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

by

D. TENNEY

Northern Affairs Program

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BULLETIN 1

Geology Section,
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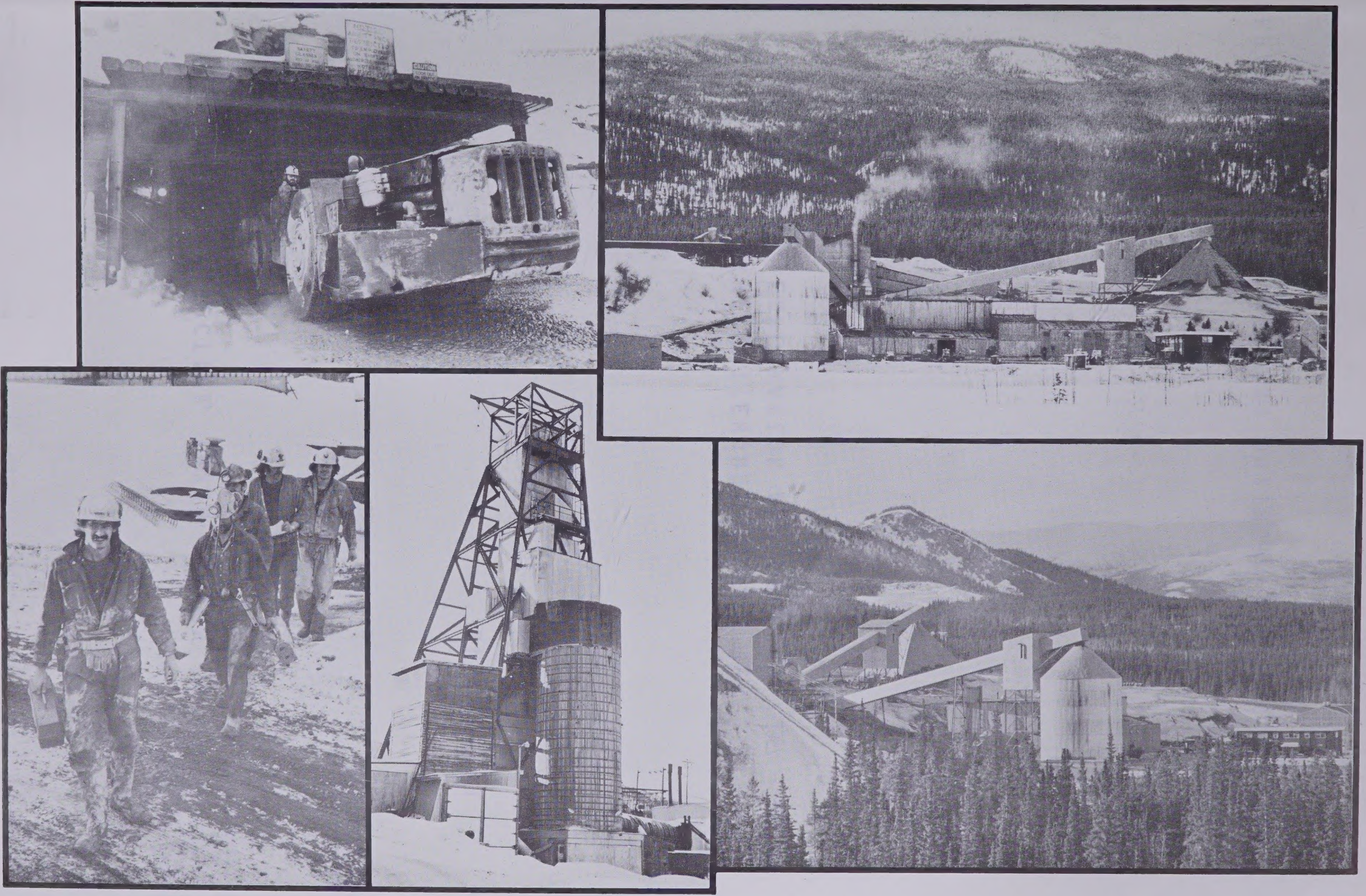
DEPARTMENT OF INDIAN AND NORTHERN AFFAIRS, CANADA



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:
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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.

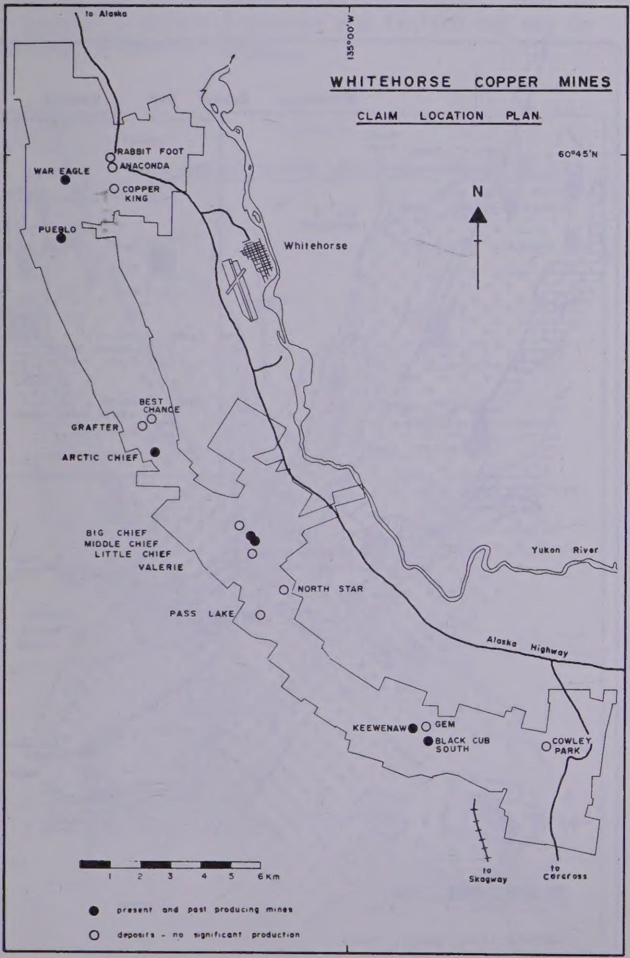

Figure 1. Map of claims held by Whitehorse Copper Mines in 1972 showing the largest copper deposits. The main producers are shown as black circles.

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 handpicked 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.

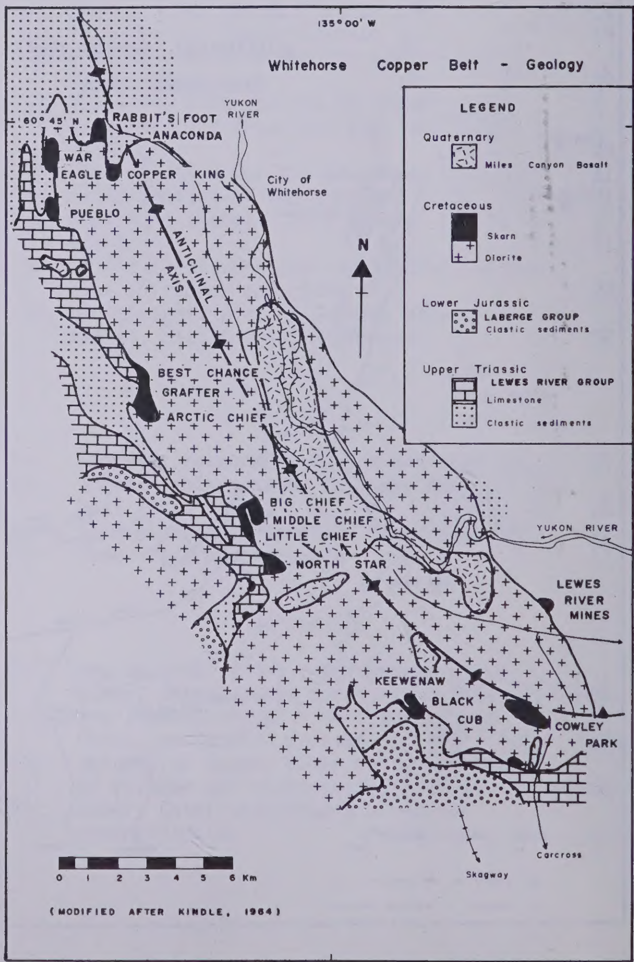

Figure 2. Regional geology of the Whitehorse Copper Belt modified from Kindle (1964).

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.

