

SUMMARY REVIEW

COGBURN CLAIM GROUP

**New Westminster Mining Division
British Columbia, Canada
NTS: 092H05E, 092H12E**

June 8, 2000

Report Text Written by the Property Owners

Gerald G. Carlson, Ph.D., P.Eng.
(KGE Management Ltd.)

&

John A. Chapman, B.Sc., P.Eng.

Report Drawings Prepared by

David K. Makepeace, M.Sc., P.Eng.

TABLE OF CONTENTS

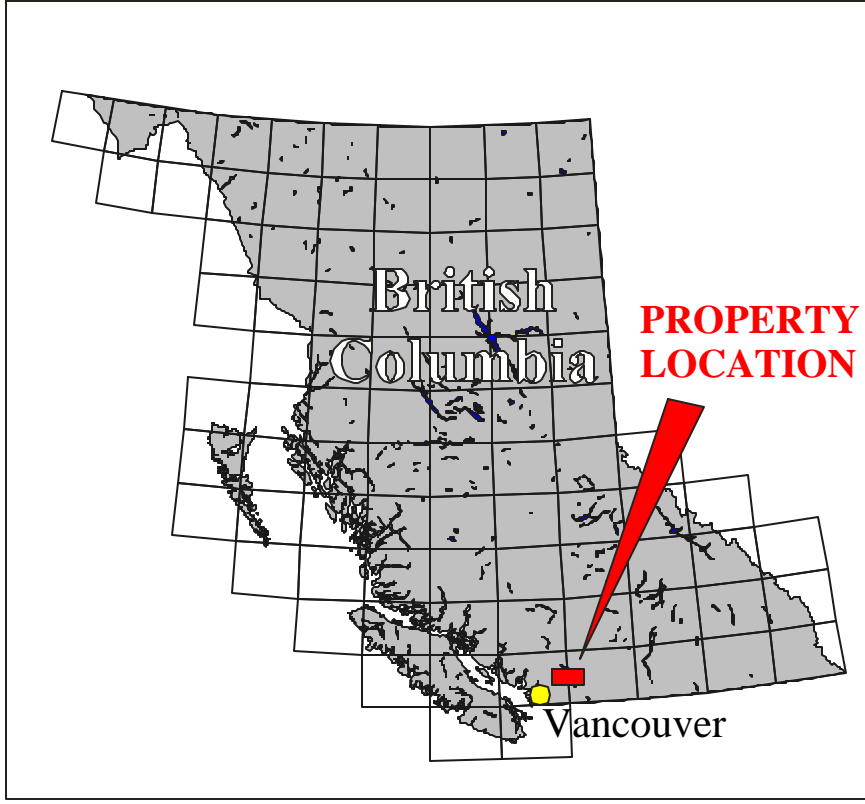
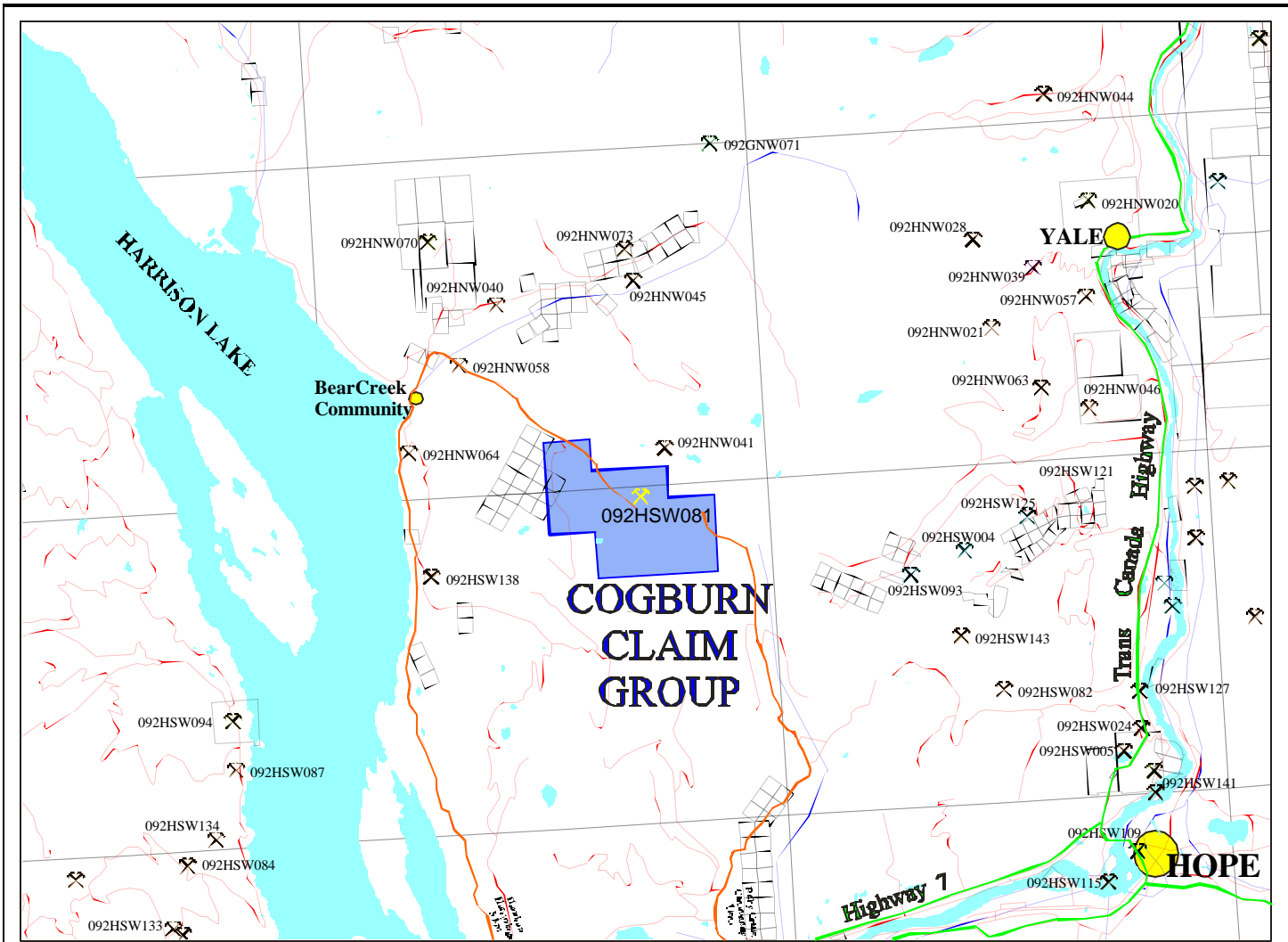
| | |
|--|--------------------------------|
| INTRODUCTION..... | 4 |
| LOCATION & ACCESS..... | 4 |
| CLIMATE | 5 |
| FLORA AND FAUNA | 5 |
| CLAIMS & OWNERSHIP | 5 |
| NATIVE LAND CLAIMS..... | 6 |
| EXPLORATION HISTORY | 6 |
| GEOLOGY | 6 |
| Regional Geology | 6 |
| Claim Geology | 7 |
| MINERALIZATION..... | 8 |
| Daioff Ridge Zone | 8 |
| Other Exploration Potential..... | 9 |
| ECONOMIC POTENTIAL | 9 |
| SOCIOECONOMIC IMPACT OF MINE DEVELOPMENT..... | 10 |
| BIBLIOGRAPHY..... | SEE APPENDIX II MINFILE |

LIST OF FIGURES

| | After Page |
|--|-------------------|
| Figure 1 – Location Map, Minfile Occurrences, Access Routes | 3 |
| Figure 2 – Mineral Claim and Metallogeny Map | 4 |
| Figure 2a – Regional Airborne Magnetometer Survey Map | Appendix I |
| Figure 3 – Property Topographic Map | Appendix I |
| Figure 4 – Property Arial Photograph Showing Review Areas | 4 |
| Figure 5 – Regional Geology Map | 5 |
| Figure 6 – Property Geology Map | Appendix I |
| Figure 7 – Regional Reconnaissance Geochemistry Map | 7 |
| Figure 8 – Property Soil and Rock Geochemistry Map | Appendix I |
| Figure 9 – Property Magnetometer & EM Anomaly Map | Appendix I |
| Figure 10 – West Talc Creek IP and Magnetometer Survey Map | Appendix I |
| Figure 11 – Property Drill Hole Location Map | Appendix I |
| Figure 12 – Property Preliminary Site Development Plan | 9 |

APPENDICES

| | |
|---------------------|---|
| APPENDIX I | Figures |
| APPENDIX II | BC Government Minfile Reports: Giant Nickel, Ni; George Cross News Letters |
| APPENDIX III | PlatSol Hydrometallurgical Process for Bulk Flotation Concentrates (Ni, Cu, Co, PGE's and precious metals); Billiton's BioNIC Process; Titan Resources NL, Bacterial Heap Leaching of Nickel |



COGBURN CLAIM GROUP

New Westminster Mining Division
British Columbia, Canada

NTS:
092H05E, 09H12E

Location Map

Figure 1

Source: TheMapPlace
Drawn By: DKM, P. Eng.
Date: May 30, 2000

INTRODUCTION

Nickel-copper mineralization was discovered in 1923 at the Pride of Emory deposit on Stulkawhits Creek, 12 km northwest of Hope and 6 km east of the Cogburn claim group, in southwestern British Columbia. Underground development was begun in 1926, but it wasn't until 1936 that commercial production was achieved. From 1936 to 1974, production totaled 26,573,090 kg of nickel and 13,212,770 kg of copper with silver, gold and cobalt credits. Production of PGE's is not recorded but early sampling yielded values from 2.74 to 3.98 gpt platinum plus palladium.

Exploration in the vicinity of the Cogburn claims (a.k.a. NI prospect, B.C. Minfile No. 092HSW081, see Appendix II) began in the late 1960's by the Nickel Syndicate (Giant Explorations Limited and Giant Mascot Mines Limited) and continued through 1975. The work focused on a series of ultramafic intrusions apparently related to the rocks that host the Pride of Emory. The Cogburn property was drilled in 1971 and was shown to contain a large volume of low-grade sulphide nickel mineralization within altered ultramafic rocks. Unfortunately, drill core, core logs and assays are all missing and today only summary information remains.

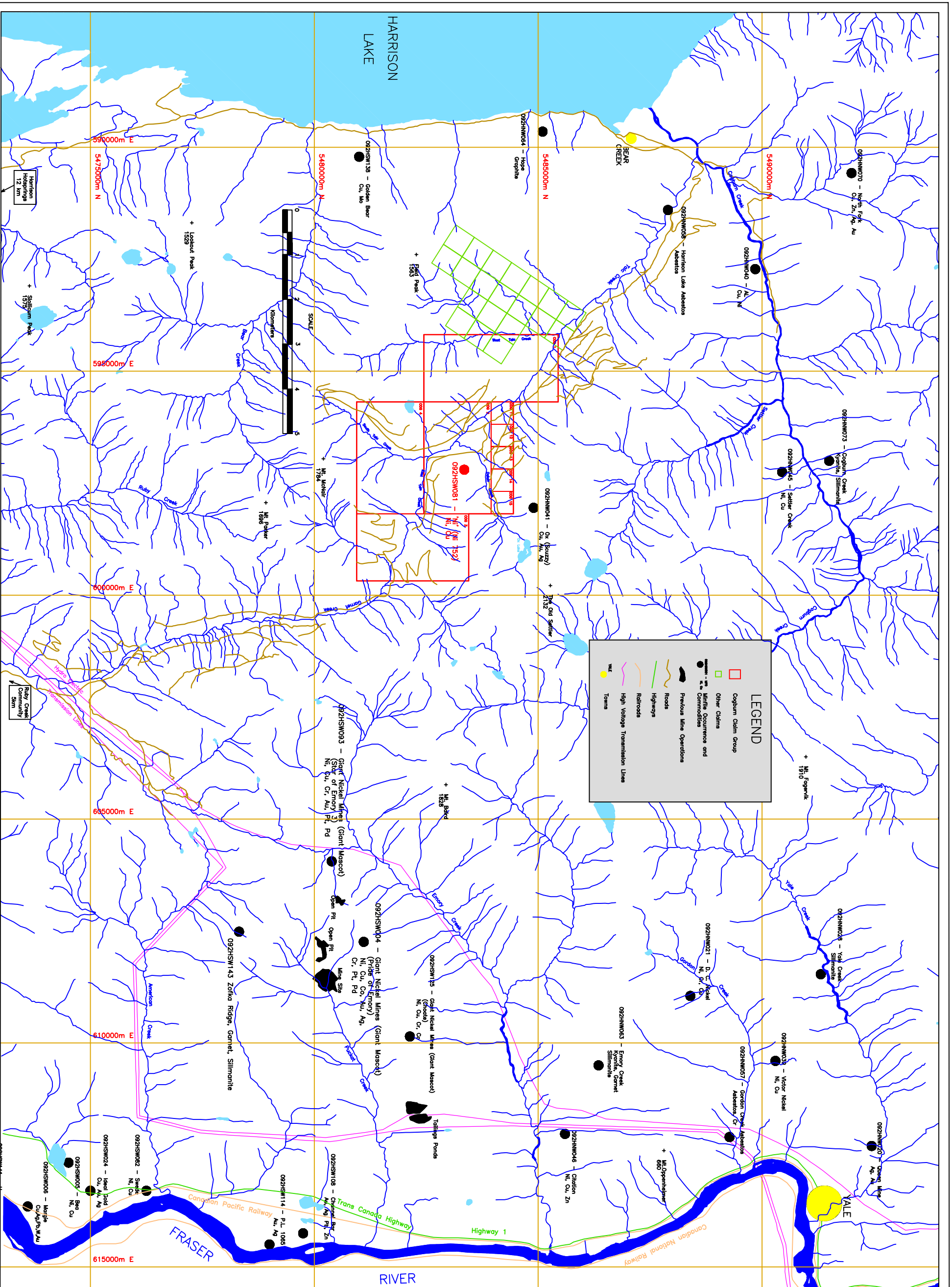
In September, 1971, the George Cross News Letter reported that results of surface sampling and three drill holes showed consistent results over an area 2,000 feet long, 800 feet wide and up to 1,600 feet vertically. Assays of rock and core showed "remarkably consistent values between 0.19% and 0.25% nickel", with ascorbic acid tests indicating that between 90% and 95% of the nickel occurred in sulphide form.

In January 2000, while researching the Pride of Emory deposit, Mr. John Chapman recognized the bulk tonnage potential of the Cogburn deposit and the property was subsequently staked in February with additional claims added in April. The authors have not had an opportunity as yet to visit the property. This report provides a summary of the geology and mineralization of the deposit, as taken from historical documents, and presents a vision of the potential of this untapped nickel resource.

LOCATION & ACCESS

The Cogburn property is located at 49° 29' 49" N latitude and 121° 39' 28" W longitude, NTS sheets 092H05E and 092H12E in southwestern British Columbia, approximately 120 km east of Vancouver. The claims are centred near the junction of Talc Creek and Daioff Creek, 8 km east of Harrison Lake. The claims are readily accessible by logging road from the east side of Harrison Lake, along Cogburn Creek and thence up Talc Creek, a road distance of 42 km from Harrison Hot Springs. The general area of the claims is also accessible by logging road from the Fraser River, both from the south and from the east. The Fraser River is a major transportation corridor with road, rail, gas and oil pipelines and power transmission lines (see Figures 1 and 2).

Access for development and mining operations would most likely be from Ruby Creek at Highway 7, along the Fraser River midway between Agassiz and Hope. The Cogburn property is located 16 km north from Ruby Creek via International Forest Products Ltd's (INTERFOR) Garnet Creek main road. The road climbs from 50 masl at Ruby Creek to 1,200 masl at the property, yielding an average road gradient of 7.2 percent.



COGBURN CLAIM GROUP

New Westminster Mining Division, British Columbia Canada

NTS:
092H05E, 092H12E

Mineral Claim Metallurgy & Access Map

Figure 2

Scale : As Shown
 Drawn By : DKM, P.Eng.
 Date : May 30, 2000

The CPR mainline is adjacent to Highway 7 at Ruby Creek. In addition barge access to Ruby Creek on the Fraser River is a possibility. There are studies now in place for dredging of a barge channel to Harrison River that could accommodate barges with up to 4,000 tonne payloads. Catherwood Towing Ltd. has indicated that there is the further potential to extend the barge channel east to Ruby Creek for barges with payloads to 2,000 tonne.

The claims (see Figure 2) are in moderately rugged, glaciated, mountainous terrain, with elevations ranging from 800 m to over 1,500 masl. Much of the area has been logged in recent years and active logging and construction of new logging road access continues (see Figures 3 & 4).

CLIMATE

The climate in the vicinity of the Cogburn deposit is typical of the area with cool summers and mild winters. Annual precipitation is approximately 300 cm, including average annual snowfall at 1,000 cm. Snowpack can reach 400 cm and remains on south slopes until April or May and on north slopes until June. Temperatures range from an average of minus one degree centigrade in winter to an average of plus 15 degrees centigrade in summer.

FLORA AND FAUNA

The Cogburn project area is in an active logging region that extends from the deposit along access corridors south to Ruby Creek at Highway 7 and north to Bear Landing at Harrison Lake. The only extraordinary environmental element within the region is the Old Settler Peak goat herd which resides at the headwaters of Daioff Creek some 2 km to the northeast of the deposit. The herd stays on the Peak year-round so there should be no impact on their habitat by exploration or mine development and operations at Cogburn.

CLAIMS & OWNERSHIP

The 1,700 hectare Cogburn property consists of 10 claims for a total of 68 units located in the New Westminster Mining Division, British Columbia, NTS sheets 92H5E and 92H12E. The claims straddle Talc Creek where it is joined by Daioff Creek, north of Mt. McNair and west of the Old Settler Mountain. The claims are owned 50% by KGE Management Ltd., (Gerald G. Carlson, President) and 50% by John A. Chapman as follows:

| <u>Claim Name</u> | <u>Tenure Number</u> | <u>NTS Map Sheet</u> | <u>Units</u> | <u>Expiry</u> |
|-------------------|----------------------|----------------------|--------------|-------------------|
| COG 1 | 374546 | 092H12E | 18 | February 19, 2001 |
| COG 2 | 374547 | 092H12E | 15 | February 21, 2001 |
| COG 3 | 375295 | 092H05E | 15 | April 8, 2001 |
| COG 4 | 375296 | 092H05E | 15 | April 7, 2001 |
| COG 11 | 375290 | 092H12E | 1 | April 8, 2001 |
| COG 12 | 375291 | 092H12E | 1 | April 8, 2001 |
| COG 13 | 375292 | 092H12E | 1 | April 8, 2001 |
| COG 14 | 375293 | 092H12E | 1 | April 8, 2001 |
| COG 15 | 375294 | 092H12E | 1 | April 8, 2001 |

COGBURN CLAIM GROUP

New Westminster
Mining Division
British Columbia
Canada

NTS:
092H05E, 09H12E

Regional Geological Map

Figure 5

Source: GSC O.F. 2490

Scale : As Shown

Drawn By: DKM, P.Eng.

