PRELIMINARY METALLURGICAL TEST RESULTS
TURNAGAIN PROJECT

Dease Lake, British Columbia, Canada

for

Canadian Metals Exploration Ltd.
#1060-1090 W. Georgia St.,
Vancouver, BC, V6E 3V7

by
Frank Wright, BBA, BSc., P.Eng.

April 21, 2001

FR Wright & Associates Consulting
427 Fairway Dr., North Vancouver, BC, V7G 1L4

TABLE OF CONTENTS

Summary

The Turnagain mineral claims are located near Dease Lake, British Columbia (BC), Canada. The property exploration and process studies are being undertaken by Canadian Metals Exploration Ltd., of Vancouver, BC, who control 100% of the project. The geological model and exploration target being developed is for a near surface, bulk tonnage, resource in excess of 250 million tons[1] (227 million tonnes). This potential resource would consist of low grade nickel and cobalt, with possible other by-product credits., most notably platinum and palladium.

A metallurgical concept is being developed with the on-going geological work, using an average anticipated grade of 0.3% nickel and 0.015% cobalt. The preliminary studies on the work to date suggests that the project offers a promising opportunity by using a high rate of production, similar to what has been used by other base metal producers in the province.

The economic viability of this approach is supported by a number of important facts including;.

- significant operating cost savings experienced by the economies of scale for open pit, bulk tonnage production; verses smaller, higher grade, underground deposits that are traditionally exploited in sulfide nickel mining.

- a gross contained metal value at Turnagain which may be 2 to 3 times greater than other base metal mines (see Figure 1, Appendix) in the province. These operating mines use similar production methods as those envisioned for Turnagain, which consist of crushing, grinding, and froth flotation.

kg batch flotation test was conducted, comprising of bulk flotation and two stages of cleaning. Further cleaning including the use of regrinding and magnetic separation was not incorporated in order to retain sufficient weight for a series of batch pressure leach tests. The baseline composite sample analyzing 0.47% Ni and 176 ppm Co produced a sulfide concentrate as follows.

Flotation Concentrate
for Pressure Leach Evaluation

Metal	Ro. Flot. % Rec.	Final Tail Grade	2nd Cleaner Conc.Grade
Ni	77.9	0.12 %	6.72 %
Co	70.6	0.007 %	0.26 %
Pd	67.0	0.02 g/t	0.61 g/t
Pt	>75	<0.01 g/t	0.52 g/t

The ratio of concentration was 23:1. Characterization of the concentrate showed that pentlandite made up approximately 32% of the concentrate weight. Other principal minerals included 25 wt% pyrrhotite, 1 wt% chalcopyrite and the remaining 39% was gangue minerals. Pressure leaching of the concentrate achieved 98% nickel recovery and >92% cobalt recovery(cobalt in pressure leach residue was below detection limit). There appeared to be no deleterious substances in solution which would complicate production of cathode product by solvent extraction and electrowinning techniques. Pressure Leach residue was hand panned and achieved precious metal concentrations of 45.1 g/t gold, 4.2 g/t platinum, and 16.9 g/t palladium. More sophisticated gravity recovery techniques using a Falcon™ centrifugal concentrator were also undertaken. The concentrate appears to be suitable for on-site metal refining using recently developed hydrometallurgical techniques.

1.0 Introduction

The Turnagain Property is owned 100% by Canadian Metals Exploration Ltd. (formerly Bren-Mar Resources Ltd.). The property is located in northern British Columbia (BC), Canada, approximately 1350 km north of the City of Vancouver, BC (see Figure 2, Appendix). Bren-Mar conducted a geological program during 1996, 1997 and 1998, including geophysical work and drilling indicating the potential of large tonnage, low grade nickel cobalt mineralization. While the property has the potential for higher grade zones, the current geological model anticipates an open pit bulk tonnage deposit averaging approximately 0.3% Ni and 0.015% Co. Further geological studies, including geophysical work and diamond drilling have been recommended for better quantifying the potential resource tonnage and grade.

The target grade at Turnagain has gross contained metal values (Ni, Co) that are higher than other large base metal mines operating in the province. However, the nickel grade is only about 1/3 that of more traditional (usually underground) nickel sulfide operations. Since Turnagain is envisioned to be a large open pit resource considerable cost savings are expected due to bulk material mining processing and the associated economies of scale. The project in this respect is more analogous to many of the large copper operations in North America.

New developments in hydrometallurgy are also anticipated to reduce costs and improve metal recovery. The metallurgy will be crucial in justification to forwarding the project. The metallurgical studies have been intermittent since October 1997, and further studies are recommended with the continuing geological program. This report highlights the preliminary metallurgical results and a perceived process concept.

2.0 Background

The Turnagain property is located 65 km east of the community of Dease Lake, BC. The property is accessed from Dease Lake via helicopter or in summer by a poorly maintained 78 km dirt road.

Turnagain is located at latitude 58 20' north and longitude 128 58' west and is comprised of 91 contiguous claims. There is no apparent land use conflict to mining on the provincial (crown) land. Topography on the property consists of forested flats along the Turnagain River, rising to a sloping plateau situated above the tree line. The elevation varies from approximately 1000 m to 1400 m. The climate is generally mild summer days with cool evenings, to harsh sub freezing temperatures in winter. The annual mean temperature range at Dease Lake (elev. =816 m) is -17 °C to +12 °C. There is an average annual precipitation accumulation of 42 cm. Snow depth in winter months typically vary between 40 to 60 cm.

Mineralized showing were first discovered on the banks of the Turnagain River in 1956. This was followed up by Falconbridge Limited, who conducted geophysical work and shallow drilling between the years 1966 to 1973. Since then only minor studies were undertaken until 1996, when Bren-Mar acquired and expanded the property, and undertook the ongoing program.

The town of Dease Lake with a population of about 600 is the major commercial hub for the region. The town is accessed by regular air service and all weather roads. The Cassiar Highway (#37) connects Dease Lake, with the Yellowhead Highway (#16), 491 km to the south. The Cassiar Highway continues 237 km north from Dease Lake, joining the Alaska Highway near the community of Watson Lake, Yukon. A railway right of way also exists to Dease Lake, and the provincial government had envisioned a branch line connecting south east to the British Columbia Railway (BCR) mainline near the city of

Prince George. Currently, while there is a rail bed in place, the track only extends to within 275 km of the town. Trucking distance from the site to the railhead is 350 km. While there have been no formal plans to extend the rail line, there is considerable interest by various business groups and governments to undertake a study for the feasibility of establishing a rail line to Alaska.

The closest nickel refinery is located at Fort Saskatchewan, Alberta, and is operated by Sherritt International Ltd. The distance from the rail head to Fort Saskatchewan is approximately 1200 km. The closest ocean port is located at Stewart, BC. Stewart is accessed west from Meziadian Junction located on the Cassiar Highway. The trucking distance from the site to Stewart would be 470 km. Other port facilities, which can be serviced by both rail and road exist at Prince Rupert, BC and North Vancouver, BC.

Power is a major consideration for developing the property. Owing to abundant hydroelectric generating capacity, BC has among the lowest industrial energy rates in the world. Currently, the provincial integrated power grid extends only to the Meziadian Junction, located overland from the site a distance of approximately 250 km to the south. While there are other mineral deposits in the area that would benefit from access to the power, the government and BC Hydro have no formal plans to extend the grid.

A 1.7 MW hydropower plant located at Dease Lake can be expanded by 4 to 6 MW, but this would not meet the anticipated process demand. Power requirements could range considerably higher depending on the design throughput of the mineral processing circuit and if a refinery is to be included. There are undeveloped hydro power generation sites in the region that could provide the necessary power demand. Recently there have been increasing reports in the media that a consortium of various groups is evaluating the feasibility of constructing a pipeline to access the natural gas fields of the Alaska North Slopes. One proposed route for such a pipeline could be sufficiently close to the property to offer an obvious means of providing a source for electrical power.

