**AMC Mining Consultants (Canada) Ltd**

BC0767129

Suite 1330, 200 Granville Street
Vancouver BC V6C 1S4
CANADA

T +1 604 669 0044
F +1 604 669 1120
E amccanada@amcconsultants.ca

# TURNAGAIN PROJECT

## HARD CREEK NICKEL CORPORATION

## PRELIMINARY ECONOMIC ASSESSMENT

**Prepared by AMC Mining Consultants (Canada) Ltd in accordance with the requirements of National Instrument 43-101, Standards for Disclosure of Mineral Projects, of the Canadian Securities Administrators.**

**Qualified Persons:**

A Riles, MAIG, AMC Mining Consultants (Canada) Ltd
M Molavi, PEng, AMC Mining Consultants (Canada) Ltd
R Simpson, PGeo GeoSim Services Inc
B Fong, PEng Moose Mountain Technical Services
John Reid, F.AusIMM, Reid Resource Consulting
Graham McTavish, Valard Construction Ltd
D Friedman, PEng Knight Piésold Ltd.

**Project 711022**

**2nd December 2011**

# 1 SUMMARY

## 1.1 Introduction

This National Instrument 43-101 compliant report has been compiled by AMC Mining Consultants (Canada) Ltd (AMC) for Hard Creek Nickel Corporation (HNC) with input from the following independent consultants:

A Riles, MAIG, AMC Mining Consultants (Canada) Ltd

M Molavi, PEng, AMC Mining Consultants (Canada) Ltd

R Simpson, PGeo GeoSim Services Inc

B Fong, PEng Moose Mountain Technical Services

J Reid, F.AusIMM, Reid Resource Consulting

Graham McTavish, Valard Construction Ltd

D Friedman, PEng Knight Piésold Ltd

The project is based on a 87,000 tpd capacity nickel sulphide flotation plant scheduled to commence production at 50% of full capacity and then expand to the full 87,000 tpd after year 5, with a total mine life of 28 years

## 1.2 Project Location

The project is located in northern British Columbia Canada, 1350 km northwest of Vancouver and 70 km east of the township of Dease Lake. Current access is by paved road to Dease Lake and then light aircraft to site, landing at a coarse gravel strip adjacent to the exploration camp. An existing gravel road provides vehicular access to the site but will need upgrading for project development requirements. Current power supply is by diesel generators.

## 1.3 History

After the initial discovery of nickel and copper sulphides in the Turnagain River in 1956, Falconbridge Nickel Mines Ltd (Falconbridge) acquired the property in 1966 and conducted various geophysical, geochemical and exploratory drilling programs up until 1973. Between 1973 and 1996 minimal exploration work was carried out and what was done focused more on platinum group elements (PGE's).

Bren-Mar Resources Ltd (Bren-Mar) optioned the property in 1996 and conducted further exploration work and some preliminary metallurgical testwork in the period 1996-98, resuming exploration activities after the name change to Canadian Metals Exploration Limited (CME) in 2002.

In 2004 after a change of management, CME became HNC and from thereon several exploration programs were conducted, including: mapping, soil and sediment sampling, geophysical surveys, metallurgical studies, diamond drilling and environmental base line studies. To date a total of 79,351m in 320 holes has been completed.

The first resource estimate for the property was produced in 2003 by N.C.Carter and several since, the most recent by Ron Simpson of Geosim in May 2009. This has been updated for this report.

Three Preliminary Economic Assessments have been prepared for the property, two by AMEC of Americas Ltd (AMEC) in 2006 and 2008 and the third, of which this report is an update, by Wardrop Engineering Inc (Wardrop) in 2010.

## 1.4 Geology and Mineralization

The Turnagain ultramafic Alaskan-type complex comprises a central core of dunite with bounding units of wehrlite, olivine clinopyroxenite, clinopyroxenite, representing crystal cumulate sequences, hornblende clinopyroxenite, and hornblende. No orthopyroxene is present. The complex is elongate and broadly conformable to the northwesterly-trending regional structural grain.

The ultramafic rocks are generally fresh-to-mildly serpentinized; however, more intense serpentinization and talc-carbonate alteration are common along faults and restricted zones within the complex. The central part of the ultramafic body is intruded by granodiorite to diorite, and hornblende–plagioclase porphyry dikes and sills.

The sulphide mineralization, which is unusual for an Alaskan-type deposit, is thought to be associated with meta-sediment wall-rock inclusions which provided the sulphur source. The sulphides are mainly pentlandite and pyrrhotite with minor amounts of chalcopyrite and pyrite, and trace bornite. Anomalous levels of platinum and palladium are also present.

## 1.5 Resource Estimate

Using a cut-off grade of 0.1% Ni, the Turnagain property contains an estimated 865 Mt of Measured and Indicated Resources at 0.21% Ni and 0.013% Co. An additional 976 Mt grading 0.20% Ni and 0.013% Co is classified as inferred. The resource estimate is presented in Table 1.1.

**Table 1.1 Mineral Resource Estimate**

| Resource Category | Tonnes ‘000’s | % Ni (T) | % Co (T) |
|---|---|---|---|
| Measured | 227,379 | 0.22 | 0.014 |
| Indicated | 638,103 | 0.21 | 0.013 |
| Measured & Indicated | 865,482 | 0.21 | 0.013 |
| Inferred | 976,295 | 0.20 | 0.013 |

## 1.6 Metallurgical Testing and Recovery Methods

The testwork of 2007-2010, as reported in the Wardrop PEA, had been unsuccessful in producing a saleable concentrate. This is believed to be due to having worked with aged crushed sample rejects as well as not having fully pursued the use of dispersants. Nevertheless some of the outcomes from the testwork are still relevant:

- The XPS mineralogy work from 2007 identified relationships between % nickel sulphides and %S and MgO/Fe ratios. These were utilized in this study to “cap” recent recovery model predictions and should be further investigated in the next phases of study.
- ACNi assays were found by XPS and Wardrop to be unreliable as a predictor of nickel recovery. However their usefulness for future geometallurgical applications shouldn’t be ignored and further investigations into improving their accuracy and their usefulness should be pursued.
- The grinding testwork and circuit design based on a conventional SABC circuit to treat 87,000 tpd is believed to be reasonable. Given the mineralization is categorized as hard to very hard and the proposed SAG mill is large, HPGR technology should be explored as being a potentially viable alternative. This alternative is subject to satisfactory resolution of any potential chrysotile fibre issues.

The most recent testwork completed since the Wardrop PEA has produced the following key outcomes:

- High grade saleable nickel concentrates (+/-20% Ni) at total nickel recoveries close to 60% appear achievable from the 10-265 hole drilled through the Horsetrail starter pit, using Calgon as a dispersant
- Results from additional testwork on sample 08-264 confirmed the findings for the metallurgical performance of sample 10-265. Although the twinned hole 08-264 iinitially gave inferior results, similar performance to 10-265 was obtained through maximizing rougher mass pulls and improved cleaner selectivity with the dispersant Cyquest 40E and low % solids
- Mineralogical studies largely confirmed the findings of the earlier XPS studies and also indicated the merit in pursuing a split cleaner concept to recover fast-floating liberated pentlandite separately from the more difficult slower floating, but ultimately still recoverable middlings material
- A bulk concentrate production test confirmed the flotation performance on a larger scale.

From the batch flotation test results and locked cycle test data,, some recovery modelling and predictions have been developed in order to:

a) predict recoveries across the full grade range of the mine plan and,

b) provide a mechanism for the pit optimizations to preferentially select higher grade blocks early in the mine life (as well as the softer 106 lithology domain) and improve the project NPV.

A minimum concentrate grade of 18% Ni will be required to meet smelter requirements for %MgO and Fe/MgO ratios. At this grade a L.o.M. nickel recovery of 56.4% is predicted.

Recovery for the first five years is predicted to be 58%, with some of the higher recoveries modelled at the higher feed grades capped to maintain a conservative approach..

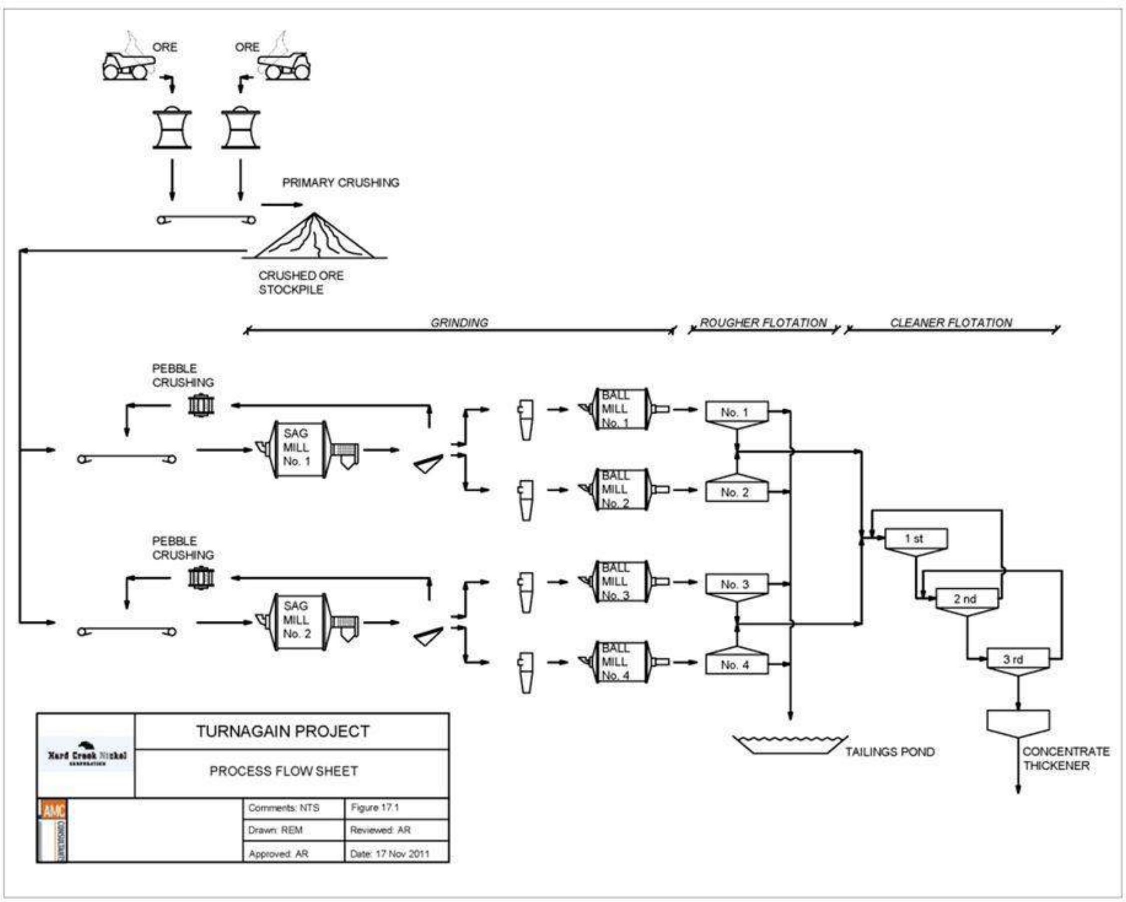
Based on both the mineralogy and the testwork results it is strongly recommended the split cleaner concept be pursued in the next phase of study to optimize concentrate grade whilst still focussing on maximizing recovery of some of the more difficult to float material.

In summary, the process plant will consist of:

- Two trains of primary crushing
- Two grinding trains, comprising a SAG ball mill pebble-crusher (SABC) circuit
- Four banks of rougher flotation, utilising the $500m^3$ tank cells expected to be commercially available in the near future
- A three – stage cleaner circuit and concentrate filtration

The flowsheet is shown in Figure 1.2. This circuit lends itself well to simple modification for the first five years at reduced capacity, reducing to one comminution train and two rougher banks with appropriate modifications to the cleaner circuit.

**Figure 1.1 Simplified Process Flowsheet**



## 1.7 Mining Methods

The Turnagain deposit will be mined using an open pit mining method, employing high volume trucks and shovels. The use of large mining equipment will achieve high mining rates and ensure the lowest possible mine operations unit costs. The waste and mineralized rock will require blasting and typical grade control methods using blast-hole sampling.

For the purpose of this study, the Horsetrail Pit is designed for 28 year life of mine, and includes the Horsetrail and Northwest mineralized zones. Previous evaluations have indicated a potential open pit resource in the Hatzl zone located on the east side of the Turnagain River, but is not included in the scope of this study. The Turnagain River is fish – bearing and considered a wildlife corridor and as such any underlying mineralized material has been excluded as potentially mineable.

The material contained in the Horsetrail Pit is summarized in Table 1.2. This pit forms the basis of the mine plan and production schedule in this study. It is contained within the optimized economic pit shell, a much larger potential open pit resource. The increment between the Horsetrail Pit and the optimized pit is also shown in Table 1.2, but is not included with the production plan in this study.

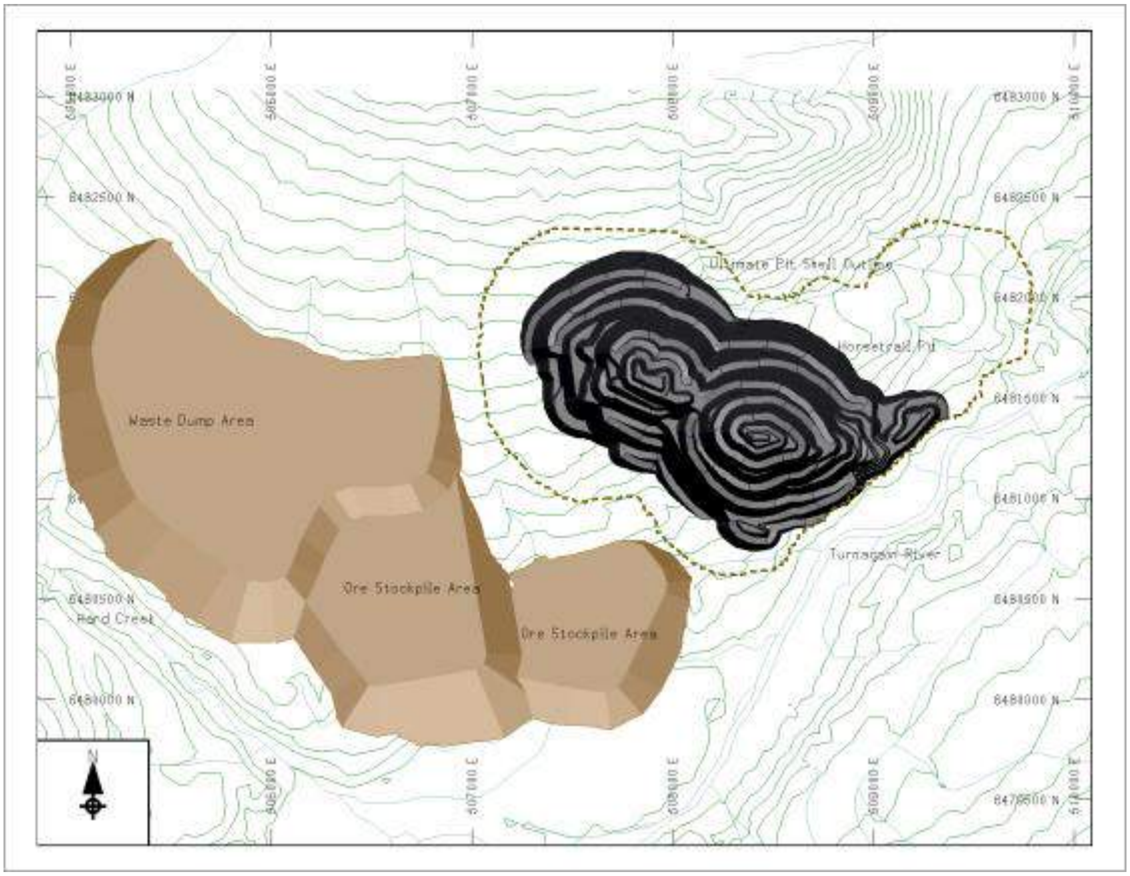
### Table 1.2 Potential In-pit Material

| | Mineralization (kt) | Waste (kt) | Strip Ratio | NSR* ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Horsetrail Pit (28 Yr LOM) | 762,896 | 317,872 | 0.42 | 21.60 | 0.230 | 0.013 | 0.69 |
| Potential Pushback to Ultimate Pit Shell | 499,269 | 641,776 | 1.29 | 18.96 | 0.209 | 0.012 | 0.63 |
| **Total** | **1,262,165** | **959,648** | **0.76** | **20.56** | **0.222** | **0.013** | **0.67** |

* NSR = net smelter return at Base Case metal pricing

Figure 1.2 shows a plan view of the preliminary design for the Horsetrail Pit as well as the optimized or 'Ultimate' pit shell outline.

**Figure 1.2 Horsetrail Pit and Waste Dump / Stockpile Design – Plan View**




The mine will feed the crusher at an average rate of 43,400 t/d during the first five years, and increased to an average of 84,600 t/d thereafter. The resource will be mined for a total of 28 years at these rates.

To access the most economic mineralization in the early years and provide a smooth strip ratio throughout the life of mine, mineralization production from the Horsetrail Pit is scheduled from five mining phases. Phase 1 will commence at centre of the Horsetrail Pit, where the highest mineralization grade and lowest strip ratio will be encountered.

An elevated cut-off grade will be employed in the initial production years to enhance the economics of the project. Mineralization lower than the elevated cut-off grade will be sent to a stockpile near the crusher and either reclaimed at the end of the mine life, or blended with the run-of-mine feed if the opportunity exists. Mineralization that is below the mine cut-off grade, but sufficiently mineralized to cover the cost of milling and handling once it is hauled out of the pit, will also be sent to the mill either directly or through the mineralization stockpile.

Pit waste material will be hauled to a waste dump southwest of the pit adjacent to the mineralization stockpile area. Current geochemistry data suggests that there is insignificant acid generating potential in the waste rock. Further studies will be undertaken to confirm that the waste rock will have minimum long-term environmental impact. Figure 1.1 also shows the conceptual waste dumping and low grade mineralization stockpiling plan.

## 1.8 Infrastructure

The main infrastructure items pertinent to this project are the power supply and waste / tailings management.

### 1.8.1 Power Supply

Since the Wardrop PEA the awarding of the construction contract by BC Hydro for the Northwest Transmission Line (NTL) to Bob Quinn and the upholding of the Federal approvals for the Red Chris project just east of Tatogga Lake have increased the certainty of power supply for the Turnagain project. The Valard study commissioned by HNC evaluated three points of connection, Bob Quinn, Tatogga Lake and Dease Lake, and also 138 kV vs 287 kV transmission voltage. It concluded that a reasonable base case for this report was a 287 kV line from Tatogga Lake following Highway 37 to a switching station just south of Dease Lake and then via the access road to step-down substations at the mine site.

The Valard study also developed the capital and operating costs for this base case as detailed in Section 1.10.

Not included in the Valard study were the BC Hydro inter-connection fees which have been the subject of separate discussions between BC Hydro and HNC and which are also included in the capital cost estimate.

### 1.8.2 Waste / Tailings Management

The major proportion of the waste is expected to be non-reactive and will be stored in conventional sub-aerial dumps adjacent to the open-pit.

From studies to date the relatively small volume of potentially reactive waste is not expected to be acid generating but could be neutral metal leaching. In any case it will be encapsulated in the non-reactive waste which is known to have high neutralization potential and appropriate water collection and control measures implemented as described in the next section.

Various tailings storage options were studied in the option studies carried out in 2006 and 2007 and reported in the Wardrop PEA. The preferred option of Flat Creek has been retained for this study, as it offers good storage efficiency in terms of the ratio of capacity to embankment volume and also has a small catchment area.

The tailings management facility construction will commence with an initial starter dam to provide two years storage at the initial production rate of 43,500 tpd and the will be raised in five stages of centerline raise construction to handle the increase in year 6 to 87,000 tpd and the ultimate storage of 757 Mt of tailings over the 28 year mine life.

## 1.9 Environmental Considerations

There are four main components to the project environmental management, as detailed below:

- The Environmental Impact Assessment process, because of the scale of the project, will require a comprehensive study and harmonized review under both the Britsih Columbia and Canadian Environmental Assessments Acts (BCEAA and CEAA respectively). The comprehensive review will involve public and First Nations consultation as well as detailed studies of baseline environmental settings and potential project impacts.
- Accordingly, baseline environmental studies were initiated in 2004 and are ongoing. Additional studies will include air quality, background noise levels and flora and fauna with special emphasis on aquatic habitat and aquatic life particularly in Flat Creek.
- Water and waste management measures will be directed towards achieving the following key objectives:
    - Adequate storage and containment in the TMF of process tailings, process water and storm run-off
    - Interception and diversion of clean waters to the extent possible
    - Collection and control of mine-affected waters including appropriate waste dump and low grade stockpile design with collection ponds and re-use and/or treatment of these run-off waters
    - Optimization of the storage and usage of water over the entire site with regard to environmental, operational and economic criteria
- Finally the reclamation and closure plan will minimize any adverse environmental and social impacts associated with the mine development, and seek to return disturbed site areas to conditions consistent with an approved end-use plan. Preliminary closure planning will be carried out concurrently with the various stages of project development and design in order to integrate the post-closure objectives into the design, construction, and operation of all mine infrastructure and facilities. The closure and reclamation plan will be developed in consultation with the HNC project team, local stakeholders, and the appropriate regulatory authorities.

## 1.10 Capital and Operating Costs

Project capital costs are summarized in Table 1.3. The estimate relied on the significant level of detail in the previous study but updated major equipment quantities, sizing and pricing.

The initial capital for the first five years at 50% capacity was estimated for the mine from the mine schedule and resulting fleet requirements, and for the processing plant the major equipment item quantities and scaling factors where appropriate.

Included in the processing capital are the power supply costs estimated by Valard and with an allowance for the inter-connection fee likely to be applied by BC Hydro.

### Table 1.3 Project Capital Cost Summary

| CAPEX Summary | Initial Capital | Year 5 Expansion | Total LoM Capital |
|---|---|---|---|
| | US$M | US$M | US$M |
| Mine | 244,055 | 68,174 | 406,054 |
| Processing | 986,474 | 405,717 | 1,392,190 |
| Other and sustaining | 94,502 | 17,924 | 477,467 |
| Working Capital | 32,189 | | 32,189 |
| **Totals** | **1,357,220** | **491,815** | **2,307,901** |

Working Capital is assumed to be 25% of yr 1 costs, i.e. equivalent to financing the first three months of operations

Operating unit costs are summarized in Table 1.4.

### Table 1.4 Unit Operating Cost Summary

| | L.o.M. | Yrs 1-5 | Yrs 6-21 |
|---|---|---|---|
| Operating Cost US$/T milled: | 7.30 | 8.37 | 7.78 |
| - mining | 2.52 | 3.11 | 3.11 |
| - processing (incl TMF) | 4.44 | 4.69 | 4.38 |
| - G&A | 0.33 | 0.57 | 0.29 |

## 1.11 Economic Evaluation

The base case for the purposes of the economic analysis was the production schedule developed in Section 16 with the first five years at approximately 50% throughput.

The core model was developed as a pre-tax model; however an after-tax model was also prepared with professional tax expertise input.

The key project outputs are summarized in Table 1.5.

### Table 1.5 Key Project Outputs

| Key Outputs | | L.o.M. | | Yrs 1-5 | Yrs 6-21 |
|---|---|---|---|---|---|
| | | Pre-tax | After tax | | |
| **Financial** | NPV (8%) (US$M) | 1295 | 724 | | |
| | IRR % | 15.9 | 13.5 | | |
| | Payback period yrs | 7.3 | | | |
| | Smelter % netback | 72.4 | | | |
| | NSR delivered $/T | 18.5 | | 21.6 | 20.2 |
| | Average operating cash flow (US$M) | 316 | | 208 | 387 |
| **Physicals** | Feed Grade | | | | |
| | - Ni | 0.23 | | 0.26 | 0.25 |
| | -Co | 0.013 | | 0.014 | 0.013 |
| | Average annual throughput Mtpa | 28.1 | | 15.8 | 31.3 |
| | Strip ratio | 0.82 | | 0.74 | 0.83 |
| | Recoveries %: | | | | |
| | -Ni | 56.4 | | 58.0 | 57.7 |
| | -Co | 56.4 | | 58.0 | 57.7 |
| | Average Annual Metal Production: | | | | |
| | -Ni (lbs x 1000) | | | 52717 | 97871 |
| | -Co (lbs x 1000) | | | 2822 | 5363 |
| | DMT Concentrate | 2032101 | | 132846 | 246633 |
| **Costs** | Operating Cost US$/T milled: | 7.30 | | 8.37 | 7.78 |
| | -mining | 2.52 | | 3.11 | 3.11 |
| | -processing (incl TMF) | 4.44 | | 4.69 | 4.38 |
| | -G&A | 0.33 | | 0.57 | 0.29 |
| | C1 cash cost $/lb payable Ni (after Co credits) | 4.26 | | 4.23 | 4.20 |

The key inputs in the above table are a nickel price of $8.50/lb, cobalt $14/lb and the C$:US$ exchange rate of 0.95.

## 1.12 Conclusions and Recommendations

The key elements in this update are the recent metallurgical work towards production of a saleable concentrate at acceptable recoveries and the increased certainty of power supply to northern BC.

Based on re-validated resource estimate and the metallurgical recovery models resulting from the testwork, a production schedule base case has been developed with an elevated cut-off grade strategy and a phased approach to capacity to deliver a 28 year mine life. The processing route is a conventional comminution and flotation plant.

Opportunities exist to prove up additional resources, including those containing anomalous levels of platinum and palladium, and to further enhance the geometallurgical knowledge base and metallurgical efficiencies, although a con-commitant risk is that the geometallurgical variability may prove greater than expected. There is also an opportunity for full mine-to-product (including tailings) project optimization as better information becomes available in the next phase of study.

This Preliminary Economic Assessment has shown that the Turnagain property is a potentially viable project at the base case parameters and on the estimated current NI 43-101 compliant resource. AMC recommends therefore that HNC carry it forward to the preliminary feasibility stage, in accordance with the budget presented in Table 1.6.

**Table 1.6 Preliminary Feasibility Study Budget**

| **Item** | **US$ '000** |
|---|---|
| Reserve Drilling | 1560 |
| Geotechnical Drilling | 360 |
| Analyses | 620 |
| Transportation Support | 1020 |
| Environmental | 125 |
| Special Engineering Studies | 400 |
| On-site Consultants | 125 |
| Metallurgy and Geometallurgy | 650 |
| Contract Salaries | 160 |
| Camp Costs | 490 |
| PFS Engineering | 500 |
| Contingency | 902 |
| **Total** | **6912** |

# CONTENTS

## TABLES

## FIGURES

## APPENDICES

**Distribution list:**

| | |
|---|---|
| 3 copies to | Mr xxx, xxx site |
| 2 copies to | Ms xxx, xxx site |
| 1 copy to | AMC xxx office |

## 2 INTRODUCTION

This NI 43-101-compliant report has been prepared by AMC Mining Consultants (Canada) Ltd for Hard Creek Nickel Corporation (HNC) as an update of the previous Preliminary Economic Assessment prepared by Wardrop in April 2010. The principal reason for the update was recent metallurgical testwork, detailed in this report had occasioned significant changes to the scope of the project, namely being able to produce saleable nickel concentrates at acceptable recoveries removed the need for a hydrometallurgical plant.

The report has inputs from the following independent consultants:

- Moose Mountain Technical Services (MMTS)
- GeoSim Services Inc. (GeoSim)
- Knight Piésold Ltd. (KP)
- Valard Construction Ltd (Valard)
- Reid Resource Consulting Pty Ltd. (Reid).

Mr. Ron Simpson (P.Geo.) of Geosim visited the site on 3 to 5 October 2005, 11 to 12 October 2006, and 16 June 2009. Mr. Bob Fong (P.Eng.) of MMTS visited the site on 9 September 2008, and 16 June 2009. Mr. Daniel Friedman (P.Eng.) of KP visited the site on 7 September 2005, and 16 June 2009. In addition, Alan Riles and Mo Molavi of AMC Consultants visited the site on 9 to 10 August 2011.

A summary of the qualified persons (QPs) responsible for each section of this report is detailed in Table 2.1. All the QP's listed are independent of Hard Creek Nickel Corporation.

**Table 2.1 Persons who Prepared or Contributed to this Technical Report**

| Qualified Person | Employer | Date of Site Visit | Professional Designation | Sections of Report |
|---|---|---|---|---|
| Daniel Friedman | Knight Piésold Ltd. | Sep 7, 2005, and Jun 16, 2009 | P.Eng. | 5, 20, parts of 18, 21 |
| Robert Fong | Moose Mountain Technical Services | Sept 9 2008 and June 16 2009 | P.Eng | 16 |
| Ron Simpson | GeoSim Services Inc. | Oct 3-5 2005, Oct 11-16 Oct 2006 and June 16 2009 | P.Geo | 4,6-12,14 |
| Alan Riles | AMC Consultants Ltd Canada | Aug 9-10 2011 | MAIG | 1, 2, 3, 13, 17, 21-27 |
| Mo Molavi | AMC Consultants Ltd Canada | Aug 9-10 2011 | P.Eng | Parts of 18 |
| John Reid | Reid Resource Consulting Pty Ltd | | | 19 |
| Graham McTavish | Valard Construction | | P.Eng. | 18.7 |

# 3 RELIANCE ON OTHER EXPERTS

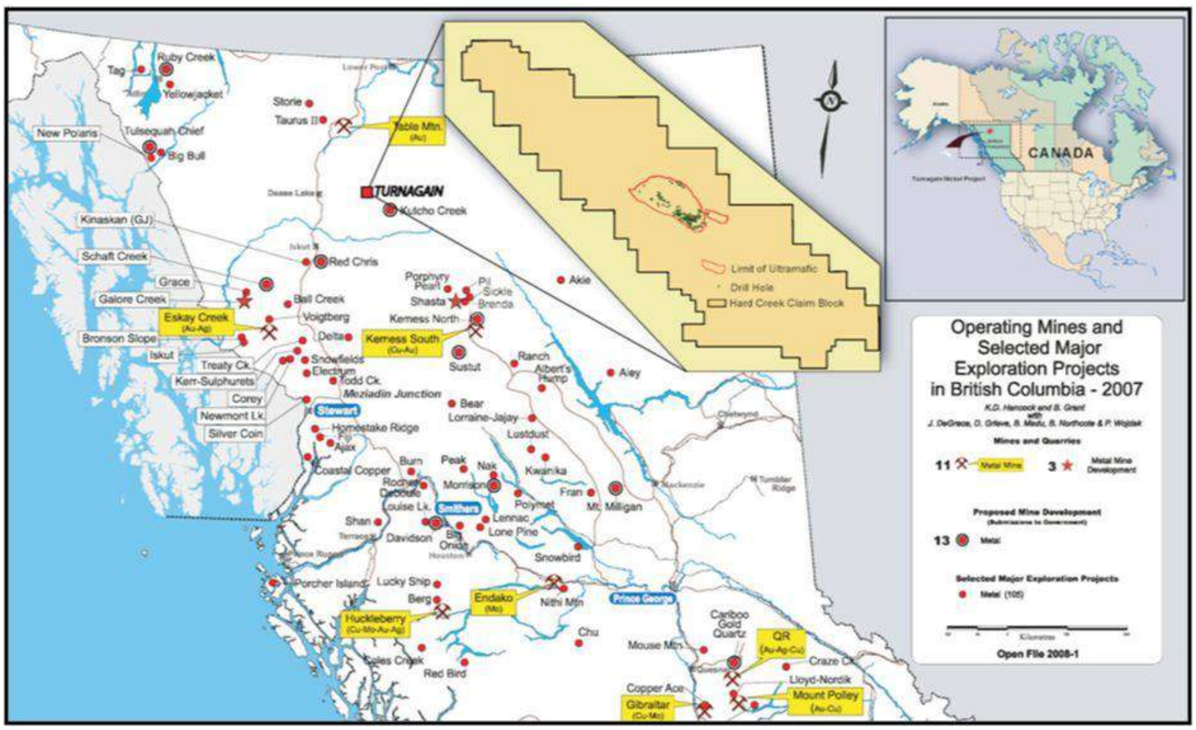
Not applicable.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Location

The Turnagain Project is located in BC, Canada, about 1,350 km north-northwest of Vancouver and 70 km east of the township of Dease Lake (Figure 4.1). The Project centre is located at approximately 58°30'N latitude and 128°45'E longitude, or UTM NAD83 Zone 9 coordinates 6,481,000 N and 508,000 E. Elevations within the Project area range between 1000 and 1800 masl.

**Figure 4.1 Location Map**




## 4.2 Mineral Rights

Mineral claims located in 1996 by J. Schussler and E. Hatzl were subsequently optioned to Bren-Mar Resources Ltd. (Bren-Mar), a predecessor company of Canadian Metals Exploration Ltd. (CME) and HNC. The original option agreement gave Bren-Mar the right to earn a 100% interest in the mineral claims in exchange for the issuance of 200,000 shares and incurring property expenditures of C$1 million within five years of acquisition. The 100% interest has been earned subject to a 4% Net Smelter Royalty (NSR) on possible future production from the mineral claim 511330. HNC retains the right to purchase all or part of this royalty for C$1 million for each 1% of the royalty.

On 28 November 2002, HNC entered into an agreement with Schussler and Hatzl to acquire an additional 34 mineral claims, adjacent to the Turnagain property, Liard Mining Division, BC, in exchange for an aggregate total of 100,000 common shares.

Between November 2003 and March 2005, additional claims were staked, enlarging the Turnagain property from 3,700 ha to approximately 27,500 ha.

The “Drift” cell mineral claims, situated northwest of Hard Lake, and the “Dinah” cell minerals claims, situated southeast of Turnagain River, were acquired early in 2005 by way of the BC Ministry of Energy and Mines online map selection process.

The 8 “Flat” cell mineral claims, located in the Flat and Blick Creek drainages, were acquired in mid-2009 by way of the BC Ministry of Energy and Mines online map selection process.

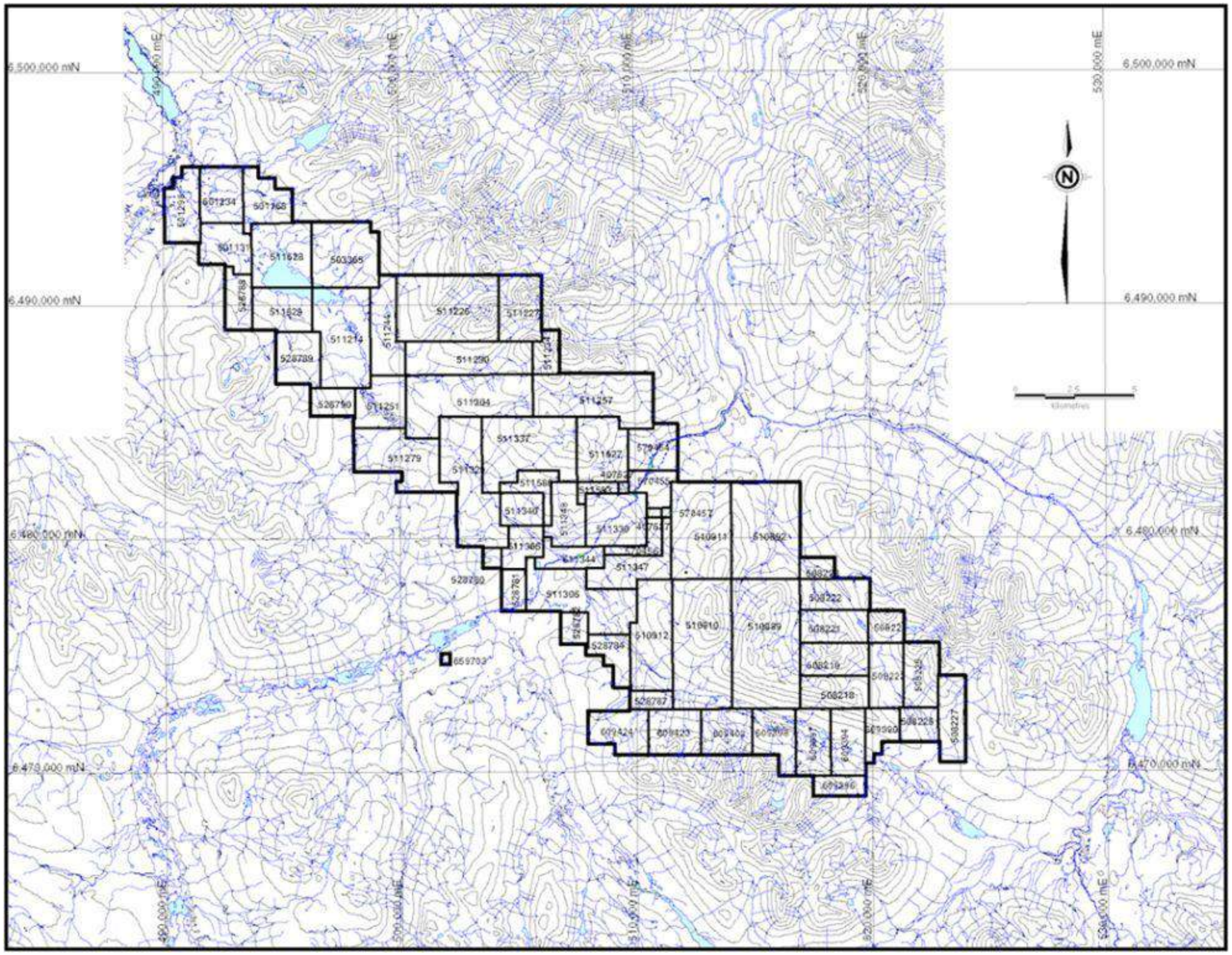
The configuration of the various mineral claims is illustrated in Figure 4.2, which incorporates information plotted on BC Mineral Titles Reference Maps M104I 045, 046, 055, and 056. Details are listed in Table 4.1.

### Table 4.1 Turnagain Mineral Claims

| Tenure Number | Claim Name | Original Legacy Names | Tenure Type | Good To Date | Area (ha) |
|---|---|---|---|---|---|
| 407627 | PUP 4 | | Mineral | 2019/jan/01 | 500 |
| 501131 | Drift 1 | | Mineral | 2019/jan/12 | 422 |
| 501168 | Drift 2 | | Mineral | 2019/jan/12 | 422 |
| 501234 | Drift 3 | | Mineral | 2019/jan/12 | 422 |
| 501298 | Drift 4 | | Mineral | 2019/jan/12 | 422 |
| 503365 | | Hard 2 | Mineral | 2019/feb/18 | 793 |
| 508218 | Dinah 1 | | Mineral | 2019/mar/03 | 407 |
| 508219 | Dinah 2 | | Mineral | 2019/mar/03 | 407 |
| 508221 | Dinah 3 | | Mineral | 2019/mar/03 | 407 |
| 508222 | Dinah 4 | | Mineral | 2019/mar/03 | 407 |
| 508223 | Dinah 5 | | Mineral | 2019/mar/03 | 407 |
| 508225 | Dinah 6 | | Mineral | 2019/mar/03 | 407 |
| 508226 | Dinah 7 | | Mineral | 2019/mar/03 | 255 |
| 508227 | Dinah 8 | | Mineral | 2019/mar/03 | 407 |
| 508228 | Dinah 9 | | Mineral | 2019/mar/03 | 136 |
| 508229 | Dinah 10 | | Mineral | 2019/mar/03 | 203 |
| 510889 | | Flat 10, 13, 15 | Mineral | 2019/apr/07 | 1628 |
| 510892 | | Flat 2, 6 | Mineral | 2019/apr/07 | 1219 |
| 510910 | | Flat 9, 12, 14 | Mineral | 2019/apr/07 | 1424 |
| 510911 | | Flat 1, 5 | Mineral | 2019/apr/07 | 1067 |
| 510912 | | Flat 8, 11 | Mineral | 2019/apr/07 | 780 |
| 511214 | | Hard 4, 6 | Mineral | 2019/feb/18 | 980 |
| 511226 | | Hill 1, 2 | Mineral | 2019/feb/18 | 1216 |
| 511227 | | Hill 3 | Mineral | 2019/feb/17 | 507 |
| 511230 | | Hill 4, 5 | Mineral | 2019/feb/17 | 760 |
| 511234 | | Hill 6 | Mineral | 2019/feb/16 | 186 |
| 511244 | | Hard 5, 7 | Mineral | 2019/feb/18 | 490 |
| 511251 | | Hard 8 | Mineral | 2019/feb/17 | 473 |
| 511257 | | Hill 9, 10 | Mineral | 2019/feb/17 | 1014 |

| Tenure Number | Claim Name | Original Legacy Names | Tenure Type | Good To Date | Area (ha) |
|---|---|---|---|---|---|
| 511279 | | Hard 9, 10 | Mineral | 2019/feb/17 | 897 |
| 511304 | | Hill 7, 8 | Mineral | 2019/feb/17 | 1150 |
| 511305 | | Hound 3 | Mineral | 2019/sep/27 | 271 |
| 511306 | | Turn 2, Flat 7 | Mineral | 2019/feb/19 | 881 |
| 511329 | | Hound 1, 2 | Mineral | 2019/sep/27 | 1015 |
| 511330 | | Cub | Mineral | 2018/dec/01 | 593 |
| 511337 | | Cub 10, 18, Pup 1 | Mineral | 2018/dec/01 | 1066 |
| 511340 | | Cub 17 | Mineral | 2018/dec/01 | 254 |
| 511344 | | Turn 1, Bear 2 | Mineral | 2019/feb/19 | 271 |
| 511347 | | Flat 3, 4 | Mineral | 2019/apr/07 | 474 |
| 511348 | | Cub 2 | Mineral | 2018/dec/01 | 389 |
| 511586 | | Pup 2 | Mineral | 2019/jan/01 | 237 |
| 511593 | | Pup 3 | Mineral | 2019/jan/01 | 102 |
| 511627 | | Cub 11 | Mineral | 2018/dec/01 | 592 |
| 511628 | | Hard 1 | Mineral | 2019/feb/18 | 709 |
| 511629 | | Hard 3 | Mineral | 2019/feb/18 | 473 |
| 528780 | T1 | | Mineral | 2019/feb/23 | 68 |
| 528781 | T2 | | Mineral | 2019/feb/23 | 203 |
| 528782 | T3 | | Mineral | 2019/feb/23 | 153 |
| 528784 | T4 | | Mineral | 2019/feb/23 | 288 |
| 528787 | T5 | | Mineral | 2019/feb/23 | 170 |
| 528788 | T6 | | Mineral | 2019/feb/23 | 270 |
| 528789 | T7 | | Mineral | 2019/feb/23 | 422 |
| 528790 | T8 | | Mineral | 2019/feb/23 | 254 |
| 570454 | | Bear 1 | Mineral | 2019/may/26 | 457 |
| 570455 | | Bear 19, Bear 21 to 28 | Mineral | 2019/may/26 | 237 |
| 570456 | | Bear 3 to 18 | Mineral | 2019/may/26 | 220 |
| 570457 | | Bear 20 | Mineral | 2019/may/26 | 17 |
| 609390 | FLAT 7 | | Mineral | 2018/sep/20 | 255 |
| 609394 | FLAT 6 | | Mineral | 2018/sep/20 | 407 |
| 609396 | FLAT 8 | | Mineral | 2018/sep/20 | 204 |
| 609397 | FLAT 5 | | Mineral | 2018/sep/20 | 407 |
| 609398 | FLAT 4 | | Mineral | 2018/sep/20 | 407 |
| 609403 | FLAT 3 | | Mineral | 2018/sep/20 | 407 |
| 609423 | FLAT 2 | | Mineral | 2018/sep/20 | 407 |
| 609424 | FLAT 1 | | Mineral | 2018sep/20 | 424 |

## Figure 4.2 Claim Location Map



Twenty-nine of the original four-post mineral claims (now termed legacy claims), northwest of the Turnagain River, were converted to cell mineral claims in April 2006. This conversion process ensured greater security of mineral title by effectively eliminating the possibility of internal and external fractions within or adjacent to the various mineral claims. Accumulated assessment work credits were also retained under the conversion system.

One 4-post claim and twenty-seven 2-post claims, located adjacent to and partially within the central part of the property holdings but outside of the prospective ultramafic rocks, were the subject of a legal dispute between HNC and Mr. Weise. On 10 July 2006, the Supreme Court of British Columbia ordered that these claims be transferred to HNC. The transfer has been completed and the claims have been included in the Turnagain property. Mr. Weise subsequently filed a Notice of Appeal of the Order; the appeal was dismissed by the British Columbia Court of Appeal on 30 April 2007.

Six placer claims were acquired by the online staking method in the Turnagain River, Hard Creek and Flat Creek drainages to reduce the opportunity for placer prospectors and miners to create surface disturbances within the area of Hard Creek's underlying mineral tenures. These placer claims are in good standing until the first half of 2012.

## 4.3 Permits and Environmental Liabilities

Exploration work on mineral properties in BC requires the filing of a Notice of Work and Reclamation with the Ministry of Energy and Mines. The issuance of a permit facilitating such work may involve the posting of a reclamation bond.

Permits for the 2003 to 2008 exploration work programs were obtained with no undue delays. Reclamation bond securities to the value of C$187,900 were held against the 2007 work program and this also covered all surface disturbances in 2008 and 2009. The work program for 2007, 2008, and 2009 were all granted under license MX-1-505. Surface disturbances were 6.78 ha and 1.42 ha for 2007 and 2008, respectively. Reclaimed areas were 2.81 ha and 0.11 ha. An Annual Summary of Exploration Work was filed with the Mines Inspector for each year.

In March 2011, the Mines branch of the Ministry of Natural Resources issued an amended permit with a multi-year term ending on 31 March 2014. An additional C$150,000 was added to the existing reclamation bond which increases it to C$337,900.

Environmental studies within the property area have been ongoing since 2003. These studies include hydrological measurements on tributary creeks, water quality sampling from creeks and drill holes, wildlife observations and determination of fish species, and the collection of meteorological site data. Multi-element analyses of soil samples have provided useful information regarding background concentrations of major and trace elements. The meteorological station was moved and upgraded in 2009.

Discussions with First Nations including Tahltan and Kaska Dena, and stakeholders that may be impacted by any proposed mining operation are at an early to intermediate stage. Such discussions will be necessary as the Project is advanced, in order to assess any socioeconomic impacts.

AMC is not aware of any specific environmental liabilities to which the various mineral claims are subject. The Turnagain property is situated in an area where mining-related activities have been underway for more than 75 years.

## 4.4 Royalties

A 4% NSR on possible future production from one mineral claim (TENURE No. 511330) is held by the original property vendors J. Schussler and E. Hatzl. HNC retains the right to purchase all or part of this royalty for C$1 million per each 1% of the royalty.

HNC intends to purchase the royalty prior to the start of production. As such, this has not been included in the financial model, nor in the Taxation and Royalties section.

# 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

## 5.1 Accessibility

The nearest airport to the project is at Dease Lake, 70 km to the west of the project. Dease Lake has scheduled airline service by Northern Thunderbird Air. Flight frequency generally depends on the prevailing demand and economic conditions.

A 900m coarse gravel airstrip immediately adjacent to the HNC exploration camp, constructed by Falconbridge Nickel Mines Ltd. (Falconbridge) in 1967, was upgraded by HNC in 2007.

Secondary roads extending easterly from Dease Lake have been used by large, articulated 4-wheel drive vehicles to convey equipment to the Turnagain property. This road is not suitable for regular vehicle traffic.

## 5.2 Climate

The climate of the area is generally characterized by cold winters, warm summers, and reasonably consistent precipitation throughout the year, although the summer months are the wettest. Annual flow patterns are typically characterized by a very pronounced period of high flows in the spring due to snowmelt and rainfall, followed by declining flows through the summer and fall, and low flows throughout the winter.

### 5.2.1 Meteorology

A Campbell Scientific automated weather station was installed at the east end of the site airstrip on 11 August 2004; data is available from that date until 9 September 2009, after which the station was relocated. The automated weather station records average hourly and daily wind direction, wind speed, temperature, precipitation, and relative humidity. The station has also been modified to include solar radiation and total precipitation since it was relocated. Almost one year of temperature and relative humidity data was lost from 20 June 2005, to 24 August 2006, due to damage to the station caused by a moose.

The temperature at the project site is on average 2°C cooler than Dease Lake, which is to be expected given the higher elevation of the project site station at 1020m compared to Dease Lake at elevation 802m. The estimated mean annual temperature for the period of record is -2.0°C and the mean monthly temperatures range from a high of 11.1°C in July to a low of -18.0°C in January.

Concurrent months of precipitation from the regional station at Dease Lake were compared with the project site data to estimate the long-term mean precipitation. The average ratio of the project site to Dease Lake values for all concurrent months of record is 1.5, indicating that the project site receives on average 50% more precipitation than Dease Lake, which has a long-term mean annual precipitation of 381 mm. Therefore, the long-term average precipitation for the project area at elevation 1020m is estimated to be 571 mm.

The mean monthly wind speeds range from 1.67 m/s in March to 0.73 m/s in July. The overall mean wind speed for the period of record is 1.08 m/s with a maximum hourly wind speed record of 7.15 m/s documented on 27 September 2004. The monthly wind direction data for the project site indicate that the predominant wind direction is from the southwest.

The mean monthly relative humidity ranges from 84.1% in October to 50.5% in July, and the overall mean annual relative humidity is 70.6%.

### 5.2.2 Hydrology

Six stream gauging stations have been installed within the project area. Automated dataloggers have been collecting data at these stations for the following periods:

- Lower Hard Creek – August 2004 to present
- Falkner Creek – August 2006 to present
- Flat Creek – August 2006 to present
- Upper Hard Creek – October 2006 to 20 October 2009 (site removed)
- Turnagain River – 11 September 2008, to present
- Furthest Hard Creek – 11 September 2008, to present

Dataloggers record water level elevations at 15 minute intervals. The most complete and continuous datasets are for the Falkner Creek and Flat Creek stations, where almost five complete years of data have been collected. Unfortunately, periods of data were lost at both the Upper and Lower Hard Creek sites due to various instrument failures.

Estimates of long-term average monthly and annual unit runoff for basins in the project area were generated by correlating short-term site data with long-term regional records. The results indicate a mean annual unit runoff of approximately 16 L/s/km2 and monthly values ranging from a low of 2.6 L/s/km$^2$ in March to a high of 54.1 L/s/km$^2$ in June. It is apparent from the estimated mean distribution that nearly 50% of the total annual flow occurs during the months of June and July, and that approximately 90% of annual flows occur between the non-freezing months of May and October.

## 5.3 Local resources

An exploration camp built on the property in April 2003 is capable of accommodating approximately 35 people, and consists of 17 walled tents, 3 trailers, and drill core storage facilities. Power is provided by an on-site diesel generator and a back-up generator.

On-site communications include satellite telephone, facsimile, and internet connections.

There are approximately 32 km of unpaved roads and trails on the property, constructed from the late 1960s to the present.

## 5.4 Infrastructure

Dease Lake (population 650) offers some supplies and services. The communities of Terrace (population 12,000) and Smithers (population 5,500), 700 and 500 km to the south respectively, offer the best range of supplies and services which can be trucked to Dease Lake via Highway 37. The closest deep water port is the bulk terminal at Stewart. There is no rail link within the Cassiar district, although there is a rail bed between Dease Lake and

Takla Landing to the south. The closest railhead for the Canadian National Railway is located at Kitwanga, approximately 485 km south of Dease Lake.

At present, the Cassiar district is not serviced by the provincial electricity grid. The 3 MW Hluey Lakes Hydro Project, supplemented by diesel generators, produces electricity for Dease Lake.

## 5.5 Physiography

The data on physiography of the Stikine region is taken from the Integrated Land Management Bureau (2007).

Between Dease Lake and the property, topography comprises mountains and wide river valleys of the Stikine Ranges. Ridges, plateaus, and summits lower than 1800m are rounded while higher summits are rugged. Valley bottoms are 1000 to 1350m elevation while the highest peak (King Mountain) is about 15 km south of the Turnagain property at 2425m elevation. Plateau surfaces are at about 1500m.

The valley bottoms and lower elevation slopes are covered with glacial drift. Esker and drumlin formations are numerous and extensive. The ranges are characterized by the occurrence of flat-topped tuyas, which are steep-sided volcanoes that erupted on the plateau surface under the ice sheet during the Pleistocene glaciations.