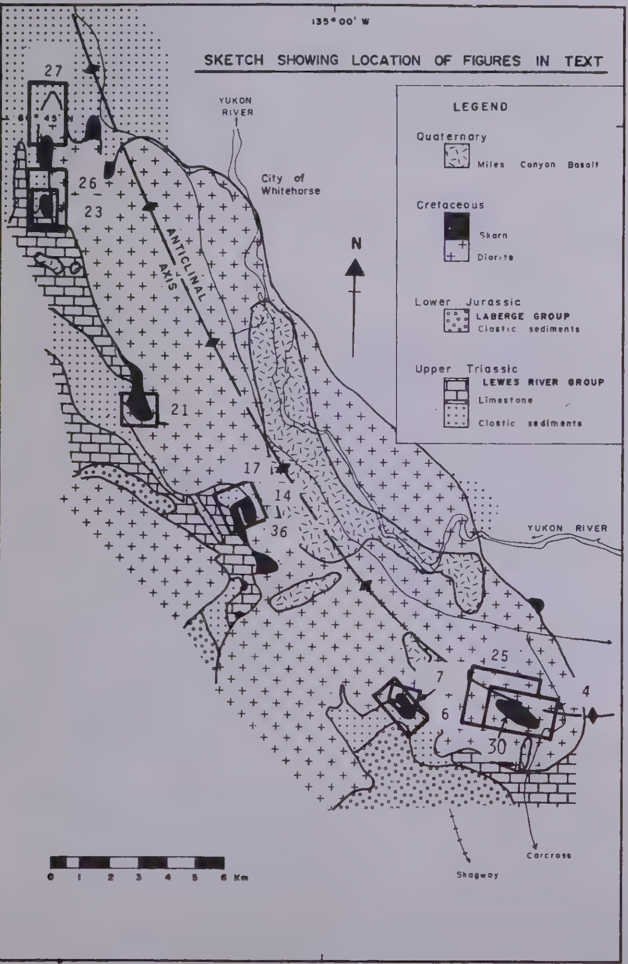

Figure 3. Regional geology of the Whitehorse Copper Belt showing the location of text figures.

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.

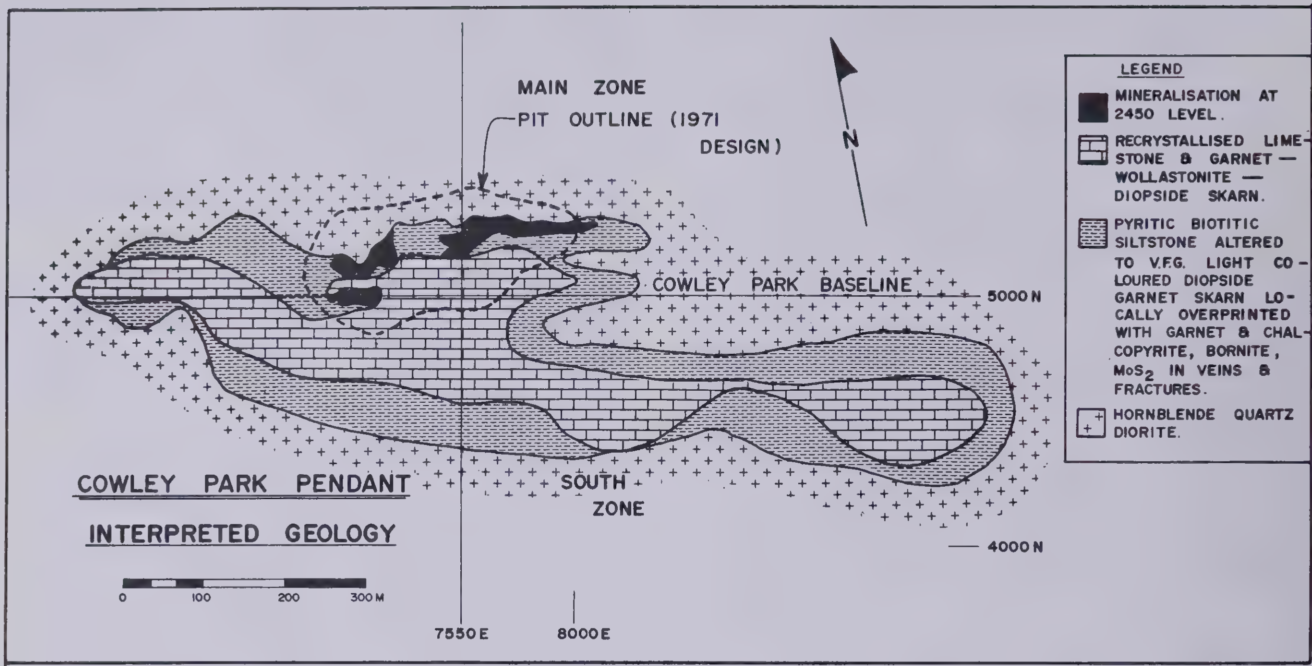

Figure 4. Geological plan of the Cowley Park roof pendant from company plans as reinterpreted by G. Morrison. Note that the Cowley Park South zone, which is the site of a moderately strong induced polarization anomaly, does not crop out at surface (i.e. 2450-2500' elevation).

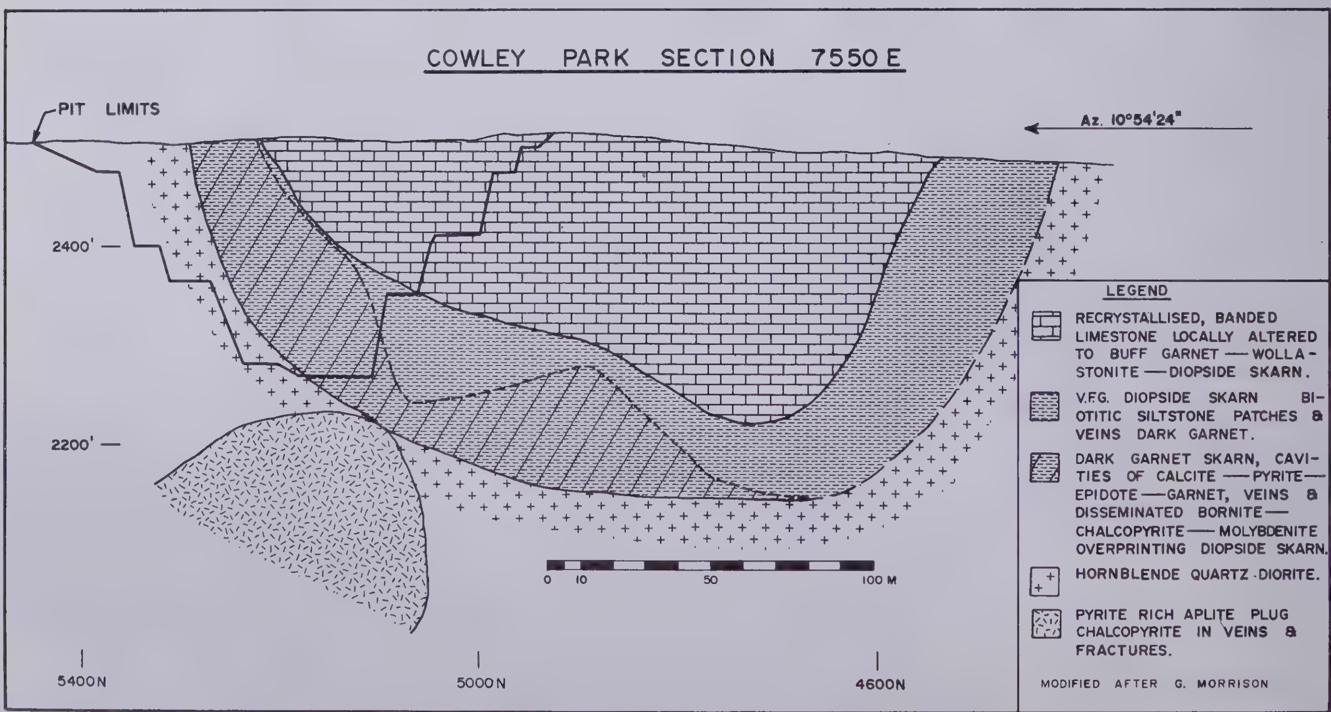

Figure 5. Geological cross-section of the Cowley Park main zone with limits of proposed (1979) pit design. The geology shown is as reinterpreted by G. Morrison.

	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 (MoS_2) 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.