Date: May 30, 2000

PARTIAL LEGEND

PLUTONIC ROCKS

OLIGOCENE (24 - 29 Ma)

Ogd Granitoids of CHALLENGER PLUTON and smaller smaller bodies

EARLY TERTIARY (40 - 64 Ma)

Egt Granitoids and monzonites (g) of NEEDLE PEAK, WISDOM ROSE, MOUNT OUTRAMP and TEXAS CREEK PLUTONS (40 - 58 Ma) and bodies of intermediate (i) composition

MID - CRETACEOUS (mainly 91 - 105 Ma; locally as old as 112 Ma)

mk Cores (k): coarse (c), granitoids (g), monzonites and diorite (d) of SPUIZZUM PLUTON and related bodies, including SPUIZZUM, CHALLENGER, POTEAU, CASTLE TOWERS, BRILLIANTITY, LALLOTT RIVER, THALYTE, MT. CLARKE, RTY LAKE, AMALBROOK, STAKE BLADE, SPITON CREEK, PRINCESS ROYAL REACH, WERTY LAKE, BIG HOLE and SOLAMON JARVIS.
Intermediate (i): granitoids (g), monzonites (m) of FALLS LAKE PLUTONIC SUITE, granite (g) of CATHEDRAL LAKES and VERDE CREEK plutons

PALEOZOIC and PROTEROZOIC

py Cores (k): diorite and gabbro of YELLOW ASTER COMPLEX

OPHIOLITE COMPLEX ROCKS

PALEOZOIC and MESOZOIC

CJH Unaltered ophiolite rocks, mostly associated with CATCH CREEK and BRIDGE RIVER and MOZAMBEK COMPLEXES and SPIDER PEAK FORMATION

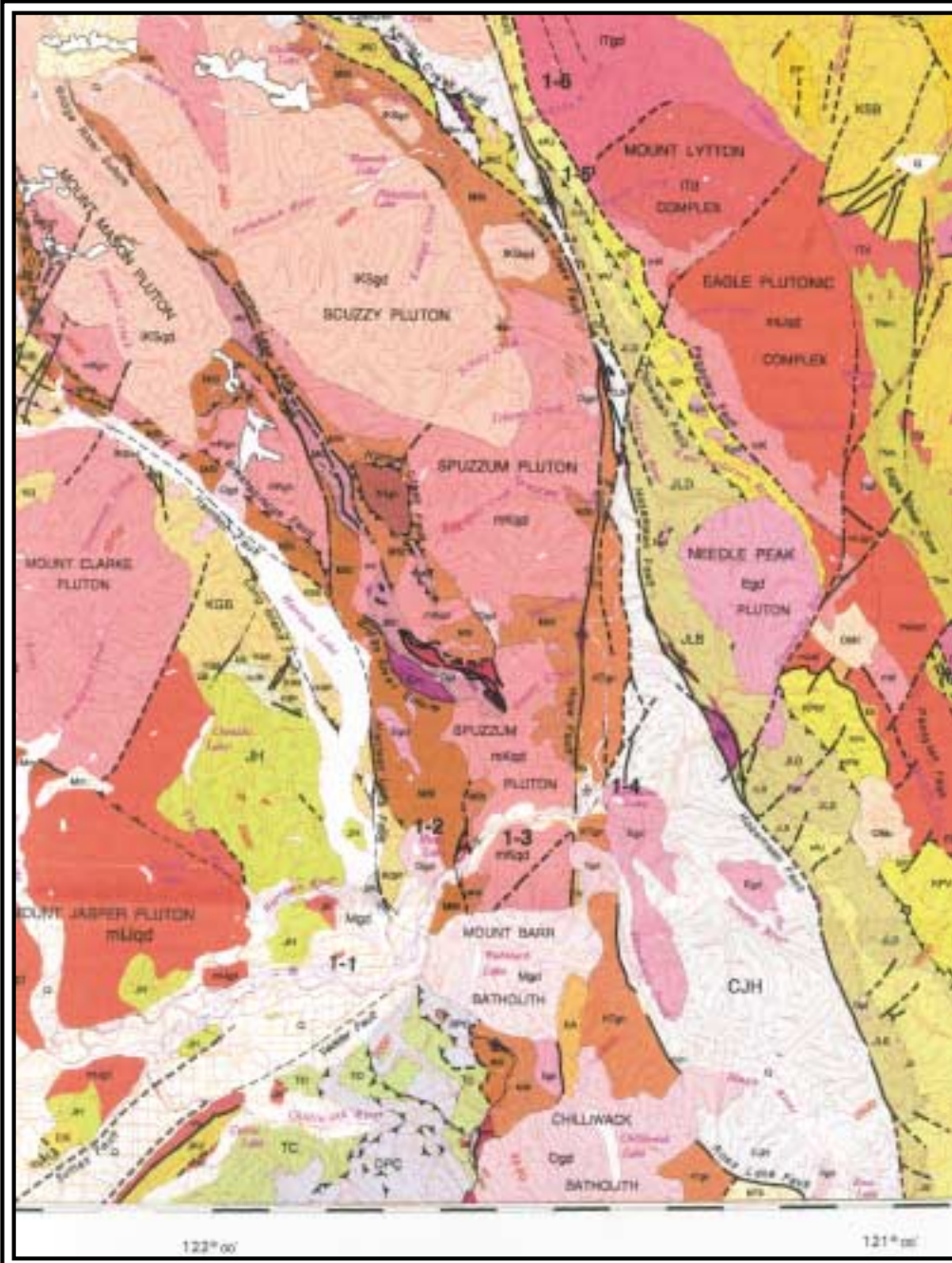
METAMORPHIC ASSEMBLAGE ROCKS

EARLY LATE CRETACEOUS and LATE CRETACEOUS

MSI SETTLER SCHIST: garnet-biotite, muscovite, quartz and clinopyroxene with amphibole, including staurolite-bearing, albite-rich; includes greenstone-granite complexes, pegs and broken formations south of Fraser River; partially altered or in part derived from the Jura-Cretaceous (AYDISH) ASSEMBLAGE metamorphosed in situ in early Late Cretaceous (94 - 100 Ma) and Late Cretaceous (98 - 99 Ma). Mineral assemblages reveal both high pressure/intermediate to high-temperature, Barrovian metamorphic field gradients (M1) ranging from middle greenschist to lower amphibolite and prongite, over- to near-Barrovian overprint (M2) or greenschist to amphibolite facies

MSI BULLOCK SCHIST: mafic, intermediate and felsic meta-volcanic rocks, pegs, minor calcane and carbonaceous argonite, partially altered or in part derived from GAMBIE and BULLOCK CREEK formations metamorphosed in situ in early Late Cretaceous (94 - 100 Ma). Mineral assemblages reveal an intermediate to high-pressure/intermediate to high-temperature Barrovian metamorphic field gradient (M1) ranging from middle greenschist to middle amphibolite facies

MC COGBURN SCHIST: unaltered pegs, amphibolite, mafic, ultramafic rocks derived in part from the BRIDGE RIVER and MOZAMBEK COMPLEXES, metamorphosed in situ in early Late Cretaceous (94 - 100 Ma). Mineral assemblages reveal a high pressure/intermediate to high-temperature Barrovian field gradient (M1) ranging from middle greenschist to amphibolite facies and a prongite, low pressure Barrovian overprint (M2) or greenschist to amphibolite facies



NATIVE LAND CLAIMS

Almost all of British Columbia's lands are subject to treaty negotiations with the Status Indians. The Cogburn deposit falls within the large "Yale" treaty area extending from the U.S.A. border in the south, north to Boston Bar, east to Manning Park and west to Chilliwack.

EXPLORATION HISTORY

Recorded exploration in the area of the Cogburn claims is restricted to the period 1969, when the NI claims were staked by the Nickel Syndicate (Giant Explorations Limited and Giant Mascot Mines Limited), to 1975. During this period, reconnaissance style exploration, including regional geological mapping, prospecting and stream sediment geochemistry was followed by a helicopter-borne magnetometer survey, detailed grid exploration and drilling.

The airborne magnetometer survey (see Figure 2a) included 60 flight lines for a total of 335 line miles, covering an area of approximately 85 square miles. The sensor was flown with a mean terrain clearance of 300 ft.

The early work resulted in the definition of eight target areas for detailed exploration. A line grid was cut over each target and the survey lines used to control geological mapping, soil sampling, rock chip sampling where outcrop was exposed and ground magnetics. Soil and rock samples were analyzed for nickel and copper.

During the summer of 1971, local IP surveys were carried out to define specific drill targets. These were followed by 20 diamond drill holes for a total of 5,760 feet. The holes tested anomalies defined on at least two of the grid areas (Areas 4 and 6; 6 is the area of the current Cogburn claim group). Unfortunately, details of the drill program were not reported. Core logs, assays and most hole locations are missing, as well as the drill core. There is also little reported on subsequent work from 1972 through to 1975.

In 1985, John McGoran collected three rock chip samples and a heavy mineral sample from Daioff Creek, within grid area 6. However, it appears that the samples collected by Mr. McGoran were taken from outside the mineralized area.

GEOLOGY

Regional Geology

The geology in the region of the claim area is complex, dominated by northwest-trending fault bounded blocks of diverse geology. There appear to be at least three separate packages or terranes that underlie the claim group. These include, from west to east, the Harrison Terrane, the Cogburn Group, which is correlated with the Bridge River and Hozameen complexes, and the Cayoosh Assemblage. These have later been intruded by middle Jurassic to cretaceous Coast Plutonic intrusives

The Harrison Terrane includes a variety of sedimentary and arc volcanic rocks of middle Triassic to Early Cretaceous age (see Figure 5). These rocks occur on the western edge of the claim group in a complex fault relationship with the Cogburn Group, as indicated by airborne magnetics.

The core of the claims is underlain by the Cogburn Group, which has been correlated with the Bridge River Complex and the Hozameen Complex to the north, mainly on the basis of similar lithology. The Cogburn Group consists of bedded chert, argillite, basic volcanics and alpine-type ultramafic rocks with minor amounts of marble. This is a fault-emplaced assemblage of oceanic crust or ophiolite. Ages of rocks within the Bridge River Complex range from Mississippian to earliest Jurassic, suggesting that it now represents the remnants of a long-lived (+190 Ma), large ocean basin.

The Settler Schist, which underlies the eastern part of the claim group, is part of the Lower to Middle Jurassic Cayoosh Assemblage, an upward-coarsening succession of argillite, siltstone and sandstone, possibly emplaced over the Bridge River Complex. The Settler Schist, believed to be a metamorphic equivalent, consists of pelitic and quartzofeldspathic schists, amphibolite and minor quartzite. The metamorphic grade is up to amphibolite with staurolite and, to the north, kyanite and sillimanite schists.

The main intrusive rocks in the area belong to the Spuzzum Pluton, a post-accretion (mid-Cretaceous, approx. 100 Ma) quartz diorite body which lies to the east of and partially encloses the Cogburn Group ultramafic package and the adjacent Settler schist. The grade of metamorphism of this latter unit varies with proximity to the contact. Within the claim group just north of Daioff Creek, a smaller body of Early Tertiary quartz-diorite to granodiorite cuts across Talc Creek in a dike-like body with a northeasterly trend.

Claim Geology

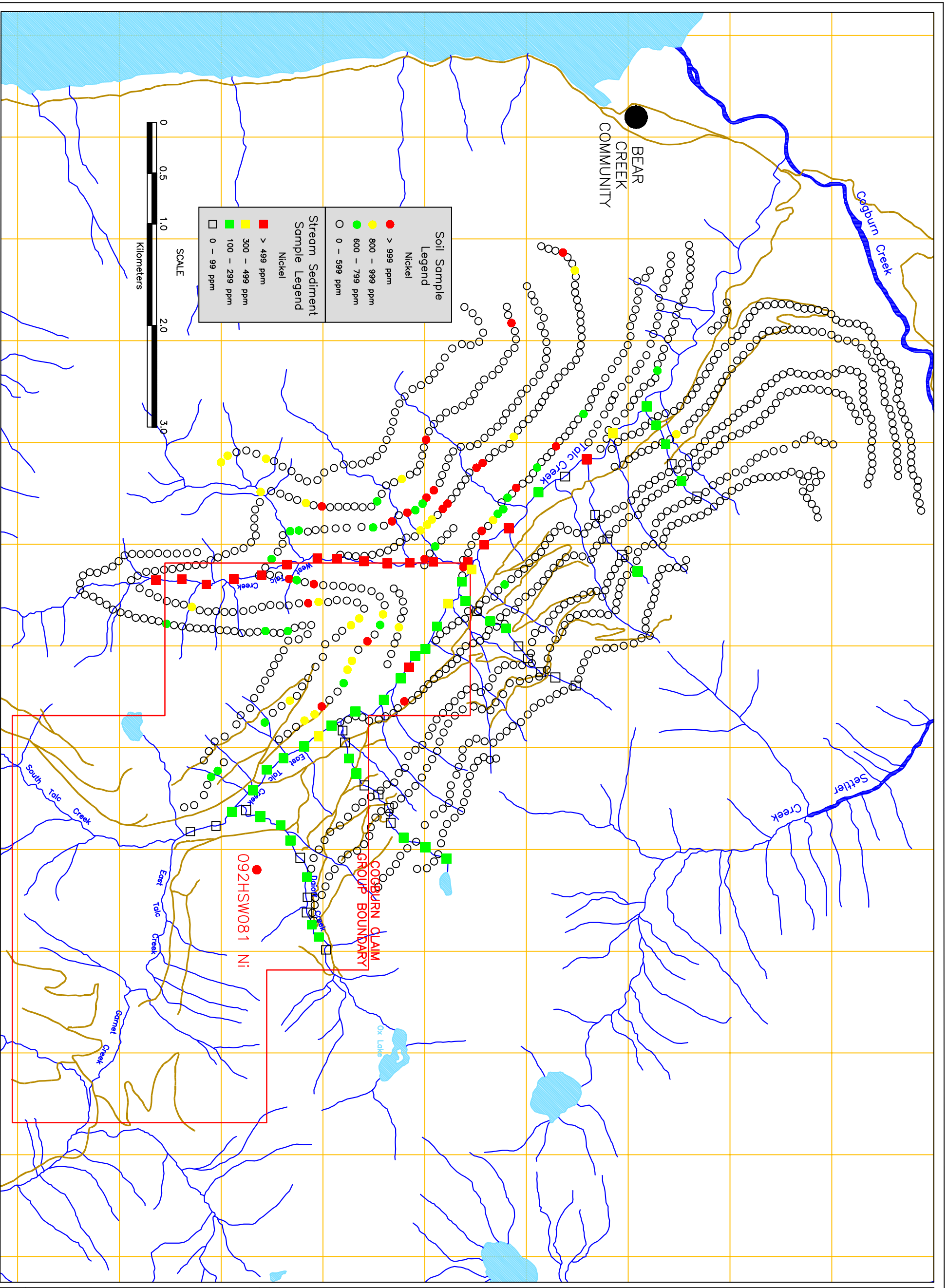
The claims are underlain by ultramafic rocks of the Cogburn Group in the central and western parts and metamorphic rocks of the Settler Schist to the east, with a younger quartz diorite intrusive that extends along a northeasterly trend in the western part of the claim group (see Figure 6).

The ultramafic rocks are comprised primarily of altered pyroxenite, with peridotite evident in talus on the south side of Daioff Ridge, but not seen in outcrop.

The pyroxenite is medium grained, greenish-grey and weathers a buff to light brown colour. It is strongly uralitized, with up to 75% of the pyroxene altered to uralite, sometimes occurring as randomly oriented blades to 1 cm long. Talc and serpentine are also observed adjacent to shears and fractures. Pyrrhotite with minor magnetite and traces of pyrite, chalcopyrite and possibly pentlandite are observed finely disseminated throughout the rock.

The peridotite is fine grained to aphanitic, dark green to black and buff weathering. It typically contains 2% to 3% disseminated magnetite in fine seams, but with little or no sulphide.

The metamorphic rocks occurring to the east of the mineralized zone are believed to be in fault contact. They are described as fine-grained, dark, altered volcanic rocks and, to the east towards Old Settler Peak, white, massive quartzite. The quartzite is sometimes schistose, with locally abundant biotite and sericite. Given the lithologic descriptions in regional syntheses, it is possible that the western metavolcanics belong to the Cogburn Group, while the metasediments are part of the Settler Schist.



COGBURN CLAIM GROUP

New Westminster Mining Division, British Columbia Canada

NTS:
092H05E, 092H12E

Reconnaissance Geochemistry Map

Figure 7

Scale : As Shown
Drawn By : DKM, P.Eng.
Date : May 30, 2000

MINERALIZATION

Daioff Ridge Zone

This summary of historical exploration results is taken from a number of assessment reports referenced in the property's appended Minfile reports 092HSW004 and 092HSW081.

A soil geochemical nickel anomaly covers most of the pyroxenite where it is known to occur within the claim group. The anomaly is approximately 1,000 m long in a north-south direction and up to 1,000 m wide, although many of the grid sites were unsampled due to outcrop and/or talus and steep topography. Nickel values within the anomaly range from 100 to 900 ppm. In the same area, copper values are typically low, less than 50 ppm, with a few spot highs. See Figures 7 and 8).

The nickel soil anomaly reflects bedrock mineralization that includes up to 4% disseminated pyrrhotite with minor pyrite, chalcopyrite and possibly pentlandite. The sulphide minerals occur as grains and blebs interstitial to the pyroxene crystals and as narrow fracture veinlets up to a few cm long.

Rock chip samples were collected over a large part of the area of missing soil values. Within an area approximately 600 m in a north-south direction and 500 m to 700 m in an east-west direction, nickel values in bedrock range from .16% to .24% nickel in 55 samples (average .22% nickel). The anomalous values continue at least another 500 m to the south in wider spaced sampling. The bedrock-mineralized zone appears to be open to the west and to the north. The farthest north samples also contain copper ranging from .19% to .24%. Copper values are not reported for the other samples.

According to geological mapping on Daioff Ridge, the nickel values occur within altered pyroxenite. This also coincides with an area of elevated magnetics, as reflected in the airborne magnetometer survey as well as the ground survey (see Figure 9). However, there does not appear to be an exact correlation here, as some of the +.2% nickel values occur in an area of low magnetic relief in the southern portion of the mineralized zone.

The details of the 1991 trenching and drilling programs were not reported in assessment reports but were mentioned in summary form in George Cross Newsletters 170(1971) and 242(1971) which are appended. In the former, the mineralization is reported over "a length of 2,000 feet, a width of 800 feet and over a vertical surface range of 700 feet". From three drill holes, the assays were reported to range between .19% and .25% nickel. Ascorbic acid tests indicated that 90% to 95% of the nickel occurs in sulphide form. In the latter report, with a total of 1,980 feet of diamond drilling, the average nickel value is reported as .22% over an expanded vertical range of 1,600 feet (see Figures 8 and 11).

