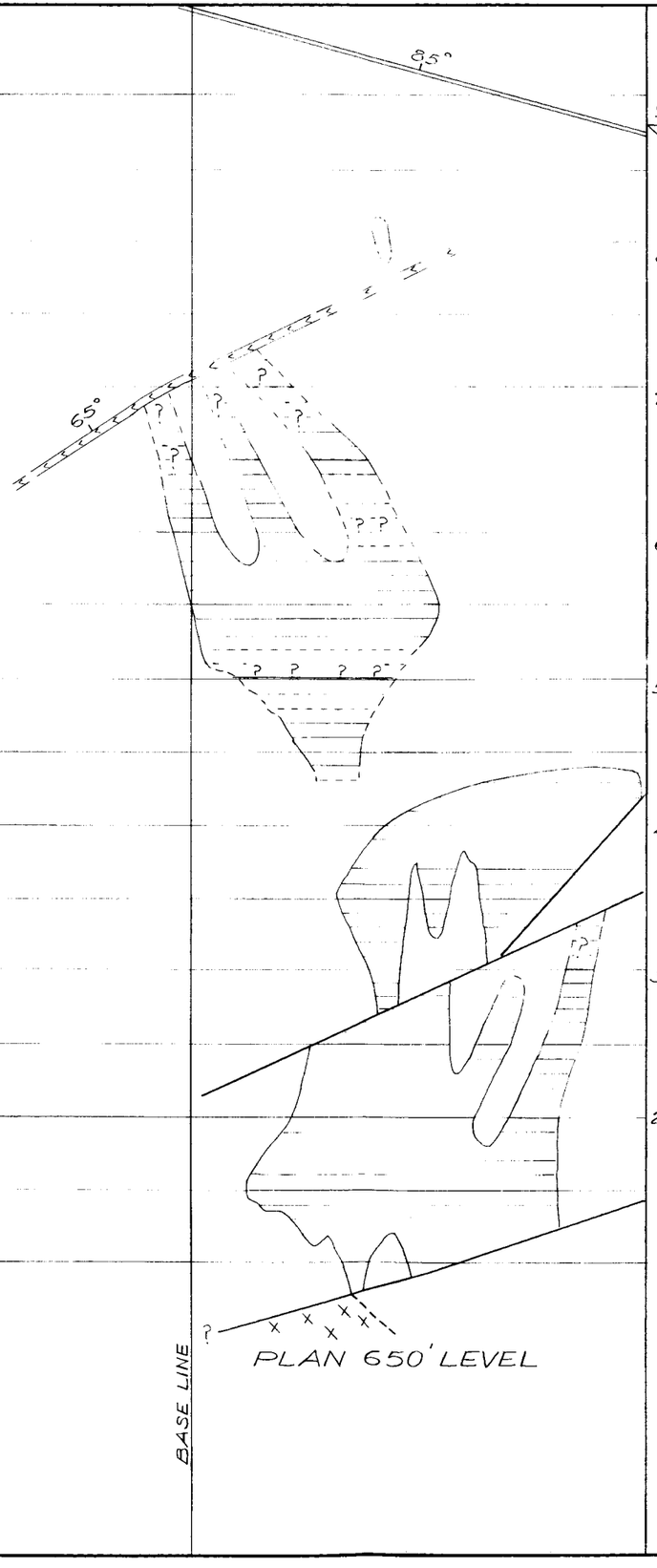
4. KEEWENAW AREA
   At about 500 feet to the northeast of the Keewenaw depost similar mineralization has been found by drilling to be associated with an I. P. anomaly. No ore grade has been located but further work is justified.

   An old trench was found about 2000 feet to the east-southeast of the main Keewenaw deposit. It is 60 feet long and extensively mineralized with malachite and some chalcopyrite in a silicate skarn. Attempts to reach bedrock in parallel and cross trenches apparently failed. This occurrence certainly warrants drilling.

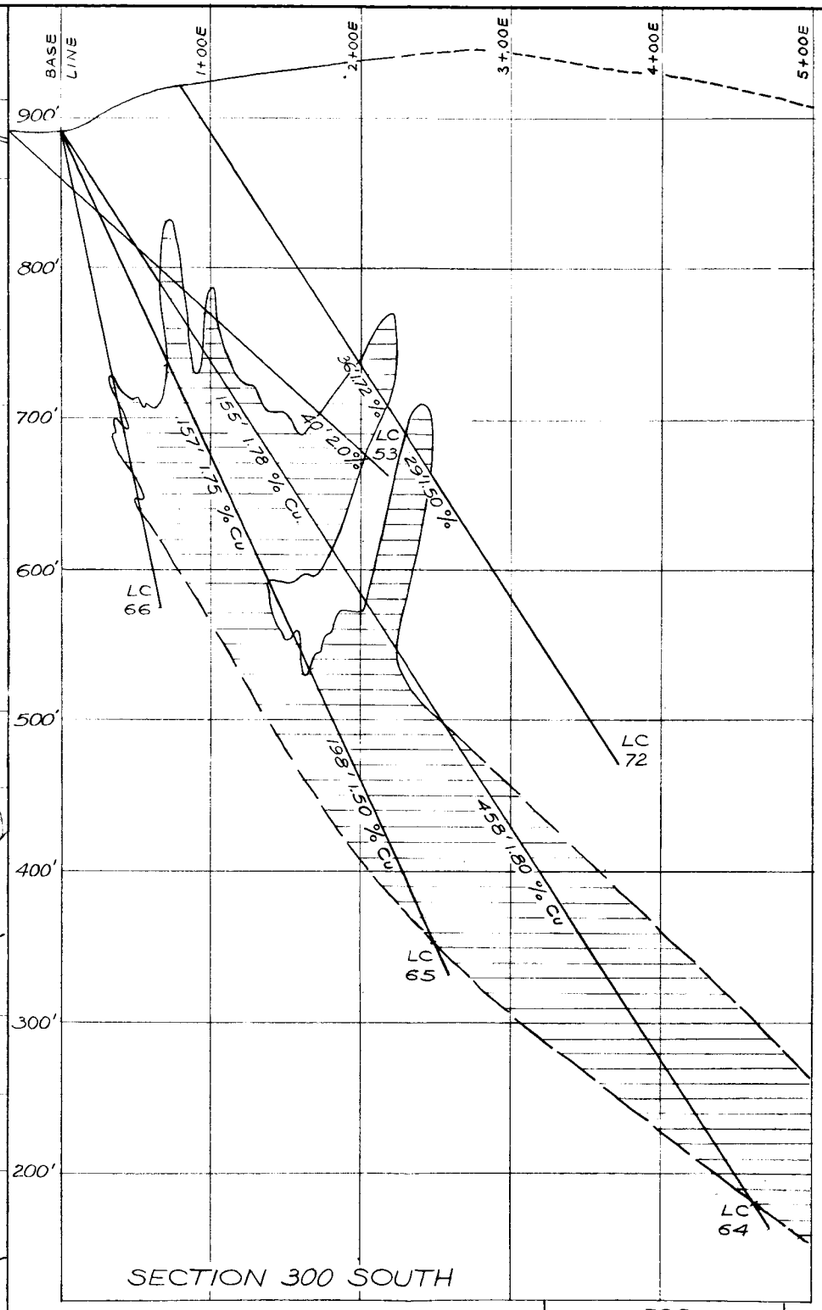
DWG. NO. 402 021 009



PLAN 850' LEVEL



PLAN 650' LEVEL



SECTION 300 SOUTH

FOR
DR. A.C. SKERL
SEPT. 1965

REFERENCE	DWG. NO.

LITTLE CHIEF LEVEL PLANS & SECTION

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

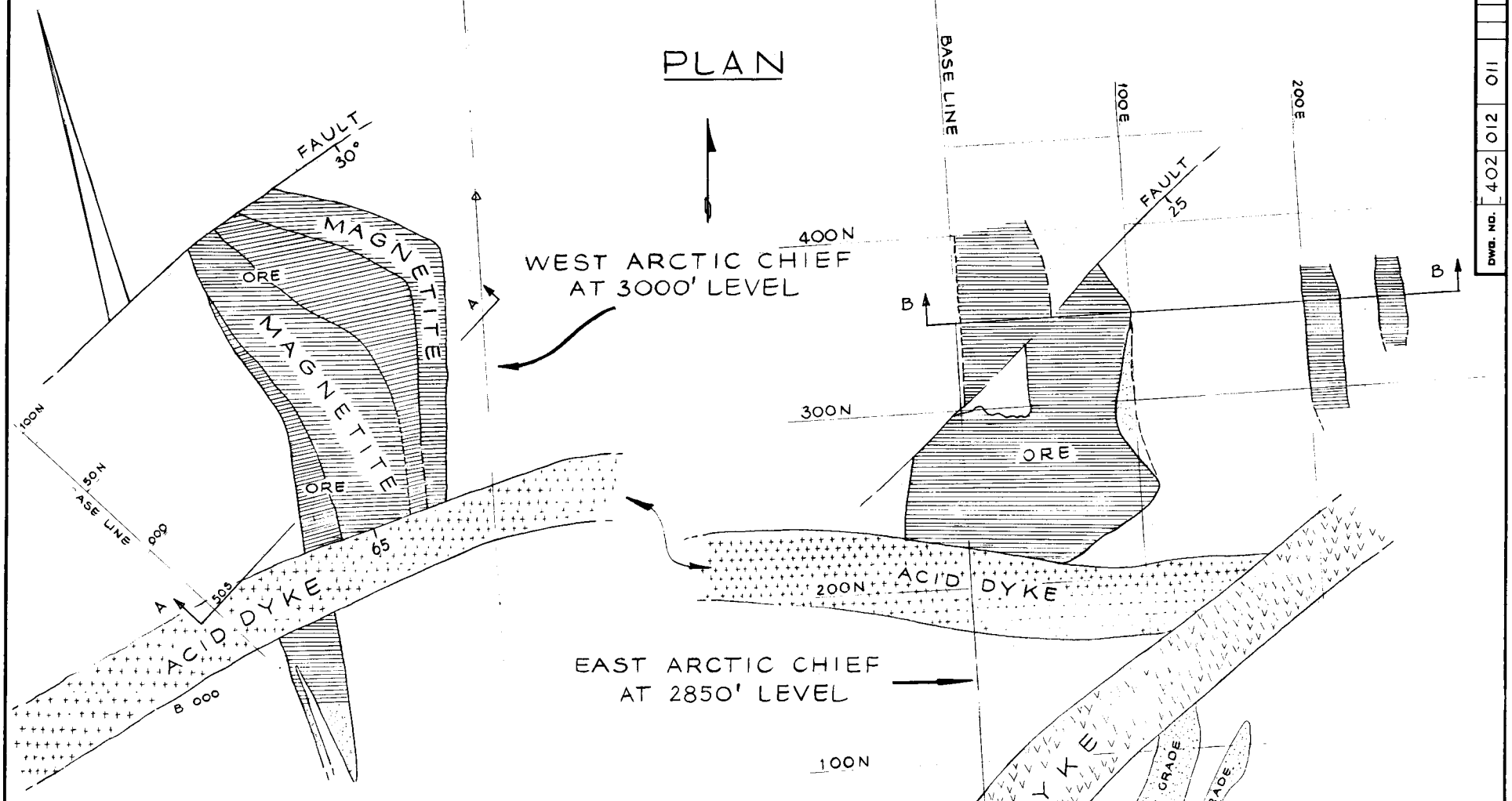
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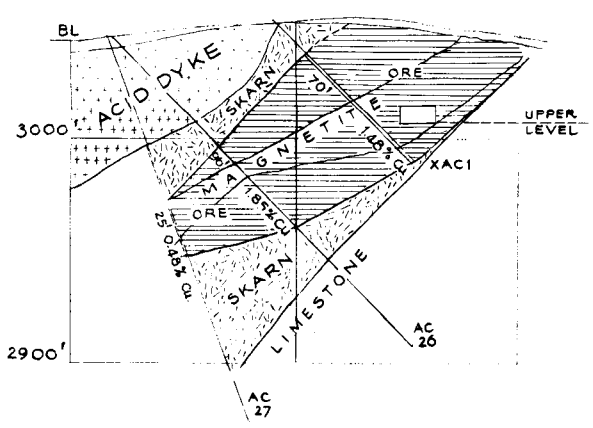
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ISSUED FOR REPORT.					

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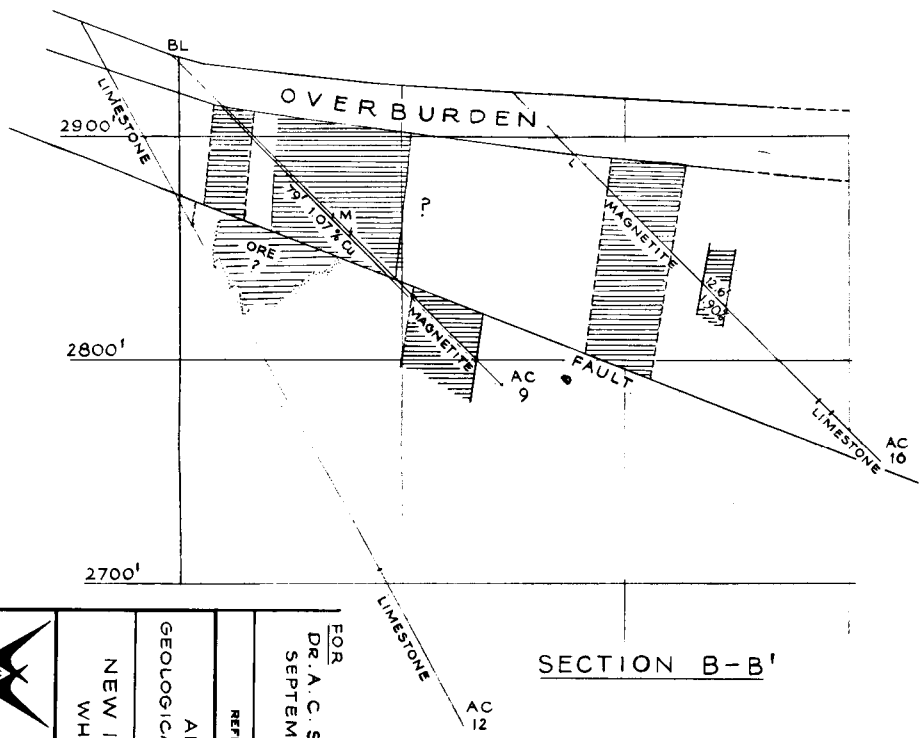
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VANCOUVER ——— CANADA

PLAN

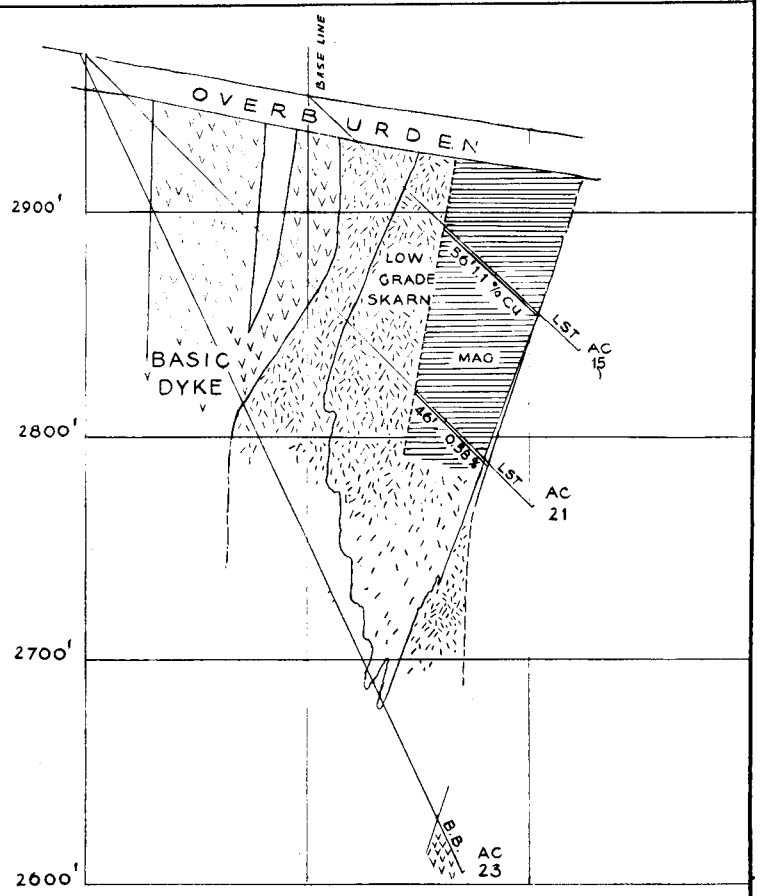




SECTION A-A'



SECTION B-B'



SECTION C-C'

LEGEND

- DIORITE
- ACID DYKE
- BASIC DYKE
- LIMESTONE
- SKARN
- MINERALIZATION
- ORE

DWG. NO. 402 012 011

APP. DATE	DESIGN	DRAWN	DIM. CHECK	DESIGN CHECK	FINAL CHECK	APPR.	ISSUED FOR	DESCRIPTION OF REVISION
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ARCTIC CHIEF
GEOLOGICAL PLAN & SECTIONS

NEW IMPERIAL MINES
WHITEHORSE YT

REFERENCE

DWG. NO.

FOR
DR. A. C. SKERL
SEPTEMBER 1965

WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

DWG. NO. 402 021 011

DWG. NO. 402 021 012

BASE LINE

000

50W

100W

150W

200W

250W

300W

200N

150N

100N

50N

000

50S

100S

150S

200S

300S

350S

400S

FOR
DR. A.C. SKERL
SEPT. 1965

LEGEND

GRANITE

LIMESTONE

SKARN

MINERALIZATION

REFERENCE	DWG. No.

BEST CHANCE
GEOLOGY 625' LEVEL

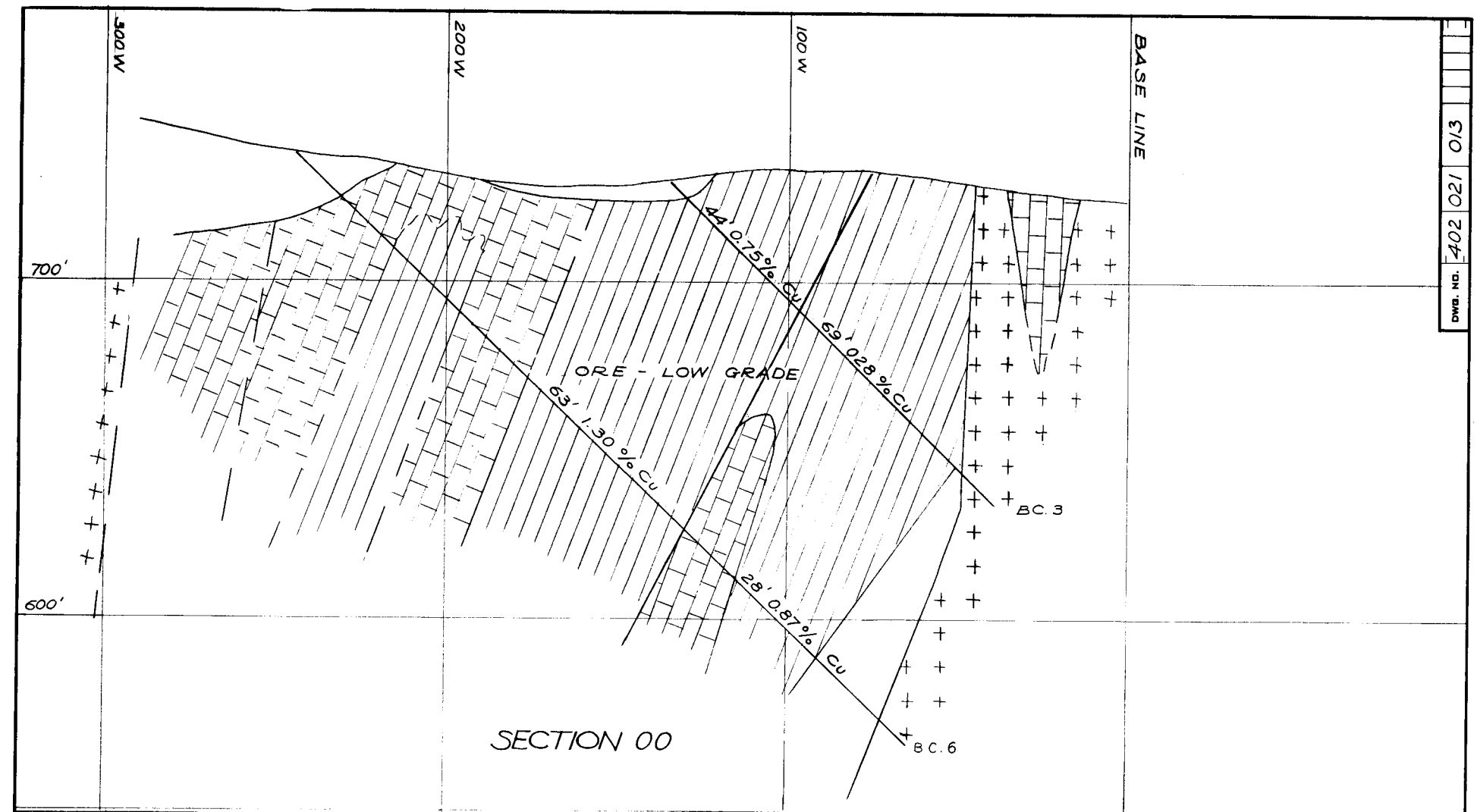
NEW IMPERIAL
MINES LTD.
WHITEHORSE, YUKON

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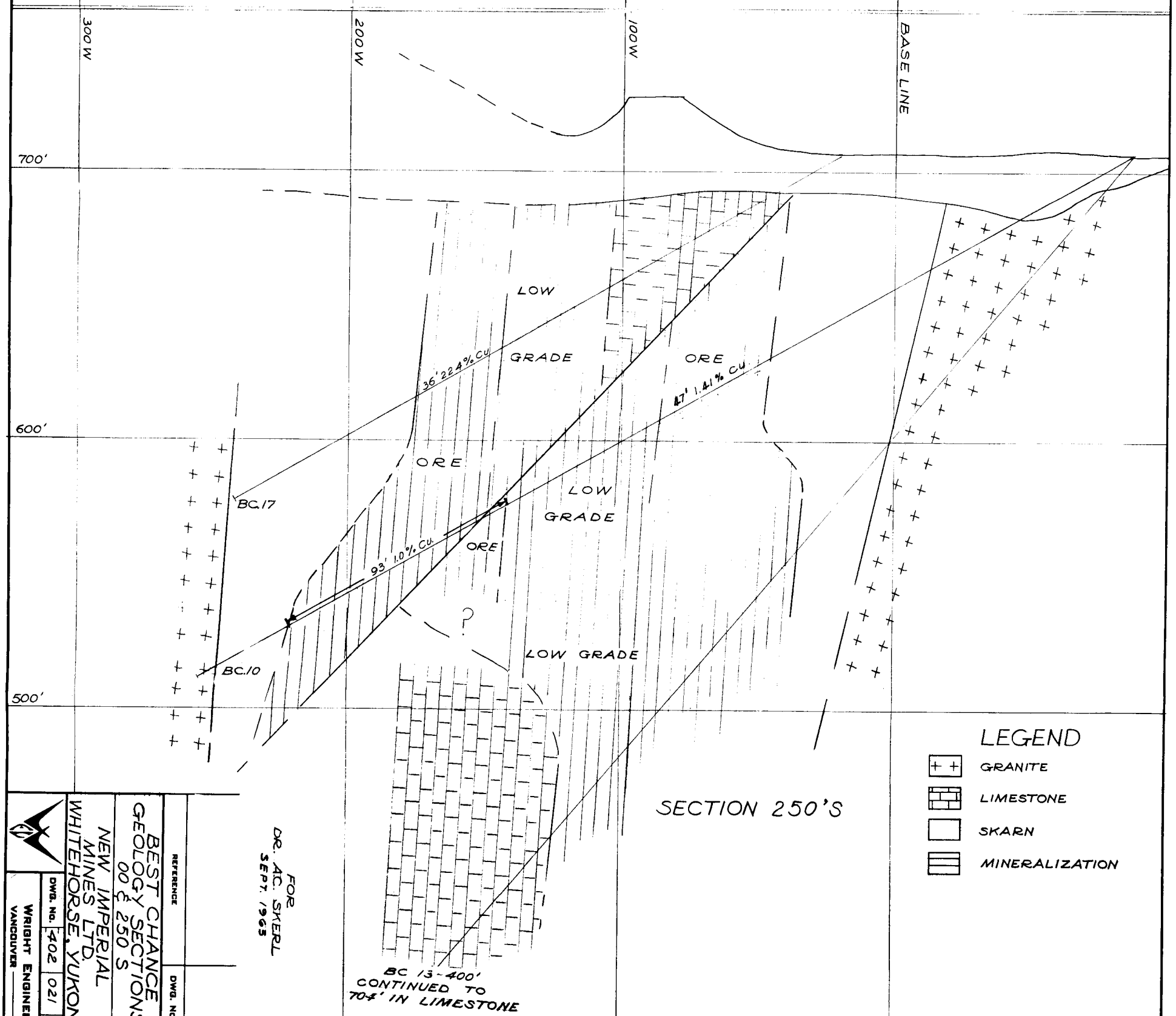
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SCALE:



SECTION 00



SECTION 250'S

LEGEND

- GRANITE
- LIMESTONE
- SKARN
- MINERALIZATION

FOR
DR. AC. SKERL
SEPT. 1965

BEST CHANCE
GEOLOGY SECTIONS
00 & 250 S

NEW IMPERIAL
MINES LTD.
WHITEHORSE, YUKON

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DWG. NO. 402 021 013

REFERENCE

DWG. NO.

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							REPORT.	

SCALE:

DWG. NO. 402 021 014

600W

500W

400W

300W

200W

100W

BASE LINE

100E

300N 400N 500N 600N 700N 800N 900N 1000N 1100N 1200N 1300N

?F

No 1 SHAFT

No 2 SHAFT

?

FOR
DR. A.C. SKERL
SEPT. 1965

LEGEND

- LIMESTONE
- SKARN & QUARTZITE
- ACID DYKE
- ORE

1400N 1500N

REFERENCE DWG. No.