3.0 Geology and Mineralogy[1,2]

The property is described as an ultramafic complex approximately 8 km in length and up to 3 km in width. It trends in a northwest-southeasterly direction, intruding and in fault contact with the Cache Creek Group meta-volcanic and meta-sedimentary rocks of the upper Palaeozoic and Triassic age The Turnagain ultramafic intrusion is of late Triassic age, with host rocks including dunite, peridotite, olivine, and pyroxenite. Magmatic disseminated to semimassive sulfides are hosted by the Turnagain complex, primarily within the olivine and pyroxenite rock. Anomalous nickel and cobalt values which are associated with the sulfides occurs over a wide area. This sulfide mineralization has also been shown to extend below the maximum drill indicated depth of 300 m. Nickel grades typically range between 0.15% to 0.6%, with some narrow intervals of up to1.4 % nickel. Cobalt grades average between 0.01% to 0.02%, with thin intersections of up to 0.07% Co. The chemical analyses on the composited samples used for the test work is provided in the Appendix.

Mineralogical studies have shown sulfides contain primarily ferronickel as pendlandite and pyrrhotite. Pyrite is only rarely reported. Chalcopyrite is noted in some samples, but is typically not reported or only present in trace amounts. The selected field samples indicate that the pendlandite is most commonly granular, occurring near the margins of pyrrhotite. Pentlandite intergrowths if present are associated with sulfides or magnetite, and not with the silicates. The pyrrhotite is often mixed at the margins with fine granular magnetite and typically is described as fine or moderately disseminated to net textured blebs, up to discrete vienlets of semi-massive sulfides within the host rock. The pyrrhotite may be intimately mixed with silicates at the margin. The ratio of pyrrhotite to pendlandite appears to vary considerably, although an approximate average ratio of 1.5:1 has been suggested. In some samples, pyrrhotite was absent, with the pendlandite present as scattered grains, 20-200 microns in size, associated with magnetite. A mineralogical report by Harris Exploration Services is included in the Appendix.

Prospecting suggests some areas of the claims may hold significant concentrations of platinum group metals (PGM), copper (Cu) and molybdenum (Mo). Gold values of up to 0.1 g/t were also reported in one drill hole. For the basis of this report, nickel and cobalt have been used as the only product metals. A stronger focus on the contained platinum and palladium values is recommended in future studies.

4.0 Laboratory Test Results

4.1 Sample Characterization

Continuous BQ diamond drill core was utilized for obtaining the mineral samples. The initial material was from a 1998 exploration program. The drill core had been split with half the core archived on site and the remainder transported to Vancouver, BC for analyses by Acme Analytical Laboratories Ltd. (Acme), as part of the geological program. Subsequently, selected intervals of the assay rejects were composited for use in the process studies. Samples were composited into a low, mid and high nickel grade, based on the exploration analyses and the geological core logging description, as outlined in Table 4.1.

Table 4.1: Composites 1998 Drill Program

Comp. #	Drill Hole (interval meters)	Description
1A	98-1 (6.7-20)	grey dunite, low Ni in sulfide stringers
1B	98-1 (31-52)	black dunite, low Ni in disseminated sulfides
1C	98-2 (6-32), 98-2 (50-112), 98-4 (22-41)	grey dunite, low Ni in disseminated sulfides
2A	98-1 (54-74), 98-4 (84-94), 98-4 (112-128)	black dunite, mid Ni in disseminated sulfides
2B	98-1 (194-218), 98-2 (32-50), 98-4 (98-108)	grey dunite, mid Ni in disseminated sulfides
3A	98-1 (218-236)	grey dunite, high Ni in disseminated sulfides

The samples were analyzed for 32 element ICP, whole ore, total carbon, total sulfur and sulfide sulfur. Nickel and cobalt were also analyzed by wet methods. Gold, platinum group and rare earth metals were analyzed by International Plasma Laboratory Ltd (IPL) for chemical analyses. The results and methodology are included in the Appendix, with a summary of the data provided below, in Table 4.2.

Table 4.2: Head Grade Analyses Composite Samples

Composite Number	Nickel %	Cobalt ppm	Sulfur % S_{total}	Iron %	Mg %
1A	0.28	153	4.00	12.9	14.6
1B	0.28	143	2.40	10.4	18.9
1C	0.27	143	1.04	8.6	23.0
2A	0.52	232	2.33	11.2	22.2
2B	0.47	176	1.17	8.3	29.0
3A	0.70	264	2.59	11.2	22.9

The nickel analyses in the composite samples ranged from 0.28% to 0.70%, agreeing well with the exploration assays on which the compositing was based. The sulfur content is notably higher in these samples than the samples which had been undertaken in some earlier scoping studies.

The chemical analyses show that nickel and cobalt were the principle products of economic interest. Other portions of the property are reported to contain elevated copper, molybdenum, and silver concentrations that were not evident in the composite samples. Potential by-products include gold, platinum, palladium, which do report to the flotation concentrate. Magnetite is also present in significant quantity and may be recovered for use in the northeast coal industry for heavy media separation. Magnesium may also offer opportunities to recover from the process circuit. However, for the purpose of this report and in developing the project viability, only nickel and cobalt are being evaluated at this time.

Four of the composite samples (1A, 1B, 2B, 3A) and one tailing sample from a subsequent flotation test on composite 2B, were submitted for mineralogical study. The analyses was conducted by Harris Exploration Services, of North Vancouver, BC. A copy of the report, along with selected photomicrographs are attached in the Appendix.

The samples are all described as sulfide bearing ultramafic rock, with minor variation in silicate mineralogy. The dominant sulfide is pyrrhotite, with pentlandite as an accessory. There was no reported pyrite and only a trace of chalcopyrite reported. The sulfides are relatively coarse grained (up 500μ or more), in the silicates, with some finer grained intergrowths noted in Samples 1A and 1B. Intergrowths within the sulfides are also described as relatively coarse, with pentlandite present as liberated grains (50-300 μ) within pyrrhotite, or occasionally associated with magnetite. It was stated that "liberation should be accomplished by relatively coarse grinding".

The comparison of nickel concentration between chemical and petrographic analyses is open to some debate, particularly for the lower grade samples. Harris suggests that nickel may also be extracted from mafic silicate gangue minerals. This is an important statement as ultramafic rock can have background nickel concentrations of 0.1 to 0.2%. If the nickel is associated with the silicates it would be unlikely to be economically extracted. Consequently, a significant number of Canadian geologists and mineralogists are of the opinion that low grade nickel deposits are not economic. This is a possible explanation why the potential of the Turnagain Project has not been fully appreciated. It is therefore important to fully evaluate nickel distribution.

There are a number of facts which tend to disprove that any significant portion of the indicated nickel content is associated with silicates for the resource zones drilled on the Turnagain project. These include as follows;

- chemical analyses by various independent laboratories whose analysts have stated the digestion procedures would not result in appreciable amounts of nickel in silicates being dissolved.

- discussions with Bruce Downing the project geologist who reported that nickel is enriched in olivine minerals only if the dunite is non-sulphide bearing. Nickel is depleted in olivine that is associated with sulphide bearing dunite. Therefore there is little nickel available for quantitative analyses in silicate minerals for the samples which have been tested.

- a quote from the Harris report that "the texturally heterogeneous character of this sample and the partial occurrence of sulfides in finely dispersed and often poorly polished form, reduces the precision of optical differentiation between pyrrhotite and pentlandite."

- the possibility of some nickel values present in nickeliferrous pyrrhotite.

- the mass balances and relatively high flotation recoveries would not account for nickel associated within silicate minerals. The simple fact is that the majority of the nickel is concentrated with sulfide flotation, in a mass relationship with the sulfide content. Further it is only this nickel production (sulphide concentrate) that the project economics are based, not any nickel lost with the mainly silicate tailing.

These facts indicate that the reported analytical content for nickel is unlikely to be significantly associated with silicates. However, to better evaluate flotation losses, and supply additional confidence to the project mineralogy, further studies (including microscopic point count) are recommended on nickel distribution for ore and tailing samples.

4.2 Concentration of Sulphides

Most of the preliminary flotation studies have focused on producing a low to medium grade nickel concentrate for use with pressure leach processing. The work on optimizing rougher flotation was conducted at PRA and are provided in the Appendix. Composite

2B was used for baseline studies as it was the most common mid-grade material available. The first three tests varied the grind and results are outlined in Table 4.3.