The sample results were confirmed by G.E.P. Eastwood, a geologist with the British Columbia Department of Mines, who collected five samples from within the mineralized zone. Eastwood's samples ranged from .18% to .25% nickel (average .22% nickel). Eastwood also did acid soluble nickel tests that showed .19% to .22% sulphide nickel in the samples tested.

Other Exploration Potential

A number of other potentially mineralized zones are indicated by high nickel values in the reconnaissance soil and stream sediment surveys. One of these, Grid 2, which also lies within the Cogburn claim group, was examined with detailed sampling, mapping, ground magnetics and an IP survey. It is believed that this target was not drill tested. The IP survey on Grid 2 shows a chargeability anomaly with coincident anomalous nickel in soils in an anomalous pattern that is open to the west (see Figure 10). According to the geophysical interpretation, the chargeability values suggest up to 4% sulphides. It should be noted that, at the time of the survey (September, 1971) a small, high-grade nickel deposit was being sought and a broader zone of disseminated sulfides was not considered to be as significant.

The results of previous exploration work suggest that there is good potential for the discovery of additional nickel-copper-cobalt-PGE mineralization within the COG claim group.

ECONOMIC POTENTIAL

The Cogburn deposit, with a resource of +100 million tonnes grading .22% nickel, has the potential to be developed as a very low strip ratio open pit mining operation feeding a +20,000 tonne per day milling and processing plant. The gross in-situ metal value, considering nickel only without regard for any possible copper, cobalt, gold, silver or PGE's is C\$29 per tonne, or C\$2.9 billion for a 100 million tonne deposit.

The deposit, which has a surface extent of approximately 30 hectares, is located on a hill between Daioff Creek and the east fork of Talc Creek at its headwaters (see Figure 12). Similar to Craigmont mine, the deposit would best be developed with a crusher at the deposit and a downhill conveyor (generating power) to the plant. The plant would be located to the west of and below the deposit at an elevation (about 950 m) that would allow a slight downhill gradient to the tailings management facility in the early years of operations.

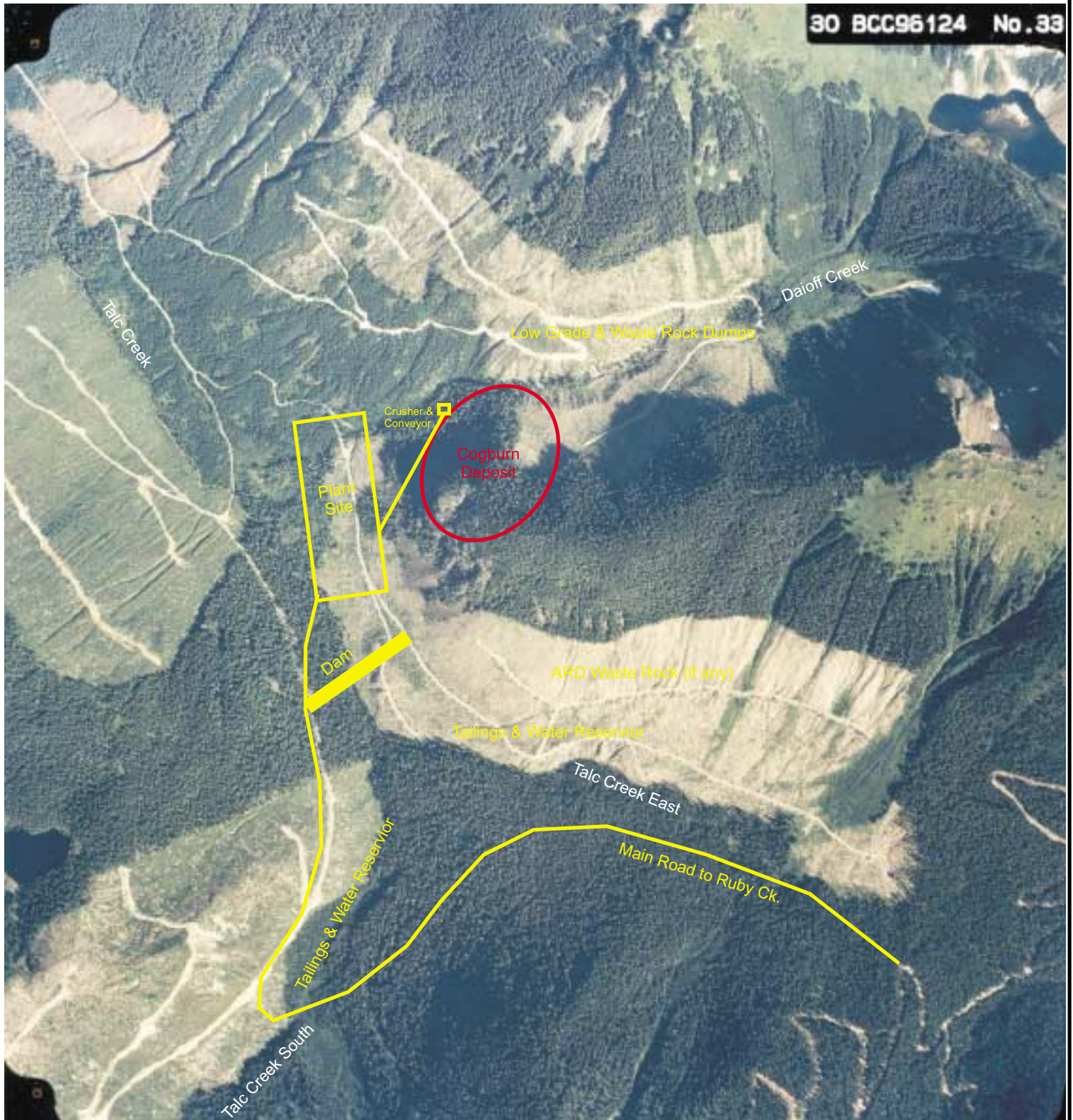
The tailings management facility would include a dam located just below the junction of Talc Creek east and south forks near the headwaters, with the toe of the dam located at the 890 m contour. The main dam fill would consist of mine waste and/or excess plant site waste rock and overburden. Building the dam several months ahead of plant startup, and as a high structure, could result in enough water supply to sustain the operations without the need for a pipeline and pump stations from either Harrison Lake (via Talc Creek) or the Fraser River (via Garnet Creek).

There are ample areas for low-grade stockpiles and waste dumps in the Daioff Creek drainage. If any acid generating waste rock is encountered it could be dumped to the south above the tailings management facility. Part of the water management could include a pump station with weir in Talc Creek just below the confluence of Daioff Creek. This Daioff pump station could deliver seepage from the dam and Daioff waters to the tailings management facility and/or directly to the mill as make-up water.

Electrical power and natural gas are available at or near Ruby Creek. Delivery of power and gas to the site would be via transmission systems installed to parallel the main Garnet Creek access road (approximately 16 km).

Cogburn Project

30 BCC96124 No.33



Preliminary Site Development Plan

The metallurgy of the Cogburn deposit at this time is unknown, as no records of the early (1971) bench scale recovery tests have been found. However, the nickel is in sulfide form (+90%) and should respond to flotation. New hydrometallurgical technology (PlatSol) has the potential to facilitate on-site production of high-purity metals (nickel, copper, cobalt, PGE's and precious metals) from low-grade flotation concentrate (with the bonus of high nickel recovery to concentrate). If the PlatSol technology developed by Dreisinger, O'Kane and Lakefield Research Limited has application at Cogburn then it may be wise to consider a high-density concentrate slurry pipeline from the site to Ruby Creek for pressure oxidation and electrowinning. Recent advances in heap and vat bioleaching of nickel sulfide ores have been reported by Billiton PLC and Titan Resources NL (see Appendix III).

The project economics have the potential to be robust with a C\$29 per tonne gross value, considering only the nickel content. The estimated project capital cost for development as an open pit mine with SAG/ball milling at 20,000 tonnes per day and flotation to produce a nickel concentrate, would be about C\$200 million. Assuming a 0.25 to 1.00 strip ratio the mining, milling and administrative costs will total about C\$8 per tonne milled. Assuming an 80% recovery of nickel to concentrate, the recovered gross value per tonne milled would be C\$23. This would leave a healthy \$15, after deduction of mine costs, to pay concentrate handling, smelting and refining, as well as provide a profit margin.

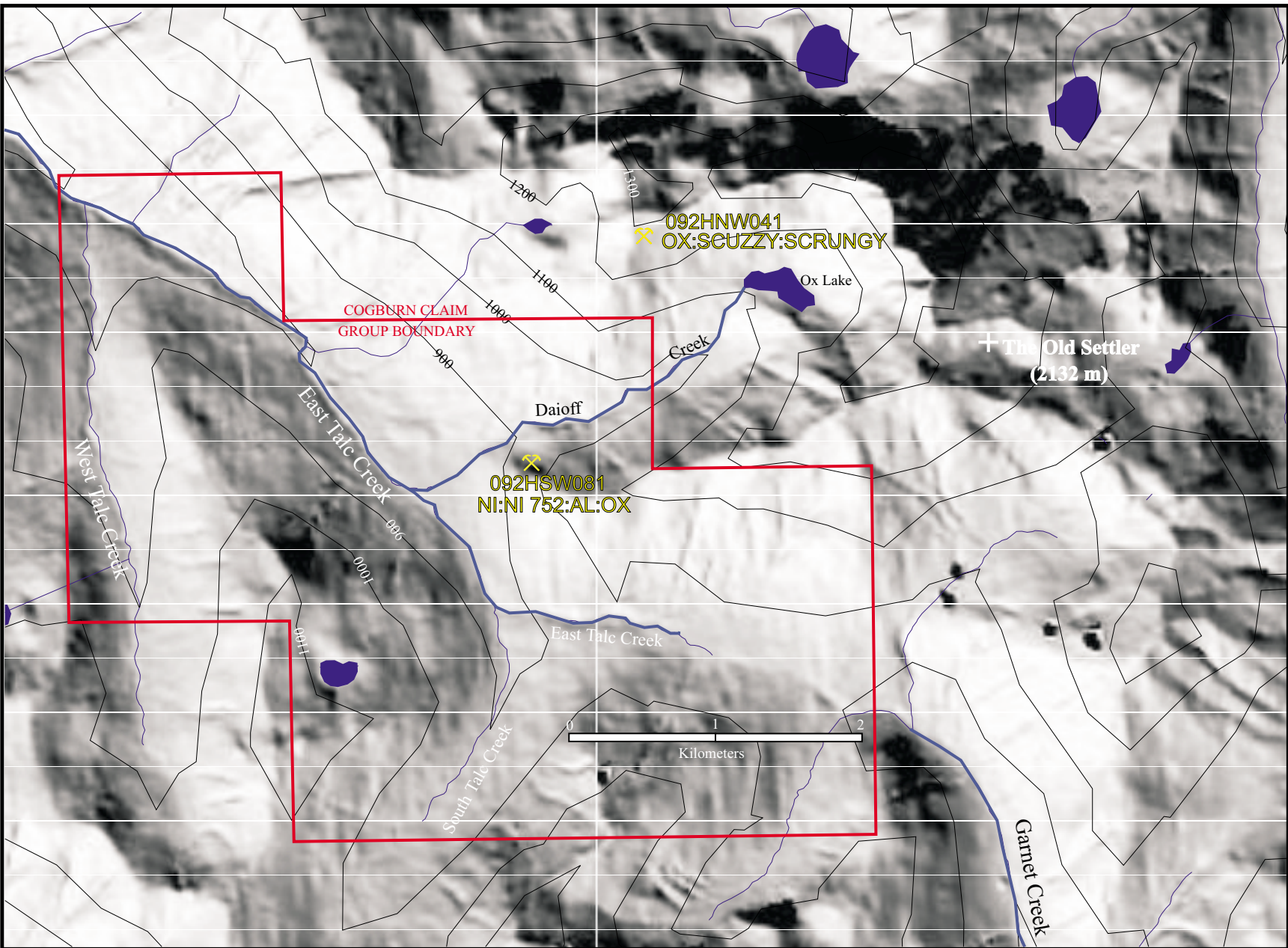
SOCIO-ECONOMIC IMPACT OF MINE DEVELOPMENT

The development of a +20,000 tpd mine operation would result in a significant positive impact on the economy of the Hope, Agassiz and Chilliwack region and the Province of British Columbia. The mine operations would sustain approximately 800 direct, indirect and induced full time highly paid jobs in the Province, over a broad range of job skills.

The operation would inject about \$50 million annually into the British Columbia economy, through expenditures on labour, goods and services. The Hope, Agassiz and Chilliwack region would be the largest beneficiary as over \$25 million would be spent there annually.

The large population and infrastructure base in the region is conducive to efficient mine development and mine operations, to the mutual benefit of the communities in the region, the Provincial and Federal governments (taxes on profits) and the Cogburn property owners.

APPENDIX I



COGBURN CLAIM GROUP

New Westminister Mining Division
 British Columbia
 Canada

NTS:
 092H05E, 09H12E

Digital Elevation Model Mosaic
 Topographical Map

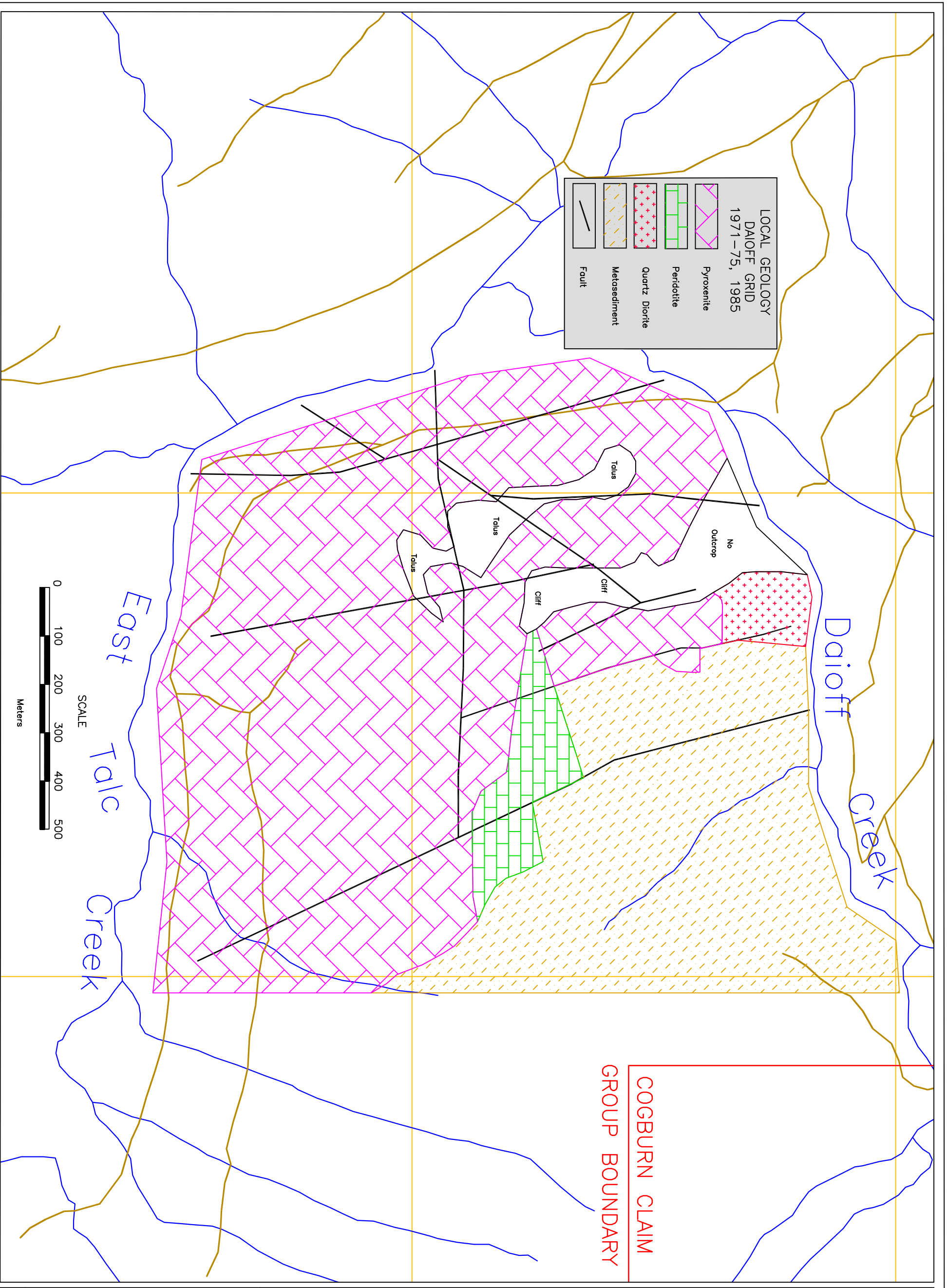
Figure 3

Source : The Map Place

Scale : As Shown

Drawn By: DKM, P.Eng.

Date: May 30, 2000



COGBURN CLAIM GROUP

New Westminster Mining Division, British Columbia Canada

NTS:
092H05E, 092H12E

Property Geology Map

Figure 6

Source : A.R. 3356, 5527, 14402

Scale : As Shown

Drawn By : DKM, P.Eng.