WAR EAGLE
GEOLOGY 600' LEVEL

NEW IMPERIAL
MINES LTD.
WHITEHORSE, YUKON

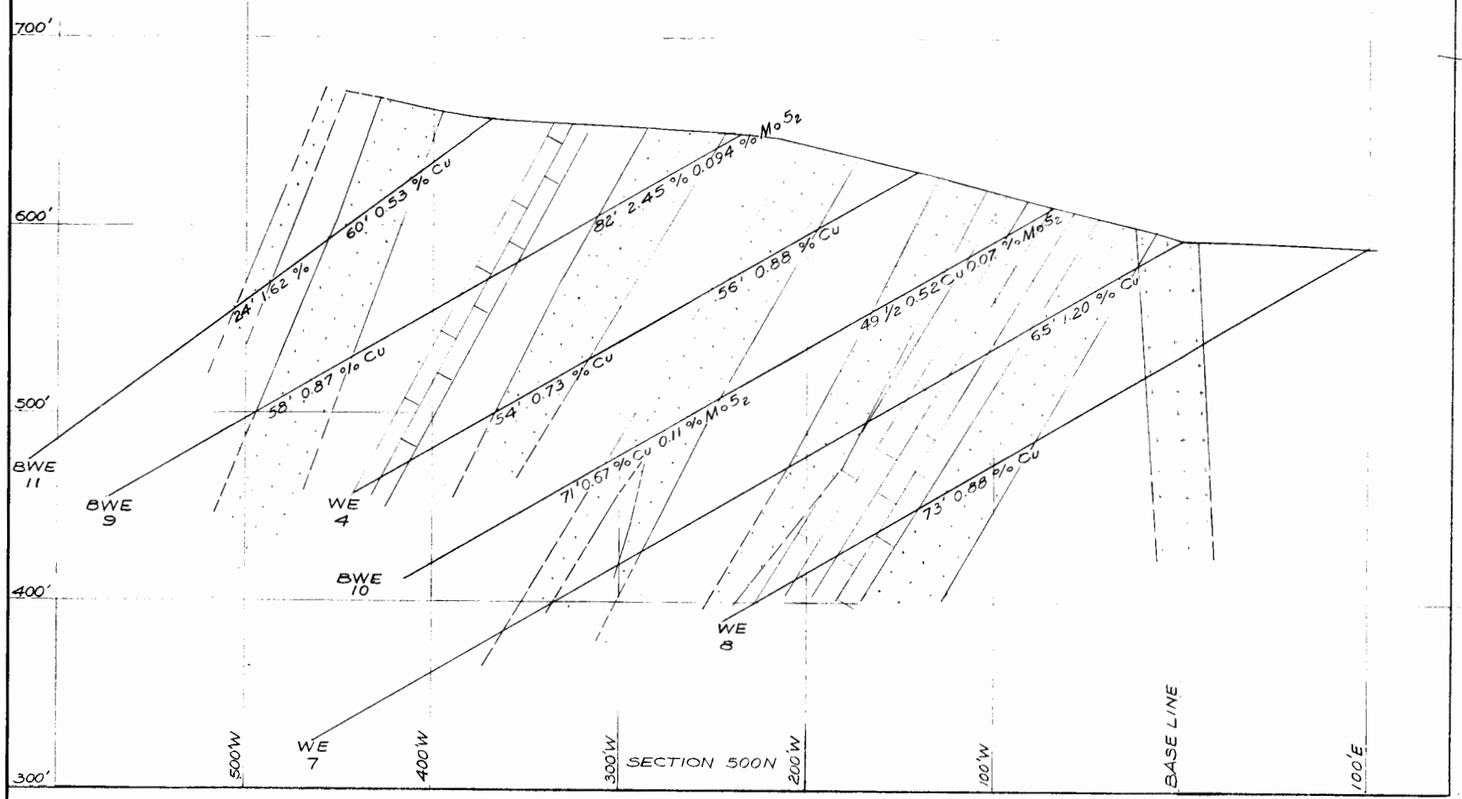
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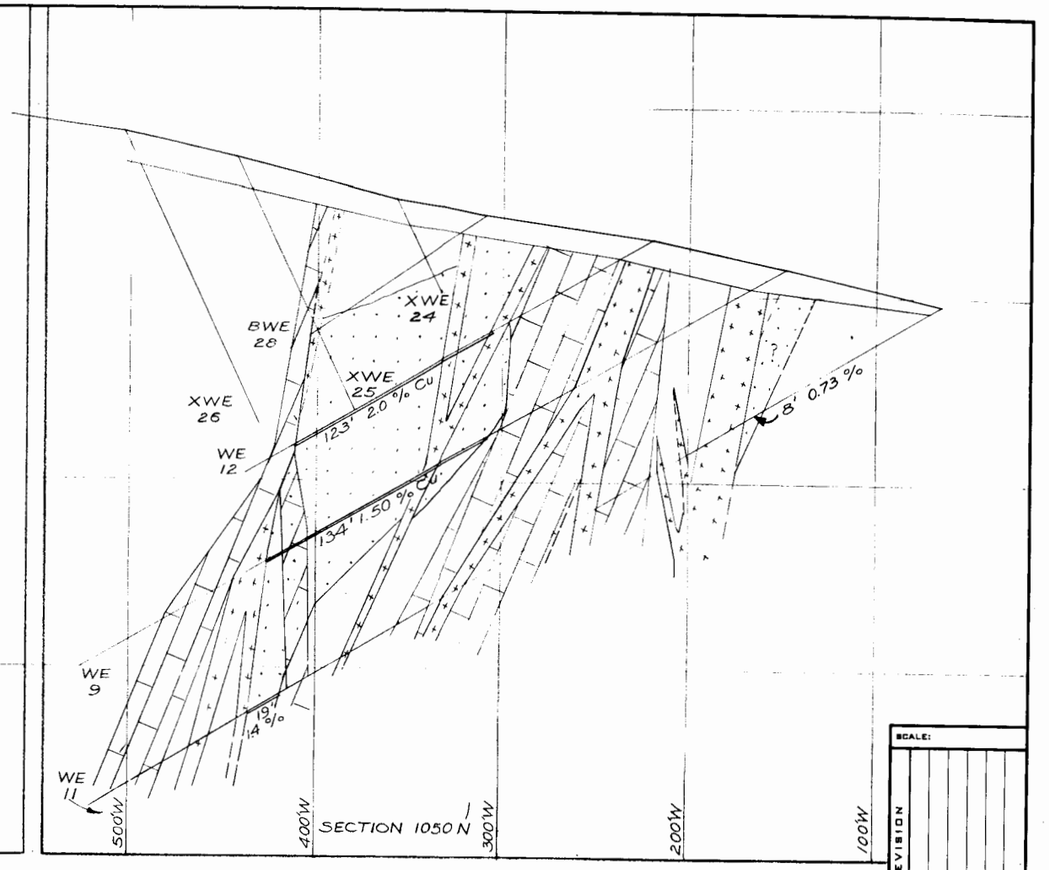
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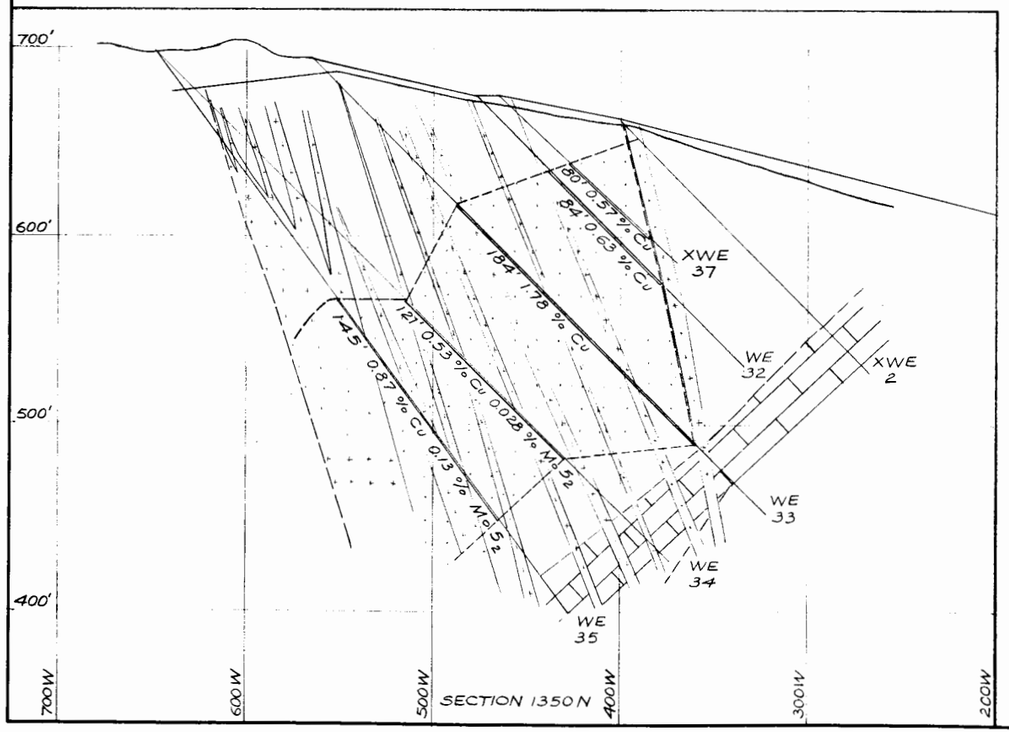
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							ISSUED FOR REPORT

DWG. NO. 402 021 015







FOR
DR. A.C. SKERL
SEPT 1965.

LEGEND

- LIMESTONE
- SKARN & QUARTZITE
- ACID DYKE
- ORE

DESCRIPTION OF REVISION	DESIGN	DRAWN	DES. CHECK	DRAFT CHECK	APPR.
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WAR EAGLE GEOLOGY SECTIONS

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 021 015

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DWG. NO. 402 021 016

700'

600'

500'

BASE LINE

SECTION 125+00 E

1+00 N

3+00 N

C.P.21

C.P.24

C.P.26

C.P.28

C.P.30

135' 0.50% Cu, 0.116% MoS2

130' 1.40% Cu, 1.14% MoS2

137' 0.83% Cu, 0.125% MoS2

126' 1.27% Cu, 0.196% MoS2

121' 0.61% Cu, 0.092% MoS2

SECTION 123

EASE LINE

1+00 N

2+00 N

3+00 N

136' 0.87% Cu, 0.098% MoS2

137' 0.53% Cu, 0.058% MoS2

145' 0.86% Cu, 0.045% MoS2

C.P.34

C.P.36

C.P.39

TO 500' C.P.41

C.P.33

200 N

100 N

BASE LINE

100 S

11,900 12,000 12,100 12,200 12,300 12,400 12,500 12,600 12,700 12,800

LEGEND.

- DIORITE
- ALTERED DIORITE
- LIMESTONE
- SKARN
- BASIC DYKE
- ORE

FOR DR. A.C. SKERL SEPT. 1965

REPORT.

ISSUED FOR

DESCRIPTION OF REVISION

SCALE:

APP. DATE | DESIGN | DRAWN | CHECK | DES. CHECK | FINAL CHECK | APPR.

REFERENCE | DWG. No.

GEOLOGY AT 650' ELEVATION
SECTIONS 123+00E & 125+00E
COWLEY PARK

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 021 016

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DWG. NO. 402 | 021 | 017

LEGEND

- GRANITE
- DIORITE
- ACID DYKE
- BASIC DYKE
- MINERALIZATION

800E

700E

600E

500E

500 CONTOUR AT BEDROCK

500 CONTOUR AT SURFACE

45°

FAULT

FAULT

VERT.

BASE LINE

200 S

300 S

400 S

SCALE:

DESCRIPTION OF REVISION	ISSUED FOR	APPR.	CHKD.	DESIGN	FINAL CHECK	DIM. CHECK	DRAWN	DESIGN	APP. DATE
	REPORT.								

FOR
DR. A.C. SKERL
SEPT. 1955

REFERENCE	DWG. No.

KEEWENAW
GEOLOGY AT 500 ELEVATION.

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

DWG. No. 402 | 021 | 017

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DWG. NO. 402 021 018

500'

400'

300'

200'

F

962

0.41% Cu

153' 1.21% Cu

77' 2.32% Cu

130' 1.86% Cu

96' 0.62% Cu

K30

K20

K4

K8

K19

K25

FAULT

BASE LINE

100'S

300'S

500'S

LEGEND

- GRANITE
- ACID DYKE
- BASIC DYKE
- MINERALIZATION
- ORE
- DIORITE

FOR
DR. A.C. SKERL
SEPT. 1965

SCALE:

DESCRIPTION OF REVISION	APPR.	DESIGN	DRAWN	CHKD.	DATE
ISSUED FOR REPORT.					

REFERENCE	DWG. No.

KEEWENAW - GEOLOGY
SECTION 650 E.

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

DWG. No. 402 021 018

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

SECTION II
MINING

DEVELOPMENT OF SITE, ROADWORK AND MINE PREPRODUCTION EXPENSES

The work estimated in this section includes the roadwork necessary to gain permanent access to the Little Chief mine site, the internal road system for haulage of ore and waste, the clearing and levelling of the mine sites, the removal of overburden and lake bottom silt prior to production.

The clearing and rough grading of the access roads must be completed prior to the commencement of pit preproduction work but it is proposed to gravel as few miles of roads as possible until the installation of the crushing plant provides a cheap source of crushed waste rock which can be used for this purpose. Wherever practical, it is proposed to use waste rock from the pits to build all roads on the property.

The exact sequence in which the clearing, roadwork and grading work should proceed will depend on the overall construction schedules. The costs have been estimated on the assumption that the company completes the work using its own mining equipment for a major part of the work outlined in this section.

During the first year of operation, it is proposed to mine only from the Little Chief ore zone. As a result, the initial preproduction preparation work is limited to the work required to bring this zone into production.

CAPITAL COST ESTIMATES

Preproduction - Little Chief Zone

Main ore and waste haulage roads, 50 feet wide, gravel surface	$ 14,000	
Access road, 30 feet wide, from millsite to Little Chief, gravel surface	3,000	
Preproduction Expenses - Mine		
Little Chief only		
Clearing 12 acres @ $200.00 per acre	2,400	
Draining lake and silt removal, 56,350 cu. yds. @ $0.80	45,080	
Overburden removal, 141,000 cu. yds. @ $0.50	70,500	
Waste rock removal, 275,000 cu. yds.	284,000	
Ore broken and stockpiled 70,000 tons	36,000	
Miscellaneous roads to powder magazines	4,500	
Preparation of coarse ore stockpile area	5,000	
Total Mine Preproduction Capital Cost		$ 464,480

In addition to the foregoing, the following work must be completed prior to commencement of production from the remaining pits. The expenditures required for this work are tabulated below in the year during which the work must be completed. The following costs do not include miscellaneous mine expense or general overhead as these items are adequately included elsewhere in the operating costs.

Second Year - Arctic Chief Zone

Road construction from millsite to Arctic Chief Zone, 3.54 miles	$ 71,540	
Clearing mine site 8 acres @ $200.00 per acre	1,600	
Overburden removal 132,800 cu. yds. @ $0.50	66,400	
Waste Rock removal 16,850 cu. yds.	23,780	
Total		$ 163,320
Contingency @ 10%		16,330
Total Second Year		$ 179,650

Third Year - War Eagle Zones

Road construction from Arctic Chief to War Eagle zones, 7.56 miles	$ 154,000	
McIntyre Creek bridge	18,000	
Clearing mine sites 10 acres @ $200.00 per acre	2,000	
Overburden removal 194,600 cu. yds. @ $0.50	97,300	
Waste rock removal 297,800 cu. yds.	265,600	
Total		$ 536,900
Contingency @ 10%		53,690
Total Third Year		$ 590,590

Fourth Year - Cowley Park Zone

New road construction from millsite to the Cowley Park zone, 9.02 miles	$ 197,700	
Wolfe Creek bridge	20,000	
Clearing mine site 10 acres @ $200.00 per acre	2,000	
Overburden removal 42,000 cu. yds. @ $0.50	21,000	
Waste rock removal 329,670 cu. yds.	298,400	
Total		$ 539,100
Contingency @ 10%		53,910
Total Fourth Year		$ 593,010

Fifth Year - Keewenaw Zone

New road construction to the Keewenaw zone, 0.23 miles	$ 5,100	
Clearing mine site 4.0 acres @ $200.00 per acre	800	
Overburden removal 39,200 cu. yds. @ $0.50	19,600	
River diversion 75,000 cu. yds.	67,900	
Total		$ 93,400
Contingency @ 10%		9,340
Total Fifth Year		$ 102,740

SUMMARY OF ORE RESERVES AND PIT DATA

Ore Zone	Little Chief	Arctic Chief	War Eagle	War Eagle South	Cowley Park	Keewenaw	Best Chance	Totals & Averages
Ore tons	1,343,900	377,300	812,750	188,890	1,193,440	433,200	169,280	4,518,760
Grade % Cu.	1.36	1.41	1.41	1.08	0.90	1.17	1.09	1.21
Dilution tons	174,710	43,450	113,780	26,440	71,610	51,980	22,000	503,970
grade % Cu	0.10	0.05	0.07	0.07	0.10	0.08	0.06	0.08
%	13.0	11.4	14.0	14.0	6.0	12.0	13.0	11.1
Total Ore tons	1,518,610	420,750	926,530	215,330	1,265,050	485,180	191,280	5,022,730
Grade % Cu.	1.21	1.27	1.24	0.96	0.86	1.05	0.98	1.10
% MoS_2					0.091			
Recoverable Gold & Silver $/ton	$1.07	$1.67	$0.73	$0.73	$0.20	$0.96	$0.26	$0.78
Waste rock cu. yds. Preproduction	275,000	16,850	233,800	64,000	329,670	75,000	50,000	1,044,320
After preproduction	1,674,800	164,610	616,550	254,530	449,750	200,090	119,250	3,479,580
Total waste rock cu. yds.	1,949,800	181,460	850,350	318,530	779,420	275,090	169,250	4,523,900
Waste/Ore Ratio cu.yds./ton	1.284	0.431	0.918	1.479	0.616	0.567	0.885	0.901
Overburden Cu. Yds.	141,000	132,800	145,700	48,900	42,000	39,200	61,900	611,500
Lake silt Cu. Yds.	56,350							56,350
Area to be cleared acres	12	8	12	3	10	4	4	53



MINING OPERATIONS

1. Pit Design

Studies have been made of each pit including such pertinent geological features as faults, types of rock, shape of ore bodies, etc. Such factors have been considered in preparing the various designs. This has resulted in different overall wall slopes, widths of berms and other design features so that all pits are not designed to a uniform set of conditions.

In general the rock conditions are very competent, which factor, coupled with the relatively small zones should permit mining at pit slopes steeper than normally considered in open pit operations.

A total of nine pits have been designed in varying degrees of detail. The Little Chief, War Eagle, Cowley Park and Keewenaw have been designed complete with berms, roads, etc. as these operations contain a major portion of the ore reserves. The other pits have been designed with overall slopes considered feasible for deposits of these types.

Additional design work may be successful in retrieving more ore but it is doubtful if the stripping ratios can be lowered significantly.

2. Production Requirements

During the preproduction period it will be necessary to mine 275,000 tons of waste rock to permit mining to continue at an adequate rate thereafter. It is estimated that prior to commencement of production 70,000 tons of ore will also be broken and available to supplement production during the first few months of operation.

Estimates have been based on the assumption that, during the first year of operation, 650,000 tons of ore will be milled and 720,000 annually thereafter. The following table schedules production from the various pits in accordance with the mining sequence previously outlined.

ORE AND WASTE PRODUCTION SCHEDULE

Year	Ore Zone	Ore Tons	Grade % Cu.	Grade MoS_2	Waste Rock Cu. Yds.	Waste/Ore Ratio Cu.Yds./Ton
Preproduction						
	Little Chief	70,000 *	1.25	-	275,000	3.929
1.	Little Chief	580,000	1.25	-	814,810	1.405
2.	Preproduction					
	Arctic Chief	-			16,850	-
	Little Chief	599,010	1.08	-	706,670	1.180
	Arctic Chief	120,990	1.27	-	76,480	0.632
	Totals & Averages	720,000	1.11	-	800,000	1.111
3.	Preproduction					
	War Eagle	-			233,800	-
	War Eagle South	-			64,000	-
	Little Chief	269,600	1.43	-	153,320	0.569
	Arctic Chief	299,760	1.27	-	88,130	0.294
	War Eagle	150,640	1.13	-	260,750	1.731
	Totals & Averages	720,000	1.30	-	800,000	1.111
4.	Preproduction					
	Cowley Park	-			329,670	-
	War Eagle	613,650	1.25	-	265,800	0.433
	War Eagle South	106,350	0.96		204,530	0.192
	Totals & Averages	720,000	1.20	-	800,000	1.111
5.	Preproduction					
	Keewenaw	-			75,000	-
	War Eagle	162,240	1.35	-	90,000	0.555
	War Eagle South	108,980	0.96	-	50,000	0.459
	Cowley Park	448,780	0.86	0.091	300,000	0.668
	Totals & Averages	720,000	0.99	0.057	515,000	0.715
6.	Preproduction					
	Best Chance	-			50,000	-
	Cowley Park	570,000	0.86	0.091	100,000	0.175
	Keewenaw	150,000	1.05	-	125,000	0.833
	Totals & Averages	720,000	0.90	0.072	275,000	0.382
7.	Cowley Park	246,270	0.86	0.091	49,750	0.202
	Keewenaw	335,180	1.05	-	75,090	0.224
	Best Chance	138,550	0.98	-	100,000	0.722
	Totals & Averages	720,000	0.97	0.031	224,840	0.312
Ore remaining after seventh year						
	Best Chance	52,730	0.98		19,250	0.365

* It is assumed the 70,000 tons of ore mined from the Little Chief during preproduction will be milled during the first year.

MINING OPERATIONS Cont'd.

3. Mining Methods

During the first year, mining must be concentrated in the north end of the Little Chief pit where the ore outcrops at the surface over a substantial area. At the south end the ore is overlain by a substantial thickness of waste rock and can only be mined after much of this material has been removed.

The stripping ratios allocated in this report are such as to maintain fairly uniform productions of ore and waste. The recommended schedule has the effect of extending production from the Little Chief zone into the third year as the ore at, and below, the 600 level can only be mined after practically all the waste rock has been removed. Unfortunately the highest grade ore lies in this lower region and can only be extracted during the last stages of production from this zone. The following data include recommendations and conditions applying to the proposed mining operation.

Production Data

Ore production - per year		720,000 tons
- per day		2,880 tons
- shifts per day		2
Working days per year		250
Working hours per week		40
Operation shifts worked	Per Day	
2 - primary drills	1.5	
2 - 2½ cu. yd. shovels	3	
Hauling	3	
General pit maintenance	1 & 2	
Other	1	

It is proposed to mine at 25 foot bench intervals leaving berms every third bench. Berm widths vary between 20 and 25 feet depending upon the competency of the rock.

Diesel powered 2½ cubic yard shovels, each operating on a three shift basis are proposed, each unit producing 2,000 tons per shift. Larger units would meet production requirements but because of the varying haul lengths and necessity of moving long distances, from pit to pit, it is recommended 2½ cubic yard units be used with the 35 ton hauling units. The following advantages favor the choice of the smaller shovel size.

(1) Lower initial capital expenditure
(2) Flexibility of operation
(3) Standardization - smaller inventory requirement
(4) Expense in moving larger than 2½ cubic yard shovel
(5) One small unit required in any case.

A rubber-mounted front-end loader is included to clean up scatter rock in the pits and waste dumps. This machine is much faster than a bulldozer for this work and is also capable of substituting for a shovel in the event of a prolonged breakdown. A spare loading unit is necessary as the shovels are programmed close to full capacity.

MINING OPERATIONS Cont'd.

Limited test work has indicated primary drilling might be done effectively with rotary machines. In the absence of conclusive data, however, these estimates are based on using "down-the-hole" percussion machines which will undoubtedly perform the work.

Two Drillmaster type rigs and one crawler mounted drill are required and will have ample capacity to maintain production. There will be numerous moves of equipment made during the mining operations and these machines have reasonable flexibility and can be moved with a low-boy trailer unit that can be rented locally.

Although 35 ton dump trucks are proposed throughout, consideration should be given to the acquisition of larger units considering the long ore hauls necessary after the first year of operations. Some reduction in the hauling costs can be expected with larger units. The units recommended are equipped with exhaust heated bodies.

MINE CAPITAL EXPENDITURES

The capital equipment required at the start of production is listed below. Additional equipment that will be needed is listed during the year in which the expenditure must be made.

Capital expenditures for shifters' offices, powder and fuse magazines are provided under the Ancillary Buildings section. Provision therein has been made for temporary buildings at each pit to serve as field offices, lunch and waiting rooms and for emergency repair work. An allowance has also been made in the section on Power for electrical installations to the Little Chief pit from the mill site. At the other pits it is proposed to provide a small diesel generator to supply lighting and power for miscellaneous uses that might be required.

Number Required	Description	Tons	Total
2	I.R. Drillmasters	39.0	$ 224,756
1	Crawler mounted drill complete	2.2	23,471
2	Jackleg rock drills complete	0.1	2,340
2	2.5 cubic yard shovels	145.0	253,352
1	Front end loader size #988	30.8	101,930
1	Size D-7-E bulldozer	25.5	57,015
5	35 ton trucks	128.5	439,260
1	600 cfm portable compressor	4.0	25,361
1	Road grader	12.5	28,866
1	Workmans bus	-	8,000
1	Service truck	-	7,500
4	Pick-up trucks	-	12,000
1	15 H.P. pump	0.1	1,235
	Small tools and miscellaneous	0.4	6,000
	Totals	388.1	$ 1,191,086

MINE CAPITAL EXPENDITURES Cont'd.

Number Required	Description	Tons	Total
	Brought forward	388.1	$ 1,191,086
	Freight 388.1 tons @ $80.00/ton		31,048
	Total F.O.B. Minesite		1,222,134
	Contingency and Engineering @ 5%		61,107
	TOTAL MINE EQUIPMENT - First Year		$ 1,283,241

ADDITIONAL EQUIPMENT REQUIRED

Number Required	Description	Tons	Total
	SECOND YEAR		
1	Service truck	-	$ 7,500
1	Pick-up truck	-	3,000
1	30 KW Diesel generator	1.5	5,947
1	set of 6 radio installations for trucks	0.25	4,595
	Totals	1.75	$ 21,042
	Freight 1.75 tons @ $80.00/ton		140
	Total F.O.B. Minesite		21,182
	Contingency and Engineering @ 5%		1,059
	TOTAL - SECOND YEAR		$ 22,241
	THIRD YEAR		
1	35 ton truck	25.7	$ 87,852
1	30 K.W. generator	1.5	5,947
	Totals	27.2	$ 93,799
	Freight 27.2 tons @ $80.00/ton		2,176
	Total F.O.B. Minesite		$ 95,975
	Contingency & Engineering @ 5%		4,799
	TOTAL THIRD YEAR		$ 100,774
	FOURTH YEAR		
3	35 ton trucks	77.1	$ 263,556
	Freight 77.1 tons @ $80.00/ton		6,168
	Total F.O.B. Minesite		$ 269,724
	Contingency & Engineering @ 5%		13,486
	TOTAL FOURTH YEAR		$ 283,210

MINE OPERATING COSTS

Following is a summary of the estimated mine operating costs.

SUMMARY OF ESTIMATED MINE OPERATING COSTS

Cost Per Ton Milled

Year	First	Second	Third	Fourth	Fifth	Sixth	Seventh
Primary Drilling	$0.140	$0.130	$0.102	$0.104	$0.100	$0.078	$0.077
Primary Blasting	0.323	0.292	0.220	0.220	0.208	0.151	0.150
Secondary & Misc. Breaking	0.074	0.067	0.067	0.067	0.067	0.067	0.067
Shovel Loading	0.298	0.275	0.203	0.195	0.187	0.132	0.132
Hauling	0.383	0.391	0.457	0.727	0.829	0.769	0.628
Mine Maintenance	0.110	0.100	0.100	0.100	0.100	0.100	0.100
Miscellaneous Mine Expense	0.126	0.153	0.168	0.168	0.168	0.168	0.168
Ore Stockpile to Crusher	0.062	0.056	0.056	0.056	0.056	0.056	0.056
Totals per Ton Milled	$1.516	$1.464	$1.373	$1.637	$1.715	$1.521	$1.378

Explanatory Notes:

1. These Costs do not include replacement of equipment nor depreciation allowances.

2. The first year is computed on the basis of milling 650,000 tons, subsequent years at 720,000 tons.

3. Costs for each year vary with the individual pits and their locations.

4. A contingency of 10% has been included on all costs to allow for remote location and severe winter conditions.

5. Costs have been based on present day prices for material and labour rates expected to apply in 1966. No allowance has been made for escalation in costs nor for slightly lower costs during the early years when the maintenance will be lower.

MINE OPERATING COSTS Cont'd.

PRIMARY DRILLING

Data

	Ore	Waste
Drill Pattern - Burden - feet	12	15
Spacing - feet	14	16
Height of Bench - feet	25	25
Sub-drilling in %	20	20
Net Cubic Yards per Hole	156	222
Drilling Speed - feet per hour	25	25

Year	Production Requirements Tons/Day		Drilling Requirements Hours/Day	Drill Shifts Required Per Day
	Ore	Waste		
1	2880	7985	25.8	3.2
2	2880	7840	25.5	3.2
3	2880	7840	25.5	3.2
4	2880	7840	25.5	3.2
5	2880	5047	19.3	2.2
6	2880	2695	14.2	1.8
7	2880	2203	12.9	1.6

Operating Costs

	Costs per Hour
Fuel 6 gal/hr. @ $0.27	$ 1.62
Lubricants	0.55
Maintenance	4.00
Bits and rods	5.50
Labour	3.95
	$15.62
Contingency @ 10%	1.56
Total	$17.18

MINE OPERATING COSTS Cont'd.

PRIMARY DRILLING Cont'd.

Operating Costs Cont'd.

Zone	Net tons per hour Ore	Cost per ton Ore	Cost per ton Waste
Little Chief	352	$0.049	$0.032
Arctic Chief	381	0.045	0.032
War Eagle	316	0.054	0.032
Cowley Park	324	0.053	0.032
Keewenaw	288	0.057	0.032
Best Chance	377	0.046	0.032

Summary of Primary Drilling Costs

Year	Cost per Year	Cost per ton milled
First	$ 91,160	$0.140
Second	93,800	0.130
Third	73,500	0.102
Fourth	75,090	0.104
Fifth	73,310	0.100
Sixth	56,080	0.078
Seventh	55,840	0.077

PRIMARY BLASTING

Data

	Ore	Waste
Pounds of explosive per net cu. yd.	1.0	1.0
Pounds of explosive per hole		
Hydromex	40	55
AN-FO	116	167
Pounds of explosive per hole blasted	156	222

MINE OPERATING COSTS Cont'd.

PRIMARY BLASTING Cont'd.