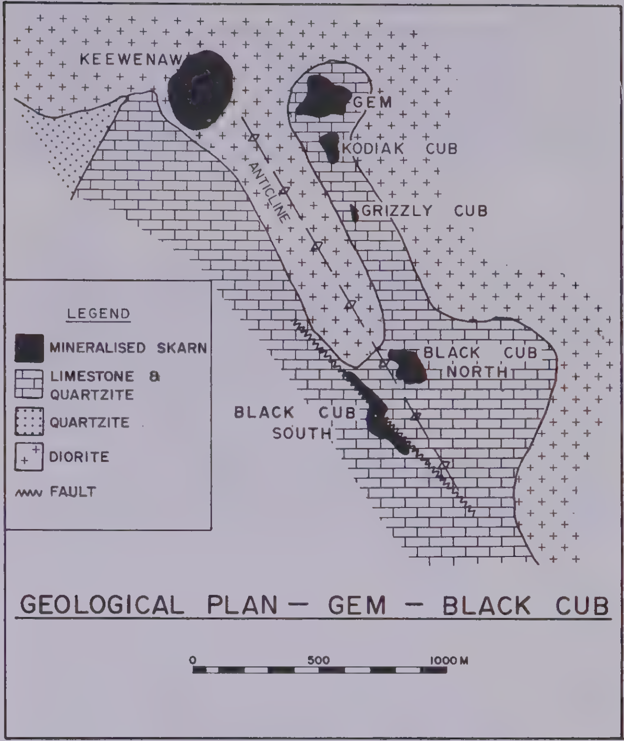

Figure 6. Geological plan of the Keewenaw and Gem-Bear Cub zones. Note the similarity in structure to Little Chief-Big Chief (see figure 17).

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.

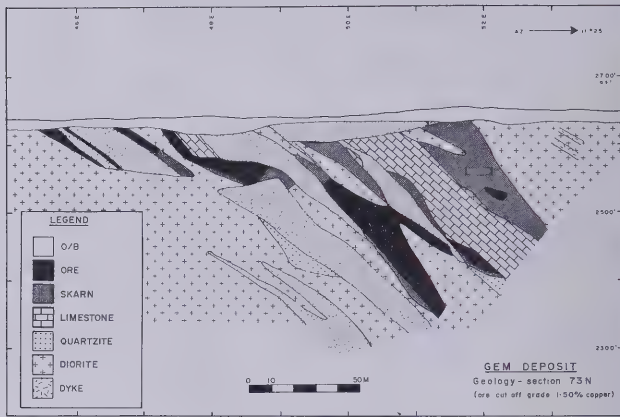

Figure 8. Geological cross-section of the Gem deposit interpreted from surface diamond drilling using a 1.50% copper cut-off grade.

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

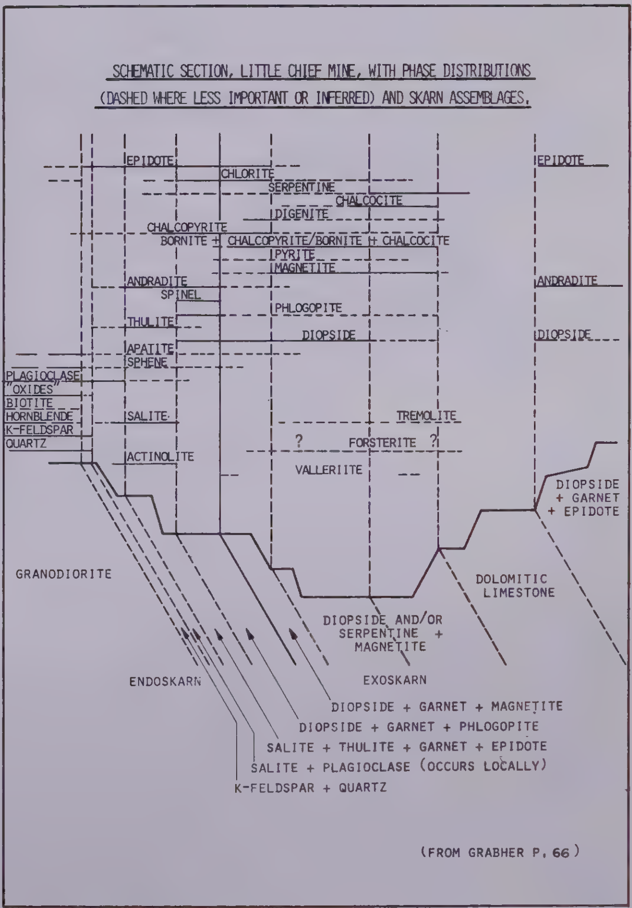

Figure 9. A zoned skarn sequence at Little Chief also showing the pit profile (from Grabher p.66).

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.

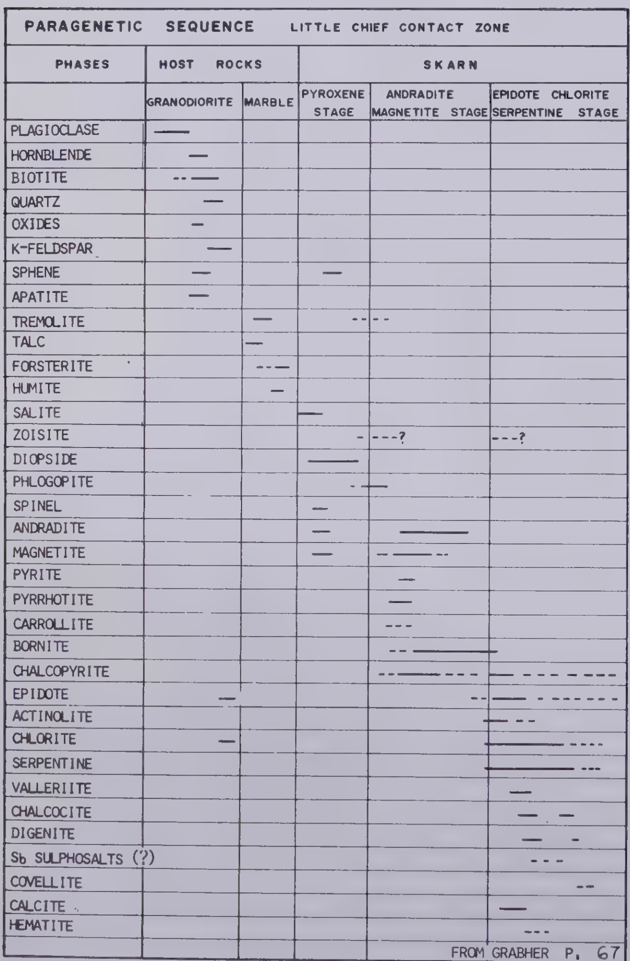

Figure 10. Paragenetic sequence at Little Chief (from Grabher p. 67).

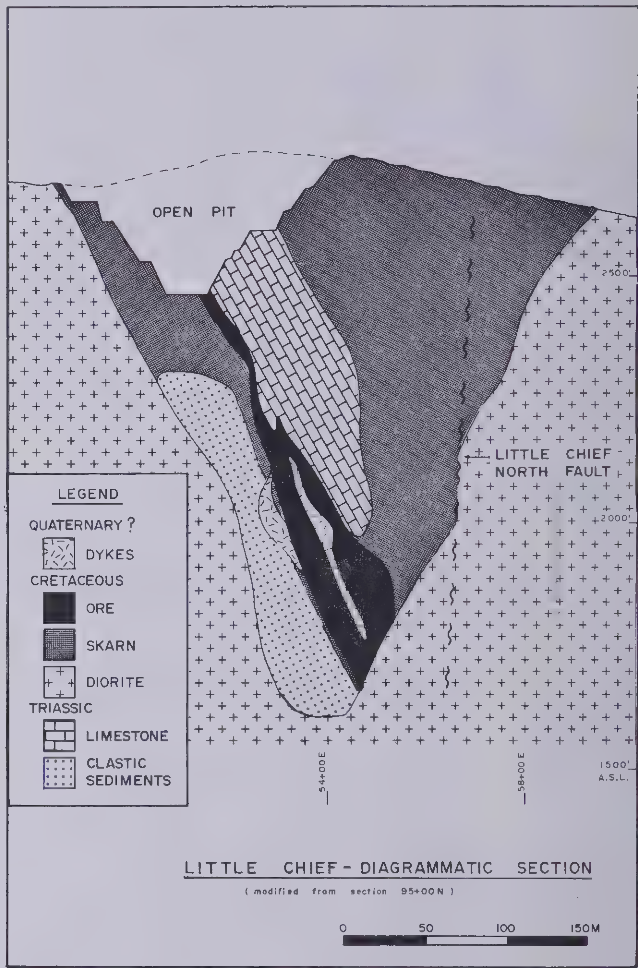

Figure 11. Schematic geological cross-section of the Little Chief flank pendant showing location of the open pit. The footwall side of the ore coincides with the limestone/clastic rocks contact. On the footwall this contact parallels the intrusive contact.