Table 4.3: Composite 2B - Rougher Flotation vs. Grind

Test #	Grind K80 u	% Ni Grade Conc	Tailing	Recovery %Ni	%Co	%S–
F1	75	2.43	0.176	69.4	64.9	64.5
F2	51	1.61	0.188	70.7	64.6	65.5
F3	99	1.93	0.194	65.2	61.7	62.8

The results show the lowest nickel tailing losses for test F1, with a K_{80} = 75 microns. The highest losses were at the coarser grind (F3). The finer grind (F2) shows a reduced nickel grade in the concentrate, likely due to increased slimes entrainment. A mineralogical examination of the F1 tailing is included in the report in the Appendix. It indicates that some losses may be attributed to sulfide intergrowths, including those with magnetite. It was also stated that significant free sulfide was noted and that improved recovery might be expected.

Modifications to the bulk flotation reagent scheme were conducted on several tests including the use of sulphidizing agents, slime dispersent, and altering the collector addition, with no appreciable improvement in metal recovery. Further modifications to the flotation procedure or perhaps the use of magnetic separation is recommended in future studies.

A large batch flotation test (F9) was undertaken in a 54 L cell to produce several kilograms of concentrate for pressure leach testing. As there was an insufficient amount of Composite 2B material to make up the required 90 kg of feed weight, additional material was blended in to approximate the original feed grade of Composite 2B. The final blend used 70 wt% Composite 2B, 20 wt% Composite 1C, and 10 wt% Composite 3A. The flotation procedure utilized two cleaner stages. The optimum conditions found from the bench program were used, but with no magnetic concentration included. The grind K_{80} was 71 microns and detailed procedures and results are included in the PRA report data dated January 12, 1999, attached in the Appendix. A summary of the metal distribution for the test F9 is given in Table 4.4, below.

Table 4.4: Flotation Concentration Metal Distribution used for Pressure Leach Testing

Metal	Rougher % Rec.	Final Tail Grade	Cleaner Conc Grade
Ni	77.9	0.12 %	6.72 %
Co	70.6	0.007 %	0.26 %
Pd	67.0	0.02 g/t	0.61 g/t
Pt	>75	<0.01 g/t	0.52 g/t
Au	66.1	0.01 g/t	0.27 g/t
Ag	25.7	5.2 g/t	9.3 g/t
Fe	31.2	8.0 %	12.0 %
Mg	19.3	21.0 %	19.3 %

The results showed an improved nickel recovery compared to batch bench scale tests conducted on Composite 2B. The nickel grade in the tailing was 0.12% compared to a range of 0.15% to 0.18% in most of the bench studies. The difference may be a result of the addition of Composite 1C and Composite 3A to the blended feed material, or from variations in using the larger scale process equipment.

The mass ratio of concentration was approximately 23:1, with 4.3% of the feed weight reporting to the second cleaner concentrate. Further upgrading can likely be achieved by additional flotation cleaning with regrinding and/or incorporating magnetic separation. Locked cycle testing will be required to provide the overall concentration recovery.

Using information generated from the Composite 2B bench program, the remaining 5 composites from the 1998 drill program were subjected to bulk flotation studies. The results are summarized in Table 4.5.

Table 4.5: Rougher Flotation Results on 1998 Drill Core Composites

Float Test #	Comp. Sample#	Calc.Hd %Ni	Grind K80 (u)	Rougher Recovery %Ni	%Co	Ro. Conc %Ni	Ro.Tail %Ni
F1	2B	0.49	75	69.4	64.9	2.43	0.176
F10	1C	0.30	70	67.7	54.9	1.05	0.12
F11	1A	0.38	75	70.4	61.3	1.12	0.15
F12	1B	0.30	80	67.7	62.1	1.00	0.12
F13	2A	0.54	72	83.1	76.3	2.04	0.12
F14	3A	0.69	74	79.3	69.3	2.40	0.18

The grade of the rougher concentrate is shown to increase with increasing nickel content in the feed. A minor trend is evident for improved recovery with higher feed grade.

Based on mineralogical examination of the F1 tailing, the use of magnetic separation was investigated. The tailing from Comp. 2B were subjected to magnetic separation using a Davis Tube (low wet intensity) with the following results.

Table 4.6: Low Intensity Magnetic Separation on Bulk Tailing

Comp 2B Tail Test #	Mag Intensity Gauss	% Ni Grade Conc	Tailing	Mag Recovery %Ni
M1 (F2 Tailing)	1000	0.472	0.164	14.0
M2 (F1 Tailing)	1000	0.320	0.160	15.7
M3 (F3 Tailing)	1000	0.280	0.174	11.0
M4 (F2 Tailing)	2000	0.388	0.152	23.0

The results show that the use of magnetic separation reduced nickel losses to the final tailing. The improvement to the overall concentration recovery at 1000 gauss was between 3.8 to 4.8 percent. Increasing the magnetic intensity to 2000 gauss resulted in improving the overall recovery for F2 tailing from 4.13% to 6.74%. A summary of the improvements to recovery are given in Table 4.7, below.

Table 4.7: Comp 2B Concentration Recovery (flotation followed by magnetic separation)

Test #'s	% Nickel Recovery		
Flot / Mag	Bulk Float	Tail (Mag)	Total
F1 / M2	69.4	4.8	74.2
F2 / M1	70.7	4.1	74.8
F2 / M4	70.7	6.7	77.7
F3 / M3	65.2	3.9	69.1

The work shows that magnetic separation can make a significant contribution to nickel recovery. Test F6 was undertaken using magnetic separation prior to flotation. The results show a higher nickel recovery (75.1%) than would have been expected without using magnetic separation. The nickel grade to the magnetic and flotation concentrates were 0.73% and 2.4%, respectively without using cleaning stages on either process. Each of these rougher concentrates accounted for about 11% of the feed weight. More optimization in magnetic separation is recommended.

4.3 Pressure Leach Testing

Pressure leaching of the concentrate is currently the principal technology considered for on-site treatment of the Turnagain concentrate. This process offers a number of potential advantages in an iron sulfate system. Depending on the process conditions used iron can be precipitated as hematite and elemental sulfur produced from sulfate. This can offer both economic and environmental advantages over alternate processes.

A variation of pressure leaching technology was tested using the CESL process, which utilizes a small amount of HCL during the leach. A series of eight batch tests was undertaken by Cominco Engineering Services Ltd.(CESL), located in Vancouver. A copy of their report, dated March 30, 1999 is included in the Appendix. The concentrate used for the program was obtained from a large batch flotation program (PRA-Test F9, described previously). Characterization of the concentrate showed that pentlandite made up approximately 32% of the concentrate weight. Other principal minerals include 25 wt% pyrrhotite, 1 wt% chalcopyrite and the remaining 39% as gangue minerals. The CESL analyses indicated a lower nickel content (6.3%) and higher cobalt content (0.31%) than the previous analyses of the concentrate, which had analyzed 6.7% Ni and 0.26% Co, respectively.

The CESL tests were run with oxygen at 200 psi, at 130°C for 60 minutes. The concentrate had been reground prior placing in the autoclave and underwent a one hour atmospheric leach following removal of the slurry from the autoclave. The optimum free acid was 60 g/L, at a pulp density of 200 g/L solids. The oxygen consumption was considered favorable, at 0.16 gms oxygen per gm of concentrate. There was approximately 98% Ni extraction and greater than 92% cobalt extraction. The calculation of the cobalt extraction was restricted by the detection limit for low cobalt content remaining in the pressure leach residue. CESL concluded that the concentrate was well suited to the process, but that some method of magnesium rejection would be required.

In April 2001 the residues from the CESL test program were composited and subjected to gravity recovery using a Falcon™ centrifugal concentrator. Overall recoveries from the Falcon were 69% Au, 28% Ag, 52% Pt, and 31% Pd. Hand panning was used for cleaning resulting in grades of 86.4 g/t Au, 246 g/t Ag, 8.3 g/t Pt, 14.7 g/t Pd, but with significant losses of recovery. The detailed results are provided in the Appendix. While the gold responded well to gravity techniques the PGM were less encouraging likely owing to being present at a very fine particle size. Flotation may offer a more promising recovery technique for upgrading PGM's from the pressure leach residue.

5.0 Recommendations

The preliminary results are encouraging with an overall indicated nickel recovery for the composite samples ranging between 65% to 80%. Cobalt extraction typically ranged between 60% to 75%, expect for Comp 1C that had a cobalt recovery of 54%. The next phase of work should better define and hopefully improve on the metal recovery indicated to date.

