# N.I. 43-101 TECHNICAL REPORT & PRELIMINARY ECONOMIC ASSESSMENT FOR THE TURNAGAIN PROJECT

British Columbia, Canada

Report date: November 18, 2020
Effective date: October 28, 2020

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# 1.0 SUMMARY

## 1.1 Introduction

This report has been compiled by Hatch Ltd. (Hatch) for Giga Metals Corporation (Giga Metals) with input from the following independent consultants:

- Blue Coast Metallurgy Ltd. (Blue Coast)
- Kirkham Geosystems Ltd. (Kirkham)
- Kerr Wood Leidal Ltd. (KWL)
- Knight Piésold Ltd. (KP)
- Wood Mackenzie (WM)

The project is based on a 90,000 t/d capacity nickel-sulphide flotation plant that is scheduled to commence production at 50% capacity (45,000 t/d) and expand to full capacity after Year 5. The current mine life is 37 years.

This report is compliant with National Instrument 43-101 (NI 43-101) disclosure standards for mineral projects in Canada.

## 1.2 Project Location

The Turnagain Project is located in northern British Columbia (BC), Canada, 1,350 km northwest of Vancouver and 65 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. A seasonal exploration trail provides vehicular access to the site; this trail will require significant upgrades to meet project requirements. Current power supply for the exploration camp 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 (PGEs).

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 to 1998, resuming exploration activities after the name change to Canadian Metals Exploration Limited (CME) in 2002.

In 2004 after a change of management, CME became Hard Creek Nickel Corporation (HNC) and from then until 2010, several exploration programs were conducted, including mapping, soil and sediment sampling, geophysical surveys, metallurgical studies, diamond drilling, and environmental baseline studies. Until 2010, 79,351 m in 320 holes had been completed.

In 2017, after a period of relative dormancy, HNC changed its name to Giga Metals Corporation (Giga Metals), and in 2018 drilled 10,835 metres in 40 drill holes and restarted metallurgical and environmental baseline studies.

To date, 90,635 m in 362 holes have been completed.

The first resource estimate for the property was produced in 2003 by N.C. Carter. Several have since been produced by Ron Simpson of Geosim, including updates dated May 2009 and December 2011. For this report Garth Kirkham of Kirkham Geosystems has prepared an updated resource estimate current to the 2018 drilling year.

Four previous Preliminary Economic Assessments have been prepared for the property, two by AMEC of Americas Ltd. (AMEC) in 2006 and 2008; a third by Wardrop Engineering Inc. (Wardrop) in 2010; and a fourth by AMC Mining Consultants (Canada) Ltd. (AMC) in December 2011.

## 1.4 Geology & Mineralisation

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 hornblendite. The complex is elongate and broadly conformable to the northwesterly-trending regional structural grain.

The ultramafic rocks are generally fresh to mildly serpentinised; however, more intense serpentinisation 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 mineralisation, 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, however, they are not considered economic at this time.

## 1.5 Resource Estimate

The mineral resources were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy's (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (December, 2019) and are reported in accordance with NI 43-101 guidelines.

The 362 drill holes in the database were supplied in electronic format by Giga Metals, 307 of which had assay values. The primary economic contributor is shown to be nickel (Ni%) content, and the secondary is cobalt (Co%). Sulphur (S%) has similarly been analysed and estimated on a block-by-block basis. Assay values were composited to 4.0 m within the mineralised domains: (1) Du-Wh-Sp (dunite, wehrlite, serpentinite); (2) cPx-oPx (clinopyroxenite, olivine, magnetite and hornblende clinopyroxenite); (3) volcanics; (4) dykes; (5) overburden.

An evaluation of the probability plots for nickel and sulphur composites suggests that no outlier values could result in an overestimation of resources.

The chosen block size was 15 m x 15 m x 15 m, roughly reflecting the drill hole spacing which is spaced at approximately 50 m centres. The resource estimation plan includes the coding of lithological zones code in each block and the estimation of nickel, cobalt, and sulphur grades using ordinary kriging. The search criteria of the estimation process were the utilisation of a 150 m omni-directional ellipse, a minimum of four composites per block, maximum of 16 and a maximum of 4 composites per drill hole.

Mineral resources are classified under the categories of measured, indicated and inferred according to CIM guidelines. Mineral resource classification was based primarily on drill hole spacing and on continuity of mineralisation. Measured resources were defined at Turnagain as blocks with a distance to three drill holes of less than ~40 m to nearest composite and an average of 80 m and occurring within the estimation domains. Indicated resources were defined as those with a distance to three drill holes of less than ~60 m and an average distance of 100 m. Inferred resources were defined as those with an average drill hole spacing of less than ~150 m and meeting additional requirements. Final resource classification shells were manually constructed on sections.

This estimate is based upon the reasonable prospect of eventual economic extraction based on continuity of an optimized pit, using estimates of operating costs and price assumptions. The “reasonable prospects for eventual economic extraction” were tested using floating cone pit shells. The Horsetrail, Northwest, and Duffy zones of the deposit are all included within the Horsetrail reasonable prospects pit shells. The pit optimization results are used solely for testing the “reasonable prospects for eventual economic extraction” and do not represent an attempt to estimate mineral reserves.

The differences between the previous resource estimate as described in a 2011 Preliminary Economic Assessment by AMC Consultants of Vancouver, BC is the inclusion of an additional 36 infill drill holes totalling 8,940 m drilled in 2018 in the areas of the conceptual open pit and updated geological modelling.

Using a cut-off grade of 0.1% Ni, the Turnagain property contains an estimated 1,073 Mt of measured and indicated resources at 0.220% Ni and 0.013% Co. An additional 1,142 Mt grading 0.217% 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 | Kilotonnes | % Ni (T) | % Co (T) |
|---|---|---|---|
| Measured | 360,913 | 0.230 | 0.014 |
| Indicated | 712,406 | 0.215 | 0.013 |
| Measured & Indicated | 1,073,319 | 0.220 | 0.013 |
| Inferred | 1,142,101 | 0.217 | 0.013 |

Notes: **(1)** All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI 43-101"). **(2)** Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate reasonable expectation of eventual economic extraction, as required under NI 43-101; mineralisation lying outside of these pit shells is not reported as a mineral resource. **Mineral resources are not mineral reserves and do not have demonstrated economic viability.** **(3)** Open pit mineral resources are reported at a cut-off grade of 0.1% Ni. Cut-off grades are based on a price of US $7.50 per pound, nickel recoveries of 60%, ore and waste mining costs of $2.80, along with milling, processing and G&A costs of $7.20. **(4)** Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated. **(5)** Due to rounding, numbers presented may not add up precisely to the totals provided and percentages my not precisely reflect absolute figures.

## 1.6 Metallurgical Testing

Numerous phases of testing have been conducted on Turnagain samples over the past two decades. Earlier mineral processing work focused on developing a low-grade concentrate suitable for on-site hydrometallurgical processing; however, a breakthrough on concentrate cleaning in 2011 created an opportunity to make flotation concentrates for direct sale. Since then, work has focused on the production of high-grade nickel-sulphide concentrates. The initial work mostly used material drilled from the south end of the Horsetrail Zone, whereas more recent work has been conducted on samples from the Horsetrail and Northwest zones.

Samples from the Turnagain deposit have undergone extensive grindability testing (i.e., at least 77 Bond work index tests, 12 semi-autogenous grinding (SAG) mill comminution tests, 5 Bond rod mill work index tests, 5 abrasion tests and 5 crusher work index tests). In addition, piston press testing has been conducted to evaluate the response of the material to comminution by high-pressure grinding rolls (HPGRs).

Turnagain material is hard, with a mean Bond ball mill work index of 19.8 kWh/t, SAG milling Axb of 27.1 and Bond rod mill work index of 18.6 kWh/t. The SAG Axb hardness of the 15$^{th}$ percentile material is 23.0, which indicates the Turnagain material is highly resistant to SAG milling. This makes HPGR a favourable alternative comminution technology for the project.

Nickel occurs in both sulphide and non-sulphide form. More than 99% of the sulphide nickel is hosted in pentlandite, with pyrrhotite hosting less than 1% of the nickel. Non-sulphide nickel is mostly hosted in olivine and serpentine. There is considerable variability in nickel deportment within pentlandite and in non-sulphide form; this is the primary driver behind nickel recovery to concentrate. The pentlandite grain size is also quite widely distributed. There is an appreciable content of fine-grained pentlandite that is only adequately liberated at grind sizes ($P_{80}$) of well below 100 µm.

The host rock is comprised of serpentine (median = 42%), olivine (29%) and clinopyroxene (12%). Talc was essentially absent from about 95% of the samples analysed, with the median content being 0.01%.

The flotation flowsheet adopted for this study includes primary grinding to 80% passing 85 µm, rougher flotation, and three or four stages of cleaner flotation. Residence times in rougher flotation and the first stage of cleaner flotation testing are 30 minutes and 15 minutes, respectively. Using an industry standard scale up factor of 2.2, the residence time in rougher flotation and the first stage of cleaner flotation used for plant design is 67, and 27 minutes, respectively. The use of reagents, including isopropyl xanthate collector (SIPX) (78 g/t), a dispersant (Calgon) (115 g/t) and methyl isobutyl carbinol (MIBC) frother (40 g/t), is conventional for nickel flotation. Dilute pulps have been shown to aid in the rejection of gangue in cleaning.

Although there is more pyrrhotite than pentlandite in the resource, pentlandite tends to float well, while pyrrhotite floats poorly, so good selectivity is achieved between pentlandite and pyrrhotite. This is a key reason why high-grade concentrates can be achieved. The other reason is the paucity of floatable gangue, so while Calgon is useful as a means of countering gangue interference in pentlandite flotation, gangue depressants are usually not needed.

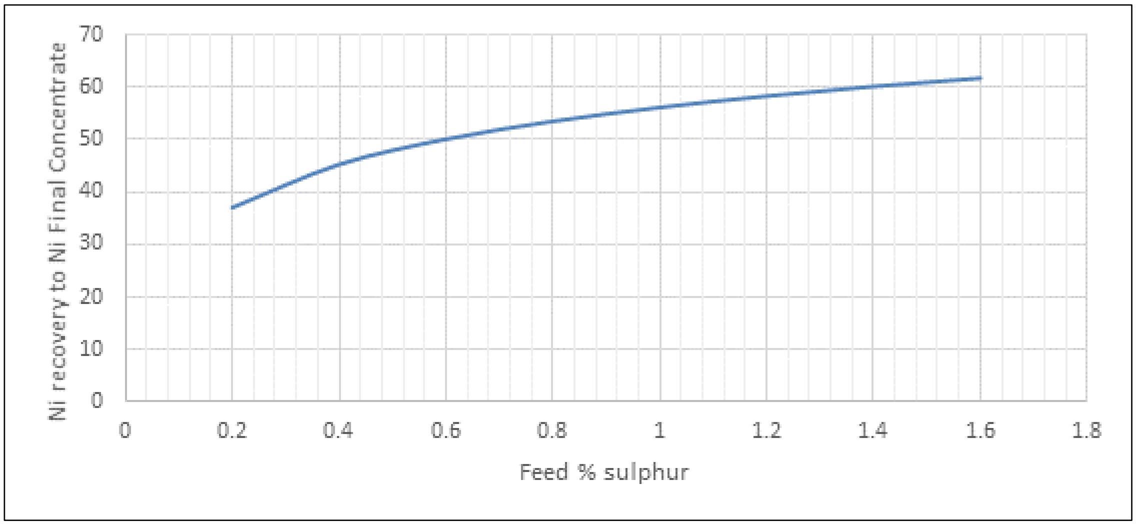
Numerous locked cycle tests have been conducted on Turnagain samples. Ten such tests that used slight variants of the optimum flowsheet yielded concentrate grades ranging from 15.3% to 21.4% nickel, averaging 19.3% nickel. Nickel recoveries have averaged 57.8%. On average, based on five multi-element analyses of locked cycle concentrates, the samples contained 1.2% cobalt, 2 g/t palladium, 1.2 g/t platinum, 32% iron and 4.4% magnesium.

Five different variability studies have been conducted on project samples since 2009. Each of these studies revealed a link between rougher nickel recovery and sulphur head grade, so these have been used to build a basic geometallurgical model for the project. Cleaner and locked cycle data have been used to convert rougher recoveries to expected recoveries to final concentrates, still as a function of sulphur head grade. This is shown in Figure 1-1.

Further work is required to refine the flowsheet. This includes investigating the use of regrinding in concentrate cleaning and determining how to tailor the production of concentrates for specific markets.

Additional work is also needed on lower sulphur-bearing samples. To date, development work has focused on samples assaying over 1.1% sulphur; however, the life-of-mine (LOM) sulphur grade is closer to 0.7%, so work is needed to confirm whether these materials can still be floated to produce premium grade nickel concentrates, and to explore if the drop in recovery with lower sulphur-bearing feeds can be better addressed (Figure 1-1).

**Figure 1-1: Nickel Recovery to Concentrate as a Function of Sulphur Head Grade**




Source: Blue Coast Metallurgy Ltd., 2020.

## 1.7 Recovery Methods

The process plant will consist of:

- one primary crusher followed by two trains of secondary crushing and HPGRs
- two grinding trains, each comprising two ball mills in series
- four banks of rougher flotation, utilising 630 $m^3$ tank cells
- two parallel lines of three-stage cleaner circuits and concentrate filtration

This circuit lends itself to simple modification for the first five years at a lowered capacity, reducing to one comminution train and two rougher banks with appropriate modifications to the cleaner circuit.

## 1.8 Mining Methods

The Turnagain deposit will be mined using open pit mining methods, 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 mineralised rock will require blasting and typical grade control methods using blast-hole sampling.

For the purpose of this study, the Horsetrail Pits are scheduled for a 37-year mine life. This includes the Horsetrail and Northwest mineralised zones, combined into one primary pit and a smaller pit located just outside the periphery to the northeast (Figure 1-2). Previous evaluations have indicated a potential open pit resource in the Hatzl Zone located on the east side of the Turnagain River, but this is not included in the scope of this study. The Turnagain River is fish-bearing and is considered a wildlife corridor. As such, any underlying mineralised material has been excluded.

The material contained and scheduled in the Horsetrail Pits is summarised in Table 1.2. These pits form the basis of the mine plan and production schedule in this study.

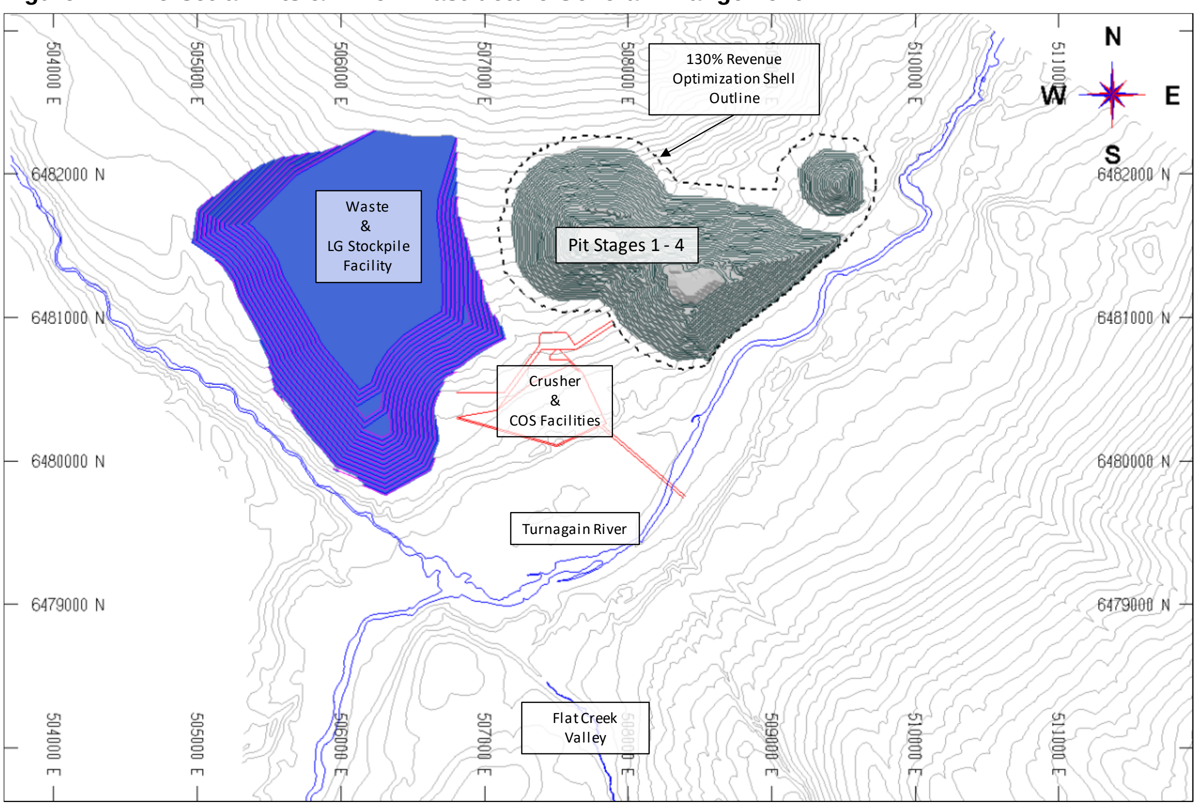
**Table 1.2: Potential In-pit Material**

| | Mineralisation (kt) | Waste (kt) | Strip Ratio | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|
| Horsetrail Pit Shells (PEA Basis, 37 Year LOM) | 1,121,980 | 207,880 | 0.19 | 0.221 | 0.013 | 0.60 |
| **Total** | **1,121,980** | **207,880** | **0.19** | **0.221** | **0.013** | **0.60** |

Notes: Includes inferred resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that these data will be realised.

Figure 1-2 shows a plan view of the scheduled pit shells and basic general arrangement of mine infrastructure.

**Figure 1-2: Horsetrail Pits & Mine Infrastructure General Arrangement**




Source: Hatch, 2020.

The annual mill steady-state feed rate for the plant Phase 1, Years 1 through 5, is set at 45 kt/d (16.4 Mt/a) and for the balance of the mine life, Years 6 through 37, 90 kt/d (32.9 Mt/a). The resource will be processed for 37 years at these rates.

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

Elevated cut-off grades will be employed in the initial production and through most of the mining life to enhance the economics of the project. Mineralisation lower than the elevated cut-off grades will be sent to stockpile in a joint waste and low-grade (LG) stockpile facility to the west of the crusher and coarse ore storage (COS) facilities. Mineralisation below the high-grade (HG) cut-off but above the respective LG cut-offs will be stockpiled. Stockpiled material will be reclaimed at the end of the mine life or blended with the run-of-mine feed when the opportunity occurs.

Pit waste material will be hauled to the waste and LG facility. 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 1-2 also shows the conceptual waste dumping and low-grade mineralisation stockpiling layout.

## 1.9 Infrastructure

On-site and off-site infrastructure is required to support a fly-in, fly-out mining operation, as described in Section 18. The two largest infrastructure requirements are related to power supply and waste/tailings management.

### 1.9.1 Power Supply

Since the 2011 PEA, BC Hydro completed the Northwest Transmission Line (NTL) from Terrace to Bob Quinn. In addition, the NTL Extension was built to Tatogga Lake to provide power for the Red Chris Mine, and a 25 kV line was built to serve the community of Iskut. The plan for power supply is to connect the proposed Turnagain Mine to the BC Hydro system with a ~160 km long (powerline route) 287 kV powerline from the existing operating BC Hydro 287 kV substation at Tatogga Lake.

### 1.9.2 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 neutralisation potential—and appropriate water collection and control measures will be implemented as described in the next section.

Various tailings management options were assessed in previous studies. The preferred option in Flat Creek has been presented in this study, as it offers good storage efficiency in terms of the ratio of capacity to embankment volume and has a relatively small catchment area.

Construction of the tailings management facility will commence with an initial starter dam to provide up to three years storage at the initial production rate of approximately 45,000 t/d and will be raised in stages using centreline construction throughout the approximately 37-year mine life. The tailings facility staging will accommodate the production rate increase to approximately 90,000 t/d in Year 6.

## 1.10 Environmental Considerations

The project is expected to be subject to both provincial and federal reviews of an environment impact assessment (EIA), which should be conducted in one review process through a substitution agreement between the provincial and federal agencies. This process will involve consultations with the public and First Nations, as well as detailed studies of baseline environmental settings and an assessment of potential project impacts. Once the EIA review is successfully completed, a number of environmental conditions stipulating preventive and mitigative measures will be mandated before allowing the project to proceed.

Baseline environmental studies to support the EIA process were initiated in 2004 and are ongoing. Additional studies will include air quality, background noise levels, vegetation, wildlife, soil quality, groundwater, and archaeology, 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 tailings management facility (TMF) of process tailings, process water and storm runoff
- 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 runoff waters
- optimisation of the storage and usage of water over the entire site with regard to environmental, operational and economic criteria

Based on the diesel and electricity consumption for mining and processing activities, the carbon intensity for the operations is estimated to average 74,428 $tCO_2e$ per year; this decreases to 23,080 $tCO_2e$ per year if an electrified fleet is used. The average project carbon intensity is estimated to 2.24 $tCO_2e$/t Ni for the base case and 0.69 $tCO_2e$/t Ni for the electrified case. Giga Metals continues to support research into mineral carbonation by the host rock of the Turnagain deposit, with an ultimate goal of better defining the quantity of $CO_2$ that can be sequestered and optimal tailings management strategies.

The project is located within the traditional territories of the Tahltan Nation and the Kaska Dena. Giga Metals has established positive engagements with the Tahltan Central Government and Kaska Dena Council via its member nation, Dease River First Nation, and will continue these respectful and ongoing engagements. Giga Metals will seek First Nations participation in ongoing baseline studies, the environmental assessment, and development of environmental and social mitigation and management plans. Giga Metals will also work in partnership with First Nations to optimise employment, training, and procurement opportunities through the project life cycle.

The reclamation and closure plan will minimise 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 Giga Metals project team, local stakeholders, and the appropriate regulatory authorities.

## 1.11 Capital & Operating Costs

The mine initial, expansion, and sustaining capital costs were estimated for the mine from the mine schedule and fleet requirements. For the processing plant, major equipment items were budget priced and a factored initial and expansion capital cost estimate was prepared. Sustaining capital was also factored. Infrastructure elements were taken from recent similar projects. As a check, the capital cost estimates were benchmarked against similar projects.

Knight Piésold provided TMF, site-wide surface water management, and reclamation/closure costs. Off-site electrical powerline costs were provided by Kerr Wood Leidal.

Project capital costs are summarised in Table 1.3.

**Table 1.3: Project Initial/Expansion Capital Cost Summary**

| Item | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Mine Directs | US$M | 133 | 45 | 178 |
| Process Plant Directs | US$M | 307 | 245 | 551 |
| Tailings Storage Facility Directs | US$M | 87 | 20 | 107 |
| On-site Infrastructure Directs | US$M | 77 | - | 77 |
| Indirects | US$M | 204 | 104 | 308 |
| Contingency | US$M | 191 | 99 | 290 |
| Owner's Cost & EA | US$M | 63 | 20 | 83 |
| Electrical Supply | US$M | 278 | - | 278 |
| Site Access Road | US$M | 42 | - | 42 |
| **Total Initial/Expansion Capital** | **US$M** | **1,381** | **532** | **1,913** |

Project sustaining capital costs, as well as closure and reclamation costs, are summarised in Table 1.4

**Table 1.4: Project Sustaining Capital & Closure & Reclamation Cost Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mine | US$M | 0 | 348 | 148 | 496 |
| Process Plant | US$M | 31 | 165 | 187 | 384 |
| Tailings Storage Facility | US$M | 107 | 377 | 335 | 819 |
| On-site Infrastructure | US$M | 8 | 23 | 26 | 57 |
| Electrical Supply (Tariff Supplement 37) | US$M | 90 | 82 | - | 172 |
| **Total Sustaining Capital** | **US$M** | **236** | **996** | **697** | **1,928** |
| **Closure & Reclamation** | **US$M** | **38** | **15** | **18** | **72** |
| **Total** | **US$M** | **274** | **1,011** | **715*** | **2,000** |

Note: * Includes $2.8 M in TMF and closure costs in Year 38.

The overall operating costs are summarised in unit cost terms in Table 1.5

**Table 1.5: Unit Operating Cost Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mining | US$/t milled | 3.52 | 2.89 | 2.46 | 2.72 |
| Processing & Site Infrastructure | US$/t milled | 4.90 | 4.39 | 4.38 | 4.42 |
| G&A | US$/t milled | 1.13 | 0.68 | 0.68 | 0.71 |
| Electrical Supply O&M | US$/t milled | 0.08 | 0.04 | 0.04 | 0.04 |
| **Total** | **US$/t milled** | **9.63** | **7.99** | **7.56** | **7.89** |

## 1.12 Economic Evaluation

The PEA case is based on the long-term nickel and cobalt prices provided by Wood Mackenzie, US$7.50/lb Ni and US$22.30/lb Co, and does not achieve a positive NPV at an 8% discount rate.

Pre-tax NPV becomes positive at the Wood Mackenzie long-term environmental, social and governance (ESG) incentive price of US$8.50/lb Ni. The returns are shown in Table 1.6.

The PEA case (Case 1) at US$7.50/lb Ni is used as the basis for this report. The nickel price is the long-term average forecast by Wood Mackenzie. PEA case returns are highly sensitive to input assumptions and should be viewed in the context of the sensitivity analysis. The PEA case economic outputs are summarised in Table 1.7.

**Table 1.6: Returns Summary**

| Item | Unit | Case 1: PEA<br>Wood Mackenzie Long-Term Price<br>US$7.50/lb Ni | Case 2:<br>Wood Mackenzie Long-Term ESG Incentive Price<br>US$8.50/lb Ni |
|---|---|---|---|
| **Pre-Tax NPV @ 8%** | **US$M** | **(269)** | **242** |
| Pre-Tax IRR | % | 6.3% | 9.4% |
| Pre-Tax Payback | years | 13.7 | 10.8 |
| **After-Tax NPV @ 8%** | **US$M** | **(443)** | **(88)** |
| After-Tax IRR | % | 4.9% | 7.4% |
| After-Tax Payback | years | 14.8 | 11.7 |
| **Nickel Price** | **US$/lb** | **7.50** | **8.50** |
| Cobalt Price | US$/lb | 22.30 | 22.30 |
| Exchange Rate | USD/CAD | 0.77 | 0.77 |

*Inverse of 1.30 CAD/USD applied.

**Table 1.7: PEA Case Key Economic Outputs**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | LOM |
|---|---|---|---|---|---|
| **Project Economics** | | | | | |
| NPV@ 8% Before Tax | US$M | - | - | - | (269) |
| NPV@ 8% After Tax | US$M | - | - | - | (443) |
| IRR Before Tax | % | - | - | - | 6.3% |
| IRR After Tax | % | - | - | - | 4.9% |
| Payback Period Before Tax | years | - | - | - | 13.7 |
| Payback Period After Tax | years | - | - | - | 14.8 |
| **Market Drivers** | | | | | |
| Nickel Price | US$/lb | 7.50 | 7.50 | 7.50 | 7.50 |
| Cobalt Price | US$/lb | 22.30 | 22.30 | 22.30 | 22.30 |
| Exchange Rate | USD/CAD | 0.77 | 0.77 | 0.77 | 0.77 |
| Nickel Payable % | % | 78% | 78% | 78% | 78% |
| Cobalt Payable % | % | 35% | 35% | 35% | 35% |
| **Physicals** | | | | | |
| Effective Strip Ratio (incl. stockpile) | t::t | 0.50 | 0.56 | 0.24 | 0.40 |
| Ore Throughput: Annual Average | Mt/a | 15.3 | 32.7 | 32.6 | 30.3 |
| Nickel Head Grade | % | 0.260% | 0.220% | 0.216% | 0.221% |
| Cobalt Head Grade | % | 0.016% | 0.013% | 0.013% | 0.013% |
| Recovery: Nickel and Cobalt | % | 57.3% | 51.6% | 46.5% | 49.6% |
| Nickel Recovered | kt | 114 | 557 | 558 | 1,229 |
| Cobalt Recovered | kt | 7 | 33 | 32 | 73 |
| **Financial** | | | | | |
| Revenue | US$M/a | **317** | **517** | **456** | **462** |
| Mining Cost | US$/t milled | 3.52 | 2.89 | 2.46 | 2.72 |
| Processing and Site infrastructure | US$/t milled | 4.90 | 4.39 | 4.38 | 4.42 |
| G&A | US$/t milled | 1.13 | 0.68 | 0.68 | 0.71 |
| Electrical Supply O&M | US$/t milled | 0.08 | 0.04 | 0.04 | 0.04 |
| **Site Operating Costs** | **US$/t milled** | **9.63** | **7.99** | **7.56** | **7.89** |
| **Site Operating Costs** | **US$/lb Ni recovered** | **2.93** | **3.20** | **3.41** | **3.27** |
| Concentrate Shipping | US$/lb Ni recovered | 0.31 | 0.31 | 0.31 | 0.31 |
| Cobalt Credit | US$/lb Ni recovered | (0.47) | (0.47) | (0.45) | (0.46) |
| **Net Operating Cost** | **US$/lb Ni recovered** | **2.77** | **3.04** | **3.27** | **3.12** |
| Construction Capital Cost | US$M | 1,381 | 532 | - | 1,913 |
| Sustaining Capital & Closure Cost | US$M | 274 | 1,011 | 715 | 2,000 |

## 1.13 Conclusions & Recommendations

The key elements in this update are additional resource drilling and metallurgical work, the future potential of non-stainless steel demand, and the increased certainty of power supply to northern British Columbia.

Based on an updated 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 37-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 concomitant risk is that the geometallurgical variability may prove greater than expected. There is also an opportunity for full mine-to-product (including tailings) project optimisation 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. It is recommended, therefore, that Giga Metals carry the project forward to the pre-feasibility stage, in accordance with the budget presented in Table 1.8.

**Table 1.8: Pre-feasibility & Early Environmental Works Study Budget (now to mid-2022)**

| **Item** | **US$000s** |
|---|---|
| Coordinating-Design Engineer | 1,280 |
| Power Studies | 240 |
| Marketing Studies | 80 |
| Metallurgical Development | 600 |
| Geotechnical | 1,200 |
| $CO_2$ Sequestration | 80 |
| ARD/ML | 160 |
| Tailings | 200 |
| Environment | 1,720 |
| Community Program | 160 |
| **Resource/Reserve Program** | 3,040 |
| Government Bonding for works | 160 |
| Field Support Costs | 520 |
| **Total PFS & Early Environmental Studies** | **9,440** |
| **Total All Activities** | **12,440** |

# 2.0 INTRODUCTION

This Preliminary Economic Assessment (PEA) report has been prepared by Hatch as an update to the previous PEA prepared by AMC in December 2011. The principal reasons for this update are to include the results of additional drilling, resource modelling and metallurgical testwork and updating cost, revenues and economics for 2020.

The following independent consultants have contributed to this report:

- Hatch
- Kirkham Geosystems Ltd.
- Knight Piésold Ltd. (KP)
- Kerr Wood Leidal (KWL)
- Blue Coast Metallurgy Ltd.
- Wood Mackenzie (WM)

Site visits are shown in Table 2.1.

A list of the qualified persons (QPs) responsible for each section of this report is provided in Table 2.1, and their QP certificates are appended to the back of this report.

All the qualified persons listed in Table 2.1 are independent of Giga Metals.

**Table 2.1: Persons Who Prepared or Contributed to this Technical Report**

| Qualified Person | Employer | Date of Site Visit | Sections of Report |
|---|---|---|---|
| Ian Thompson, P.Eng. | Hatch | October 9 to10, 2018 | 1.1, 1.2, 1.3, 1.8, 1.9, 1.11, 1.13, 2, 3, 4.4, 5.1, 5.4, 15, 16, 18.3, 21 (except for 21.1.2, 21.1.3.1, 21.1.4, 21.2.3, 21.2.6), 24, 25.2, 25.8.1.2, 25.8.2.2, 26.1, 26.3 and 27 |
| Persio Rosario, P.Eng. | Hatch | | 1.7, 17, 21.1.2, 21.2.3, 25.4, 25.8.1.3, 25.8.2.4, 26.5 |
| Evan Jones, P.Eng. | Hatch | | 1.10, 4.3, 5.2, 5.3, 5.6, 20, 25.8.1.5, 25.8.2.7 and 26.8 |
| Gerald (Gerry) Schwab, P.Eng. | Hatch | | 5.5, 18,1, 18.5.5, 18.5.6, 18.5.7, 18.6, 18.7, 18.9, 18.10, 25.8.1.4, 25.8.2.5 and 26.6 |
| Stefan Hlouschko, P.Eng. | Hatch | | 1.12, 22, 25.7 |
| Garth Kirkham, P.Geo. | Kirkham Geosystems | October 9 to10, 2018 | 1.4, 1.5, 4.1, 4.2, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.1, 25.8.1.1, 25.8.2.1 and 26.2 |
| Daniel Friedman, P.Eng. | Knight Piésold Ltd. | September 7, 2005 and June 16, 2009 | 1.9.2, 18.2, 18.4, 18.5.1, 18.5.2, 18.5.3, 18.5.4, 21.1.4, 21.2.6 and 25.5 |
| Ron Monk, P.Eng. | Kerr Wood Leidal | | 1.9.1, 18.8, 21.1.3.1, 25.6, 25.8.2.6 and 26.7 |
| Chris Martin, C.Eng., MIMMM | Blue Coast Metallurgy | | 1.6,13, 25.3, 25.8.2.3 and 26.4 |
| Andrew Mitchell, PhD, C.Eng. | Wood Mackenzie | | 19 |

# 3.0 RELIANCE ON OTHER EXPERTS

This report is based on information provided by Giga Metals and other specialists throughout the course of the study. The qualified persons have taken reasonable measures to confirm information provided by others and have taken responsibility for the information.

The following specialists, who are not qualified persons for the purposes of this report, were relied upon for specific advice:

- Cameron McCarthy, President, Swiftwater Consulting, P.Eng./P.Geo. (BC), provided the update and review of sections on climate (Section 5.2), surface water quality (Section 20.1.2), groundwater flows (Section 20.1.4) and groundwater quality (Section 20.1.5).
- Richard Pope, Partner, Dillon Consulting Ltd, RPBio (BC), provided the update and review of sections related to environmental baseline studies (Section 20.1), aquatic life (Section 20.1.1), soil quality (Section 20.1.3), and archaeological studies (Section 20.1.6).
- Laura Laurenzi, Hydrogeochemist, BGC Engineering, P.Geo. (BC), reviewed and updated the discussion on metal leaching and acid rock drainage potential (Section 18.3.2).
- Trevor Crozier, Principal Hydrogeological Engineer, BGC Engineering, P.Eng. (BC), reviewed and updated Hydrogeology,
- Cathy (Catherine) Mackay, COO and Senior Biologist, EDI, MSc., RPBio., P.Ag. (BC), provided advice on water quality baseline locations and sampling (Section 20.1.2), wildlife monitoring, wildlife management plan preparation and prepared annual wildlife monitoring reports (Section 20.1).

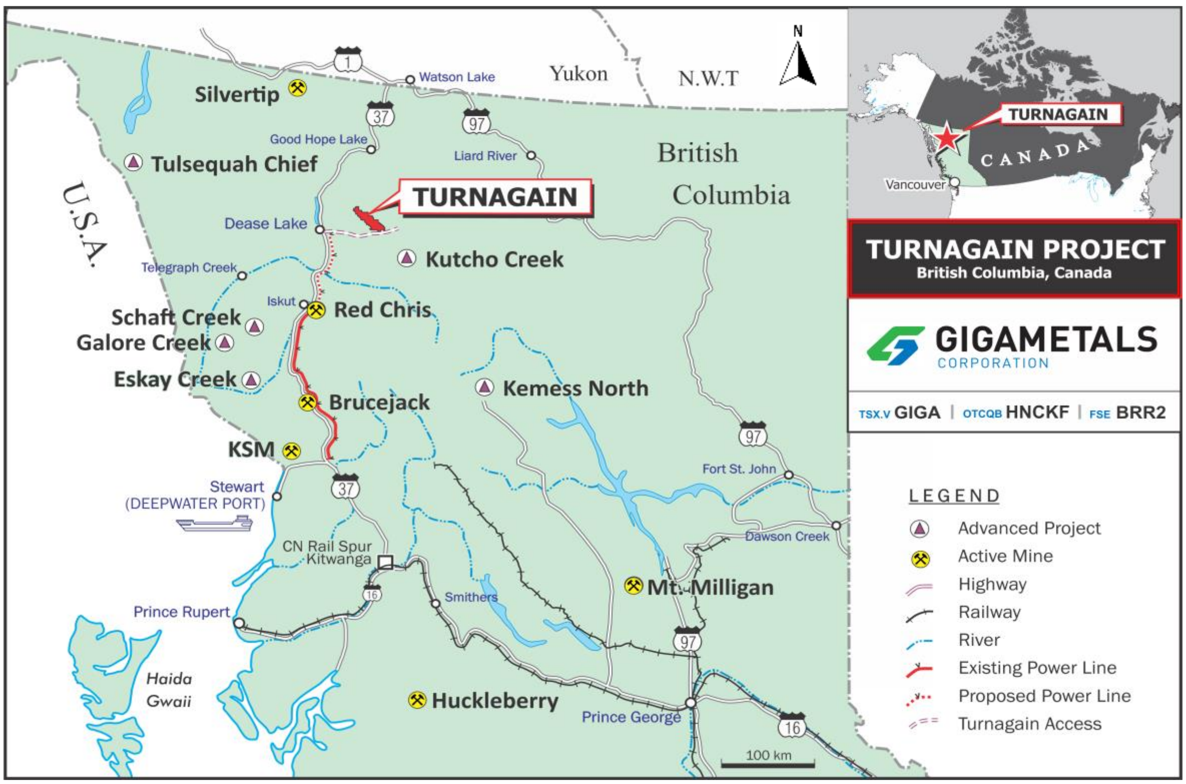
The qualified persons responsible for these sections used their experience to determine if the information from the specialists was accurate.

# 4.0 PROPERTY DESCRIPTION & LOCATION

## 4.1 Location

The project is located approximately 65 km east of Dease Lake in the Liard Mining Division of northwestern British Columbia (Figure 4-1). The deposit is approximately centred at UTM NAD83 Zone 9 coordinates 508,000 m E and 6,481,000 m N (58°28'10''N latitude and 128°51'46''W longitude). In the central claims, elevations range from about 1,000 metres above sea level (masl) along the Turnagain River to 1,800 masl at an unnamed summit in the central property area. The property is accessible via a 900 m gravel airstrip and a seasonal exploration trail from Highway 37, which is suitable for off-highway vehicle use during summer months.

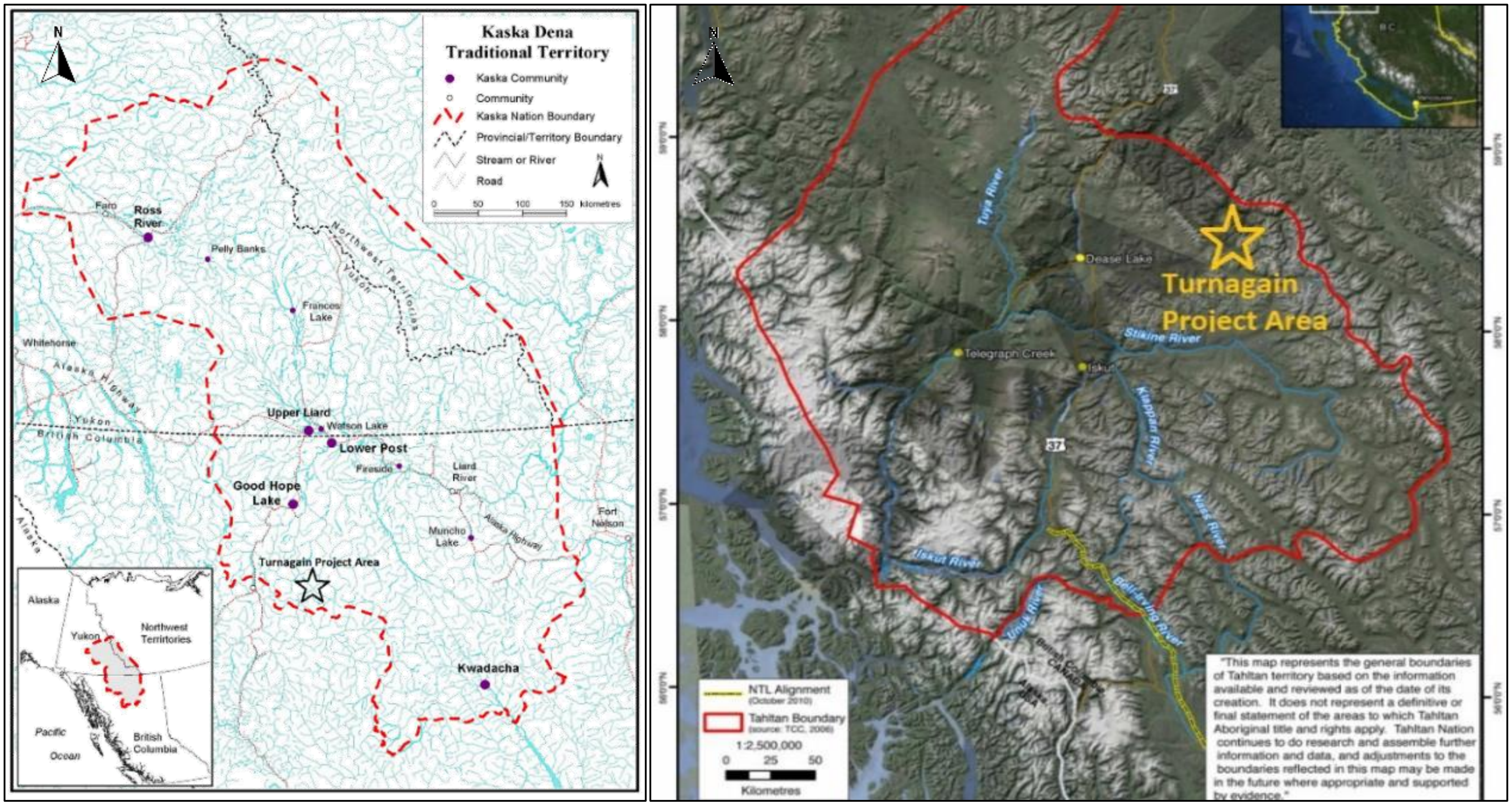
**Figure 4-1: Turnagain Project Location Overview Map**




Source: Giga Metals, 2020.

The closest community to the project is the community of Dease Lake (unincorporated), which is a town of approximately 335 people (BC Census Data, 2016) located on Highway 37 at the south end of Dease Lake. Other local communities include Telegraph Creek, Iskut, and Good Hope Lake. There are no residences near the mine site. The property lies within the traditional territorial claims of the both the Tahltan Nation and Kaska Dena (Figure 4-2).

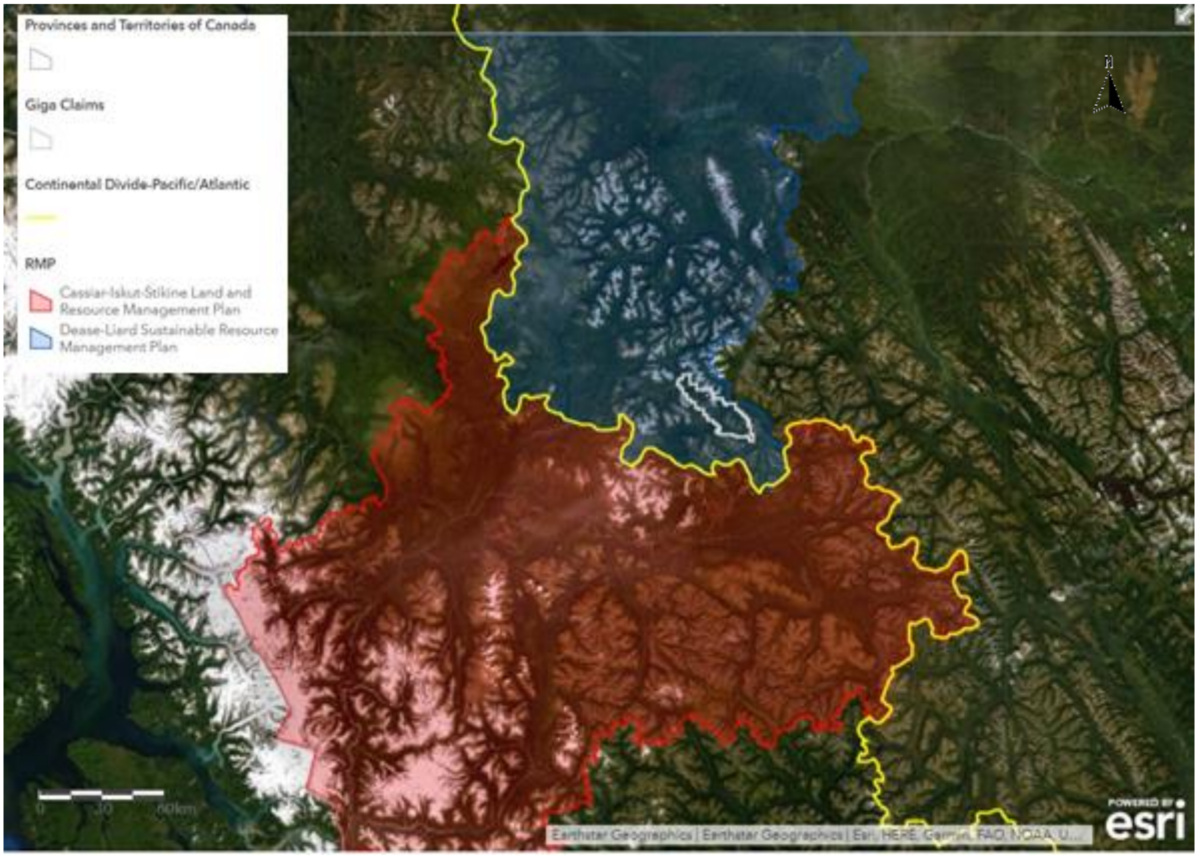
**Figure 4-2: Turnagain Project Relative to Traditional Territories of Kaska Dena & Tahltan Nation**




Source: Giga Metals, 2020.

The project is located within the Liard River watershed on the Arctic side of the Pacific-Arctic continental divide (yellow line), as shown in Figure 4-3.

**Figure 4-3: Turnagain Project Area Relative to Land Management Areas**



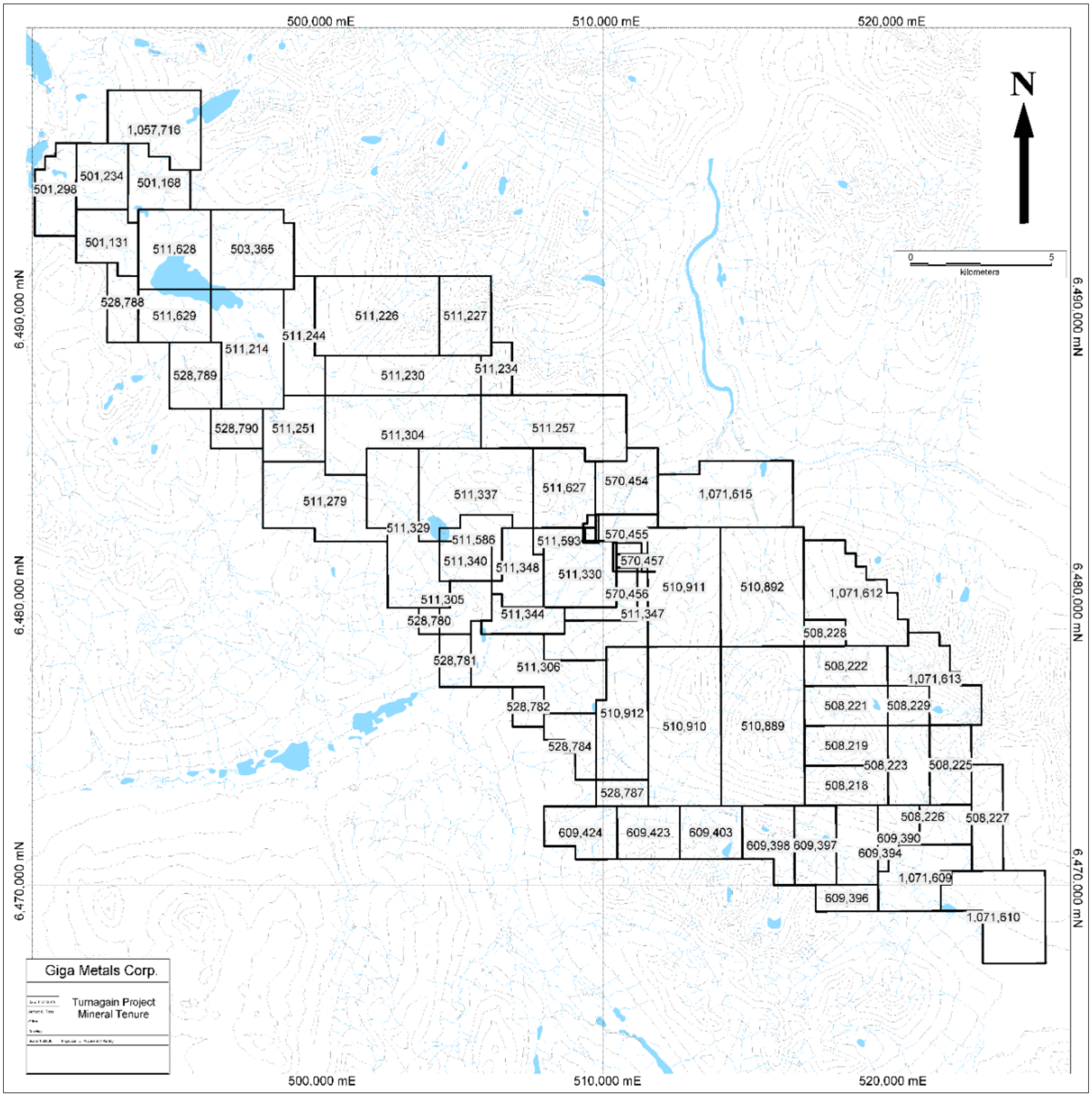

Source: Giga Metals, 2020.

The Turnagain property is found within the Dease-Liard Sustainable Resource Management Plan (BC MSRM 2004, 2012), while the access trail also passes through the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) (BC MSRM 2000).

## 4.2 Mineral Claims

The Turnagain Project is wholly owned by Giga Metals and consists of 71 mineral claims covering an area of approximately 38,000 ha (Figure 4-4). All claims are contiguous and occur within the BC Liard Mining Division in the Stikine region of northwest BC. The configuration of the various mineral claims is illustrated in Figure 4-4, which incorporates information plotted on BC Mineral Titles Reference Maps M104I-036 to -38, -045 to -047, -055 to -057, and 065. Details are listed in Table 4.1.

**Figure 4-4: Turnagain Project Claims**




Source: Giga Metals, 2020.

**Table 4.1: Mineral Claim Details**

| Tenure Number | Claim Name | Original Legacy Name(s) | Tenure Type | Good to Date | Area /ha |
|---|---|---|---|---|---|
| 407627 | PUP 4 | | Mineral | Dec-01-2029 | 500 |
| 501131 | Drift 1 | | Mineral | May-01-2027 | 422 |
| 501168 | Drift 2 | | Mineral | May-01-2027 | 421.8 |
| 501234 | Drift 3 | | Mineral | May-01-2027 | 421.7 |
| 501298 | Drift 4 | | Mineral | May-01-2027 | 421.8 |
| 503365 | | Hard 2 | Mineral | May-01-2027 | 793.3 |
| 508218 | Dinah 1 | | Mineral | May-01-2027 | 407.2 |
| 508219 | Dinah 2 | | Mineral | May-01-2027 | 407.1 |
| 508221 | Dinah 3 | | Mineral | May-01-2027 | 406.9 |
| 508222 | Dinah 4 | | Mineral | May-01-2027 | 406.7 |
| 508223 | Dinah 5 | | Mineral | May-01-2027 | 407.1 |
| 508225 | Dinah 6 | | Mineral | May-01-2027 | 407.1 |
| 508226 | Dinah 7 | | Mineral | May-01-2027 | 254.6 |
| 508227 | Dinah 8 | | Mineral | May-01-2027 | 407.3 |
| 508228 | Dinah 9 | | Mineral | May-01-2027 | 135.5 |
| 508229 | Dinah 10 | | Mineral | May-01-2027 | 203.4 |
| 510889 | | Flat 10, 13, 15 | Mineral | May-01-2027 | 1627.9 |
| 510892 | | Flat 2, 6 | Mineral | Dec-01-2029 | 1219.3 |
| 510910 | | Flat 9, 12, 14 | Mineral | Dec-01-2028 | 1424.3 |
| 510911 | | Flat 1, 5 | Mineral | Dec-01-2029 | 1066.9 |
| 510912 | | Flat 8, 11 | Mineral | May-01-2027 | 779.9 |
| 511214 | | Hard 4, 6 | Mineral | May-01-2027 | 979.9 |
| 511226 | | Hill 1, 2 | Mineral | May-01-2027 | 1216.1 |
| 511227 | | Hill 3 | Mineral | May-01-2027 | 506.7 |
| 511230 | | Hill 4, 5 | Mineral | May-01-2027 | 760.5 |
| 511234 | | Hill 6 | Mineral | May-01-2027 | 185.9 |
| 511244 | | Hard 5, 7 | Mineral | May-01-2027 | 489.9 |
| 511251 | | Hard 8 | Mineral | May-01-2027 | 473.4 |
| 511257 | | Hill 9, 10 | Mineral | Dec-01-2029 | 1014.4 |
| 511279 | | Hard 9, 10 | Mineral | Dec-01-2029 | 896.7 |
| 511304 | | Hill 7, 8 | Mineral | Sep-01-2029 | 1149.7 |
| 511305 | | Hound 3 | Mineral | Dec-01-2029 | 271 |
| 511306 | | Turn 2, Flat 7 | Mineral | Dec-01-2029 | 881.2 |
| 511329 | | Hound 1, 2 | Mineral | Feb-01-2029 | 1015.4 |
| 511330 | | Cub | Mineral | Oct-01-2029 | 592.6 |
| 511337 | | Cub 10, 18, Pup 1 | Mineral | Oct-01-2029 | 1065.8 |
| 511340 | | Cub 17 | Mineral | Dec-01-2029 | 253.9 |
| 511344 | | Turn 1, Bear 2 | Mineral | Dec-01-2029 | 271 |
| 511347 | | Flat 3, 4 | Mineral | Dec-01-2029 | 474.3 |
| 511348 | | Cub 2 | Mineral | Oct-01-2029 | 389.4 |
| 511586 | | Pup 2 | Mineral | Dec-01-2029 | 236.9 |
| 511593 | | Pup 3 | Mineral | Oct-01-2029 | 101.5 |
| 511627 | | Cub 11 | Mineral | Dec-01-2029 | 592.1 |

| Tenure Number | Claim Name | Original Legacy Name(s) | Tenure Type | Good to Date | Area /ha |
|---|---|---|---|---|---|
| 511628 | | Hard 1 | Mineral | May-01-2027 | 709 |
| 511629 | | Hard 3 | Mineral | May-01-2027 | 472.9 |
| 528780 | T1 | | Mineral | Dec-01-2028 | 67.7 |
| 528781 | T2 | | Mineral | Dec-01-2028 | 203.3 |
| 528782 | T3 | | Mineral | May-01-2028 | 152.6 |
| 528784 | T4 | | Mineral | May-01-2028 | 288.3 |
| 528787 | T5 | | Mineral | May-01-2027 | 169.6 |
| 528788 | T6 | | Mineral | May-01-2027 | 270.2 |
| 528789 | T7 | | Mineral | May-01-2027 | 422.5 |
| 528790 | T8 | | Mineral | May-01-2028 | 253.6 |
| 570454 | | Bear 1 | Mineral | Dec-01-2029 | 456.8 |
| 570455 | | Bear 19, Bear 21 to 28 | Mineral | Dec-01-2029 | 237 |
| 570456 | | Bear 3 to 18 | Mineral | Dec-01-2029 | 220.2 |
| 570457 | | Bear 20 | Mineral | Dec-01-2029 | 16.9 |
| 609390 | FLAT 7 | | Mineral | May-01-2027 | 254.6 |
| 609394 | FLAT 6 | | Mineral | May-01-2027 | 407.4 |
| 609396 | FLAT 8 | | Mineral | May-01-2027 | 203.8 |
| 609397 | FLAT 5 | | Mineral | May-01-2027 | 407.4 |
| 609398 | FLAT 4 | | Mineral | May-01-2027 | 407.4 |
| 609403 | FLAT 3 | | Mineral | May-01-2027 | 407.3 |
| 609423 | FLAT 2 | | Mineral | May-01-2027 | 407.3 |
| 609424 | FLAT 1 | | Mineral | May-01-2027 | 424.2 |
| 1057716 | NWMAG | | Mineral | Aug-01-2023 | 741.9 |
| 1071609 | BLICK 1 | | Mineral | Oct-04-2020 | 560.3 |
| 1071610 | BLICK 2 | | Mineral | Oct-04-2020 | 900.1 |
| 1071612 | FAULKNER 1 | | Mineral | Oct-04-2020 | 897.5 |
| 1071613 | FAULKNER 2 | | Mineral | Oct-04-2020 | 627 |
| 1071615 | FAULKNER 3 | | Mineral | Oct-04-2020 | 1032.1 |

Mineral claims staked 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), Hard Creek Nickel Corp. (HNC) and Giga Metals. The original option agreement gave Bren-Mar the right to earn a 100% interest in the mineral claims in exchange for 200,000 shares and incurring property expenditures of C$1 M within five years of acquisition. The 100% interest was earned subject to a 4% net smelter royalty (NSR) on possible future production from the mineral claim 511330. Giga Metals retains the right to purchase all or part of this royalty for C$1 M for each 1% of the royalty.

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

Between November 2003 and October 2019, additional claims were staked at various times, some of which were subsequently forfeited by way of the BC Ministry of Energy and Mines online map selection process, enlarging the Turnagain property to its current configuration of 71 claims covering approximately 38,000 ha.

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 four-post claim and 27 two-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 July 10, 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 April 30, 2007. All subsequent claim acquisitions for various exploration and access considerations were made using BC Ministry of Energy and Mines online map selection process. Most recently, in 2019, five additional claims contiguous with the Turnagain Property totalling 4,017 ha were staked in the southeastern portion of the project area in the Blick and Faulkner creek valleys.

## 4.3 Permits & Environmental Liabilities

Exploration work on mineral properties in BC requires a Notice of Work and Reclamation to be filed with the Ministry of Energy and Mines. Obtaining a permit to facilitate such work may require a reclamation security to be posted. The value of Giga Metals' Turnagain Project reclamation security is C$232,000, although this could be amended.

The project will require several permits, approvals, and authorisations from provincial and federal agencies, which are summarised in Section 20.

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 and further upgraded in-place in 2018 and 2019.

There is no knowledge 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 (511330) is held by the original property vendors J. Schussler and E. Hatzl. Giga Metals retains the right to purchase all or part of this royalty for C$1 million per each 1% of the royalty.

A 2% NSR on all future metal production is held by Conic Metals Corp. Giga Metals has a one-time option to repurchase 0.5% of the 2% royalty for US$20 million prior to the 5th anniversary of the NSR Agreement (i.e., July 12, 2023), which would leave Conic with a 1.5% NSR. Conic will have a right of first refusal on any future sale by Giga Metals of a royalty or product stream or similar instrument.

# 5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

## 5.1 Accessibility

The nearest airport to the project is at Dease Lake, 65 km by air 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. Fixed-wing charter flights are also available from Whitehorse (430 km northwest), Terrace and Smithers (440 km south and 425 km south-southeast, respectively), and Prince George (630 km southeast).

A 900 m coarse gravel airstrip immediately adjacent to the Turnagain exploration camp, constructed by Falconbridge in 1967, was upgraded by HNC in 2007 and has been used regularly as recently as summer 2019.

An exploration trail, known as the Boulder Trail, extending east from Highway 37 8 km south of Dease Lake has been used by large, articulated four- and six-wheel drive vehicles to convey equipment to mining and exploration operations in the region. The length of Boulder Trail plus 5 km spur to the Turnagain Camp is approximately 78 km, for a total distance from Dease Lake of 86 km. (Figure 5-1).

**Figure 5-1: Turnagain Project Location & Access**


Source: Giga Metals, 2020.

Trail access is not suitable for regular vehicle traffic. The Boulder Trail is also the primary access trunk to the Boulder City placer camp (10 km southwest), Kutcho Copper (40 km southeast) and

District Copper (immediately west) properties, as well as several jade operations in the vicinity. The BC government has recently convened a multi-stakeholder group to work together on Boulder Trail permitting, maintenance, and upgrades.

## 5.2 Climate

The climate of the area is generally characterised by cold winters, warm summers, and reasonably consistent precipitation throughout the year, although the summer months are the wettest. The nearest Government of Canada weather station is located at Dease Lake.

Climate monitoring commenced with the installation of the TURNMET climate station, a Campbell Scientific automated weather station installed at the east end of the site airstrip on August 11, 2004; climate data are available from that date until September 9, 2009. The station was then relocated several hundred metres west-southwest of the Hard Creek campsite and renamed TURNMET2. Several of the instruments were replaced and a new total precipitation gauge and solar radiation sensor were installed. TURNMET2 recorded average hourly and daily wind direction, wind speed, temperature, precipitation, and relative humidity. The station functioned well, but approximately one year of temperature and relative humidity data were lost from June 20, 2005 to August 24, 2006 due to damage to the station caused by a moose. The climate station was overhauled in mid-2018 and renamed "BC400972". The installation utilised the existing 10 m tower, but included the addition of new instrumentation, including barometric pressure and snow depth. The upgraded station also included GOES satellite telemetry, with data accessible through a web-based portal and updated hourly. Finally, a Class A evaporation pan was installed in 2019 and integrated with the station instrumentation. The geographical coordinates of the meteorological station are shown in Table 5.1.

**Table 5.1: Location of Meteorology Stations**

| Station Name | Year | Easting | Northing | Elevation (masl) |
|---|---|---|---|---|
| BC400972 | 2018-Present | 508064 | 6480139 | 1,020 |
| TURNMET2 | 2009-2011 | 5080644 | 6480139 | 1,020 |
| TURNMET | 2004-2009 | 508386 | 6480221 | 1,015 |
| Dease Lake | 1957-Present | 440983 | 6476843 | 802 |

Notes: The geographical coordinates are approximate location based on the text from the reports reviewed. NAD 1983 UTM Zone 9N

The estimated mean annual temperature for the period of record (as of the 2011 PEA report) 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. The temperature at the project site is approximately 2°C cooler than Dease Lake, which is to be expected given the higher elevation (1,020 masl vs. 802 masl).

As part of the 2011 PEA, concurrent months of precipitation from the Dease Lake climate station were compared with project site data to estimate the long-term average precipitation. The average ratio of precipitation between the project site and Dease Lake was 1.5, indicating that the project site receives approximately 50% more precipitation than Dease Lake. On average, Dease Lake has 381 mm of precipitation annually, so the long-term average precipitation for the project site is estimated to be 571 mm.

Mean monthly wind speeds range from 1.67 m/s in March to 0.73 m/s in July, as reported in the 2011 PEA. 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 recorded on September 27, 2004. The monthly wind direction data for the project site indicate that the predominant wind direction is from the southwest, which is expected due to the orientation of the Turnagain River valley at the project site. Wind speeds are expected to be higher at elevated sections of the project site.

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%, as reported in the 2011 PEA.

The mean annual lake evaporation for the project site is estimated to be 291 mm at 1,020 masl.

## 5.3 Hydrology

The project is located near the headwaters of the Turnagain River, which flows generally east and north to join the Kechika River, a tributary of the Liard River, which itself is a tributary of the Mackenzie River, which empties into the Arctic Ocean. Several small creeks flow into the Turnagain River at or near the project site, including Hard Creek, Flat Creek, and Faulkner Creek. Figure 5-2 and Figure 5-3 show Turnagain River near the existing airstrip and Hard Creek just before the confluence with the Turnagain River.

Annual flow patterns are typically characterised 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.

Hydrometric (streamflow) monitoring stations have been operated in the project area. Continuous monitoring has been done at these stations for the following periods:

- Lower Hard Creek – September 2005 to August 2008, July 2011 to July 2012 and 2018 to present (instantaneous manual flow measurements are ongoing for this site)
- Upper Hard Creek – August 2006 to October 2009 (but not ongoing)
- Farthest Hard Creek – September 2008 to August 2011 (but not ongoing)
- Faulkner Creek – August 2006 to August 2011 and June 2018 to present
- Flat Creek – August 2006 to August 2011 and June 2018 to present
- Turnagain River – September 2008 to August 2011 and June 2018 to present.

**Figure 5-2: Turnagain River at Project Location & Airstrip (looking Northeast)**

Source: Giga Metals, 2020.

**Figure 5-3: Hard Creek flowing into Turnagain River (looking Northwest)**

Note: Red dot represents Lower Hard Creek. Source: Giga Metals, 2020.

Installed dataloggers and pressure transducers record water level and temperature at 15-minute intervals, and environmental technicians complete monitoring activities that build the relationship between water level in the streams and flow. The most complete and continuous datasets are for the Faulkner Creek and Flat Creek stations, where over six complete years of data had been collected as of 2011. Unfortunately, periods of data were lost at both the Upper and Lower Hard Creek sites due to 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 in the 2011 PEA. The results indicate a mean annual unit runoff of approximately 16 L/s/km$^2$ 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.

## 5.4 Local Resources

An exploration camp built on the property in April 2003 can accommodate approximately 35 people. The camp consists of 17 walled tents, three trailers, and drill core logging and storage facilities. Power is provided by an on-site diesel generator and a backup generator.

On-site communications include satellite telephone and internet connections.

There are approximately 36 km of exploration trails on the property, constructed from the late 1960s to the present.

## 5.5 Infrastructure

Dease Lake (population 335) offers some supplies and services. The cities of Terrace (population 15,700) and Smithers (population 10,600), 580 and 600 km to the south of Dease Lake, 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 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. In 2014, BC Hydro completed the extension of the provincial transmission grid to Tatogga, approximately 100 km south of Dease Lake. This line is powered at 287 kV and includes interconnections to run-of-river hydro projects in western BC at Forest Kerr, McLymont Creek and Volcano Creek.

## 5.6 Physiography

The data on physiography of the Stikine region are 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 1,800 m are rounded while higher summits are rugged. Valley bottoms are 1,000 to 1,350 m elevation, while the highest peak (King Mountain) is about 15 km south of the Turnagain property at 2,425 m elevation. Plateau surfaces are at about 1,500 m.

The valley bottoms and lower elevation slopes are covered with glacial drift. Esker and drumlin formations are numerous and extensive. The ranges are characterised 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.

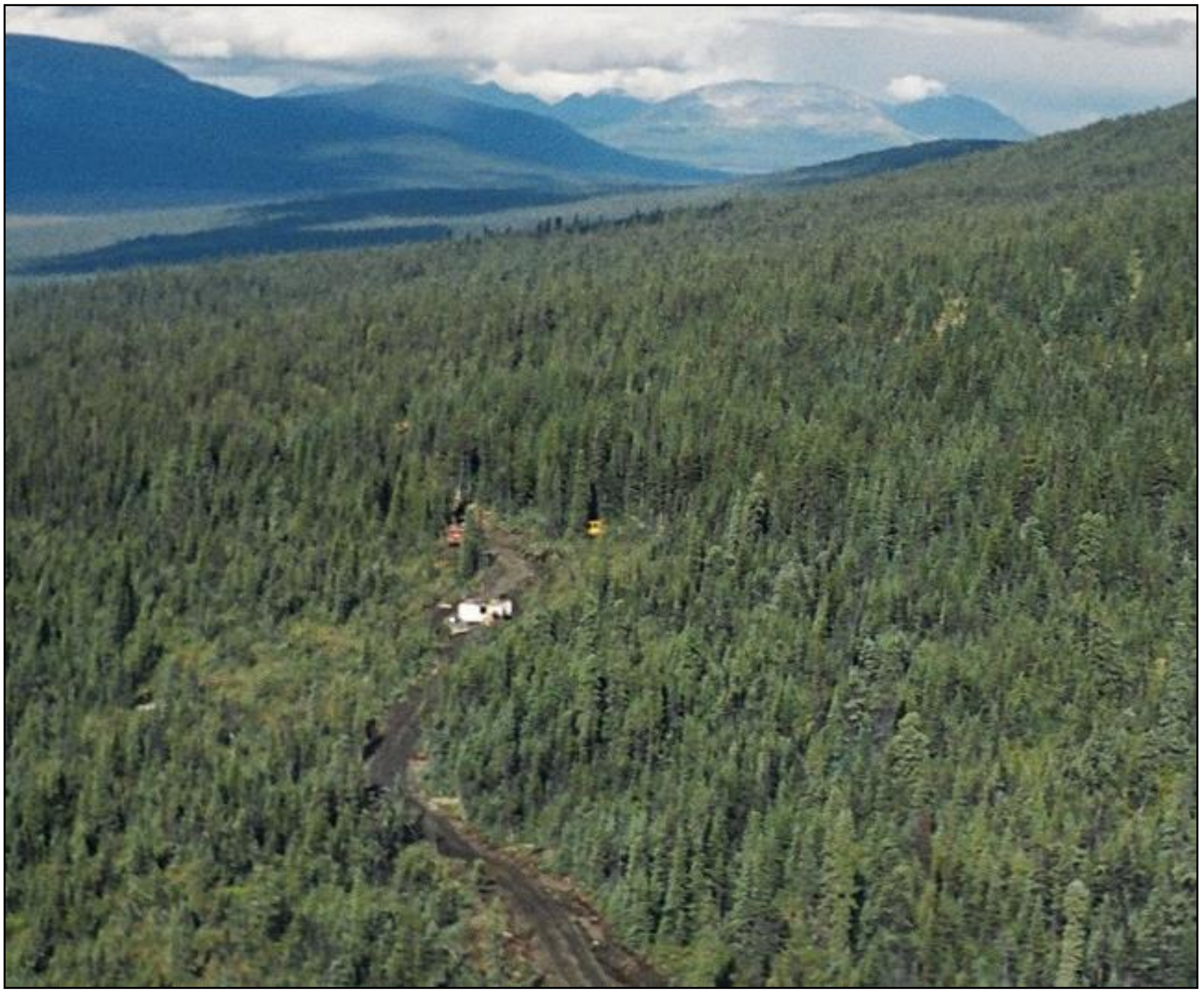
Several species of large mammal including grizzly bear, black bear, wolf, moose, caribou, mountain goat, and sheep can be found in the Cassiar Mountains. Bird species noted in the mountains include gyrfalcon, golden eagle, willow ptarmigan, least sandpiper, red-necked phalarope, snow bunting, and Smith‘s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

The Turnagain Project straddles the Turnagain River near its confluence with Hard Creek. The project area covers north-, south-, east-, and west-facing slopes northwest and southeast of the Turnagain River and alpine terrain above the tree line. Elevations range from about 1,000 m along the Turnagain River in the central claims area to 1,800 m at an unnamed summit in the central property area. The general site topography and environmental setting are shown in Figure 5-4 and Figure 5-5.

**Figure 5-4: Turnagain Project Area & Access (Transport Trucks on Boulder Trail, looking South)**

Source: Giga Metals, 2020.

**Figure 5-5: Turnagain Project General Setting (Exploration Drill Hole, looking Northwest)**



Source: Giga Metals, 2020.

# 6.0 HISTORY

The description of the property exploration history is based on work by Nixon (1998) and Baldys et al., (2006), as reported in prior Turnagain studies.

Nickel and copper sulphides were first recognised 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 (2,895 m). The work identified several sulphide “showings”. The exploration program tested many of the mineralised 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, several of the showings were re-staked and tested with short, small diameter core holes by an unnamed party. Three EX-sized core holes, totalling 55.5 m, 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, S Bridcut drilled a single drill hole (17 m) near the east bank of the Turnagain River and intersected unmineralised 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 drill holes (3,889 m), geological prospecting and sampling, down-hole pulse electromagnetic surveys in four of the 1997-1998 drill holes, 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 polarisation (IP) and ground magnetic survey followed by 1,687 m of diamond drilling in seven holes. Drilling continued in 2003, with 23 holes (including deepening of one of the 2002 drill holes) completed for 8,769 m. 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, including:

- geological mapping
- bedrock, stream sediment, and soil sampling
- surface, borehole, and airborne geophysical surveys
- mineralogical, metallurgical, and analytical studies
- 172 diamond drill holes for 41,502 m of drilling.

## 6.1 2006-2008

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; therefore, the 2006 resource estimate was reported in terms of sulphide nickel. Sulphide nickel was determined using an 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 Economic Assessment (PEA) on the project. A key assumption of the PEA was that a 0.10% sulphide nickel analysis cut-off was economically reasonable for the project. This cut-off was determined based on parameters selected for pit optimisation. Resources in the PEA 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 completed a second PEA, 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, results from the 2007 drill program were not available.

In June 2008, AMEC released an interim resource estimate that included results of all 2007 drill holes. In the same year, HNC completed an additional 16 core holes totalling 4,105 m.

## 6.2 2009-2011

In April 2010, Wardrop released another PEA, based on an updated resource that included the results of the 70 holes (21,099 m) drilled in late 2007 and 2008, as well as additional metallurgical work (production of a bulk flotation concentrate to feed a hydrometallurgical treatment plant). The project scope at that time consisted of an 87,000 t/d flotation plant employing the Outotec nickel chloride leach process to produce 35,000 t/a LME grade nickel metal and 2,000 t/a cobalt as a hydroxide.

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

In 2010-2011, a metallurgical testwork program at SGS Vancouver led to a breakthrough in reagent selection, resulting in repeatable recovery of high-grade flotation concentrates, with concentrate grades over 18% nickel and recoveries of total nickel in the 50% to 65% range.

In 2011, AMC completed a PEA based on a two-phase facility culminating in an 87,000 t/d flotation plant producing high-grade nickel concentrate (>18% nickel).

## 6.3 2012-2019

In 2017, HNC changed its name to “Giga Metals Corporation” to reflect the company’s intent to become a mineral firm providing nickel, cobalt, and potentially other raw materials for use in electric vehicles (EV) and battery energy storage markets. Giga Metals is a publicly traded company, headquartered in Vancouver, BC (a registered BC company), and listed on the TSX Venture Exchange.

In 2018, a diamond drilling program (40 holes, 10,835 m) was completed, along with a metallurgical test program and engineering studies.

The drilling program had two primary purposes: carry out infill drilling over the previously defined resource to update the resource database and retrieve fresh samples for metallurgical testing; and explore the very promising geophysical anomalies at the northern end of the property. The program is summarised below.

- 13 metallurgical infill holes totalling 3,073 m within the Horsetrail and Northwest zones of the Turnagain deposit
- 23 infill holes totalling 5,867 m sited between the Horsetrail and Northwest zones of the Turnagain deposit
- 4 exploration holes totalling 1,895 m in the MAG Zone and platinum-enriched Attic Zone, northwest of the Horsetrail Zone.

The 2018 metallurgical program, overseen by Blue Coast Metallurgy, was focused on a master composite created from five lithology composites originating from Hole 10-266, which was a horizontal drill hole, drilled through the southwest portion of the Horsetrail resource, dissecting what is likely to be the heart of the early production resource. The comprehensive program included a suite of comminution tests, mineralogy, and detailed flotation testing. Hole 10-266 was drilled in 2010; fresh drill core was not available for this study.

In 2018, Hatch completed a conceptual engineering study looking at replacing the semi-autogenous grinding step of prior PEA studies with more energy-efficient, high-pressure grinding rolls.

Engineering, resource evaluation, and metallurgical works were continued in 2019. Hatch conducted a process plant site location trade-off study. Blue Coast Research began a detailed flotation study, on fresh material, examining a wider range of variability samples and targeting the production of a bulk concentrate for marketing purposes. Kirkham Geosystems Ltd. completed an updated resource estimate, with 1.07 Bt of measured and indicated resources and a further 1.1 Bt of inferred resources, with average grades of 0.22% nickel and 0.013% cobalt.

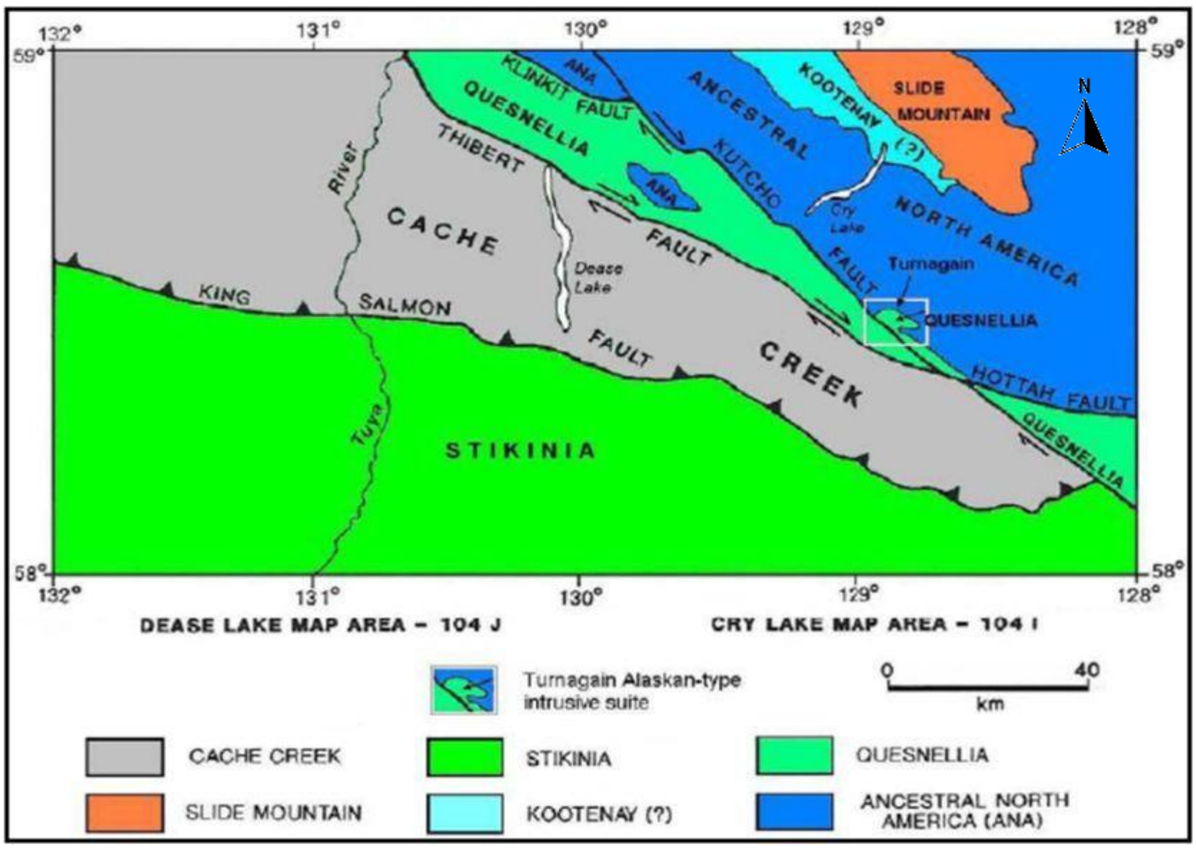
# 7.0 GEOLOGICAL SETTING & MINERALISATION

## 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), AMEC (2006) and AMC (2011). 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 ultramafic 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 on the property.