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).

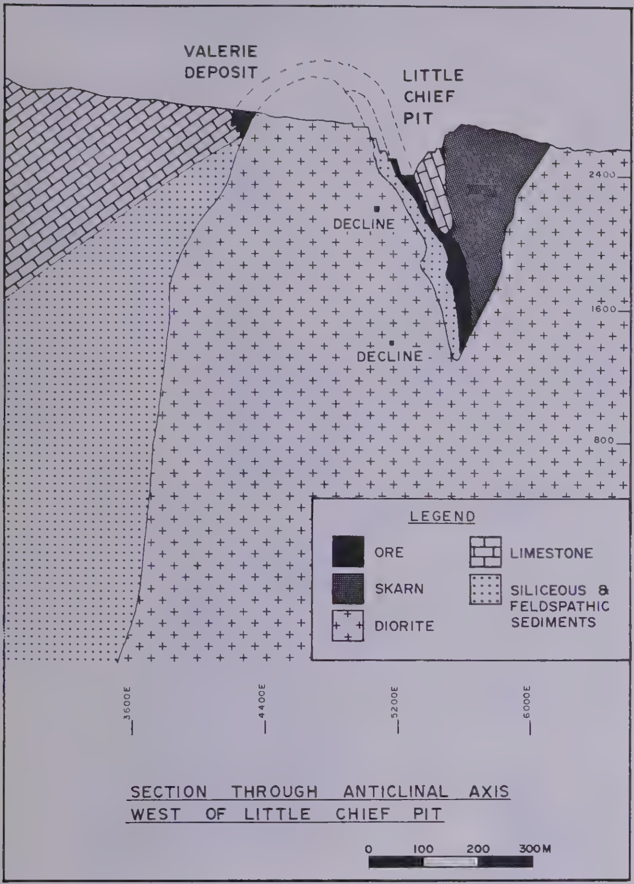

Figure 12. Geological cross-section through the anticline between Little Chief and Valerie. The diorite contact beneath the Valerie showings is vertical which places the Little Chief pendant on the flank of the Whitehorse Batholith. (c.f. Bear Cub deposits Fig. 6).

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

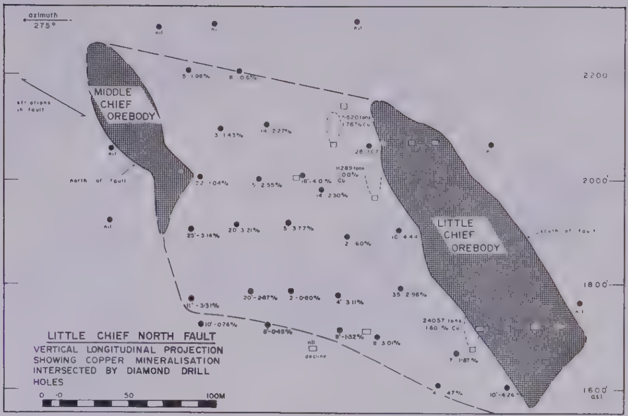

Figure 13. Vertical longitudinal prjection along the Little Chief North Fault (see Fig. 14 & 15) showing the location of ore dragged along the fault between Little Chief and Middle Chief ("B" zone). Some of this ore has been mined by block caving and the grade and tonnage extracted is shown. The direction 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.

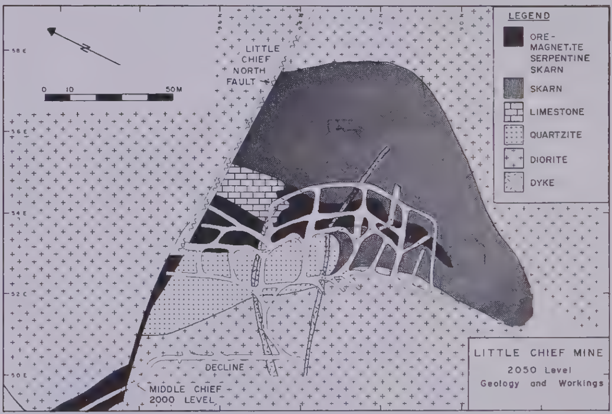

Figure 14. Geological plan of the Little Chief 2050 level showing mine workings and the relationship of "quartzite", ore, skarn and hangingwall limestone in the Little Chief flank pendant. The Middle Chief "B" zone lies north of the Little Chief North Fault. Ore dragged along the fault is also shown.

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.

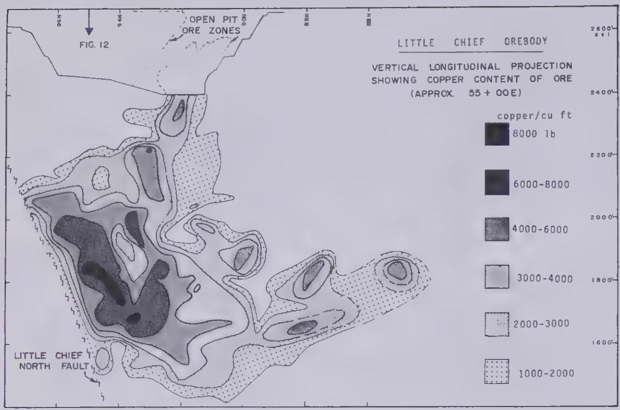

Figure 15. Vertical longitudinal projection of the Little Chief orebody showing the copper content of the ore. This is obtained at the location of each diamond drill hole by multiplying estimated true width of the ore by the grade of the intersection.

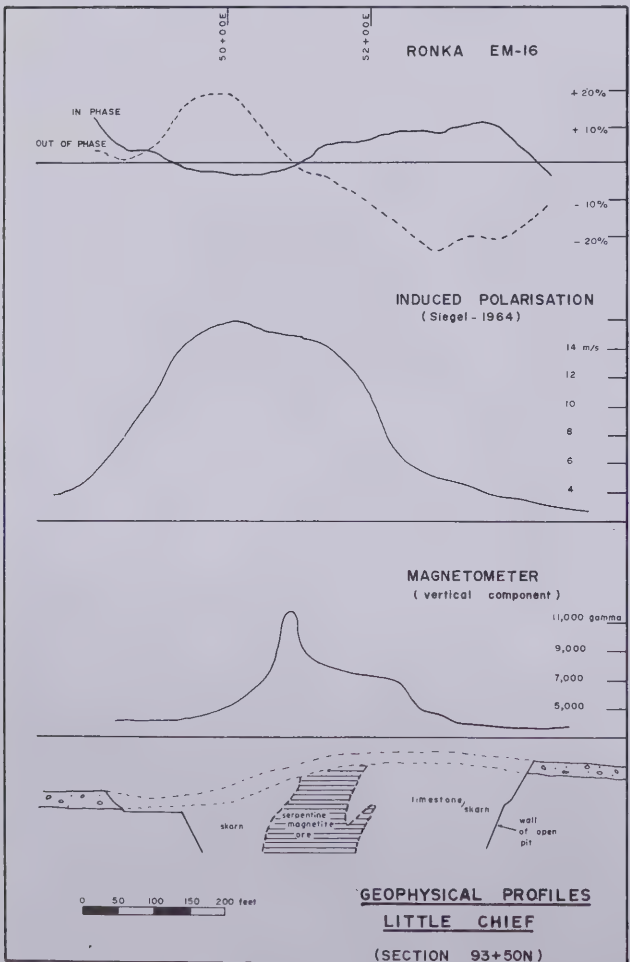

Figure 16. Geophysical profiles across the Little Chief open pit. The magnetometer and IP responses are characteristic. The E.M. - 16 profile is weakly anomalous.