The following metallurgical studies will be required for developing the flowsheet and prior to undertaking a pre-feasibility study.

- Crushing and grinding work indexes of several representative samples should be performed.

- There are still substantial sulfide (and accompanying metal) losses to bulk flotation tailing. An earlier test program achieved lower tailing nickel losses than the most recent study using similar conditions. There is a need to confirm if the difference is due strictly to mineralogy of the different samples, or to testwork techniques. This should include examination of the analytical (digestion) procedures used. Similar programs including incorporating magnetic separation using one sample composite should be performed at two or more laboratories.

- Cleaning of the bulk concentrate needs to be defined, including the use of regrinding and slime dispersent. A pentlandite pyrrhotite differential flotation should also be investigated. Locked cycle testing is required to determine overall flotation recovery, and to establish a grade verses recovery curve.

- Further pressure leach studies, including continuous mini-piloting. Research into alternate leaching techniques, including biological leaching, may be warranted.

- Evaluation for recovery of potential by-product credits, particularly platinum and palladium. Gold, silver, magnesium and magnetite also offer additional value. Other areas of the property may have increased copper and molybdenum content, which could alter circuit design requirements.

- Begin preliminary work on metal recovery from solutions including, ion exchange, solvent extraction and electrowinning. While these processes are fairly standard for the industry, the effect of possible detrimental elements on cathode quality will need to be determined.

- Ongoing testwork, should be conducted on representative samples with the developing geology in defining the mineable grade, tonnage, and metallurgical response. Additional mineralogical examinations of the various product streams is recommended to determine where improvements to recovery might be achieved.

A budget of $350,000 is estimated to be required for metallurgical testing for establishing pre-feasibility data requirements. This does not include costs for sample collection or engineering.

6.0 Conclusions

A preliminary test program has shown that the Turnagain drill core samples provide a positive response to conventional mineral processing techniques. A primary grind (K_{80}) of 74 microns (200 mesh) appears sufficient for optimum metal recovery. Nickel recoveries to the rougher flotation concentrate ranged from 67% to 83%, and cobalt recoveries ranged from 55 to 76%. Nickel grades to a cleaner concentrate ranged up to 14% for nickel, but with significant losses during open cycle cleaning. There are indications further improvements can be made with the use of magnetic separation and additional flotation cleaning, including the use of regrinding the bulk concentrate and or middling streams.

Pressure leaching a low grade sulfide concentrate resulted in 98% nickel extraction and greater than 92% cobalt extraction. The concentrate appeared to be well suited to pressure leaching. There appear to be no serious environmental concerns relating to waste rock or flotation tailing quality.

Infrastructure, including power supply will be of paramount importance. The project geology would need to establish that a near surface, bulk tonnage deposit can be developed. This would allow for an operation to process low grade ore to produce a nickel sulfide concentrate, with possible cobalt, gold and PGM credits. Recently developed hydrometallurgical technology appears to offer a cost effective option to treat the concentrate on-site. Further study on the Turnagain Project is warranted.

References

1. Turnagain Nickel-Cobalt Project Report, Bren-Mar Resources Ltd., April 1999 (Draft), Downing B.

2. Geophysical and Drilling Report on the Turnagain Project, Gamah International Limited, Vancouver, November 1997, Downing B.

3. Personnel Communication (Verbal) with Blair Salter, Manager Customer Development, BC Hydro, June 16,1999

4. Personnel Communications (Written and Verbal) with Bill Harland, Director Power Division, Stothert Group, Vancouver, May / June 1999

5. A Qualification of and Underlying Factors for Differences in Mine Operating Costs in Selected Countries and Sates or Provinces of the American Continents, Behre Dolbear & Co., New York, Presented January 1994 Cordilleran Roundup, Vancouver, Schreiber H., Kuestermeyer A.

6. Royal Oak Mines Inc., 5501 Lakeview Dr., Kirkland Washington, 1996, 1997, 1998 Annual Reports and Corporate New Releases.

7. A Preliminary Evaluation of the Metallurgical Approach in Treating the Old Nick Deposit, Applied Mine Technologies Inc., Vancouver BC, Wright F.R.

8. Dominion Mining Limited, Perth West Australia. Annual Report 1996, Yakabindie Nickel Project Brochures, Feb 1996, November 1996. Western Minerals Technology Pty. Ltd. Randol Vancouver 1996.

9. CESL Process for Nickel - Cobalt - Copper Sulfides., Cominco Engineering Services Ltd., Richmond BC, May 1998, Jones D., Hestrin J., Moore R.

10. Process Development Concept and Economic Potential for the Turnagain Project, FR Wright & Associates Consulting, November 2000

STATEMENT OF QUALIFICATIONS

I, **Frank R. Wright** do hereby certify:

I am a Consulting Metallurgical Engineer, practicing at 427 Fairway Dr., North Vancouver, BC, Canada

I graduated with a Bachelor of Science, in Metallurgical Engineering, obtained in 1979 from the University of Alberta., Edmonton Alberta. I also obtained a degree with a Bachelor of Business Administration, from Simon Fraser University, Burnaby BC, in 1984.

I am a registered member in good standing with the Association of Professional Engineers and Geoscientists of British Columbia, Registration No. 15747.

I have continuously practiced my profession for over 20 years.

This report dated April 21, 2001, has been issued to Canadian Metals Exploration Ltd., relating to the Turnagain mineral exploration property, British Columbia, Canada. My written report is based on personnel knowledge of the project and on metallurgical information generated and supplied by other parties as outlined in the report.

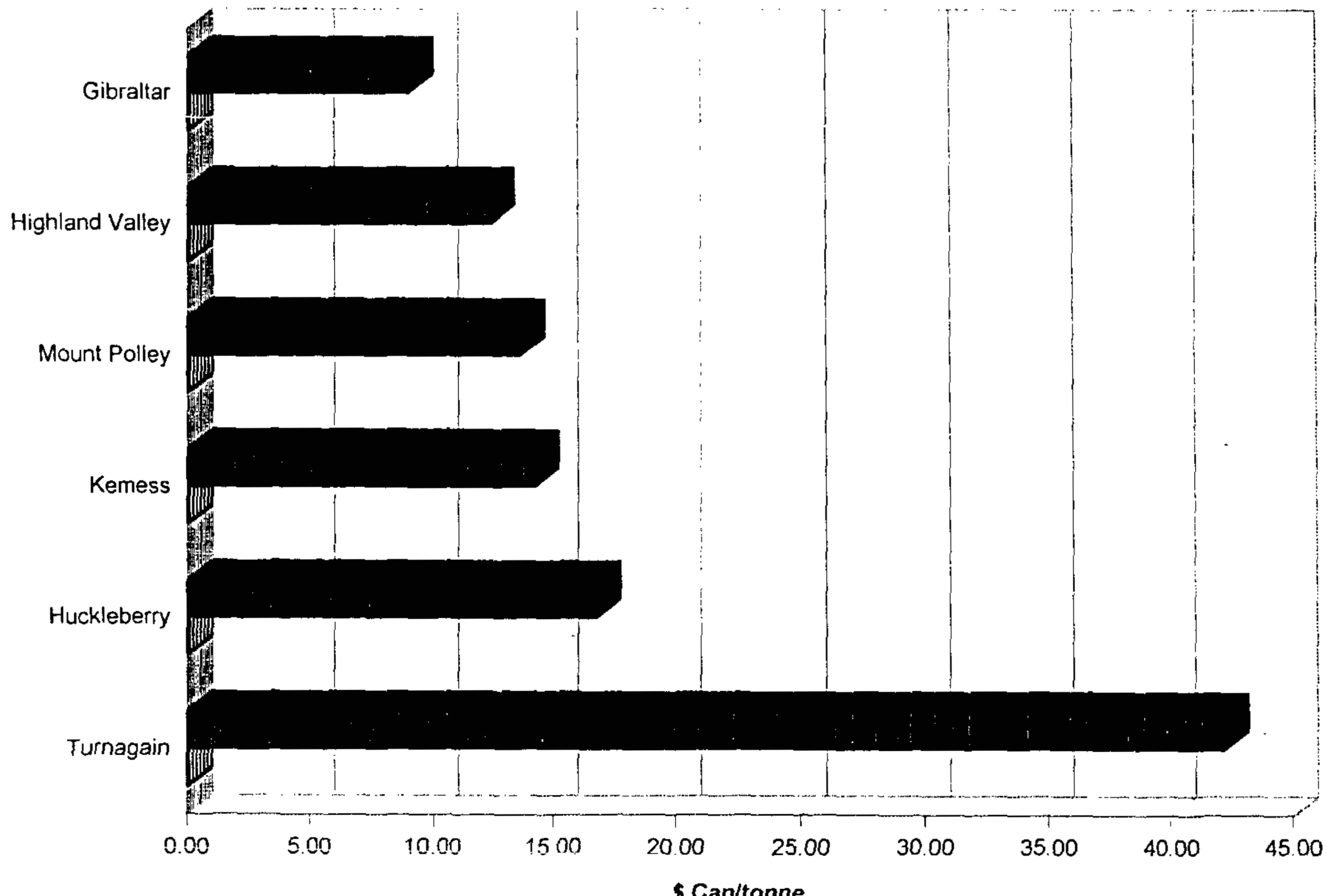
I confirm that I have not received any interest in the Turnagain Project, or that I own directly or indirectly any securities of Canadian Metals Exploration Ltd.