Operating Costs

	Ore	Waste
Materials per hole		
40 lbs. of Hydromex @ 28.89/cwt.	$11.56	
116 lbs. of AN-FO @ $0.089/lb.	10.32	
55 lbs. of Hydromex @ $28.89/cwt.		$15.89
167 lbs. of AN-FO @ $0.089/lb.		14.86
Primer, fuse, liner, cap and wire	6.08	6.08
	27.96	36.83
Contingency @ 10%	2.80	3.68
Total material per hole	$30.76	$40.51

Labour	
Cost per day	$ 84.48
Cost per ton broken	0.008

Zone	Net tons per hole		Cost per ton	
	Ore	Waste	Ore	Waste
Little Chief	441	544	$0.078	$0.082
Arctic Chief	478	544	0.072	0.082
War Eagle	396	544	0.086	0.082
Cowley Park	407	544	0.084	0.082
Keewenaw	362	544	0.093	0.082
Best Chance	472	544	0.073	0.082

Summary of Primary Blasting Costs

Year	Cost per year	Cost per ton milled
First	$209,930	$0.323
Second	210,230	0.292
Third	157,010	0.220
Fourth	156,930	0.220
Fifth	149,900	0.208
Sixth	108,420	0.151
Seventh	107,880	0.150

MINE OPERATING COSTS Cont'd.

SECONDARY & MISCELLANEOUS BREAKING

		Per Hour		Per Year
Materials				
Crawler mounted drill				
Rods, bits, hoses, etc.				
Cost per foot drilled	$0.20			
Cost at 25 feet per hour		$5.00		
Repairs		1.32	6.32	
Compressor operation				
Fuel - 6 gal./hr. @ $0.27		1.62		
Lubricant and repairs		.43	2.05	
Explosives				
Assume 0/75#/ton				
(includes fuse, caps & miscellaneous blasting)				
@ $0.34 per lb. for 94 lbs./hr.			23.97	
Total material for 1000 hrs./yr.			$32.34	$32,340
Labour				11,480
Total				$43,820
Contingency @ 10%				4,382
Total Secondary and Miscellaneous Breaking				$48,202
Cost per ton milled - first year				$0.074
- subsequent years				0.067

SHOVEL LOADING

Data

Two 2½ cubic yard diesel powered shovels	
Production per 8 hour shift	2,000 tons
Production per day on 2 shifts per machine	4,000 tons
Production per day on 3 shifts	5,400 tons
Total available capacity per day	10,800 tons
Total available capacity per year	2,700,000 tons

MINE OPERATING COSTS Cont'd.

SHOVEL LOADING Cont'd.

Operating Cost

	Per Hour	Per Year
Fuel 6.25 gallons per hour @ $0.27	$1.69	
Oil and grease	0.60	
Shovel teeth	1.13	
Wire rope and miscellaneous	2.31	
Maintenance	3.06	
	$8.79	
Contingency @ 10%	.88	
	$9.67	
Total for six shifts @ 6.67 hours per shift		$96,748
Labour		$95,687
Total Shovel Loading		$192,435
Cost per ton loaded		$0.075

Summary of Shovel Operating Costs

Year	Daily Production Required Tons		Daily Machine Shifts	Daily Shovel Capacity		Cost Per Ton Milled
	Ore	Waste		Ore	Waste	
First	2880	7985	6	2880	7920	$0.298
Second	2880	7840	6	2880	7920	0.275
Third	2880	7840	6	2880	7920	0.203
Fourth	2880	7840	6	2880	7920	0.195
Fifth	2880	5047	4	2880	5120	0.187
Sixth	2880	2695	* 2	2880	1120	0.132
Seventh	2880	2203	** 2	2880	1120	0.132

* 6th year - third shift required for 9.5 months
** 7th year - third shift required for 6.5 months

MINE OPERATING COSTS Cont'd.

HAULING

Data

First Year
- Recommend 35 ton rear dump trucks
- Average load - 35 tons
- Three shift operation to service shovel units

Number of trucks required	4
Spare truck required	1
Total required, years 1 & 2	5

- Additional requirements shown in table on page 2-10
- Truck requirements based on 50 minute hour and 85% truck/shovel efficiency
- For years 2 to 7, number of trucks include requirements for pre-production work but operating costs for this work are included under Preproduction Expenses

Operating Cost per Hour per Truck

	Per Hour
Materials & Supplies	
Fuel 8 gallons per hour @ $0.27	$2.16
Tire replacement	3.60
Tire repairs	0.54
Repairs materials	3.15
Lubricants	0.25
Total Materials	$9.70
Labour	$4.75
	$14.45
Contingency @ 10%	1.445
Total Hauling	$15.895

Summary of Hauling Operating Costs

Year	Truck Hours Per Year	Number of Trucks Required	Cost per Year	Cost/ton Milled
First	15,683	5	249,280	$0.383
Second	17,710	5	281,500	0.391
Third	20,699	6	329,010	0.457
Fourth	32,943	9	523,630	0.727
Fifth	37,589	9	597,470	0.829
Sixth	34,833	8	553,670	0.769
Seventh	28,450	7	452,210	0.628

MINE OPERATING COSTS Cont'd.

MINE MAINTENANCE

	Per Year	
Bulldozer		
Fuel and material @ $5.24 per hour for 1 shift operation @ 6.67 hours per shift	$ 8,736	
Labour	9,620	
Total Bulldozer		$ 18,356
Grader		
Fuel and material @ $2.57 per hour for 1 shift operation @ 6.67 hours per shift	$ 4,285	
Labour	6,500	
Total Grader		$ 10,785
Front End Loader		
Fuel and material @ $8.93 per hour for 1 shift operation @ 6.67 hours per shift	$ 14,891	
Labour	15,820	
Total Front End Loader		$ 30,711
Scaling and miscellaneous		$ 5,400
Total		$ 65,252
Contingency @ 10%		6,525
Total Mine Maintenance		$ 71,777
Cost per ton milled - first year		$0.110
- subsequent years		$0.100

MINE OPERATING COSTS Cont'd.

MISCELLANEOUS MINE EXPENSE

	Per Year	
Service Truck		
Fuel and materials	$ 3,035	
Labour	6,870	
Total Service Truck		$ 9,905
Pick-up Truck		
Fuel and materials	$ 1,784	
Labour	4,400	
Total Pick-up Truck		$ 6,184
Transportation - Men and Equipment		
Fuel and materials	$ 1,625	
Labour - included elsewhere	-	
Total Transportation		$ 1,625
Pumping and Miscellaneous		$ 2,500
Supervision		
Material	$ 2,500	
2 pick-up trucks	5,352	
Labour	37,200	
Total Supervision		$ 45,052
Miscellaneous Garage and Services		
Material	$ 4,400	
Labour	4,800	
Total Miscellaneous Garage and Services		$ 9,200
Total		$ 74,466
Contingency @ 10%		7,447
Total Miscellaneous Mine Expense		$ 81,913
Cost per ton milled - first year		$0.126
- second year		$0.153
- subsequent years		$0.168

POSSIBLE UNDERGROUND DEVELOPMENT

Little Chief Ore Zone

Deep drilling in the Little Chief Ore Zone on Sections 2+50S, 3+00S, and 4+00S indicate that the ore zone below the pit floor could be of a size and grade to consider an underground mining operation in this area. On Section 3+00S good values persist in diamond drill holes number 64 and 65 to 200 feet, or 400 feet below the pit floor.

In the geological section of this report, Dr. Skerl has illustrated the results obtained on the drawing the 3+00S section.

More diamond drilling will have to be done to delineate the ore and to determine tons and grade before serious consideration can be given to mining underground.

To serve as a rough guide for this possibility, the following estimate of the cost of equipment, development and preparation work to sustain an underground mining operation, is included. In addition an estimated mining cost, based on what could be a comparable situation, is shown.

The mining costs used are direct underground costs and include only the cost of placing the ore in the ore bin at the shaft. No overhead or administration costs are included.

Mining

Means of Entry

It is reasonable to consider sinking a shaft from the pit floor. The best ore is in Sections 3+00S and 3+50S and there is not enough room on the pit floor to sink too close to these two sections. The remaining pit floor climbs 50 feet to the 650 foot elevation. Future work may make this approach impracticable because of ore extensions, however for the time being, the shaft will be considered as located at 1+50S and 2+00E with the collar at 650 foot elevation.

Size of Shaft

As the economics of the underground mining will depend on the grade of ore and the tonnage hoisted, the shaft should be large enough to hoist 1000 to 1800 tons of ore per day and to service the mine. Minimum requirements would probably be a three compartment shaft with compartments 6 feet by 5 feet 6 inches. This would mean an overall size outside of timber of 20 feet 6 inches by 9 feet 6 inches.

Three levels would be driven at the 450 foot, 300 foot and 150 foot horizons. To allow for a loading pocket, spill pocket and sumps the minimum length of the shaft would be 675 feet.

POSSIBLE UNDERGROUND DEVELOPMENT Cont'd.

Mine Development

If Section 3+00S were taken as the center of the ore and the shaft is located at 1+50S, drifts along the ore zone would be 300 feet long. Cross cuts from the drifts to the ore zone would be probably 75 feet in length.

A study of Section 3+00S with the suggested width and the flat dip, suggests that the ore could be stoped transversely leaving a number of buttress pillars against the hanging wall - particularly from the 500 foot horizon down the dip.

Preparation

Scram drifts could be driven above the haulage levels and the ore scraped to one chute located near the footwall side. Mining would retreat from the hanging wall toward the footwall.

Sub drifts driven off raises at 45 feet above the level would serve as sill drifts and drilling could be done from this sub and from one placed 60 feet above the sill.

Stoping

Breaking could be done by long hole machines drilling from the sill elevation and the sub level 60 feet above. Stopes widths with two walls in ore, could be 50 feet and pillars between stopes 30 feet wide. Combined with the loss in sill pillars and scram drifts the overall extraction would not be more than 65% of the ore.

Because of the irregularities in the Little Chief Ore Zone shown to date, transverse stoping would retard dilution. The final decision as to choice of mining method will have to wait for the results of detailed drill assessment.

The cost of an underground mining plan such as the above would probably be as follows:

Summary of Pre-production Costs

Equipment	$807,153
Shaft including head frame, pockets, stations, etc.	303,750
Development	312,750
Preparation	639,750
Underground Excavations	65,572
Underground Installations	95,160
Miscellaneous Mine Expense	132,780
Services	276,336
Total	$2,633,251
Contingency @ 10%	263,325
Total all Pre-production Costs	$2,896,576
Estimated Mining Costs Per Ton	$2.70

DWG. NO. 402 022 001

2 +00 N | BASE LINE | 2 +00 S | 4 +00 S | 6 +00 S

A — 40' ROAD — A

B — ROCK SURFACE — B

20' BERMS

875 BERM

25' BERM

800 BERM

725 BERM

650 BERM

ROAD - 12% GRADE

TO CRUSHING PLANT

TO WASTE DUMP

50' ROAD

PIT FLOOR
EL. 650

PIT FLOOR
EL. 600

ROAD - 8% GRADE

4 +00 E

2 +00 E

BASE LINE

2 +00 W

825 | 800 | 775 | 750 | 725 | 700 | 675 | 650 | 625 | 600 | 850 | 875

50 FT. 0 50 FT. 100 FT.

ORIGINAL SCALE 1" = 40'

NO.	DESCRIPTION OF REVISION	APP'D
	ISSUED FOR REPORT.	

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REFERENCE | DWG. NO.

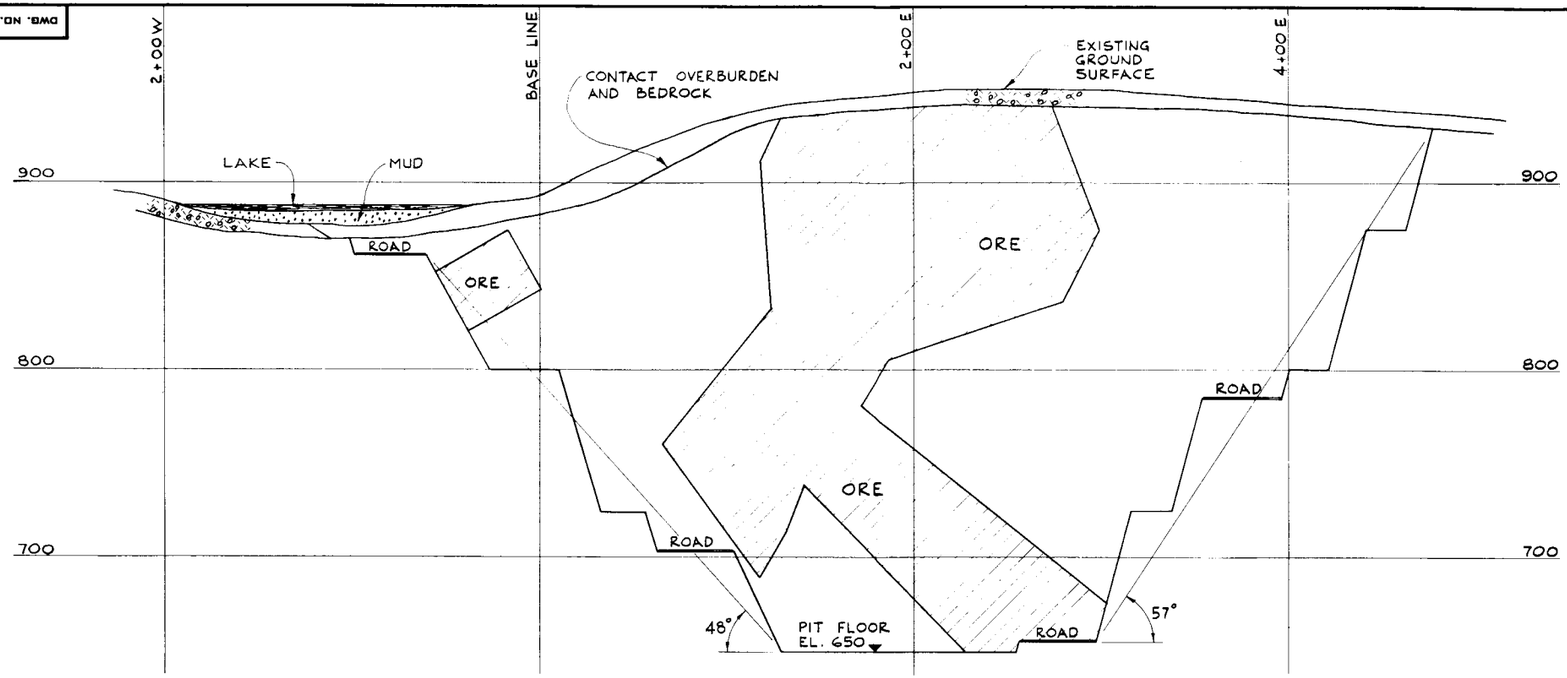
LITTLE CHIEF ZONE
OPEN PIT PLAN

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

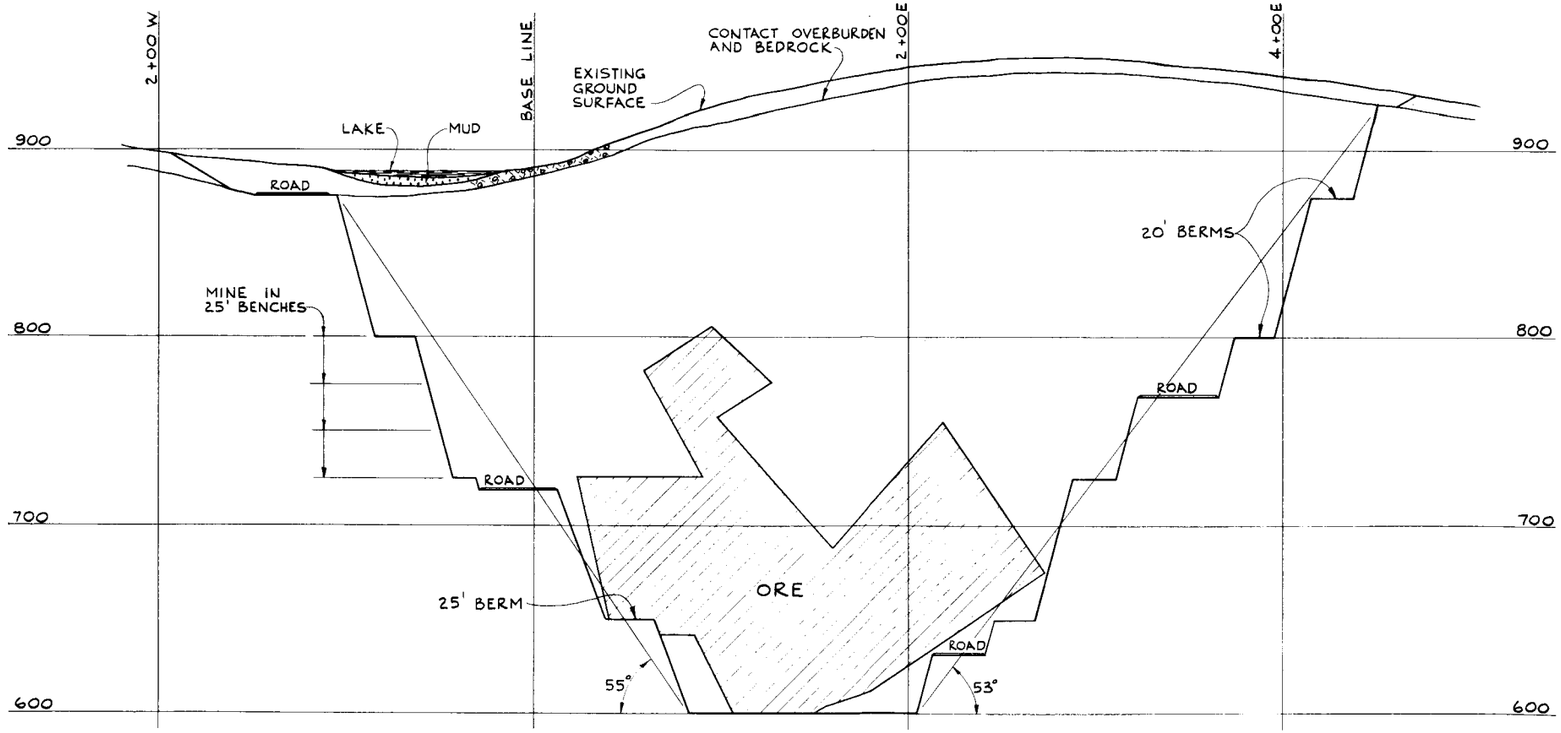
DWG. NO. 402 022 001

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DWG. NO. 402 022 002



SECTION A-A



SECTION B-B

ORIGINAL SCALE: 1" = 40'

NO.	DESCRIPTION OF REVISION	ISSUED FOR	APP'D	FINAL CHECK
		REPORT		

50FT. 0 50FT. 100FT

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REFERENCE | DWG. No.

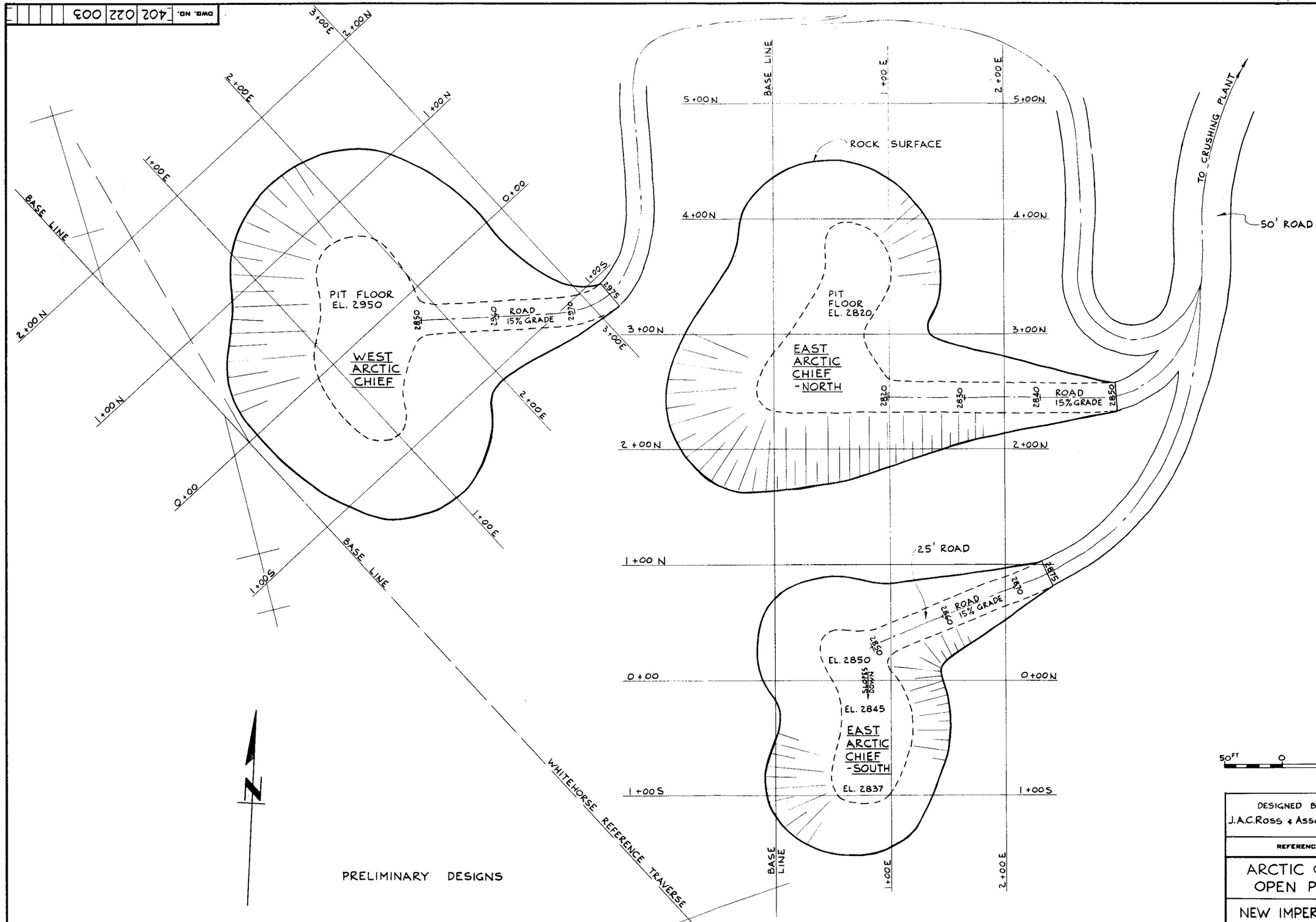
LITTLE CHIEF ZONE
OPEN PIT SECTIONS

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 022 002

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DWG. NO. 402 022 003



ORIGINAL SCALE: 1" = 40'

DESCRIPTION OF REVISION	DATE	DESIGN	DRAWN	CHKD.	APPD.
ISSUED FOR REPORT.	MAY 1965		DJH		

DESIGNED BY
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REFERENCE	DWG. No.

ARCTIC CHIEF ZONE OPEN PIT PLANS

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 022 003

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DWG. NO. 402 022 005

120+00E | 122+00E | 124+00E | 40' ROAD | 126+00E | ROCK SURFACE | 128+00E | 130+00E

4+00N

25' BERMS

750 | 675 | 650 | 625 | 600 | 575 | 550 | 725 | 700

PIT FLOOR EL. 600

PIT FLOOR EL. 550

675 BERM

750 BERM

2+00N

BASE LINE

2+00S

TO CRUSHING PLANT

50' ROAD

ROAD - 8% GRADE

760

50FT 0 50FT 100FT.

ORIGINAL SCALE: 1" = 40'

DESCRIPTION OF REVISION

ISSUED FOR REPORT

APPR. | FINAL CHECK | DESIGN CHECK | DRAWN | DETAIL CHECK | APPR. | DESIGN

DESIGNED BY
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REFERENCE | DWG. No.

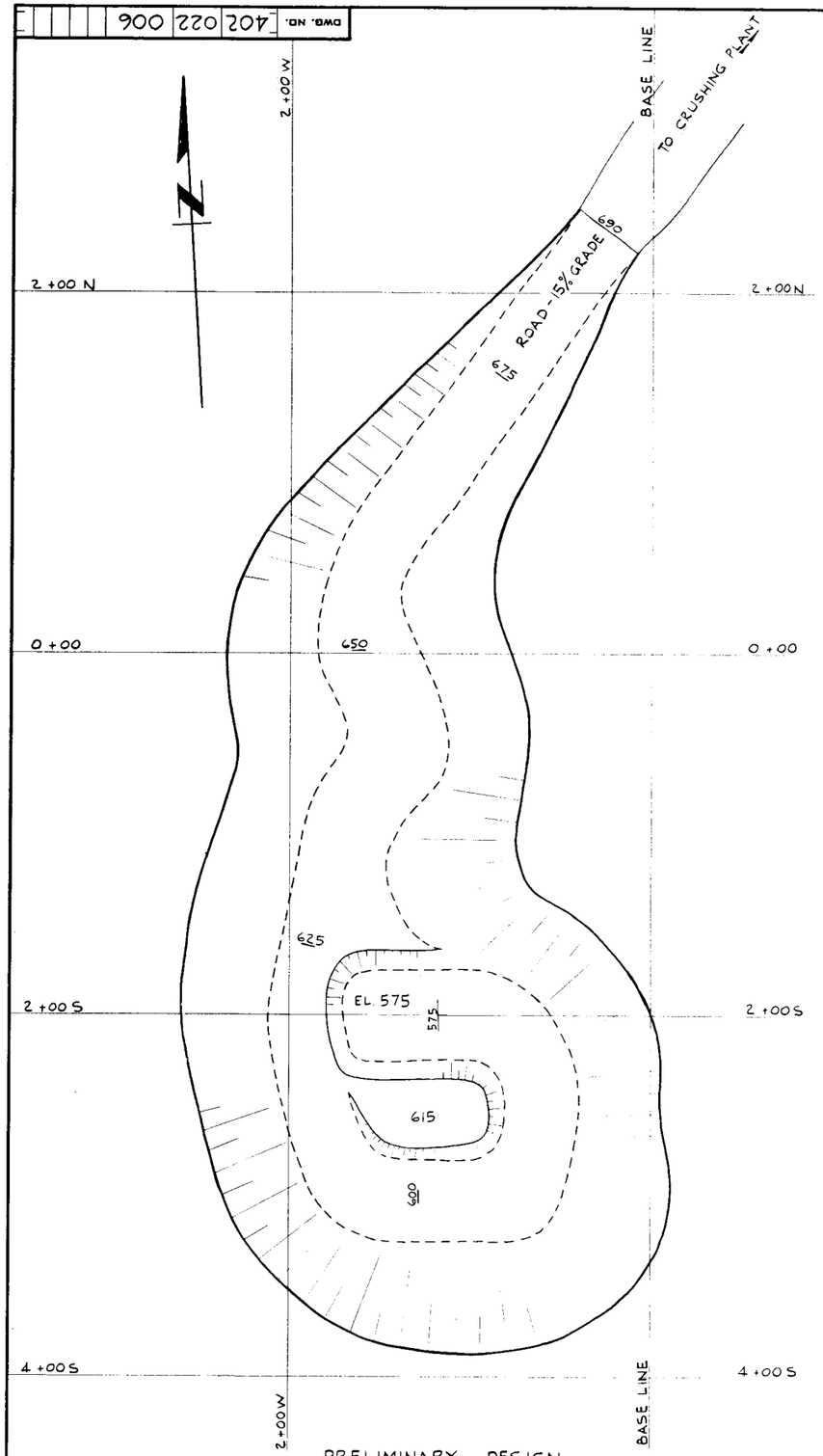
COWLEY PARK ZONE OPEN PIT PLAN

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 022 005

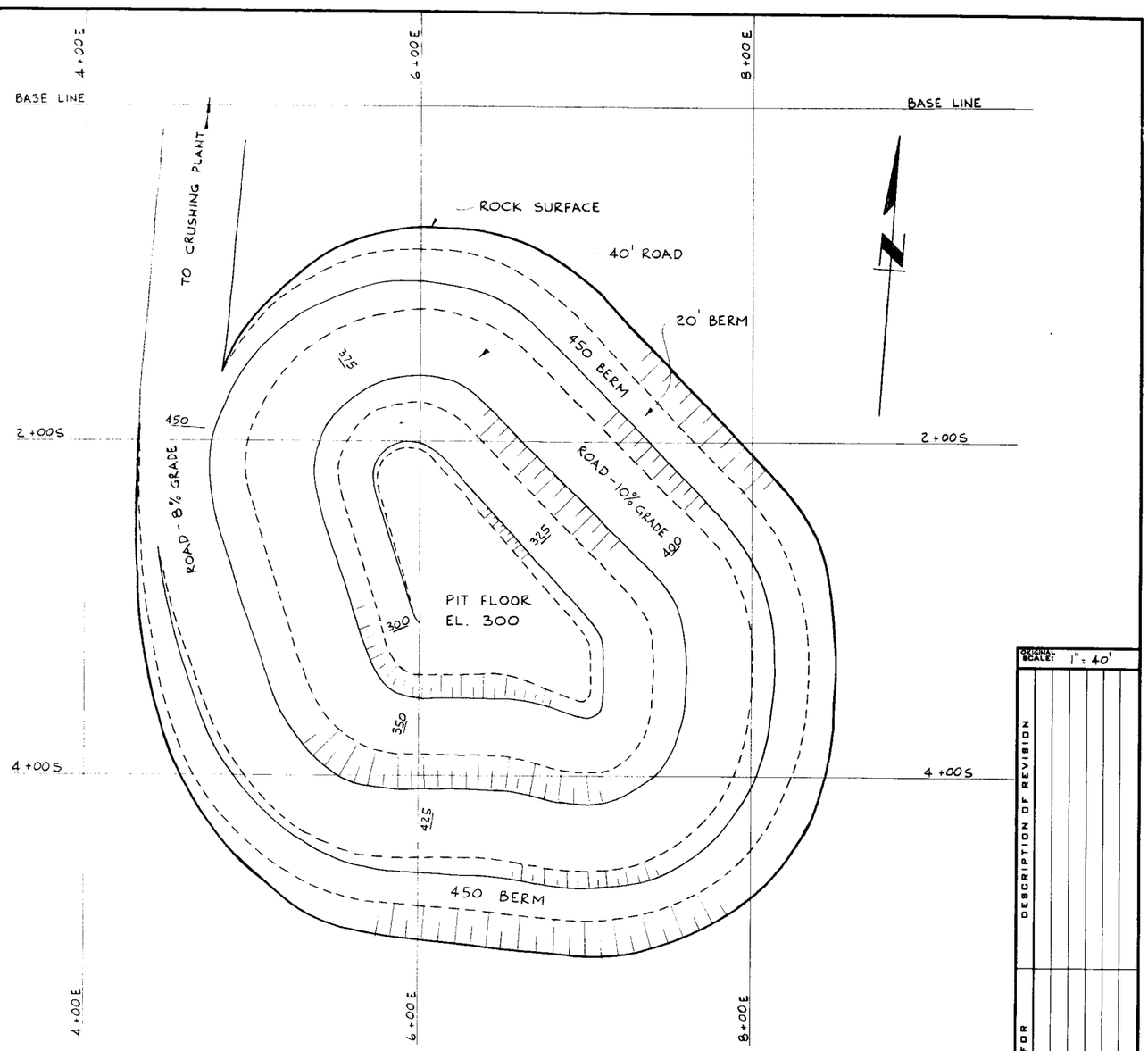
WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

DWG. NO. 402 022 006



PRELIMINARY DESIGN

BEST CHANCE



PRELIMINARY DESIGN

KEEWENAW

50FT. 0 50FT. 100FT.