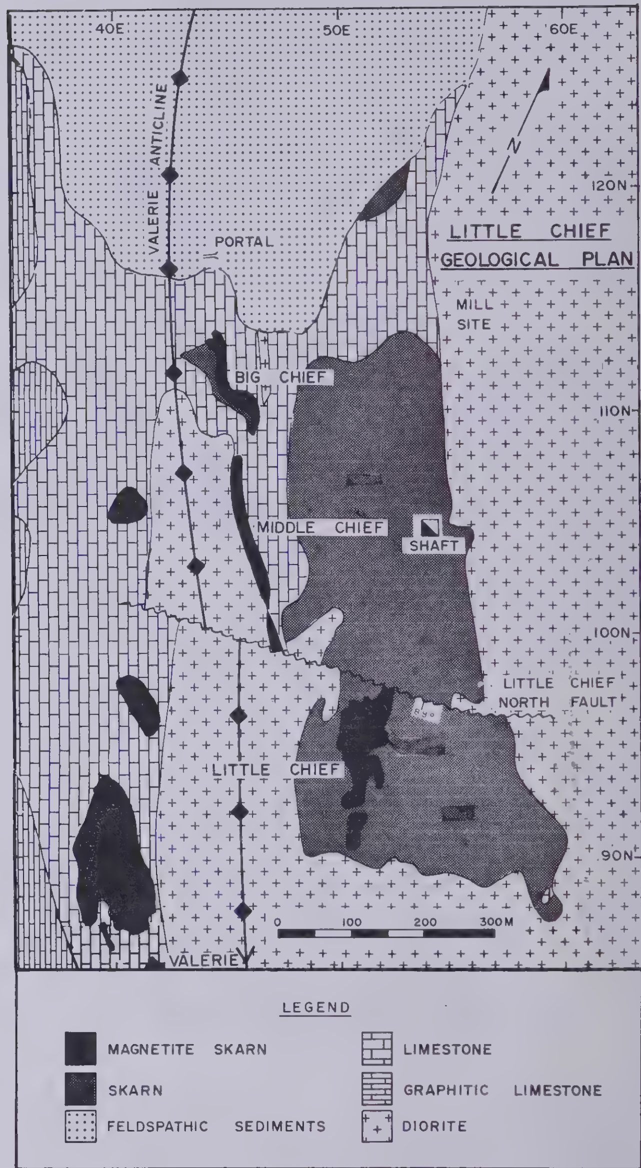

Figure 17. Geological plan of the Little Chief flank pendant, showing the Valerie Anticline. The Valerie serpentine magnetite skarns: they contain less magnetite than Little Chief and more serpentine. Note that the hangingwall limestone at Little Chief does not crop out at surface.

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.

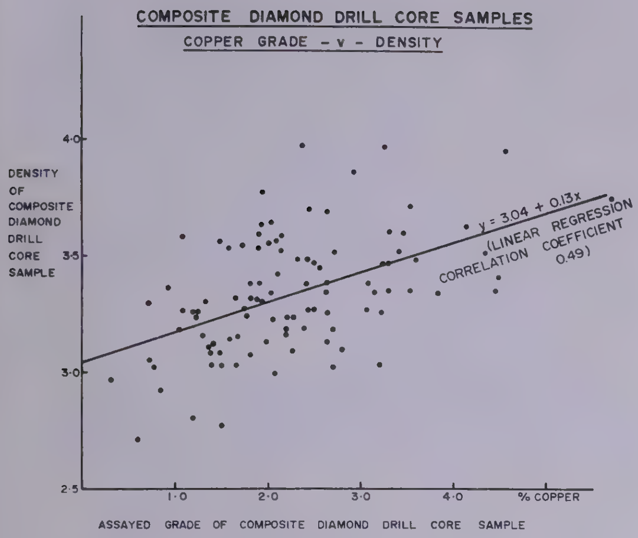

Figure 18. Pot of copper grade against density for composite underground diamond drill core samples. Note the regression line is almost the same as that in figure 32, suggesting that dilution has little influence on density of ore delivered to the mill.

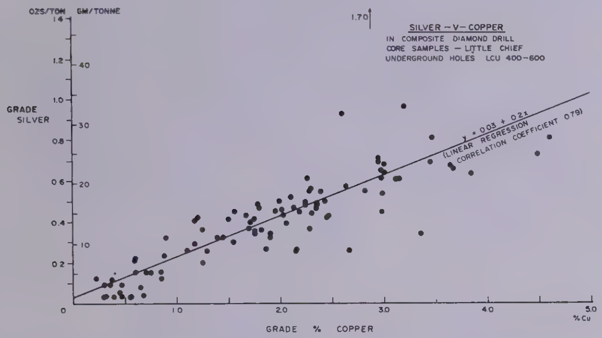

Figure 19. Plot of silver grade against copper grade in composite diamond drill core samples from the Little Chief underground mine. The strong linear correlation (factor 0.79) and the lack of identifiable silver minerals in the ore suggests that silver is present in solid solution in the copper sulphides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

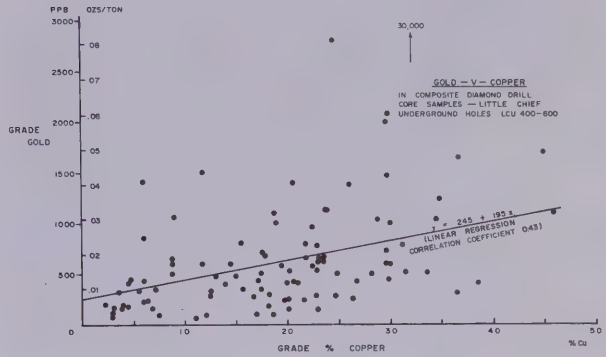

Figure 20. Plot of gold against copper for the same samples plotted in figure 19. The relationship is not as strong as for silver (linear regression correlation coefficient 0.43) which confirms limited visual observations of native gold in diamond drill core samples not being associated with unusual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

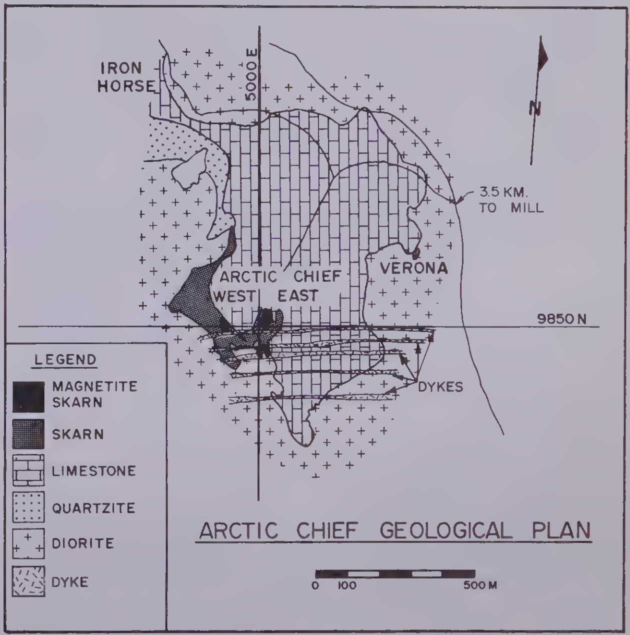

Figure 21. Geological plan of the Arctic Chief pendant.

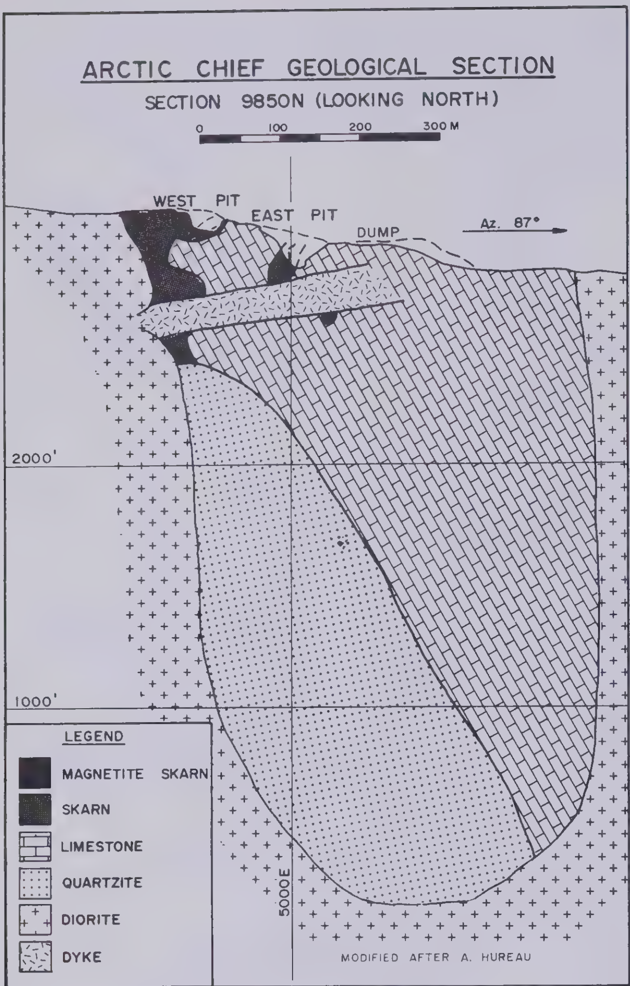

Figure 22. Geological cross-section of the Arctic Chief pendant. The East and West Arctic orebodies are further above the roots of the pendant than the ore at Little Chief or Gem/Bear Cub. The limestone/sediment contact is not parallel to surrounding intrusive contacts and no large orebodies are expected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