**Figure 7-1: Regional Structural Setting – Turnagain Property**




Source: 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). 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.

A series of undated sedimentary rocks, possibly volcaniclastic, lies south of the Turnagain complex. This series may represent rocks of the Lay Range assemblage of the Quesnel terrane (Figure 7-2). 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.

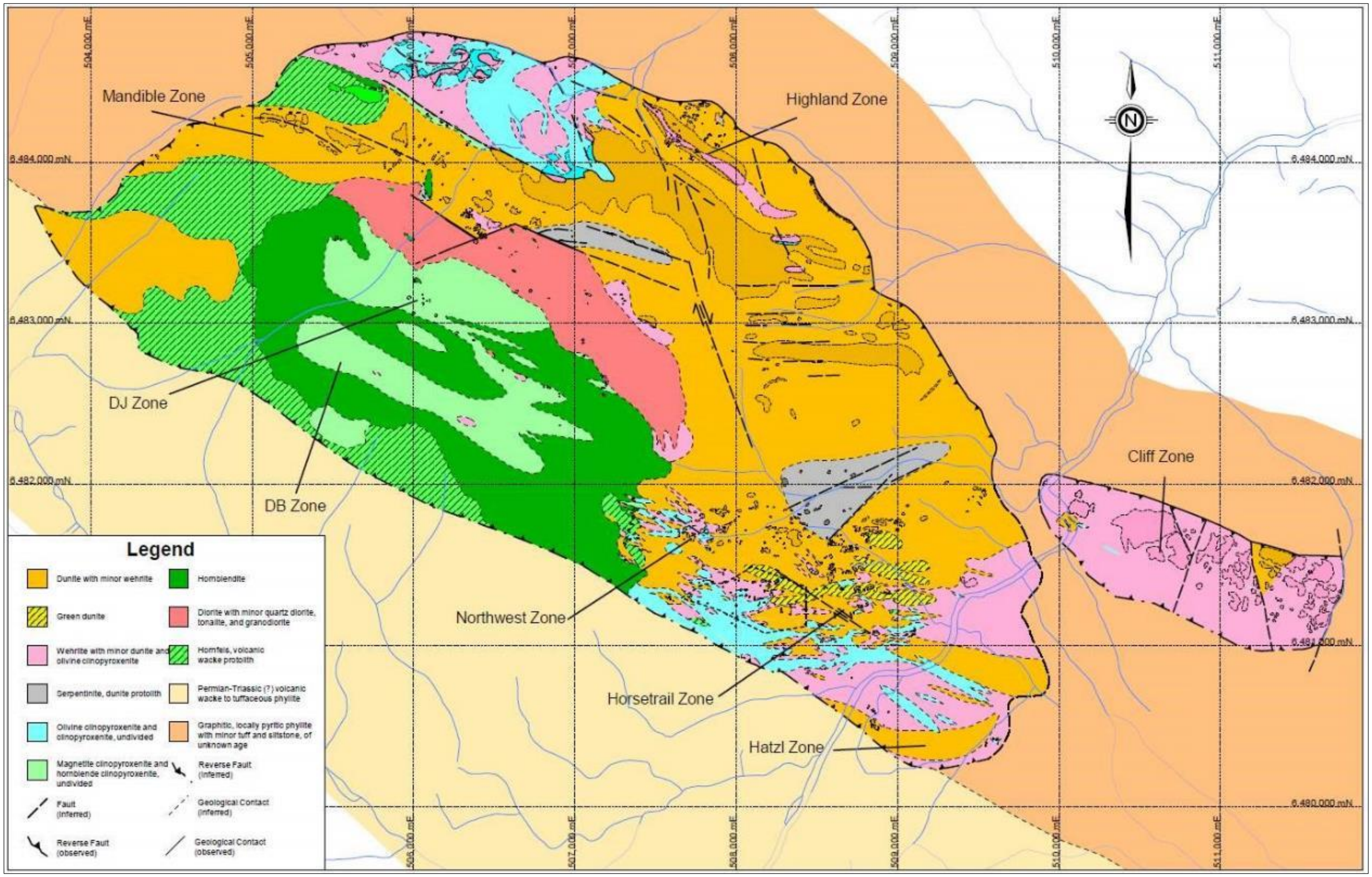
## 7.2 Property Geology

The Early Jurassic (190 ±1 million years ago (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 7-2). The complex is elongate and broadly conformable to the northwesterly trending regional structural grain.

The ultramafic rocks are generally fresh to mildly serpentinised; however, more intense serpentinisation and talc-carbonate alteration occur 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 variable dips and is truncated by the faulted eastern boundary of the complex. Despite localised zones of well-developed cumulate layering, way-up direction indicators are inconclusive and the internal structure of the Turnagain complex is poorly understood (Nixon, 1998).

**Figure 7-2: Property Geology**




Source: Kirkham 2020.

The descriptions of lithologies in the following subsections are 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 remobilised and intruded by thin dunite dikes.

Serpentinisation volumes are highly variable, but generally are no more than about 10% of the rock by volume. The degree of overall serpentinisation is higher in the Horsetrail, Northwest, and Hatzl zones. Secondary magnetite is abundant where serpentinisation is pervasive. Some dunite that is proximal to massive sulphide mineralisation 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 clinopyroxene, and may reflect magmatic layering.

### 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 clinopyroxene. 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. Both types of wehrlite commonly contain abundant serpentine up to 85% of the rock by volume.

### 7.2.3 Olivine Clinopyroxenite & Clinopyroxenite

These rock types mostly crop out in the northwestern part of the intrusion and commonly comprise around 85% cumulus clinopyroxene and smaller amounts of cumulus olivine. These rocks are also common along the southern margin of the Horsetrail and Northwest 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 dikes are commonly found adjacent to the cumulate clinopyroxenite or intruding more olivine-rich lithologies in the Horsetrail and Northwest zones. These latter intrusions are interpreted to be late-stage injections of trapped liquid through olivine-rich cumulates.

### 7.2.4 Hornblende Clinopyroxenite & 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 & Hornblende Clinopyroxenite

Generally found in the southwestern 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 km x 300 m 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 dikes, about 1 m to 2 m wide and clearly post-mineral, have been noted cutting wehrlites and clinopyroxenites in drill core. Some dikes may be up to 20 m wide and all dikes 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, like those seen marginal to the ultramafic intrusion, are present within the ultramafic intrusive rocks. These inclusions are thought to be the sulphur source responsible for the sulphide mineralisation in the Turnagain intrusion and are sourced from the wall rocks.

## 7.3 Mineralisation

Showings of semi-massive and massive sulphides have been identified by work to date. These semi-massive and massive zones, plus broad zones of disseminated sulphides, are generally hosted by dunite and wehrlite near the southern and eastern margins of the ultramafic body. The central and northern dunite is largely devoid of sulphide minerals although their highly magnesian olivine is more enriched in nickel (up to 0.20 to 0.30 weight percentage) than the olivine in the peridotites and pyroxenites of the Horsetrail and Northwest zones, which may be nickel-depleted in areas of sulphide mineralisation. Nixon (1998) suggests that these features are further evidence of fractional crystallisation of the ultramafic magma.

Primary sulphide minerals consist of pyrrhotite with lesser pentlandite (iron-nickel sulphide) and minor chalcopyrite. Some bornite has been reported. 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 remobilisation of primary sulphides adjacent to dikes, altered xenoliths, and serpentinised areas.

Secondary nickel and copper sulphides, including violarite and valleriite, have been noted in serpentinised zones and both primary and secondary sulphides are associated with graphite (Nixon, 1998). Microscope 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 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 mineralisation in both dunite and wehrlite and serpentinised 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. 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 mineralisation. 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 mineralisation 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%. Total cobalt grades range from 0.001% to 0.166%. The Horsetrail and the Northwest Zones form a zone approximately 2,000 m long in the east-west direction, and 550 m wide from north to south and have been tested by 251 drill holes.
- The Hatzl Zone mineralisation consists of disseminated and net textured pyrrhotite and pentlandite hosted by dunite and wehrlite. This mineralisation is similar to, and may be continuous with, the Horsetrail Zone. 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 mineralisation. The Hatzl Zone is 1,150 m long in a northeast direction and 300 m wide in a northwest direction and has been tested by 17 drill holes.
- The Duffy Zone mineralisation lies 500 m 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. The Duffy Zone is 300 m in diameter and does not crop out. It was discovered by exploration drilling in 2006. The zone has been tested by six drill holes.

Other mineralised zones are exploration targets include:

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

# 8.0 DEPOSIT TYPES

The geological setting of the sulphide mineralisation 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 mineralisation at Turnagain is hosted in dunite, wehrlite, olivine clinopyroxenite and clinopyroxenite and serpentinised equivalents. Sulphides include pyrrhotite, pentlandite, chalcopyrite, and trace bornite. Valleriite occurs where serpentinisation is intense.

# 9.0 EXPLORATION

Section 6 of this report summarises the early exploration work carried out between 1957 and 1995 and presents an overview of work completed by Giga Metals and its predecessor companies since acquisition of the project in 1996. This section presents more detail on exploration since 1996.

## 9.1 Geological Mapping

Sulphide-bearing outcrops of the Davis, Horsetrail, Discovery, and Cliff zones 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 drill holes, claim posts, and other geographical positions.

Detailed geological mapping was undertaken by Clark (1976) at various scales from 1:50 (inches to feet) to 1:1,000 as part of his Ph.D. thesis work. Additional mapping was completed by Giga Metals' geologists and Scheel (2007) at metric scales ranging from 1:1,000 to 1:10,000.

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. Giga Metals 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 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 (61 m) intervals along grid lines spaced 400 ft (122 m) apart. The samples were analysed for nickel, copper, and cobalt. Values greater than 650 ppm nickel and 300 ppm copper were considered anomalous; cobalt values were erratic. The best results were obtained from a 900 m x 450 m area west of the Discovery Zone where anomalous nickel values ranged from 800 ppm to 2000 ppm.

A geochemical sampling program carried out in 2003 consisted of the collection and analysis of 250 soil samples at a 100 m spacing along four topographic contour lines between 1,300 m and 1,460 m elevation, northwest and upslope of the principal mineralised zones. An analysis and interpretation of the results obtained from these samples was undertaken by Dr. Colin E. Dunn, P. Geo. on behalf of Giga Metals in early 2004 (Carter, 2005).

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

More than 2,000 soil samples during the 2004 and 2005 programs were collected at 50 m intervals along survey lines spaced 200 m apart within an area of 15 km$^2$. More detailed sampling at 25 m intervals on lines spaced 50 m 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 m x 1,100 m and 900 m x 600 m. These anomalous areas flank the hornblende diorite-granodiorite intrusion within 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 analysis 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.

## 9.3 Geophysical Surveys

The following discussion, modified from Carter (2005), is considered thorough and reasonably representative of 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 utilised a helicopter borne AeroTEMII 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 m to 200 m spacing. Precise locations were established using a GPS.

Two geophysically anomalous areas within the ultramafic rocks were surveyed along lines on 50 m spacings. Terrain clearance was 30 m 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 and 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 100 m spacing with stations every 25 m along lines.

A total of 12.3 line-km were surveyed within an approximate 1 $km^2$ area. Several magnetic highs identified from the survey were correlated with pod-like serpentinised and magnetite-banded peridotite intrusions; however, four of the magnetic anomalies were potentially due to the presence of sulphides.

Grid B comprised 100 m spaced east-west lines with stations at 25 m 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 mineralised 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 Zone).

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

### 9.3.3 Down-hole Geophysics

Borehole pulse electromagnetic surveys were undertaken on four drill holes (97-9, 98-1, 98-4, and 98-5) in 1998. 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, correlated with zones of sulphide mineralisation containing elevated (+0.30%) nickel values, and with talc/serpentinite zones.

In 2004, down-hole geophysical surveys were completed on another six drill holes in conjunction with surface transient electromagnetic, very low frequency (VLF), and magnetic surveys over an 800 m by 900 m grid centred on the Horsetrail Zone. Several 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 then-potential tailings management areas in the Hard Creek Valley, as well as in the vicinity of the current low-grade stockpile and waste dump sites, to determine depth to bedrock and type of overburden.

## 9.4 Drilling

Since 1966, the Turnagain Project has been tested by 90,635 m of diamond drilling in 362 holes (Table 9.1). Analytical results for the last 10,835 m of diamond drilling in 40 holes drilled in 2018 are reported in Section 11.0. Results for earlier holes have been published in previous technical reports by Simpson (2006), AMEC (2007) and AMC (2011).

**Table 9.1: Summary of Drill Programs**

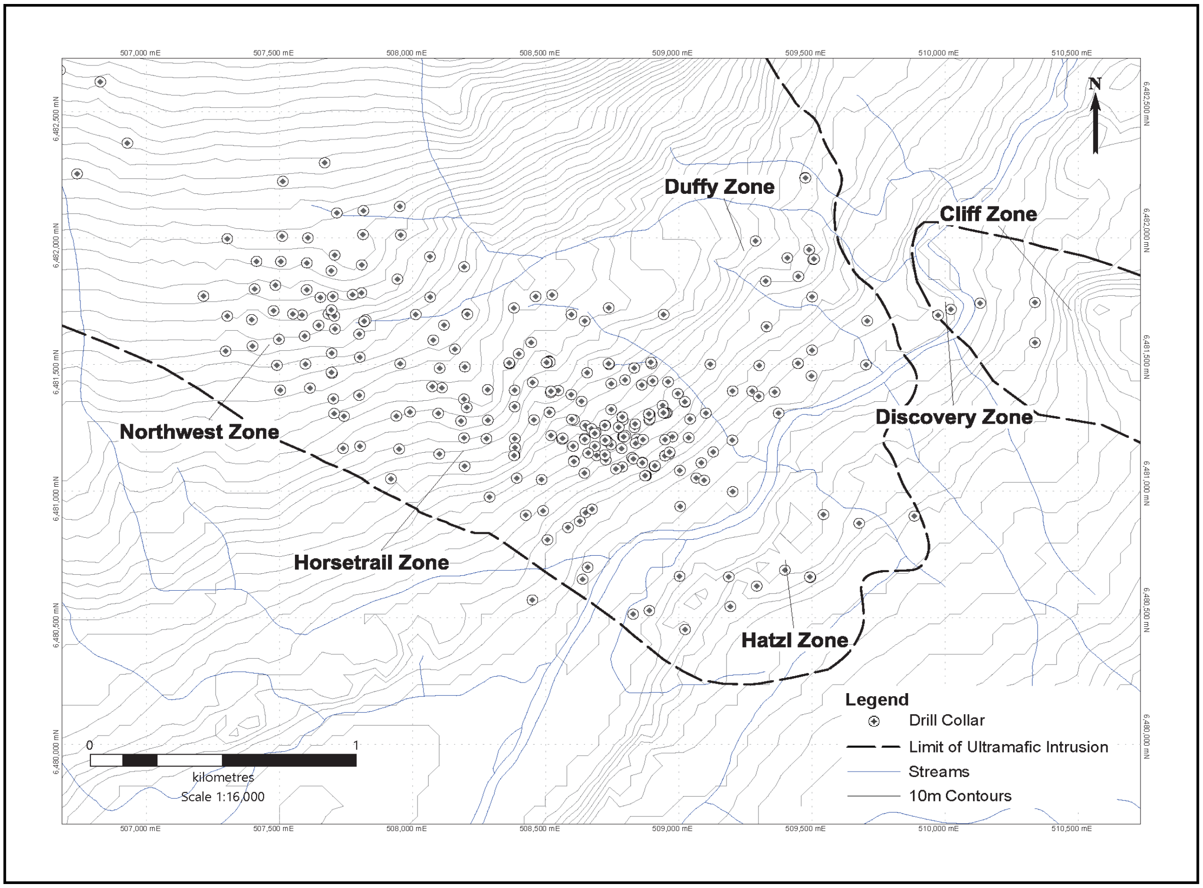
| Year* | Operator | No. Holes | Metres** |
|---|---|---|---|
| 1967 | Falconbridge | 13 | 1,310 |
| 1970 | Falconbridge | 15 | 1,457 |
| 1996 | Bren-Mar | 5 | 793 |
| 1997 | Bren-Mar | 9 | 1,855 |
| 1998 | Bren-Mar | 5 | 1,264 |
| 2002 | Canadian Metals | 7 | 1,938 |
| 2003 | Canadian Metals | 22 | 8,419 |
| 2004 | Hard Creek Nickel | 49 | 7,633 |
| 2005 | Hard Creek Nickel | 37 | 7,541 |
| 2006 | Hard Creek Nickel | 68 | 19,173 |
| 2007 | Hard Creek Nickel | 74 | 23,928 |
| 2008 | Hard Creek Nickel | 16 | 4,105 |
| 2010 | Hard Creek Nickel | 2 | 384 |
| 2018 | Giga Metals | 40 | 10,835 |
| **Total** | | **362** | **90,635** |

Notes: *Five holes were extended in subsequent years. Extension metres in this table listed under the year of holes' original collaring. ** Rounding to nearest metre may occur.

# 10.0 DRILLING

The Turnagain drill hole database contains 362 drill holes totalling 90,635 m of drilling. The previous technical report (AMC, 2011) reported on all drilling completed up to, and including, 2010. No drilling took place from 2011 to 2017. In 2018, Giga Metals completed 40 drill holes totalling 10,835 m. Hole locations within the defined resource are shown in Figure 10-1.

**Figure 10-1: Drill Hole Location Plan**




Source: Kirkham 2020.

Most of the holes drilled to date have been moderately to steeply inclined. Since 2004, contractor DJ Drilling (2004) Ltd. 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 2 m. Since 2006, core sampling has been completed predominantly on 4 m intervals.

Table 10.1 lists the drill holes along with location and orientation.

**Table 10.1: Drill Hole Collars, Location & Orientation**

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH02-01 | 509329.3 | 6481650 | 1070 | 203.3 | 0 | -90 |
| DDH02-02 | 508737.5 | 6481501 | 1130 | 213.1 | 0 | -85 |
| DDH02-03 | 508737.5 | 6481501 | 1130 | 318.21 | 180 | -50 |
| DDH02-04 | 508737.5 | 6481501 | 1130 | 149 | 0 | -50 |
| DDH02-05 | 508512 | 6481514 | 1160 | 152.4 | 0 | -90 |
| DDH02-06 | 508512.31 | 6481513.43 | 1162.68 | 485.2 | 181.82 | -50.43 |
| DDH02-07 | 508507 | 6481510 | 1160 | 416.4 | 225 | -50 |
| DDH03-01 | 508515 | 6481510 | 1160 | 501.7 | 240 | -50 |
| DDH03-02 | 508117.1 | 6481656 | 1230 | 532.2 | 225 | -45 |
| DDH03-03 | 507601.7 | 6481902 | 1302 | 462.1 | 180 | -50 |
| DDH03-04 | 507601.6 | 6481902 | 1302 | 334.4 | 180 | -70 |
| DDH03-05 | 508654.78 | 6481467.3 | 1138.29 | 590.7 | 168.93 | -47.53 |
| DDH03-06 | 508563.8 | 6481206.63 | 1104.66 | 523 | 180 | -50 |
| DDH03-07 | 508583.25 | 6480856.29 | 1036.14 | 434.4 | 169.45 | -47.07 |
| DDH03-08 | 509447.08 | 6481503.95 | 1027.51 | 477.3 | 206.08 | -56.43 |
| DDH03-09 | 509447.38 | 6481504.58 | 1027.51 | 252.1 | 200 | -85 |
| DDH03-10 | 508503.89 | 6481506.83 | 1162.67 | 577.9 | 197 | -51.73 |
| DDH03-11 | 508502.8 | 6481508 | 1160 | 249.9 | 0 | -55 |
| DDH03-12 | 508730 | 6481199.11 | 1086.75 | 349.6 | 350.63 | -76.1 |
| DDH03-13 | 508560.64 | 6481206.26 | 1104.27 | 322 | 6.75 | -49.75 |
| DDH03-14 | 508560.59 | 6481205.13 | 1104.23 | 261.6 | 6.75 | -84.17 |
| DDH03-15 | 508484.38 | 6481045.69 | 1090.17 | 508.1 | 21.78 | -64.95 |
| DDH03-16 | 508690.89 | 6481140.65 | 1079.91 | 369.1 | 34.52 | -49.28 |
| DDH03-17 | 508832.99 | 6481251.47 | 1083.42 | 296.6 | 41.4 | -51.05 |
| DDH03-18 | 508890.97 | 6481301.36 | 1082.83 | 242.95 | 50.83 | -50.3 |
| DDH03-19 | 508862.34 | 6481113.97 | 1058.1 | 303.3 | 43.57 | -49.72 |
| DDH03-20 | 508975.32 | 6481214.47 | 1061.59 | 214.6 | 45.65 | -46.8 |
| DDH03-21 | 508605.95 | 6481280.83 | 1117.19 | 333.75 | 32.17 | -46.77 |
| DDH03-22 | 508382.53 | 6481398.5 | 1167.89 | 281.9 | 53.15 | -48.62 |
| DDH04-23 | 508744.58 | 6481183.8 | 1085.01 | 413.3 | 46.15 | -49.75 |
| DDH04-24 | 508789.59 | 6481215.51 | 1083.65 | 370.65 | 46.4 | -46.9 |
| DDH04-25 | 508954.57 | 6481309.43 | 1081.56 | 221.3 | 40.72 | -49.53 |
| DDH04-26 | 508950.95 | 6481308.42 | 1081.88 | 178.6 | 28.8 | -49.23 |
| DDH04-27 | 508888.25 | 6481299.62 | 1082.97 | 264 | 29.67 | -47.97 |
| DDH04-28 | 508727.68 | 6481198.15 | 1086.84 | 233.15 | 5.03 | -43.37 |
| DDH04-29 | 508824.15 | 6481132.97 | 1063.56 | 245.7 | 45.32 | -46.35 |
| DDH04-30 | 508907.97 | 6481098.29 | 1044.31 | 215.2 | 39.47 | -45.1 |
| DDH04-31 | 508784.37 | 6481168.53 | 1077.37 | 113.7 | 52.87 | -50.22 |
| DDH04-32 | 508841.5 | 6481206.01 | 1078.79 | 115.8 | 51.72 | -50.37 |
| DDH04-33 | 508948.04 | 6481140.95 | 1048.16 | 135.95 | 43.2 | -48.12 |
| DDH04-34 | 508646.17 | 6481208.47 | 1094.97 | 120.4 | 349.28 | -47.63 |
| DDH04-35 | 508789.37 | 6481280.96 | 1090.38 | 166.4 | 355.8 | -49.3 |
| DDH04-36 | 508722.854 | 6481259.948 | 1098.507 | 120.7 | 354.37 | -50 |
| DDH04-37 | 508722.39 | 6481259.19 | 1098.07 | 117.65 | 176.67 | -49.35 |
| DDH04-38 | 508192.22 | 6481363.82 | 1189.48 | 120.7 | 45.32 | -49.43 |
| DDH04-39 | 507938.03 | 6481297.05 | 1178.66 | 148.2 | 48.98 | -49.33 |
| DDH04-40 | 508546.41 | 6481397.69 | 1145.26 | 118.9 | 48.28 | -53.7 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH04-41 | 509359.38 | 6481391.57 | 1028.07 | 96.32 | 22 | -50 |
| DDH04-42 | 506927.578 | 6482375.041 | 1371.506 | 76.5 | 0 | -90 |
| DDH04-43 | 506826.179 | 6482617.075 | 1401.576 | 158.8 | 0 | -90 |
| DDH04-44 | 506674.127 | 6482662.745 | 1411.06 | 124.1 | 0 | -90 |
| DDH04-45 | 506460.975 | 6482749.722 | 1423.752 | 186.25 | 0 | -90 |
| DDH04-46 | 506468.861 | 6482806.906 | 1436.257 | 145.4 | 0 | -90 |
| DDH04-47 | 506267.563 | 6482843.425 | 1425.672 | 184.4 | 0 | -90 |
| DDH04-48 | 506254.412 | 6483031.55 | 1459.441 | 166.75 | 0 | -90 |
| DDH04-49 | 506254.439 | 6483032.547 | 1459.557 | 75.3 | 180 | -50 |
| DDH04-50 | 506254.229 | 6483030.535 | 1459.82 | 169.15 | 0 | -60 |
| DDH04-51 | 506284.205 | 6483023.234 | 1462.76 | 153.9 | 0 | -60 |
| DDH04-52 | 506073.263 | 6483173.319 | 1456.806 | 112.15 | 0 | -90 |
| DDH04-53 | 506073.364 | 6483174.533 | 1456.413 | 114.3 | 0 | -50 |
| DDH04-54 | 506100.176 | 6483128.014 | 1453.201 | 149.95 | 180 | -50 |
| DDH04-55 | 506280.841 | 6483021.702 | 1462.377 | 60.05 | 0 | -90 |
| DDH04-56 | 506084.006 | 6482880.609 | 1414.473 | 100 | 0 | -90 |
| DDH04-57 | 506263.32 | 6482932.592 | 1440.86 | 111.85 | 0 | -90 |
| DDH04-58 | 506010.275 | 6483207.197 | 1454.881 | 111.25 | 35 | -80 |
| DDH04-59 | 506006.854 | 6483289.47 | 1462.932 | 110.95 | 0 | -80 |
| DDH04-60 | 507822.68 | 6481676.69 | 1252.38 | 201.8 | 44.28 | -48.67 |
| DDH04-61 | 507816.43 | 6481669.74 | 1252.84 | 114.3 | 226.6 | -47.02 |
| DDH04-62 | 507817.31 | 6481670.61 | 1252.68 | 67.95 | 231.8 | -85.88 |
| DDH04-63 | 508398.42 | 6481542.92 | 1181.64 | 181.35 | 43.67 | -47.88 |
| DDH04-64 | 508444.72 | 6481587.21 | 1175.77 | 138.7 | 43.57 | -47.27 |
| DDH04-65 | 508363.16 | 6481504.35 | 1185.19 | 135.95 | 41.75 | -43.45 |
| DDH04-66 | 508684.02 | 6481176.52 | 1087.79 | 179.2 | 7.63 | -48.08 |
| DDH04-67 | 508686.65 | 6481232.12 | 1099.81 | 145.1 | 1.9 | -48.82 |
| DDH04-68 | 508787.18 | 6481096.41 | 1062.72 | 90.2 | 45.05 | -50.22 |
| DDH04-69 | 509003.72 | 6481081.96 | 1025.14 | 196.9 | 45.53 | -50.2 |
| DDH04-70 | 509275 | 6481395 | 1043 | 132.9 | 22 | -50 |
| DDH04-71 | 508724.67 | 6481123.5 | 1073.92 | 221.3 | 48.18 | -47.98 |
| DDH05-100 | 508276.8 | 6481207.62 | 1153.87 | 190.8 | 180.16 | -49.97 |
| DDH05-101 | 505395.456 | 6482884.279 | 1336.749 | 184.7 | 40.83 | -65 |
| DDH05-102 | 506345.951 | 6482434.504 | 1366.166 | 337.4 | 219.3 | -51.4 |
| DDH05-103 | 508192.1 | 6481210.66 | 1163.27 | 400.8 | 180.67 | -49.2 |
| DDH05-104 | 508181.06 | 6481277.02 | 1178.39 | 233.2 | 175.5 | -46.83 |
| DDH05-105 | 508281.93 | 6481280.63 | 1174.16 | 288.1 | 189.41 | -47.7 |
| DDH05-106 | 508648.65 | 6481259.3 | 1110.69 | 257.25 | 184.72 | -49.53 |
| DDH05-107 | 508519.47 | 6481219.64 | 1112.16 | 199.35 | 179.5 | -47.3 |
| DDH05-108 | 508962.95 | 6481146.32 | 1047.28 | 217.65 | 192.5 | -50.18 |
| DDH05-72 | 508721.87 | 6481143.62 | 1076.71 | 186.4 | 181 | -63.42 |
| DDH05-73 | 508288.23 | 6480976.89 | 1100.23 | 152.4 | 218.63 | -48.33 |
| DDH05-74 | 508829.11 | 6481127.35 | 1063.48 | 223.7 | 171.35 | -48.18 |
| DDH05-75 | 508938.42 | 6481338.89 | 1087.44 | 217.7 | 48.82 | -49.62 |
| DDH05-76 | 508513.13 | 6481312.23 | 1128.21 | 223.7 | 183.87 | -47.63 |
| DDH05-77 | 508382.7 | 6481333.23 | 1162.41 | 223.7 | 182.43 | -47.65 |
| DDH05-78 | 508099.23 | 6481147.15 | 1148.45 | 147.5 | 220 | -49.12 |
| DDH05-79 | 507989.59 | 6481312.85 | 1180.23 | 211.3 | 228.73 | -50.5 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH05-80 | 507741.69 | 6481296.02 | 1192.71 | 199.4 | 32.08 | -50.22 |
| DDH05-81 | 507213.487 | 6481770.476 | 1284.038 | 181.4 | 44.91 | -50 |
| DDH05-82 | 507213.487 | 6481770.476 | 1284.038 | 62.2 | 224.91 | -50 |
| DDH05-83 | 506528.861 | 6482326.024 | 1352.83 | 172 | 224.5 | -52 |
| DDH05-84 | 505995.658 | 6482358.196 | 1350.069 | 189.6 | 46.68 | -52 |
| DDH05-85 | 507741.07 | 6484040.09 | 1676.198 | 140.2 | 232.21 | -52.5 |
| DDH05-86 | 507508.732 | 6484239.893 | 1656.923 | 129 | 225.51 | -53.25 |
| DDH05-87 | 506938.064 | 6484389.522 | 1665.308 | 143.25 | 48.44 | -49.75 |
| DDH05-88 | 505395.456 | 6482884.273 | 1336.379 | 172.2 | 40.55 | -50.5 |
| DDH05-89 | 505925.463 | 6482544.446 | 1356.583 | 166.1 | 40.77 | -50.75 |
| DDH05-90 | 508363.05 | 6481504.93 | 1185.44 | 193.55 | 224.25 | -49.37 |
| DDH05-91 | 509023.24 | 6481351.37 | 1081.62 | 217.95 | 49.67 | -49.57 |
| DDH05-92 | 508995.35 | 6481385.63 | 1082.8 | 205.75 | 51.23 | -47.48 |
| DDH05-93 | 508910.18 | 6481099.06 | 1044.86 | 202.7 | 182 | -51.13 |
| DDH05-94 | 508910.17 | 6481099.84 | 1044.78 | 245.7 | 191.92 | -80 |
| DDH05-95 | 508763.68 | 6481087.3 | 1062.85 | 240.5 | 182.5 | -50 |
| DDH05-96 | 508659.52 | 6481150.19 | 1088.31 | 211.85 | 174.75 | -49 |
| DDH05-97 | 508382.48 | 6481141.45 | 1136.57 | 185.3 | 180.58 | -49.2 |
| DDH05-98 | 508383.52 | 6481207.3 | 1143.95 | 199.65 | 187 | -48.25 |
| DDH05-99 | 508383.77 | 6481171.7 | 1140.03 | 187.2 | 183.17 | -49.6 |
| DDH06-109 | 509301.8 | 6481495.9 | 1049.7 | 324.9 | 180 | -84.06 |
| DDH06-110 | 509117.8 | 6481501.7 | 1078.3 | 285 | 180 | -84 |
| DDH06-111 | 508897.7 | 6481503.7 | 1115.6 | 297.2 | 180 | -84.07 |
| DDH06-112 | 508596.4 | 6481697.9 | 1175.1 | 260.9 | 180 | -85.2 |
| DDH06-113 | 508095.2 | 6481307.3 | 1176.6 | 276.15 | 195.35 | -50.14 |
| DDH06-114 | 508109.6 | 6481408.4 | 1197.4 | 193.55 | 193.23 | -49.42 |
| DDH06-115 | 508281.4 | 6481401 | 1190.3 | 243 | 168.53 | -50.37 |
| DDH06-116 | 508195.3 | 6481491.6 | 1202.5 | 202.7 | 183.88 | -48.16 |
| DDH06-117 | 508204.4 | 6481699.7 | 1224.3 | 202.7 | 176.14 | -47.57 |
| DDH06-118 | 508205.1 | 6481698.7 | 1224.3 | 217.95 | 358.7 | -50.2 |
| DDH06-119 | 508506.3 | 6480807.9 | 1041.3 | 187.45 | 180.77 | -48.1 |
| DDH06-120 | 508011.5 | 6481697.8 | 1239.1 | 175.25 | 179.49 | -50.11 |
| DDH06-121 | 507504 | 6481907.6 | 1304.8 | 297.7 | 183.78 | -49.81 |
| DDH06-122 | 507694.3 | 6481870.7 | 1306.4 | 301 | 177.1 | -49.9 |
| DDH06-123 | 508489.7 | 6480922.8 | 1065.8 | 265.2 | 177.78 | -48.43 |
| DDH06-124 | 507606.1 | 6482000.1 | 1313.5 | 230.15 | 181.54 | -48.92 |
| DDH06-125 | 507812.4 | 6482013.8 | 1312.6 | 381.9 | 180.39 | -50.03 |
| DDH06-126 | 507593.5 | 6481612.7 | 1259.1 | 501.4 | 178.23 | -48.74 |
| DDH06-127 | 507476.5 | 6481712.7 | 1276.8 | 367.3 | 180.49 | -49.97 |
| DDH06-128 | 507695.6 | 6481544.7 | 1242.8 | 92.95 | 176.9 | -47.8 |
| DDH06-128A | 507695.6 | 6481544.6 | 1242.1 | 297.2 | 177.78 | -58.8 |
| DDH06-129 | 509474.651 | 6482238.889 | 1094.762 | 227.1 | 40.33 | -60.7 |
| DDH06-130 | 509475.448 | 6482237.402 | 1096.448 | 214.9 | 85.73 | -49.9 |
| DDH06-131 | 509448.9 | 6481848 | 1093 | 388.6 | 39.78 | -50.47 |
| DDH06-132 | 509504.4 | 6481915.1 | 1088.6 | 266.85 | 39.31 | -49.93 |
| DDH06-133 | 509507.6 | 6481917 | 1088.7 | 208.8 | 156.76 | -49.16 |
| DDH06-134 | 509407.4 | 6481921.6 | 1101.5 | 343.5 | 34.4 | -85.78 |
| DDH06-135 | 507635.672 | 6484060.985 | 1676.879 | 197.2 | 221.22 | -49.9 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH06-136 | 507654.31 | 6484263.878 | 1652.724 | 196.9 | 43.06 | -48.6 |
| DDH06-137 | 507811.303 | 6484165.356 | 1647.189 | 219.74 | 38.2 | -48.5 |
| DDH06-138 | 507853.055 | 6484049.305 | 1651.67 | 224.65 | 217.76 | -48.2 |
| DDH06-139 | 508009.612 | 6483910.46 | 1607.373 | 212.45 | 220 | -48.5 |
| DDH06-140 | 508097.717 | 6483845.603 | 1573.11 | 233.65 | 219.99 | -48.9 |
| DDH06-141 | 504420.795 | 6483501.092 | 1276.527 | 255.1 | 30.55 | -50 |
| DDH06-142 | 504419.585 | 6483496.857 | 1275.62 | 245.95 | 210.55 | -50 |
| DDH06-143 | 504769.638 | 6483975.363 | 1402.336 | 340.45 | 31.98 | -50 |
| DDH06-144 | 507511.9 | 6482223.7 | 1343.7 | 194.4 | 40 | -83.9 |
| DDH06-145 | 507668.8 | 6482297 | 1355.3 | 161.6 | 40 | -85.2 |
| DDH06-146 | 506738.64 | 6482253.57 | 1341.3 | 251.45 | 179.67 | -49.3 |
| DDH06-147 | 505937.98 | 6483466.01 | 1475.38 | 377.05 | 40 | -50.9 |
| DDH06-148 | 506101.49 | 6483388.7 | 1483.54 | 139.3 | 40 | -50 |
| DDH06-149 | 505911.96 | 6483180.07 | 1435.8 | 315.45 | 41.88 | -49.5 |
| DDH06-150 | 506032.27 | 6483009.63 | 1428.63 | 257.6 | 44.7 | -49.8 |
| DDH06-151 | 509493.5 | 6480658.8 | 1058.7 | 292.55 | 166.63 | -50.19 |
| DDH06-152 | 509492.2 | 6480661.7 | 1058.8 | 324.6 | 347.79 | -50.27 |
| DDH06-153 | 509492.3 | 6480661 | 1058.6 | 233.8 | 347.79 | -85.68 |
| DDH06-154 | 509193 | 6480544.3 | 1070.9 | 298.1 | 190.54 | -63.53 |
| DDH06-155 | 509543 | 6480907.8 | 1040.7 | 303.9 | 173.95 | -50.2 |
| DDH06-156 | 509884.5 | 6480901.4 | 1081.4 | 298.1 | 178 | -61.5 |
| DDH06-157 | 509677.4 | 6480873.1 | 1059.1 | 260.6 | 178.15 | -63.76 |
| DDH06-158 | 509023.1 | 6480453.7 | 1037 | 161.35 | 175.6 | -57.3 |
| DDH06-159 | 509186.9 | 6480661.9 | 1040.9 | 330.9 | 172 | -58.6 |
| DDH06-160 | 505442.82 | 6482934.05 | 1349.18 | 309.4 | 41.49 | -49.1 |
| DDH06-161 | 505431.81 | 6482833.59 | 1338.16 | 285 | 41.41 | -48.5 |
| DDH06-162 | 507395.1 | 6481677.5 | 1267.2 | 358.15 | 178.45 | -50.47 |
| DDH06-163 | 507497 | 6481599.8 | 1255.2 | 339.85 | 187.67 | -49.46 |
| DDH06-164 | 507596.8 | 6481501.8 | 1235.2 | 349 | 178.72 | -48.83 |
| DDH06-165 | 507801.4 | 6481529.4 | 1232.8 | 379.45 | 164.16 | -50.01 |
| DDH06-166 | 507412.8 | 6481908.2 | 1300.7 | 305.7 | 176.63 | -49.17 |
| DDH06-167 | 507699.7 | 6481768.5 | 1277.9 | 352.65 | 175.54 | -47.93 |
| DDH06-168 | 509326.3 | 6481829.4 | 1090 | 520 | 184.42 | -84.58 |
| DDH06-169 | 509288.1 | 6481988 | 1118.6 | 199.95 | 177.7 | -85.55 |
| DDH06-170 | 509202.1 | 6481395.4 | 1052.1 | 434.5 | 175.56 | -51.5 |
| DDH06-171 | 507583.8 | 6481695.2 | 1269.4 | 333.75 | 175.36 | -50.02 |
| DDH06-172 | 509100.8 | 6481308.1 | 1052.1 | 409.95 | 179.04 | -50.31 |
| DDH06-173 | 509292 | 6480624.9 | 1049.6 | 330.7 | 183.3 | -59.01 |
| DDH06-174 | 509398 | 6480688.4 | 1045.3 | 349 | 173.4 | -58.2 |
| DDH06-175 | 509003.1 | 6480663.9 | 1020 | 337.4 | 181.5 | -62 |
| DDH07-176 | 508837.2 | 6481189.1 | 1076.7 | 337.1 | 175.1 | -47.8 |
| DDH07-177 | 508513.7 | 6481311 | 1128.3 | 361.8 | 353.6 | -59.3 |
| DDH07-178 | 508888.8 | 6481280.3 | 1083.6 | 483.1 | 180.3 | -57.6 |
| DDH07-179 | 508518.3 | 6481389.8 | 1145.2 | 423 | 179.1 | -49.8 |
| DDH07-179a | 508519.448 | 6481394.787 | 1158.876 | 0 | 180.8 | -60 |
| DDH07-180 | 509022.6 | 6481354 | 1081.5 | 419.1 | 181.4 | -48.4 |
| DDH07-181 | 508634.2 | 6481354.8 | 1125.1 | 394.7 | 178.9 | -48.6 |
| DDH07-182 | 509299.6 | 6481374 | 1031 | 336.5 | 182.2 | -50.7 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH07-183 | 509200.9 | 6481201.5 | 1018.8 | 365.45 | 170.1 | -60.4 |
| DDH07-184 | 508362.3 | 6481506.9 | 1185.2 | 537.95 | 178.4 | -48.3 |
| DDH07-185 | 509005.1 | 6480939.7 | 1011.7 | 468.8 | 175.7 | -50.7 |
| DDH07-186 | 508072.8 | 6481413.2 | 1196.5 | 366.05 | 180.7 | -50.5 |
| DDH07-187 | 509202.1 | 6480998 | 1008 | 492.55 | 180.4 | -58.8 |
| DDH07-188 | 508194.909 | 6481098.062 | 1148.764 | 0 | 179.45 | -50 |
| DDH07-189 | 509500.3 | 6481557 | 1029.6 | 379.5 | 176.6 | -58.5 |
| DDH07-190 | 508077 | 6481596.8 | 1222.8 | 363.95 | 174.3 | -48.2 |
| DDH07-191 | 509498.1 | 6481455.1 | 1026.9 | 420.3 | 175.1 | -59.3 |
| DDH07-192 | 507798.5 | 6481620.5 | 1249.7 | 400.8 | 178.3 | -49 |
| DDH07-193 | 509702.698 | 6481499.295 | 1022.007 | 0 | 177.33 | -60 |
| DDH07-194 | 509489.2 | 6481953.4 | 1092.9 | 382.85 | 38.6 | -49.7 |
| DDH07-195 | 507707.6 | 6481640.6 | 1260.1 | 458.7 | 177.8 | -54.8 |
| DDH07-196 | 507613.3 | 6481406.9 | 1220.5 | 218.85 | 180.2 | -49.3 |
| DDH07-197 | 507397.6 | 6481573 | 1256.3 | 240.2 | 177.6 | -47.4 |
| DDH07-198 | 507302.8 | 6481997.3 | 1312.3 | 166.1 | 177.3 | -47.4 |
| DDH07-199 | 507509.3 | 6482006.9 | 1315.4 | 366.65 | 179 | -47.7 |
| DDH07-200 | 509500.1 | 6481767.6 | 1066.5 | 312.75 | 179.6 | -59.4 |
| DDH07-201 | 508735.6 | 6481504.2 | 1132.2 | 666.6 | 185.9 | -67.2 |
| DDH07-202 | 507405.1 | 6481798.8 | 1283.9 | 341.7 | 172.4 | -48.6 |
| DDH07-203 | 509707.6 | 6481672.6 | 1025.8 | 453.25 | 177.6 | -59.9 |
| DDH07-204 | 507484.2 | 6481813 | 1287.9 | 499.25 | 180.6 | -48.4 |
| DDH07-205 | 507601.5 | 6481795.6 | 1285.1 | 438.4 | 180 | -50.4 |
| DDH07-206 | 509974.136 | 6481695.047 | 1017.406 | 358.75 | 69.83 | -57.4 |
| DDH07-207 | 505919.144 | 6482546.345 | 1357.219 | 311.8 | 189.34 | -49.9 |
| DDH07-208 | 508945.321 | 6481308.01 | 1083.107 | 58.5 | 180 | -75 |
| DDH07-209 | 505740.869 | 6482652.644 | 1354.576 | 394.1 | 40 | -49.8 |
| DDH07-210 | 505337.937 | 6482829.584 | 1323.158 | 413 | 45.75 | -50.1 |
| DDH07-211 | 505337.416 | 6482829.353 | 1323.364 | 276.15 | 45.75 | -68.3 |
| DDH07-212 | 508671.1 | 6481246.8 | 1105.4 | 137.15 | 177.3 | -70.9 |
| DDH07-213 | 508865.518 | 6481203.695 | 1079.977 | 56.7 | 168.3 | -60 |
| DDH07-214 | 505331.921 | 6482901.219 | 1325.874 | 348.7 | 44.69 | -49.1 |
| DDH07-215 | 508835.8 | 6481265.9 | 1084.1 | 413.3 | 180.1 | -50.1 |
| DDH07-216 | 506607.846 | 6483119.019 | 1506.123 | 245.05 | 180 | -49 |
| DDH07-217 | 507955.3 | 6482011.7 | 1295.3 | 403.55 | 178.6 | -49.4 |
| DDH07-218 | 508605 | 6481119.6 | 1088.8 | 400.5 | 181.1 | -44.8 |
| DDH07-219 | 507808.2 | 6481894 | 1313.3 | 424.9 | 180.6 | -48.6 |
| DDH07-220 | 508787.111 | 6481293.113 | 1091.48 | 119.2 | 180 | -59.3 |
| DDH07-221 | 508787.4 | 6481293 | 1092.1 | 410.85 | 177 | -50.7 |
| DDH07-222 | 508450.5 | 6481429.5 | 1154 | 412.4 | 179.5 | -50.4 |
| DDH07-223 | 507706.6 | 6481932.8 | 1311.1 | 403.55 | 177.8 | -49.4 |
| DDH07-224 | 508832 | 6481487 | 1121.7 | 42.65 | 180.54 | -85 |
| DDH07-225 | 508832 | 6481486.3 | 1122 | 675.15 | 178.8 | -58.8 |
| DDH07-226 | 507303 | 6481691 | 1267.6 | 330.1 | 180.1 | -48.7 |
| DDH07-227 | 508597.8 | 6481283.9 | 1117.5 | 358.75 | 184.1 | -48.8 |
| DDH07-228 | 507696.4 | 6481465.5 | 1225.9 | 330.1 | 177.7 | -46.4 |
| DDH07-229 | 508602.3 | 6481177.3 | 1095 | 266.4 | 182.8 | -50.1 |
| DDH07-230 | 505618 | 6484204 | 1630 | 239.6 | 180 | -60 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH07-231 | 505905.246 | 6484236.244 | 1633.723 | 245.05 | 180 | -60 |
| DDH07-232 | 506369 | 6483714 | 1586.92 | 223.4 | 180 | -60 |
| DDH07-233 | 507376 | 6483374 | 1575 | 224.95 | 225 | -60 |
| DDH07-234 | 506765 | 6483656 | 1627.7 | 246.3 | 180 | -70 |
| DDH07-235 | 507376 | 6483374 | 1575 | 207.5 | 45 | -60 |
| DDH07-236 | 511185 | 6481355 | 1170 | 230.85 | 180 | -85 |
| DDH07-237 | 510889 | 6481643 | 1125.23 | 305.1 | 180 | -58.8 |
| DDH07-238 | 510889 | 6481643 | 1125.23 | 26.5 | 180 | -70 |
| DDH07-239 | 510889 | 6481643 | 1125.23 | 167.05 | 180 | -85 |
| DDH07-240 | 510337 | 6481587 | 1050 | 308.45 | 180 | -60 |
| DDH07-241 | 507806.4 | 6481783.2 | 1284.8 | 398.35 | 174.7 | -48.6 |
| DDH07-242 | 508391.1 | 6481051.4 | 1101.9 | 300.85 | 177.8 | -48.6 |
| DDH07-243 | 508942.8 | 6481698.2 | 1147.6 | 416.05 | 355.2 | -49.8 |
| DDH07-244 | 508455.4 | 6481281.9 | 1137.7 | 355.1 | 179.6 | -47.9 |
| DDH07-245 | 508645 | 6481071.2 | 1078.2 | 286.4 | 184.4 | -45.8 |
| DDH07-246 | 509042.2 | 6481284.8 | 1062.9 | 351.45 | 181.8 | -52.1 |
| DDH07-247 | 508895.2 | 6481509.6 | 1116.4 | 355.1 | 359.2 | -49.9 |
| DDH07-248 | 509036.9 | 6481211.6 | 1047.9 | 352.65 | 182.3 | -48.4 |
| DDH08-249 | 511004.622 | 6481593.399 | 1149.571 | 370.35 | 180 | -50 |
| DDH08-250 | 510798.399 | 6481718.991 | 1130.682 | 181.95 | 181.39 | -50 |
| DDH08-251 | 510337.404 | 6481745.664 | 1041.766 | 0 | 179.93 | -85 |
| DDH08-252 | 511298.952 | 6481511.305 | 1165.016 | 254.8 | 178.63 | -50 |
| DDH08-253 | 510907.369 | 6481451.983 | 1137.353 | 273.7 | 180.48 | -50 |
| DDH08-254 | 508860.4 | 6481419.5 | 1103.7 | 318.5 | 181.5 | -47.9 |
| DDH08-255 | 508797.2 | 6481439.3 | 1114.7 | 358.15 | 181.4 | -50.7 |
| DDH08-256 | 508745.6 | 6481424.7 | 1122.8 | 327.65 | 180.9 | -49.7 |
| DDH08-257 | 508901.4 | 6481435.9 | 1103.2 | 273.4 | 181.3 | -49.5 |
| DDH08-258 | 508959.7 | 6481431.4 | 1092 | 275.85 | 180.5 | -51.6 |
| DDH08-259 | 508941.6 | 6481309.7 | 1082.6 | 300.85 | 180.9 | -47.3 |
| DDH08-260 | 508745.5 | 6481322.5 | 1104.9 | 333.75 | 178.6 | -50.2 |
| DDH08-261 | 508595.342 | 6481381.69 | 1130.513 | 419.1 | 179.7 | -50.7 |
| DDH08-262 | 507844.9 | 6479426.5 | 1018.9 | 20.11 | 177.18 | -90 |
| DDH08-263 | 509128.7 | 6481155.4 | 1022.6 | 152.1 | 179.4 | -89 |
| DDH08-264 | 508876 | 6481060.9 | 1038.5 | 245.05 | 358.3 | -4.5 |
| DDH10-265 | 508877 | 6481060.9 | 1038.5 | 179.85 | 358.3 | -4.5 |
| DDH10-266 | 508875 | 6481060.9 | 1038.5 | 204.2 | 358.3 | -4.5 |
| DDH18-267 | 508192.648 | 6481886.288 | 1237.4835 | 126.19 | 180.3 | -50 |
| DDH18-268 | 508065.329 | 6481766.371 | 1242.2328 | 449.89 | 179.8 | -50 |
| DDH18-269 | 508202.257 | 6481330.308 | 1186.8103 | 221.59 | 173.6 | -75 |
| DDH18-270 | 507801.569 | 6481177.718 | 1169.5199 | 251.76 | 359.9 | -50 |
| DDH18-271 | 508158.435 | 6481561.245 | 1211.8988 | 374.6 | 179.8 | -80 |
| DDH18-272 | 508100.873 | 6481485.62 | 1208.5846 | 289.26 | 0.2 | -80 |
| DDH18-273 | 507948.828 | 6481166.556 | 1157.8566 | 102.41 | 181.2 | -60 |
| DDH18-274 | 507952.732 | 6481504.278 | 1214.1073 | 317.6 | 179.7 | -55 |
| DDH18-275 | 507704.48 | 6481306.77 | 1200 | 71.6 | 180 | -50 |
| DDH18-276 | 507701.722 | 6481364.622 | 1211.3117 | 404.47 | 359.7 | -55 |
| DDH18-277 | 508523.514 | 6481775.278 | 1189.9196 | 99.67 | 179.9 | -50 |
| DDH18-278 | 508461.379 | 6481769.414 | 1192.5322 | 99.97 | 179.5 | -60 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH18-279 | 508645.637 | 6481672.432 | 1173.7759 | 151.49 | 179.8 | -50 |
| DDH18-280 | 508736.498 | 6481724.672 | 1174.8224 | 151.49 | 179.8 | -50 |
| DDH18-281 | 507951.011 | 6482125.106 | 1308.5202 | 200.25 | 179.8 | -50 |
| DDH18-282 | 507798.175 | 6481378.918 | 1207.8642 | 468.17 | 0.3 | -60 |
| DDH18-283 | 507814.078 | 6482107.939 | 1314.8282 | 224.64 | 180.2 | -50 |
| DDH18-284 | 507714.718 | 6482098.915 | 1320.231 | 215.49 | 180 | -50 |
| DDH18-285 | 508064.361 | 6481926.143 | 1272.972 | 450.19 | 180.2 | -60 |
| DDH18-286 | 508380.249 | 6481723.453 | 1193.138 | 333.76 | 179.8 | -50 |
| DDH18-287 | 507942.058 | 6481837.576 | 1284.8462 | 599 | 180 | -50 |
| DDH18-288 | 504245 | 6486190 | 1568 | 319.13 | 0 | -90 |
| DDH18-289 | 507502.35 | 6481398.932 | 1226.1087 | 114.91 | 180.2 | -50 |
| DDH18-290 | 507489.694 | 6481497.544 | 1238.8894 | 148.44 | 179.7 | -50 |
| DDH18-291 | 502950 | 6484800 | 1295 | 455.98 | 40 | -67 |
| DDH18-292 | 508889.996 | 6481285.312 | 1084.0467 | 249.02 | 180.3 | -67 |
| DDH18-293 | 508862.419 | 6481111.87 | 1058.752 | 196.6 | 19.9 | -70 |
| DDH18-294 | 508722.665 | 6481201.768 | 1087.9228 | 197.51 | 359.9 | -60 |
| DDH18-295 | 508683.485 | 6481228.277 | 1100.8466 | 236.83 | 30.4 | -60 |
| DDH18-296 | 508646.427 | 6481204.37 | 1095.6497 | 188.67 | 0.2 | -83 |
| DDH18-297 | 508562.847 | 6481209.363 | 1105.4211 | 185.62 | 50.2 | -80 |
| DDH18-298 | 508380.428 | 6481141.002 | 1136.7752 | 252.07 | 0.3 | -77 |
| DDH18-299 | 508604.724 | 6481117.454 | 1089.6392 | 322.78 | 0.2 | -85 |
| DDH18-300 | 508948.001 | 6481139.352 | 1048.5672 | 160.63 | 42.8 | -60 |
| DDH18-301 | 505568.093 | 6483040.201 | 1375.1747 | 584.3 | 223.8 | -55 |
| DDH18-302 | 508872.745 | 6481060.6 | 1038.1867 | 391.06 | 329.9 | -4 |
| DDH18-303 | 505575.687 | 6483039.499 | 1375.2562 | 535.53 | 313.6 | -60 |
| DDH18-304 | 507773.807 | 6481776.752 | 1285.2929 | 242.93 | 179.7 | -55 |
| DDH18-305 | 507694.657 | 6481469.64 | 1225.7932 | 249.33 | 180 | -85 |
| DDH18-306 | 507652.891 | 6481765.92 | 1277.736 | 199.95 | 180.3 | -75 |
| DDH67-01 | 509065 | 6481052 | 1015 | 152.4 | 25 | -35 |
| DDH67-02 | 509065 | 6481052 | 1015 | 124.05 | 25 | -60 |
| DDH67-03 | 509065 | 6481052 | 1015 | 123.44 | 205 | -35 |
| DDH67-04 | 509084 | 6481112 | 1021.51 | 14.33 | 205 | -40 |
| DDH67-05 | 509084 | 6481112 | 1021.51 | 21.03 | 205 | -55 |
| DDH67-06 | 509084 | 6481112 | 1021.51 | 114.3 | 25 | -35 |
| DDH67-07 | 508948 | 6481203 | 1063.25 | 156.97 | 25 | -35 |
| DDH67-08 | 508948 | 6481203 | 1063.25 | 136.7 | 25 | -60 |
| DDH67-09 | 508798 | 6481220 | 1085 | 154.53 | 25 | -35 |
| DDH67-10 | 508798 | 6481220 | 1085 | 152.4 | 25 | -60 |
| DDH67-11 | 508774 | 6481253 | 1090 | 110.64 | 25 | -40 |
| DDH67-12 | 508774 | 6481253 | 1090 | 38.4 | 25 | -35 |
| DDH67-13 | 508774 | 6481253 | 1090 | 10.67 | 295 | -35 |
| DDH70-14 | 508828 | 6480514 | 1015 | 139 | 25 | -40 |
| DDH70-15 | 508828 | 6480514 | 1015 | 87.8 | 215 | -60 |
| DDH70-16 | 508450 | 6480570 | 1032.01 | 122.83 | 19 | -40 |
| DDH70-17 | 508425 | 6480905 | 1074.13 | 123.44 | 212 | -60 |
| DDH70-18 | 508425 | 6480905 | 1074.13 | 15.84 | 212 | -30 |
| DDH70-19 | 508627 | 6480881 | 1035 | 118.87 | 34 | -40 |
| DDH70-20 | 508649 | 6480915 | 1039.26 | 106.1 | 22 | -41 |

| Hole | UTM East | UTM North | UTM Elevation | Length (m) | Azimuth (°) | Dip (°) |
|---|---|---|---|---|---|---|
| DDH70-21 | 507918 | 6481049 | 1132.92 | 201.5 | 25 | -40 |
| DDH70-22 | 507738 | 6481169 | 1176.12 | 109.73 | 25 | -40 |
| DDH70-23 | 507297 | 6481553 | 1250 | 122.83 | 25 | -40 |
| DDH70-24 | 507646 | 6481655 | 1260 | 60.65 | 25 | -34 |
| DDH70-25 | 507646 | 6481655 | 1260 | 16.2 | 25 | -65 |
| DDH70-26 | 507683 | 6481700 | 1270 | 77.1 | 25 | -38 |
| DDH70-27 | 507705 | 6481692 | 1270 | 61.9 | 25 | -38 |
| DDH70-28 | 507550 | 6481700 | 1270.66 | 93.3 | 25 | -38 |
| DDH96-01 | 509375 | 6481308 | 1010.61 | 184.4 | 22 | -45 |
| DDH96-02 | 508638 | 6480652 | 1016.25 | 178.6 | 290 | -60 |
| DDH96-03 | 508889 | 6480528 | 1019.42 | 137.5 | 20 | -60 |
| DDH96-04 | 508889 | 6480528 | 1019.42 | 137.5 | 200 | -60 |
| DDH96-05 | 508889 | 6480528 | 1019.42 | 154.6 | 290 | -60 |
| DDH97-01 | 511509 | 6481520 | 1210 | 160 | 45 | -60 |
| DDH97-02 | 507098 | 6484190 | 1690 | 190.5 | 0 | -60 |
| DDH97-03 | 507094 | 6484219 | 1685 | 133.2 | 0 | -50 |
| DDH97-04 | 507694 | 6481716 | 1270 | 163.7 | 210 | -50 |
| DDH97-05 | 507694 | 6481716 | 1270 | 130.1 | 210 | -65 |
| DDH97-06 | 510022 | 6481718 | 1014.26 | 197.2 | 45 | -65 |
| DDH97-07 | 510132 | 6481743 | 1005.71 | 166.7 | 5 | -60 |
| DDH97-08 | 508657 | 6480699 | 1015.52 | 220.7 | 290 | -60 |
| DDH97-09 | 509094.74 | 6481042.52 | 1010.26 | 493.2 | 345.02 | -45.07 |
| DDH98-01 | 508791.86 | 6481213.83 | 1083.73 | 288 | 340.7 | -57.07 |
| DDH98-02 | 508793.1 | 6481215 | 1085 | 184.7 | 170 | -56 |
| DDH98-03 | 508887.1 | 6481305 | 1085 | 203 | 355 | -56 |
| DDH98-04 | 508965.06 | 6481154.48 | 1047.27 | 292.6 | 344.63 | -56.43 |
| DDH98-05 | 508673.34 | 6480928.77 | 1036.31 | 295.7 | 324.75 | -57.83 |

## 10.1 Collar Surveying

Giga Metals planned drill holes in advance and then spotted the collar in the field using a Differential Global Positioning System (DGPS). Because of the high magnetic background resulting in magnetic compass inaccuracy, Giga Metals set the direction of drilling using either foresights and backsights spotted by staff using a DGPS, or using a north-seeking Reflex TN14 GyroCompass. After completion of the hole, the collar location was resurveyed using a DGPS; the collar dip measurement was taken from the down-hole survey measurement nearest to the top of the hole. Most casings remain intact with semi-permanent markers. Many pre-2018 collars have had their location, azimuth, and dip 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 from 2004 to 2010. 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 and Northwest zones, 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. A Reflex EZ Gyro was used to survey drill hole deviations during the 2018 drill season.

# 11.0 SAMPLE PREPARATION, ANALYSES & SECURITY

This section provides an overview of the sample preparation, analysis, and security procedures used by Giga Metals. Where available, similar information is also provided for Giga Metals' predecessor companies.

Sample preparation and analysis programs have been undertaken by a variety of operators during various drill campaigns. This section summarises the verification work and practices employed by each of the operators. The independent qualified person (QP) responsible for Section 11 of this report, Garth Kirkham, P. Geo., believes that the sample collection, preparation, analysis and security procedures for all Giga Metals' drilling are consistent with industry standards and best practices. This supports their use in mineral resource and mineral reserve estimation as detailed in this study.

## 11.1 Security & Chain of Custody

The drill contractor transported drill core from the drill site to the exploration camp for processing. Split core samples were numbered, bagged and sealed, and transported from site by helicopter or fixed-wing airplane to Dease Lake (or similarly by plane to Smithers) in 300 to 350 kg lots. The samples were then shipped by commercial transport to a primary preparation facility as follows:

- Acme Laboratories in Vancouver (2003 to 2005) or Smithers (2006 to 2010)
- ALS in Terrace (2018).

Requisition forms were transmitted to the Giga Metals Vancouver office with the date and number of samples shipped, and Acme notified Giga Metals upon receipt of the 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 the 2003 to 2018 programs is stored in sturdy racks, stacked either in neat rows or cross-stacked near the camp on the Turnagain Property. Sample security and core storage conform to industry standards.

## 11.2 Sampling Methods

Since 1967, multiple drilling and sampling methods have been used by prior operators and Giga Metals and its predecessors. Sampling and logging methods, as well as quality control measures, have varied over this time.

### 11.2.1 Geotechnical Data

In 2007, Giga Metals contracted Piteau and Associates Engineering Ltd. (Piteau) to provide geotechnical core logging guidance. Piteau provided Giga Metals' geologists with instructions for recording core rock quality designation (RQD), recovery, joint frequency, joint condition, fracture density and orientation, hardness, and weathering. In addition, Giga Metals' geologists collected over 7,000 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 Piésold Consulting. Geotechnical logging between 2002 and 2004 included RQD and recovery only.

### 11.2.2 Geological Data

In 2006, Giga Metals 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 realigned, and driller block measurements are converted to metres. Drill core was sampled at 2 m intervals or less during the 2004 and 2005 programs and predominantly on 4 m intervals since 2006, although sample breaks are often inserted at significant changes in lithology or mineralisation. Following core logging, sample intervals are marked with a red or yellow marker and sample numbers are assigned from a pre-printed analytical laboratories sample tag book. Core is digitally photographed three boxes at a time on the logging rack in the core shack. Core samples are halved by a hydraulic core splitter and/or diamond saw. In most cases, half the core is stored in boxes on site and half is sent for analysis. In the case of 2018; however, 13 core samples from 40 holes were halved, and then one half was again split into quarters. One quarter was sent for analysis; one quarter was stored in the box; and the remaining half of the core was stored for future metallurgical testing.

### 11.2.3 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 2010 programs, received by Acme (now Bureau Veritas) in Smithers and Vancouver, BC, were checked against requisition documents prior to being dried, weighed, crushed, split, and pulverised. They were then subjected to a variety of analytical techniques. Acme is a certified ISO:9000 facility. Drill core samples from the 2018 program were similarly processed almost entirely by ALS Laboratories facilities in Terrace, Kamloops and Vancouver, BC; and two batches by TSL Laboratories in Saskatoon, SK. ALS Laboratories is a certified ISO:17025 facility; however, TSL Laboratories is no longer accredited.

Prior to 2004, samples were analysed 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 2018 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 2004 and 2005, sulphur content was analysed by the Leco furnace method. In 2006, sulphur content was analysed by ICP-ES after a four-acid digestion. Since 2007, sulphur content has been analysed by both ICP-ES after a four-acid digestion, and Leco furnace.

Some exploration drill holes prior to 2004, and all drill holes since 2004, were analysed for platinum, palladium, and gold by lead-collection fire-assay fusion followed by ICP-ES.

### 11.2.4 Density

Giga Metals collects bulk specific gravity measurements by water immersion method every 20 samples, using up to 50 cm of unsplit core. A protocol for specific gravity measurements is posted in the logging tent.

Specific gravity is calculated as follows:

*(SG) specific gravity = weight in air / (weight in air – weight in water)*

Prior to 2018, data were recorded manually on paper and later transferred to a digital file. Data entry errors due to transposition of numbers or poor written records were possible. AMEC (2007) recommended double data entry for any manual entry of data into a database and suggested that Giga Metals create a density standard to use periodically to ensure the scale is working properly. Since 2008, mass standards have been used to calibrate the scale at least once each day and immersion water replaced periodically to ensure accuracy of measurements. In 2018, data were entered directly upon measurement into an Access database.

## 11.3 Quality Assurance / Quality Control

Laboratory quality control since 2004 has been maintained by routinely inserting and analysing internal standards, sample blanks, and duplicate samples. Giga Metals geologists also insert reference sample pulps in the field every 20 samples, and blank samples are inserted every 30 samples. Laboratories are instructed to create and analyse duplicate pulps from crushed core every 30$^{th}$ sample. Pulps from every 10$^{th}$ sample are sent to a check laboratory. In 2018, SGS Laboratories in Burnaby was used as a check laboratory to analyse pulps for total nickel, sulphur, platinum, palladium, and gold, among other elements. From 2007 to 2010, International Plasma Laboratories Ltd. (IPL) in Richmond was used as the check laboratory. Prior to 2007, ALS Chemex in Vancouver was used as a check laboratory. At the times of these analyses, SGS was ISO:17025 certified, IPL was ISO:9001 certified, and ALS Chemex was ISO:9001 certified.

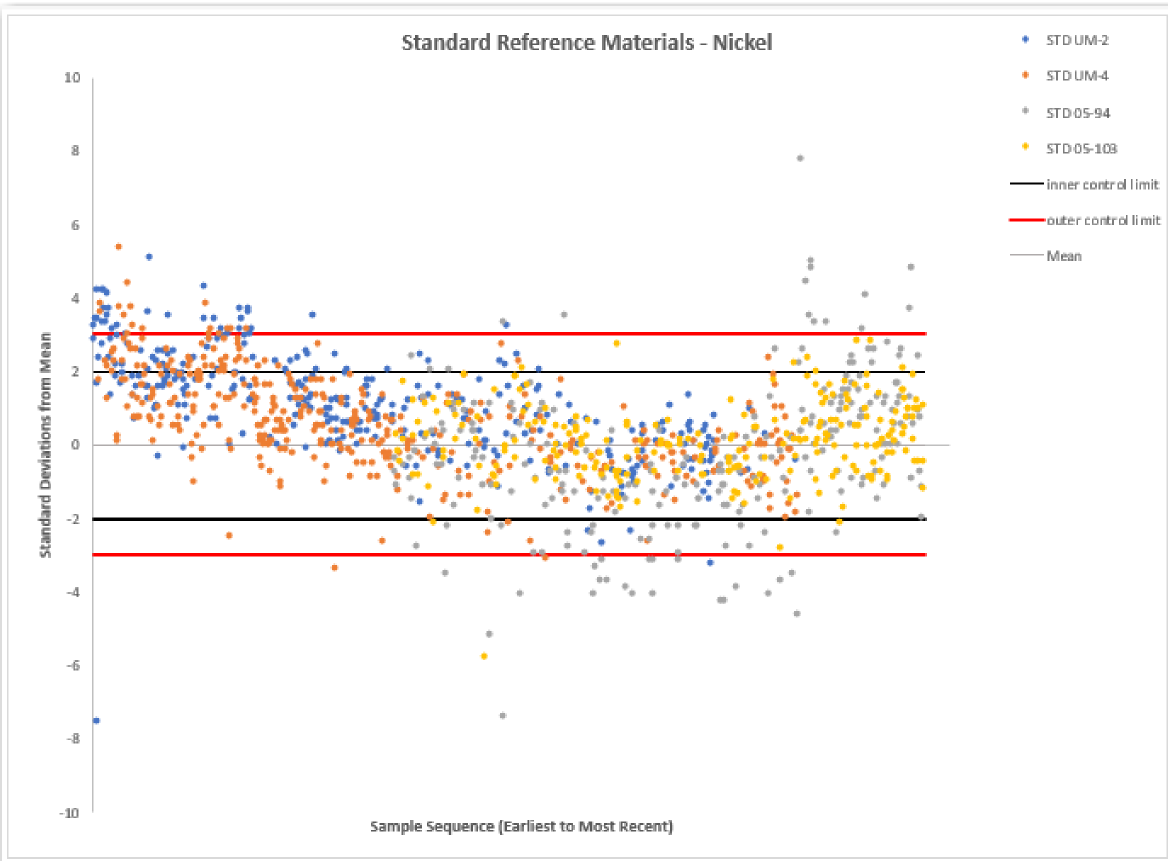
Giga Metals' standard reference materials used from 2004 to 2010 for Ni, Cu, and Co include two Canada Centre for Mineral and Energy Technology (CANMET) reference samples labelled UM-2 and UM-4 (Cameron, 1975). Both were derived from small, lenticular masses of peridotite that occur along a major east-west fault zone in the Werner Lake District of northwestern Ontario. CANMET analysed the material for ascorbic acid-hydrogen peroxide soluble nickel and, by use of four-acid digestion, for total Ni content. The CANMET certification of these materials was completed in 1974; however, it is not supported by current industry standards requiring a round-robin approach using several laboratories.

Giga Metals has two reference materials (05-94 and 05-103) prepared from mineralised 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 recertification program for all four reference materials.

Five other standard reference materials—PGMS-1, ME-1309, and ME-1310 from CDN Labs; and WGB-1, WMG-1 from Natural Resources Canada—have been used for monitoring platinum and palladium concentrations for exploration purposes. However, platinum and palladium quality assurance and quality control are not detailed in this report, as assay concentrations are not economic at this time and not the subject of this resource estimate reported herein.

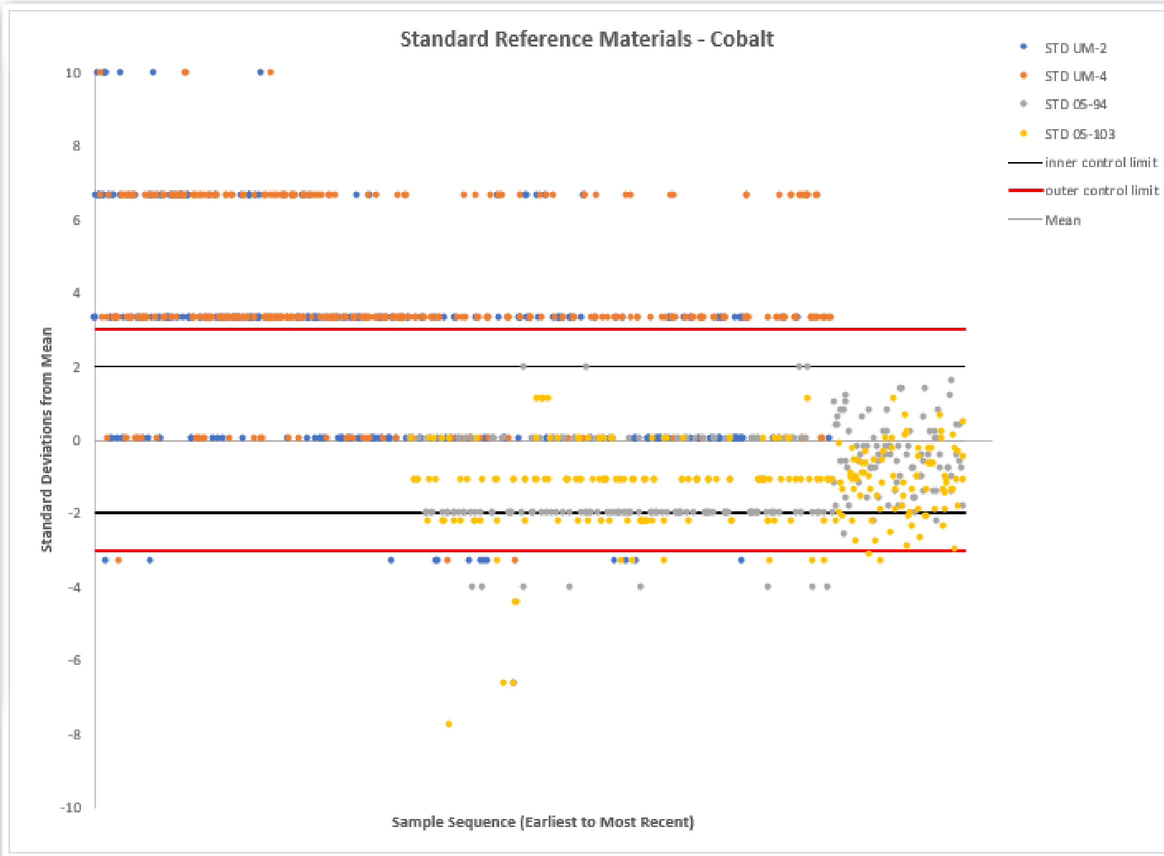
The performance of the nickel and cobalt standards is summarised in Figures 11-1 and 11-2, respectively.

**Figure 11-1: Nickel Performance of Standard Reference Materials**




Source: Kirkham, 2020.

**Figure 11-2: Cobalt Performance of Standard Reference Materials**




Source: Kirkham, 2020.

Figure 11-1 shows that early performance of the UM-2 and UM-4 appear to show a high bias that was investigated and resulted in the creation and use of the STD05-94 and STD05-103 standards (which are now the predominant standard). Although there appears to be a high bias, the check lab results during the same period show a correlation of 0.96, which illustrates that the results are repeatable and verifiable. AGORATEK recommended that the 2004, 2005 and 2006 data be adjusted to account for this relatively minor high bias; however, the author believes that this is not warranted and has not applied this arbitrary adjustment to the raw data. The cobalt standards performance shown in Figure 11-2 shows good results, although it is difficult to attain precision due to the very low values that are being measured and the sensitivity of the instrumentation in and around the non-detect ranges.

The source of the field blank used up to, and including, 2005 is not known. The field blank material used from 2006 to 2010 was crushed granite gneiss obtained from Squamish and crushed silica glass. During 2018, a crushed glass blank was used.

## 11.4 Adequacy Statement

It is the opinion of the QP, Garth Kirkham, P.Geo., that the sampling preparation, security, analytical procedures and quality control protocols used are consistent with generally accepted industry best practices and therefore reliable for the purpose of resource estimation.

# 12.0 DATA VERIFICATION

## 12.1 Site Visit & Verification

The qualified person visited the site, facilities and surrounding areas on October 9 to 10, 2018.

The tour of the offices, core logging, and storage facilities showed a clean, well-organised, professional environment. Giga Metals' geological staff and on-site personnel led Kirkham through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are to industry standards and best practices, and no issues were identified.

Several complete drill holes were selected by Kirkham and laid out at the core storage area. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, Kirkham toured the complete core storage facilities. No issues were identified, and core recoveries appeared to be very good.

The 2018 site visit entailed inspection of the workshops, offices, reclaimed drill sites, the Northwest and Horsetrail mineral resource areas along with the outcrops, historic drill collars, and areas of potential disturbance for potential future mining operations. In addition, the site visit included a tour of the most likely populated area to be affected by any potential mining operation along with surrounding environs. The drilling, logging and sample handling operations were conducted in a professional manner to industry standards and the on-site facilities were clean, well organised and of professional norms.

Kirkham reviewed the geological information from various programs and other relevant data available in the Giga Metals offices and is of the opinion that the programs were conducted and the data gathered in a professional and ethical manner.

Data validation and verification programs have been undertaken by numerous independent consultants as well as Giga Metals personnel, as discussed in previous NI 43-101 technical reports (AMC 2011, AMEC 2007, AGORATEK 2011) and performed subsequently including a database review by Kirkham Geosystems. The independent qualified person responsible for Section 12 of this report, Garth Kirkham, P. Geo., believes that the datasets are validated and verified sufficiently to support their use in mineral resource and mineral reserve estimation for each of the respective deposits.

Kirkham is confident that the data and results are valid based on the site visits and inspection of all aspects of the project, including the methods and procedures used. It is the opinion of Kirkham that all work, procedures, and results have adhered to best practices and industry standards as required by NI 43-101. No duplicate samples were taken to verify assay results, but Kirkham is of the opinion that the work is being performed by a well-respected company that employs competent professionals that adhere to best practices and standards. Kirkham also notes that authors of prior technical reports (AMC 2011) collected duplicate samples and had no issues.

The datasets employed for use in the mineral resource estimates are a mix of historic data and recent data. There is always a concern regarding the validity of historic data. Extensive validation and verification must be performed to ensure that the data may be relied upon.

Kirkham reviewed extensive validation and verification studies along with procedures performed by external consultants and Giga Metals to ensure the validity of the mineral resource estimates.

It is the opinion of Kirkham that the data used for estimating the current mineral resources for the Northwest and Horsetrail deposits is adequate for this PEA and may be relied upon to report the mineral resources and mineral reserves contained in this report.