ORIGINAL SCALE: 1" = 40'

DESCRIPTION OF REVISION	ISSUED FOR	APP.	FINAL CHECK	DESIGN CHECK	DIR. CHECK	DRAWN	DESIGN	APP. DATE	ENGR. DATE
	REPORT								

DESIGNED BY
J.A.C. ROSS & ASSOCIATES LTD.

REFERENCE | DWG. No.

BEST CHANCE AND KEEWENAW ZONES OPEN PIT PLANS

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON

DWG. No. 402 022 006

WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

SECTION III
METALLURGY

METALLURGY

Introduction

The metallurgical flowsheet was developed from testwork carried out on drill core composite samples from each of the six ore bodies. This testwork is reported from the laboratories concerned as follows:-

(1) Britton Research Laboratories, Vancouver.

(a) Concentration Tests on Samples of Copper-Iron Ore (Arctic Chief) (24/10/63).

(b) Tests on Composite Sample from Best Chance Ore body (2/1/64).

(c) Indicated Results from Treating Ore from the Little Chief Ore body (15/5/65).

(d) Progress Report No. 5, War Eagle (16/7/65).

(e) Progress Report No. 6, Keewenaw, (16/7/65).

(f) Progress Report No. 7, Cowley Park, (20/9/65).

(2) Sumitomo Metal Mining Co. Ltd., Tokyo.

(a) Testwork on Little Chief, Arctic Chief and Best Chance, (5/12/65).

(b) Testwork on War Eagle and Cowley Park, (5/12/64).

(c) Testwork on Keewenaw, (5/12/64).

(3) Department of Mines and Technical Surveys, Ottawa.

(a) Letter 26/5/65 - Little Chief Testwork.

(b) Letter 27/11/65- Little Chief Testwork.

The previous feasibility report was concerned with the recovery of copper, iron and associated gold and silver values from the Arctic Chief, Little Chief and Best Chance ore bodies. The terms of reference for the present report exclude the recovery of iron but the previous testwork for recovery of copper values is still valid. No testwork for iron recovery was carried out on the Cowley Park, Keewenaw and War Eagle ore bodies. Preliminary testwork has been performed on the concentration of molybdenite which is expected to be economically recoverable from the Cowley Park ore body.

Conclusions

The ore from the six deposits has similar grinding and metallurgical characteristics and the copper, gold and silver values can be readily concentrated by flotation of the copper sulphides. The grade of the copper concentrate produced is mainly dependent on the bornite content of the ore being treated but is expected to average about 35% copper at an average recovery of 85%. The grade will probably range between 28 and 45% copper at recoveries between 76% and 95%. Gold recoveries will range between 30 and 85% although this is difficult to predict from testwork due to the small concentrations present. Silver recoveries will range between 69 and 90%.

The copper is mineralized as the sulphides, chalcopyrite, bornite and chalcocite and major oxidation has occurred only in the Keewenaw ore body where chrysocolla and a little malachite account for 20% of the total copper. Attempts to float the oxides were unsuccessful.

Preliminary molybdenite flotation tests on the Cowley Park ore indicated that 65% of the MoS_2 can be recovered in a concentrate grading 90% MoS_2. Further testwork is required to develop a flowsheet.

Testwork indicates a power requirement for the primary grinding varying between 10.3 and 11.7 kwh/ton. Although the work index varies between 12.0 and 15.0, this variation is compensated by the differing mesh of grinds required (50 to 70% - 200 mesh). Variations in the work index of up to 1.0 kwh/ton were recorded from separate composite samples from the same ore body within the range 12.0 to 15.0, and it is thought that in actual plant practice adjustments to the mill operation, such as alteration of rod and ball load, will allow sufficient grinding control.

The proposed smelter schedule as expected, confirms the advantage of aiming for a high grade concentrate consistent with recovery. Testwork, particularly by Sumitomo, on the Little Chief ore body (Tests LCN4 and LCN5) indicates that regrinding of the copper rougher concentrates to 90% - 200 mesh in conjunction with a gangue depressant will increase the grade of the concentrates by up to 10% copper.

Testwork by Sumitomo and the Department of Mines checked the use of a gangue depressant particularly on the Little Chief ore where talc is present. Little success was reported with sodium silicate or Aero 610 but significant improvements in grade were recorded with the use of quartec. It is expected in plant practice that quartec additions will be confined to the cleaning sections due to the relatively high cost of the reagent.

The proposed method of treatment is as follows:-

(1) Crushing to 5/8"

(2) Rod milling to approximately 10 mesh.

(3) Ball milling to approximately 65% - 200 mesh.

(4) Flotation of a copper-gold-silver rougher concentrate.

(5) Flotation of a copper scavenger concentrate which will be recirculated to the primary ball mill.

(6) Regrinding and double cleaning of the copper-gold-silver rougher concentrate. (Flotation of a MoS_2 Concentrate.)

(7) Recirculation of the cleaner tailing to the primary ball mill.

(8) Recirculation of the recleaner tailing to the cleaner circuit.

(9) Thickening, filtering and drying of the copper-gold-silver concentrate.

Test Results

Metallurgical results from the laboratories concerned are summarized in Tables 1 and 2. Detailed flotation results from The Britton Research Laboratories on the three main ore bodies, Little Chief, War Eagle and Cowley Park, are shown in Tables 3, 4, and 5. Grinding characteristics are compared in Table 6.

TABLE I - Summary of Test Results - Copper.

ORE BODY	BRITTON RESEARCH					SUMITOMO					DEPT. OF MINES				
	GRADE % CU		CU REC %	SIZE FLOT FEED %-200	SIZE FINAL CONC. %-325	GRADE % CU		CU REC %	SIZE FLOT FEED %-200	SIZE FINAL CONC. %-325	GRADE % CU		CU REC %	SIZE FLOT FEED %-200	SIZE FINAL CONC. %-325
	FEED	CONC.				FEED	CONC.				FEED	CONC.			
Arctic Chief	0.88	28	83	70	-	0.79	36	89	60	-					
Best Chance	0.85	25	90	61	-	0.61	45	83	57	-					
Little Chief	1.20	25	83	65	70	1.37	45	80	62	98	1.29	30 to 34	86 to 88	75	-
War Eagle	1.25	35	94	50	-	N) 1.41 S) 0.78	42 35	95 81	70 68	- -					
Keewenaw	1.31	28	76	55	-	1.38	55	79	50	85					
Cowley Park	0.92	36	80	59	93	0.80	37	86	65	77					

TABLE 2 - Summary of Test Results - Associated Minerals

Ore Body	BRITTON RESEARCH						SUMITOMO						DEPT. OF MINES					
	Au		Ag		MoS_2		Au		Ag		MoS_2		Au		Ag		MoS_2	
	Feed Oz/Ton	Rec %	Feed Oz/Ton	Rec %	Feed %MoS_2	Rec %	Feed Oz/Ton	Rec %	Feed Oz/Ton	Rec %	Feed % MoS_2	Rec %	Feed Oz/Ton	Rec %	Feed Oz/Ton	Rec %	Feed MoS_2	Rec %
Arctic Chief	0.03	78	0.22	76	-	-	0.023	-	0.26	-	-	-	-	-	-	-	-	-
Best Chance	0.003	30	0.17	90	-	-	0.003	-	0.22	-	-	-	-	-	-	-	-	-
Little Chief	0.023	84	0.30	80	-	-	0.029	-	0.45	-	-	-	0.045	82 to 87	0.55	80 to 90	-	-
War Eagle	0.01	65	0.46	88	0.03	-	N) 0.006	-	0.51	-	0.005	-	-	-	-	-	-	-
							S) 0.006	-	0.23	-	0.041	-	-	-	-	-	-	-
Keewenaw	0.03	82	0.27	69	-	-	0.039	-	0.32	-	0.002	-	-	-	-	-	-	-
Cowley Park	0.004	55	0.15	70	0.11	65	0.003	-	0.12	-	0.097	-	-	-	-	-	-	-

TABLE 3 - BRITTON RESEARCH LABORATORIES

LITTLE CHIEF ORE (Second Composite)

Flotation Results
(Derived from Test L.C.7)

No.	Product	Weight %	Assays Au Oz/Ton	Assays Ag Oz/Ton	Assays Cu %	Distribution % Au	Distribution % Ag	Distribution % Cu
1	1st Cu Concentrate	4.13	0.38	5.90	25.28	75.3	71.5	77.2
2	2nd Cu Concentrate	0.60)	0.03	0.84	4.22)	2.2	3.7	1.8
3	2nd Cu Cleaner Tail	0.92)			1.60)			1.1
4	1st Cu Cleaner Tail	4.64	0.03	0.43	1.35	6.7	5.9	4.7
5	Scav. Cu Concentrate	3.51	0.02	0.36	1.11	3.4	3.7	2.8
6	Final Tails	86.20	0.003	0.06	0.19	12.4	15.2	12.4
7.	Head (calculated)	100.00	0.021	0.34	1.35	100.0	100.0	100.0
	Head (direct assays)		0.026	0.31	1.35			
	Calculated Results							
1 + 2	Combined Cu Concentrates	4.73	-	-	22.61	-	-	79.0
2 + 3	2nd Cu Conc. & 2nd Cleaner Tail	1.52	0.03	0.84	2.63	2.2	3.7	2.9
1 to 3	Cu Conc. after 1 cleaning	5.65	0.29	4.54	19.19	77.5	75.2	80.1
1 to 4	Cu rougher Concentrate	10.29	0.17	2.69	11.14	84.2	81.1	84.8
1 to 5	Cu rougher & Scav.Concentrate	13.80	0.13	2.09	8.59	87.6	84.8	87.6

TABLE 4 - BRITTON RESEARCH LABORATORIES

WAR EAGLE ORE

Flotation Results
Test W.E. 5

No.	Product	Weight %	Assays Oz. per Ton Au	Assays Oz. per Ton Ag	Assays % Cu	Assays % MoS_2	Distribution % Au	Distribution % Ag	Distribution % Cu	Distribution % MoS_2
1	Final Concentrate	3.16	0.20	11.90	35.91	0.27	46.2	83.7	91.5	47.9
2	2nd Cleaner Tailing	0.95)			2.91)				2.3)	
3	1st Cleaner Tailing	2.14)	0.12	0.70	0.82)	0.17	33.4	5.9	1.4)	36.4
4	Scavenger Concentrate	0.72)			1.20)				0.7)	
5	Scavenger Tailing	93.03	0.003	0.05	0.055	0.003	20.4	10.4	4.1	15.7
6	Head (calculated)	100.00	0.014	0.45	1.24	0.018	100.0	100.0	100.0	100.0
	Calculated Results									
1 + 2	Conc. after 1 cleaning	4.11			28.28				93.8	
1 to 3	Rougher Concentrate	6.25			18.88				95.2	
1 to 4	Rougher & scav. concs.	6.97	0.16	5.80	17.05	0.22	79.6	89.6	95.9	84.3
2 to 4	Middlings	3.81	0.12	0.70	1.41	0.17	33.4	5.9	4.4	36.4

Additional assays on final concentrate: Fe 19.58%, SiO_2 9.72%, CaO 3.77%, MgO 1.23%, Al_2O_3 1.70%, S 27.32%.

TABLE 5 - BRITTON RESEARCH LABORATORIES

COWLEY PARK ORE

Flotation Results
Test C.P. 3

No.	Product	Weight		Assays				Distribution %			
		Grams	%	Oz. per Ton Au	Oz. per Ton Ag	% Cu	% MoS_2	Au	Ag	Cu	MoS_2
1	MoS_2 Concentrate	1.46	0.037			0.47%	95.53			-	30.4
2	" Middling	1.18	0.030)			2.84%	81.45)			0.1	21.1
3	2nd Cu Concentrate	4.09	0.102)	0.10	4.9	25.93%	13.03)	51.7	67.7	3.0	11.4
4	1st Cu Concentrate	78.6	1.967			37.59	0.48			80.5	8.2
5	3rd Cleaner Tailing	13.2	0.330)			6.00%	0.85)			2.1	2.4
6	2nd Cleaner Tailing	24.8	0.621)			2.57%	0.35)			1.7	1.8
7	1st Cleaner Tailing	230.0	5.756)	0.002*	0.05*	0.86%	0.07)	48.3*	32.3*	5.4	3.5
8	Rougher tailing	3642.6	91.157)			0.07%	0.027)			6.9	21.2
9	Cu in solution	-	-			0.12%	-			0.3	-
10	Head (calculated)	3995.93	100.000			0.92%	0.116	100.0	100.0	100.0	100.0

*Including MoS_2 Concentrate

Calculated Results

1 + 2	MoS_2 Conc. & Middling	2.64	0.067			1.55	89.24			0.1	51.5
3 + 4	Combined Cu Concs.	82.69	2.069			37.02	1.10			83.5	19.6
1 to 7 + 9	Rougher Concentrate	353.33	8.843			9.67	1.04			93.1	78.8

TABLE 6

Comparison of Grinding Characteristics and Power Requirements

	Arctic Chief	Best Chance	Little Chief	War Eagle	Keewenaw	Cowley Park
M.O.G. % - 200 Mesh	70	61	65	50	55	59
Rod Mill						
Wi	12.1	13.4	12.0	15.0	14.0	13.0
Fu	9,400	9,400	9,400	9,400	9,400	9,400
Pu	455	455	455	455	455	455
W Kwh/Ton	4.43	4.90	4.39	5.49	5.12	4.75
HP for 2000 TPD	494.5	547.7	490.4	613.1	572.2	531.3
Mill Size	10' x 12'	10' x 12'	10' x 12'	10' x 14'	10' x 14'	10' x 12'
Ball Mill						
Wi (allow 1.25 increase)	13.35	14.65	13.25	16.25	15.25	14.25
Fu	455	455	455	455	455	455
Pu	100	115	100	170	140	120
W Kwh/Ton	7.09	6.80	7.03	4.84	5.73	6.32
HP for 2000 TPD	792.3	759.5	786.4	541.2	640.3	706.3
Mill Size	10' x 14'	10' x 14'	10' x 14'	10' x 10'	10' x 10'	10' x 10'
Total HP	1286.8	1307.2	1276.8	1154.3	1212.5	1237.6
Total W Kwh/Ton	11.52	11.70	11.42	10.33	10.85	11.07
TPD to M.O.G. on L.C. HP	1983	1952	2000	2211	2105	2063

Design Data

Most of the design data are based on results of testwork from the Little Chief ore body. This is the largest ore body of the six and will be the first pit mined. Grinding power requirements are close to the weighted average for the six ore bodies and flotation conditions require the maximum cell volume.

Assumed Assay of Ore

Copper	1.2%
Gold	0.023 oz/ton
Silver	0.3 oz/ton

Estimated Grinding Power

Primary rod and ball milling (Feed size 5/8 in.)	11.4
Copper concentrate regrinding	1.5
Total	12.9 KWH/ton

Work index of ore: - Wi = 12 KWH/ton

Specific gravity of ore: - 3.4

Screen Analyses (Percent passing)

Product	Mesh Size (Tyler) 28	48	100	200	325
Primary ball mill feed	48%	32%	18%	10%	6%
Primary flotation feed	100%	100%	94%	65%	45%
Final copper concentrate	100%	100%	100%	95%	80%

80% Passing Sizes	Microns
Primary ball mill feed	455
Primary flotation feed	100
Final copper concentrate	40

Weights of Concentrates (Percent of mill feed)

Rougher concentrate	10%
Cleaner concentrate	6%
Final concentrate	4%
Scavenger concentrate	4½%
Cleaner tails	6%
Recleaner tails	1½%

Assay of Concentrates - (From Britton Research Laboratories Test L.C. 7)

Chemical Analysis

Copper (Cu)	25.%
Gold (Au)	0.46 oz/ton
Silver (Ag)	5.9 oz/ton
Iron (total) (Fe)	12.%
Iron (acid sol.) (Fe)	11.%
Sulphur (S)	12.%
Silica (SiO_2)	22.%
Moisture (H_2O)	12.%

Assay of Concentrates - Cont'd.

Spectrographic Analysis

Alumina (Al_2O_3)	0.5%
Antimony (Sb)	Nil
Arsenic (As)	Nil
Barium (Ba)	Trace
Beryllium (Be)	Nil
Bismuth (Bi)	Nil
Boron (B)	0.002%
Cadmium (Cd)	Nil
Calciumoxide (CaO)	5%
Chromium (Cr)	Trace
Cobalt (Co)	0.003%
Gallium (Ga)	Nil
Lead (Pb)	0.04%
Magnesiumoxide (MgO)	15%
Manganese (Mn)	0.1%
Molybdenite (MoS_2)	0.03%
Niobium (Nb)	Nil
Nickel (Ni)	0.01%
Strontium (Sr)	Trace
Tantalum (Ta)	Nil
Tin (Sn)	Trace
Titaniumoxide (TiO_2)	0.3%
Tungsten (W)	Nil
Vanadium (V)	0.02%
Zinc (Zn)	0.1%
Zirconium (Zr)	Nil

Conditioning and Flotation Contact Periods and Percent Solids

	Retention Time Minutes	Percent Solids (approx.)
Conditioner (before rougher flotation)	5	33
Rougher flotation	10	33
Scavenger flotation	4	32
Cleaner flotation	6	11
Recleaner flotation	4	8

Assumed Recoveries

Copper	83.3%
Gold	80%
Silver	78.7%

Reagent Consumption

Potassium amyl xanthate	0.18 Lbs./Ton
Pine oil	0.16 Lbs./Ton
Quartec	0.05 Lbs./Ton

In practice, it is probable that less reagents will be required.

Grinding and Flotation Circuits
Metallurgical and Water Balances

The determination of grinding power requirements is shown in Table 7. A work index of 19 and a product size of 40 microns has been assumed for the regrinding of the rougher concentrates. Testwork indicates that the Little Chief ore will require the greatest amount of regrinding due to the close mineral association with magnetite and gangue and it is probable that power requirements will be reduced on the other ores.

The determination of pulp volumes and flotation cell sizes are shown in Table 8 and the metallurgical balance in Table 9. The water balance is shown in Table 10.

TABLE 7

Grinding Data and Power

	80% passing Size Microns		Approx. Work Index (Rel.)	Work Input Kwh/Ton of		Mill Size		% of Critical
	Feed	Product	Kwh/Ton	Mill Feed	of 2000 TPD	Ft	Motor HP	Speed
Rod Mill	9,400	455	12.0	4.4	4.4	10 x 12	500	65
Primary Ball Mill	455	100	13.25	7.0	7.0	10 x 14	800	65
Regrind Ball Mill	140	40	19.0	13.9	1.5	7 x 7	200	
Total					12.9		1,500	

TABLE 8

Flotation Circuit

	Feed					
	Dry Solids TPD	Pulp I GPM	% Solids	Contact Time Min.	Pulp Volume Cu.Ft.	Cells installed No. and Cu. ft.
Rougher Flotation	2210	703	33.3	10	1127	24-50 cu.ft.
Scavenger Flotation	1990	646	32.7	4	416	8-50 cu.ft.
Cleaner Flotation	250	290	11.0	6	280	12-24 cu.ft.
Recleaner Flotation	130	151	11.0	4	97	4-24 cu.ft.
Total cells installed			32 cells, 50 cu.ft. each 16 cells, 24 cu.ft. each			

TABLE 9

Assumed Metallurgy in the Flotation Circuit

(From Britton Research Laboratories Test L.C. 7)

	Weight %	Cu %	Au Oz/Ton	Ag Oz/Ton	Distribution % Cu	Au	Ag
Head Feed	100.0	1.20	0.023	0.30	100.0	100.0	100.0
Rougher Concs.	10.0	10.75	0.20	2.55	89.5	87	85
Final Concs.	4.0	25.00	0.46	5.90	83.3	80	78.7
Final Tails	96.0	0.21	0.005	0.07	16.7	20	21.3

TABLE 10

Total Water Introduced into the Mill Circuit

Point of Addition	I GPM
Rod Mill	119.0
Primary Ball Mill	217.3
Regrind Mill	25.4
Cleaner Cells	80.8
Recleaner Cells	123.8
Total	566.3

Total Water Losses without Water Reclamation

Final Copper concentrate	17.0
Final Tails	549.3
	566.3

Molybdenite Flotation

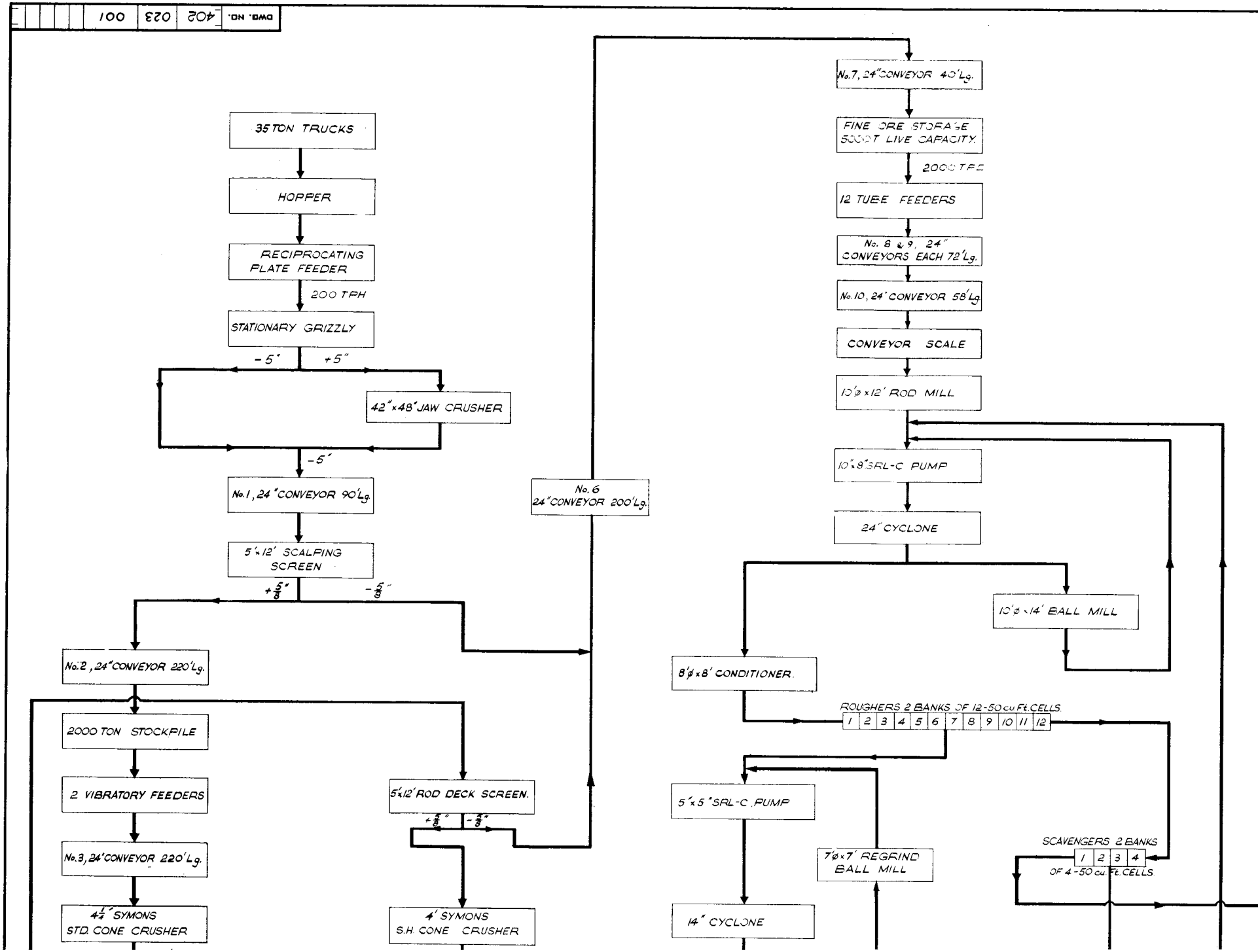
Preliminary tests have indicated that a 90% MoS_2 concentrate containing less than 1% copper can be obtained at a 65% recovery, but further testwork is required to determine optimum conditions and reduce reagent consumption. Detailed results are as follows:

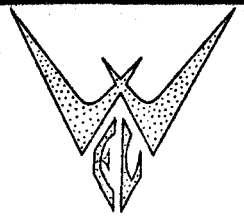
MoS_2 Concentrate

Test	Wt. %	Assay % MoS_2	Assay % Cu	Rec %	%-325 Mesh	Reagent Addns. Lbs./Ton orig.ore NaCN	NaHS	K4(Fe(CN6)	NaCLO
Britton									
CP1	0.079	89.6	0.92	65.2	48	0.60			
CP2	0.090	94.9	0.47	77.7	94	0.95			
CP3)	0.037	95.5	0.47	30.4	93	0.20	0.10		
)Midd.	0.030	81.4	2.84	21.1					
Sumitomo									
	0.06	96.2	0.20	47.6	77	0.69		0.03	0.04
(Midd.)	0.11	29.5	15.60	26.8					

It is not expected that regrinding of the copper cleaner concentrates will be required and a saleable grade concentrate will probably be produced by flotation of a rougher molybdenite concentrate followed by four to five stages of cleaning.

DWG. NO. 402 023 001





SECTION IV
SITE DEVELOPMENT, ROADS AND CONCENTRATE MOVEMENT

SITE DEVELOPMENT

All clearing, grubbing and rough excavation costs are included in Section VII except for the pits which are included in Section II.

ROADS

Mine road costs are included in Section II. The roads in the millsite complex are included in Section VII. The 2-1/4 mile access road from MacRae to millsite is costed in this section.

CONCENTRATE MOVEMENT

It is expected that the Whitepass and Yukon Route will transport the copper concentrate from the concentrator to shipboard in Skagway. A preliminary survey by the Whitepass and Yukon Route, reported in a letter to New Imperial Mines Limited, dated 13th October, 1965, estimates the freight rate at $11.85 per short ton. Although this rate is not firm, it is used in this report. The freight rate includes truck transportation in containers from the concentrator to the rail siding at McRae, rail transportation to Skagway, handling, storage at Skagway and loading to ship.

Of the two sizes of collapsible containers available it is proposed to use the smaller size (12' x 4' x 3') for ease of handling. This container has a maximum capacity of 10 tons. It may be necessary to line the containers with paper to prevent the concentrates sticking under freezing conditions. It may also be necessary to cover the containers with plastic sheeting to prevent wind losses in transportation. No allowance has been made for the lining and sheeting in the operating cost.