Boreal white spruce and lodgepole pine forest occur on valley bottoms, where they are interspersed with wetlands. At higher elevations, the boreal forest gives way to sub-alpine fir and scrub birch in open forests and woodlands. In areas of cold-air ponding and in upper elevation exposed areas, the forest gives way to sub-alpine shrub and grassland and scrub vegetation. Alpine shrub-land, heath, and tundra occur above the tree line. Bedrock is reasonably well exposed in the areas above the tree line and along drainage divides.

Several species of large mammal including grizzly bear, black bear, wolf, moose, caribou, mountain goat, and sheep can be found in the Cassiar Mountains. Bird species noted in the mountains include gyrfalcon, golden eagle, willow ptarmigan, least sandpiper, red-necked phalarope, snow bunting, and Smith's longspur.

The Turnagain Project straddles the Turnagain River near its confluence with Hard Creek. The project area covers north, west, and east-facing slopes northwest and southeast of the Turnagain River and alpine terrain above the tree line. Elevations range from about 1000 masl along the Turnagain River in the central claims area to 1800m at an unnamed summit in the central property area.

## 6 HISTORY

The description of the property exploration history is based on work by Nixon (1998) and Baldys et al., (2006).

Nickel and copper sulphides were first recognized in rusty weathering exposures at the Discovery zone on the Turnagain River in about 1956. Falconbridge Nickel Mines Ltd. (Falconbridge) acquired the property in 1966 and, during the period 1966–1973, completed an airborne geophysical survey, ground geophysical surveys, geological mapping, geochemical surveys, and 28 wide-spaced diamond holes (2,895m). The work identified a number of sulphide "showings". The exploration program tested many of the mineralized outcrops by "packsack" drilling; the Discovery outcrop was not successfully drilled.

During the early 1970s, adjacent claims were investigated with a geochemical survey by Union Miniere Exploration and Mining Corporation Ltd. (UMEX). Once the Falconbridge and UMEX claims expired, a number of the showings were re-staked and tested with short, small diameter core holes by an unnamed party. Three EX-sized core holes, totalling 55.5m, were drilled on the west bank of the Turnagain River in 1977. No significant intersections were reported and the collars have not been located. In 1979, a single drillhole (17m) was drilled by S. Bridcut near the east bank of the Turnagain River and intersected unmineralized quartz diorite.

The commodity focus for exploration shifted to platinum group elements (PGEs) in the mid-1980s. A geochemical survey for PGEs was conducted for Equinox Resources Ltd. in 1986, and Bridcut re-sampled the Falconbridge core in 1988.

In 1996, Bren-Mar optioned the Cub claims from Schussler and Hatzl. From 1996 to 1998, Bren-Mar completed an airborne magnetic survey over 45 $km^2$ (400 line-km of survey), 19 diamond drillholes (3,889m), geological prospecting and sampling, down-hole pulse electromagnetic surveys in 4 of the 1997-1998 drillholes, and preliminary metallurgical testwork on drill core composite samples.

Bren-Mar changed its name to Canadian Metals Exploration Limited (CME), and resumed exploration in 2002 with an induced polarization (IP) and ground magnetic survey followed by 1,687m of diamond drilling in 7 holes. Drilling continued in 2003, with 23 holes (including deepening of one of the 2002 drillholes) completed for 8,769m. Additional exploration included geological mapping and prospecting as well as bedrock, stream sediment, and soil sampling.

In 2004, CME changed its name to Hard Creek Nickel Corporation and recommenced work on the property.

Up to the end of 2007, HNC had completed the following:

- geological mapping
- bedrock, stream sediment, and soil sampling
- surface, borehole, and airborne geophysical surveys
- mineralogical, metallurgical, and analytical studies

- 172 diamond drillholes for 41,502m of drilling

In 2006 HNC reported a Measured and Indicated Resource estimate inside a 0.2% sulphide nickel grade shell. Only the sulphide minerals were considered recoverable into a saleable product and therefore the 2006 resource estimate was reported in terms of sulphide nickel. Sulphide nickel was determined using ammonium-citrate hydrogen peroxide partial extraction procedure. The estimate was completed by Geosim of Vancouver (Simpson, 2006).

Later in 2006, HNC reported results of the first Preliminary Assessment (PA) on the Project. A key assumption of the PA was that a 0.10% sulphide nickel analysis cut-off was economically reasonable for the Project. This cut-off was determined as a result of parameters selected for pit optimization. Resources in the PA were reported in terms of sulphide and total nickel.

In 2007, HNC reported a new Measured and Indicated Resource estimate in terms of sulphide and total nickel inside a 0.10% sulphide nickel grade shell. This estimate was completed by Geosim and resulted in a significant increase in the tonnes of the deposit (Simpson, 2007).

Resource estimates reported in 2006 and March 2007 were constrained using sulphide nickel grade shells. The restriction on grade shells was appropriate given that no geological domains had been defined at that time.

In January 2008, AMEC Americas Ltd. (AMEC) completed a second PA, which included an updated resource estimate constrained by lithologic domains based on the nearest-neighbour interpolation of geology from drill logs. At the time the resource estimate was carried out, complete results from the 2007 drill program were not available.

In June 2008, AMEC released an interim resource estimate that included results of all 2007 drillholes.

In 2008, HNC completed an additional 16 core holes totalling 4,105m.

In April 2010 Wardrop released a further PEA, based on an updated resource with the 70 holes (21,098.9m) drilled in late 2007 and 2008 included and with additional metallurgical work towards the production of a bulk flotation concentrate to feed a hydrometallurgical treatment plant.

The project scope at that time was around a 87,000 tpd flotation plant and the Outotec Nickel Chloride Leach process to produce 35,000 tpa LME grade nickel metal and 2,000 tpa cobalt as a hydroxide.

In 2010, HNC completed two core holes totalling 384m to recover 3,530 kg of core for metallurgical testing.

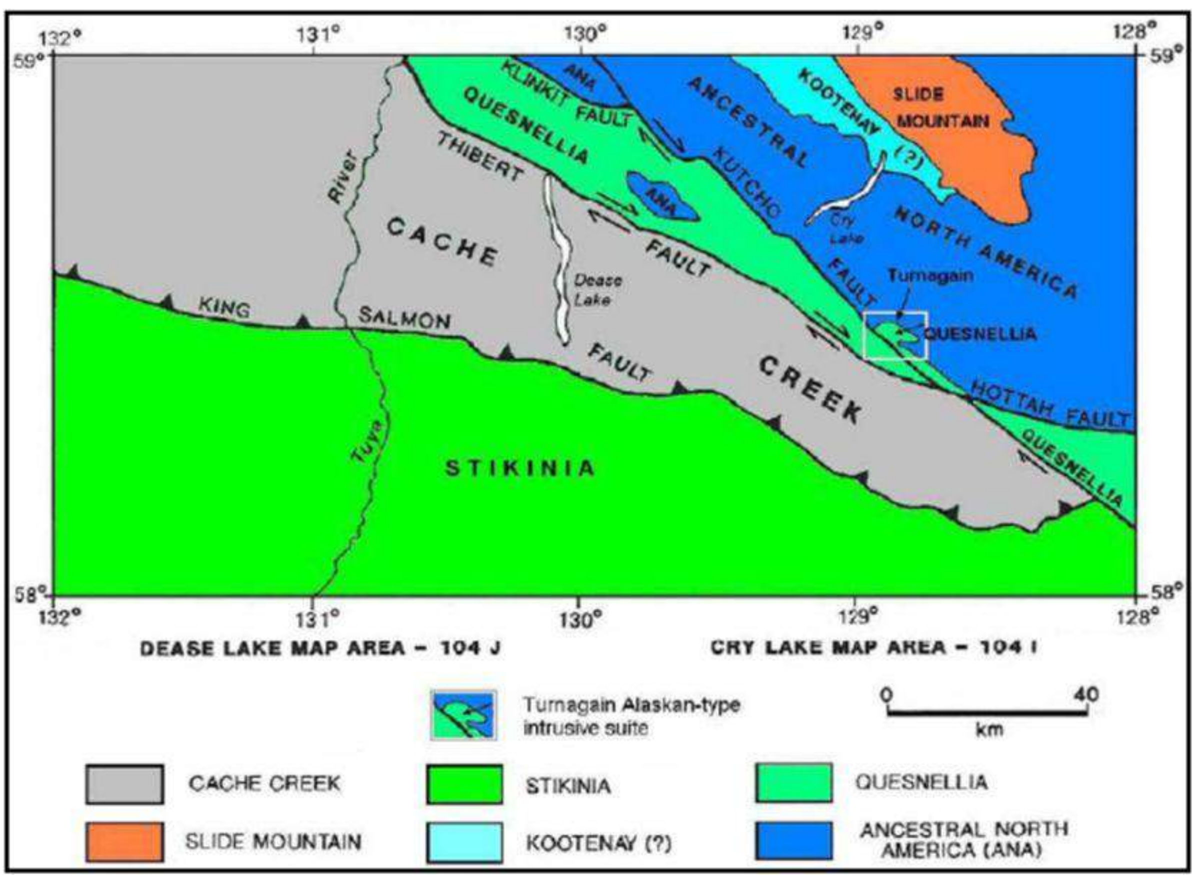
# 7 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 Regional Geology

The regional geology of the Turnagain property has been described by Nixon (1997, 1998), Scheel et al. (2005), and in Technical Reports by Geosim (2006, 2007) and AMEC (2006). The regional description provided here is based on work by Scheel et al. (2005), Scheel (2007), and Nixon (1998). The geological understanding of the region and the setting of the deposit continues to be refined with additional information from drilling and exploration programs.

The property encompasses the Turnagain ultramafic complex and its host rocks, and the ultramatic rocks may be hosted within either the Yukon-Tanana terrane or the Quesnel terrane. The Turnagain complex is fault-bounded, has dimensions of about 3.5 km x 8 km, and lies to the north of two major fault systems — the Kutcho and Thibert–Hottah Faults (Figure 7.1). Neither fault system is exposed in the property.

**Figure 7.1 Regional Structural Setting – Turnagain Property**



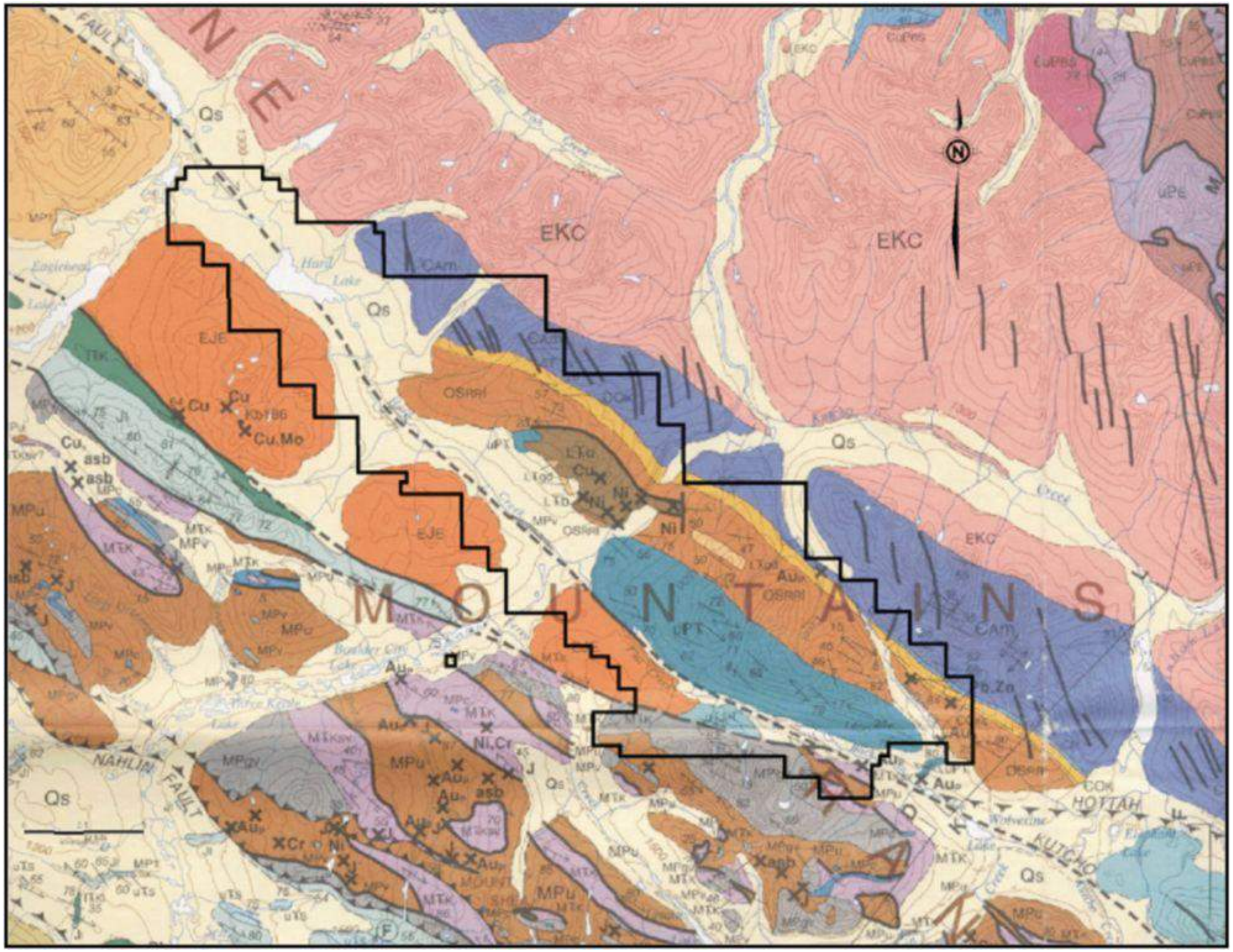

Note: Modified from Gabrielse (1998)

The western, northern, and eastern margins of the complex abut rocks attributed to the Lower Ordovician Road River Formation and the Mississippian Earn Group (Figure 7.2 and Figure 7.3). The Road River and Earn Group rocks comprise graphitic phyllite, which can be strongly pyritic and graphitic near the Turnagain complex intercalated with lesser quartz-rich and calc-silicate tuff layers. The graphitic phyllite in the vicinity of the property remains directly and biostratigraphically undated. Metamorphism in the phyllites regionally reaches greenschist facies. No contact hornfelsing has been mapped adjacent to the northern or eastern contacts with the Turnagain complex.

South of the Turnagain complex is a series of undated sedimentary rocks, possibly volcaniclastic, that may represent rocks of the Lay Range assemblage of the Quesnel terrane (Figure 7.2 and Figure 7.3). On the south side of the Kutcho Fault, dioritic to granodioritic rocks from the early Jurassic Eaglehead Pluton crop out.

The regional setting and method of emplacement of the Turnagain complex is still being established. Gabrielse (1998) postulates that the Turnagain complex intrudes rocks of the miogeoclinal margin of ancestral North America, indicating that a supra-subduction setting was operational at the cratonic margin at the time of emplacement. An alternative view (Scheel et al., 2005; Nixon, 1998) places the Turnagain complex within an imbricated set of rocks that was thrust eastward onto the margin of the North American craton.

## Figure 7.2 Regional Geology



## Figure 7.3 Geological Legend

## 7.2 Property Geology

The 190 ±1 Ma (Scheel 2007) Turnagain complex comprises a central core of dunite with bounding units of wehrlite, olivine clinopyroxenite, clinopyroxenite, representing crystal cumulate sequences, hornblende clinopyroxenite and hornblendite (Figure 7.4). No orthopyroxene is present. The complex is elongate and broadly conformable to the northwesterly-trending regional structural grain.

The ultramafic rocks are generally fresh-to-mildly serpentinized; however, more intense serpentinization and talc-carbonate alteration are common along faults and restricted zones within the complex. The central part of the ultramafic body is intruded by granodiorite to diorite, and hornblende–plagioclase porphyry dikes and sills.

Primary layering in clinopyroxene-rich cumulates, reflecting variations in the modal abundance of olivine and pyroxene, is visible in outcrop. The layering has moderate to steep dips and is truncated by the faulted eastern boundary of the complex. Despite localized zones of well-developed layering, way up criteria are inconclusive and the internal structure of the Turnagain complex is poorly understood (Nixon, 1998).

The following description of lithologies is modified from Scheel et al. (2005).

### 7.2.1 Dunite

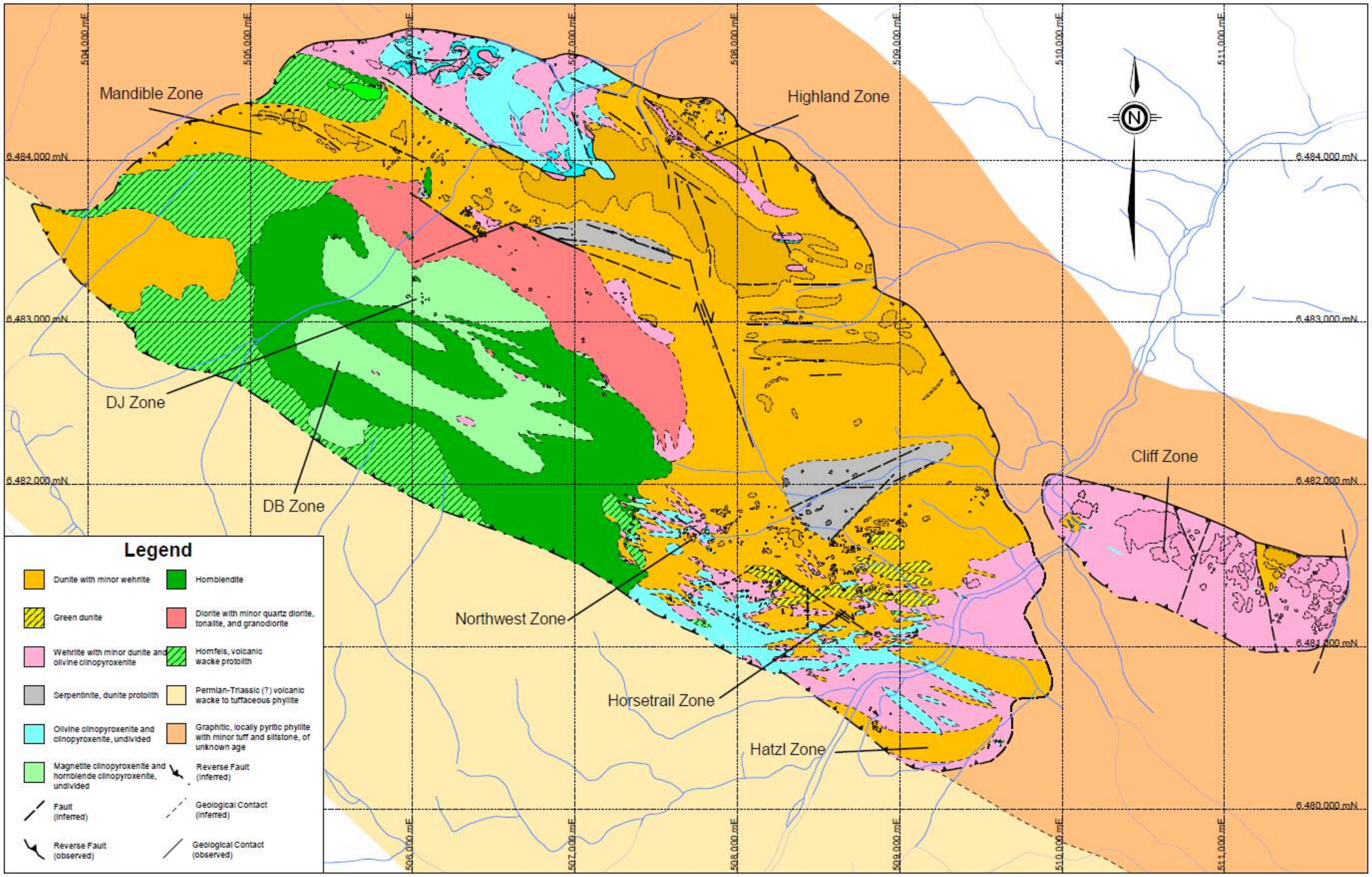
Dunite is primarily found in the eastern and central portions of the complex. It is mainly composed of cumulus olivine, minor amounts of chromite and pyroxene, and trace amounts of primary phlogopite. Dunite commonly hosts grains of poikilitic green diopside, either as discrete, centimetre-scale crystals or elongate aggregations. The latter are interpreted to be small dikes resulting from the escape of trapped liquid.

Millimetre- to centimetre- scale layering in the dunite core is evident locally where concentrations of chromite crystals have accumulated. These chromitite horizons are discontinuous and commonly remobilized and intruded by thin dunite dikes.

Serpentinization volumes are highly variable but generally are no more than about 10% of the rock mass by volume. The degree of overall serpentinized is higher in the Horsetrail, Northwest, and Hatzl Zones. Large amounts of secondary magnetite are found where serpentinization is pervasive. Some dunite that is proximal to massive sulphide mineralization commonly contains some alteration to grey tremolite.

Contacts between wehrlite and dunite are sharp to gradational over short distances, represented by a slight change in the size and modal abundance of pyroxene, and may reflect magmatic layering.

## Figure 7.4 Property Geology



### 7.2.2 Wehrlite

Two different wehrlite types have been identified. On the west side of the Turnagain River, the wehrlite is mainly composed of cumulus olivine with a sizable proportion of interstitial clinopyroxene and minor amounts of cumulus pyroxene. On the east side of the river, and in the far northwest of the intrusion, cumulus clinopyroxene reaches approximately 40% by volume of the rock mass, cumulus clinopyroxene is typically prismatic and finer grained than coexisting olivine.

### 7.2.3 Olivine Clinopyroxenite and Clinopyroxenite

These rock types commonly crop out in the northwestern part of the intrusion and commonly comprise around 85% cumulus pyroxene and smaller amounts of cumulus olivine. These rocks are also common along the southern margin of the HT-NW Zones. Depending on location within the complex, the clinopyroxenites can be either an original magma differentiate or an intrusive; in the northwestern portion of the complex, they appear to be related to the original magma, further to the east, they are brecciated and intrusive in nature. Pegmatitic clinopyroxenite dykes are commonly found adjacent to the cumulate clinopyroxenite or intruding more magnesium lithologies on the Horsetrail-Northwest Zones. The latter intrusions are interpreted to be late-stage injections of trapped liquid through olivine-rich cumulates.

### 7.2.4 Hornblende Clinopyroxenite and Clinopyroxenite

These rock types are generally restricted to the west central portion of the Turnagain intrusion, and coincide with a copper-platinum-palladium soil anomaly. They are very poorly exposed and their relationships to other units in the Turnagain complex are not well-defined. Some of these rocks contain angular, altered clasts of former dunite and wehrlite.

### 7.2.5 Magmatic Hornblendite and Hornblende Clinopyroxenite

Generally found in the south western portion of the complex, these rock types contain amphibole crystals that typically range from less than one centimetre to up to three centimetres in length. The crystals appear to be cumulus, but in some cases they replace pyroxene. Most hornblende-bearing ultramafic rocks in the Turnagain complex are associated with large amounts of magnetite that is interpreted to be cumulus in origin.

### 7.2.6 Hornblende Diorite

A 2,000m x 300m elongate hornblende diorite to granodiorite body, offset by an east-northeast striking fault, intrudes hornblendite and dunite in the central part of the intrusive suite. Narrow porphyritic granitic dykes, about 1-2m wide and clearly post-mineral, were noted cutting wehrlites and clinopyroxenites in drill core, some dykes may be up to 20m wide and all dykes are spatially associated with the large hornblende diorite intrusion.

### 7.2.7 Metasediments

Numerous inliers, xenoliths and small inclusions of hornfelsed, calc-silicate metasedimentary rocks, similar to those seen marginal to the ultramafic intrusion, are present within the ultramafic intrusive rocks. These inclusions are are thought to be the sulphur source responsible for the sulphide mineralization in the Turnagain intrusion and are sourced from the wall rocks.

## 7.3 Mineralization

Showings of semi-massive and massive sulphides have been identified by work to date (Figure 7.4). These semi-massive and massive zones, plus broad zones of disseminated sulphides, are invariably hosted by dunite and wehrlite near the southern and eastern margins of the ultramafic body. The central and northern dunite is essentially devoid of sulphide minerals although it is worthy of note that the highly magnesian olivine is more enriched in nickel (up to 0.20-0.30 weight %) than the olivine in the peridotites and pyroxenites, which may be depleted in nickel in areas of sulphide mineralization. Nixon (1998) suggests that these features are further evidence of fractional crystallization of the ultramafic magma.

Primary sulphide minerals consist mainly of pyrrhotite with lesser pentlandite (iron-nickel sulphide) and minor chalcopyrite. Some bornite has been reported (Geosim concurs with other investigators that these are magmatic sulphides). Interstitial and blebby sulphides, with grain sizes ranging from 1 to 4 mm, are evident in widespread disseminated zones seen in drill cores. With increasing concentrations, these intercumulus sulphide grains coalesce to form net-textured sulphides. Semi-massive and massive sulphides, and rare sulphide matrix breccias, were also noted in drill cores over intervals not exceeding a few tens of centimetres.

Narrow fracture-filling sulphide lenses, commonly featuring chalcopyrite and minor pentlandite along with the more prevalent pyrrhotite, appear to be products of remobilization of primary sulphides adjacent to dykes, altered xenoliths, and serpentinized areas.

Secondary nickel and copper sulphides, including violarite and valleriite, have been noted in serpentinized zones and both primary and secondary sulphides are associated with graphite (Nixon, 1998). Recent microscopic and microprobe studies of drill core samples from the Horsetrail zone (Kucha, 2005) have identified additional nickel sulphide minerals including mackinawite, heazlewoodite, godlevskite, and millerite. Platinum group element minerals identified to date include vysotskite, a palladium-iron-nickel sulphide, and sperrylite, a platinum arsenide mineral.

The principal mineral zones identified to date on the Turnagain property (Figure 7.4) include the following:

- The Horsetrail Zone and surrounding area have been the focus of most of the historic and recent diamond drilling. Results to date suggest a northwest to west-northwest trend for these zones which consist of broadly dispersed, disseminated to intercumulus sulphide mineralization in both dunite and wehrlite and serpentinized equivalents. Sulphide grains range in size from 0.5 to 5 mm and commonly occupy interstices between olivine grains. Drill core samples from the Horsetrail Zone have a median of 0.23% total nickel with grades ranging from 0.01% to 4.89% total nickel. AC-Ni-based (ammonium-citrate–hydrogen peroxide leaching analytical method) grades range from 0.01% to 4.06% and have a median grade of 0.14%. Total cobalt grades range from 0.001% to 0.480% with a median of 0.013% Co. There appears to be a spatial relationship between graphitic xenoliths, increasing clinopyroxene content in the ultramafic host rocks and the incidence of sulphide mineralization. Where present, chalcopyrite occurs along the margins of pyrrhotite and in narrow veinlets. Relatively unaltered dunite adjacent to the Horsetrail Zone may contain total nickel values of 0.20% to 0.30%, virtually all of which is in the crystal lattices of the silicate mineral olivine and consequently is not of economic importance.

- The Northwest Zone is contiguous with, and lies northwest of, the Horsetrail Zone. This zone has mineralization styles and grades similar to the Horsetrail Zone but is intruded by several mafic and felsic dikes which dilute the overall grade. Drill core samples from the Northwest Zone have a median grade of 0.20% total nickel with grades ranging from 0.01% to 2.86%. AC-Ni-based grades range from 0.01% to 0.76% and have a median of 0.14%. Total cobalt grades range from 0.001% to 0.166%. The Horsetrail and the Northwest Zones form a zone approximately 2,000m long in the east-west direction, and 550m wide from north to south and have been tested by 228 drillholes.
- The Hatzl Zone mineralization consists of disseminated and net textured pyrrhotite and pentlandite hosted by dunite and wehrlite. This mineralization is similar to the Horsetrail Zone and may be continuous with Horsetrail. The Turnagain River flows between the two zones and the region below the river has not been sufficiently drill-tested to exclude the potential of additional mineralization. The Hatzl Zone is 1,150m long in a northeast direction and 300m wide in a northwest direction and has been tested by 17 drillholes.
- The Duffy Zone mineralization lies 500m northeast of the Horsetrail Zone and consists of disseminated sulphides similar to those within the Horsetrail Zone. Grades range from 0.014% to 0.525% total nickel and 0.007% to 0.388% AC-Ni. The Duffy Zone is 300m in diameter, lies 70m below the surface topography, does not crop out and was discovered by exploration drilling in 2006. The zone has been tested by six drillholes.

Other mineralized zones are exploration targets undergoing initial drilling, including:

- the Bench, DJ, and DB prospects, which host platinum group element (PGE) mineralization
- the Mandible, Davis, Highland, and Discovery prospects, which host Ni-Co mineralization
- the Cliff and Central area prospects, which host Ni-Co and PGE mineralization.

## 8 DEPOSIT TYPES

The geological setting of the sulphide mineralization at the Turnagain deposit is unusual, in that it is hosted by an Alaskan-type complex, which is a magmatic environment that is not generally noted for its sulphide potential. Nixon (1998) concluded that the iron-nickel-copper (Fe-Ni-Cu) sulphides in the Turnagain complex are of magmatic origin, and that wall rock inclusions observed in drill core may have provided a mechanism for sulphur saturation and precipitation of Fe-Ni-Cu sulphides. This has been confirmed by sulphur and lead isotope results reported by Scheel (2007).

Disseminated and rare net-textured mineralization at Turnagain is hosted in dunite, wehrlite, olivine clinopyroxenite and clinopyroxenite and serpentinized equivalents. Sulphides comprise pyrrhotite, pentlandite, chalcopyrite and trace bornite. Valleriite occurs where serpentinization is intense.

# 9 EXPLORATION

Section 6 of this report summarizes the early exploration work carried out between 1957 and 1995, and presents an overview of work completed by HNC and its precursor companies since acquisition of the Project in 1996. This section presents more detail on HNC's exploration.

## 9.1 Geological Mapping

Sulphide-bearing outcrops of the Davis, Horsetrail, Discovery, and Cliff showings were relocated, and then prospected and mapped in 1996.

In 1998, a global positioning survey (GPS) was undertaken by Bren-Mar personnel using a Trimble Geoexplore 2 instrument to locate drillholes, claim posts, and other geographical positions.

Detailed geological mapping was undertaken by Clark (1976) at various scales from 1 inch:50 ft to 1 inch:1,000 ft as part of his Ph.D. thesis work. Additional mapping was completed by HNC geologists and Scheel (2007) at metric scales ranging from 1:1000 to 1:10000.

In 2005, Thurber Engineering Ltd. (Thurber) completed a surficial geology map of the Hard Creek drainage from air photos. The interpretation of surficial geology was extended across the Turnagain River, to cover the Flat Creek drainage, by Thurber in 2009. HNC conducted bedrock mapping and small test pits to aid Thurber's surficial interpretation.

## 9.2 Geochemical Surveys

The following discussion, modified from Carter (2005), is considered thorough; it is believed to reasonably represent the surface geochemical soil sampling programs completed on the property.

Of particular importance are the results of a 1971 soil geochemical survey conducted by UMEX over mineral claims contiguous with Falconbridge claims, and covering the northeastern margin of the ultramafic complex and the Cliff Zone east of Turnagain River. More than 800 samples were collected from B and C soil horizons at 200 ft intervals along grid lines spaced 400 ft (122m) apart. The samples were analyzed for nickel, copper, and cobalt. Values greater than 650 ppm nickel and 300 ppm copper were considered to be distinctly anomalous; cobalt values were erratic. The best results were obtained from a 900 x 450m area west of the Discovery zone where anomalous nickel values ranged from 800 to 2000 ppm.

A geochemical sampling program carried out in 2003 consisted of the collection and analyses of 250 soil samples at a 100m spacing along four topographic contour lines between 1300m and 1460m elevation, northwest and upslope of the principal mineralized zones. An analysis and interpretation of the results obtained from these samples was undertaken by Dr. Colin E. Dunn (P.Geo.) on behalf of HNC in early 2004 (Carter, 2005).

Results for copper, nickel, cobalt, and platinum+palladium were kriged and contoured at 90th, 80th, 70th and 50th percentiles. Coincident high copper, cobalt, and platinum+palladium values are concentrated within a poorly-explored area between 3 and

4 km west-northwest of the Horsetrail zone. Elevated nickel values in soils are more widespread and are coincident with the Horsetrail zone and immediately northwest of the copper, cobalt, and platinum+palladium anomalies.

A reconnaissance biogeochemical survey carried out in April 2004 consisted of the collection of 132 twig and bark samples along four transects over the Turnagain ultramafic intrusion. Analytical results were not as definitive as those obtained from previous soil sampling and a comprehensive geochemical soil sampling program was initiated in mid-2004 to follow up and expand upon results of the 2003 surveys.

The 2004/2005 program also consisted of the collection of more than 2,000 soil samples collected at 50m intervals along survey lines spaced 200m apart within an area of 15 $km^2$. More detailed sampling at 25m intervals on lines spaced 50m apart was undertaken in areas yielding anomalous base and precious metals results. Results of this survey highlighted two strong copper-in-soil anomalies 2.5 km northwest of the Horsetrail zone with values exceeding 430 ppm copper with peaks to 3,219 ppm copper over areas of 1,500 x 1,100m and 900 x 600m. These anomalous areas flank the hornblende diorite-granodiorite intrusion that cuts the older ultramafic rocks in this area. Anomalous platinum-palladium values in soils, in part coincident with the DJ zone, extend from the northern part of the larger copper-in-soils anomaly. Anomalous nickel values in soils are widespread over the northern part of the Turnagain ultramafic intrusion and within and adjacent to the Horsetrail zone. The geochemical interpretation requires that anomalous nickel values in soils are paired with copper so that the highly mobile nickel originating from olivine can be screened. Copper occurs only in sulphide minerals and when present in ultramafic rocks with nickel can be used successfully to indicate nickel anomalies of exploration significance.

The 2004 geochemical program also included the collection and analyses of 330 rock float and 243 bedrock samples from within, and adjacent to, the soil geochemical grid. Results for total nickel and platinum+palladium indicated significant total nickel results (>0.20% to a maximum of 1.9%) in both float and bedrock samples, which are mainly clustered in the area of the Horsetrail zone and in a smaller area north of the DJ zone, known as the Central area.

## 9.3 Geophysical Surveys

The following discussion, modified from Carter (2005), is considered thorough and to reasonably represent the geophysical survey programs completed on the property.

### 9.3.1 Airborne Surveys

Scintrex Ltd. (Scintrex) completed a helicopter-borne electromagnetic (HEM) and magnetic survey for Falconbridge in July 1969 (680 line-km), and Questor Surveys Ltd. (Questor) completed a fixed wing “high resolution” magnetic survey for Bren-Mar in August 1996 (400 line-km).

A third airborne geophysical survey was completed over the Turnagain property by AeroQuest Ltd. (AeroQuest) in late September 2004. The AeroQuest survey utilized a helicopter-borne AeroTEM II time domain electromagnetic system and a high sensitivity caesium vapour magnetometer. Continuous readings on both instruments were obtained from northeast-southwest oriented survey lines at 100 to 200m spacing; precise locations were established using a GPS.

Two geophysically anomalous areas within the ultramafic rocks were surveyed along lines on 50m centres. Terrain clearance was 30m and the survey totalled 1,866 line-km. The AeroQuest magnetic response confirmed the results of earlier surveys, accurately outlining the limits of the Turnagain ultramafic intrusion. Magnetic data ranged from lows of 55,000 nanoteslas (nT) to highs of 63,000 nT; the average background was 57,800 nT. The AeroQuest survey also highlighted electromagnetic anomalies within the ultramafic intrusion.

### 9.3.2 Ground Magnetic Surveys

Ground magnetic surveys using an Overhauser magnetometer commenced in 1997–1998 to further define two of the airborne anomalies, Davis (Grid A) and Northwest (Grid B).

The Grid A survey used north-south lines at 100m spacing with stations every 25m along lines. A total of 12.3 line-km were surveyed within an approximate 1 $km^2$ area. A number of magnetic highs identified from the survey were correlated with pod-like serpentinized and magnetite-banded peridotite intrusions; however, four of the magnetic anomalies were considered to be potentially due to the presence of sulphides.

Grid B comprised 100m-spaced east-west lines with stations at 25m along lines for a total survey distance of 5.6 line-km. The survey identified a strong positive magnetic anomaly.

Results of the grid-based surveys showed that the areas of high total field magnetic readings do not necessarily coincide with sulphide-rich rocks as there appears to be little correlation between trends and known mineralized showings. Some prospects are on magnetic highs (i.e. the Northwest zone), some on magnetic lows (i.e. the Discovery zone), and others in areas of mixed magnetic response (i.e. the Horsetrail and Fishing Rock zones).

In 2011,Frontier Geoscience Inc. completed a 75.5 line-kilometre ground magnetic survey over the DJ-DB area, centred 2.5 km northwest of the Horsetrail deposit. With magnetic readings every 25m on 50m spaced lines, the survey provided detailed information on distribution of buried lithology and intrusive contacts.

### 9.3.3 Down-hole Geophysics

Borehole pulse electromagnetic surveys were undertaken on four drillholes (97-9, 98-1, -4, and -5) in 1998. All of these holes were drilled to test the southern part of the Horsetrail zone. Major in-hole anomalies were interpreted as being caused by two sheet-like, shallowly south-dipping conductive horizons that, in part, correlate with zones of sulphide mineralization containing elevated (+0.30%) nickel values and with talc/serpentinite zones.

In 2004, down-hole geophysical surveys were completed on another six drillholes in conjunction with surface transient electromagnetic, very low frequency (VLF), and magnetic surveys over an 800m by 900m grid centered on the Horsetrail area. A number of prominent conductors were identified.

Between 2004 and 2007, S.J. Geophysics Ltd. conducted several 3D magnetic inversion studies of selected areas from the 2004 Aeroquest airborne magnetics to determine depths to source of magnetic anomalies and thickness of the Turnagain ultramafic intrusions.

Subsequent drill testing confirmed the interpretations of single magnetic anomalies but was not successful in areas of multiple overlapping anomalies.

### 9.3.4 Seismic Survey

In 2008, a 7.6 km seismic refraction survey was carried out over potential tailings management areas and waste dump sites to determine depth to bedrock and type of overburden.

## 9.4 Drilling

The Turnagain ultramafic intrusion has been tested by 79,351m of diamond drilling in 322 holes since 1966 as listed in Table 9.1. Analytical results for the last 54 holes drilled in 2007 and 16 holes drilled in 2008 are reported in Section 11.0. Results for earlier holes have been published in previous technical reports by Simpson (2006) and AMEC (2007).

**Table 9.1 Summary of Drill Programs**

| Year | Operator | No. Holes | Metres |
|---|---|---|---|
| 1967 | Falconbridge | 13 | 1,304.9 |
| 1970 | Falconbridge | 15 | 1,458.0 |
| 1996 | Bren-Mar | 5 | 795.3 |
| 1997 | Bren-Mar | 9 | 1,855.3 |
| 1998 | Bren-Mar | 5 | 1,264.1 |
| 2002 | CME | 7 | 1,686.6 |
| 2003 | CME | 22* | 8,672.0 |
| 2004 | HNC | 49 | 7,633.4 |
| 2005 | HNC | 37 | 7,143.1 |
| 2006 | HNC | 68** | 19,121.8 |
| 2007 | HNC | 74 | 23,927.1 |
| 2008 | HNC | 16*** | 4,105.3 |
| 2010 | HNC | 2 | 384.1 |
| **Total** | | **320** | **79,351.0** |

One 2003 drillhole was extended.
** One 2005 drillhole was extended.
*** Three earlier drillholes were extended.

## 9.5 Other Studies

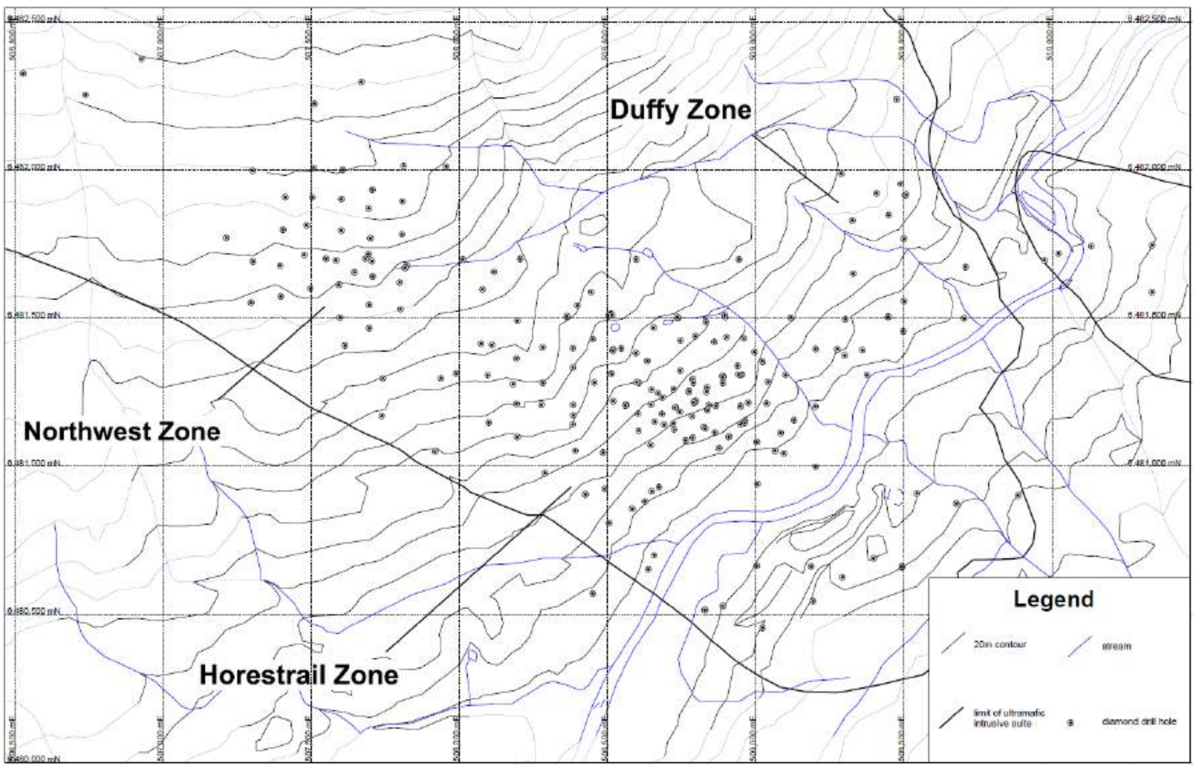
During 1972–1975, a Ph.D. dissertation was completed on the geology of the Turnagain intrusion (Clark, 1976). The work comprised geological mapping, lithogeochemical sampling, and petrographic studies.

In 2007, a master thesis was completed on the age and origin of the Turnagain intrusion and associated mineralization (Scheel, 2007).

## 10 DRILLING

The Turnagain drill hole database contains a total of 322 core holes totalling 79,351m of drilling. The previous technical report (AMEC, 2007) reported on the first 20 holes completed in 2007. An additional 54 holes (16,993.6m) were completed by the end of 2007 and 16 holes (4,105.3m) were drilled in the 2008 season, including two geotechnical holes for groundwater monitoring. Two horizontal HQ- size holes (384.05m) were drilled in the Horsetrail deposit to obtain samples for metallurgical testing. Hole locations within the Horsetrail zone are shown in Figure 10.1.

**Figure 10.1 Drill Hole Location Plan**




Most of the holes drilled to date have been inclined. Since 2004, contractors DJ Drilling (2004). Ltd. (DJ Drilling) has recovered NQ size (47.6 mm) core. Part of the 2007 drill program included PQ size (85 mm) core collected for metallurgical purposes. Core recoveries are excellent, averaging 95%. Prior to 2006, most drill core sample intervals were 2m. Since 2006, core sampling has been completed predominantly on 4m intervals.

### 10.1 Collar Surveying

HNC planned drill holes in advance and then spotted the collar in the field using a backpack-mounted Trimble Differential Global Positioning System (DGPS). Because of the high magnetic background, HNC set the direction of drilling using foresights and backsights, which were also spotted by HNC staff using a backpack portable DGPS. After completion of the hole, the collar location was resurveyed using the same backpack Trimble DGPS; the dip was taken from the first Reflex Maxibor® II down-hole survey measurement. Finally the collar location, azimuth, and dip were surveyed by Gabriel Aucoin (Commissioned Land Surveyor [CLS]) of Aucoin Surveys Limited. Data in the collar table for holes used in the resource estimate are the best available method for each attribute of each hole.

## 10.2 Downhole Surveying

A Reflex Maxibor® II unit was used for most downhole surveying since 2004. Where casing was intact, 2002 and 2003 holes were re-entered and surveyed with the Maxibor® II instrument. A number of holes were not surveyed either because they were initial exploration holes drilled outside of the Horsetrail area, damaged or missing casing prevented re-entry, or the survey tool was not available. Where Maxibor® II surveys were not conducted, acid dip tests provided limited control on hole orientation.

# 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

## 11.1 Sampling Method

### 11.1.1 Geotechnical Data

In 2007, HNC contracted Piteau and Associates Engineering Ltd. (Piteau) to provide geotechnical core logging guidance. Piteau provided HNC geologists with instructions for recording core rock quality designation (RQD), recovery, joint frequency, joint condition, fracture density and orientation, hardness, and weathering. HNC has posted the geotechnical protocol in several places in the core shack and use standardized geotechnical core logs. In addition, HNC geologists are performing point load tests on core, following the instructions set out by Piteau. During the 2005 and 2006 drill programs, the geotechnical core logging protocol was designed by Knight Piesold Consulting Geotechnical logging between 2002 and 2004 included RQD and recovery only.

### 11.1.2 Geological Data

In 2006, HNC established a core logging and sampling protocol that is posted as a flowsheet in the core shack. Prior to any geological logging, the core is re-aligned and driller block measurements are converted to metres. Drill core was sampled at 2m intervals or less during the 2004 and 2005 programs and on 4m intervals since 2006. Following core logging, sample intervals are marked with a red or yellow marker and sample numbers are assigned from a pre-printed Acme Analytical Laboratories Ltd. (Acme) assay tag book. Core is photographed three boxes at a time on the logging rack in the core shack with a digital camera.

Core is halved by use of a hydraulic core splitter and/or a diamond saw. Half of the core is stored in core boxes on site and half is sent for analysis.

### 11.1.3 Density

HNC collects bulk density measurements by water immersion method every 20 samples, using up to 50 cm of un-split core. A protocol for density measurements is posted in the logging tent. Density is calculated as follows:

$$\text{Density} = \text{weight in air} / (\text{weight in air} - \text{weight in water})$$

Data is recorded manually on paper and later transferred to a digital file. Data entry errors due to transposition of numbers or poor written records are possible. AMEC (2007) recommended double data entry for any manual entry of data into a database and also suggested that HNC create a density standard to use periodically to ensure the scale is working properly.

## 11.2 Sample Preparation and Analyses

No information is available regarding sample preparation, analytical procedures, or quality assurance/quality control (QA/QC) measures in place during the 1967-1998 exploration programs. As none of this data was used in resource estimation, this is not considered significant.

Drill core samples from the 2002 to 2008 programs, received by Acme in Vancouver, were checked against requisition documents prior to being dried, weighed, crushed, split, and pulverized before being subjected to a variety of analytical techniques. Acme is a certified ISO:9000 facility.

Prior to 2004, samples were analyzed for nickel, copper, cobalt, and approximately 20 major and minor elements by aqua regia digestion followed by an inductively coupled plasma emission spectroscopy (ICP-ES) finish. Samples collected from the 2004 to 2008 programs were subjected to a four-acid ($HNO_3$-$HClO_4$-HF and HCl) digestion followed by ICP-ES analyses to determine values for total nickel, copper, cobalt, and 22 other elements, including sulphur.

In 2003, to assist in distinguishing nickel in sulphide from nickel in silicate phases, every 10th sample was analyzed for nickel, cobalt, and magnesium with an ammonium citrate-hydrogen peroxide leach. Beginning in 2004, all new samples and most of the older samples have been analyzed with AC-Ni.

In 2004 and 2005, sulphur content was analyzed by the Leco furnace method. In 2006, sulphur content was analyzed by ICP-ES after a four-acid digestion. In 2007/2008, sulphur content was analyzed by Leco and ICP-ES.

For some core, particularly outside the resource area, samples were analyzed for platinum, palladium, and gold by lead-collection fire-assay fusion followed by ICP-ES and results reported in parts per billion (ppb).

## 11.3 Quality Assurance / Quality Control

Laboratory quality control since 2004 has been maintained by routinely analyzing internal standards, sample blanks, and duplicate samples. HNC staff also insert reference sample pulps in the field as samples ending in 00, 01, 25, 26, 50, 51, 75, 76, 100, 101, and blank samples are inserted every 30 samples. Acme is instructed to create and analyze duplicate pulps from crushed core every 30th sample. Pulps from every 10th sample are sent to a check laboratory. Since 2007, International Plasma Laboratories Ltd. (IPL) in Richmond has been used as a check laboratory and analyzed pulps for total nickel, sulphide nickel, and sulphur. Prior to 2007, ALS Chemex in Vancouver was used as a check laboratory and pulps were analyzed for total nickel and sulphur. IPL is ISO:9001 certified; the ALS Chemex laboratory is ISO:9002 certified.

The HNC reference standards used for Ni, Cu, and Co include two Canada Centre for Mineral and Energy Technology (CANMET) reference samples labelled “U.M. 2” and “U.M. 4” (Cameron, 1975). Both were derived from small lenticular masses of peridotite that occur along a major east-west fault zone in the Werner Lake District of northwestern Ontario. CANMET analyzed the material for ascorbic acid-hydrogen peroxide soluble nickel and, by use of a four-acid digest, for total Ni contents. The CANMET certification was completed in 1974 and is not supported by current industry standards requiring a round-robin approach using several laboratories.

HNC has two reference materials (05-94 and 05-103) prepared from mineralized drill core from the resource area. These standards were initially certified by Smee & Associates Consulting Ltd. through a round-robin process for total digestion nickel, iron, copper, and sulphur. In 2009, AGORATEK International (AGORATEK) supervised a standard re-certification program for all four standards.

Three other standards labelled PGMS-1, WGB-1, and WMG-1 were used for monitoring platinum and palladium.

The field blank material used since 2006 was crushed granite gneiss obtained from Squamish.

HNC inspects quality control samples on receipt of Acme certificates. Failures are identified by inspection of values. Standards and blanks are reviewed based on acceptable limits and duplicates based on straight line graph. When failures are identified, Acme is notified to do re-run analyses on the batch.

### 11.4 Sampling Security

Drill core was transported from the drill site to the exploration camp for logging by the drill contractor. Split core samples were numbered, bagged, and transported from the site by helicopter to Dease Lake in sealed numbered bags in 300 to 350 kg lots, or by plane to Smithers. The samples were then shipped by commercial transport to the primary laboratory (Acme) in Vancouver. Requisition forms were faxed to the HNC Vancouver office with the date and number of samples shipped, and Acme notified HNC upon receipt of samples.

Drill core from holes drilled between 1996 and 2002 is stored in racks at the Boulder camp on Wheaton Creek, 15 km west of the property. Core recovered from all the 2003 to 2008 programs is stored in sturdy racks at the camp on the property. Sample security and core storage are considered to conform to industry standards.

# 12 DATA VERIFICATION

## 12.1 Geosim Site Visits

The QP (Ronald G. Simpson) has visited the Turnagain property on three occasions – October 2005, October 2006, and June 2009. The site inspections included the examination of drill sites, drill core, and surface outcrops as well as observation of sample preparation and QA/QC procedures. He has also reviewed the geological information from previous programs and other relevant data available in the HNC office. He is of the opinion that the programs have been conducted and the data gathered in a professional and ethical manner and conform to standards acceptable within the industry.

## 12.2 Database Validation

AMEC (2007) completed an independent review of the Turnagain Project database in 2007 and concluded that it was sufficiently precise and accurate to support resource estimation. Recommendations regarding the documentation of collar and downhole surveys were followed in subsequent database updates.

Geosim has reviewed the updated database for consistency, accuracy, and precision and has found no issues of significance.