# 13.0 MINERAL PROCESSING & 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 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 byproduct credits.

The main lithological domains are pyroxenite-dominated, green dunite and wehrlite/ dunite with various degrees of serpentinisation. The pyroxenite lithotype component accounts for less than 10% of the overall deposit tonnage.

A history of the metallurgical testwork conducted until 2010 was summarised in AMEC's 2007 NI 43-101 report, the Wardrop's 2010 Preliminary Economic Assessment NI 43-101 and AMC's 2011 Preliminary Economic Assessment. Work pertinent to the current study is referred to in this report.

Prior to 2010, the focus of mineral processing work was to create a concentrate suitable for on-site hydrometallurgical processing. In this early work, little test data showed the potential to make a high-grade saleable concentrate, while the sheer tonnage of the deposit and the attendant potentially high nickel production rate made inclusion of on-site high-capital hydrometallurgical processing more attractive. In 2010, testing at SGS working in parallel with G&T Metallurgical Services (G&T) started to expose the potential for the production of high-grade nickel concentrates, assaying 15-25% nickel. This led to the preparation of the 2011 technical report.

After a seven-year hiatus in metallurgical testing, work was re-started at Blue Coast Research Ltd. (BCR) in 2018. Initial work was conducted on samples drilled prior to and during 2010 (and stored as drill core under ambient conditions), while work in 2019 has been conducted on samples freshly drilled in 2018. The most recent work is probably the most valuable work supporting the 2020 PEA; however, early work is still of significant value and has been referred to where appropriate.

## 13.2 Sources of Information for this Study

The key sources of metallurgical information referred to in this study are listed below.

- G&T Metallurgical Services Ltd (2008) Metallurgical Development for the Hard Creek Project, Dease Lake, British Columbia, Canada, Project No. KM2181, October 31, 2008
- Xstrata Process Support (2008) Turnagain Ore Characterisation Project, Phase 1A, Hard Creek Nickel Corporation, Turnagain Project No. 09001823-09010824, Falconbridge, ON, June 17, 2008. This variability study of 17 samples included rougher flotation only.
- SGS Canada Inc. (2019) An Investigation into Flotation Flowsheet Development Testing on the Turnagain Deposit, Report 17124-01, March 6, 2019. This program was executed in 2010 and 2011 but was only recently reported. It included two separate variability studies: Rougher

flotation on 16 variability samples using a soda ash-based flowsheet, and cleaner flotation on 38 variability samples and 2 composites.

- Blue Coast Research (2019) PJ5252 Giga Metals – Turnagain Project Pre-feasibility Study Testwork Report, January 18, 2019. A Study of a master composite including seven locked cycle tests, plus a variability program using 11 samples.
- Blue Coast Research (2019) PJ5280 Giga Metals – Review of Turnagain Metallurgical Flowsheet Development Summary Report, October 2, 2019. As of the end of 2019, work included a preliminary variability program on 24 samples sourced from throughout the deposit, an ongoing optimisation program, two major (16-cycle) locked cycle programs and one regular (6 cycle) locked cycle test.

## 13.3 Grindability

Turnagain material is hard, with an average SAG grindability (Axb) of 27.1 and a Bond ball mill work index of 19.8 kWh/t, when ground to a closing screen size of 75 µm (see Table 13.1).

**Table 13.1: Grindability Data from the Turnagain Project**

| | # Samples | Mean | 85th Percentile | 15th Percentile |
|---|---|---|---|---|
| JK SMC | | | | |
| - A | 10 | 92.5 | 100.0 | 79.6 |
| - b | 10 | 0.31 | 0.41 | 0.24 |
| - Axb | 10 | 27.1 | 32.9 | 23.0 |
| - ta | 10 | 0.27 | 0.37 | 0.20 |
| Bond Rod Mill Work Index | | | | |
| - kWh/t | 5 | 18.6 | 22.3 | 14.1 |
| - P80 | 5 | 901 | 925 | 873 |
| Bond Ball Mill Work Index | | | | |
| - kWh/t | 77 | 19.8 | 22.5 | 14.1 |
| - $P_{80}$ | 47* | 83.8 | 111.1 | 71.0 |
| Abrasion test | 9 | 0.23 | 0.38 | 0.10 |
| Crusher Work Index | | | | |
| - kWh/t | 5 | 14.2 | 18.2 | 10.0 |

Note: *Not all BWI product sizes have been reported.

For the design criteria, Hatch will typically use values at the 85th percentile for testwork that might not have sufficient samples or show a large standard deviation in value since the ore type may not be well defined.

For testwork that has a high count of samples and +85% confidence interval, Hatch will determine that the testwork for the representative ore type is well defined and the average value can be used for design basis.

The abrasion index used in the design criteria was different from Table 13.1, since previous testwork showed a slightly more conservative value of 0.26.

HPGR testwork was conducted at UBC on five samples at four different forces. The results were then modelled to predict the reduction ratio, specific energy consumption, and product size distribution. The mean and variance data are shown in Table 13.2. The results suggest significant variability in hardness between the five tested samples.

**Table 13.2: HPGR Data**

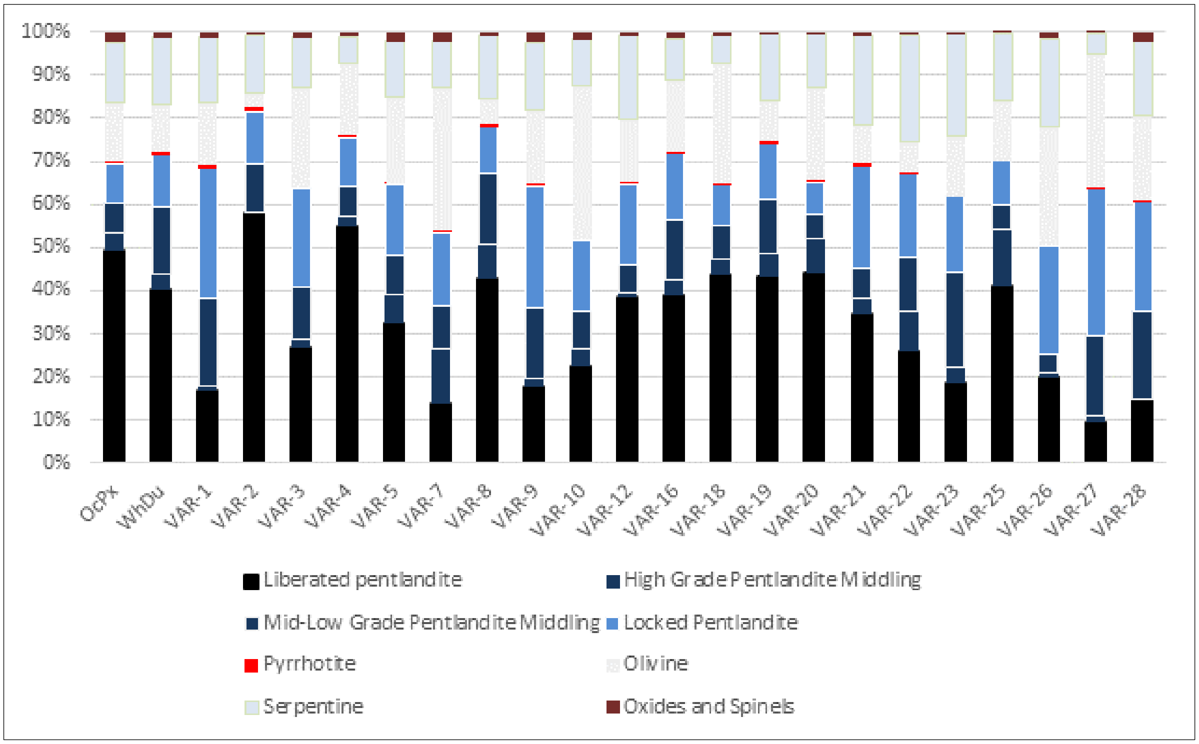
| | 1 $N/mm^2$ | 2 $N/mm^2$ | 3 $N/mm^2$ | 4 $N/mm^2$ |
|---|---|---|---|---|
| **Reduction ratio ($F_{50}/P_{50}$)** | | | | |
| Mean | 4.13 | 5.59 | 7.05 | 8.53 |
| Std. dev. | 1.46 | 2.13 | 2.81 | 3.50 |
| Maximum | 6.50 | 9.16 | 11.83 | 14.49 |
| Minimum | 2.70 | 3.77 | 4.83 | 5.90 |
| **Specific energy consumption (open circuit), kWh/t** | | | | |
| Mean | 1.41 | 1.87 | 2.33 | 2.80 |
| Std. dev. | 0.20 | 0.26 | 0.33 | 0.39 |
| Maximum | 1.61 | 2.15 | 2.65 | 3.22 |
| Minimum | 1.11 | 1.47 | 1.84 | 2.21 |
| **Specific energy consumption (closed-circuit with 6 mm screen), kWh/t** | | | | |
| Mean | 2.76 | 3.22 | 3.62 | 4.01 |
| Std. dev. | 0.65 | 0.74 | 0.79 | 0.85 |
| Maximum | 3.56 | 4.11 | 4.51 | 4.91 |
| Minimum | 1.82 | 2.14 | 2.45 | 2.75 |

## 13.4 Mineralogy

Nickel deportment is distributed between pentlandite, pyrrhotite, olivine, serpentine and various oxides and spinels.

Based on QEMSCAN and probe analyses from 23 samples, assaying on average 0.25% nickel and 1.11% sulphur, and located throughout the resource, the average nickel deportment as pentlandite is 66%, with just 0.42% present in pyrrhotite (Figure 13-1). The remainder is present in non-sulphide minerals, on average 17% as olivine and 14% as serpentine. A small amount is contained with oxides and spinels. The presence of nickel as pentlandite is closely linked to the sulphur assay in the feed, with the samples assaying more and less than 1% sulphur containing 71% and 61% nickel as pentlandite, respectively. Further, pentlandite liberation is linked to sulphur assay, with 78% of the pentlandite being present in liberated or mid/high-grade middling form in samples assaying over 1% sulphur. That number dropped to 66% for samples assaying below 1% sulphur.

**Figure 13-1: Nickel Speciation in 23 Samples Tested during 2019**




Source: Blue Coast Metallurgy, 2020.

For the current technical report, the only economically recoverable nickel is hosted in pentlandite. At a grind of 80% passing 80 µm, roughly half is liberated (32% of all nickel) and another 4% is hosted in high-grade middlings. If not slimed in grinding, this nickel will float quickly and has the potential to create very high-grade concentrates. Another 12% of the nickel is present as mid- or lower-grade middlings. These should also respond well to flotation. The remaining pentlandite, comprising about 18% of the nickel, is locked in silicates and will be poorly recovered. Typically, these will be mostly recoverable using longer flotation residence times and more collector. The degree of nickel locking suggests that at a grind of 80% passing 80 µm, 80% to 90% of the pentlandite is floatable. As 66% of the nickel is present as pentlandite, this suggests a theoretical nickel recovery ceiling of 55% to 60% based on these 23 samples.

The proportion of nickel present in poorly liberated or locked form, averaging 30% but ranging from 13% to 52%, points to a material that will benefit from fine grinding to achieve good liberation.

However, the comminution-resistant material, and the propensity for the softer pentlandite[1] to slime in such an extreme grinding environment, all conflict with this assumption. Optimising the trade-off in grinding power will be a key step in the Turnagain Project.

Olivine and serpentine are important hosts of nickel that is not economically recoverable by flotation. Pyrrhotite is a very minor host of nickel and its recovery is not economic, so the flotation process has been developed to maximise its rejection to tails.

The median, 95$^{th}$, 80$^{th}$, 20$^{th}$ and 5$^{th}$ percentile host rock mineralogy, as determined by QEMSCAN, is shown in Table 13.3. Talc is commonly problematic with mafic nickel ores, as it is can be free-floating and contains high levels of MgO, which is challenging for smelters to process. However, the Turnagain deposit is largely free from talc (a small proportion of the samples contained talc at levels that would require significant depressant doses).

**Table 13.3: Modal Abundance from QEMSCAN Analysis of 34 Samples Studied by XPS in 2018 & 2019**

| | **Median** | **95$^{th}$ Percentile** | **80$^{th}$ Percentile** | **20$^{th}$ Percentile** | **5$^{th}$ Percentile** |
|---|---|---|---|---|---|
| Pentlandite | 0.59 | 0.97 | 0.74 | 0.44 | 0.28 |
| Chalcopyrite/Valleriite | 0.09 | 0.45 | 0.19 | 0.03 | 0.02 |
| Pyrrhotite | 2.8 | 5.92 | 3.8 | 1.0 | 0.39 |
| **Total Sulphides** | **3.5** | **7.01** | **4.6** | **1.6** | **0.93** |
| Olivine | 29.4 | 52.85 | 42.5 | 19.0 | 13.30 |
| Serpentine | 41.5 | 69.48 | 62.5 | 29.3 | 25.42 |
| Quartz | 0.00 | 0.01 | 0.01 | 0.00 | 0.00 |
| Feldspar | 0.02 | 0.56 | 0.17 | 0.00 | 0.00 |
| Orthopyroxene | 0.05 | 17.07 | 0.62 | 0.02 | 0.01 |
| Clinopyroxene | 11.9 | 22.53 | 18.6 | 2.2 | 1.14 |
| Amphibole | 0.07 | 1.34 | 0.32 | 0.03 | 0.01 |
| Chlorite | 0.36 | 1.31 | 0.85 | 0.09 | 0.01 |
| Epidote | 0.14 | 0.73 | 0.44 | 0.03 | 0.01 |
| Mica | 0.23 | 1.19 | 0.66 | 0.01 | 0.00 |
| Talc | 0.01 | 3.51 | 0.04 | 0.00 | 0.00 |
| Oxides and Spinels | 5.1 | 7.30 | 6.4 | 3.6 | 3.08 |
| Carbonates | 0.14 | 0.59 | 0.37 | 0.09 | 0.04 |
| Other | 0.09 | 0.16 | 0.11 | 0.07 | 0.05 |
| **Total NSG** | **96.5** | **99.07** | **98.4** | **95.4** | **92.99** |

[1] Slimed pentlandite floats particularly poorly.

Pyrrhotite is another potentially problematic mineral, owing to challenges that can occur in separating pentlandite from pyrrhotite. The mass ratio of pyrrhotite to pentlandite, which averages about 5:1, means that pyrrhotite rejection is a requirement for the production of a saleable nickel concentrate. Fortunately, the Turnagain deposit hosts, by nickel industry norms, relatively little pyrrhotite and the pyrrhotite is reasonably easy to reject in flotation. This is a key feature of the mineralisation that allows for production of a superior grade nickel concentrate.

## 13.5 Flotation

The Turnagain resource is an example of a low-sulphur, nickel-bearing mafic or ultramafic deposit, the treatment of which tends to follow well-established schemes. Recoverable nickel is contained within pentlandite with very minor pyrrhotite, the latter hosting nickel at a grade that is too low to warrant its shipment to smelters. The pentlandite grain size range is moderately fine, but varies widely meaning recovery is quite dependent on primary grind size. Host rock mineralisation is predominantly benign silicates (from a flotation sense) such as olivine and serpentine. Some tested samples have contained a more problematic abundance of talc suggesting that this exists, rarely, within the resource, while one sample out of the 118 variability samples tested since 2007 contained some active carbon. The flowsheet as developed does not handle higher levels of talc or active carbon, rather the much more process-friendly mineral mix, which constitutes the overwhelming majority of samples tested to date.

Typical pentlandite flotation schemes, when present in a low-sulphur host mineralisation, tend to include moderate doses of xanthate collectors and reagents designed to both disperse and in some cases depress the silicates. Raising the pH is only necessary if the pyrrhotite proves to be highly floatable.

In the case of Turnagain flotation, the following principals have been used in flotation flowsheet development:

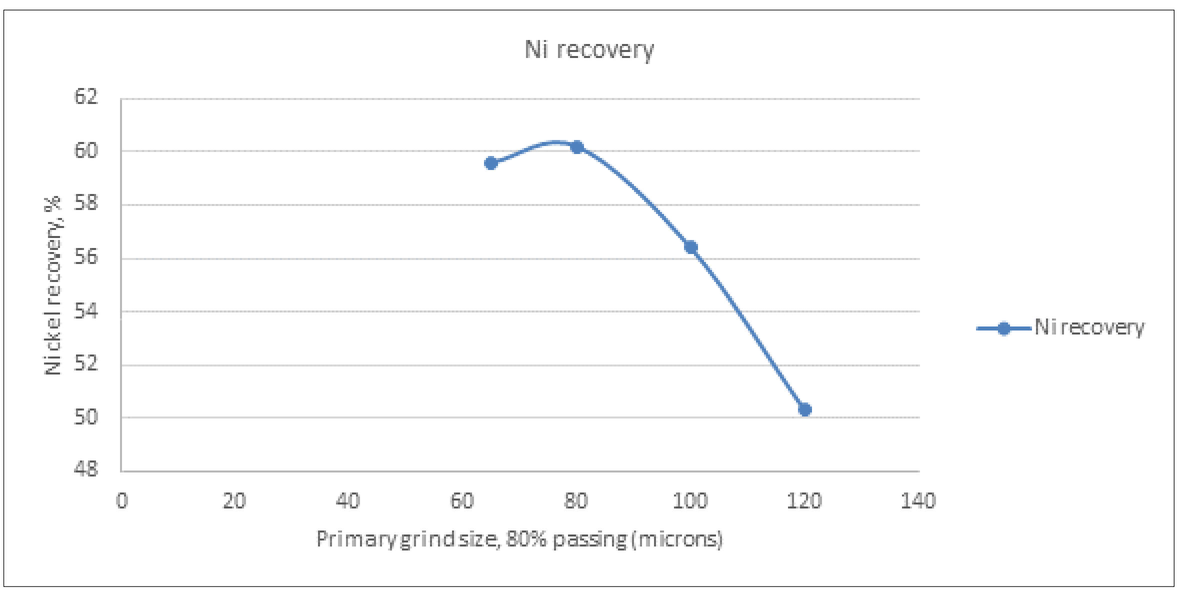
- Pyrrhotite has proven to be poorly floatable with recoveries to final concentrates usually in the range of 3-7%, so pH manipulation is not needed to assist in pyrrhotite rejection. Consequently, flotation is effected at natural pH (pH 8.5-10).
- The silicate mix usually contains little or no talc, so gangue floatability is weak. This eliminates the need for large doses of gangue depressants, although some of the tested treatment schemes have used modest doses of polymeric guar gum and/or carboxymethyl cellulose (CMC) based depressants in testing of Turnagain samples from time to time.
- However, the silicates can interfere with pentlandite flotation, presumably by interacting with the pentlandite surfaces. Hexametaphosphate dispersants, such as Calgon, are widely used to address this and have been adopted for this flowsheet. Calgon enhances pentlandite floatability, but too much also increases the challenge of gangue rejection in cleaning, so they are used sparingly, especially in cleaner flotation.
- Pulp density: The use of dilute pulps has proven advantageous in flotation of Turnagain samples.

- Xanthate is highly effective as a baseline reagent for Turnagain flotation. Isopropyl or isobutyl xanthate both work well. However, adding a secondary collector can be useful in nickel flotation and Turnagain is a candidate for this. Adding the alkyl thionocarbamate collector AERO 3894 at a fraction of the xanthate dose may enhance pentlandite floatability.
- MIBC is used as a frother. The silicate-dominant froths have some degree of inherent stability so strong frothers are not needed, while MIBC volatilises in the process, thereby reducing the risk of frother buildup in latter stages of cleaning.

Primary grind size has a strong influence on nickel recovery. A series of four locked cycle tests were run at different grind sizes in 2018 (Figure 13-2). They pointed to an optimal recovery at about 80 µm. Turnagain material is very hard and grinding costs will be high; however, the steepness of the recovery curve meant that the finer grind was still justified, though grinding finer than 80 µm had a tendency of over-sliming the soft pentlandite, adversely affecting recovery. Therefore, for the design criteria, it is recommended that the target grind size should be between a $P_{80}$ of 80 to 85 µm.

So far, testwork employing concentrate regrinding has failed to yield improved metallurgy, so the flowsheet includes only grinding prior to rougher flotation.

**Figure 13-2: Link between Primary Grind size & Nickel Recovery to Final Concentrate by Locked Cycle Test**




Source: Blue Coast Metallurgy, 2020.

Table 13.4, from the latter cycles of the second super cycle test, shows a typical treatment scheme used since 2018 for six-cycle locked cycle and 16-cycle super cycle tests. The latter were 10 kg production runs, run in closed circuit and aimed at producing concentrate for marketing purposes. The composites were created to reflect geologist visual estimates of talc grade, though in reality very little talc has been seen in processing any of them.

**Table 13.4: Typical Conditions used in Closed Circuit Flotation Testing at BCR**

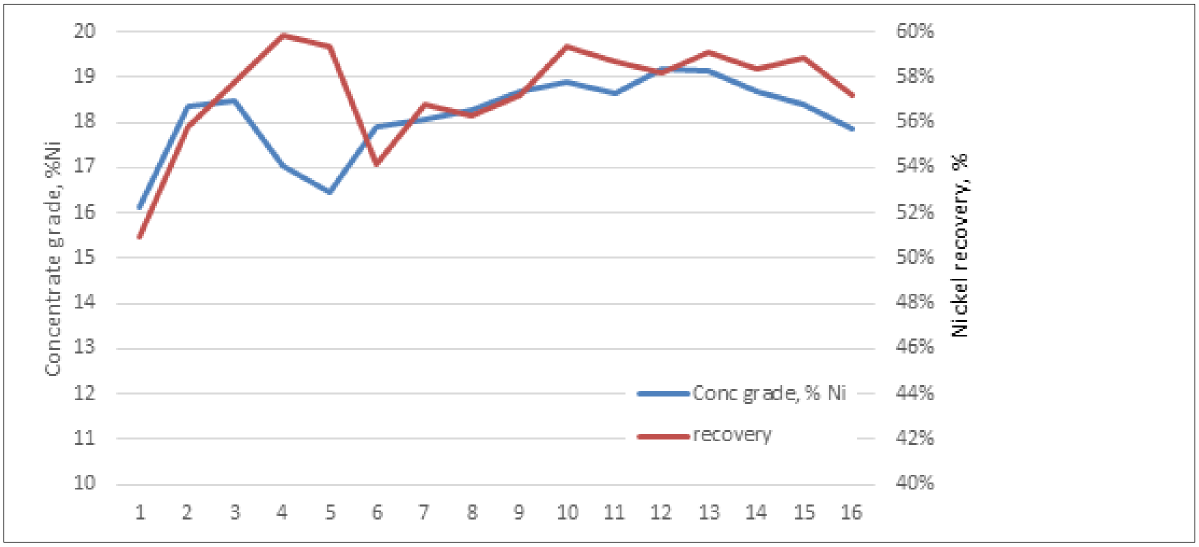
| Stage | Reagents (g/t) | | | | Time, Minutes | |
|---|---|---|---|---|---|---|
| | Calgon | SIPX | MIBC | Grind (µm) | Cond. | Froth |
| Primary Grind | 33 | | | 80 | | |
| Ni Rougher 1 | | 10 | 16 | | 1 | 3 |
| Ni Rougher 2 | 7 | 10 | 8 | | 1 | 3 |
| Ni Rougher 3 | 7 | 10 | 16 | | 1 | 8 |
| Ni Rougher 4 | 13 | 20 | 32 | | 1 | 21 |
| Rougher Total | 60 | 50 | 73 | 80 | 4 | 35 |
| Ni Cleaner 1 | | 18 | 5 | | 2 | 15 |
| Ni Cleaner 1 Scavenger | | | | | | 5 |
| Ni Cleaner 2 | | 6 | 3 | | 1 | 10 |
| Ni Cleaner 3 | | 3.5 | | | 1 | 9 |
| Ni Cleaner 4 | | 1 | | | 1 | 4 |
| Cleaner Total | | 29 | 8 | | 5 | 43 |

Metallurgy in this super cycle test stabilised in the second half of the test; the first half being partly used to test different process parameters. During the last eight cycles, nickel recoveries were close to 59%, to a concentrate assaying 18% to 19% nickel (Figure 13-3).

Metallurgical projections from ten locked and super cycle tests run using optimal flowsheets are shown in Table 13.5. All the locked-cycle testwork shown in Table 13.5 has been conducted on samples assaying over 1.1% sulphur and 0.26% nickel; samples lower in nickel and sulphur have been tested in batch flotation.

No recent locked cycle work has been done on low nickel or sulphur samples; the only previous work done at SGS consists of two locked cycle tests yielding recoveries of less than 40% to lower grade nickel concentrates. These tests did not implement many of the process improvements made in more recent testwork, so the data should be interpreted with caution.

**Figure 13-3: Nickel Concentrate Grades & Recoveries per Cycle in the Talc 2-3 Super Cycle Test at BCR**




Source: Blue Coast Metallurgy, 2020.

**Table 13.5: Summary of Locked Cycle Test Data using Optimised Treatment Schemes**

| Lab | Locked Cycle Test | Feed Grade | | Conc. Grade | | | Ni Recovery |
|---|---|---|---|---|---|---|---|
| | | Ni, % | S, % | Ni, % | Fe, % | MgO, % | |
| SGS | 08-264 LCT 4 | 0.31 | 1.15 | 21.4 | 34.4 | 6.6 | 50.8 |
| SGS | 08-264 LCT 5 | 0.31 | 1.15 | 19.4 | 28.9 | 9.6 | 51.0 |
| SGS | 10-265 LCT3 | 0.32 | 1.15 | 20.9 | 32.3 | 7.1 | 61.7 |
| SGS | 10-265 LCT6 | 0.32 | 1.15 | 20.3 | 32.9 | 8.3 | 63.6 |
| SGS | 10-265 Bulk LCT | 0.32 | 1.15 | 19.7 | n/a | n/a | 57.9 |
| BCR | 10-266 LCT2 | 0.30 | 1.26 | 15.3 | 38.5 | 7.4 | 59.8 |
| BCR | 10-266 LCT3 | 0.30 | 1.26 | 18.3 | 37.6 | 6.1 | 60.2 |
| BCR | Litho comp LCT1 | 0.26 | 1.14 | 19.2 | 30.3 | 8.4 | 57.0 |
| BCR | Talc 1, SCT | 0.30 | 1.48 | 16.4 | n/a | n/a | 60.5 |
| BCR | Talc 2-3, SCT | 0.30 | 1.44 | 19.0 | n/a | n/a | 58.6 |
| Average | | | | 19.3 | | | 57.8 |

## 13.6 Other Treatment Schemes

### 13.6.1 Two Concentrates

Work has been conducted at different laboratories to explore the potential to produce two concentrates. It has been repeatedly demonstrated that high-grade concentrates in excess of 20% nickel can be produced. Previous tests at BCR have produced premium grade concentrates, while locked cycle tests at SGS have produced concentrates assaying up to 28% nickel, albeit at 49% nickel recovery. While this concentrate may have a virtually unique quality and may attract premium terms, the key is to produce a second, lower grade concentrate that is still reliably saleable. To date this has not been demonstrated consistently enough to adopt the flowsheet with two concentrates as the baseline process.

### 13.6.2 Hydrometallurgical Processing

Historically, hydrometallurgical testwork has been completed on lower-grade Turnagain concentrates in four separate testing periods. Work with Cominco Engineering Services Ltd. (CESL) in 1999, as reported in the 2006 AMEC PEA, achieved extractions of nickel and cobalt of approximately 97% and 93%, respectively. Work with Outokumpu in the late 2000s, reported in the 2010 Wardrop PEA, noted chloride leach extractions from a 4% nickel concentrate of 98% nickel and 97% cobalt. Subsequent work on sulphate leach flowsheets was completed at SGS in 2007 and 2008 on concentrate samples grading 4-5% nickel and 10% nickel. This work reported leach extractions of over 97.5% for nickel and cobalt from finely ground concentrates at leach conditions of 110°C to 220°C.

Additionally, one of these test programs by SGS in 2008 demonstrated the complete flowsheet, treating leach solution by iron removal, copper solvent extraction, mixed hydroxide precipitation, and magnesium removal, producing a mixed hydroxide primary product grading over 45% nickel.

Although no hydrometallurgical testwork has been completed on the higher-grade nickel concentrates produced since the mineral processing breakthroughs of 2010-2011 achieved commercial grade concentrates, it is expected that the current Turnagain concentrates would have equal or better performance due to lower levels of impurities.

## 13.7 Metallurgical Forecast

### 13.7.1 Introduction & Background

Nickel recovery represents the greatest source of variability for metallurgical forecasting. Concentrate grades of 18-25% nickel have been repeatedly achieved in each study since 2010, and while there is a trade-off between concentrate grade and recovery, a concentrate grade of 18% nickel is consistently achievable, typically at high cleaner circuit recoveries (90% or higher).

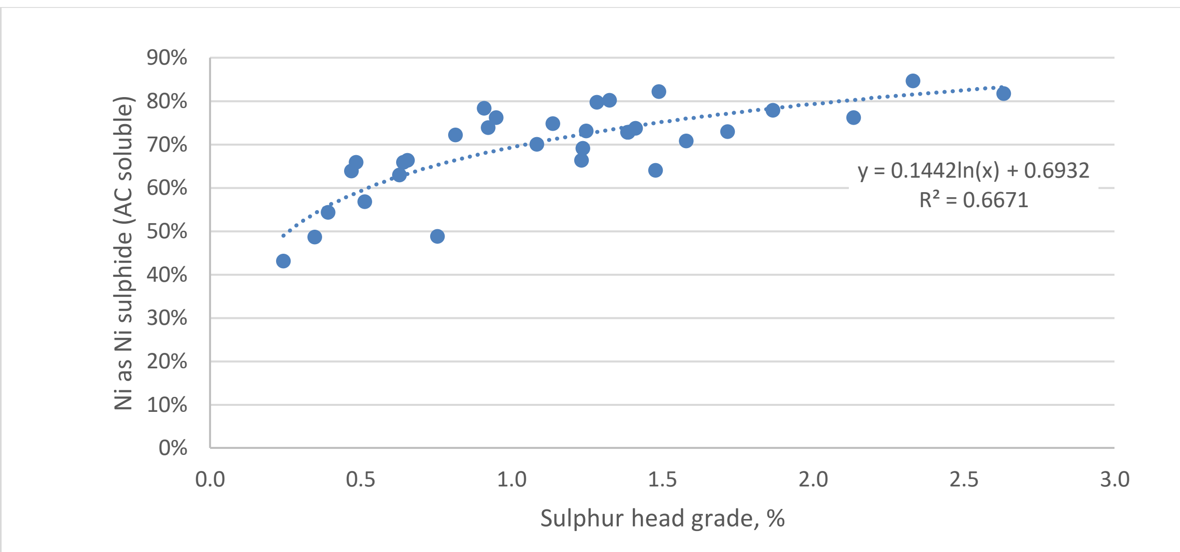
Past studies have recognised nickel recovery as the primary source of metallurgical variability. Work by G&T in 2009 led to the creation of a multi-variate regression driven by the proportion of nickel in sulphide form and the sulphur grade, while the 2010 PEA study employed a metallurgical forecast that was largely based on a nickel head grade/recovery relationship. That study chose to filter out much of the low recovery data on the assumption that they were not optimally tested; however, the weight of additional data points to the presence of samples of lower nickel recovery, making the elimination of these data more difficult to justify.

The present approach recognises that the bulk of the variability in recovery occurs in rougher flotation, while the recovery of nickel from rougher concentrate to final concentrate tends to be quite high and quite consistent. Accordingly, a parameter is first developed to predict rougher flotation recovery.

### 13.7.2 Rougher Flotation Recovery

As described earlier in this section, nickel is deported within two broad host types, namely (1) pentlandite (recoverable), and (2) non-sulphides (non-recoverable). The distribution of nickel between these hosts is the primary cause of metallurgical variability for the project[2]. Various diagnostic leaches exist that provide some insight into the sulphide and non-sulphide hosted nickel. Ammonium citrate digestion is one of these and has been used in this study as a guide for pentlandite abundance both in metallurgical testing and, to a limited extent, in resource modelling (Figure 13-4). However, the method has a propensity to partially leach some silicate minerals, releasing nickel that is unrecoverable by flotation. This makes it somewhat unreliable and prone to error associated with minor variations in procedure between laboratories.

**Figure 13-4: Link between Sample S Assay & Ni as Ni(s) Assay as determined by Ammonium Citrate Method**




Source: Blue Coast Metallurgy, 2020.

Sulphur grades perhaps offer a better parameter to predict nickel rougher recovery. It follows from a geological perspective that more sulphidic samples have a higher proportion of sulphide nickel, and hence more recoverable nickel. In fact, data from recent studies at Blue Coast bear this out,

[2] Pentlandite grain size is probably an additional factor. At this time, its effect on nickel recovery and how it can be modelled in the resource is yet to be established.

where a logarithmic fit can be used to link sulphur assays with the proportion of nickel as a sulphide with reasonable accuracy[3]:

A data resource comprising the six different variability studies listed earlier and 128 samples has been used to evaluate possible parameters affecting nickel flotation recovery.

Each of these studies employed different rougher and cleaner flotation procedures, leading to different mass pull rates to the rougher concentrate. Driven by flowsheet selection at each laboratory, these varied from 5% to 25% and with them, nickel recoveries varied. Accordingly, the data cannot be combined into a single dataset. However, they provide an opportunity to repeatedly, and independently, explore different candidate parameters driving nickel recovery when each dataset is examined in isolation.

Table 13.6 lists correlation coefficients as determined from each of the study datasets between nickel rougher and cleaner recoveries, and some typical candidate geometallurgical parameters. Total and sulphide nickel head grades are not well connected with nickel recovery. Nickel recovery is somewhat more reliably linked with the proportion of nickel in sulphide form, and better still with sulphur assay in the feed. Study 3a is an outlier to these trends for reasons that are presently not well understood. This has been excluded in calculating the average numbers.

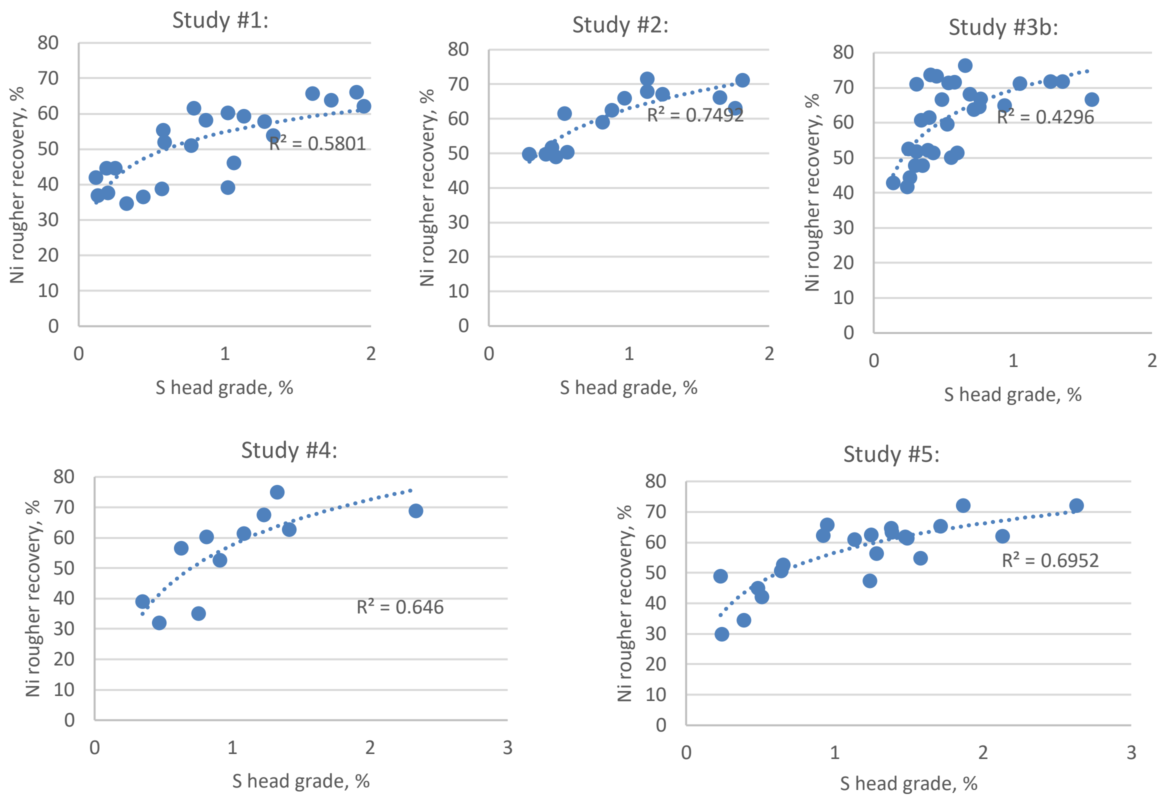
**Table 13.6: Correlation Coefficients between Four Candidate Geomet Parameters & Nickel Recovery (R-squared)**

| Parameter | Study 1 | | Study 2 | | Study 3a | | Study 3b | | Study 4 | | Study 5 | | Average | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Rghr | Clnr | Rghr | Clnr | Rghr | Clnr | Rghr | Clnr | Rghr | Clnr | Rghr | Clnr | Rghr | Clnr |
| Ni Head Grade | 0.01 | 0.05 | 0.04 | n/a | 0.20 | 0.21 | 0.01 | 0.04 | 0.18 | 0.05 | 0.03 | 0.02 | 0.05 | 0.04 |
| Ni(s) Head Grade | 0.10 | 0.27 | 0.26 | n/a | 0.23 | 0.20 | 0.41 | 0.17 | 0.37 | 0.12 | 0.43 | 0.35 | 0.31 | 0.23 |
| S Head Grade | 0.58 | 0.80 | 0.75 | n/a | 0.17 | 0.07 | 0.45 | 0.56 | 0.65 | 0.50 | 0.70 | 0.64 | 0.63 | 0.63 |
| % Ni as Sulphide | 0.63 | 0.38 | 0.36 | n/a | n/a | n/a | 0.58 | 0.50 | 0.58 | 0.25 | 0.67 | 0.68 | 0.56 | 0.45 |

Relationships between the sulphur head grade and the recovery of nickel to the rougher concentrate from Studies 1, 2, 3b, 4 and 5 are shown in Figure 13-5. While none of the relationships is tight, they are all quite similar, making sulphur grade a consistent if somewhat crude indicator of nickel recovery. The degree of scatter in the relationships points to the presence of as-yet undetermined additional factors affecting nickel recovery, pointing to the need for more in depth geometallurgical work in the future as the project progresses.

[3] The nickel sulphide assays were all completed by the Blue Coast assay laboratory.

**Figure 13-5: Regression Analysis of Feed Sulphur Grade & Nickel Rougher Recovery[4] from Five Variability Studies**

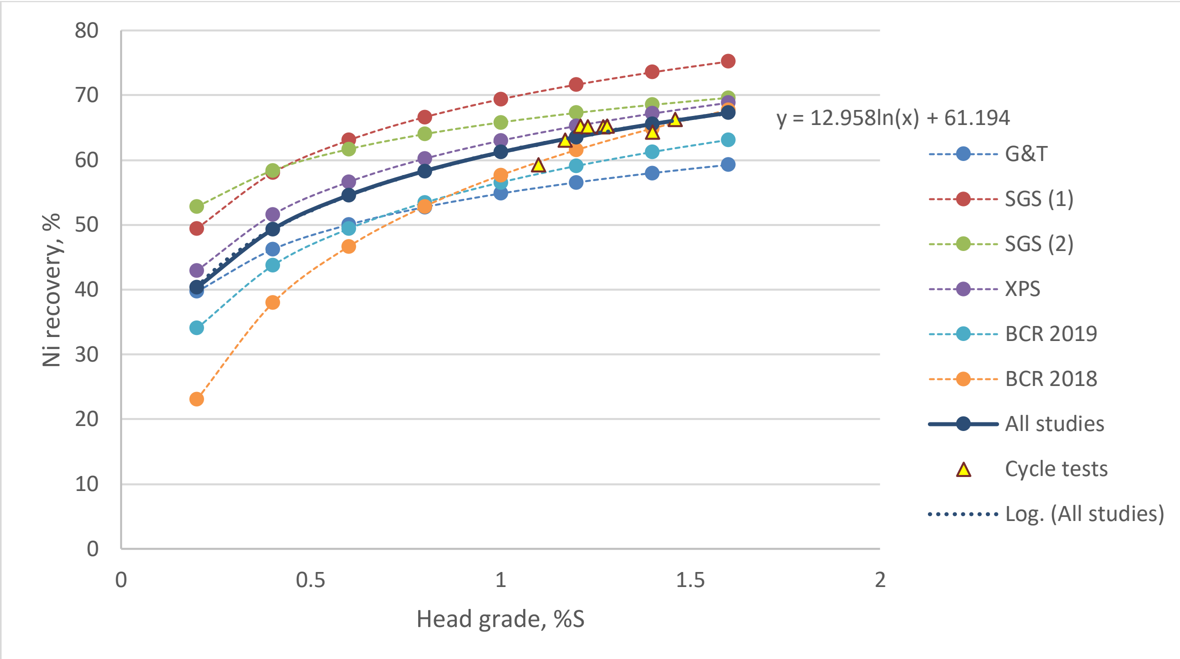



Source: Blue Coast Metallurgy, 2020.

Each of the regression fits are described by logarithmic equations. Plotting the logarithm curves from each of these variability studies on the same graph reveals a series of S grade vs. Ni rougher recovery curves that, for the most part, run parallel with each other (Figure 13-6). As described earlier in this section, the offset between them is due to the different flowsheets used for each of the programs, and the resulting rougher mass pull rates, which varied from an average of 5% to 25%.

[4] Nickel cleaner recoveries are based on mill feed.

**Figure 13-6: Algorithms Fitting S Head Grade vs. Ni Rougher Recovery from the Five Variability Studies**




Source: Blue Coast Metallurgy, 2020.

For this study, the sulphur head grade/Ni rougher recovery relationship used for metallurgical forecasting was nominated to be the mean from all the different datasets. This was established at:

*Ni rougher recovery (%) = 12.958 x ln [sulphur head grade (%)] + 61.194*

To check the validity of this curve, average rougher recoveries were plotted against sulphur grades from a host of locked cycle tests and super-cycle tests. As each of these tests consists of between 6 and 16 rougher flotation tests making them statistically robust, and as they were all on composites of material sourced from around the resource, they represent a good quality check of the proposed curve. They are shown in Figure 13-7 as triangles and coincide well with the chosen curve, so validating the rougher recovery relationship.

The reader should be aware that the reliability of the data declines at lower feed sulphur grades, as (1) there is no locked cycle data to validate cleaner performance for samples below 1% sulphur, (2) the best quality data, which is from the most recent studies, contained a paucity of data points below 0.5% sulphur, and (3) the flowsheets have been developed for sulphur grades in excess of 1% and are believed to be less than optimal for lower feed sulphur grades. As such application of the regressions on feed sulphur grades below 0.5% sulphur for the sake of financial forecasting is not recommended.

### 13.7.3 Recovery to Final Concentrate

Data from the same locked cycle tests and super cycle tests can also be used to establish the cleaner stage recovery of nickel from rougher concentrate to final concentrate. Four of the five tests exhibited quite similar cleaner stage recoveries varying from 88% to 92%, with just one test being higher, at 96%.

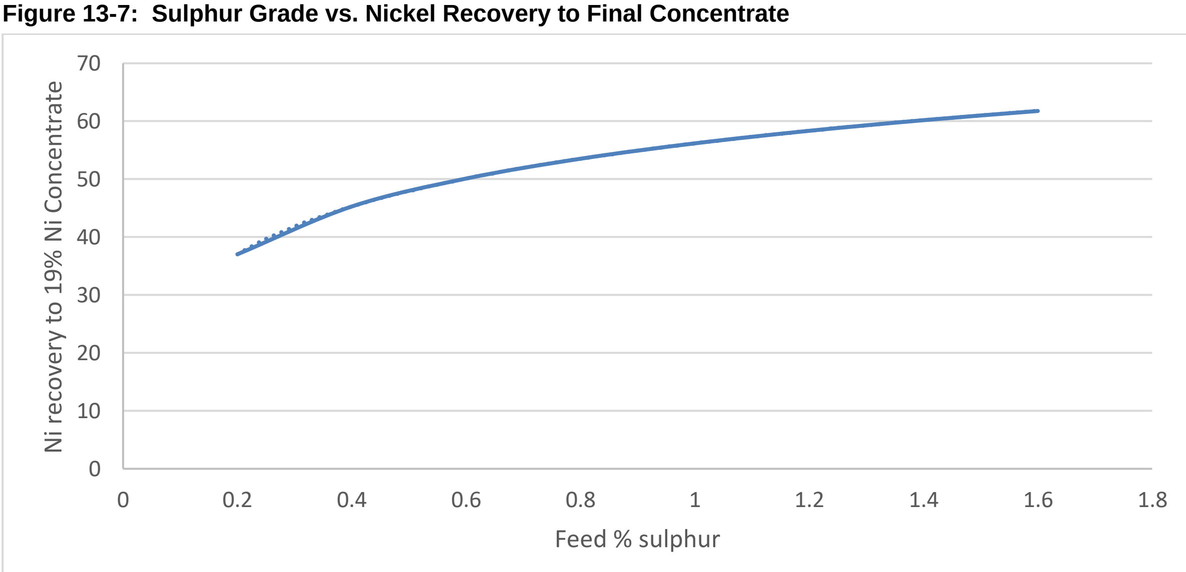
There is inadequate evidence to indicate any relationship between Ni or S grade, or any other tested parameter, and cleaner stage recovery, so a fixed recovery has been assumed, to be the average from the five tests (91.8%). The recoveries are shown in Table 13.7.

**Table 13.7: Cleaner Stage Recoveries from Five Recent Locked & Super Cycle Tests**

| | % Ni | Final Conc. | Rougher | Cleaner Stage Recovery |
|---|---|---|---|---|
| LCT1: 2018 | 19.2 | 57.4 | 65.3 | 87.9% |
| LCT3- 2018 | 18.3 | 59.8 | 65.2 | 91.7% |
| SCT Talc 1 – 2019 | 17.4 | 60.9 | 66.2 | 92.0% |
| SCT Talc 2-3 – 2019 | 19.0 | 58.6 | 64.1 | 91.4% |
| LCT1 – 2019 | 19.2 | 57.0 | 59.3 | 96.1% |
| Average | 18.6 | 58.7 | 64.0 | 91.8% |

Applying this cleaner recovery factor to the regression curve for rougher flotation yields the following S head grade vs nickel recovery curve, to a concentrate grade of 18.6% nickel, as shown in Figure 13-7.

**Figure 13-7: Sulphur Grade vs. Nickel Recovery to Final Concentrate**




Source: Blue Coast Metallurgy, 2020.

The respective curve, and therefore the equation used for recovery forecasting for this study, is:

*Ni recovery to 18% Ni concentrate = (12.958 x ln [feed sulphur(%)] + 61.194) x 0.918*

## 13.8 Final Concentrate Specifications

Multi-element scans have been conducted on saleable nickel concentrates with grades close to the expected life-of-mine product, from five different locked cycle tests on different samples and composites. The average for each element from the dataset is shown in Table 13.8.

**Table 13.8: Average Assay of 39 Elements in Five LCT Concentrates**

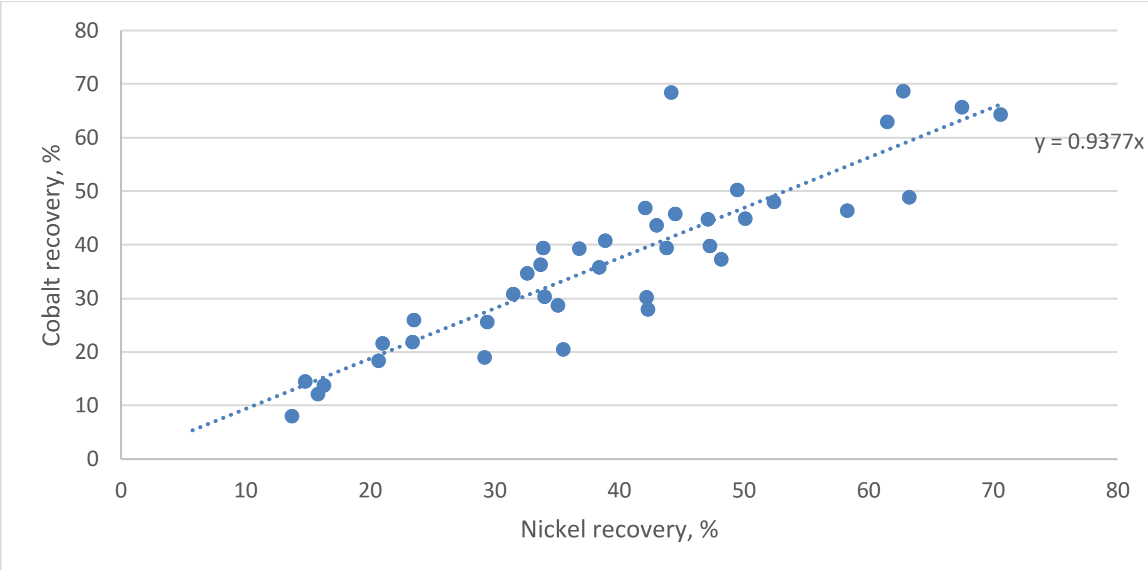
| Element | Assay | Element | Assay | Element | Assay | Element | Assay |
|---|---|---|---|---|---|---|---|
| Ni % | 19.7 | As % | <0.001 | F % | <0.01 | Se g/t | 78.4 |
| Co % | 1.2 | Ag g/t | 9.4 | Hg g/t | 0.7 | Sn g/t | <20 |
| Cu % | 0.46 | Al % | 0.56 | K % | 0.09 | Sr g/t | 24 |
| Fe % | 32.3 | Ba g/t | 81 | Li g/t | < 5 | Te g/t | <60 |
| S % | 25.9 | Be g/t | 1 | Mn g/t | 280 | Ti g/t | 210 |
| $SiO_2$ % | 6.4 | Bi g/t | <20 | Mo g/t | 53 | Tl g/t | <60 |
| Mg % | 4.4 | Ca % | 0.48 | Na g/t | 540 | U % | <0.005 |
| Pt g/t | 1.1 | Cd g/t | <3 | P g/t | <100 | V g/t | 48 |
| Pd g/t | 2 | Cl g/t | 67 | Pb g/t | 380 | Y g/t | 2.1 |
| | | Cr % | 0.1 | Sb % | <0.002 | Zn g/t | 210 |

Pentlandite hosts both cobalt and palladium, such that for any given sample cobalt recovery equates to roughly 93.8% of nickel recovery (Figure 13-8).

The assay ratio of cobalt to nickel is quite consistent at 0.6% Co per 10% Ni, and that of palladium is 1 g/t Pd for every 10% Ni. Platinum roughly follows palladium at 0.5 g/t Pt for every 1 g/t Pd.

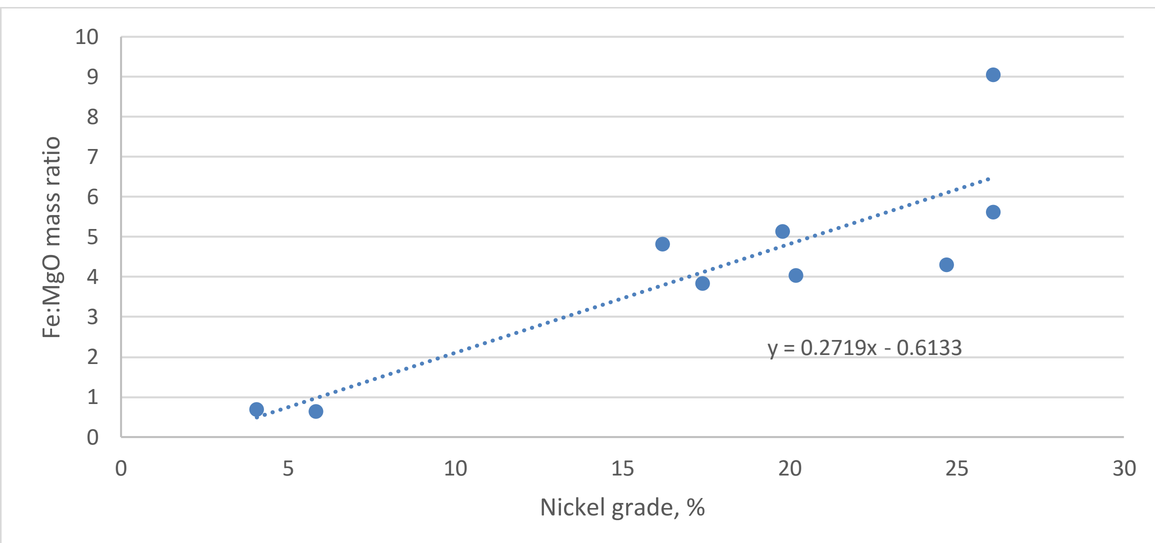
The relationship between nickel grade and Fe:MgO ratio is shown in Figure 13-9. The assay ratio of Fe:MgO is a key criterion in the marketability of many nickel flotation concentrates, as it impacts metallurgy in nickel smelting. This is described further in Section 19, Market Studies and Contracts.

**Figure 13-8: Cobalt Recovery vs. Nickel Recovery**




Source: Blue Coast Metallurgy, 2020.

**Figure 13-9: Relationship between Ni Grade & Fe/MgO Ratio**




Source: Blue Coast Metallurgy, 2020.

## 13.9 Recovery Methods & Equipment Selection

The process recommended to Hatch for plant design includes primary comminution to 80% passing 85 µm. Subject to successful resolution of any dust handling issues when crushing dry, potentially fibrous Turnagain materials, the suggested comminution process would involve crushing using high-pressure grinding rolls, followed by tumbling and/or stirred milling technologies to achieve the final primary grind size.

For this study, rougher flotation would employ conventional tank cell flotation technology, although future consideration should be given to newer flotation technologies (e.g., Woodgrove SFR technology) as a substitute for early-stage rougher flotation. This may lower capital costs, will likely lower operating costs, and may enhance final concentrate grades. In our experience, with slower floating sulphides tank cells are still needed after SFR flotation to ensure optimal recoveries. This requires testing prior to being included in the design, so the more conservative all tank cell option has been adopted for this study.

Relatively low pulp densities have been included, reflecting test results that have indicated that a more dilute pulp yields better rougher metallurgy. At the designed tonnages, this is an expensive option, so more testing and analysis are warranted. The residence time reflects laboratory residence times in an 8 L cell scaled up by a factor of 2.2, which is industry practice for such applications.

No regrinding is recommended at this stage. Regrinding is being tested, but no recent data have indicated any advantage can be gained from its use.

It is recommended that concentrate cleaning is conducted in three stages in the plant using conventional mechanical flotation cells throughout. Column or Jameson Cell flotation could offer advantages, but would need to be tested in a pilot plant, as for the most part they have not been successful commercially in nickel flotation (due to poor recoveries). Mechanical cells represent a safe selection, but using them may forego some upside in concentrate grades (although they will still match what is achieved in the laboratory). The residence time scale up factors for the first cleaner and cleaner scavenger circuit are average (1.8 x) and high (3 x) respectively. Past experience from benchmarking operating plants against lab or pilot plant cells has shown that slower floating material, as found in the cleaner scavenger flotation stage, requires a bigger scale-up factor to the plant due to the lower energy intensity employed commercially.

The second and third cleaner stage scale-up times are 3:1. This is quite high, but the higher residence time mitigates against the use of just three cells at each stage and resulting potential for short-circuiting to tails.

# 14.0 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

The purpose of this report is to document the resource estimations for the Turnagain deposit. This section describes the work undertaken by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd., and includes key assumptions and parameters used to prepare the mineral resource models for Northwest and Horsetail zones in addition to extensions of the Duffy and Hatzl zones, together with appropriate commentary regarding the merits and possible limitations of such assumptions.

The Mineral Resource Statement presented herein represents an updated mineral resource evaluation prepared for Giga Metals in accordance with the Canadian Securities Administrators' NI 43-101.

This section describes the mineral resource estimation methodology and summarises the key assumptions. In the opinion of qualified person Garth Kirkham, P.Geo., the mineral resource estimates reported herein are a reasonable representation of the mineral resources found within the project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted CIM guidelines ("Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines", December 2019) and are reported in accordance with NI 43-101 guidelines. <u>It is important to note that mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated</u>.

The mineral resource evaluation reported herein for Turnagain is current and supersedes earlier mineral resource estimates completed for Hard Creek Nickel Corp. including:

- Preliminary Economic Assessment Technical Report for the Turnagain Project, December 2011 (AMC, 2011).

The mineral resource estimates were prepared, reviewed and verified by Garth Kirkham, P.Geo., the independent qualified person for the mineral resource estimates included in this report. Giga Metals' field work on the Project from 2018, including drilling, was carried out under the supervision of Greg Ross, P.Geo., who is Giga Metals' senior geologist.

The general mineral resource estimation methodology for the deposit involved the following procedures:

- database verification and validation
- data exploration, compositing and evaluation of outliers
- construction of estimation domains

- spatial statistics
- block modelling and grade interpolation
- mineral resource classification and validation
- assessment of “reasonable prospects for eventual economic extraction”
- preparation of the mineral resource statement

## 14.2 Data

The 362 drill holes in the database were supplied in electronic format by Giga Metals, 307 of which had assay values. This included collars, downhole surveys, lithology data and assay data of varying vintages and analysis types.

Prior to 2004, samples were analysed 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 2018 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. Drill holes drilled prior to 2018 were also analysed by a ‘sulphide-specific, partial leach’ ammonium citrate-hydrogen peroxide method, followed by ICP-ES finish for Ni, Cu, Cu, Mg and S (the AC method).

In 2004 and 2005, sulphur content was analysed by the Leco furnace method. In 2006, sulphur content was analysed by ICP-ES after a four-acid digestion. Since 2007, sulphur content has been analysed by both ICP-ES after a four-acid digestion, and Leco furnace.

Some exploration drill holes prior to 2004, and all drill holes since 2004, were analysed for platinum, palladium, and gold by lead-collection fire-assay (FA) fusion followed by ICP-ES.

Table 14.1 lists the elements and analyses namely FA (Au, Pt, Pd), AC (Ni, Co, Cu, Mg), ICP (26 element) and LECO (S) along with a listing of ‘best’ values which refers to the final to be used when multiple vintages and methods are in the database. Table 14.1 shows the list of valid elements and analyses methods.

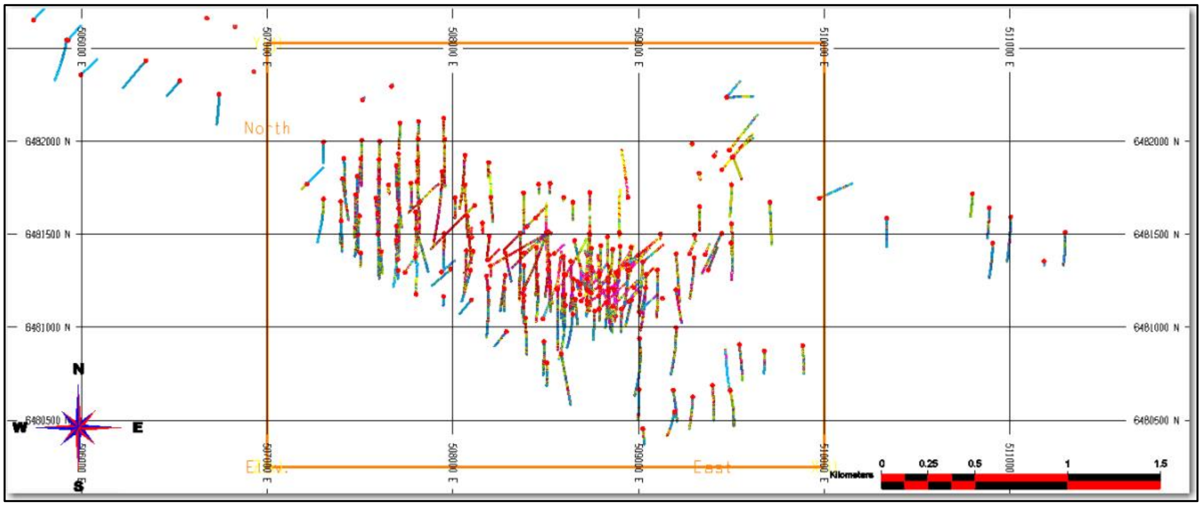
**Table 14.1: List of Analysis Elements & Method**

| ELEMENT | VALUE |
|---|---|
| Ag_ICP_ppm | x |
| Al_ICP_pct | x |
| As_ICP_ppm | x |
| Be_ICP_ppm | x |
| Bi_ICP_ppm | x |
| Ca_ICP_pct | x |
| Cd_ICP_ppm | x |
| Co_ICP_ppm | x |
| Cr_ICP_ppm | x |
| Cu_ICP_ppm | x |
| Fe_ICP_pct | x |
| K_ICP_pct | x |
| Mg_ICP_pct | x |
| Mn_ICP_ppm | x |
| Mo_ICP_ppm | x |
| Na_ICP_pct | x |
| Ni_ICP_pct | x |
| P_ICP_pct | x |
| Pb_ICP_ppm | x |
| S_ICP_pct | x |
| Sb_ICP_ppm | x |
| Sr_ICP_ppm | x |
| Ti_ICP_pct | x |
| V_ICP_ppm | x |
| W_ICP_ppm | x |
| Zn_ICP_ppm | x |

| ELEMENT | VALUE |
|---|---|
| Au_FA_ppm | x |
| Pd_FA_ppb | x |
| Pt_FA_ppb | x |
| S_LECO_pct | x |
| Ni_AC_pct | x |
| Co_AC_pct | x |
| Cu_AC_pct | x |
| Mg_AC_pct | x |
| S_Best_pct | x |

Validation and verification checks were performed during importation of data to ensure there were no overlapping intervals, typographic errors or anomalous entries. None were found. Figure 14-1 shows a plan view of the supplied drill holes.

**Figure 14-1: Plan View of Turnagain Drill Holes & Model Limits**




Source: Kirkham Geosystems, 2020.

## 14.3 Data Analysis

Statistics were run to evaluate the elements of primary potential economic and geometallurgical interest namely Ni%, NiAC%, Co%, CoAC%, Cu%, CuAC%, Mg, MgAC%, Fe%, Pt ppm, Pd ppm, S%, S% (Leco), Au and Ag. The primary economic contributor is shown to be nickel content whilst the secondary is cobalt. The relative concentrations of platinum, palladium, gold and silver are very low and considered not to be economic at this time and although not the subject of this resource estimate and not reported, they have been estimated. However, they may be payable depending upon mineral processing and concentrate treatment methods and terms. The NiAC%, Mg, MgAC%, Cu, CuAC%, CoAC%, Fe%, S% and S%(Leco) have similarly been analysed and estimated on a block by block basis which are useful from a geometallurgical standpoint. However, they are not reported within the resource statement.

The statistical analysis was grouped by lithology as logged by the site geologists and supplied in the database. The lithology codes and descriptions are listed in Table 14.2. There was only one occurrence of Bx (breccia) and SMS (semi-massive sulphide) so they are not listed.

Table 14.3 details the statistical analyses for Ni% and Co% for each of the individual lithologic units. Note that a large percentage of the data is associated with the Wehrlite, Dunite, Serpentinite, and Green Dunite lithologies which range in mean grades between 0.2% and 0.25% nickel followed by the Pyroxenites which are lower grade at approximately 0.11% - 0.2% nickel. However, the mean cobalt grades are consistently in the 0.011% - 0.014% range for all.

In addition, the coefficients of variability for nickel, cobalt and sulphur are all very low for all lithology units which indicates very low variability and risk. The coefficient of variation is defined as CV=σ/m (standard deviation/mean) and represents a measure of variability that is unit independent. This variability index can be used to compare different and unrelated distributions.

**Table 14.2: Lithology Codes & Descriptions for Statistical Grouping**

| Lithology | Description |
|---|---|
| cPx | Clinopyroxenite |
| CS | Calc-Silicate |
| Di | Diorite |
| Dk | Dyke |
| Du | Dunite |
| flt | Fault |
| gDi | Granodiorite |
| gDu | Green Dunite |
| GS | Graphite-Sulphide |
| Hb | Hornblendite |
| hbcPx | Hornblende Clinopyroxenite |
| Hfs | Hornfels |
| Inc | Inclusion |
| MS | Massive Sulphide |
| MSD | Metasediment |
| mtcPx | Magnetite Clinopyroxenite |
| MV | Metavolcanics |
| ocPx | Olivine Clinopyroxenite |
| Ovb | Overburden |
| Phy | Phyllite |
| Sp | Serpentinite |
| Um | Undifferentiated Ultramafic |
| Wh | Wehrlite |

Source: Kirkham Geosystems, 2020.

**Table 14.3: Statistics for Nickel, Cobalt & Sulphur**

| Lith | Lith Code | Value | Valid | Length (m) | Min | Max | Mean | SD | CV | Ni:S |
|---|---|---|---|---|---|---|---|---|---|---|
| cPx | 100 | NI | 1,987 | 3,648.4 | 0.001 | 1.051 | 0.074 | 0.087 | 1.2 | |
| | | CO | 1,987 | 3,648.4 | 0.001 | 0.077 | 0.010 | 0.005 | 0.5 | |
| | | S | 1,040 | 2,161.0 | 0.010 | 6.530 | **1.084** | 0.975 | 0.9 | 0.1 |
| CS | 50 | NI | 125 | 220.5 | 0.003 | 0.355 | 0.082 | 0.068 | 0.8 | |
| | | CO | 125 | 220.5 | 0.001 | 0.027 | 0.008 | 0.004 | 0.6 | |
| | | S | 46 | 116.6 | 0.020 | 1.360 | 0.441 | 0.401 | 0.9 | 0.2 |
| Di | 90 | NI | 398 | 995.7 | 0.000 | 0.313 | 0.034 | 0.048 | 1.4 | |
| | | CO | 398 | 995.7 | 0.000 | 0.016 | 0.005 | 0.002 | 0.5 | |
| | | S | 350 | 904.9 | 0.010 | 3.090 | 0.562 | 0.472 | 0.8 | 0.1 |
| Dk | 20 | NI | 452 | 849.7 | 0.001 | 0.604 | 0.071 | 0.078 | 1.1 | |
| | | CO | 452 | 849.7 | 0.001 | 0.032 | 0.006 | 0.004 | 0.7 | |
| | | S | 363 | 735.6 | 0.010 | 3.370 | 0.394 | 0.381 | 1.0 | 0.2 |
| Du | 110 | NI | 14,819 | 30,439.0 | 0.001 | 2.861 | **0.231** | 0.088 | 0.4 | |
| | | CO | 14,819 | 30,439.0 | 0.001 | 0.166 | 0.014 | 0.005 | 0.3 | |
| | | S | 12,046 | 26,201.7 | 0.010 | 12.040 | **0.469** | 0.561 | 1.2 | 0.5 |
| flt | 7 | NI | 515 | 1,057.9 | 0.005 | 0.581 | 0.163 | 0.079 | 0.5 | |
| | | CO | 515 | 1,057.9 | 0.001 | 0.026 | 0.011 | 0.004 | 0.4 | |
| | | S | 451 | 963.2 | 0.010 | 5.030 | 0.593 | 0.564 | 1.0 | 0.3 |
| qDi | 91 | NI | 18 | 35.2 | 0.001 | 0.024 | 0.005 | 0.006 | 1.1 | |
| | | CO | 18 | 35.2 | 0.001 | 0.005 | 0.002 | 0.001 | 0.5 | |
| | | S | 13 | 28.6 | 0.150 | 0.600 | 0.375 | 0.144 | 0.4 | 0.0 |
| gDu | 120 | NI | 2,053 | 4,021.9 | 0.001 | 0.750 | **0.250** | 0.053 | 0.2 | |
| | | CO | 2,053 | 4,021.9 | 0.001 | 0.030 | 0.013 | 0.002 | 0.2 | |
| | | S | 1,917 | 3,794.3 | 0.010 | 3.370 | **0.110** | 0.232 | 2.1 | 2.3 |
| GS | 30 | NI | 40 | 67.8 | 0.038 | 0.546 | **0.228** | 0.124 | 0.5 | |
| | | CO | 40 | 67.8 | 0.008 | 0.041 | 0.019 | 0.008 | 0.4 | |
| | | S | 30 | 56.7 | 1.340 | 8.100 | **3.764** | 1.970 | 0.5 | 0.1 |

| Lith | Lith Code | Value | Valid | Length (m) | Min | Max | Mean | SD | CV | Ni:S |
|---|---|---|---|---|---|---|---|---|---|---|
| Hb | 109 | NI | 914 | 1,761.8 | 0.001 | 0.210 | 0.019 | 0.026 | 1.4 | |
| | | CO | 914 | 1,761.8 | 0.002 | 0.033 | 0.007 | 0.003 | 0.4 | |
| | | S | 579 | 1,196.0 | 0.010 | 5.710 | 0.874 | 0.887 | 1.0 | 0.0 |
| hbcPx | 108 | NI | 772 | 1,604.2 | 0.002 | 0.188 | 0.020 | 0.020 | 1.0 | |
| | | CO | 772 | 1,604.2 | 0.003 | 0.033 | 0.007 | 0.003 | 0.4 | |
| | | S | 421 | 967.8 | 0.010 | 2.980 | 0.610 | 0.511 | 0.8 | 0.0 |
| Hfs | 51 | NI | 291 | 596.9 | 0.001 | 0.276 | 0.024 | 0.044 | 1.9 | |
| | | CO | 291 | 596.9 | 0.001 | 0.030 | 0.004 | 0.003 | 0.8 | |
| | | S | 217 | 453.3 | 0.050 | 4.580 | 1.156 | 0.903 | 0.8 | 0.0 |
| Inc | 8 | NI | 28 | 46.3 | 0.010 | 0.122 | 0.052 | 0.038 | 0.7 | |
| | | CO | 28 | 46.3 | 0.006 | 0.020 | 0.011 | 0.004 | 0.4 | |
| | | S | 27 | 45.7 | 0.010 | 6.690 | 2.464 | 2.068 | 0.8 | 0.0 |
| MGS | 31 | NI | 4 | 7.4 | 0.275 | 0.599 | **0.497** | 0.135 | 0.3 | |
| | | CO | 4 | 7.4 | 0.025 | 0.052 | 0.044 | 0.012 | 0.3 | |
| | | S | 4 | 7.4 | 1.730 | 6.570 | **4.943** | 2.106 | 0.4 | 0.1 |
| MSD | 52 | NI | 195 | 402.1 | 0.002 | 0.287 | 0.054 | 0.051 | 0.9 | |
| | | CO | 195 | 402.1 | 0.001 | 0.014 | 0.005 | 0.003 | 0.6 | |
| | | S | 191 | 399.9 | 0.010 | 8.390 | 1.410 | 1.714 | 1.2 | 0.0 |
| MS | 32 | NI | 6 | 11.0 | 0.038 | 1.987 | 0.286 | 0.425 | 1.5 | |
| | | CO | 6 | 11.0 | 0.003 | 0.149 | 0.026 | 0.032 | 1.3 | |
| | | S | 5 | 10.0 | 0.690 | 14.060 | 3.056 | 3.173 | 1.0 | 0.1 |
| mtcPx | 103 | NI | 1,560 | 2,683.2 | 0.001 | 0.140 | 0.024 | 0.014 | 0.6 | |
| | | CO | 1,560 | 2,683.2 | 0.001 | 0.024 | 0.007 | 0.003 | 0.4 | |
| | | S | 846 | 1,521.2 | 0.010 | 5.060 | 0.594 | 0.759 | 1.3 | 0.0 |
| MV | 17 | NI | 8 | 21.4 | 0.002 | 0.074 | 0.024 | 0.025 | 1.1 | |
| | | CO | 8 | 21.4 | 0.001 | 0.005 | 0.003 | 0.001 | 0.5 | |
| | | S | 8 | 21.4 | 0.330 | 0.790 | 0.547 | 0.156 | 0.3 | 0.0 |

| Lith | Lith Code | Value | Valid | Length (m) | Min | Max | Mean | SD | CV | Ni:S |
|---|---|---|---|---|---|---|---|---|---|---|
| ocPx | 101 | NI | 2,920 | 5,836.9 | 0.003 | 5.148 | **0.157** | 0.145 | 0.9 | |
| | | CO | 2,920 | 5,836.9 | 0.003 | 0.146 | 0.012 | 0.006 | 0.5 | |
| | | S | 2,184 | 4,726.8 | 0.010 | 11.550 | **0.962** | 0.908 | 0.9 | 0.2 |
| Ovb | 10 | NI | 15 | 21.2 | 0.080 | 0.303 | 0.204 | 0.073 | 0.4 | |
| | | CO | 15 | 21.2 | 0.010 | 0.021 | 0.015 | 0.004 | 0.2 | |
| | | S | 2 | 1.6 | 0.160 | 0.480 | 0.440 | 0.106 | 0.2 | 0.5 |
| oxPx | 102 | NI | 11 | 20.1 | 0.038 | 0.367 | **0.202** | 0.096 | 0.5 | |
| | | CO | 11 | 20.1 | 0.009 | 0.024 | 0.014 | 0.005 | 0.3 | |
| | | S | 8 | 16.0 | 0.310 | 1.130 | **0.660** | 0.296 | 0.4 | 0.3 |
| Phy | 53 | NI | 82 | 232.1 | 0.002 | 0.184 | 0.018 | 0.032 | 1.8 | |
| | | CO | 82 | 232.1 | 0.001 | 0.011 | 0.002 | 0.002 | 0.9 | |
| | | S | 82 | 232.1 | 0.110 | 2.870 | 1.071 | 0.757 | 0.7 | 0.0 |
| Sp | 115 | NI | 1,124 | 2,043.5 | 0.004 | 1.646 | **0.227** | 0.096 | 0.4 | |
| | | CO | 1,124 | 2,043.5 | 0.001 | 0.097 | 0.013 | 0.005 | 0.4 | |
| | | S | 799 | 1,564.1 | 0.010 | 3.670 | **0.395** | 0.465 | 1.2 | 0.6 |
| SMS | 33 | NI | 1 | 2.0 | 0.534 | 0.534 | 0.534 | 0.000 | 0.0 | |
| | | CO | 1 | 2.0 | 0.057 | 0.057 | 0.057 | 0.000 | 0.0 | |
| | | S | 1 | 2.0 | 7.670 | 7.670 | 7.670 | 0.000 | 0.0 | 0.1 |
| Um | 130 | NI | 53 | 129.3 | 0.036 | 0.192 | 0.097 | 0.035 | 0.4 | |
| | | CO | 53 | 129.3 | 0.005 | 0.013 | 0.009 | 0.002 | 0.3 | |
| | | S | 53 | 129.3 | 0.010 | 1.910 | 0.751 | 0.449 | 0.6 | 0.1 |
| Wh | 111 | NI | 10,644 | 21,957.0 | 0.001 | 2.587 | **0.224** | 0.099 | 0.4 | |
| | | CO | 10,644 | 21,957.0 | 0.001 | 0.141 | 0.014 | 0.005 | 0.4 | |
| | | S | 8,706 | 19,030.4 | 0.010 | 11.810 | **0.614** | 0.665 | 1.1 | 0.4 |
| Total | Total | NI | 39,078 | 78,806.4 | 0.000 | 5.148 | 0.192 | 0.116 | 0.6 | |
| | | CO | 39,078 | 78,806.4 | 0.000 | 0.166 | 0.013 | 0.005 | 0.4 | |
| | | S | 30,406 | 65,333.0 | 0.010 | 14.060 | 0.578 | 0.703 | 1.2 | 0.3 |

Source: Kirkham Geosystems, 2020.

The box plots illustrate the various lithologic units and their statistical relationship to each other. The box plots also show that there are grade similarities that justify the grouping of particular lithologic units. Therefore, it is acceptable to treat them in a similar manner both geologically and statistically. Box plots for all lithologic units are shown in Figure 14-2 through Figure 14-4 for nickel, cobalt and sulphur, respectively. The lithology units displayed represent greater than 92% of the samples and do not list all units as shown in Table 14.3 for the sake of brevity and significance. These include the dunite, wehrlite, green dunite, serpentinite and pyroxenites.

In order to evaluate lithologies in further granularity for the purpose of determining if there are logical, justifiable groupings, dunite, green dunite, wehrlite and serpentinite are shown in Figure 14-5 through Figure 14-7 for nickel, cobalt and sulphur, respectively. Nickel and cobalt box plots show a very good correlation and support for grouping however the sulphur makes it is clear that the green dunite should be estimated separately.

In addition, the clinopyroxenite, olivine clinopyroxenite, magnetite clinopyroxenite and hornblende clinopyroxenite were also statistically evaluated to determine if there are logical, justifiable groupings. The box plots are shown in Figure 14-8 through Figure 14-10 for nickel, cobalt and sulphur, respectively. It appears that there is no justification to separate the sub-groups during estimation.

To further support the statistical groupings during estimation of nickel and sulphur, a useful analysis is the comparison of the nickel-to-sulphur ratio against the nickel grades by lithologic unit as shown in the plot in Figure 14-11. It is clear that the dunite, wehrlite and serpentinite are very similar with the clinopyroxenites having lesser but similar characteristics. It is very clear that the green dunite is statistically different from all other zones and as such understandable that it is estimated separately.