The preliminary survey does not indicate where the concentrates are to be weighed but it is presumed that the Whitepass and Yukon Route will provide such a facility.

A preliminary concentrate production rate of 80 tons per day was reported to the Whitepass and Yukon Route for estimating purposes. It is now apparent that the average rate will be closer to 60 tons per day and will range between 43 and 73 tons per day at an assumed concentrate grade of 35% copper.

Construction & Installation	Quantity	@$	Cost $
Access Road MacRae to Millsite	2-1/4 miles	$24,000	$54,000

SECTION V
ANCILLARY BUILDINGS

ANCILLARY BUILDINGS

General

General comments regarding Ancillary Buildings made in the initial report still stand. However there has been a steep upward trend in tendered prices lately which cannot be justified solely by known increases in labour rates and prices of materials. The buildings therefore have been re-estimated at current unit prices for the same class of work in districts of similar location.

The mill site is close enough to Whitehorse that no provision need be made for housing at the site.

The repair shops of the White Pass and Yukon Route in Whitehorse may be relied on as a base for major equipment repairs. For this reason, an elaborately equipped machine shop is not considered necessary.

The few necessary buildings have been designed with a maximum economy of space and material in mind.

Office and Warm Warehouse

This building is designed as a conventional timber framed structure on concrete footings with a concrete slab forming the warehouse floor. The multiplicity of partitions renders a standard steel building unsuitable for this purpose. The offices are placed on the upper floor which enables a comprehensive view of the whole plant to be obtained from the Manager's office. The vault is placed over the furnace room and is accessible to both engineering and office staff. The furnace room and vault enclosure is constructed of concrete blocks. In the event of fire destroying the building, the vault should remain standing with records remaining intact. Combining the warehouse and office accommodation allows maximum economy in excavation and footings.

Mine Office and Dry

The mine office and dry is located at the plant site where it shares water, sewer, lighting and fire protection services with other plant buildings. A conventional timber framed building with pitched roof and concrete footings is recommended. A total of 60 lockers has been provided to allow service and surface personnel as well as the mine personnel to use the changehouse facilities.

Machine Shop and Cold Warehouse

The increased number of vehicles proposed in the present report has required an increase in the size of the maintenance area. A standard steel building with an overhead crane 40 ft. wide, similar to the previous scheme, has been proposed. An additional bay, approximately 25 ft. wide for the length of the building, has been added and the tool room, electrical shop, office and washroom have been relocated in this area, leaving the area travelled by the crane entirely clear.

The building is fully insulated and has an inner lining of galvanized sheet up to 8 feet for protection of the vapour barrier and insulation material. Heating is by unit heaters or infra red heaters. The crane hoist would be hand operated but the beam travel would be powered by electric motor. Washroom facilities are included within the building to provide for inclement weather and for time saving. The roof slope is extended on one side to provide cold warehousing requirements and a wire mesh enclosure is all that is required. Any racking required in this area should be provided by staff on site and no allowance has been made for this. A separate enclosed oil storage has been added under this roof slope.

Pit Shack, Pit Garages, Powder Magazine & Cap & Fuse Shed

A pit shack 12 ft. x 20 ft. mounted on skids provides a lunch room, one end of which is partitioned off for a washroom and toilet facilities. This can be moved from pit to pit as necessary. Four pit garages each 20 ft. wide, 30 ft. long and 14 ft. high are provided.

A powder magazine and cap and fuse shed complete with the necessary earthworks as illustrated are provided but the core shed and nitrate storage shed are omitted from the present scheme.

Sewage Disposal

Washroom facilities in both the general plant area and the pit area require two separate sewage systems. The plant site sewage system would consist of a septic tank and a leaching cesspit. At the pit shack, a simple septic tank discharging to surface drainage should suffice because this building will be relocated after three years.

SUMMARY OF CAPITAL ESTIMATE

	Building Construction	Equipment	Freight	Total
(a) Office & Warm Warehouse	$ 43,200.00	$10,000.00	$ 4,000.00	$ 57,200.00
(b) Maintenance Building & Open Storage	60,000.00	22,000.00	4,000.00	86,000.00
(c) Mine Office & Dry	30,500.00	6,500.00	5,200.00	42,200.00
(d) Miscellaneous Small Buildings, Sewage, Etc.	40,000.00	-	2,000.00	42,000.00
	$173,700.00	$38,500.00	$15,200.00	$227,400.00

ANCILLARY BUILDINGS

Office & Warm Warehouse - 2,700 sq.ft.

Building at$16.00 per sq.ft. including heating & lighting.	=	$ 43,200.00	$ 43,200.00

Equipment

Desks & Chairs		
Drafting Tables		
Typewriters & Machines		
Filing Cabinets	6,300.00	
Warehouse racks etc.	3,700.00	
		10,000.00

Freight

50 Tons @ $80.00	4,000.00	4,000.00
Total To Summary		$ 57,200.00

MAINTENANCE BUILDING & OPEN STORAGE

Prefab. Building - Delivered & Erected	$ 32,000.00	
Foundations, Partitions, Services, Etc.	$ 28,000.00	
		$ 60,000.00

Equipment

Office	700.00	
5 Ton Crane	13,000.00	
1 Electric Welder		
1 Set Acelylene torches		
1 Bench Table		
1 Bench Hacksaw		
1 Set pipe dies		
Small Tools	8,300.00	
		22,000.00

Freight

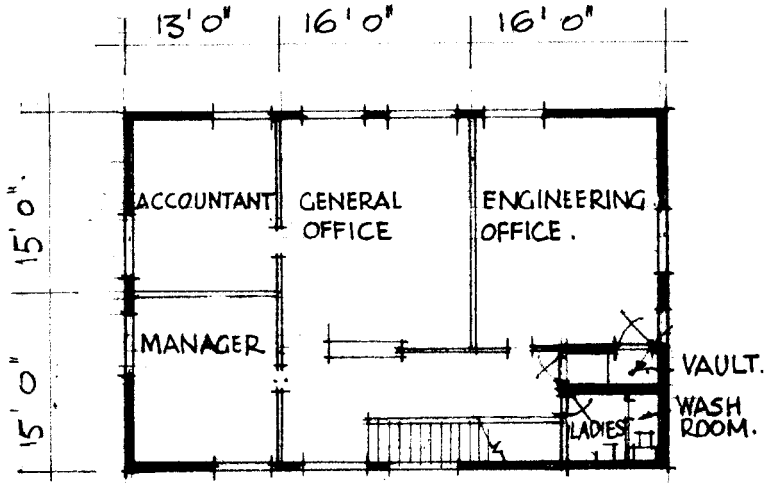
50 Tons @ $80.00	4,000.00	4,000.00
Total To Summary		$ 86,000.00

MINE OFFICE, SMALL PARTS STORE & DRY - 1,680 sq.ft.

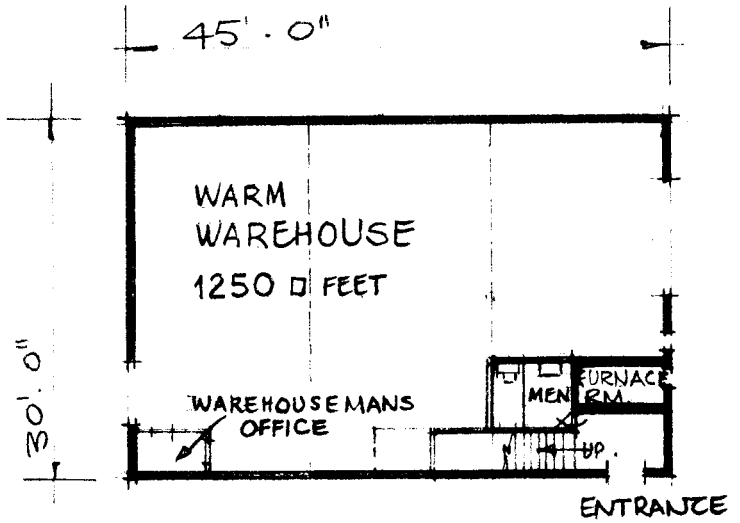
Building at $18.00 per sq.ft. including heating & lighting	=	$ 30,500.00	
Equipment			
Desks & Chairs Drafting Tables & Stools Filing Cabinets Typewriter Storage racks Lockers		6,500.00	
Freight			
67.5 Tons @ $80.00		5,200.00	
Total To Summary			$ 42,200.00

MISCELLANEOUS SMALL BUILDINGS, Etc.

Powder Magazine and Cap and Fuse Shed	$ 12,600.00	
Pit Shack on skids	2,400.00	
4 Pit Garages @ $5,000. each	20,000.00	
Sewage Disposal allowed	5,000.00	
		$ 40,000.00
Freight		
25 Tons @ $80.00	2,000.00	2,000.00
Total To Summary		$ 42,000.00



SECOND FLOOR - OFFICES



MAIN FLOOR - WAREHOUSE.

TWO STOREY BUILDING.
2700 SQ.FT. TOTAL.

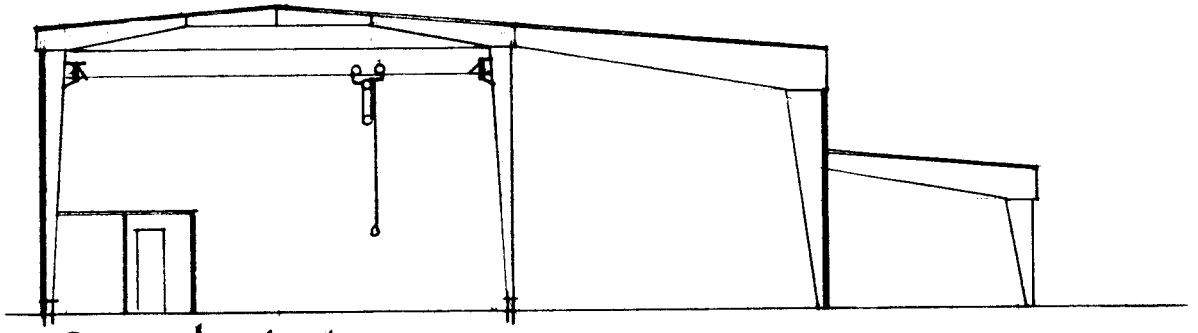
ADMIN. OFFICES & WARM WAREHOUSE

NEW IMPERIAL MINES LIMITED, WHITEHORSE, YUKON.

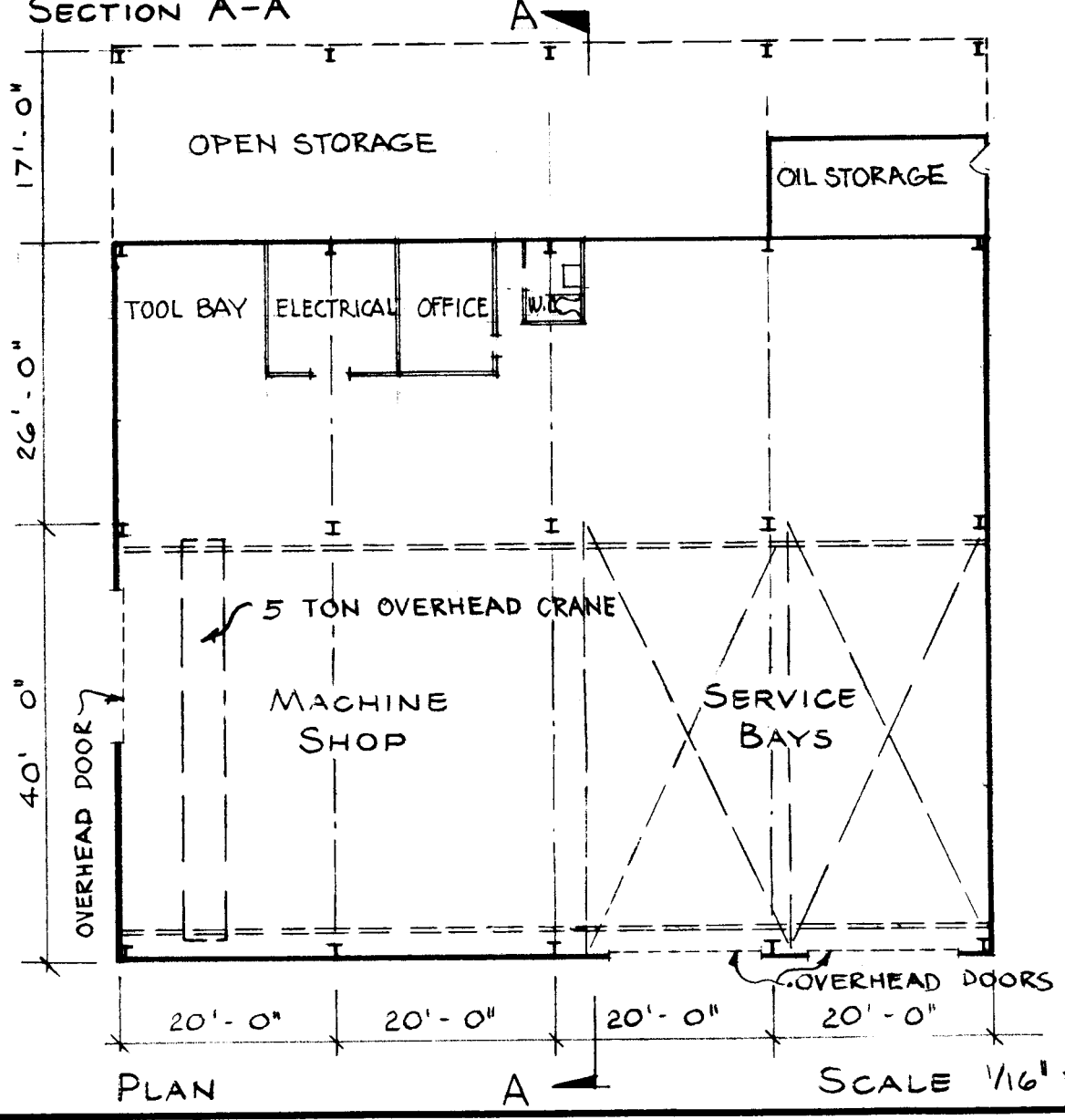
WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

DRAWN BY RC
DATE NOV 1965

DRAWING NO. 402 | 025 | 001



SECTION A-A



PLAN

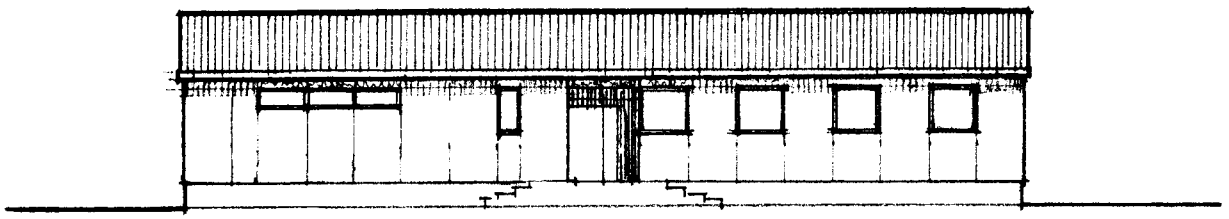
SCALE 1/16" = 1'-0"

MAINTENANCE BUILDING & COLD WAREHOUSE
NEW IMPERIAL MINES, LIMITED. WHITEHORSE, YUKON

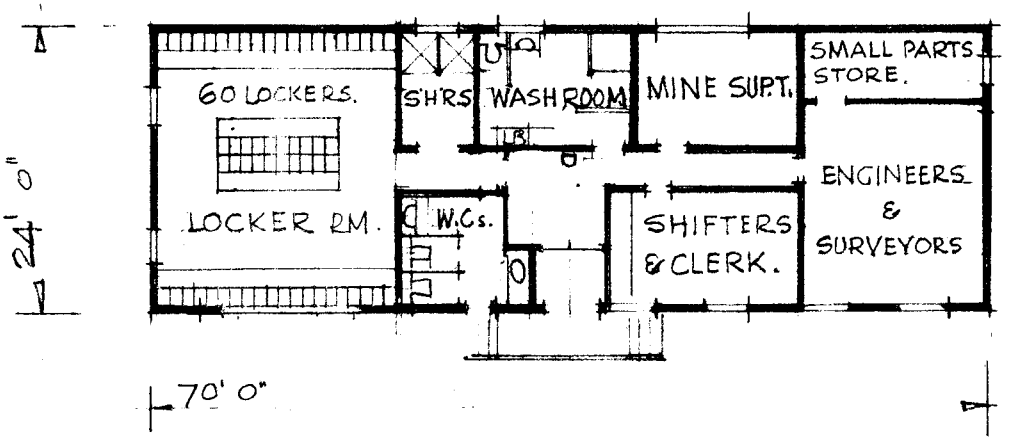
WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

DRAWN BY
DATE NOV 65

DRAWING NO. 403 025 002



ELEVATION.



1680 SQ.FT.

ACCOMMODATION FOR MINE SUPERINTENDENT.
2 SHIFTERS
CLERK.
2 ENGINEERS
SURVEYOR & HELPER
60 OPERATIVES.
SMALL PARTS & SURVEY EQUIP.T.

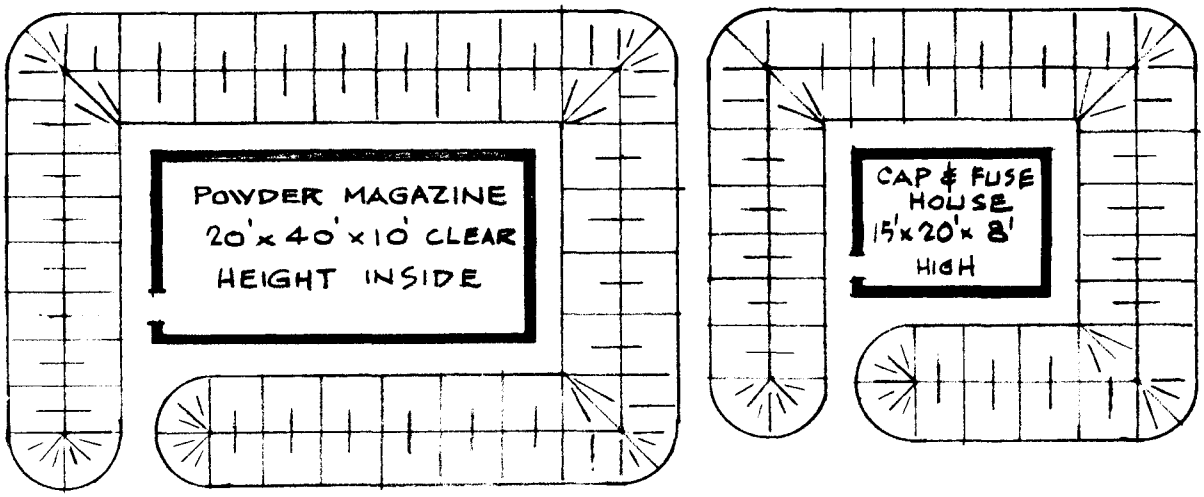
MINE OFFICE DRY & SMALL PARTS STORE

NEW IMPERIAL MINES, LIMITED. WHITEHORSE, YUKON.

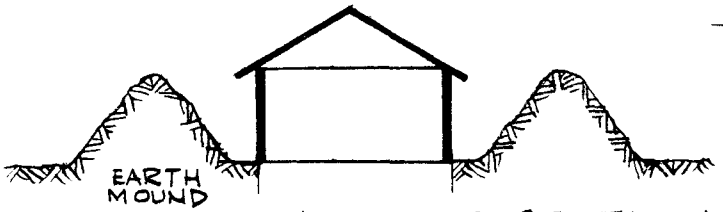
WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DRAWN BY RG
DATE NOV 1965

DRAWING NO.
402 | 025 | 003



PLAN



TYPICAL CROSS SECTION

NOTE:-

WALLS & ROOF MUST BE BULLET PROOF. BUILDINGS IN SEPARATE LOCATIONS.

SCALE:- 1" = 20'-0"

POWDER MAGAZINE & CAP & FUSE HOUSE

NEW IMPERIAL MINES, LIMITED. WHITEHORSE, YUKON

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DRAWN BY F.J.P
DATE NOV - 65.

DRAWING NO. 402 025 004

SECTION VI
POWER

ELECTRIC POWER

The estimated total electrical load demand for the mine, milling, water supply, crushing plant and auxiliary service operations is estimated to be approximately 2200 KVA for a production rate of 2000 TPD.

The overall load factor for the operation of the mill is estimated at 79%. The monthly energy consumption for 2000 TPD operation amounts to 1,240,000 KWH.

Power will be supplied from a 34KV transmission line from the Whitehorse Rapids plant to a location near the mill site. The Yukon Electrical Company Ltd., has agreed to provide a substation having a capacity of 2500 KVA, three phase, 60 cycle to deliver 2300/4160 volts. Metering will be installed on the secondary side of the transformer preferably in the main metalclad switchboard.

The billing demand offered by the Yukon Electrical Company Ltd., states that the monthly billing demand upon which the bill in any month or billing period is computed, will be the maximum demand established in the current month or any previous month of the preceding eleven months. This clause could be expensive as applied to the customer. For example, let us say that the ordinary maximum demand as estimated is 2200 KVA, but during a winter month the plant peak demand reached 2500 KVA. The normal monthly billing, based on 2200 KVA demand (assuming .85 power factor) is estimated at $18,898. However, the minimum monthly billing for all of the eleven months subsequent to this peak month would be based on 2500 KVA of peak demand and would never be less including relative excessive charges than $19,250 regardless of the actual demand or energy consumption during these months.

POWER SUMMARY

Plant Section	HP Connected	Demand Factor in %	HP Consumed	KW	Remarks
Crushing	630	80	504	377	2shifts/day-5days/W
Concentrator	1,917.25	90	1,720	1,282	3shifts/day-7days/W
Mining	30	70	21	15.6	3shifts/day-5days/W
Water Supply	200	90	180	134	3shifts/day-7days/W
Lighting(PFO.9)	175 KVA	65		156.8	3shifts/day-7days/W
Total KW				1,965.4	

Assuming Diversity Factor 1.05

Estimated Peak Demand $\frac{1,965}{1.05} = 1,870$ KW

Assuming Plant Power factor at .85

Estimated Peak Demand $\frac{1,871}{.85} = 2,200$ KVA

The estimates for overhead power lines are based on standard wood pole construction and allowances have been made for clearing of right-of-way of suitable width for the voltage class of power line concerned.

POWER CONSUMPTION

Plant Section	KW Consumed	Operating Hrs/Week	KWH/Week
Crushing	377	80	30,160
Concentrator	1,282	168	215,200
Mining	15.6	120	1,872
Water Supply	134	168	22,510
Lighting	156.8	168	26,400
Total			296,142

POWER COST

Re: The Yukon Electrical Company Ltd's. letter to New Imperial Mines Ltd., dated July 26, 1965.

Demand Charge - $7.5 KVA of billing demand to include 500 KWH per KVA of peak demand per month.

Excess Emergency Charge - All KWH's used in excess of 500 KWH'S/KVA/Month at 1.38¢.

Demand Charge - Max. demand (KVA) x 7.50 2200 x 7.50 = $16,570.

KWH Allowance - 2200 x 500 = 1,100,000 KWH/Month.

Excess energy charge with regard to days in Month.

$\frac{\text{Total KWH/Week}}{\text{7 Days}} = \frac{296,142}{7} =$ 42,306 KWH Daily.

1 Month of 28 Days	42,306x28-1,100,000 = 84,068x1 @ 1.38¢ =	$1,160
4 Months of 30 Days	42,306x30-1,100,000 =168,680x4 @ 1.38¢ =	$9,311
6 Months of 31 Days	42,306x31-1,100,000 =210,986x6 @ 1.38¢ =	$17,469
1 Month/2 Weeks Vacation	No Surplus Charge Yearly Cost:	$27,940

Average/Month: $\frac{27,940}{12} =$ $ 2,328

Demand Charge: 16,570

$18,898

DESCRIPTION

1. Metalclad switchgear made up of a total of 7 cells and including the following:

Metalclad switchgear shall be rated for 5KV 150 MVA with a bus work rated for 1000 amp.

Cell # 1

One 1200A 5 KV 60 cycles 150 MVA power circuit breaker removable element type complete with one ammeter and 3 phase ammeter transfer switch, three current transformers, two relays inst. and time overcurrent and one relay, ground censor.

Cell # 2

Metering unit with provision for current and potential transformers and meters.

Cells 3 to 7

5 fused load interrupters, quick make, quick break 3 pole group operated, front operated handle mechanism, switch rated 600 amper continuous 600 amperes interrupting 40,000 RMS amps, complete with 3 power fuse holders, with mufflers.

6. 2 - 750 KVA 3 phase, 60 cycles 2400/600 V power transformers ONS type (oil immersed, air cooled) complete with tap changer.

7. 2 - Air circuit breakers stationary type 50 KA interrupting capacity at 600 volt, 800 amp, 3 spst manually operated with adjustable inverse time and short time delay trips.

8. 2 - Motor control centres shall consist of fused combination starters ranging from CEMA size 1 to CEMA size 5, main bus work shall be rated for 1000 A 600 volt.

9. One synchronous motor starter 5 KV 600 HP, 8 pf with current limiting fuses and contactor complete with all auxiliary relays as inst. overcurrent, inst. (ground sensor) field removal, incomplete sequence inst. undervoltage and ammeter wattmeter and varmeter.

10. One synchronous motor starter 5 KV 1000 HP .8pf same as described under (9).

16. Grounding shall consist of bare stranded soft annealed copper conductors and cadweld connectors.

17 & 18. 2 - operator control consoles consist of push buttons and pilot lights.

19. 2.4 KV overhead line using #4 bare stranded copper wire poles spaced @ 80 ft complete with all auxiliary equipment required for distribution.

DESCRIPTION Cont'd.

Lighting; Mercury vapour hi-bay low bay lamp fittings for 600 volt.

Lamp Fittings; Fluorescent and incandescent industrial types.

Cables

Thermoplastic insultated 0-600 volts.

Single conductor, #10 and smaller shall be solid while #8 and larger shall be stranded annealed. Copper wire general purpose, moisture resistant and flame retarding building wire to be suitable for open wiring, flexible or rigid conduit wiring or metal troughs and for dry and wet locations.

Current ratings to be supplied based on ambient temperature 30°C (86°F).

No. 3 Armoured Cable 0 - 3000 volts

Multi-conductor, standard stranded tinned copper conductors insulated with heat, moisture and ozone resistant rubber. The conductors cabled together with rubber fillers which completely fill the cable interior. Neoprane jacketed with interlocking aluminum sheath and P.V.C. sheath overall.

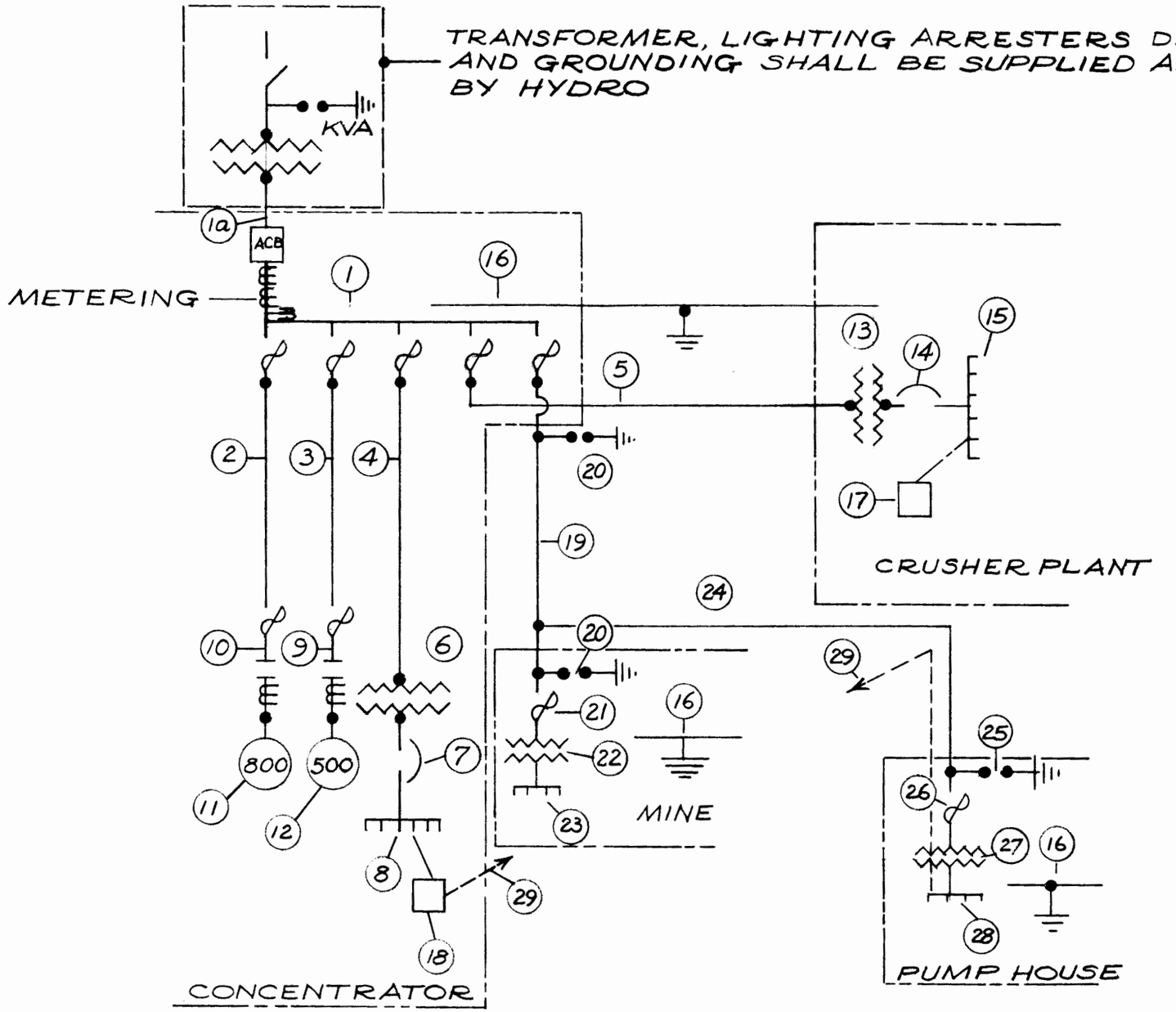
Ground conductor to be included with line conductors.