Date : May 30, 2000



COGBURN CLAIM GROUP

New Westminster Mining Division,
British Columbia
Canada

NTS:
092H05E, 092H12E

Property
Soil and Rock
Geochemistry
Map

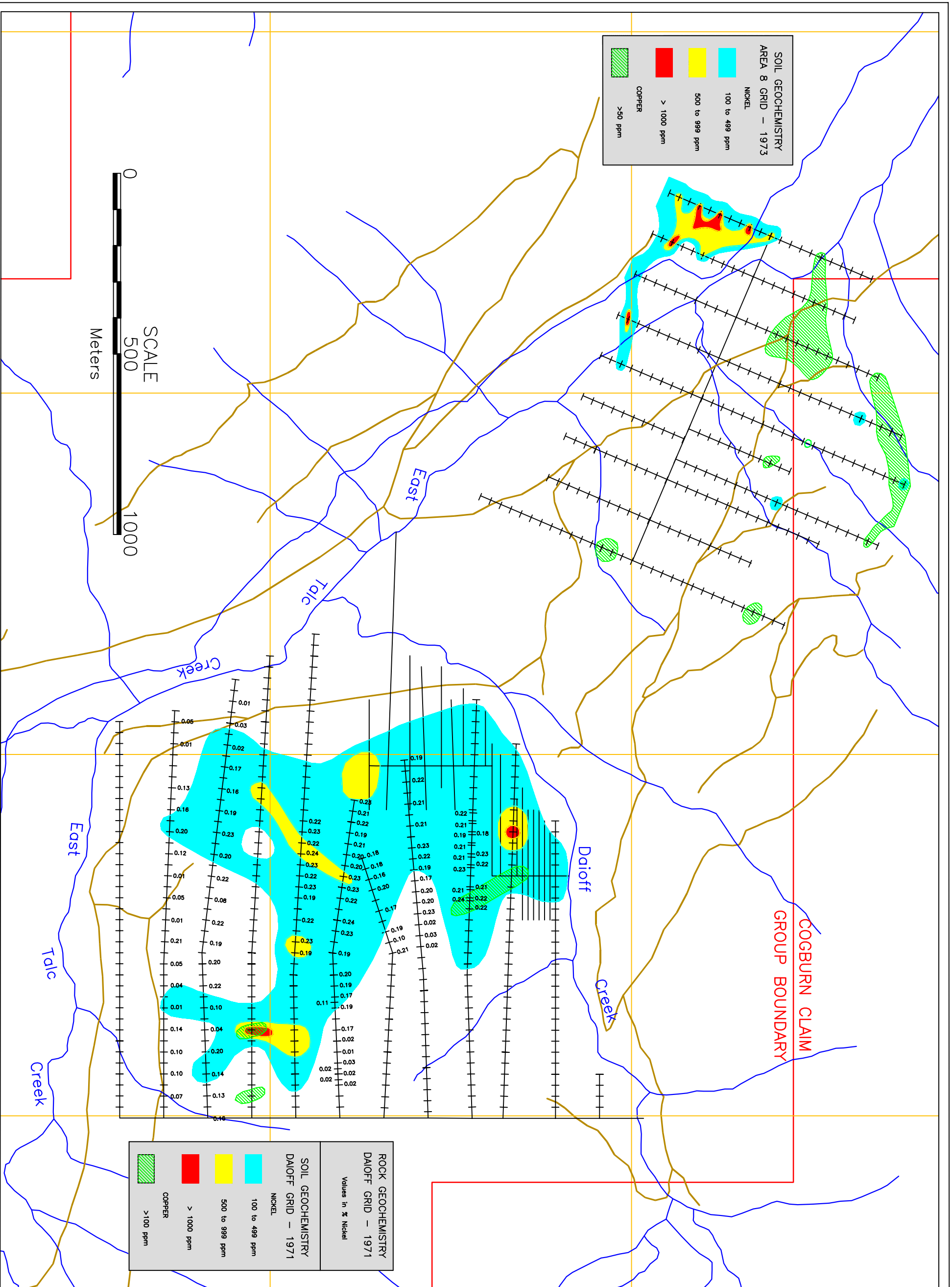
Figure 8

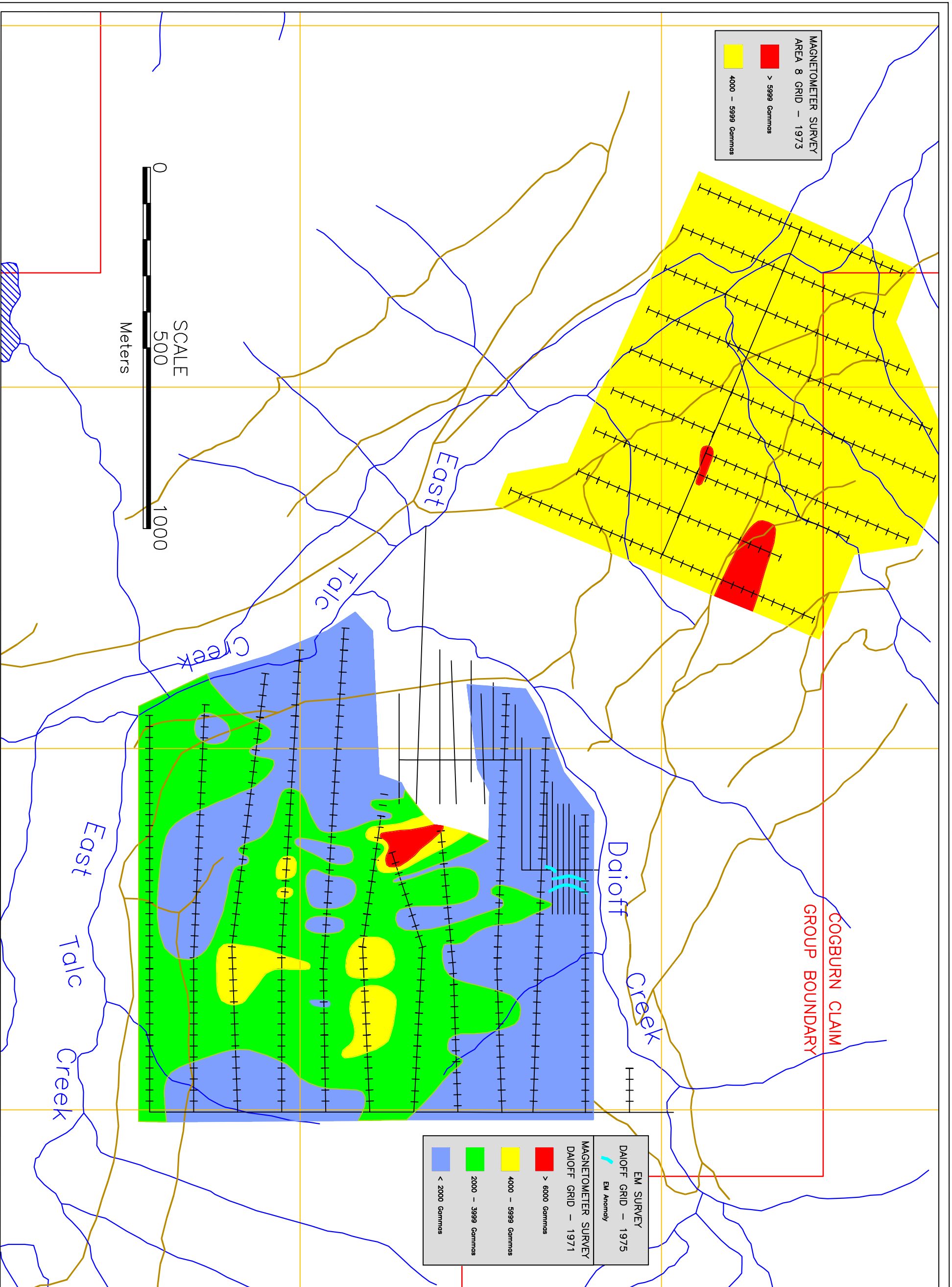
Source : A.R. 3356, 4536

Scale : As Shown

Drawn By : DKM, P.Eng.

Date : May 30, 2000





MAGNETOMETER SURVEY
 AREA 8 GRID - 1973

- > 5999 Gammas
- 4000 - 5999 Gammas

EM SURVEY
 DAIOFF GRID - 1975

EM Anomaly

MAGNETOMETER SURVEY
 DAIOFF GRID - 1971

- < 2000 Gammas
- 2000 - 3999 Gammas
- 4000 - 5999 Gammas
- > 6000 Gammas



**COGBURN
 CLAIM
 GROUP**

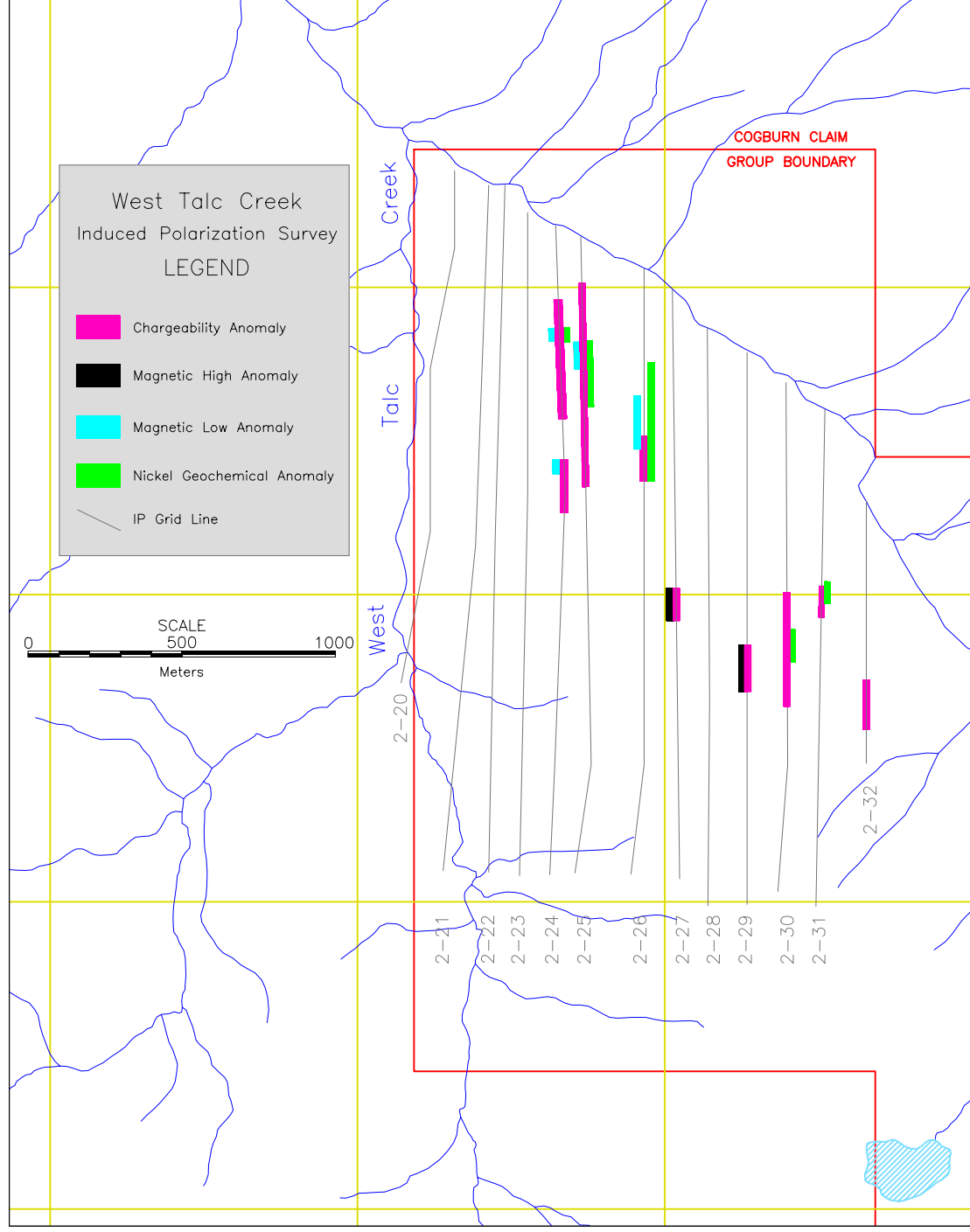
New Westminster
 Mining Division,
 British Columbia
 Canada

NTS:
 092H05E, 092H12E

Property
 Magnetometer
 & EM Anomaly
 Map

Figure 9

Scale : As Shown
 Drawn By : DKM, P.Eng.
 Date : May 30, 2000



COGBURN CLAIM GROUP

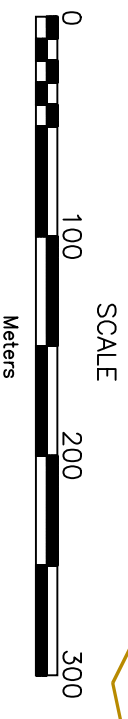
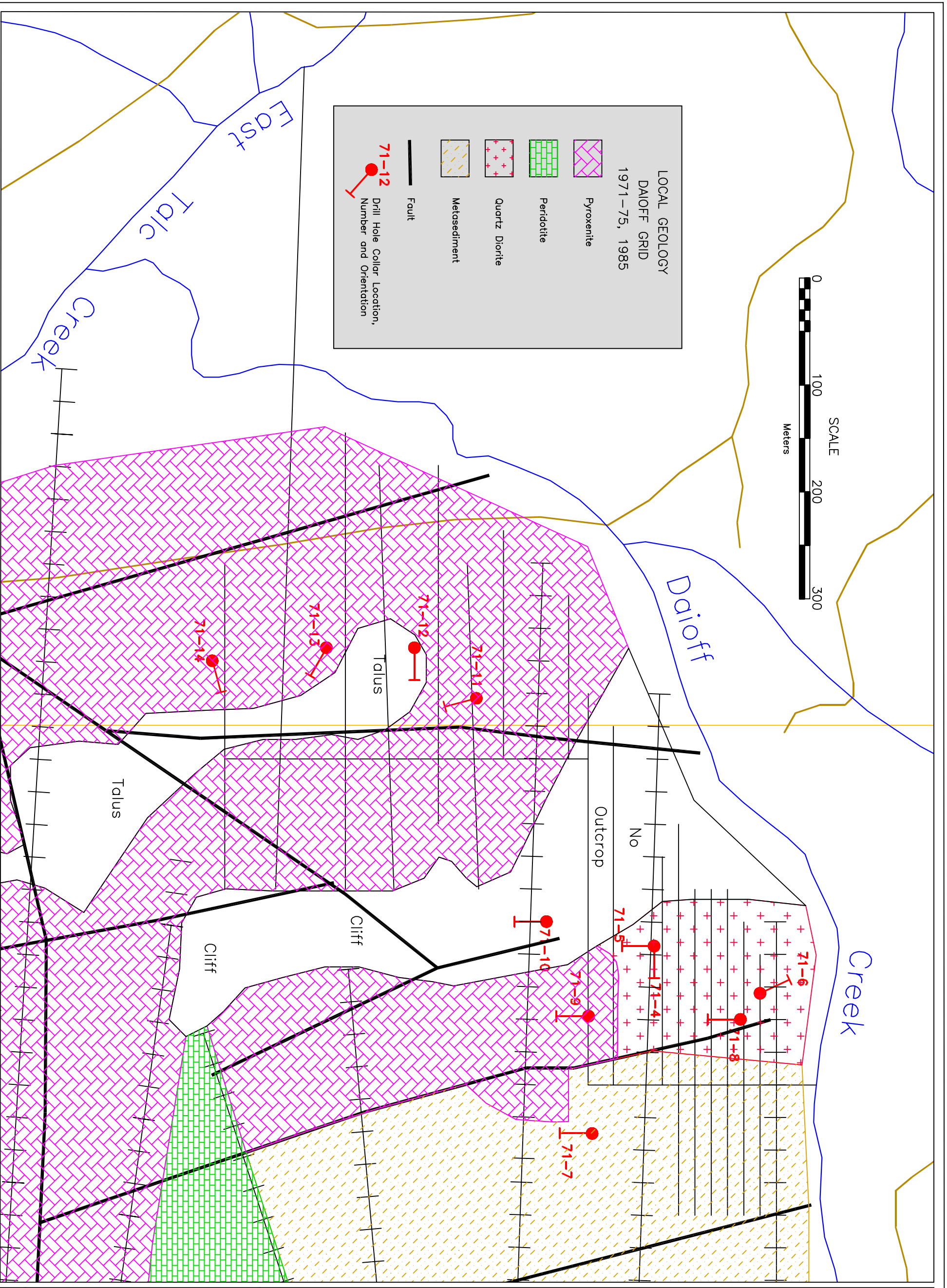
New Westminster Mining Division,
British Columbia
Canada

NTS:
092H05E, 092H12E

West Talc Creek
Geophysics
Anomaly
Map

Figure 10

Source : A.R. 3614
Scale : As Shown
Drawn By : DKM, P.Eng.
Date : May 30, 2000



LOCAL GEOLOGY
 DAIOFF GRID
 1971-75, 1985

- Pyroxenite
- Peridotite
- Quartz Diorite
- Metasediment
- Fault
- 71-12 Drill Hole Collar Location, Number and Orientation



COGBURN CLAIM GROUP

New Westminster Mining Division, British Columbia Canada

NTS: 092H05E, 092H12E

Drill Hole Location Map

Figure 11

Source : A.R. 3356, 5527, 14402

Scale : As Shown

Drawn By : DKM, P.Eng.

Date : May 30, 2000

APPENDIX II

COGBURN PROPERTY

Fact Sheet, August 2000

- Location**
- 120 km east of Vancouver, near Hope, British Columbia, Canada. Reference NTS maps 092H05E and 092H12E.
- Access**
- By logging road from Ruby Creek at Highway 7 then up Garnet Creek to its headwaters (16km), or from Harrison Hot Springs by logging road up Cogburn Creek and then to the headwaters of Talc Creek (42km).
- Land Package Ownership**
- Nine contiguous Mineral Claims consisting of 68 Units (1,700ha) in the New Westminster Mining Division.
 - John A. Chapman (50%) and KGE Management Ltd. (50%). Gerald G. Carlson is the President and major shareholder of KGE Management Ltd.
- Completed Exploration**
- In 1971, Giant Explorations Ltd. (a subsidiary of Giant Mascot Mines Ltd.) discovered the Cogburn nickel deposit while conducting a wide area airborne geophysical and stream silt geochemistry program. The survey area, which identified a number of ultramafic intrusions, covered a 12km wide swath from the Giant Mascot nickel, copper, cobalt mine northeast 25km to Harrison Lake. This preliminary work was followed by grid surveys over the Cogburn deposit area, including soil geochemistry, magnetics and rock chip sampling and then core drilling (1971 to 1975).
- Resources**
- The only mention of resource potential was in the George Cross News Letter of September 1, 1971, "Present indications are in the order of 200,000 tons per vertical foot, which gives 100 million tons per 500 feet of depth ... A number of the samples from the property have been subjected to carefully controlled ascorbic acid tests which indicate that between 90% and 95% of the nickel is in sulfide form ... All of the samples from within the discovery area have presented a remarkably consistent value between 0.19% and 0.25% nickel". G.E. Eastwood, a geologist with the BC Department of Mines, in his property report (EMPR GEM 1971 pgs 258-264) stated, "samples from southeast of Daioff Creek (Cogburn deposit) contained 0.19% to 0.22% sulfide nickel These results are to be compared with Nickel Syndicate averages of 0.22% nickel obtained from systematic rock chip sampling over an area of approximately 80 acres and 0.20% from diamond-drill core ... Southeast of Daioff Creek 17 holes were diamond drilled to an aggregate length of over 4,000 feet". PGE values within the deposit are not known (no assays).
- Area Production**
- Eight kilometers to the east of Cogburn is the former Pride of Emory mine that was operated by Giant Mascot Mines Ltd. from 1958 to 1974. The mine produced 4.3 million tonnes of ore and recovered 26.6 million kg of nickel, 13.2 million kg of copper, 140 thousand kg of cobalt and minor gold and silver. At closure in 1974 the resource estimate was 863 thousand tonnes grading 0.75% nickel, 0.30% copper and 0.03% cobalt. Zones within the Pride of Emory deposit were reported to contain PGE's. **The Minfile No. 092HSW004 reports, "In 1936, one 22.7 tonne bulk sample taken from the 488 metre (1,600 feet) crosscut averaged 2.74 grams per tonne platinum and palladium and 0.68 grams per tonne gold. Early samples of ore yielded 3.98 grams per tonne platinum and palladium and 7.89 grams per tonne gold In 1978, three samples collected on the surface were anomalous in platinum and yielded 1.17, 1.61 and 1.61 grams per tonne platinum respectively ... One high-grade sample from the bottom of the "1500" orebody assayed 2.85 grams per tonne platinum and 4.94 grams per tonne palladium".**
- Geology**
- The Cogburn deposit, similar to the Pride of Emory, is within ultramafic rocks composed primarily of altered pyroxenite and peridotite. The pyroxenite is strongly unaltered and contains pyrrhotite with minor magnetite and traces of pyrite, chalcopyrite and possibly pentlandite that are all finely disseminated throughout the rock.
- Environment**
- The deposit is located between 800 meters and 1,500 meters elevation on the headwaters of Talc and Garnet Creeks. The area is being actively logged. In the past 10 years approximately 60% of the claim area has been clear-cut.
- Potential**
- The Cogburn deposit and surrounding area has potential for discovery of further nickel, copper, cobalt and PGE's. The deposit setting lends it to development as a large low-grade open pit operation. Major highways, high capacity electric transmission lines and a high capacity natural gas pipeline are all located within 16 kilometers of the deposit.
- Area Claims**
- Santoy Resources Ltd. a Ron Netolitzky Group Company has recently staked 6,500 hectares of claims nearby to the east boundary of the Cogburn property. A private group of mining professionals has staked 10,400 hectares in the past eight weeks to the immediate north of the Cogburn property.
- Status**
- The Cogburn claims are available for option. Contact John Chapman at 604-536-8356 (E-mail: jacms1@sprynet.com), or Gerald Carlson at 604-688-0833 (E-mail: gcarlson@copper-ridge.com).