AGORATEK has carried out a comprehensive database audit. (AGORATEK 2011)

## 12.3 QA/QC Review

### 12.3.1 Certified Reference Materials

Laboratory quality control was maintained by routinely analyzing internal standards, sample blanks, and duplicate samples. HNC staff also inserted standard reference samples with known nickel content in the sample sequence every 25 samples to monitor laboratory accuracy. Field blanks were also inserted in sample batches to check for possible laboratory contamination.

Two purchased reference standards labelled UM-2 and UM-4 have been used since 2004. In the latter half of 2006, CDN Resource Laboratories Ltd. (CDN) in Vancouver created site specific standard pulps from drill core rejects. One sample was from drillhole 05-94 (108-156m) and the other from hole 05-103 (107-124m). The drill core rejects represented typical material types with variable sulphur and nickel grades.

The bulk standards were prepared and packaged by CDN of Delta, BC. Each bulk sample was pulverized in a large rod mill, screened through 200 mesh using an electric sieve, and homogenized in a large rotating mixer. Each standard was sealed in plastic (5g) to prevent gravity separation and oxidation. The program was carried out under the supervision of Smee & Associates Consulting Ltd.

In 2007, AGORATEK supervised a standard re-certification program for all four standards. The following laboratories received 20 samples of each standard, to be assayed in 4 batches.

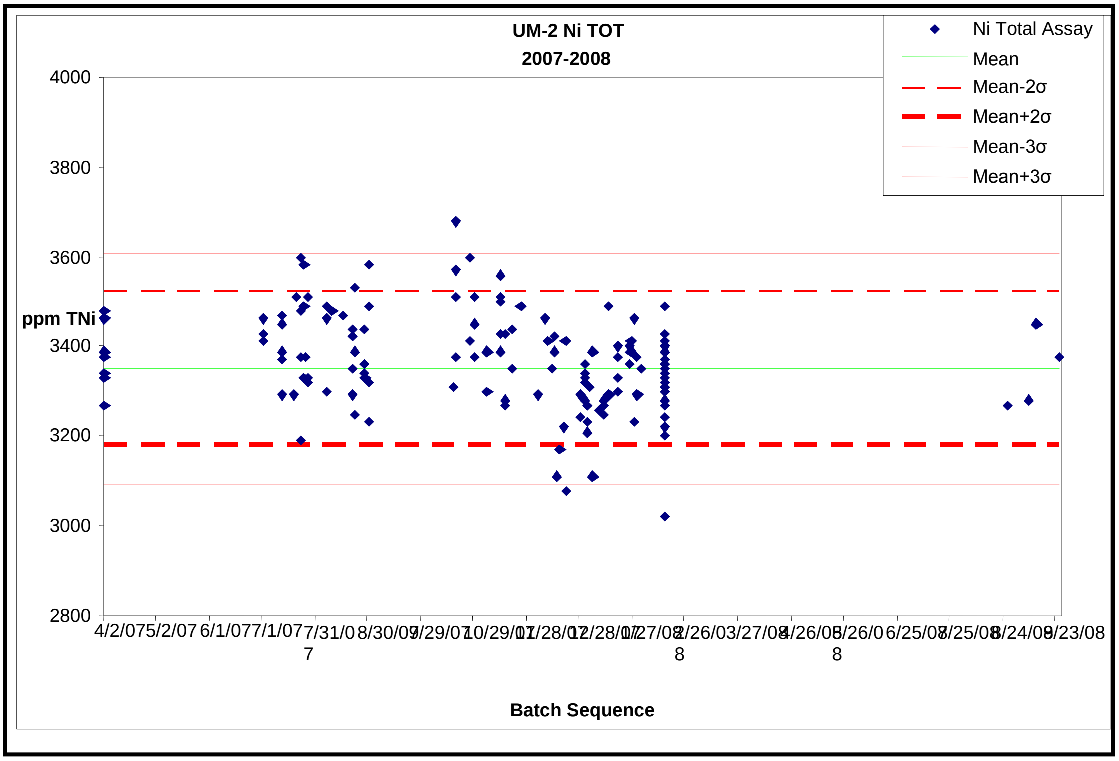
- ALS Laboratory Group (ALS), Vancouver
- International Plasma Labs Ltd. (iPL), Vancouver
- Assayers Canada Ltd. (Assayers Canada), Vancouver
- Genalysis Laboratory Services Pty Ltd. (Genalysis), Perth
- G&T Metallurgical Services Ltd. (G&T), Kamloops

Results of the round-robin test are summarized in Table 12.1. The overall standard performance for total Ni for the 2007-08 drill programs is judged as acceptable (Figure 12.1 to 12.4). However, results for standard 05-94 (Figure 12.3) do show a slight low bias averaging 0.248 compared with the accepted standard mean of 0.257. When adjusted for bias, the assays fall within acceptable limits.

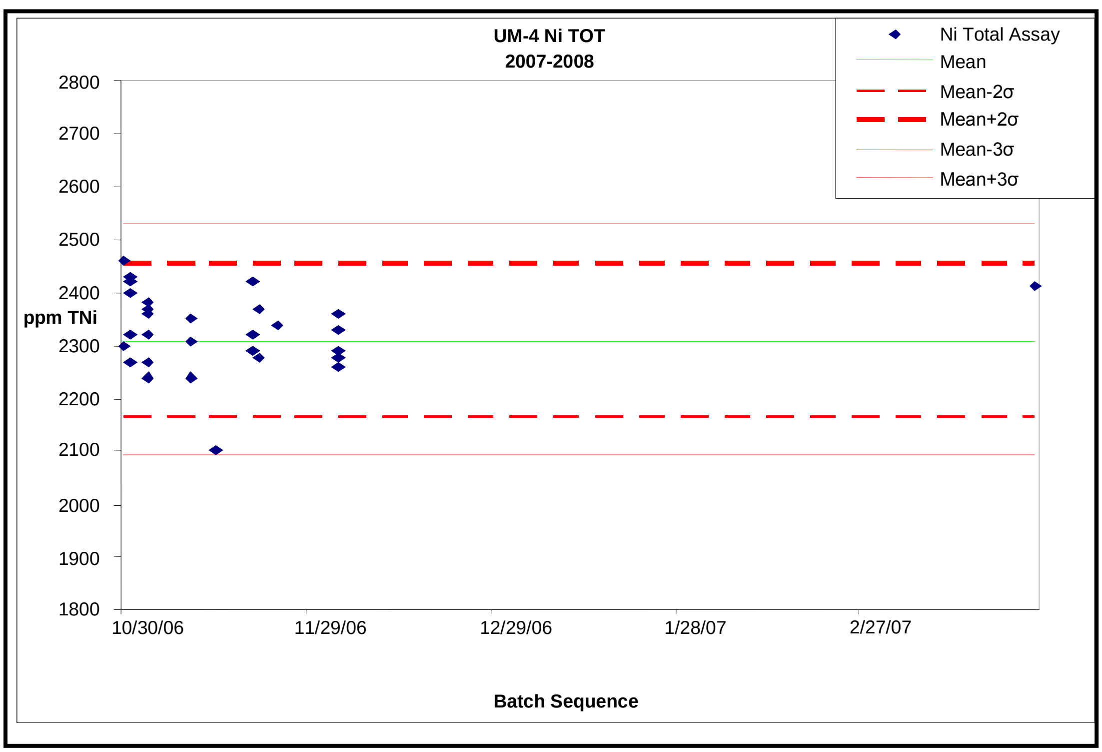
## Table 12.1 Standard Re-certification Statistics

| Standard | Statistic | AC-Ni | Ni-4AD | AC-Cu | AC-Co | Cu-4AD | Co-4AD | Mg-4AD | Fe-4AD | S-Leco |
|---|---|---|---|---|---|---|---|---|---|---|
| 05-94 | Recommended Mean | 0.204 | 0.257 | 0.028 | 0.0116 | 0.0317 | 0.017 | 24.504 | 8.555 | 1.436 |
| | Standard Error of Mean | 0.0021 | 0.003 | 0.0006 | 0.00003 | 0.0008 | 0.0005 | 0.9302 | 0.0441 | 0.0089 |
| | Standard Deviation (1) at one Lab (grade %) | 0.0052 | 0.0048 | 0.0013 | 0.00096 | 0.0008 | 0.0005 | 0.715 | 0.1877 | 0.025 |
| 05-103 | Recommended Mean | 0.369 | 0.403 | 0.045 | 0.024 | 0.049 | 0.028 | 21.539 | 12.494 | 4.41 |
| | Standard Error of Mean | 0.0017 | 0.0045 | 0.0013 | 0.001 | 0.0014 | 0.0007 | 0.8297 | 0.04 | 0.0128 |
| | Standard Deviation (1) at one Lab (grade %) | 0.0111 | 0.0122 | 0.0024 | 0.001 | 0.001 | 0.0009 | 0.8004 | 0.3325 | 0.0449 |
| UM-2 | Recommended Mean | 0.225 | 0.335 | 0.08 | 0.01 | 0.095 | 0.015 | 14.951 | 8.829 | 1.062 |
| | Standard Error of Mean | 0.0056 | 0.0035 | 0.0005 | 0.0002 | 0.0031 | 0.0001 | 0.5005 | 0.095 | 0.0203 |
| | Standard Deviation (1) at one Lab (grade %) | 0.0072 | 0.0086 | 0.005 | 0.0005 | 0.002 | 0.0003 | 0.4353 | 0.1599 | 0.0201 |
| UM-4 | Recommended Mean | 0.177 | 0.231 | 0.052 | 0.007 | 0.056 | 0.01 | 13.687 | 7.959 | 0.495 |
| | Standard Error of Mean | 0.0048 | 0.0025 | 0.002 | 0.0002 | 0.0015 | 0.0001 | 0.4361 | 0.0873 | 0.0057 |
| | Standard Deviation (1) at one Lab (grade %) | 0.0036 | 0.0073 | 0.0033 | 0.0005 | 0.003 | 0.0003 | 0.4616 | 0.2745 | 0.0144 |

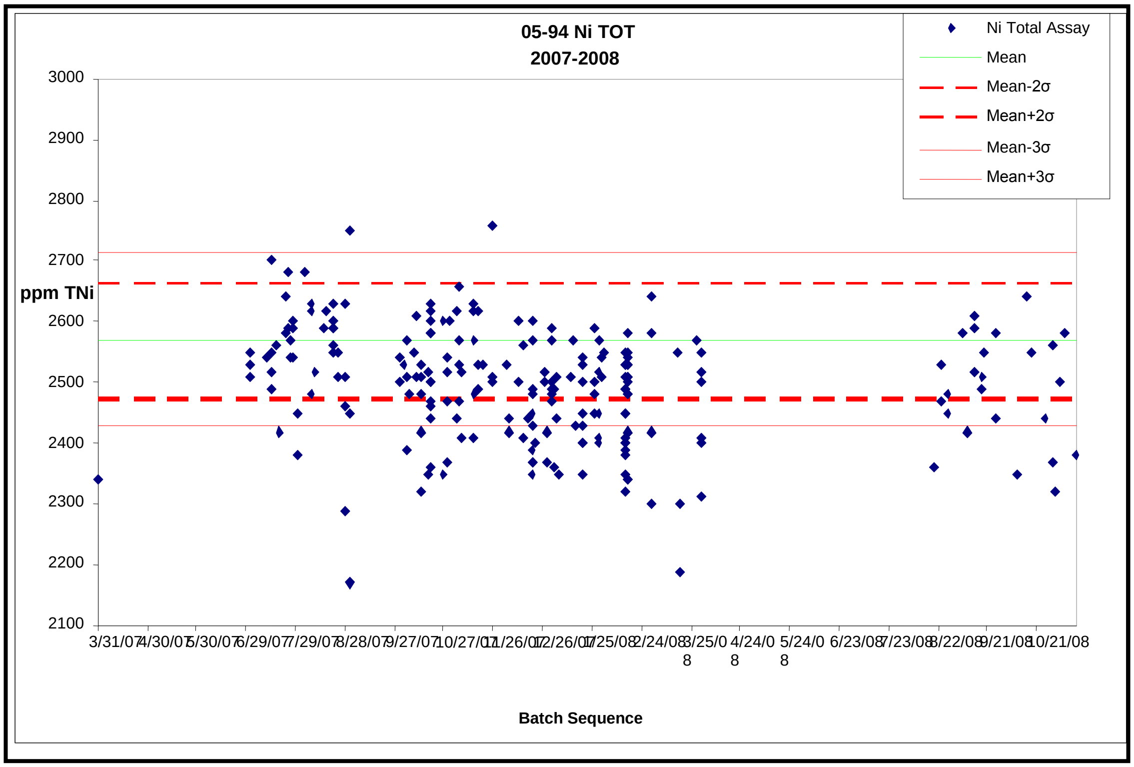
## Figure 12.1 Standard UM-2 Control Chart for Total Ni 2007-2008



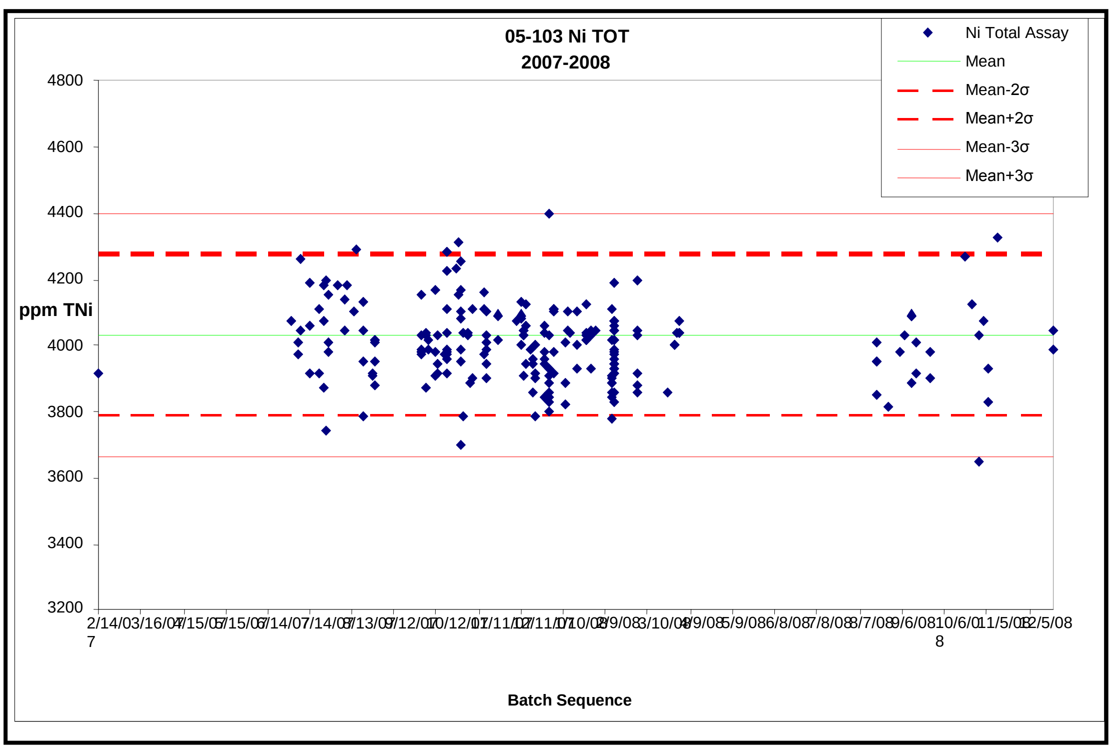

## Figure 12.2 Standard UM-4 Control Chart for Total Ni 2007-2008



## Figure 12.3 Standard 05-94 Control Chart for Total Ni 2007-2008




## Figure 12.4 Standard 05-103 Control Chart for Total Ni 2007-2008



The standard re-certification included AC Ni for the first time. Previous studies included comparison with results from a secondary laboratory (IPL) and showed reasonable correlation. However the re-certified results suggest that Acme has a high bias ranging from 7% to 13% for UM-2 and UM-4 between 2004 and 2008, and between 7% and 9% for the new standards between 2007 and 2008. In view of these results, it was decided not to use AC Ni as the primary grade item in the present resource estimate.

### 12.3.2 Check Assays

A study of historic and recent check assays up to March 2009 was carried out by AGORATEK as part of an ongoing QA/QC evaluation. Although certain biases were identified in this and previous reports, AGORATEK noted that:

> *"As of 2007, none of the problems identified by AMEC were considered critical for the suitability of the database to support HCN's resource estimates. As was found by AGORATEK International, the quality of the assaying, if anything, has been improving ever since."*

The preliminary conclusions from the AGORATEK study are outlined in the following sections.

#### 12.3.2.1 Pre-2007 Data (ALS Check Assays)

For Ni, ALS and Acme were actually both biased in opposite directions, in smaller amounts. Standards were used to estimate the actual Acme bias and it was confirmed that Acme was biased on AC Ni by about 9% to 15% and on totalNi by 2% to 5%.

For Leco S, standards revealed Acme to be biased by less than 2%, which is acceptable, especially at those low levels.

For Co, the large relative bias is a consequence of the very low average grade level. In absolute terms, the bias between the two laboratories only amounts to 0.001% Co (i.e. a negligible quantity). Additionally, the bias, if any, is conservative.

#### 12.3.2.2 Post-2006 Data (IPL Check Assays)

The negative S bias was confirmed for Acme at around 4% to 6%. The Co biases were not confirmed; iPL may have been biased slightly lower.

Unfortunately, the lack of appreciable bias for AC Ni above 0.12% Ni was due to a common bias at both laboratories. Acme revealed itself biased on AC Ni by about 6% to 9% at the 0.20-0.237% Ni grade level. The bias below 0.12% Ni could not be checked by lack of adequate standards but is not economically relevant. The gap observed in the data around that level should nevertheless be investigated as it reveals a concerning and questionable calibration situation at Acme.

Finally, standards revealed no appreciable bias on total Ni at the 0.23-0.37% Ni grade level.

### 12.3.3 Bulk Density

The Turnagain database contains 1,214 density measurements representing all lithologies intersected in the drilling programs. AMEC (2007) carried out checks of comparable coarse rejects from pre-2007 drilling and concluded that the data was sufficiently precise and accurate to support resource estimation. AMC considers that the density results collected since 2007 are consistent with previous measurements and equally acceptable.

## 12.4 Conclusions

The data collection to date is considered acceptable to support resource estimation.

Due to an apparent high bias in Acme results for AC Ni, this item is not being used as the primary grade item in the present resource estimation.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

The Turnagain deposit is a large, low grade ultramafic deposit containing nickel and cobalt-bearing pentlandite and pyrrhotite as well as minor amounts of chalcopyrite and pyrite. It also hosts anomalous levels of platinum and palladium as well as trace amounts of silver, gold and native copper. The main economic value is in the nickel with some modest cobalt by-product credits.

The main lithological domains are: pyroxenite-dominated (101), green dunite (105), wherlite/ dunite (104) and serpentinized dunite/ wherlite (106) with these last two accounting for 39% and 53% respectively of the ultimate pit resource.

Typically the metallurgy of these deposits is challenging due to largely unrecoverable nickel in the ultramafic gangue and at the same time dilution of the flotation concentrate due to entrainment (at head grade) of this nickel-bearing silicate gangue.

A history of the metallurgical testwork conducted up until 2007 was summarized in the AMEC 2007 NI 43-101 report. The Wardrop 2010 Preliminary Assessment provided a detailed account of the more recent (2007-2010) testwork. This is summarized in Section 13.2 with particular emphasis on the key outcomes pertinent to this present update focussing on production of a saleable concentrate based on the most recent work conducted during 2010-2011.

## 13.2 Summary of Previous Testwork (2007-2010)

A summary of sample provenance and the various programs conducted is tabulated in Table 13.1

**Table 13.1 2007-2010 Testwork Programs**

| Date | Lab | Sample Provenance | Testwork |
|---|---|---|---|
| Nov 2007 | XPS | Starter – pit (yrs 1-5) Variability composites 1-16<br>Domain 104 composite A<br>Domain 106 composite B | Mineralogy<br>Preliminary flotation tests |
| Nov 2007 | SGS Lakefield | Variability composite 1-16 | Preliminary grindability characterization |
| Jan 2008 | SGS Lakefield | 2800 kg PQ core | Definitive grindability characterization |
| Mar 2008 | G&T (KM 2181) | Variability composites 1-16<br>5 additional low S samples<br>Composites A and B | Flotation amenability and reagent testwork |
| Feb 2009 | G&T (KM 2348) | PQ core crushed reject samples<br>Composites 104A, 104B, 106A, 106B<br>08-264 “starter-pit” hole | Flotation test program |
| Mar 2009<br>Jan 2010 | G&T (KM 2348), and WMT[1] | 500 kg composite C<br>(equivalent to composite A) | Flotation test program |
| Aug 2009 | G&T | 7 additional variability samples | Completion of flotation program |
| Nov 2009 | G&T | Composite C | Pilot plant test program |

[1] Western Minerals Technology Ltd, Perth Australia

The key conclusions of the two G&T programs on which the 2010 study was largely based were:

- Flotation response was highly variable, but in any case only moderate concentrate grades (7-10% Ni) at unacceptably low nickel recoveries (45%) and high MgO levels (>8%) were achieved.
- The use of a dispersant (PE26) appeared beneficial in reducing the amount of entrained gangue.
- Grinding to a finer primary grind size produced no improvement and it was also concluded there was little benefit in regrinding.
- Production of a bulk concentrate assaying 4% Ni corresponding to 55% nickel recovery appeared achievable (planned feedstock to the hydrometallurgical process route also included in the 2010 study).

As the main thrust of this previous work was aimed at producing this bulk low-grade concentrate it is not considered in any detail in this update although some of the reasons behind the flotation results are discussed in Section 13.3 in the context of the most recent testwork towards a saleable concentrate.

However, the XPS mineralogical investigations and the SGS grindability testwork are still very pertinent to this update with some key outcomes listed below:

- From the XPS mineralogy:
  - The issues reported in the 2007 AMEC report regarding ammonium citrate analysis for sulphide nickel (ACNi) being problematic in compiling metallurgical balances were confirmed; XPS found that ACNi recovery correlated poorly with both sulphur content and the percentage of nickel sulphides as determined by mineralogy. AMC's observation is that low ACNi/(NiT) ratios provide a reasonable predictor of % nickel sulphides but then at higher levels (closer to the normal range in fact) there is a very poor correlation.
  - Total nickel recovery using the standard four multi-acid digestion method was found to correlate moderately with the sulphur content.
  - Pentlandite was associated at coarser sizes mainly with pyrrhotite and magnetite whereas the associations with serpentinite and olivine were much finer. An optimum grind size of 90µ was indicated.
  - Pure pentlandite assayed 29.6% Ni (and 1.92% Co) and the next most nickel-rich mineral was the iron-rich forsterite (0.14% Ni). Pyrrhotite assayed 0.04% Ni.
  - XPS presented some thorough mineralogical examination results which have been further analysed by AMC. See Table 13.2, modified from the Wardrop report.

**Table 13.2 Nickel Deportment**

| Variable Composite | Description | Domain | %NiS | %S | %Fe | %MgO | MgO/Fe |
|---|---|---|---|---|---|---|---|
| 1 | HiNiS-HiS | 106 | 83.5 | 1.73 | 9.4 | 43.4 | 4.62 |
| 2 | HiNiS-LowS | 106 | 61.3 | 0.53 | 6.8 | 39.6 | 5.82 |
| 3 | LowNiS-HiS | 106 | 81.0 | 1.58 | 9.2 | 36.4 | 3.96 |
| 4 | LowNiS-LowS | 106 | 46.2 | 0.37 | 6.9 | 44.4 | 6.43 |
| 5 | NiNiS-HiS | 104 | 74.8 | 1.46 | 8.7 | 43.9 | 5.05 |
| 6 | HiNiS-LowS | 104 | 60.3 | 0.85 | 7.9 | 45.2 | 5.72 |
| 7 | LowNiS-HiS | 104 | 67.2 | 1.22 | 8.3 | 32.1 | 3.87 |
| 8 | LoNiS-LowS | 104 | 38.9 | 0.42 | 6.8 | 44.5 | 6.54 |
| 9 | HiNiS-LowS | 105 | 64.4 | 0.65 | 7.4 | 39.9 | 5.39 |
| 10 | LowNiS-LowS | 105 | 45.3 | 0.27 | 6.8 | 42.5 | 6.25 |
| 11 | HiNiS-HiS | 101 | 82.8 | 1.70 | 8.0 | 31.5 | 3.94 |
| 12 | LowNiS-LowS | 101 | 61.7 | 0.73 | 7.9 | 32.8 | 4.15 |
| 13 | HiNiS-HiMg | Mix | 70.5 | 1.14 | 8.0 | 40.8 | 5.10 |
| 14 | LowNiS-HiMg | Mix | 51.8 | 0.48 | 8.0 | 44.4 | 5.55 |
| 15 | HiNiS-LoMg | Mix | 60.6 | 0.7 | 6.8 | 33.1 | 4.87 |
| 16 | LowNiS-LowMg | Mix | 64.5% | 0.83 | 8.0 | 35.1 | 4.39 |

From this table some useful relationships can be derived with respect to the fundamental mineralogy and nickel deportment, in particular the percentage of nickel sulphides, as expressed by % NiS:

% NiS = 0.254 x %S + 0.401

; with a correlation coefficient $r^2 = 0.88$

There is no stoichiometric relationship behind this equation as there is generally more than enough sulphur to account for the nickel:sulphur stoichiometry. It is probably more related to the reaction kinetics associated with the nickel “scavenging” role the sulphur plays.

Also, although there was no apparent relationship between % NiS and % MgO there was a modest correlation with the ratio MgO/Fe (perhaps not surprisingly as there was a modest inverse correlation between %S and MgO/Fe) and a multiple linear regression yielded the following:

% NiS = -0.0322 x MgO/Fe + 0.210 x %S + 0.606

$r^2 = 0.90$

For resource-typical values of 5.0 +/- 0.5 for MgO/Fe and 0.65% +/- 0.1% for S then this last relationship yields %NiS values in the range 54% to 62%, giving a guide to likely maximum recoveries achievable to a sulphide concentrate. This is used in Section 13.4 to better inform the recovery predictions derived from the current testwork.

- From the SGS grindability testwork:
  - The standard grinding test results are summarized in Table 13.3

**Table 13.3 Summary of Grinding Test Results**

| Sample Domain | JKTech Drop Weight Test | | | S.G. | RWI | BWI (150µ) | Ai (g) |
|---|---|---|---|---|---|---|---|
| | A | b | Axb | | | | |
| 102 (incl in 101) | 84.3 | 0.24 | 20.2 | 2.95 | 24.9 | 23.2 | 0.290 |
| 104A | 100.0 | 0.25 | 25.0 | 3.11 | 19.6 | 17.9 | 0.295 |
| 104B | 100.0 | 0.24 | 24.0 | 3.02 | 20.6 | 20.2 | 0.402 |
| 106A | 78.0 | 0.45 | 35.1 | 2.92 | 14.5 | 16.4 | 0.084 |
| 106B | 82.4 | 0.4 | 33.0 | 3.07 | 13.6 | 13.8 | 0.132 |
| 104 (variability averages) | | | | | | *18.9*[1] | |
| 106 (variability averages) | | | | | | *19.8*[1] | |

[1] the BWI tests for the variability samples were carried out to a closing size of 106µ.

It was concluded that the mineralized rock fell into the hard to very hard category (although only moderately abrasive) and would be a potential candidate for high-pressure grinding rolls (HPGR’s). However, the reported presence of chrysotile fibres could possibly militate against the use of multi-stage crushing. At this stage a conventional SAG-ball mill circuit was preferred. AMC concurs with the above but also notes the following:

- The process risk with the vary hard mineralized rockis compounded by the size of the SAG mills contemplated (40’) so a trade-off study with HPGR’s is recommended, although a risk assessment related to the chrysotile fibres would be an essential pre-requisite prior to any HPGR testwork.
- The variability test averages for 104 and 106 lithology domains did not confirm the differential hardness between them shown by the composites and indeed showed the mineralized rock on average to have a slightly higher BWi (this may be due at least in part to the finer closing size used in the variability sample tests, sometimes an important consideration).

The key outcomes with respect to the grinding circuit design are:

- From a JKSimMet/SGS-Lakefield grinding simulation study a SAG/ Ball mill/ pebble Crusher (SABC) circuit was designed based on either two or three parallel grinding lines.
- The two-line circuit was preferred, even though it would be based on the largest SAG mill currently available (40’), and would grind an average of 86,700 tpd to a grind size of 80% passing 80µ, based on the differential 104 and 106 domain composite grinding indices and weighted for their relative proportions.
- This was subsequently revised by Wardrop to address the power balance between the SAG and ball mills and to upsize the ball mill slightly as the circuit was ball mill power-limited.
- Each line would consist of one 12.2 x 6.71m SAG mill with a 17.65 MW motor and two 7.93 x 12.5m ball mills each with two 6.6 MW motors.

AMC considers the revision by Wardrop to be prudent especially in light of the previous observation that the variability sample average BWI values were about 10% higher than the indices derived from the composites.

AMC also notes that although the BWI basis for the grinding circuit design requires additional testwork and re-evaluation, the circuit and the installed power should be adequate as the optimum grind size is likely to be coarser at 100µ, as discussed in the subsequent section. The very large SAG mill required however does re-inforce the recommendation regarding HPGR’s as an option.

## 13.3 Current Testwork Program (2010-2011)

The metallurgical breakthrough towards a saleable concentrate resulted from testwork carried out at SGS Vancouver in late 2010 and early 2011 on bulk samples from drillhole 10-265 (a twin of the earlier 08-264 drillhole) drilled horizontally through the main Horsetrail “starter pit”, and also some additional work on hole 08-264.

This work was supervised by Jake Lang, B.E.Sc of SGS Canada and the general program was directed by a metallurgical steering committee consisting of Project Metallurgist, Mike Ounpuu, Professor David Dreisinger, UBC University, Gary Johnson of Strategic Metallurgy PTY Ltd, John Hoffert, P.Eng of Hoffert Processing Services Ltd, and Chris Martin, C.Eng of Blue Coast Metallurgy Ltd. The work was reported by Mike Ounpuu in a series of three internal memos to HNC, summarized briefly below.

The "Metallurgical Progress Report from Sample 10-265 and Initial Variability Tests" reported on the initial test program consisting of ten batch flotation tests, culminating in two locked cycle tests (LCT's) the results of which are summarized in Table 13.4 below:

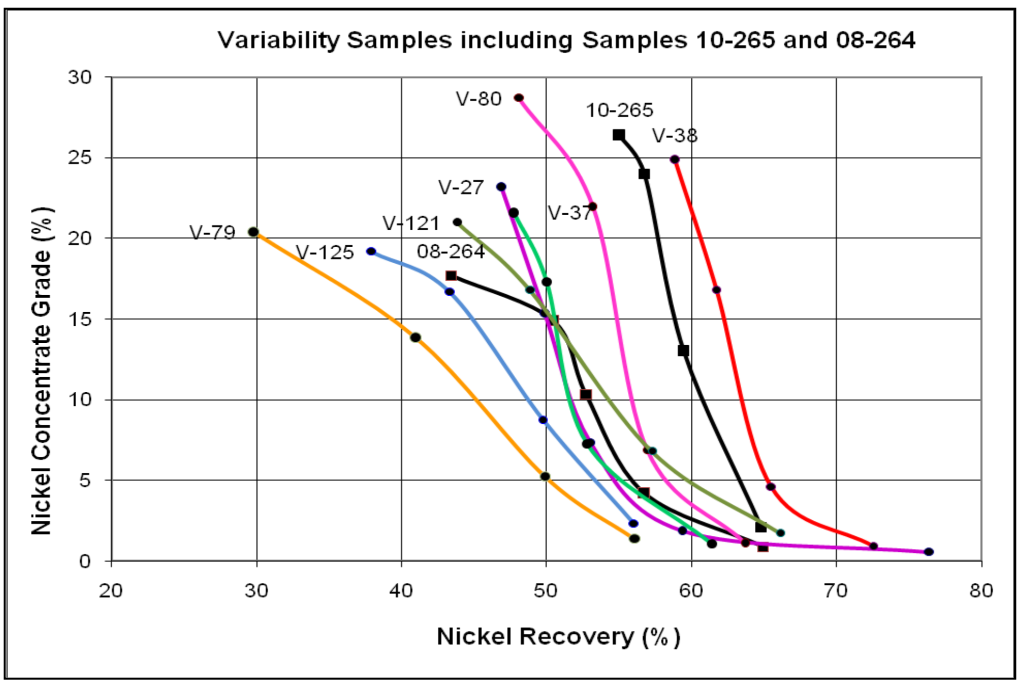
**Table 13.4 Sample 10-265 Locked Cycle Test results**

| | | Head Assay | | Concentrate | | | |
|---|---|---|---|---|---|---|---|
| **Test** | | **%Ni** | **%MgO** | **Ni % recovery** | **%Ni** | **%MgO** | **Fe:MgO** |
| LCT-1 | | 0.31 | 40.8 | 65.0 | 16.0 | 7.8 | 4.7 |
| LCT-2 (split cleaner) | Combined fast and slow float | 0.33 | 37.6 | 62.1 | 21.3 | 7.0 | 4.6 |
| | - Fast float (70% of nickel) | | | 49.7 | 25.5 | 3.4 | 9.7 |
| | - Slow float (30% of nickel) | | | 12.4 | 12.9 | 14.3 | 2.2 |

Grade-recovery curves from this data, including the batch tests, would indicate that a 20% Ni concentrate could be achieved at around 60% total nickel recovery. The curve steepens sharply indicating a cap to nickel recovery at around 65%. This is consistent with the XPS mineralogical findings reported earlier where a sample like 10-265 containing 1.15% S would likely contain 68% nickel sulphides.

Initial variability testing results (batch only) suggested that the new procedure would work reliably across the deposit to produce a nickel concentrate grade exceeding 15% Ni with nickel recoveries of 50% or higher. Figures from batch tests usually under-estimate the ultimate recovery when middlings streams are recycled. These are shown in grade-recovery terms in Figure 13.1 (from Ounpuu's report).

**Figure 13.1 Variability Batch Flotation Grade vs Recovery**



These results are significantly better than those from the previous programs reported in Section 13.2 which typically achieved 55-60% nickel recovery to 3-4%Ni concentrates with the best being two tests reporting 56.6% and 51.4% recoveries to 9.0%Ni and 11.4%Ni concentrates respectively. Achieving acceptable MgO levels may require moving along the grade recovery to close to 20% Ni (with a commensurate loss of recovery). This issue is addressed in more detail in this and the subsequent section.

The remainder of the report discussed some of the possible reasons for the improvement in metallurgical performance, and the slightly inferior results obtained for sample 08-264 compared to sample 10-265.

A partial explanation for the recent improved performance is the use of a dispersant (Calgon, hexa-metaphosphate) instead of CMC.The earlier G&T work used CMC to assist with de-agglomerating the minerals in the pulp. However some comparative tests on the effect of procedures and possible laboratory differences showed both labs and both procedures appeared to operate along similar grade-recovery curves.

One key factor identified from further comparative tests was the majority of the G&T work was carried out on 2007 crushed assay reject samples which results in poor control of sample grain size for flotation batch testing. The fineness of the assay reject samples could also be resulting in an aging effect of the samples.

A further “Progress Report on Sample 10-265 Metallurgical Testing” focussed on the inferior performance of hole 08-264 drill core composite samples compared to “fresh” hole 10-265 drill core composite samples. Various optimization exercises on this more difficult sample were also pursued. The difficulty appeared to rest in the greater tendency for sample 08-264 to create viscous/agglomerated pulps, It was later discovered the compositing had not been performed exactly as instructed at the SGS Vancouver laboratory. The sample 08-264 composite included a much longer drill hole interval than the 10-265 composite with more silicates. This was in fact validated by the lower head assay.

The key outcomes of this work were:

- Dispersants and depressants gave similar rougher performance but it was evident cleaner performance improved in the order of Cyquest 40E>Calgon>CMC>water.
- Grade-recovery (but not ultimate recovery) improved with a coarser grind (up to 125µ) lending further support to a split cleaner flowsheet with a coarse primary grind (say 100µ) to a fast-floating high grade rougher concentrate followed by a scavenger float and regrind to a lower grade concentrate (see third report on mineralogy).
- Dispersant dosages did not appear to be critical, nor did collector selection although SIBX may offer a slight improvement compared to SIPX.
- Critical were % solids, especially in the first cleaners (15% maximum recommended).

The overall conclusion was the optimization exercises on the more difficult 08-264 sample had approached the 10-265 results with application of the following key parameters:

- Maintaining high rougher mass pull to maximize initial recovery

- Chasing improved cleaner selectivity through the dispersant 40E and low % solids.

The third report "Progress Report on Process Mineralogy Examination" covered the results of the mineralogical investigations on samples from the 10-265 (LCT-1) and 08-264 (F-22 kinetic rougher test) flotation testwork. Feed and critical flotation product stream samples were examined. The key outcomes were:

- 08-264 and 10-265 samples were generally similar having 80% of the nickel as pentlandite although 08-264 did contain more pyroxenes
- Recovery of liberated pentlandite was rapid (80% in initial 1.5 minutes of flotation) and very high ultimate recoveries were achieved, hence the appeal of a split cleaner circuit to recover this liberated material as fast as possible and remove it from any agglomerating tendencies in the pulp that would impact adversely on selectivity
- However middlings pentlandite showed much lower recoveries (20% for the predominant silicate associations, 70% for the more minor pyrrhotite/magnetite associations) and may benefit from regrinding
- Pyrrhotite recovery is not required for nickel recovery

In summary these results were generally in agreement with the XPS findings although 80% nickel sulphides is higher than would be predicted from the contained % S based on the XPS work.

Some additional variability testwork not listed above was completed during 2011.A lthough this work has not been formally reported the results have been incorporated into the recovery predictions presented in Section 13.4.

In mid 2011 a Bulk Concentrate Production Testwork Program was performed, at SGS Vancouver utilizing sample 10-265. The results have been reported by Mark Urbani of Strategic Metallurgy Pty Ltd of Perth Australia. The objectives of this program were to generate sufficient concentrate mass to meet marketing requirements by repeating the locked cycle tests on a larger scale using larger laboratory flotation cells. This would also provide insight into the impacts on the concentrate grade and nickel recovery of recycling middling streams.

The key outcomes were:

- Flotation performance was slightly improved with the larger cells, showing no ill effects of scaling
- The bulk concentrate batch and locked cycle test results are summarized for comparison in Table 13.5 below

**Table 13.5 Bulk Concentrate Production Test Result**

| Test | Ni Recovery | Concentrate | | |
|---|---|---|---|---|
| | | % Ni | % MgO | Fe:MgO |
| Batch | 49.7 | 23.9 | 5.9 | 5.5 |
| Locked Cycle | 58.0 | 19.9 | 10.3 | 3.2 |

- Targeting a 20% Ni concentrate in the bulk concentrate locked cycle tests resulted in a lower quality concentrate with respect to %MgO and the Fe:MgO ratio. As stated by Urbani, this was attributed to:
  - A coarser grind for the locked cycle tests (104µ vs 86µ)
  - Pulling the cleaners too hard and/or excess reagent dosing associated with recycled streams

AMC's comment on this is as follows:

- The coarser grind did not likely contribute to lower quality concentrate. Previous work by Ounpuu and confirmed by AMC 's analysis was the ultimate grade-recovery performance was relatively insensitive to grind size up to about 100µ. Pulling rates and effects of excess reagents are effectively moving the operating point to a higher recovery-lower grade point on essentially the same grade-recovery curve and this is discussed in more detail in the next section on recovery predictions.
- It is possible the optimum concentrate grade will need to be greater than 15% Ni to meet smelter requirements of <6% MgO and >4.5 Fe/MgO in concentrate. This is discussed further in the next section.

Notwithstanding some of the concerns expressed above, this bulk concentrate test provided valuable confirmation and assurance a saleable concentrate can be produced at an acceptable recovery from the Horsetrail starter pit sample.

## 13.4 Recovery Predictions

A practical model for recovery predictions for the Turnagain deposit must consider the following:

- Given the early stage nature of this project, the overall number and representativeness of locked cycle tests are limited.
- Batch flotation tests consistently under-estimate actual recoveries. Middlings streams in cleaner tails are recycled in an industrial circuit which are not accounted for in batch tests. One means of allowing for this, which had been suggested by Oupuu is to match a cleaner concentrate grade with the recoveries one or two stages prior, i.e. 4th cleaner % Ni with 2nd cleaner nickel recovery and 3rd cleaner % Ni with 1st cleaner nickel recovery. AMC agrees this appears to correlate well with locked cycle data, where available, for the same sample and using a pair of points also gives a grade-recovery "line" to model with.
- The definitive 10-265 sample is of significantly higher grade (0.31-0.33% Ni in the various tests) compared to the resource grade of 0.21% Ni. This concern has been alleviated to a certain extent by the re-optimization of the mine plan at higher cut-off grades to provide mill feed in the critical first few years in the 0.25-0.28% Ni range. It is essential to develop some means of predicting grade-recovery performance at feed grades encompassed by the span of the resource average grade and the sample 10-265 grade.

- The results of the variability test program are "noisy" as testwork wasn't tightly enough controlled with respect to consistency of test conditions and control of grind size. It was hoped the variability test program carried out during 2011 would assist in developing feed grade relationships and in quantifying some of the key geometallurgical parameters such as host lithology, degree of alteration (especially serpentinization), and sulphur content that one would expect to impact on the metallurgy. Despite the resulting "noise" in the data some careful filtering has enabled some useful information to be extracted from the variability data especially with respect to performance at lower feed grades.
- It is challenging to derive similarly robust relationships for the concentrate %MgO levels and Fe:MgO ratios with limited test data.

The locked cycle tests provide the initial basis for predicting recoveries. The following graphs illustrate the grade-recovery performance followed by tables showing the predictions from the regression equations.

**Figure 13.2 Grade-Recovery Plots for Sample 10-265 Lock Cycle Tests**

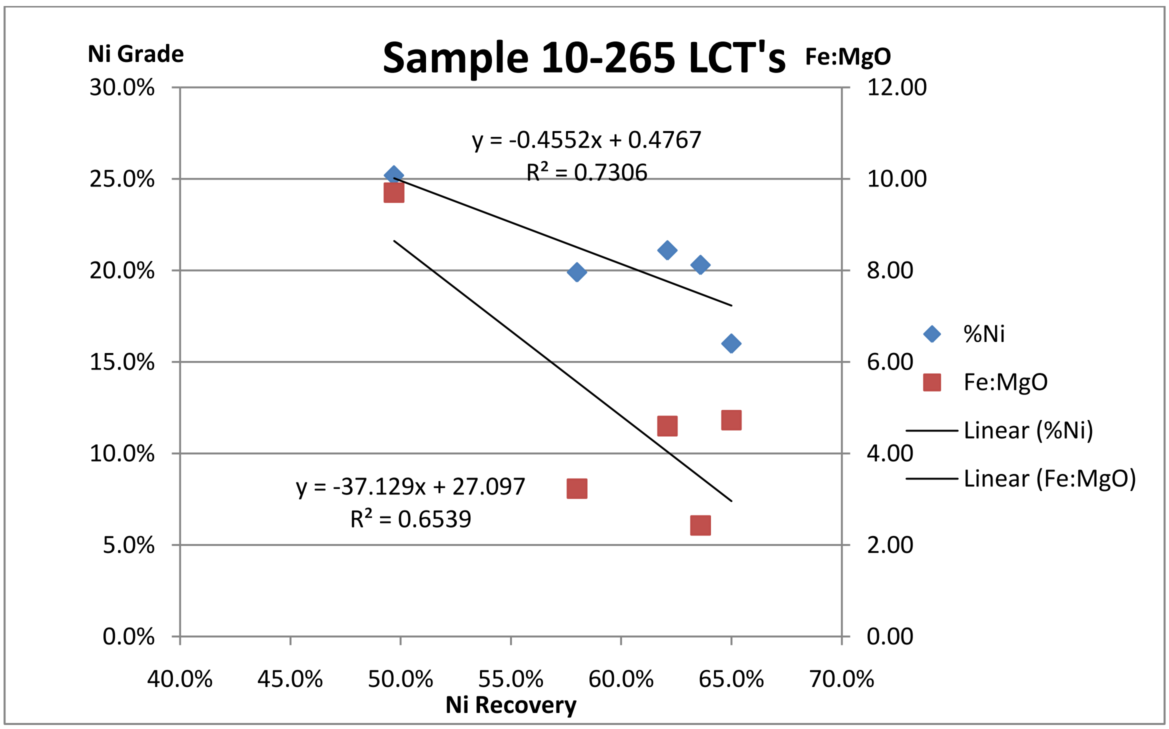



**Table 13.6 Grade-Recovery Predictions for Sample 10-265**

| Ni recovery | Conc % Ni | Conc Fe:MgO |
|---|---|---|
| 50% | 25.1% | 8.53 |
| 55% | 22.8% | 6.68 |
| 60% | 20.6% | 4.82 |

**Figure 13.3 Grade-Recovery Plot for Bulk Concentrate tests**

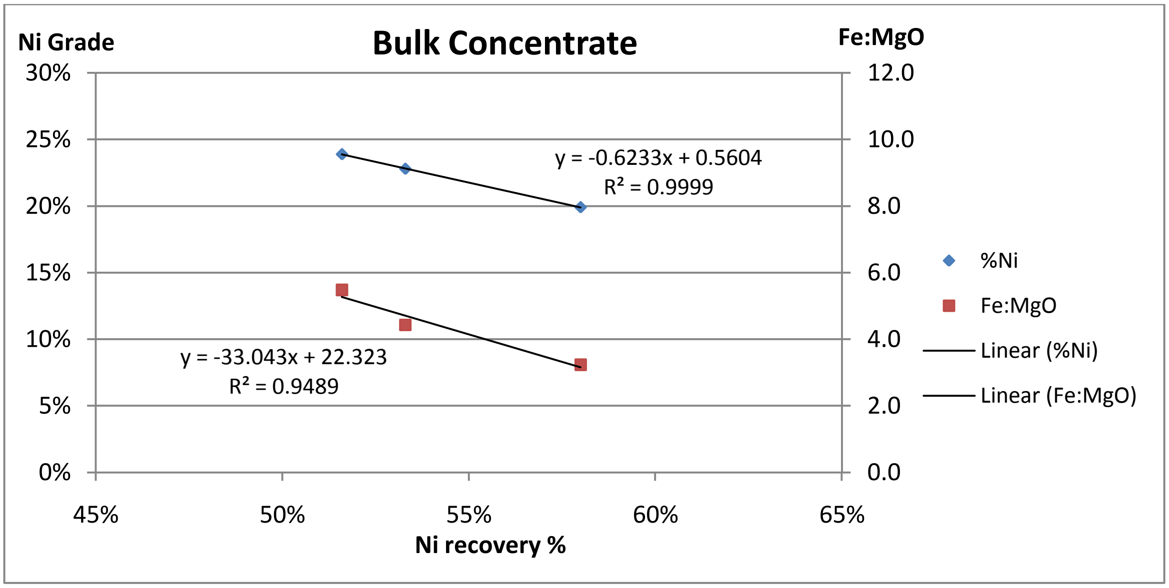



**Table 13.7 Grade-Recovery Predictions for Bulk Concentrate**

| Ni recovery | Conc % Ni | Conc Fe:MgO |
|---|---|---|
| 50% | 24.9% | 5.80 |
| 55% | 21.8% | 4.15 |
| 60% | 18.6% | 2.50 |

**Figure 13.4 Grade-Recovery Plots for Combined LCT Tests and Select Variability Tests (average 0.28% Ni)**

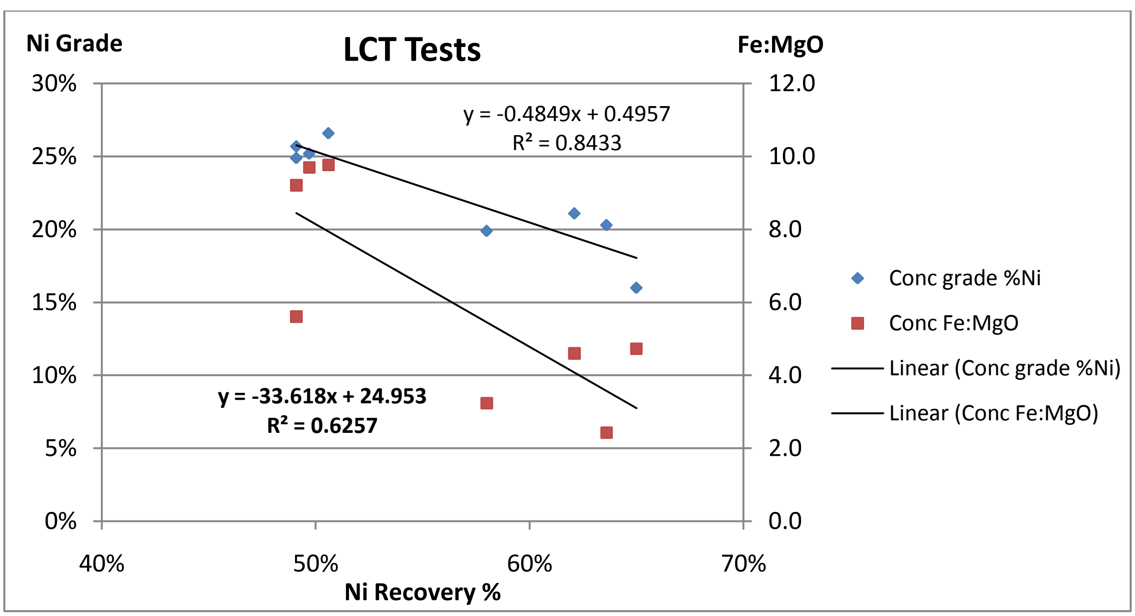


**Table 13.8 Grade-Recovery Predictions for Bulk Concentrate**

| Ni recovery | Conc % Ni | Conc Fe:MgO |
|---|---|---|
| 50% | 23.3% | 8.14 |
| 55% | 20.9% | 6.46 |
| 60% | 18.5% | 4.78 |

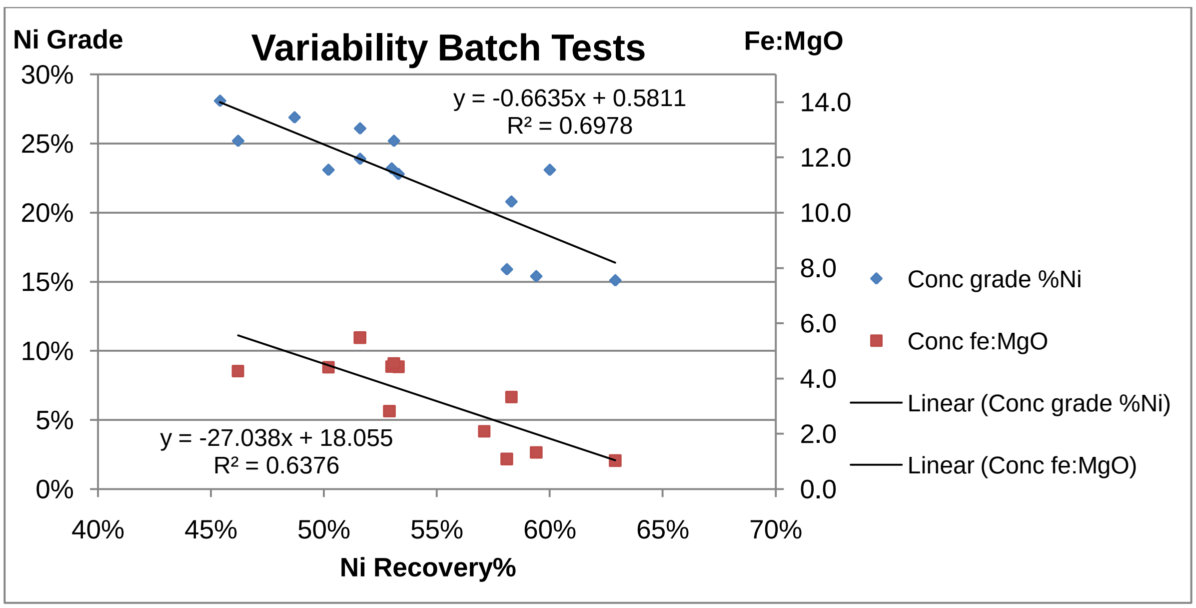
Note the LCT tests shown in Figure 13.4 and Table 13.8 include variability samples in the +/-0.25% Ni range.