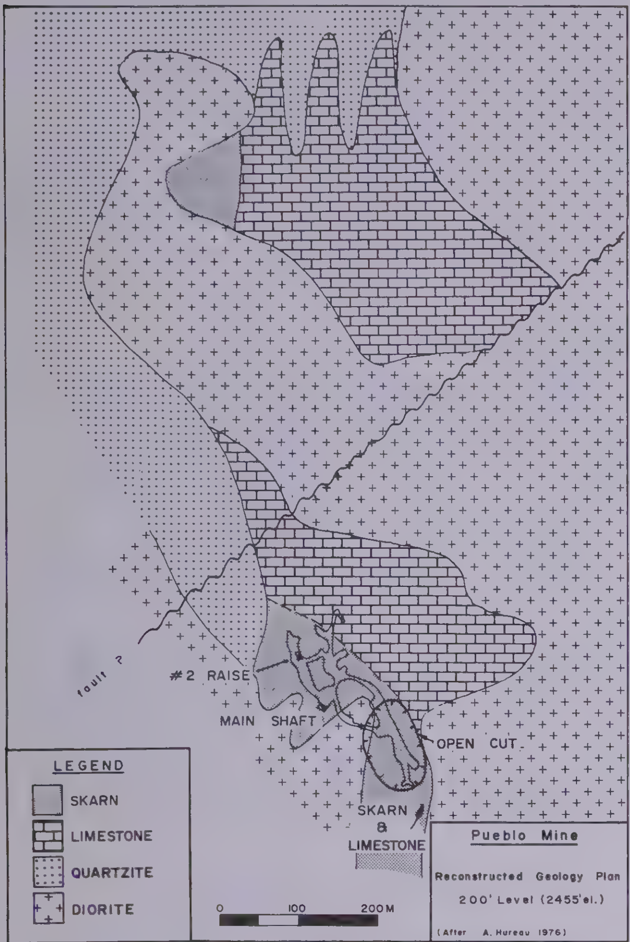

Figure 23. Geological plan of the Pueblo - Gulch area showing the Pueblo pendant, open pit and underground workings. The fault shown is speculative with unknown movement.

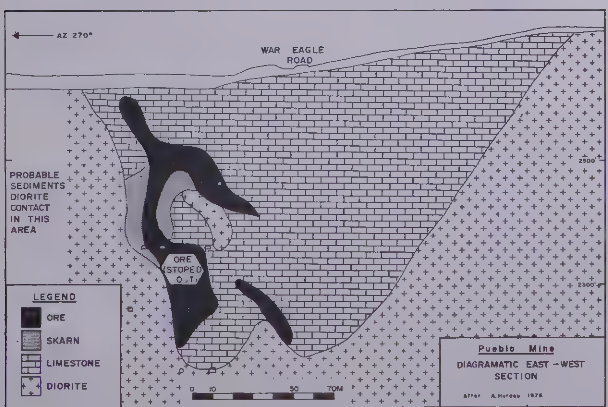

Figure 24. Schematic geological cross-section of the Pueblo pendant. Down dip potential for more ore is limited as the diorite contact cuts off the sedimentary rocks (after A. Hureau, 1976).

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 magnetmeter 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		Comments
	Peak	Average	
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 strategically located in the western hemisphere. Frequencies in use, however, are high by geophysical stan-

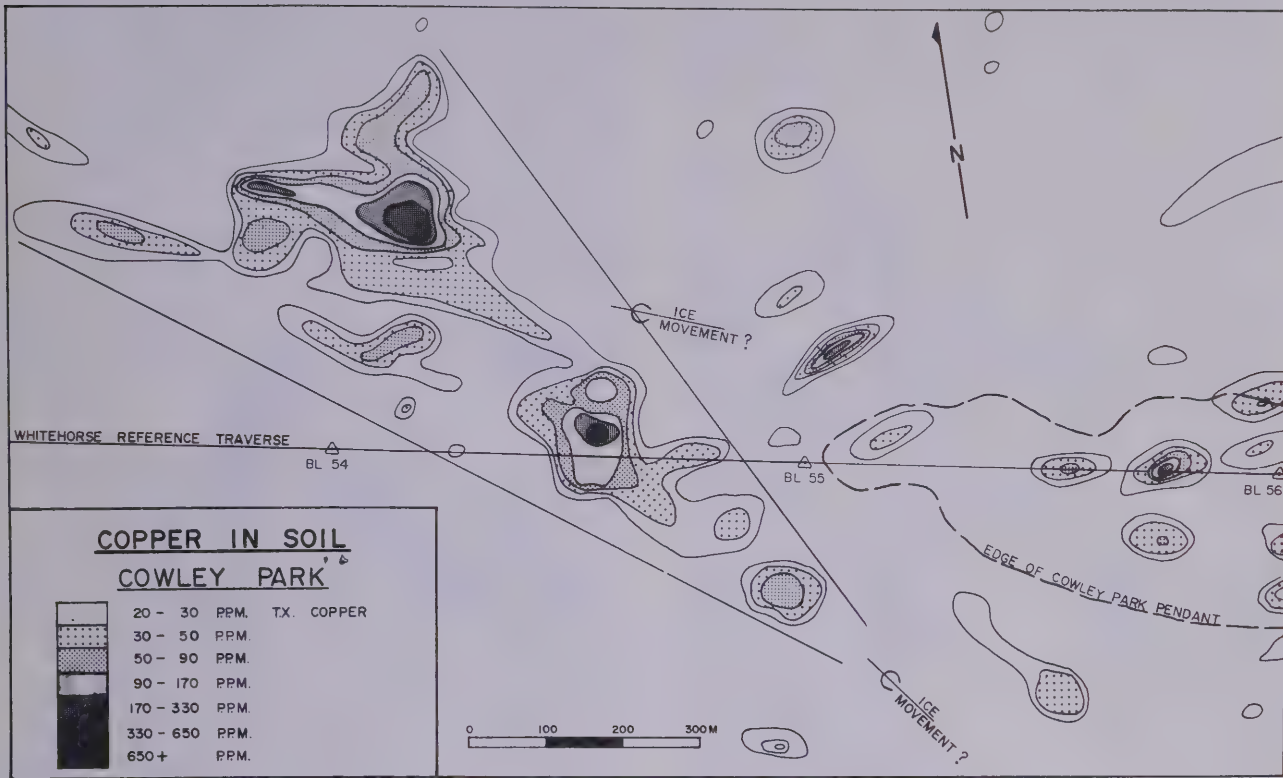

Figure 25. Anomalous fan shaped copper dispersion pattern in overburden at Cowley Park. Two possible directions of ice movement based on this data are indicated. The edge of the Cowley Park roof pendant is shown as it is the likely source of the anomalous copper. These copper anomalies would have led 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

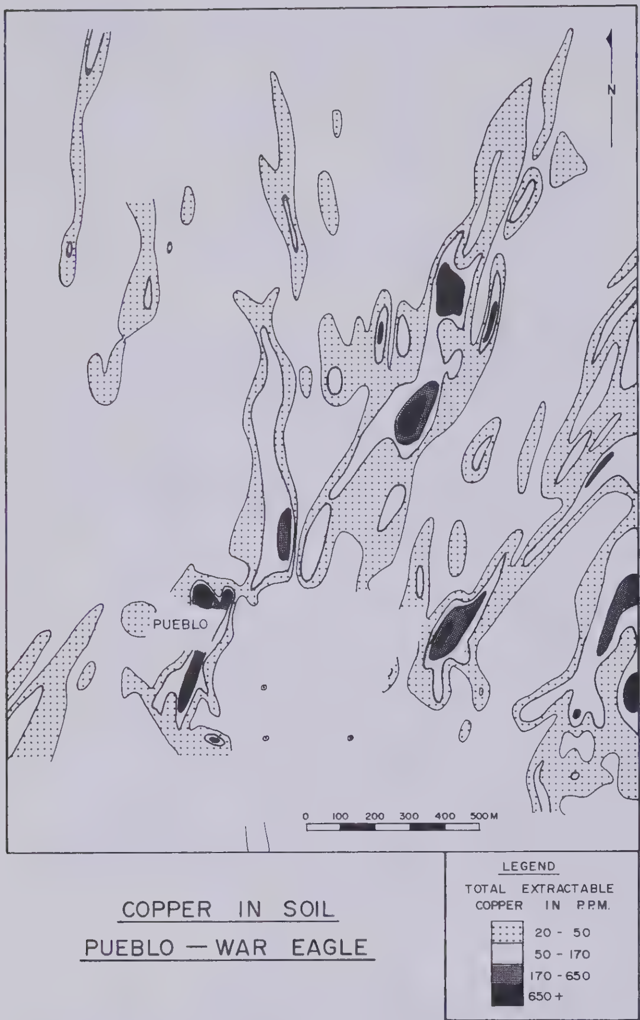

Figure 26. Anomalous train shaped copper dispersion patterns in overburden at Pueblo. These patterns, which may be modified by glacio-fluvial action, indicate the direction of ice movement during the last retreat of Wisconsin ice.