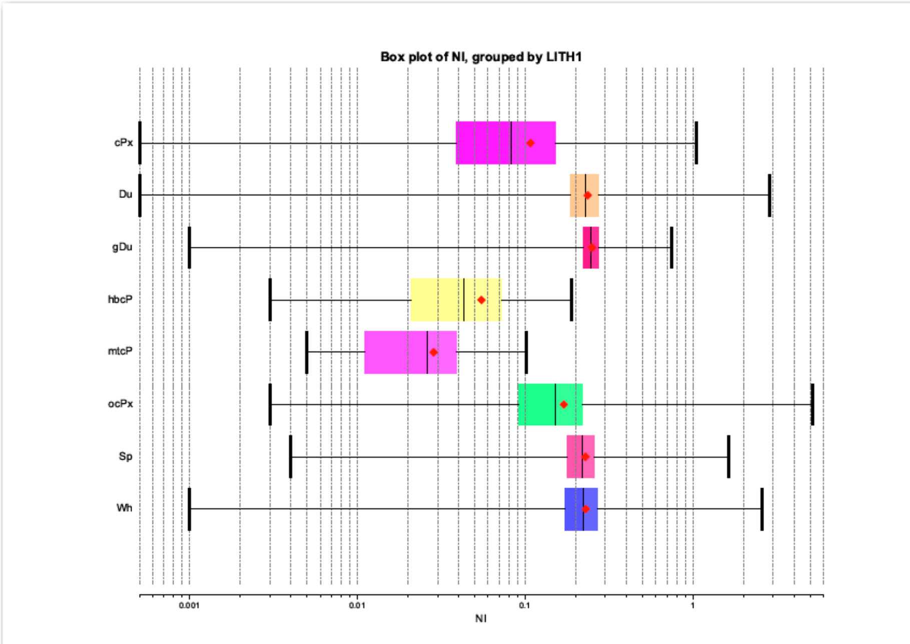
## 14.4 Geology Model

The mineral resource estimate is based on the validated drill hole database, interpreted three-dimensional geological model, digitised as-built data of historical workings, and topographic data. The geologic modelling was completed using the commercially available software Seequent Leapfrog Geo 4.3. The estimation of mineral resources was completed using commercial three-dimensional block modelling and mine planning software Hexagon Minesight$^{TM}$ MS3D Version 15.50.

Solid models (Figure 14-12) were created from coded drill hole intersections based primarily on lithology and site knowledge. It is important to note that the understanding and interpretation has evolved relatively flat lying units intruded by late dykes and sub-volcanics.

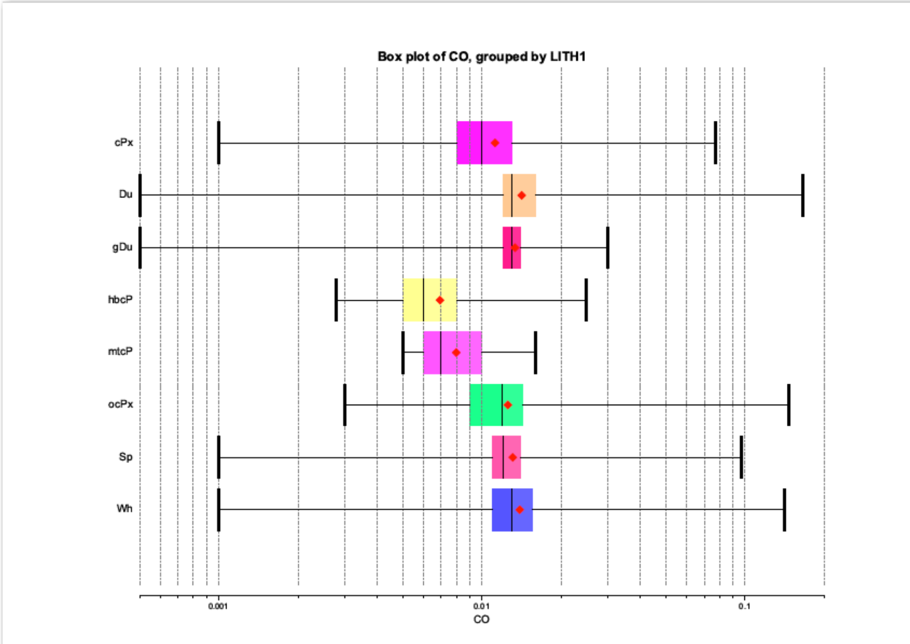
The domain models were modelled based on current drilling and assay data using Seequent Leapfrog Geo 4.3 and then imported into Minesight$^{TM}$ MS3D Version 15.50 for interpretation and refinement.

**Figure 14-2: Box Plot of Nickel Composites by Lithology**



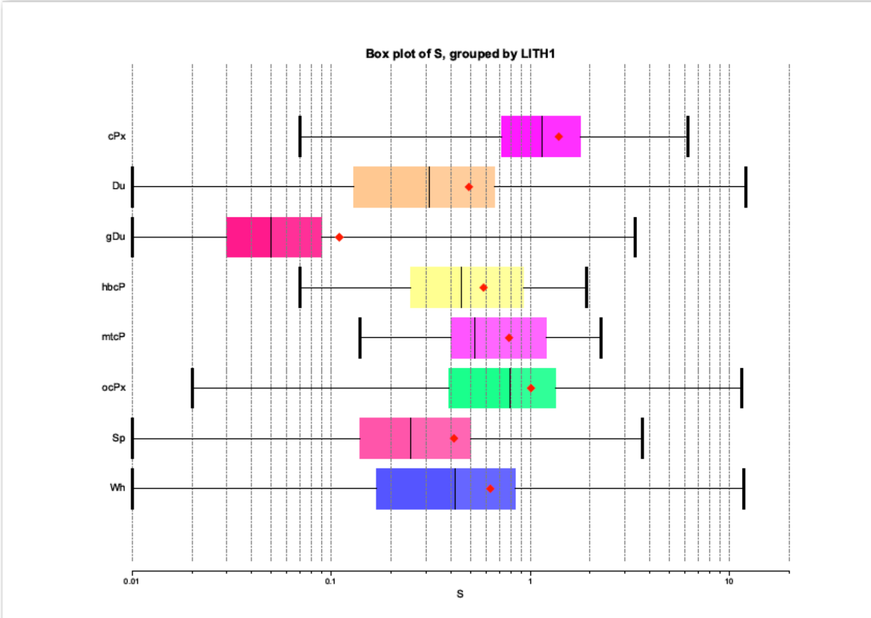

Source: Kirkham Geosystems, 2020.

**Figure 14-3: Box Plot of Cobalt Composites by Lithology**



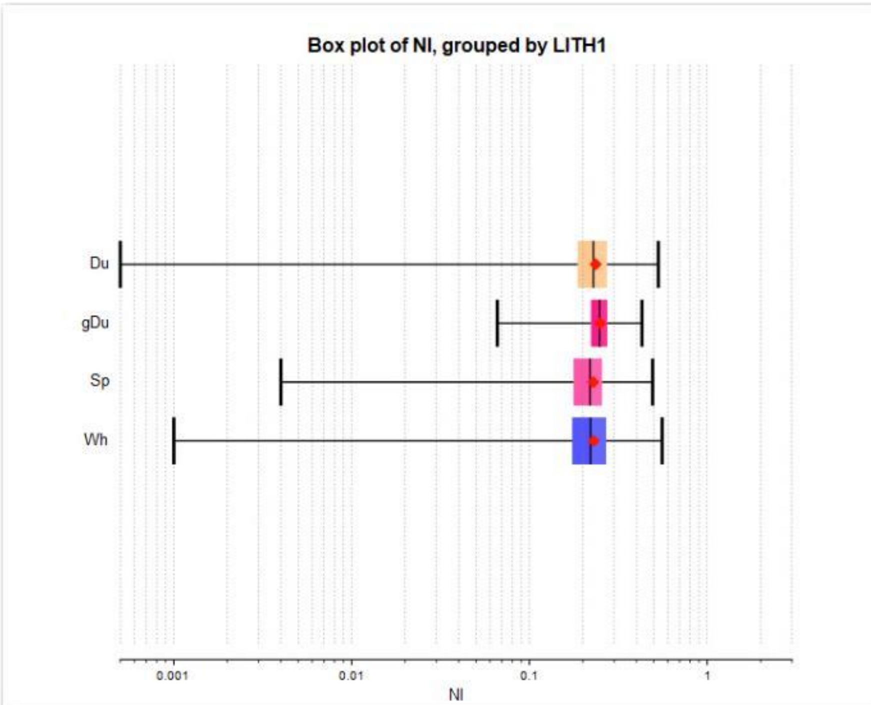

Source: Kirkham Geosystems, 2020.

**Figure 14-4: Box Plot of Sulphur Composites by Lithology**



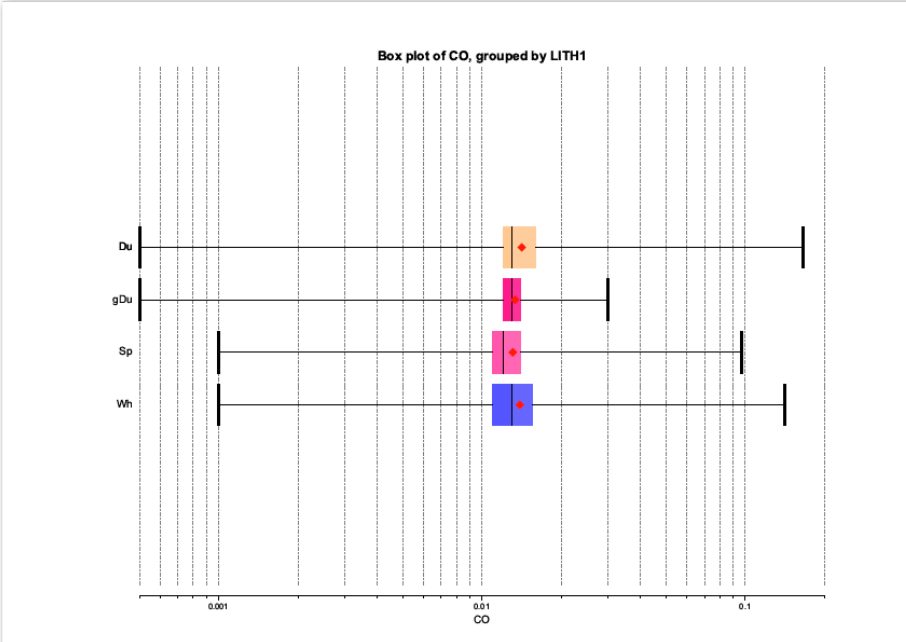

Source: Kirkham Geosystems, 2020.

**Figure 14-5: Box Plot of Nickel Composites for Dunite, Green Dunite, Wehrlite, Serpentinite**



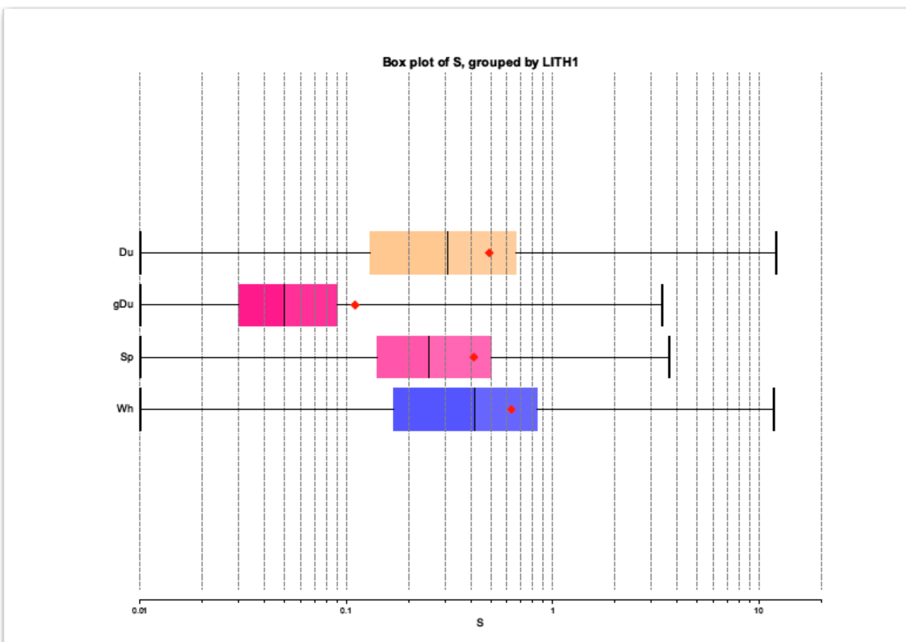

Source: Kirkham Geosystems, 2020.

**Figure 14-6: Box Plot of Cobalt Composites for Dunite, Green Dunite, Wehrlite, Serpentinite**



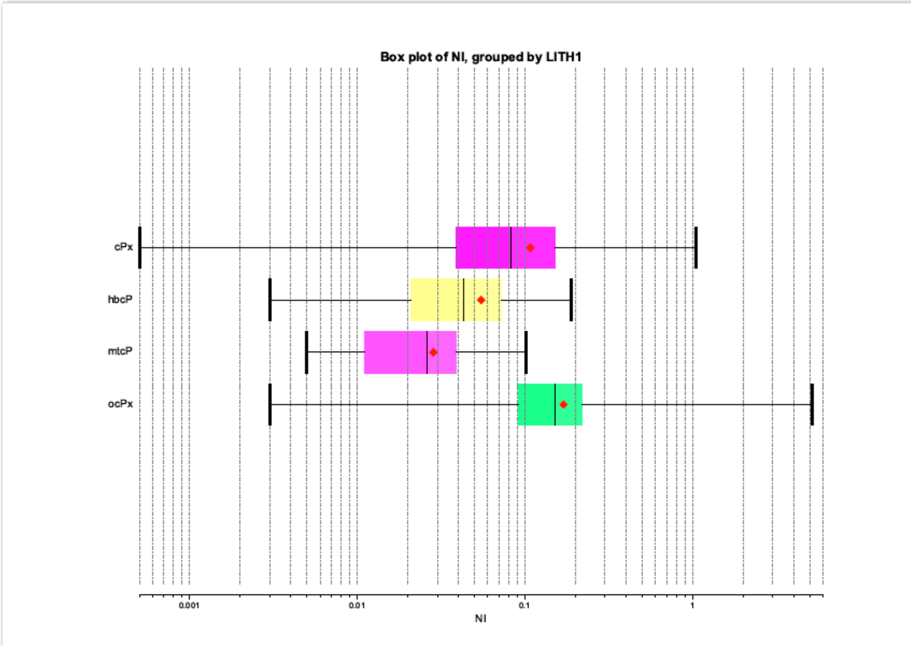

Source: Kirkham Geosystems, 2020.

**Figure 14-7: Box Plot of Sulphur Composites for Dunite, Green Dunite, Wehrlite, Serpentinite**



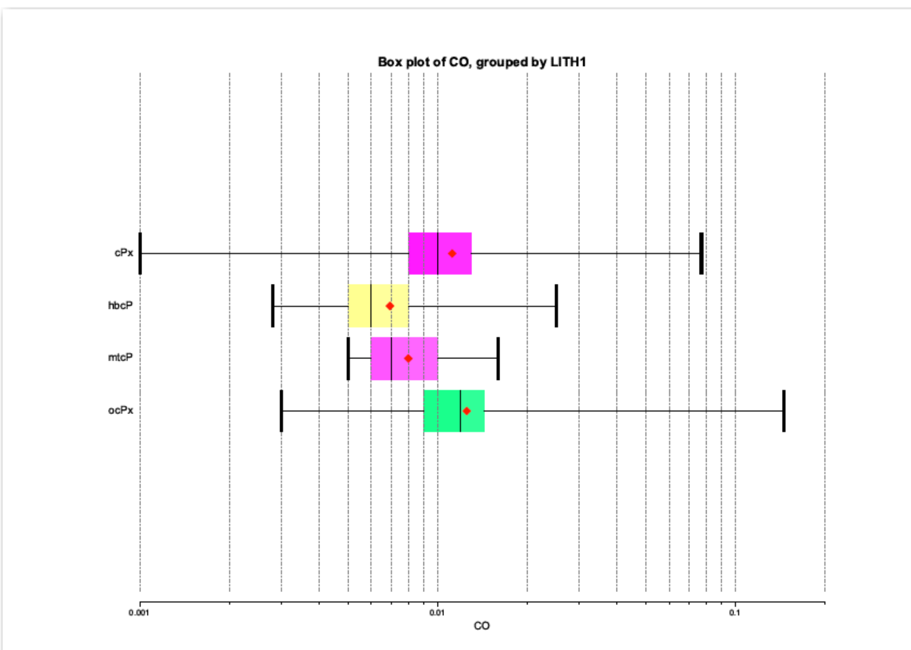

Source: Kirkham Geosystems, 2020.

**Figure 14-8: Box Plot of Nickel Composites by Lithology Unit**



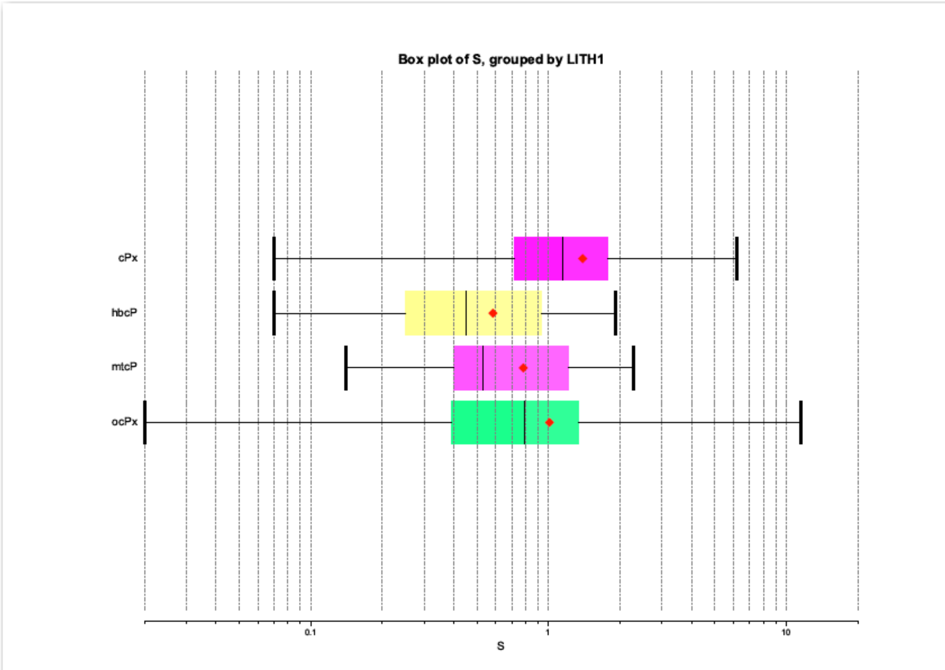

Source: Kirkham Geosystems, 2020.

**Figure 14-9: Box Plot of Cobalt Composites by Lithology Unit**



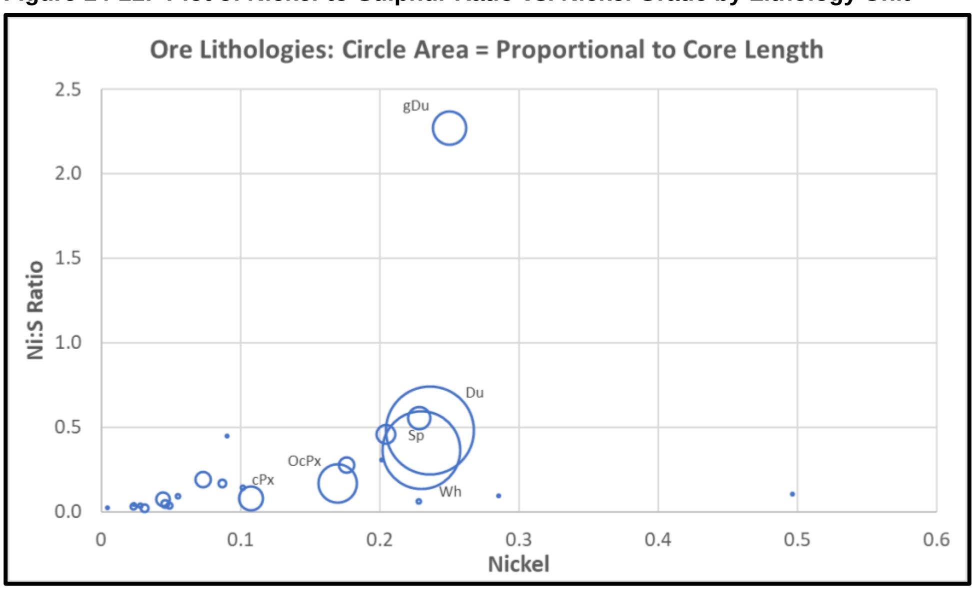

Source: Kirkham Geosystems, 2020.

**Figure 14-10: Box Plot of Sulphur Composites by Lithology**



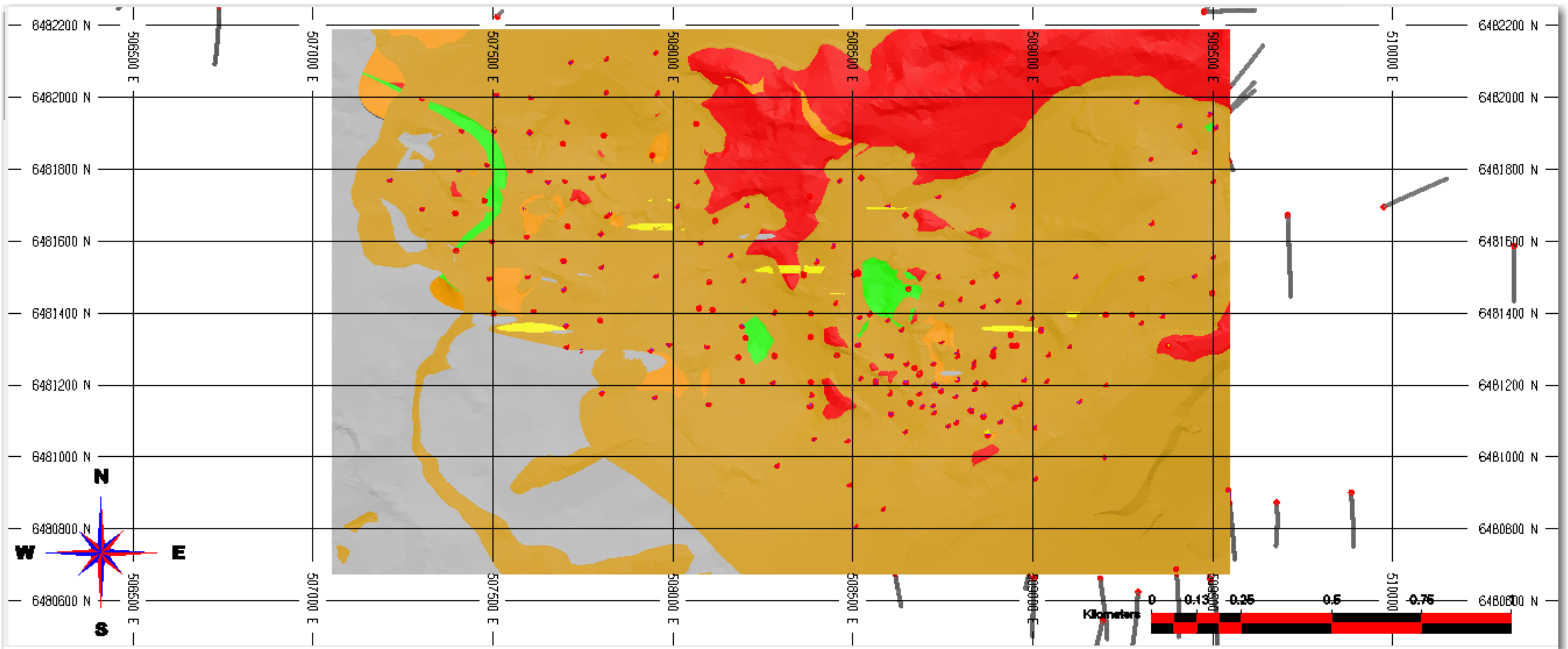

Source: Kirkham Geosystems, 2020.

**Figure 14-11: Plot of Nickel-to-Sulphur Ratio vs. Nickel Grade by Lithology Unit**




Source: Kirkham Geosystems, 2020.

**Figure 14-12: Plan View of Turnagain Deposit Geology Domains & Drill Holes**



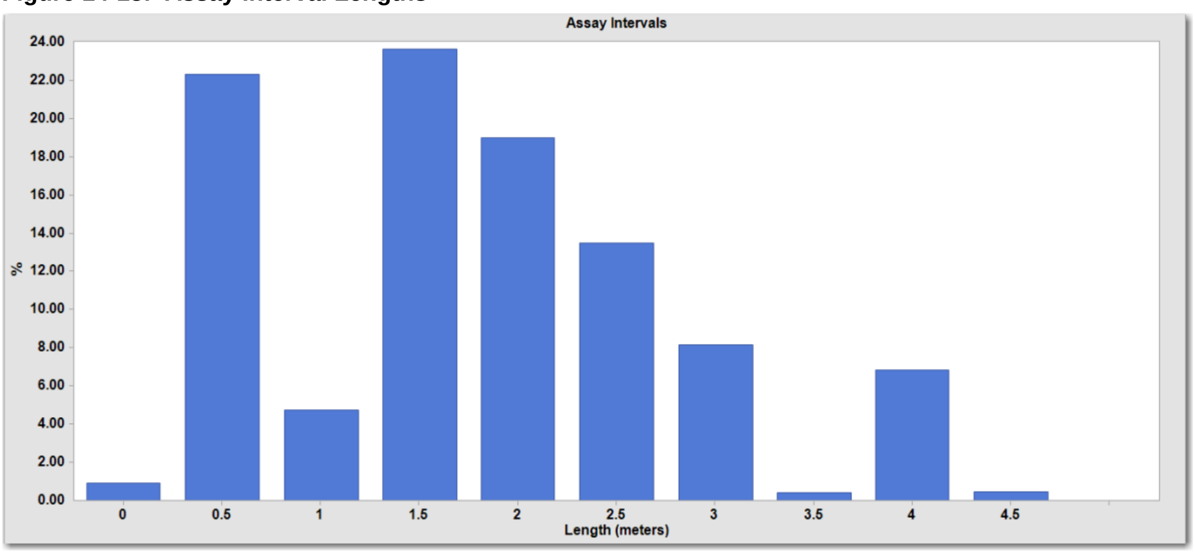

Notes: Grey = volcanics; brown = overburden; red = Wh-Du-Sp; green = green dunite; yellow = dykes; orange = pyroxenes. Source: Kirkham Geosystems, 2020.

The database was numerically coded by solids for the various zones. Intersections were inspected to ensure approximate agreement with the solids and then manually adjusted to match the drill intercepts where required. Once the solid model was created, it was used to code the drill hole assays and composites for subsequent statistical and geostatistical analyses. The solid zones were used to constrain the block model by matching assays to those within the zones. The orientation and ranges (distances) used for search ellipsoids in the estimation process were derived from strike and dip of the mineralised zone, site knowledge and on-site observations by Giga Metals geological staff. It is important to note that the block model is coded with the solids on a whole block majority basis which results in smoothing of the coded blocks and exclusion of thin stringers.

## 14.5 Composites

It was determined that a 4.0 m composite length offered the best balance between supplying common support for samples and minimising the smoothing of the grades. The 4.0 m sample length also was consistent with the distribution of sample lengths within the mineralised domains as shown in the histogram of assay lengths in Figure 14-13.

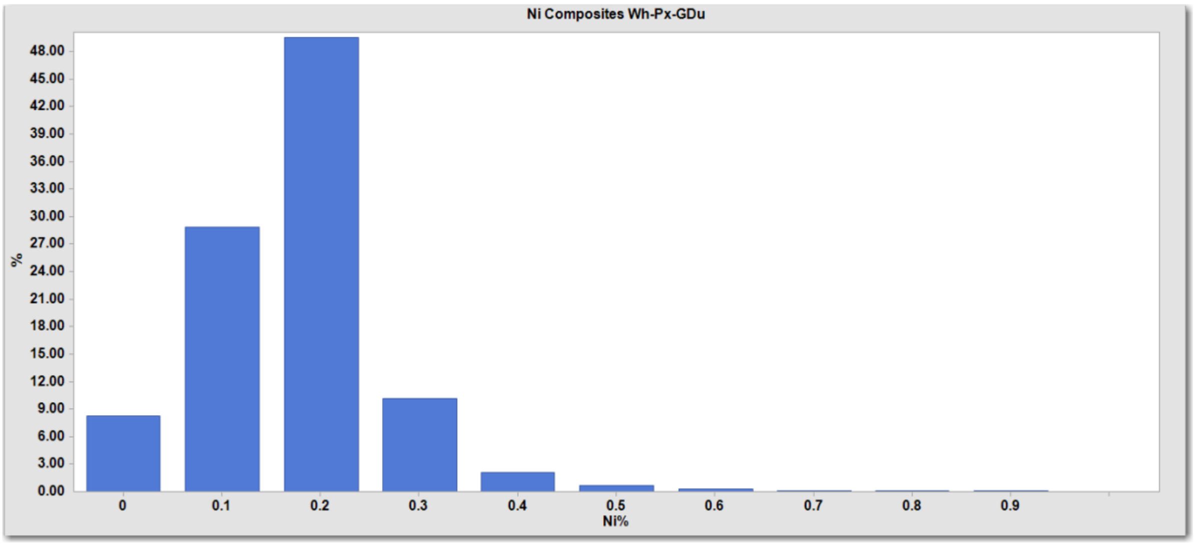
**Figure 14-13: Assay Interval Lengths**




Source: Kirkham Geosystems, 2020.

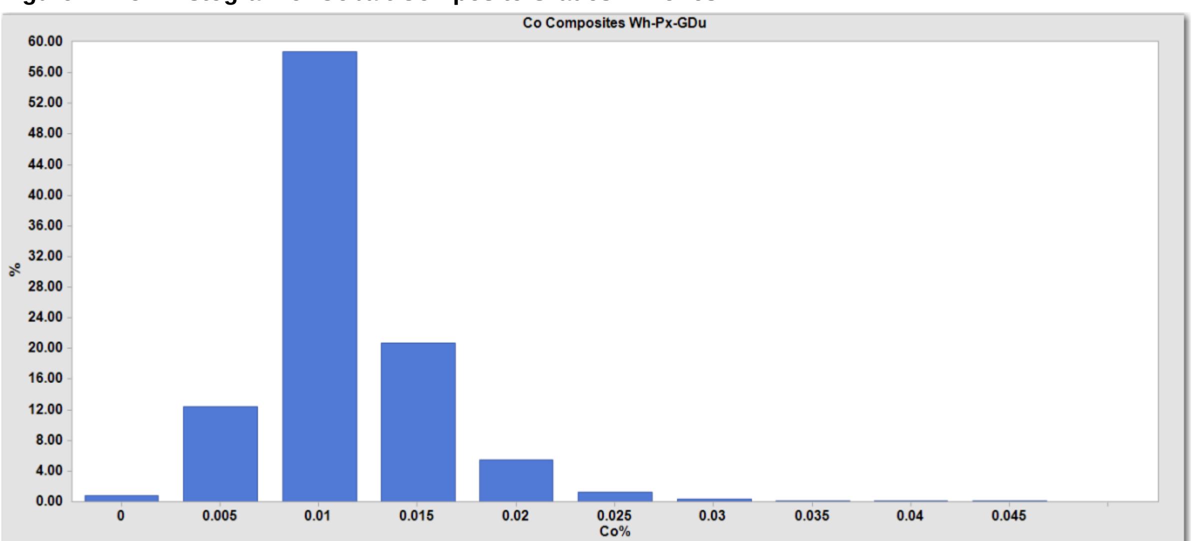
Figures 14-14 through 14-16 show the histograms for nickel, cobalt and sulphur, respectively, within the mineralised solids for all zones which demonstrate well-formed normal distribution for nickel and cobalt and log-normal distribution for sulphur.

**Figure 14-14: Histogram of Nickel Composite Grades in Zones**



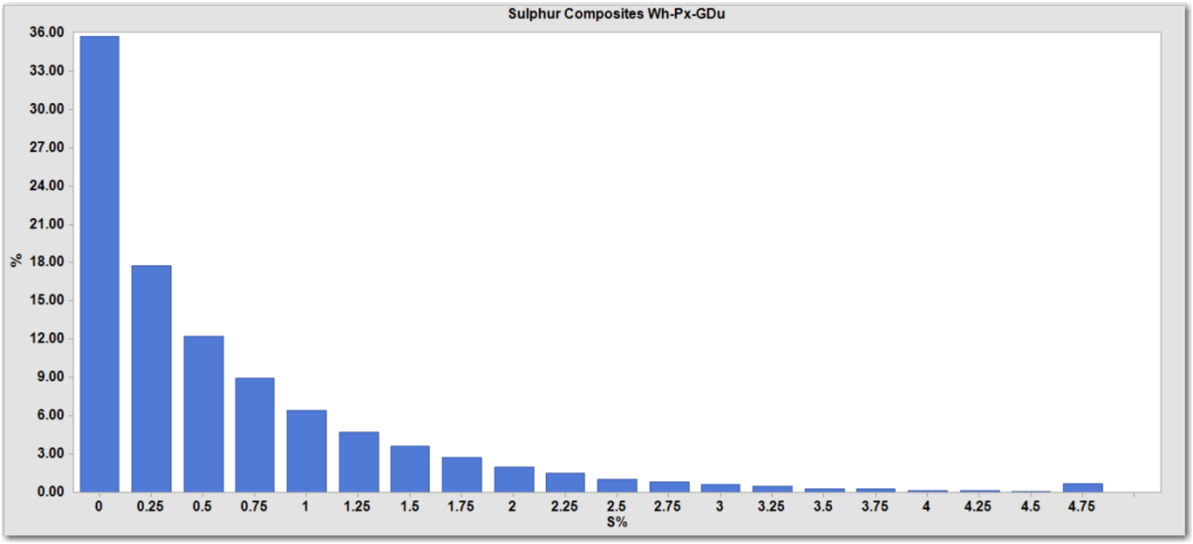

Source: Kirkham Geosystems, 2020.

**Figure 14-15: Histogram of Cobalt Composite Grades in Zones**




Source: Kirkham Geosystems, 2020.

**Figure 14-16: Histogram of Sulphur Composite Grades in Zones**




Source: Kirkham Geosystems, 2020.

Table 14.4 shows the basic statistics for the 4.0 m composite grades within the mineralised domains: (1) Du-Wh-Sp (dunite, wehrlite, serpentinite); (2) cPx-oPx (clinopyroxenite, olivine, magnetite and hornblende clinopyroxenite); (3) Volcanics; (4) Dykes; (5) Overburden. It should be noted that although 4.0 m is the composite length, any residual composites of lengths greater than 2.0 m and less than 4.0 m were retained to represent a composite, while any composite residuals less than 2.0 m were combined with the previous composite.

There is a total of 2,017 composites, with the averages (i.e., Ni%, NiAC%, Co%, CoAC%, Cu%, CuAC%, Mg, MgAC%, Fe%, Pt ppm, Pd ppm, S%, S% (Leco), Au, Ag), respectively, shown in Table 14.4.

**Table 14.4: Composite Statistics Weighted by Length**

| | Litho Zone | Valid | Count | Min | Max | Mean | SD | CV |
|---|---|---|---|---|---|---|---|---|
| NI | Wh-Du-Sp | 9,114 | 36,345.3 | 0.00 | 1.21 | 0.23 | 0.09 | 0.4 |
| | gDu | 3,132 | 12,483.2 | 0.01 | 2.73 | 0.24 | 0.10 | 0.4 |
| | cPx-oPx | 2,679 | 10,643.2 | 0.00 | 0.97 | 0.18 | 0.10 | 0.6 |
| | Dyke | 76 | 293.1 | 0.00 | 0.65 | 0.10 | 0.13 | 1.3 |
| | Volcanics | 184 | 725.4 | 0.00 | 0.32 | 0.07 | 0.08 | 1.1 |
| | Overburden | 43 | 113.7 | 0.09 | 0.40 | 0.24 | 0.08 | 0.3 |
| | **Total** | 15,228 | 60,603.9 | 0.00 | 2.73 | 0.22 | 0.10 | 0.4 |
| | **All** | 20,141 | 79,923.4 | 0.00 | 2.73 | 0.19 | 0.11 | 0.6 |
| NIAC | Wh-Du-Sp | 7,645 | 30,481.1 | 0.00 | 1.26 | 0.17 | 0.09 | 0.6 |
| | gDu | 2,773 | 11,053.8 | 0.00 | 2.79 | 0.13 | 0.10 | 0.8 |
| | cPx-oPx | 2,352 | 9,340.2 | 0.00 | 0.96 | 0.14 | 0.08 | 0.6 |
| | Dyke | 68 | 261.2 | 0.00 | 0.68 | 0.08 | 0.12 | 1.5 |
| | Volcanics | 114 | 447.6 | 0.00 | 0.27 | 0.07 | 0.07 | 1.0 |
| | Overburden | 33 | 79.9 | 0.04 | 0.29 | 0.17 | 0.07 | 0.4 |
| | **Total** | 12,985 | 51,663.8 | 0.00 | 2.79 | 0.15 | 0.10 | 0.6 |
| | **All** | 16,997 | 67,435.5 | 0.00 | 2.79 | 0.14 | 0.10 | 0.7 |
| CO | Wh-Du-Sp | 9,114 | 36,345.3 | 0.00 | 0.09 | 0.01 | 0.00 | 0.3 |
| | gDu | 3,132 | 12,483.2 | 0.001 | 0.071 | 0.013 | 0.004 | 0.3 |
| | cPx-oPx | 2,679 | 10,643.2 | 0.001 | 0.049 | 0.013 | 0.005 | 0.4 |
| | Dyke | 76 | 293.1 | 0.001 | 0.021 | 0.006 | 0.005 | 0.8 |
| | Volcanics | 184 | 725.4 | 0.000 | 0.023 | 0.006 | 0.004 | 0.6 |
| | Overburden | 43 | 113.7 | 0.006 | 0.024 | 0.015 | 0.003 | 0.2 |
| | **Total** | 15,228 | 60,603.9 | 0.000 | 0.086 | 0.014 | 0.005 | 0.3 |
| | **All** | 20,141 | 79,923.4 | 0.000 | 0.086 | 0.013 | 0.005 | 0.4 |
| COAC | Wh-Du-Sp | 7,645 | 30,481.1 | 0.001 | 0.088 | 0.009 | 0.005 | 0.6 |
| | gDu | 2,773 | 11,053.8 | 0.001 | 0.076 | 0.007 | 0.004 | 0.6 |
| | cPx-oPx | 2,352 | 9,340.2 | 0.001 | 0.042 | 0.009 | 0.005 | 0.5 |
| | Dyke | 68 | 261.2 | 0.001 | 0.015 | 0.004 | 0.004 | 1.0 |
| | Volcanics | 114 | 447.6 | 0.001 | 0.020 | 0.005 | 0.004 | 0.8 |
| | Overburden | 33 | 79.9 | 0.003 | 0.017 | 0.010 | 0.003 | 0.3 |
| | **Total** | 12,985 | 51,663.8 | 0.001 | 0.088 | 0.009 | 0.005 | 0.6 |
| | **All** | 16,997 | 67,435.5 | 0.001 | 0.088 | 0.008 | 0.005 | 0.6 |

| | Litho Zone | Valid | Count | Min | Max | Mean | SD | CV |
|---|---|---|---|---|---|---|---|---|
| CU | Wh-Du-Sp | 9,114 | 36,345.3 | 0.000 | 0.394 | 0.024 | 0.028 | 1.2 |
| | gDu | 3,132 | 12,483.2 | 0.000 | 0.271 | 0.017 | 0.025 | 1.5 |
| | cPx-oPx | 2,679 | 10,643.2 | 0.000 | 0.374 | 0.025 | 0.029 | 1.1 |
| | Dyke | 76 | 293.1 | 0.001 | 0.054 | 0.013 | 0.013 | 1.0 |
| | Volcanics | 184 | 725.4 | 0.002 | 0.090 | 0.016 | 0.010 | 0.7 |
| | Overburden | 43 | 113.7 | 0.000 | 0.076 | 0.023 | 0.019 | 0.9 |
| | **Total** | 15,228 | 60,603.9 | 0.000 | 0.394 | 0.022 | 0.027 | 1.2 |
| | **All** | 20,141 | 79,923.4 | 0.000 | 0.414 | 0.023 | 0.030 | 1.3 |
| CUAC | Wh-Du-Sp | 7,645 | 30,481.1 | 0.001 | 0.362 | 0.023 | 0.027 | 1.2 |
| | gDu | 2,773 | 11,053.8 | 0.001 | 0.251 | 0.016 | 0.024 | 1.5 |
| | cPx-oPx | 2,352 | 9,340.2 | 0.001 | 0.331 | 0.023 | 0.024 | 1.0 |
| | Dyke | 68 | 261.2 | 0.001 | 0.051 | 0.013 | 0.012 | 1.0 |
| | Volcanics | 114 | 447.6 | 0.003 | 0.044 | 0.016 | 0.008 | 0.5 |
| | Overburden | 33 | 79.9 | 0.002 | 0.065 | 0.025 | 0.018 | 0.7 |
| | **Total** | 12,985 | 51,663.8 | 0.001 | 0.362 | 0.022 | 0.026 | 1.2 |
| | **All** | 16,997 | 67,435.5 | 0.001 | 0.419 | 0.022 | 0.028 | 1.3 |
| MG | Wh-Du-Sp | 9,112 | 36,337.3 | 1.38 | 35.16 | 23.36 | 4.68 | 0.2 |
| | gDu | 3,132 | 12,483.2 | 0.73 | 33.71 | 24.54 | 4.52 | 0.2 |
| | cPx-oPx | 2,679 | 10,643.2 | 2.53 | 33.48 | 20.38 | 5.58 | 0.3 |
| | Dyke | 76 | 293.1 | 0.45 | 28.89 | 10.34 | 8.96 | 0.9 |
| | Volcanics | 184 | 725.4 | 0.47 | 26.58 | 9.69 | 6.58 | 0.7 |
| | Overburden | 43 | 113.7 | 3.55 | 31.34 | 24.33 | 3.47 | 0.1 |
| | **Total** | 15,226 | 60,595.9 | 0.45 | 35.16 | 22.85 | 5.33 | 0.2 |
| | **All** | 20,139 | 79,915.4 | 0.01 | 35.16 | 20.99 | 6.66 | 0.3 |
| MGAC | Wh-Du-Sp | 6,612 | 26,395.7 | 0.04 | 9.5 | 2.1 | 1.1 | 0.5 |
| | gDu | 2,442 | 9,732.4 | 0.01 | 7.4 | 2.4 | 1.1 | 0.5 |
| | cPx-oPx | 1,970 | 7,826.8 | 0.04 | 5.6 | 1.5 | 1.1 | 0.8 |
| | Dyke | 46 | 181.1 | 0.02 | 5.2 | 0.8 | 1.2 | 1.4 |
| | Volcanics | 113 | 444.1 | 0.05 | 4.9 | 0.7 | 0.8 | 1.3 |
| | Overburden | 25 | 58.9 | 0.48 | 3.6 | 1.6 | 0.7 | 0.5 |
| | **Total** | 11,208 | 44,638.9 | 0.01 | 9.5 | 2.0 | 1.1 | 0.6 |
| | **All** | 14,985 | 59,516.9 | 0.01 | 9.5 | 1.8 | 1.2 | 0.7 |

| | Litho Zone | Valid | Count | Min | Max | Mean | SD | CV |
|---|---|---|---|---|---|---|---|---|
| S | Wh-Du-Sp | 9,114 | 36,345.3 | 0.01 | 11.53 | 0.72 | 0.87 | 1.2 |
| | gDu | 3,132 | 12,483.2 | 0.01 | 13.14 | 0.47 | 0.72 | 1.5 |
| | cPx-oPx | 2,679 | 10,643.2 | 0.01 | 9.33 | 1.11 | 1.03 | 0.9 |
| | Dyke | 76 | 293.1 | 0.01 | 1.45 | 0.37 | 0.30 | 0.8 |
| | Volcanics | 184 | 725.4 | 0.09 | 3.64 | 0.91 | 0.79 | 0.9 |
| | Overburden | 43 | 113.7 | 0.01 | 2.27 | 0.63 | 0.51 | 0.8 |
| | **Total** | 15,228 | 60,603.9 | 0.01 | 13.14 | 0.74 | 0.89 | 1.2 |
| | **All** | 20,141 | 79,923.4 | 0.01 | 13.14 | 0.71 | 0.88 | 1.2 |
| SICP | Wh-Du-Sp | 9,044 | 36,064.9 | 0.01 | 11.53 | 0.64 | 0.74 | 1.2 |
| | gDu | 3,126 | 12,458.2 | 0.01 | 6.97 | 0.42 | 0.60 | 1.4 |
| | cPx-oPx | 2,671 | 10,613.2 | 0.01 | 9.33 | 1.00 | 0.92 | 0.9 |
| | Dyke | 76 | 293.1 | 0.01 | 1.20 | 0.34 | 0.26 | 0.8 |
| | Volcanics | 184 | 725.4 | 0.09 | 3.32 | 0.84 | 0.70 | 0.8 |
| | Overburden | 42 | 109.6 | 0.01 | 2.27 | 0.62 | 0.47 | 0.8 |
| | **Total** | 15,143 | 60,264.4 | 0.01 | 11.53 | 0.66 | 0.77 | 1.2 |
| | **All** | 20,056 | 79,583.9 | 0.01 | 11.53 | 0.64 | 0.76 | 1.2 |
| SLECO | Wh-Du-Sp | 6,838 | 27,264.8 | 0.01 | 11.53 | 0.78 | 0.94 | 1.2 |
| | gDu | 2,339 | 9,323.5 | 0.01 | 13.14 | 0.52 | 0.79 | 1.5 |
| | cPx-oPx | 2,131 | 8,471.3 | 0.01 | 9.33 | 1.20 | 1.07 | 0.9 |
| | Dyke | 58 | 219.9 | 0.01 | 1.45 | 0.40 | 0.33 | 0.8 |
| | Volcanics | 138 | 542.5 | 0.11 | 3.64 | 1.05 | 0.85 | 0.8 |
| | Overburden | 33 | 85.2 | 0.01 | 2.13 | 0.60 | 0.56 | 0.9 |
| | **Total** | 11,537 | 45,907.1 | 0.01 | 13.14 | 0.81 | 0.96 | 1.2 |
| | **All** | 14,438 | 57,337.4 | 0.01 | 13.14 | 0.78 | 0.95 | 1.2 |
| FE | Wh-Du-Sp | 9,114 | 36,345.3 | 0.77 | 26.50 | 7.95 | 1.74 | 0.2 |
| | gDu | 3,132 | 12,483.2 | 0.46 | 22.82 | 7.48 | 1.77 | 0.2 |
| | cPx-oPx | 2,679 | 10,643.2 | 4.18 | 19.31 | 8.52 | 1.80 | 0.2 |
| | Dyke | 76 | 293.1 | 1.95 | 11.00 | 5.91 | 2.38 | 0.4 |
| | Volcanics | 184 | 725.4 | 1.93 | 13.45 | 7.67 | 2.19 | 0.3 |
| | Overburden | 43 | 113.7 | 0.79 | 12.23 | 7.81 | 1.38 | 0.2 |
| | **Total** | 15,228 | 60,603.9 | 0.46 | 26.50 | 7.94 | 1.80 | 0.2 |
| | **All** | 20,141 | 79,923.4 | 0.01 | 26.50 | 8.02 | 2.00 | 0.2 |

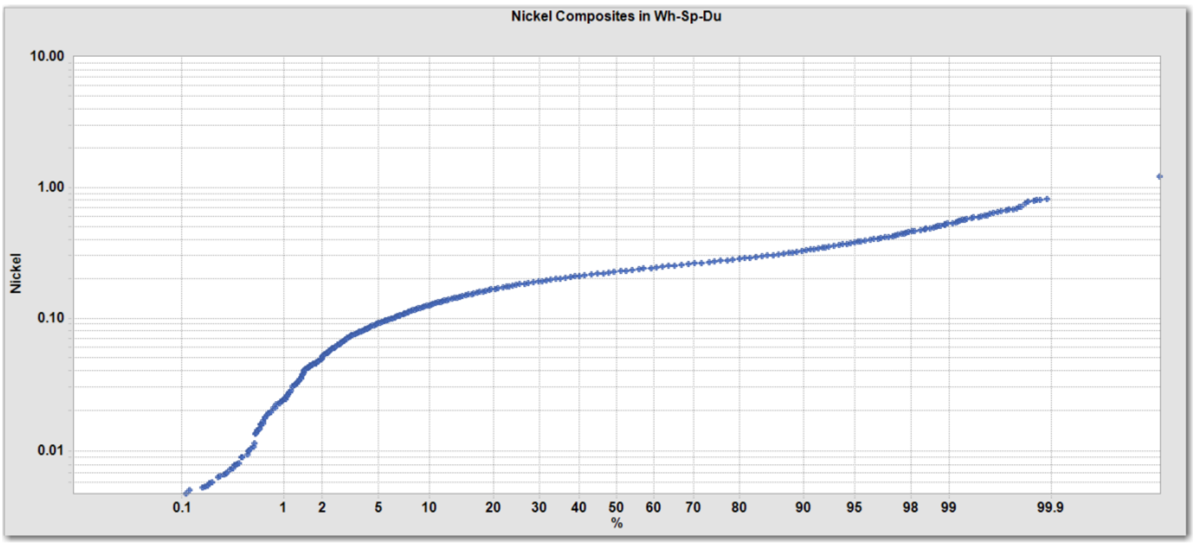
| | Litho Zone | Valid | Count | Min | Max | Mean | SD | CV |
|---|---|---|---|---|---|---|---|---|
| PT | Wh-Du-Sp | 8,203 | 32,720.2 | 1 | 1,054 | 22 | 32 | 1.4 |
| | gDu | 2,808 | 11,188.4 | 1 | 647 | 21 | 34 | 1.6 |
| | cPx-oPx | 2,406 | 9,566.3 | 1 | 213 | 18 | 23 | 1.2 |
| | Dyke | 66 | 253.0 | 1 | 198 | 16 | 32 | 2.0 |
| | Volcanics | 168 | 659.8 | 1 | 171 | 11 | 16 | 1.4 |
| | Overburden | 37 | 95.1 | 2 | 65 | 23 | 13 | 0.6 |
| | **Total** | 13,688 | 54,482.8 | 1 | 1,054 | 21 | 31 | 1.5 |
| | **All** | 18,592 | 73,769.7 | 1 | 1,974 | 24 | 45 | 1.9 |
| PD | Wh-Du-Sp | 8,441 | 33,669.7 | 1 | 1,192 | 23 | 33 | 1.4 |
| | gDu | 2,883 | 11,489.4 | 1 | 707 | 21 | 35 | 1.7 |
| | cPx-oPx | 2,460 | 9,780.6 | 1 | 284 | 20 | 23 | 1.2 |
| | Dyke | 66 | 253.0 | 1 | 201 | 16 | 34 | 2.1 |
| | Volcanics | 183 | 721.4 | 1 | 125 | 10 | 13 | 1.2 |
| | Overburden | 37 | 95.1 | 1 | 98 | 24 | 16 | 0.6 |
| | **Total** | 14,070 | 56,009.2 | 1 | 1,192 | 22 | 32 | 1.4 |
| | **All** | 18,980 | 75,320.0 | 1 | 1,602 | 25 | 43 | 1.7 |
| AU | Wh-Du-Sp | 7,668 | 30,590.2 | 0.001 | 1.525 | 0.005 | 0.023 | 4.4 |
| | gDu | 2,677 | 10,663.9 | 0.001 | 0.164 | 0.004 | 0.008 | 1.9 |
| | cPx-oPx | 2,283 | 9,071.8 | 0.001 | 0.219 | 0.005 | 0.008 | 1.7 |
| | Dyke | 59 | 225.1 | 0.001 | 0.044 | 0.004 | 0.006 | 1.8 |
| | Volcanics | 149 | 586.3 | 0.001 | 0.018 | 0.002 | 0.002 | 1.0 |
| | Overburden | 33 | 81.7 | 0.001 | 0.027 | 0.005 | 0.007 | 1.5 |
| | **Total** | 12,869 | 51,219.1 | 0.001 | 1.525 | 0.005 | 0.018 | 3.8 |
| | **All** | 17,541 | 69,582.9 | 0.001 | 1.525 | 0.005 | 0.017 | 3.8 |
| AG | Wh-Du-Sp | 7,828 | 31,210.8 | 0.27 | 26.4 | 1.1 | 0.6 | 0.6 |
| | gDu | 2,786 | 11,105.8 | 0.15 | 29.0 | 1.1 | 0.7 | 0.6 |
| | cPx-oPx | 2,400 | 9,530.6 | 0.5 | 17.9 | 1.1 | 0.5 | 0.5 |
| | Dyke | 68 | 261.2 | 1 | 2.0 | 1.0 | 0.2 | 0.2 |
| | Volcanics | 143 | 560.5 | 0.5 | 3.0 | 1.1 | 0.3 | 0.3 |
| | Overburden | 33 | 79.9 | 0.15 | 2.0 | 1.0 | 0.2 | 0.2 |
| | **Total** | 13,258 | 52,748.8 | 0.15 | 29.0 | 1.1 | 0.6 | 0.6 |
| | **All** | 17,901 | 70,993.3 | 0.15 | 152.0 | 1.1 | 1.4 | 1.2 |

Source: Kirkham Geosystems, 2020.

## 14.6 Evaluation of Outlier Assay Values

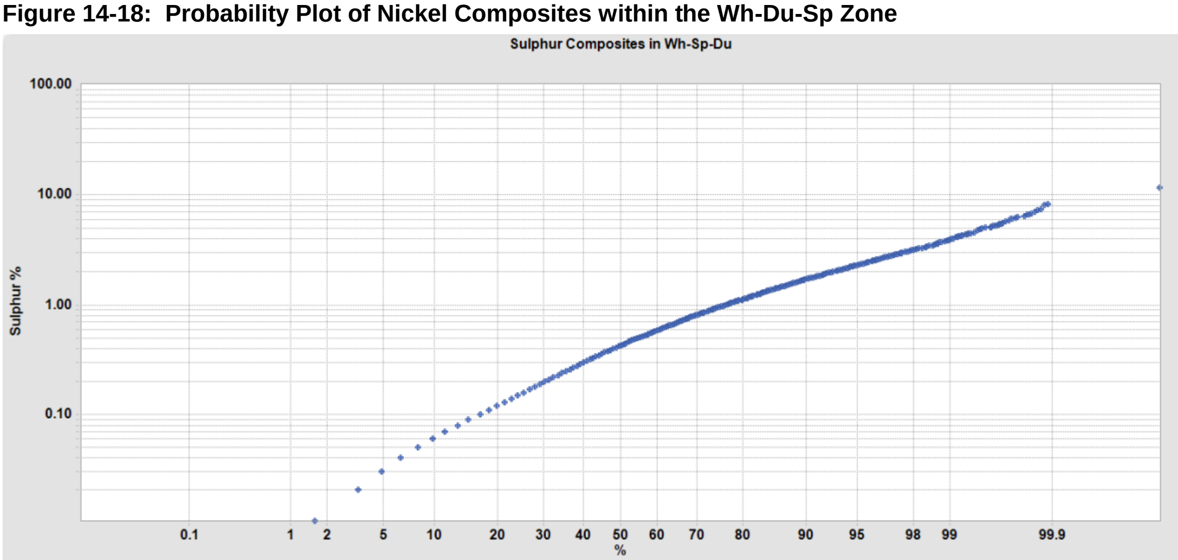
An evaluation of the probability plot of nickel and sulphur composites suggests there are no outlier values that could result in an overestimation of resources. Figure 14-17 and 14-18 illustrate that there are no distinct "breaks" in the cumulative frequency plots with which to determine the potential of an outlier population.

**Figure 14-17: Probability Plot of Nickel Composites within the Wh-Du-Sp Zone**




Source: Kirkham Geosystems, 2020.

**Figure 14-18: Probability Plot of Nickel Composites within the Wh-Du-Sp Zone**




Source: Kirkham Geosystems, 2020.

## 14.7 Bulk Density Estimation

Bulk density measurements were carried out on 1,318 core samples collected between 2004 and 2018. 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:

$$\textit{Bulk density} = [\textit{weight in air}/(\textit{weight in air} - \textit{weight in water})] * 1\ t/m^3$$

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 1,318 measurements were done by the immersion method on whole core and 312 measurements carried out on crushed samples using the pycnometric procedure.

Bulk densities were based on 1,630 measurements taken by personnel throughout the zones. These density values ranged from 2.24 t/m$^3$ to 5.36 t/m$^3$**.** Density was assigned to the model blocks based on the mean value for the corresponding lithology, as listed in Table 14.5.

**Table 14.5: Bulk Density Statistics by Zone**

| | | | Valid | Min | Max | Mean | SD | CV |
|---|---|---|---|---|---|---|---|---|
| SG | **1** | **DuWhSp** | 997 | 1.81 | 4.32 | 3.05 | 0.20 | 0.1 |
| | **2** | **gDu** | 276 | 2.64 | 4.49 | 3.08 | 0.22 | 0.1 |
| | **3** | **cPx/ocPx** | 313 | 2.24 | 5.36 | 3.07 | 0.28 | 0.1 |
| | **8** | **Dyke** | 11 | 2.71 | 3.21 | 2.99 | 0.14 | 0.0 |
| | **9** | **Volcanic** | 18 | 2.74 | 3.2 | 3.00 | 0.15 | 0.1 |
| | **10** | **OB** | 3 | 2.87 | 2.91 | 2.89 | 0.02 | 0.0 |
| | **Total** | | 1,618 | 1.81 | 5.36 | 3.06 | 0.22 | 0.1 |
| | **All** | | 2,035 | 1.81 | 5.36 | 3.07 | 0.22 | 0.1 |

Source: Kirkham Geosystems, 2020.

## 14.8 Variography

The degree of spatial variability and continuity in a mineral deposit depend on both the distance and direction between points of comparison. Typically, the variability between samples is proportionate to the distance between samples. If the variability is related to the direction of comparison, then the deposit is said to exhibit *anisotropic* tendencies which can be summarised by an ellipse fitted to the ranges in the different directions. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include the nugget, the sill, and the range. Often samples compared over very short distances (including samples from the same location) show some degree of variability. As a result, the curve of the variogram often begins at a point on the y-axis above the origin; this point is called the *nugget*. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability which can be introduced due to errors during sample collection, preparation, and assaying.

Typically, the amount of variability between samples increases as the distance between the samples increase. Eventually, the degree of variability between samples reaches a constant or maximum value; this is called the *sill*, and the distance between samples at which this occurs is called the *range*.

The spatial evaluation of the data was conducted using a correlogram instead of the traditional variogram. The correlogram is normalised to the variance of the data and is less sensitive to outlier values; this generally gives cleaner results.

Experimental variograms and variogram models in the form of correlograms were generated for nickel and cobalt along with sulphur.

Correlograms were generated for the distribution of nickel and cobalt along with sulphur in the various areas using the commercial software package Sage 2001© developed by Isaacs & Co. Correlogram model data is shown in Table 14.6 for nickel, cobalt and sulphur along with platinum and palladium in Table 14.7.

**Table 14.6: Variography for Nickel, Cobalt & Sulphur by Zone**

| Ni | | Px | gDu | Wh/Du |
|---|---|---|---|---|
| | C0 | 0.294 | 0.103 | 0.267 |
| | C1 | 0.225 | 0.567 | 0.498 |
| | C2 | 0.481 | 0.33 | 0.235 |
| 1st Structure | DY | 16.6 | 15.5 | 20.3 |
| | DX | 136.8 | 32.2 | 32.8 |
| | DZ | 124.5 | 11.3 | 44 |
| | R1 | 30 | 85 | -49 |
| | R2 | 16 | 6 | -6 |
| | R3 | -61 | -32 | 3 |
| 2nd Structure | DY | 85.3 | 81.7 | 102.3 |
| | DX | 62.3 | 34.3 | 175 |
| | DZ | 326.5 | 273.2 | 282.2 |
| | R1 | 33 | -22 | 43 |
| | R2 | 28 | 13 | -4 |
| | R3 | -25 | 0 | -46 |

| Co | | Px | gDu | Wh/Du |
|---|---|---|---|---|
| | C0 | 0.332 | 0.329 | 0.381 |
| | C1 | 0.45 | 0.477 | 0.454 |
| | C2 | 0.218 | 0.194 | 0.165 |
| 1st Structure | DY | 38.4 | 28.5 | 96.5 |
| | DX | 25.4 | 17.6 | 31.6 |
| | DZ | 128.34 | 79 | 27.4 |
| | R1 | 41 | -16 | 0 |
| | R2 | 10 | 10 | 65 |
| | R3 | -9 | -12 | -71 |
| 2nd Structure | DY | 85.4 | 410.1 | 354.4 |
| | DX | 220.4 | 119.5 | 109.7 |
| | DZ | 889.8 | 246.4 | 449.1 |
| | R1 | 81 | 14 | -73 |
| | R2 | 4 | -15 | 20 |
| | R3 | -60 | -65 | 14 |

| S | | Px | gDu | Wh/Du |
|---|---|---|---|---|
| | C0 | 0.294 | 0.076 | 0.277 |
| | C1 | 0.299 | 0.802 | 0.543 |
| | C2 | 0.407 | 0.122 | 0.18 |
| 1st Structure | DY | 14.6 | 320.6 | 37.5 |
| | DX | 10 | 33.4 | 46.3 |
| | DZ | 35.5 | 211 | 131.9 |
| | R1 | -22 | 10 | 3 |
| | R2 | -39 | 33 | 0 |
| | R3 | -6 | 7 | -1 |
| 2nd Structure | DY | 81.9 | 62.1 | 246.6 |
| | DX | 71.1 | 187.4 | 965.5 |
| | DZ | 1596.9 | 248 | 719.6 |
| | R1 | 42 | -7 | 20 |
| | R2 | 36 | 31 | -4 |
| | R3 | -30 | -73 | -80 |

Source: Kirkham Geosystems, 2020.

**Table 14.7: Variography for Platinum & Palladium by Zone**

| Pt | | Px | gDu | Wh/Du |
|---|---|---|---|---|
| | C0 | 0.311 | 0.411 | 0.44 |
| | C1 | 0.435 | 0.451 | 0.43 |
| | C2 | 0.254 | 0.138 | 0.13 |
| 1st Structure | DY | 17.2 | 17.7 | 15.5 |
| | DX | 11.6 | 48.5 | 25.5 |
| | DZ | 21.6 | 13.3 | 52.1 |
| | R1 | 93 | -27 | 79 |
| | R2 | -70 | 27 | -1 |
| | R3 | -23 | 63 | -4 |
| 2nd Structure | DY | 46.8 | 227 | 179.8 |
| | DX | 115.9 | 113 | 139.9 |
| | DZ | 813.1 | 482.7 | 392.8 |
| | R1 | 71 | -10 | -39 |
| | R2 | 4 | 133 | 33 |
| | R3 | -18 | -21 | -21 |

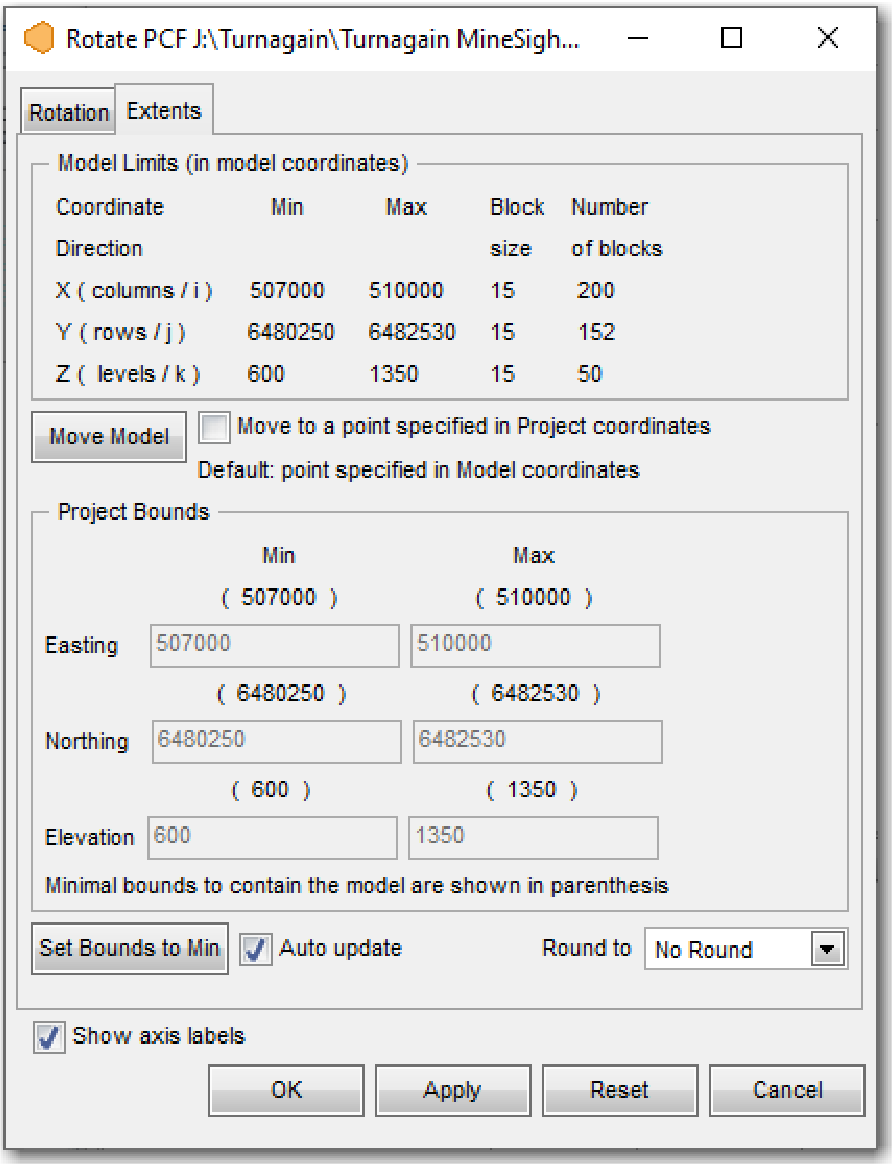
| Pd | | Px | gDu | Wh/Du |
|---|---|---|---|---|
| | C0 | 0.404 | 0.349 | 0.384 |
| | C1 | 0.391 | 0.48 | 0.433 |
| | C2 | 0.205 | 0.171 | 0.183 |
| 1st Structure | DY | 11.1 | 16.2 | 14.9 |
| | DX | 12.6 | 51 | 49.7 |
| | DZ | 74.3 | 17.8 | 20.8 |
| | R1 | -16 | -65 | 66 |
| | R2 | -1 | 31 | -2 |
| | R3 | 1 | 76 | 77 |
| 2nd Structure | DY | 76.3 | 727.6 | 133.3 |
| | DX | 149.2 | 193.1 | 176.7 |
| | DZ | 675.7 | 138.9 | 562.8 |
| | R1 | 56 | -28 | 36 |
| | R2 | 3 | 42 | -10 |
| | R3 | -20 | -57 | -40 |

Source: Kirkham Geosystems, 2020.

## 14.9 Block Model Definition

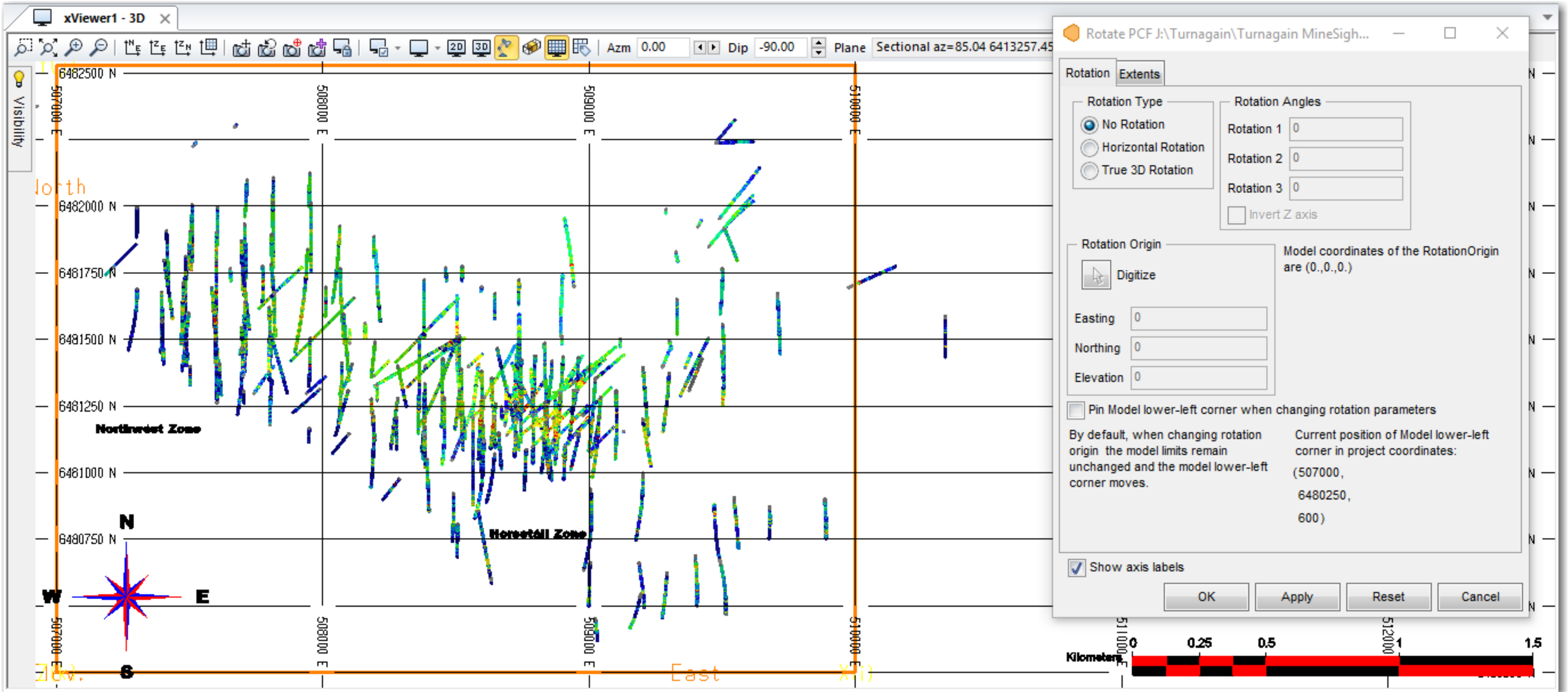
The block model used to estimate the resources was defined according to the limits specified in Figures 14-19 and 14-20. The block model is orthogonal and non-rotated, reflecting the orientation of the deposit. The chosen block size was 15 m x 15 m x 15 m, roughly reflecting the drill hole spacing (i.e., 4 to 6 blocks between drill holes) which is spaced at approximately 50 m centres. Note: MineSight™ uses the centroid of the blocks as the origin.

**Figure 14-19: Dimensions for the Turnagain Block Model**




Source: Kirkham Geosystems, 2020.

**Figure 14-20: Origin & Orientation for the Turnagain Block Model**




Source: Kirkham Geosystems, 2020.

## 14.10 Resource Estimation Methodology

The resource estimation plan includes the following items:

- lithological zone code in each block
- estimated block nickel, cobalt, sulphur grades using ordinary kriging
- estimated block platinum, palladium grades using ordinary kriging

Table 14.8 summarises the search ellipse dimensions for the one estimation pass for each zone by majority code.

**Table 14.8: Search Ellipse Parameters for the Turnagain Deposit**

| Major Axis | Semi-Major Axis | Minor Axis | 1st Rotation Angle Azimuth | 2nd Rotation Angle Dip | 3rd Rotation Angle | Min. No. Of Comps | Max. No. Of Comps | Max. Samples per Drill Hole |
|---|---|---|---|---|---|---|---|---|
| 150 | 150 | 150 | 0 | 90 | 0 | 2 | 16 | 4 |

Source: Kirkham Geosystems, 2020.

## 14.11 Resource Validation

A graphical validation was completed on the block model. This type of validation serves the following purposes:

- checks the reasonableness of the estimated grades based on the estimation plan and the nearby composites
- checks that the general drift and the local grade trends compare to the drift and local grade trends of the composites
- ensures that all blocks in the core of the deposit have been estimated
- checks that topography has been properly accounted for
- checks against manual approximate estimates of tonnages to determine reasonableness
- inspects for and explains potentially high-grade block estimates in the neighbourhood of the extremely high assays

A full set of cross sections, long sections and plans were used to digitally check the block model; these showed the block grades and composites. There was no indication that a block was wrongly estimated, and it appears that every block grade could be explained as a function of the surrounding composites and the applied estimation plan.

The validation techniques included the following:

- visual inspections on a section-by-section and plan-by-plan basis
- use of grade-tonnage curves

- swath plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates
- inspection of histograms showing distance from first composite to nearest block, and average distance to blocks for all composites (this gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources)

## 14.12 Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources may be impacted by further infill and exploration drilling that may result in an increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Mineral resources for the Turnagain deposit were classified according to the CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) by Garth Kirkham, P.Geo., an independent qualified person as defined by NI 43-101 guidelines.

Drill hole spacing in the Turnagain deposit is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. Kirkham Geosystems, therefore, is of the opinion that the amount of sample data is adequate to demonstrate very good confidence in the grade estimates for the deposit.

The estimated blocks were classified according to the following:

- confidence in interpretation of the mineralised zones
- number of data used to estimate a block
- number of composites allowed per drill hole
- distance to nearest composite used to estimate a block

The classification of resources was based primarily on distance to the nearest composite; however, all of the quantitative measures, as listed here, were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

The mineral resource estimates for Turnagain were prepared to industry standards and best practices using commercial mine-modelling and geostatistical software. Garth Kirkham, P.Geo., is the qualified person responsible for the Turnagain mineral resource estimates for the purposes of NI 43-101.

Mineral resources are classified under the categories of *measured*, *indicated* and *inferred* according to CIM guidelines. Mineral resource classification was based primarily on drill hole spacing and on continuity of mineralisation. Measured resources were defined at Turnagain as blocks with a distance to three drill holes of less than ~40 m to nearest composite and an average of 80 m and occurring within the estimation domains. Indicated resources were defined as those with a distance to three drill holes of less than ~60 m and an average distance of 100 m. Inferred resources were defined as those with an average drill hole spacing of less than ~150 m and meeting additional requirements. Final resource classification shells were manually constructed on sections.

Furthermore, an interpreted boundary was created for the measured, indicated and inferred thresholds in order to exclude orphans and reduce “spotted dog” effect. The remaining blocks were unclassified and may be considered as geologic potential for further exploration.

## 14.13 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 14.9 shows the total resources for all metals at varying Ni% cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grades.

Note: The base case cut-off grades presented in Table 14.9 are based on potentially open pit resources at the base case of 0.1% Ni.

## 14.14 Mineral Resource Statement

Table 14.10 shows the mineral resource statement for the Turnagain deposit.

This estimate is based upon the reasonable prospect of eventual economic extraction based on continuity an optimized pit, using estimates of operating costs and price assumptions. The “reasonable prospects for eventual economic extraction” were tested using floating cone pit shells. The Horsetrail, Northwest, and Duffy zones of the deposit are all included within the Horsetrail reasonable prospects pit shells. The pit optimization results are used solely for testing the “reasonable prospects for eventual economic extraction” and do not represent an attempt to estimate Mineral Reserves.

Differences from the previous resource estimate described in AMC’s 2011 PEA are the inclusion of an additional 36 infill drill holes totalling 8,940 m drilled in 2018 in the areas of the conceptual open pit and updated geological modelling.

**Table 14.9: Sensitivity Analyses of Global Tonnage & Grades at Various Ni% Cut-off Grades**

| Classification | Cut-off | Tonnage (000s) | Ni Grade (%) | Co Grade (%) |
|---|---|---|---|---|
| Measured | 0.05 | 364,997 | 0.228 | 0.0137 |
| | 0.10 | 360,913 | 0.230 | 0.0138 |
| | 0.15 | 341,518 | 0.236 | 0.0140 |
| | 0.20 | 271,773 | 0.251 | 0.0143 |
| | 0.25 | 111,963 | 0.285 | 0.0151 |
| | 0.30 | 24,868 | 0.340 | 0.0173 |
| Indicated | 0.05 | 744,776 | 0.209 | 0.0127 |
| | 0.10 | 712,406 | 0.215 | 0.0129 |
| | 0.15 | 632,964 | 0.226 | 0.0133 |
| | 0.20 | 468,776 | 0.242 | 0.0136 |
| | 0.25 | 158,978 | 0.274 | 0.0140 |
| | 0.30 | 17,020 | 0.326 | 0.0153 |
| Inferred | 0.05 | 1,219,566 | 0.211 | 0.0128 |
| | 0.10 | 1,142,101 | 0.217 | 0.0130 |
| | 0.15 | 1,041,600 | 0.226 | 0.0133 |
| | 0.20 | 791,347 | 0.241 | 0.0137 |
| | 0.25 | 243,580 | 0.277 | 0.0144 |
| | 0.30 | 27,211 | 0.347 | 0.0159 |

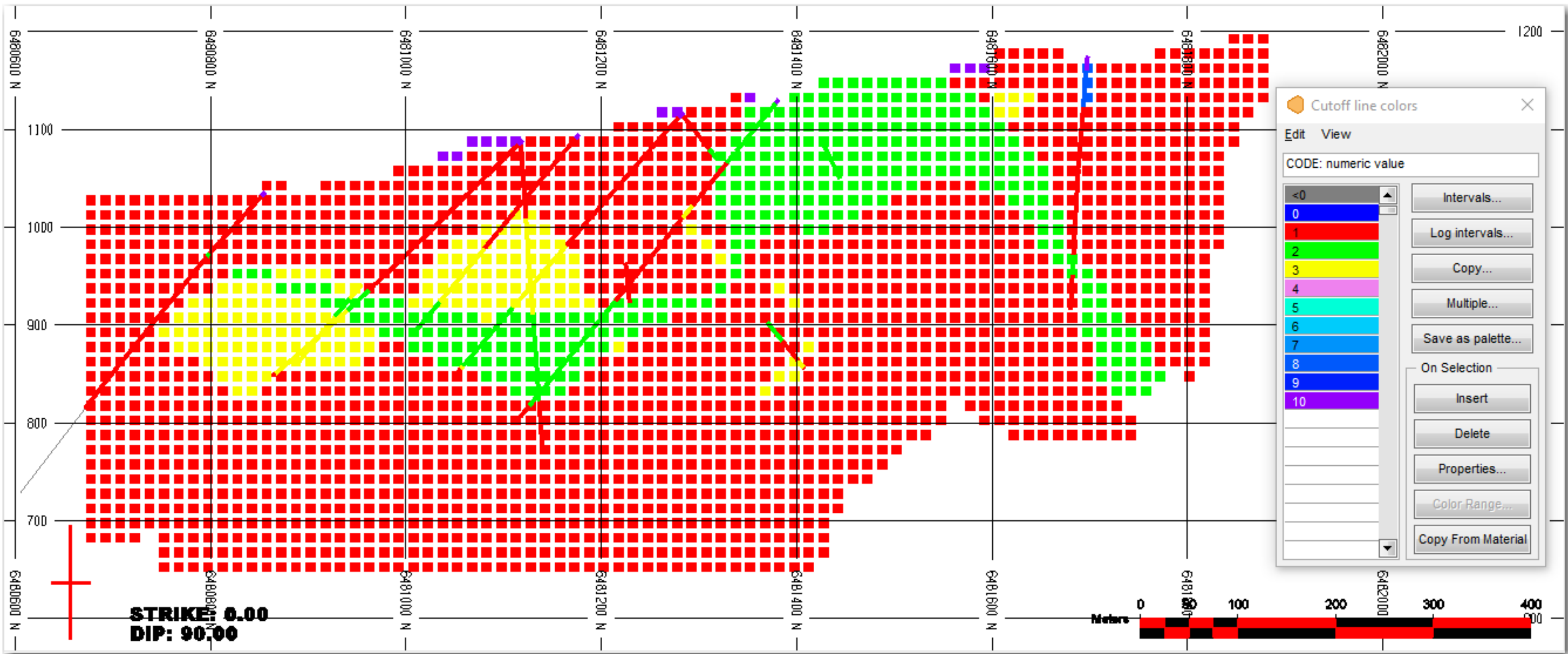
Notes: **(1)** The current resource estimate was prepared by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd. **(2)** All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum definitions, as required under National Instrument 43-101. **(3)** Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate reasonable expectation of eventual economic extraction, as required under NI 43-101; mineralisation lying outside of these pit shells is not reported as a mineral resource. **Mineral resources are not mineral reserves and do not have demonstrated economic viability.** All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely. **(4)** Open pit mineral resources are reported at a cut-off grade of 0.1% Ni. Cut-off grades are based on a price of US $7.50 per pound, nickel recoveries of 60%, ore and waste mining costs of $2.80, along with milling, processing and G&A costs of $7.20. **(5)** Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated. **(6)** Due to rounding, numbers presented may not add up precisely to the totals provided and percentages my not precisely reflect absolute figures. Source: Kirkham Geosystems, 2020.