Although there are moderately good correlations for nickel grade-recovery relationships and reasonable consistency across the three series, the Fe:MgO correlates less well with nickel recovery and there is a marked deterioration in the Fe:MgO ratio in the bulk concentrate result.

A similar grade–recovery plot and predictive table were developed using "filtered" variability test results. Test results were selected if the batch variability tests met the 55% recovery and 15% Ni concentrate criteria using the previously mentioned construct of matching grades with recoveries two stages prior, The selected tests covered the feed grade range of 0.2-0.3% Ni. The plot and predictive table are shown below.

Similar predictions for the nickel grade-recovery relationship were obtained as for the locked cycle tests but with inferior Fe:MgO ratios.

**Figure 13.5 Grade-Recovery Plots for Variability Batch Tests**



### Table 13.9 Grade-Recovery Predictions for Variability Batch Tests

| Ni recovery | Conc % Ni | Conc Fe:MgO |
|---|---|---|
| 50% | 24.9% | 4.54 |
| 55% | 21.6% | 3.19 |
| 60% | 18.3% | 1.83 |

At the time of setting the parameters for the pit optimizations and based primarily on the LCT data, the minimum concentrate % Ni grade to achieve acceptable Fe:MgO levels should be 18% not 15% (with sensitivity to increasing to 20% Ni) This was carried forward to the next phase of determining the relationship of the grade-recovery performance to feed grade.

The initial pit optimization work without elevated cut-off grades showed a resource block grade tightly distributed around the 0.21% Ni average (0-18-0.24% Ni range). This identifies aneed to model the grade-recovery dependence on feed grade.

In the absence of good fundamental geometallurgical data at this stage in the project, an empirical approach was adopted, recognizing as more extensive and more reliable geometallurgical data became available this empirical approach should be modified and perhaps eventually replaced by a more fundamental model.

Mineral processing separation processes can be modelled according to a general equation of the form:

**c/f = 1 + 1/k (1 – exp(k(1-r))),** referred to as the separation equation

where:

c = concentrate grade

f= feed grade (NB: %Ni(T)), i.e. c/f is the upgrade ratio

r = concentration ratio (tonnes of feed / tonnes of concentrate)

(note that recovery =upgrade ratio / concentration ratio i.e. = (c/f)/r)

k = model constant

The derivation of this equation is as follows, illustrated in the graph below too:

The c/f vs r operating line is described by an equation of the general form:

c/f = A – B exp (-kr)

A, B are constants, c,f,r are as per the list above

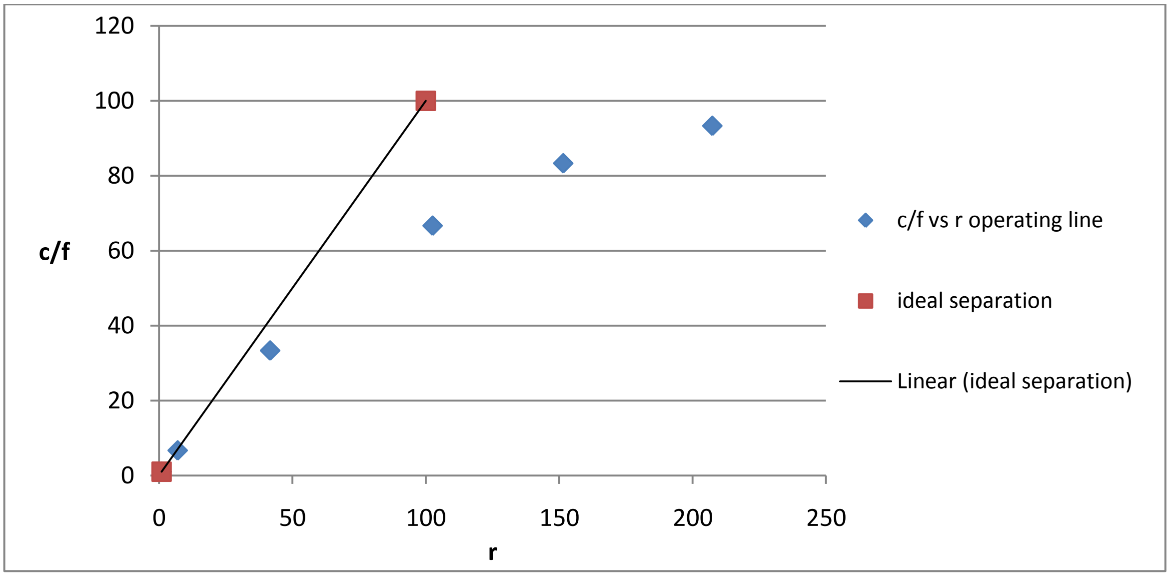
Applying the following boundary conditions:

c/f tends to 1 as r tends to 1

d(c/f)/dr tends to 1 as r tends to 1

yields the separation equation shown above

**Figure 13.6 General Form of the Separation Equation**



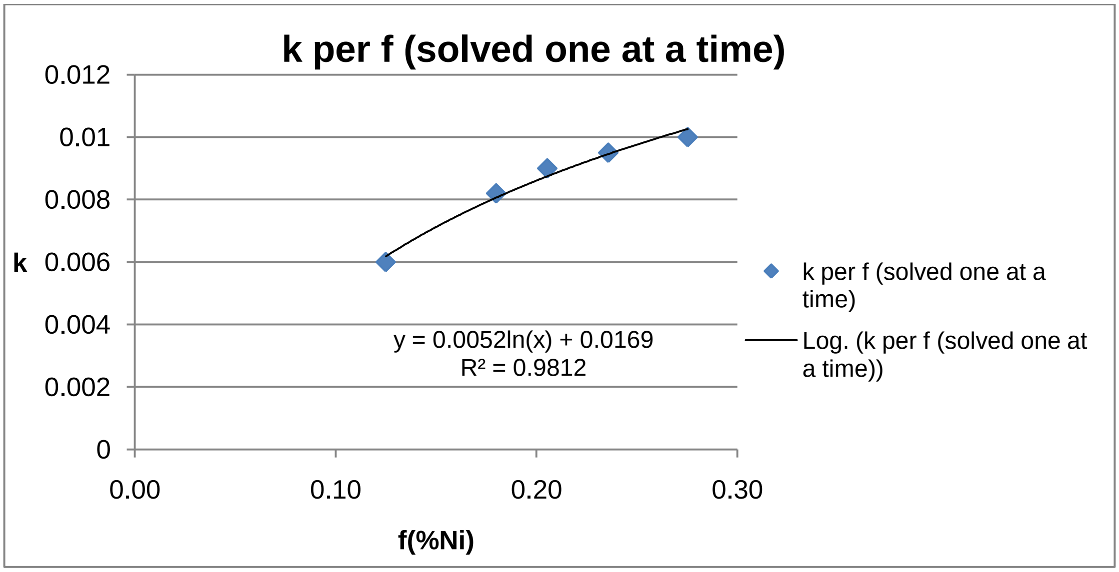

This equation is not solvable analytically, but from a body of data at various feed grades such as the grade-recovery data already derived, the model constant k can be determined iteratively by a minimization of the sum of the squares approach. The lower the value of k the better the separation i.e. on a superior grade-recovery curve. Once k is determined then the concentrate grades and recoveries at various feed grades can calculated using goal seek in Excel.

This equation has proved robust in flotation circuits (although generally with much lower upgrade ratios).

Initial attempts to model with a single k value did not provide a good fit with the locked cycle grade-recovery data; it appeared that the model under-estimated the separation efficiency at lower feed grades.

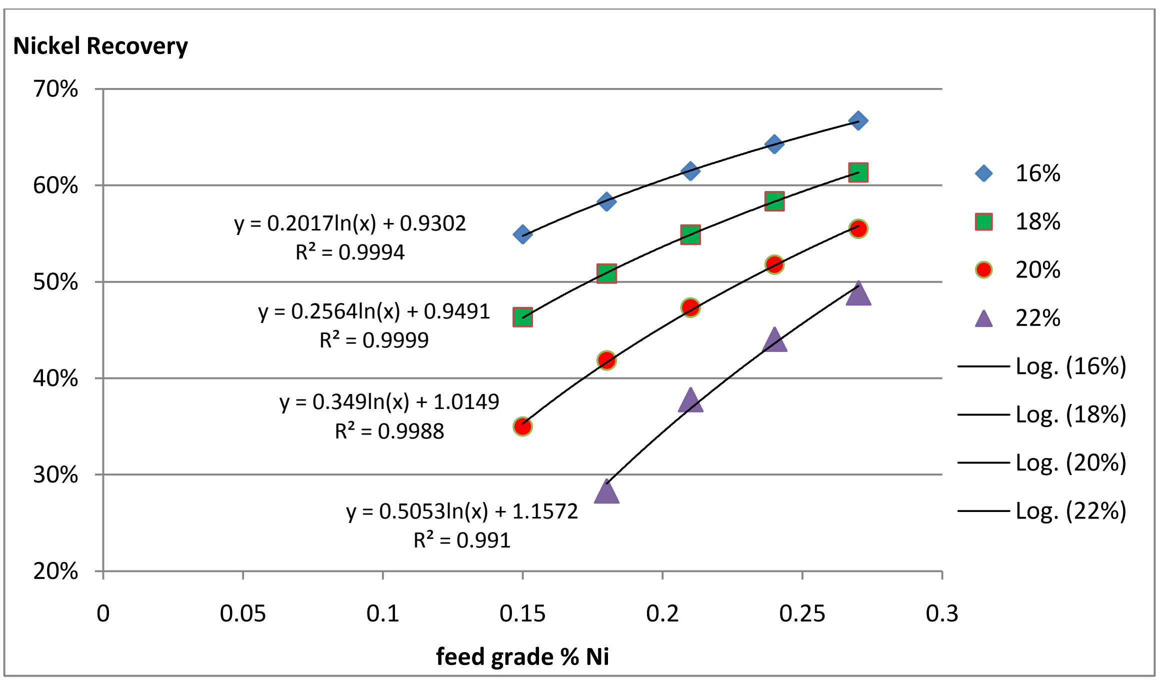
From an analysis of the variability batch data, and in conjunction with Mike Ounpuu in organising this data into grade bins each with a k value, the following relationship was derived showing how k varied with the feed grade:

**Figure 13.7 k as a Function of Nickel Feed Grade**




Applying this to an array of data with varying feed grades and a range of concentrate grades and solving the separation equation for recovery using the Excel function Goal Seek yields the graph shown below in Figure 13.8

**Figure 13.8 Nickel Recovery vs %Ni Feed Grade at Various Concentrate Grades**




From the regression equations fitted to the above curves, the following table has been constructed as a summary and also to compare with the LCT grade-recovery data previously presented, and with recoveries at a single concentrate grade of 20% Ni predicted separately by Oupuu:

### Table 13.10 Summary of Feed Grade and Concentrate Grade-Recovery Relationships

| | % Recovery to Concentrate Grade | | | | |
|---|---|---|---|---|---|
| **Feed % Ni** | **16% Ni** | **18% Ni** | **20% Ni** | **20% Ni (from Ounpuu)** | **22% Ni** |
| 0.22 | 62.5% | **56.1%** | **48.6%** | 48.3 | 39.2% |
| 0.25 | 65.1% | **59.4%** | **53.1%** | 51.5 | 45.7% |
| 0.28 | 67.3% | **62.3%** | **57.1%** | 54.7 | 51.4% |
| LCT (0.28 average) | 65.1% | **61.0%** | **56.9%** | | 52.7% |
| 0.31 | *69.4%*[1] | **64.9%** | **60.6%** | 58.0 | 56.5% |
| 10-265 (0.31) | *70.1%*[1] | **65.6%** | **61.2%** | | 56.8% |
| Bulk Conc (0.31) | 64.2% | **61.0%** | **57.8%** | | 54.6% |

[1] This value from the 10-265 LCT (and the recovery predicted at 0.31% nickel and 16% Ni conc grade) is getting into the realms of extrapolation and probably into the grade-recovery region where the slope steepens considerably as recovery asymptotes to a maximum dictated by a mineralogical characteristic like %S. This is discussed in more detail subsequently.

The separation equation has provided a robust means of modelling the variation of grade-recovery with feed grade which appears to fit well with the locked cycle test data available at the time, although the subsequent single bulk concentrate test has given slightly inferior results.

The key regression equations from the area highlighted in red in the table above and used in the pit optimizations and financial modelling are:

% Ni recovery (18%Ni) = 0.2564 x ln (% Ni (feed)) + 0.9491 (in pit optimizations)

% Ni recovery (20%Ni) = 0.349 x ln (% Ni (feed)) + 1.0149 (financial model sensitivity)

These equations have been useful in the pit optimizations, not only in applying some correction to recoveries for the lower average feed grade, but also in bringing in a sensitivity to feed grade that “forces” the optimization to chase higher feed grades with higher recoveries. The models are empirical and have a limited range of application and in particular there needs to be some recognition of phenomenological constraints e.g. from the mineralogy or other assay components like sulphur.

It had been hoped the correlation between %S and the mineralogy from the XPS work mentioned earlier could have been extended from the recent variability work. However not only was the data noisy but there were some instances e.g. sample V-5 where a very low sulphur sample (0.26%S) gave nickel recoveries to acceptable grades that were outside not only the expected % Ni sulphide content from the correlation but also beyond stoichiometric constraints too, assuming nickel as pentlandite. Clearly other factors were at play, either some issues with sulphur assays or the nickel occurring e.g. as millerite.

It is also prudent to apply some constraint on the modelling to recognise the importance of the sulphur content. It has so far not proved possible to derive a satisfactory algorithm from the recent data, however what has been applied is a cap to recovery dictated by the %

nickel sulphides as calculated from the in-pit resource average sulphur grade. At an average 0.69% S in the pit, the cap is set at 58% nickel recovery. This has not been used in the pit optimizations but has been applied in the financial model. The impact (discussed also in the financial evaluation section of this report) is to reduce the modelled recovery in the critical first 5 years from 60.5% to perhaps a more realistic (given the status of the testwork and this study) 58%, but without changing the logic of the pit optimizations to favour higher grade mineralized rock closer to the grades tested and with recoveries in which there is a higher degree of confidence.

Tthere is reasonable confidence that an 18% Ni concentrate can be produced at acceptable recoveries (58% in first 5 years, 56.4% life of mine) based on the testwork to date and the recovery modelling described.

Outputs from the financial model (Section 22) are summarized in Table 13.11:

**Table 13.11 Nickel Recoveries over L.o.M. and at 18% and 20% Ni Concentrate Grades**

| | L.o.M* (Incl mining of stockpiles) | Yrs 1-5 | Yrs 6-21 |
|---|---|---|---|
| **% Ni feed** | 0.230 | 0.261 | 0.246 |
| **Base Case Recovery (18% Ni concentrate)** | 56.4 | 58.0 | 57.7 |
| **Sensitivity (20% Ni concentrate)** | 50.6 | 54.6 | 52.5 |

## 13.5 Concentrate Quality

Clearly the quality of the concentrate produced from Turnagain and its acceptability to potential smelter customers is a critical item.

The analysis in the previous section has demonstrated a high grade (in % Ni terms) concentrate can be produced at an acceptable recovery. Indeed an 18% Ni concentrate would be considered highly desirable in the market. However the other key factor in concentrate quality is the % MgO levels and especially Fe:MgO ratios as these are critical to the smelter slag chemistry. Conventional smelters require MgO typically in the 5% range with a maximum of 8% and Fe:MgO ratios > 4.5. This last parameter has been one mainly used to evaluate the Turnagain concentrate quality and, as already explained, was the basis to stipulate a minimum 18% Ni concentrate grade.

Taking into account previous discussion on the mineralogy, the concept of the split cleaner circuit should be vigorously pursued. The results indicate that 80% of the recoverable nickel can be recovered to a fast floating initial clean concentrate and the remainder recovered in a rougher-scavenger circuit plus possible regrind to a lower grade higher %MgO concentrate. Although these will probably be blended back together as a single final concentrate, the advantage of the split cleaner circuit is in separating as soon as possible the high grade material from the potentially deleterious pulp chemistry and then having conditions tailored to the recovery of the more difficult material.

### 13.6 Conclusions and Recommendations

- The testwork of 2007-2010 had been unsuccessful in producing a saleable concentrate and is believed to be due to having worked with aged crushed sample rejects as well as to not having fully pursued the use of dispersants. Nevertheless some of the outcomes are still relevant:
  - From the XPS mineralogy work from 2007 some relationships between % nickel sulphides and %S and MgO/Fe ratios have been derived which should be further investigated in the next phases of study, but which have still been valuable in "capping" recent recovery model predictions.
  - Although ACNi assays were found by XPS and Wardrop to be unreliable as a predictor of nickel recovery, there also is scope for further investigations into improving their accuracy and their usefulness.
  - The grinding testwork and circuit design based on a conventional SABC circuit to treat 87000 tpd is believed to be reasonable, although the mineralized rock being categorized hard to very hard and the size of the SAG mill both point to HPGR technology as being a potentially viable alternative, subject to satisfactory resolution of any chrysotile fibre issues.
- The current round of testwork has produced the following key outcomes:
  - High grade (+/-20% Ni) concentrates at recoveries close to 60% appear achievable from the 10-265 hole drilled through the Horsetrail starter pit, using Calgon as a dispersant.
  - Although the hole 08-264 in the same location initially gave inferior results, similar performance to 10-265 was obtained through maximizing rougher mass pulls and improved cleaner selectivity with the dispersant Cyquest 40E and low % solids.
  - Mineralogical studies largely confirmed the findings of the earlier XPS studies and also indicated the merit in pursuing a split cleaner concept to recover fast-floating liberated pentlandite separately from the more difficult slower floating, but ultimately still recoverable middlings material.
  - A bulk concentrate production test confirmed the flotation performance on a larger scale.
- Based on the locked cycle test data supplemented by batch data adjusted to reflect the impact on recovery of recycling middling streams as detailed in section 13.4, some recovery modelling and predictions have been carried out in order to: a) address concerns regarding the resource grade vs sample grade, and b) provide a mechanism for the pit optimizations to preferentially select higher grade blocks early in the mine life ( as well as the softer 106 lithology domain) and improve the project NPV.
- A minimum concentrate grade of 18% Ni will be required to meet smelter requirements for %MgO and Fe/MgO ratios and at that grade a L.o.M recovery of 56.4% is predicted with 58% (capped to allow for expected correlation between %S and % nickel sulphides as a constraint on recoverable nickel) in the first 5 years.
- Based on both the mineralogy and the testwork results it is strongly recommended that the split cleaner concept be pursued in the next phase of study in the interests

of optimizing concentrate grade whilst still focussing on maximizing recovery of some of the more difficult to float material.

- Some key elements for the next phase of study include:
  - Review assaying procedures and establish a sound platform of reliable assay protocols e.g. ACNi, %S, or other methods to quantify % nickel sulphides
  - Review mineralization characterization e.g. serpentinization
  - Review and if necessary revise mineralization - type definition
  - Review and if necessary revise laboratory procedures
  - Establish metallurgical response for each mineralization-type and identify the key geometallurgical drivers for each mineralization type
  - Conduct additional grindability tests (review appropriateness of the various methods) per mineralization-type
  - Then, and only then, conduct variability testing to investigate mineralization variability testing (spatially and temporally with respect to a preliminary mine plan)
  - Revise pit optimizations with additional geometallurgical inputs (not only hardness) and extend optimization exercise to the full Whittle Enterprise Optimization process (including large capital expense items such as the tailings dam)
  - Confirm mineralization response, flowsheet and key design criteria on some early years production composites

# 14 MINERAL RESOURCE ESTIMATES

## 14.1 Exploratory Data Analysis

The current database for the Turnagain Nickel Deposit consists of 25,308 analyzed intervals in 273 drill holes representing 70,570 metres of core. Forty-seven holes drilled prior to 2002 were excluded as it was mostly small diameter core, and sampling was incomplete. Furthermore, the areas drilled prior to 2002 have been covered reasonably well by later drill programs. Composited data from 204 drill holes were used directly for block grade estimation comprising 20,542 assayed intervals (57,746m).

In accordance with recommendations made by AGORATEK (2011) the following adjustments were made to database assays prior to resource estimation:

**Table 14.1 Adjustment Factors Applied to Historic Analyses**

| | Year of Analysis | | | |
|---|---|---|---|---|
| | **2004** | **2005** | **2006** | **2007 and later** |
| **Ni TOT** | x 0.933 | x 0.941 | x 0.972 | unchanged |
| **Co TOT** | x 0.910 | x 0.910 | x 0.939 | unchanged |
| **S Leco** | unchanged | unchanged | unchanged | 1.059* |

** Since June 2006*

AGORATEK (2011) also recognized reliability problems with the ammonium citrate analyses so AC Ni and AC Co are no longer being used in the resource estimate.

The descriptive statistics for the adjusted analyzed intervals within the main lithologic domains used in the present resource model are shown in Tables 14.2 to 14.4.

**Table 14.2 Descriptive Statistics of Adjusted Raw Assay Data (Ni, Co, S)**

| Domain | Pyroxenites (101) | | | Dunite Wehrlite (104) | | | Green Dunite (105) | | |
|---|---|---|---|---|---|---|---|---|---|
| **Variable** | **%Ni** | **%Co** | **% S** | **%Ni** | **%Co** | **% S** | **%Ni** | **%Co** | **% S** |
| Count | 1657 | 1657 | 1657 | 16796 | 16792 | 16861 | 1116 | 1115 | 1117 |
| Min | 0.002 | 0.001 | 0.01 | 0.001 | 0.001 | 0.010 | 0.001 | 0.006 | 0.01 |
| Max | 5.148 | 0.149 | 26.33 | 3.926 | 0.156 | 21.900 | 0.972 | 0.032 | 2.83 |
| Mean | 0.140 | 0.011 | 1.81 | 0.216 | 0.013 | 0.834 | 0.252 | 0.013 | 0.18 |
| Median | 0.122 | 0.010 | 1.36 | 0.214 | 0.013 | 0.520 | 0.252 | 0.013 | 0.09 |
| Variance | 0.034 | 0.000 | 3.05 | 0.012 | 0.000 | 1.236 | 0.004 | 0.000 | 0.06 |
| Std Dev | 0.185 | 0.007 | 1.75 | 0.111 | 0.006 | 1.112 | 0.061 | 0.002 | 0.25 |
| COV | 1.326 | 0.626 | 0.97 | 0.516 | 0.428 | 1.333 | 0.244 | 0.161 | 1.40 |

**Table 14.3 Descriptive Statistics of Raw Assay Data (Fe and Mg)**

| Domain | Pyroxenites (101) | | Dunite Wehrlite (104) | | Green Dunite (105) | |
|---|---|---|---|---|---|---|
| **Variable** | **%Fe** | **%Mg** | **%Fe** | **%Mg** | **%Fe** | **%Mg** |
| Count | 1864 | 1864 | 17237 | 17237 | 1496 | 1496 |
| Min | 2.86 | 2.15 | 0.01 | 0.01 | 2.15 | 0.68 |
| Max | 41.76 | 30.76 | 48.65 | 36.05 | 12.05 | 33.9 |
| Mean | 9.19 | 17.26 | 8.16 | 22.92 | 7 | 26.52 |
| Median | 8.78 | 17.26 | 7.95 | 24.47 | 6.62 | 26.74 |
| Variance | 6.79 | 20.37 | 4.47 | 32.6 | 1.99 | 6.21 |
| Std Dev | 2.61 | 4.51 | 2.11 | 5.71 | 1.41 | 2.49 |
| COV | 0.28 | 0.26 | 0.26 | 0.25 | 0.202 | 0.094 |

**Table 14.4 Descriptive Statistics of Raw Assay Data (Au, Pt, Pd)**

| Domain | Pyroxenites (101) | | | Dunite Wehrlite (104) | | | Green Dunite (105) | | |
|---|---|---|---|---|---|---|---|---|---|
| **Variable** | **ppb Au** | **ppb Pt** | **ppb Pd** | **ppb Au** | **ppb Pt** | **ppb Pd** | **ppb Au** | **ppb Pt** | **ppb Pd** |
| Count | 1649 | 1648 | 1648 | 15304 | 15304 | 15304 | 1344 | 1344 | 1344 |
| Min | 0.5 | 0.5 | 1 | 0.5 | 1 | 1 | 1 | 1 | 1 |
| Max | 607 | 1067 | 964 | 3041 | 1067 | 964 | 344 | 615 | 306 |
| Mean | 5 | 14 | 15 | 5 | 21 | 23 | 4 | 19 | 20 |
| Median | 2 | 7 | 8 | 2 | 12 | 13 | 2 | 10 | 11 |
| Variance | 267 | 1415 | 1526 | 951 | 1266 | 1143 | 149 | 1211 | 851 |
| Std Dev | 16 | 38 | 39 | 31 | 36 | 34 | 12 | 35 | 29 |
| COV | 4 | 3 | 3 | 6 | 2 | 1 | 3 | 2 | 1 |

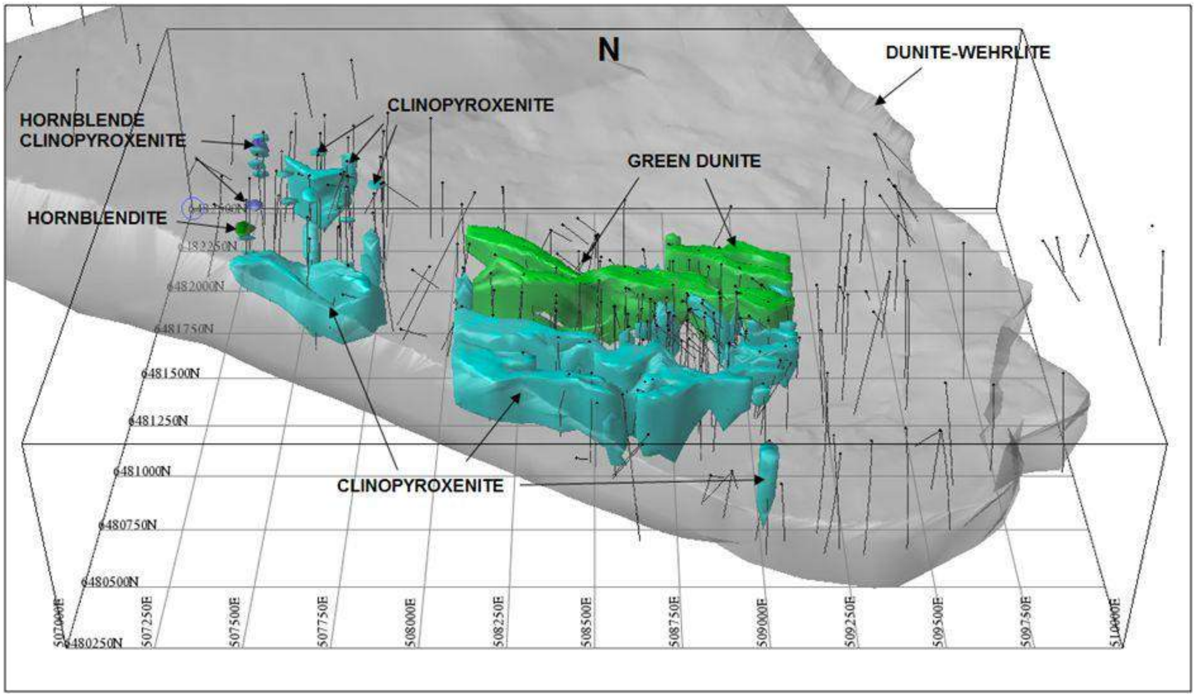
## 14.2 Lithologic Model

The previous resource model (AMEC, 2007) used geologic domains determined by nearest neighbor interpolation of drill hole lithology. Company geologic staff has re-interpreted the lithology in cross section following the 2008 drill program and created solid models of the major domains. The main domains were coded as shown in the following table and illustrated in Figure 14.1. These domains were used as hard boundaries to constrain grade estimation in the model.

**Table 14.5 Model Lithologic Domain Codes**

| Code | Lithology | SG |
|---|---|---|
| 101 | cPx/ ocPx | 3.16 |
| 104 | DuWh | 3.11 |
| 105 | gDu | 3.12 |
| 106 | Sp/SpDI/SpWh | 3.00 |
| 107 | Dk | 3.20 |
| 108 | MSD | 2.95 |

**Figure 14.1 Model Lithologic Domains**




Serpentinization, which affects all primary lithologies, was modeled using indicator kriging and blocks were assigned a value between 0 and 1. This value represents the estimated percent of the block within the serpentinite domain (code 106) where 0 is absent and 1 is 100%. Blocks with a value exceeding 0.5 were assigned to this domain. It should be noted that this does not represent an absolute degree of serpentinization since the domain includes partially serpentinized dunite and wehrlite. The degree of serpentinization was not used as a factor in grade estimation but blocks coded as 106 were assigned a lower density (SG) value of 3.0.

## 14.3 Density

Bulk density measurements were carried out on 1184 core samples collected between 2004 and 2006. Bulk density of core samples was measured in the field by the immersion method. A piece of whole core up to 50 cm in length was weighed in air and in water and the density calculated using the following formula:

Density = weight in air/ ( weight in air – weight in water)

As part of the metallurgical test program, Process Research Associates Ltd. (PRA), measured bulk density using the pycnometric method with -10 Tyler mesh assay rejects. Their results were within 5% of density determinations measured by ACME Laboratory in 2007 using the same method. A total of 810 measurements were done by the immersion method on whole core and 312 measurements carried out on crushed samples using the pycnometric procedure.

Density was assigned to the model blocks based on the median value for the corresponding lithology as listed in Table 14.5. Blocks that were estimated to be >50% within the serpentinite domain were assigned a Bulk Density of 3.0 t/m$^3$.

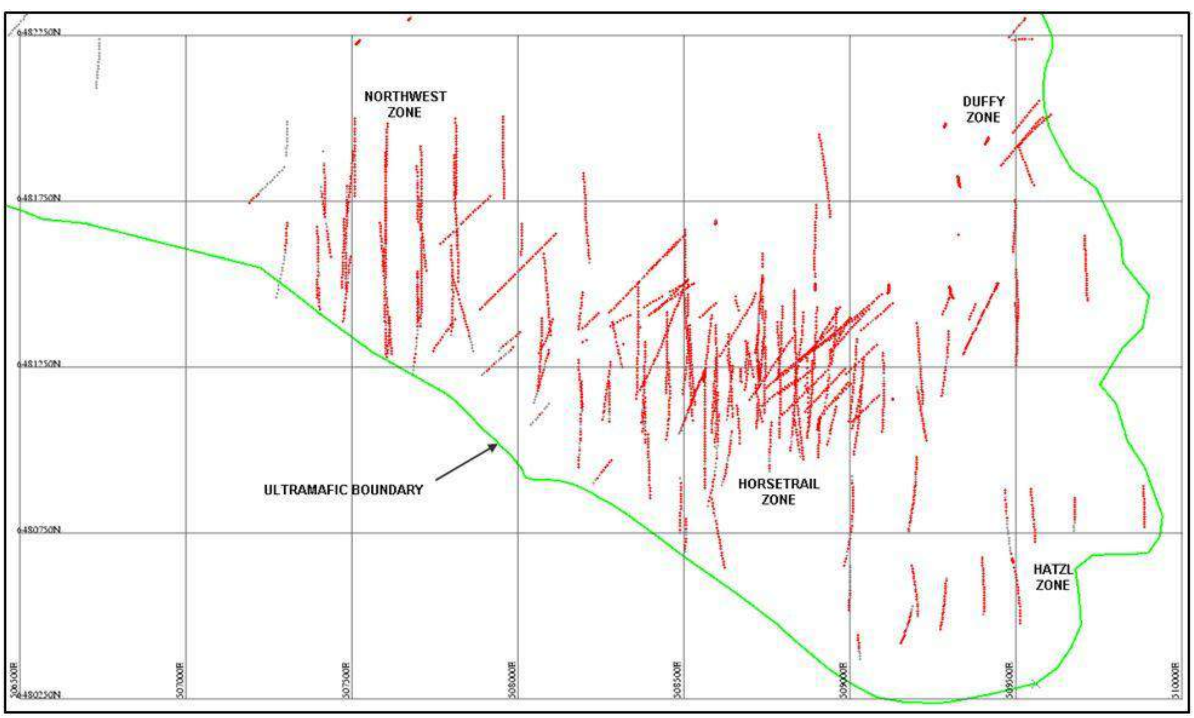
## 14.4 Compositing

Drill hole assays within the separate domains were composited downhole at intervals of 15m honoring domain boundaries. Partial composites were allowed if they were at least 7.5 metres in length. Composites from 204 drill holes were used in the final resource estimation. Table 14.6 shows the statistics by domain for AC-Ni, Ni and Co. Composite location is illustrated in Figure 14.2.

**Table 14.6 Composite Statistics for Ni, Co and S by Domain**

| Variable | Ni % | | | Co % | | | S % | | |
|---|---|---|---|---|---|---|---|---|---|
| **Domain** | **101** | **104** | **105** | **101** | **104** | **105** | **101** | **104** | **105** |
| Count | 298 | 3206 | 223 | 298 | 3206 | 223 | 298 | 3206 | 223 |
| Min | 0.01 | 0.00 | 0.14 | 41 | 10 | 100 | 0.23 | 0.01 | 0.01 |
| Max | 0.94 | 0.68 | 0.46 | 298 | 431 | 184 | 6.19 | 7.05 | 1.17 |
| 1st Quartile | 0.09 | 0.17 | 0.23 | 88 | 110 | 120 | 0.9425 | 0.22 | 0.04 |
| Median | 0.13 | 0.22 | 0.25 | 102 | 127 | 126 | 1.415 | 0.5 | 0.09 |
| 3rd Quartile | 0.16 | 0.26 | 0.27 | 118 | 148 | 134 | 2.0725 | 0.99 | 0.19 |
| Mean | 0.13 | 0.21 | 0.25 | 107 | 131 | 128 | 1.61 | 0.73 | 0.16 |
| Variance | 0.01 | 0.01 | 0.00 | 1087 | 1516 | 177 | 0.88 | 0.55 | 0.04 |
| Std Dev | 0.09 | 0.08 | 0.04 | 33 | 39 | 13 | 0.94 | 0.74 | 0.19 |
| COV | 0.696 | 0.374 | 0.160 | 0.307 | 0.297 | 0.104 | 0.582 | 1.023 | 1.189 |

**Figure 14.2 Plan View of Composites Grading ≥ 0.10% Ni Shown in Red**



## 14.5 Extreme Grades

Most elements, except for sulphur, have highly symmetrical, low skewed distributions, with moderate to low COVs; because of this, it is considered that extreme grades are not a cause of concern for interpolation, and it was decided not to perform a high-grade capping (or top-cutting) on any of the elements for all rock types. AMEC (2007) used an outlier restriction or "dynamic capping" in order to limit the influence of higher grade composites over longer search distances. This was also deemed an appropriate procedure for the present model update and similar methodology was incorporated.

## 14.6 Variography

Directional semi-variograms were modeled for all elements in the principal dunite-wehrlite lithologic domain (code 104) in order to have sufficient samples to define the structures. The parameters were then used in separate kriging runs using hard boundaries for each domain.

For Ni, a nested spherical model was obtained with a maximum range of 250m and strong anisotropy in the order of 3:1. For Co a single spherical structure was modelled with a range of 110m. Table 14.7 shows the variogram models for these and the additional elements.

### Table 14.7 Semi-Variogram Models

| Item | Domain | Axis | direction | co | c1 | a1 | c2 | a2 |
|---|---|---|---|---|---|---|---|---|
| **Ni** | **104 Dun/Whl** | major | 0->105 | 0.0007 | 0.0035 | 100 | 0.002 | 250 |
| | | s-major | -80->015 | 0.0007 | 0.0035 | 100 | 0.002 | 250 |
| | | minor | 10->015 | 0.0007 | 0.0035 | 30 | 0.002 | 90 |
| **Cu** | **104 Dun/Whl** | major | 0->105 | 7731 | 45030 | 72 | 18603 | 145 |
| | | s-major | -80->015 | 7731 | 45030 | 72 | 18603 | 145 |
| | | minor | 10->015 | 7731 | 45030 | 30 | 18603 | 60 |
| **Co** | **104 Dun/Whl** | major | 0->105 | 257 | 1230 | 110 | | |
| | | s-major | -80->015 | 257 | 1230 | 110 | | |
| | | minor | 10->015 | 257 | 1230 | 50 | | |
| **Fe** | **104 Dun/Whl** | major | 74->144 | 0.422 | 1.632 | 105 | | |
| | | s-major | 12->283 | 0.422 | 1.632 | 90 | | |
| | | minor | -10->195 | 0.422 | 1.632 | 60 | | |
| **Mg** | **104 Dun/Whl** | major | 77->153 | 2.1 | 4.179 | 57.4 | 9.505 | 193 |
| | | s-major | 11->295 | 2.1 | 4.179 | 35 | 9.505 | 120 |
| | | minor | -8->206 | 2.1 | 4.179 | | 9.505 | |
| **S** | **104 Dun/Whl** | major | 0->105 | 0.201 | 0.507 | 110 | | |
| | | s-major | -80->015 | 0.201 | 0.507 | 110 | | |
| | | minor | 10->015 | 0.201 | 0.507 | 80 | | |
| **Pt/Pd** | **104 Dun/Whl** | major | 0->105 | 31.5 | 184.1 | 50 | 245.2 | 120 |
| | | s-major | -80->015 | 31.5 | 184.1 | 110 | 245.2 | 120 |
| | | minor | 10->015 | 31.5 | 184.1 | 80 | 245.2 | 75 |
| **Au** | **104 Dun/Whl** | major | 0->105 | 0.172 | 0.184 | 100 | | |
| | | s-major | -80->015 | 0.172 | 0.184 | 100 | | |
| | | minor | 10->015 | 0.172 | 0.184 | 50 | | |

## 14.7 Grade Estimation

A block model with dimensions of 15x15x15 metres was created using Gemcom Surpac Vision© software. The block size was decreased in the x and y directions from the previous model in order to achieve more accurate definition between the dunite-wehrlite and the narrower clinopyroxenite and green dunite domains. Extents of the model are shown in the following table:

### Table 14.8 Block Model Extents

| | Min | Max | Extent | Size | Number |
|---|---|---|---|---|---|
| x | 507000 | 510000 | 3000 | 15 | 200 |
| y | 6480250 | 6482530 | 2280 | 15 | 152 |
| z | 600 | 1350 | 750 | 15 | 50 |

Blocks were estimated by ordinary kriging in three passes. Domain envelopes were treated as hard boundaries for all items. Search parameters for the various items estimated are summarized in the following table.

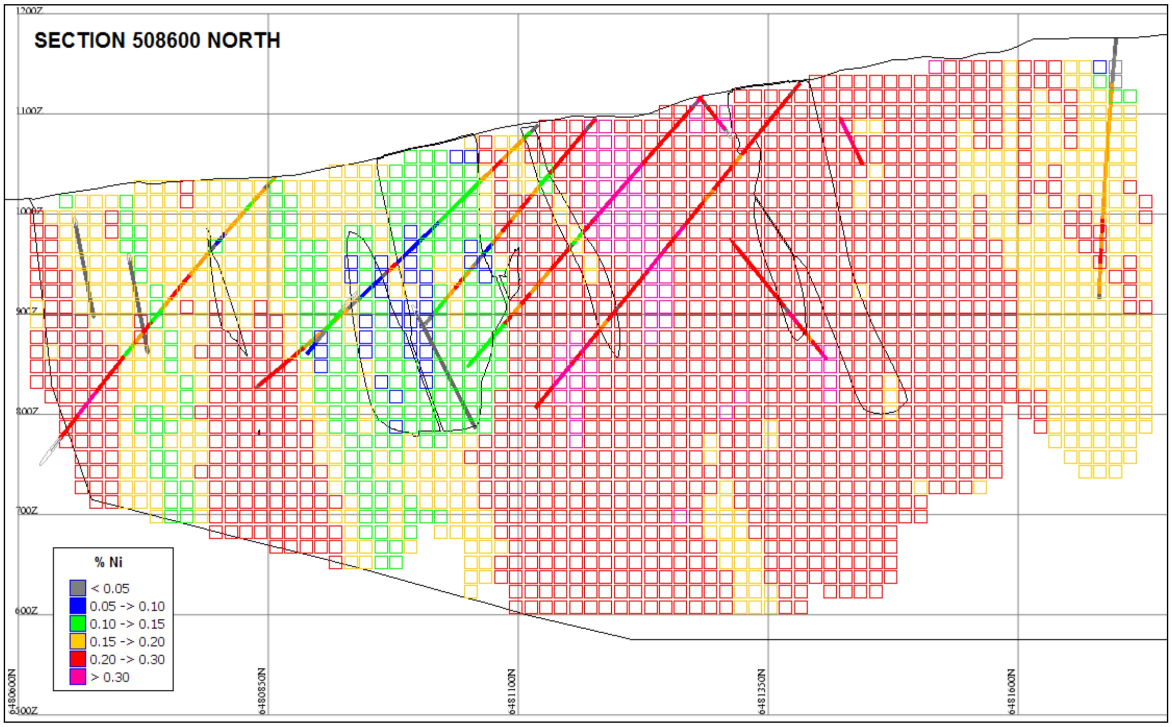
## Table 14.9 Block Model Search Parameters

| Grade Item | Pass | Search Distance | min comp | max comp | Max Comp / Hole | Topcut Level | | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | 101 DuWh | 104 cPx/ocPx | 105 gDu |
| AC-Ni | 1 | 80 | 3 | 12 | 2 | 0.49 | - | - |
| | 2 | 170 | 3 | 8 | 2 | 0.49 | 0.41 | 0.32 |
| | 3 | 250 | 3 | 8 | 2 | 0.28 | 0.37 | 0.27 |
| Ni | 1 | 80 | 3 | 12 | 2 | 0.51 | - | - |
| | 2 | 170 | 3 | 8 | 2 | 0.51 | 0.46 | 0.39 |
| | 3 | 250 | 3 | 8 | 2 | 0.33 | 0.41 | 0.36 |
| Cu | 1 | 80 | 3 | 12 | 2 | 0.11 | 0.12 | 0.07 |
| | 2 | 170 | 3 | 8 | 2 | 0.11 | 0.12 | 0.07 |
| | 3 | 250 | 3 | 8 | 2 | 0.09 | 0.09 | 0.05 |
| Co | 1 | 80 | 3 | 12 | 2 | 0.025 | 0.026 | 0.02 |
| | 2 | 170 | 3 | 8 | 2 | 0.025 | 0.026 | 0.02 |
| | 3 | 250 | 3 | 8 | 2 | 0.021 | 0.023 | 0.019 |
| Pt | 1 | 80 | 3 | 12 | 2 | 107 | 98 | 109 |
| | 2 | 170 | 3 | 8 | 2 | 107 | 98 | 109 |
| | 3 | 250 | 3 | 8 | 2 | 65 | 82 | 78 |
| Pd | 1 | 80 | 3 | 12 | 2 | 120 | 107 | 98 |
| | 2 | 170 | 3 | 8 | 2 | 120 | 107 | 98 |
| | 3 | 250 | 3 | 8 | 2 | 64 | 85 | 77 |
| Au | 1 | 80 | 3 | 12 | 2 | 25 | 33 | 36 |
| | 2 | 170 | 3 | 8 | 2 | 25 | 33 | 36 |
| | 3 | 250 | 3 | 8 | 2 | 22 | 21 | 21 |
| S | 1 | 80 | 3 | 12 | 2 | - | - | - |
| | 2 | 170 | 3 | 8 | 2 | - | - | - |
| | 3 | 300 | 3 | 8 | 2 | - | - | - |
| Fe | 1 | 80 | 3 | 12 | 2 | - | - | - |
| | 2 | 170 | 3 | 8 | 2 | - | - | - |
| | 3 | 300 | 3 | 8 | 2 | - | - | - |
| Mg | 1 | 80 | 3 | 12 | 2 | - | - | - |
| | 2 | 170 | 3 | 8 | 3 | - | - | - |
| | 3 | 350 | 3 | 8 | 3 | - | - | - |

Partial block weighting was not used in this model. Only blocks with ≥50% of their volume within a domain were estimated.

Figures 14.3 to 14.8 illustrate the block grade distributions in plan and section views.

### Figure 14.3 Section 508600 E - Ni Grades in Composites and Block Model




### Figure 14.4 Section 508600 E - Co Grades in Composites and Block Model

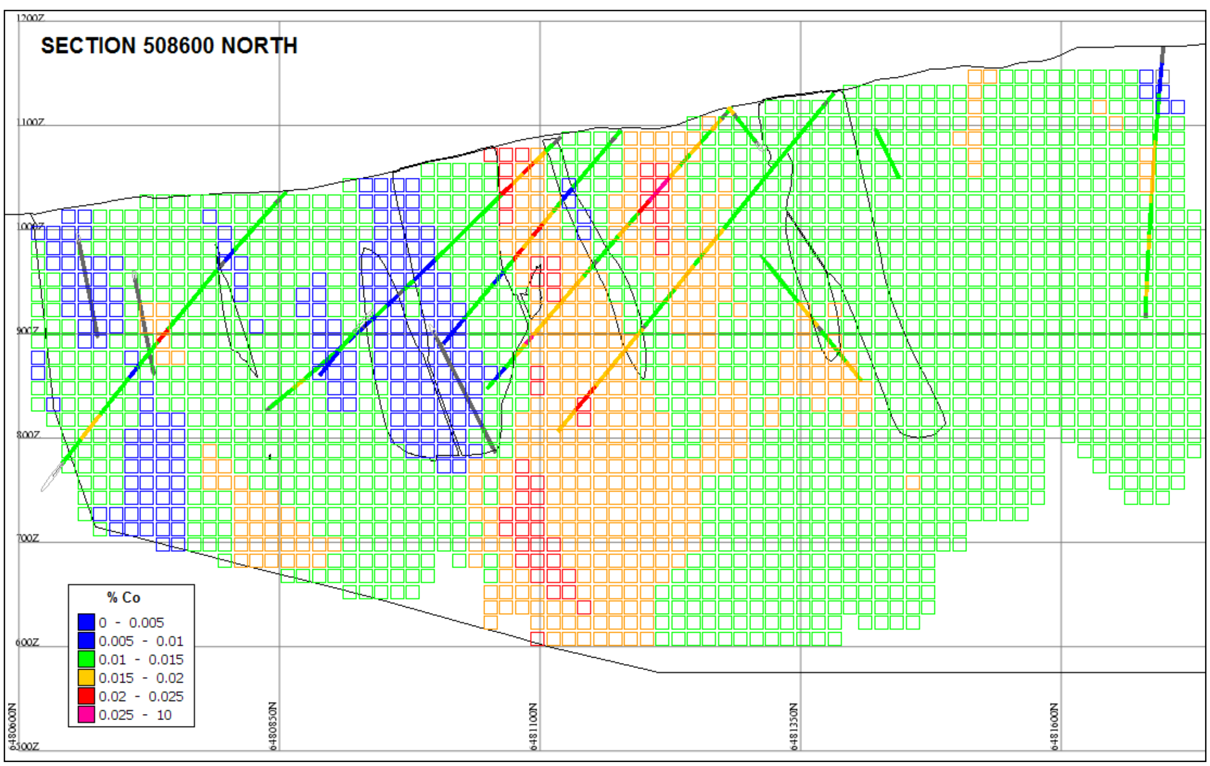


**Figure 14.5 Section 508600 E - S Grades in Composites and Block Model**

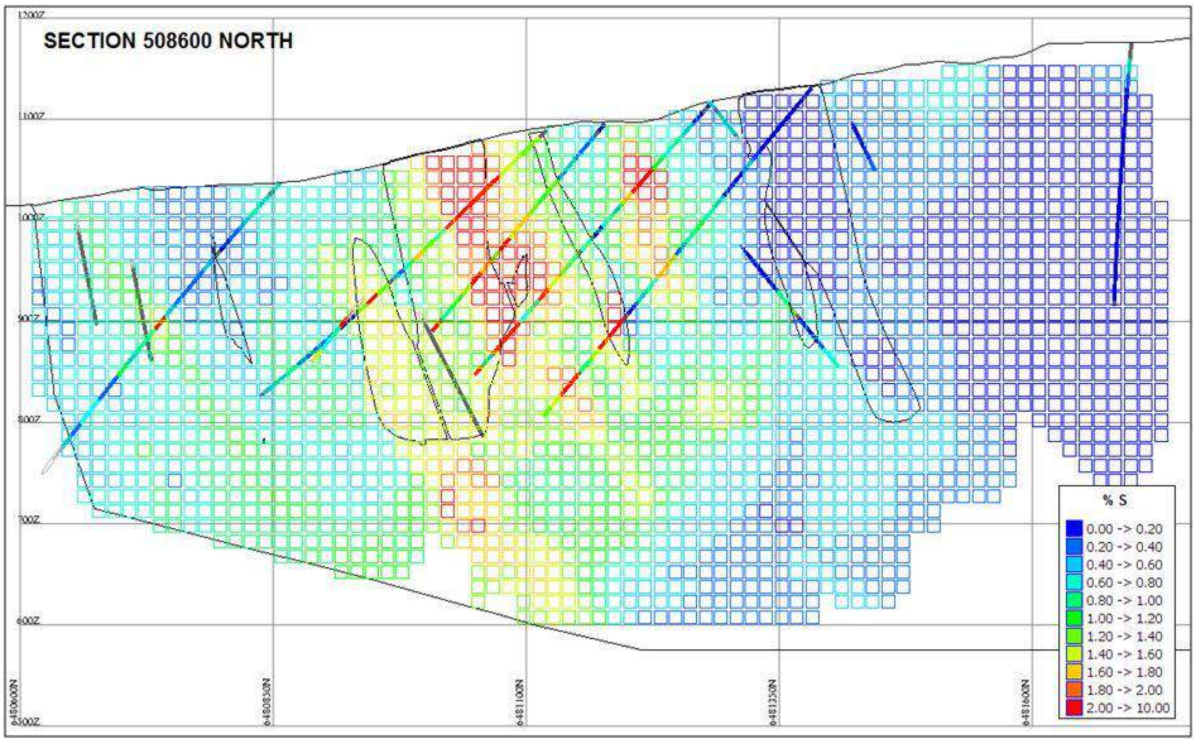
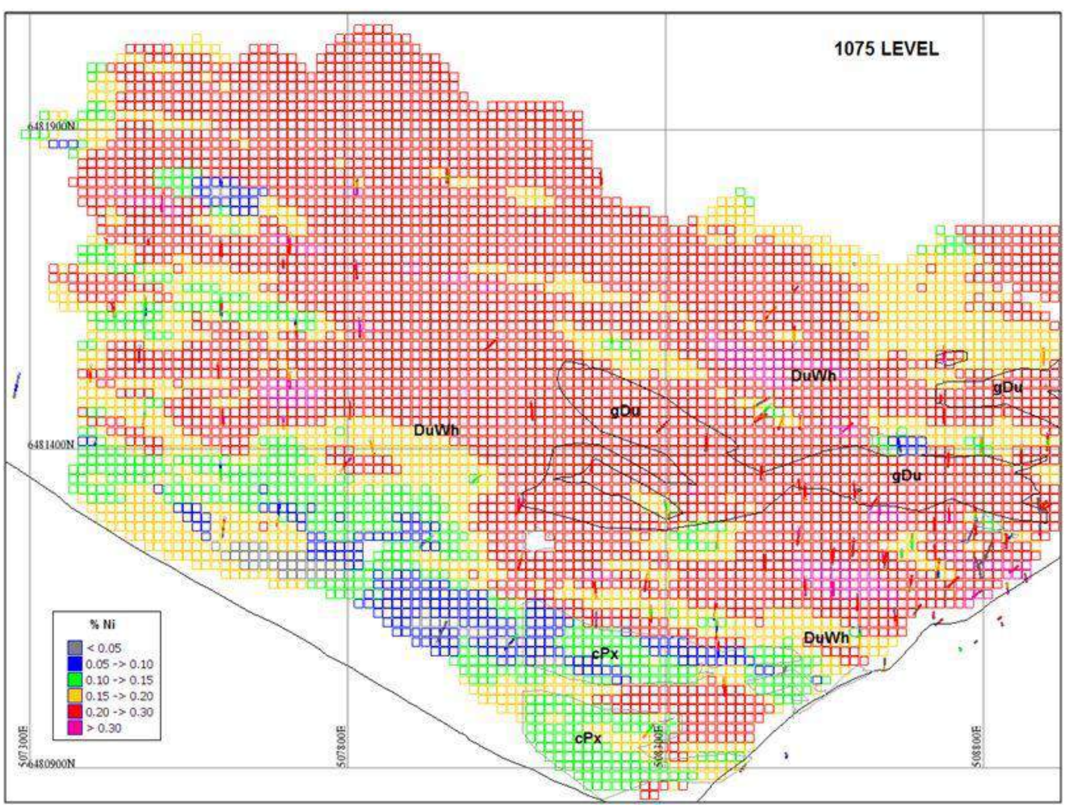
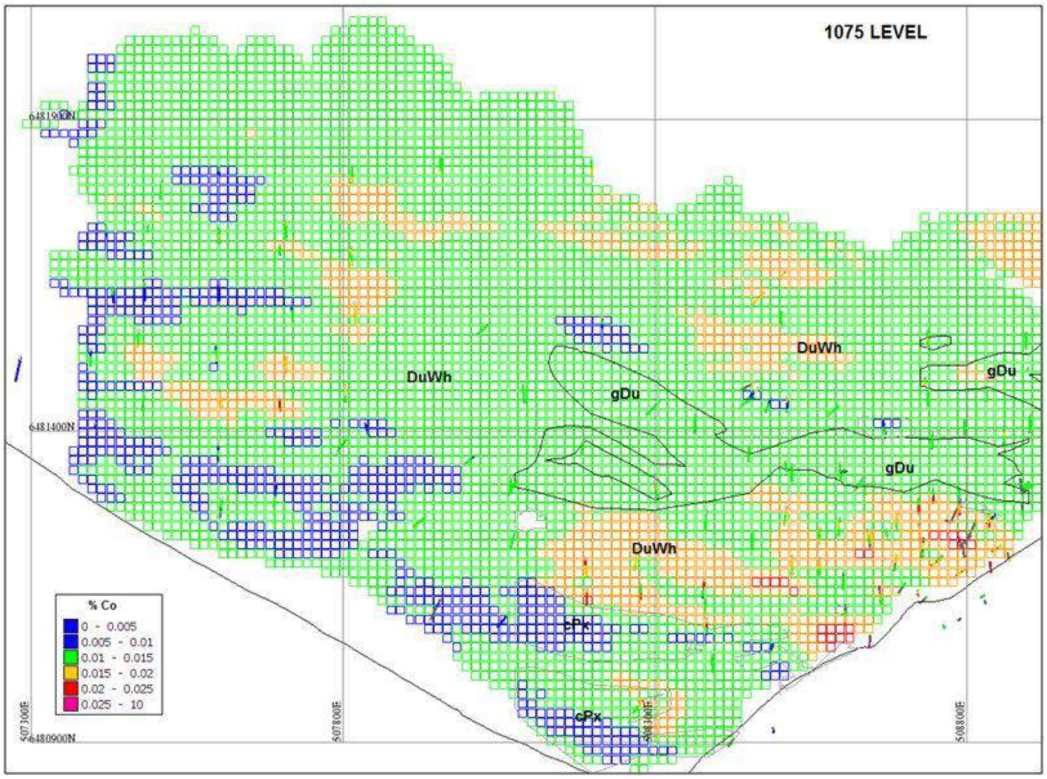


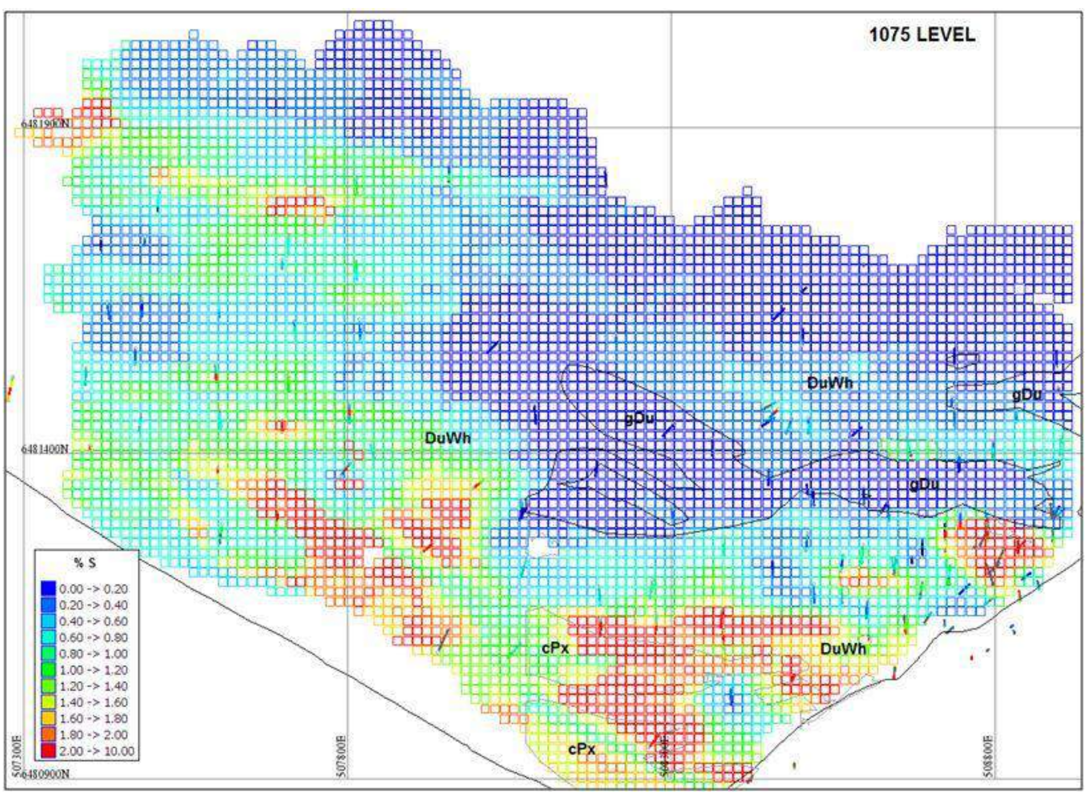

**Figure 14.6 1075 Level - Ni Grades in Composites and Block model**



### Figure 14.7 1075 Level - Co Grades in Composites and Block Model 1075 Level




### Figure 14.8 1075 Level - S Grades in Composites and Block Model 1075 Level



### 14.8 Validation

Model verification was carried out by visual comparison of color coded blocks and composites on plans and sections.