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.

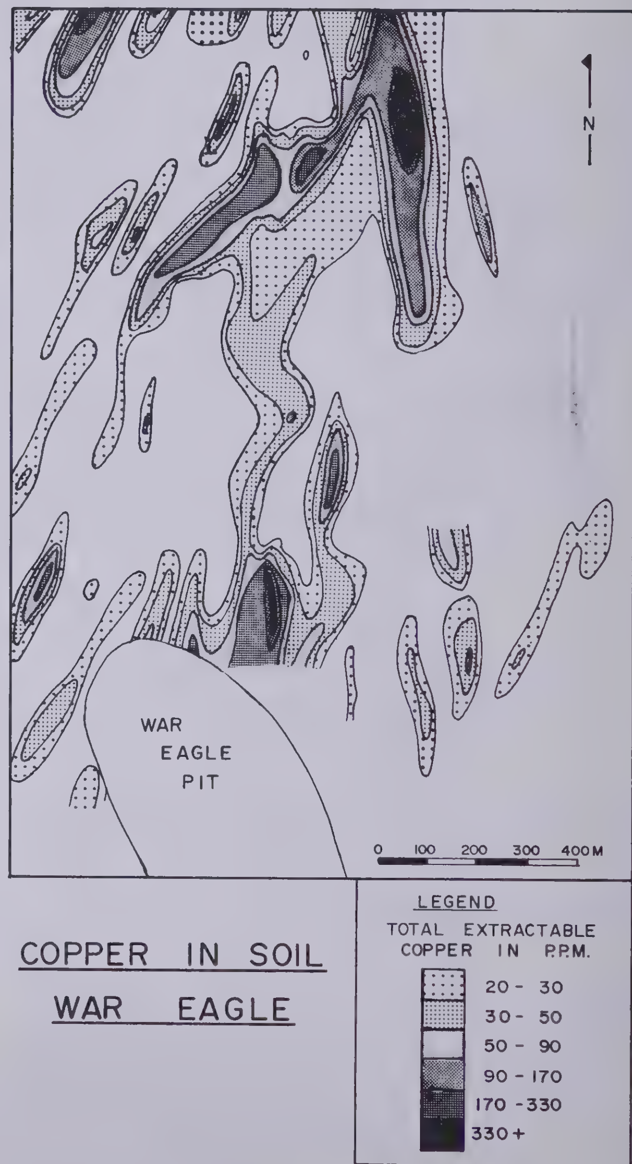

Figure 27. Anomalous train shaped copper dispersion patterns in overburden north of the War Eagle pit indicating the direction of retreat of the last ice to cover this area. The anomalies are interpreted as glacial smear from the War Eagle deposit.

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

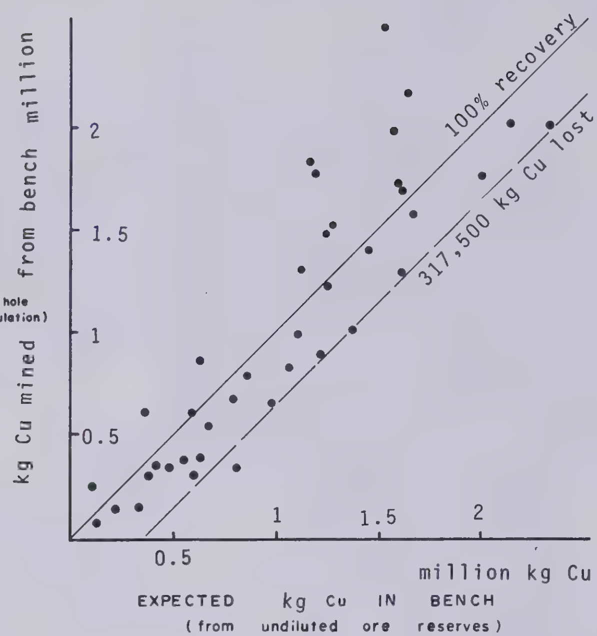

Figure 28. Open Pit ore reconciliation by benches showing the quantity of copper in undiluted ore reserves plotted against the quantity of copper mined according to uncorrected blasthole production calculations.

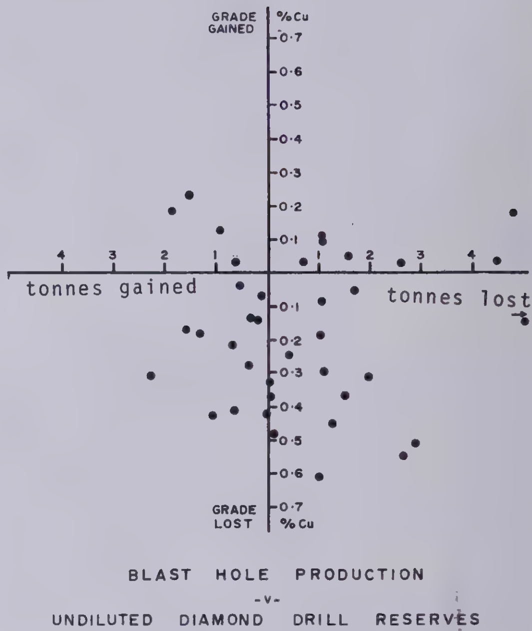

Figure 29. Open pit reconciliation by benches showing blasthole indicated grade and tonnage produced as a gain (upwards and right) or loss (downwards and left) when compared with undiluted diamond drill reserve grade and tonnage.

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

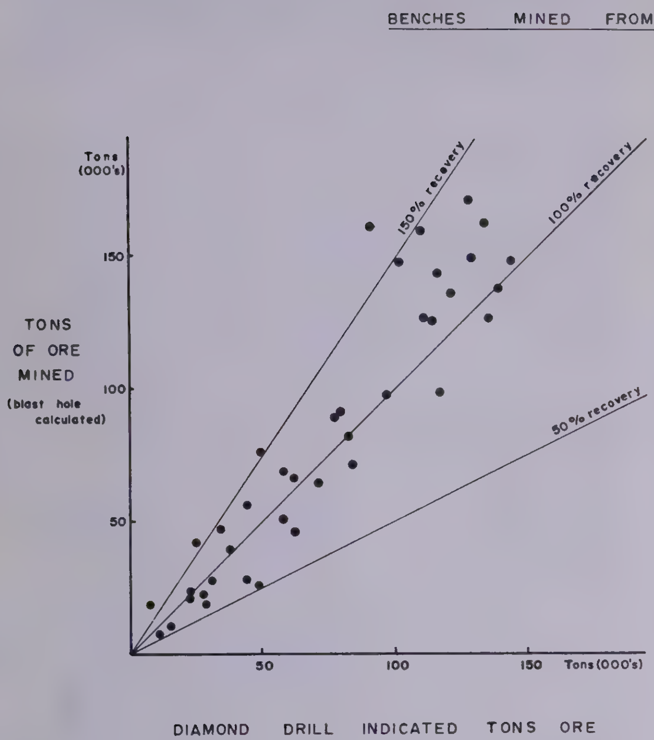

Figure 30. Open pit ore reconciliation by benches showing tonnage in undiluted diamond drill reserves against tonnage mined. Large tonnage benches produce as much, or more ore than predicted while smaller benches show some of the largest losses. This indicates that the geological interpretation of ore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

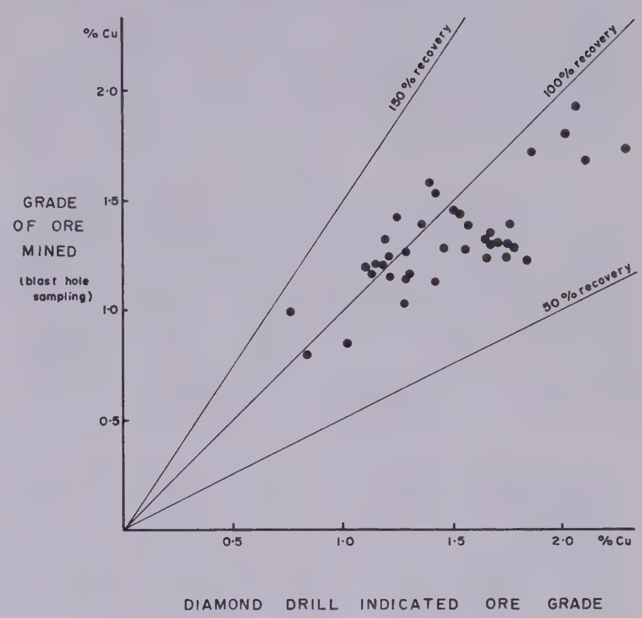

Figure 31. Open pit ore reconciliation by benches showing undiluted diamond drill reserve grade against grade mined according to blasthole production. Grade is between 70% and 120% of that predicted, but this needs correction (see Table IV).