**Table 14.10: Open Pit Mineral Resource Statement (1,2,3,4,5) for the Turnagain Project. Base Case Estimate at 0.1% Nickel Cut-off Grade**

| Classification | Tonnage (000s) | Ni Grade (%) | Contained Ni (klbs) | Co Grade (%) | Contained Co (klbs) |
|---|---|---|---|---|---|
| **Measured** | 360,913 | 0.230 | 1,832,424 | 0.0138 | 109,802 |
| **Indicated** | 712,406 | 0.215 | 3,373,585 | 0.0129 | 202,604 |
| Measured & Indicated | 1,073,319 | 0.220 | 5,206,009 | 0.0132 | 312,406 |
| **Inferred** | 1,142,101 | 0.217 | 5,473,862 | 0.0130 | 327,324 |

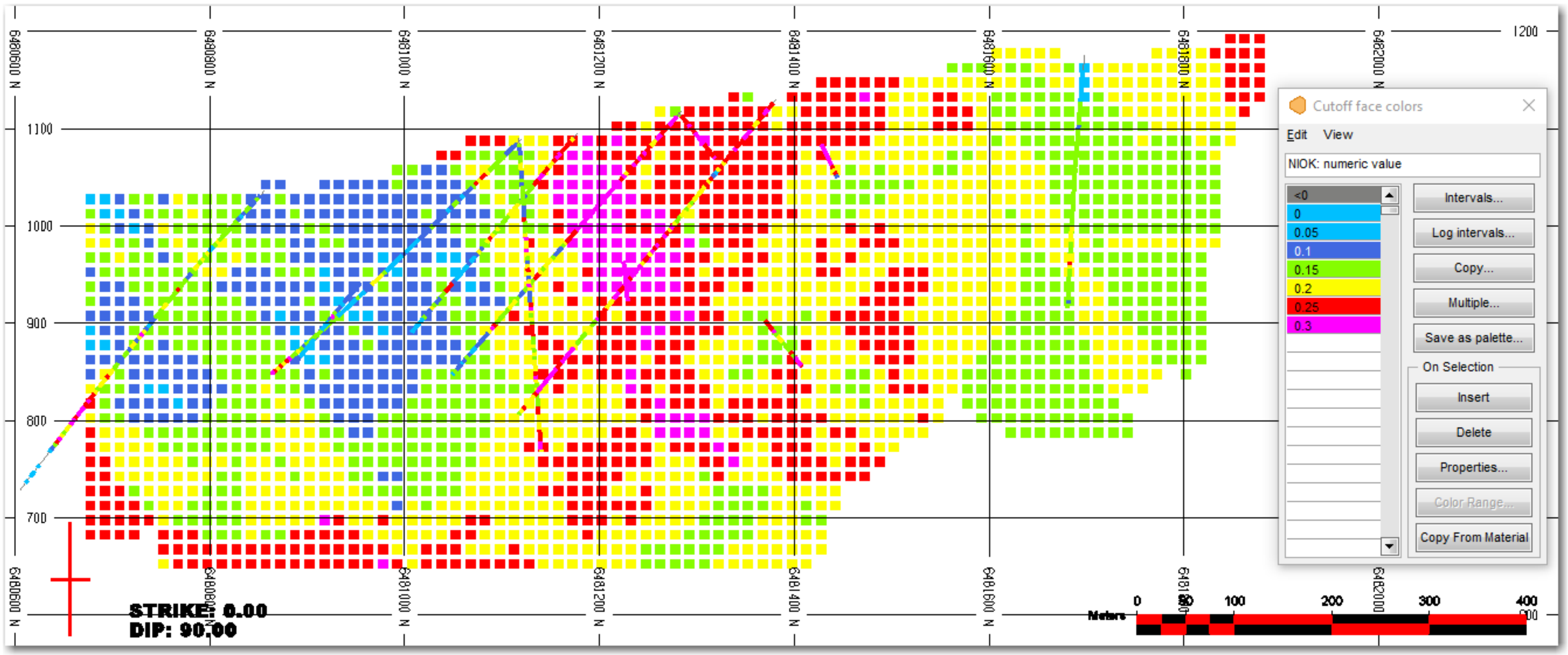
Notes: **(1)** All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum definitions, as required under National Instrument 43-101. **(2)** Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate reasonable expectation of eventual economic extraction, as required under NI 43-101; mineralisation lying outside of these pit shells is not reported as a mineral resource. **Mineral resources are not mineral reserves & do not have demonstrated economic viability.** **(3)** Open pit mineral resources are reported at a cut-off grade of 0.1% Ni. Cut-off grades are based on a price of US $7.50 per pound, nickel recoveries of 60%, ore and waste mining costs of $2.80, along with milling, processing and G&A costs of $7.20. **(4)** Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated. **(5)** Due to rounding, numbers presented may not add up precisely to the totals provided and percentages my not precisely reflect absolute figures. Source: Kirkham Geosystems, 2020.

**Figure 14-21: Section View of Lithology Coded Blocks & Drill Holes for the Block Model**



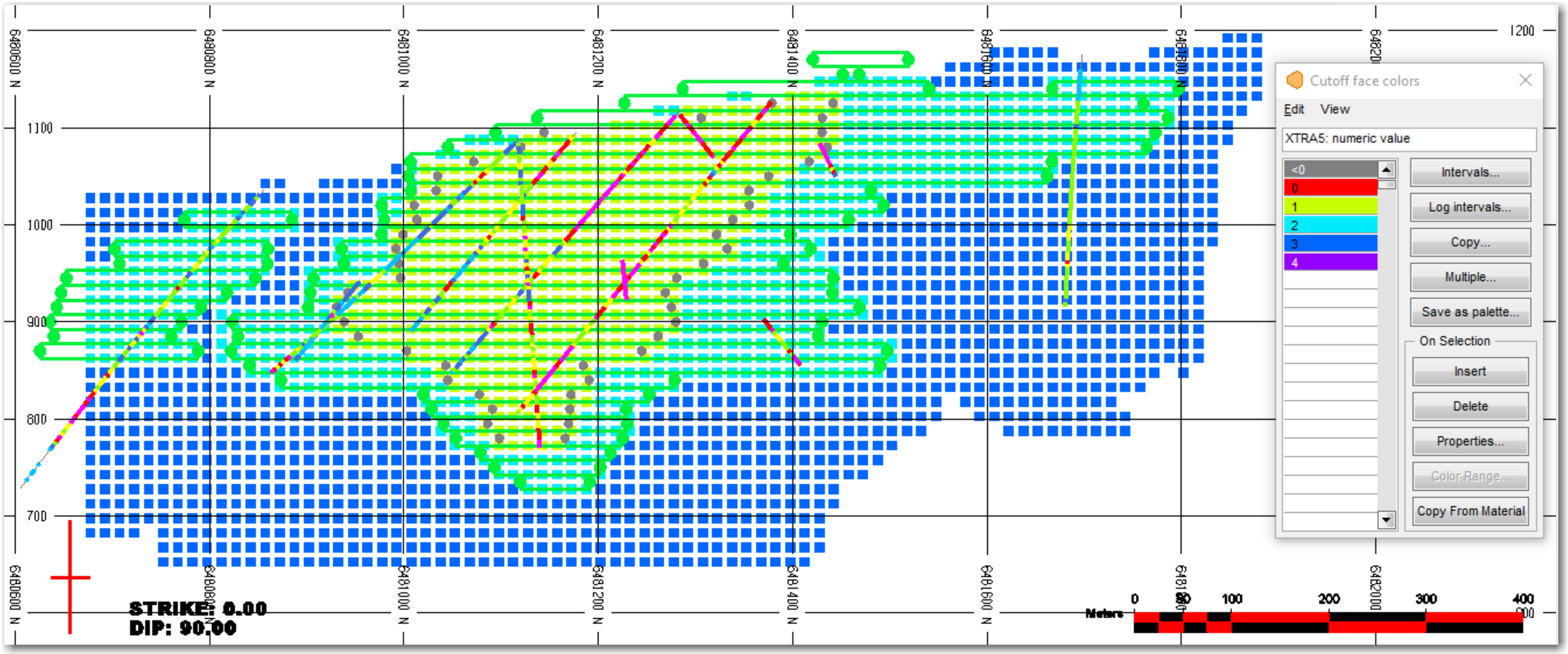

Note: 1 = Wehrlite-Dunite-Serpentinite (red), 2 = Green Dunite (green), 3 = Pyroxenite-Clinopyroxenite-Olivine Clinopyroxenite (yellow), 8 = Dyke (light blue), 9 – Volcanics (dark blue), 10 = Overburden (purple). Source: Kirkham Geosystems, 2020.

**Figure 14-22: Section View of Ni% & Drill Holes for the Block Model**




Source: Kirkham Geosystems, 2020.

**Figure 14-23: Section View of Classification of Blocks & Drill Holes**




Note: 1 = Measured (green), 2 = Indicated (light blue), 3 = Inferred (blue). Source: Kirkham Geosystems, 2020.

## 14.15 Comparison to Previous 2011 Mineral Resource Estimate

Since the 2011 mineral resource update, Giga Metals has performed infill drilling and updated its geological modelling. As a result, the measured plus indicated resources have grown due to infill drilling, but the inferred resources have also grown because the volumes of the ultimate conceptual pit have grown. This comparison is provided for information purposes only. This comparison should not be interpreted as a statement of mineral reserves; mineral reserves can only be defined in a pre-feasibility or feasibility study.

**Table 14.11: Comparison of 2019 & 2011 Consolidated Mineral Resource Statement(1,2,3,4,5) for the Turnagain Project**

| Classification | Tonnage (000s) | Ni Grade (%) | Contained Ni (000s lbs) | Co Grade (%) | Contained Co (000s lbs) |
|---|---|---|---|---|---|
| 2019 Update | | | | | |
| Measured and Indicated | 1,073,885 | 0.22 | 5,206,051 | 0.013 | 312,550 |
| Inferred(4) | 1,163,515 | 0.2176 | 5,581,627 | 0.013 | 333,461 |
| 2011 Estimate | | | | | |
| Measured and Indicated | 865,482 | 0.213 | 4,057,052 | 0.013 | 253,061 |
| Inferred(4) | 976,295 | 0.2 | 4,304,719 | 0.013 | 279,807 |
| 2011-2019 Change | | | | | |
| Measured and Indicated | 208,403 | 0.007 | 1,148,999 | unchanged | 59,489 |
| Inferred(4) | 187,220 | 0.018 | 1,276,908 | unchanged | 53,654 |
| Percentage Change | | | | | |
| Measured and Indicated | 24.08% | 3.50% | 28.32% | unchanged | 23.51% |
| Inferred | 19.18% | 8.70% | 29.66% | unchanged | 19.18% |

Notes: **(1)** All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum definitions, as required under National Instrument 43-101. **(2)** Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate reasonable expectation of eventual economic extraction, as required under NI 43-101; mineralisation lying outside of these pit shells is not reported as a mineral resource. **Mineral resources are not mineral reserves & do not have demonstrated economic viability.** All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely. **(3)** Open pit mineral resources are reported at a cut-off grade of 0.1% Ni. Cut-off grades are based on a price of US $7.50 per pound, nickel recoveries of 60%, ore and waste mining costs of $2.80, along with milling, processing and G&A costs of $7.20. **(4)** Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated. **(5)** Due to rounding, numbers presented may not add up precisely to the totals provided and percentages my not precisely reflect absolute figures. Source: Kirkham Geosystems, 2020.

# 15.0 MINERAL RESERVE ESTIMATES

There are no reserves to report at this stage.

# 16.0 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 categorised as mineral reserves, and there is no certainty that the PEA results based on these resources will be realised.

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 operational unit costs. The waste and mineralisation will require blasting and typical grade control methods using blast-hole sampling.

For the purpose of this study, extraction from the Horsetrail, Northwest, and Duffy mineralised zones are incorporated in the LOM mine plan with two separate excavations. Previous evaluations have indicated a potential open pit resource in the Hatzl Zone 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, so the underlying mineralisation is excluded.

The potential resource contained in the Horsetrail pit is summarised in Table 16.1. These pit shells form the basis of the mine production schedule in this study. It is defined by the 95% sensitivity revenue base case optimisation shell.

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

| | Mineralisation (kt) | Waste (kt) | Strip Ratio (t:t) | MHV* ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Horsetrail Pit Shells (PEA basis, 38-year LOM) | 1,121,980 | 207,880 | 0.19 | 8.86 | 0.221 | 0.013 | 0.60 |
| **Total** | **1,121,980** | **207,880** | **0.19** | **8.86** | **0.221** | **0.013** | **0.60** |

Notes: (1) * MHV = mill head value at base case metal pricing. (2) Note: includes inferred resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that these data will be realised.

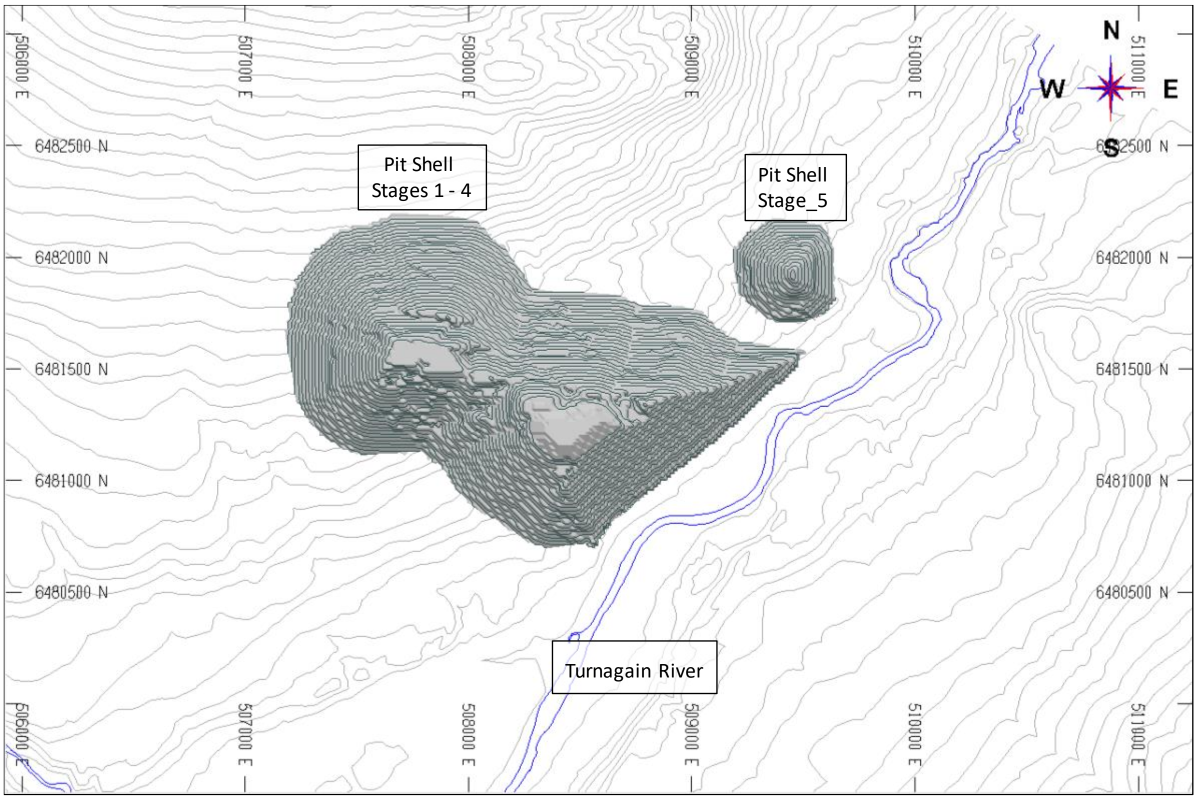
MHV calculations and values described in this section were calculated and utilised for the development of the mine plan with the input parameters listed in this section, and are not necessarily consistent with final values utilised in the site assessment evaluation described in Section 22.

Figure 16-1 shows a plan view of the selected ultimate Horsetrail pit shells used for PEA mine scheduling.

The mine will feed the crusher at an average rate of 45,000 t/d during the first five years and increase to an average of 90,000 t/d thereafter. The resource will be mined for 37 years at these rates based on a 365-day production year.

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

**Figure 16-1: Horsetrail Scheduled Pit Shells – Plan View**

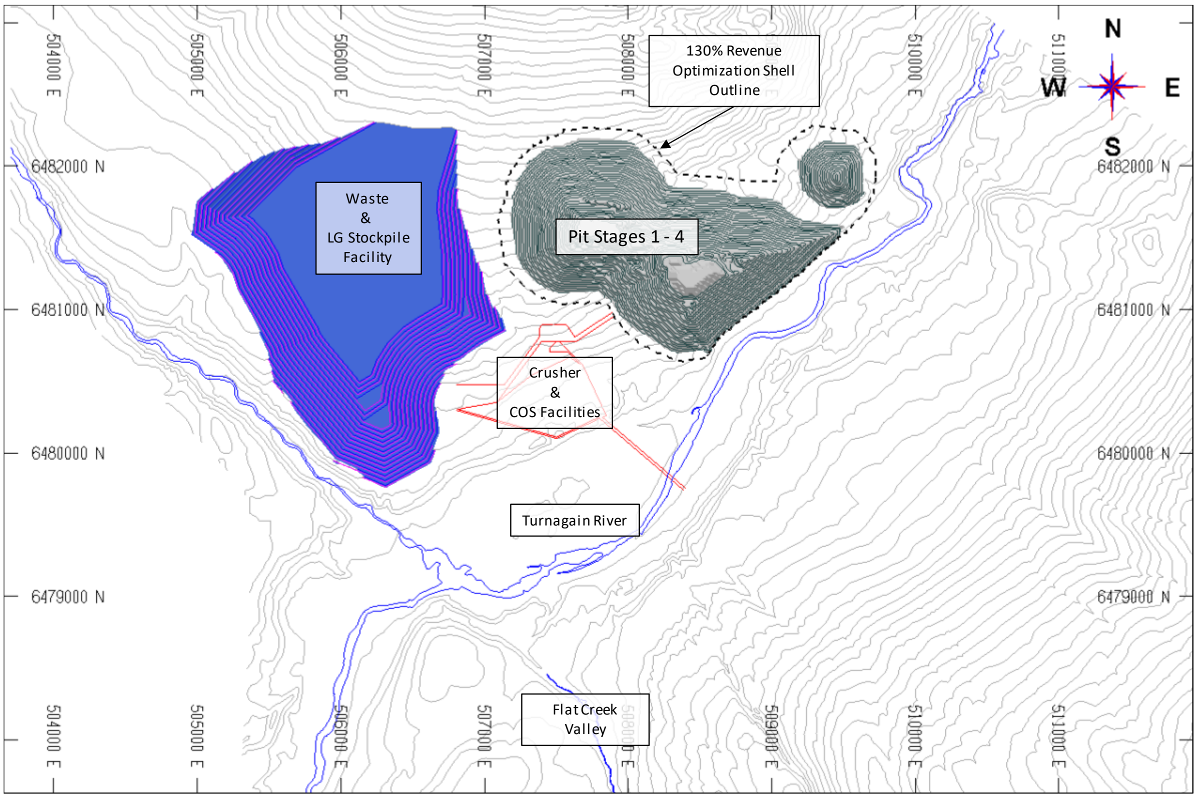



Source: Hatch, 2020.

An elevated cut-off grade will be employed in the initial production years to enhance the economics of the project. Mineralisation that is below the mine high-grade (HG) direct feed cut-off, but of sufficient grade to cover the cost of milling and rehandling once it is hauled out of the pit, will be classified and stockpiled by categorised value for subsequent rehandling. Stockpile reclaim will occur at the end of the mine life or it will be blended with the ROM feed as the appropriate opportunity arises. Low-grade (LG) stockpiles will be placed in the same facility as the waste dump to avoid additional costs.

Pit waste and LG material will be hauled to a waste dump and stockpile complex southwest of the pit, past the crusher and coarse ore stockpile facilities. The design shown in Figure 16-2 is capable of containing all scheduled waste and LG material in the LOM plan. Current geochemistry data suggest there is insignificant acid generating potential in the waste rock. There is also the potential that if deemed cost effective and geochemically acceptable, waste rock can be utilised as buttress material for the TMF in Flat Creek valley, thus it is conceivable that the waste and LG storage facilities will have minimal impact at the end of mine life. Further studies will be undertaken to confirm this possibility. Figure 16-2 shows the conceptual waste and LG maximum extents dumping plan.

**Figure 16-2: Waste Disposal & Mineralisation Stockpile General Arrangement**




Source: Hatch, 2020.

## 16.2 Open Pit Optimisation Study

The optimisation study methodology that was used is widely accepted in the mining industry for preliminary assessments of open pit mining potential. The potential resource for the open pit was evaluated by undertaking pit optimisation studies on the geological model using Hexagon MineSight® Pseudoflow software, which produces equivalent results as the industry-standard Lerchs-Grossman computer algorithm. The Pseudoflow programming produces results significantly faster than Lerchs-Grossman runs. Various preliminary pit shells were generated from the simulations and analysed. Selected shells were assessed for whether they are appropriate to use for framing the ultimate pit and mining phases.

### 16.2.1 Optimisation Parameters

Economic values are assigned on measured, indicated, and inferred resource classes as categorised in the resource block model. The preliminary input parameters applied on the optimised pit (Table 16.2) are estimated based on previous studies and discussions with Giga Metals. Costs, exchange rate, recoveries, and pit slope angles are preliminary and are specific for this optimisation study only. Values utilised in the pit optimisation process were determined and applied early in this PEA study, and are not necessarily consistent with final values utilised in the cashflow and site assessment evaluation.

**Table 16.2: Preliminary Design Parameters for Optimised Pit**

| | Units | Value |
|---|---|---|
| Exchange Rate | (CAD/US) | 0.75 |
| Nickel Price | US$/lb | 8.00 |
| Cobalt Price | US$/lb | 20.00 |
| Concentrate Grade | (%) | 18.0 |
| Concentrate Transport Cost | (US$/dmt) | 169.20 |
| Payable* (of recovered metal value) | (%) | 70.56 |
| Total Milling & Site (incl. G&A) Unit Operating Cost | (US$/t milled) | 6.29 |
| Total Base Unit Mining Cost | (US/$t mined) | 2.29 |
| Incremental Bench Unit Costs | (US$/t/bench) | 0.04 |
| Stockpile Rehandling | (US$/t moved) | 0.92 |
| Overall Wall Angles | (degrees or °) | 45° |
| River Boundary Offset | (m) | 65 |

Note: *Payable is inclusive of metals payable, treatment and refining costs.

Process recoveries for nickel and cobalt were provided by Giga Metals (see Section 13 for a description of the process used to determine recoveries). The nickel recovery formula used for an 18% concentrate for the optimisation sensitivity runs was:

$$\%Ni\_Recovery = 2.293*(\%S)3 - 17.079*(\%S)2 + 42.382*(\%S) + 28.916$$

$$Total\ recovery\ cap\ (maximum) = 63.96\%$$

$$\%Co\_Recovery = \%Ni\_Recovery$$

No other byproduct metals were included in the valuations, although copper, palladium and platinum were included in previous project cashflow evaluations.

### 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 in previous evaluations was deemed to be 1,015 masl along the river banks. The pit crest lines are offset from this contour by 65 m, which includes 50 m for a no-disturbance zone and 15 m for an access corridor. These preliminary estimates will be updated as necessary.

### 16.2.3 Pit Optimisation Variable – Net Metal Value

The pits were optimised 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 mineralisation. It was calculated in US 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 16.2. The net metal value calculated for the base case is carried in the model as the NSR value. A mill head value (MHV) method was used in the evaluations, wherein the MHV represents the total value per tonne milled if it were to be processed (i.e., the value at the crusher pocket) and is the net value of NSR less mill processing, TMF and G&A.

#### 16.2.3.1 Pit Optimisation Results

Table 16.3 shows the mineralisation and waste quantities contained in the optimised pit shells generated against the net metal value variable. The base case is Shell 39, where the input metal price is US$8.00/lb for nickel and US$20.00/lb for cobalt, and the shell contains 1.26 billion tonnes of mineralisation at a 0.15 to 1 strip ratio. The cut-off grade applied is on a MHV value of $0.01/t (i.e., breakeven grade cut-off).

The other shells were generated by varying net metal value to test the sensitivity of the resource, assess possible pit phases and the opportunities for expansion. Figure 16-3 illustrates the contained mineralisation in the LG shells.

The results from this set of optimised pits indicate that the Turnagain deposit total feed is insensitive to metal prices until nickel and cobalt prices exceed around 60% of the base case. The 95% revenue case was selected as the ultimate shell. Larger pits mine significant material at no value, while smaller ones would leave a lot of value behind. Of note, the ultimate shell selected in this evaluation did bottom out at the lower limit of modelled nickel grades.

#### 16.2.3.2 Updated Planning Parameters Used for Mine Scheduling

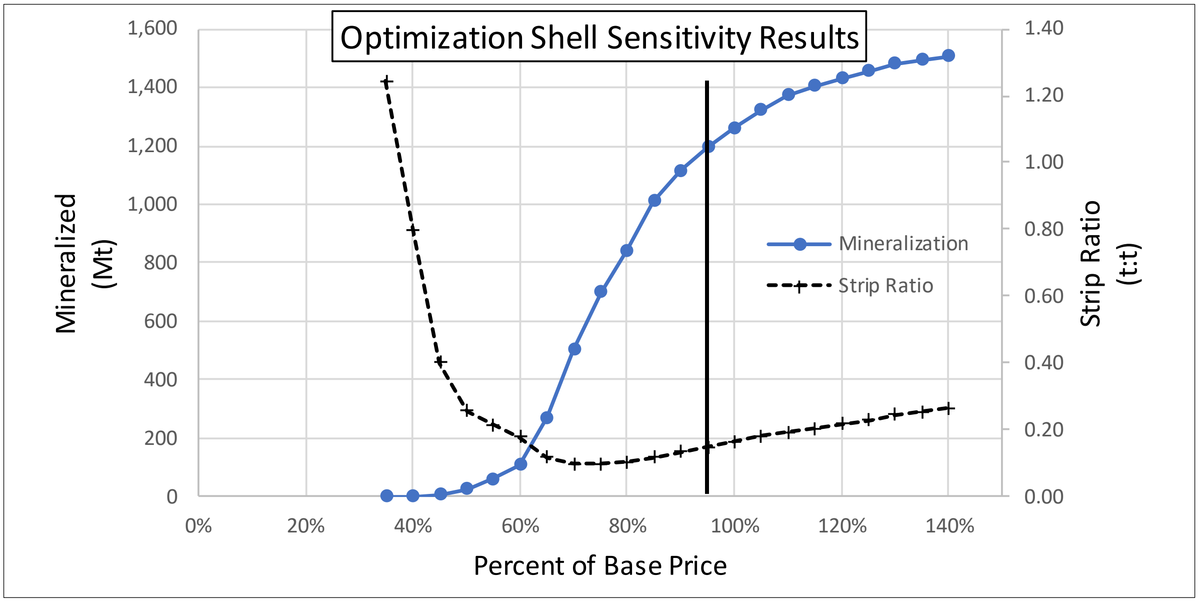
During the course of this PEA, the metal recovery formula was revised, base metal pricing was updated, and mining costs were refined based on the mine production schedule. As a confirmation check, the base case optimised pit was re-generated with the updated parameters and costs for comparison against results of the preliminary input criteria. The revised mining and processing costs are shown in Table 16.4.

**Table 16.3: Optimised Pit Shells (Cut-off on $0.01/t Milled MHV Value)**

| % of Metal Price (Base Case) | Optimised Shell | Mineralisation (Mt) | Ni (%) | Co (%) | S (%) | MHV (US$/t Milled) | Strip (Mt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|---|---|
| 35% | Pit_52 | 0 | 0.37 | 0.02 | 1.65 | 23.64 | 0.1 | 1.24 |
| 40% | Pit_51 | 0 | 0.33 | 0.02 | 1.71 | 21.13 | 0.3 | 0.79 |
| 45% | Pit_50 | 7 | 0.29 | 0.02 | 1.19 | 15.95 | 2.7 | 0.40 |
| 50% | Pit_49 | 25 | 0.28 | 0.02 | 1.13 | 14.32 | 6.5 | 0.26 |
| 55% | Pit_48 | 59 | 0.26 | 0.02 | 1.06 | 12.73 | 12.6 | 0.21 |
| 60% | Pit_47 | 106 | 0.25 | 0.01 | 1.00 | 11.49 | 18.9 | 0.18 |
| 65% | Pit_46 | 270 | 0.23 | 0.01 | 0.82 | 9.54 | 31.7 | 0.12 |
| 70% | Pit_45 | 500 | 0.23 | 0.01 | 0.75 | 8.59 | 49.1 | 0.10 |
| 75% | Pit_44 | 698 | 0.23 | 0.01 | 0.68 | 8.00 | 67.7 | 0.10 |
| 80% | Pit_43 | 842 | 0.23 | 0.01 | 0.65 | 7.62 | 86.9 | 0.10 |
| 85% | Pit_42 | 1,010 | 0.22 | 0.01 | 0.61 | 7.21 | 118.3 | 0.12 |
| 90% | Pit_41 | 1,115 | 0.22 | 0.01 | 0.58 | 6.99 | 147.0 | 0.13 |
| 95% | Pit_40 | 1,196 | 0.22 | 0.01 | 0.57 | 6.82 | 175.7 | 0.15 |
| Base Case | Pit_39 | 1,262 | 0.22 | 0.01 | 0.55 | 6.68 | 205.9 | 0.16 |
| 105% | Pit_38 | 1,322 | 0.22 | 0.01 | 0.54 | 6.55 | 239.1 | 0.18 |
| 110% | Pit_37 | 1,373 | 0.22 | 0.01 | 0.53 | 6.43 | 264.7 | 0.19 |
| 115% | Pit_36 | 1,405 | 0.22 | 0.01 | 0.52 | 6.36 | 285.5 | 0.20 |
| 120% | Pit_35 | 1,432 | 0.22 | 0.01 | 0.52 | 6.31 | 311.3 | 0.22 |
| 125% | Pit_34 | 1,457 | 0.22 | 0.01 | 0.52 | 6.25 | 333.0 | 0.23 |
| 130% | Pit_33 | 1,481 | 0.22 | 0.01 | 0.51 | 6.20 | 360.2 | 0.24 |
| 135% | Pit_32 | 1,493 | 0.22 | 0.01 | 0.51 | 6.18 | 376.2 | 0.25 |
| 140% | Pit_31 | 1,506 | 0.22 | 0.01 | 0.51 | 6.15 | 396.5 | 0.26 |

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

**Figure 16-3: Sensitivity of Contained In-Pit Resource & Respective Strip Ratios**




Source: Hatch, 2020.

**Table 16.4: Updated Design Parameters for Optimised Pit – Base Case**

| Parameter | Units | Amount |
|---|---|---|
| Exchange Rate | (CAD/US) | 0.77 |
| Nickel Price | US$/lb | 9.00 |
| Cobalt Price | US$/lb | 20.00 |
| Concentrate Grade | (%) | 18 |
| Concentrate Transport Cost | (US$/dmt) | 170 |
| Payable* (of recovered metal value) | (%) | 70.56 |
| Total Milling & Site (incl. G&A) Unit Operating Cost | (US$/t milled) | 5.80 |
| Total Base Unit Mining Cost | (US/$t mined) | $2.31 |
| Incremental Bench Unit Costs | (US$/t/bench) | $0.03 |
| Stockpile Rehandling | (US$/t moved) | $0.92 |
| Overall Wall Angles | (degrees or °) | 45° |
| River Boundary Offset | (m) | 65 |
| % Ni Recovery | (%) | =11.895*LN(%S)+56.176 |

Note: *Payable is inclusive of metals payable, treatment and refining costs.

The contained mineralisation and grades for the revised base case optimised pit is summarised in Table 16.5. It is reported on the respective MHV cut-off values of $0.01/t for each scenario.

**Table 16.5: Base Case Optimised Pit – Updated Cost Parameters**

| Scenario | Mineralisation (Mt) | Ni (%) | Co (%) | S (%) | MHV (US$/t Milled) | Strip (Mt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|---|
| **Preliminary Base Case** | 1,262 | 0.223 | 0.013 | 0.552 | 6.68 | 205.9 | 0.16 |
| **Updated Base Case** | 1,251 | 0.217 | 0.013 | 0.588 | 8.43 | 281.7 | 0.23 |
| Difference (Update - Preliminary) | -12 | -0.005 | 0.000 | 0.037 | 1.75 | 75.8 | 0.06 |
| % Variance | -1% | -2% | 0% | 7% | 26% | 37% | 38% |

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

The results indicate that the optimised pit with the updated input criteria is a slightly larger excavation but with marginally fewer feed tonnes (-1%) and an incremental strip ratio increase of 0.06 (0.23 versus 0.16). The results confirm the applicability of the preliminary shell selection process for scheduling and PEA evaluation purposes.

The updated parameters were implemented in model valuations for mine production scheduling as discussed in Section 16.5. The process for selecting the ultimate pit should be further refined at the pre-feasibility study stage relative to measured, indicated and inferred mineralisation categories and should include sensitivity analysis and discounting. It is important to note that the contained mineralisation in this simulated pit shell is only an indication of the potential open pit mineable resource and should not be relied on as a recoverable resource.

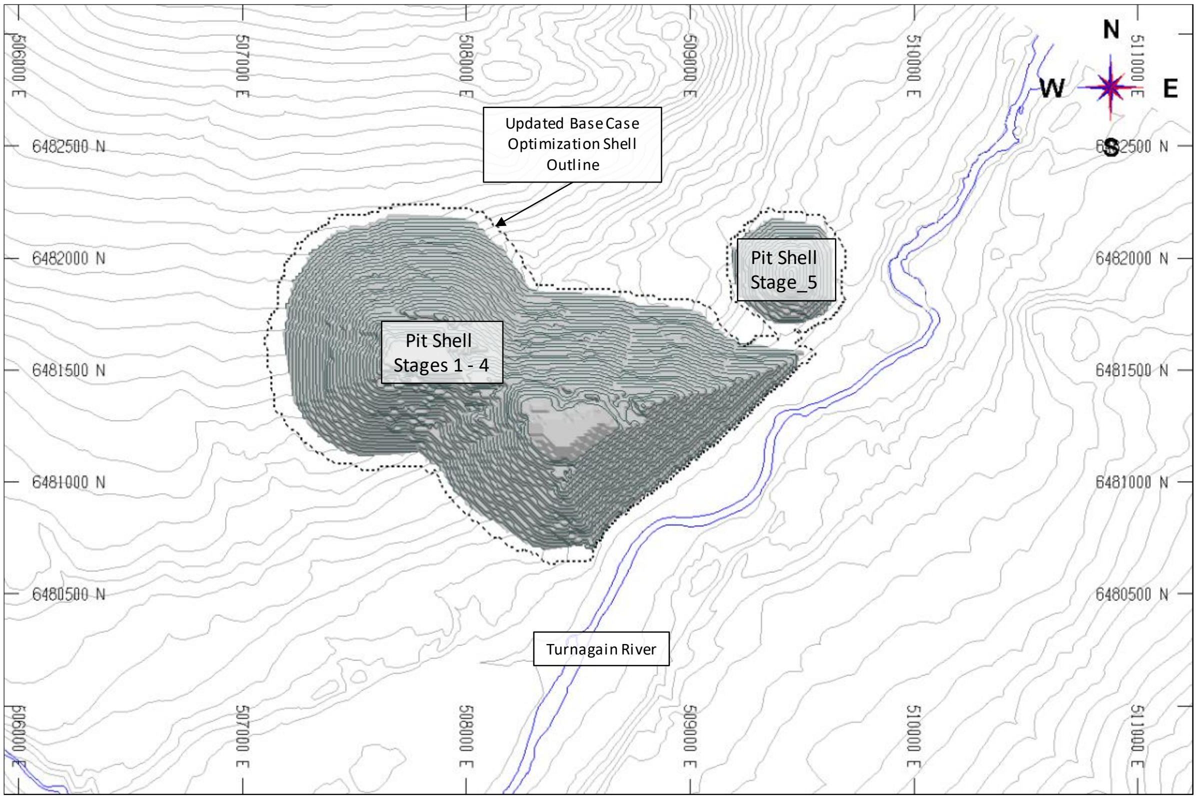
Figure 16-4 on the following page is a plan view of the revised base case optimised pit shell outline, compared to the ultimate selected shell (Pit 40) for mine planning in this PEA.

## 16.3 Pit Designs

Smoothed pit designs were not developed for this study, as optimised pit shells are sufficient for this PEA analysis and detailed pit designs at this stage will not necessarily improve mine planning accuracy, given the relatively large size of excavations anticipated along with the preliminary level of current pit slope geotechnical analysis.

Pit stages were identified from the pit optimisation sensitivity runs based on step changes in pit values. Additionally, a minimum 100 m pushback width was incorporated to facilitate productive efficient mining operations. The ultimate selected shell provides sufficient material for a mine life of 37 years (1.12 Bt), and for this study, five stages are planned. As the project develops and further knowledge is gained regarding the basic economics, pit geotechnical and geometallurgy, smoothed designs and more refined stage selection can be implemented.

**Figure 16-4: Plan View of Updated Base Case Shell Outline compared to PEA Selected Planning Shells**

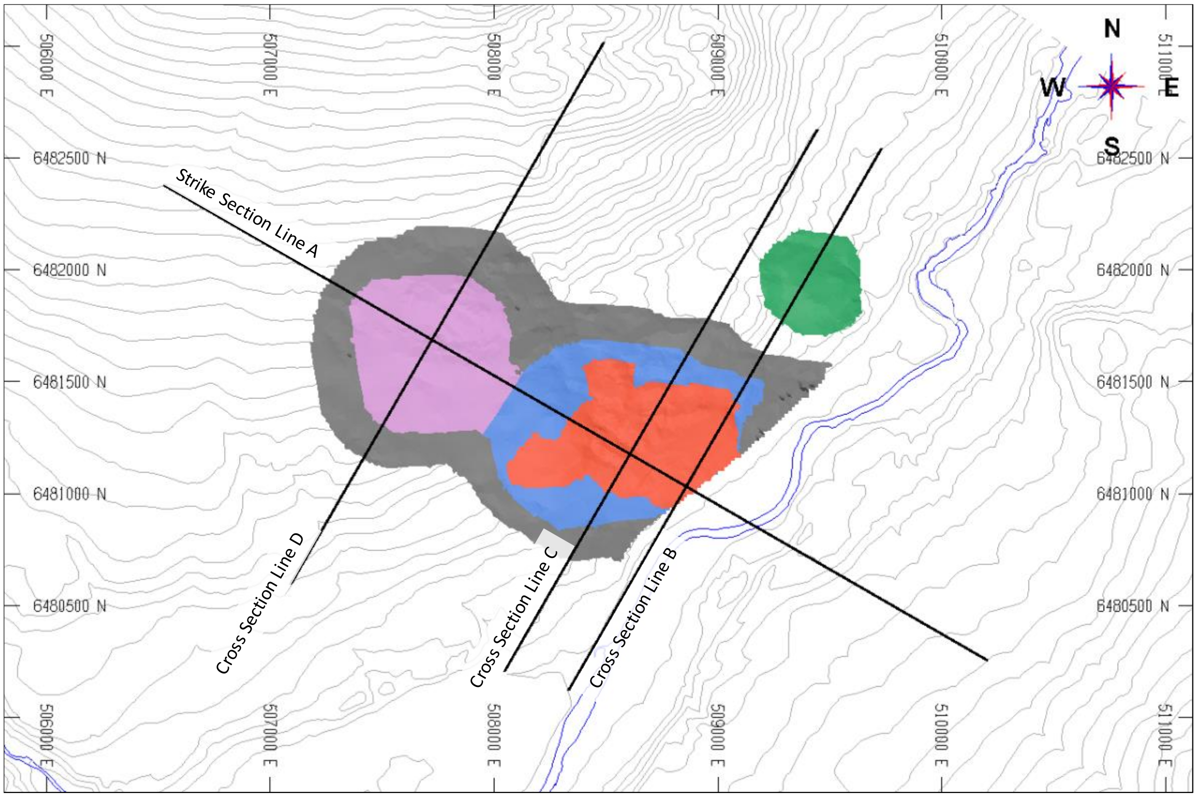



Source: Hatch, 2020.

The dimensions of the primary excavation containing Stages 1 through 4 (Figure 16-1) are approximately 2 km in length, and 1.1 km wide. Stage 5 is a circular pit with top diameter of 0.5 km. The bottom of the Stage 4 pit is at 645 m elevation; the highest point along the pit rim is 1,340 m elevation at the northwest end.

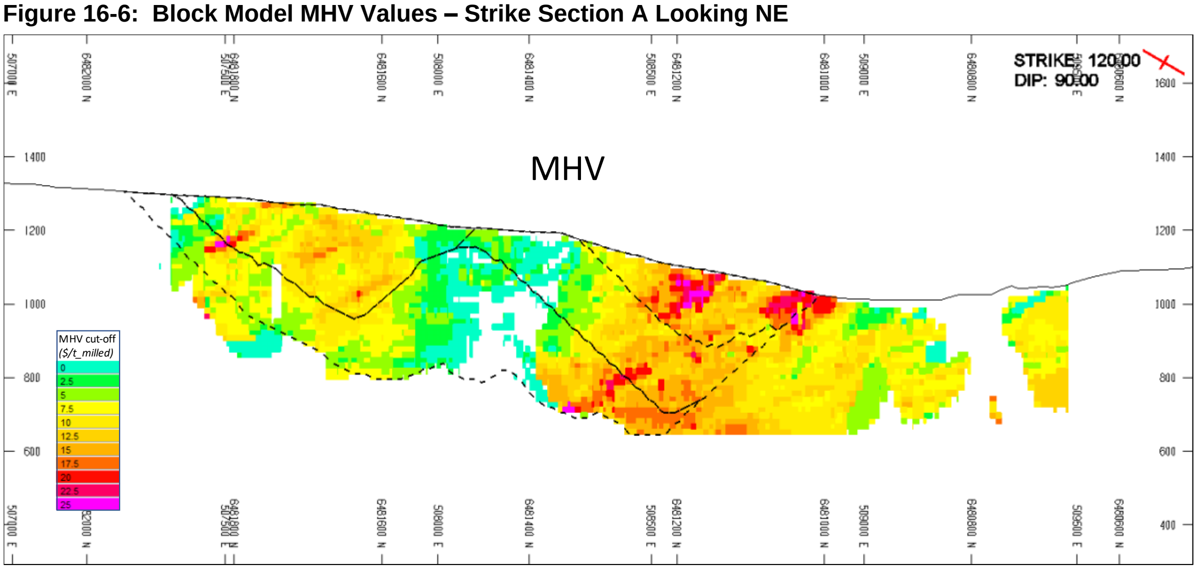
Figure 16-5 is a reference for the locations of the non-orthogonal cross-sectional views in Figures 16-6 to 16-11. The figures show the block model MHV (scheduled) and five scheduled pit stage shells overlain in all sections. Nickel (Ni%) and sulphur (S%) grades are shown in two respective figures to illustrate the impact of these grades on the MHV values along the primary deposit strike length (Section A).

**Figure 16-5: Horsetrail & Ultimate Pit Shell Outlines with Reference Lines**



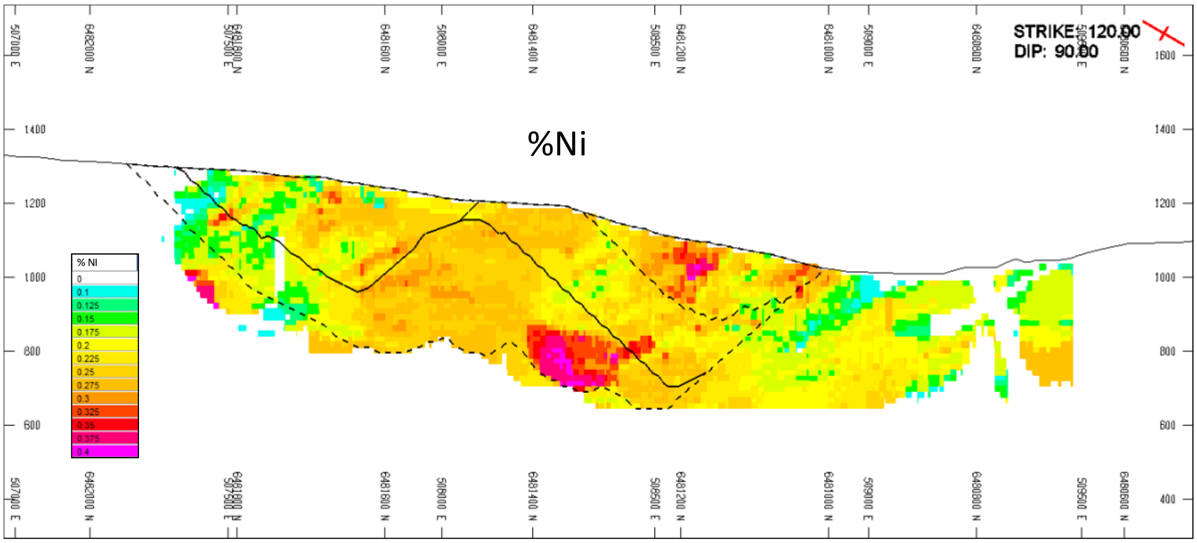

Source: Hatch, 2020.

**Figure 16-6: Block Model MHV Values – Strike Section A Looking NE**



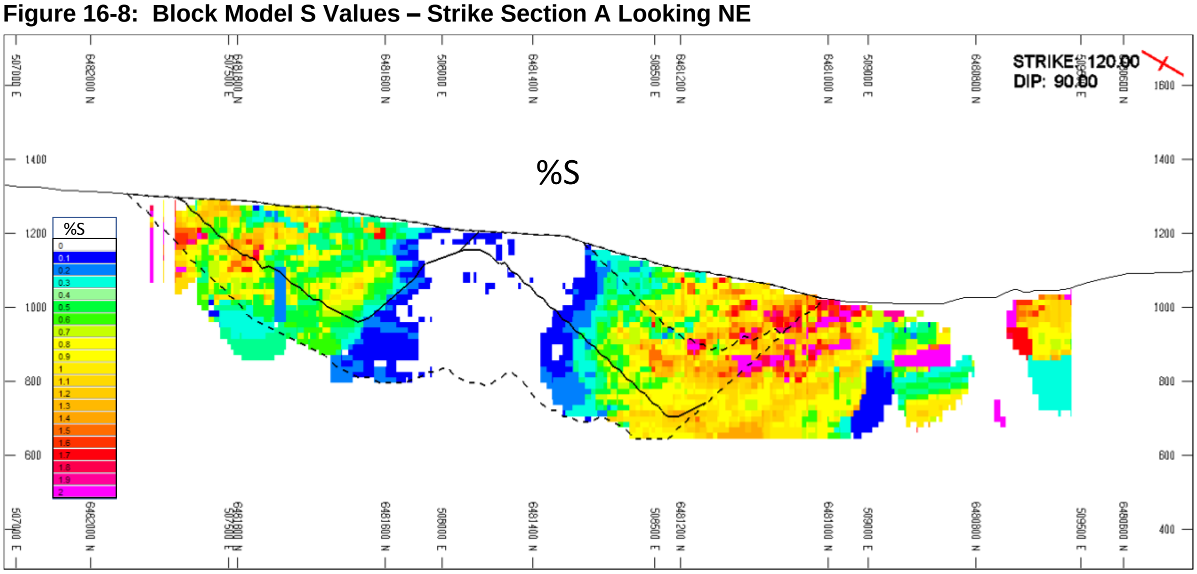

Source: Hatch, 2020.

**Figure 16-7: Block Model Ni Values – Strike Section A Looking NE**



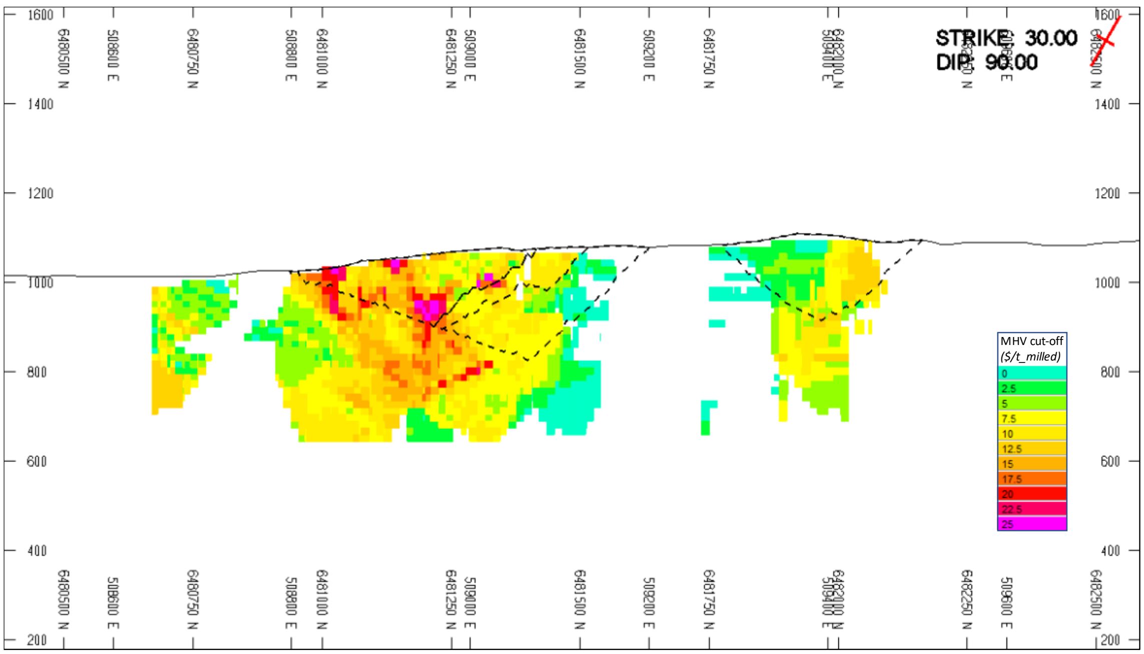

Source: Hatch, 2020.

**Figure 16-8: Block Model S Values – Strike Section A Looking NE**



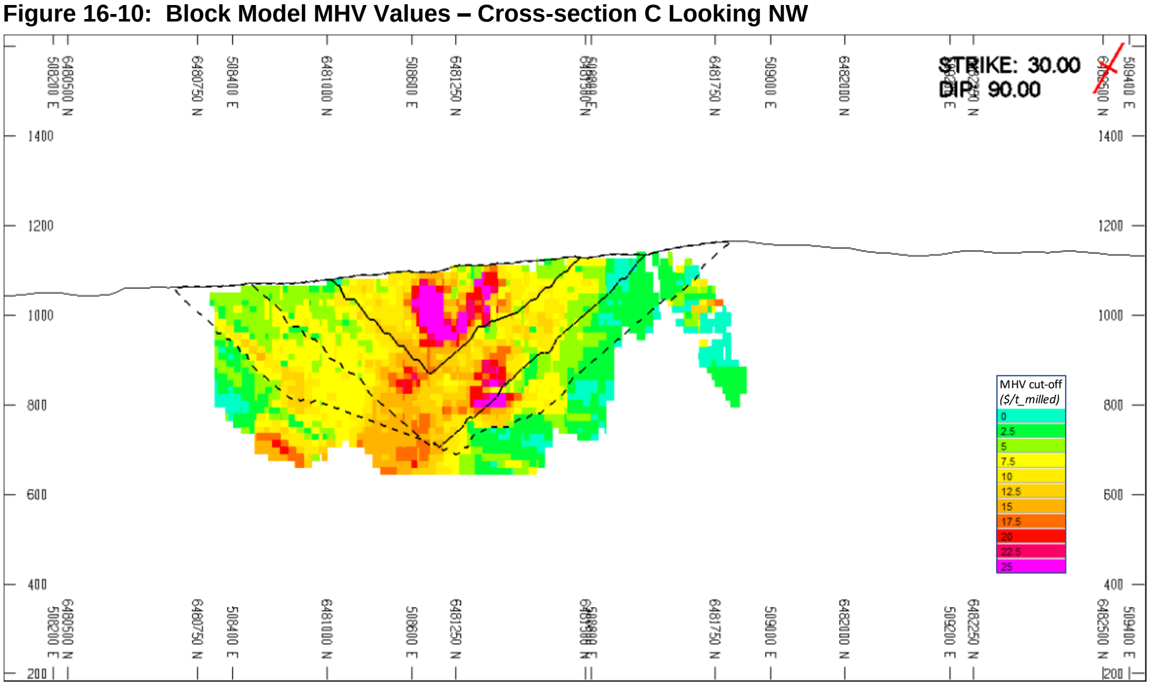

Source: Hatch, 2020.

**Figure 16-9: Block Model MHV Values – Cross-section B Looking NW**



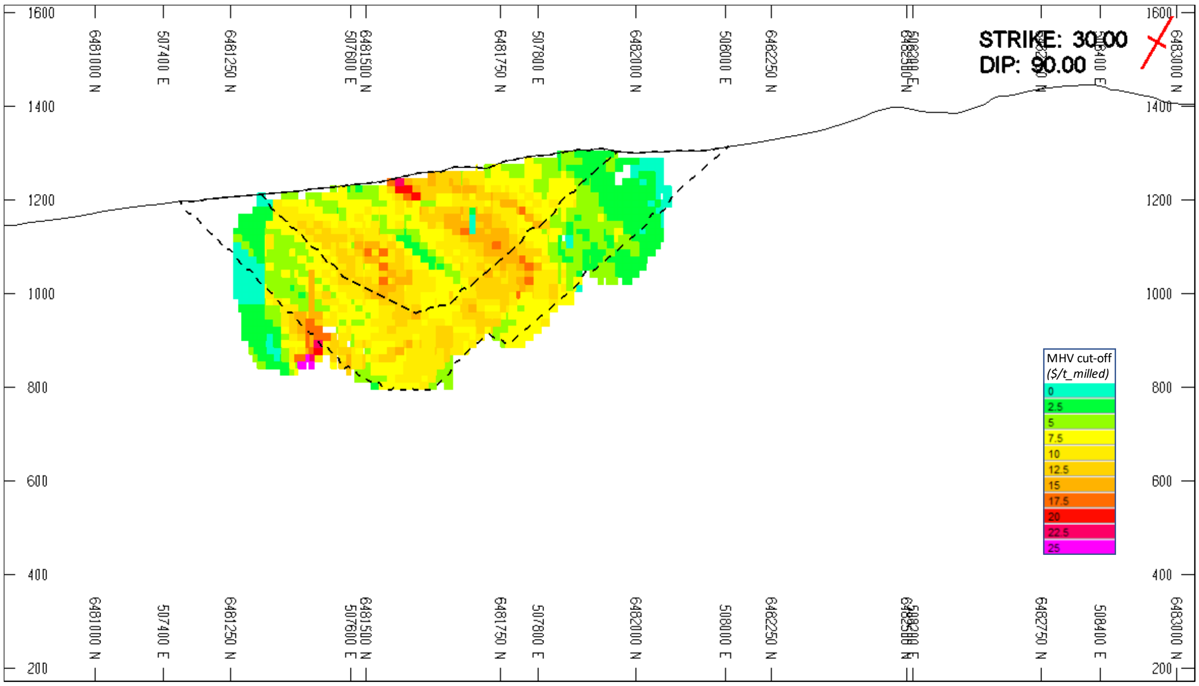

Source: Hatch, 2020.

**Figure 16-10: Block Model MHV Values – Cross-section C Looking NW**




Source: Hatch, 2020.

**Figure 16-11: Block Model MHV Values – Cross-section D Looking NW**




Source: Hatch, 2020.

### 16.3.1 Pit Slope Assumptions

The overall pit slopes have been assigned to an overall maximum angle of 45°. For preliminary open pit analysis, 45° is a typically assumed value and preliminary geotechnical assessments (Piteau, 2008) support this assumption. Pit slope angles and design parameters will require further review and geotechnical assessments for the next level of study.

Groundwater conditions will affect the stability of the pit slopes. It is anticipated that the bedrock is competent and the highwalls will be stable if an effective dewatering plan is implemented to keep them well drained. Limited hydrogeological studies have been undertaken; these will be important during the next level of study, particularly with regard to the proximity of Turnagain River.

### 16.3.2 In-Pit Resource Potential

The potential resource estimate by mineralisation domain and classification contained within the ultimate scheduled Horsetrail pit is summarised in Table 16.6.

Scheduled tonnes and grades are different from the values listed in the pit optimisation shell selection phase (Section 16.2.1), because of the changes in input parameters during the intermediate parameter and recovery update that occurred during the course of this PEA. Additionally the application of a LG minimum cut-off accounting for rehandling affected the total tonnes available for scheduling.

**Table 16.6: Horsetrail Pit Potential Resource**

| Domain | Classification | Mineralisation (Mt) | MHV ($/t Milled) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|
| Dunites and Serpentinised_ Wehrlites | Measured | 142 | 12.21 | 0.235 | 0.014 | 0.796 |
| | Indicated | 388 | 9.38 | 0.222 | 0.013 | 0.610 |
| | Inferred | 248 | 7.66 | 0.230 | 0.013 | 0.396 |
| Green Dunite | Measured | 61 | 9.93 | 0.235 | 0.014 | 0.601 |
| | Indicated | 92 | 7.33 | 0.231 | 0.013 | 0.409 |
| | Inferred | 47 | 4.78 | 0.233 | 0.013 | 0.159 |
| Pyroxenes | Measured | 44 | 11.28 | 0.202 | 0.014 | 1.086 |
| | Indicated | 63 | 7.84 | 0.170 | 0.012 | 1.006 |
| | Inferred | 20 | 6.14 | 0.158 | 0.012 | 0.988 |
| Dyke | Measured | 0 | 3.99 | 0.135 | 0.009 | 0.599 |
| | Indicated | 3 | 4.69 | 0.165 | 0.007 | 0.411 |
| | Inferred | 0 | 3.16 | 0.146 | 0.007 | 0.315 |
| Volcanics | Measured | 1 | 2.92 | 0.120 | 0.008 | 0.653 |
| | Indicated | 6 | 3.50 | 0.129 | 0.009 | 0.608 |
| | Inferred | 7 | 2.79 | 0.100 | 0.009 | 1.130 |
| **Grand Total** | **Measured** | **249** | **11.42** | **0.228** | **0.014** | **0.798** |
| | **Indicated** | **551** | **8.78** | **0.216** | **0.013** | **0.621** |
| | **Inferred** | **322** | **7.03** | **0.223** | **0.013** | **0.414** |

### 16.3.3 Reported Grade Items

The reported grade items are:

- MHV: this is the mill head value combined net smelter return value for contained metal as $/t mineralisation less processing, TMF and G&A unit costs (i.e., the total value at the crusher pocket)
- Ni: total nickel grade percentage
- Co: cobalt percentage
- S: sulphur percentage

### 16.3.4 Cut-off Grade

The scheduled minimum MHV cut-off grade is estimated to be the minimum value of the mineralisation contained in the designed ultimate pit that is sufficient to cover the cost of stockpile handling, stockpile maintenance, and the incremental costs for the haul distance from the stockpile to the crusher. The potential pit scheduled resource estimate is calculated as:

*MHV LG (minimum) cut-off grade = $1.20/t.*

For the purposes of mine production scheduling, elevated (HG) cut-off grades were implemented based on total in-shell tonnes/grade resources ratios and expected mine fleet productivities. The respective applied schedule cut-off grades are summarised in Table 16.7.

**Table 16.7: Scheduling MHV Material Cut-off Grades**

| Pit Stage | LG Cut-off (MHV $/t) | HG Cut-off (MHV $/t) |
|---|---|---|
| Stage_1 | 1.20 | 10.00 |
| Stage_2 | 1.20 | 8.00 |
| Stage_3 | 1.20 | 6.50 |
| Stage_4 | 1.20 | 4.00 |
| Stage_5 | | 1.20 |

During scheduling two categories of LG were applied with the intention of making higher value stockpiled material available for periods where strip ratios were naturally higher and to supplement grades when direct feed grades dipped. The bulk of LG material is intended for processing during the late stages of the mine life.

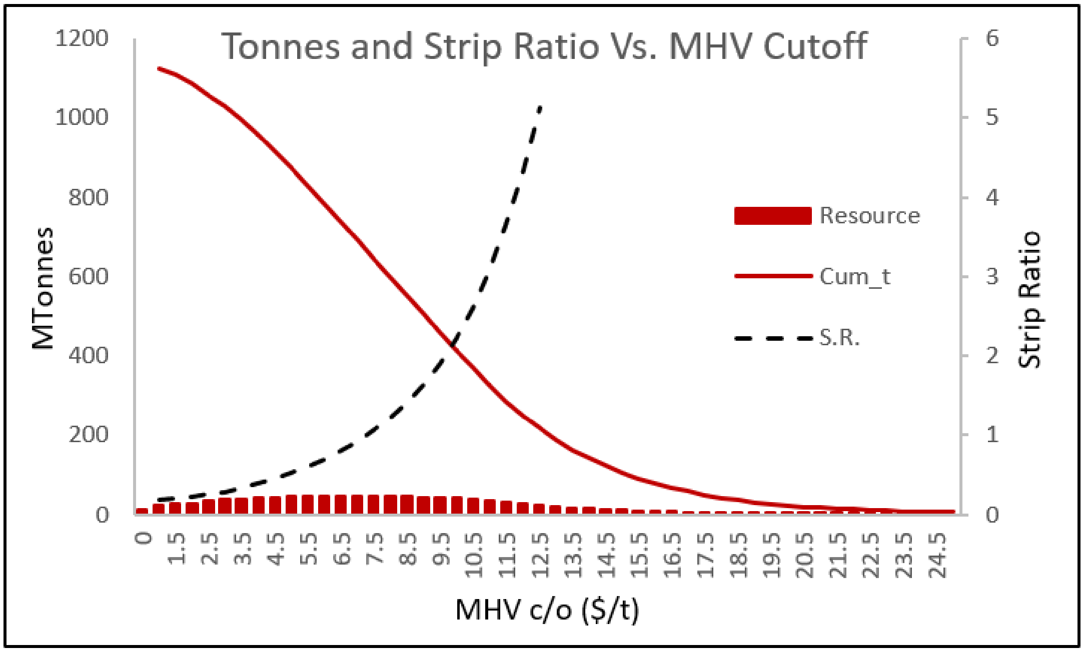
The relatively low strip ratio of the Horsetrail deposit provides significant opportunity to utilise strategic stockpiling to improve dynamic feed values as the pits are developed over the life of mine. However, the opportunity is relatively limited to the immediate circumstances involved, as illustrated by the grade tonnage and strip ratio curves shown in Figure 16-12. The strip ratio curve is extremely steep particularly after a cut-off grade of $7.50 MHV, where the effective strip ratio is 1:1 (i.e., stripping ratios are very sensitive to increased cut-off grades above MHVs of $7.50).

Figure 16-12 summarises the total scheduled tonnes contained in the ultimate shell, while the applied cut-offs listed in Table 16.7 were determined through the respective shell tonne grade curves. The analysis on the cut-off grades will be re-assessed in future studies.

### 16.3.5 Mineralisation 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 mineralisation will require blasting and typical grade control methods using blast-hole sampling. Some dilution is anticipated, specifically when waste is mixed with mineralisation during blasting and excavation activities. The modelled block dimensions are 15 m x 15 m x 15 m, which in consideration of the fleet unit sizes is a mineable unit; therefore, dilution and recovery factors for this evaluation have been set at 0% and 100%, respectively.

**Figure 16-12: Scheduled Ultimate Shell Grade & Tonnage Relationships**




Source: Hatch, 2020.

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 MHV value that is above the calculated economic cut-off value. Therefore, potential 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 mineralisation 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, all pit development will be on the west side of Turnagain River. Pit phases are determined to allow for mine development that will generate maximum cash flow in the initial years and balanced pushbacks thereafter.

The first pit stage, Stage_1, will be in the central zone of the Horsetrail pit with minimal waste stripping. This stage targets near-surface higher value material (Figure 16-6 and Figure 16-10). The second phase, Stage_2, will be a second separate excavation phase to the north that targets the next most accessible source of higher value material. Stage_3 is a pushback of Stage_1 that connects with Stage_2, while Stage_4 is the final pushback encapsulating the previous mine phases and limited by the Turnagain River boundary. Stage_5 is a standalone pit located to the southeast of the main Horsetrail pit. The Stage_5 is the lowest grade (value) phase of the five.

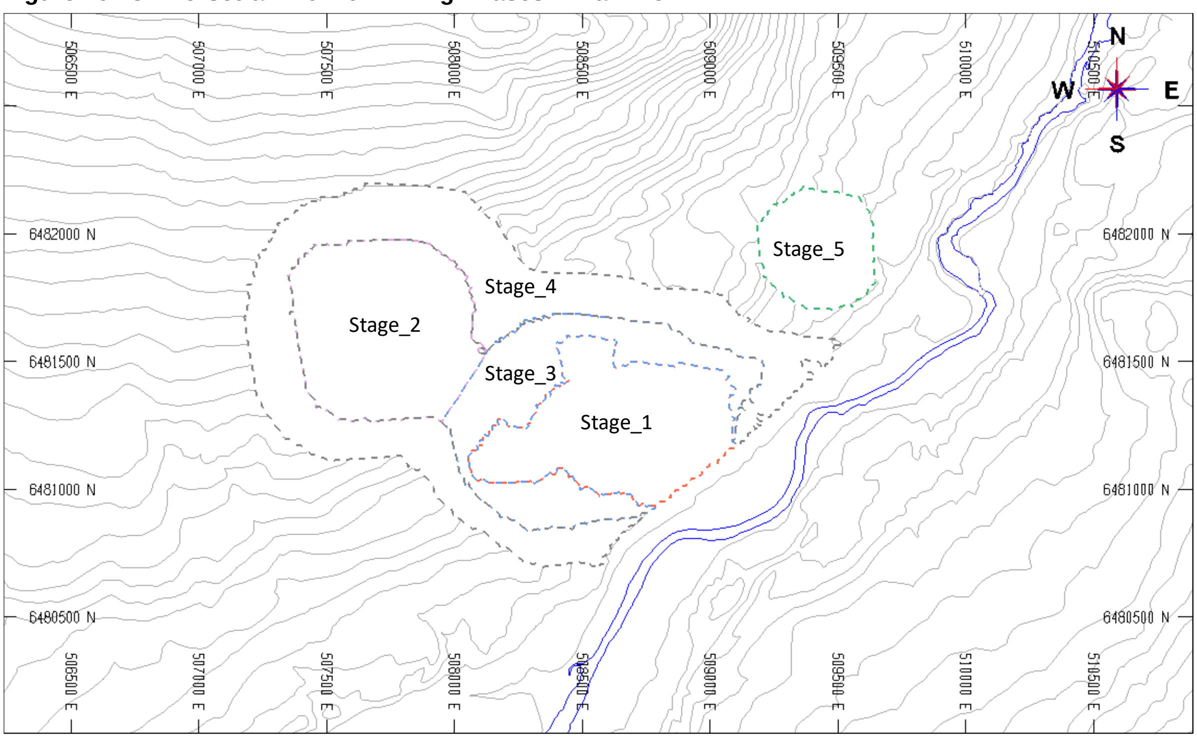
The potential and scheduled Horsetrail pit resource by phase is shown in Table 16.8 and Figure 16-13. A further 66 Mt of material above break-even MHV, but below the applied LG cut-off value (Table 16.7), is also contained within the scheduled optimisation shells.

**Table 16.8: Potential Resource by Horsetrail Pit Phases**

| Pit Phase | Mineralisation (kt) | Waste (kt) | SR (t:t) | MHV ($/t) | Ni (%) | Co (%) | S (%) |
|---|---|---|---|---|---|---|---|
| Stage_1 | 102,322 | 7,195 | 0.07 | 14.1 | 0.25 | 0.02 | 0.99 |
| Stage_2 | 155,009 | 10,183 | 0.07 | 10.3 | 0.22 | 0.01 | 0.69 |
| Stage_3 | 237,436 | 18,084 | 0.08 | 10.0 | 0.22 | 0.01 | 0.72 |
| Stage_4 | 596,710 | 159,292 | 0.27 | 7.3 | 0.22 | 0.01 | 0.48 |
| Stage_5 | 30,498 | 13,125 | 0.43 | 6.2 | 0.22 | 0.01 | 0.28 |
| **Total Horsetrail** | **1,121,976** | **207,880** | **0.19** | **8.86** | **0.22** | **0.01** | **0.60** |

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

**Figure 16-13: Horsetrail Pit with Mining Phases – Plan View**




Source: Hatch, 2020.

## 16.5 Mine Production Schedule

The annual mill steady state feed rate for the plant Phase_1, Years 1 through 5, is set at 45 kt/d (16.4 Mt/a) and for the balance of the mine life, Years 6 through 37, at 90 kt/d (32.9 Mt/a). Previous studies implemented variable feed rates with the SAG mill comminution circuit configuration, while for this study an HPGR circuit has been implemented. This is understood to be less sensitive to material hardness as per throughput rates.

Implementing a cut-off grade strategy is commonly used to optimise the mineralisation feed to the mill so that the NPV of the project is maximised. During the life of mine, mineralisation mined above the respective pit shell cut-off grades listed in Table 16.7 will be directly fed to the mill. Mineralisation below the HG cut-off but above the respective LG cut-offs will be stockpiled. In developing the mine production schedule, LG has been further split into material above $6.00 MHV and below. The higher value LG material is scheduled for rehandling intermittently when beneficial to boost grades and or reduce required stripping. The lower value LG material is scheduled for rehandling after completion of the pits. Due to the low strip ratio of the deposit, it is foreseen that operators will be able to take advantage of a variable direct feed cut-off based on equipment productivities on site and the active proportion of waste material for any given period to maximise income.

Oxidation of mineralisation 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(s) and stockpiling strategy is viable.

A preliminary production forecast is shown in Table 16.9. Pre-stripping will not be necessary, as the initial mineralisation feed will be near the surface and accessible when the plant starts up. Over the life of the pit, approximately 1,122 Mt of mineralisation will be fed to the mill, of which 881 Mt will be directly from the pit and 241 Mt will be stockpiled and reclaimed largely at the end of mine life when the pits are depleted. The average strip ratio (waste/mineralisation) is 0.2 for LOM and peaks in Year 17 at 0.7. This ratio may be somewhat misleading, as mineralisation stockpiling activity is not represented. Table 16.10 summarises the actual and effective strip ratios by grouped periods incorporating stockpile material.

**Table 16.9: Production Forecast**

| Period | Direct Feed (Mt) | To Stockpile (Mt) | Stockpile Reclaim (Mt) | Plant Feed (Mt) | Ni (%) | Co (%) | S (%) | Waste (Mt) | Recovered Ni (kt) | Strip Ratio (t:t) |
|---|---|---|---|---|---|---|---|---|---|---|
| Pre-prod | - | - | - | - | - | - | - | - | - | |
| 1 | 10.68 | 5.62 | - | 10.68 | 0.252 | 0.015 | 0.782 | 4.16 | 14.3 | 0.3 |
| 2 | 16.43 | 7.39 | - | 16.43 | 0.253 | 0.015 | 0.978 | 2.35 | 23.2 | 0.1 |
| 3 | 16.43 | 4.97 | - | 16.43 | 0.26 | 0.016 | 1.148 | 0.67 | 24.7 | 0 |
| 4 | 16.43 | 2.74 | - | 16.43 | 0.261 | 0.016 | 1.313 | 0.01 | 25.4 | 0 |
| 5 | 16.43 | 7.46 | - | 16.43 | 0.272 | 0.016 | 1.206 | 3.13 | 26.1 | 0.1 |
| 6 | 31.21 | 10.5 | - | 31.21 | 0.234 | 0.014 | 0.703 | 3.65 | 38.0 | 0.1 |
| 7 | 32.85 | 13.21 | - | 32.85 | 0.231 | 0.013 | 0.761 | 1.96 | 40.1 | 0 |
| 8 | 32.85 | 8.3 | - | 32.85 | 0.232 | 0.013 | 0.808 | 2.82 | 40.8 | 0.1 |
| 9 | 32.85 | 11.2 | - | 32.85 | 0.241 | 0.014 | 0.586 | 5.04 | 39.4 | 0.1 |
| 10 | 32.85 | 15.18 | - | 32.85 | 0.224 | 0.013 | 0.577 | 7.67 | 36.5 | 0.2 |
| 11 | 32.85 | 14.09 | - | 32.85 | 0.211 | 0.013 | 0.767 | 3.36 | 36.7 | 0.1 |
| 12 | 32.85 | 7.86 | - | 32.85 | 0.214 | 0.014 | 0.811 | 0.64 | 37.8 | 0 |
| 13 | 32.85 | 2.14 | - | 32.85 | 0.224 | 0.014 | 0.893 | 0.01 | 40.4 | 0 |
| 14 | 32.85 | 4.35 | - | 32.85 | 0.242 | 0.015 | 1.046 | 9.35 | 45.1 | 0.3 |
| 15 | 27.85 | 9.14 | 5 | 32.85 | 0.232 | 0.014 | 0.606 | 25.4 | 38.2 | 0.7 |
| 16 | 17.85 | 7.62 | 15 | 32.85 | 0.211 | 0.012 | 0.44 | 14.61 | 32.2 | 0.6 |
| 17 | 12.85 | 9.46 | 20 | 32.85 | 0.206 | 0.012 | 0.507 | 16.63 | 32.6 | 0.7 |
| 18 | 25.35 | 9.85 | 7.5 | 32.85 | 0.202 | 0.012 | 0.554 | 13.33 | 32.6 | 0.4 |
| 19 | 32.85 | 14.24 | - | 32.85 | 0.194 | 0.012 | 0.676 | 9.08 | 32.9 | 0.2 |
| 20 | 32.85 | 14.66 | - | 32.85 | 0.201 | 0.012 | 0.675 | 10.22 | 34.0 | 0.2 |
| 21 | 32.85 | 15.53 | - | 32.85 | 0.208 | 0.012 | 0.575 | 11.53 | 33.9 | 0.2 |
| 22 | 32.85 | 12.89 | - | 32.85 | 0.211 | 0.012 | 0.509 | 9.11 | 33.5 | 0.2 |
| 23 | 32.85 | 5.6 | - | 32.85 | 0.212 | 0.012 | 0.516 | 3.94 | 33.6 | 0.1 |
| 24 | 32.85 | 9.95 | - | 32.85 | 0.212 | 0.012 | 0.532 | 6.54 | 33.9 | 0.2 |

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| 25 | 32.85 | 3.61 | - | 32.85 | 0.216 | 0.012 | 0.522 | 3.11 | 34.3 | 0.1 |
| 26 | 32.85 | 5.9 | - | 32.85 | 0.22 | 0.013 | 0.5 | 6.01 | 34.7 | 0.2 |
| 27 | 32.85 | 3.52 | - | 32.85 | 0.225 | 0.013 | 0.482 | 5.89 | 35.1 | 0.2 |
| 28 | 32.85 | 2.83 | - | 32.85 | 0.232 | 0.013 | 0.497 | 7.26 | 36.5 | 0.2 |
| 29 | 32.85 | 0.7 | - | 32.85 | 0.251 | 0.014 | 0.519 | 3.86 | 40.0 | 0.1 |
| 30 | 32.85 | 0.61 | - | 32.85 | 0.266 | 0.015 | 0.612 | 7.27 | 44.0 | 0.2 |
| 31 | 30.09 | - | 2.76 | 32.85 | 0.23 | 0.014 | 0.341 | 9.28 | 32.7 | 0.3 |
| 32 | 2.28 | - | 30.57 | 32.85 | 0.201 | 0.012 | 0.355 | - | 28.9 | 0 |
| 33 | - | - | 32.85 | 32.85 | 0.199 | 0.012 | 0.33 | - | 28.1 | |
| 34 | - | - | 32.85 | 32.85 | 0.199 | 0.012 | 0.33 | - | 28.1 | |
| 35 | - | - | 32.85 | 32.85 | 0.199 | 0.012 | 0.33 | - | 28.1 | |
| 36 | - | - | 32.85 | 32.85 | 0.199 | 0.012 | 0.33 | - | 28.1 | |
| 37 | - | - | 28.89 | 28.89 | 0.199 | 0.012 | 0.33 | - | 24.7 | |
| 38 | - | - | - | - | - | - | - | - | | |
| **Total** | **880.85** | **241.12** | **241.12** | **1,121.98** | **0.221** | **0.013** | **0.601** | **207.88** | **1229.0** | **0.2** |

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

**Table 16.10: Strip Ratio – Actual & Effective by Grouped Periods**

| Mining Strip Ratio | Ave LOM (t:t) | Max (t:t) | Years 1-5 (t:t) | Years 6-15 (t:t) | Years 16-21 (t:t) | Years 22-31 (t:t) | Years 32-38 (t:t) |
|---|---|---|---|---|---|---|---|
| Actual (waste/mineralisation) | 0.19 | 0.7 | 0.10 | 0.14 | 0.33 | 0.17 | 0.00 |
| Effective (waste+stockpile/feed) | 0.40 | 1.1 | 0.50 | 0.48 | 0.74 | 0.30 | 0.00 |

## 16.6 Mine Equipment

### 16.6.1 Mobile Fleet

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

Table 16.11 summarises 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 | Units on Site | |
|---|---|---|---|---|
| | | | Years 1-2 | Max. |
| Blast-hole Electric Drill | Primary drill | 311 mm | 2 | 5 |
| Surface Crawler Percussion Drill | Pioneering/wall control | 155 mm | 1 | 1 |
| Cable Shovel | Production loading | 27 $m^3$ | 2 | 3 |
| Wheel Loader | Backup loader & stockpile handling | 21 $m^3$ | 1 | 1 |
| Haul Trucks | Production haulage | 221 t | 5 | 22 |
| Track Dozer (D10 Equivalent) | Road development & maintenance | 425 kW | 2 | 3 |
| Track Dozer (D9 Equivalent) | Pit maintenance | 302 kW | 1 | 1 |
| RT Wheel Dozer | Shovel support | 597 kW | 1 | 2 |
| Grader (24' Blade) | Road maintenance | 518 kW | 1 | 2 |
| Grader (16' Blade) | Road maintenance | 216 kW | 1 | 1 |
| Water/Sanding Truck | Road maintenance | 20,000 L | 1 | 2 |

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 27 $m^3$ electric rope shovels and 220 t trucks. The shovel size was selected to meet the annual mineralisation and waste production requirements from the pits. Each unit has the capacity to produce an average of 22 Mt/a. It is estimated that a maximum of three shovels will be required during the life of mine. Electric rope shovels were chosen over hydraulic excavators due to their lower operating costs, long life and on the assumption that grid power will be provided to site. A production class front-end loader (21 $m^3$) has been included in the fleet as a backup, construction, and ROM stockpile digging unit.