Statistical comparisons of global block grades and composite grades show excellent correlation (Figure 14.9).

**Table 14.10 Global Mean Grade Comparison**

| Type | Ni | Co |
|---|---|---|
| Sample | 0.208 | 0.013 |
| Composite | 0.209 | 0.013 |
| Kriged grade | 0.208 | 0.013 |
| NN grade | 0.204 | 0.013 |

Swath plots were generated for all major orientations in order to test for local bias in the estimate. This was accomplished by selecting 50 metre-wide panels of blocks in three N-S cross sections, one longitudinal (E-W) section and one level plan. The block estimates for sulphide nickel using kriging and nearest neighbor methods were then averaged and plotted (Figure 14.10 to Figure 14.12). No significant local biases were identified.

**Figure 14.9 Swath Plot - 1005 Level**

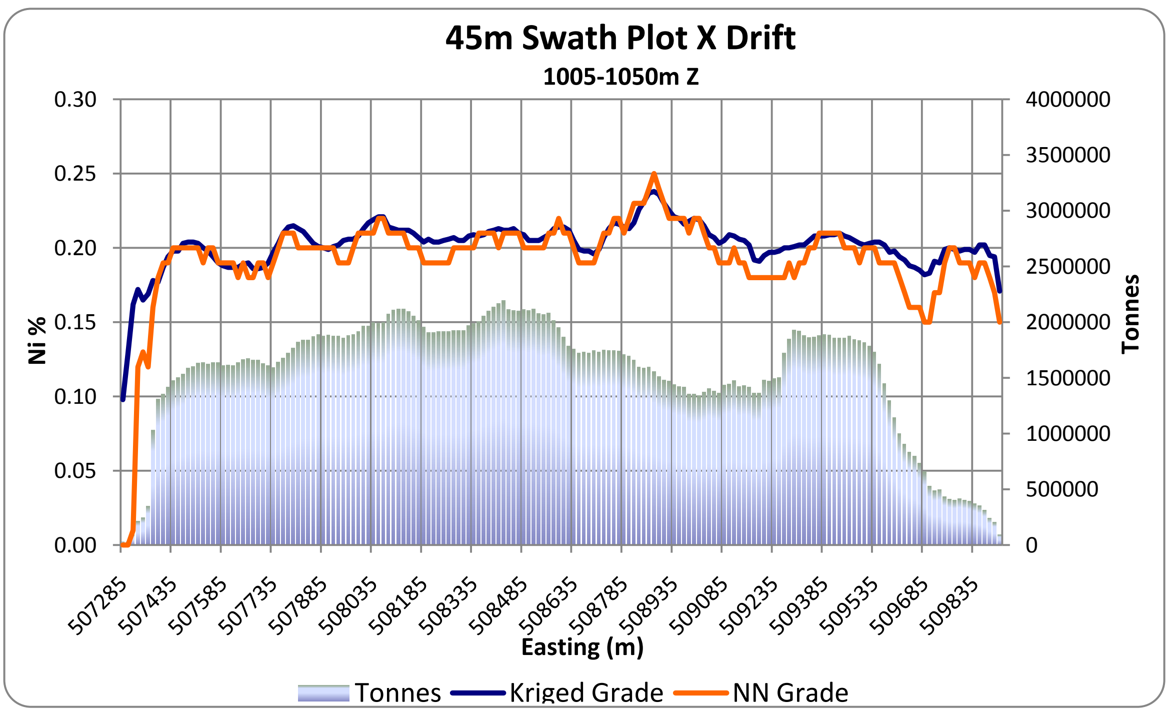


**Figure 14.10 Swath Plot - Section 507667 E**

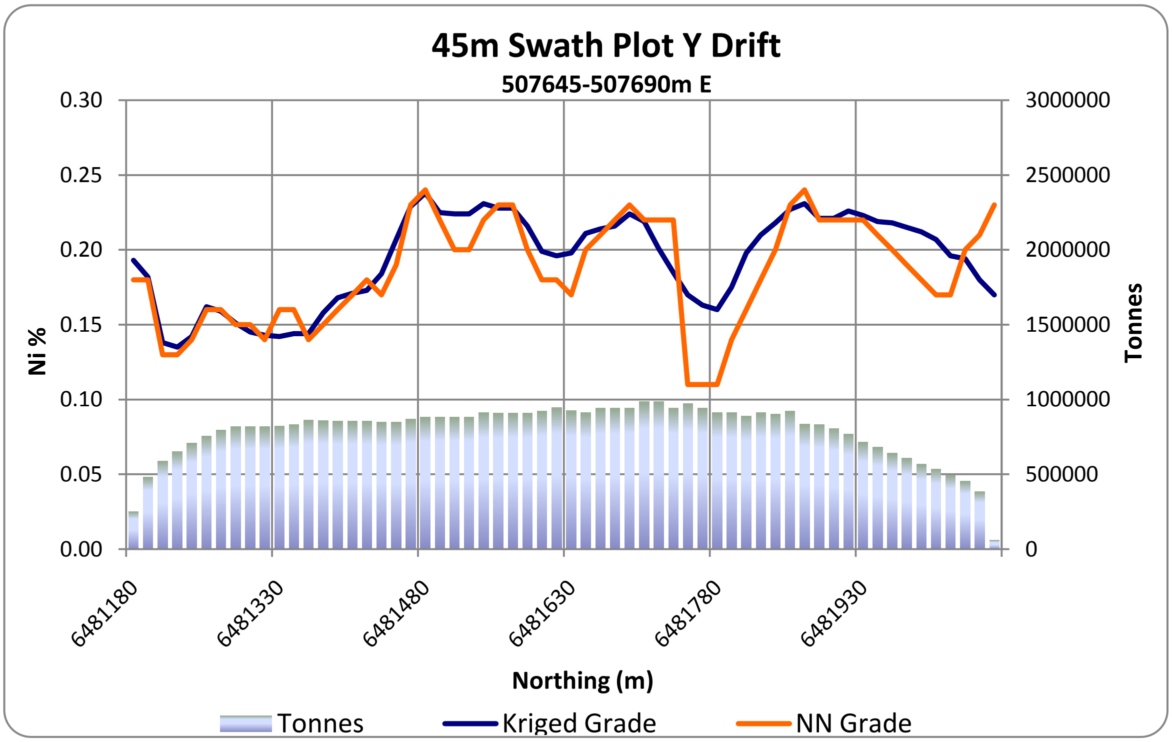
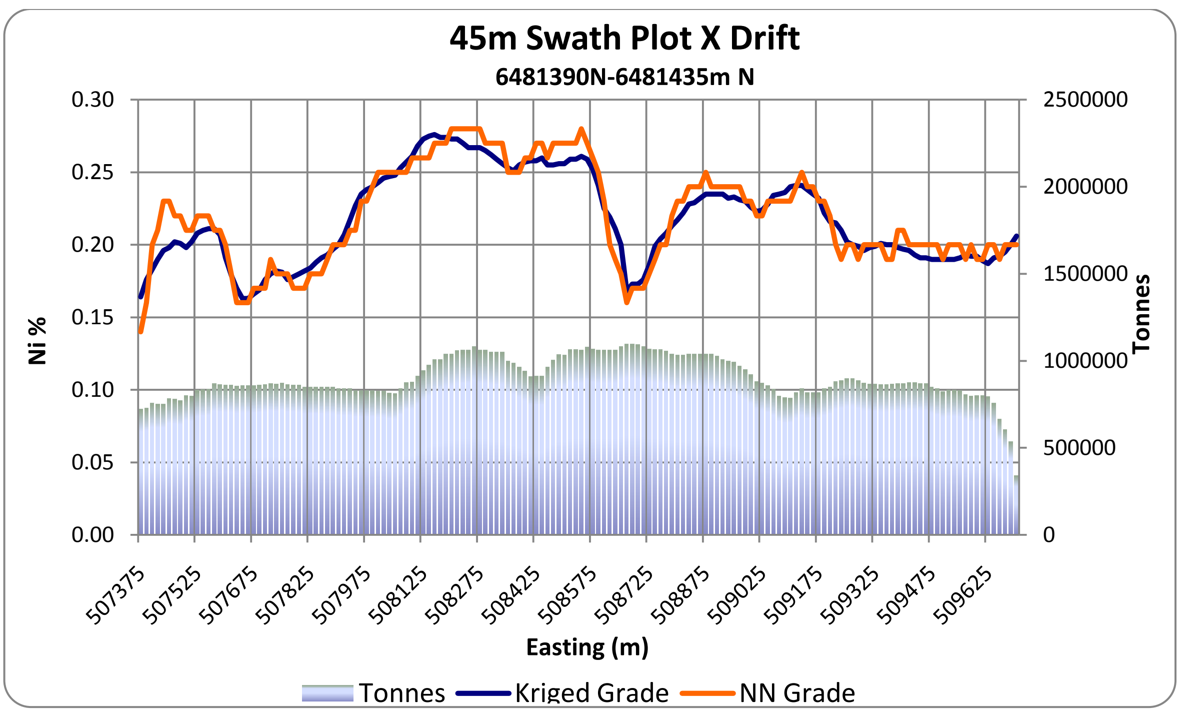



**Figure 14.11 Swath Plot - Section 6481410 N**



## 14.9 Cut-off Grade Determination

Table 14.10 shows mining and economic assumptions used for pit optimization in 2009 and the resulting external cut-off of 0.11% equivalent Ni. Since Co is expected to contribute at least 10% to the overall metal value the selected base case cut-off grade was set at 0.1% Ni.

### Table 14.11 External Cut-off Determination

| | |
|---|---|
| CDN$/tonne milled | $5.50 |
| CDN$/tonne mined | $1.40 |
| CDN$/US$1 | $1.18 |
| % recovery | 49 |
| % metal price paid for transportation, treatment and refining costs | 33.33 |
| lb/tonne | 2204.62 |
| nickel price (US$/lb) | $7.50 |
| Equivalent Ni cut-off % | 0.11% |

## 14.10 Mineral Resource Classification

Resource classifications used in this study conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves as referenced in NI 43-101. The classification parameters were consistent with the previous resource estimate (AMEC, 2007) with the main result being an increase in the measured category tonnes due to infill drilling.

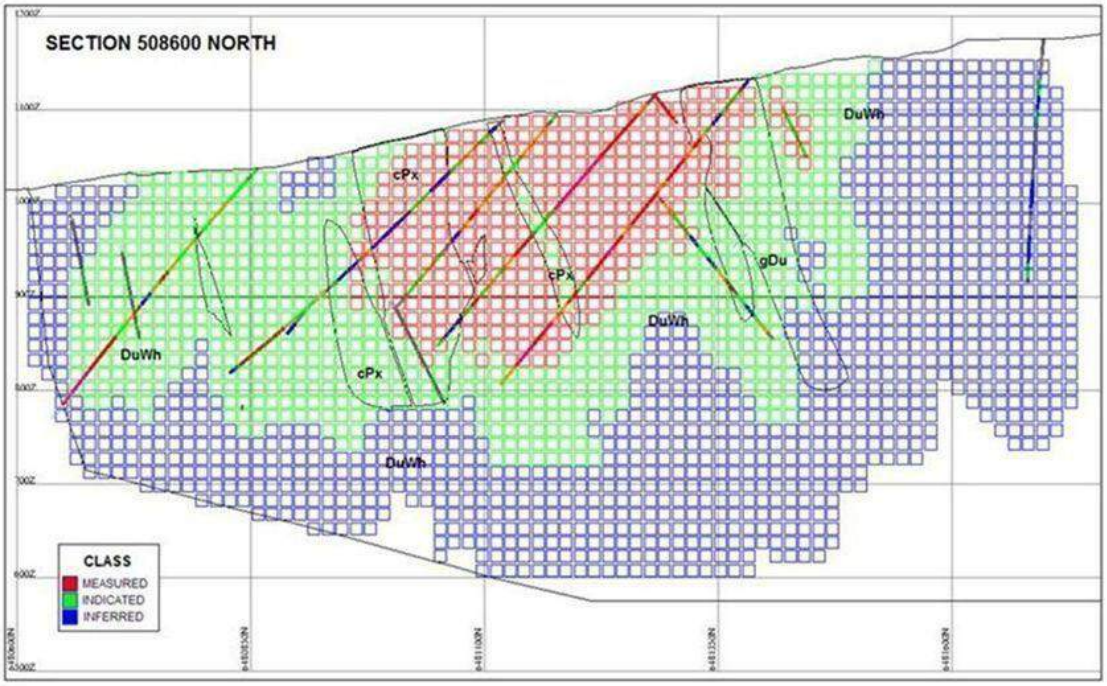
Blocks classified as Measured were restricted to the areas where the drill spacing is on approximate 75 x 50m centres and the distance from a block centroid to the closest composite was less than 50m.

Blocks not classified as Measured were assigned to the Indicated category if they were located in areas with drill spacing on approximate 100 x 100m centres and the distance to the nearest composite was within 100m.

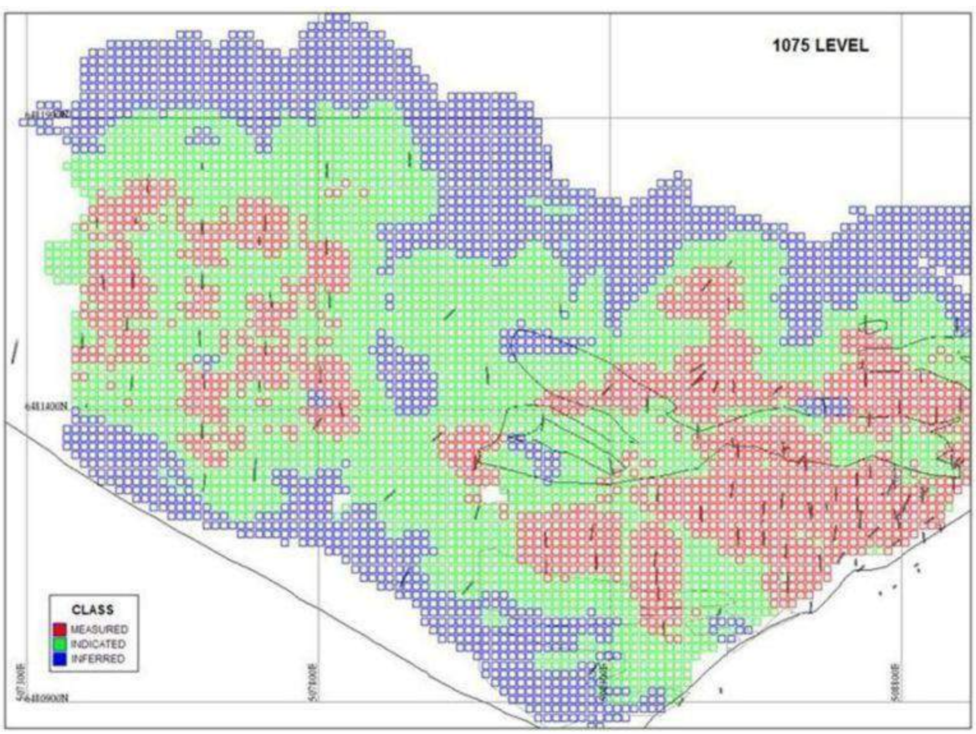
Remaining estimated blocks that did not meet the criteria for Measured or Indicated Mineral Resource were classified as Inferred if they fell within 250m of two drill holes and were within mineralized lithologic domains.

The following figures illustrate the distribution of the three classes in plan view and cross section.

### Figure 14.12 Block Classification –Section 508600 North




### Figure 14.13 Block classification - 1075 Level



### 14.11 Mineral Resource Summary

In order to be consistent with previous resource estimates and in accordance with CIM best practice of reporting resources which have a reasonable expectation of economic extraction, the classified blocks in this model were constrained by a Lerchs-Grossman optimized pit shell. The pit optimization was carried out by Moose Mountain Technical Services (MMTS).

The January 2008, AMEC Preliminary Assessment assumed a 100m buffer along the river, which reduced Measured and Indicated Resources from 489M tonnes to 348M tonnes, a decrease of 17%. The buffer also resulted in a 70% reduction of Inferred Resources from 560M tonnes to 170M tonnes. The 2008 Preliminary Assessment identified a 29 year mine life based on mineral resource not impacted by the buffer zone. The actual buffer and final resource will be dependent on geotechnical and hydrological studies, as well as any permitting restrictions. The new resources stated in this news release have not been constrained by the 100m buffer and when considered will also likely result in a similar reduction in the resource statement.

**Table 14.12 Pit Optimization Assumptions**

| | Units | |
|---|---|---|
| Exchange Rate | C$/US$ | 1.00 |
| Nickel Price (London Metal Exchange) | US$/lb | 9.50 |
| Cobalt Price (London Metal Exchange) | US$/lb | 14.00 |
| Nickel Offsite Charges | C$/lb | 2.35 |
| Cobalt Offsite Charges | C$/lb | 2.50 |
| Processing Cost | | |
| 101 & 104 Mineralization | C$/lb | 5.55 |
| 105 Mineralization | C$/lb | 4.87 |
| 106 Mineralization | C$/lb | 4.37 |
| G&A Cost | C$/t | 0.26 |
| Mining Cost – Mineralization | C$/t | 1.79 |
| Mining Cost – Waste | C$/t | 1.79 |
| Incremental Bench Cost (below 1140m) | C$/t | 0.025 |
| Stockpile and Re-handle Cost | C$/t | 0.80 |
| Nickel Pay Factor | % | 95 |
| Cobalt Pay Factor | % | 50 |
| Overall Pit Slope Angle with Ramp | degrees | 45 |

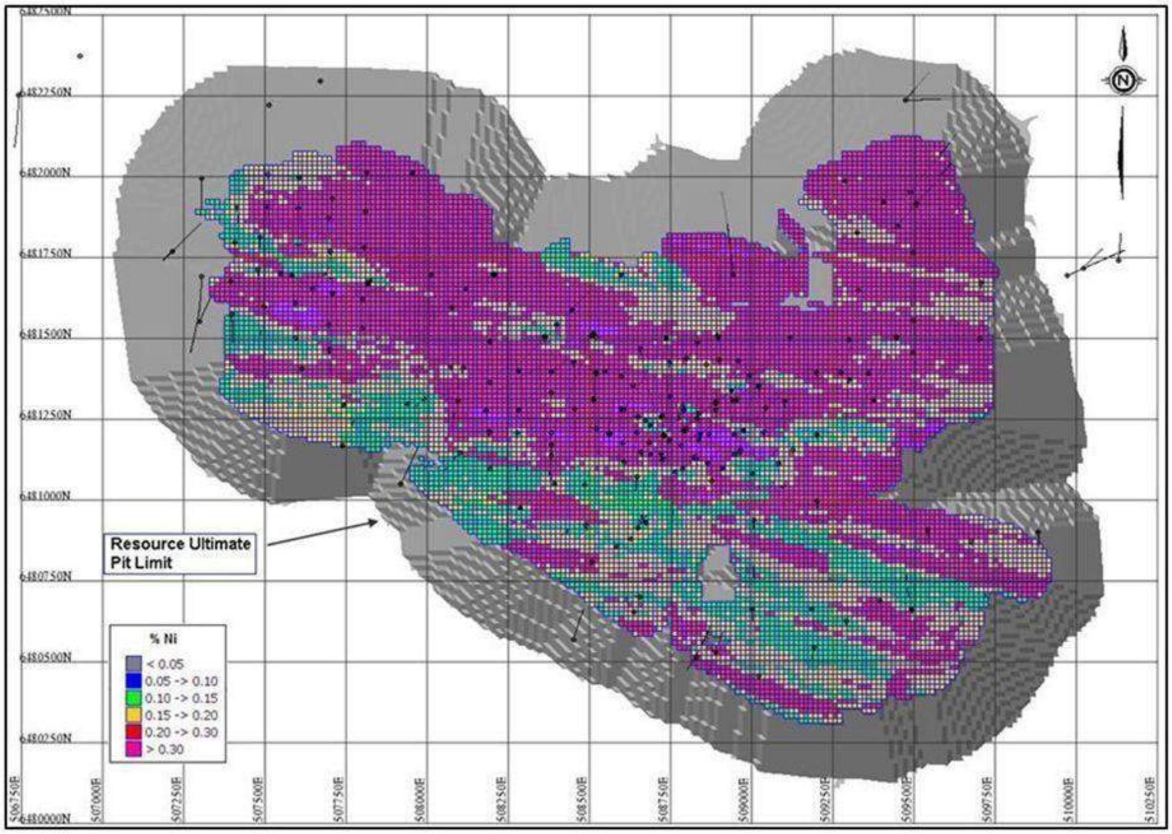
Using the preceding assumptions and a cut-off grade of 0.1% Ni, the Turnagain property contains an estimated 865 Mt of Measured and Indicated Resources at 0.21% Ni and 0.013% Co. An additional 976 Mt grading 0.20% Ni and 0.013% Co is classified as Inferred. The resources for a range of cut-off grades are presented in Table 14.13 and Figure 14.14.

**Table 14.13 Mineral Resource Estimate; Effective Date 30 June 2011, R. Simpson QP**

| COG Ni % | Measured | | | Indicated | | | Measured & Indicated | | |
|---|---|---|---|---|---|---|---|---|---|
| | Tonnes 000's | Total Ni % | Total Co % | Tonnes 000's | Total Ni % | Total Co % | Tonnes 000's | Total Ni % | Total Co % |
| 0.20 | 145,870 | 0.25 | 0.015 | 355,910 | 0.24 | 0.014 | 501,780 | 0.24 | 0.014 |
| 0.18 | 175,240 | 0.24 | 0.015 | 436,621 | 0.23 | 0.014 | 611,861 | 0.24 | 0.014 |
| 0.16 | 195,309 | 0.24 | 0.014 | 503,476 | 0.22 | 0.013 | 698,785 | 0.23 | 0.013 |
| 0.14 | 210,085 | 0.23 | 0.014 | 559,319 | 0.22 | 0.013 | 769,404 | 0.22 | 0.013 |
| 0.12 | 220,896 | 0.23 | 0.014 | 604,254 | 0.21 | 0.013 | 825,150 | 0.21 | 0.013 |
| 0.10 | 227,379 | 0.22 | 0.014 | 638,103 | 0.21 | 0.013 | 865,482 | 0.21 | 0.013 |

| COG Ni % | Inferred | | |
|---|---|---|---|
| | Tonnes 000's | Total Ni % | Total Co % |
| 0.20 | 543,624 | 0.24 | 0.014 |
| 0.18 | 690,692 | 0.23 | 0.014 |
| 0.16 | 806,945 | 0.22 | 0.013 |
| 0.14 | 889,170 | 0.21 | 0.013 |
| 0.12 | 944,549 | 0.21 | 0.013 |
| 0.10 | 976,295 | 0.20 | 0.013 |

## Figure 14.14 Mineral Resource Plan



## 15 MINERAL RESERVE ESTIMATES

There are no reserves to report at this stage.

# 16 MINING METHODS

## 16.1 Summary

This section includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA based on these resources will be realized.

The Turnagain deposit will be mined using an open pit mining method, employing high volume trucks and shovels. The use of large mining equipment will achieve high mining rates and ensure the lowest possible mine operations unit costs. The waste and mineralization will require blasting and typical grade control methods using blast-hole sampling.

For the purpose of this study, the Horsetrail Pit is designed for 28 year life of mine, and includes the Horsetrail and northwest mineralized zones. Previous evaluations have indicated a potential open pit resource in the Hatzl zone located on the east side of the Turnagain River, but that opportunity is not included in the scope of this study. The Turnagain River is a fish habitat and wildlife corridor and the underlying mineralization is also excluded.

The potential resource contained in the Horsetrail Pit is summarized in Table 16.1. This pit forms the basis of the mine plan and production schedule in this study. It is contained within the optimized economic pit shell, a much larger potential open pit resource. The increment between the Horsetrail Pit and the optimized pit is also shown in Table 16.1, but is not included with the production plan in this study.

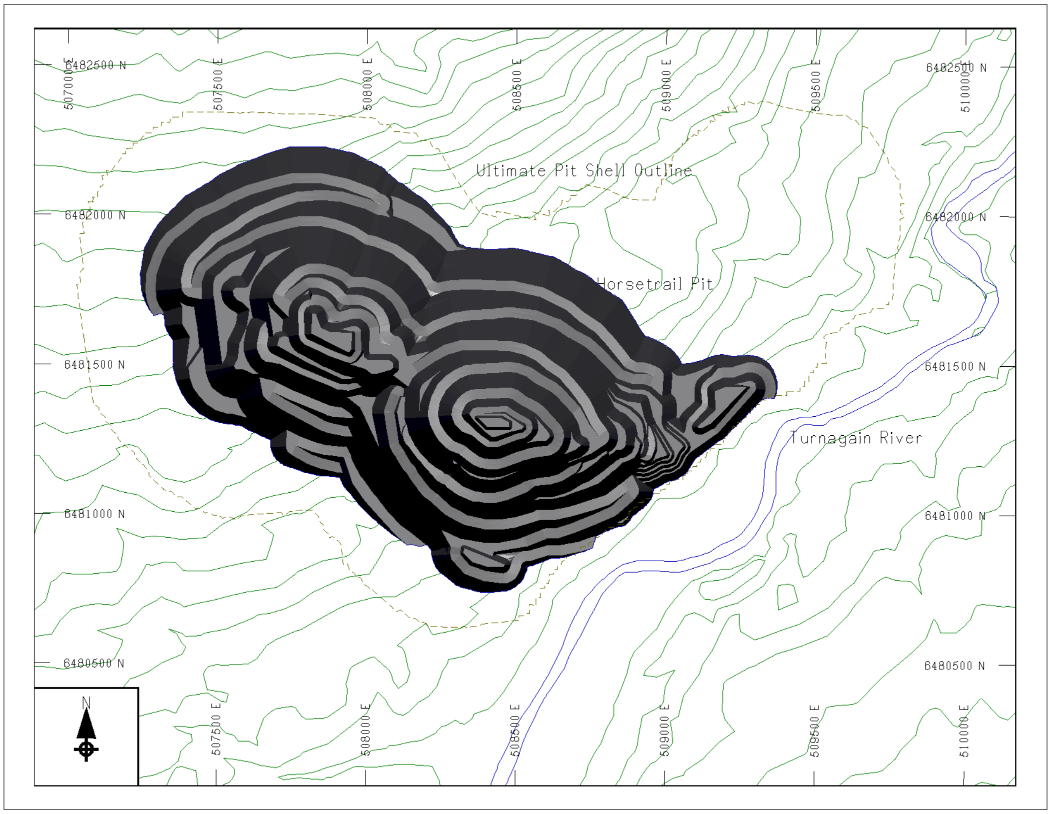
**Table 16.1 Potential In-Pit Resource Estimate**

| | Mineralization (kt) | Waste (kt) | Strip Ratio | NSR* ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Horsetrail Pit (PEA basis, 28 Yr LOM) | 762,896 | 317,872 | 0.42 | 21.60 | 0.230 | 0.013 | 0.69 |
| Potential Pushback to Ultimate Pit Shell | 499,269 | 641,776 | 1.29 | 18.96 | 0.209 | 0.012 | 0.63 |
| **Total** | **1,262,165** | **959,648** | **0.76** | **20.56** | **0.222** | **0.013** | **0.67** |

* NSR = net smelter return at Base Case metal pricing
Note: includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized. (Details of Measured and Indicated Resources in PEA basis pit are shown in Table 16.7).

Figure 16.1 shows a plan view of the preliminary design for the Horsetrail Pit as well as the optimized, or Ultimate pit shell outline.

### Figure 16.1 Horsetrail Pit Design – Plan View



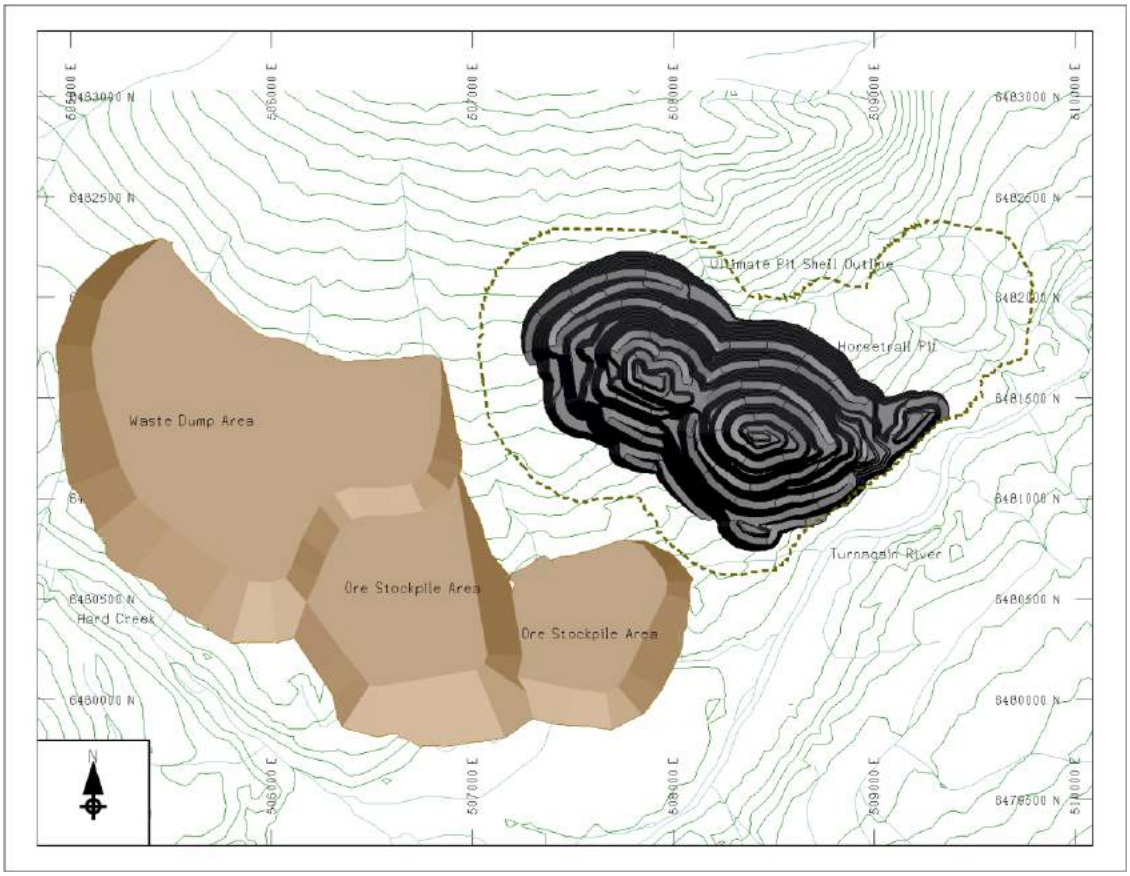

The mine will feed the crusher at an average rate of 43,400 t/d during the first five years, and increase to an average of 84,600 t/d thereafter. The resource will be mined for a total of 28 years at these rates.

To access the most economic mineralization in the early years and provide a smooth strip ratio throughout the life of mine, mineralization production from the Horsetrail Pit is scheduled from five mining phases. Phase 1 will commence at the centre of the Horsetrail Pit, where the highest grade and lowest strip ratio will be encountered.

An elevated cut-off grade will be employed in the initial production years to enhance the economics of the project. Mineralization that is below the cut-off grade will be sent to a stockpile near the crusher and either reclaimed at the end of the mine life, or blended with the run-of-mine feed if an appropriate opportunity arises. Mineralization that is below the mine cut-off grade, but of sufficient grade to cover the cost of milling and handling once it is hauled out of the pit, will also be sent to the mill either directly or through the mineralization stockpile.

Pit waste material will be hauled to a waste dump southwest of the pit adjacent to the mineralization stockpile area. Current geochemistry data suggests that there is insignificant acid generating potential in the waste rock. Further studies will be undertaken to confirm that the waste rock will have minimum long-term environmental impact. Figure 16.2 shows the conceptual waste dumping plan.

**Figure 16.2 Waste Disposal and Mineralization Stockpile General Arrangement**




## 16.2 Open Pit Optimization Study

The optimization study methodology used is described below and is common and widely accepted in the mining industry for preliminary assessments of open pit mining potential. The potential resource for the open pit is initially evaluated by undertaking pit optimization studies on the geological model using the Lerchs-Grossman (LG) computer algorithm. Various preliminary pit shells are generated from the simulations and analyzed. Selected shells are assessed for whether they are appropriate to use for framing the ultimate pit and mining phases.

### 16.2.1 Design Parameters

Economic values are assigned on Measured, Indicated, and Inferred Resource classes as categorized in the resource block model. The preliminary input parameters applied on the optimized pit (Table 16.2) are estimated based on previous studies and discussions with HNC. Costs, exchange rate, recoveries, and pit slope angles are preliminary and are specific for this optimization study only.

### Table 16.2 Preliminary Design Parameters for Optimized Pit

| | Units | |
|---|---|---|
| Exchange Rate | C$/US$ | 1.00 |
| Nickel Price (London Metal Exchange) | US$/lb | 9.50 |
| Cobalt Price (London Metal Exchange) | US$/lb | 14.00 |
| Nickel Offsite Charges | C$/lb | 2.35 |
| Cobalt Offsite Charges | C$/lb | 2.50 |
| Processing Cost | C$/t | 5.50 |
| G&A Cost | C$/t | 0.26 |
| Mining Cost – Mineralization | C$/t | 1.24 |
| Mining Cost – Waste | C$/t | 1.24 |
| Incremental Bench Cost (below 1140 m) | C$/t | 0.025 |
| Stockpile and Re-handle Cost | C$/t | 0.60 |
| Nickel Pay Factor | % | 95 |
| Cobalt Pay Factor | % | 50 |
| Overall Pit Slope Angle with Ramp | degrees | 45 |

Process recoveries for nickel and cobalt were provided by HNC and the processes for deriving them have been described in Section13. The nickel recovery formula to produce 18% concentrate is:

$$= 0.2564 \times \ln(\%Ni_T) + 0.9491, \text{ and } = 0 \text{ for ACNi grades } <0.1\%.$$

Process recovery on the green dunite mineralization (lithology code 105) is further reduced by lowering its Ni% grade to 75% of the value in the block model. Cobalt recovery is set equal to the nickel recovery in the same block of the model.

### 16.2.2 Turnagain River Restriction

A mining restriction will limit mining activity to either side of the Turnagain River. Until further studies are carried out to assess other mining scenarios, the river will not be disturbed. The limit is determined by approximating the high water level, which HNC estimated to be the 1015 masl along the river banks. The pit crest lines are offset from this contour by 65m, which includes 50m for a no-disturbance zone and an additional 15m for an access corridor. These are preliminary estimates and will be updated as necessary.

### 16.2.3 Pit Optimization Variable – Net Metal Value

The pits were optimized on the net metal value that is calculated from the Ni% and Co% grades in the 3D block model. This variable represents the combined net metal values for nickel and cobalt in the mineralization. It is calculated in Canadian dollars, and is the sum of the net nickel value and net cobalt value. They are derived as follows:

- Net metal value ($/t) nickel = 2,204.6 lb/t x Ni grade (%) x payable nickel (%) x net nickel price ($/lb) at mine gate x process recovery (%)
- Net metal value ($/t) cobalt = 2,204.6 lb/t x Co grade (%) x payable cobalt (%) x net cobalt price ($/lb) at mine gate x process recovery (%)

The parameters that were used to calculate the net metal value for the Base Case are provided in Table 16.3. The net metal value calculated for the Based Case is carried in the model as the NSR value.

### Table 16.3 NSR Parameters

| Parameter | Units | |
|---|---|---|
| **Metal Price (London Metal Exchange)** | | |
| Nickel | US$/lb | 9.50 |
| Cobalt | US$/lb | 14.00 |
| Nickel | C$/lb | 9.50 |
| Cobalt | C$/lb | 14.00 |
| **Offsite Charges** | | |
| Nickel | C$/lb | 2.35 |
| Cobalt | C$/lb | 2.50 |
| **Net Metal Price at Mine Gate** | | |
| Nickel | C$/lb | 7.15 |
| Cobalt | C$/lb | 11.50 |
| **Payables** | | |
| Payable Nickel | % | 95.00 |
| Payable Cobalt | % | 50.00 |

**Note:** exchange rate (C$/US$) = 1.00.

#### 16.2.3.1 Pit Optimization Results

Table 16.4 shows the mineralization and waste quantities contained in the optimized pit shells generated against the net metal value variable. The Base Case is LG Shell 31, where the input metal price is US$9.50/lb for nickel and US$14.00/lb for cobalt, and the shell contains 1.46 billion tonnes of mineralization at 0.8 to 1 strip ratio. The cut-off grade applied is on an NSR value of $7.30/t; this is a conservative estimate relative to the parameters shown in Table 1.2 and is used for this analysis only.

The other LG shells were generated by varying net metal value variable to test the sensitivity of the resource, assess possible pit phases and the opportunities for expansion. Figure 16.3 provides a graphical illustration of contained mineralization in the LG shells.

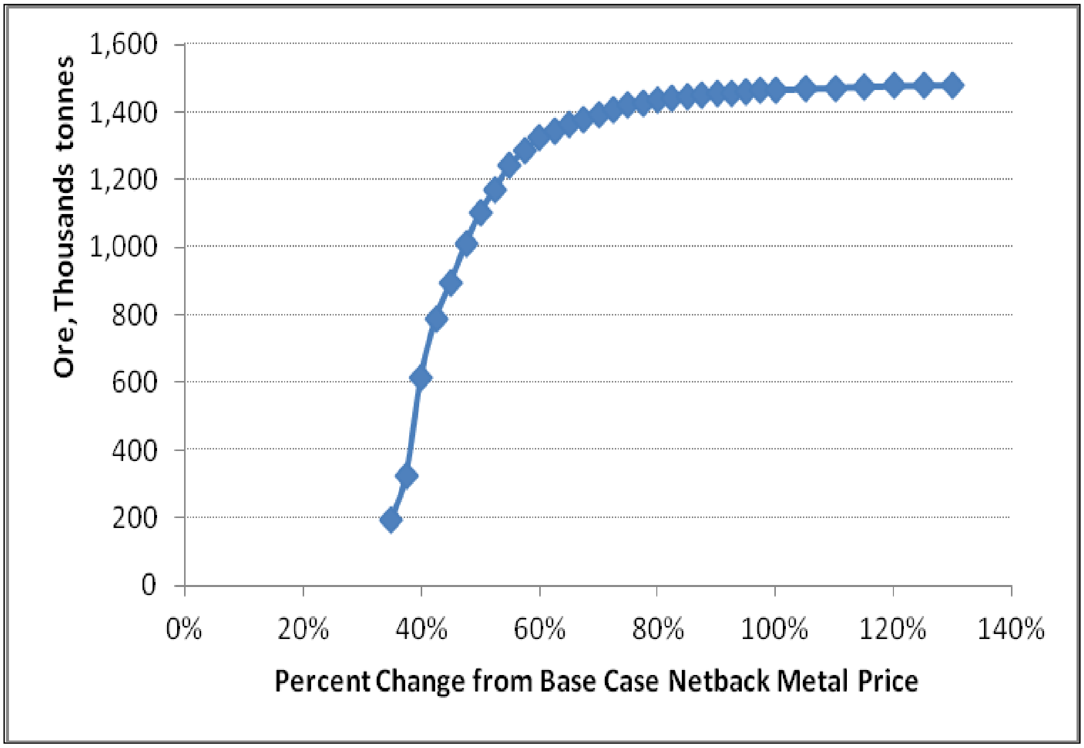
The results from this set of optimized pits indicate that the Turnagain deposit is insensitive to metal prices once nickel and cobalt prices exceed around 60% of the Base Case.

## Table 16.4 LG Optimized Pit Shells (Cut-off on $7.30/t NSR Value)

| % of Base Case metal price | LG Shell | Waste (kt) | Mineralization (kt) | Ni % | Co % | NSR ($/t) | Strip Ratio |
|---|---|---|---|---|---|---|---|
| 35% | 5 | 34,426 | 192,226 | 0.248 | 0.014 | 24.00 | 0.18 |
| 37% | 6 | 77,817 | 323,613 | 0.241 | 0.013 | 23.08 | 0.24 |
| 40% | 7 | 191,058 | 614,027 | 0.234 | 0.013 | 22.02 | 0.31 |
| 43% | 8 | 291,293 | 789,335 | 0.231 | 0.013 | 21.69 | 0.37 |
| 45% | 9 | 357,038 | 896,644 | 0.229 | 0.013 | 21.42 | 0.4 |
| 48% | 10 | 430,975 | 1,009,585 | 0.226 | 0.013 | 21.09 | 0.43 |
| 50% | 11 | 510,377 | 1,104,922 | 0.224 | 0.013 | 20.85 | 0.46 |
| 52% | 12 | 574,022 | 1,170,050 | 0.223 | 0.013 | 20.69 | 0.49 |
| 55% | 13 | 684,874 | 1,242,560 | 0.222 | 0.013 | 20.58 | 0.55 |
| 57% | 14 | 749,844 | 1,284,722 | 0.221 | 0.013 | 20.50 | 0.58 |
| 60% | 15 | 823,982 | 1,324,441 | 0.221 | 0.013 | 20.43 | 0.62 |
| 63% | 16 | 858,588 | 1,344,623 | 0.220 | 0.013 | 20.38 | 0.64 |
| 65% | 17 | 899,313 | 1,361,863 | 0.220 | 0.013 | 20.35 | 0.66 |
| 68% | 18 | 932,659 | 1,377,651 | 0.220 | 0.013 | 20.31 | 0.68 |
| 70% | 19 | 972,071 | 1,394,055 | 0.219 | 0.013 | 20.27 | 0.7 |
| 72% | 20 | 1,015,222 | 1,409,622 | 0.219 | 0.013 | 20.23 | 0.72 |
| 75% | 21 | 1,049,557 | 1,422,136 | 0.219 | 0.013 | 20.20 | 0.74 |
| 77% | 22 | 1,062,098 | 1,425,929 | 0.219 | 0.013 | 20.19 | 0.74 |
| 80% | 23 | 1,095,416 | 1,435,604 | 0.219 | 0.013 | 20.17 | 0.76 |
| 83% | 24 | 1,124,691 | 1,442,334 | 0.218 | 0.013 | 20.16 | 0.78 |
| 85% | 25 | 1,138,823 | 1,445,691 | 0.218 | 0.013 | 20.15 | 0.79 |
| 88% | 26 | 1,155,236 | 1,450,241 | 0.218 | 0.013 | 20.14 | 0.8 |
| 90% | 27 | 1,169,923 | 1,454,275 | 0.218 | 0.013 | 20.13 | 0.8 |
| 92% | 28 | 1,185,894 | 1,457,602 | 0.218 | 0.013 | 20.12 | 0.81 |
| 95% | 29 | 1,194,280 | 1,459,865 | 0.218 | 0.013 | 20.11 | 0.82 |
| 97% | 30 | 1,208,529 | 1,462,880 | 0.218 | 0.013 | 20.10 | 0.83 |
| Base Case | **31** | 1,220,494 | 1,464,911 | 0.218 | 0.013 | 20.10 | 0.83 |
| 105% | 32 | 1,240,043 | 1,468,520 | 0.218 | 0.013 | 20.09 | 0.84 |
| 110% | 33 | 1,259,805 | 1,471,448 | 0.218 | 0.013 | 20.08 | 0.86 |
| 115% | 34 | 1,286,074 | 1,475,753 | 0.218 | 0.013 | 20.07 | 0.87 |
| 120% | 35 | 1,300,291 | 1,477,661 | 0.218 | 0.013 | 20.06 | 0.88 |
| 125% | 36 | 1,313,658 | 1,479,141 | 0.218 | 0.013 | 20.06 | 0.89 |
| 130% | 37 | 1,326,684 | 1,480,402 | 0.218 | 0.013 | 20.06 | 0.9 |

Note: includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

**Figure 16.3 Sensitivity of Contained in-Pit Resource**




Note: includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

During the course of this PEA mining study, estimated mining costs were developed from the detail in the design and plan. The base case optimized pit was re-generated with the updated costs to compare against the result from using the preliminary input cost estimates. Processing costs by mineralization types were also applied instead of a single average cost for all mineralization types. The revised mining and processing costs are shown in Table 16.5.

### Table 16.5 Updated Design Parameters for Optimized Pit – Base Case

| | Units | |
|---|---|---|
| Exchange Rate | C$/US$ | 1.00 |
| Nickel Price (London Metal Exchange) | US$/lb | 9.50 |
| Cobalt Price (London Metal Exchange) | US$/lb | 14.00 |
| Nickel Offsite Charges | C$/lb | 2.35 |
| Cobalt Offsite Charges | C$/lb | 2.50 |
| Processing Cost | | |
| 101 & 104 Mineralization | C$/lb | 5.55 |
| 105 Mineralization | C$/lb | 4.87 |
| 106 Mineralization | C$/lb | 4.37 |
| G&A Cost | C$/t | 0.26 |
| Mining Cost – Mineralization | C$/t | 1.79 |
| Mining Cost – Waste | C$/t | 1.79 |
| Incremental Bench Cost (below 1140m) | C$/t | 0.025 |
| Stockpile and Re-handle Cost | C$/t | 0.80 |
| Nickel Pay Factor | % | 95 |
| Cobalt Pay Factor | % | 50 |
| Overall Pit Slope Angle with Ramp | degrees | 45 |

The contained mineralization and grades for the revised Base Case optimized pit is summarized in Table 16.6. It is reported on an NSR cut-off value of $7.00/t. The result indicates that the optimized pit with the updated and higher mining cost is smaller, containing 14% less mineralization. For this study, this pit shell will be referred to as the optimized or Ultimate pit shell. The process of selection of the Ultimate pit should be further refined at the pre-feasibility study stage relative to Measured, Indicated and Inferred mineralization categories and should include sensitivity analysis and discounting. It is important to note that the contained mineralization in this simulated pit shell is only an indication of the potential open pittable resource and should not be relied on as a recoverable resource.