To be used for control, lighting and power circuits in wet locations and to be suitable for direct earth burial.

Current ratings to be supplied based on ambient temperature of 30°C. (86°F).

Item No.	Concentrator	Weight lbs.	Material $	Labour $
2	100' 4/0 Teck Cable 3KV 3 cond.	400	600	450
3	100' 1/0 Teck Cable 3KV 3 cond.	240	450	300
4	60' 3/0 Teck Cable 3KV 3 cond.	200	325	250
6	750 KVA 2400/600 V 3 phase ONS Transformer	8,500	6,200	1,500
7	A.C.B. Stationary Type 50 ka i.c./600V 3 phase	900	1,600	400
8	Motor control centre complete wired to individual motors 6 Vertical Section	5,400	20,100	20,100
9	2500 V Synchrs. Motor starter limit amp 150 MVA 600 HP	4,000	5,000	500
10	2500 V Synchrs. Motor starter limit amp 150 MVA 1000 HP	4,000	5,500	500
11	800 HP Synchrs. Motor all auxiliaries incl. & wired to starter		20,000	15,000
12	500 HP Synchrs. Motor all auxiliaries incl. & wired to starter.		10,200	10,000
18	Operator Control Console Concentrator Plant	600	1,200	1,200
		24,240	71,175	50,200
	Freight 12 tons @ $80		960	
	Tax @ 11% on $71,175		7,830	
			79,965	50,200
	Sub-Total to Section VII			$ 130,200
	Crusher Plant			
5	240' 3/0 Teck Cable 3 KV 3 cond.	780	1,300	1,000
13	750 KVA 2400/600V 3 phase ONS Transformer	8,500	6,200	1,500
14	A.C.B. Stationary Type 50 KA I.C./600V 3 phase	900	1,600	400
15	Motor Control Centre Complete wired to individual motor 5 Vertical Sections	4,500	17,500	17,500
17	Operator Control Console Crusher Plant	600	1,200	1,200
		15,280	27,800	21,600
	Freight 8 tons @ $80		640	
	Tax @ 11% on $27,800		3,058	
			31,498	21,600
	Sub-Total to Section VII			$ 53,100

Item No.	Water Supply	Weight lbs.	Material $	Labour $
24	2.4KV overhead line complete instal. 136 poles appr. 13,500' @ $1.42/ft. 40,500 ft. wire	127,665	8,200	11,000
25) 26) 27) 28) 29)	Lightning arrestors, cutouts, 3 single phase transformers mounted on a platform 50KVA 2400/600V distribution equip. incl. starters & aux. equip. wired to individual motors. C Control cable with guide wire	2,140	8,500	8,500
		129,805	16,700	19,500
	Freight 65 tons @ $80		5,200	
	Tax @ 11% on $16,700		1,840	
			23,740	19,500
	Sub-Total to Section VIII			$ 43,200
	Mine & General			
1	Main 2,400 V metalclad switchboard 7 cells	15,000	11,900	3,000
1A	100' 350 MCM Teck Cable 3 cond. 3 KV	560	1,050	800
16	Grounding system mill, crusher plant, Pump station and Mine		800	1,400
19	2.4 KV Overhead Line Approx. 1000' 11 poles 3000'wire	10,290	800	800
20) 21) 22) 23)	Lightning arrestors, cut outs, 3 phase transformer 45 KVA, 2400/600 V. Distribution equip. incl. starters. disc. switches wired to individual motors	820	3,600	3,000
	Indoor lighting mill and crusher plant, pump house.		15,000	15,000
	Electrical heating mill and crusher plant, pump house.		4,500	4,500
	Outdoor lighting including mine.		6,800	6,800
		26,670	44,450	35,300
	Freight 13 tons @ $80		1,040	
	Tax @ 11% on $44,450		4,900	
	Sub-Totals to Summary Sheet		$50,390	$ 35,300



	NEW IMPERIAL MINES WHITEHORSE, YUKON	ELECTRICAL DISTRIBUTION SYSTEM	
	WRIGHT ENGINEERS LIMITED VANCOUVER——CANADA	DRAWN BY LMC DATE NOV. 65	DRAWING NO. 402 026 001

SECTION VII
CRUSHING AND CONCENTRATING PLANT

CRUSHING PLANT

The proposed crushing plant is sized to crush 200 ton/hour, 14 hours/day, 5 days/week. The plant, for ease of operating and due to site conditions, encloses in one building the primary, secondary, and tertiary crushers. The primary crusher circuit operates as part of the open pit, and comprises a truck dump, hopper, feeder, jaw crusher (with 5" setting), screen (5/8" opening), and a coarse ore stock pile. The secondary, tertiary circuit reclaims 5" material from the stock pile, and reduces it to 1 1/2" in a standard cone crusher in open circuit, and to 5/8" in a short-head cone crusher, in closed circuit with a single rod deck screen. The-5/8" ore is screened out after each stage of crushing, and by-passed to the fine ore bins.

The coarse ore pile has a live capacity of 2,000 tons, which is adequate for surge storage ahead of the secondary crusher. Moreover it has a total storage of approximately 14,500 tons of-5" material, which can be reclaimed if inclement weather interrupts mining. It is not considered necessary to cover the pile inasmuch as, precipitation and snowfall is low, the roof structure is expensive and becomes a hazard if bull-dozing of the pile is required. Excavation and foundation costs assume that foundations are laid on bed rock and that over burden is about one foot on the average.

The magnetite in the ore and non magnetic steel from the open pit operation makes a suspended magnet unsuitable for removal of tramp iron, consequently a metal detector is allowed for in the capital cost estimate.

CONCENTRATOR

The concentrator is designed to treat 2000 TPD of feed on a continuous basis to produce daily a maximum of 80 dry short tons of copper concentrate containing 35% copper. The concentrator is situated at elevation 2605 ft. and some 200 ft. to the northeast of, and 55 ft. below the crusher building. The site consists of gravel flats which have a light soil cover with some erratic boulders. After the trees are cleared and stumped little levelling and excavation will be required.

Several alternative schemes for fine ore storage were considered. The drill cores indicate that copper values will be fairly erratic (also the molybdenite in the Cowley Park ore) and blending of the ores is desirable to obtain an even grade of feed to the concentrator. Blending is also desirable when treating a mixture of ores from several pits having slightly different grinding characteristics. A single rectangular bin with multiple draw off points and fed by shuttle conveyor would blend the mixed ore better than the two circular 2,500 ton capacity bins proposed, but the higher capital cost of the rectangular bin cannot be justified. The two circular bins will blend the mixed ore better than a single circular bin at a small increase in capital cost.

Fine ore is drawn from the bins through tube feeders (6 per bin) on to two variable speed conveyor belts. A single conveyor collects the fine ore and transfers it to the 10'ø x 12' rod mill. The feed rate is measured and controlled by a weightometer on the rod mill feed belt. The rod mill discharge is pumped to a cyclone in closed circuit with a 10'ø x 14' ball mill. The cyclone overflow drains to an 8'ø x 8' conditioning tank ahead of rougher flotation. The rougher concentrate is pumped to a cyclone in closed circuit with a 7'ø x 7' regrind ball mill and the cyclone overflow is fed to the cleaner and recleaner cells. The tailing from the cleaner cells is combined with the scavenger concentrate and returned to the primary grinding circuit. The tailing from the recleaner cells is recirculated to the cleaner cells.

After the final copper concentrate is thickened it is stored in a 10'ø x 10' stock tank from which it is pumped to a 6'ø x 6 disc filter. The filtered product is dried in a 4'ø x 28' oil fired rotary drier from which it is transported via a conveyor to concentrate storage. Concentrate is reclaimed from the 300 ton stockpile by a front end loader and loaded into containers supplied by the Whitepass and Yukon Railway.

Storage is provided in the building for 12 - 12' x 4' x 3'containers stacked three deep. A 10 ton crane is installed to service the concentrator and will be used to move the containers to filling, storage and loading on to trucks.

The concentrate will be dried to 8% or 9% moisture during the summer months but during freezing conditions in the winter the moisture will be reduced to between 3% and 4%.

It is not possible at present to design the molybdenite circuit for the Cowley Park ore. It is expected however that a rougher molybdenite concentrate will be floated from the copper recleaner concentrates and can be raised to marketable grade with four or five stages of cleaning. An extra thickener, filter, and drier will be required. It is not expected that extra regrinding will be necessary.

CRUSHING PLANT

Equipment	Weight lbs.	H.P.	Cost $
Reciprocating Plate Feeder	20,000	25 HP	$ 11,000
Stationary Grizzly c/w Chute	37,000		22,000
Jaw Crusher 42" x 48"	145,000	150 HP	82,400
Discharge Chute	3,000		1,800
Screen Feed Box	2,000		1,200
Scalping Screen 5'x12' Single Deck	10,000	15 HP	7,500
Vibratory Feeders (2)	10,600	6 HP	9,000
Crusher Feed Box	3,000		1,800
4 1/4' Std. Cone Crusher	50,000	150 HP	44,200
Discharge Chute	4,000		2,400
Transfer Chute	6,000		3,600
Screen Feed Box	3,000		1,800
5'x12' Rod Deck Screen	17,500	10 HP	13,500
Screen Discharge Chute	10,000		6,000
Crusher Feed Box	3,000		1,800
4' Short-Head Cone Crusher	47,400	150 HP	46,000
Discharge Chute	3,000		1,800
Metal Detector	500		5,200
Dust Control System	12,000	15 HP	18,500
10 Ton Travelling Crane	20,000	5 HP	15,000
Conveyor No. 1 90' long	8,300	10 HP	9,000
No. 2 220' long	18,600	20 HP	17,000
No. 3 220' long	18,400	15 HP	16,800
No. 4 140' long	12,800	15 HP	13,500
No. 5 100' long	10,000	15 HP	11,000
No. 6 200' long	17,000	20 HP	14,700
Transfer Chute	6,000		3,600
SUB-TOTAL -- To Summary Sheet			$ 382,100

CRUSHER BUILDING

Construction and Installation	Quantity	@ $	Cost $
Excavation			
Rock	4,000 cu.yd.	3.5	14,000
Footings	150 cu.yd.	20	3,000
Back Fill Tamped	220 cu.yd.	2.5	550
Concrete Slab	80 cu.yd.	130	10,200
Walls	250 cu.yd.	140	35,000
Foundations	170 cu.yd.	130	22,000
Steel Columns	36,000 lb..	.40	14,400
Beams & Trusses	25,000 lb.	.40	10,000
Purlins	25,000 lb.	.40	10,000
Cladding	11,000 sq.ft.	1.10	12,100
Miscellaneous Items			
Platforms, stairs & handrails	2,500 sq.ft.	8	20,000
Conveyor Galleries	450 ft.	100	45,000
Towers and Cantilever	Allowance		20,000
Switch Room	500 sq.ft.	7	3,500
Wash Room	120 sq.ft.	20	2,400
Fire Protection	4,000 sq.ft.	.30	1,200
Windows & Doors	Allowance		5,000
Coarse ore Stock Pile			
Excavation	500 cu.yd.	3.5	1,750
Concrete Tunnel	140 cu.yd.	140	19,600
Freight and Handling	220 tons	80	17,600
Equip. installation excluding electrical	$382,100 @	25%	95,500
Federal Tax - Equipment	$382,100 @	11%	42,000
- Construction	$230,400 @	5 1/2%	12,700
Electrical Equipment and Installation including freight and tax (From Section VI) (excluding lighting and heating)			53,100
SUB-TOTAL -- To Summary Sheet			$ 470,600

CRUSHER & CONCENTRATOR OPERATING COSTS

Supplies & Power

Primary Crushing	Cost/Yr. $	
Liners @ 0.04 lb./ton @ 42¢/lb.	12,100	
Misc. material @ .4¢/ton	2,880	
Power 0.63 KWH @ 1.5¢/KWH	6,800	
		21,780
Secondary Crushing		
Liners @ 0.05 lb./ton @ 52¢/lb.	18,700	
Misc. maintenance @ 1.0¢/ton	7,200	
Screening, feeders, chutes, etc. @ 1.0¢/ton	7,200	
Conveyors @ 1.0¢/ton	7,200	
Motors, lubrication, dust collection, etc., @ .5¢/ton	3,600	
Power 1.47 KWH @ 1.5¢/KWH	15,850	
		59,750
Primary Grinding		
Rods 0.75 lb/ton @ 12¢/lb.	64,800	
Liners & Material @ .05 lb/ton @ 24¢	8,640	
Balls 1.0 lb./ton @ 13¢	93,400	
Liners & maint. @ .09 lb./ton @ 20¢	12,960	
Conveyors, chutes, general @ 1.0¢/ton	7,200	
Pumps, cyclones, etc. @ 0.5¢/ton	3,600	
Power 11.19 KWH/ton @ 1.5¢/KWH	120,860	
		311,460
Flotation & Regrind		
Reagents @ 20¢/ton	144,000	
Cell repair & Pumps, etc. @ 1.5¢/ton	10,800	
Regrind, liners, maint. etc. @ .10 lb. @ 20¢	1,600	
Balls .5 lb. @ 14¢ on 79,000 tons	5,500	
Power 3.51 KWH/ton @ 1.5¢/KWH	37,930	
		199,830
Filtering & Drying		
Filter maint. @ 4¢/ton on 30,000 tons	1,200	
Fuel 2.5 gal./ton @ 27¢/gal.	20,200	
Dryer maint. @ 4¢/ton	1,200	
Power 0.83 KWH/ton @ 1.5¢/KWH	9,000	
		31,600

CRUSHER BUILDING

Construction and Installation	Quantity	@ $	Cost $
Excavation			
Rock	4,000 cu.yd.	3.5	14,000
Footings	150 cu.yd.	20	3,000
Back Fill Tamped	220 cu.yd.	2.5	550
Concrete Slab	80 cu.yd.	130	10,200
Walls	250 cu.yd.	140	35,000
Foundations	170 cu.yd.	130	22,000
Steel Columns	36,000 lb..	.40	14,400
Beams & Trusses	25,000 lb.	.40	10,000
Purlins	25,000 lb.	.40	10,000
Cladding	11,000 sq.ft.	1.10	12,100
Miscellaneous Items			
Platforms, stairs & handrails	2,500 sq.ft.	8	20,000
Conveyor Galleries	450 ft.	100	45,000
Towers and Cantilever	Allowance		20,000
Switch Room	500 sq.ft.	7	3,500
Wash Room	120 sq.ft.	20	2,400
Fire Protection	4,000 sq.ft.	.30	1,200
Windows & Doors	Allowance		5,000
Coarse ore Stock Pile			
Excavation	500 cu.yd.	3.5	1,750
Concrete Tunnel	140 cu.yd.	140	19,600
Freight and Handling	220 tons	80	17,600
Equip. installation excluding electrical	$382,100 @	25%	95,500
Federal Tax - Equipment	$382,100 @	11%	42,000
- Construction	$230,400 @	5 1/2%	12,700
Electrical Equipment and Installation including freight and tax (From Section VI) (excluding lighting and heating)			53,100
SUB-TOTAL -- To Summary Sheet			$ 470,600

FINE ORE BINS AND CONCENTRATOR

Equipment	Weight lbs.	H.P.	Cost $
Conveyor No. 7 40' long	5,400	5	6,000
12 Tube Feeders	6,000		4,800
12 Chutes	7,200		4,500
Conveyor No. 8 72' long	7,500	5	8,700
Conveyor No. 9 72' long	7,500	5	8,700
2 Transfer Chutes	6,000		3,600
Conveyor No. 10 58' long	6,800	5	6,800
Weightometer			5,000
10'ø x 12' Rod Mill including charge	285,000	500	113,900
10'ø x 14' Ball Mill including charge	315,000	800	124,700
7'ø x 7' Ball Mill including charge	116,000	200	54,400
Rod Mill feed chute	1,200		600
Rod Mill pump box	7,000		2,000
Rod Charger			3,500
2 - 10" x 8" SRL-C Pumps	11,000	50	10,300
2 - 24"ø Cyclones incl. piping	2,500		7,000
Cyclone underflow box	1,200		2,000
8'ø x 8' Conditioner	4,000	5	1,900
2 - 16 cell # 24 Flotation Machines	102,400	160	53,400
Blower	1,500	5	2,100
2 - 8 Cell # 18s Flotation Machines	30,100	40	17,400
Distributor	800		400
14" Cyclone including piping	1,500		3,000
Launders	4,000		5,000
2 - 5" x 5" SRL Pumps	3,000	20	4,000
4 Pump boxes	4,000		3,000
2 - 2" x 2" SRL Pumps	1,400	10	1,600
28'ø Concentrate Thickener	38,000	1 1/2	12,000
10'ø x 10' Stock Tank	2,000	7 1/2	2,900
6'ø x 6 Disc Filter (incl. vac. equip.)	11,000	48	13,900
4'ø x 28' Rotary Dryer	25,000	30	28,000
Refractories			1,000
Fuel Storage Tank	2,000		1,000
Reagent Distribution (allowance)			6,000
Conveyor No. 11 34' long	5,000	5	3,000
2 Automatic Samplers	1,000		2,500
Sump Pump	1,000	3	1,500
10/2 ton Crane 86' span	45,000		20,000
General Piping (allowance)	10,000		30,000
Welding set and tools (allowance)			3,000
SUB-TOTAL -- To Summary Sheet			$583,100

FINE ORE BINS AND CONCENTRATOR

Construction and Installation	Quantity	@ $	Cost $
Site Roads (allowance)			20,000
Clearing and Grubbing	9 acres	1,500	13,500
Rough Excavation and Backfilling (allowance)			20,000
Fine Ore Bins:-			
Excavating footings	100 cu.yds	20.	2,000
Concrete:			
Structural slab	180 cu.yds	130	23,400
Columns and Foundations	130 cu.yds	140	18,200
Base slab	40 cu.yds	130	5,200
Steel:-			
Roof and supports	65,000 lbs.	40¢	34,500
Bins	242,000 lbs.	40¢	96,800
Roof Cladding	4,200 sq.ft	1.10	4,600
Wall Cladding (allowance)			5,000
Concentrator:			
Excavating Foundations (allowance)			5,000
Concrete:-			
10'ø x 12' Rod Mill Foundations	190 cu.yds	130	24,700
10'ø x 14' Ball Mill Foundations	210 cu.yds	130	27,300
7'ø x 7' Ball Mill Foundations	80 cu.yds	130	10,400
Walls	130 cu.yds	140	18,200
Flooring	230 cu.yds	130	29,900
Concentrate Retaining Walls	110 cu.yds	130	14,300
Crane Columns			2,000
Steel:-			
Main building structural frame	431,200 cu.ft	40¢	172,500
Cladding (allowance)			30,000
Office, assay laboratory building	660 sq.ft	14	9,200
Office, dry, assay laboratory equipment (allowance)			20,000
Sprinkler system	12,300 sq.ft	30¢	3,700
Electrical Control Room (allowance)			2,500
Handrails, stairs and platforms	8,800 sq.ft	10	88,000
Freight and Handling	539 tons	80	43,100
Equipment Installation	$538,100 @	25%	145,800
Federal Tax- Equipment	$583,100 @	11%	64,100
- Construction	$633,200 @	5 1/2%	34,800
Electrical Equipment and Installation including freight and tax (from Section VI) (excluding lighting and heating).			130,200
SUB-TOTAL -- To Summary Sheet.			$1,118,900

CRUSHER & CONCENTRATOR OPERATING COSTS - Cont'd.

Supplies & Power

Miscellaneous	Cost/Yr. $	
Assay supplies	4,800	
Tailing disposal @ 1¢/ton	7,200	
Mill general @ 9.0¢/ton	64,800	
Light, heat @ 1.5 KWH @ 1.5¢/KWH	16,200	
Power for water supply 1.63 KWH @ 1.5¢/KWH	17,640	
		110,640
Total Supplies & Power		735,060
Total Labour & Supervision		187,600
Total Cost/Year		$ 922,660

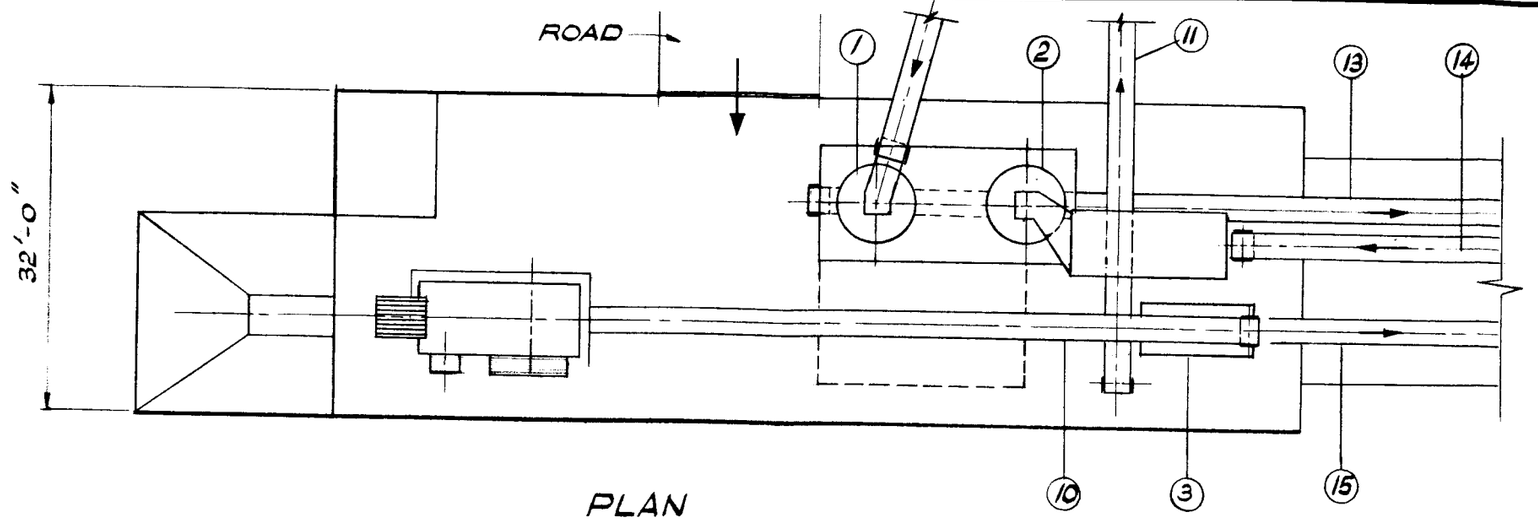
Total Operating Cost/Ton $\frac{922,660}{720,000}$ = $ 1.281/Ton @ 720,000 Tons/Yr.

$\frac{922,660}{650,000}$ = $ 1.419/Ton @ 650,000 Tons/Yr. (1st. Yr.)

CRUSHER & CONCENTRATOR LABOUR REQUIREMENTS

	No. Men	Shifts /Day	Men /Day	Days /Year	Rate $/Day	Cost $/Year
Crushing Plant						
Operator	1	2	2	260	18.88	9,820
Helper	1	2	2	260	18.32	9,530
Mill						
Grinding	1	3	3	360	18.88	20,390
Flotation	1	3	3	360	21.04	22,720
Filter & dryer handled by Shift Boss						
Loading	2	1/2	1	260	18.32	4,760
Reagents & Balls	2	1/2	1	260	18.32	4,760
Maintenance 1st	2	1	2	260	22.08	11,480
Maintenance 2nd	2	1	2	260	20.32	10,570
Labour	4	1	4	260	17.28	17,970
Supervision						
Superintendent	1	1	1	Salary	1,000	12,000
Foreman	1	1	1	"	700	8,400
Shift Bosses	1	3	4	"	600	28,800
Metallurgist	1	1	1	"	700	8,400
Chief Chemist	1	1	1	"	600	7,200
Chemist	1	1	1	"	500	6,000
Helper	1	1	1	"	400	4,800
Total working direct labour			30 Men			$ 187,600

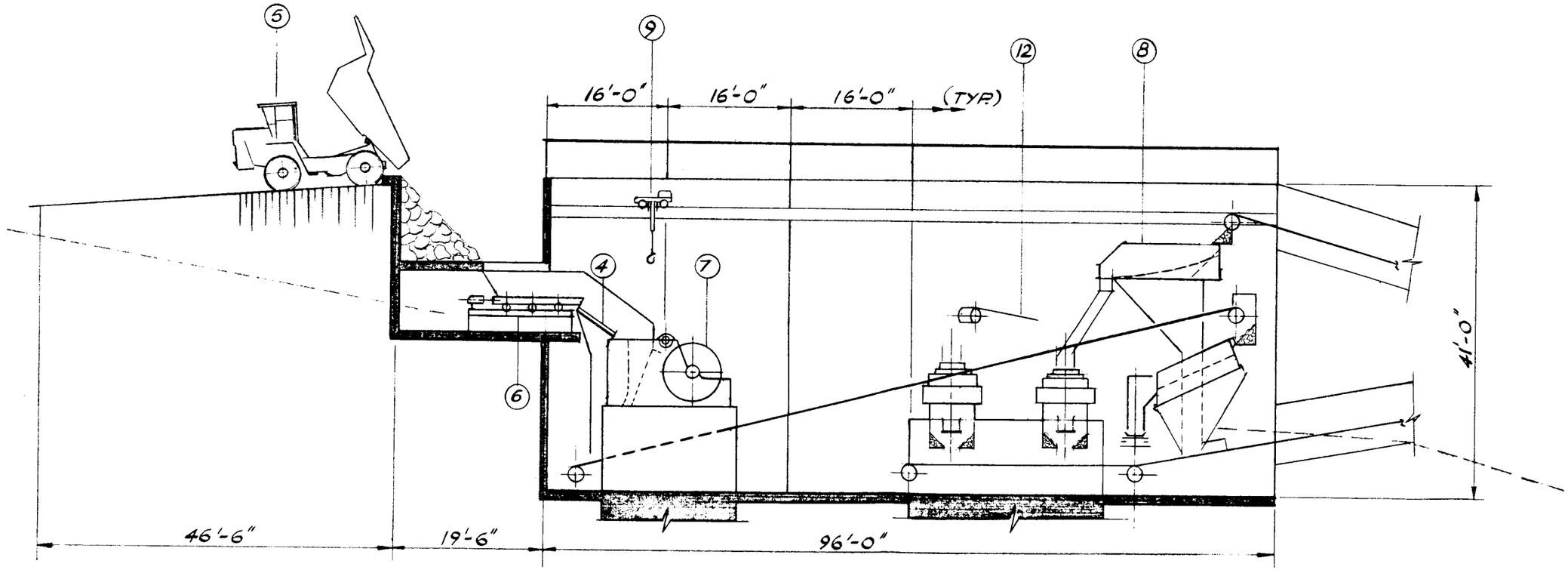
$ 0.289/Ton milled @ 650,000 Tons/Yr.
$ 0.261/Ton milled @ 720,000 Tons/Yr.