MINFILE Capsule Geology and Bibliography

[MINFILE Home page](#)

[Main Search Menu](#)

[Back to Search Results](#)

[ARIS Home page](#)

Capsule Geology and Bibliography

Next Hit

092HSW004

[Production Report](#)

[Inventory Report](#)

| | | | |
|-------------------------------|---|------------------------|--|
| Name | PRIDE OF EMORY | Mining Division | New Westminster |
| Status | Past Producer | NTS | 092H05E NAD 27 |
| Latitude Longitude | 49 28 30 N 121 30 46 W | UTM | 10 5481105 607742 |
| Commodities | Nickel Copper Cobalt Gold Silver Chromium Platinum Palladium Zinc | Deposit Types | M02 : Tholeiitic intrusion-hosted Ni-Cu. |
| Tectonic Belt | Coast Crystalline | Terranes | Plutonic Rocks. Undivided Metamorphic Assembl.. |

| | |
|------------------------|---|
| Capsule Geology | <p>The Pride of Emory mine is located at the head of Texas Creek, approximately 12 kilometres north of Hope on the west side of the Fraser River. Access is by mine road from the Trans-Canada Highway. The mine operated during the period 1958 to 1974 and is presently undergoing re-evaluation by Corona Corporation.</p> <p>The property lies within an ultramafic complex between the southern tip of the Tertiary-Jurassic Coast Plutonic Complex and the northern end of a belt of intrusions termed the Chelan batholith. The intrusive rocks within this belt are granites, granodiorites and quartz diorites of Jurassic age and younger. They form the core of an uplifted block of regionally metamorphosed upper Paleozoic rocks which trend north and are bounded by the Fraser River fault system on the east and on the west by somewhat less metamorphosed Mesozoic rocks.</p> <p>The ultramafic complex hosting the mineralized zones is composed of hypersthene diorite and quartz diorites, norites and ultramafic rocks, termed the Pacific Nickel Complex, which intrudes schists and earlier intrusive rocks. The older noritic rocks are found northwest and southwest of the ultramafic complex. Potassium-argon ages from the ultramafic complex range from about 120 to 95 million years. The older ages were obtained from the hornblende pyroxenite phase with late hornblende dikes having the youngest ages. The ultramafite is truncated by diorite of the Late Cretaceous Spuzzum intrusions, at 79 to 89 Ma (McLeod, J.A. et. al., 1976).</p> <p>The ultramafic rocks of the Pacific Nickel Complex form an irregular stock-like mass about 3 kilometres across. The northeastern half of the stock consists of barren pyroxenites and peridotites which contain little or no hornblende. The southwestern half of the stock is a highly</p> |
|------------------------|---|

variable, hornblende-rich assemblage of peridotites and pyroxenites which are mineralized and contain some seventeen orebodies associated with the mine. These orebodies are scattered along a line trending about 285 degrees.

Ore is associated with pipe-like concentrations of enstatite, olivine and hypersthene containing in order of abundance: pyrrhotite, pentlandite, chalcopyrite, magnetite, pyrite and sphalerite with lesser amounts of chromite and cobalt minerals which include linnaeite and morenosite. The orebodies fall into two types which grade into one another: zoned, with gradational contacts and disseminated sulphides; unzoned, massive sulphides with sharp contacts.

The zoned orebody is the most common with the sulphide present as interstitial grains between olivine and orthopyroxene. Sulphide concentration is the greatest in the core and decreasing outward to minor, patchy disseminations.

Massive orebodies exhibit consistent composition and grade and consists of orthopyroxene with olivine in a groundmass of sulphides. Contacts between the ore and the country rock are sharp. This type of orebody is generally irregular in plan and plunges steeply. The Pride of Emory orebody is composed of massive and zoned parts so that it appears the two types may grade into one another.

Faulting is widespread throughout the underground workings and may be grouped into two broad categories. The first group strikes 310 degrees and dips 50 to 75 degrees northeast. The second group strikes between 335 and 025 degrees with steep dips to the east or west. At least five northwest trending fault zones have been traced up to 609 metres; these are thought to be post-ore features.

Alteration on the property does not appear to be related to mineralization or mineralizing processes. Talc-amphibole alteration produces a soft grey rock which is resistant to weathering. It appears to be related to faults or fractures. Crumbly alteration found in the peridotites appears to be fault related and is recognized by the development of talc along cracks and grain boundaries. This crumbly alteration produces soft rocks which weather deeply and is found in the underground workings. Uralitization (propylitization) is related to faulting and is accompanied by small pegmatitic and aplitic veinlets. Serpentine has developed in some of the peridotites and along strong faults. Violarite is secondary after pentlandite and occurs bordering cracks and cleavages of that mineral. Limonite occurs in narrow, sinuous veinlets that cut both sulphide and gangue minerals. Greenish white stains and crusts of the hydrous nickel sulphate, morenosite, have been identified.

From 1936 to 1974, 4,319,976 tonnes of ore was mined yielding 16,516 grams of silver, 1,026 grams gold, 140,700 kilograms cobalt, 13,212,770 kilograms copper and 26,573,090 nickel. Ore graded about 0.77 per cent nickel and 0.34 per cent copper with cobalt as a byproduct. However, chrome oxide, platinum, gold and silver are also present (Assessment Report 16553).

In 1936, one 22.7 tonne bulk sample taken from the 488 metre (1600 feet) crosscut averaged 2.74 grams per tonne platinum and palladium and 0.68 gram per tonne gold. Early records of samples of ore yielded 3.98 grams per tonne platinum and palladium and 7.89 grams per tonne gold. The chrome content of the ore averaged 0.2 to 0.4 per cent.

In 1987, 63 samples were collected and all were anomalous in chromium with assays up to 1.28 per cent. Three samples collected on the surface were anomalous in platinum and yielded 1.17, 1.61 and 1.61 grams per tonne platinum, respectively. One high-grade sample from the bottom of the "1500" orebody assayed 2.85 grams per tonne platinum and 4.94 grams per tonne palladium. The best gold assay from the rock samples was 0.93 gram per tonne gold.

Exploration and development of the Pride of Emory deposit has had a long history. The original showings were staked and prospected in 1923. In 1926, the B.C. Nickel Co. Ltd. began underground development of the original showings. Further development continued through to 1938 until poor market conditions forced a closure. In 1952, Newmont Mining Co. and Pacific Nickel Mines Ltd. formed Western Nickel Mines Ltd. to reopen the workings and drive the 2600 main haulage level and other adits. Mine and mill development proceeded to 1958 and commercial production began. Production continued from 1958 to 1974, at which time the

economic limit of mining was reached. The scattered nature of the mineralized zones and the fact that most of them were blind, forced the closure of the mine.

Combined (proven/probable) reserves in 15 zones are 863,000 tonnes grading 0.75 per cent nickel, 0.3 per cent copper and 0.03 per cent cobalt (Property File - Christopher, P.A., 1975).

See also Star of Emory 3 (092HSW093) and Choate (092HSW125).

Bibliography

EMPR AR 1924-137; 1928-227; 1929-239; 1930-204; 1934-F17,F19; 1935-F58; 1936-F64; 1937-F37; 1952-208; 1953-158; *1954-160-163; 1957-66; 1958-55; 1959-124-127; 1960-87-88; 1961-86-88; 1962-91; 1963-89; *1964-137-142; *1965-213-217; 1966-58; 1967-63; 1968-76; 1969-A54; 1970-A53; 1971-A53; 1972-A53; 1973-A54
 EMPR ASS RPT [5385](#), [*16553](#), [19035](#)
 EMPR BC METAL MM00225
 EMPR FIELDWORK 1974, pp. 17-21; 1975, pp. 49-52
 EMPR GEM 1969-196; 1970-248; 1971-267; 1972-117; 1973-131,132; *1974-105-113
 EMPR INDEX 3-188; 4-124
 EMPR MAP 65 (1989)
 EMPR OF 1986-7, p. 30; 1990-27; 1992-1
 EMPR P 1991-4, pp. 134-137
 EMPR PF (Miscellaneous References, maps and photos; Miscellaneous published documents and inter-office correspondence; Giant Mascot Mines Information Circular; Giant Explorations Ltd.-Fifth Annual Report Nov.30, 1970; Giant Mascot Mines - Twenty First Annual Report 1971; Grove, E.W., et al: Report on the Giant Mascot Mine-With an Appraisal of the Present Situation and Future Outlook, Apr.23, 1974; Grove, E.W.: Proposal on the Giant Mascot Ultrabasic Project, May 1974; Starck, L.P.: Supplement to Sept. '74-Giant Mascot Mines Ltd.; Starck, L.P.: Report on Giant Mascot Mines Ltd and Its Subsidiaries and Associates up to Feb.28, 1974; Christopher, P.A.: Report on the Giant Mascot Ultrabasic Project-Summary Report 1974, Jan. 1975)
 EMR MP CORPFILE (B.C. Nickel Mines Ltd.; Pacific Nickel Mines Ltd.; Western Nickel Mines Ltd.; Giant Mascot Mines Ltd.; Newmont Mining Corporation of Canada Ltd.; Granby Mining Company Ltd.)
 GSC MAP 12-1969; 737A; 1386A; 41-1989
 GSC MEM *190, pp. 1-15
 GSC P *69-47, pp. 63,64; 90-1E, pp. 183-195
 GSC SUM RPT *1924 Part A, pp. 100-106; *1933 Part A, pp. 62-68
 CANMET IR 763 (1935), p. 320; 688 (1936), pp. 43-82
 CIM *Vol.2 (1957), pp. 27-36
 CJES No. 13, 1976, pp. 1152-1154
 ECON GEOL *Vol.51 (1956), pp. 448-481
 GCNL #17, 1961; #96, 1968
 W MINER *Vol.44 (1971), pp. 23-61; Jun. 1969, pp. 41-46; Vol.33, Nov. 1960, pp. 39-42
 Muir, (1972): A Study on the Petrology and Ore Genesis of the Giant Nickel 4600 Orebody, Hope, British Columbia, unpublished M.Sc. Thesis, University of Toronto
 Western Canadian Investments No. 39, 1973

Database last posted: May 02, 2000

Go to: [Main Search Menu](#); [MINFILE Name/No. Search](#); [Commodity/Status/NTS Search](#); [Deposit Type Search](#); [Tectonic Belt/Terrane/Latitude/Longitude Search](#)



MINFILE Capsule Geology and Bibliography

[MINFILE Home page](#)

[Main Search Menu](#)

[Back to Search Results](#)

[ARIS Home page](#)

Capsule Geology and Bibliography

092HSW081

| | | | |
|-------------------------------|---|------------------------|--|
| Name | NI | Mining Division | New Westminster |
| Status | Prospect | NTS | 092H05E ^{NAD 27} |
| Latitude Longitude | 49 29 49 N 121 39 28 W | UTM | 10 5483350 597200 |
| Commodities | Nickel Copper | Deposit Types | M02 : Tholeiitic intrusion-hosted Ni-Cu. |
| Tectonic Belt | Coast Crystalline | Terranes | Bridge River. |

| | |
|------------------------|---|
| Capsule Geology | <p>The area of the Ni prospect is underlain by a metamorphic package of Paleozoic and/or Mesozoic rocks, possibly correlative with the Permian-Jurassic Hozameen Complex, that were metamorphosed in the Cretaceous. These rocks comprise quartz-muscovite-garnet schist and metavolcanics consisting of hornblende and hornblende feldspar schists. The regional foliation strikes from 320 degrees with a 45 degree northeast dip to 290 degrees with an 85 degree northeast dip, and is generally parallel to layering although some foliation across layering has suggested the occurrence of isoclinal folding. Intimately associated with these Hozameen-derived rocks is a northwest trending fault-bound body of Mesozoic and/or Paleozoic serpentinized pyroxenites and peridotites stretching for about 10 kilometres.</p> <p>The country rocks are intruded by quartz diorite plugs and dikes associated with the Late Cretaceous Spuzzum pluton and Monger has mapped a small stock of Oligocene granodiorite just north of the prospect (GSC Map 41-1989).</p> <p>Mineralization identified on surface and drillholes consists of pyrrhotite, up to 4 per cent, with trace chalcopyrite and pyrite. Pyrrhotite occurs widely but sparingly in the ultramafics. It occurs as grains and blebs interstitial to pyroxene crystals, as scattered grains along microshears, and as narrow fracture veinlets less than a few centimetres in length. Chalcopyrite has been found sparsely disseminated in the pyrrhotite. Pyrite is sparingly distributed through most of the rocks of the area, but is lacking in parts of the ultramafic body. Pentlandite has been reported to occur with the above minerals as disseminations (Geology, Exploration and Mining in B.C. 1975, page 74). Samples of mineralized rock yielded 0.19 to 0.22 per cent nickel and trace copper (Geology, Exploration and Mining in B.C. 1971, page 262).</p> <p>Exploration, which commenced on the Ni claims in 1969 and proceeded through to 1975, was conducted by the Nickel Syndicate, a joint venture between Giant Explorations Limited and Giant Mascot Mines Limited. Extensive geochemical, geophysical and sampling programs were followed by diamond drilling at favorable locations.</p> |
|------------------------|---|

| | |
|---------------------|--|
| Bibliography | EMPR GEM 1970-248; *1971-258-264; 1972-116; 1973-153; EMPR EXPL 1975-E74 EMPR ASS RPT 2469 , 2583 , 2801 , 3155 , 3280 , 3356 , 3442 , 3614 , 3615 , 3901 , 3956 , 4070 , 4071 , 4536 , 5527 , 14402 EMPR OF *1990-27, pp. 45,47 GSC P 69-47 GSC MAP 12-1969; 737A; 1069A; 41-1989 |
|---------------------|--|

Database last posted: May 02, 2000

Go to: [Main Search Menu](#); [MINFILE Name/No. Search](#); [Commodity/Status/NTS Search](#); [Deposit Type Search](#); [Tectonic Belt/Terrane/Latitude/Longitude Search](#)

[MINFILE Home page](#)

This page last updated: November 29, 1999



538 HOWE STREET
SUITE 203
VANCOUVER 1, B.C.
683-7265
(AREA CODE 604)

COPYRIGHT
ALL REPRODUCTION
RIGHT RESERVED
PUBLISHED DAILY
SUBSCRIPTION RATE

\$160.00 per year
NO.170(1971)
SEPTEMBER 1,1971

George Cross News Letter

"Reliable Reporting"

WESTERN CANADIAN INVESTMENTS
GIANT EXPLORATIONS LTD.

0.170(1971)
SEPTEMBER 1,1971

+ Major Tonnage of Low Grade Nickel Discovered at Harrison Lake, B.C.

Initial diamond drill holes and widespread surface sampling at the Nickel Syndicate, Harrison Lake, B.C. joint venture exploration project of Giant Explorations Ltd. and Giant Mascot Mines Ltd. has discovered a major tonnage of low grade nickel mineralization. Mineralization is over a length of 2,000 feet, a width of 800 feet and over a vertical surface range of 700 feet. The nickel mineralization has an indicated gross value of \$5.00 per ton, before considering recoverable values.

Interest in the immediate area of the discovery, which is named 2A, started with favourable aerial magnetometer readings, and was reinforced by initial soil sampling results. The recent work has included collection and assaying of in excess of 50 surface rock samples on a grid over the favourable, altered pyroxinite geology. To date, three diamond drill holes have been completed carrying the same values as the surface sampling. A second diamond drilling machine has been moved to the area since the discovery. The drilling is planned to expand the known depth and size of the discovery area. Present indications are in the order of 200,000 tons per vertical foot, which gives 100,000,000 tons per 500 feet of depth. The mineralized area is open to possible extension to the south and east.

A number of samples from the property have been subjected to carefully controlled ascorbic acid tests which indicate that between 90% and 95% of the nickel is in sulphide form.

Metallurgical testing will be undertaken to determine the recoverability of the minerals. Work to date has shown an almost complete absence of copper but suggests possible other minerals of interest. All of the samples from within the discovery area have presented a remarkably consistent value between 0.19% and 0.25% nickel. The consistency of the values, as much as the grade, is the outstanding feature.

A logging road, built several years ago, provides access to within several hundred feet of the discovery area.

There is almost no overburden in the area and the discovery is situated on a steeply sloping ridge ideal for an openpit layout.

In announcing the discovery on the large 564 claim property, a joint venture between, Giant Mascot Mines Ltd. and Giant Explorations, L.P. Starck, president of both companies, stated that the pace of activities in the 2A area has been stepped up to increase the rate of collection of data.

A subsequent report will be issued in the next few weeks when additional assays have been received and evaluated, he stated.

GIANT EXPLORATIONS LIMITEDGIANT MASCOT MINES LIMITED

EXPLORATION PROGRAM REVIEWED - The 1971 exploration program of Giant Explorations Limited and Giant Mascot Mines Limited on the 583 claim Nickel Syndicate property located on the east side of Harrison Lake, B.C., was completed in mid-November. Over 100 miles of surveyed grid lines in 6 selected areas provided control for geological mapping, geochemical and magnetometer surveys, with 10.6 line miles of induced polarization surveys covering specific targets. Nickel-copper-iron mineralization was discovered in typical ultramafic host rocks in all of these areas and 5,760 feet of diamond drilling were done in 20 holes to test certain sectors. Reconnaissance exploration of the intermediate ground between these 6 selected area points to other areas requiring detailed examination.