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.

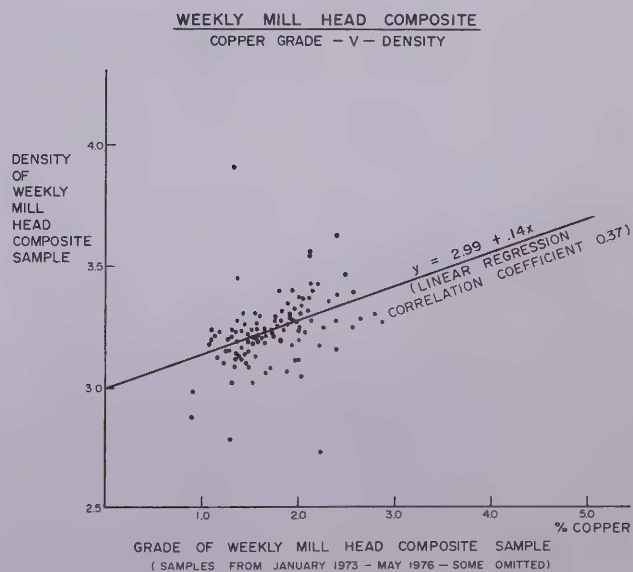

Figure 32. Copper grade plotted against density for weekly mill head composite sample (see also figure 18).

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.

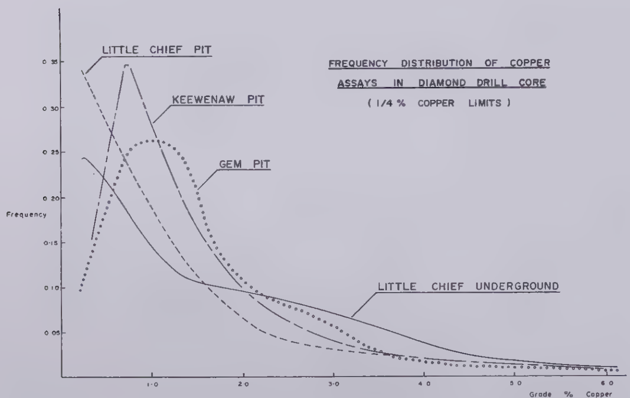

Figure 33. Frequency distributions of diamond drill core assays from selected open pits and the Little Chief underground mine. Curves are skewed positively. The Gem pit looks most attractive on this basis but mining difficulties offset this.

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^{0}.

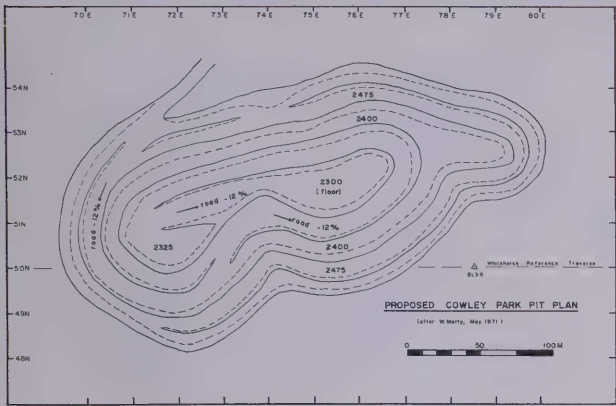

Figure 34. Proposed 1971 Cowley Park pit by W. Marty. Road gradients are not normally as steep as 12%.

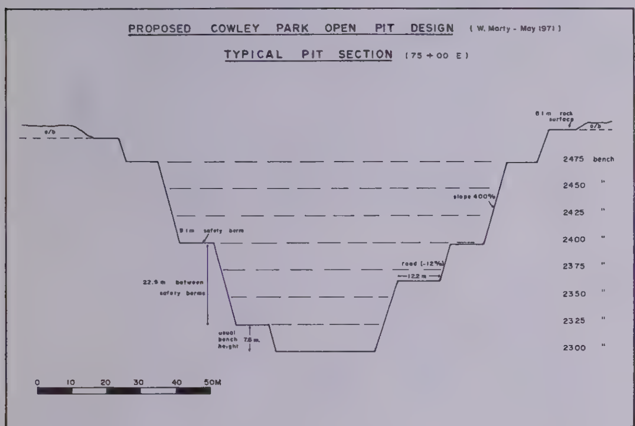

Figure 35. Typical open pit cross-section (proposed 1971 Cowley Park pit by W. Marty) showing width of safety berms and roads, and slope of walls.

These specifications were subject to modification depending on local circumstances. The average slope of the walls of the Little Chief Pit is 52^{0} 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.

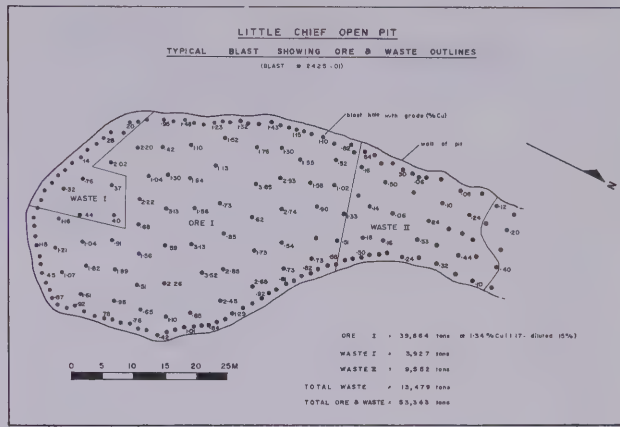

Figure 36. Typical open pit blasthole layout (the Little Chief 2425 - 01 blast) showing the spacing of drill holes, and preshear holes around the pit wall. Areas of waste and ore planimetered converted to waste and ore tonnage blasted are also shown.

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.

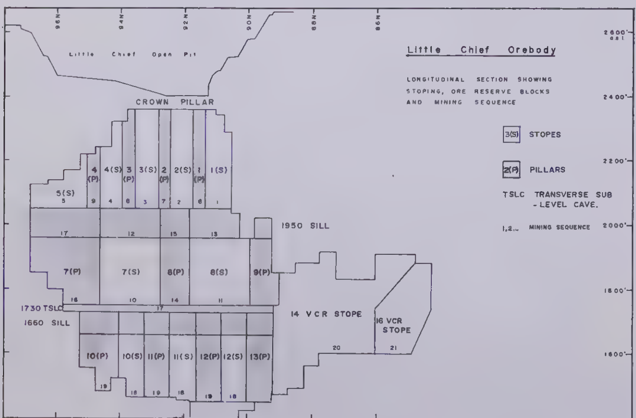

Figure 37. Vertical longitudinal projection of the Little Chief underground mine showing stopes and pillars with mining sequence. This section should be viewed in conjuction with the Little Chief underground ore recondiliation (Appendix IV) to judge the effectiveness of the mining methods used.

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.

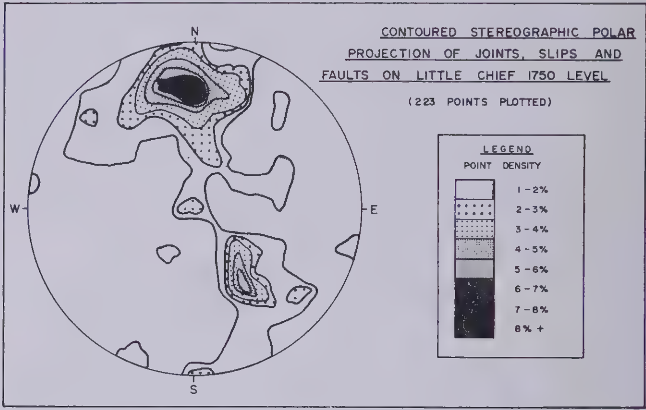

Figure 38. Contoured stereographic polar projection of joints, slips and faults Little Chief underground 1750 level. Two sets of joints dipping approximately 70° south and 50° north are apparent. The intersection of these two joint sets on the hangingwall side of the stopes induced caving out into the 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 $
<u>MINING</u>								
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
<u>MILLING</u>								
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