### Table 16.6 Base Case Optimized Pit – Updated Cost Parameters

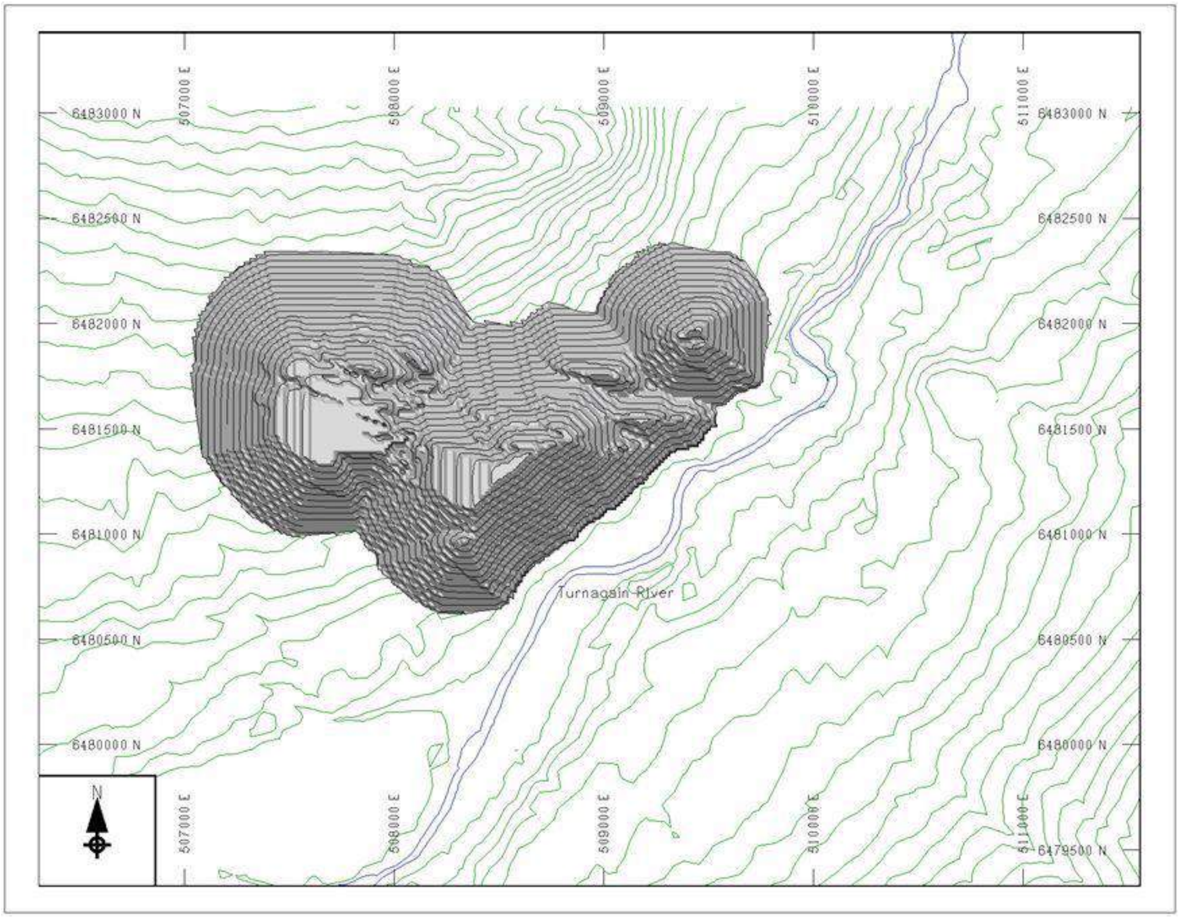
| | Waste (kt) | Mineralization (kt) | Strip Ratio | NSR ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Base Case - Preliminary Cost Parameters | 1,220,494 | 1,464,911 | 0.42 | 20.09 | 0.218 | 0.013 | 0.71 |
| Base Case - Updated Cost Parameters | 959,648 | 1,262,165 | 0.76 | 20.56 | 0.222 | 0.013 | 0.67 |

Note: includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

Figure 16.4 is a plan view of the revised Base Case optimized pit, or Ultimate pit shell. The potential pit resource contained in this optimized pit shell would sustain a mine operation for well over 40 years which is beyond the planning and design scope of this study. A smaller optimized pit shell is selected for the purpose of this study to design a pit with 25 to

30 year life of mine. Therefore, the revised cost inputs for the optimized pit have no impact on the pit designed in this study.

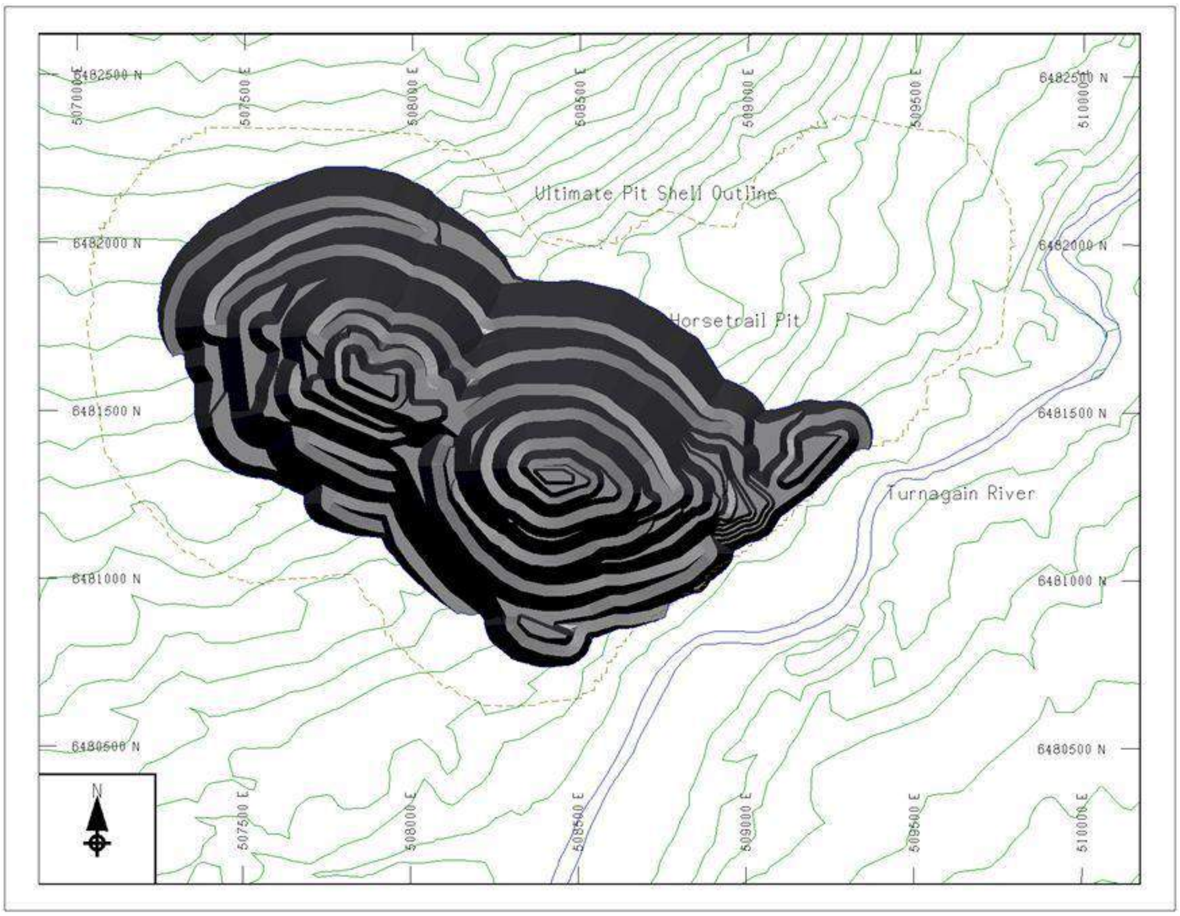
**Figure 16.4 Plan View of Optimized Pit (Ultimate Pit Shell)**



### 16.3 Preliminary Pit Designs

The pit optimization study indicates that under the update Base Case economic parameters, the potential open pit resource approaches 1.3 billion tonnes of mineralization. However, for the purpose of this study, the Horsetrail pit is sized to produce for 25 to 30 years life of mine. Using a smaller optimized pit shell – LG Shell 08 as a design guideline, a preliminary pit was created to incorporate high wall ramps, smooth pit walls, and workable mining phases. Figure 16.5 shows a plan view of the designed pit.

**Figure 16.5 Horsetrail Pit Design – Plan View**




The dimensions are approximately 1.8 km in length, and 1.1 km at the widest section. The bottom of the pit is at 630m elevation; the highest point along the pit rim is 1340m elevation at the west end.

Figure 16.6 is a reference for the locations of the non-orthogonal cross sectional views in Figures 16.7 to 16.9. The sections show the block model nickel grades (Ni%) overlain by the Horsetrail Pit (inner pit wall) and the Ultimate pit shell (outer pit wall).

**Figure 16.6 Horsetrail and Ultimate Pit Shell Outlines with Reference Lines**

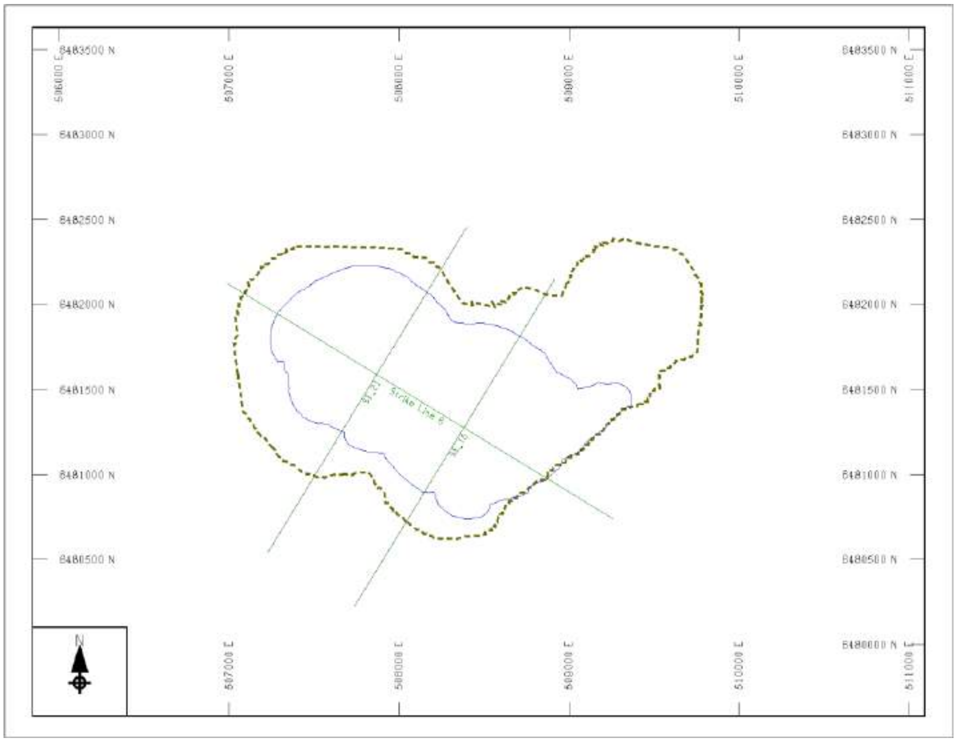


**Figure 16.7 Block Model - Ni values - Section SE_15 Looking NW**

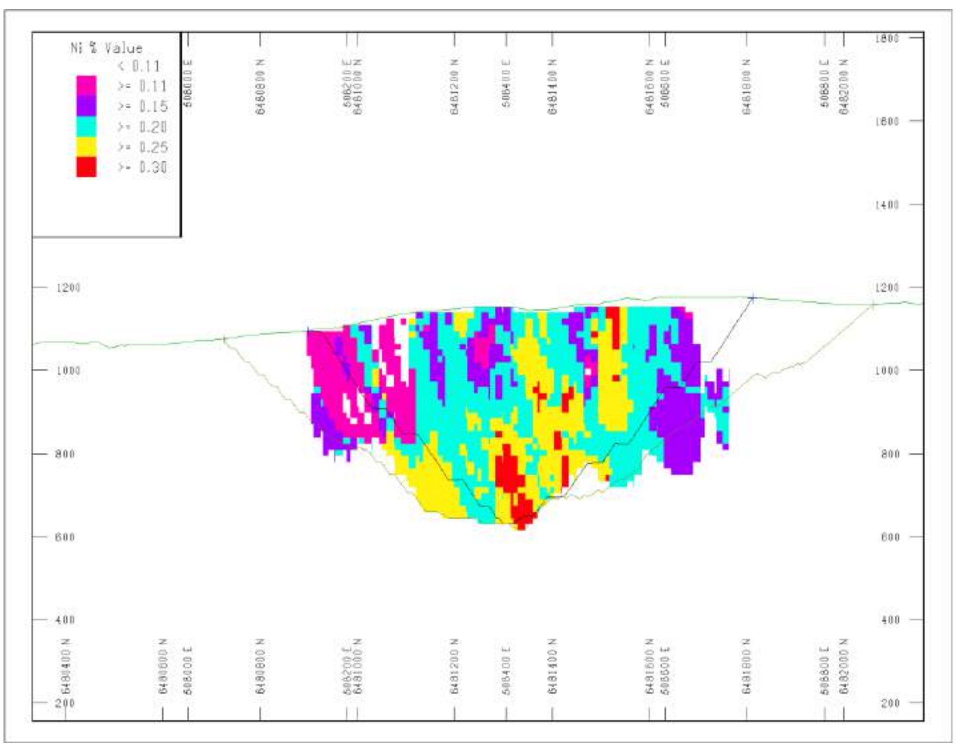


**Figure 16.8 Block Model - Ni values - Section SE_21 Looking NW**

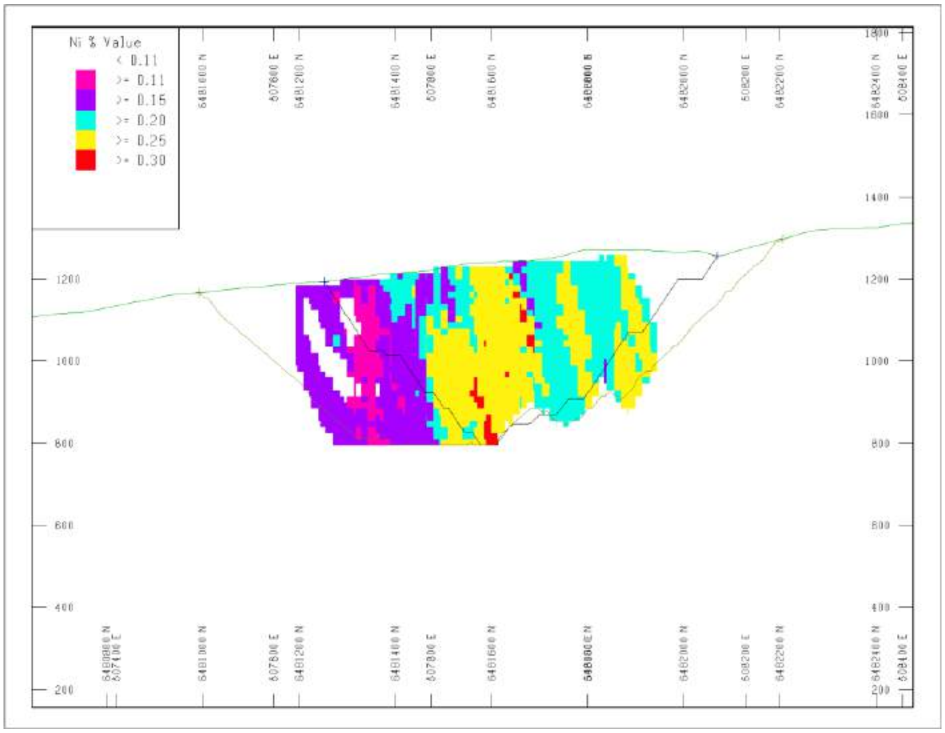
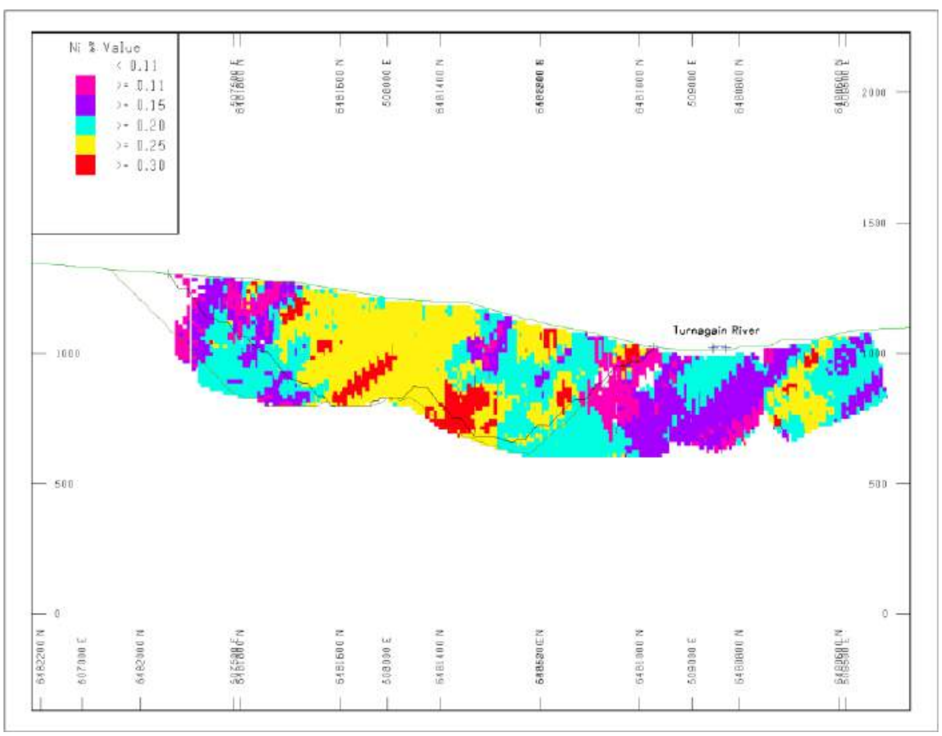



**Figure 16.9 Block Model - Ni values – Longitudinal Section Looking NE**



### 16.3.1 Pit Slope Assumptions

The overall pit slope angle in the designed pit is estimated to be 45°. The ramp width is designed at 36m to include the minimum truck running surface, a ditch allowance, and berms. The assumptions on the pit slopes will require further review as geotechnical assessments have not yet been undertaken.

Groundwater conditions will affect the stability of the pit slopes. It is anticipated that the bedrock is competent and the highwalls will be stable, provided that an effective dewatering plan is implemented to keep them well drained. There have been only limited hydro-geological studies undertaken; these will be very important for the next level of study, and with particular reference to the proximity of the Turnagain River.

### 16.3.2 In-pit Resource Potential

The potential resource estimate by mineralization domain and classification contained within the designed Horsetrail Pit is summarized in Table 16.7.

The mineralized domains, or lithologies, are:

- 101: clinopyroxenes
- 104: dunites and wehrlites
- 105: green dunite
- 106: serpentinized dunites and wehrlites.

**Table 16.7 Horsetrail Pit Potential Resource**

| Domain | Category | Mineralization (kt) | NSR ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|
| 101 | Measured | 5,410 | 10.61 | 0.141 | 0.011 | 1.85 |
| | Indicated | 12,234 | 10.78 | 0.143 | 0.011 | 1.64 |
| | Inferred | 5,246 | 12.10 | 0.154 | 0.011 | 1.53 |
| 104 | Measured | 99,765 | 22.44 | 0.233 | 0.014 | 0.84 |
| | Indicated | 184,426 | 22.09 | 0.232 | 0.013 | 0.63 |
| | Inferred | 131,519 | 23.45 | 0.242 | 0.013 | 0.47 |
| 105 | Measured | 9,265 | 16.27 | 0.250 | 0.013 | 0.22 |
| | Indicated | 14,457 | 17.40 | 0.262 | 0.013 | 0.32 |
| | Inferred | 2,030 | 18.23 | 0.271 | 0.013 | 0.21 |
| 106 | Measured | 91,820 | 22.40 | 0.233 | 0.014 | 0.89 |
| | Indicated | 144,634 | 20.75 | 0.222 | 0.013 | 0.73 |
| | Inferred | 62,091 | 21.42 | 0.227 | 0.013 | 0.47 |
| **Total Mineralization** | Measured | 206,260 | 21.83 | 0.231 | 0.014 | 0.86 |
| | Indicated | 355,751 | 20.97 | 0.226 | 0.013 | 0.69 |
| | Inferred | 200,886 | 22.47 | 0.235 | 0.013 | 0.50 |

### 16.3.3 Reported Grade items

The reported grade items are:

- NSR: combined net smelter return value for contained metal as $/t mineralization
- Ni: total nickel grade percentage
- Co: cobalt percentage
- S: sulphur percentage

### 16.3.4 Cut-off Grade

The NSR cut-off grade is estimated to be the minimum value of the mineralization contained in the designed ultimate pit that is sufficient to cover the cost of milling, G&A, stockpile handling, stockpile maintenance, and the incremental costs for the haul distance from the stockpile from the crusher. The potential pit resource estimate is calculated on an NSR cut-off grade of $7.00/t. This is a conservative value, as the calculated average economic cut-off value is estimated to be around $6.00/t, based on the costs that are available at the early stage of this study (see Table 16.5). The analysis on the cut-off grades will be re-assessed in future studies.

In general, the lower grade mineralization (metal values close to the cut-off grade) mined in the initial years will be stockpiled near the crusher, and then reclaimed as scheduled to blend with the run of mine mineralization feed during the late stages of the mine life.

### 16.3.5 Mineralization Dilution

The Turnagain deposit will be mined by open pit with large trucks and shovels. Large mining equipment will be used to achieve high mining rates ensuring the lowest possible unit costs for mine operations. The waste and mineralization will require blasting and typical grade control methods using blast-hole sampling. Some dilution is anticipated, specifically when waste is mixed in with mineralization during blasting and excavation activities.

Waste is defined to be material below the cut-off grade. The resource model suggests that the dilution material will generally consist of metal grades that are marginally less than economic. The minimum cut-off grade applied in this study is an NSR value that is above the calculated economic cut-off value. Therefore, the dilution material will most likely be of economic value. It is anticipated that the overall grade of the in-pit resource will not be significantly impacted by dilution. Therefore, for this scoping-level study, it is assumed that dilution and mineralization losses will not be material. This assumption will be evaluated in more detail at the next level of study.

## 16.4 Mine Plan Development

### 16.4.1 Preliminary Mining Phases

For the purpose of this study, the Horsetrail pit is sized to produce mineralization for 25 to 30 years life of mine. All pit development will be on the west side of the Turnagain River. Pit phases are designed to allow for mine development that will generate maximum cash flow in the initial years, and balanced pushbacks thereafter.

The starter pit, consisting an east and west area, will be located in the central zone of the Horsetrail Pit — the areas with minimum waste stripping and better mineralization grades. The second phase will be a pushback phase to the north. Phases 3 and 4 will push the wall to the southwest and southeast, respectively, while Phase 5 will push the wall to the north limit of the Horsetrail Pit. These phases have slightly lower nickel grades and higher strip ratios as they approach the higher ground elevations.

The potential Horsetrail pit resource by phase are shown in Table 16.8.

### Table 16.8 Potential Resource by Horsetrail Pit Phases

| Pit Phase | Mineralization (kt) | Waste (kt) | SR | NSR ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| 1 | 100,339 | 16,436 | 0.16 | 23.75 | 0.246 | 0.014 | 0.74 |
| 2 | 124,288 | 57,097 | 0.46 | 22.21 | 0.237 | 0.013 | 0.57 |
| 3 | 146,131 | 41,356 | 0.28 | 20.54 | 0.221 | 0.012 | 0.72 |
| 4 | 148,844 | 45,516 | 0.31 | 19.24 | 0.210 | 0.013 | 1.02 |
| 5 | 243,294 | 157,467 | 0.65 | 22.47 | 0.237 | 0.013 | 0.50 |
| **Total Horsetrail** | 762,896 | 317,872 | 0.42 | 21.60 | 0.230 | 0.013 | 0.69 |

Note: Includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

Table 16.9 summarizes the additional pit resource potential beyond the Horsetrail Pit to the limits of the Ultimate Pit shell. The strip ratio is significantly higher, and the average grades and NSR value are lower in this potential incremental phase. The majority of the resources – greater than 70%, is in the inferred category.

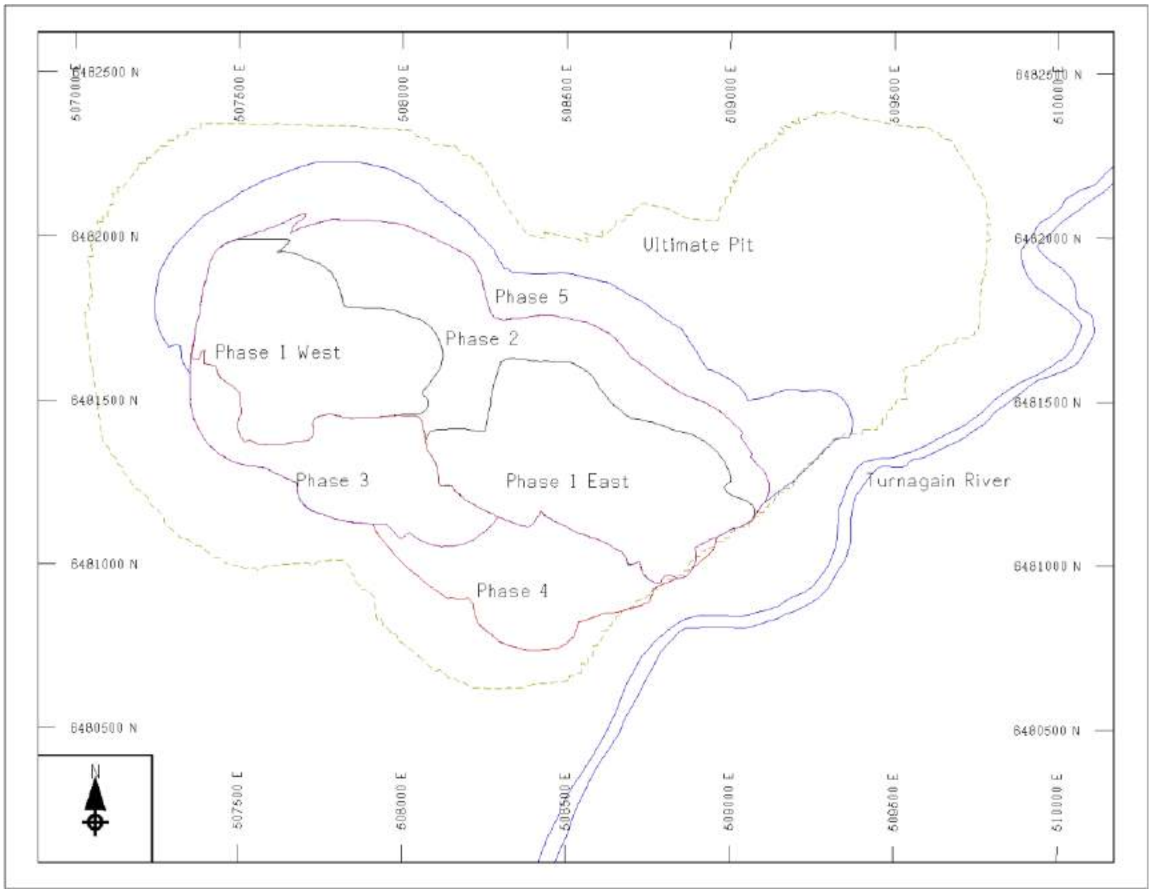
### Table 16.9 Additional Resource Potential to Ultimate Pit Shell

| | Mineralization (kt) | Waste (kt) | SR | NSR ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Measured | 21,119 | | | 17.21 | 0.196 | 0.012 | 0.73 |
| Indicated | 116,890 | | | 17.18 | 0.195 | 0.012 | 0.75 |
| Inferred | 361,259 | | | 19.64 | 0.214 | 0.012 | 0.59 |
| **Total** | 499,269 | 641,776 | 1.29 | 18.96 | 0.209 | 0.012 | 0.63 |

Note: Includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

Figure 16.10 shows a plan view of the pit phases and the outline of the Ultimate pit shell that expands the Horsetrail Pit in all directions except where it is limited by the river.

## Figure 16.10 Horsetrail Pit with Mining Phases – Plan View



## 16.5 Mine Production Schedule

The annual mill feed rate will be variable and dependent on quantities of the dominant mineralization types mined from the pit, specifically the wehrlite-dunite (domain 104) and serpentinized dunite/wehrlite (domain 106). These two mineralization types represent 94% of the total mineralization feed. The latter is a softer mineralization requiring less grinding and therefore mineralization will have a higher throughput rate, while the former is a harder mineralization requiring more grinding time in the mill and will have a lower throughput rate. The limitation will be the total available mill hours per annum estimated at 8,760 hrs. During the first 5 years, the mill rate will be reduced, operating at half the operating rate. Preliminary mill throughput rates of 78,346 tpd and 99,099 tpd were assigned to the 104 and 106 domains respectively, for scheduling purposes. It should be understood that these rates are not necessarily the same as the throughput rates that are the outcome of this PEA study.

Implementing a cut-off grade strategy is commonly used to optimize the mineralization feed to the mill so that the net present value (NPV) of the project is maximized. During the life of mine, mineralization mined at an NSR cut-off value between $17.00/t to 22.00/t will be directly fed to the mill, and mineralization below that cut-off and above $7.00/t will be stockpiled and reclaimed in later years. Elevating the NSR cut-off value further will increase the feed grade but the additional material that will have to be mined and stockpiled will exceed the capacity of the selected loading equipment fleet.

Oxidation of mineralization that has been stockpiled over a long period may be a concern as process recoveries on the reclaimed material may be affected. Metallurgical tests will determine whether the assumed recoveries are sustainable after re-handling and long-term storage, and verify that the cut-off grade and stockpiling strategy is viable.

A preliminary production forecast is shown in Table 16.10. Over the life of the envisaged pit, approximately 763 million tonnes of mineralization will be fed to the mill, of which 535 million tonnes will be directly from the pit. 228 million tonnes of mineralization will be stockpiled and reclaimed at the end of mine life when the pit is depleted. The average strip ratio (waste/mineralization feed to mill) for the first 5 years is 0.6, and varies from 0.2 to 1.4 during the remaining life of the pit. This ratio maybe somewhat misleading as the mineralization stockpile activity is not represented

## Table 16.10 Production Forecast

| Period | Pit Mineralization to Mill (kt) | Pit Mineralization to Stockpile (kt) | Stockpile Mineralization to Mill (kt) | Total Mill Feed (kt) | Nickel (Ni) Feed Grade (%) | Cobalt Feed Grade (%) | Sulphur in Feed (%) | Waste Rock (kt) | Strip Ratio Waste/ Mineralization Feed (t/t) |
|---|---|---|---|---|---|---|---|---|---|
| Pre-production | - | - | - | - | - | - | - | - | - |
| 1 | 15,875 | 5,693 | - | 15,875 | 0.247 | 0.013 | 0.632 | 6,391 | 0.3 |
| 2 | 16,186 | 6,137 | - | 16,186 | 0.258 | 0.013 | 0.479 | 16,965 | 0.8 |
| 3 | 15,258 | 6,505 | - | 15,258 | 0.259 | 0.014 | 0.462 | 8,333 | 0.4 |
| 4 | 15,755 | 3,948 | - | 15,755 | 0.269 | 0.015 | 0.911 | 6,938 | 0.4 |
| 5 | 15,811 | 2,924 | - | 15,811 | 0.273 | 0.015 | 0.952 | 7,446 | 0.4 |
| 6 | 30,813 | 25,042 | - | 30,813 | 0.258 | 0.014 | 0.507 | 13,782 | 0.2 |
| 7 | 31,696 | 21,361 | - | 31,696 | 0.247 | 0.013 | 0.579 | 19,646 | 0.4 |
| 8 | 32,010 | 25,959 | - | 32,010 | 0.254 | 0.014 | 0.660 | 12,683 | 0.2 |
| 9 | 31,335 | 24,206 | - | 31,335 | 0.261 | 0.014 | 0.654 | 15,737 | 0.3 |
| 10 | 31,110 | 24,571 | - | 31,110 | 0.266 | 0.013 | 0.435 | 16,319 | 0.3 |
| 11 | 20,687 | 19,559 | 13,000 | 33,687 | 0.220 | 0.013 | 0.786 | 16,534 | 0.4 |
| 12 | 32,280 | 18,812 | - | 32,280 | 0.238 | 0.014 | 0.825 | 24,859 | 0.5 |
| 13 | 32,037 | 15,672 | - | 32,037 | 0.250 | 0.015 | 0.940 | 27,992 | 0.6 |
| 14 | 20,008 | 5,284 | 14,704 | 34,712 | 0.221 | 0.012 | 0.559 | 33,575 | 1.3 |
| 15 | 20,181 | 3,817 | 10,964 | 31,144 | 0.238 | 0.013 | 0.582 | 28,007 | 1.2 |
| 16 | 30,953 | 6,852 | - | 30,953 | 0.232 | 0.013 | 0.482 | 26,036 | 0.7 |
| 17 | 30,598 | 5,886 | - | 30,598 | 0.236 | 0.013 | 0.417 | 13,253 | 0.4 |
| 18 | 30,162 | 3,052 | - | 30,162 | 0.244 | 0.013 | 0.384 | 7,626 | 0.2 |
| 19 | 29,671 | 1,420 | - | 29,671 | 0.256 | 0.014 | 0.420 | 8,339 | 0.3 |
| 20 | 29,301 | 944 | - | 29,301 | 0.268 | 0.014 | 0.601 | 6,632 | 0.2 |
| 21 | 23,116 | 404 | 5,957 | 29,074 | 0.251 | 0.014 | 0.754 | 777 | 0.0 |
| 22 | - | - | 31,985 | 31,985 | 0.187 | 0.012 | 0.789 | - | - |
| 23 | - | - | 36,171 | 36,171 | 0.168 | 0.011 | 0.854 | - | - |
| 24 | - | - | 31,885 | 31,885 | 0.197 | 0.012 | 0.646 | - | - |
| 25 | - | - | 28,597 | 28,597 | 0.170 | 0.011 | 0.894 | - | - |
| 26 | - | - | 28,597 | 28,597 | 0.170 | 0.011 | 0.894 | - | - |
| 27 | - | - | 22,239 | 22,239 | 0.146 | 0.011 | 1.506 | - | - |
| 28 | - | - | 3,952 | 3,952 | 0.140 | 0.010 | 1.665 | - | - |
| **Total** | **534,844** | **228,051** | **228,051** | **762,895** | **0.230** | **0.013** | **0.686** | **317,871** | **0.4** |

Note: includes Inferred Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that these data will be realized.

## 16.6 Mine Equipment

### 16.6.1 Mobile Fleet

The mine fleet consists of the mobile equipment operating from the pit to the tipping point at the mineralization crusher, and to the waste dump. Crushing and conveying equipment have been included under the process plant section of the PEA.

Table 16.11 summarizes the major equipment fleet, the number of units required at start-up, and the maximum fleet size during the life of mine.

**Table 16.11 Major Mine Equipment Fleets**

| Major Mine Equipment | Purpose | Size | No. Units | |
|---|---|---|---|---|
| | | | Y1 | Max. |
| Blast-hole Electric Drill | Primary Drill | 311 mm | 2 | 3 |
| Diesel Track Drill | Highwall Drill | 155 mm | 1 | 1 |
| Cable Shovel | Production Loading | 36 $m^3$ | 2 | 3 |
| Wheel Loader | Backup Loader & Stockpile Handling | 18 $m^3$ | 1 | 1 |
| Haul Trucks | Production Haulage | 218 t | 15 | 27 |
| Track Dozer (D10 Equivalent) | Road Development & Maintenance | 425 kW | 2 | 2 |
| Track Dozer (D9 Equivalent) | Pit Maintenance | 302 kW | 1 | 3 |
| RT Wheel Dozer | Shovel Support | 597 kW | 1 | 2 |
| Hydraulic Backhoe | Ditch Construction & Maintenance | 5 $m^3$ | 1 | 1 |
| | Pit Maintenance | 3 $m^3$ | 1 | 1 |
| Grader (16M Equivalent) | Road Maintenance | 205 kW | 3 | 3 |
| Water/Sanding Truck | Road Maintenance | 20,000 gal | 2 | 3 |

The size and production rate of the mine will accommodate large mobile equipment, and will lead to lower unit mining costs. The primary load and haul equipment fleet will consist of 36$m^3$ electric cable shovels and 218t trucks. The shovel size was selected to meet the annual mineralization and waste production requirements from the pits. Each unit has the capacity to produce an average of 23 million tonnes per year. It is estimated that a maximum of three shovels will be required during the life of mine. Electric cable shovels were chosen over hydraulic excavators due to their lower operating costs. It will therefore be necessary to have power supplied to the pit area.

The size of the trucks has been selected to match the shovel output so that no more than four passes are necessary to fully load the truck. The size of the fleet is determined by estimating the haulage productivities for mineralization and waste. Preliminary estimates on the haulage productivities indicate that 15 units will be required by Year 2, increasing to a maximum of 27 units by Year 16. The drill fleet will consist of three 311 mm diameter electric blast-hole drills for production drilling. One 155 mm diameter diesel track drill will be required for pre-shear holes and pioneering.

Pit support equipment will include rubber tire dozers for pit floor maintenance near the shovel faces, track dozers, and backhoes for road development, maintenance, and ditching. The road maintenance fleet will also include motor graders and a water/sanding truck.

Ancillary mine equipment will include light duty vehicles, service trucks, cranes, utility backhoes, blast-hole stemmers, lighting plants, and other equipment required to support the mine and maintenance areas of the operation.

### 16.6.2 Mine Buildings

On-site mine service buildings will include a heavy-duty truck shop, mine dry, light duty vehicle shop, wash bay, warehouse, fuel depot and distribution, assay laboratory facility, process control room, and administration building,

Blasting explosives will be manufactured on-site, and the explosives plant will be housed in a secure structure. The plant and storage facilities will be located a minimum distance away from the central plant site and pit, in compliance with regulatory requirements.

## 16.7 Waste Rock Management Facility

### 16.7.1 Waste Disposal Strategy

Mine waste rock will be initially hauled from the Horsetrail Pit and placed on a waste dump located south west of the pit. The preliminary waste dump design is constrained by the Turnagain River and Hard Creek. Currently-available rock chemistry data indicates the waste rock to be non-potential acid generating. The majority of the rock is assumed to be competent and able to be end-dumped on 60m lift intervals. Poor-quality material will be blended with competent waste rock at an acceptable ratio, or placed in a contained area within the dump. The foundation under the waste dump has not been analyzed for stability. It may be necessary for the waste dump to be further offset from the river and creek if future geotechnical studies raise concerns over the ground conditions.

Waste rock will be required for tailings dam construction, and will be delivered from the pit each year from Year 1 to Year 13. Approximately 111 million tonnes of suitable waste rock will be required over this period.

The starter dam will be constructed during the pre-production period using material from a nearby borrow pit. There will not be sufficient waste rock readily available from the pit without stockpiling significant mineralization. It is also anticipated that the mine mobile equipment fleet will not be available at this early stage of the project to perform the work, and excavating and hauling activities will be carried out by a mining contractor.

## 16.8 Capital Cost Estimate

The summary for the mine capital costs is shown in Table 16.12. All costs in this section are provided in 2011 CDN dollars.

### Table 16.12 Mine Capital Cost Estimate

| MINE CAPITAL COST ESTIMATE | | M$ |
|---|---|---|
| Pre-Strip | | 0.0 |
| Mine Equipment Fleet | | 147.4 |
| Mine Buildings Costs | | |
| | Heavy Duty Truck Maintenance Shop | 17.0 |
| | Light Duty Vehicle Shop and Warehouse | 4.0 |
| | Fuel Depot, Distribution and Storage | 2.5 |
| | Explosives Facility | 2.0 |
| | Administration, Dry, and other Buildings | 4.0 |
| | Cold Storage Building - Warehouse | 3.9 |
| | Control Room PLC | 0.7 |
| Subtotal | | 34.1 |
| Other Costs | | |
| | On-site Power | 5.3 |
| | Site Prep | 5.5 |
| | Engineering | 7.0 |
| | Mine Inventory | 14.7 |
| Subtotal | | 32.6 |
| Totals | | 214.1 |
| Contingency | @20% | 42.8 |
| Total Mine Capital | | 256.9 |

Pre-stripping will not be necessary as the initial mineralization feed will be near surface and accessible when the plant starts up. Material for the tailings starter dam will be sourced from a borrow pit close to the dam site rather than from the pit. As indicated above, it is anticipated that the mine equipment fleet will not be available on site at this early stage of the project and a mining contractor will be procured for this work. The costs for the construction of the initial tailings dam and the haul road to the dam are not in the mining capital costs, and are included in the process plant capital costs.

The cost for the initial mine equipment includes the fleet requirement to meet the production in Year 1.

The equipment pricing is based on new units delivered to the mine, with all transportation and erection costs included. Most unit prices are based on recent vendor budgetary quotations. Others are sourced from the MMTS equipment database. Used equipment, if available, will reduce these equipment capital costs, and have not been considered for this study.

The cost estimate includes a 25 kV line and sub-stations required to supply power to the mine, primarily to operate the cable shovels and the blast-hole drills. The origin of supply for this estimate is the process plant.

The site-prep cost is an allowance for clearing and grubbing, site drainage, initial access roads, and the main haul road from the pit to the crusher, mineralization stockpile and waste dump. No detailed designs have been undertaken for this estimate.

The engineering capital cost is an allowance for expenditures for computer supplies, surveying equipment, truck dispatch system, geotechnical, environmental and mine design studies.

In the capital costs, 10% of the equipment capital is included for spare parts such as truck tires, loading buckets, shovel teeth, drill bits, shovel cables, drill cables, and other significant consumable items such as fuel.

## 16.9 Operating Cost Estimate

All costs in this section are provided in 2011 CDN dollars. The average mine operating cost is estimated to be US$1.87/t material mined or US$2.65/t mineralization milled over the life of mine. The average operating cost for material moved is $1.55/t and this cost includes the material movement to and from the mineralization stockpiles. The cost estimate consists of all mining activities from the pit to the tipping point at the mineralization and waste truck dumps. Mine and maintenance activities associated with loading, hauling, drilling, blasting, pit support, mine maintenance support, general mine expense, and engineering are included. Table 16.13 shows the estimated operating costs for each of the areas.

### Table 16.13 Mine Operating Cost Estimate

| | $/t Mined | $/t Mineralization Milled |
|---|---|---|
| Drilling | 0.07 | 0.11 |
| Blasting | 0.23 | 0.33 |
| Loading | 0.21 | 0.30 |
| Hauling | 1.00 | 1.42 |
| Pit Support | 0.17 | 0.24 |
| Mine Maintenance Support | 0.03 | 0.04 |
| General Mine Expense | 0.15 | 0.20 |
| Engineering | 0.01 | 0.01 |
| **Totals** | **1.87** | **2. 65** |

Preliminary equipment productivities are generated and applied against the annual production quantities to estimate equipment operating hours. Consumption rates for consumables and unit operating costs are applied to the equipment hours to calculate the total equipment operating costs for each period. The cost of parts and repairs are included in the operating costs for the major mining equipment. Major part replacements are based on manufacturer's recommendations and are expensed in the year in which they are forecasted.

The operating costs fluctuate annually and are a reflection of the total material mined and haulage distances. Balancing the waste quantities and the haulage destinations will smooth the operating costs and minimize fluctuation. Some smoothing is applied to the production schedule in this study.

The costs for power and diesel, respectively, are $0.0411/kWh and $1.05/L. These unit prices are consistent with those assumed by AMC for application in other areas of the operation. Mine operations power costs are calculated utilizing the estimated kilowatt hours for each year of operation. Peak annual power consumption is estimated to be 31 million kilowatt hours, with the average being 26 million kilowatt hours per year during the high usage period between Years 6 to 21. Fuel consumption for explosives and the mine equipment fleet is estimated to average 24 million litres per annum during the Years 6 to 21 when the mine operation is at full capacity.

Explosives quantities were calculated using typical powder factors from existing operations and projects similar in nature. Recently-attained explosives and accessories unit costs for mines in BC were applied to the projected quantities to estimate costs for blasting materials. It is estimated that the average annual explosives consumption will be 15 million kg during the high consumption period from Years 6 to 21. Ammonium nitrate/fuel oil (ANFO) usage has been assumed to be 75%, based on successful dewatering to ensure dry blast-holes most of the time. If dewatering is not successfully implemented, emulsion explosives usage and blasting costs will increase substantially.

Salaries and hourly labour rates are based on previous estimates from the 2009 PEA study, adjusted by 3% for inflation. The labour rates were applied to the operating and maintenance workforce generated from the equipment fleet to determine the total hourly labour cost. Salaries were applied to the total staff estimate to arrive at the salaried cost.

Dewatering costs include only allowances for in-pit pumping. Only limited hydro-geological studies to determine whether water inflows will be significant have been undertaken, and further work should be carried out in the next study phase to provide a more definitive cost estimate.

### 16.9.1 Mining Labour

The mine workforce estimate is summarized in Table 16.14. The hourly labour workforce reflects the annual quantity of material mined. During the first five years at the lower production rate the total mine labour count averages 134. It increases to an average of 240 during next five year period when the mineralization production rate is doubled. When the pit is completed and the mill is fed from the mineralization stockpiles after year 21, the mine labour force will be reduced significantly.

### Table 16.14 Mine Operations Labour – Average for Periods

| | Years | | | |
|---|---|---|---|---|
| | **1 to 5** | **6 to 10** | **11 to 21** | **After 21** |
| **Hourly Labour** | | | | |
| Equipment Operators | 77 | 142 | 139 | 29 |
| Mine Maintenance | 29 | 59 | 55 | 14 |
| **Sub-total** | **106** | **201** | **194** | **43** |
| **Salaried Staff – Mine and Maintenance Operations** | | | | |
| Mine Superintendent | 1 | 1 | 1 | 1 |
| Maintenance Superintendent | 1 | 1 | 1 | 0 |
| Mine & Maintenance General Foremen | 2 | 2 | 2 | 1 |
| Shift Foremen/Team Leaders | 10 | 12 | 12 | 6 |
| Trainers | 1 | 1 | 1 | 0 |
| Maintenance Planners | 1 | 1 | 1 | 1 |
| Clerks/Dispatchers | 4 | 8 | 8 | 4 |
| **Sub-total** | **20** | **26** | **26** | **13** |
| **Mine Technical** | | | | |
| Chief Engineer | 1 | 1 | 1 | 1 |
| Geologists | 2 | 3 | 3 | 1 |
| Mine Engineers | 3 | 5 | 5 | 1 |
| Technicians/Surveyors | 2 | 4 | 4 | 2 |
| **Sub-total** | **8** | **13** | **13** | **5** |
| **Total Salaried Staff** | **28** | **39** | **39** | **18** |
| **Total Mine Workforce** | **134** | **240** | **233** | **61** |

## 16.10 Mining Related Opportunities and Risks

## 16.11 Increased Production Rate with Mine Resource Expansion

As indicated, a significantly larger potential resource exists within the optimized economic pit shell, and with the inclusion of the Hatzl deposit. Increasing the size of the mine and production capacity will capture that opportunity, and by extension, generate higher revenues. The limiting factor is the high infrastructure cost of power supply to meet the demand of incremental production.

### 16.11.1 Geotechnical and Hydrogeological Risks

A limited number of geotechnical and hydrogeological studies have been conducted on pit wall stability and waste disposal locations. Pit slopes may have to be reduced, thereby increasing the strip ratio. It is assumed that waste dumps will be constructed by low-cost end-dumping methods. Operating costs will increase if the waste dumps have to be constructed in limited-height lifts due to foundation weakness or poor material quality issues.

Groundwater sources, and particularly relative to the proximity of the Turnagain River, have to be studied in order to implement a dewatering plan. The low-lying areas within the potential pit limits are receptors for water draining off the slopes, and are deemed to be saturated. Hydrogeological studies will be required to determine the degree of dewatering necessary to keep the pit under dry operating.

### 16.11.2 Other Risks

There is an increased cost risk relative to potential for oxidized mineralization in stockpiles.

There is some risk to mining plan and schedule relative to about 26% of projected mineralization in the Horsetrail Pit being inferred material, but this is to be expected for this level of study.

# 17 RECOVERY METHODS

## 17.1 Introduction

The 2010 Wardrop PEA had contemplated a conventional crush-grind-flotation flowsheet to treat 87,000 tpd of mineralized rock and produce a 4% Ni concentrate as feed to the hydrometallurgical refinery.

Although the metallurgical work described in Section 13 of this report has resulted in a significant project scope change by being able to produce a saleable concentrate therefore eliminating the refinery, the impact on the concentrator design is not that significant. The crushing, grinding and rougher flotation stages have to meet very similar design criteria and the only significant changes are in the cleaner flotation section.

One significant change is the phased approach whereby the mine starts up at 50% of full capacity i.e. 43,500 tpd for the first five years and then is expanded to the nominal 87,000 tpd thereafter. The implications of this are discussed briefly in Section 17.4

## 17.2 Process Flowsheet

A simplfied process flowsheet is shown in Figure 17.1.

As no new comminution testwork has been carried out and rougher concentrate mass flows are very similar, the front-end of the circuit remains essentially unchanged relative to the previous study.

The elevated cut-off grade strategy described under mining methods in Section 16 effectively reduced the mineralization types in the mill feed to two, domain 104 dunite/wherlite and domain 106, serpentinised dunite/wherlite. The significantly harder grinding indices of the 104 domain were factored in to the pit optimization process to favour the selection of the softer 106 material.

The process plant will consist of the following unit operations:

- Run of Mine (ROM) crusher feed hopper
- Two parallel trains of primary crushing:
  - Conveyors
  - Crushed mineralization stockpile
- Stockpile reclaim
- A SAG –Ball Mill-Pebble crusher (SABC) circuit comprising:
  - 2 parallel trains of:
    - 1 12.2 x 6.7m SAG mill with 17.65 MW ring motor
    - 2 7.9 x 12.5m ball mills each with a 6.6 MW motor
    - Cyclone classification
- Four parallel trains of rougher flotation:

    - each with 6 x 500$m^3$ tank cells (note that the opportunity has been taken in this update to utilize the largest flotation cells available (compared to the 300 $m^3$ cells in the previous study in order to gain some capital costs and power consumption efficiencies)
- One train of cleaner flotation in three stages:
    - 8 x 300$m^3$ first cleaners
    - 9 x 200$m^3$ second cleaners
    - 5 x 100$m^3$ third cleaners

Note that excess cleaner capacity has been retained in light of comments made in Section 13 regarding the merits of a split cleaner circuit; also even with the single cleaner circuit as set out it is probable that a cleaner tail scavenger train may be a better option than merely recycling cleaner tails to the roughers. This is a topic for further investigation in the next phase of study.

- Concentrate thickening and filtration
- High head tailings pumping system (considered and costed under the tailings management facility category)
- Tailings facility reclaim water system (similarly dealt with in theTMF section) for process water
- Fresh water will be used for pump gland seals, reagent mixing and other special requirements e.g. hydraulic unit cooling

A schematic plant layout is shown in Figure 17.2. Note that all services (Including camp) are grouped together in the interests of effective heating and personnel transit during cold weather.