The size of the trucks has been selected to match the shovel output for four-pass loading. The size of the fleet is determined by estimating the haulage productivities for mineralisation and waste. Preliminary estimates on the haulage productivities indicate that 5 units will be required by Year 2, increasing to a maximum of 22 units by Year 15. 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 pioneering and potentially pre-shearing.

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 water/sanding trucks.

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.

Further description of the respective mine support infrastructure is included in Section 18.0.

## 16.7 Waste Disposal & LG Stockpile Management Facility

Mine waste rock and long-term stockpile material will be hauled from the Horsetrail pit and placed on a waste and stockpile facility located southwest of the pit. The preliminary waste dump and LG stockpile design is constrained by Turnagain River, Hard Creek, and planned site infrastructure (Figure 16-2). Rock chemistry data indicate the waste rock to be non-potentially acid generating. The majority of the rock is assumed to be competent and able to be end-dumped on high lift intervals. The foundation under the waste dump has not been analysed for stability. It is necessary to confirm the geotechnical viability of the dumpsite and material geochemistry for the next level of study particularly with the proximity of Turnagain and Hard Creek rivers.

The end-of-mine waste dump will be approximately 45% of the volume shown in Figure 16-2 once all LG has been reclaimed. Previous studies had scheduled waste rock to be hauled and placed as buttress material for the TMF main embankment located 4 km upstream from the mouth of Flat Creek in the Flat Creek Valley. Further analysis and trade-off studies are required to determine the viability of such a scenario; although if confirmed, the mine waste dump complex at the end of mine life might reduce to 50 Mt of waste or a reduction of 40% volume, as shown in Figure 18-2.

Overburden and soil will be stockpiled in separate piles accessible for ongoing or end-of-mine reclamation activities.

A ROM crusher stockpile will be located in close proximity to the primary crusher providing temporary storage for material not fed to the crusher directly (for various operational reasons).

# 17.0 RECOVERY METHODS

## 17.1 Introduction

The processing plant proposed for the Turnagain project was designed to overcome the challenges imposed by its highly competent nickel ore. To address these challenges, appropriate comminution equipment selection is vital to the overall economics of this project by ensuring proper size reduction and recovery of nickel-bearing minerals. Building from the previous 2011 AMC PEA and Hatch's previous 2018 conceptual study, the mineral processing plant was designed to treat 90,000 t/d of ROM ore and consists of a crush-grind-flotation flowsheet.

Previous studies performed by Hatch for the design of the Turnagain concentrator considered the use of a semi-autogenous grinding (SAG) mill as the principal method of coarse grinding. This study considers the use of a high-pressure grinding roll (HPGR) as a tertiary crushing circuit and the use of two ball mill circuits in series for grinding in order to reach the particle sizes required for flotation.

HPGR technology has been selected primarily due to higher energy efficiency compared to SAG milling. In recent years, HPGR technology has seen wider acceptance and use in operations, specifically the application of large, high throughput HPGR systems. Recent operations, such as Freeport-McMoRan's Metcalf concentrator, have seen successful implementation of large HPGR systems as the principal comminution equipment.

One key consideration in the design of the processing plant is the phased concentrator construction approach, whereby the concentrator starts up at 50% capacity before expanding to full capacity (i.e., 45,000 t/d for the first five years, and 90,000 t/d thereafter). The implications of this are discussed briefly in Section 17.4.

## 17.2 Process Design Criteria

The process design criteria detail mineral processing design considerations for the Turnagain concentrator including annual ore and product capabilities, plant availability, and capacities. Table 17.1 provides a summary of key design criteria.

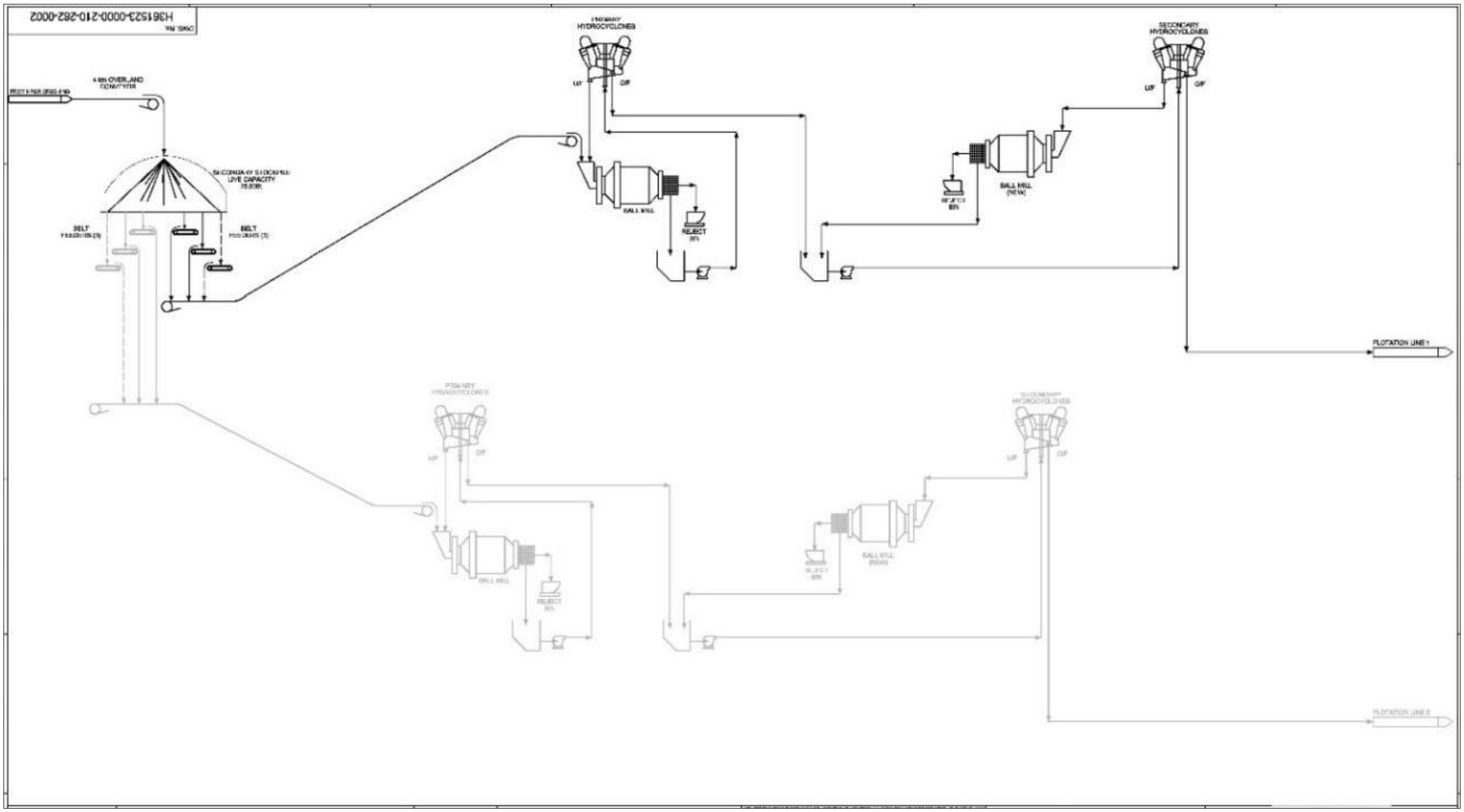
## 17.3 Process Description

The process flowsheets for the proposed plant are shown in Figure 7-1, Figure 7-2, and Figure 7-3.

**Table 17.1: Key Process Design Criteria**

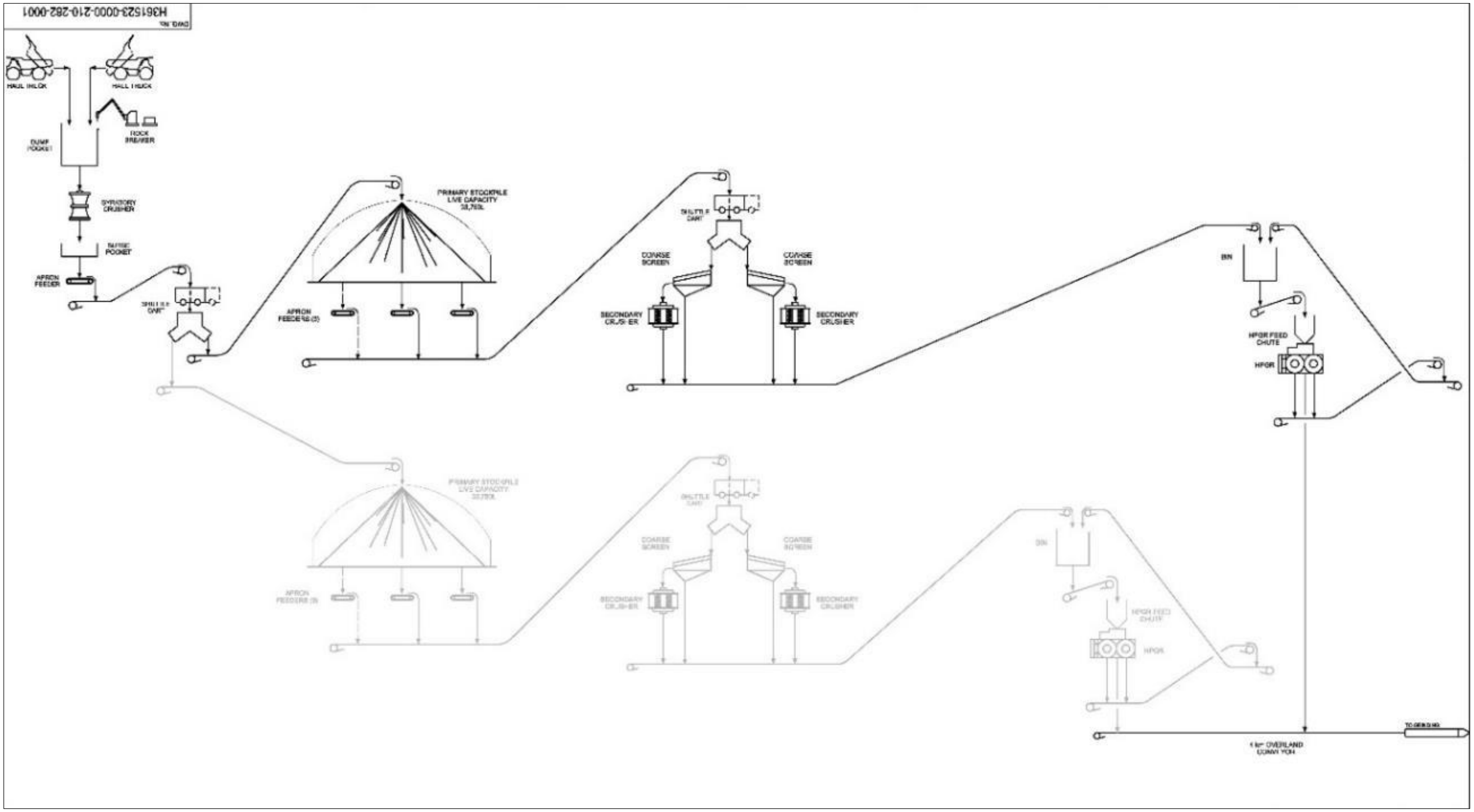
| Area | Criteria | Unit | Nominal Value |
|---|---|---|---|
| General | Primary Crushing Availability | % | 70 |
| | Secondary Crushing Availability | % | 88 |
| | HPGR Circuit Availability | % | 88 |
| | Concentrator Availability | % | 94 |
| | Nickel Head Grade | | |
| | 45 kt/d (Year 2-5 Average) | % | 0.261 |
| | 90 kt/d (Year 6-10 Average) | % | 0.232 |
| | Sulphur Head Grade | | |
| | 45 kt/d (Year 2-5 Average) | % | 1.16 |
| | 90 kt/d (Year 6-10 Average) | % | 0.69 |
| | Average Nickel Recovery | | |
| | 45 kt/d (Year 2-5 Average) | % | 57.9 |
| | 90 kt/d (Year 6-10 Average) | % | 51.6 |
| | Average Annual Concentrate Production (18% Ni) | | |
| | 45 kt/d (Year 2-5 Average, 9% moisture) | t/a | 151,814 |
| | 90 kt/d (Year 6-10 Average, 9% moisture) | t/a | 237,902 |
| | Average Annual Nickel Production | | |
| | 45 kt/d (Year 2-5 Average) | t/a | 24,867 |
| | 90 kt/d (Year 6-10 Average) | t/a | 38,968 |
| Crushing | Crusher Work Index | kWh/t | 18.2 |
| | ROM Ore Top size | mm | 1000 |
| | Crusher + HPGR Circuit Product size ($P_{80}$) | mm | 4.0 |
| | Primary Stockpile Capacity (live) | hr | 18 |
| | Secondary Stockpile Capacity (live) | hr | 13.3 |
| Grinding | Bond Ball Mill Work Index | kWh/t | 19.8 |
| | Grinding Circuit Product size ($P_{80}$) | µm | 80 to 85 |
| Flotation | Rougher Flotation Design Retention Time | min | 67 |
| | 1$^{st}$ Cleaner Design Retention Time | min | 27 |
| | 1$^{st}$ Cleaner Scavenger Design Retention Time | min | 16 |
| | 2$^{nd}$ Cleaner Design Retention Time | min | 32 |
| | 3$^{rd}$ Cleaner Design Retention Time | min | 35 |
| Concentrate Thickening & Filtration | Thickener Solids Loading | $t/h/m^2$ | 0.25 |
| | Thickener Underflow Density | Wt% | 55 |
| | Filter Cake Moisture | Wt% | 9 |

**Figure 17-1: Process Flow Diagram – Crushing Circuit**



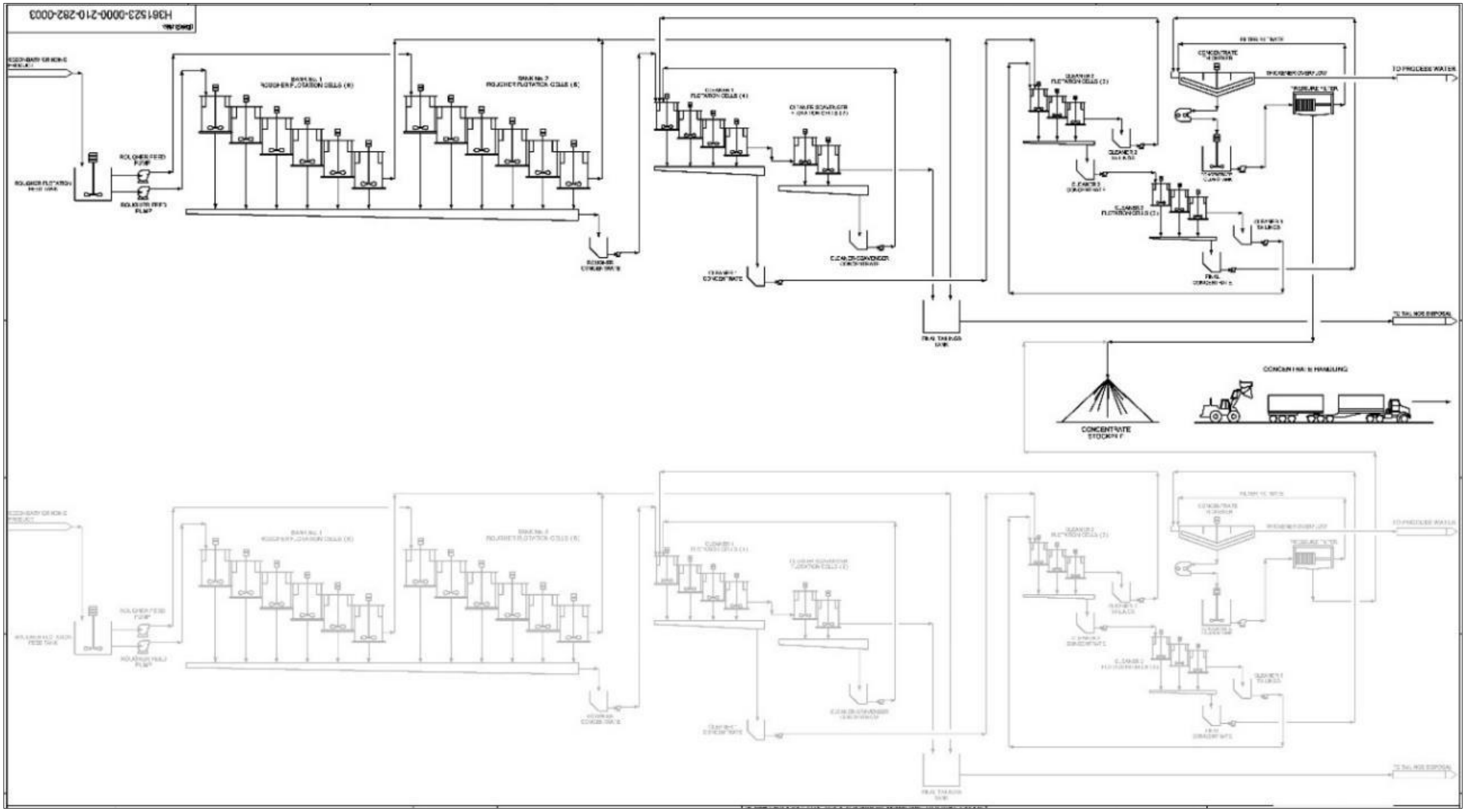

Source: Hatch, 2020.

**Figure 17-2: Process Flow Diagram – Grinding Circuit**




Source: Hatch, 2020.

**Figure 17-3: Process Flow Diagram – Flotation Circuit**




Source: Hatch, 2020.

The process plant, when running at full capacity (90,000 t/d), will consist of the following major unit operations:

- ROM crusher feed hopper
- one primary crusher and two crushed ore stockpiles including:
  - conveyors
  - ore reclaiming systems
- two parallel secondary crushing circuits, with each circuit consisting of:
  - two 3.0 x 7.3 m double-deck coarse screens in parallel
  - two cone crushers in parallel, each with 933 kW motors
- two parallel tertiary crushing HPGR circuits, with each circuit consisting of:
  - 2.6 m diameter x 2.4 m wide HPGR with 9,000 kW of installed power
  - one HPGR edge recycle conveying system
- two parallel primary ball mill grinding circuits, with each circuit consisting of:
  - 7.9 m diameter x 13.7 m long (26  x 45 ft) ball mill with 18,000 kW of installed power
  - one 14-place cluster of 0.84 m diameter cyclones
- two parallel secondary ball mill grinding circuits, with each circuit consisting of:
  - 7.9 m diameter x 13.7 m long (26  x 45 ft) ball mill with 18,000 kW of installed power
  - one 10-place cluster of 0.66 m diameter cyclones
- four parallel trains of rougher flotation, each with 6 x 630 m$^3$ tank cells
- two parallel trains of cleaner flotation in three stages
- concentrate thickening and filtration
- tailings facility reclaim water system (part of the TMF) for process water
- fresh water will be used for pump gland seals, reagent mixing and other special requirements (e.g., hydraulic unit cooling).

All buildings and conveyors are enclosed with proper dust suppression and collection systems to address exposure to any silicates or fibrous materials.

ROM plant feed will be delivered to a dump pocket by haul trucks on top of the gyratory crusher. Crushed material from the gyratory crusher will be reclaimed by an apron feeder and conveyed to the primary stockpile.

Ore from the primary stockpile will be reclaimed using apron feeders and transported to the secondary crushing splitter chute via conveyor belt. A self-cleaning permanent magnet will be installed along the conveyor belt to remove unwanted tramp metals from the bulk materials ahead of the secondary crushing circuit. The two secondary cone crushers are operated in an open circuit with fines scalping and parallel configuration. There is a double-deck, multi-slope, vibrating screen installed ahead of each of the secondary cone crushers with the crusher fed directly from the screen oversize. Conveyors will have variable speed drives to facilitate the crusher choke feed requirement. Subsequently, the crushed product will join the screen undersize onto a conveyor belt before being transported to the HPGR feed bin. The edge material from the HPGR discharge will be recycled back to the HPGR feed, while the HPGR product (centre) will proceed to the secondary stockpile at the concentrator plant via a 4 km overland conveyor.

Each grinding circuit consists of two closed-circuit (with hydrocyclones) ball mills operated in series for primary and secondary grinding. The primary ball mill will be operated in a “direct” closed-circuit configuration with a cyclone cluster; the primary ball mill will receive feed from the secondary stockpile reclaim. The primary ball mill discharge will pass through a trommel for ball scats removal and into the pump box prior to entering the primary cyclone cluster for size classification. Primary cyclone underflow will feed back to the primary ball mill. Primary cyclone overflow will flow by gravity to a pump box and be pumped to the secondary cyclone cluster. Secondary cyclone underflow will flow to the feed of the secondary ball mill. The secondary ball mill will be operated in an “indirect” configuration, as the secondary ball mill feed must first pass through the secondary cyclone cluster to scalp final-grind material to the overflow stream. The secondary ball mill product will pass through a trommel before entering back to the pump box ahead of the secondary cyclone cluster. Secondary cyclone cluster overflow will report to the rougher feed conditioning tank prior to being pumped to the rougher flotation circuit.

A conventional sequential flotation flowsheet consisting of a rougher, cleaner and cleaner scavenger will be used to produce the final Ni concentrate. Regrinding is not currently included. Limited testwork has shown a potential increase in recovery, and ongoing testwork is intended to determine the magnitude of any such improvement. This will enable trade-off studies to be completed on possibly adding regrinding to the flowsheet at a later stage. The concentrate cleaning has been designed with three stages in the plant, using conventional mechanical flotation cells.

The concentrate products will be dewatered in high-rate thickeners with the underflow feeding filter feed stock tanks. The pressure filter will dewater the Ni concentrate to a moisture content of 9%. The Ni concentrate will then be discharged directly from the pressure filter to the concentrate shed for truck load-out. The rougher tailings and cleaner scavenger tailings will flow by gravity to the TMF for deposition. Reagents consumed within the flotation circuits are prepared within the reagent handling and make-up area. This facility includes mixing and storage for MIBC as frother, SIPX as collector, Calgon as depressant, and flocculant.

## 17.4 Phased Expansion

The mineral processing plant is designed for two grinding trains, four rougher flotation trains, and two cleaner and cleaner scavenger trains at full capacity. This configuration allows for phased construction of the mill. The first phase at 50% of the full capacity will simply comprise one crushing train, one grinding train, two rougher flotation trains, and one cleaner and cleaner-scavenger train. For the first five years, the plant throughput will average 16.4 Mt/a. Thereafter, for Years 6 to 35, the average throughput will be 32.9 Mt/a.

The primary crushing circuit is designed to operate during both phases, as it is larger than required for the first phase throughput. This design introduces the opportunity for optimising the operating schedule of the primary crusher during the first phase (to save on power and maintenance costs) which can be explored in future studies. The overland conveyor and secondary stockpile have also been designed to operate during both phases of plant operation.

This phased approach will provide site experience for the construction and ramp-up of the proposed concentrator design, allowing for optimisation of construction and ramp-up for the second phase expansion. There is potential to improve the economics of this project by altering this phasing plan, either by constructing at full scale immediately or by decreasing the time between construction of the two concentrator phases, in order to increase production rates as quickly as possible.

Other opportunities for improvements and alternatives to the Turnagain concentrator design are presented in Section 25.8.

# 18.0 PROJECT INFRASTRUCTURE

## 18.1 Site Access Road

The communities of Terrace and Smithers in BC, and Whitehorse in the Yukon, are all several hundred kilometres away and offer the best range of supplies and services. Supplies can be trucked via the Stewart-Cassiar Highway (Highway 37) to 8 km south of Dease Lake, a small community in northwest BC, approximately 250 km south of the Yukon border. The Turnagain property can be accessed by a trail extending east from Highway 37. The trail has been used by large, articulated four-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 trail network extends into the Turnagain property.

The road distance to the site from Highway 37 is approximately 78 km. The access trail from Highway 37 to the property will require upgrading. There are two major stream crossings and approximately 50 other minor crossings en route to the site. These crossings are currently passable with light vehicles, but will need upgrading for larger trucks. Bridges are required for the major crossings; culverts will likely be required for the minor crossings.

For safe travel with the anticipated volume of traffic, especially considering the potential for multiple uses of the road, an 8 m wide all-weather road built to BC Forestry standards to allow two-way traffic is envisioned. The existing access road would be widened and upgraded.

As shown on Figure 18-1, there is the possibility of an alternate route on the north side of the Turnagain River (instead of following the trail currently on the south side) for the last 16 km to the site to reduce the number of Turnagain River crossings (bridges) and reduce cost. This will be further considered in the next study stage of the project.

The progress of the Kutcho project and road user discussion group outcomes will also be considered. There may be an opportunity to share the cost of upgrading 54 km of this road with Kutcho Copper Corporation. With the current trail and road route, two river crossings will require bridges: one on the main access road and the other to the open pit side of the river. To be able to efficiently access all areas of the planned site, a network of internal roads will also 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.

**Figure 18-1: Potential Access Road along North Side of Turnagain River**


Source: Giga Metals, 2020.

## 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 TMF sites in the vicinity of the Turnagain deposit (e.g., Flat Creek Valley was acknowledged as a potential option for a tailings impoundment). In July 2007, a preliminary mine development alternatives assessment was carried out (Ref. No. VA07-01017) to explore several mine development alternatives, including the location of the plant site, waste dumps, low-grade stockpile, haul and construction access roads, and tailings and water reclaim pipelines. This assessment of alternatives should be updated during the next phase of study.

In 2019, further site configuration and trade-off analysis was performed by Hatch (Siting Location Trade-off Study, H355439-00000-210-230-0001). Primary objectives of the study included the following:

- placement of the plant site within the same catchment as the TMF
- minimise footprint with regards to civil/earthworks
- incorporate dust considerations in site selection
- consider plant expandability
- minimise power consumption demands

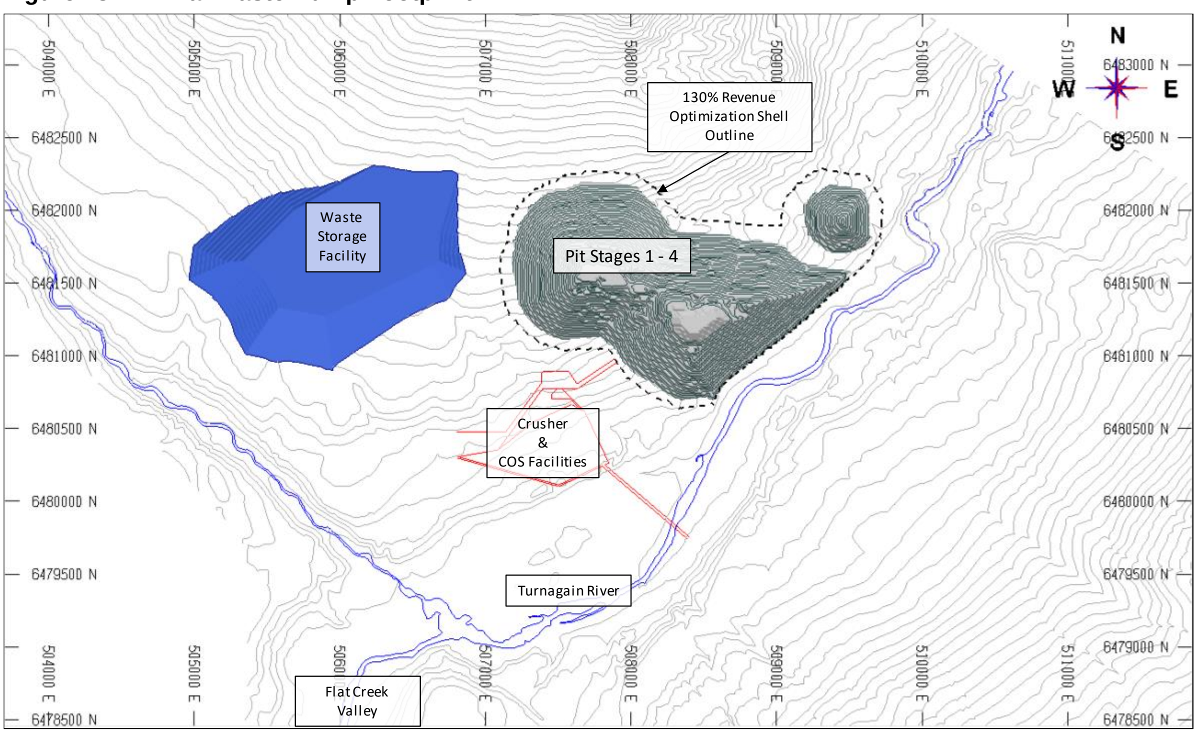
Based on the study results, the plant site was relocated upslope to the north of the TMF main abutment to facilitate gravity tails flow, and the HPGR crushing facility was placed with the primary crusher system in the valley near the open pit operations. The HPGR product will be conveyed across the Turnagain River and upslope to the processing plant. Conveying instead of high-pressure slurry pumping across the Turnagain River reduces the risk of significant spills as well as power demand.

## 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. The waste dump is sized for the entirety of the expected waste rock; possible use of waste rock for TMF construction will reduce the required size of the waste dump. The potential re-use of waste rock will be investigated further (geotechnical, geochemical, economic) in the next stage of engineering.

**Figure 18-2: Final Waste Dump Footprint**




Source: Hatch, 2020.

### 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 characterisation to date is largely inconclusive on the acid generating potential of the waste rock, since most of the testwork was carried out on mineralised material. Although a small percentage of the waste may have acid generating potential, it is expected to be insignificant with respect to the overall neutralising 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 neutralising 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.

Additional characterisation work is under way but was not available for this study. This work will be incorporated in the next phase of study.

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 & Operating Criteria

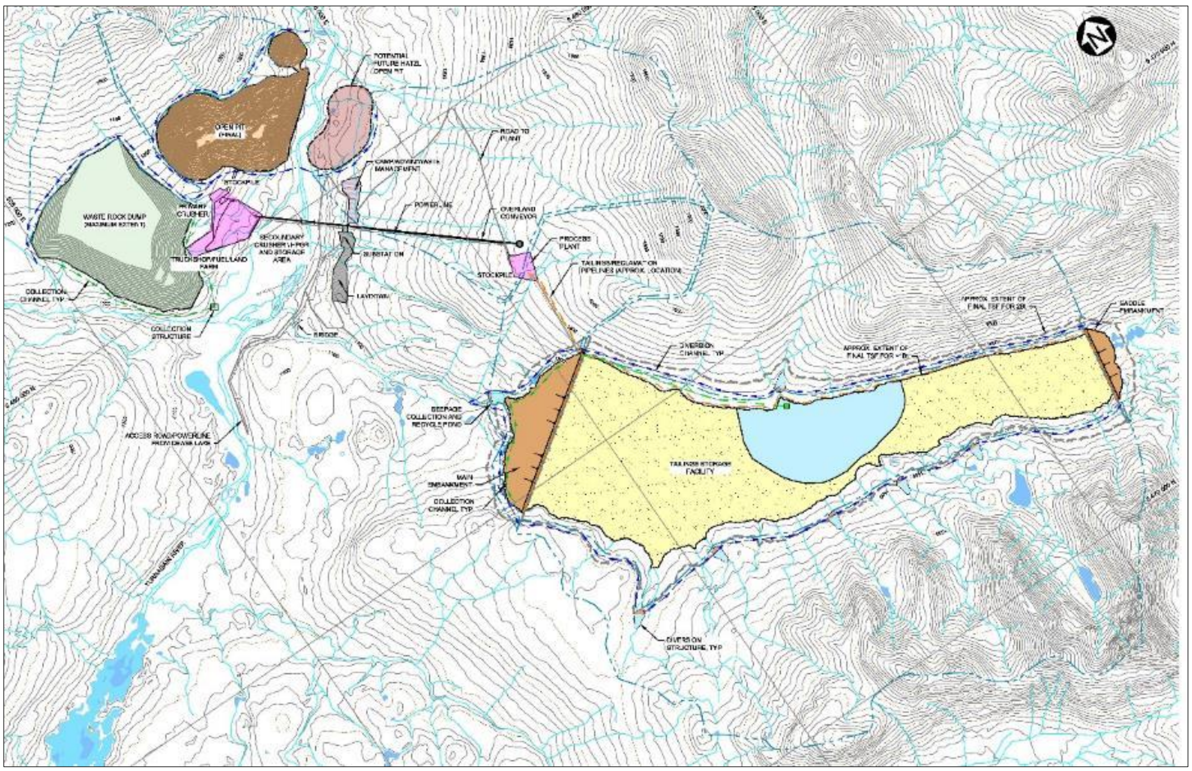
The principal objective of the design and operation of the TMF is to ensure secure containment for tailings solids and impounded process water. The TMF will serve as the primary water management facility for the project, providing a buffering volume for the mill process water demands, as well as collecting and storing the necessary quantities of precipitation and runoff for start-up and operations. The proposed TMF is located in Flat Creek Valley, as shown on Figure 18-3.

The mill throughput will be approximately 45,000 t/d for the first five years of operation and approximately 90,000 t/d starting in Year 6. The total tailings production is assumed to be approximately 1,122 Mt over 37 years of operation. Tailings from the mill will be discharged to the TMF as a slurry at an average un-thickened solids content of approximately 25% (by weight).

The TMF starter embankment is sized to provide water storage for start-up and to store the estimated volume of tailings produced during the first three years of operation.

The final facility is sized to store the estimated 1,122 Mt of tailings produced over the planned mine life. The facility makes use of the favourable topography to yield a relatively large storage volume, compared with the quantity of material required for embankment construction. Likewise, significant increases in storage capacity can be realised with moderate increases in the elevation of the confining embankments. The TMF storage capacity could be substantially expanded within the proposed location. A 2 Bt capacity scenario has been conceptually fit with the basic configuration shown on Figure 18-3.

**Figure 18-3: TMF General Arrangement**



Source: KP, 2020.

The presence of minerals in the tailings that can react with $CO_2$ is expected to provide a range of benefits to the project, as noted in Section 20.6. The potential presence of minor amounts of fibrous mineral forms of magnesium silicate in the tailings may require dust management on exposed beach areas through seasonal irrigation, additives, or other means. Although not part of the current design, the reaction with $CO_2$ tends to create hardened surfaces that reduce dust generation and create a stronger tailings material that is more resistant to low-stress flow. This will be further investigated in future studies.

### 18.4.2 Layout & Operating Strategy

#### 18.4.2.1 Tailings Management Facility Embankments

The main TMF embankment will be raised in 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. The staged design of the embankments will be reviewed and updated annually to accommodate the actual mine production, availability of construction materials, and to incorporate experience gained with local conditions and constraints.

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

A drainage network of 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 larger main collector pipes to transport embankment drainage and intercepted 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.

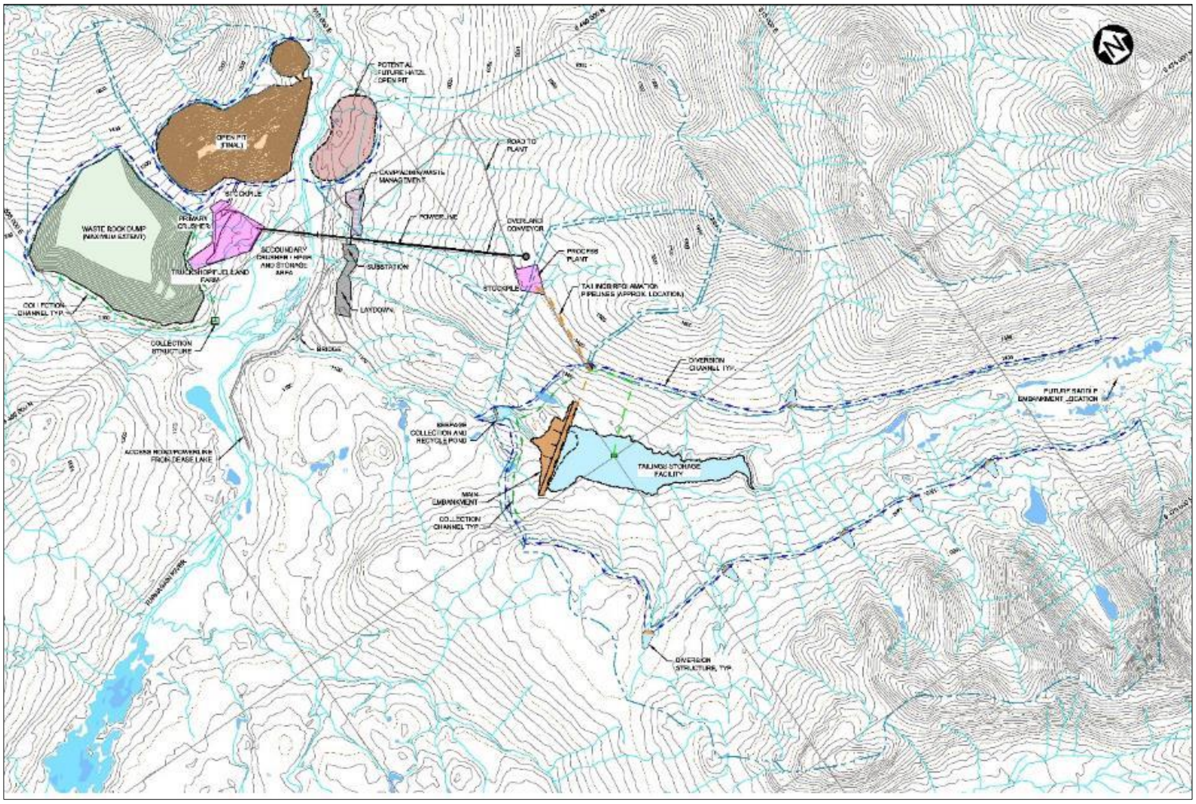
The TMF embankment construction will begin with a starter dam located at the northwest end of Flat Creek Valley, as shown on Figure 18-4. The starter embankment will be built as a water-retaining structure with 2H:1V upstream and 3.5H:1V downstream slopes.

The starter dam will be constructed in two phases (Stage 1a and Stage 1b) that are scheduled to provide storage for the first three years of operation. 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. A low permeability 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 may 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 material and 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 of both the northwest main embankment and the southeast saddle embankment. Each stage will have a minimum horizontal width of approximately 60 m to allow placement with large haul trucks. Rock will be dumped, spread, and compacted to the specified density. Both embankments will have a minimum final crest width of 30 m to allow for vehicle access and pipelines. The embankments will be designed as free-draining structures above the starter dam to enhance stability by promoting well-drained conditions in the dam fill material and the tailings beach upstream of the embankment. Large tailings beaches will be developed and the supernatant pond will be kept small and remote from the embankments during operations.

**Figure 18-4: TMF Starter Embankment**



Source: KP, 2020.

Appropriate filter zones will be raised with each stage of the dams to separate the tailings beach from the coarser rockfill in the downstream embankment shell. It is envisaged that these will be constructed in the summer months by a specialised contractor using locally sourced borrow material. Ongoing construction of the main northwest dam shell zone will make use of suitable low-sulphur, geochemically innocuous rock from a quarry within the TMF catchment area that will be progressively inundated by the stored tailings. The shell zones will need to be constructed prior to beginning work on the crest and filter zones to allow access for equipment. The final general arrangement at the end of the mine life is shown on Figure 18-2.

A smaller saddle embankment will be constructed at the southeast end of the TMF. It will be built using local borrow material in the same manner as the main embankment. The first stage of the saddle dam will need to be completed by about Year 15 of mine operations. The saddle dam will be constructed using the same centreline method as the main dam and will comprise 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 minimise 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 rockfill that makes up the majority of the embankment volume can be placed year-round and in most climatic conditions. The finer-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. It is anticipated that the proposed project schedule will provide sufficient time to place the required volume of material estimated for the current TMF design.

#### 18.4.2.2 Mill Tailings Transport & Deposition System

The tailings transport system will be constructed in stages throughout the life of the project. Tailings will initially flow by gravity from the mill to the TMF main embankment and will be distributed from off-takes (spigots) located along the embankment crest as shown on Figure 18-2 and Figure 18-3. Tailings will also be deposited along the northern shore of the TMF and the saddle embankment beginning in approximately Year 6. Although not included in the current mine layout, tailings discharge from the southern shore of the TMF would be possible if this is found to be a more practical approach during future studies. Two parallel pipelines will be operated concurrently for each 45,000 t/d mill train. Tailings pipelines will be constructed using HDPE wherever practical.

Gravity flow to the main embankment will be possible for the entire mine life. Tailings pumping may be required to discharge tailings to the saddle embankment after approximately Year 25 when the dam is raised past El. 1340 m. A booster pump station that can operate on one of the tailings pipelines (22,500 t/d) would be required in Year 25 and has been included in the conceptual design.

Tailings will initially be deposited from a series of valved off-takes along the embankments to develop long, low-angle sloped beaches away from the embankments. Beach development will be managed through rotational deposition to maintain the pond remote from the embankments. This approach has the added benefit of promoting $CO_2$ sequestration through the presence of freshly deposited tailings in contact with the atmosphere.

The advantages of the proposed layout are:

- Gravity discharge is possible through the entire mine life to the main embankment and most of the north side of the TMF.
- Pipelines are located within the TMF catchment and any leakage would be contained in the impoundment or the main embankment seepage collection pond.
- Pipelines can be drained and flushed by gravity into the TMF.

#### 18.4.2.3 Reclaim Water System

The reclaim water system is illustrated on Figures 18-2 and 18-3. Supernatant reclaim water will be pumped from the TMF to a head tank at the mill site. Reclaim pumps will be mounted on a floating barge in the TMF supernatant pond and will operate on level control from the head tank.

Total pump power for the water reclaim operation will increase in Year 6 when the second mill train comes online, and gradually decrease over the remaining LOM as the TMF is raised. Additional pumps will be added in Year 5 to accommodate the increased throughput at the mill. The water reclaim pipeline will be constructed using a combination of steel and HDPE. Steel will be used for part of the higher pressure section near the barge for the starter layout. The required pumping pressure will be reduced as the supernatant water level in the TMF rises throughout the mine life and extensions to the reclaim pipelines will comprise HDPE.

## 18.5 Water & Waste Management

### 18.5.1 Site Water Management

Water management is an important component of the overall design, and the objectives include:

- provide adequate storage and freeboard in the TMF for secure containment of all process water and storm runoff
- intercept and divert clean water to the extent possible
- adequately collect and control of water within the mine affected area
- mitigate environmental impacts to the extent possible
- optimise the storage and use of water over the entire site to satisfy environmental, operational, and economic criteria

### 18.5.2 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 $Mm^3$ is assumed to provide sufficient buffering volume to satisfy water requirements and account for seasonal variations. Twenty-five million cubic metres corresponds to approximately three months of reclaimed water required for a full mill production rate of approximately 90,000 t/d. Water runoff will be collected from the TMF catchment runoff during pre-production years to satisfy start-up pond requirements. A more detailed water balance will be completed in the next phase of study.

The diversion channels shown on Figure 18-3 and Figure 18-4 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 minimise 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 to the extent practical.

The minimum freeboard requirement for the TMF is assumed to be 5 m. Further studies, including determining the inflow design flood and potential seismic deformations from the maximum design earthquake, will be needed to define the required freeboard throughout the life of the facility.

### 18.5.3 Waste Dump, Low-Grade Stockpile, Open Pit & 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 mineralisation 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 a collection pond. The volume of in-pit dewatering will generally be dependent on direct precipitation and groundwater, and will be 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 ore stockpiles will be collected in channels along the downstream toes and diverted into a collection pond located at topographic low points below the facilities.

A gravity-fed pipeline will be available to convey water from the waste dump and low-grade mineralisation stockpile collection ponds to the main pit collection pond. Water collected in the site water management ponds will be pumped to the plant site for reuse, or for treatment and discharge.

Surface water diversion channels will be constructed upstream of the waste dump, low-grade mineralisation stockpiles, and open pit to minimise 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 will be sized to carry the estimated 1-in-10-year peak instantaneous runoff.

### 18.5.4 Site 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 available information regarding annual hydrologic conditions were used as the basis for the modelling. Precipitation parameters are considered to represent reasonable site conditions; however, confirmation of these parameters and more detailed water balance calculations will be required in the next stage of the study.

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. This is beneficial to the project, as it is unlikely that surface water discharge from the site will be required during operations. It is similarly unlikely that additional make-up water will be required from outside the project area. It is assumed that the 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 during operations. It is estimated that approximately 20% to 30% of runoff collected by the diversion channels would be sufficient to satisfy make-up water requirements on mean annual basis. This system, combined with good water balance planning during operations, would provide substantial flexibility to maintain a water-neutral condition at the site.

### 18.5.5 Water Supply

It is assumed that fresh water will be collected from alluvial groundwater wells just north of the plant site near Turnagain River. This would supply the fresh water requirements at the mill as well as other mine facilities. Water will be pumped to a storage tank at the mill. The fresh water demand is currently estimated at approximately 13 $m^3/h$.

### 18.5.6 Sewage Disposal

A sewage treatment plant is included in the mine infrastructure. Non-process wastewater 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 containerised 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 to the environment in accordance with the requirements of the Environmental Impact Assessment and effluent permits or approvals.

### 18.5.7 Refuse Disposal

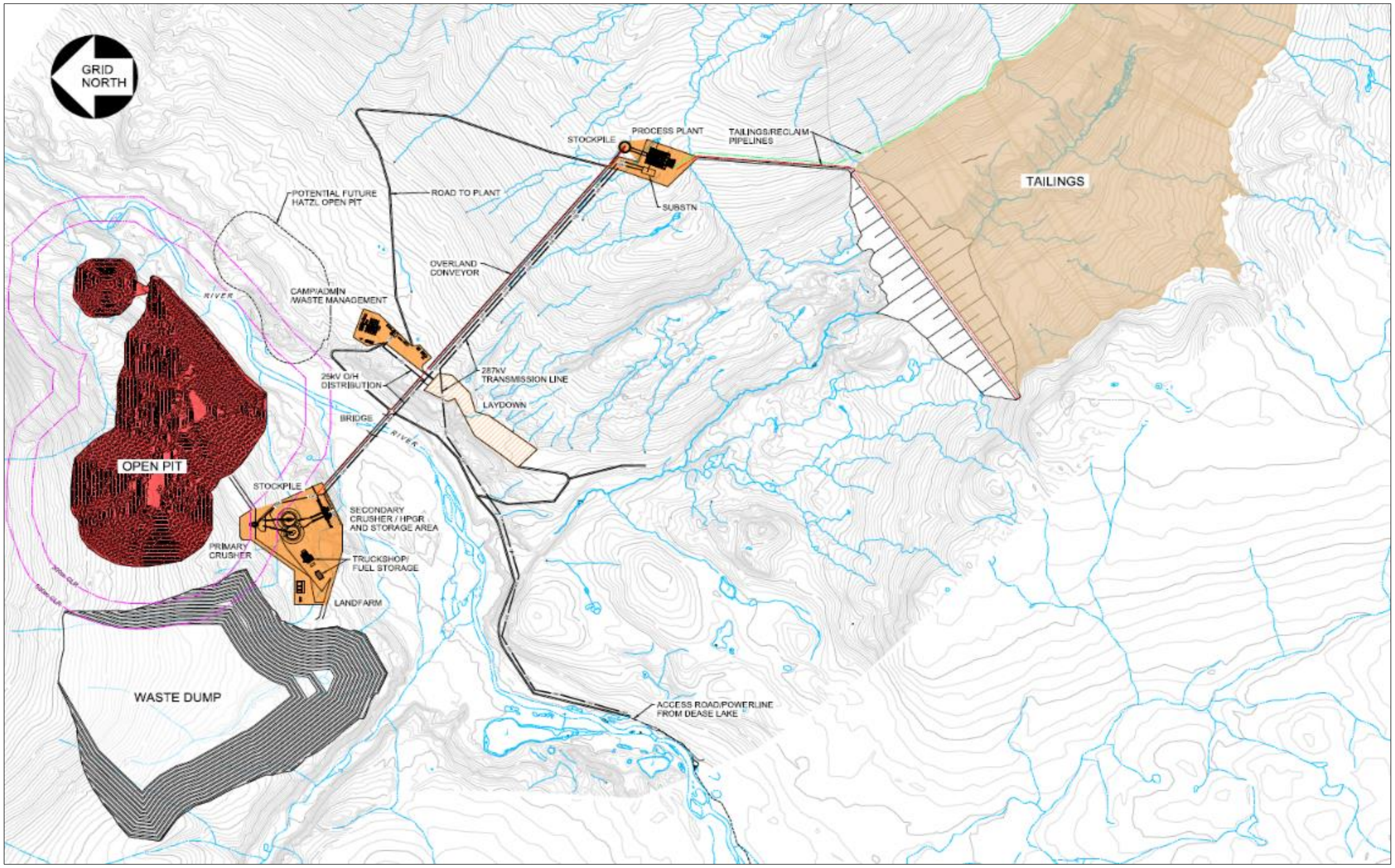
Several forms of domestic and industrial solid waste will be generated over the life of the mine. All avenues of reuse, reduction, and recycling of materials will be examined and implemented prior to any waste disposal.

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

## 18.6 Plant Site Layout

The plant site layout in Figure 18-5 shows the open pit area, camp infrastructure area, and main concentrator plant. Site development is divided into these three areas to place facilities in the most efficient location for site operation; mine infrastructure area (including truck shop and fuel storage), camp infrastructure area (including camp, administration buildings, waste management) and process plant infrastructure (main substation, process water and tailings).

**Figure 18-5: Plant Site Layout**




Source: Hatch, 2020.

## 18.7 Ancillary Facilities

Project infrastructure and services have been designed to support a 90,000 t/d operation. Infrastructure and ancillary facilities will consist of the following modular, pre-engineered, 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 facilities, including fuel station
- security / gatehouse
- power supply and distribution
- communications system
- sewage system
- water supply

The facilities will be located to minimise the overall footprint and excavation effort and maximise operational efficiency. Wherever possible, the layout takes advantage of the flattest slope in the area.

### 18.7.1 Administration Building

The administration building will be a modularised structure that provides working space for management, supervisors, geology, engineering, and other operations support staff.

### 18.7.2 Camp

A modularised accommodations facility will be located in the infrastructure area between the open pit facilities and the concentrator. This camp will be installed for construction personnel, and re-furbished as necessary at the start of operations.

### 18.7.3 Mine Dry

A mine dry facility, including lockers and shower facilities, will be provided. The mine dry will be a modularised structure located near the camp.

### 18.7.4 Truck Shop

The truck shop building will be a pre-engineered building with an overhead clearance of at least 10 m. 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 erected at the crushing plant area near the mining haul road and the warehouse.

The truck shop will house light- and heavy-vehicle maintenance bays, 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, and an oil separator. Waste oil will be used as fuel in the refuse incinerator with any remaining oil removed and discarded at an approved facility. The tire change area will be nearby, but set apart from the other areas for safety.

### 18.7.5 Light Vehicle Maintenance/ Warehouse

A light vehicle and plant equipment maintenance/warehouse facility will be provided near the open pit and crushers. The maintenance area will be equipped with a crane. The warehouse area will be sized to accommodate all the mine fleet equipment spares and maintenance shop supplies. Process materials will be stored separately near the concentrator building.

### 18.7.6 Open Area Storage

Open area storage areas will be provided for construction laydown and operations, as well as maintenance storage for equipment and materials at the mine site area, camp and infrastructure area, and at the concentrator area.

### 18.7.7 Assay Laboratory

An assay/environmental laboratory will be located in a separate modular building. The laboratory will be a single-storey structure equipped to perform daily analyses of mine and process samples.

### 18.7.8 Fuel Storage & Distribution

An area will be designated near the truck shop for fuel storage and dispensing. The fuel storage and dispensing facility will have a lined containment area, so that spills are confined and can readily be cleaned up. This will prevent the need for costly remediation during site closure.

Diesel fuel will be required for mobile mine equipment, some small trucks, and surface vehicles. The pumping station allows both light-duty vehicles and heavy-duty mining equipment to be refuelled. Approximately two weeks of fuel storage will be provided to accommodate interruptions in supply due to weather or road issues.

Propane storage vessels will be provided for field maintenance and space heating fuel.

#### 18.7.9 Security/Gatehouse

A security/gatehouse will be located on the site access road at the plant site.

### 18.8 Power Supply

Giga Metals Corporation commissioned Kerr Wood Leidal Associates Ltd. (KWL) to review power supply options and undertake a preliminary assessment of power supply options to support the development and operation of the Turnagain Mine.

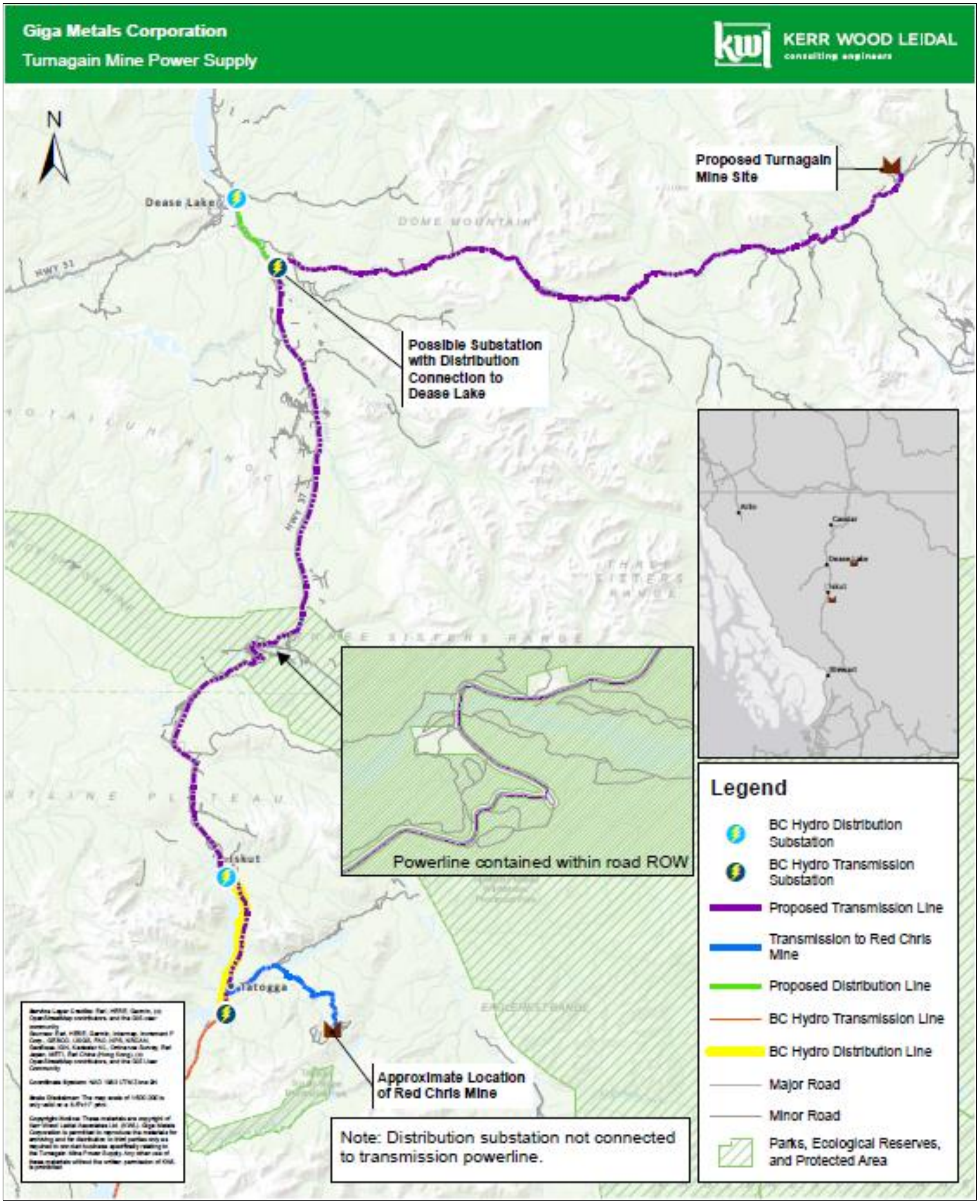
Since the 2011 PEA, BC Hydro completed the Northwest Transmission Line (NTL) from Terrace to Bob Quinn. In addition, the NTL Extension was built to Tatogga Lake to provide power for the Red Chris Mine, and a 25 kV line was built to serve the community of Iskut. The proposed Turnagain Mine is approximately 160 km (powerline route) from an existing operating BC Hydro 287 kV substation at Tatogga Lake (see Figure 18-6).

It is proposed that a 287 kV transmission line be built between the existing BC Hydro substation at Tatogga Lake and a new substation to be built south of Dease Lake near the mine access road. Once built, this line would be sold at a negligible price to BC Hydro to own and operate (sale of the transmission line is not a source of revenue). From the new substation near Dease Lake, a 25 kV distribution line could be built to serve the community of Dease Lake, and the 287 kV line extended to the mine site. The point of interconnection to BC Hydro's system would be the proposed new substation near Dease Lake. An operating voltage of 287 kV for the line has been assumed, as a lower voltage, such as 138 kV, would not be adequate for the Turnagain Mine electrical load in Phase 2.

In addition to the option of connecting to BC Hydro's grid through an extension of the NTL, the option of powering the mine with a natural gas-fired power plant was also assessed. In this option, liquefied natural gas (LNG) would be trucked to the mine site, gasified, and used to produce power through combustion turbines. A range of renewable power options was examined for use as potential fuel savers or alternatives to a BC Hydro connection or LNG combustion turbines. These options include variable wind, geothermal, hydroelectric, and connection to hydroelectric projects in Alaska. The initial screening of these options did not indicate they would be economically competitive with interconnection to the NTL or on-site natural gas-fired generation.

On-site LNG-fired power options have a higher cost and higher greenhouse gas emissions ($CO_2e$) than the BC Hydro tie-in. Although viable, this approach is not recommended unless regulatory hurdles for a BC Hydro tie-in are insurmountable (this is not expected). Renewable options, such as wind, could offer lower operating cost power than LNG with increased capital cost, and should be investigated as an operating cost reduction if the LNG power option is pursued. Alternative LNG supply options in northeast BC and coastal liquefaction facilities in the Prince Rupert-Kitimat area should also be reviewed if the LNG power option is pursued.

**Figure 18-6: Proposed Transmission Line**




Source: KWL, 2020.

Based on the NTL and other powerlines in BC, the construction of a 287 kV transmission line from Tatogga Lake to the Turnagain Mine is technically feasible. Optimisation of the transmission line route is required as additional project information becomes available. See Section 21 for capital and operating costs, and Section 26 for recommendations for further study.

## 18.9 Site Power Distribution

From the main substation (25 kV line up) located at the concentrator (which has the highest power draw), 25 kV overhead lines will deliver power to the following:

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

From the 25 kV line up, 25 kV overhead lines will also 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 the electrical rooms will be located the relevant 4 kV and 600V electrical equipment, and the 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, three-phase, four-wire, high-resistance grounded. Note that the ball mills will likely require 13.8 kV.
- Low voltage: motors larger than 0.5 hp 575 V, three-phase, three-wire 5A resistance grounded
- Area lighting: interior and exterior 120 V or 347 V, single phase, solidly grounded
- Room lighting: 120 V, single phase, solidly grounded
- Control voltage: A 120 V, single phase, solidly grounded
- Control voltage: B 24 V, direct current (DC) (if required to suit OEM equipment)
- Instrumentation loop voltage: 24 V direct current (VDC), 4 to 20 mA.

## 18.10 Communication System

The site communications systems will be supplied as a design-build package; the scope will be defined in future project phases.

# 19.0 MARKET STUDIES & CONTRACTS

## 19.1 Nickel Markets

### 19.1.1 Overview

After many years of annual surpluses, the global nickel market moved into deficit in 2016. Although the shortfall in that year was moderate, it was followed by larger deficits in 2017 and 2018. However, the ongoing ramp-up of nickel pig iron (NPI) production in Indonesia and China led to a small global surplus in 2019. With further capacity expansion in Indonesia and the impact of the Covid-19 pandemic on global nickel demand, the nickel market will be in significant surplus in 2020—a situation that is expected to prevail for much of the decade.

It is projected that supply will start to fall behind growth in demand in 2029 with the need for new nickel supply from unidentified resources beginning in 2032. The market requirement for new supply will increase rapidly from 133 kt (including probable projects) by 2030 to 1.30 Mt by 2040. To encourage investment to bring these projects to market, it is forecast that nickel prices will need to average at least US$16,500/t (US$7.50/lb) in the long term.

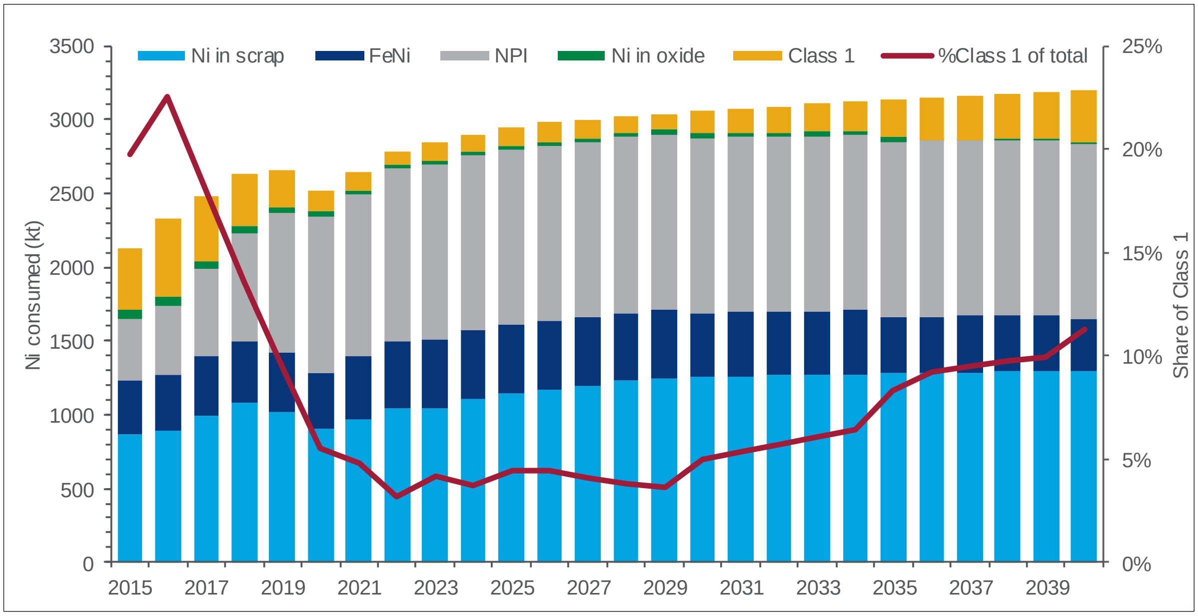
### 19.1.2 Uses & Marketing

Nickel is predominantly used in the production of stainless steel (~70% of nickel demand), with the balance being consumed for high nickel alloys, chemicals, plating, and batteries. The expected demand for nickel in stainless and non-stainless end uses is shown in Figure 19-1. Note that the scrap nickel inputs to stainless steel is not new nickel introduced to the market, but recycled nickel units, and is not counted in any other supply-demand calculations. At present and over the next few years, the portion of world nickel demand that will go to EV batteries remains comparatively small; it is only after 2025, and especially from 2030, that the potential quantities involved become more critical for the nickel market in general (Figure 19-2).

### 19.1.3 Supply/Demand

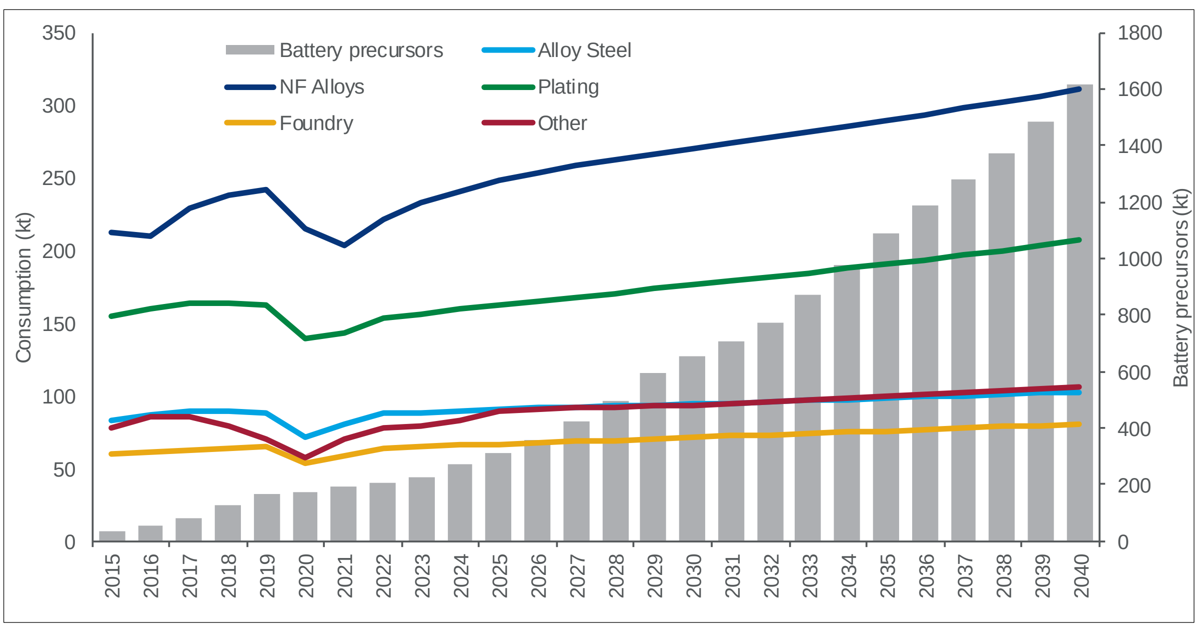
World nickel demand increased by 5.2% in 2019 to 2.43 Mt. However, the global impact of Covid-19 will lead to a forecast 4.7% contraction to 2.31 Mt in 2020. While China is forecast to show 0.9% growth, the world ex-China is hardest hit with a contraction of 12% (Figure 19-3). Despite this, growth at a CAGR of 2.7% will raise world demand to 2.78 Mt in 2025. Over the long term, a CAGR of 2.9% raises nickel demand to 3.15 Mt in 2030 and 4.33 Mt in 2040. Over the next 20 years, growth in nickel demand becomes increasingly dominated by an accelerating uptake in the battery segment due to the anticipated expansion in global electrification of transportation and energy storage systems (ESS). At the same time, primary nickel uptake in stainless will decelerate due to greater use of scrap in China. As a result, the stainless share of global nickel demand is projected to decline from 65% to 70% at present to less than 50% of demand in 2040, whereas that of batteries for EVs and ESSs will increase from 7% to 37% over the same period.

**Figure 19-1: Consumption of Nickel in Stainless Steel by Nickel Product Type**



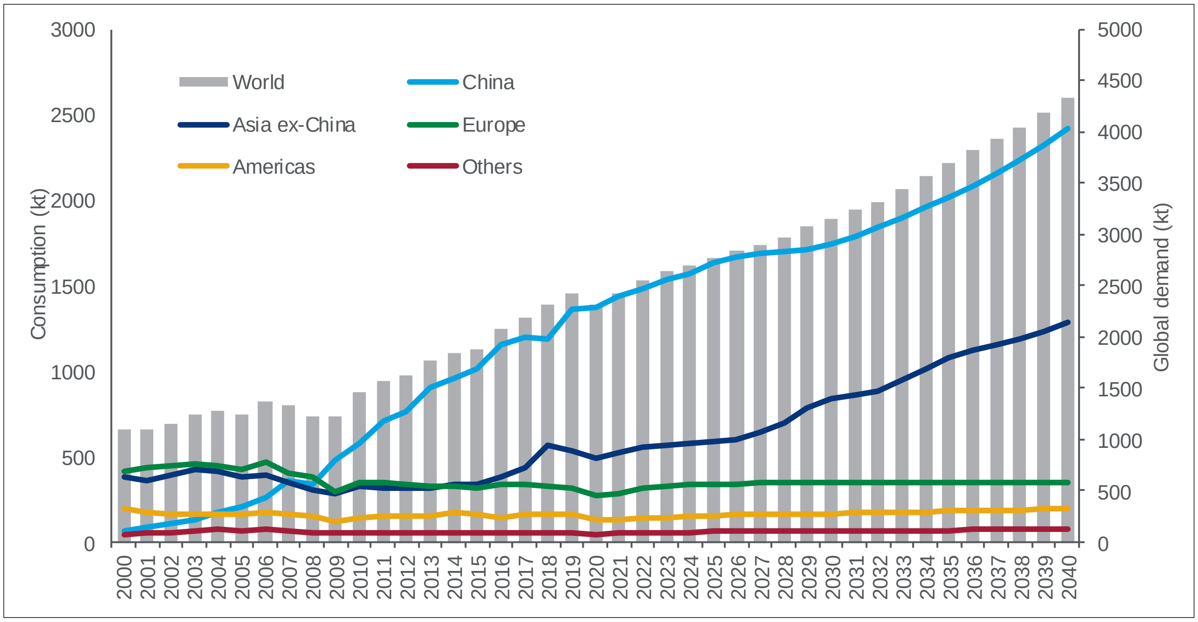

Source: Wood Mackenzie, 2020.

**Figure 19-2: Nickel Demand for Non-Stainless Steel Uses**




Source: Wood Mackenzie, 2020.

**Figure 19-3: Global Nickel Consumption by Region**




Source: Wood Mackenzie, 2020.

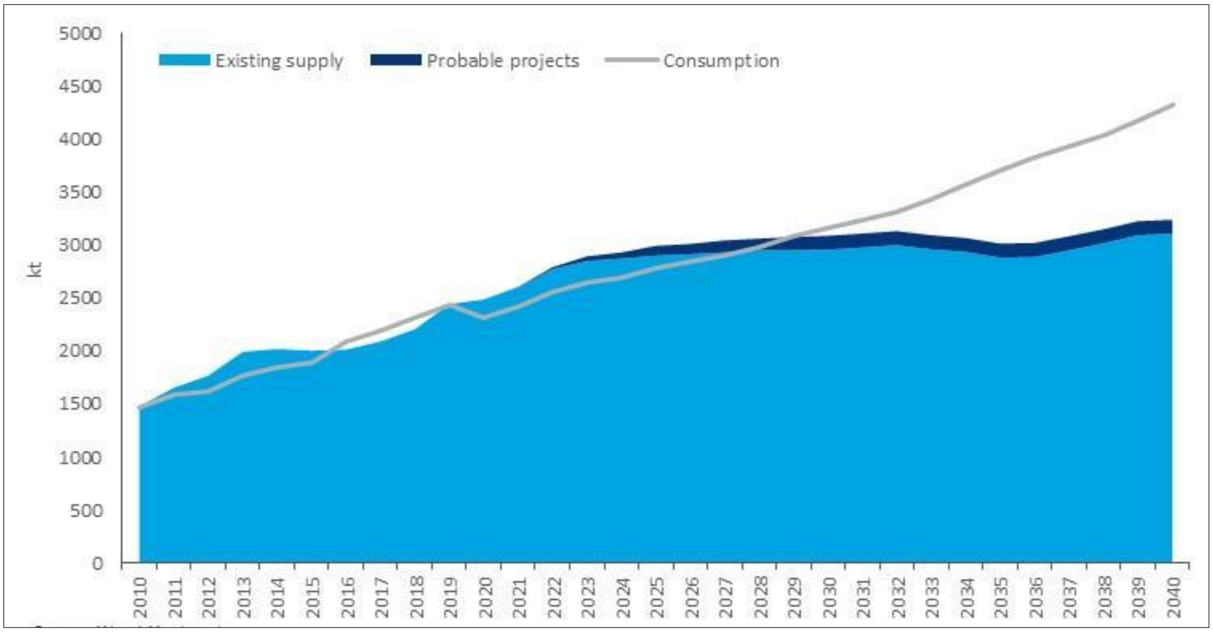
The forecast average annual increase in nickel demand in EV/ESS battery use (as Class 1 or chemicals) from 2025 to 2040 is 84 kt/a; nearly two Turnagain projects (at Phase 2 rates) every year.

Over the midterm, the combined impact of sustained expansion in Indonesian NPI and declining growth in primary nickel use in Chinese stainless as scrap use accelerates is that the world nickel market is in surplus from 2020 through to 2028. However, from around 2027 we project a surge in demand for nickel in EV batteries, helping the market to return to deficit by 2029. This is exacerbated by anticipated closures, due to exhaustion of currently defined economic reserves, at Cerro Matoso (in 2029), Nickel West (2032), Murrin, Koniambo and Long Harbour (all 2034).

By 2030, the market needs around 133 kt of new supply which can be provided by projects in the Wood Mackenzie probable projects list. However, by 2035, a further 510 kt of new nickel supply will be required from (as-yet) unidentified resources. By 2040, the figures increase to 1.3 Mt of nickel, of which 1.17 Mt would need to come from unidentified resources. The forecast average annual increase in total nickel supply required from 2025 to 2040 is over 80 kt/a, or almost two Turnagain projects (at Phase 2 rates) every year.

Figure 19-4 shows that the long-term shortfall in nickel supply is 1.3 Mt/a. To supply such a large quantity of nickel by then will be a considerable challenge for investors and producers alike, especially given that the typical time required for development, construction and ramp-up of a new facility can take eight to ten years, and 2030 is only 10 years away. It is also noteworthy that this substantial shortfall is arrived at with moderate demand growth over the long term and a reasonably conservative EV demand forecast. Clearly, more bullish estimates will only make the situation more untenable.

**Figure 19-4: Long-term Nickel Supply & Demand**




Source: Wood Mackenzie, 2020.

One main risk to this outlook is the considerable NPI expansion potential in Indonesia, where there is about 350 kt/a nickel capacity identified in proposed possible and probable projects. With the establishment of large industrial parks, such as Morowali and Weda Bay, brownfield capacity can now be added in less than 12 months and ramp-up completed in as little as three. Thus, new capacity can be added quickly with only modest capital outlay (e.g., Nickel Mines Australia's RKEF lines in Morowali were added for around US$10,000/t nickel a year).

One other risk is that our assumptions on EV battery recycling rates prove optimistic. We currently assume that ~25 kt Ni will be recycled in 2025, growing rapidly to 84 kt by 2030, 240 kt by 2035 and 540 kt by 2040. This is a significant tonnage of nickel to be returned to the supply pool. Without this recycle stream, the requirement for nickel by 2035 and beyond would be significantly higher than shown and would result in a higher long-term price requirement.

Aside from NPI, the main recent development signalling where future nickel supplies may come from, specifically those needed by the new EV battery segment, is the ongoing construction of three HPAL facilities in Indonesia (as announced in late 2018): QMB and Huaqing, both at Tsingshan's Indonesia Morowali Industrial Park (IMIP); and Lygend on Obi Island. Planned capacity at each site is 50 to 60 kt/a nickel in either nickel sulphate or an intermediate for conversion to nickel sulphate. Officially reported schedules assume these facilities will enter production in 2021, although this may prove to be optimistic. None of the projects has obtained permits for tailings impoundment or disposal, and recent news suggests that QMB is currently essentially on hold. The process flowsheet for the plants is apparently based around that of Ramu (PNG). It should also be noted that ore feed composition in Indonesia will be different to that at Ramu and will vary depending on the contracted ore source. This in turn could result in prolonged ramp-up periods due to ore variability and process instability. Two other HPAL projects have also been announced—a Sumitomo Metal Mining/Vale joint venture at Pomalaa, and PT Ceria at Kolaka—but neither are yet committed and will not be in production before 2025 at the earliest.