Dated this 21st day of April, 2001 at North Vancouver, BC

Frank Wright, P.Eng.

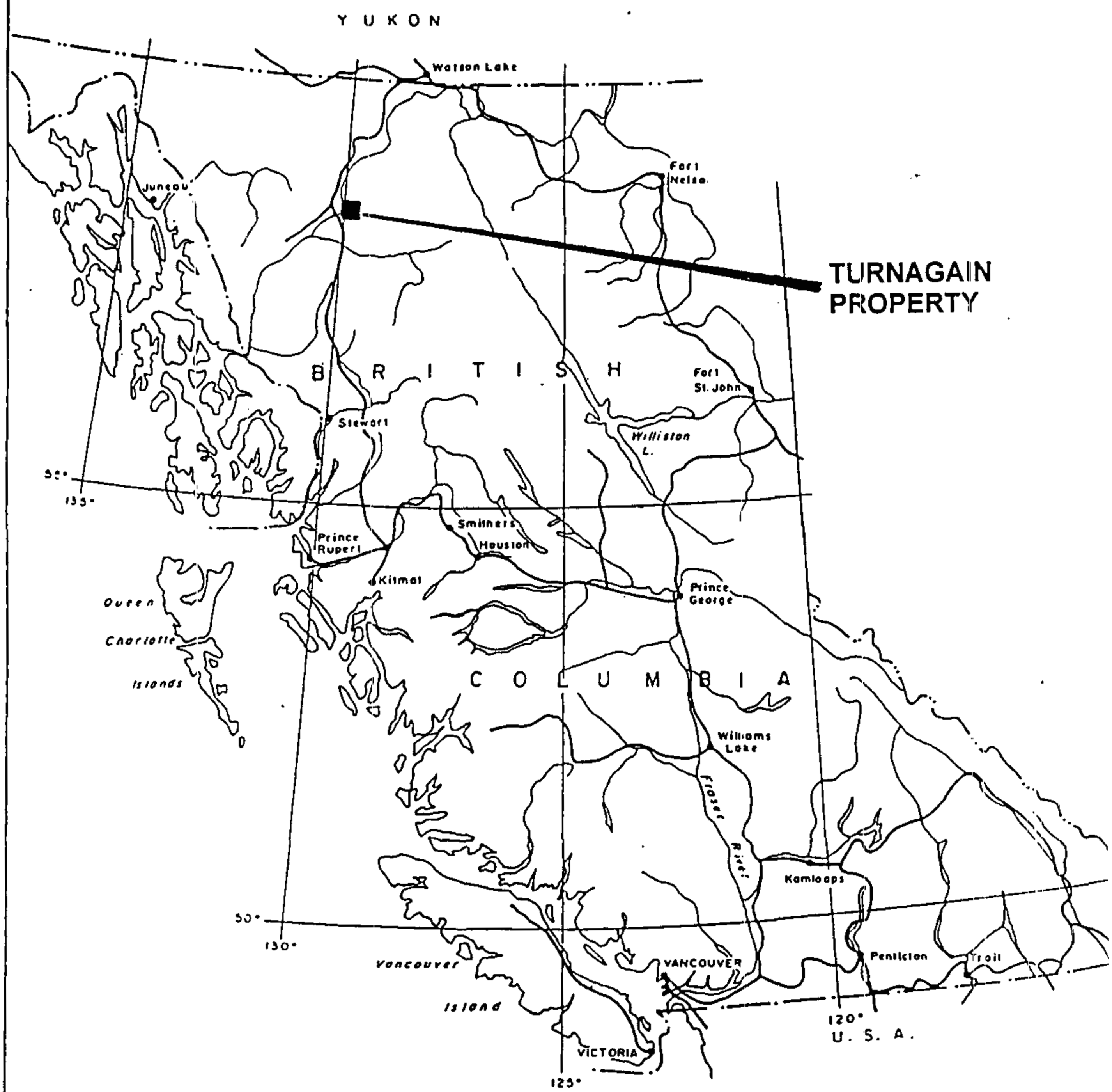
Price Ref: Internet q[illegible]n E-MJ com

Figure 1:
British Columbia Mineral Deposits
Open pit bulk tonnage
Ranked by Deposit Gross Unit Metal Value



Metal Price (US$)
Cobalt ($US/lb): = 15.00
Copper ($US/lb): = 0.83
Gold ($US/oz): = 268
Moly ($US/lb): = 2.60
Nickel ($US/lb) = 3.50

October, 2000
FW/Brenmar/graphvalu



BREN-MAR RESOURCES LTD.

TURNAGAIN PROPERTY

LOCATION MAP

N.T.S. 104 I-7

LIARD M.D., B.C.

0 100 200 300 400 500 KM.

DRAWN BY : E.L.	DATE : JULY 1996
SCALE : AS SHOWN	FIGURE Nº. 2

CHONG

CERTIFICATE OF ANALYSIS
iPL 98L1286

2036 Columbia Street
Vancouver, B.C
Canada V5Y 3E1
Phone (604) 879-7878
Fax (604) 879-7898

Process Research Associates Ltd

Project: 98-044

Out: Dec 11, 1998
In : Dec 07, 1998

[128614:36:43:89121198]

5 Samples 5=Pulp

Symbol	Unit	Pulp 1A Head	Pulp 1B Head	Pulp 1C Head	Pulp 2A Head	Limit Low	Limit High
S(tot)	%	4.32	2.51	1.02	2.42	0.01	100.00
S(-2)	%	4.21	2.35	2.71	2.51	0.01	100.00
Co	%	0.02	0.02	0.02	0.03	0.01	100.00
Fe	%	12.91	10.35	8.64	11.16	0.01	100.00
Mg	%	14.56	18.90	22.97	22.22	0.01	100.00
Ni	%	0.29	0.29	0.28	0.58	0.01	100.00
Al	ppm	5841.	3012.	1343.	1051.	100.	50000.
Sb	ppm	<5.	<5.	<5.	<5.	5.	1000.
As	ppm	<5.	<5.	<5.	<5.	5.	10000.
Ba	ppm	32.	24.	<2.	<2.	2.	10000.
Bi	ppm	<2.	<2.	<2.	<2.	2.	10000.
Cd	ppm	16.9	14.2	12.1	14.9	0.1	100.0
Ca	ppm	3616.	804.	2131.	715.	100.	100000.
Cr	ppm	593.	816.	279.	762.	1.	10000.
Co	ppm	153.	143.	143.	232.	1.	10000.
Cu	ppm	668.	379.	318.	761.	1.	20000.
Fe	ppm	12%.	9.8%	8.5%	10%.	100.	50000.
La	ppm	<2.	<2.	<2.	<2.	2.	10000.
Pb	ppm	22.	22.	20.	26.	2.	20000.
Mg	ppm	12%.	15%.	19%.	19%.	100.	100000.
Mn	ppm	736.	787.	868.	976.	1.	10000.
Hg	ppm	<3.	<3.	<3.	<3.	3.	10000.
Mo	ppm	23.	11.	3.	5.	1.	1000.
Ni	ppm	3234.	3139.	3074.	5836.	1.	10000.
P	ppm	<100.	<100.	<100.	<100.	100.	50000.
K	ppm	2763.	497.	<100.	<100.	100.	100000.
Sc	ppm	4.	4.	3.	4.	1.	10000
Ag	ppm	0.7	0.4	0.3	0.6	0.1	100.0
Na	ppm	<100.	<100.	204.	<100.	100.	50000.
Sr	ppm	12.	2.	9.	<1.	1.	10000.
Tl	ppm	<10.	<10.	<10.	<10.	10.	1000.
Ti	ppm	139.	182.	<100.	<100.	100.	10000.
W	ppm	<5.	<5.	<5.	<5.	5.	1000.
V	ppm	113.	56.	13.	26.	2.	10000.
Zn	ppm	83.	79.	38.	47.	1.	20000.
Zr	ppm	2.	2.	1.	2.	1.	10000.

Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

BC Certified Assayer: David Chiu

iPL
INTERNATIONAL PLASMA LABORATORY LTD.

CERTIFICATE OF ANALYSIS
iPL 98K1238

2036 Columbia Street
Vancouver, B.C.
Canada V5Y 3E1
Phone (604) 879-7878
Fax (604) 879-7898

Process Research Associates Ltd

Project: 98-044

Out: Nov 25, 1998
In : Nov 18, 1998

[123811:13:53:8911:2598]

1 Samples 1=Pulp

Symbol	Unit	Pulp 2B	Limit Low	Limit High
Au	g/mt	0.02	0.01	9999.00
Ag	g/mt	0.6	0.3	99999.0
Pt	g/mt	<0.01	0.01	9999.00
Pd	g/mt	0.03	0.01	9999.00
Al	ppm	509.	100.	50000.
Sb	ppm	<5.	5.	1000.
As	ppm	<5.	5.	10000.
Ba	ppm	3.	2.	10000.
Bi	ppm	<2.	2.	10000.
Cd	ppm	14.1	0.1	100.0
Ca	ppm	601.	100.	100000.
Cr	ppm	371.	1.	10000.
Co	ppm	176.	1.	10000.
Cu	ppm	541.	1.	20000.
Fe	ppm	8.3%	100.	50000.
La	ppm	<2.	2.	10000.
Pb	ppm	20.	2.	20000.
Mg	ppm	29%.	100.	100000.
Mn	ppm	907.	1.	10000.
Hg	ppm	<3.	3.	10000.
Mo	ppm	4.	1.	1000.
Ni	ppm	4596.	1.	10000.
P	ppm	<100.	100.	50000.
K	ppm	<100.	100.	100000.
Sc	ppm	3.	1.	10000.
Ag	ppm	1.3	0.1	100.0
Na	ppm	<100.	100.	50000.
Sr	ppm	<1.	1.	10000.
Tl	ppm	<10.	10.	1000.
Ti	ppm	<100.	100.	10000.
W	ppm	<5.	5.	1000.
V	ppm	12.	2.	10000.
Zn	ppm	49.	1.	20000.
Zr	ppm	1.	1.	10000.

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

BC Certified Assayer: David Chiu ____________

CERTIFICATE OF ANALYSIS
iPL 98L1286

2036 Columbia Street
Vancouver, B.C.
Canada V5Y 3E1
Phone (604) 879-7878
Fax (604) 879-7898

INTERNATIONAL PLASMA LABORATORY LTD.

Process Research Associates Ltd

Project: 98-044

Out: Dec 11, 1998
In : Dec 07, 1998

5 Samples 5=Pulp

[128614:36:43:89121198]

Symbol	Unit	Pulp 3A Head	Limit Low	Limit High
S(tot)	%	2.79	0.01	100.00
S(-2)	%	0.92	0.01	100.00
Co	%	0.03	0.01	100.00
Fe	%	11.20	0.01	100.00
Mg	%	22.89	0.01	100.00
Ni	%	0.77	0.01	100.00
Al	ppm	922.	100.	50000.
Sb	ppm	<5.	5.	1000.
As	ppm	<5.	5.	10000.
Ba	ppm	2.	2.	10000.
Bi	ppm	<2.	2.	10000.
Cd	ppm	15.9	0.1	100.0
Ca	ppm	547.	100.	100000.
Cr	ppm	798.	1.	10000.
Co	ppm	264.	1.	10000.
Cu	ppm	1740.	1.	20000.
Fe	ppm	10%.	100.	50000.
La	ppm	<2.	2.	10000.
Pb	ppm	27.	2.	20000.
Mg	ppm	19%.	100.	100000.
Mn	ppm	885.	1.	10000.
Hg	ppm	<3.	3.	10000.
Mo	ppm	3.	1.	1000.
Ni	ppm	7782.	1.	10000.
P	ppm	<100.	100.	50000.
K	ppm	<100.	100.	100000.
Sc	ppm	3.	1.	10000.
Ag	ppm	1.1	0.1	100.0
Na	ppm	<100.	100.	50000.
Sr	ppm	<1.	1.	10000.
Tl	ppm	<10.	10.	1000.
Ti	ppm	<100.	100.	10000.
W	ppm	<5.	5.	1000.
V	ppm	20.	2.	10000.
Zn	ppm	45.	1.	20000.
Zr	ppm	1.	1.	10000.

—=No Test Ins=Insufficient Sample Del=Delay Max=No Estimate Rec=ReCheck m=x1000 %=Estimate % NS=No Sample

BC Certified Assayer: David Chiu

Harris
EXPLORATION SERVICES

MINERALOGY AND GEOCHEMISTRY

534 ELLIS STREET, NORTH VANCOUVER, B.C., CANADA V7H 2G6 TELEPHONE (604) 929-5867

Report for: F.R. Wright & Associates,
427 Fairway Drive,
NORTH VANCOUVER, B.C.
V7G 1L4

Report 98-124

January 18, 1999

MINERALOGICAL EXAMINATION OF METALLURGICAL TEST SAMPLES FROM THE TURNAGAIN PROJECT

Introduction:

5 samples, labelled as below, were submitted by Frank Wright. Portions of each were prepared for examination as smear-mount polished thin sections (slide numbers as shown).

Sample		Slide no.
98-044	Tails Fl-1	99-398
98-044	Head 1A	99-399
98-044	Head 1B	99-400
98-044	Head 2B	99-401
98-044	Head 3A	99-402

Summary:

The head samples all consist of sulfide-bearing ultramafic rock material; however, they show some variations in silicate mineralogy. 1A is distinct from the others of the suite in consisting of a mixture of amphibole, pyroxene and serpentine, with only minor olivine. 1B is composed of olivine and serpentine in approximately equal proportions, whilst in 2B and 3A the silicates are dominantly fresh olivine, with only minor accessory serpentine.

In all cases the sulfides consist dominantly of pyrrhotite, with pentlandite as an accessory. Traces of chalcopyrite are also seen.

Estimated total sulfide contents range from 6-10%. The ratio of pentlandite to pyrrhotite appears relatively low in 1A and 1B (in the 0.05 - 0.07 range), but is signficantly higher in 2B (estimated 0.2) and strikingly so in 3A (pentlandite:pyrrhotite ratio of c. 0.4).

For the most part, the sulfides exhibit a relatively coarse-grained textural mode - liberated grains of 500 microns or more being not uncommon in the head samples. Good overall liberation of sulfides from silicates should be readily attainable without excessively fine grinding. It also appears that mutual intergrowths within the sulfides are typically on a relatively coarse scale, and the bulk of the pentlandite seen in these head samples is as liberated grains or simple composites (on a scale of 50 - 300 microns) with pyrrhotite - or, occasionally, with the magnetite which is an ubiquitous minor accessory.

Samples 1A and 1B are distinctive in containing a proportion of the sulfides (estimated 10-20% in total) as intimate fine-grained intergrowths with silicates which will not be fully liberatable. Such intergrowths appear rare to absent in 2B and 3A.

The Tails sample (stated to be derived from the treatment of Head 2B) contains an estimated 1.5% unrecovered sulfides - representing 20% of the concentration in the head. These sulfides are almost entirely pyrrhotite.