Because of interesting mineralizing and attendant geological conditions, two sectors received most of the follow-up work in the form of diamond drilling and induced polarization surveys. In Area 2A-6, which is centrally located on the property, a 2,500 square foot mineralized zone of altered pyroxenite grading 0.22% nickel, with negligible copper values, was indicated by surface sampling and 1,980 feet of diamond drilling, over a vertical range of 1,600 feet. The implications of a high tonnage open pit situation in this area indicated the necessity for metallurgical testing to determine recoverable nickel values. Some preliminary testing has been completed.

In Area 4, on the south side of Cogburn Creek, a total of 3,455 feet of diamond drilling in 9 holes returned several assays up to 0.5% combined nickel and copper over narrow widths in the ultrabasics.

The results of the 1971 reconnaissance exploration work of the Hedley Syndicate, a 50-50 participation with Pacific Petroleum Limited, is presently being correlated.

On the Mahwitte Lake property the several induced polarization anomalous zones on the "Norman" grid, at the west end of the property, were examined on the ground. Sulphide mineralization in these localities was largely pyrite, but two zones of intrusive rocks warrant further investigation.

APPENDIX III

HYDROMETALLURGICAL PROCESS DEVELOPMENT AT POLYMET MINING CORPORATION'S NORTHMET PROJECT

P.T. O'Kane, O'Kane Consultants Inc., Vancouver, BC
C.L. Brierley, Brierley Consultancy LLC, Highlands Ranch, CO
D.B. Dreisinger, University of British Columbia, Vancouver, BC
D.W. Gentry, PolyMet Mining Corporation, Golden, CO

March 9, 2000

ABSTRACT

PolyMet Mining Corporation's large, low-grade, disseminated, polymetallic project (NorthMet) is representative of other Cu-Ni deposits in Minnesota's Duluth Complex. These unique mineralized bodies, often enriched with PGMs, have been exploration/development targets for some 30 years, because of the magnitude of the resource, which includes copper, nickel, cobalt, palladium, platinum, rhodium, ruthenium, iridium and osmium. However, these types of deposits have not been viable until now, because saleable, high-quality concentrates could not be produced and traditional metallurgical processes rendered the projects uneconomic. Technology advances during the last five years in two metallurgical processes, pressure oxidation and bioleaching, offer a solution for economically viable recovery of metals from deposits in the Duluth Complex. This paper describes pressure oxidation and bioleach testwork performed on a bulk sulfide concentrate from the NorthMet Project, illustrates viable flowsheet options, including a "hybrid" flowsheet that couples pressure oxidation and bioleaching, and provides the overall economics of each process option.

BACKGROUND

The Cu-Ni Pt-Pd deposits located in the Duluth Complex in Minnesota have been known for at least three decades. The Duluth complex is a large series of mafic to felsic intrusive bodies that host these often large, low-grade, disseminated polymetallic deposits, some of which are enriched with platinum group metals (PGMs). These deposits represent a unique class of orebody, magmatic sulfide deposits, which are the principal concentrations of group VIII transition metals - iron (Fe), cobalt (Co), nickel (Ni), palladium (Pd), platinum (Pt), rhodium (Rh), ruthenium (Ru), iridium (Ir), and osmium (Os).

The two larger and better-known of these deposits – NorthMet (formerly called Dunka Road) and Babbitt (subsequently the Minnamax Project) – were discovered in the late 1960's by US Steel (USX) and Kennecott Copper Corporation (Bear Creek Mining Company) respectively. USX drilled 112 diamond drill holes into the NorthMet deposit along a geophysical grid on 183-meter centers. The Babbitt or Minnamax Project, as it became known in 1974 when AMAX became partners with Kennecott, was extensively drilled to a “reserve” status in the late 1970's. In addition, a 12-ft diameter shaft was sunk to the 1700-ft level in a higher-grade portion of the deposit for the purposes of determining continuity of mineralization by drifting through the mineralization, coupled by detailed underground fan drilling, and the collection of a bulk sample for metallurgical sampling.

In both cases, the enormous magnitude of the mineralized resource led property owners to think in terms of large, underground mining operations at appropriately high cut-off grades, with the sale of concentrates to a U.S. or Canadian smelter for processing. Unfortunately, neither USX nor AMAX were able to produce satisfactory concentrate for sale. A bulk concentrate was too low in grade to be marketable, and clean, high-quality copper and nickel concentrates could not be produced consistently.

Although alternative metallurgical processes were tested throughout the intervening years, none was found that made these projects economically viable. In 1976, USX performed a few analyses for PGMs, but the project remained uneconomic due to low metals prices and poor metallurgical recoveries. The Minnamax Project was abandoned in 1982 for essentially the same reasons. Unfortunately, sporadic and poorly defined metallurgical testing of various types occurred well into the 1990's with very limited success. Thus these polymetallic deposits continued to suffer from a reputation of being too complex metallurgically to ever be economic.

In 1987 the Minnesota Natural Resources Research Institute published PGM data collected from a number of the Duluth Complex deposits and noted the possibility for a large resource of PGMs along the base of the Complex. Occasional metallurgical testing to recover PGM values in subsequent years by various leaseholders proved largely unsuccessful in solving the metals recovery problems that surrounded these deposits in the Duluth Complex.

In 1989, Fleck Resources (now PolyMet Mining Corporation) leased the NorthMet Project from USX. In 1997-98, PolyMet's management came to believe that technological advances made during the last 6-8 years in the general areas of hydrometallurgy offered the opportunity to solve previous metallurgical recovery problems associated with these types of deposits and thus make its NorthMet deposit economic. The Company placed into action a plan to test these beliefs and to provide a solution to previous metallurgical recovery problems. Constraints placed on the

testing program by PolyMet’s management resulted from its desires not to attempt to permit a smelter in northern Minnesota, not to sell concentrate to a toll smelter, and to produce metals on site to the maximum extent possible.

METALLURGICAL TESTING PROGRAM

The metallurgical testing program planned provided for production of a bulk sulfide concentrate, oxidation and leaching of sulfide minerals via pressure oxidation and/or bio-leaching, and on-site metal recovery to the extent possible

Concentrate Production

The first step in the testing program called for the production of concentrate from mineralized materials believed to represent run-of-mine material. It was decided to focus on producing a bulk sulfide concentrate at a rather coarse grind that resulted in maximum sulfide recovery for subsequent hydrometallurgical test work. Two additional objectives were:

- to reduce the pyrrhotite content of the concentrate, and
- to demonstrate the possibility of copper/nickel separation by flotation.

Lakefield Research Limited, located near Peterborough, Ontario was selected to perform the concentrate testing program based on their previous experience with mineralized material from both the NorthMet and adjacent Minnamax deposits. Lakefield crushed, ground and blended mineralized reverse circulation drill cuttings to produce a sample that weighed 23.8 tonnes and had the grades shown in Table 1. This material was treated in a pilot plant run over a period of 42 hours. The resulting bulk concentrate grades are provided in Table 1, along with metal recoveries. The concentration ratio was approximately 40:1, and the weight of the concentrate was approximately 2.5% of the ore.

Table 1

| Production of Bulk Concentrate | | | |
|---------------------------------------|------------------------------|---------------------------------------|-------------------------------------|
| | Feed (Head) Grade | Bulk Concentrate Grade | Metal Recoveries (%) |
| Copper | 0.43% | 15.5% | 94.6 |
| Nickel | 0.12% | 3.69% | 77.2 |
| Cobalt | 0.009% | 0.149% | 46.4 |
| Platinum | 0.08 g/t | 2.49 g/t | 76.4 |
| Palladium | 0.37 g/t | 11.1 g/t | 75.8 |
| Gold | 0.06 g/t | 2.80 g/t | 76.6 |
| Silver | 1.48 g/t | 41.5 g/t | 69.9 |

As a general rule, lower grade ores often demonstrate better economics from a coarse grind, as the incremental grinding costs to achieve finer grinds are not offset by the improved metallurgical recoveries. Optimization testing demonstrated the economic optimum grind to be between 200 and 250µm, a range of grinding typical of porphyry copper ores. The optimum economic grind target for NorthMet mineralization was chosen to be a K80 of 200µm.

Relative to program objectives, pyrrhotite recovery to concentrate was reduced through development of a new flowsheet and modifications to reagents. The new flowsheet has closed circuit cleaning with shorter flotation time and reduced collector addition than flowsheets developed previously.

Unlike previously unsuccessful attempts, this testing program achieved copper-nickel separation with a simple high lime and Na₂SO₃ reagent scheme. Good quality copper concentrate was produced (25% Cu, 0.5% Ni @ 70% copper recovery).

The bulk sulfide concentrate produced in this phase of the testing program became the feed for subsequent hydrometallurgical testing using pressure oxidation (POX) and bio-oxidation processes. PolyMet’s management believed that both processes, properly applied, could unlock the metallurgical challenges facing these types of deposits.

CONCENTRATE LEACHING AND METALS RECOVERY

Polymet, in 1998, commissioned limited test work to determine the amenability of pressure oxidation and bioleaching to treatment of NorthMet bulk concentrate. Process Research Associates (PRA) of Vancouver conducted the pressure oxidation (POX) tests while RPC of New Brunswick carried out bioleaching tests.

Extraction of copper and nickel exceeded 95% for both methods, while subsequent cyanidation of the leach residues yielded rather mediocre extractions for gold, platinum and palladium at about 90, 20 and 50% respectively.

A screening study was undertaken to guide the metallurgical test program based on the conceptual flowsheet shown in Figure 1. The premise was to utilize either bioleaching or POX followed by recovery of the copper and nickel using proven solvent extraction/electrowinning techniques plus cyanidation of the leach residue to recover gold and PGMs. This study showed there was, in fact, little difference in total project capital cost regardless of the leaching process used, primarily because the leach step represents less than 20% of the total, as shown in the high temperature POX example in Table 2 below. The balance of the Capex is common to all leach variations.

Table 2
**Order of Magnitude Capex – 50,000 t/d Throughput
High Temperature POX**

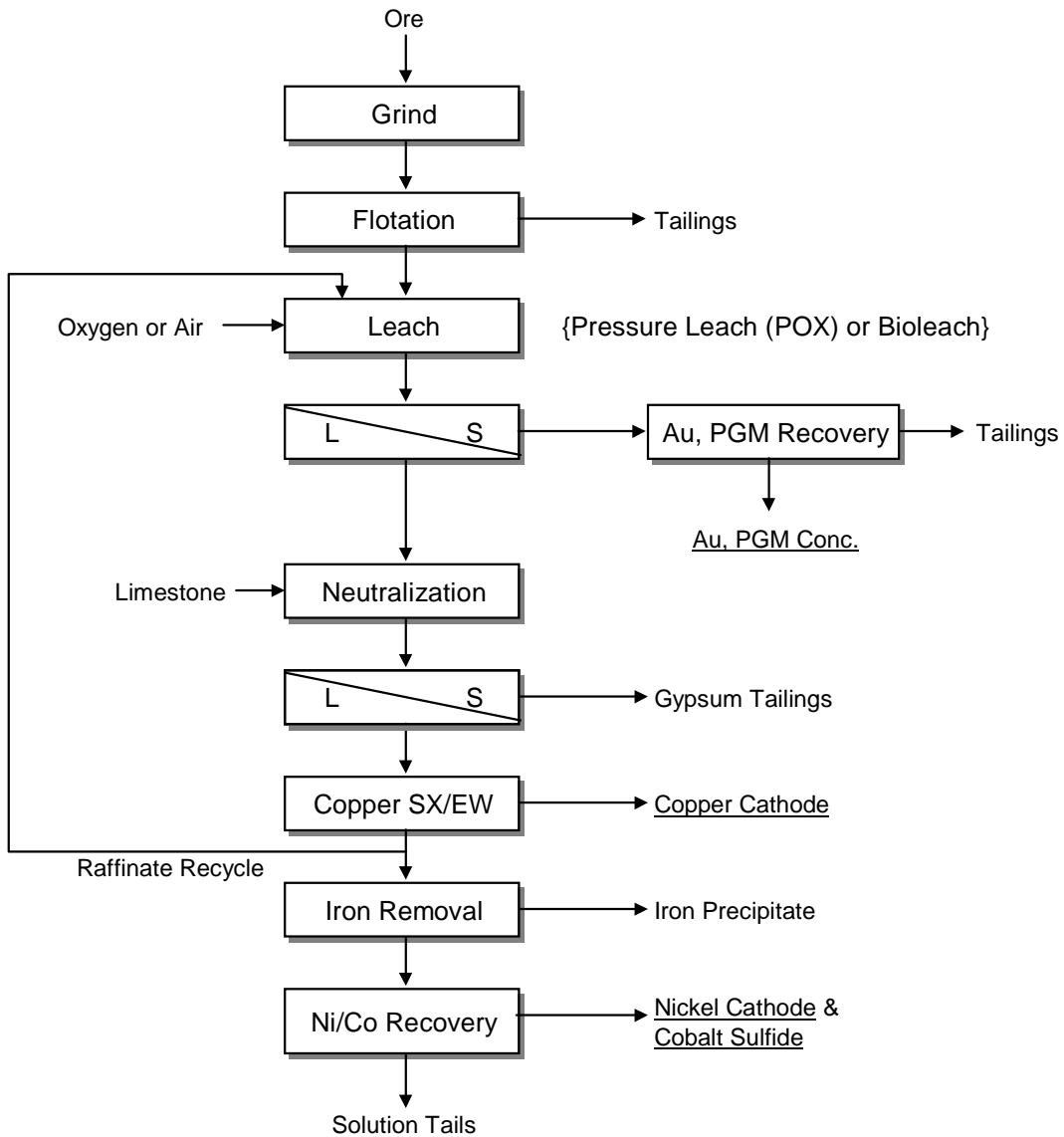
| Area | US\$ Million |
|--------------------------------------|---------------------|
| Mine | \$ 38 |
| Concentrator | 84 |
| Leaching & L/S Separation | 54 |
| Metals Recovery | 50 |
| Tailings & Infrastructure | 68 |
| Total Direct Cost | 300 |
| Indirect Costs, Contingency | 200 |
| Total Project Capital Cost | \$ 500 |

The cash operating cost will vary with the degree of sulfur oxidation, but again, leaching cost is less than 20% of the total per Table 3.

Table 3
**Cash Operating Cost
High Temperature POX**

| Area | US\$/tonne Ore |
|----------------------------|-----------------------|
| G& A | \$0.50 |
| Mining | 1.70 |
| Concentrator | 2.90 |
| POX, Neutralisation | 1.50 |
| Metals Recovery | 1.50 |
| Tailings Disposal | 0.10 |
| Total | \$8.20 |

Figure 1
NorthMet Conceptual Flowsheet



It follows that metal recoveries will be the predominant variable in determining which process should be selected for the NorthMet Project. The importance of metal recoveries is demonstrated in Table 4, which shows what the value would be of the various metals the flotation recoveries to concentrate shown in Table 1 and assuming 95% hydrometallurgical process recovery for both base and precious metals.

Table 4
Value of Recovered Metals

| Metal | Price | Value – US\$/t Ore |
|--------------------------------|--------------|---------------------------|
| Copper | \$0.80/lb | \$ 6.90 |
| Nickel | \$3.00/lb | 5.80 |
| Cobalt | \$10.00/lb | 0.50 |
| Sub-total Base Metals | | 13.20 |
| Gold | \$300/oz. | 0.40 |
| Platinum | \$350/oz. | 0.65 |
| Palladium | \$350/oz. | 3.00 |
| Other PGM | Allowance | 0.50 |
| Sub-total Au & PGMs | | 4.55 |
| Total | | \$17.55 |

Based on the preliminary 1998 tests, gold and PGM recovered value was lower than the above by more than \$2.00/t ore at 90% Au, 20% Pt and 50% Pd extraction. Hence the importance of improving precious metals recoveries.

The metallurgical test program was thus set up to test base and precious metal extractions using several variations of pressure oxidation and bioleaching. Also, recovery of the various base and precious metals was to be demonstrated. Lakefield Research conducted the majority of the POX tests. Dynatec Corporation, Fort Saskatchewan, Alberta, tested their proprietary pressure leach process. The bioleaching tests were performed by the joint partnership of BacTech (Australia) Pty Ltd, Perth and Mintek (Johannesburg). All test work was conducted during the period January to May 1999.

The following sections describe the results.

PRESSURE OXIDATION AT LAKEFIELD RESEARCH

Initial testing focussed on standard pressure leaching to dissolve the Cu, Ni and Co, followed by HCl/Cl₂ leaching of the residue to recover the PGMs.¹⁾ Pressure leaching was investigated under both mild and aggressive leaching conditions. The base metals dissolved efficiently (>90%) under mild pressure leaching conditions (100°C, 100 psi O₂ pressure). However, the kinetics were very slow (t = 6 hrs) when treating the float concentrate directly, without fine re-grinding, and the leach slurry was also very slow to settle and filter. Only about 60% of the sulfides were oxidized to soluble sulfate under these conditions, but more than 40% of the iron reported to the leach liquor as ferrous sulfate.

Conducting the pressure leaching under more severe conditions (220°C, 100 psi O₂) dissolved >95% of the Cu and Ni in 1 hour and left most of the iron (~90%) in the residue. The sulfides were almost completely oxidized to sulfates under these conditions. It was interesting to observe that ~60% of the palladium also dissolved under these conditions, but no gold or platinum. Results from leaches carried out under these two sets of conditions are summarized in Table 5.

Table 5 – Low & High Temperature POX Results

| Element | Low Temperature POX | | | High Temperature POX | |
|------------------|---------------------|---------|------------|----------------------|------------|
| | Feed | Residue | Ext'n % | Residue | Ext'n % |
| Cu % | 13.8 | 2.38 | 90.0 | 0.16 | 99.3 |
| Ni % | 3.5 | 0.14 | 97.7 | 0.23 | 95.9 |
| Co % | 0.15 | <0.02 | >92 | <0.02 | >92.0 |
| Fe % | 28.7 | 28.6 | 42 | 40.8 | 11.5 |
| S _T % | 25.6 | 18.9 | 57.1 | 4.4 | 91.5 |
| Au g/t | 2.24 | 3.21 | ~0 | 3.14 | ~0 |
| Pt g/t | 1.75 | 2.45 | ~0 | 2.15 | ~0 |
| Pd g/t | 8.91 | 13.1 | ~0 | 5.36 | 61.1 |

The main drawback to this approach, however, was the poor response of the pressure leach residue to PGM recovery by the conventional HCl/Cl₂ leach process. Various experimental conditions were tested, including concentrated HCl and/or NaCl; NaOCl or Cl₂ additions to potentials of >1000 mV (Ag/AgCl); fine grinding to a K₈₀ of ~10 microns, and temperatures of up to 80°C. Although Au and Pd recoveries of >90% were achievable under aggressive conditions, platinum recovery was always <50%, and usually ~30%. Not only were the platinum recoveries deemed unacceptable, but also the aggressive leaching conditions would have led to unacceptably high reagent consumption and costs.