## Figure 17.1 Simplified Process Flowsheet

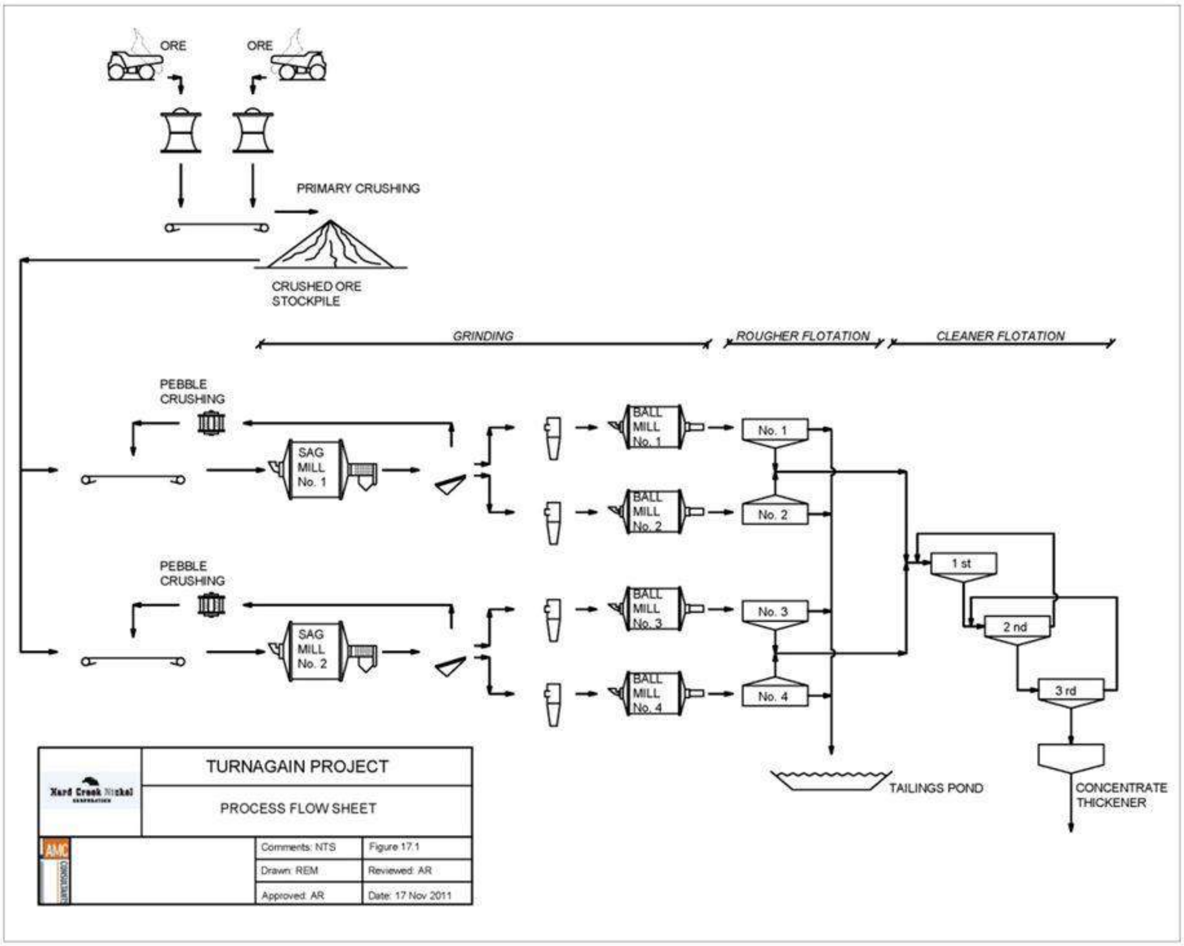
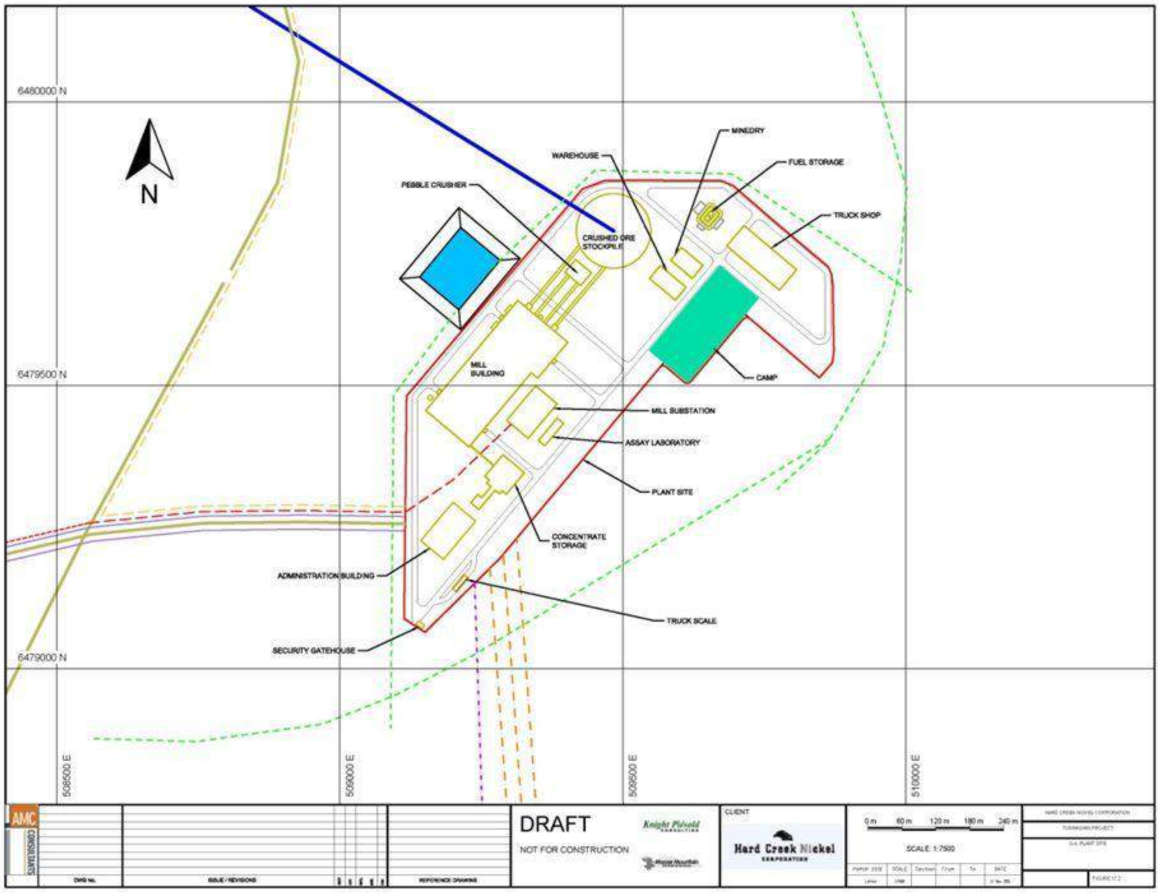


**Figure 17.2 Schematic Plant Layout**




## 17.3 Phased Expansion

The circuit, as originally designed with two grinding trains and four rougher flotation trains, lends itself well to a phased expansion.

The first phase at 50% of the full capacity will simply comprise one grinding train and two rougher flotation trains. The single cleaner circuit will be reduced to the minimum number of cells in series (typically 5) to prevent short-circuiting with space allowed for additional cells as the second phase. In the case of the first and second cleaners this approximates to half capacity but the third cleaners remain at the original capacity and froth crowders will probably be necessary to promote froth discharge at the reduced throughput.

For the first five years the plant throughput will average 15.8 Mtpa and then for years 6-21 the average throughput will be 31.3 Mtpa. For years 22-28 the plant will be fed with low grade stockpiles of low grade, generally harder mineralization (as per the pit optimization process) at an average throughput of 29.9 Mtpa.

# 18 PROJECT INFRASTRUCTURE

## 18.1 Site Access Road

The communities of Terrace and Smithers in BC, and Whitehorse in Yukon, are all several hundred kilometers away and offer the best range of supplies and services, which can be trucked to Dease Lake via Highway 37. Dease Lake is a small community located in northwest BC, and west of the Stewart-Cassiar Highway 37, approximately 250 km south of the Yukon border. The property can be accessed by a secondary road extending east from Dease Lake. The road has been used by large articulated 4-wheel drive vehicles to access the Kutcho Creek area local jade extraction, Eaglehead gold-copper exploration, and to supply gold operations at Wheaton Creek. A branch of this road network extends into the Turnagain property.

Road distance to Dease Lake from site is approximately 71 km. The access road from Highway 37 to the property will require upgrading. There is one major stream crossing and approximately 50 other minor crossings. These crossings are currently passable with light vehicles but will need upgrading for larger trucks. The major crossing will require a bridge, however culverts are suggested for the minor crossings.The existing access road will be upgraded and widened to 5m private mine access single lane road with intervisible turnouts and radio control. There would be an opportunity to share the cost of upgrading 64 km of this road with Capstone Mining Corporation.

The haul road from the open pit to the TMF is approximately 4 km, and will be upgraded to 36m wide to accommodate mine trucks. There are two stream crossings that will require bridges. one for the main access road and another for the haul road. To be able to efficiently access all areas of the planned site, a network of internal roads will be required. These roads will access the open pit mine, waste dumps, tailings storage facility, crusher building, plant site, and all permanent and service facilities that would support the mine.

## 18.2 Waste Management Facility Alternatives

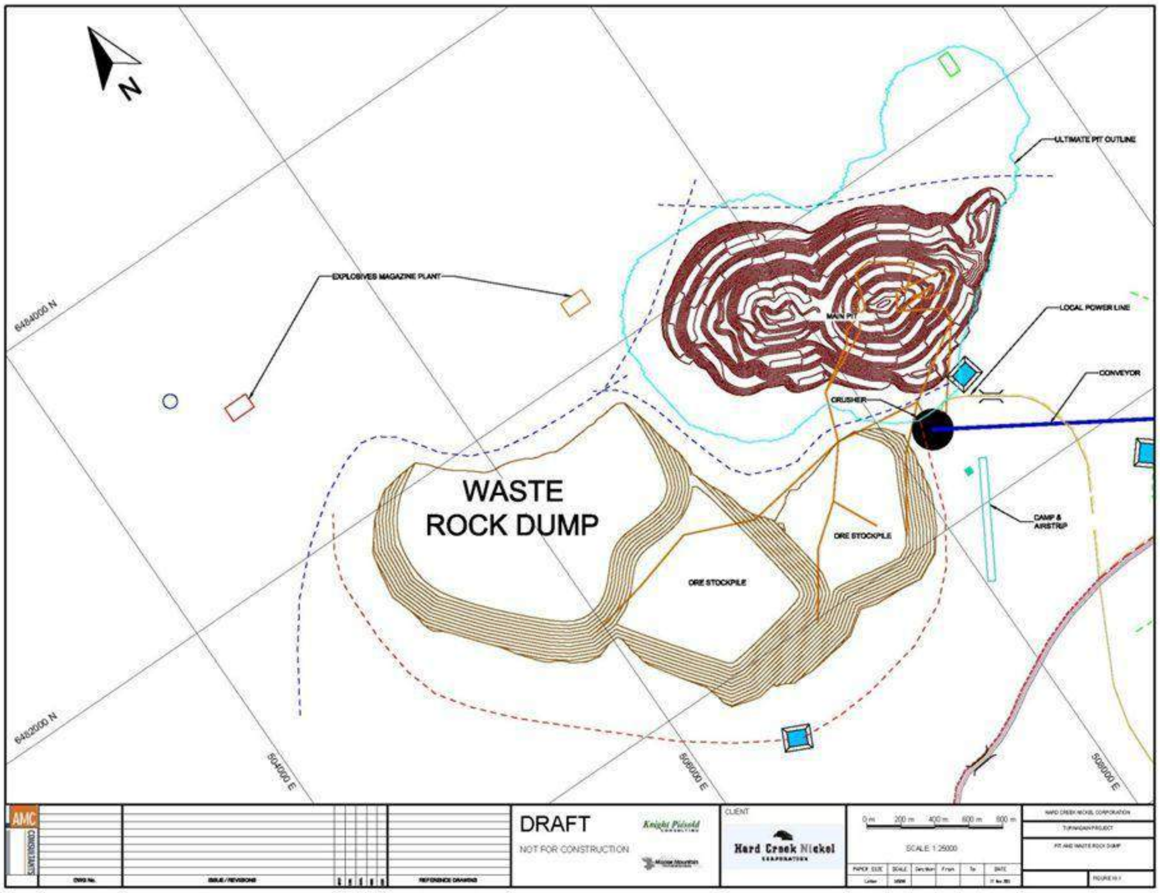
In April 2006, KP completed a preliminary waste management facility alternatives study (Ref. No. VA06-00593), which identified a number of potential tailings management facility (TMF) sites in the vicinity of the Turnagain deposit. Flat Creek valley was acknowledged as a potential option for a tailings impoundment for the Turnagain Project. In July 2007, a preliminary mine development alternatives assessment was subsequently carried out (Ref. No. VA07-01017), which explored several mine development alternatives including the location of the plant site, waste dumps, a low grade stockpile, haul and construction access roads, and tailings and water reclaim pipelines.

## 18.3 Waste Rock Management Facility

### 18.3.1 Non-Reactive Waste Rock

Non-reactive waste rock will be stored in a conventional sub-aerial (surface) dump or within the mined-out open pit later in the mine life. The approximate final waste dump footprint is shown on Figure 18.1

**Figure 18.1 Final Waste Dump Footprint**




### 18.3.2 Potentially Reactive Waste Rock

Potentially reactive waste rock will be stored in the surface waste dump and within the mined-out open pits using best management practices. The waste rock is generally not expected to exhibit acid generating properties but may be neutral metal leaching.

The waste rock characterization to date is largely inconclusive on the acid generating potential of the waste rock, since most of the testwork was carried out on mineralized material. Although a small percentage of the waste may have acid generating potential, it is expected to be insignificant with respect to the overall neutralizing potential of the waste rock pile. It is assumed that any potentially acid generating (PAG) material encapsulated in the dump would be surrounded by a sufficient quantity of rock with high neutralizing potential. It is therefore highly unlikely that acid rock drainage would emanate from the dump, both during operations and post-closure. The risk posed by PAG waste for the project is accordingly low.

If necessary, a treatment process will be incorporated into the water management plan early in the mine life, to allow for treatment of any effluent from the waste dumps that may be affected by neutral metal leaching. The plant will produce a metal oxide precipitate with total metal content too low for effective recovery and therefore will be added to the plant tailings for pumping to the tailings management facility.

## 18.4 Tailings Management Facility

### 18.4.1 Design Basis and Operating Criteria

The principal objective of the design and operation of the TMF is to ensure secure containment for tailings solids and impounded process water. The TMF will serve as the primary water management facility for the project, providing a buffering volume for the mill process water demands, as well as collecting and storing the necessary quantities of precipitation and runoff.

The mill throughput is assumed to be approximately 43,000 t/d for the first five years of operation and approximately 87,000 t/d thereafter. The total tailings production is assumed to be about 87,000 t/d, or 100% of the milled mineralized rock. Tailings from the mill will be discharged to the TMF as a slurry at an average unthickened solids content of approximately 35% (by weight).

The starter TMF is sized to store the estimated volume of tailings produced in the first two years of operation. The final facility is sized to store the estimated 757 Mt of tailings produced over the planned mine life of 28 years.

The potential presence of minor amounts of fibrous mineral forms of magnesium silicate in the tailings necessitates that the tailings solids be kept submerged to the extent possible during operations. The supernatant pond will be maintained over the full extent of the facility with a minimum of exposed beach area during operations.

The facility makes use of the favourable topography to yield a relatively large storage volume as compared with the quantity of material required for embankment construction. Likewise, significant increases in storage capacity can be realized with moderate increases in the elevation of the confining embankments.

### 18.4.2 Layout and Operating Strategy

#### 18.4.2.1 Tailings Management Facility Embankments

The main TMF embankment will be raised in five stages, with each stage providing the required capacity for that particular period until the next stage is completed, while always maintaining minimum storm water storage, wave run-up, and freeboard requirements. It is expected that the staged design of the embankment will be reviewed annually and refined as required, to accommodate the availability of construction materials and to incorporate experience gained with local conditions and constraints.

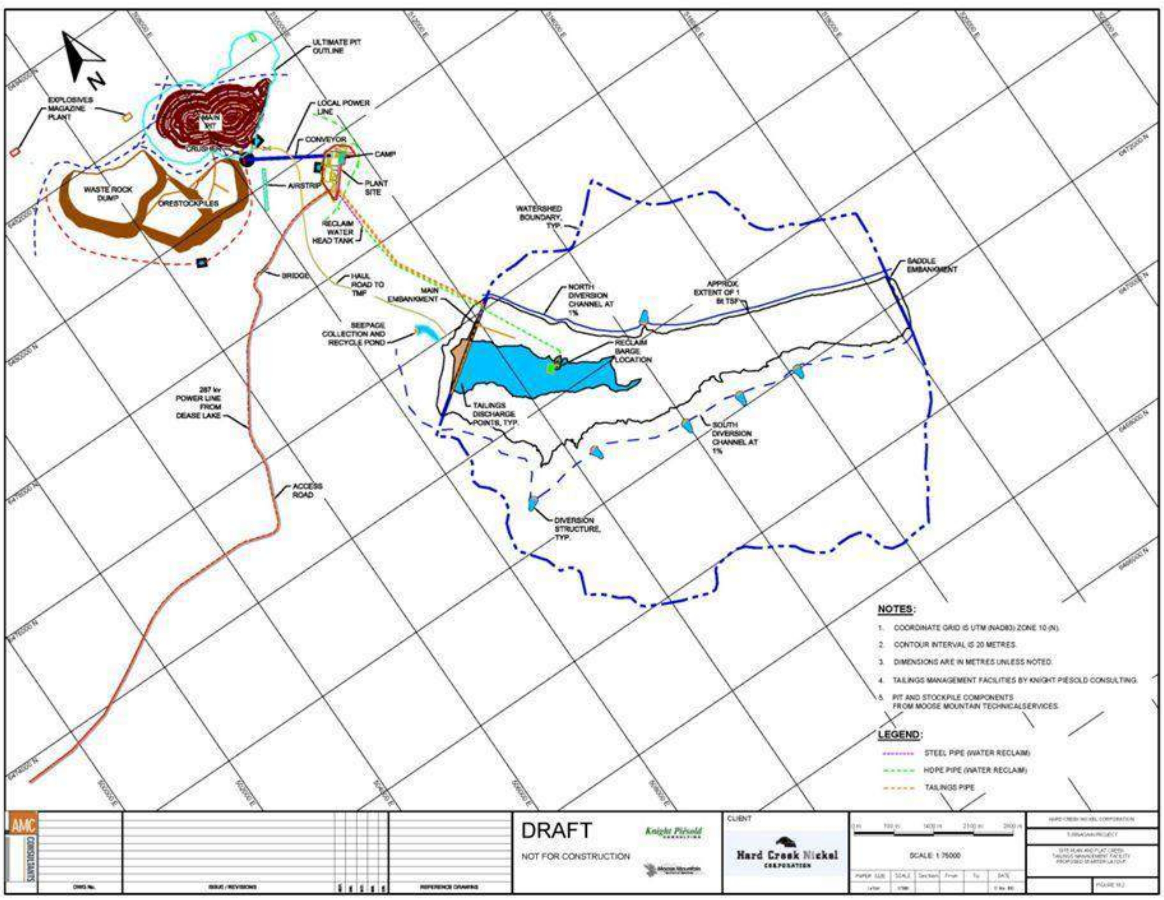
A small temporary coffer dam will be constructed on Flat Creek upstream of the main TMF embankment footprint. This dam will allow the TMF starter dam foundation area to be dewatered, cleared, and stripped prior to preparation for construction of the embankment.

A network of 150 mm-diameter perforated high-density polyethylene (HDPE) interceptor pipes placed in a dendritic or herringbone pattern will underlie each dam foundation. The drains will be surrounded by appropriate filter and drainage materials. The individual interceptor drains will connect to 300 mm diameter HDPE main collector pipes to transport seepage to recycle ponds located at the topographic low points below each embankment.

The underdrain network will be expanded as the staged embankments are constructed. They will also provide foundation dewatering during initial construction.

The TMF embankment construction will begin with a two-phase starter dam. The starter dam will be located at the northwest end of the Flat Creek valley, and built as a water retaining structure with a 2.5H:1V upstream and 2.25H:1V downstream slope, as shown in plan on Figure 18.2.

**Figure 18.2 TMF Starter Dam**



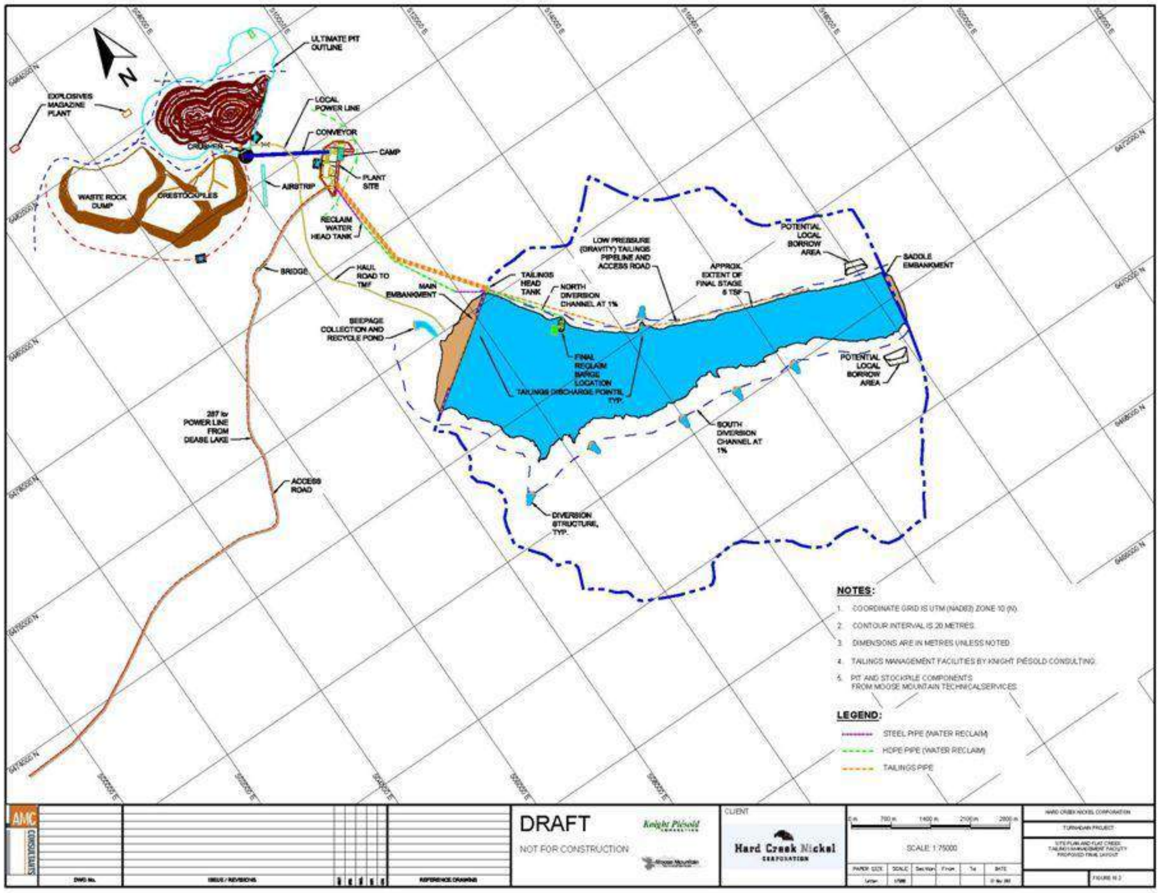

The dam will be constructed in two phases (Stage 1a and Stage 1b) to provide storage for the Year 1 and Year 2 tailings, respectively. Stage 1a will be completed in Years -2 and -1, before mill start-up, and Stage 1b will be built as a downstream raise before the beginning of Year 2. Stage 2 must be completed prior to the start of Year 3 of operations. A synthetic liner system will be placed on the upstream face of the starter dam to provide a continuous hydraulic cut-off. The liner will tie into a concrete plinth or slurry trench along the upstream toe of the starter dam. Blanket and curtain grouting will be employed up to the final elevation of the starter dam as required to ensure continuity of the hydraulic cut-off.

The initial embankment will be built using a combination of local borrow materialand material from the mill site excavation. Suitable material will be placed and compacted to achieve the required permeability and satisfy embankment stability criteria. The embankment will impound an initial fresh water pond prior to start-up of processing operations.

The centreline method of embankment construction will be used for ongoing raises (Stage 2 through Stage 5) of both the northwest main embankment and the southeast saddle embankment. Each stage has a minimum horizontal width of 55m to allow placement of waste rock by the mine fleet. Rock will be dumped, spread, and compacted to the specified density. Both embankments will have a minimum final crest width of 40m to allow for vehicle access and pipelines. The dams will be designed as water retaining structures to allow the supernatant pond to submerge the tailings beaches to the extent possible. A central low-permeability core zone will provide a positive hydraulic cut-off in the dam.

Appropriate filter zones will be placed downstream of the low-permeability core. The core and filter zones will be raised with each stage of the dams. It is envisaged that these will be constructed in the summer months by a specialized contractor using locally sourced borrow material. Ongoing construction of the main northwest dam shell zone will make use of suitable low-sulphur, geochemically innocuous waste rock from the open pit. The shell zones will need to be constructed prior to beginning work on the crest, core, and filter zones to allow access for equipment. Waste rock will be hauled to the dam by the mine fleet. Both geochemical and geomechanical waste rock characterizations are needed to confirm that this is technically and economically feasible. The final general arrangement at the end of the mine life is shown in Figure 18.3.

**Figure 18.3 TMF Final Arrangment**



A much smaller saddle embankment will be constructed at the southeast end of the TMF. It will be built using local borrow material in much the same manner as the main embankment. The first stage of the saddle dam will need to be completed by the beginning of Year 8 of mine operations. The saddle dam will be constructed using the same centreline method as the main dam, and will be comprised of similar fill zones. All materials used in construction of the saddle dam will be sourced from local borrow. Borrow material will be taken from within the TMF impoundment area or diversion channel excavations where possible to minimize the total project footprint. Borrow areas may also be developed in the hill slopes immediately above the TMF.

Seasonal construction restrictions for the various fill materials and zones should be considered. The coarse, free-draining rock fill that makes up the majority of the embankment volume can be placed year-round and in most climatic conditions. The fine-grained but free-draining filter zones should only be placed when temperatures are above freezing. Ice crystals in sandy materials are difficult to detect, even with manual-visual techniques, and must not be included in the embankment fill. The fine-grained, low-permeability core zone should only be placed during periods of above-freezing temperature. Placement of fine-grained soils will also be less efficient during times of heavy and frequent precipitation. The best months for core zone construction are anticipated to be late April, May, August, September, and early October. It is anticipated that this will provide sufficient time to place the required volume of material estimated for the current dam design.

#### 18.4.2.2 Mill Tailings Transport and Deposition System

The tailings transport system will be constructed in stages throughout the life of the project. Tailings will initially be pumped from the mill directly to the main TSF embankment through a 560 mm-diameter rubber lined steel pipeline; a 560 mm-diameter HDPE pipeline will be used to distribute tailings to offtakes located along the embankment. Beginning in Year 6, slurry tailings will be pumped from the mill site to a head tank at an approximate initial elevation of 1264m through twinned 560 mm-diameter rubber lined steel pipelines. The high-pressure system will be designed with 50% back-up redundancy in the event that one pipeline is unavailable. Tailings will then flow by gravity through a single 1,067 mm-diameter HDPE pipeline towards the southeast side of the facility along the southwest bank of the TMF. The head tank and low pressure pipeline will be moved up the hillside as the TMF elevation rises. A preliminary trade-off study suggested that the added cost of relocating the pipeline is offset by the savings in reduced pumping power requirements. It is anticipated that the pipeline will be completely replaced during the relocation, minimizing interruptions to the operation of the mill. The staging of the tailings system corresponds with the average anticipated HDPE tailings pipeline replacement schedule. The advantages of a staged layout are:

- reduced overall pumping power costs
- staged construction of tailings pump station
- reduced quantities of costly high pressure pipe, valves and fittings
- reduced size of the required drain-back pond at the pump station
- gravity drainage of the majority of the pipeline towards TMF following loss of pumping power
- secondary containment ditches generally drain towards the TMF in the event of pipeline leakage or failure

Tailings will initially be discharged from a series of valved offtakes along the northwest embankment, and along the southwest bank of the TMF after Year 6. Tailings slurry will be deposited onto small, low-angle sloped beaches. Beach development will be managed to ensure constant saturation of the tailings solids, minimizing the potential for desiccation and dusting.

The rising elevation of the TMF with time will need to be considered in more detail in future designs, as the tailings delivery system layout and hydraulic requirements will change throughout the project life.

#### 18.4.2.3 Reclaim Water System

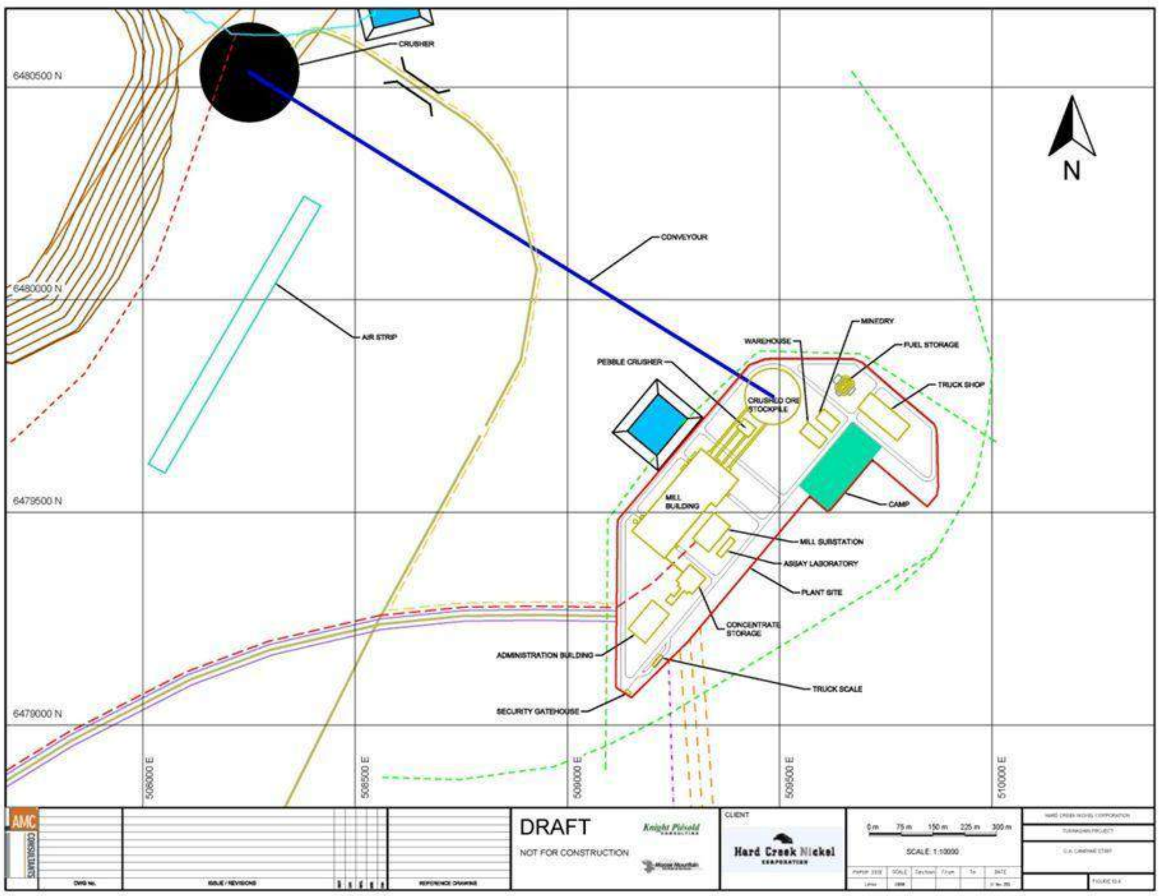
The reclaim water system illustrated in Figures 18.2 and 18.3 will comprise both pumped and gravity flow components. Supernatant reclaim water from the TMF will be pumped over the dam to a head tank directly above the mill site at an elevation of roughly 1200m. This will provide approximately 100 psi (75m) of pressure at the mill. As the supernatant water level in the TMF rises throughout the mine life, pressure in excess of the required 75m will be developed at the head tank. This excess pressure, which will reach 185 psi (130m) in the later stages of the project, will require energy dissipation by installation of an orifice valve or other throttling device.

Reclaim pumps will be mounted on a floating barge in the TMF supernatant pond and will operate on level control from the head tank. Additional pumps will be added in Year 5 to accommodate the increased throughput at the mill.

## 18.5 Plant Site Layout

Figure 18.4 shows the plant site layout including the main concentrator plant, substations, and all associated ancillary buildings.

**Figure 18.4 Plant Site Layout**




## 18.6 Ancillary Facilities

The project infrastructure and services have been designed to support an operation of 87,000 t/d of nickel/cobalt mineralized rock to produce concentrate. Infrastructure and ancillary facilities will be comprised of the following pre-engineered structures or stick-built structures:

- administration building
- Camp

- warehouse building
- mine dry
- open storage area
- truck shop
- assay lab and metallurgical laboratory
- fuel storage and distribution including fuel station
- security/ gatehouse
- power supply and distribution
- communications system
- sewage system
- water supply

Careful attention was given to the placement of the facilities in order to minimize the overall footprint and required excavation. The layout as a whole takes advantage of the natural slope in the area.

### 18.6.1 Administration Building

The administration building will be a modularized structure that will provide working space for management, supervision, geology, engineering, and other operations support staff.

### 18.6.2 Camp

There are provisions for a modularized structure construction camp in the capital estimate. At a later stage this structure can be converted into a temporary or permanent accommodation facility for the workforce. The construction camp is located at the plant site.

### 18.6.3 Maintenance/ Warehouse

A stick-built maintenance/warehouse facility will be provided to service the mobile equipment and for storage of equipment spares. One repair bay will be provided for servicing light vehicles, which will also be used for tune-ups. Small tools and equipment will be provided. A waste oil system, exhaust system, lube-oil system, water system, small machine shop and equipment, and welding bay will be included.

The maintenance area will be equipped with a crane. The warehouse area will be sized to accommodate process materials and the maintenance shop supplies.

### 18.6.4 Mine Dry

A separate mine dry facility, including lockers and shower facilities, will be provided. The mine dry will be a modularized structure located at the plant site.

### 18.6.5 Open Area Storgage

No open area storage area has been specifically allocated; however, there is area within the plant site that could be fenced off to provide extra storage for equipment and materials, if required.

### 18.6.6 Truck Shop

The truck shop building will be a pre-engineered building, with an overhead clearance of at least 10m. The building will be designed to provide facilities for maintenance and repair, minor office space, clean and dry areas, and general storage. It will be at the plant site near the mining haul road. The truck shop will house two maintenance bays, one light vehicle repair bay, a welding and machine shop, and an electrical and instrument shop. The truck wash and tire change building is included within the truck shop. The building will contain a wash bay, maintenance bay, tool crib, compressor room, hot water pressure system, tire change, and an oil separator. Waste oil will be disposed of in the refuse incinerator with any remaining oil removed and discarded at an approved facility.

### 18.6.7 Assay Laboratory

An assay laboratory will be located in a separate modular building at the southeast side of the mill building. The laboratory will be a single-storey structure equipped to perform daily analyses of mine and process samples.

### 18.6.8 Fuel Storage and Distribution

An area has been designated near the truck shop for the storage and dispensing of fuel. The fuel storage and dispensing facility will include a lined containment area so that spills are confined and can readily be cleaned, and so that the need for extensive and costly remediation work can be avoided during site closure.

Diesel fuel will be required for the mobile mine equipment, some small trucks, and surface vehicles. The pumping station allows for refueling of both light vehicles and heavy-duty mining equipment.

### 18.6.9 Security/Gatehouse

A security/gatehouse will be located on the site access road at the plant site. The access road at Highway 37 will have a manual gate with signage indicating that they are now entering private property.

## 18.7 Power Supply

### 18.7.1 Introduction

HNC commissioned Valard Construction Ltd to review available information and compile a preliminary study for the transmission line to support the development and operation of the Turnagain Mine. The full report is to be found in the supporting documentation and the following summary has been abstracted from it.

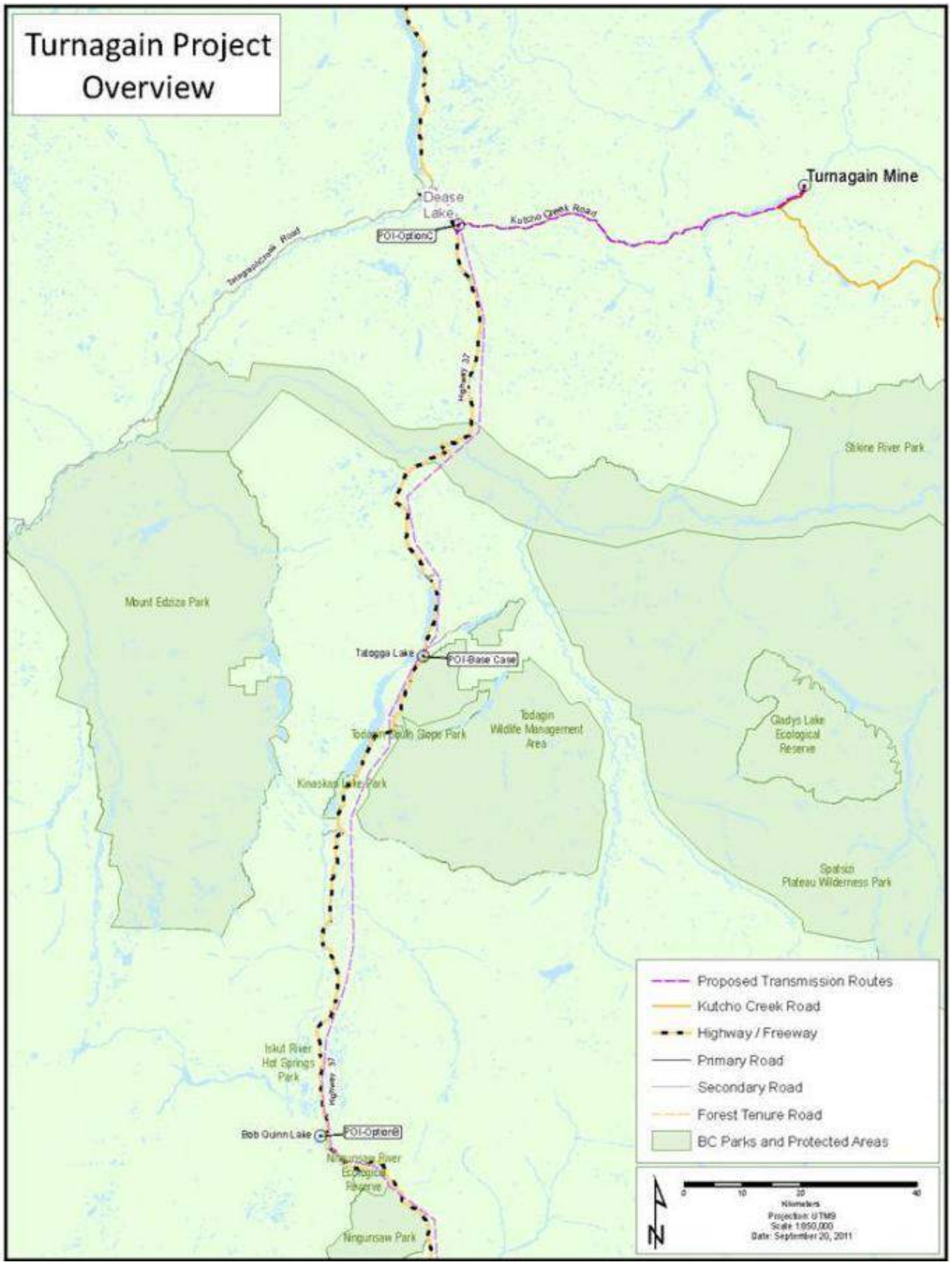
The Wardrop PA considered a transmission line interconnection at Dease Lake for the Turnagain Mine, with HNC constructing the transmission line from Bob Quinn Lake to Dease Lake and transferring this to BC Hydro. At the time of the Wardrop PA, the contract to construct the Northwest Transmission Line was not awarded and other developments in the area (Red Chris Mine, among others) were less certain. Since the Wardrop PA report, the Construction Contract for the Northwest Transmission Line (NTL) has been awarded to a Design-Build consortium led by Valard Construction Ltd. In addition, the Federal Government upheld the previous Federal approvals with respect to the Red Chris Project. The NTL will extend the BC Hydro transmission system northwards to Bob Quinn Lake, and operate at a voltage of 287 kV, and the increased certainty of approvals allows the Red Chris Project to move forward. It is understood the target commercial date for production at the Red Chris Mine is 2014 (Imperial Metals Corp., 2011), and also that it is reasonable to expect the NTL could be extended as far as Tatogga lake to supply this project.

Based on the current transmission situation in northwest BC, Valard reviewed three potential Points of Interconnection (POI) to the BC Hydro grid for the Turnagain mine. These are Tatogga Lake (near the Red Chris Mine), Bob Quinn Lake and Dease Lake. Additionally two operating voltages, 138 kV and 287 kV, for the transmission line are also reviewed. All scenarios consider the Northwest Transmission Line to Bob Quinn as complete, currently under construction by Valard. These options are shown in Table 18.1 and Figure 18.5.

**Table 18.1 Turnagain Transmission Line POI and Route Options**

| | Option A: Tatogga Lake POI (Base Case) | Option B: Bob Quinn POI | Option C: Dease Lake POI |
|---|---|---|---|
| **Total transmission line length** | 85+65 = 150 km | 95+85+65 = 255 km | 65 km |
| **Total length of new access trail to right-of-way** | 23 km | 37 km | 18 km |
| **Total width of right-of-way** | 40m @ 287 kV<br>30m @ 138 kV | 40m @ 287 kV<br>30m @ 138 kV | 40m @ 287 kV<br>30m @ 138 kV |
| **Approximate Forest Clearing** | 620 ha @ 287 kV<br>465ha @ 138 kV | 980 ha @ 287 kV<br>735 ha @ 138 kV | 280 ha @ 287 kV<br>210 ha @ 138 kV |

**Figure 18.5 Turnagain Transmission Line POI and Route Options**




The selection of a preferred POI and operating voltage relies on many factors including: capital construction costs; operating costs; line losses; future mining and transmission development in the area; potential BC Hydro infrastructure upgrades to support mine operating power load; and geography. The costs for the alternatives are summarized in Table 18.2.

### Table 18.2 Comparison of Interconnection Alternatives for Route Options

| | | **Option A: POI at Tatogga Lake** | **Option B: POI at Bob Quinn Lake** | **Option C: POI at Dease Lake** |
|---|---|---|---|---|
| **287 kV** | Capital Cost | $73,454,000 | $124,896,000 | $31,929,000 |
| | Substation Costs | $26,500,000 | $56,500,000 | $26,500,000 |
| | Losses (NPV) | $4,998,000 | $8,496,000 | $2,166,000 |
| | Operating Costs (NPV) | $3,086,000 | $5,177,000 | $1,407,000 |
| | BCH Studies/Upgrades | $6,250,000 | $6,250,000 | $6,250,000 |
| | Cost Escalation to 2015 | $17,512,000 | $26,425,000 | $10,195,000 |
| | **TOTAL** | ***$131,800,000*** | **$227,743,000** | **$78,446,000** |
| **138 kV** | Capital Cost | $55,160,000 | $94,158,000 | $23,966,000 |
| | Substation Costs | $37,500,000 | $47,500,000 | $17,500,000 |
| | Losses (NPV) | $13,950,000 | $23,715,000 | $6,045,000 |
| | Operating Costs (NPV) | $2,325,000 | $3,900,000 | $1,063,000 |
| | BCH Studies/Upgrades | $4,250,000 | $4,250,000 | $4,250,000 |
| | Cost Escalation to 2015 | $15,855,000 | $21,422,000 | $7,206,000 |
| | **TOTAL** | **$129,040,000** | **$194,945,000** | **$60,029,000** |

Ultimately, based on these criteria the base case scenario for the transmission was selected as a POI at Tatogga Lake operating at 287 kV.

Based on preliminary cost estimates the Tatogga Lake Transmission option at 287 kV would require $123,716,000 (+/- 30%) in capital expenditures. The net present value of the operating cost of this transmission line, over an assumed 25 year life, would be $8,084,000 (+/-30%). The capital costs include substations and series capacitors as required for the efficient electrical transmission. The operating costs consider both electrical losses as well as line maintenance, vegetation management, and other costs.

Based on Valard's work on the Northwest Transmission Line and other lines in northern BC and elsewhere in Canada, the construction of a 287 kV transmission line from Tatogga Lake to the Turnagain Mine is technically feasible. Confirmation of Tatogga Lake as the preferable POI and 287 kV as the optimal operating voltage is required as additional project information becomes available.

In order to address risks associated with the transmission line and avoid unnecessary delays in approval processes, key recommendations have also been made in this report. These recommendations include:

- A BC Hydro study is necessary to determine the availability of power at the Skeena Substation, and other potential constraints;
- Routing studies should be carried out prior to formally entering the Environmental Assessment Process;
- Engaging First Nations and local stakeholders is important for the development of the transmission line; and
- Revision of cost estimates periodically during the planning of the mine is important to incorporate additional information as it becomes available.

Assumptions were made in the preliminary cost estimates of the transmission line based on information currently available and based on Valard Construction LP's expertise and experience on comparable projects. As the project develops, these assumptions will need to be reviewed and cost estimates revised as more information becomes available.

Separately HNC have had discussions with BC Hydro with regard to any inter-connection fee applicable to connecting to the Northwest Transmission Line. Preliminary advice is that an interconnection fee of $114M would apply and this has been factored in to the capital cost estimation.

## 18.8 Site Power Distribution

From the main substations (see Section 18.7), 25 kV line up, 25 kV power cables will extend to deliver power to:

- 2 sets of electrical distribution equipment for 18 MW SAG mills
- 4 sets of motor starters for 13.2 MW ball mills
- 3 step down locations consisting of 4 kV transformers, power distribution equipment, 4 kV motor control, and 4 kV variable frequency drives
- 3 step down locations consisting of 600V transformers, power distribution centres, and 600V motor control centres (MCCs)

From the 25 kV line up, 25 kV overhead lines will extend to deliver power to:

- pit and mining equipment loads
- tailings pumping and reclaim areas
- camp and miscellaneous service facilities

In locations where loads are logically grouped, electrical rooms will be provided with area step down transformers located outside the exterior walls. Within these areas electrical rooms will be concentrated the relevant 4 kV, 600V, and process control equipment.

A 4 kV emergency power system is provided to support critical process area loads as identified in ongoing project planning. A Critical Process MCC is provided in each electrical room and connected to the area's stand-alone generator so that, should utility power fail, the critical equipment can be restarted after the emergency generator comes online.

Frequency of the power supply is 60 Hz alternating current (AC). Operating voltage levels are as follows:

- medium voltage: original equipment manufacturer (OEM) equipment 4.16 kV, 3 phase, 4 wire, high resistance grounded
- low voltage: motors larger than 0.5 hp 575V, 3-phase, 3-wire 5 A resistance grounded
- area lighting: interior and exterior 120V or 347V, single phase, solidly grounded
- room lighting: 120V, single phase, solidly grounded
- control voltage: A 120V, single phase, solidly grounded
- control voltage: B 24V, direct current (DC) (if required to suit OEM equipment)
- instrumentation loop voltage: 24V direct current (VDC), 4 to 20 milliamperes (mA).

## 18.9 Communication System

The site communications systems will be supplied as a design build package and the scope defined in further project phases.

# 19 MARKET STUDIES AND CONTRACTS

## 19.1 Introduction

At this early stage in the project no material contracts are in place, and therefore there is no consideration of contracts in this section.

However in lieu of a smelter contract some benchmark smelter terms are applied in the financial model; the key elements of these terms are summarized in Table 19.1.

**Table 19.1 Benchmark smelter terms**

| Metal | TC US$/DMT | RC US$/lb or oz | % payable |
|---|---|---|---|
| | 250 | | |
| Ni | | 0.6 | 90 |
| Co | | 2.5 | 50 |
| Cu | | 0.4 | 90 |
| Pt | | 20 | 80 |
| Pd | | 20 | 80 |

Note that no price participation is considered applicable, in line with current market advice.

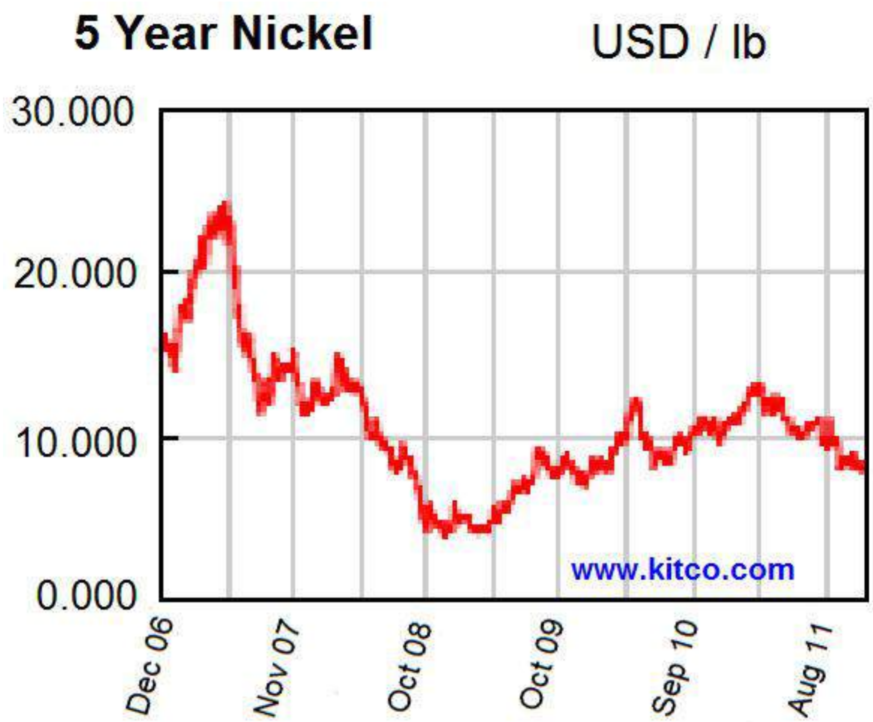
## 19.2 Market Studies

A comprehensive nickel and cobalt market study has been carried out by Reid Resource Consulting Pty Ltd (RRC). The full report is included in the supporting documentation and there follows a summary abstracted from it.

### 19.2.1 Nickel Market

The current 5 year nickel spot price history is shown in Table 19.1. After years of strong nickel consumption growth and resulting nickel price rises, the global collapse in 2008 hit the nickel industry particularly hard with a slump in the demand for stainless steel, which then constituted about 65% of nickel consumption. Following the immediate collapse to US$4.50-5.50/lb, the nickel price has rebounded and for the last two years has been trading in the US$8-12/lb range.

**Figure 19.1 Five Year Nickel Spot Price (source Kitcometals)**




The key elements in the nickel market of recent years have been:

- The growth in the Chinese market
- The use of nickel in stainless steel production, (60-70% of total use)
- Nickel pig iron (NPI) in China

These elements are demonstrated in Tables 19.2 and 19.3.

**Table 19.2 Stainless Steel Production by Country/Region ('000t)** (source RRC)

| | 2007 | 2008 | 2009 | 2010 | 2011 Estimate |
|---|---|---|---|---|---|
| **USA** | 2,171 | 1,925 | 1,618 | 2,200 | 2,250 |
| **Japan** | 3,885 | 3,567 | 2,600 | 3,400 | 3,315 |
| **Europe** | 8,093 | 7,819 | 5,970 | 7,500 | 7,600 |
| **China** | 7,560 | 7,057 | 9,650 | 12,300 | 14,180 |
| **Other Countries** | 6,939 | 6,225 | 6,550 | 6,830 | 7,120 |
| **World Total** | 28,648 | 26,593 | 26,388 | 32,230 | 34,465 |

**Table 19.3 Chinese Nickel Supply and Demand ('000t)** (source RRC)

| Production | 2003 | 2004 | 2005 | 2006 | 2007 | 2008 | 2009 | 2010 | 2011 Estimate |
|---|---|---|---|---|---|---|---|---|---|
| **Jinchuan** | 60 | 80 | 100 | 105 | 100 | 104 | 131 | 130 | 133 |
| **Ni Pig Iron** | 0 | 0 | 5 | 35 | 85 | 50 | 109 | 160 | 220 |
| **Other** | - | - | - | - | 15 | 24 | 28 | 40 | 40 |
| **Total Production** | 60 | 80 | 105 | 145 | 200 | 178 | 268 | 330 | 393 |
| **Imports** | 80 | 70 | 105 | 115 | 130 | 142 | 182 | 215 | 267 |
| **Consumption** | 140 | 150 | 210 | 260 | 330 | 320 | 450 | 545 | 660 |

The first appearance in China of the new product "nickel pig iron" in 2005 took the world by surprise. Nickel pig iron (NPI) production was initiated in idled blast furnaces fed with imported low-grade nickel laterite ore and coke, with the pig iron containing 4 to 8% nickel and >80% iron. However, in the view of RRC, Chinese NPI production provides some stability to the nickel market by capping the price on the upside and providing a floor of ~US$8.50/lb, keeping in mind that costs in China are rising relatively rapidly.