PLAN

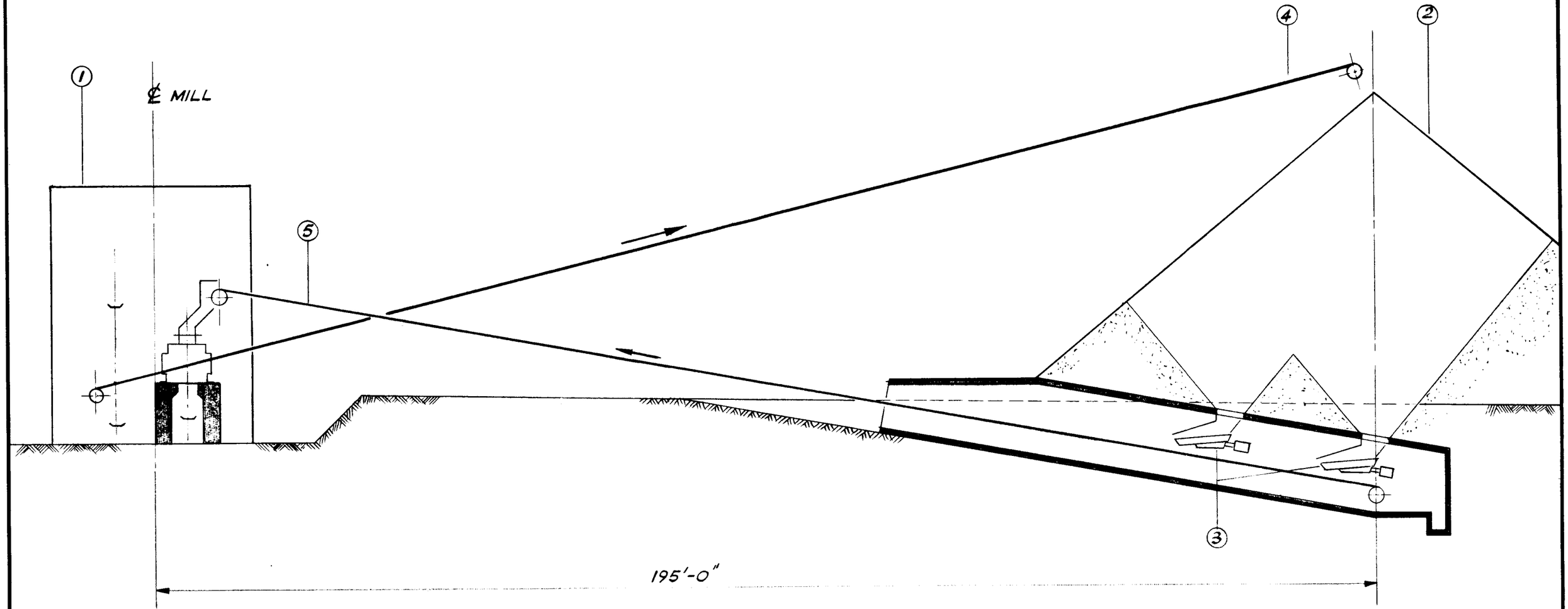
LEGEND.

1. 4¼ Ft SYMONS STD. CONE CRUSHER.
2. 4Ft. SYMONS S.H. CONE CRUSHER.
3. 5×12 SINGLE DECK SCREEN
4. STATIONARY GRIZZLY
5. 35T. TRUCK
6. RECIPROCATING PLATE FEEDER.
7. 42"×48" JAW CRUSHER.
8. 5'×12' ROD DECK SCREEN.
9. 10 TON CRANE.
10. CONVEYOR No. 1
11. CONVEYOR No. 2
12. CONVEYOR No. 3
13. CONVEYOR No. 4
14. CONVEYOR No. 5
15. CONVEYOR No. 6



SCALE : 1/16" = 1'-0"

	NEW IMPERIAL MINES LTD. WHITEHORSE, YUKON.	CRUSHING & SCREENING PLANT.
	WRIGHT ENGINEERS LIMITED VANCOUVER——CANADA	DRAWN BY C.Mc. DATE Nov. 1965
		DRAWING No. 402 027 001



LEGEND.

1. CRUSHING & SCREENING PLANT
2. COARSE ORE STOCKPILE
3. VIBRATORY FEEDERS
4. CONVEYOR No. 2
5. CONVEYOR No. 3

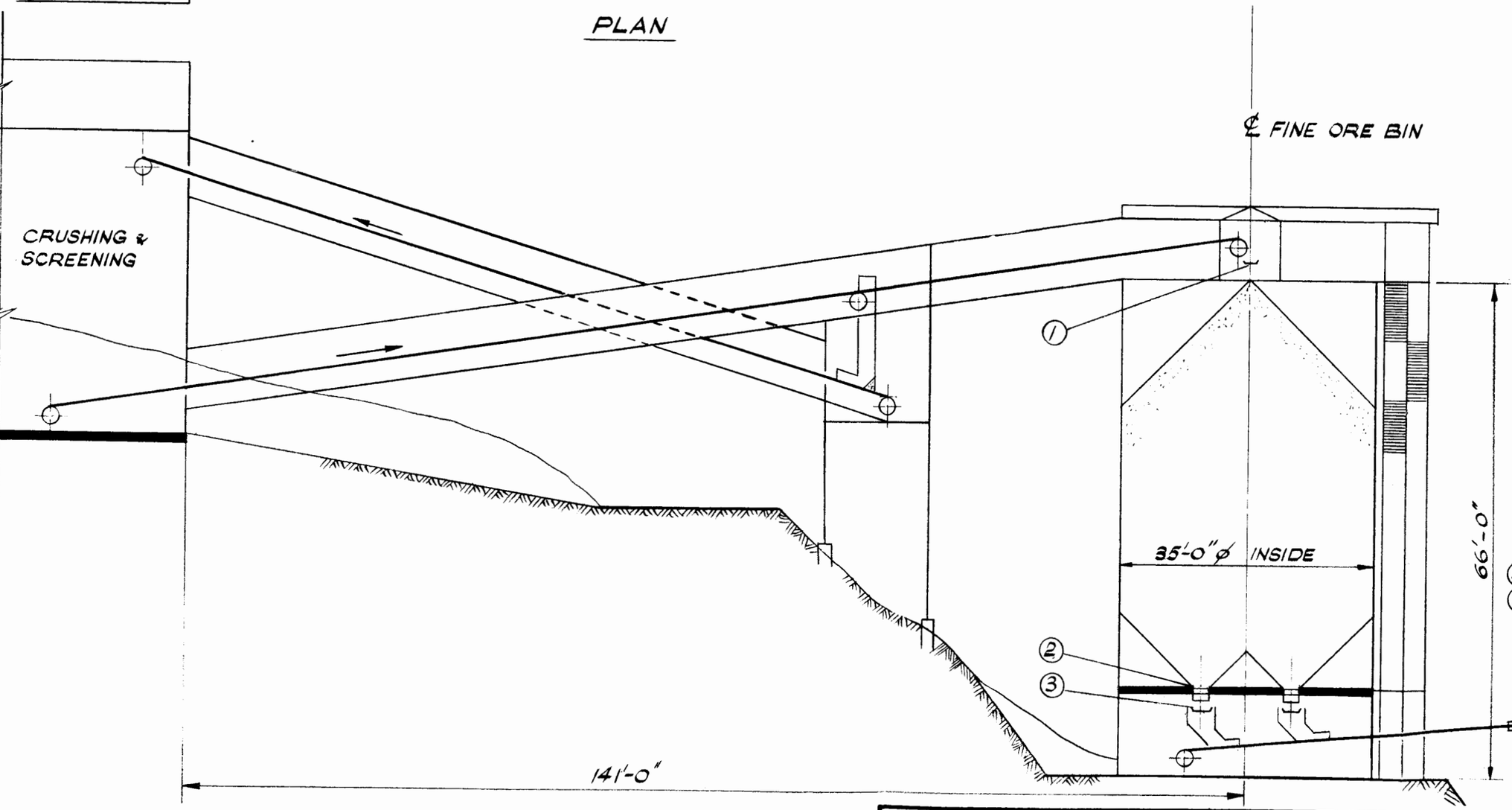
SCALE : 1/16" = 1'-0"

	NEW IMPERIAL MINES LTD. WHITEHORSE, YUKON.	SECTION SHOWING COARSE ORE STOCKPILE.
WRIGHT ENGINEERS LIMITED VANCOUVER——CANADA	DRAWN BY C. Mc. DATE Nov. 1965	DRAWING NO. 402 027 002

LEGEND

1. CONVEYOR No.7 (REVERSIBLE)
2. TUBE FEEDER
3. CONVEYOR No. 8 & No. 9
4. CONVEYOR No. 10
5. WEIGHTOMETER

PLAN



SCALE: 1/16" = 1'-0"

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

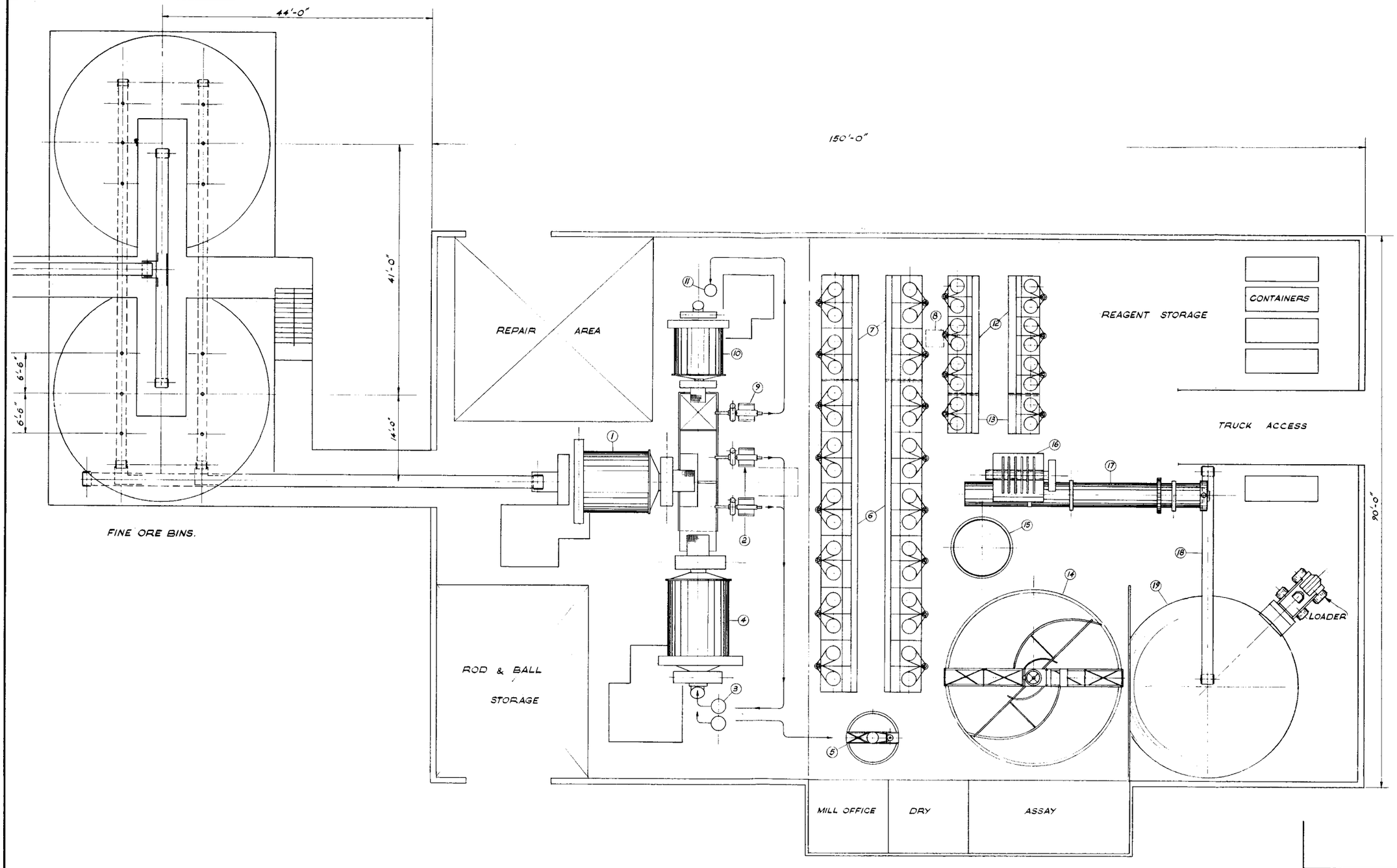
SECTION
TRANSFER HOUSE & F.O. BIN.

WRIGHT ENGINEERS LIMITED
VANCOUVER——————CANADA

DRAWN BY C. Mc.
DATE Nov. 1965

DRAWING NO. 402 027 003

DWG. NO. 402 027 004



LEGEND.

1. 10'ø x 12' ROD MILL
2. 10"x 8" S.R.L. C-PUMP
3. 24' CYCLONE
4. 10'ø x 14' BALL MILL
5. 8'ø x 8' CONDITIONER
6. ROUGHERS 2 BANKS OF 12-50 cu Ft. CELLS.
7. SCAVENGERS 2 BANKS OF 4-50 cu Ft. CELLS.
8. 5"x 5" S.R.L. C-PUMP
9. 5"x 5" S.R.L. C-PUMP
10. 7'ø x 7' REGRIND BALL MILL.
11. 14" CYCLONE
12. CLEANERS 2 BANKS OF 6-24 cu Ft. CELLS.
13. RECLEANERS 2 BANKS OF 2-24 cu Ft. CELLS.
14. 28'ø CONC. THICKENER.
15. 10'ø x 10' STOCK TANK.
16. 6'ø x 6 DISC. FILTER.
17. 4'ø 28' ROTARY DRYER
18. CONVEYOR # 11 , 18'
19. 300 TON STOCKPILE.

SCALE: 1/16" = 1'-0"

DESCRIPTION OF REVISION	ISSUED FOR	APPR.	DESIGN FINAL CHECK	DIM. CHECK	DRAWN	DESIGN	APP.
	REPORT.				C.M.		NOV 1968

REFERENCE	DWG. No.

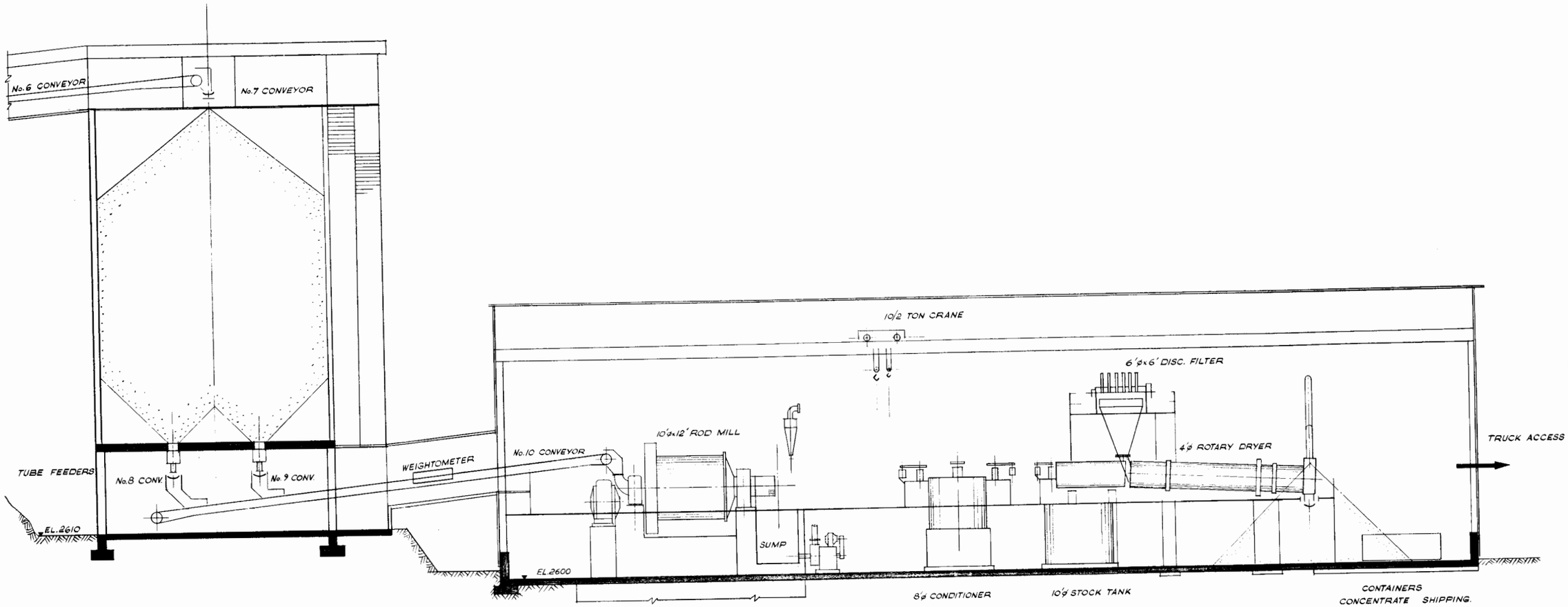
CONCENTRATOR
GENERAL ARRANGEMENT.

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

DWG. No. 402 027 004

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

DWG. NO. 402 027 005



FINE ORE BINS.

SCALE: 1/8" = 1'-0"

NO.	DESCRIPTION OF REVISION	APPR.	DATE
	ISSUED FOR REPORT.		

APP.			
DATE			
DESIGN			
DRAWN	JE		
DIM. CHECK			
DESIGN CHECK			
FINAL CHECK			
APPR.			

REFERENCE | DWG. No.

SECTION
FINE ORE BINS & CONCENTRATOR.

NEW IMPERIAL MINES LTD.
WHITEHORSE, YUKON.

DWG. No. 402 027 005

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

SECTION VIII
WATER SUPPLY AND TAILINGS DISPOSAL

WATER SUPPLY

An accurate economic comparison of alternative sources of water for the plant water supply is difficult due to the lack of long term records of stream run off in the vicinity of the plant site. Because of this lack of data and the lack of assurance that plant water requirements can be obtained from nearby sources of water, the proposed water supply system is based on a year round pumped supply from the Yukon River supplemented by water drained from Parker Lake for 9 months of the year. An assured year round supply of water could also be obtained either from McIntyre Creek or Wolf Creek but development of an assured water supply from these creeks would be more costly than from the Yukon River.

The proposed Yukon River system would require three pumps to deliver 900 gallons per minute through 13,500 feet of 10" pipe to a 10,000 gallon mill head tank situated uphill from the mill. During the period from May to December, one or two of these pumps could be shut down and process and domestic water requirements could be supplemented by water pumped from the Little Chief reservoir which would be filled from Parker Lake through a diversion ditch. A diesel fire pump drawing from the Little Chief reservoir would be installed as fire protection for the plant.

Because the cost of pumping process water from the Yukon River would be high, alternative sources of water have been studied. Field investigation and stream measurements will be necessary before any of these sources can be considered as either replacing or supplementing the Yukon River system.

Although the annual rainfall at Whitehorse is only ten inches, the reduced transpiration and evaporation in this area may result in useful sources of water from small streams or groundwater. Small streams do not flow between January and April and a water supply drawn from these streams must be combined with a reservoir of sufficient storage to supply the plant requirements for a four month period. It is proposed to build a road over fill across the north end of Little Chief Lake to provide access to the pits to the north west of the plant. This earth fill dam will form a reservoir in the Little Chief Lake which could supply the plant for 30 days at a rate of 900 g.p.m.

This Little Chief reservoir could be filled from the Parker Lake system by a diversion ditch from Mill Feed Lake but the supply of water from Parker Lake would be inadequate to meet the total plant requirements throughout the year. This reservoir could serve as a supplementary supply of water to the Yukon River system or to an alternative ground water system.

Drainage to Parker Lake could be increased by diverting the creek 3 miles south of the plant site but until stream flow measurements of these creeks are made during the winter months, it is not known whether this system could provide for all the plant requirements on a year round basis.

Because Mill Feed Lake is at a higher elevation than the plant, pumping from the Parker Lake system would not be necessary, operating costs would be low, and matching of supply to a fluctuating demand would be simple. If winter stream flow measurements indicate that this system could supply all the plant requirements on a year round basis, a pipeline from Mill Feed Lake would be preferable to a diversion ditch, because the pressure head available from Mill Feed Lake would be adequate for fire protection as well as the mill water pressure requirements. In this case it would be necessary to size the pipeline from Mill Feed Lake to meet the maximum fire demand plus the process water demand.

There are indications that groundwater may be available near the plant site but it is not certain whether the groundwater aquifer has sufficient storage to supply all the plant requirements on a year round basis. Because development of an adequate groundwater supply would provide the least costly year round supply of water for the plant, drilling tests for groundwater should be conducted as soon as possible. If, however, sufficient groundwater supplies cannot be developed, it would be necessary to revert to the Yukon River as a dependable year round source of water. It is recommended that not more than $20,000 be spent in field investigations of ground water supplies, because unless an adequate source of ground water can be developed to replace the Yukon River supply, costly investigation of ground water supplies will not be warranted.

WATER SUPPLY AND FIRE PROTECTION

Equipment	Weight lbs.	Cost $
3 - 350 IGPM 75 H.P. Vertical Pumps	12,000	$ 10,500
13,500 ft. 10" Ductile Iron Pipe	428,000	53,500
1 - 50 H.P. Diesel Fire Pump	12,000	7,500
1 - 50 H.P. Electric Fire Pump	4,000	3,000
2 - 10 H.P. Vertical Pumps Jockey & Mill	4,000	1,800
10,000 gallon Mill Tank	5,000	2,400
1000 ft. 8" Ductile Iron Pipe	23,000	3,000
Valves, hydrants and fittings (allowance)	27,000	11,500
Sub-Total -- To Summary Sheet	515,000	$ 93,200

Construction & Installation	Quantity	@ $	Cost $
Yukon River Pumphouse			$13,000
Installation of Insulated Pipeline from Yukon River	13,500 ft.	5	67,500
Reservoir Pumphouse			11,000
Mill Tank Installation and piping			12,500
Mill Feed Lake Diversion Ditch			15,000
Freight and Handling	258 tons	80	20,600
Federal Tax - Equipment	$93,200	11%	10,300
- Construction	$104,000	5 1/2%	5,200
Electrical Equipment and Installation including freight and tax (from Section VI) (excluding lighting and heating)			43,200
Sub-Total -- To Summary Sheet			$198,300

TAILINGS DISPOSAL

It is proposed to store tailings in the shallow valley just east of the plant by constructing dams to a crest elevation of 2550 feet from overburden stripped from the Little Chief orebody. This tailings basin would be adequate to store tailings for about five years at the production rate of 1950 tons per day. An additional five years of storage could be attained by building up the dams to an elevation of 2570 feet. It is assumed that these dams will be raised either with classified sands from the tailings or with locally available fill. Because the cost of raising the dams will be an operating cost which will be incurred several years after the start of production, these costs are not included in the initial capital cost estimate.

Two alternative areas for tailings storage were studied. The canyon 4000 feet east of the plant site would require a dam at least 85 feet high to attain 5-1/2 years of storage. The canyon 5000 feet north east of the plant site would require a dam 60 feet high to attain 6 years of storage. In both these areas, the volume of fill required to build the dams, and the distance required to transport the fill available from stripping the Little Chief orebody made these areas less desirable for tailings disposal than the shallow valley east of the plant. If it becomes feasible in the future to reclaim the tailings for iron recovery,or to reclaim the tailings water, the shallow valley east of the plant would be much more desirable for tailings storage than the canyons because of the shorter pumping distance and the lower pumping pressure required.

The initial development of the proposed tailings area in the shallow valley would not be costly because the tailings pipeline would be short, and because the additional haulage distance required to haul the overburden from the Little Chief orebody to the dams would not appreciably increase haulage costs. The attached cost estimate for building the dams from pit overburden includes only the extra costs of hauling, spreading, and compacting overburden at the dams instead of dumping the pit overburden in the most suitable area adjacent to the pit. A soil study should be made along the site of the proposed dams to ascertain the suitability of the subsoil for dam construction. Because of the relatively low dam heights required for the initial tailings area, it is not expected that suitable subsoil conditions will be difficult to attain in the general area of the dams.

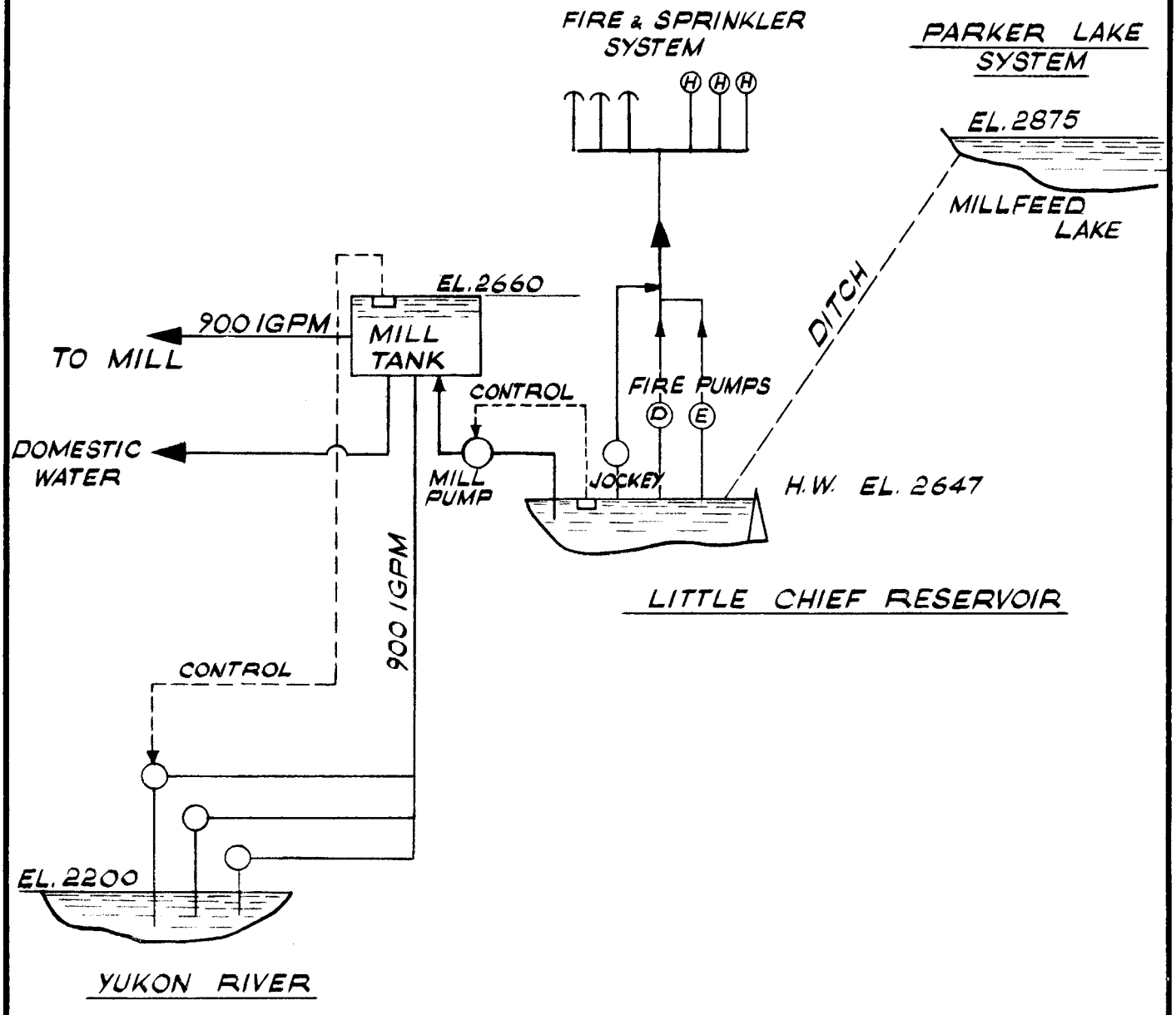
The ground slope east of the mill site is suitable for gravity drainage of tailings through trestle supported wood stave pipe. Water in the tailings pond would be decanted through concrete decantation towers to a 16 inch reinforced concrete pipe laid on the valley floor. This concrete pipe would decant the tailings water through the base of the dam and discharge it into the canyon north of the tailings area. Any fine solids that might be carried into the decant pipe would settle out in the canyon where they would be inoffensive.