High pressure, (ambient temperature) intensive cyanidation of the pressure leach residue was also briefly studied. The best results achieved were 99% gold and 90% palladium dissolution, but only 35% platinum. Therefore, this approach was no more successful than the HCl/Cl₂ leach, although it probably would have proven to be a more economical process.

Application of the PlatSol[®] Process. ²⁾

This process was developed at Lakefield Research for the NorthMet concentrate, and is owned by a private company, International PGM Technologies Ltd. The fundamental difference between the PlatSol[®] Process and the conventional high temperature pressure oxidation process is the addition of chloride ions to the autoclave feed. It was discovered in this project that the addition of chloride at concentrations as low as 3 g/l resulted in the dissolution of most of the PGMs in the autoclave. By conducting the chloride leach in the autoclave, not only did Pt extraction improve from ~ 20% to ~98% under optimum conditions, but significant capital and operating cost savings can be achieved by leaching the PGMs in the same vessel and within the same reaction time as the Cu and Ni, under essentially the same conditions needed for efficient base metal extraction.

The oxidation of gold, palladium, platinum and other PGMs is favored by the presence of small amounts of chloride in solution. The chloride stabilizes the various platinum group elements as dissolved chloro complexes.

A number of tests were carried out aimed at optimizing parameters, such as the particle size of the feed, the autoclave temperature and the chloride concentration. The test conditions are summarized in Table 6 and the metallurgical results in Table 7.

Table 6 - Pressure Leaching Test Conditions

| Test No. | Concentrate Reground | Feed K ₈₀ μm | Autoclave Temp °C | O ₂ Pressure psi | Time h | Chloride g/L |
|----------|----------------------|-------------------------|-------------------|-----------------------------|--------|--------------|
| 1 | No | 32 | 220 | 100 | 2 | 0 |
| 2 | No | 32 | 220 | 100 | 2 | 6 |
| 3 | Yes | 15-20 | 220 | 100 | 2 | 3 |
| 4 | Yes | 15-20 | 200 | 100 | 2 | 6 |
| 5 | Yes | 15-20 | 220 | 100 | 2 | 6 |
| 6 | Yes | 15-20 | 220 | 100 | 2 | 6 |

Table 7 - Pressure Leaching Metallurgical Results

| Test No. | Cu | | Ni | | Au | | Pt | | Pd | |
|----------|---------|-------|---------|-------|-----------|-------|-----------|-------|-----------|-------|
| | Assay % | Ext % | Assay % | Ext % | Assay g/t | Ext % | Assay g/t | Ext % | Assay g/t | Ext % |
| Feed | 13.8 | | 3.52 | | 2.24 | | 1.75 | | 8.91 | |
| 1 | 0.16 | 99.3 | 0.23 | 97.7 | 3.32 | ~0 | 2.15 | ~0 | 5.36 | 61 |
| 2 | 0.05 | 99.7 | 0.31 | 93.4 | 0.27 | 91 | 0.49 | 79 | 1.37 | 88 |
| 3 | 0.12 | 99.4 | 0.27 | 94.3 | 0.64 | 79 | 0.16 | 93 | 1.01 | 92 |
| 4 | 0.28 | 98.3 | 0.38 | 90.8 | 2.71 | ~0 | 1.97 | 4 | 10.9 | ~0 |
| 5 | 0.11 | 99.4 | 0.31 | 93.3 | 0.13 | 96 | 0.06 | 98 | 0.72 | 94 |
| 6 | 0.10 | 99.4 | 0.26 | 94.3 | 0.13 | 96 | 0.06 | 98 | 0.64 | 95 |

Cu and Ni extraction was high in all the tests, but PGM and gold extraction was found to be a sensitive function of the operating conditions. For example;

- Reducing the autoclave temperature from 220°C to 200°C decreased PGM extraction from >90% to essentially zero. (Tests 3 and 4)
- Increasing the chloride concentration from 0 (Test 1) to 3 g/l (Test 3) increased gold extraction from ~0 to 79%, platinum from ~0 to 93% and palladium from 61 to 92%. Further chloride addition to 5 g/l (Tests 5 &6) did not have a dramatic effect, but improved Pd extraction to 95%.
- Regrinding the float concentrate from a K₈₀ of 32 microns (Test 2) to 15-20 microns (Test 3) improved Pt extraction from 79 to 93% and Pd from 88 to 92%, but actually resulted in a decrease in gold extraction from 91 to 79%. This was thought to be due to the cementation of gold chloride onto residual flakes of iron powder (introduced during fine grinding in a stirred mill with steel grinding media). This was corrected by regrinding with ceramic media.

The effect of acid concentration in the autoclave was not assessed in this testwork. Under the conditions tested, the feed to the autoclave contained ~25 g/L free H₂SO₄ (due to recycling a portion of the autoclave liquor after copper solvent extraction, to build up Ni concentration and to provide autoclave coolant) and the discharge contained 55-65 g/L H₂SO₄.

Gold and PGM Recovery from the Autoclave Product

Two approaches have been tested; precipitation with sulfide ions and adsorption on activated carbon. In both cases the autoclave liquor was treated directly without neutralization of the 55-65 g/L H₂SO₄ in solution. This will be an important requirement of any precious metal recovery process, as it was shown in the testwork that gold and the PGMs partially precipitated during neutralization. The sulfide precipitation flowsheet will obviously require solid/liquid separation prior to PGM extraction, whereas the activated carbon flowsheet could be applied to solutions or pulps.

Precipitation with sodium hydrogen sulfide (NaHS) was very effective, particularly when the pregnant leach liquor was first treated with SO₂ or metabisulfite to reduce the Fe³⁺ to Fe²⁺ and lower the emf to ~400 mV (Ag/AgCl). Results from a precipitation test are presented in Table 8.

Table 8 - NaHS Precipitation of Gold and the PGMs from Pre-Reduced Pregnant Solution

| | | Cu | Ni | Fe | Au | Pt | Pd |
|--------------------------|------|-----------|-----------|-----------|-----------|-----------|-----------|
| Preg solution | mg/L | 17000 | 19900 | 1550 | 0.32 | 0.34 | 1.23 |
| Barren solution | mg/L | 14300 | 18200 | 1340 | 0.01 | 0.00 | 0.01 |
| Precipitate | | 61.8% | 0.19% | 0.37% | 92 g/t | 102 g/t | 484 g/t |
| Precipitation Efficiency | % | 16 | <0.1 | 1 | 97 | ~100 | 99 |

No attempt was made to optimize the amount of NaHS added in this phase of testwork, and the concentration of copper in the final precipitate could probably be reduced quite significantly without compromising PGM precipitation efficiency. The PGM precipitate would be pretreated in a small pressure leach reactor to dissolve the copper and produce a very high grade PGM residue (>50% PGMs) for sale to a PGM refinery.

Gold and PGM recovery using activated carbon was also tested briefly, with promising results. It was shown that gold and PGMs could be loaded onto activated carbon.

The greatest potential advantage of the activated carbon variant will be the option to recover gold and PGMs directly from the acidic autoclave pulp.

A more detailed evaluation of the NaHS precipitation and carbon adsorption processes for gold and PGM extraction will be undertaken in the next phase of the feasibility study on this project.

Overall Flowsheet

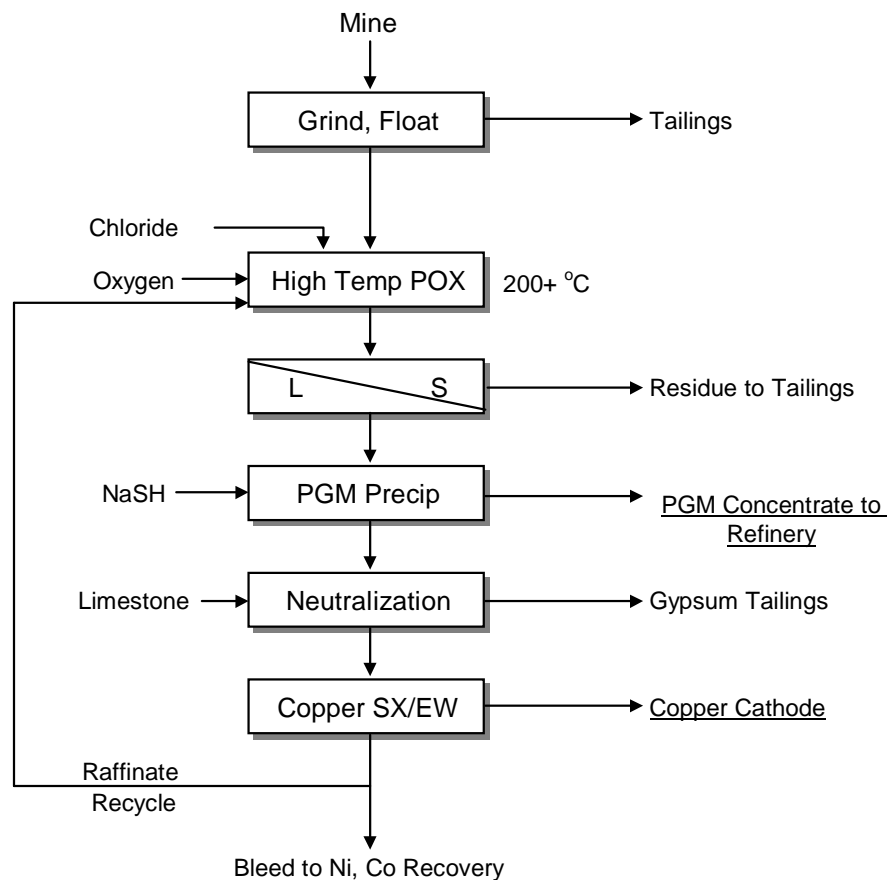
A limited amount of testwork has been done on base metal recovery. Copper recovery by conventional solvent extraction/electrowinning was very efficient. In two stages of countercurrent extraction with LIX984, it was possible to reduce copper in solution from 9.1 g/l to less than 0.4 g/l, producing an organic phase containing 16 g/l Cu. The presence of ~6 g/l Cl in the pregnant solution had no adverse effect on copper extraction or selectivity, with the loaded organic containing copper at a purity of >99.98%.

In addition, various schemes were tested for the treatment of the copper SX raffinate solution to recover Ni (~17 g/l) and cobalt (~0.6g/l). The testwork demonstrated the technical viability of all the flowsheets considered, recovering high purity nickel and cobalt products at high yield.

A simplified conceptual flowsheet incorporating sulfide precipitation is presented in Figure 2.

Figure 2

Simplified Block Flow Diagram for Gold, PGM and Base Metal Recoveries from NorthMet Concentrates PlatSol Process



PRESSURE OXIDATION AT DYNATEC

Dynatec has developed a medium temperature pressure oxidation process³⁾ for leaching chalcopyrite containing concentrates at ~150 °C which involves the use of coal to catalyze the reaction. Dynatec conducted several tests on the NorthMet concentrate. Base metal extractions were good, basically matching the high temperature POX results. The results for cyanide leaching of the leach residue, however, were well short of recoveries achieved by the PlatSol[®] process.

BIOLEACHING PROCESS OPTIONS

Two process alternatives, relying exclusively on bioleaching as a method for extracting base metals and concentrating precious metals in the solid residue, were evaluated. This segment of the paper describes these bioleaching process approaches and summarizes the results and conclusions of the amenability testwork performed by the joint partnership of BacTech (Australia) Pty. Ltd. (Perth) and Mintek (Johannesburg). These organizations have commercialized the use of bioleaching in the gold industry and have development programs for commercializing bioleach technology for base metals.

Description of the Bioleaching Process Options and Testwork Objectives

Two bioleaching process routes were evaluated:

- One route, termed a "full bioleach", maximized the extraction of copper, nickel and cobalt values in a bioleach operation with base metal recovery from the leach solution achieved using conventional hydrometallurgical practices of solvent extraction and precipitation. The PGMs and precious metals were extracted by cyanidation from the bioleach solid residue.
- The second process route, called the "selective partial bioleach", maximized the leaching of nickel and cobalt in a bioleach operation, while minimizing the leaching of copper (the majority of which resides in chalcopyrite). The bioleach residue, upgraded in copper would contain all the gold and PGM values, which could be recovered by smelting.

The overall objectives of the testwork were to determine the amenability of the concentrate to bacterial leaching and to translate the results into preliminary process designs. The NorthMet concentrate was characterized by size analysis, mineralogy and chemical analyses. Acid consumption was determined and the concentrate was subjected to ferric leaching to give support information for the planned bioleach tests.

Bioleaching batch tests were performed using mixed cultures of mesophilic *Thiobacillus* and *Leptospirillum* species, moderately thermophilic leaching bacteria, and the extremely thermophilic *Archaea*. Iron rejection and solvent extraction studies were conducted to demonstrate selective iron removal and downstream metal recovery. In this initial testwork a single non-optimized test was conducted to investigate PGM and precious metal recoveries from bioleach residues using cyanidation. A scoping study to derive capital and operating costs was completed on the most promising bioleach route.

Results and Conclusions of Bioleaching Testwork

The particle size of the as-received flotation concentrate was $P_{90}<70\ \mu\text{m}$. The principal sulfide mineral was chalcopyrite (30%), and the concentrate assayed 12.8% copper, 3.58% nickel, 0.14% cobalt and had a gangue content of 38%. Most bioleaching testwork was completed on the concentrate milled to two different sizes: $P_{90}<22\ \mu\text{m}$ and $P_{90}<14\ \mu\text{m}$. Acid consumption of the concentrate was 236 kg/t.

Ferric iron leaching, conducted using several temperature, redox and grind size conditions in the absence of bacteria, gave a maximum release of only 92% copper, 89% nickel and 73% cobalt. The value of the overall results from the matrix of ferric leach tests performed was the suggestion that selective nickel and cobalt extraction over copper might be achievable by operating at a lower temperature and coarser particle size. However, the results also suggested that overall nickel and cobalt recoveries might be limited if a "selective partial bioleach" route was adopted.

A total of nine bioleach tests were conducted in batch reactors using either mesophilic bacteria (35°C), moderately thermophilic bacteria (50°C) or extremely thermophilic *Archaea* (70°C). Test conditions were designed to either promote chalcopyrite leaching for a "full bioleach" or to inhibit copper leaching for a "selective partial bioleach". Table 10 summarizes the metal extractions and levels of oxidation achieved based on residue analysis for the "selective partial bioleach" tests. The Table 9 results illustrate that, with the concentrate tested, it was not possible

through "selective partial bioleaching" to achieve an adequate copper grade in the bioleach residue that would satisfy the criteria for downstream smelting. The extent of nickel contamination in the bioleach residue, as revealed by the nickel/copper ratio (Table 9), may also have resulted in a penalty by the smelter.

Table 9
Summary of extraction results from "selective partial bioleach" tests to minimize chalcopyrite leaching

| Test No. | Culture | Feed size (μm) | %Cu extract | %Ni extract | %Co extract | %S oxidized | Residue analysis for Ni/Cu ratio | | | |
|----------|-----------|-----------------------------|-------------|-------------|-------------|-------------|----------------------------------|-------|------------------|--------|
| | | | | | | | %Ni | %Cu | %S ²⁻ | %Ni/Cu |
| 1 | Meso-35°C | $P_{90}<70$ | 39 | 92 | 65 | 63 | | | | |
| 2 | Meso-35°C | $P_{90}<70$ | 37 | 91 | 72 | 63 | | | | |
| 3 | Meso-35°C | $P_{90}<22$ | 24 | 34 | -- | 35 | | | | |
| 4 | MTh-50°C | $P_{90}<70$ | 40 | 91 | 88 | 66 | 0.45 | 10.80 | 12.5 | 4.1 |
| 5 | MTh-50°C | $P_{90}<22$ | 71 | 98 | 97 | 82 | 0.11 | 7.07 | 9 | 1.6 |
| 6 | MTh-50°C | $P_{90}<14$ | 86 | 97 | 95 | 88 | 0.19 | 3.76 | 6.6 | 5.1 |

Table 10 summarizes the results from the three "full bioleach" tests conducted using the three different cultures, each at its optimum temperature. These tests were conducted with concentrate ground to $P_{90}<14\ \mu\text{m}$. Excellent results were obtained for base metal extraction from the "full bioleach" tests. Moderately thermophilic bacteria (Test 8) and extremely thermophilic bacteria (Test 9) were found to be superior to the mesophilic bacteria (Test 7). A maximum release of 99% copper, 97% nickel and 94% cobalt was achieved using material ground to $P_{90}<14\ \mu\text{m}$ (Test 8). A feature of the testwork, shown in Table 9, was to maximize in-situ acid production during

bioleaching, because of the high (236 kg/t) acid consumption of the concentrate, which contained pyrrhotite and other acid labile species.

Table 10 - Summary of extraction results for "full bioleach" tests

| Test No. | Culture | Feed size (µm) | %Cu extract | %Ni extract | %Co extract | %S oxidized |
|----------|-----------|---------------------|----------------|----------------|----------------|----------------|
| 7 | Meso-35°C | P ₉₀ <14 | 82 | 83 | 75 | 79 |
| 8 | MTh-50°C | P ₉₀ <14 | 99 | 97 | 94 | 94 |
| 9 | ExTh-70°C | P ₉₀ <14 | 95 | 83 | 83 | 94 |

Preliminary downstream testing for iron rejection and base metal recovery from the bioleach liquor (Test 8, Table 10) gave good results. Adjusting the pH to 2.7 with 350 kg/t limestone achieved 83% iron rejection. At this pH, a 1.8% copper loss was sustained, but with additional testing for confirmation, adjusting the pH somewhat lower would very likely reduce the copper loss and still result in good iron rejection. Copper was removed from the limestone-treated solution using LIX 622 at ambient temperature, leaving a relatively pure cobalt/nickel stream for precipitation to produce product metals.

A cyanide leach test was performed on the residue from Test No. 8 ("full bioleach" with moderate thermophiles) to determine precious metal and PGM recoveries from the bioleach residue. Table 11 summarizes these results. Although excellent base metal recoveries were achieved using the "full bioleach" approach and good extraction of gold and palladium were obtained when the bioleach residue was leached with cyanide, disappointing platinum recoveries spawned a third approach - a hybrid bioleach and pressure oxidation route - for processing the NorthMet concentrate. This process option would entail bioleaching the readily oxidizable sulfide minerals (approximately 50% of the total sulfides) and the bioleach residue, after thickening, would be fed to a pressure autoclave to complete sulphide oxidation and extract PGMs and precious metals.