Pentlandite (except for a single liberated grain) appears to be virtually entirely present as sparse inclusions, 10 - 50 microns in size, in the pyrrhotite.

The sulfides in the tails occur dominantly (estimated 95%) as liberated grains, so potential clearly exists for improved total sulfide recovery in flotation.

Although the observed pentlandite concentration in this tails is <0.1%, the sample is stated to assay 0.17% Ni. This suggests that the assay method used must also extract Ni from the mafic silicates. Ni contents of 0.1 - 0.2% are not abnormal for ultramafic olivines and serpentine.

Similar discrepancies between the optically estimated pentlandite contents and the Ni assays also exist in the head samples.

Individual sample descriptions, plus a set of photomicrographs (illustrating the modes of occurrence of pentlandite in Head sample 2B and the derived tails) are attached.

J.F. Harris Ph.D.

FLOTATION TEST METALLURGICAL BALANCE

Date: 12-Jan-99
Project: 98-044

Client: Bren-Mar
Test: F9
Sample: Comp (70/20/10, 2B/1C/3A)

Objective: To produce rougher concentrate for pressure leaching

Products	Weight (g)	Weight (%)	Assay (%) Ni	Assay (%) Co	Assay (%) Fe	Distribution (%) Ni	Distribution (%) Co	Distribution (%) Fe
Cleaner Concentrate	3627	4.3	6.72	0.257	30.0	67.1	61.4	14.2
Cleaner Tail	15900	17.9	0.26	0.009	6.60	10.8	9.2	17.0
Total Flotation Concentrate	**19727**	**22.2**	**1.51**	**0.057**	**12.8**	**77.9**	**70.6**	**31.2**
Final Tails	69273	77.8	0.12	0.007	8.00	22.1	29.4	68.8
Assay Head	**89000**	**100.0**	**0.43**	**0.018**	**9.05**	**100.0**	**100.0**	**100.0**
Measured Head								

Products	Weight (g)	Weight (%)	Assay (g/t) Au	Assay (g/t) Ag	Assay (g/t) Pt	Distribution (%) Au	Distribution (%) Ag	Distribution (%) Pt
Cleaner Concentrate	3627	4.3	0.27	9.30	0.52	50.6	7.3	100.0
Cleaner Tail	15900	17.9	0.02	5.30	<0.01	15.6	18.4	0.0
Total Flotation Concentrate	**19727**	**22.2**	0.07	**6.32**	**0.10**	**66.1**	**25.7**	**100.0**
Final Tails	69273	77.8	[illegible]	5.20	<0.01	33.9	74.3	0.0
Assay Head	**89000**	**100.0**	**0.02**	**5.45**	0.02	**100.0**	**100.0**	**100.0**
Measured Head								

Products	Weight (g)	Weight (%)	Assay Pd (g/t)	Assay Rh (g/t)	Assay Mg (%)	Distribution (%) Pd (g/t)	Distribution (%) Rh (g/t)	Distribution (%) Mg (%)
Cleaner Concentrate	3327	4.3	0.61	0.020	8.03	55.6	100.0	1.7
Cleaner Tail	15900	17.9	0.03	<0.01	20.0	11.4	0.0	17.6
Total Flotation Concentrate	**19727**	[illegible]	**0.1[illegible]**	**0.004**	**17.7**	67.0	100.0	19.3
Final Tails	69273	[illegible]	[illegible]	<0.0[illegible]	21.0	33.0	0.0	80.7
Assay Head	**89000**	100.0	**0.05**	0.001	20.[illegible]	100.0	100.0	100.0
Measured Head								

FLOTATION TEST METALLURGICAL BALANCE

Client: Bren-Mar
Test: F10
Sample: Comp #1C

Date: 25-Jan-99
Project: 98-044

bjective: Variability Study

Products	Weight		Assay (%)				Distribution (%)	
	(g)	(%)	Ni	Co	Mg	$S_{(T)}$	Ni	Co
Concentrate 1	45.8	4.7	1.88				28.8	
Concentrate 2	84.3	8.6	1.08				30.5	
Concentrate 1 + 2	**130.1**	**13.3**	**1.36**				**59.3**	
Concentrate 3	37.5	3.8	0.45				5.6	
Concentrate 1 +2 +3	**167.6**	**17.1**	**1.16**				**65.0**	
Concentrate 4	24.4	2.5	0.34				2.8	
Total Flotation Concentrate	**192.1**	**19.6**	**1.05**	0.060			**67.7**	**54.9**
Final Tails	789.1	80.4	0.12	0.012	16.8	0.31	32.3	45.1
Assay Head	***981.2***	***100.0***	***0.30***	***0.021***			***100.0***	***100.0***
Measured Head								

FLOTATION TEST METALLURGICAL BALANCE

Client: Bren-Mar
Test: F12
Sample: Comp #1B

Date: 01-Feb-99
Project: 98-044

Objective: Variability Study

Products	Weight		Assay (%)				Distribution (%)	
	(g)	(%)	Ni	Co	Mg	S^{2-}	Ni	Co
Total Flotation Concentrate	196.9	20.1	1.00	0.052			67.7	62.1
Final Tails	782.7	79.9	0.12	0.008	16.0	1.19	32.3	37.9
Assay Head	**979.6**	**100.0**	**0.30**	**0.017**			**100.0**	**100.0**
Measured Head								

FLOTATION TEST METALLURGICAL BALANCE

Client: Bren-Mar

Date: 01-Feb-99

Test: F13

Project: 98-044

Sample: Comp #2A

bjective: Variability Study

Products	Weight		Assay (%)				Distribution (%)	
	(g)	(%)	Ni	Co	Mg	S^{2-}	Ni	Co
Total Flotation Concentra	214.2	21.8	2.04	0.092			83.1	76.3
Final Tails	767.2	78.2	0.12	0.008	16.6	1.02	16.9	23.7
Assay Head	**981.4**	**100.0**	**0.54**	**0.026**			**100.0**	**100.0**
Measured Head								

FLOTATION TEST METALLURGICAL BALANCE

Client: Bren-Mar **Date:** 01-Feb-99
Test: F14 **Project:** 98-044
Sample: Comp #3A

Objective: Variability Study

Products	Weight		Assay (%)				Distribution (%)		
	(g)	(%)	Ni	Co	Mg	S^{2-}	Ni	Co	Mg
Total Flotation Concentrate	224.3	22.8	2.40	0.092	14.4		79.3	69.3	19.6
Final Tails	761.7	77.2	0.18	0.012	17.4	1.07	20.7	30.7	80.4
Assay Head	**986.0**	**100.0**	**0.69**	**0.030**	**16.7**		**100.0**	**100.0**	**100.0**
Measured Head									

Summary

Eight pressure oxidation tests were done on a sample of Turnagain concentrate in which free acid concentration and solids density were varied. High nickel and cobalt extractions were achieved with low reagent (oxygen) consumption.

Key Results

- >97.0% optimum nickel extraction

- >92% cobalt extraction (probably higher but residue cobalt at or below detection limit).

- 0.16 grams of oxygen consumed per gram of concentrate (low).

- *magnesium extraction was 65%.*

So far the Turnagain concentrate seems well suited to the CESL Process. Some method for method for magnesium rejection will be required.

Background

Turnagain is a nickel-cobalt deposit located in northern British Columbia. The testing program was commissioned by Frank Wright, acting consultant for Bren-Mar Resources.

Sample (3.8 kg) arrived at CESL's Heather Street complex on January 26, 1999 and testing began immediately after.

Objectives

The objective of the test work was to assess the applicability of the CESL Process to the Turnagain concentrate. In particular to determine:

- nickel and cobalt recovery for a single Turnagain concentrate sample,
- oxygen consumption,
- magnesium extraction to solution,
- any unforeseen problems.

Initial Characterization

The concentrate sample shipped to CESL for testing had a wet weight of 4.4 kg (13.7% moisture, 3.8 kg dry). The sample was assayed three times, in-house, as part of the initial characterization process. The average of the three assays are reported in Table 1. For a compilation of all of the assays done on Turnagain, see Appendix A.