If these first three projects are successful, there is likely to be widespread, Chinese-led enthusiasm to invest in similar ventures, and these could provide the nickel the market will need over the next 10 years, perhaps even exceed requirements. However, although proliferation is a possibility, the challenge of waste disposal will be a major constraint. HPAL produces more waste than the ore feed it consumes (50 kt/a of Ni will produce approximately 5 to 6 Mt/a of waste slurry), and waste can only be discarded behind land-based tailings dams, requiring a huge amount of space (for fine chemically precipitated tailings materials in tectonically active areas), or by deep sea disposal. Both are likely to provoke considerable environmental concern warranting challenging legislation.

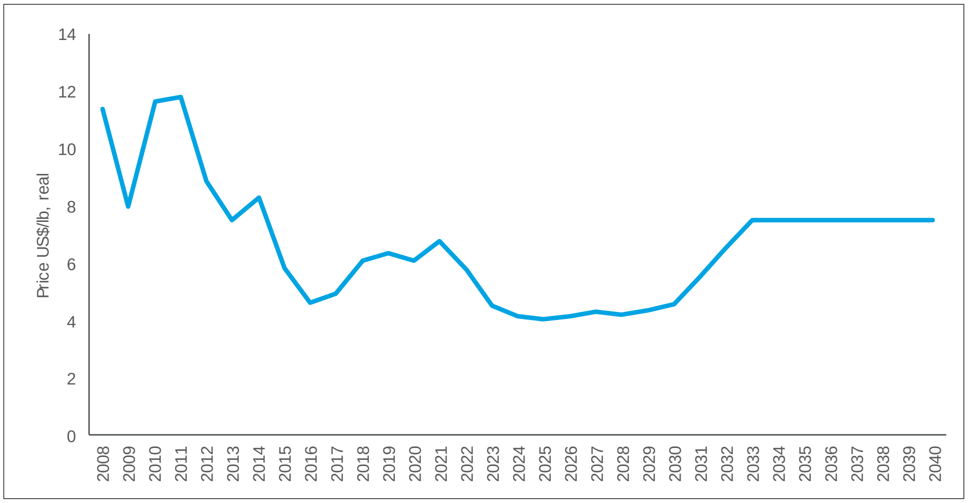
If these HPAL projects are delayed further by unforeseen technical issues, one alternative for those companies seeking to supply nickel to the battery segment could be to convert FeNi or NPI into a matte that can then be refined to nickel sulphate. This is an option under consideration by Huayou at Weda Bay. Certainly, there would be no shortage of a raw material for this process route, as there appears to be little stopping the continued expansion of NPI capacity, even if it is all currently earmarked for use in stainless steel. However, this process option does require additional smelting and refining facilities and has additional issues with slag and precipitated iron residues resulting from the high iron-to-nickel ratio in NPI.

### 19.1.4 Pricing

As shown in Figure 19-5, supply starts to fall behind growth in demand from 2029 with the need for a new nickel supply from unidentified resources in 2032. As noted above, the market requirement for new supply increases rapidly to 133 kt (including probable projects) by 2030 and 1.71 Mt by 2040. To encourage the investment to bring these projects to market, it is forecast that nickel prices will need to average at least US$16,500/t (US$7.50/lb) in the long term.

If one considers the growing requirement to source commodities that meet specific environmental, social and governance (ESG) criteria, then we are of the opinion that the long-term nickel price would need to be higher than our current base case forecast of US$16,500/t to incentivise the development of such projects. Thus, narrowing down the projects evaluated based on jurisdiction, land-based tailings disposal and access to predominantly non-diesel generated power, we estimate that a long-term nickel price of at least US$18,700/t (US$8.50/lb) would be necessary to encourage sufficient capacity to the market to both meet our market requirement and satisfy ESG requirements.

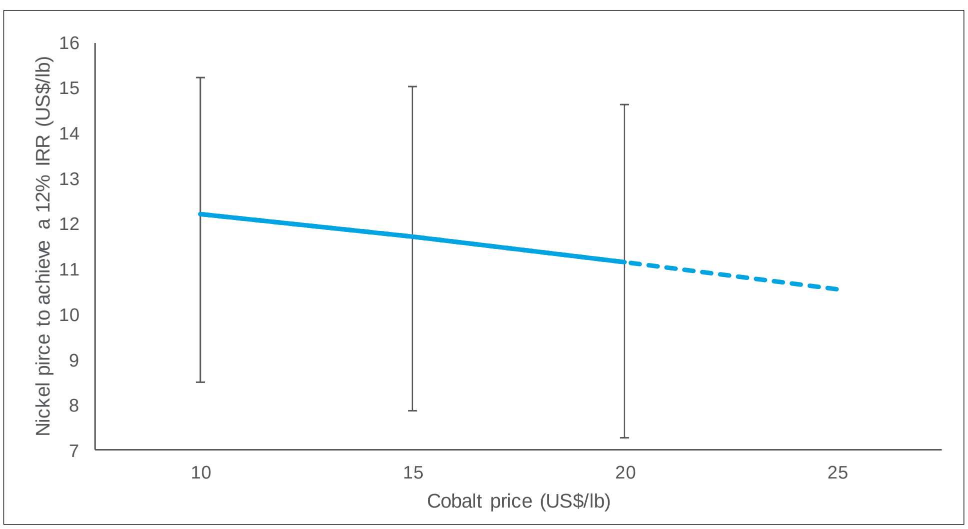
**Figure 19-5: Nickel Price Forecast**




Source: Wood Mackenzie, 2020.

Looking at current demand in the battery sector, approximately 30% of nickel units consumed are derived from laterite ore sources, and the bulk of this from intermediates produced from high-pressure acid leach (HPAL) operations (e.g., Ramu, Ravensthorpe, Gordes and VNC). Given that the bulk of nickel resources are in the form of laterite ores, it appears likely that much of the future supply requirement will be sourced from laterites and therefore through the adoption of HPAL technology. Wood Mackenzie has assessed a number of laterite projects (four in Australia and one in the Philippines). Figure 19-6 shows the average and range of nickel prices required by these projects to generate a 12% IRR with an extrapolation to the base long-term cobalt price of US$22.30/lb.

**Figure 19-6: HPAL Incentive Price**




Source: Wood Mackenzie, 2020.

While the lower end of the nickel price range fits broadly with the ESG nickel price noted above, on average they are considerably higher. Wood Mackenzie would expect the long-term nickel price would need to be approximately US$23,950/t (US$10.86/lb) for the development of new nickel supply solely from HPAL projects.

## 19.2 Cobalt Markets

### 19.2.1 Uses & Marketing

Cobalt is used in a variety of applications broadly split into two groups: metallurgical (superalloys, high-strength steels, magnets, cemented carbides) and chemical (batteries, catalysts, paints, ceramics). The rechargeable lithium-ion battery sector accounts for the largest share of consumption at present and has by far the greatest growth potential.

We estimate refined cobalt demand reached 126 kt in 2019, up by 6% from 2018 levels. Of this total, we expect demand from batteries accounted for a 55% share. Consumer electronics remains the largest sub-sector for cobalt use within batteries; however, we expect electric vehicles (EVs) to surpass this in the next two years.

Cobalt mineralisation is focused around the African copper belt spanning the Democratic Republic of Congo (DRC) and Zambia; sulphide ore deposits in Canada, Scandinavia, Russia and Australia; and laterite ore deposits in Cuba and Asia Pacific. Though it is widespread, cobalt typically occurs at such low concentrations it is uneconomical to produce on its own. As such, it is mined mainly as a byproduct of other metals, primarily copper and nickel.

Traditionally, all cobalt was sold at so-called ‘producer prices’ set by the major refiners of the metal located in Zambia and the DRC (then Zaire). However, a deteriorating political situation in the DRC in the early 1990s and corresponding disruptions to production saw these reference prices start to lose relevance in the international market.

Eventually the producer pricing system broke down when Zambian producers started selling based on the Metal Bulletin prices for 99.8% (high grade) and 99.3% (low grade) metal. These prices, generated using journalistic price discovery methods, have become the benchmark reference prices in the industry since then. In 2010 the LME launched 2010 cash and futures contracts for cobalt.

### 19.2.2 Supply/Demand

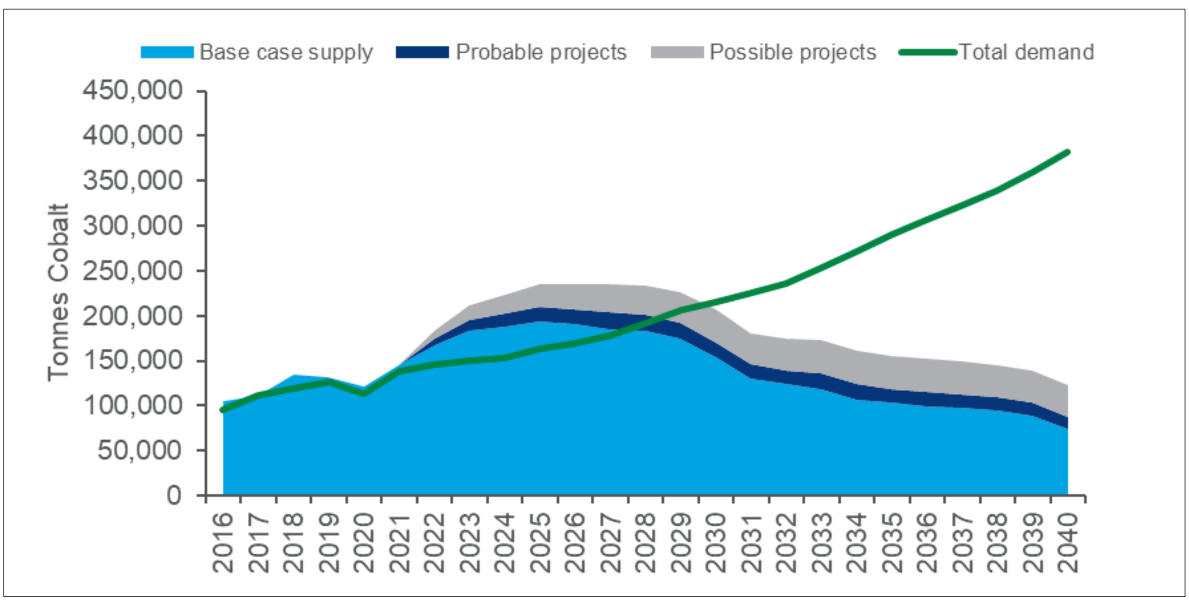
Supply growth will outpace demand over the medium term, as a number of key copper-cobalt operations ramp up in the DRC. The DRC’s share of global mined cobalt supply reached 71% last year, compared to 63% in 2015. This oversupply narrative over the medium term has been aggravated by a faster adoption of high-nickel battery chemistries. While the large surpluses we project will not materialise, the last year has demonstrated that large stocks of intermediates can build up through the supply chain and keep pressure on prices. Additionally, we expect this prolonged period of excess supply to see smaller suppliers increasingly squeezed out of the market, as lower prices render operations uneconomical and environmental, social and governance (ESG) risks affect refinery procurement strategies.

Long-term cobalt demand will be predominantly driven by growth in the lithium-ion battery sector. Portable electronics initiated the first 'battery boom' over a decade ago and we expect strong growth through the forecast. However, battery demand from the portable electronics and ESS sectors will be completely overshadowed by the scale of EV growth over the long term.

Under the Wood Mackenzie base case electric vehicle forecast, we expect passenger car EV (full-electric and plug-in hybrids) sales to reach 7% of total passenger car sales by 2025, and 14% by 2030. Importantly, nearly all OEMs have committed to utilising ternary battery chemistries (nickel-manganese-cobalt, NMC or nickel-cobalt-aluminum, NCA)—which contain cobalt—for their higher energy density and other properties. Different ternary cathode chemistries have varying lithium, nickel and cobalt consumption on a per kilowatt-hour basis. The trend towards NMC 811 will increase nickel and slow cobalt demand. Yet the overall shift towards ternary cathodes provides upside for all three metals.

Over the longer term, we see a notional deficit emerging in the cobalt market as EV-led demand starts to accelerate rapidly (Figure 19-7). We currently expect a deficit to emerge in the cobalt market by 2029, although this could be pushed out slightly by high stocks accumulated in preceding years. However, we would highlight this is predicated on a relatively conservative outlook for EV penetration, uninterrupted DRC mine supply and continued thrifting of cobalt from cell chemistries.

**Figure 19-7: Global Cobalt Supply/Demand Balance**




Source: Wood Mackenzie, 2020.

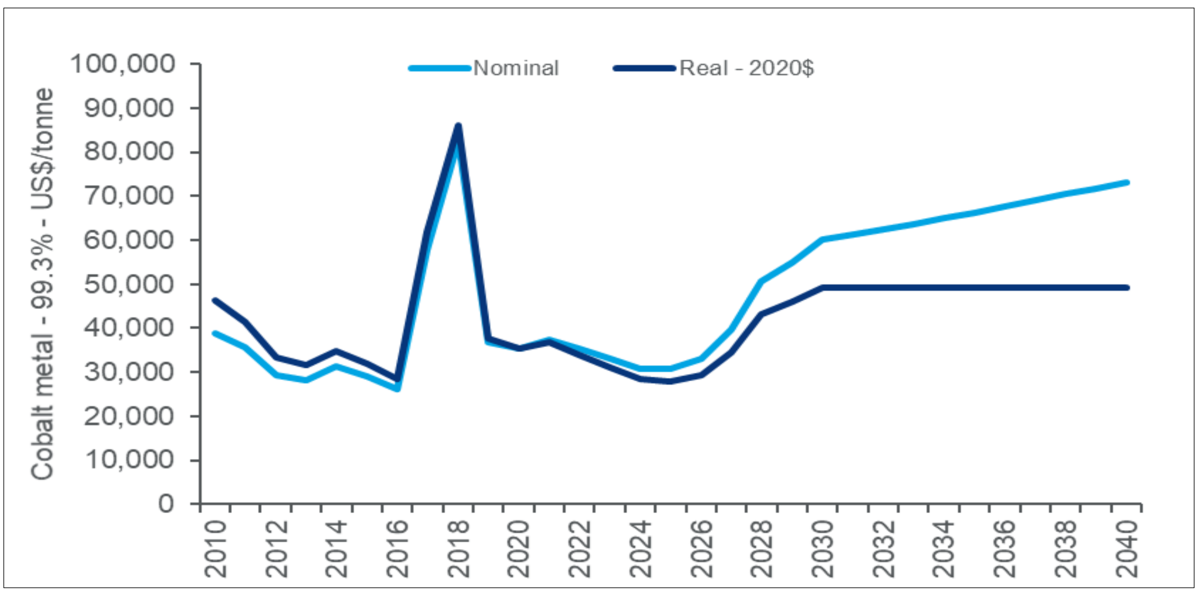
By 2030 cobalt demand reaches 216 kt. Factoring in the ‘probable’ and ‘possible’ projects we are just about able to meet demand, and the secondary supply of cobalt from recycling will become increasingly important to the industry.

With demand growing at ~13 kt/a between 2025 and 2035, the industry will essentially need a new greenfield mine to start up each year. Putting financing to one side, this assumes suitable cobalt deposits can be found to allow for this. Meanwhile, developments such as the DRC’s mining code are already discouraging long-term investment in cobalt.

### 19.2.3 Pricing

In terms of pricing, marginal cost and incentive pricing models are not appropriate for cobalt given it is mostly produced as byproduct of nickel and copper operations. As such, the economics of projects are much more dependent on nickel and copper prices than cobalt. Instead we look at the long-term average in real terms. Since 1950 this has averaged $49,163/t ($22.30/lb) and functions as Wood Mackenzie’s long-term price from 2028 (see Figure 19-8).

**Figure 19-8: Cobalt Price**




Source: Wood Mackenzie, 2020.

## 19.3 Concentrate Markets

### 19.3.1 Nickel Concentrate Supply/Demand

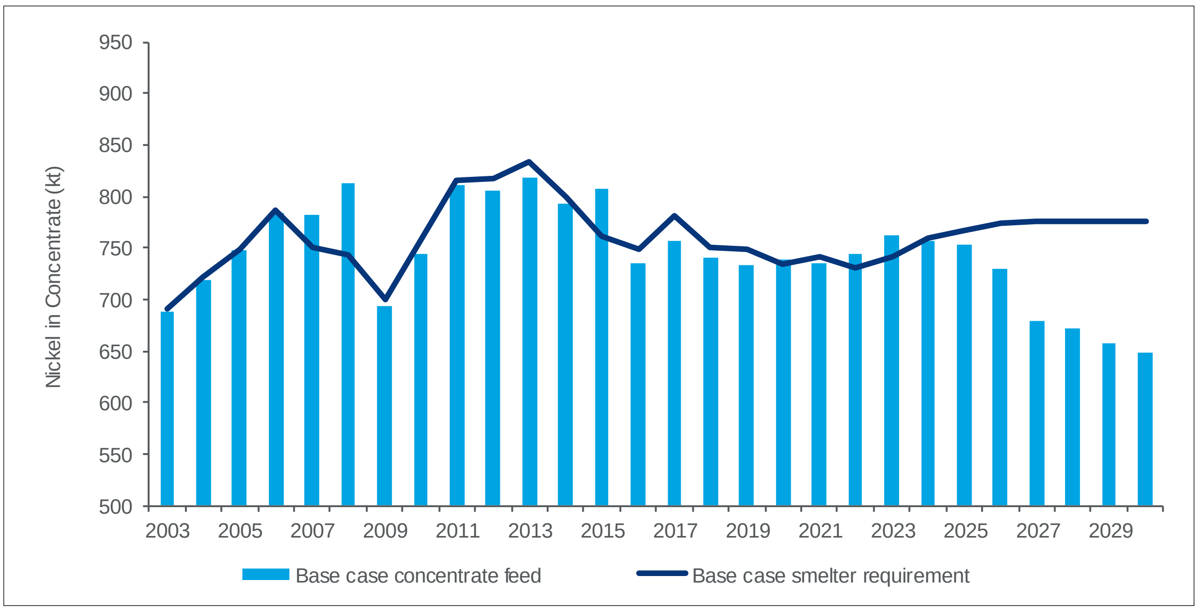
There are essentially 14 sulphide smelters in operation capable of processing concentrate from Turnagain. Of these, seven are actively procuring third-party concentrates and one has the potential to require material.

The concentrate analysis forecasts a balanced concentrate market through to 2024, at which point the market starts to turn to significant deficits (Figure 19-8). There are projects currently under evaluation that are considered probable projects, which could provide sufficient concentrate feed to maintain a balanced market from that time. Nevertheless, new sulphide mine development is needed to meet the forecast smelter output, be that from the current probable projects list or others.

### 19.3.2 Indicative Terms for Turnagain Concentrate

Historically, contracts were developed to match the specific smelter's abilities with respect to metal recoveries and detrimental elements. For example, BHPB Group's Kwinana refinery has no cobalt, copper or PGM (platinum group metal) recovery circuit, other than as a copper sulphide and mixed Ni:Co sulphides. Therefore, the plant has offered lower cobalt, copper and PGM payment terms than, for example, those offered by Falconbridge (now Glencore) or Inco (now Vale), both of which have cobalt, copper and PGM refining capability.

**Figure 19-9: Global Nickel Concentrate Treated & Net Required**




Source: Wood Mackenzie, 2020.

What is apparent from the available information is that tenders for concentrate have become more favourable to the miner in recent years. In the late 1990s, the combined treatment and refining charges (TC/RC) were equivalent to between 45% and 55% of the nickel price before the addition of transportation. More recently, this figure is in the range of 22% to 35%, and is a consequence, presumably, of smelters needing additional feed rather than taking concentrate because it is on offer.

Nickel concentrate contracts have taken several forms, such as a fixed treatment charge with a variable refining charge related to the LME nickel price, a variable percentage of nickel price payable, and a payable nickel percentage with a variable refining charge. This latter form may or may not also include a fixed treatment charge. The straight percentage of nickel price charge related to the LME price was favoured by the Chinese in their off-take deals and appears to have become the “benchmark” for current tenders.

Unlike, for example, the copper market, there are no global benchmark treatment and refining charges set for nickel concentrates, and each contract is negotiated on an individual, private, and confidential basis. As a result, it is generally difficult to obtain details relating to concentrate off-take agreements. The following indicative terms should therefore be viewed on a “best endeavours” basis and have been determined through various conversations with both buyers and sellers of concentrates.

The current terms for nickel concentrates indicate 78% payability for nickel, 30% to 40% pay on copper and 35% pay for cobalt (if above 0.3% in the concentrate, zero pay if below 0.3% Co), all CIF China port. There was no indication that higher grade nickel concentrate would command any preferential (higher) payabilities. However, it was acknowledged that there is currently a paucity of such material and therefore it has not been necessary to address this potential in the market. Concentrates with high Fe:MgO ratios are definitely seen as being desirable (e.g., Western Areas Flying Fox concentrates), as blending of such material improves the capability to take high MgO concentrates on which offered terms would be lower than those indicated and potentially a penalty applied for the elevated MgO.

While the above appear to be the current benchmark, there were clear indications that such high payabilities on nickel may not be sustainable going forward. While specific details as to why this was the case were not provided, it seemed to relate to the potential for the concentrate market to move to surplus in the next few years, making it more difficult to miners to place material at smelters with available capacity. As such, we would recommend evaluating the Turnagain project at nickel payabilities in the range of 68% to 75% to ensure robust economics under such circumstances.

At the currently envisaged initial production level of 22 kt Ni/a in concentrate by 2025, it could be possible to have a single off-take agreement with any of the indicative preferred smelters of Jinchuan, Boliden (Harjavalta), Glencore (Sudbury), and Vale (Sudbury). However, a combination of at least two off-takers would be the most likely viable option; this is typical for current third-party sellers at such levels of nickel in concentrate production.

BHP Group (Kalgoorlie) may also be a viable off-take option. Tsingshan, which was processing sulphide concentrates through a roaster at its Fuan plant as feed to produce nickel pig iron, has ceased to be active in the market, but could provide an opportunity in the future.

Should production levels be raised to the proposed 44 kt Ni/a in concentrate by 2030, then numerous off-takes would almost certainly be necessary to place the material successfully. Wood

Mackenzie's current forecasts indicate that there will be sufficient demand for such tonnages of nickel within the concentrate market at that time.

One factor that Wood Mackenzie has considered is the potential for high-grade (~20% Ni) concentrates to be treated directly by hydrometallurgical processing. This has been demonstrated commercially at Sherritt's Fort Saskatchewan facility (which treated concentrate from 1954 to 1990), at BHP's (then Western Mining Corporation) Kwinana refinery, and is currently being employed at Vale's Long Harbour refinery (which is treating 100% of the Voisey's Bay concentrate). Hydrometallurgical processing has also been evaluated more recently by Independence Group and Western Areas, both in Australia, to process all or part of their concentrate production from the Nova-Bollinger and Flying Fox operations. The processing of high-grade nickel concentrates to directly produce nickel chemicals or Class 1 nickel has the potential to improve the revenue generated from a mine's output. However, this option has been excluded from the scope of this study, as insufficient data currently exist on the commercial and technical feasibility to allow a sound evaluation of the economic viability.

# 20.0 ENVIRONMENTAL BASELINE STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

## 20.1 Environmental Baseline Studies

In preparation for an environmental assessment of the project, baseline environmental studies were initiated in 2004 and are ongoing. Several additional baseline environmental studies will be required to fully satisfy the requirements of an environmental impact assessment, as well as to support the necessary permits, approvals, and licences for the project. This will include studies or monitoring of fish and aquatic habitat, wildlife, vegetation, soil, noise, groundwater, air quality, 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 potential issues as they are identified. The environmental programs carried out to date are described in the subsections below.

### 20.1.1 Aquatic Life

In 2007, an initial fish and fish habitat baseline study that was conducted for late summer/fall provided an initial understanding of species diversity and abundance as well as habitat features. Additional fisheries and aquatic studies will be required during the environmental assessment process and other permitting processes. The proposed conceptual design for waste management involves displacement and alteration of productive aquatic habitat in Flat Creek Valley. To meet the requirements of the "no net loss of fish habitat" principle of Fisheries and Oceans Canada (DFO), mitigation and compensation measures will be required to replace any lost productivity. Additional seasons and detailed studies of the existing fish species, their life cycle, and utilisation of the physical aquatic habitat will be necessary to design and propose acceptable fisheries compensation plans. The following investigations may be required:

- investigation, mapping, and quantification of existing aquatic habitat and its utilisation (spawning, rearing, overwintering) by resident species in Flat Creek and its fish-bearing tributaries
- estimation of existing number of fishes 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) to provide support for fish communities in perpetuity. Depending on negotiations with the BC Ministry of Environment and Climate Change Strategy (MOE) and DFO, additional fisheries compensation options may be expected. To prepare for this outcome, a list of potential compensation options will 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.

### 20.1.2 Surface Water Quality

Preliminary surface water quality surveys were completed in the project area between 2003 and 2011. Monthly surface water sampling was re-started in late 2018 and 2019 at 13 sites and ongoing monitoring is planned. In general, surface water in the area can be characterised as having a neutral pH, low to moderate hardness, and low dissolved and suspended solids concentrations. All nutrient and anion concentrations were relatively low.

### 20.1.3 Soil Quality

Soil geochemical surveys were first conducted by Falconbridge in 1971 and by Bren-Mar in 2003, followed by a comprehensive geochemical soil sampling program initiated by HNC in 2004 involving the collection of more than 2,000 soil samples. Results highlighted two strong copper-in-soil anomalies 2.5 km northwest of the Horsetrail Zone with values exceeding 430 ppm copper and peaks to 3,219 ppm copper. 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 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”. Future investigations of soil quality are required to support the environmental impact assessment and permitting efforts.

### 20.1.4 Groundwater Flows

Groundwater levels in the project area were collected in September 2008 and June 2009 at 93 geotechnical boreholes within the main pit with negligible information collected from beyond the pit area. Being spatially constrained, there is insufficient groundwater level data across the project site to enable a characterisation of groundwater resources and flow across the local study and regional study areas (e.g., to identify recharge / discharge zones, or to quantify mine / pit de-watering rates). Although data was collected over two individual years, the data set does not capture the seasonal variation within a particular year. Due to the age of the data (collected more than 10 years ago), information may not be representative of current groundwater flow conditions. Future hydrogeological investigations are required to support the environmental impact assessment and permitting efforts.

### 20.1.5 Groundwater Quality

Eight groundwater wells, also predominantly from the pit area, were monitored for water quality (i.e., physical parameters, anions and nutrients, and total and dissolved metals) between 2004 and 2011, and limited groundwater quality samples were collected in 2018 and 2019. Groundwater sites have a more basic pH, with variable hardness. Anions and nutrients are typically low except for fluoride concentrations. Metal criteria anomalies (with respect to British

Columbia Water Quality Guidelines for the Protection of Aquatic Life) were observed for cadmium, copper, selenium, aluminum, and nickel at some sampling locations. Although data were collected over seven years at different locations in the Hard Creek Sub-Catchment, the data set does not capture seasonal variation within a particular year. Based on a preliminary review, groundwater quality data are insufficient to establish baseline groundwater quality conditions across the local study and regional study areas, or to understand the interaction between surface and groundwater quality emergent zones. Additional groundwater quality monitoring is required to establish baseline groundwater quality conditions and support the environmental impact assessment and permitting efforts.

### 20.1.6 Archaeological Studies

Exploration pre-clearing was completed in 2018 and 2019. Archaeological impact assessment studies are being conducted with the help of Tahltan and Kaska members.

## 20.2 Regulatory Requirements

### 20.2.1 Environmental Impact Assessment

The project is subject to a provincial review under the *British Columbia Environmental Assessment Act* as it exceeds the following threshold under Part 3 (Table 6) of the Reviewable Projects Regulation: “A new mine facility that, during operations, will have a production capacity of >75,000 tonnes/year of mineral ore”. The project will also be subject to a review under the federal *Impact Assessment Act*, as it exceeds the following threshold prescribed in the Schedule of Physical Activities specified in the Physical Activities Regulations (SOR/2019-285): Section 18(c): “The construction, operation, decommissioning and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 t/day or more”. The Environmental Impact Assessment (EIA) process is expected to be conducted in one review process through a substitution agreement between the provincial and federal agencies, and will involve public and First Nations consultation as well as detailed studies of baseline environmental settings and an assessment of potential project impacts. A comprehensive study of potential impacts of the project and its facilities and components on the surrounding environment was initiated by HNC in 2004, as well as public and First Nations consultation efforts.

### 20.2.2 Permits & Approvals

Options for tailings storage facilities at Hard Creek and Flat Creek were evaluated during the preliminary assessment and conceptual design stages, all of which involved alteration and possible loss of aquatic habitat. Based on the analysis and a mill throughput of 87,000 t/d over a mine life of 25 years, preferred options for a tailings management facility (TMF) have been identified at both Hard Creek and Flat Creek. The possible impact on the either drainage will require authorisation under Section 35(2) of the federal *Fisheries Act* for harmful alteration, disruption, or destruction (HADD) of fish habitat. As part of the authorisation 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 satisfy the regulatory bodies that all possible design alternatives have been considered and evaluated, and that fish utilisation and connectivity would be maintained through implementation of appropriate mitigation and compensation strategies.

The proposed TMF design will likely also require an amendment to Schedule 2 of the Metal Mine Effluent Regulations (MMER) of the *Fisheries Act*, to 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.

Other permits, approvals, and regulatory requirements that may apply to the project at various stages include, but are not limited to, the following:

- Migratory Birds Convention Act Authorisation, for vegetation clearing during migratory bird nesting season
- Species at Risk Act Permit, for activities that may affect a listed species or its habitat
- Explosives Act License and Permit, for explosives transportation and magazines
- Transportation of Dangerous Goods Act and Regulations Permits, for transport of dangerous goods by rail, road or air
- Mines Act Permit, for approval of the Mine Plan and Remediation and Closure Plan
- Environmental Management Act Waste Discharge Permit, Waste Storage Approval, for authorisation of waste storage and discharge of wastes to water, land or air
- Heritage Conservation Act Concurrence Letter stating that the assessment is complete
- Wildlife Permit Act Approvals for wildlife salvages and bird nest removal or relocation
- Drinking Water Protection Act Permits for potable water wells, water system construction, and water system operations
- Water Sustainability Act Approvals and Licenses for changes in and about a stream, management of nuisance water from mining operations, and activities requiring surface or groundwater resources for potable or process water
- Forest Act Licence to Cut and Special Use Permit for harvesting, and for use of Crown Land within a Provincial Forest
- Transportation Act Industrial Access Permit, for new roads joining onto public roads
- Public Health Act Permit for regulated activity such as managing septic systems and processing wastewater.

## 20.3 Indigenous Groups & Local Communities

The project is located within the traditional territories of both the Tahltan First Nation and the Kaska Dena, and roughly 65 km east of the Township of Dease Lake.

Giga Metals is committed to creating and sustaining constructive dialogue and relationships with indigenous groups and local and regional stakeholders to support the environmental, social, and economic sustainability of the project. Key components of Giga Metals' consultation and engagement strategy will include:

- communication of information in a timely, consistent manner to indigenous groups, local community members, stakeholders and regulators throughout the lifecycle of the project
- early identification and understanding of project issues and concerns, as well as responsive engagement regarding indigenous groups and stakeholder interests
- early, frequent, open, and honest communication to build strong relationships with interested parties, particularly those who will be most affected by the project
- fostering strong, collaborative, long-term partnerships with indigenous groups, regulators, community groups, and other stakeholders

Giga Metals has established consultation and engagement processes with indigenous groups, communities, and stakeholder groups. Further consultation and engagement will be required for the purposes of conducting baseline socioeconomic studies; assessing and reviewing potential environmental effects during the environmental assessment process; developing agreements; developing environmental and social management plans; closure planning; environmental monitoring; providing regular project development updates; and ensuring optimal economic development opportunities and participation during construction and operations.

### 20.3.1 Indigenous Groups

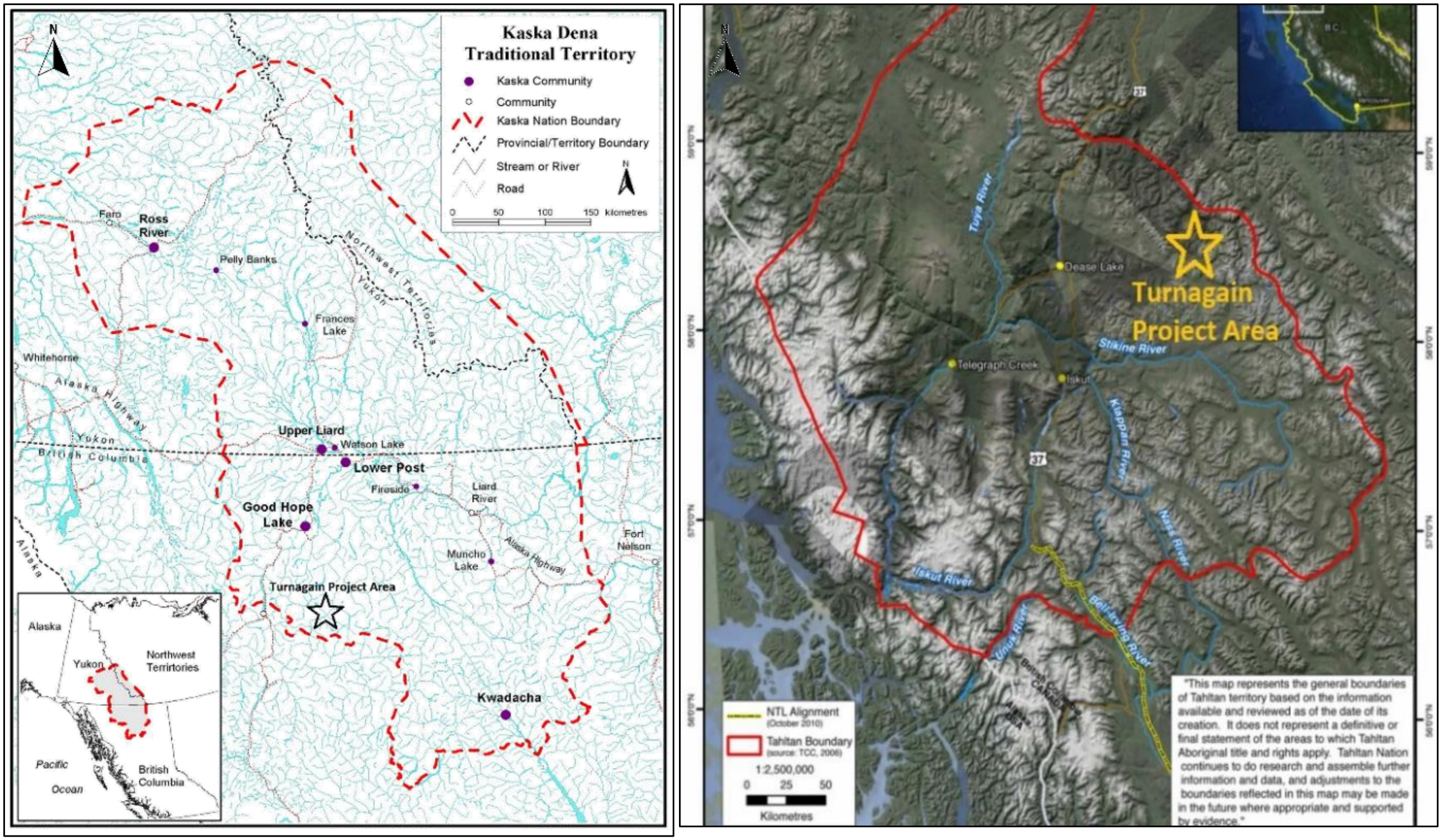
The Tahltan Nation is represented by the Tahltan Central Government (TCG) for aspects related to indigenous rights and title, and natural resource development within Tahltan traditional territory (Tahltan Territory). The TCG represents the Tahltan Band, Iskut Band, and roughly 5,000 members of the Tahltan Nation, and governs according to the principles enshrined in the Declaration of the Tahltan Tribe (1910). Tahltan Nation members comprise over half of the residents in Tahltan Territory, and primarily live in three communities: Telegraph Creek, Dease Lake, and Iskut. The TCG is located in Dease Lake.

Tahltan Territory is unceded, and comprises roughly 95,933 km², or 11% of British Columbia. Tahltan Territory includes the Stikine Watershed and headwaters, ranging from the lower Yukon boreal forest to the Cassiar Mountains to the east, the Skeena and Nass River headwaters in the south, and the Central Mountains in the West.

The Kaska Dena traditional territory and people are represented and governed by the Kaska Dena Council, which is headquartered in Lower Post and represents the five bands of the roughly 3,000 Kaska Dena people, located in the communities of Lower Post, Good Hope Lake, Watson Lake, Ross River and Kwadacha. The area covered by the Kaska Dena traditional territories includes portions of the Yukon Territory, Northwest Territories, and British Columbia. Within BC, the territory range includes Lower Post, Good Hope Lake, and Kwadacha.

Figure 20-1 shows the project location in relation to the Tahltan and Kaska Dena traditional territories.

**Figure 20-1: Turnagain Project Relative to Traditional Territories of Kaska Dena & Tahltan**




Source: Giga Metals, 2020.

The Kaska Dena Council and TCG are increasingly entering into shared decision-making agreements with the province of BC, regarding land and resource use development in their territory. The Kaska Dena Council (including Dease Lake First Nation) and the TCG have plans, agreements, and protocols to govern land and resource use in their asserted traditional territories. These include the Kaska Dena Land Use Framework; the FLNRO Dease-Liard Sustainable Resource Management Plan (2012), which includes the Dease River First Nation's Principles and Policy For Mineral Exploration and Mine Development Outline; the Kaska Dena Ne'āh', Gu Cha Duga, and the Strategic Land Use Plan Agreement; and the Tahltan Nation's Klappan Plan (2017) to guide land and resource management in the Klappan Area, among others.

The Klappan Plan defines and describes areas of Tahltan traditional territory that are considered acceptable or unacceptable for resource development activities, and defines activities (such as jade and placer mining) that are considered unacceptable for their territory. The Turnagain Project is north of Klappan Area Sacred Headwaters Zone and Zone B North. No industrial activities are considered acceptable for Sacred Headwaters. Industrial activity and resource extraction could occur in Zones B and C, subject to extensive engagement with Tahltan Nation. Tahltan Nation is developing its Tahltan Land Stewardship Plan, with a land use planning process currently under way.

Giga Metals' Indigenous Engagement strategy will be multifaceted and developed in consultation with Indigenous Groups to incorporate their engagement priorities and timelines. A detailed Indigenous Engagement Plan will be developed to formalise the following elements of Giga Metals' Engagement Strategy: Indigenous Group identification; participation in Technical Working Groups; in-person meetings; Open Houses; Notifications (including newsletters); documentation review; issues tracking and reporting; participation in assessments; and mechanisms for capacity building for both Giga Metals (including cultural awareness training) and indigenous groups.

As part of the project's environmental assessment, detailed Indigenous Knowledge and Traditional Land Use (TLU) studies will be completed in partnership with indigenous groups. Indigenous Knowledge studies completed previously for the project (e.g., Dena Kayeh Institute, 2010) will be updated as required. All archaeological and heritage assessments will be conducted in partnership with indigenous groups. Giga Metals will respect and work in partnership with indigenous groups to incorporate into baseline data, the environmental assessment, and mitigation and management plans any information provided by indigenous knowledge holders and keepers, land guardians, elders, and representatives.

Treaty 8 First Nations and the Métis Nation of BC may also be interested in the project. The project is located inside the Western boundary of Treaty 8 traditional territory, as recently determined by the BC Supreme Court (*West Moberly First Nations v. British Columbia*, 2017).

Figure 20-1 shows the Turnagain Project location in relation to this Western boundary of Treaty 8 traditional territory. Treaty 8 representatives participated as observers on the BC Muskwa-Kechika Advisory Board to which Giga Metals presented the project in October 2018.

#### 20.3.1.1 Indigenous Consultation & Engagement Activities

Hard Creek Nickel Corporation first initiated engagement with the Tahltan Nation and Kaska Dena in 2004, during development of the preliminary environmental baseline program. Engagement since then has included in-person meetings, telephone calls, letters and email correspondence.

Giga Metals has entered into a Communications Agreement with the TCG that establishes a framework for communications with the Tahltan Nation, and is renewed annually. Giga Metals also has an Opportunities Sharing Agreement with the TCG that establishes a framework for communications and cooperation on employment, contracting and community opportunities. This agreement was transferred from Hard Creek Nickel Corporation to Giga Metals and was updated in April 2020.

In October 2008, the Daylu Dena Council, Dease River Band Council, Kwadacha First Nation and Kaska Dena Council, and Hard Creek Nickel Corporation entered into a Cornerstone Agreement that outlined the intentions and obligations of the parties regarding BC Kaska's initial participation in the Turnagain Project. The Cornerstone Agreement expired on December 31, 2010.

Hard Creek Nickel Corporation had previously entered into a Protocol with the Daylu Dena Council, Kaska Dena Council, Dease River Indian Band, Kwadacha First Nation, and the Dena Keyeh Institute (signed June 9, 2009) which included commitments by the parties regarding environmental protection, economic opportunities and benefits, Kaska knowledge, education and training. This Protocol expired when the Cornerstone Agreement expired.

Following the Cornerstone Agreement expiration, Giga Metals is negotiating an exploration agreement with Kaska Dena Council, with Dease River First Nation leading the negotiations on behalf of Kaska Dena Council. Giga Metals is maintaining a positive working relationship with the Dease River First Nation and the Kaska Dena Council.

Giga Metals has shared draft management plans with the Tahltan Nation and Kaska Dena for their review and input. For example, a draft Wildlife Mitigation Plan prepared by EDI on behalf of Giga Metals and an Archaeological Chance Find Procedure were provided to the Tahltan, Kaska Dena, and provincial government for review.

### 20.3.2 Local Communities

The closest community to the project is Dease Lake (unincorporated), which is a town of approximately 335 people (BC Census Data, 2016) located on Highway 37 at the south end of Dease Lake. Other local communities include Telegraph Creek, Iskut, and Good Hope Lake. There are no residences near the project property.

The cities of Terrace (population 15,700) and Smithers (population 10,600) are 580 km and 600 km to the south of Dease Lake, respectively.

Throughout the environmental assessment process, Giga Metals will engage with local and regional stakeholders interested in and potentially affected by the project. These will include local communities, local and regional governments, recreational organisations and users, community groups, rights holders, utilities, and local and regional businesses. Giga Metals will solicit input on the project through project notifications, a project website, Open Houses, and in-person meetings.

## 20.4 Government Agencies Consultation & Engagement

Consultation and engagement with federal and provincial government agencies will continue during the environmental assessment process through to construction, operations and decommissioning. Table 20.1 provides a record of government consultations to date.

**Table 20.1: Government & Stakeholder Consultation & Engagement Completed to Date**

| Government Organisation | Consultation Method(s) | Date | Summary |
|---|---|---|---|
| Impact Assessment Agency of Canada (formerly CEAA) | Presentation | May 2008<br>August 2018 | Discussions regarding federal involvement in the project |
| BC Environmental Assessment Office | Presentation | November 2007<br>June 2018 | Introductory project meetings. Inquiries about project guidance |
| BC Ministry of Energy, Mines and Petroleum Resources (MEM) | In-person meetings | 2018 and 2019 | Project discussions, and intent to enter the EA process for major projects |
| BC Ministry of Environment and Climate Change Strategy (MOE) | In-person meeting | March 2019 | Introduction of the company and project. |
| BC Ministry of Forests, Lands, Natural Resource Operations and Rural Development (FLNRORD) | Telephone meetings | 2018 and 2019 | Permit discussions<br>Indigenous Referral discussions |
| BC Muskwa-Kechika Advisory Board | Presentation in person | October 23, 2018 | Introduction to Giga Metals and the Turnagain Project |
| Department of Fisheries and Oceans (DFO) | In-person meeting | January 2008 | Discussions of best management practices, and the terms of reference and timeline for major projects. |

## 20.5 Reclamation & Closure Plan

The main objective of closure is to minimise 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 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 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 stabilisation 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.

## 20.6 Greenhouse Gases (GHGs)

### 20.6.1 Sequestration of Carbon Dioxide with Mine Tailings

The host rocks of the Turnagain deposit contain a variety of minerals, such as olivine and serpentine, which are known to react with carbon dioxide ($CO_2$) under atmospheric conditions to sequester the greenhouse gas as mineral carbonates over geological timeframes. For individual minerals, the reaction rate is influenced by a variety of factors, including particle size (specific surface area), temperature, $CO_2$ pressure, moisture, and water chemistry. With the fine grinding of the Turnagain material for froth flotation and deposition of the tailings sub-aerially in a large tailings management facility, some level of carbonation is expected. At this time, insufficient information is available to quantify the expected sequestration, and no approved quantification protocol exists that would allow an economic valuation, but $CO_2$ sequestration by the TMF is expected to help achieve the goal of $CO_2$ neutrality in future years. The company is actively working on quantifying $CO_2$ sequestration rates through ongoing independent scientific research at the University of British Columbia by Dr. Greg Dipple, with an ultimate goal of better defining the quantity of $CO_2$ that can be sequestered and the optimal tailings management strategies. Giga Metals will seek to develop a sequestration offset protocol under either regulatory (provincial or federal) or commercial (i.e., Canadian Standards Association, the Gold Standard, American Carbon Registry) standards. Demonstration of sequestration of significant quantities of $CO_2$ with appropriate management of the Turnagain mine tailings may also provide benefits through enhanced social acceptance of the project with local stakeholders, international stakeholders, and customers.

### 20.6.2 GHG Reduction Opportunity through Fleet Electrification

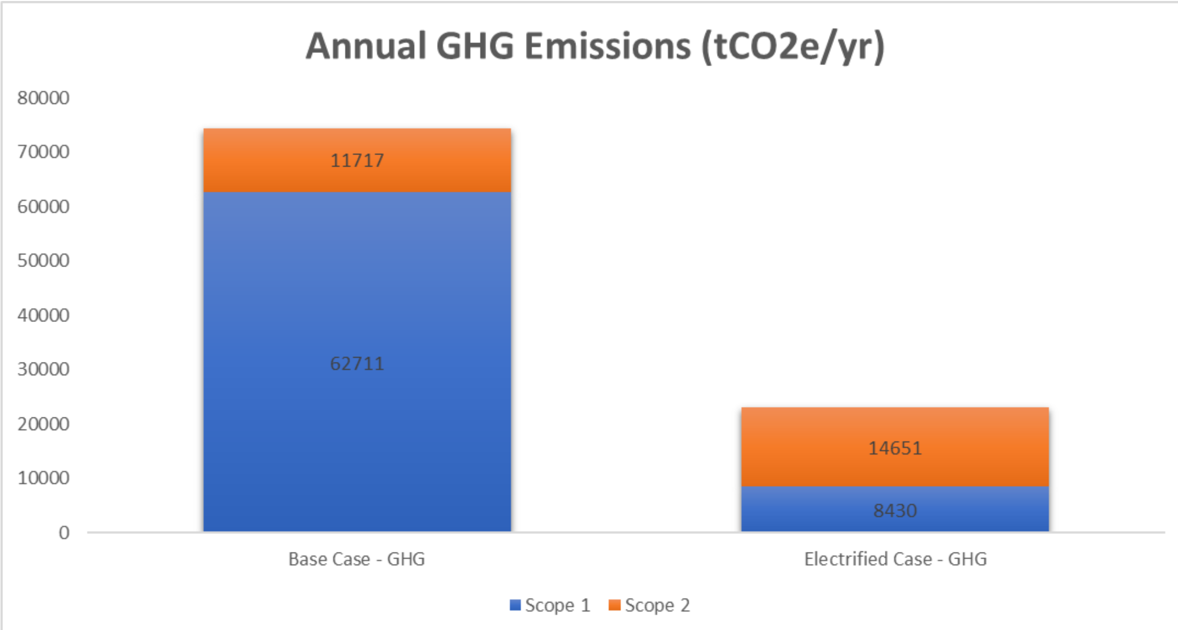
Based on the diesel and electricity consumption for mining and processing activities, the carbon intensity calculated for the operations is estimated to average 74,428 $tCO_2e$ per year. Most of the emissions are direct emissions (scope 1) coming from diesel consumption of the mining fleet. There is an opportunity to reduce the carbon intensity through electrification of the mining fleet and shift the majority of the fleet emissions to indirect emissions (scope 2).

In an electrified fleet scenario, the model assumes that mining trucks and loaders would run on hydrogen, and the support fleet will be fully battery electric, based on technological feasibility, upcoming market offerings, and recognising the limitations of battery-only powertrains for such heavy duty applications.

These opportunities could result in reducing the carbon emissions to 23,080 $tCO_2e$ per year. Most of the emissions for the electrification case would be indirect, coming from electricity generation from the BC grid, which has a grid emissions intensity factor of 12 tCO2e/GWh based on the rolling average of the last three years for average emission factor (AEF) for BC electricity consumption. The average emission factors were published in Canada's National Inventory Report (NIR).

It should be noted that BC has published a different grid factor than the average emission factor for the integrated grid of 29.9 $tCO_2e$/GWh (British Columbia, 2020). The published factors, which are different than those used for public sector entities, are to be used by grid-connected entities in quantifying and the official reporting of GHG emissions of electricity which is not self-generated. Hatch has not confirmed with the authorities which emission factors should be used for GHG quantification. It is understood that some of the electricity consumed in BC can be purchased from less clean generation sources in other jurisdictions, which can greatly influence the grid emissions factor. If these neighbouring jurisdictions reduce their grid carbon intensity, it would be expected that the reporting differential would be reduced over time. Figure 20-2 shows the annual GHG emissions for both scenarios using the 12 $tCO_2e$/GWh grid factor from NIR. The average project carbon intensity is estimated to 2.24 $tCO_2e$/t Ni for the base case and 0.69 $tCO_2e$/t Ni for the electrified case.

**Figure 20-2: GHG Reduction Opportunity through Fleet Electrification ($tCO_2e$ per year)**




Source: Hatch, 2000

# 21.0 CAPITAL & OPERATING COSTS

## 21.1 Capital Cost Estimate

The capital cost estimate is in the range of an AACE Class 5 Capital Cost Estimate with a targeted level of accuracy of +30 to +50% and -20% to -30%. All costs are provided in Q1 2020 US dollars. The initial and expansion capital is summarised in Table 21.1.

**Table 21.1: Project Initial/Expansion Capital Cost Summary**

| Item | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Mine Directs | US$M | 133 | 45 | 178 |
| Process Plant Directs | US$M | 307 | 245 | 551 |
| Tailings Storage Facility Directs | US$M | 87 | 20 | 107 |
| On-site Infrastructure Directs | US$M | 77 | - | 77 |
| Indirects | US$M | 204 | 104 | 308 |
| Contingency | US$M | 191 | 99 | 290 |
| Owner's Cost and EA | US$M | 63 | 20 | 83 |
| Electrical Supply | US$M | 278 | - | 278 |
| Site Access Road | US$M | 42 | - | 42 |
| **Total Initial/Expansion Capital** | **US$M** | **1,381** | **532** | **1,913** |

The sustaining and closure and reclamation capital is summarised in Table 21.2.

**Table 21.2: Project Sustaining Capital & Closure & Reclamation Cost Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mine | US$M | 0 | 348 | 148 | 496 |
| Process Plant | US$M | 31 | 165 | 187 | 384 |
| Tailings Storage Facility | US$M | 107 | 377 | 335 | 819 |
| On-site Infrastructure | US$M | 8 | 23 | 26 | 57 |
| Electrical Supply (Tariff Supplement 37) | US$M | 90 | 82 | - | 172 |
| **Total Sustaining Capital** | **US$M** | **236** | **996** | **697** | **1,928** |
| Closure and Reclamation | US$M | 38 | 15 | 18 | 72 |
| **Total** | **US$M** | **274** | **1,011** | **715*** | **2,000** |

* Includes $2.8M in TMF and closure costs in Year 38.

### 21.1.1 Mining

The summary for the mine capital costs is shown in Table 21.3 and Table 21.4.

**Table 21.3: Mine Capital Cost Estimate**

| Item | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Mining Equipment | US$M | 123 | 45 | 168 |
| Mining Pre-production Costs | US$M | 9 | - | 9 |
| **Total** | **US$M** | **133** | **45** | **178** |

**Table 21.4: Mining Sustaining Cost Estimate**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mining Sustaining | US$M | 0 | 348 | 148 | 496 |

The cost for the initial mine equipment includes the fleet requirements to meet steady-state production in Year 2.

Most unit prices are based on Hatch internal and external benchmark data including recent budgetary pricing for the major production units. Equipment pricing is based on new units delivered to the mine, with transportation and erection costs included. Used equipment, if available, will reduce these equipment capital costs, and have not been considered for this study.

Pre-stripping will not be necessary, as the initial mineralisation feed will be near surface and accessible when the plant starts up. It is anticipated that land clearing and road access work will be performed by a mining contractor(s). Site preparation includes clearing and grubbing, site drainage, initial access roads, and the main haul road from the pit to the crusher. No detailed designs have been undertaken for this estimate.

A 10% factor of the support equipment fleet totals is included to account for ancillary items such as engineering supplies, light plants and pumps. A 5% critical spares factor was added to the initial equipment capital acquisitions.

### 21.1.2 On-Site Infrastructure & Process Plant

The on-site infrastructure and process plant direct capital costs are summarised in Table 21.5.

The concentrator capital cost estimate was prepared as a factored cost estimate based on a priced mechanical equipment list:

- Vendor budget prices provided 47% of the equipment costs.
- The remaining equipment was priced using Hatch database prices or factors to cover a percentage of minor equipment
- Earthworks, concrete, equipment support steel, pre-engineered buildings, piping, electrical and instrumentation/control costs were factored per area.

Infrastructure costs were taken from a recent similar project and allowances were made for some items.

**Table 21.5: On-site Infrastructure & Process Plant Direct Capital Cost**

| Items | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Truck Shop/ANFO Plant | US$M | 21 | - | 21 |
| Site Infrastructure (Camp, Utilities, etc.) | US$M | 57 | - | 57 |
| **Direct On-site Infrastructure Capital Cost** | US$M | **77** | **-** | **77** |
| | | | | |
| Crushing/HPGR/Conveyor | US$M | 133 | 77 | 210 |
| Grinding/Flotation/Concentrate | US$M | 174 | 168 | 342 |
| **Direct Process Plant Capital Cost** | US$M | **307** | **245** | **551** |

### 21.1.3 Off-Site Infrastructure

#### 21.1.3.1 Electrical Power Line

Based on preliminary cost estimates, the 287 kV transmission line connection to BC Hydro, including the transmission line and substations, would require approximately US$278 M in capital expenditure. The annual operating cost of Giga Metals' portion of the transmission line would be approximately US$1.2 M/a. The operating costs include electrical losses as well as line and vegetation management.

BC Hydro's Tariff Supplement 37 requires customers connected to the NTL to pay a charge linearly scaled to their usage compared to the capacity of the line. The above capital cost is based on Tariff Supplement 37 costs being financed through BC Hydro with monthly payments over the first five years of Phases 1 and 2.

Electricity would be purchased from BC Hydro under Rate Schedule 1823 – Transmission Service – Stepped Rate, with a demand charge of C$8.609/kVA and an energy charge of C$50.47/MWh. The resulting blended rate is approximately C$65/MWh (US$51/MWh). The annual cost of electricity from BC Hydro is estimated at US$56 M/a at full build-out. Refer to Table 21.6 and Table 21.7 for a summary of these costs.

**Table 21.6: Electrical Supply Capital Cost**

| Electrical Supply | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Transmission Line from Tatogga to Turnagain | US$M | 278 | - | 278 |
| Tariff Supplement 6 Allowance | US$M | - | - | - |
| Tariff Supplement 37 | US$M | 90 | 82 | 172 |
| **Total** | **US$M** | **368** | **82** | **450** |

**Table 21.7: Electrical Supply Sustaining Capital Cost**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Electrical Supply (Tariff Supplement 37) | US$M | 90 | 82 | - | 172 |
| **Total** | **US$M** | **90** | **82** | **-** | **172** |

#### 21.1.3.2 Site Access Road

The project's contribution to the cost of the access road was estimated as a direct cost of approximately C$500,000 per kilometre. No engineering was done for the access road. This should be reviewed in more detail in the next phase study.

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

The total initial capital costs associated with the TMF and other environmental management items are summarised in Table 21.8 and Table 21.9. The main components of the costs are listed by principal category in the following subsections.

**Table 21.8: TMF Capital Costs**

| Item | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Tailings Management Facility | US$M | 75 | 20 | 95 |
| Water Management | US$M | 12 | - | 12 |
| **Direct** | **US$M** | **87** | **20** | **107** |

**Table 21.9: TMF Sustaining Capital & Closure & Reclamation Costs**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-38) | Life of Mine (Y1-38) |
|---|---|---|---|---|---|
| Tailings Storage Facility | US$M | 107 | 377 | 335 | 819 |
| Closure and Reclamation | US$M | 38 | 15 | 18 | 72 |
| **Total** | **US$M** | **145** | **392** | **353** | **891** |

#### 21.1.4.1 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
- tailings pipelines and fittings
- reclaim water system (including pipes and pumps)
- local 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 related to mitigating unmapped terrain hazards associated with debris flows, avalanches, major stream crossings, and other geo-hazards have not been considered in this estimate. A project contingency is included to account for unanticipated items.

#### 21.1.4.2 Site-wide Water Supply & Water Management

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

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

#### 21.1.4.3 Closure & 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:

- mobilisation and demobilisation
- 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.

## 21.2 Operating Cost Estimate

The overall operating costs are summarised in unit cost terms in Table 21.10, which shows the life-of-mine costs as well as the costs during Years 1 to 5 at reduced throughput and Years 6 to 20 and 21 to 37 at full capacity.

**Table 21.10: Unit Operating Cost Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mining | US$/t milled | 3.52 | 2.89 | 2.46 | 2.72 |
| Processing & Site Infrastructure | US$/t milled | 4.90 | 4.39 | 4.38 | 4.42 |
| G&A | US$/t milled | 1.13 | 0.68 | 0.68 | 0.71 |
| Electrical Supply O&M | US$/t milled | 0.08 | 0.04 | 0.04 | 0.04 |
| **Total** | **US$/t milled** | **9.63** | **7.99** | **7.56** | **7.89** |

### 21.2.1 Staffing Numbers

The Turnagain mine will be a fly-in/fly-out camp operation. Use of available local labour will be prioritised. Mine labour estimates are based on a four-crew rotation with management and technical staff on a two-crew rotation. The hourly labour workforce reflects estimated equipment hours and annual quantity of material mined. During the first five years of lower production, the total mine labour count averages 233; it then increases to an average of 324 during the peak period between Years 14 to 33. When the pit is completed and the mill is fed from the mineralisation stockpiles after Year 32, the mine labour force will be reduced significantly.

The mine staffing numbers developed in Section 16 are summarised in Table 21.11.

**Table 21.11: Mine Operations Staffing – Average for Periods**

| Department | Years | | | |
|---|---|---|---|---|
| | 1 to 5 | 6 to 13 | 14 to 33 | 34 to 37 |
| **Hourly Labour** | | | | |
| Mine Operations (hourly) | 113 | 158 | 175 | 67 |
| Mine Maintenance (hourly) | 54 | 77 | 85 | 34 |
| **Subtotal** | **168** | **235** | **259** | **101** |
| **Salaried Staff – Mine & Maintenance Operations** | | | | |
| Senior Mine Management | 4 | 4 | 4 | 2 |
| Senior Maintenance Management | 2 | 2 | 2 | 2 |
| Mine & Maintenance General Foremen | 4 | 4 | 4 | 0 |
| Mine & Maintenance Foreman/Lead Hands | 13 | 13 | 13 | 5 |
| Mine & Maintenance Dispatch/Clerks/Planners | 16 | 16 | 16 | 4 |
| **Subtotal** | **39** | **39** | **39** | **12** |
| **Mine Technical** | | | | |
| Supervisor/Senior Engineers & Geologists | 6 | 6 | 6 | 1 |
| Engineers & Geologists | 4 | 4 | 4 | 1 |
| Surveyors, Assayers, Helpers & Staff Coverage | 16 | 16 | 16 | 3 |
| **Subtotal** | **26** | **26** | **26** | **5** |
| **Total Salaried Staff** | **65** | **65** | **65** | **17** |
| **Total Mine Workforce** | **233** | **300** | **324** | **118** |

Processing staffing numbers at full capacity are summarised in Table 21.12.

**Table 21.12: Processing Staffing Numbers**

| Job Description | Hourly | Staff |
|---|---|---|
| Mill Operations Staff | | 7 |
| Mill Maintenance Staff | | 9 |
| Maintenance | 40 | |
| Electrical | 10 | |
| Milling - Operations | 44 | |
| Metallurgy | | 8 |
| Assay Lab | 6 | |
| **Total** | **100** | **24** |

### 21.2.2 Mining

The average mine operating cost is estimated to be US$2.30 per tonne of material mined (including life-of-mine rehandling costs) or US$2.72 per tonne milled over the life of mine. The cost estimate consists of all mining activities from the pit to the crusher pocket berm, and includes the waste and low-grade stockpile facilities. Table 21.13 shows the estimated operating costs for each of the areas.

**Table 21.13: Mine Operating Cost Estimate**

| Item | Unit Cost ($/t Mined) | Unit Cost ($/t Milled) |
|---|---|---|
| Drilling | 0.16 | 0.19 |
| Blasting | 0.22 | 0.26 |
| Production Loading | 0.14 | 0.16 |
| Production Hauling | 0.86 | 1.02 |
| Pit Operations Support | 0.16 | 0.19 |
| Shop Equipment | 0.02 | 0.03 |
| **Subtotal Cost Centre (excluding labour)** | 1.56 | 1.84 |
| Mine Operations Labour | 0.35 | 0.41 |
| Operations Technical/Supervision Labour | 0.11 | 0.12 |
| Coverage & Training - Mine Department | 0.06 | 0.07 |
| Maintenance Labour | 0.18 | 0.22 |
| Maintenance Supervision Labour | 0.02 | 0.02 |
| Coverage & Training - Maintenance Department | 0.03 | 0.03 |
| **Subtotal Cost Labour** | 0.74 | 0.87 |
| **Total (including rehandling costs)** | 2.30 | 2.72 |

Preliminary equipment productivities were generated and applied against the annual production quantities to estimate equipment operating hours. Consumption rates for consumables and unit operating costs were 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.

Operating costs fluctuate annually and reflect the total material mined and haulage distances. Balancing waste quantities and haulage destinations will smooth the operating costs and minimise fluctuation. Some smoothing was applied to the production schedule in this study.

The costs for power and diesel are $.050/kWh and $0.90/L, respectively. These unit prices are consistent with those assumed in other areas of operation. Mine operations power costs were calculated utilising the estimated kilowatt-hours for each year of operation of the production electrified equipment. Peak annual power consumption is estimated to be 34 million kilowatt-hours (MkWh), with the average being 27 MkWh/a during the high usage period between Years 14 to 25. Peak annual diesel consumption for explosives and the mine equipment fleet is estimated to be 32 ML, with the average being 24 ML during Years 14 to 25.

Explosives quantities were calculated using typical powder factors from existing operations and projects similar in nature. Recently attained explosives and accessories unit costs were applied to the projected quantities to estimate costs for blasting materials. Ammonium nitrate/fuel oil (ANFO) usage has been assumed to be 75%, based on successful dewatering to ensure dry blastholes 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 Hatch's internal data sources. 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.

### 21.2.3 Concentrator

Concentrator operating costs are summarised in Table 21.14.

**Table 21.14: Concentrator Operating Costs (Steady State)**

| Item | Units | Phase 1 | Phase 2 |
|---|---|---|---|
| Operation Labour | US$/t milled | 0.28 | 0.16 |
| Maintenance Labour | US$/t milled | 0.27 | 0.15 |
| Power | US$/t milled | 1.59 | 1.56 |
| Heating (Propane) | US$/t milled | 0.13 | 0.09 |
| Operating Consumables | US$/t milled | 2.11 | 2.11 |
| Maintenance | US$/t milled | 0.31 | 0.27 |
| G&A | US$/t milled | 1.05 | 0.68 |
| **Total** | **US$/t milled** | 5.73 | 5.02 |

### 21.2.4 Off-site Charges

Apart from the smelter terms detailed in Section 19, other off-site charges include concentrate trucking and shipping, amounting to approximately US$108/wmt plus insurance equal to 0.15% of freight value. The main components are trucking to Stewart at US$62.50/wmt, port charges and sampling at US$13.50/wmt and ocean freight at US$30.00/wmt.

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

G&A costs were estimated at US$22.2 million at full capacity based on a labour complement of 46 staff, including access road maintenance and crew transportation.

### 21.2.6 Tailings Management Facility

The estimated life-of-mine power operating cost for the tailings management facility is US$53 million.

# 22.0 ECONOMIC ANALYSIS

## 22.1 Introduction

The preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

The economic analysis is based on a discounted cash flow model. The model includes the life-of-mine production plan, operating costs, capital costs, and market assumptions discussed in this report, in addition to financial assumptions introduced in this section. Net present value (NPV), internal rate of return (IRR) and payback period are calculated in the model and reported before and after taxes.

Returns are highly sensitive to input assumptions and should be viewed in the context of the sensitivity analysis provided in this section.

## 22.2 Returns Summary

The PEA case is based on the long-term nickel and cobalt prices of US$7.50/lb Ni and US$22.30/lb Co, as provided by Wood Mackenzie, and does not achieve a positive NPV at an 8% discount rate. Pre-tax NPV becomes positive at the Wood Mackenzie long-term ESG incentive price of US$8.50/lb Ni. The returns are shown in Table 22.1.

**Table 22.1: Returns Summary**

| Item | Unit | Case 1: PEA<br>Wood Mackenzie Long-Term Price US$7.50/lb Ni | Case 2:<br>Wood Mackenzie Long-Term ESG Incentive Price US$8.50/lb Ni |
|---|---|---|---|
| **Pre-Tax NPV @ 8%** | **US$M** | **(269)** | **242** |
| Pre-Tax IRR | % | 6.3% | 9.4% |
| Pre-Tax Payback | years | 13.7 | 10.8 |
| **After-Tax NPV @ 8%** | **US$M** | **(443)** | **(88)** |
| After-Tax IRR | % | 4.9% | 7.4% |
| After-Tax Payback | years | 14.8 | 11.7 |
| **Ni Price** | **US$/lb** | **7.50** | **8.50** |
| Co Price | US$/lb | 22.30 | 22.30 |
| Exchange Rate | USD/CAD | 0.77 | 0.77* |

*Inverse of 1.30 CAD/USD applied.

The PEA case (Case 1) at US$7.50/lb Ni is used as the basis for this report. The nickel price is the long-term average forecast by Wood Mackenzie. The PEA case economic outputs are summarised in Table 22.2.

**Table 22.2: PEA Case Key Economic Outputs**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | LOM |
|---|---|---|---|---|---|
| **Project Economics** | | | | | |
| NPV@ 8% Before Tax | US$M | - | - | - | (269) |
| NPV@ 8% After Tax | US$M | - | - | - | (443) |
| IRR Before Tax | % | - | - | - | 6.3% |
| IRR After Tax | % | - | - | - | 4.9% |
| Payback Period Before Tax | years | - | - | - | 13.7 |
| Payback Period After Tax | years | - | - | - | 14.8 |
| **Market Drivers** | | | | | |
| Nickel Price | US$/lb | 7.50 | 7.50 | 7.50 | 7.50 |
| Cobalt Price | US$/lb | 22.30 | 22.30 | 22.30 | 22.30 |
| Exchange Rate | USD/CAD | 0.77 | 0.77 | 0.77 | 0.77 |
| Nickel Payable % | % | 78% | 78% | 78% | 78% |
| Cobalt Payable % | % | 35% | 35% | 35% | 35% |
| **Physicals** | | | | | |
| Effective Strip Ratio (incl. stockpile) | t::t | 0.50 | 0.56 | 0.24 | 0.40 |
| Ore Throughput: Annual Average | Mt/a | 15.3 | 32.7 | 32.6 | 30.3 |
| Nickel Head Grade | % | 0.260% | 0.220% | 0.216% | 0.221% |
| Cobalt Head Grade | % | 0.016% | 0.013% | 0.013% | 0.013% |
| Recovery: Nickel and Cobalt | % | 57.3% | 51.6% | 46.5% | 49.6% |
| Nickel Recovered | kt | 114 | 557 | 558 | 1,229 |
| Cobalt Recovered | kt | 7 | 33 | 32 | 73 |
| **Financial** | | | | | |
| Revenue | US$M/a | **317** | **517** | **456** | **462** |
| Mining Cost | US$/t milled | 3.52 | 2.89 | 2.46 | 2.72 |
| Processing and Site infrastructure | US$/t milled | 4.90 | 4.39 | 4.38 | 4.42 |
| G&A | US$/t milled | 1.13 | 0.68 | 0.68 | 0.71 |
| Electrical Supply O&M | US$/t milled | 0.08 | 0.04 | 0.04 | 0.04 |
| **Site Operating Costs** | **US$/t milled** | **9.63** | **7.99** | **7.56** | **7.89** |
| **Site Operating Costs** | **US$/lb Ni recovered** | **2.93** | **3.20** | **3.41** | **3.27** |
| Concentrate Shipping | US$/lb Ni recovered | 0.31 | 0.31 | 0.31 | 0.31 |
| Cobalt Credit | US$/lb Ni recovered | (0.47) | (0.47) | (0.45) | (0.46) |
| **Net Operating Cost** | **US$/lb Ni recovered** | **2.77** | **3.04** | **3.27** | **3.12** |
| Construction Capital Cost | US$M | 1,381 | 532 | - | 1,913 |
| Sustaining Capital & Closure Cost | US$M | 274 | 1,011 | 715 | 2,000 |

## 22.3 PEA Case Assumptions & Inputs

### 22.3.1 General

The following general assumptions and criteria form part of this analysis:

- real 2020 US Dollars; no inflation applied
- costs in Canadian dollars converted to US dollars at a 1.30 CAD/USD exchange rate, equivalent to ~0.77 USD/CAD
- three-year Phase 1 construction period from Year -3 through Year -1, and a two-year Phase 2 construction period in Years 4 and Year 5
- mid-year discounting for NPV calculation
- 100% equity financing

### 22.3.2 Nickel & Cobalt Pricing

Wood Mackenzie's long-term prices of US$7.50/lb Ni and US$22.30/lb Co have been applied.

### 22.3.3 Ore Production & Processing

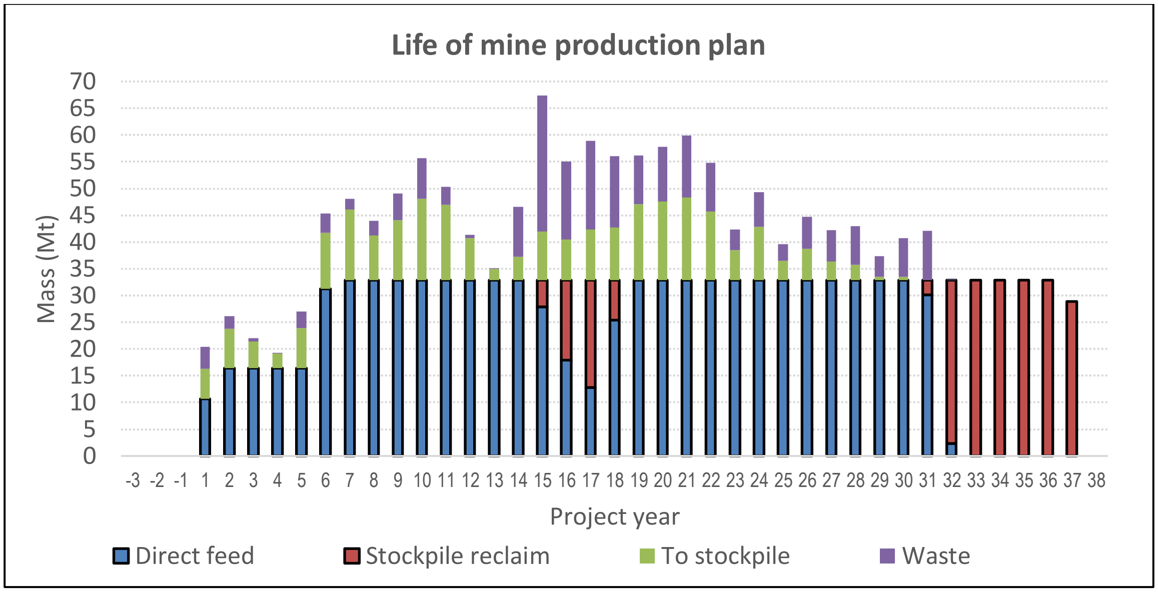
Average production statistics for Phase 1, Phase 2, and the life of mine are shown in Table 22.3.

**Table 22.3: Average Production Statistics**

| Item | Unit | Phase 1 Operation (Y1-5) | Phase 2 Operation (Y6-20) | Phase 2 Operation (Y21-37) | LOM |
|---|---|---|---|---|---|
| Phase Duration | years | 5 | 15 | 17 | 37 |
| Material Mined | Mt | 115 | 719 | 496 | 1,330 |
| Material Moved | Mt | 115 | 767 | 689 | 1,571 |
| Effective Strip Ratio (incl. stockpile) | t:t | 0.50 | 0.56 | 0.24 | 0.40 |
| Ore Processed | Mt | 76 | 491 | 554 | 1,122 |
| Ore Processed (Annual) | Mt/a | 15.3 | 32.7 | 32.6 | 30.3 |
| Nickel Head Grade | % | 0.260% | 0.220% | 0.216% | 0.221% |
| Cobalt Head Grade | % | 0.016% | 0.013% | 0.013% | 0.013% |
| Recovery: Nickel and Cobalt | % | 57.3% | 51.6% | 46.5% | 49.6% |
| Nickel Contained | kt | 199 | 1,080 | 1,200 | 2,479 |
| Nickel Recovered | kt | 114 | 557 | 558 | 1,229 |
| Nickel Payable | kt | 89 | 435 | 435 | 959 |
| Cobalt Contained | kt | 12 | 65 | 70 | 146 |
| Cobalt Recovered | kt | 7 | 33 | 32 | 73 |
| Cobalt Payable | kt | 2 | 12 | 11 | 25 |
| Concentrate Produced | k dmt | 632 | 3,096 | 3,100 | 6,828 |
| Concentrate Produced | k dmt/a. | 126 | 206 | 182 | 185 |
| Concentrate Nickel Grade | % | 18.0% | 18.0% | 18.0% | 18.0% |
| Concentrate Cobalt Grade | % | 1.09% | 1.08% | 1.04% | 1.06% |
| Concentrate Moisture Content | % | 9.0% | 9.0% | 9.0% | 9.0% |

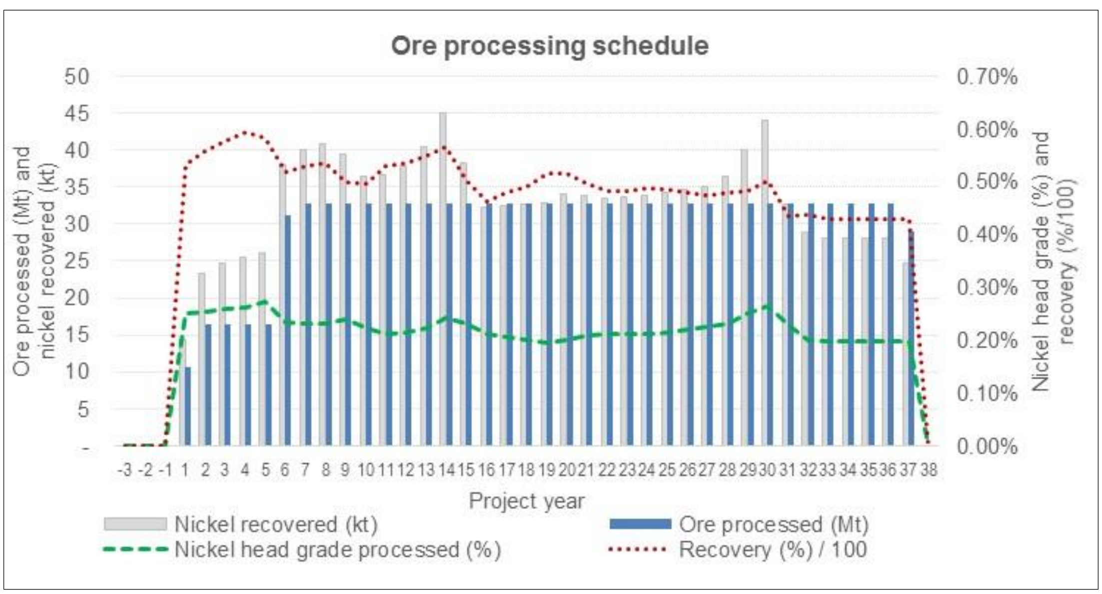
The life of mine production plan applied in the model from Section 16 is shown in Figure 22-1. The ore processing schedule applied in the model corresponding to the life of mine plan is shown in Figure 22-2.

**Figure 22-1: Life-of-Mine Production Plan**




Source: Hatch, 2020.

**Figure 22-2: Ore Processing Schedule**




Source: Hatch, 2020.

### 22.3.4 Concentrate Terms & Transportation Costs

The concentrate terms and transportation costs applied in the model are shown in Table 22.4. The concentrate terms are based on Wood Mackenzie's knowledge of nickel concentrate treatment via smelting and refining to Class I nickel. Nickel processing advancements in the nickel intermediate market (mixed sulphides, mixed hydroxides and higher-grade concentrates) may impact future terms. Section 19 states that current terms for nickel concentrates indicate 78% payability for nickel and 35% payability for cobalt (if above 0.3% in the concentrate). These payables are applied in the PEA base case. Copper payables are not applied as copper is not included in the PEA life-of-mine plan. The copper contribution to revenue would be marginal.