### 19.2.2 Nickel Price Prediction

It is RRC'c view that the current strength of the nickel price is likely to continue (with short term peaks and troughs) under the influence of the combination of strong Chinese demand and the high capital cost of greenfield projects outside China. The long established nickel sulphide ore province of Sudbury in Canada is in decline, as is the high-grade nickel saprolite ore from New Caledonia. The relatively low Capex of the still to start 32,000 t/y nickel Ramu project in PNG, at ~US$1.4B, is an exception to the experience of other laterite greenfield projects, presumably as the construction company was Chinese. Ramu is the first large Chinese metal project constructed outside China, to this time.

On a production unit basis, the capital cost to bring new western designed and built projects to the production stage has continued to escalate (approaching twice that for Ramu) and continued production expansion can only be supported by a correspondingly high nickel price.

RRC's view for future nickel prices, as shown below, is based on the above considerations. Equally, RRC believes that it would be counterproductive for the price to exceed ~US$14.lb, as happened in 2007, as this would again result in product substitution and rapid consumption decline. The production of NPI in China is having the impact of capping the upside, but equally importantly limits the downside. RRC believes that Chinese NPI production will continue to grow strongly at LME nickel prices higher than US$9.00 – 10.00/lb.

### Table 19.4 Actual and Projected Nickel Price: 2007 - 2013

| Year | 2007 | 2008 | 2009 | 2010 | 2011E | 2012E | 2013E | 2014E |
|---|---|---|---|---|---|---|---|---|
| **LME US$/lb** | 16.86 | 9.53 | 6.66 | 9.90 | *9.00* | *8.50* | *9.50* | *9.50* |

The LME nickel price at this time ( late September 2011) is US$8.50/lb.

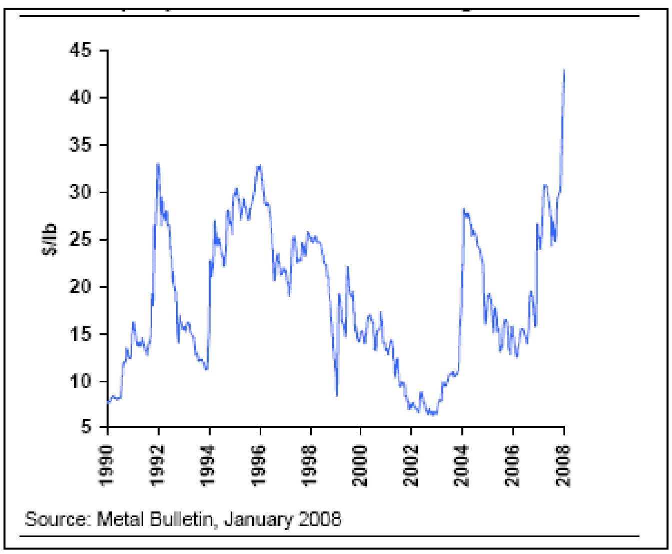
### 19.2.3 Cobalt Market

Unlike nickel, cobalt has not been traded on the LME until recently and the 99.8% Co price had been quoted in the London Metal Bulletin (LMB) for at least 20 years. About 90% cobalt production around the world is as a by-product of nickel production (55%) and copper (35%) and is therefore relatively insensitive to price. Production took a jump in 2009 with the commencement of stage 1 of Freeport's, 110,000 t/y copper and 8,000 t/y cobalt, Tenke Fungurume operation in the Congo, the largest producer to come on line for many years. More cobalt production is under construction or planning in the Congo and is expected to lead to a considerable supply surplus in the next few years. Africa contains about 52% of total world cobalt reserves (Australia 24%, Americas 17%, Asia 7%).

Global cobalt consumption in 2010 was about 65,000t and production was estimated as about 75,000t. The traditional uses have been in superalloys, magnets, chemicals, ceramics and cemented carbides and these continue to grow, but the big growth market for cobalt has been Li-Ion batteries that are now extensively used in laptop computers, cell phones and increasingly electric cars, due to their high power to weight ratio. It is estimated that cobalt use in batteries in 2010 was 17,000t, or 26% of total cobalt demand, having climbed from 3% in 1995. In 2010 usage in metal alloys, including superalloys was estimated as 12,000t and Hard Materials (carbides) consumed 8,000 t.

The price history of cobalt between 1990 and 2008 highlights the classic price volatility characteristic of a commodity produced as a by-product, as shown in the graph below. In May 2008 the price peaked at US$48/lb.

### Figure 19.2 Cobalt Price (LMB 99.8%): 1990 - 2008 (High volatility typical of by-product insensitivity to price)



In the period between late 2009 to the present the price volatility has been subdued, with most trade in the US$15 - $20 range, as shown below.

**Figure 19.3 LMB Cobalt Price: (Oct 2008 – May 2011)**


The price cobalt for 2011 YTD is US$17.00/lb and the current price is ~US$15.00/lb and it is expected that the current surplus of production over supply will continue in the mid to long term, placing downward pressure on price and RRC views US$14/lb as a reasonable price expectation for the mid term. Lower prices will undoubtedly promote the use of Li-Ion batteries in electric vehicles.

# 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

## 20.1 Data Requirements

Baseline environmental studies on the project were initiated in 2004 and are ongoing. A number of additional environmental studies will be required to support the necessary permits, approvals, and licences for the project. These studies include fisheries and aquatic studies, wildlife and wildlife habitat (including vegetation and listed species if applicable), air quality studies and modelling and background noise level measurement.

Fisheries and aquatic studies are expected to comprise the most significant part of the environmental studies during the environmental assessment process. The following is a brief summary of studies and data requirements for fisheries and aquatic studies.

### 20.1.1 Aquatic Habitat

Key aspects of the aquatic abiotic environment that will be considered during detailed baseline studies include:

- **Mesohabitat Assessment:** Involves an inventory of the available habitat in the project area, and mapping available aquatic habitat according to provincial standards.
- **Surface Water Quality:** The comprehensive surface water sampling program will be maintained to establish baseline levels and/or propose site specific water quality criteria (for parameters showing exceedances from available guidelines, if applicable). This program will include sampling sites in Flat Creek, its tributaries, and control sites.
- **Groundwater Quality:** The groundwater quality program will be maintained to establish a baseline in the project area. Additionally, groundwater seepage sites will be investigated, since they may play a key role in providing suitable habitat for spawning, incubation and rearing for some fish species.

### 20.1.2 Aquatic Life

The proposed conceptual design for waste management will involve possible displacement and alteration of productive aquatic habitat in Flat Creek valley. In order to meet the requirements of the “no net loss” of fish habitat principle of the Department of Fisheries and Oceans (DFO), mitigation and compensation measures will be required to replace lost productivity. No aquatic habitat studies have been conducted for Flat Creek.

Detailed studies of the existing fish species, their life cycle, and utilization of the physical aquatic habitat would be necessary in order to design and propose acceptable fisheries compensation plans. The following is a list of potential investigations required:

- Investigation, mapping, and quantification of existing aquatic habitat and its utilization (spawning, rearing, overwintering) by resident species in Flat Creek and its fish bearing tributaries
- Estimation of existing number of fish in Flat Creek and population composition by species

- Estimation of present level of production in the aquatic environment (primary production, fish production, etc.) to establish baseline levels and to set goals for mitigation/compensation plans
- Based on fish species inventory in the project area, appropriate design criteria should be incorporated in diversion channel(s) in order to provide support for fish communities in perpetuity
- Depending on negotiations with the Ministry of Environment (MoE) and DFO, additional fisheries compensation options may be expected; in this case, a list of potential options should be compiled during baseline studies
- A habitat balance analysis will be required to delineate achievement of the "no net loss" principle for the project and its components

Various follow up and monitoring programs will be required to monitor different environmental aspects of the project such as fish and aquatic habitat, water quality, wildlife, vegetation, and archaeology. Monitoring programs have various timelines and some may be required to be in effect for the life of the project.

An adaptive management program may also be required to address some potential issues as they are identified.

## 20.2 Environmental Impact Assessment

The project exceeds the reviewable project regulation of a new mining facility production capacity of greater than or equal to 75,000 t/a of mineral ore under the British Columbia Environmental Assessment Act (BCEAA). The throughput of the project also exceeds the Canadian Environmental Assessment Act (CEAA) comprehensive study regulations of greater than or equal to 3,000 t/d, triggering a comprehensive study. Therefore, a harmonized review will be required under both BCEAA and CEAA.

A comprehensive study requires extensive public and First Nation consultation as well as a detailed study of environmental baseline settings and assessment of potential impacts of the project and its facilities and components on the surrounding environment. HNC initiated these processes in 2004.

Any resource project that would undergo a comprehensive study under CEAA will be guided by Major Project Management Office (MPMO) of Natural Resources Canada (NRC).

A number of options for the tailings disposal facility were evaluated during the preliminary assessment and conceptual design for the project, all of which involved alteration and possible loss of aquatic habitat.

The possible impact on the Flat Creek drainage will require authorization under Section 35(2) of the federal Fisheries Act for harmful alteration, disruption or destruction (HADD) of fish habitat. As part of the authorization process, mitigation and compensation measures as guided by DFO's "no net loss" principle will be required. Supporting data and scientific evidence will be provided to ensure the regulatory bodies that all possible design alternatives have been considered and evaluated, and fish utilization and connectivity of Flat Creek is maintained through implementation of appropriate and efficient mitigation and compensation strategies.

The proposed TMF design will also require an amendment to Schedule 2 of the Metal Mine Effluent Regulations (MMER) of the Fisheries Act, which will allow the deposition of tailings to fish bearing waters. An application for amendment of MMER Schedule 2 must be submitted subsequent to environmental assessment certification of the proposed project.

## 20.3 Water and Waste Management

### 20.3.1 Site Water Management

Water management is an important component of the overall design, and the objectives are as follows:

- adequate storage and freeboard in the TMF for secure containment of all process water and storm runoff
- interception and diversion of clean water to the extent possible
- adequate collection and control of mine affected water
- mitigate environmental impacts to the extent possible
- optimize the storage and use of water over the entire site to satisfy environmental, operational, and economic criteria.

The conceptual site water management plan includes the necessary hydraulic structures and operational procedures to achieve these objectives.

### 20.3.2 TMF Water Management

The TMF supernatant pond serves as the primary component in site water management, providing a buffering volume for process water, direct precipitation, and storm runoff. A supernatant pond volume of between approximately 10 and 30 million cubic metres is assumed to provide sufficient detention time for tailings solids to settle, as well as provide a buffering volume for process water requirements, while maintaining sufficient freeboard and providing water cover for the tailings solids. Ten million cubic metres corresponds to approximately four months of process water, and 30 million cubic metres is subsequently sufficient for a full year of operation at the full mill production rate of approximately 87,000 t/d. More detailed water balances to support the phased expansion will be required in the next phase of study.

The diversion channels shown on Figures 18.2 and 18.3 will be required along both banks of the TMF to maintain a neutral water balance condition in the facility. The water will be diverted to and released downstream of the TMF directly to Flat Creek to minimize impacts to the natural downstream flow regime. Seepage through the TMF dams will be intercepted and collected by the embankment underdrain and seepage collection systems.

The minimum freeboard requirement for the TMF is assumed to be five metres. Further studies, including determining the inflow design flood and potential seismic deformations from the maximum design earthquake, will be needed to more precisely define the required freeboard throughout the life of the facility.

### 20.3.3 Waste Dump, Low Grade Stockpile, Open Pit, and Plant Site Water Management

Collection and control of the surface and groundwater at all mine facilities is an important part of the overall water management plan. The water collected at the waste dump, low grade mineralization stockpiles, open pits, and plant site represents a significant portion of the overall site water balance.

Out-of-pit dewatering wells will be pumped to local collection ponds for water quality monitoring. It is anticipated that the water will be of sufficient quality for discharge directly to the environment.

In-pit water will be pumped to three main collection ponds outside the pits. The volume of in-pit dewatering will generally be dependent on direct precipitation and groundwater inflows. The collection ponds for each of the two pit areas (Horsetrail and Northwest) are sized to contain the 1-in-100 year return period rainfall event that would report to each pit.

Runoff from the waste rock dump and low grade or stockpiles will be collected in channels along the downstream toes. The channels will divert the water into a collection pond located at topographic low points below the facilities. The channels are sized to carry the 1-in-10 year peak instantaneous runoff as estimated using the method presented by Obedkoff (2001). The collection pond is sized respectively. The quality of the collected water will be monitored during operations and will either report to a water treatment plant, or be discharged directly to the Turnagain River if discharge water quality criteria are being met. Provisions will also be made for transferring water from the collection ponds directly back to the plant site for process use.

The systems for removing water from each of the ponds are sized to drain each pond within approximately seven days. This capacity is sufficient for about 150% of the mean monthly runoff in the wettest month, typically May, due to the combination of rainfall and snowmelt.

All water collected in the open pit ponds will be pumped to the plant site through a 450 mm diameter pipeline for use in the process, or for treatment and release. A 300 mm diameter gravity fed pipeline will be available to convey water from the waste dump and low grade mineralization stockpile collection ponds to the main pit collection pond in the event that additional water is needed at the plant site.

Surface water diversion channels will be constructed upstream of the waste dump, low grade mineralization stockpiles, and open pit to minimize the quantity of contact water to be managed at the site. The channels will divert runoff away from the mine facilities and back into nearby existing drainages. These channels have been sized to carry the estimated 1-in-10 year peak instantaneous runoff.

### 20.3.4 Water Balance

A water balance was completed to estimate the mean annual surplus or deficit that may be expected at the project site. The water balance model includes the mill, waste dump, open pits, and the TMF, as well as the external contributing catchments for each of the mine components. The conceptual design presented in this report, along with the estimated mean annual hydrologic conditions were used as the basis for the modelling, although as mentioned previously more detailed water balance calculations will be required in the next stage of the study as the timing and details of phased approach to the production schedule is confirmed.

The results of the water balance model suggest that the site could be operated in a water-neutral condition for most of the mine life, given the assumed climatic factors, production schedule, and facilities layout. It is assumed that the entire catchment upstream of the final TMF impoundment area will be diverted to Flat Creek downstream of the main dam. These diversions could be deactivated if additional water is needed at the TMF.

### 20.3.5 Water Supply

It is assumed that fresh water will be collected from alluvial groundwater wells just north of the plant site in the vicinity of the Turnagain River. This would supply the fresh water requirements at the mill as well as other mine facilities. Water will be pumped approximately 120m up to a storage tank at the mill. The fresh water demand is currently estimated at approximately 170 L/s.

### 20.3.6 Sewage Disposal

A sewage treatment plant is included in the mine infrastructure. Non-process waste water from some of the site facilities, such as the mine dry and offices, would be treated in this plant.

The sewage treatment plant will be a pre-packaged Rotating Biological Contactor (RBC) system. The plant will be manufactured off site and containerized for simple connection to the collection system on site. The solid and liquid material will be separated in the treatment plant and the sewage treatment plant effluent will be discharged into the environment in accordance with the requirements of the Environmental Impact Assessment.

### 20.3.7 Refuse Disposal

Several forms of domestic and industrial solid waste will be generated over the life cycle of the mine. All avenues of reuse, reduction, and recycling of materials will be examined and implemented prior to disposal of any waste.

Domestic waste will be incinerated on site, with clean efficient combustion supported by a waste oil-fuelled dual chamber incinerator.

## 20.4 Reclamation and Closure Plan

The main objective of closure is to minimize any adverse environmental and social impacts associated with the mine development, and to return disturbed site areas to conditions consistent with an approved end-use plan.

Preliminary closure planning will be carried out concurrently with the various stages of project development and design in order to integrate the post-closure objectives into the design, construction, and operation of all mine infrastructure and facilities. The closure and reclamation plan will be developed in consultation with the HNC project team, local stakeholders, and the appropriate regulatory authorities.

It is anticipated that the following objectives will be incorporated into the design of the project facilities in order to facilitate an acceptable closure and reclamation plan:

- Long-term stability of the embankments and other engineered structures, including the waste rock dump
- Long-term preservation of water quality within and downstream of decommissioned operations
- Construction of a spillway at the TMF
- Construction of a protective berm or wildlife fence around the open pits
- Removal and proper disposal of all access roads, pipelines, structures, and equipment not required beyond the end of mine life
- Long-term stabilization of all exposed erodible materials
- Natural integration of disturbed lands into surrounding landscape, and restoration of the natural appearance of the area after mining ceases, to the extent possible
- Establishment of a self-sustaining vegetative cover consistent with existing wildlife needs
- Routine monitoring to evaluate facility performance

Groundwater monitoring wells and geotechnical instrumentation will be retained for long-term monitoring and performance assessment.

Post-closure requirements will include annual inspections of the TMF and waste rock dumps, and ongoing evaluation of water quality, flow rates, and instrumentation records to confirm the design assumptions adopted for closure.

Approximate bonding requirements for premature closure, final closure, and post-closure have been included in the capital cost estimate based on the objectives outlined above.

# 21 CAPITAL AND OPERATING COSTS

## 21.1 Capital Cost Estimate

Overall capital cost is summarized in Table 21.1, and then the various main components are further detailed in the subsequent sections.

**Table 21.1 Project Capital Cost Summary**

| CAPEX Summary | Initial Capital | Yr5 expansion | Total LoM Capital |
|---|---|---|---|
| | US$M | US$M | US$M |
| Mine | 244,055 | 68,174 | 406,054 |
| Processing | 986,474 | 405,717 | 1,392,190 |
| Other and sustaining | 94,502 | 17,924 | 477,467 |
| Working Capital | 32,189 | | 32,189 |
| **Total** | **1,357,220** | **491,815** | **2,307,901** |

Note Working Capital is assumed to be 25% of yr 1 costs, i.e. equivalent to financing the first three months of operations.

### 21.1.1 Mining

Mining capital costs and their derivation have been detailed in Section 16 Mining but are summarized here in Table 21.2.

**Table 21.2 Mining Capital Costs**

| Item | Initial Capital Cost US$M | Sustaining Capital US$M | |
|---|---|---|---|
| | | Yr 5 Expansion | Thereafter |
| Mine Equipment Fleet | 140.124 | 68.174 | |
| Mine Building Costs | 32.395 | | |
| Mine On-Site Power | 5.035 | | |
| Site Preparation | 5.225 | | |
| Engineering | 6.650 | | |
| Mine Inventory | 13.965 | | |
| Contingency | 40.661 | | |
| **Total** | **244.055** | **68.174** | **93.825** |

The mining capital cost estimate in Section 16 was in C$ based on the assumption of an exchange rate of C$:US$ of 1.00 and has been converted to US$ at the exchange rate of 0.95 used in the Economic Analysis section of this report.

### 21.1.2 Concentrator and Site Infrastructure

The concentrator and site infrastructure costs are summarized in Table 21.3.

### Table 21.3 Concentrator and Site Infrastructure Capital Costs

| | Initial Capital US$M | Total Capital US$M (incl yr 5 expansion) |
|---|---|---|
| **Directs** | | |
| Site Works | 34.100 | 36.594 |
| Crushing | 34.348 | 57.693 |
| Coarse mineralization Stockpile | 42.042 | 69.731 |
| Process Plant | 204.485 | 353.912 |
| Services/Utilities/Ancillaries | 7.444 | 12.007 |
| Temporary Services | 35.797 | 54.258 |
| Offsite Infrastructure (Power Supply incl Interconnect Fee) | 253.122 | 253.122 |
| Access Road | 17.438 | 17.438 |
| **Indirects** | | |
| Construction Indirects | 105.426 | 159.797 |
| EPCM (15% of Directs) | 59.381 | 90.005 |
| Owners Costs | 47.478 | 71.962 |
| Contingency (20%) | 145.412 | 215.672 |
| | | |
| **Total** | **986.474** | **1,392.190** |

The original Wardrop concentrator and infrastructure capital cost estimate was developed to a relatively high level of detail, with comprehensive equipment lists and pricing. This was used as the basis for the current estimate and revised as follows:

- Items in both direct and indirect costs specific to the hydrometallurgical plant were backed out
- Adjustments were made to e.g. site works and temporary services to reflect the smaller footprint and reduced scope of the project construction work
- Some double-counting of equipment items between the concentrator and mine and tailings facility was eliminated
- The process plant capital was re-estimated as follows:
  - Major equipment items e.g. mills, and flotation cells were revised as described in Section 17 and updated quotes obtained from a major supplier,
  - The remainder of the costs were factored in line with the Wardrop estimate
- The offsite infrastructure costs, mainly related to the power supply, were derived from the Valard study presented in this report
- The access road costs were recalculated, drawing on the recent information available on the Kutcho Creek project to the south and which shares the first half of the access road
- EPCM costs were recalculated on the basis of 15% of direct costs
- Owners costs were retained from the original estimate with the exception of a BC Hydro cost item covered by the Valard estimate

- The original study had a contingency allowance based on individual discipline estimates which averaged 15.9% of direct and indirect costs. In view of the revision process and the level of study and particularly in light of the scaling described below for the phased expansion, this has been increased to 20%.
- With respect to exchange rate, the original estimate was presented in US$ but based on an exchange rate of C$:US$ of 0.9. It was assumed that the equipment items were based on US$ quotes etc but that the construction labour, local equipment and other local content costs were C$ based. The current estimate was based on US$ quotes for the mills but C$ quotes for the flotation cells and therefore, together with the local content costs previously mentioned, adjusted as requiredfor the economic analysis exchange rate of 0.95 to arrive at US$.
- Sustaining capital was estimated at 2% of processing costs, amounting to US$68.580M over the life of mine

The phased expansion element of the capital cost estimate was derived as follows:

- As described in Section 17, the quantities for major equipment items e.g. crushers, mills, flotation cells were revised to reflect the 50% capacity for the first five years, and costed based on the quotes referred to above
- Other process plant costs were factored as described above.
- The remainder of the costs were scaled using an exponent scaling factor of 0.6, with the exception of areas e.g. site works, power supply, access road, where the full project scope would apply.

### 21.1.3 Waste / Tailings Management Facilities (TMF)

The total capital costs associated with the TMF and other environmental management items are summarized in Table 21.4, and the main components of the costs are listed by principal category in the subsequent sub-sections.

**Table 21.4 TMF and Environmental Management Capital Costs**

| Item | Initial Capital Cost US$M | Sustaining Capital US$M |
|---|---|---|
| Tailings management facility | 54.7 | 200.6 |
| Water management | 17.7 | 12.3 |
| Permitting and Closure/Reclamation | 22.1 | 101.5 |
| **Total** | **94.5** | **314.4** |

#### 21.1.3.1 TMF

Capital cost estimates have been completed for the following components of the TMF:

- earthworks and foundation preparation for the main (northwest) and saddle (southeast) dams
- foundation preparation and cover requirements for the refinery tailings storage area
- tailings pipelines and fittings (excludes tailings pump station)
- reclaim water system (including pipes and pumps)

- roads for TMF access, construction, and borrow source development
- seepage control and sediment control for both dams
- geotechnical and hydrogeological instrumentation
- surface water diversions

Costs associated with terrain hazards have not been considered in this estimate. Following more detailed site investigations, costs will need to be developed for mitigating hazards associated with debris flows, avalanches, major stream crossings, and other geo-hazards.

#### 21.1.3.2 Site wide water supply and water management

Capital costs for site-wide water management include the following items:

- surface water diversion channels
- collection channels
- collection ponds
- transfer pipelines
- water treatment plant
- pumps.
- groundwater wells and pumps
- pipelines

#### 21.1.3.3 Closure and Reclamation

Direct costs for closure and reclamation include the following items:

- TMF spillway
- building demolition and removal
- pipeworks removal
- re-sloping of waste rock dump
- rock and soil haulage and revegetation
- environmental monitoring during active reclamation.
- Indirect costs include:
- mobilization and demobilization
- agency administration
- site labour and management
- materials and service (power, insurance, etc.)
- engineering and specialist services
- Annual post-closure operating expenses include:
- annual environmental monitoring

- annual site maintenance costs
- annual water treatment costs

The construction closure bond is estimated based on the closure costs at the end of the construction period, or the beginning of Year 1. An annual bond contribution for premature closure has been estimated based on expenses that would be incurred at the end of each successive five year period following start up, including an allowance for expenses that are incurred in perpetuity. A discount rate of 4.3% has been assumed for bonding cost calculations in line with recent environmental assessments accepted in BC.

## 21.2 Operating Cost Estimate

The overall operating costs are summarized in unit cost terms Table 21.5, showing the life of mine costs as well as the costs during years 1-5 at reduced throughput and years 6-21 at full capacity. Note that the life of mine costs reflect a low cost base in years 22-28 when the mill is fed from the low – grade stockpile and there are no mining costs.

**Table 21.5 Unit Operating cost Summary**

| | **L.o.M.** | **Yrs 1-5** | **Yrs 6-21** |
|---|---|---|---|
| Operating Cost US$/T milled: | 7.30 | 8.37 | 7.78 |
| -mining | 2.52 | 3.11 | 3.11 |
| -processing (incl TMF) | 4.44 | 4.69 | 4.38 |
| -G&A | 0.33 | 0.57 | 0.29 |

### 21.2.1 Manning numbers

The mine manning numbers developed in Section 16 are summarized in Table 21.6.

**Table 21.6 Mine Operations Labour –Average for Periods**

| | **Years** | | | |
|---|---|---|---|---|
| | **1 to 5** | **6 to 10** | **11 to 21** | **After 21** |
| **Hourly Labour** | | | | |
| Equipment Operators | 77 | 142 | 139 | 29 |
| Mine Maintenance | 29 | 59 | 55 | 14 |
| **Sub-total** | **106** | **201** | **194** | **43** |
| **Salaried Staff – Mine and Maintenance Operations** | | | | |
| Mine Superintendent | 1 | 1 | 1 | 1 |
| Maintenance Superintendent | 1 | 1 | 1 | 0 |
| Mine & Maintenance General Foremen | 2 | 2 | 2 | 1 |
| Shift Foremen/Team Leaders | 10 | 12 | 12 | 6 |
| Trainers | 1 | 1 | 1 | 0 |
| Maintenance Planners | 1 | 1 | 1 | 1 |
| Clerks/Dispatchers | 4 | 8 | 8 | 4 |
| **Sub-total** | **20** | **26** | **26** | **13** |
| **Mine Technical** | | | | |
| Chief Engineer | 1 | 1 | 1 | 1 |

| | Years | | | |
|---|---|---|---|---|
| | **1 to 5** | **6 to 10** | **11 to 21** | **After 21** |
| Geologists | 2 | 3 | 3 | 1 |
| Mine Engineers | 3 | 5 | 5 | 1 |
| Technicians/Surveyors | 2 | 4 | 4 | 2 |
| **Sub-total** | **8** | **13** | **13** | **5** |
| **Total Salaried Staff** | **28** | **39** | **39** | **18** |
| **Total Mine Workforce** | **134** | **240** | **233** | **61** |

Processing manning numbers are summarized in Table 21.7.

**Table 21.7 Processing Manning numbers**

| Job Description | Hourly | Staff |
|---|---|---|
| Mill Operations Staff | | 8 |
| Mill Maintenance Staff | | 6 |
| Maintenance | 28 | |
| Electrical | 14 | |
| Milling - Operations | 50 | |
| Metallurgy | | 8 |
| Assay Lab | | 11 |
| VS&A Contingency | 5 | 9 |
| Total | 97 | 47 |

### 21.2.2 Mining

The average mining operating costs are tabulated in Table 21.8.

**Table 21.8 Mine Operating Cost Estimate**

| | $/t Mined | $/t Mineralization Milled |
|---|---|---|
| Drilling | 0.07 | 0.11 |
| Blasting | 0.23 | 0.33 |
| Loading | 0.21 | 0.30 |
| Hauling | 1.00 | 1.42 |
| Pit Support | 0.17 | 0.24 |
| Mine Maintenance Support | 0.03 | 0.04 |
| General Mine Expense | 0.15 | 0.20 |
| Engineering | 0.01 | 0.01 |
| **Total** | **1.87** | **2. 65** |

### 21.2.3 Concentrator

Concentrator operating costs are summarized in Table 21.9.

### Table 21.9 Concentrator Operating Costs (at full capacity)

| | US$/t milled |
|---|---|
| Power | 1.46 |
| Consumables | 0.66 |
| Grinding media | 1.63 |
| Labour | 0.40 |
| Maintenance | 0.07 |
| Other | 0.06 |
| **Total** | **4.22** |

### 21.2.4 Offsite Charges

Apart from the smelter terms detailed in section 19, other offsite charges include concentrate trucking and shipping, amounting to US$154/WMT, the main components of which are trucking to Stewart at US$62.50/WMT, port charges and sampling at US$13.50/WMT and ocean freight at US$66.50/WMT.

### 21.2.5 General and Administration (G&A) Costs

G&A costs were estimated at US$12.127M based on a manning complement of 34 staff and including access road maintenance and crew transportation.

### 21.2.6 Tailings Management facility

The LOM average TMF operating cost is calculated to be US$0.22/t processed.

Operating expenses for the TMF include electrical power for tailings, reclaim water, and seepage collection water pumping. Electrical power costs are estimated using US$0.04/kWh.

# 22 ECONOMIC ANALYSIS

## 22.1 Key project Outputs

The base case for the purposes of the economic analysis was the production schedule developed in Section 16 with the first five years at approximately 50% throughput.

The core model was developed as a pre-tax model; however an after-tax model was also prepared with professional tax expertise input. Tax inputs are summarized in section 22.3.

The key project outputs are summarized in Table 22.1.

### Table 22.1 Key Project Outputs

| Key Outputs | | L.o.M. | | Yrs 1-5 | Yrs 6-21 |
|---|---|---|---|---|---|
| | | Pre-tax | After tax | | |
| **Financial** | NPV (US$M) | 1295 | 724 | | |
| | IRR % | 15.9 | 13.5 | | |
| | Payback period yrs | 7.3 | | | |
| | Smelter % netback | 72.4 | | | |
| | NSR delivered $/T | 18.5 | | 21.6 | 20.2 |
| | Average operating cash flow (US$M) | 316 | | 208 | 387 |
| **Physicals** | Feed Grade %Ni | 0.23 | | 0.26 | 0.25 |
| | Average annual throughput Mtpa | 28.1 | | 15.8 | 31.3 |
| | Strip ratio | 0.82 | | 0.74 | 0.83 |
| | Recoveries %: | | | | |
| | Ni | 56.4 | | 58.0 | 57.7 |
| | Average Annual Metal Production: | | | | |
| | -Ni (lbs x 1000) | | | 52717 | 97871 |
| | -Co (lbs x 1000) | | | 2822 | 5363 |
| | DMT Concentrate | 2032101 | | 132846 | 246633 |
| **Costs:** | Operating Cost US$/T milled: | 7.30 | | 8.37 | 7.78 |
| | -mining | 2.52 | | 3.11 | 3.11 |
| | -processing (incl TMF) | 4.44 | | 4.69 | 4.38 |
| | -G&A | 0.33 | | 0.57 | 0.29 |
| | C1 cash cost $/lb payable Ni (after Co credits) | 4.26 | | 4.23 | 4.20 |

Note that the key inputs in the above table are a nickel price of $8.50/lb, cobalt $14/lb and the C$:US$ exchange rate of 0.95

## 22.2 Sensitivity Analysis

Key sensitivities are shown in Table 22.2.

### Table 22.2 Sensitivity Analysis

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| **Ni Price** | -30% | -20% | -10% | **0%** | 10% | 20% | 30% |
| NPV | 461,323 | 124,263 | 709,850 | **1,295,436** | 1,881,022 | 2,466,608 | 3,052,195 |
| Ni price | 5.95 | 6.80 | 7.65 | **8.50** | 9.35 | 10.20 | 11.05 |
| IRR | 4.4% | 8.9% | 12.6% | **15.9%** | 19.0% | 21.9% | 24.6% |
| NPV FX corrected[2] | 31,717 | 452,957 | 874,196 | **1,295,436** | 1,716,675 | 2,137,915 | 2,559,155 |
| **Ni Recovery[1]** | -15% | -10% | -5% | **0%** | 5% | 10% | 15% |
| NPV | 573,168 | 813,396 | 1,054,206 | **1,295,436** | 1,536,913 | 1,778,917 | 2,021,303 |
| Ni recovery | 41% | 46% | 51% | **56.4%** | 62% | 68% | 75% |
| IRR | 11.8% | 13.2% | 14.6% | **15.9%** | 17.2% | 18.5% | 19.7% |
| **Processing Capex** | -30% | -20% | -10% | **0%** | 10% | 20% | 30% |
| NPV | 1,611,775 | 1,506,328 | 1,400,882 | **1,295,436** | 1,189,990 | 1,084,543 | 979,097 |
| capex | 894,733 | 1,022,552 | 1,150,371 | **1,278,190** | 1,406,010 | 1,533,829 | 1,661,648 |
| IRR | 19.7% | 18.3% | 17.0% | **15.9%** | 14.9% | 14.0% | 13.2% |
| **FX[1]** | -15% | -10% | -5% | **0%** | 5% | 10% | 15% |
| NPV | 1,788,476 | 1,624,129 | 1,459,782 | **1,295,436** | 1,131,089 | 966,743 | 802,396 |
| FX | 0.808 | 0.855 | 0.903 | **0.950** | 0.998 | 1.045 | 1.093 |
| IRR | 19.7% | 18.4% | 17.1% | **15.9%** | 14.8% | 13.7% | 12.7% |
| **Discount Rate** | -30% | -20% | -10% | **0%** | 10% | 20% | 30% |
| NPV | 2,126,601 | 1,811,723 | 1,536,551 | **1,295,436** | 1,083,617 | 897,070 | 732,384 |
| discount rate | 5.6% | 6.4% | 7.2% | **8.0%** | 8.8% | 9.6% | 10.4% |

[1] In the above table Ni recovery and FX are in the range +/-15% to simulate a probable range, compared to the other factors at the more normal +/- 30% range.

[2] The line NPV FX corrected assumes that FX varies by half the amount of the Ni price variation, as commonly observed in commodity economies.

Another key sensitivity explored was that to project scale. The analysis was of an approximate nature only, being based on a factored approach to the base case, and on a pre-tax basis only. The results are presented in Table 26.3 and compared to the base case.

### Table 22.3 Sensitivity to Project Scale

| | Initial Capital US$Bn | Pre-tax NPV (8% US$Bn) | Pre-tax IRR % | Payback yrs | L.o.M. yrs |
|---|---|---|---|---|---|
| **Base Case** | 1.36 | 1.30 | 16.0 | 7.4 | 27.2 |
| **Constant 15.8 Mtpa** | 1.20 | 0.73 | 13.5 | 6.9 | 49.5 |
| **Constant 31.3 Mtpa** | 1.85 | 1.57 | 17.7 | 5.1 | 24.7 |

AMC notes that even at the constant reduced throughput the project still delivers an acceptable IRR despite the significant reduction in the NPV. It is also apparent from the improved payback in this case that there is an optimization opportunity with the base case with respect to the timing of the capacity expansion.

## 22.3 Taxation and Royalties

The Turnagain project is defined a "new mine" for the purposes of income and mineral tax and nickel, as a base metal, is taxable under the Mineral Tax Act.

The tax model takes account of existing laws to reduce the federal corporate tax rate to 15% effective 1 January 2012 and the BC income tax rate to 10%, already effective 1 January 2011.

No royalties are applicable in BC.

However a Provincial Resource Tax is in place, applied in two stages, an initial stage with a low "holiday" rate of 2% and then a longer term rate of 13% once the Capital Expenditure Account returns a positive balance.

An Investment Allowance credit applies, currently at a rate of 1.25% per the BC Finance Mineral Tax website.

## 23 ADJACENT PROPERTIES

This section is not applicable

## 24 OTHER RELEVANT DATA AND INFORMATION

Project data and information is detailed in the appropriate sections.

# 25 INTERPRETATION AND CONCLUSIONS

Based on the outcomes of this Preliminary Economic Assessment AMC has drawn the following key conclusions.

## 25.1 Geology

Relative to the Wardrop 2010 study, further data verification has been carried out which confirms the integrity of the Turnagain resource estimate. 74% of the 28 year mine life in-pit resource is in the Measured and Indicated categories.

## 25.2 Metallurgy

The key outcome of the laboratory scale metallurgical testwork perfomed during 2010-11 was that a potentially saleable 18%Ni concentrate could be produced at an average 56% Ni recovery. This was confirmed by locked cycle and bulk concentrate tests. Further work is required to investigate the variability of the mineralization and its geometallurgical characteristics.

## 25.3 Mining

Elevated cut-off grades combined with stockpiling was applied to pit scheduling optimizations which also considered the hardness of the mineralization as well as Ni grade. From this, a mining strategy was developed which delivered higher grade metallurgically more amenable mineralized material to the mill in the early years of the mine life. During the later years in the 28 year life, the mill would be fed with low grade stockpile material.

This approach lends itself to a phased approach to the production schedule whereby the project commences at a lower throughput level of 43,200 tpd for the first five years, with an associated 35% reduction in initial capital requirements.

Further geotechnical and hydrogeological studies are still required in the next phase of study to support the proposed mine design.

## 25.4 Processing

Although there are inconsistencies in some of the grindability testwork results, this update confirms that the Turnagain mineralization can be treated in an 87,000 tpd conventional SABC circuit followed by froth flotation along the lines of the Wardrop study, with modifications to the cleaner circuit only to reflect the production of a high grade concentrate instead of the bulk concentrate originally proposed.

The very large SAG mill indicated (40’) in the flowsheet design does merit further critical review, including the evaluation of high pressure grinding rolls (HPGR’s) as an alternative.

The two-train comminution circuit followed by four banks of rougher flotation also lends itself well to the phased approach referred to above and the capital cost estimate has been developed accordingly.

## 25.5 Tailings and Water Management

The work completed to date suggests that the current waste and water management concept is practicable and should be carried forward to the next level of design (preliminary feasibility).

The conceptual design presented in this report has potential for optimization as the design basis and operating criteria are further refined.

## 25.6 Power Supply

With the awarding of the contract for the NTL to Bob Quinn and upholding of the Federal approvals for the Imperial Metals Red Chris project, there is increased certainty of power supply to the Turnagain project and increased confidence in associated capital cost estimates.

## 25.7 Economic Analysis

A financial model has been developed based on the inputs set out in this report and it shows that the project can deliver acceptable returns at a nickel price of $8.50/lb and an exchange rate of C$:US$ of 0.95. It also demonstrates a robustness with respect to project scale.

## 25.8 Opportunities and Risks

AMC notes the following potential project opportunities:

- Proving up the additional resource contained within the ultimate pit
- Additional resources in the Hatzl and Cliff areas, with the latter offering potential for additional platinum and palladium values
- Enhanced geometallurgical knowledge of the mineralization as an aid to pit optimization
- Further metallurgical improvements, particularly related to the split cleaner concept
- Full project optimization taking into account not only the conventional pit optimization processes but also the geometallurgical parameters of the mineralization noted above, the tailings facility construction as a function of the waste stripping schedule and the timing of the capacity expansion
- Reduction of initial capital cost with the reduction in the projected BC Hydro interconnection tariff fees. HNC has assumed the worst case scenario which incorporates 114 M$ initial capital for interconnection fees. This tariff has yet to be approved and placed into legislation by the BC government. BC Hydro has also presented the opportunity of paying the fee over a five year operating span rather than an upfront fee (eg. move the fee from initial capital to an early year operating fee)
- Shared access development costs with the potential development of the Kutcho Creek Project further to the west

And also the following project risks:

- The outcomes of geotechnical and hydrogeological investigations not supporting the proposed design, although with the base case pit being developed within the ultimate pit shell, AMC consider this risk to be of low impact as waste stripping would not be significantly increased if more conservative pit slopes were deemed necessary.
- The variability of the mineralization proving greater than expected as the geometallurgical knowledge base is improved; AMC considers this to be a relatively low risk in the Horsetail pit where the initial mine development occurs but increasing in the north-west zone and towards the northern limits of the proposed pit.

## 26 RECOMMENDATIONS

This Preliminary Economic Assessment has shown that the Turnagain property is a potentially viable project at the base case parameters and based on the current NI 43-101 compliant resource. AMC recommends therefore that HNC carry it forward to the preliminary feasibility stage, in accordance with the budget presented in Table 26.1.

AMC notes that the drilling requirements have been based on:

- The current relatively high percentage of mineralization in the Measured and Indicated categories.
- A reasonable assessment of the drilling required to convert the remainder of the Inferred Resources in the current 28 year life in-pit resource; there is no allowance for extending the resource at pre-feasibility study level.
- The current geotechnical database from resource drilling and logging to date which contains comprehensive rock strength data but lacks orientation data, hence there is a minimum requirement for some geotechnical oriented core drillholes.

**Table 26.1 Preliminary Feasibility Study Budget**

| **Item** | **US$ '000** |
|---|---|
| Reserve Drilling | 1560 |
| Geotechnical Drilling | 360 |
| Analyses | 620 |
| Transportation | 1020 |
| Environmental | 125 |
| Special Engineering Studies | 400 |
| On-site Consultants | 125 |
| Metallurgy and Geometallurgy | 650 |
| Contract Salaries | 160 |
| Camp Costs | 490 |
| PFS Engineering | 500 |
| Contingency | 902 |
| **Total** | **6912** |

## 27 REFERENCES

AMEC (2008), NI 43-101 Technical Report on Preliminary Assessment – Hard Creek Nickel Corporation, Turnagain Nickel Project, British Columbia, Project No. 155915, AMEC Americas Ltd., (25 September 2007 and January 2008)

Aeroquest Ltd (2005) Report on Helicopter-Borne AeroTEM II Electromagnetic & Magnetometer Survey, Turnagain Project, Job #04023 January 2005

BC Hydro (2011) Proposed NTL Tariff Presentation to Potential Mines, Power Point Presentation by BC Hydro Economic & Business Development, 7 October , 2011

Baldys, C., Hitchins, A. and Ross, G. (2006) "2006 Diamond Drilling Report on the Turnagain Property"

Carter, N.C. (2005) Report on the 2004 Exploration Program, Turnagain Nickel Property, Turnagain River Area, Liard Mining Division, British Columbia, private report for Hard Creek Nickel Corporation, 15 June, 2005

CIM Definition Standards (2010) For Mineral Resources and Mineral Reserves, CIM 27 November, 2010

Dunn, E.D, and Scagel, R. (2004) Biogeochemical Orientation Survey, Turnagain River Property, B.C. June 2004

Dunn, E.D, (2006) Soil Geochemistry, Turnagain River Property, B.C. Summary Report, March 2006

Francois-Bongarcon, D., (2007) AGORATEK International Office visit conclusions – revised Version #2: unpublished internal memorandum to HCNC July 12, 2007.

Francois-Bongarcon, D. (2009) AGORATEK International Reference Materials 05-94, 05-103, UM-2 and UM-4 Hard Creek Nickel Re-certification Report Revision 5, 6-18 March, 2009

Francois-Bongarcon, D. (2010) AGORATEC International QA-QC Draft Report, Hard Creek Nickel Turnagain Project, Internal Review, March 2010

Francois-Bongarcon, D. (2011) AGORATEK International QA-QC Final Review, Hard Creek Nickel Turnagain Project, Internal Review Summary and Memorandum, 20 November, 2011

Frontier Geosciences Inc. (2008) Report of Seismic Refraction Investigation Proposed Mine Site Facilities, Turnagain Project, Hard Creek Nickel Corporation, Project FGI-1050, November 2008

Frontier Geosciences Inc. (2011) Report of Total Field Magnetometer Survey, Turnagain Project, Hard Creek Nickel Corporation, Project FGI-1202, June 2011

G & T Metallurgical Services Ltd., Kamloops, British Columbia (31 October 2008) Project No. KM2181, Metallurgical Development for the Hard Creek Project, Dease Lake, BRITISH COLUMBIA Canada

G & T Metallurgical Services Ltd., Kamloops, British Columbia (17 November, 2009) Advanced Flowsheet Development for the Turnagain Nickel Project, Dease Lake, British Columbia, Canada, Project No. KM2348

Gabrielse, H., (1998) Geology of Cry Lake and Dease Lake map areas, north-central British Columbia, Geological Survey of Canada Bulletin 504.

Gaham International Ltd (1997) Geophysics Report on the Cub Claim Group, Turnagain River Area, B.C. for Bren-Mar Resources Ltd., August 1997

Knight Piesold (2005) Hard Creek Nickel Corporation, Turnagain Nickel, Baseline Environmental Studies 2004/2005, Ref VA102-141/2-1 13 July, 2005

Knight Piesold (2006) Turnagain Project – Preliminary Tailings Storage Facility Alternatives Study, Ref VA06-00593 4 April, 2006

Knight Piesold (2007) Turnagain Project – Preliminary Mine Development Alternatives Assessment, Ref VA07-01017 July 2007

Knight Piesold (2007) Hard Creek Nickel Corporation, Turnagain Nickel, Hydrometeorology Report, Ref VA102-141/03-3 21 December, 2007

Knight Piesold (2007) Hard Creek Nickel Corporation, Turnagain Nickel, Water Quality Summary Report, Ref VA102-141/3-5 21 December, 2007

Knight Piesold (2008) Hard Creek Nickel Corporation, Turnagain Nickel, 2007 Environmental Summary Report, Ref VA102-141/3-4 2 January, 2008

Knight Piesold (2008) Hard Creek Nickel Corporation, Turnagain Nickel, Project Description Report, Ref VA102-141/03-6 22 February, 2008

Knight Piesold (2008) Hard Creek Nickel Corporation, Turnagain Nickel, 2008 Surface Water and Groundwater Quality Summary Report, Ref VA102-141/4-4 23 December, 2008

Knight Piesold (2010) Flat Creek Hydrologic Report, Hard Creek Nickel Corporation, Memorandum, Ref VA102-141/4-A.01 23 February, 2010

Knight Piesold (2010) Hydrology and Climatology Site Visit, Hard Creek Nickel Corporation, Memorandum, Ref VA102-141/4-A.00 19 August, 2010

Nixon, G.T. (1998): Ni-Cu Mineralization in the Turnagain Alaskan-Type Complex: A Unique Magmatic Environment in Geological Fieldwork 1997, p. 18-1-18-10.

Ounpuu, M. (2011) Metallurgical Progress Report for Sample 10-265 and Variability Tests, February 2011, draft internal memorandum

Ounpuu, M. (2011) Progress Report on Sample 10-265 Metallurgical Testing January 2011, draft internal memorandum

Ounpuu, M. (2011) Progress report on Process Mineralogy Examination, March 2011, draft internal memorandum

Questor Surveys Ltd (1996) High Resolution Aeromagnetic Survey, Turnagain River Area for Bren-Mar Resources Ltd, Ron Sheldrake, Project #Q7619, August 1996

Reid, J. (2011) Reid Resource Consulting PTY Ltd, The Nickel Market prepared for Hard Creek Nickel Corporation, 30 September, 2011

Ross, G. and Froc, N. (2011) Critical Review of Quality Assurance – Quality Control, Turnagain Nickel Project, Internal Report, October 2011

SGS Lakefield (2008) Proposed Grinding Systems for Turnagain Project Based on Small Scale Data, Project 11813-001, 26 August, 2008

S.J. Geophysics Ltd, Pezzot, E.T. (2006) Turnagain Magnetic Study – Cliff Zone Anomaly 3D Magnetic Inversion, 18 May, 2006

S.J. Geophysics Ltd, Pezzot, E.T. (2007) Turnagain Magnetic Study – NWZ and Hatzl Drilling 3D Magnetic Inversion, 12 May, 2007

Scheel, J.E., Nixon, G.T. and Scoates, J.S. (2005) New Observations on the Geology of the Turnagain Alaskan-Type Ultramafic Intrusive Suite and Associated Ni-Cu-PGE Mineralization, British Columbia, BC Ministry of Energy and Mines Geological Fieldwork 2004, Paper 2005-1, pp. 167-176

Scheel, J.E. (2007): Age and origin of the Turnagain Alaskan-type intrusion and associated Ni-sulphide mineralization, North-Central British Columbia, Canada. M.Sc.thesis, University of British Columbia, Vancouver, British Columbia

Simpson, R. (Geosim) (2006): Technical Report and Mineral Resource Estimate, Turnagain Nickel Project, Turnagain River Area, Liard Mining Division, British Columbia: Technical Report prepared for HCNC, effective date 13 April 2006.

Smee, B., (undated 2006) Certificate of Analysis, Hard Creek Nickel Standard UM2; Certificate of Analysis, Hard Creek Nickel Standard UM2; Certificate of Analysis, Hard Creek Nickel Standard UM4; Certificate of Analysis, Hard Creek Nickel Standard 05-94; Certificate of Analysis, Hard Creek Nickel Standard 05-103: unpublished report to HCNC, updated, 12 p.

Simpson, R (Geosim) (2007) Mineral Resource Update, Turnagain Nickel Project, Turnagain River Area, Liard Mining Division, British Columbia: Technical Report prepared for HCNC, effective date 29 March 2007

Thurber Engineering (2005) Geological Air Photo Interpretation, Hard Creek Exploration Property, File 19-4510-0 17 January, 2005

Thurber Engineering (2009) Geological Air Photo Interpretation, Flat Creek Watershed, Turnagain River Area, B.C., File 19-4510-1 4 September, 2009

Urbani, M. (2011) Technical Report for Bulk Concentrate Production Testwork Program, Turnagain Project, Hard Creek Nickel Corporation, Strategic Metallurgy Pty Ltd, August 2011

Valard Construction Ltd (2011) Transmission Options and Preliminary Cost Estimates, Turnagain Mine, Hard Creek Nickel Corporation, 24 September, 2011

Valard Construction Ltd (2011) Internal Memorandum on Power Supply and Interconnection Fees, Turnagain Mine, Hard Creek Nickel Corporation, 30 September, 2011

Wardrop (2010) NI 43-101 Technical Report on the Turnagain Project, Preliminary Assessment, Project No. 0955080100-REP-R0001-01, April 2010

Western Minerals Technology Pty Ltd., Osborne Park, Australia (31 July 2009) Mineralogical Report for Hard Creek Nickel Corporation, Mineralogical Liberation Analysis of Turnagain Composite C Rougher Flotation Products, Job Number J3028

Western Mineral Technology Pty Ltd., Osborne Park, Western Australia (12 January 2010) Report to Hard Creek Nickel Corporation; Conceptual Design and Supporting Test Work of a Grinding and Flotation Circuit for the Turnagain Nickel Project, Job No. J3028

Western Mineral Technology Pty Ltd., Osborne Park, Western Australia (3 February 2010) Report to Hard Creek Nickel Corporation; Conceptual Design and Supporting Test Work of a Grinding and Flotation Circuit for the Turnagain Nickel Project – Project Extension, Job No. J3049

Woods, D.V (1999) Geophysical Report on a Borehole Transient EM Survey, Turnagain Project, June 17, 1999

Woods, D.V. (2003) Geophysical Interpretation Report on Magnetic and Induced Polarization/Resistivity Surveys, Turnagain Project, Dease Lake , B.C. 13 January, 2003

Woods, D.V. (2004) Preliminary Interpretation of Detailed Geophysical Survey Results over the Horsetrail Zone, Turnagain Project, Dease Lake, B.C. Memorandum 28 May, 2004

Xstrata Process Support (2008) Turnagain Ore Characterization Project, Interim Mineralogical Report, Phase 1A, Hard Creek Nickel Corporation, Turnagain Project No. 09001823-09010824, Falconbridge, ON (7 March 2008)

Xstrata Process Support (2008) Turnagain Ore Characterization Project, Phase 1A, Hard Creek Nickel Corporation, Turnagain Project No. 09001823-09010824, Falconbridge, ON (17 June 2008)