Table 1: *Metal Composition of Turnagain Concentrate*

Nickel	6.31%
Magnesium	7.16%
Cobalt	0.31%
Copper	0.30%
Iron	32.3%
Sulphur	20.6%

The concentrate sample was analyzed microscopically by a qualified mineralogist. His analysis together with the assays were used to estimate the mineral concentrations given in Table 2. All original data as well as reconciliation calculations are included in Appendix A. The sulphide minerals in concentrate were mainly pentlandite and pyrrhotite.

Table 2: Mineralogy of Turnagain Concentrate

Pyrrhotite	25.4%
Pentlandite	31.6%
Chalcopyrite	1.00%
Gangue	39.8%

Procedure

The Turnagain test program was restricted to the Pressure Oxidation and Atmospheric Leach unit operations of the CESL Process, as described below.

Pressure Oxidation (PO)

After initial characterisation, the sample was ground in a small rod mill. Concentrate was then used in a series of pressure leaching tests conducted in a 2 L bench autoclave. For each test a sample of wet concentrate, of known dry weight, was charged to the autoclave along with a prepared feed solution. Once the bomb was assembled it was heated to operating temperature by an electrical heating jacket. The retention time lasted from the time that oxygen was injected until the oxygen valve was closed. Once the retention time was spent the autoclave was immediately cooled via pressurised water, fed through internal cooling coils. The product slurry of each test was pressure filtered and the residue was washed with tap water. The filtrates were then assayed for free acid, chloride, copper and iron.

Atmospheric Leach (AL)

Following pressure oxidation, the residue was re-slurried with tap water in an agitated beaker and subjected to an atmospheric leach. A hot plate was used to heat the reaction to 40°C and the pH was maintained at 1.6, for one hour, by the addition of concentrated sulphuric acid. The slurry was then filtered and washed with tap water in a pressure filter. Filtrates were assayed for free acid, chloride, copper and iron. A sample of the residue was dried and assayed for copper, iron, total and elemental sulfur.

Table 3:
Turnagain Pressure Oxidation Tests Summary

	Operating Parameters		Details of Nickel Recovery		Details of Cobalt Recovery		Details of Magnesium Recovery		PO discharge	NET	mass loss (gain)
TEST	FA	solids	extraction	residue	extraction	residue	extraction	residue		O2 ratio	measured
#	g/L	g/L	%	%	%	%	%	%	pH	g/g	%
1405	31	210	91.9%	0.50%	93.0%	0.02%	38.4%	4.19%	4.5	0.21	-5%
1406	40	200	94.3%	0.34%	92.8%	0.02%	60.8%	2.59%	3.2	0.19	-8%
1409	50	200	96.7%	0.22%	0.0%	<0.02	60.4%	2.96%	3.0	0.18	4%
1426	63	200	98.2%	0.11%	0.0%	<0.02	68.5%	2.17%	2.1	0.16	-4%
1427	70	200	97.2%	0.17%	92.9%	0.02%	68.5%	2.11%	2.2	0.16	-7%
1430	71	250	95.6%	0.26%	0.0%	<0.02	60.3%	2.63%	2.8	0.20	-8%
1431	90	293	95.5%	0.26%	0.0%	<0.02	59.9%	2.61%	3.0	0.23	-10%
1433	66	200	96.8%	0.19%					2.2	0.19	-7%

Notes:

1 All tests were run with 200 psi pressure, 130° operating temperature and 60 minute retention time.

2 Sox SO4 is sulphur ox. calculated from a sulphate balance; Sox O2 is calculated from oxygen consumption.

3 Sample compostion: 10 minute grind used for all tests.

pen	cp	pyrr	gangue
31.6%	1.00%	25.4%	39.8%

Cu	0.3%
S	20.8%
Fe	32.9%
Ni	6.41%
Mg	7.16%

GRAVITY CONCENTRATION TEST REPORT

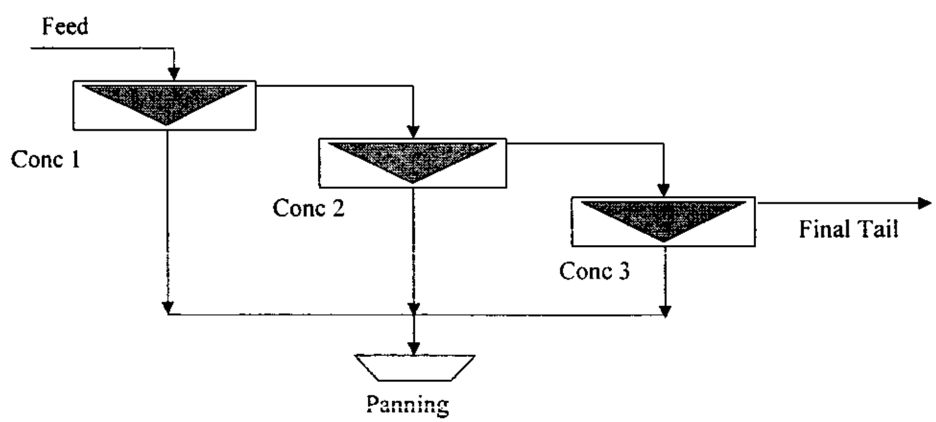
Client: Canadian Metal Exporation
Test: GSB1
Sample: Comp PLR-1

Date: 19-Apr-01
Project: 0102304

Objective: To recover PGM by gravity using Falcon laboratory SB40 concentrator

Products	Weight		Assay						Distribution					
	(g)	(%)	Au (g/t)	Ag (g/t)	Pt (g/t)	Pd (g/t)	Rh (g/t)	Ir (g/t)	Au (%)	Ag (%)	Pt (%)	Pd (%)	Rh (%)	Ir (%)
Pan Concentrate	3.67	0.36	86.4	246	8.32	14.7	<0.01	<0.01	30.9	6.5	5.2	4.6		
Pan Tails	165	16.4	2.36	18.1	1.66	1.84			38.0	21.4	46.4	26.1		
Total SB-4 Concentrate	**169**	**16.7**	**4.19**	**23.1**	**1.81**	**2.12**			**68.9**	**27.8**	**51.6**	**30.7**		
SB-4 Tails	840	83.3	0.38	12.0	0.34	0.96			31.1	72.2	48.4	69.3		
Total Measured	**1,008**	**100.0**	**1.02**	**13.8**	**0.58**	**1.15**			100.0	100.0	100.0	100.0		

3 Pass Test Conditions				
Feed pulp Density	**Pressure**	**Bowl**	**Rotation Speed**	
20%	1.5 psi	14°	60 Hz	154 G



STATEMENT OF EXPENSES

Report writing	
Frank Wright, P.Eng (7 days@ $500/day)	$3,500.00
Analyses	$750.00
Report collation/reproduction	
Bruce Downing, P.Geo. 1/2 day @ $500/day	$250.00
TOTAL	**$4,500.00**

Process Research Associates Ltd.
Metallurgical and Environmental Testing for the Mining Industry
9145 Shaughnessy Street, Vancouver, B.C., Canada V6P 6R9
Phone : (604) 322-0118 Fax : (604) 322-0181 Email : pra@pralab.com

Project no : 0102304

Invoice no : 2190-Adv
Date : 19-Apr-01

ADVANCE

TO : Canadian Metals Explorations
1060-1090 West Georgia Street,
Vancouver, B.C.
V6E 3V7

RE : **PROFESSIONAL SERVICES**
Advance

Advance Payment	$750.00
Sub-total	**$750.00**
G.S.T. (7%)	**$0.00**
	$750.00

FR Wright & Associates Consulting
427 Fairway Dr., North Vancouver, BC, Canada V7G 1L4
Phone:(604) 802-4449 Fax:929-5812 Email: fwright-nvan@msn.com
Mineral Processing / Hydrometallurgy / Environmental

July 9, 2001

Bruce Downing
Via Fax: 940-3233

Dear Bruce:

Re: Turnagain Project-Metallurgical Program

Regarding your call earlier today, the laboratory program conducted earlier this year for Canadian Metals Exploration was performed on mineral samples of Turnagain sulfide concentrate. This was tested at Process Research Associates Ltd., (PRA) for recovery of precious and platinum group metals using gravity recovery methods. Specifically this consisted of testing a Falcon centrifugal concentrator under various process conditions. The cost was approximately $750 and I had earlier requested to PRA that they forward you a copy of their invoice.

Please contact me if you require any further information.

Best Regards
Frank Wright, P.Eng.