TABLE 11. Results from cyanidation of "full bioleach" residue

| Head value (ppm) | | | | Residue values (ppm) | | | | Extraction (%) | | | |
|------------------|------|------|------|----------------------|----|------|------|----------------|----|-----|----|
| Au | Ag | Pt | Pd | Au | Ag | Pt | Pd | Au | Ag | Pt | Pd |
| 2.78 | 54.6 | 3.12 | 10.2 | 0.29 | 37 | 3.11 | 2.23 | 91 | 40 | 0.7 | 80 |

ECONOMICS OF THE PLATSOL[®] PROCESS

The high gold and PGM extractions achieved with the PlatSol[®] Process clearly indicated that this approach will be the most economic on a stand-alone basis. An alternative "Hybrid" approach combining the best features of bioleaching and PlatSol[®] is also examined in the next section of this paper.

The PlatSol[®] Process preliminary economics are shown in Table 12. They are based on the bench scale test results and on the following assumptions:

| | |
|-----------------------|---|
| Resource | 800 million tonnes |
| Mine Production | 50,000 t/day |
| Stripping ratio | 1:1 years 1 to 6, 2:1 years 7 to 14, 3:1 thereafter |
| Power cost | \$0.034/kwh |
| Limestone cost | \$13.00/tonne |
| Depreciation | 10% straight line |
| Depletion | 50% |
| Federal & State Taxes | 44% |

Table 12 – PlatSol[®] Process Economics

| | Cu | Ni | Co | Au | Pt | Pd |
|-------------------------------|-----------|-----------|-----------|---------------|-----------|-----------|
| Resource Grade - % or g/t | 0.43 | 0.12 | 0.01 | 0.088 | 0.143 | 0.389 |
| Flotation Recovery | 94.6 | 77.2 | 46.4 | 76.6 | 76.4 | 75.8 |
| Process Recovery | 97.5 | 94.5 | 92.5 | 94 | 94 | 94 |
| Overall Recovery | 92.2 | 73.0 | 42.9 | 72.0 | 71.8 | 71.3 |
| Production –t/yr | 72,000 | 16,000 | 750 | | | |
| oz/yr | | | | 37,000 | 60,000 | 162,000 |
| Metal Prices \$/lb or oz. | 0.80 | 3.00 | 10.00 | 300 | 350 | 350 |
| Capex – m\$ | | | | \$492 | | |
| Opex - \$/t ore | | | | \$8.20 | | |
| NPV @ 10% Discount – | | | | | | |
| m\$ | | | | \$780 | | |
| DCF Rate of Return - % | | | | 26% | | |

HYBRID PROCESS OPTION: BIOLEACH AND PRESSURE OXIDATION

Based on the results of the "selective partial bioleach" (Table 10) and "full bioleach" (Table 11) batch tests, a conceptual process, which unites bioleaching and pressure oxidation into a hybrid process, was devised. This is shown in Figure 3. The hybrid process is based on a partial bio-oxidation, leaching the relatively amenable sulphide minerals and yielding only partial extraction of the base metals. The readily leachable sulphides account for about 50% of the total sulphides in the NorthMet concentrate.

The design specification for the bioleach part of the hybrid process was based on a three-day residence time at 22.5% pulp density through a two-stage continuous stirred-tank reactor system treating 1,200 t/d of concentrate to oxidize 50% of the sulfur load. A feed size of P₈₀<15 µm was based on the requirement for the downstream pressure oxidation part of the hybrid system. The bioleach residue would be thickened prior to further oxidation in the autoclave. The leach liquor from the bioleach circuit would be combined with the final leach liquor from the autoclave for

base metal recovery. A cost study for the bioleach section of the hybrid circuit was prepared by Signet Technology, Inc. Detailed design criteria and an equipment list were developed to provide support data for deriving capital and operating cost estimates.

Economic Comparison of the Hybrid Approach with Platsol® Pressure Oxidation.

Capital and operating costs were estimated for a modified high temperature pressure oxidation circuit involving pre-oxidation of bulk concentrate by bio-oxidation prior to high temperature pressure oxidation.

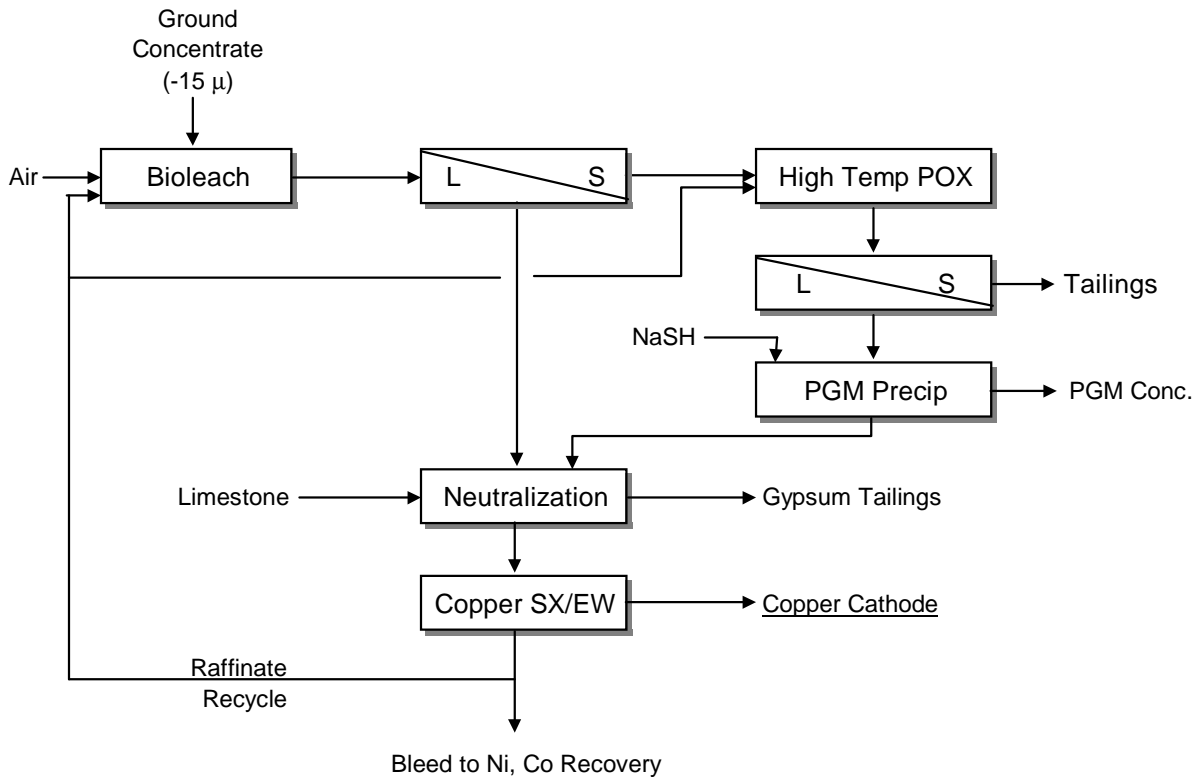
The summary of the estimates is as follows in Table 13, figures in \$US:

Table 13 – Capex & Opex Comparison

| Leach Process | PlatSol® Only | Bioleach + PlatSol® |
|----------------------|------------------|------------------------|
| Capex | \$492.4 m | \$489.0 m |
| Opex, per tonne feed | \$8.20 | \$8.10 |

Figure 3

**Simplified Block Flow Diagram
'Hybrid' Flowsheet**



CONCLUSIONS

- The PlatSol[®] Process, a combination of high temperature pressure oxidation augmented with a complexing reagent such as chloride ion, has proven to be effective in achieving high extractions of both base and precious metals simultaneously in the autoclave. Selective precipitation of gold & PGMs and separate recovery of copper, nickel and cobalt has been demonstrated.

Based on preliminary figures, The NorthMet Project appears to be very attractive economically using the PlatSol[®] Process. The economics are sufficiently attractive to justify carrying on with the next stage of development of the project.

- The Platsol process was selected over the stand-alone bioleach process option, because higher metal values were achieved as a result of higher PGM recovery.
- A bioleach-POX hybrid process combining a first stage of bioleaching followed by a second stage of pressure leaching also appears attractive, although not yet specifically tested. BacTech will soon pilot a new reactor design which, if successful, could enhance the attractiveness of the hybrid approach.
- Although the bioleach-POX hybrid process option is certainly applicable to large, low-grade, disseminated polymetallic deposits characteristic of the Duluth Complex, the hybrid option was not selected for the initial development of NorthMet, because of the probable time, cost and complexity of permitting two processes. However, a future expansion of NorthMet or commercial production at other Duluth Complex deposits may economically benefit by utilizing the hybrid process format.
- Thus, the recent developments in leaching, based fundamentally on the success of autoclaves in treating refractory gold ores and concentrates and the subsequent recovery of the base and precious metals using commercially proven unit operations and processes shows great potential in making the NorthMet project both technically and economically viable.

FUTURE PROGRAM

Polymet recently completed a major drilling program to upgrade the quality of the resource and to collect a representative sample for a continuous integrated pilot plant campaign.

It is expected that the flotation pilot plant to produce the bulk concentrate for the hydrometallurgical pilot plant campaign will be run during May, 2000. The hydromet pilot plant will follow, based on the stand alone PlatSol[®] Process.

The estimates for Pre-feasibility study will be completed following the pilot plant campaigns. The pre-feasibility study should be completed in the 3rd quarter, 2000.

A bankable feasibility study will ultimately be required assuming the pre-feasibility study continues to indicate good economics.

ACKNOWLEDGEMENTS

The authors are grateful to PolyMet Mining Corp. for permission to publish details of the work carried out on their NorthMet Project.

The authors also acknowledge, with thanks, the contributions made by the staffs of BacTech, Dynatec, Lakefield Research and Mintek to the success of the metallurgical development program. The contributions of Dr. C.A. Fleming and Dr. C.J. Ferron of Lakefield Research in particular were very significant and are much appreciated.

REFERENCES

1. Fleming, C.A., Ferron, C.J., Dreisinger, D.B., and O’Kane, P.T. “A Process for the Simultaneous Leaching and Recovery of Gold, Platinum Group Metals and Base Metals From Ores and Concentrates” EPD Congress 2000, the Minerals, Metals and Materials Society (TMS), 2000 TMS Annual Meeting, March 12-16, 2000, Editor: P.R. Taylor, pp. 419-431.
2. Dreisinger, D.B., Fleming, C.A., and O’Kane, P.T. “Oxidative Pressure Leach Recovery Using Halide Ions”. U.S. Patent Application, April, 28, 1999.
3. Collins, M.J., Kofluk, D.K. “Hydrometallurgical Process for the Extraction of Copper From Sulphide Concentrates”, U.S. Patent # 5,730,776, 24 March, 1998.

Whilst the eyes of most in the nickel industry have been focused on the progress being made by the various Australian laterite projects using high pressure acid leaching, it should be remembered that important technical innovations are under way in the processing of nickel sulphides as well. The laterite projects may be revolutionising the industry as a whole, but the new developments in the processing of sulphides could have an equally startling impact. Most of these new processes revolve around the oxidation of nickel sulphides using methods other than pyrometallurgy.

Of the various processes under development perhaps the most advanced is Billiton's BioNIC™ process. BioNIC™ has its origins in the BIOX™ process developed by Gencor for the treatment of refractory gold ores in South Africa during the 1980s (*MJ*, January 12, 1990, p.29) when it commissioned its first commercial-scale processing plant at the Fairview gold mine in 1986. Since then, BIOX™ has been used at the Wiluna mine in Western Australia, at the Sao Bento mine in Brazil and, most notably, at Ashanti's Obuasi mine in Ghana.

Billiton's endeavours

BIOX™ has spawned three other processes, BioNIC™, and BioCOP™ (*MJ*, March 10, p.189) and BioZINC™, all now under development by Billiton (formed in 1997 from Gencor's coal and base metal assets). BioNIC™ is the most advanced and, according to QNI, Billiton's nickel subsidiary, it will go into commercial use once a large enough nickel deposit is found.

At the core of the BioNIC™ process are bacteria. At present, QNI is using mesophiles – bacteria that live quite happily at relatively normal temperatures. However, the company is investigating the use of thermophile strains of bacteria whose natural environments are volcanic hot springs. These high-temperature strains can improve metal recoveries, and QNI believes that by using them it can boost nickel recoveries to an average of 97% against 92-95% achieved using mesophiles.

The bacteria are kept in a reactor cascade under acid conditions (pH 1.0 to 2.0) at a controlled temperature (30-45°C), and are provided with a steady supply of oxygen, carbon dioxide and nutrients. A concentrate slurry is passed through the reactor cascade where the bacteria digest the sulphides and produce nickel sulphate in solution.

After bio-leaching, the pregnant solution in the BioNIC™ process goes through an iron precipitation stage in which ferric iron is removed using limestone. The solution is then passed through a 'polishing' plant where the iron concentration is reduced to less than 2 parts per million.

Once the iron has been removed the solution undergoes solvent extraction; first the

New solutions for nickel sulphides

nickel is removed and then cobalt, along with copper. The nickel liquor then passes to an electrowinning circuit to produce nickel cathode. The cobalt-copper solution requires further treatment to separate the metals. Alternatively, rather than undergoing solvent extraction and electrowinning (SX-EW), mixed sulphides can be produced and refined using modified Caron refining technology (using ammonia and steam to produce a nickel carbonate which is then reduced to produce a nickel oxide product).

According to QNI, BioNIC™ offers a number of important advantages for the processing of nickel sulphide when compared with the conventional pyrometallurgical route, in terms of both cost and environmental impact. QNI estimates that, for an operation processing 167,000 t/y of concentrate, the cost of a BioNIC™ plant producing mixed sulphides would be 25-50% that of a smelter processing the same amount of ore to produce nickel matte. BioNIC™'s operating costs are also said to be lower – some 85-95% of those of a smelter. As a result, QNI says that total operating costs (allowing for amortisation) are in the range of US\$0.70-1.00/lb lower than those of smelting. Because of the cost savings, it believes that relatively small nickel sulphide deposits could support a BioNIC™ processing plant.

The process offers obvious environmental benefits. Most notably, and unlike smelters, there are no sulphur dioxide and dust emissions to contend with. Furthermore, the inert leach residue produced by the process has been shown to conform to US Environmental Protection Agency standards for stability, and QNI expects that BioNIC™ effluents will also conform.

Titan tests heap leaching

Another important bioleaching development is taking place in Western Australia. Titan Resources is developing a heap-leach process for its Radio Hill mine for the treatment of relatively low-grade sulphide ores.

The initial work was carried out in Canada by the Research Productivity Council (RPC) of New Brunswick, which

isolated a strain of bacteria that has a particular affinity for chalcopyrite and pentlandite. This work is now being developed by Titan through a JV company, Pacific Ore Technology Pty Ltd, in which Titan has a 75% interest. The remainder is held by Dr Colin Hunter, a consulting metallurgist who was head of process research at Gencor and who was involved in the original work on the process with the RPC. Batch tests of milled ore have achieved nickel recoveries of over 90%, and column tests of crushed ore have produced nickel recoveries in excess of 80%.

However, for heap leaching, Titan is expecting recoveries of 70%, and to test this the company has constructed a pilot plant at its Radio Hill mine. The plant will test metal recoveries from a maximum of five 5,000 t heaps. After crushed ore has been placed on a pad it is irrigated with an acidic solution containing nutrients to provide the optimum conditions for bacterial reproduction. The bacteria adhere to the ore where they use enzymes to break down the sulphides (including chalcopyrite and pentlandite). In addition, iron-oxidising bacteria oxidise soluble ferrous iron to ferric iron.

This, in turn, attacks the sulphide minerals and oxidises them. The combination of direct and indirect attack dissolves the sulphides and puts copper, nickel, cobalt and other metals into solution. The resulting solution undergoes iron precipitation to

remove the bulk of the iron. At the pilot plant, Electro Metals electrowinning (EMEW) cells will facilitate copper recovery from solution under conditions where conventional SX-EW cells are ineffective. Nickel and cobalt are precipitated from the solution as hydroxides and sulphides.

Titan does not expect to complete test work on the first trial heaps at Radio Hill until early 2001. If the results are positive, it plans to establish a 50,000 t heap and eventually, by the middle of 2001, begin a feasibility study into the development of a 1 Mt/y operation. According to Titan's managing director, William Ryan, such a project would cost about A\$25 million and could produce nickel at less than US\$1.00/lb (including by-product credits).

Inco opts for PAL

Another area under investigation for the processing of nickel sulphides is the use of hydrometallurgy. Inco is investigating pressure leaching of sulphides, which it believes is applicable to the giant Voisey's Bay deposit. Unlike the pressure acid leach (PAL) projects being used in Australia to treat lateritic nickel, the Inco process does not require acid to be introduced to the pressure vessel (autoclave). Instead, a slur-

ry of sulphide concentrate is pumped into the reaction vessel which is pressurised by oxygen. The sulphides react exothermically with the oxygen, generating sulphuric acid and large amounts of heat, and the contained metals go into solution.

After solid-liquid separation, the solution is partially neutralised and, using the solvent extraction process similar to that developed for use at Inco's Goro lateritic nickel project, copper is then separated and recovered. After purification of the resulting solution, hydrochloric acid is used to strip the cobalt and nickel from the solution, and a new solvent extraction reagent, Cyanex 301, is used to recover the two metals. Another solvent extraction stage is used to separate nickel and cobalt. Electrowinning is then used to recover the nickel as cathode. The cobalt can also be recovered by electrowinning. Alternatively, the process can produce either a cobalt chloride or a cobalt carbonate intermediate product.

Perhaps one of the most novel ideas for treating nickel sulphides has come from Australia's Minproc. The company has been heavily involved in the development of the various PAL projects in Australia, and it is its involvement in these that has led it to suggest that the co-treatment of nickel sulphides and laterites may prove to be feasible. According to Minproc, there is an obvi-

ous synergy between the leaching of a sulphide concentrate (typically containing 10-20% Ni) and lower-grade laterite ores or beneficiated concentrates (0.7-2% Ni).

The key to the co-treatment route is that laterite leaching requires energy and acid, and that sulphide leaching generates acid and is highly exothermic. As a result, a co-treatment PAL operation offers a number of advantages over a laterite PAL – the use of sulphides reduces acid and energy consumption, and the overall capital cost can be reduced as smaller autoclaves can be used to produce the same amount of metal. The process would also enable the treatment of lower grade sulphide concentrates and boost the recovery of cobalt when compared to the traditional smelting route.

However, there are also disadvantages: there is a high unit cost associated with producing sulphide concentrates; sulphide and laterite deposits are not usually found close together hence transportation costs for the concentrates could be higher; and under oxidising conditions there is a higher dissolution of impurities from sulphide concentrates. Despite these disadvantages, Minproc believes that the co-treatment route has potential as it could reduce unit capital costs and operating costs, while at the same time boosting cobalt recovery from sulphides. □