TAILINGS DISPOSAL PHASE (1)

Materials	Quantity	@ $	Cost $
12" dia. Wood Stave Pipe	1000 ft.	$ 3.02	$3,020
Reinforced Concrete Pipe 16"ø	800 ft.	$ 2.20	1,760
Concrete Pipe 20"ø	100 ft.	$ 3.70	370
Add Federal Tax			600
Add Freight	45 tons	$80.00	3,440
Sub-Total - to Summary Sheet			$9,190

Construction & Installation

Clearing	3.2 acres	$1000.00	$3,200
Concrete Decantation Tower	15 cu.yds	$ 110.00	1,650
Trestling allowance			8,000
Laying Wood Stave Pipe	1000 ft.	$ 2.60	2,600
Laying Concrete Pipe	800 ft.	$ 4.00	3,200
Decantation Tower Boxes	14 MBF	$ 1.50	2,100
Stripping and Scraping	3500 cu.yds	.50	1,750
Transporting Fill	180,000 tons	0.19	32,500
Spreading and Compacting Fill	180,000 tons	.12	21,600
Tax Allowance			1,000
Sub-Total - To Summary Sheet			$77,600



	NEW IMPERIAL MINES LTD. WHITEHORSE, YUKON.	WATER SUPPLY SYSTEM. SCHEME No. 1
	WRIGHT ENGINEERS LIMITED VANCOUVER —— CANADA	DRAWN BY C.Mc. DATE Nov. 1965
		DRAWING NO. 402 028 001

DWG. NO. 402 028 002

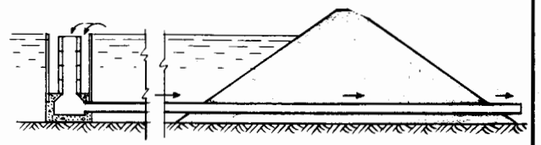


DIAGRAM SHOWING
DECANT TOWER & DAM

2400
2450
2500
2550
2600

3 FUTURE TAILING AREA (6 YEARS)

DECANT TOWERS

1 TAILING AREA (3 TO 5 YEARS)

2 FUTURE TAILING AREA (4 YEARS)

WHITEHORSE

MINE PLANT

N

0 400 800
FEET

DESCRIPTION OF REVISION	APPR.	DESIGN	DRAWN	CHECK	FINAL CHECK
ISSUED FOR REPORT					

SCALE:

REFERENCE | DWG. No.

LOCATION OF
PROBABLE TAILING AREAS

NEW IMPERIAL MINES,
WHITEHORSE, YUKON

DWG. No. 402 028 002

WRIGHT ENGINEERS LIMITED
VANCOUVER —— CANADA

SECTION IX
ADMINISTRATION AND GENERAL COSTS

ADMINISTRATION AND GENERAL

SUMMARY OF ORGANIZATION AND PERSONNEL

ADMINISTRATION	MINE	CONCENTRATING	MAINTENANCE
Accounting	Pit Operation	Primary Crushing	Maintenance Shop
Warehousing	Waste Removal	Secondary Crushing	Electrical Shop
	Engineering	Coarse Ore Storage	
		Fine Ore Storage	
		Grinding	
		Flotation	
		Concentrate Loading	
		Tailing Disposal	
		Assaying	
		Water Supply	

SALARIED STAFF

Manager	Mine Superintend.	Superintendent	Plant Superintend.
Accountant	Chief Geologist	Metallurgist	Foreman
Asst. Accountant	Geologist	Foreman	Chief Electrician
Timekeeper	Pit Engineer	Shift Bosses (4)	
Warehouseman	Surveyor	Chief Chemist	
Clerk	Shift Bosses (3)	Chemist	
Stenographer	Survey Helpers(2)		
Janitor	Sorters (2)		
First-Aid & Personnel	Samplers (2)		
Total - 9.	Total - 14.	Total - 9.	Total - 3.

Total Salaried Staff - 35.

HOURLY PAID PERSONNEL

MINE		CONCENTRATING		MAINTENANCE	
Shovel Operator	6	Crusher Operators	2	Machinist	1
Shovel Oilers	6	Crusher Helpers	2	Welder	1
Blaster Scaler	2	Mill Operators	4	Electrician 1st Class	1
Drill Operators	5	Flotation Operators	4	Electrician 2nd Class	1
Air Truck Operators	1	Concentrate Handlers)	4	Mechanic 1st Class	1
Truck Drivers	12	Reagent & Cells)		Mechanic 2nd Class	1
Service Truck Driver	2	Filter & Dryer Handling		Truck Driver	1
Grader Operator	2	Shift Bosses		Helper	1
Front-End Loader Op.	3	Maintenance 1st Class	2	Loader Operator	1
Helpers	9	Maintenance 2nd Class	2	Labourer	3
Mechanics 1st Class	3	Labourers	4		
2nd Class	1				
3rd Class	1				
Welder 1st Class	1				
Total	54	Total	24	Total	12

Total Personnel

Salaried	35
Hourly Paid	90
Total	125

CAPITAL COST ESTIMATE

The estimate includes equipment that cannot be allocated to a particular section of the operation, an allowance for spare parts and warehouse inventory and an allowance for assistance to personnel for housing. Office, garage and machine shop equipment is costed in Section V.

The spare parts and warehouse inventory cannot be costed accurately and the allowance is based on published figures for similar operations. It includes $100,000 for mine requirements. Small items will be available from Whitehorse at short notice and large stocks need not be carried.

The costs include construction of three houses for senior personnel and an advancement for the initial mortgage payment for married personnel. Allowance has been made in the operating cost for relocation of single personnel.

The front end loader is intended mainly for loading concentrate into containers but would be available for general work around the mill complex.

	Cost $
Warehouse inventory and spare parts (allowance)	300,000
Company houses 3 @ $20,000	60,000
Housing loan to married personnel 40 @ $900	36,000
Pick up truck	4,300
Jeep	2,800
5 Ton Truck	7,200
1 cu.yd. Front End Loader	21,800
Total (to summary)	$ 432,100

OPERATING COST ESTIMATE

The estimate includes the personnel costs of administrative and general maintenance services that cannot be allocated to a particular section of the operation. General maintenance and service supplies are also included.

No cost is included for head office expenses and it is assumed that the Vancouver office will be closed on start up of operations.

Compensation and silicosis are handled by the employers liability insurance and rates similar to Canada Tungsten in 1964 are assumed.

It is assumed that some assistance in relocation expenses will be given to single personnel to attract suitable men.

Power and heat costs are for the general office, garage, maintenance building, and mine office.

Summary

		$/Yr
Administration and General Office		106,150
Fringe Benefits		108,000
General Maintenance and Services		127,860
Engineering		58,200
Total.		400,210
Cost per ton milled on 720,000 tons	=	55.6¢ton
Cost per ton milled on 650,000 tons (1st yr.)	=	61.6¢ton

ADMINISTRATION

	Cost/Month $	Cost/Year $	
1 Manager	1,500	17,000	
1 Chief Accountant	775	9,300	

GENERAL OFFICE

1 Assistant Accountant	600	7,200	
1 Warehouse Man	500	6,000	
1 Timekeeper	450	5,400	
1 Clerk	325	4,000	
1 First-Aid & Personnel	550	6,600	
1 Stenographer	300	3,600	
1 Janitor	300	3,600	
Sub Total			$62,700

OFFICE & ADMINISTRATION EXPENSES

	Cost/Year $	
Stationery and Office Supplies	6,000	
Jeep	1,250	
Telephone & Telegraph	7,200	
Permits, Licences & Fees	3,500	
Property Taxes	3,500	
Insurance	14,000	
Travelling Expenses	5,000	
Legal & Audit	3,000	
Sub Total		$ 43,450
Total to Summary Sheet		$106,150

FRINGE BENEFITS

	$	
Holidays 4% wage payroll	19,000	
Statutory Holidays 3.2%	15,200	
Overtime 10% Mechanical & Electrical Payroll	10,300	
Shift Differential	6,000	
Group Insurance	2,500	
Medical Scheme Contributions	5,000	
Hospitalization Scheme	5,000	
Unemployment Insurance	4,200	
Safety & First-Aid Supplies	1,500	
Silicosis Assessment) 4.15% of total pay-	29,300	
Workmens Compensation) roll		
Relocation Allowance for single personnel	10,000	
Total to Summary Sheet		$ 108,000

ENGINEERING STAFF

	Cost/Month $	Cost/Year $
1 Chief Geologist	800	9,600
1 Geologist	700	8,400
1 Pit Engineer	700	8,400
1 Surveyor	550	6,600
2 Helpers	700	8,400
2 Sorters	700	8,400
2 Samplers	700	8,400
Total to Summary Sheet		$58,200

GENERAL MAINTENANCE AND SERVICE

	Cost/Month $	Cost/Year $
1 Plant Superintendent	1,000	12,000
1 Foreman	700	8,400
1 Chief Electrician	700	8,400
1 Machinist	@ 2.92 $/Hr.	5,840
1 Welder	@ 2.92. $/Hr.	5,840
1 Electrician 1st Class	@ 3.10 $/Hr.	6,200
1 Mechanic 1st Class	@ 2.92 $/Hr.	5,840
1 Mechanic 2nd Class	@ 2.54 $/Hr.	5,080
1 Electrician 2nd Class	@ 2.54 $/Hr.	5,080
1 Truck Driver	@ 2.63 $/Hr.	5,260
1 Helper	@ 2.20 $/Hr.	4,400
1 Loader Driver	@ 2.76 $/Hr.	5,520
3 Labourers	@ 2.20 $/Hr.	13,200
Sub-Total		$91,060

GENERAL MAINTENANCE AND SERVICE SUPPLIES

	Cost/Year $	
Mechanical Supplies	14,700	
Pick-Up Truck	1,400	
5 Ton Truck	2,000	
Tractor	2,500	
Electrical Supplies	8,200	
Power and Heat	5,000	
Engineering Supplies	3,000	
Sub-Total		$36,800
Total to Summary Sheet		$127,860

Project No. 402

By:

DR. A.C. SKERL, P. Eng.,
Consulting Mining Geologist.

J.A.C. ROSS, P. Eng.,
Consulting Mining Engineer.

WRIGHT ENGINEERS LIMITED,

M.I. CALLOW.

T.A. O'HARA, P. Eng.

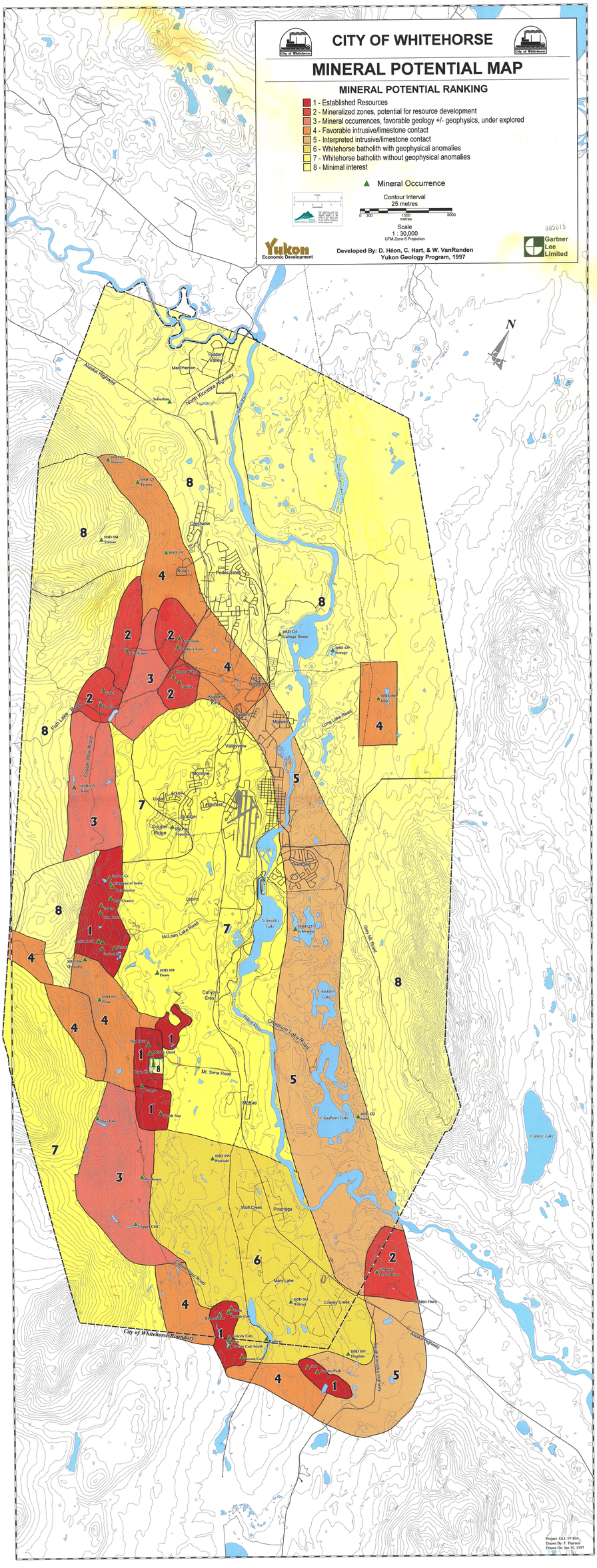
Vancouver, British Columbia,
November 17, 1965.

APPENDIX IV

MINERAL POTENTIAL MAP 1997

AND

ACCOMPANYING ASSESSMENT DOCUMENTS


CITY OF WHITEHORSE
MINERAL POTENTIAL MAP
MINERAL POTENTIAL RANKING
1 - Established Resources
2 - Mineralized zones, potential for resource development
3 - Mineral occurrences, favorable geology +/- geophysics, under explored
4 - Favorable intrusive/limestone contact
5 - Interpreted intrusive/limestone contact
6 - Whitehorse batholith with geophysical anomalies
7 - Whitehorse batholith without geophysical anomalies
8 - Minimal interest
Mineral Occurrence
Contour Interval
25 metres
0 1500 3000
metres
Scale
1 : 30,000
UTM Zone 8 Projection
005015
Yukon
Economic Development
Developed By: D. Héon, C. Hart, & W. VanRanden
Yukon Geology Program, 1997
Gartner Lee Limited
N
Alaska Highway
North Klondike Highway
Hidden Valley
MacPherson
Takhini River
Takhini
Forestdale
Kulan
Copper Haul Road
Fish Lake Road
Long Lake Road
Valleyview
Mountview
Logan
Granger
Copper Ridge
Downtown
Riverdale
Porter Creek
Upland
McLean Lake Road
Mt. Sima Road
Chadburn Lake Road
Crestview
Wolf Creek
Pineridge
Mary Lake
Golden Horn
Copper Ridge Road
Alaska Highway
City of Whitehorse Boundary
Scwatka Lake
Chadburn Lake
Cassiar Lake
Project: GLL 97-804
Drawn By: F. Pearson
Drawn On: Jan 30, 1997

DH wrote this

004871

MINERAL ASSESSMENT – CITY OF WHITEHORSE

INTRODUCTION

This mineral assessment was undertaken for the City of Whitehorse to provide them with a tool to incorporate mineral resource values in their land use planning. A review of the available geoscientific information included: existing geological maps, government reports and assessment reports. Private archival documents (maps, exploration summaries, etc) were graciously provided by Hudson Bay Exploration and Development (HBED), the current owners of the old Whitehorse Copper Mines (WCM) ground and the largest land holders in the Copper Belt.

Most of the Whitehorse Copper Belt is included within the city limits. This 32 km long belt is defined by a linear series of copper skarn occurrences. This type of mineralization forms preferentially where the Cretaceous Whitehorse batholith intrudes the partially dolomitized limestones and clastic sediments of the Triassic Lewes River Group. Economic deposits form where irregularities in the contact produce pendants or embayments of sediments within the intrusive. Of all the deposits and showings in the Whitehorse Copper Belt, 28 have been mined intermittently from the turn of the century till the early 1980's. It is estimated that current reserves in the Copper Belt stand at 3 million tons of ore grading 1% copper.

METHODOLOGY

Geological and geophysical data as well as information on mineral occurrences were compiled in preparation for the assessment. The geochemical data was not compiled since the locally thick overburden masks the geochemical signature of the deposits.

A digital geology map at a scale to 1:20 000 was compiled by Craig Hart. The detailed WCM geology maps of various scales were incorporated to the pre-existing 1:50 000 regional mapping and the resulting linework was registered to a topographic base provided by the city. The computer drafting was done by Will van Randen.

All available geophysical data was provided to Amerok Geosciences Ltd. It was established that the compilation of all this data would be an immense job, well beyond the time and budget limitations of this project. Such detailed information could be studied on a case by case basis if detailed information about a specific deposit was required. The scope of this project was, therefore, better served by an overall look at the most recent surveys. An airborne electromagnetic (EM) survey flown in 1982 by and an induced polarization (IP) survey conducted in 1975 were chosen as the most recent, pertinent surveys with the best coverage.

This choice turned out to have severe limitations. The parameters for the EM survey as well as the original data were unavailable and the hand-drawn plots of the associated magnetic survey were of little use. The available plots were interpretations of EM data and coincident magnetic anomalies, there was little overlap between that survey and the 1975 IP survey. Nevertheless, tabulation and ranking of areas of geophysical interest was achieved by interpreting the clustering of anomalies, and noting if they had any magnetic coincidence or a favorable geological setting. IP surveys are the best geophysical tool to locate skarn deposits; the most recent surveys would need to be compiled at a property wide-scale in order to provide a regional perspective.

Much time was spent in trying to register both the geological and geophysical data to actual topographic coordinates.

Mineral occurrence information was compiled from: Minfile, existing geological maps, published reports by Dave Tenney and Pat Watson (DIAND), assessment reports, theses, private studies, internal property descriptions, and exploration summaries and proposals. A summary of the most recent exploration targets and results was tabulated for each area/deposit. It was concluded that outside of the deposits that were actively mined, the level of exploration was at a quite preliminary stage.

ASSESSMENT PANEL

On October 16, 1997 an expert panel was convened to assess the relative mineral potential of tracts within the City of Whitehorse. Members of the panel were: Dave Tenney, former chief geologist for Whitehorse Copper Mines; Gerry Bidwell and Robert Stroshein, both employees of HBED at the time of that company's exploration activity in the Copper Belt; Craig Hart, project geologist with the Yukon Geology Program. All of these participants have extensive expertise with the geology and mineral deposits of the Whitehorse area. Other panelists included Jeff Mitchell, geophysicist contracted through Amerok Geosciences to evaluate the geophysical database and Danièle Héon, mineral assessment geologist with YTG. Andy Hureau, former exploration geologist for WCM, was reached by phone at the end of the session and was asked to comment and to supply additional information.

The task of the panel was to divide the study area into tracts of similar geology and then rank those tracts on the basis of their relative potential to host metallic mineral deposits. The potential for limestone was not evaluated due to the lack of information and general expertise within the group. None of the other industrial minerals was thought to occur in significant quantities within the city limits.

Information relevant to each mineral deposit/area was reviewed. The study area was then divided into tracts which are numbered from 1 to 36. Tract definition was influenced by the types of mineral deposits occurring or expected to occur in each area. Since skarn deposits are the most significant targets within Whitehorse, tracts were drawn overlapping the prospective contact and allowing for the uncertainty of the location of that contact. Through the ranking process, the tracts were grouped in 8 categories of mineral potential. The criteria defining each category and the resulting ranking of tracts are outlined below in table 1.

Table 1.

		ranking	category	tracks
very high potential	highest	1	Established Resources	2, 4, 12, 14, 20, 36
		2	Mineralized zones, potential for resource development	27, 32, 5, 29, 31
		3	Mineral occurrences, favorable geology +/- geophysics, under-explored	22, 28, 8
more work needed		4	Favorable intrusive/ limestone contact	30,3, 26, 6, 16, 15, 18, 33
		5	Interpreted intrusive/limestone contact	10, 24, 1
		6	Whitehorse batholith with geophysical anomalies	7, 23
minimal interest	lowest	7	Whitehorse batholith without geophysical anomalies	11, 9
		8	minimal interest	13, 19, 21, 35, 17, 25, 34

Categories 1 to 3 represent ground of very high mineral potential. Any withdrawal of these tracts from exploration activities is predicted to meet with resistance and criticism from the mineral industry. Tracts belonging in categories 4 through 6 have undergone various levels of exploration; our panel recommends further studies if any of these areas are to be considered for withdrawal. Categories 7 and 8 represent areas of minimal interest to the industry. Withdrawal of these lands is predicted to be the least contentious.

Table 2 lists the tracts in numerical order with their respective ranking.

Tract no.	ranking priority	includes	tract no.	ranking priority	includes
1	5		19	8	
2	1	Cowley Park	20	1	Arctic Chief/ Grafter/ Best Chance
3	4		21	8	
4	1	Gem	22	3	Minfile 105D 055
5	2	Lewes River (Minfile 105D 062)	23	7	
6	4		24	5	
7	6		25	8	
8	3	Copper Cliff, Pass Lake	26	4	
9	7		27	2	Pueblo
10	5		28	3	Reservoir Lake
11	7		29	2	Copper King, Carlisle
12	1	North Star	30	4	
13	8	mined out Little Chief pit	31	2	Rabbit's Foot
14	1	Middle Chief	32	2	War Eagle
15	4		33	4	
16	4		34	8	
17	8		35	8	
18	4		36	1	Tailings compound

It must be emphasized that these conclusions are the best estimate at the time of the assessment. Conclusions should be revised with time as new discoveries are made within the City and elsewhere.

Danièle Héon
Mineral Assessment Geologist
October 20 1997

004900

Mr. Rob Roycroft
Director, Community Services
City of Whitehorse
2121 Second Avenue
Whitehorse, Yukon
Y1A 1C2

October 20, 1997

Re: City of Whitehorse- Mineral Resource Assessment

Dear Mr. Roycroft,

Please find enclosed a disk containing the mineral potential map of the City of Whitehorse as well as a table listing the tracts and their respective ranking.

The study area is divided into geological "tracts" which are polygons of similar geology. These tracts are numbered from 1 to 36 and the ID number appears in the polygon on the map, along with the known mineral occurrences. Throughout the assessment process (see attachment), the tracts were grouped in 8 categories of mineral potential, with category 1 being the highest potential and category 8 the lowest. This ranking is not portrayed on the map. The Autocad file is arranged in layers, one layer for each level of ranking. Layer 1 contains tracts of ranking category "1", and layer 8 contains tracts of ranking category "8". This data should be manipulated by your drafting staff in order to be incorporated and displayed appropriately in your GIS.

Categories 1 to 3 represent ground of very high mineral potential. Any withdrawal of these tracts from exploration activities is predicted to meet with resistance and criticism from the mineral industry. Tracts belonging in categories 4 through 6 have undergone various levels of exploration; our panel recommends further studies if any of these areas are to be considered for withdrawal. Categories 7 and 8 represent areas of minimal interest to the industry. Withdrawal of these lands is predicted to be the least contentious.

The members of the panel were satisfied that their conclusions represented the best estimation possible of the relative mineral potential of the individual tracts, given the present state of knowledge and the scope of this project. If high potential areas are still targeted for withdrawal, more detailed assessment work will be needed.

Let me know if you need more information concerning this product, and if we can be of further assistance.

Danièle Héon
YTG- Economic Development
Box 2703, F-3
Whitehorse, YT
Y1A 2C6 667-5398

Yukon

MINERAL ASSESSMENT – CITY OF WHITEHORSE

Economic Development
Box 2703, Whitehorse, Yukon Y1A 2C6

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METHODOLOGY

Geological and geophysical data as well as information on mineral occurrences were compiled in preparation for the assessment. The geochemical data was not compiled since the locally thick overburden masks the geochemical signature of the deposits.

A digital geology map at a scale to 1:20 000 was compiled by Craig Hart. The detailed WCM geology maps of various scales were incorporated to the pre-existing 1:50 000 regional mapping and the resulting linework was registered to a topographic base provided by the city. The computer drafting was done by Will van Randen.

All available geophysical data was provided to Amerok Geosciences Ltd. It was established that the compilation of all this data would be an immense job, well beyond the time and budget limitations of this project. Such detailed information could be studied on a case by case basis if detailed information about a specific deposit was required. The scope of this project was, therefore, better served by an overall look at the most recent surveys. An airborne electromagnetic (EM) survey flown in 1982 by and an induced polarization (IP) survey conducted in 1975 were chosen as the most recent, pertinent surveys with the best coverage.

This choice turned out to have severe limitations. The parameters for the EM survey as well as the original data were unavailable and the hand-drawn plots of the associated magnetic survey were of little use. The available plots were interpretations of EM data and coincident magnetic anomalies, there was little overlap between that survey and the 1975 IP survey. Nevertheless, tabulation and ranking of areas of geophysical interest was achieved by interpreting the clustering of anomalies, and noting if they had any magnetic coincidence or a favorable geological setting. IP surveys are the best geophysical tool to locate skarn deposits; the most recent surveys would need to be compiled at a property wide-scale in order to provide a regional perspective.



KLONDIKE
GOLD RUSH
CENTENNIAL

Much time was spent in trying to register both the geological and geophysical data to actual topographic coordinates.

Mineral occurrence information was compiled from: Minfile, existing geological maps, published reports by Dave Tenney and Pat Watson (DIAND), assessment reports, theses, private studies, internal property descriptions, and exploration summaries and proposals. A summary of the most recent exploration targets and results was tabulated for each area/deposit. It was concluded that outside of the deposits that were actively mined, the level of exploration was at a quite preliminary stage.

ASSESSMENT PANEL

On October 16, 1997 an expert panel was convened to assess the relative mineral potential of tracts within the City of Whitehorse. Members of the panel were: Dave Tenney, former chief geologist for Whitehorse Copper Mines; Gerry Bidwell and Robert Stroshein, both employees of HBED at the time of that company's exploration activity in the Copper Belt; Craig Hart, project geologist with the Yukon Geology Program. All of these participants have extensive expertise with the geology and mineral deposits of the Whitehorse area. Other panelists included Jeff Mitchell, geophysicist contracted through Amerok Geosciences to evaluate the geophysical database and Danièle Héon, mineral assessment geologist with YTG. Andy Hureau, former exploration geologist for WCM, was reached by phone at the end of the session and was asked to comment and to supply additional information.

The task of the panel was to divide the study area into tracts of similar geology and then rank those tracts on the basis of their relative potential to host metallic mineral deposits. The potential for limestone was not evaluated due to the lack of information and general expertise within the group. None of the other industrial minerals was thought to occur in significant quantities within the city limits.

Information relevant to each mineral deposit/area was reviewed. The study area was then divided into tracts which are numbered from 1 to 36. Tract definition was influenced by the types of mineral deposits occurring or expected to occur In each area. Since skarn deposits are the most significant targets within Whitehorse, tracts were drawn overlapping the prospective contact and allowing for the uncertainty of the location of that contact. Through the ranking process, the tracts were grouped in 8 categories of mineral potential. The criteria defining each category and the resulting ranking of tracts are outlined below in table 1.

Table 1.

		ranking	category	tracks
very high potential	highest	1	Established Resources	2, 4, 12, 14, 20, 36
		2	Mineralized zones, potential for resource development	27, 32, 5, 29, 31, 3
		3	Mineral occurrences, favorable geology +/- geophysics, under-explored	22, 28, 8
more work needed		4	Favorable Intrusive/ limestone contact	30,3, 6, 16, 15, 18, 33
		5	Interpreted Intrusive/limestone contact	10, 24, 1
		6	Whitehorse batholith with geophysical anomalies	7, 23
minimal interest	lowest	7	Whitehorse batholith without geophysical anomalies	11, 9
		8	minimal interest	13, 19, 21, 35, 17, 25, 34

Categories 1 to 3 represent ground of very high mineral potential. Any withdrawal of these tracts from exploration activities is predicted to meet with resistance and criticism from the mineral industry. Tracts belonging in categories 4 through 6 have undergone various levels of exploration; our panel recommends further studies if any of these areas are to be considered for withdrawal. Categories 7 and 8 represent areas of minimal interest to the industry. Withdrawal of these lands is predicted to be the least contentious.

Table 2 lists the tracts in numerical order with their respective ranking.

tract no.	ranking priority	includes	tract no.	ranking priority	includes
1	5		19	8	
2	1	Cowley Park	20	1	Arctic Chief/ Grafter/ Best Chance
3	4		21	8	
4	1	Gem	22	3	Minfile 105D 055
5	2	Lewes River (Minfile 105D 062)	23	7	
6	4		24	5	
7	6		25	8	
8	3	Copper Cliff, Pass Lake	26	4	
9	7		27	2	Pueblo
10	5		28	3	Reservoir Lake
11	7		29	2	Copper King
12	1	North Star	30	4	
13	8	mined out Little Chief pit	31	2	Rabbit's Foot
14	1	Middle Chief	32	2	War Eagle
15	4		33	4	
16	4		34	8	
17	8		35	8	
18	4		36	1	Tailings compound

It must be emphasized that these conclusions are the best estimate at the time of the assessment. Conclusions should be revised with time as new discoveries are made within the City and elsewhere.

Danièle Héon
Mineral Assessment Geologist
October 20 1997

Tract no	ranking priority	includes
1	5	
2	1	Cowley Park
3	4	
4	1	Gem
5	2	Lewes River (Minfile 105D 062)
6	4	
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8	3	Copper Cliff, Pass Lake
9	7	
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15	4	
16	4	
17	8	
18	4	
19	8	
20	1	Arctic Chief/ Grafter/ Best Chance
21	8	
22	3	Minfile 105D 055
23	7	
24	5	
25	8	
26	4	
27	2	Pueblo
28	3	Reservoir Lake
29	2	Copper King
30	4	
31	2	Rabbit's Foot
32	2	War Eagle
33	4	
34	8	
35	8	
36	1	Tailings compound

Tracts ranked in numerical sequence