**Table 22.4: Concentrate Terms & Transportation Costs**

| Item | Unit | Applied | Notes |
|---|---|---|---|
| Nickel Payable | % | 78% | Source: Wood Mackenzie |
| Cobalt Payable | % | 35% | Source: Wood Mackenzie.<br>Applied above 0.3% Co grade |
| Treatment Charges | US$/dmt | - | Part of payable deductions |
| **Concentrate Transportation** | **US$/wmt** | **112.66** | China port |
| Trucking | US$/wmt | 62.50 | Source: Hatch |
| Port Charges | US$/wmt | 15.38 | Source: Hatch |
| Ocean Freight | US$/wmt | 30.00 | Source: Hatch |
| Insurance | % of value | 0.15% | Source: Giga Metals |

### 22.3.5 Site Operating Costs

The average unit operating costs in the model are shown in Table 22.5.

**Table 22.5: Site Operating Cost per Tonne of Ore Milled Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | LOM |
|---|---|---|---|---|---|
| Mining | US$/t milled | 3.52 | 2.89 | 2.46 | 2.72 |
| Processing & Site Infrastructure* | US$/t milled | 4.90 | 4.39 | 4.38 | 4.42 |
| Site Level G&A | US$/t milled | 1.13 | 0.68 | 0.68 | 0.71 |
| Electrical Supply O&M | US$/t milled | 0.08 | 0.04 | 0.04 | 0.04 |
| **Total Site Operating Cost** | **US$/t** milled | **9.63** | **7.99** | **7.56** | **7.89** |

*Includes tailings electricity costs.

### 22.3.6 Capital Costs

The construction capital costs from Section 21 applied in the model are shown in Table 22.6. The Phase 1 initial capital costs occur over three years from Year -3 to Year -1. The Phase 2 expansion capital costs occur from Year 4 to Year 5.

**Table 22.6: Project Initial/Expansion Capital Cost Summary**

| Item | Units | Phase 1 | Phase 2 | Life of Mine |
|---|---|---|---|---|
| Mine Directs | US$M | 133 | 45 | 178 |
| Process Plant Directs | US$M | 307 | 245 | 551 |
| Tailings Storage Facility Directs | US$M | 87 | 20 | 107 |
| On-site Infrastructure Directs | US$M | 77 | - | 77 |
| Indirects | US$M | 204 | 104 | 308 |
| Contingency | US$M | 191 | 99 | 290 |
| Owner's Cost and EA | US$M | 63 | 20 | 83 |
| Electrical Supply | US$M | 278 | - | 278 |
| Site Access Road | US$M | 42 | - | 42 |
| **Total Initial/Expansion Capital** | **US$M** | **1,381** | **532** | **1,913** |

Project sustaining capital costs, as well as closure and reclamation costs, are summarised in Table 22.7. Closure and reclamation costs correspond to the incremental growth of the reclamation liability for each period under a bonding scenario.

**Table 22.7: Project Sustaining Capital & Closure & Reclamation Cost Summary**

| Item | Units | Phase 1 (Y1-5) | Phase 2 (Y6-20) | Phase 2 (Y21-37) | Life of Mine (Y1-37) |
|---|---|---|---|---|---|
| Mine | US$M | 0 | 348 | 148 | 496 |
| Process Plant | US$M | 31 | 165 | 187 | 384 |
| Tailings Storage Facility | US$M | 107 | 377 | 335 | 819 |
| On-site Infrastructure | US$M | 8 | 23 | 26 | 57 |
| Electrical Supply (Tariff Supplement 37) | US$M | 90 | 82 | - | 172 |
| **Total Sustaining Capital** | **US$M** | **236** | **996** | **697** | **1,928** |
| **Closure and Reclamation** | **US$M** | **38** | **15** | **18** | **72** |
| **Total** | **US$M** | **274** | **1,011** | **715*** | **2,000** |

*Includes $2.8 M in TMF and closure costs in Year 38.

### 22.3.7 Royalties

The following royalties are applied:

- Conic Metals royalty equal to 2% of net smelter return (NSR) for the life of mine
- Giga Metals buyout of the Schussler-Hatzl royalty discussed in Section 4 of this report: C$4.0 M (US$3.1 M) in Year 1

### 22.3.8 Taxes

Preliminary tax calculations are appropriate at the PEA stage and are applied at the project level. The BC mineral tax and federal and provincial corporate taxes are applied.

The BC mineral tax includes a 2% tax on net current proceeds and a 13% tax on net revenues after the current expenditures account balance becomes positive. An investment allowance rate based on the Bank of Canada rate is applied on the current expenditures account.

Federal and provincial corporate taxes are based on a 15% federal and 12% provincial tax rate. Capital cost allowance (CCA) Class 41 depreciation at 25% is applied and tax losses are carried forward.

The company's tax advisors reviewed the model tax calculation at a high level and did not identify any additional material tax deductions appropriate for a PEA level analysis, including the company's existing tax pools, which are not applied.

### 22.3.9 Working Capital

Working capital is based on 90 days of accounts receivable, 60 days of accounts payable, and 30 days of inventory. Working capital is reflected in the cash flow as changes in net working capital.

## 22.4 Cash Flow

### 22.4.1 Annual Cash Flow Summary

Tables 22.8 and 22.9 show the cash flow summary for the PEA case.

**Table 22.8: PEA Case Cash Flow Summary – Years 1 to 29**

| Item | Units | LOM | -3 | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Prices** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Nickel Price | US$/lb | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 |
| Cobalt Price | US$/lb | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 |
| USD/CAD Exchange Rate | USD/CAD | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 |
| **Physicals** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Material Mined | Mt | 1,330 | - | - | - | 20.5 | 26.2 | 22.1 | 19.2 | 27.0 | 45.4 | 48.0 | 44.0 | 49.1 | 55.7 | 50.3 | 41.3 | 35.0 | 46.5 | 62.4 | 40.1 | 38.9 | 48.5 | 56.2 | 57.7 | 59.9 | 54.8 | 42.4 | 49.3 | 39.6 | 44.8 | 42.3 | 42.9 | 37.4 |
| Effective Strip Ratio (incl. stockpile) | t:t | 0.40 | - | - | - | 0.92 | 0.59 | 0.34 | 0.17 | 0.64 | 0.45 | 0.46 | 0.34 | 0.49 | 0.70 | 0.53 | 0.26 | 0.07 | 0.42 | 1.05 | 0.68 | 0.79 | 0.71 | 0.71 | 0.76 | 0.82 | 0.67 | 0.29 | 0.50 | 0.20 | 0.36 | 0.29 | 0.31 | 0.14 |
| Ore Processed | Mt | 1,122 | - | - | - | 10.7 | 16.4 | 16.4 | 16.4 | 16.4 | 31.2 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 |
| Nickel Grade | % | 0.221% | - | - | - | 0.252 | 0.253 | 0.260 | 0.261 | 0.272 | 0.234 | 0.231 | 0.232 | 0.241 | 0.224 | 0.211 | 0.214 | 0.224 | 0.242 | 0.232 | 0.211 | 0.206 | 0.202 | 0.194 | 0.201 | 0.208 | 0.211 | 0.212 | 0.212 | 0.216 | 0.220 | 0.225 | 0.232 | 0.251 |
| Recovery | % | 49.6% | - | - | - | 53.3 | 55.9 | 57.8 | 59.4 | 58.4 | 52.0 | 52.9 | 53.6 | 49.8 | 49.6 | 53.0 | 53.7 | 54.8 | 56.7 | 50.2 | 46.4 | 48.1 | 49.2 | 51.5 | 51.5 | 49.6 | 48.2 | 48.3 | 48.7 | 48.4 | 47.9 | 47.5 | 47.9 | 48.4 |
| Nickel Payable % | % | 78.0% | - | - | - | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 | 78.0 |
| Nickel Payable | Mlbs | 2,113 | - | - | - | 24.6 | 39.9 | 42.4 | 43.8 | 44.9 | 65.3 | 68.9 | 70.2 | 67.8 | 62.8 | 63.1 | 65.0 | 69.4 | 77.5 | 65.8 | 55.4 | 56.0 | 56.0 | 56.5 | 58.5 | 58.4 | 57.5 | 57.8 | 58.3 | 59.0 | 59.6 | 60.3 | 62.8 | 68.7 |
| Cobalt Grade | % | 0.013% | - | - | - | 0.015 | 0.015 | 0.016 | 0.016 | 0.016 | 0.014 | 0.013 | 0.013 | 0.014 | 0.013 | 0.013 | 0.014 | 0.014 | 0.015 | 0.014 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.012 | 0.013 | 0.013 | 0.013 | 0.014 |
| Recovery | % | 49.6% | - | - | - | 53.3 | 55.9 | 57.8 | 59.4 | 58.4 | 52.0 | 52.9 | 53.6 | 49.8 | 49.6 | 53.0 | 53.7 | 54.8 | 56.7 | 50.2 | 46.4 | 48.1 | 49.2 | 51.5 | 51.5 | 49.6 | 48.2 | 48.3 | 48.7 | 48.4 | 47.9 | 47.5 | 47.9 | 48.4 |
| Cobalt Payable % | % | 35% | - | - | - | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 | 35 |
| Cobalt Payable | Mlbs | 56 | - | - | - | 0.7 | 1.1 | 1.2 | 1.2 | 1.2 | 1.7 | 1.7 | 1.8 | 1.7 | 1.7 | 1.8 | 1.9 | 1.9 | 2.1 | 1.7 | 1.5 | 1.5 | 1.5 | 1.6 | 1.6 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.6 | 1.7 |
| Concentrate Produced | k dmt | 6,828 | - | - | - | 79 | 129 | 137 | 141 | 145 | 211 | 223 | 227 | 219 | 203 | 204 | 210 | 224 | 250 | 212 | 179 | 181 | 181 | 183 | 189 | 189 | 186 | 187 | 188 | 191 | 193 | 195 | 203 | 222 |
| Concentrate Produced | k wmt | 7,503 | - | - | - | 87 | 142 | 151 | 155 | 159 | 232 | 245 | 249 | 241 | 223 | 224 | 231 | 246 | 275 | 233 | 197 | 199 | 199 | 201 | 208 | 207 | 204 | 205 | 207 | 210 | 212 | 214 | 223 | 244 |
| **Cash Flow** | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
| Nickel Revenue | US$M | 15,850 | - | - | - | 184.4 | 299.6 | 318.2 | 328.2 | 336.9 | 489.9 | 516.9 | 526.6 | 508.2 | 471.3 | 473.3 | 487.2 | 520.4 | 581.1 | 493.2 | 415.6 | 419.9 | 420.2 | 424.0 | 438.8 | 437.7 | 431.4 | 433.3 | 437.2 | 442.6 | 447.1 | 452.3 | 470.8 | 515.3 |
| Cobalt Revenue | US$M | 1,249 | - | - | - | 15.0 | 24.4 | 25.8 | 26.8 | 26.7 | 37.8 | 39.0 | 40.6 | 38.6 | 37.1 | 40.2 | 41.8 | 42.8 | 47.2 | 38.4 | 32.4 | 33.4 | 34.1 | 35.4 | 35.3 | 34.5 | 33.8 | 33.9 | 34.2 | 34.0 | 34.0 | 34.1 | 34.8 | 37.1 |
| **Revenue Total** | **US$M** | **17,099** | **-** | **-** | **-** | **199.4** | **324.0** | **344.0** | **355.0** | **363.6** | **527.8** | **555.9** | **567.2** | **546.7** | **508.4** | **513.5** | **529.0** | **563.2** | **628.4** | **531.6** | **448.0** | **453.3** | **454.3** | **459.4** | **474.1** | **472.2** | **465.2** | **467.2** | **471.3** | **476.6** | **481.1** | **486.4** | **505.6** | **552.3** |
| Treatment & Refining | US$M | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Concentrate Transportation | US$M | (845) | - | - | - | (9.8) | (16.0) | (17.0) | (17.5) | (18.0) | (26.1) | (27.6) | (28.1) | (27.1) | (25.1) | (25.3) | (26.0) | (27.8) | (31.0) | (26.3) | (22.2) | (22.4) | (22.4) | (22.6) | (23.4) | (23.3) | (23.0) | (23.1) | (23.3) | (23.6) | (23.8) | (24.1) | (25.1) | (27.5) |
| **Net Smelter Return (NSR)** | **US$M** | **16,254** | **-** | **-** | **-** | **189.6** | **308.0** | **327.1** | **337.5** | **345.6** | **501.6** | **528.3** | **539.1** | **519.6** | **483.2** | **488.2** | **503.0** | **535.4** | **597.4** | **505.3** | **425.8** | **430.9** | **431.9** | **436.8** | **450.7** | **448.9** | **442.2** | **444.1** | **448.0** | **453.0** | **457.3** | **462.3** | **480.5** | **524.9** |
| Conic Metals Royalty | US$M | (325) | - | - | - | (3.8) | (6.2) | (6.5) | (6.7) | (6.9) | (10.0) | (10.6) | (10.8) | (10.4) | (9.7) | (9.8) | (10.1) | (10.7) | (11.9) | (10.1) | (8.5) | (8.6) | (8.6) | (8.7) | (9.0) | (9.0) | (8.8) | (8.9) | (9.0) | (9.1) | (9.1) | (9.2) | (9.6) | (10.5) |
| **Operating Costs** | **US$M** | **(8,852)** | **-** | **-** | **-** | **(123.9)** | **(148.5)** | **(151.7)** | **(143.8)** | **(167.8)** | **(248.1)** | **(246.0)** | **(240.3)** | **(252.4)** | **(267.6)** | **(264.4)** | **(251.7)** | **(254.0)** | **(280.9)** | **(300.1)** | **(257.5)** | **(259.9)** | **(258.0)** | **(266.9)** | **(277.5)** | **(287.4)** | **(277.8)** | **(245.0)** | **(276.1)** | **(247.1)** | **(279.9)** | **(272.9)** | **(277.4)** | **(256.3)** |
| Mining | US$M | (3,048) | - | - | - | (46.2) | (51.3) | (54.5) | (46.6) | (70.7) | (85.9) | (78.3) | (72.6) | (84.7) | (100.0) | (96.8) | (84.1) | (86.4) | (113.4) | (132.6) | (90.0) | (92.4) | (90.5) | (99.4) | (110.1) | (119.9) | (110.4) | (77.6) | (108.7) | (79.5) | (112.4) | (105.4) | (109.9) | (89.2) |
| Processing & Site Infrastructure | US$M | (4,962) | - | - | - | (59.2) | (78.8) | (78.7) | (78.7) | (78.6) | (138.8) | (144.3) | (144.3) | (144.3) | (144.2) | (144.2) | (144.2) | (144.2) | (144.2) | (144.1) | (144.1) | (144.1) | (144.1) | (144.1) | (144.0) | (144.0) | (144.0) | (144.0) | (143.9) | (144.1) | (144.1) | (144.1) | (144.1) | (143.7) |
| G&A | US$M | (797) | - | - | - | (17.2) | (17.2) | (17.2) | (17.2) | (17.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) |
| Electrical Supply O&M | US$M | (45) | - | - | - | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) |
| **EBITDA** | **US$M** | **7,077** | **-** | **-** | **-** | **61.9** | **153.3** | **168.8** | **187.0** | **170.9** | **243.5** | **271.7** | **288.0** | **256.8** | **205.9** | **214.1** | **241.2** | **270.7** | **304.5** | **195.1** | **159.8** | **162.3** | **165.2** | **161.1** | **164.2** | **152.5** | **155.5** | **190.3** | **163.0** | **196.8** | **168.2** | **180.1** | **193.5** | **258.1** |
| Changes in Net Working Capital | US$M | - | - | - | - | (39.0) | (28.7) | (4.7) | (3.4) | (0.1) | (33.9) | (7.1) | (3.2) | 6.0 | 10.7 | (1.5) | (4.9) | (8.3) | (13.9) | 25.4 | 17.1 | (1.1) | (0.4) | (0.5) | (2.8) | 1.3 | 1.0 | (3.2) | 1.5 | (3.7) | 1.6 | (1.9) | (4.4) | (13.3) |
| Phase 1 Initial Capital Cost | US$M | (1,381) | (138.7) | (502.5) | (739.4) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Phase 2 Expansion Capital Cost | US$M | (532) | - | - | - | - | - | - | (106.6) | (425.7) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Sustaining Capital Cost | US$M | (1,928) | - | - | - | (43.3) | (53.5) | (43.4) | (41.7) | (54.2) | (59.7) | (56.3) | (64.5) | (102.5) | (113.0) | (42.7) | (37.9) | (109.4) | (55.6) | (45.6) | (34.8) | (54.8) | (41.2) | (83.2) | (94.7) | (35.0) | (33.7) | (61.7) | (69.7) | (52.5) | (51.7) | (70.9) | (43.4) | (58.6) |
| Closure & Reclamation | US$M | (72) | - | (4.8) | (4.8) | (5.7) | (5.7) | (5.7) | (5.7) | (5.7) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) |
| Schussler-Hatzl Royalty Buyout | US$M | (3) | - | - | - | (3.1) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Pre-Tax Cash Flow** | **US$M** | **3,161** | **(138.7)** | **(507.3)** | **(744.1)** | **(29.1)** | **65.5** | **115.0** | **29.7** | **(314.9)** | **148.9** | **207.4** | **219.2** | **159.3** | **102.7** | **168.8** | **197.4** | **152.0** | **234.0** | **173.9** | **141.1** | **105.5** | **122.6** | **76.4** | **65.7** | **117.8** | **121.7** | **124.4** | **93.8** | **139.6** | **117.1** | **106.3** | **144.8** | **185.3** |
| BC Mineral Tax | US$M | (372) | - | - | - | (1.3) | (3.2) | (3.5) | (3.9) | (3.6) | (5.1) | (5.6) | (6.0) | (5.3) | (4.3) | (4.5) | (5.0) | (5.6) | (6.3) | (4.1) | (3.4) | (3.4) | (3.5) | (3.4) | (3.5) | (3.2) | (3.3) | (11.1) | (13.2) | (19.8) | (16.2) | (15.3) | (20.6) | (27.2) |
| Corporate Tax | US$M | (779) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | (13.3) | (47.1) | (56.5) | (30.1) | (23.3) | (25.6) | (27.4) | (26.1) | (25.0) | (22.1) | (25.0) | (33.0) | (24.4) | (31.6) | (25.3) | (28.5) | (30.7) | (46.9) |
| **After-Tax Cash Flow** | **US$M** | **2,010** | **(138.7)** | **(507.3)** | **(744.1)** | **(30.4)** | **62.3** | **111.5** | **25.9** | **(318.4)** | **143.8** | **201.7** | **213.3** | **154.0** | **98.3** | **164.3** | **179.0** | **99.3** | **171.2** | **139.7** | **114.4** | **76.4** | **91.7** | **46.9** | **37.3** | **92.5** | **93.4** | **80.3** | **56.2** | **88.2** | **75.6** | **62.5** | **93.4** | **111.2** |

**Table 22.9: PEA Case Cash Flow Summary – Years 30 to 38**

| Item | Units | LOM | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Prices** | | | | | | | | | | | |
| Nickel Price | US$/lb | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 | 7.50 |
| Cobalt Price | US$/lb | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 | 22.30 |
| USD/CAD Exchange Rate | USD/CAD | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 | 0.77 |
| **Physicals** | | | | | | | | | | | |
| Material Mined | Mt | 1,330 | 40.7 | 39.4 | 2.3 | - | - | - | - | - | - |
| Effective Strip Ratio (incl. stockpile) | t:t | 0.40 | 0.24 | 0.28 | 0.00 | - | - | - | - | - | - |
| Ore Processed | Mt | 1,122 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 28.9 | - |
| Nickel Grade | % | 0.221% | 0.266% | 0.230% | 0.201% | 0.199% | 0.199% | 0.199% | 0.199% | 0.199% | - |
| Recovery | % | 49.6% | 50.3% | 43.4% | 43.8% | 43.0% | 43.0% | 43.0% | 43.0% | 43.0% | - |
| Nickel Payable % | % | 78.0% | 78.0% | 78.0% | 78.0% | 78.0% | 78.0% | 78.0% | 78.0% | 78.0% | - |
| Nickel Payable | Mlbs | 2,113 | 75.7 | 56.3 | 49.7 | 48.2 | 48.2 | 48.2 | 48.2 | 42.4 | - |
| Cobalt Grade | % | 0.013% | 0.015% | 0.014% | 0.012% | 0.012% | 0.012% | 0.012% | 0.012% | 0.012% | - |
| Recovery | % | 49.6% | 50.3% | 43.4% | 43.8% | 43.0% | 43.0% | 43.0% | 43.0% | 43.0% | - |
| Cobalt Payable % | % | 35% | 35% | 35% | 35% | 35% | 35% | 35% | 35% | 35% | - |
| Cobalt Payable | Mlbs | 56 | 1.9 | 1.5 | 1.3 | 1.3 | 1.3 | 1.3 | 1.3 | 1.1 | - |
| Concentrate Produced | k dmt | 6,828 | 245 | 182 | 161 | 156 | 156 | 156 | 156 | 137 | - |
| Concentrate Produced | k wmt | 7,503 | 269 | 200 | 176 | 171 | 171 | 171 | 171 | 151 | - |
| **Cash Flow** | | | | | | | | | | | |
| Nickel Revenue | US$M | 15,850 | 567.8 | 422.3 | 372.7 | 361.8 | 361.8 | 361.8 | 361.8 | 318.2 | - |
| Cobalt Revenue | US$M | 1,249 | 41.9 | 33.5 | 29.8 | 28.9 | 28.9 | 28.9 | 28.9 | 25.4 | - |
| **Revenue Total** | **US$M** | **17,099** | **609.7** | **455.8** | **402.5** | **390.7** | **390.7** | **390.7** | **390.7** | **343.6** | **-** |
| Treatment & Refining | US$M | - | - | - | - | - | - | - | - | - | - |
| Concentrate Transportation | US$M | (845) | (30.3) | (22.5) | (19.9) | (19.3) | (19.3) | (19.3) | (19.3) | (17.0) | - |
| **Net Smelter Return (NSR)** | **US$M** | **16,254** | **579.4** | **433.2** | **382.6** | **371.4** | **371.4** | **371.4** | **371.4** | **326.7** | **-** |
| Conic Metals Royalty | US$M | (325) | (11.6) | (8.7) | (7.7) | (7.4) | (7.4) | (7.4) | (7.4) | (6.5) | - |
| **Operating Costs** | **US$M** | **(8,852)** | **(278.2)** | **(264.8)** | **(215.0)** | **(209.4)** | **(205.7)** | **(204.8)** | **(204.8)** | **(188.3)** | - |
| Mining | US$M | (3,048) | (111.1) | (97.8) | (48.0) | (42.4) | (38.8) | (37.9) | (37.9) | (34.8) | - |
| Processing & Site Infrastructure | US$M | (4,962) | (143.6) | (143.6) | (143.6) | (143.6) | (143.6) | (143.5) | (143.5) | (130.1) | - |
| G&A | US$M | (797) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | (22.2) | - |
| Electrical Supply O&M | US$M | (45) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | (1.2) | - |
| **EBITDA** | **US$M** | **7,077** | **289.6** | **159.8** | **160.0** | **154.6** | **158.2** | **159.2** | **159.2** | **131.9** | **-** |
| Changes in Net Working Capital | US$M | - | (12.3) | 36.9 | 9.0 | 2.4 | (0.3) | (0.1) | (0.0) | 10.2 | 69.3 |
| Phase 1 Initial Capital Cost | US$M | (1,381) | - | - | - | - | - | - | - | - | - |
| Phase 2 Expansion Capital Cost | US$M | (532) | - | - | - | - | - | - | - | - | - |
| Sustaining Capital Cost | US$M | (1,928) | (48.7) | (41.0) | (31.5) | (30.5) | (14.6) | (22.5) | (14.4) | (14.4) | (1.8) |
| Closure & Reclamation | US$M | (72) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) | (1.0) |
| Schussler-Hatzl Royalty Buyout | US$M | (3) | - | - | - | - | - | - | - | - | - |
| **Pre-Tax Cash Flow** | **US$M** | **3,161** | **227.6** | **154.5** | **136.5** | **125.5** | **142.4** | **135.6** | **143.8** | **126.7** | **66.4** |
| BC Mineral Tax | US$M | (372) | (32.7) | (16.4) | (17.6) | (17.0) | (19.5) | (18.6) | (19.7) | (16.0) | - |
| Corporate Tax | US$M | (779) | (54.1) | (24.1) | (25.0) | (24.9) | (26.7) | (28.6) | (29.3) | (24.0) | - |
| **After-Tax Cash Flow** | **US$M** | **2,010** | **140.8** | **114.0** | **93.9** | **83.6** | **96.2** | **88.4** | **94.8** | **86.7** | **66.4** |

### 22.4.2 Unit Cash Flow Summary

Table 22.10 summarises the life-of-mine cash flow on a unit basis in terms of US$/t ore milled and US$/lb of payable nickel.

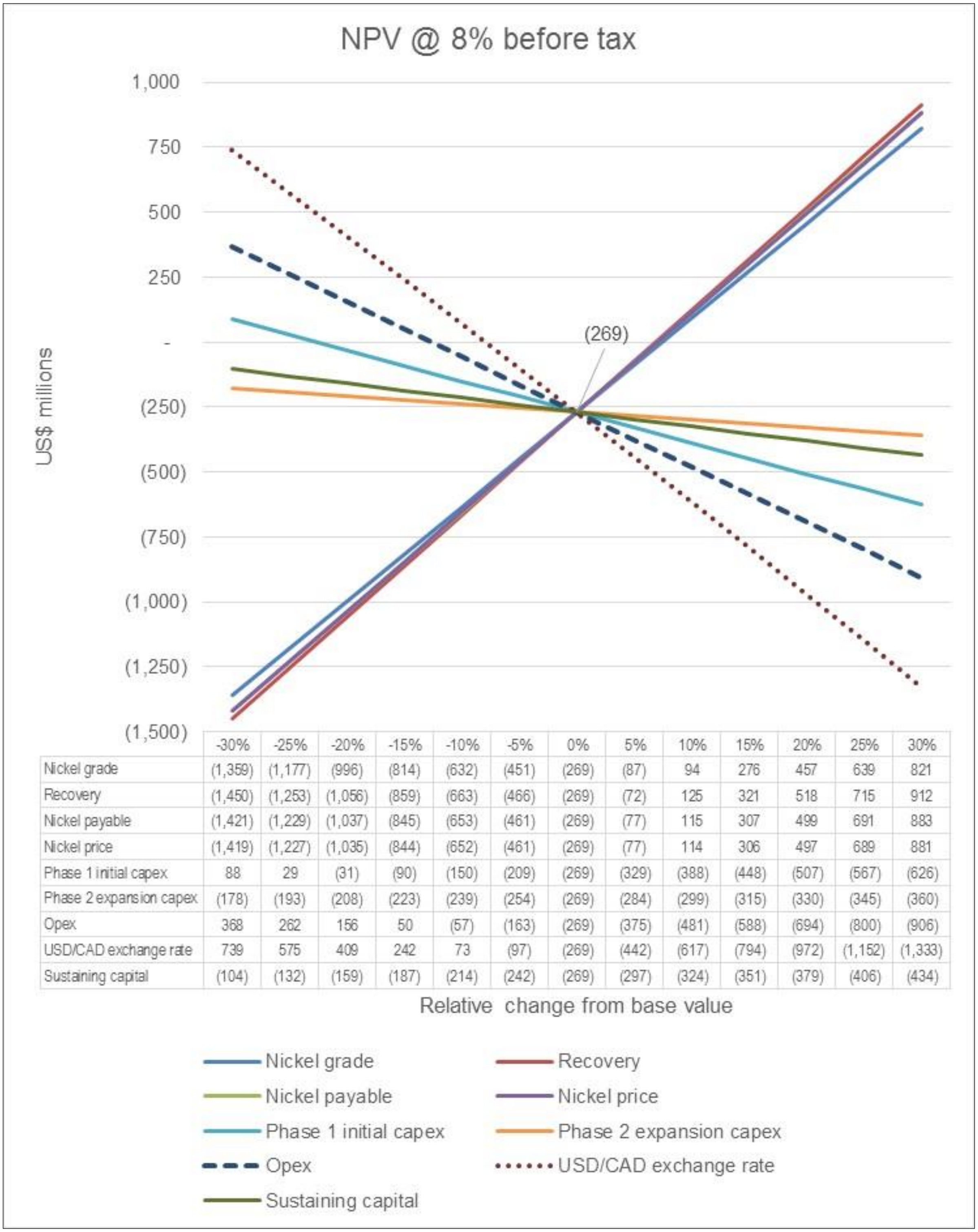
**Table 22.10: Unit Cash Flow Summary**

| Item | US$/t Milled | US$/lb Ni Payable | LOM US$M |
|---|---|---|---|
| Nickel Revenue | 14.13 | 7.50 | 15,850 |
| Cobalt Revenue | 1.11 | 0.59 | 1,249 |
| **Total Revenue** | **15.24** | **8.09** | **17,099** |
| Treatment & Refining | - | - | - |
| Concentrate Transportation | (0.75) | (0.40) | (845) |
| **Net Smelter Return (NSR)** | **14.49** | **7.69** | **16,254** |
| Conic Metals Royalty | (0.29) | (0.15) | (325) |
| Mining Operating Cost | (2.72) | (1.44) | (3,048) |
| Processing & Site Infrastructure Operating Cost | (4.42) | (2.35) | (4,962) |
| Site Level G&A | (0.71) | (0.38) | (797) |
| Electrical Supply O&M | (0.04) | (0.02) | (45) |
| **EBITDA** | **6.31** | **3.35** | **7,077** |
| Phase 1 Initial Capital | (1.23) | (0.65) | (1,381) |
| Phase 2 Expansion Capital | (0.47) | (0.25) | (532) |
| Sustaining Capital | (1.72) | (0.91) | (1,928) |
| Closure & Reclamation Costs | (0.06) | (0.03) | (72) |
| Schussler-Hatzl Royalty Buyout | (0.00) | (0.00) | (3) |
| **Pre-Tax Cash Flow** | **2.82** | **1.50** | **3,161** |
| BC Mineral Tax | (0.33) | (0.18) | (372) |
| Corporate Tax | (0.69) | (0.37) | (779) |
| **After-Tax Cash Flow** | **1.79** | **0.95** | **2,010** |
| *C1 Cost (Excluding Royalty)* | | *4.00* | |

## 22.5 Sensitivity Analysis

A sensitivity analysis was performed to identify key variables with significant impacts on project returns. The nickel price, exchange rate, nickel payable percent, nickel grade, recovery, capital costs, and operating costs were each varied independently on an annual basis and the resulting NPV @ 8% and IRR are shown in Figures 22-3 through 22-6 before and after taxes. The parameter ranges shown do not represent the range limits for each parameter. NPV is most sensitive to recovery, nickel payable percent, nickel price, nickel grade, and exchange rate.

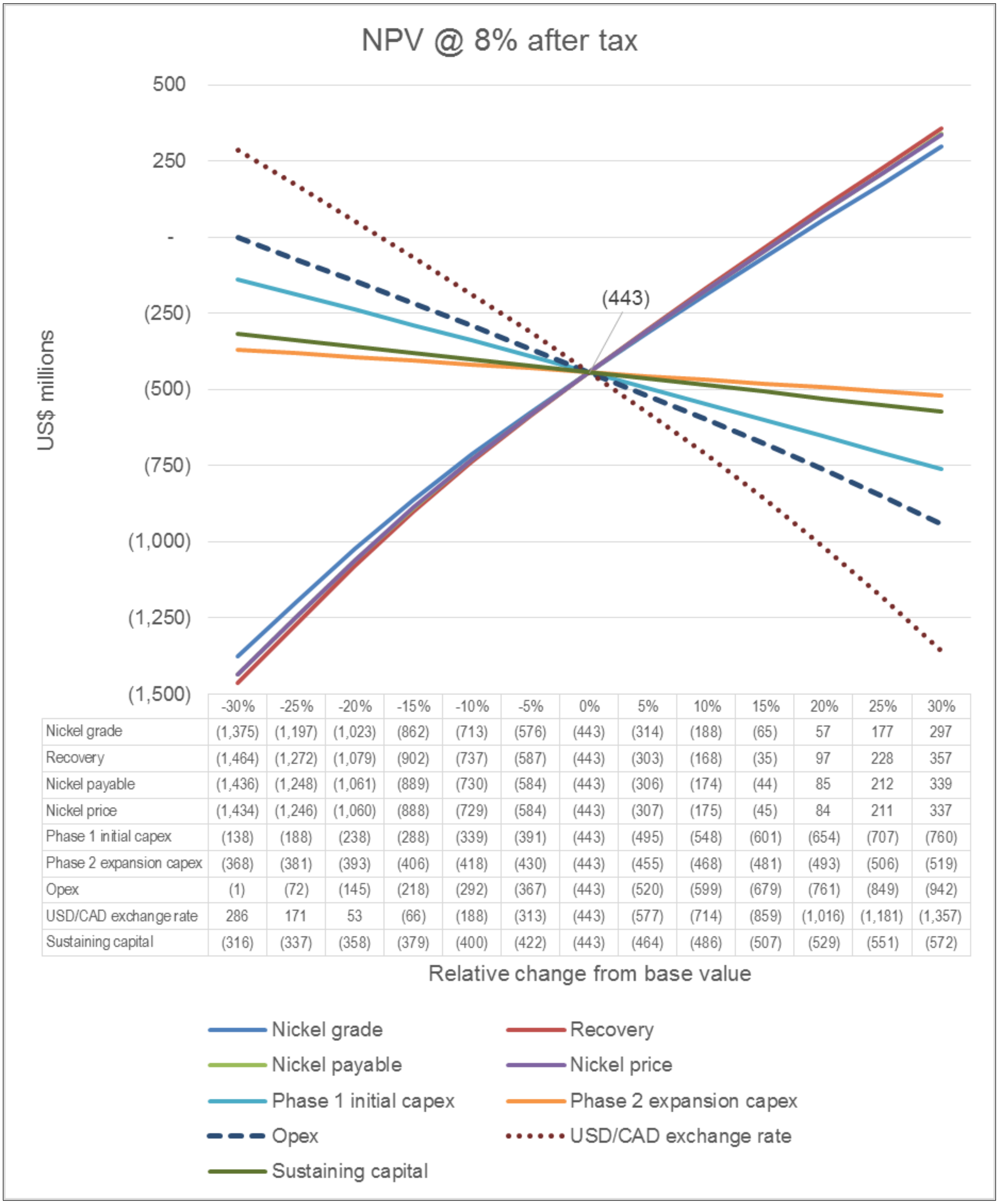
**Figure 22-3: NPV Pre-Tax @ 8% Sensitivity**




| | -30% | -25% | -20% | -15% | -10% | -5% | 0% | 5% | 10% | 15% | 20% | 25% | 30% |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nickel grade | (1,359) | (1,177) | (996) | (814) | (632) | (451) | (269) | (87) | 94 | 276 | 457 | 639 | 821 |
| Recovery | (1,450) | (1,253) | (1,056) | (859) | (663) | (466) | (269) | (72) | 125 | 321 | 518 | 715 | 912 |
| Nickel payable | (1,421) | (1,229) | (1,037) | (845) | (653) | (461) | (269) | (77) | 115 | 307 | 499 | 691 | 883 |
| Nickel price | (1,419) | (1,227) | (1,035) | (844) | (652) | (461) | (269) | (77) | 114 | 306 | 497 | 689 | 881 |
| Phase 1 initial capex | 88 | 29 | (31) | (90) | (150) | (209) | (269) | (329) | (388) | (448) | (507) | (567) | (626) |
| Phase 2 expansion capex | (178) | (193) | (208) | (223) | (239) | (254) | (269) | (284) | (299) | (315) | (330) | (345) | (360) |
| Opex | 368 | 262 | 156 | 50 | (57) | (163) | (269) | (375) | (481) | (588) | (694) | (800) | (906) |
| USD/CAD exchange rate | 739 | 575 | 409 | 242 | 73 | (97) | (269) | (442) | (617) | (794) | (972) | (1,152) | (1,333) |
| Sustaining capital | (104) | (132) | (159) | (187) | (214) | (242) | (269) | (297) | (324) | (351) | (379) | (406) | (434) |

Source: Hatch, 2020.

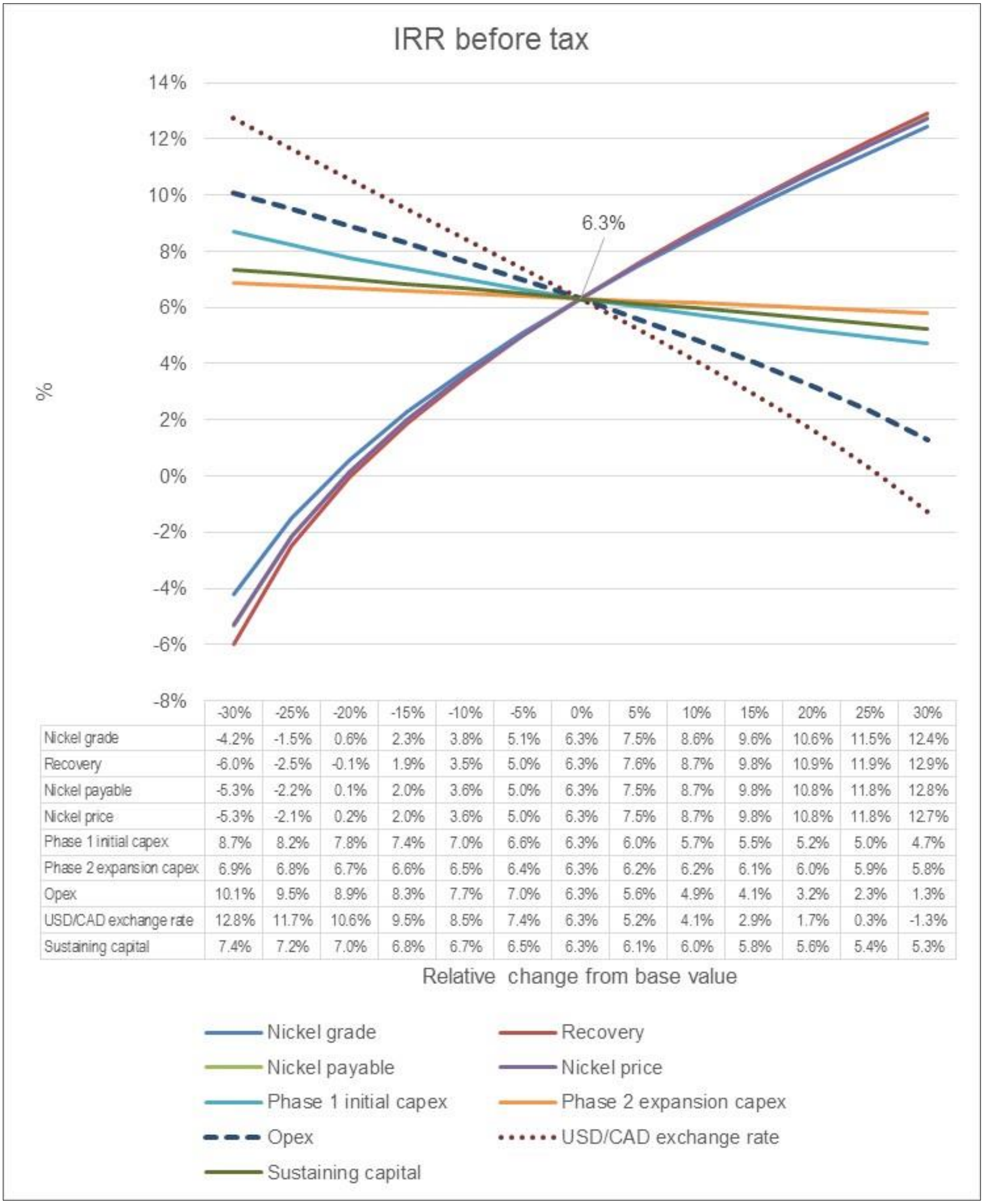
**Figure 22-4: NPV After-Tax @ 8% Sensitivity**




| | -30% | -25% | -20% | -15% | -10% | -5% | 0% | 5% | 10% | 15% | 20% | 25% | 30% |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nickel grade | (1,375) | (1,197) | (1,023) | (862) | (713) | (576) | (443) | (314) | (188) | (65) | 57 | 177 | 297 |
| Recovery | (1,464) | (1,272) | (1,079) | (902) | (737) | (587) | (443) | (303) | (168) | (35) | 97 | 228 | 357 |
| Nickel payable | (1,436) | (1,248) | (1,061) | (889) | (730) | (584) | (443) | (306) | (174) | (44) | 85 | 212 | 339 |
| Nickel price | (1,434) | (1,246) | (1,060) | (888) | (729) | (584) | (443) | (307) | (175) | (45) | 84 | 211 | 337 |
| Phase 1 initial capex | (138) | (188) | (238) | (288) | (339) | (391) | (443) | (495) | (548) | (601) | (654) | (707) | (760) |
| Phase 2 expansion capex | (368) | (381) | (393) | (406) | (418) | (430) | (443) | (455) | (468) | (481) | (493) | (506) | (519) |
| Opex | (1) | (72) | (145) | (218) | (292) | (367) | (443) | (520) | (599) | (679) | (761) | (849) | (942) |
| USD/CAD exchange rate | 286 | 171 | 53 | (66) | (188) | (313) | (443) | (577) | (714) | (859) | (1,016) | (1,181) | (1,357) |
| Sustaining capital | (316) | (337) | (358) | (379) | (400) | (422) | (443) | (464) | (486) | (507) | (529) | (551) | (572) |

Source: Hatch, 2020.

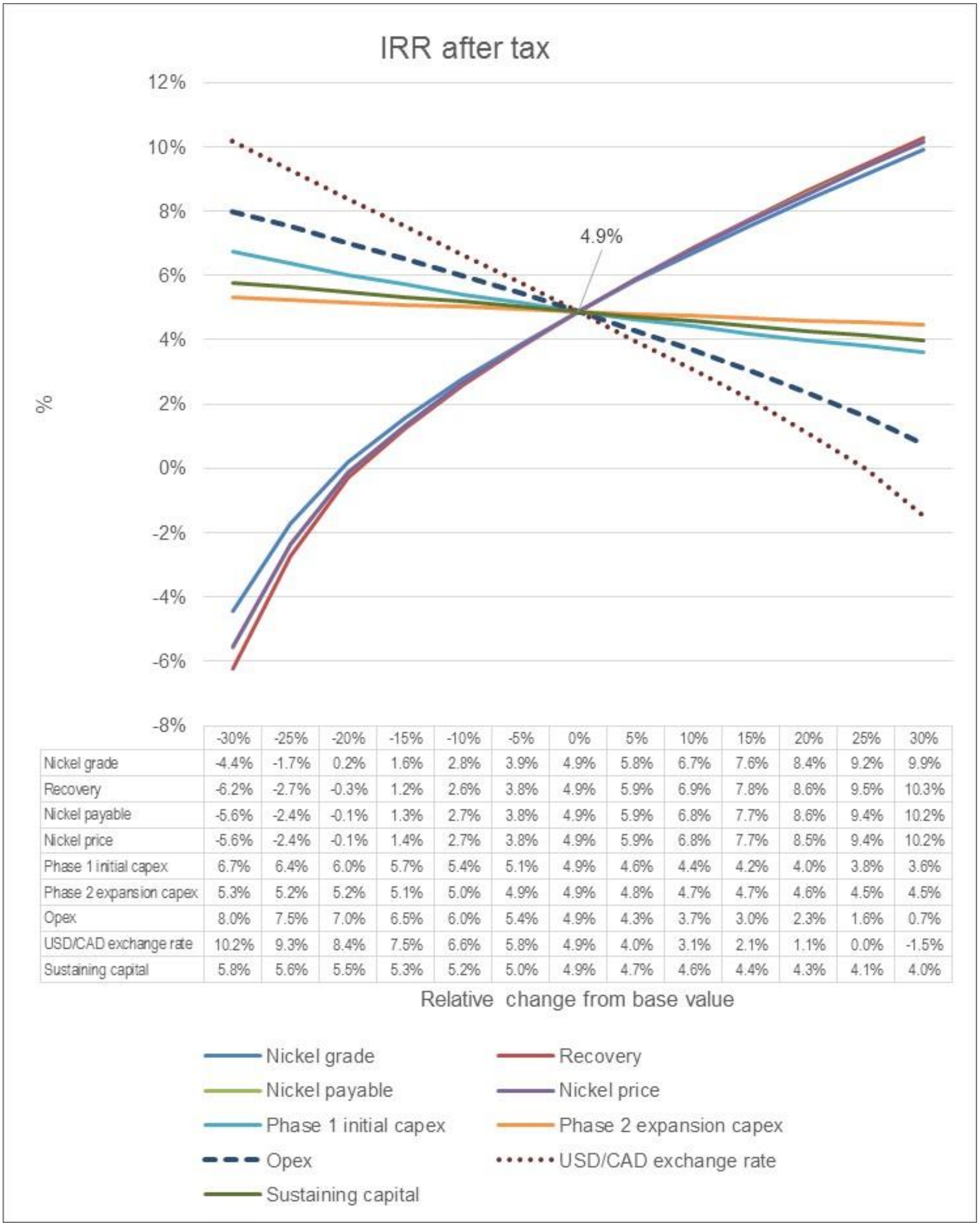
**Figure 22-5: IRR Pre-Tax Sensitivity**




| | -30% | -25% | -20% | -15% | -10% | -5% | 0% | 5% | 10% | 15% | 20% | 25% | 30% |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nickel grade | -4.2% | -1.5% | 0.6% | 2.3% | 3.8% | 5.1% | 6.3% | 7.5% | 8.6% | 9.6% | 10.6% | 11.5% | 12.4% |
| Recovery | -6.0% | -2.5% | -0.1% | 1.9% | 3.5% | 5.0% | 6.3% | 7.6% | 8.7% | 9.8% | 10.9% | 11.9% | 12.9% |
| Nickel payable | -5.3% | -2.2% | 0.1% | 2.0% | 3.6% | 5.0% | 6.3% | 7.5% | 8.7% | 9.8% | 10.8% | 11.8% | 12.8% |
| Nickel price | -5.3% | -2.1% | 0.2% | 2.0% | 3.6% | 5.0% | 6.3% | 7.5% | 8.7% | 9.8% | 10.8% | 11.8% | 12.7% |
| Phase 1 initial capex | 8.7% | 8.2% | 7.8% | 7.4% | 7.0% | 6.6% | 6.3% | 6.0% | 5.7% | 5.5% | 5.2% | 5.0% | 4.7% |
| Phase 2 expansion capex | 6.9% | 6.8% | 6.7% | 6.6% | 6.5% | 6.4% | 6.3% | 6.2% | 6.2% | 6.1% | 6.0% | 5.9% | 5.8% |
| Opex | 10.1% | 9.5% | 8.9% | 8.3% | 7.7% | 7.0% | 6.3% | 5.6% | 4.9% | 4.1% | 3.2% | 2.3% | 1.3% |
| USD/CAD exchange rate | 12.8% | 11.7% | 10.6% | 9.5% | 8.5% | 7.4% | 6.3% | 5.2% | 4.1% | 2.9% | 1.7% | 0.3% | -1.3% |
| Sustaining capital | 7.4% | 7.2% | 7.0% | 6.8% | 6.7% | 6.5% | 6.3% | 6.1% | 6.0% | 5.8% | 5.6% | 5.4% | 5.3% |

Source: Hatch, 2020.

**Figure 22-6: IRR After-Tax Sensitivity**

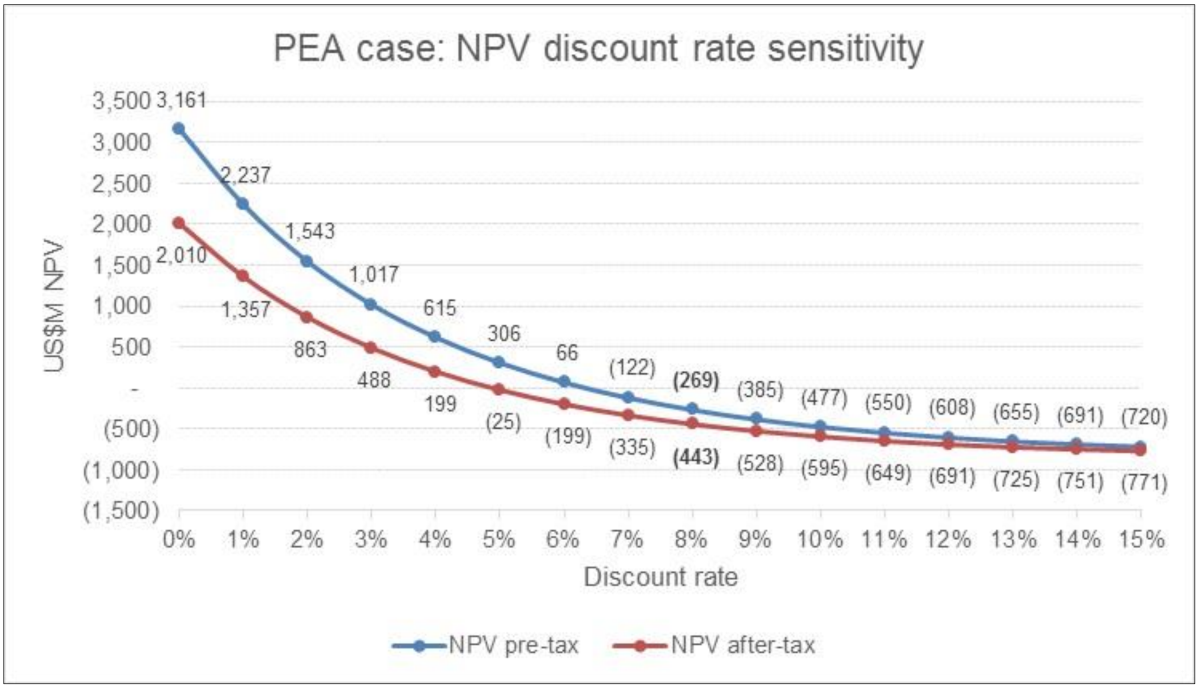



| | -30% | -25% | -20% | -15% | -10% | -5% | 0% | 5% | 10% | 15% | 20% | 25% | 30% |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Nickel grade | -4.4% | -1.7% | 0.2% | 1.6% | 2.8% | 3.9% | 4.9% | 5.8% | 6.7% | 7.6% | 8.4% | 9.2% | 9.9% |
| Recovery | -6.2% | -2.7% | -0.3% | 1.2% | 2.6% | 3.8% | 4.9% | 5.9% | 6.9% | 7.8% | 8.6% | 9.5% | 10.3% |
| Nickel payable | -5.6% | -2.4% | -0.1% | 1.3% | 2.7% | 3.8% | 4.9% | 5.9% | 6.8% | 7.7% | 8.6% | 9.4% | 10.2% |
| Nickel price | -5.6% | -2.4% | -0.1% | 1.4% | 2.7% | 3.8% | 4.9% | 5.9% | 6.8% | 7.7% | 8.5% | 9.4% | 10.2% |
| Phase 1 initial capex | 6.7% | 6.4% | 6.0% | 5.7% | 5.4% | 5.1% | 4.9% | 4.6% | 4.4% | 4.2% | 4.0% | 3.8% | 3.6% |
| Phase 2 expansion capex | 5.3% | 5.2% | 5.2% | 5.1% | 5.0% | 4.9% | 4.9% | 4.8% | 4.7% | 4.7% | 4.6% | 4.5% | 4.5% |
| Opex | 8.0% | 7.5% | 7.0% | 6.5% | 6.0% | 5.4% | 4.9% | 4.3% | 3.7% | 3.0% | 2.3% | 1.6% | 0.7% |
| USD/CAD exchange rate | 10.2% | 9.3% | 8.4% | 7.5% | 6.6% | 5.8% | 4.9% | 4.0% | 3.1% | 2.1% | 1.1% | 0.0% | -1.5% |
| Sustaining capital | 5.8% | 5.6% | 5.5% | 5.3% | 5.2% | 5.0% | 4.9% | 4.7% | 4.6% | 4.4% | 4.3% | 4.1% | 4.0% |

Source: Hatch, 2020.

The NPV sensitivity to discount rate is shown in Figure 22-7.

**Figure 22-7: NPV Discount Rate Sensitivity**




Source: Hatch, 2020.

Tables 22.11 and 22.12 summarise the sensitivity of NPV @ 8% before and after taxes to changes nickel prices and USD/CAD exchange rate. The green shaded cell is at current PEA case pricing, while the blue shaded cell is at Wood Mackenzie ESG incentive pricing.

Table 22.13 and Table 22.14 show the same sensitivity tables as above in terms of pre-tax and after-tax IRR.

Table 22.15 and Table 22.16 summarise the sensitivity of NPV @ 8% before and after taxes to changes in nickel price and nickel payables. The market study recommends a downside nickel payable sensitivity from 68% to 75%. The green shaded cells are current PEA case pricing. The blue shaded cells are at Wood Mackenzie ESG incentive pricing.

Table 22.17 and Table 22.18 show the same sensitivity tables as above in terms of pre-tax and after-tax IRR.

**Table 22.11: NPV Pre-Tax @ 8% Sensitivity to Nickel Price & USD/CAD Exchange Rate**

| NPV Pre-Tax (US$M) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| USD/CAD Exchange Rate | 0.70 | (727) | (472) | (216) | 39 | 295 | 550 | 806 | 1,061 | 1,317 | 1,572 | 1,828 |
| | 0.71 | (771) | (516) | (260) | (5) | 251 | 506 | 762 | 1,017 | 1,273 | 1,528 | 1,784 |
| | 0.72 | (816) | (560) | (305) | (49) | 206 | 462 | 717 | 973 | 1,228 | 1,484 | 1,739 |
| | 0.73 | (860) | (605) | (349) | (94) | 162 | 417 | 673 | 928 | 1,184 | 1,439 | 1,695 |
| | 0.74 | (905) | (649) | (394) | (138) | 117 | 373 | 628 | 884 | 1,139 | 1,395 | 1,650 |
| | 0.75 | (949) | (694) | (438) | (183) | 73 | 328 | 584 | 839 | 1,095 | 1,350 | 1,606 |
| | 0.76 | (994) | (739) | (483) | (228) | 28 | 283 | 539 | 794 | 1,050 | 1,305 | 1,561 |
| | 0.77* | (1,035) | (780) | (525) | (269) | (14) | 242 | 497 | 753 | 1,008 | 1,264 | 1,519 |
| | 0.78 | (1,084) | (828) | (573) | (317) | (62) | 194 | 449 | 705 | 960 | 1,216 | 1,471 |
| | 0.79 | (1,129) | (873) | (618) | (362) | (107) | 148 | 404 | 659 | 915 | 1,170 | 1,426 |
| | 0.80 | (1,174) | (919) | (663) | (408) | (152) | 103 | 359 | 614 | 870 | 1,125 | 1,381 |

Notes: *Inverse of 1.30 CAD/USD applied. Colour legend: PEA case price, ESG incentive price.

**Table 22.12: NPV After-Tax @ 8% Sensitivity to Nickel Price & USD/CAD Exchange Rate**

| NPV After-Tax (US$M) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| USD/CAD Exchange Rate | 0.70 | (780) | (580) | (392) | (213) | (39) | 132 | 300 | 468 | 634 | 799 | 963 |
| | 0.71 | (819) | (615) | (426) | (245) | (71) | 100 | 269 | 437 | 604 | 769 | 933 |
| | 0.72 | (858) | (650) | (460) | (278) | (103) | 69 | 238 | 407 | 573 | 739 | 903 |
| | 0.73 | (898) | (686) | (494) | (311) | (135) | 37 | 207 | 376 | 542 | 708 | 873 |
| | 0.74 | (939) | (722) | (529) | (344) | (167) | 5 | 176 | 345 | 512 | 678 | 843 |
| | 0.75 | (980) | (759) | (564) | (378) | (199) | (26) | 145 | 313 | 481 | 647 | 812 |
| | 0.76 | (1,021) | (797) | (599) | (411) | (232) | (58) | 113 | 282 | 450 | 616 | 782 |
| | 0.77* | (1,060) | (833) | (631) | (443) | (262) | (88) | 84 | 253 | 421 | 588 | 753 |
| | 0.78 | (1,106) | (876) | (670) | (480) | (298) | (123) | 49 | 219 | 388 | 554 | 720 |
| | 0.79 | (1,151) | (916) | (706) | (515) | (331) | (155) | 17 | 187 | 356 | 523 | 689 |
| | 0.80 | (1,195) | (957) | (742) | (550) | (365) | (188) | (15) | 156 | 324 | 492 | 658 |

Notes: *Inverse of 1.30 CAD/USD applied. Colour legend: PEA case price, ESG incentive price.

**Table 22.13: IRR Pre-Tax Sensitivity to Nickel Price & USD/CAD Exchange Rate**

| IRR Pre-Tax (%) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| USD/CAD Exchange Rate | 0.70 | 2.7% | 4.8% | 6.6% | 8.2% | 9.8% | 11.2% | 12.6% | 13.9% | 15.1% | 16.3% | 17.5% |
| | 0.71 | 2.4% | 4.5% | 6.3% | 8.0% | 9.5% | 10.9% | 12.3% | 13.6% | 14.9% | 16.1% | 17.3% |
| | 0.72 | 2.0% | 4.2% | 6.0% | 7.7% | 9.2% | 10.7% | 12.1% | 13.4% | 14.6% | 15.8% | 17.0% |
| | 0.73 | 1.7% | 3.8% | 5.7% | 7.4% | 9.0% | 10.4% | 11.8% | 13.1% | 14.4% | 15.6% | 16.7% |
| | 0.74 | 1.3% | 3.5% | 5.4% | 7.1% | 8.7% | 10.2% | 11.5% | 12.8% | 14.1% | 15.3% | 16.5% |
| | 0.75 | 0.9% | 3.2% | 5.1% | 6.9% | 8.4% | 9.9% | 11.3% | 12.6% | 13.9% | 15.1% | 16.2% |
| | 0.76 | 0.5% | 2.9% | 4.8% | 6.6% | 8.2% | 9.6% | 11.0% | 12.3% | 13.6% | 14.8% | 16.0% |
| | 0.77* | 0.2% | 2.6% | 4.6% | 6.3% | 7.9% | 9.4% | 10.8% | 12.1% | 13.4% | 14.6% | 15.7% |
| | 0.78 | -0.3% | 2.2% | 4.2% | 6.0% | 7.6% | 9.1% | 10.5% | 11.8% | 13.1% | 14.3% | 15.5% |
| | 0.79 | -0.7% | 1.9% | 3.9% | 5.7% | 7.4% | 8.9% | 10.3% | 11.6% | 12.8% | 14.1% | 15.2% |
| | 0.80 | -1.2% | 1.5% | 3.6% | 5.5% | 7.1% | 8.6% | 10.0% | 11.3% | 12.6% | 13.8% | 15.0% |

Notes: *Inverse of 1.30 CAD/USD applied. Colour legend: PEA case price, ESG incentive price.

**Table 22.14: IRR After-Tax Sensitivity to Nickel Price & USD/CAD Exchange Rate**

| IRR After-Tax (%) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| USD/CAD Exchange Rate | 0.70 | 1.9% | 3.6% | 5.1% | 6.5% | 7.7% | 8.9% | 10.0% | 11.1% | 12.1% | 13.1% | 14.0% |
| | 0.71 | 1.7% | 3.4% | 4.9% | 6.2% | 7.5% | 8.7% | 9.8% | 10.9% | 11.9% | 12.9% | 13.8% |
| | 0.72 | 1.4% | 3.1% | 4.6% | 6.0% | 7.3% | 8.5% | 9.6% | 10.7% | 11.7% | 12.7% | 13.6% |
| | 0.73 | 1.1% | 2.9% | 4.4% | 5.8% | 7.1% | 8.3% | 9.4% | 10.5% | 11.5% | 12.5% | 13.4% |
| | 0.74 | 0.8% | 2.6% | 4.1% | 5.6% | 6.8% | 8.0% | 9.2% | 10.2% | 11.3% | 12.3% | 13.2% |
| | 0.75 | 0.5% | 2.3% | 3.9% | 5.3% | 6.6% | 7.8% | 9.0% | 10.0% | 11.1% | 12.0% | 13.0% |
| | 0.76 | 0.2% | 2.1% | 3.7% | 5.1% | 6.4% | 7.6% | 8.7% | 9.8% | 10.9% | 11.8% | 12.8% |
| | 0.77* | -0.1% | 1.8% | 3.4% | 4.9% | 6.2% | 7.4% | 8.5% | 9.6% | 10.7% | 11.6% | 12.6% |
| | 0.78 | -0.5% | 1.5% | 3.2% | 4.6% | 6.0% | 7.2% | 8.3% | 9.4% | 10.4% | 11.4% | 12.4% |
| | 0.79 | -0.9% | 1.2% | 2.9% | 4.4% | 5.7% | 7.0% | 8.1% | 9.2% | 10.2% | 11.2% | 12.2% |
| | 0.80 | -1.4% | 0.9% | 2.7% | 4.2% | 5.5% | 6.7% | 7.9% | 9.0% | 10.0% | 11.0% | 12.0% |

Notes: *Inverse of 1.30 CAD/USD applied. Colour legend: PEA case price, ESG incentive price.

**Table 22.15: NPV Pre-Tax @ 8% Sensitivity to Nickel Price & Nickel Payable %**

| NPV Pre-Tax (US$M) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| Ni Payable % | 65% | (1,547) | (1,335) | (1,122) | (909) | (696) | (483) | (271) | (58) | 155 | 368 | 581 |
| | 66% | (1,508) | (1,292) | (1,076) | (860) | (644) | (428) | (211) | 5 | 221 | 437 | 653 |
| | 67% | (1,469) | (1,249) | (1,030) | (811) | (591) | (372) | (152) | 67 | 286 | 506 | 725 |
| | 68% | (1,429) | (1,207) | (984) | (761) | (539) | (316) | (93) | 129 | 352 | 575 | 797 |
| | 69% | (1,390) | (1,164) | (938) | (712) | (486) | (260) | (34) | 192 | 418 | 644 | 870 |
| | 70% | (1,351) | (1,121) | (892) | (663) | (434) | (204) | 25 | 254 | 483 | 713 | 942 |
| | 71% | (1,311) | (1,079) | (846) | (614) | (381) | (149) | 84 | 316 | 549 | 781 | 1,014 |
| | 72% | (1,272) | (1,036) | (800) | (564) | (329) | (93) | 143 | 379 | 615 | 850 | 1,086 |
| | 73% | (1,232) | (993) | (754) | (515) | (276) | (37) | 202 | 441 | 680 | 919 | 1,158 |
| | 74% | (1,193) | (951) | (708) | (466) | (224) | 19 | 261 | 504 | 746 | 988 | 1,231 |
| | 75% | (1,154) | (908) | (662) | (417) | (171) | 75 | 320 | 566 | 812 | 1,057 | 1,303 |
| | 76% | (1,114) | (865) | (616) | (367) | (119) | 130 | 379 | 628 | 877 | 1,126 | 1,375 |
| | 77% | (1,075) | (823) | (570) | (318) | (66) | 186 | 438 | 691 | 943 | 1,195 | 1,447 |
| | 78% | (1,035) | (780) | (525) | (269) | (14) | 242 | 497 | 753 | 1,008 | 1,264 | 1,519 |
| | 79% | (996) | (737) | (479) | (220) | 39 | 298 | 557 | 815 | 1,074 | 1,333 | 1,592 |
| | 80% | (957) | (695) | (433) | (171) | 91 | 354 | 616 | 878 | 1,140 | 1,402 | 1,664 |

Note: Colour legend: PEA case price, ESG incentive price.

**Table 22.16: NPV After-Tax @ 8% Sensitivity to Nickel Price & Nickel Payable %**

| NPV After-Tax (US$M) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| Ni Payable % | 65% | (1,560) | (1,352) | (1,143) | (945) | (764) | (601) | (444) | (293) | (147) | (3) | 139 |
| | 66% | (1,522) | (1,310) | (1,098) | (902) | (722) | (559) | (402) | (250) | (102) | 43 | 187 |
| | 67% | (1,483) | (1,268) | (1,055) | (859) | (682) | (518) | (359) | (207) | (58) | 89 | 235 |
| | 68% | (1,444) | (1,226) | (1,013) | (817) | (642) | (477) | (318) | (164) | (14) | 135 | 282 |
| | 69% | (1,406) | (1,185) | (971) | (777) | (603) | (437) | (277) | (122) | 30 | 181 | 330 |
| | 70% | (1,367) | (1,143) | (930) | (737) | (563) | (396) | (236) | (80) | 74 | 226 | 377 |
| | 71% | (1,329) | (1,101) | (890) | (699) | (525) | (357) | (195) | (38) | 118 | 272 | 425 |
| | 72% | (1,290) | (1,060) | (850) | (661) | (486) | (317) | (155) | 4 | 162 | 317 | 472 |
| | 73% | (1,252) | (1,021) | (811) | (624) | (448) | (279) | (115) | 46 | 205 | 363 | 519 |
| | 74% | (1,213) | (983) | (774) | (588) | (410) | (240) | (75) | 88 | 248 | 408 | 566 |
| | 75% | (1,174) | (945) | (737) | (551) | (373) | (202) | (35) | 129 | 292 | 453 | 613 |
| | 76% | (1,136) | (907) | (701) | (515) | (336) | (164) | 4 | 171 | 335 | 498 | 660 |
| | 77% | (1,097) | (870) | (666) | (479) | (299) | (126) | 44 | 212 | 378 | 543 | 707 |
| | 78% | (1,060) | (833) | (631) | (443) | (262) | (88) | 84 | 253 | 421 | 588 | 753 |
| | 79% | (1,024) | (797) | (597) | (408) | (226) | (50) | 123 | 294 | 464 | 633 | 800 |
| | 80% | (988) | (763) | (563) | (372) | (190) | (13) | 162 | 335 | 507 | 677 | 846 |

Note: Colour legend: PEA case price, ESG incentive price.

**Table 22.17: IRR Pre-Tax Sensitivity to Nickel Price & Nickel Payable %**

| IRR Pre-Tax (%) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ni Payable % | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| | 65% | -8.5% | -3.8% | -0.8% | 1.4% | 3.3% | 4.9% | 6.3% | 7.7% | 8.9% | 10.1% | 11.2% |
| | 66% | -7.4% | -3.1% | -0.3% | 1.9% | 3.7% | 5.3% | 6.7% | 8.0% | 9.3% | 10.5% | 11.6% |
| | 67% | -6.4% | -2.4% | 0.2% | 2.3% | 4.1% | 5.6% | 7.1% | 8.4% | 9.6% | 10.8% | 12.0% |
| | 68% | -5.5% | -1.9% | 0.7% | 2.7% | 4.5% | 6.0% | 7.4% | 8.8% | 10.0% | 11.2% | 12.3% |
| | 69% | -4.7% | -1.3% | 1.1% | 3.1% | 4.8% | 6.4% | 7.8% | 9.1% | 10.4% | 11.5% | 12.7% |
| | 70% | -4.0% | -0.8% | 1.6% | 3.5% | 5.2% | 6.7% | 8.1% | 9.5% | 10.7% | 11.9% | 13.0% |
| | 71% | -3.4% | -0.3% | 2.0% | 3.9% | 5.6% | 7.1% | 8.5% | 9.8% | 11.1% | 12.2% | 13.4% |
| | 72% | -2.8% | 0.2% | 2.4% | 4.3% | 5.9% | 7.4% | 8.8% | 10.2% | 11.4% | 12.6% | 13.7% |
| | 73% | -2.2% | 0.6% | 2.8% | 4.6% | 6.3% | 7.8% | 9.2% | 10.5% | 11.7% | 12.9% | 14.1% |
| | 74% | -1.7% | 1.0% | 3.2% | 5.0% | 6.6% | 8.1% | 9.5% | 10.8% | 12.1% | 13.3% | 14.4% |
| | 75% | -1.2% | 1.4% | 3.5% | 5.3% | 7.0% | 8.4% | 9.8% | 11.1% | 12.4% | 13.6% | 14.8% |
| | 76% | -0.7% | 1.8% | 3.9% | 5.7% | 7.3% | 8.8% | 10.2% | 11.5% | 12.7% | 13.9% | 15.1% |
| | 77% | -0.3% | 2.2% | 4.2% | 6.0% | 7.6% | 9.1% | 10.5% | 11.8% | 13.0% | 14.3% | 15.4% |
| | 78% | 0.2% | 2.6% | 4.6% | 6.3% | 7.9% | 9.4% | 10.8% | 12.1% | 13.4% | 14.6% | 15.7% |
| | 79% | 0.6% | 2.9% | 4.9% | 6.6% | 8.2% | 9.7% | 11.1% | 12.4% | 13.7% | 14.9% | 16.1% |
| | 80% | 1.0% | 3.3% | 5.2% | 7.0% | 8.5% | 10.0% | 11.4% | 12.7% | 14.0% | 15.2% | 16.4% |

Note: Colour legend: PEA case price, ESG incentive price.

**Table 22.18: IRR After-Tax Sensitivity to Nickel Price & Nickel Payable %**

| IRR After-Tax (%) | | Nickel Price - Flat (US$/lb) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ni Payable % | | 6.00 | 6.50 | 7.00 | 7.50 | 8.00 | 8.50 | 9.00 | 9.50 | 10.00 | 10.50 | 11.00 |
| | 65% | -8.8% | -4.0% | -1.0% | 0.9% | 2.4% | 3.7% | 4.9% | 6.0% | 7.0% | 8.0% | 8.9% |
| | 66% | -7.7% | -3.3% | -0.5% | 1.2% | 2.7% | 4.0% | 5.2% | 6.3% | 7.3% | 8.3% | 9.2% |
| | 67% | -6.7% | -2.7% | -0.1% | 1.6% | 3.0% | 4.3% | 5.5% | 6.6% | 7.6% | 8.6% | 9.5% |
| | 68% | -5.8% | -2.1% | 0.3% | 2.0% | 3.4% | 4.6% | 5.8% | 6.9% | 7.9% | 8.9% | 9.8% |
| | 69% | -5.0% | -1.5% | 0.6% | 2.3% | 3.7% | 4.9% | 6.1% | 7.2% | 8.2% | 9.2% | 10.1% |
| | 70% | -4.3% | -1.0% | 1.0% | 2.6% | 4.0% | 5.2% | 6.4% | 7.5% | 8.5% | 9.5% | 10.4% |
| | 71% | -3.6% | -0.5% | 1.3% | 2.9% | 4.3% | 5.5% | 6.7% | 7.7% | 8.8% | 9.7% | 10.7% |
| | 72% | -3.0% | -0.1% | 1.7% | 3.2% | 4.6% | 5.8% | 6.9% | 8.0% | 9.1% | 10.0% | 11.0% |
| | 73% | -2.4% | 0.2% | 2.0% | 3.5% | 4.8% | 6.1% | 7.2% | 8.3% | 9.3% | 10.3% | 11.2% |
| | 74% | -1.9% | 0.5% | 2.3% | 3.8% | 5.1% | 6.4% | 7.5% | 8.6% | 9.6% | 10.6% | 11.5% |
| | 75% | -1.4% | 0.9% | 2.6% | 4.1% | 5.4% | 6.6% | 7.8% | 8.8% | 9.9% | 10.9% | 11.8% |
| | 76% | -0.9% | 1.2% | 2.9% | 4.3% | 5.7% | 6.9% | 8.0% | 9.1% | 10.1% | 11.1% | 12.1% |
| | 77% | -0.5% | 1.5% | 3.2% | 4.6% | 5.9% | 7.2% | 8.3% | 9.4% | 10.4% | 11.4% | 12.3% |
| | 78% | -0.1% | 1.8% | 3.4% | 4.9% | 6.2% | 7.4% | 8.5% | 9.6% | 10.7% | 11.6% | 12.6% |
| | 79% | 0.2% | 2.1% | 3.7% | 5.1% | 6.5% | 7.7% | 8.8% | 9.9% | 10.9% | 11.9% | 12.9% |
| | 80% | 0.5% | 2.4% | 4.0% | 5.4% | 6.7% | 7.9% | 9.1% | 10.1% | 11.2% | 12.2% | 13.1% |

Note: Colour legend: PEA case price, ESG incentive price.

# 23.0 ADJACENT PROPERTIES

There are two adjacent properties of note, the Kutcho and Eaglehead properties. These are described briefly below.

## 23.1 Kutcho Copper-Zinc Deposit

The Kutcho copper-zinc deposit lies approximately 40 km southeast of the Turnagain deposit. Kutcho Copper Corporation (Kutcho) owns 100% of the project as of October 2020.

A pre-feasibility study was completed in 2017, and Kutcho is currently conducting a feasibility study.

Kutcho recently received a Section 11 order, allowing it to begin the environmental assessment process in British Columbia. If both the Kutcho and Turnagain facilities are constructed, approximately 60 to 70 km of road access will be shared, depending on the final route selected for each company.

## 23.2 Eaglehead Gold-Copper Deposit

The Eaglehead gold-copper deposit lies approximately 14 km west of the Turnagain deposit, and was the subject of a 2012 Technical Report.

District Copper owns a 100% interest in this porphyry Cu-Mo-Au project as of December 2019. Giga Metals and District Copper have discussed potential areas of project coordination, including sharing infrastructure assets. Giga Metals and District Copper have executed a letter of agreement allowing Giga Metals to conduct minor physical works on District Copper’s claims to support Turnagain project development. Should Eaglehead and Turnagain both become commercial projects, approximately 70 km of road access will be in common (depending on the final route selected for each company).

There are a number of jade and placer mining operations in the area. These facilities are not expected to provide any significant benefit or risk to the Turnagain project.

# 24.0 OTHER RELEVANT DATA & INFORMATION

The completion of additional technical programs, engineering studies, and environmental studies is expected to take a minimum of 3.0 years. Following successful financing, project construction is expected to take an additional 2.5 to 3.0 years, depending on leading activities.

A preliminary project execution plan will be prepared in the next study phase.

# 25.0 INTERPRETATION & CONCLUSIONS

Based on the outcomes of this PEA, the contributors have drawn the following key conclusions.

## 25.1 Geology & Mineral Resources

The mineral resources were estimated in conformity with CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (December, 2019) and are reported in accordance with NI 43-101 guidelines.

The 362 drill holes in the database were supplied in electronic format by Giga Metals, 307 of which had assay values. The primary economic contributor is shown to be nickel (Ni%) content, and the secondary is cobalt (Co%). Sulphur (S%) has similarly been analysed and estimated on a block-by-block basis. Assay values were composited to 4.0 m within the mineralised domains: (1) Du-Wh-Sp (dunite, wehrlite, serpentinite); (2) cPx-oPx (clinopyroxenite, olivine, magnetite and hornblende clinopyroxenite); (3) volcanics; (4) dykes; (5) overburden.

The mineralised domains were re-interpreted and refined including the current drilling data. The mineral resource estimate is based on an additional 36 infill drill holes totalling 8,940 metres drilled in 2018 in the areas of the conceptual open pit in addition to updated geological modelling.

Mineral resources are classified under the categories of *measured*, *indicated* and *inferred* according to CIM guidelines. The author evaluated the resource in order to ensure that it meets the condition of "reasonable prospects of eventual economic extraction" as suggested under NI 43-101. The criteria considered were confidence, continuity and economic cut-off in addition to considering constringed the resources within an optimised pit shell based on.

Using a cut-off grade of 0.1% Ni, the Turnagain property contains an estimated 1,073 Mt of measured and indicated resources at 0.220% Ni and 0.013% Co. An additional 1,142 Mt grading 0.217% Ni and 0.013% Co is classified as inferred. The resource estimate is presented in Table 25.1.

**Table 25.1: Mineral Resource Estimate**

| Resource Category | Kilotonnes | % Ni (T) | % Co (T) |
|---|---|---|---|
| Measured | 360,913 | 0.230 | 0.014 |
| Indicated | 712,406 | 0.215 | 0.013 |
| Measured & Indicated | 1,073,319 | 0.220 | 0.013 |
| Inferred | 1,142,101 | 0.217 | 0.013 |

Notes: **(1)** All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI 43-101"). **(2)** Mineral resources are reported in relation to a conceptual pit shell in order to demonstrate reasonable expectation of eventual economic extraction, as required under NI 43-101; mineralisation lying outside of these pit shells is not reported as a mineral resource. **Mineral resources are not mineral reserves & do not have demonstrated economic viability.** **(3)** Open pit mineral resources are reported at a cut-off grade of 0.1% Ni. Cut-off grades are based on a price of US $7.50 per pound, nickel recoveries of 60%, ore and waste mining costs of $2.80, along with milling, processing and G&A costs of $7.20. **(4)** Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as mineral reserves. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated. **(5)** Due to rounding, numbers presented may not add up precisely to the totals provided and percentages my not precisely reflect absolute figures.

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 resource. It is recommended that the project be advanced to the pre-feasibility study stage.

## 25.2 Mining

The Turnagain deposit will be mined using conventional open pit mining methods 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. A mining strategy was developed that delivers higher value mineralised material to the mill in the early years of the mine life. During the later years in the 37-year life, the mill will be fed with low-grade stockpile material. This strategy involves a phased approach to the production schedule, whereby the project commences at a lower throughput level of 45,000 t/d for the first five years, and expands to a 90,000 t/d operation for the remainder of the mine life.

The mine production plan is primarily based on respective value cut-off grades. Variations in throughput, energy draw, and recovery are critical factors in actual unit costs. Variations in these factors have the potential to impact the mine development plan/design/sequence.

Further geotechnical, geomechanical (open pit), and hydrogeological studies are required to support the detailed mine design and site configurations.

## 25.3 Metallurgical Testing

Five different variability studies have been conducted on project samples since 2009. Each of these studies revealed a link between rougher nickel recovery and sulphur head grade, so these have been used to build a basic geometallurgical model for the project.

Further work is required to refine the flowsheet. This work includes investigating the use of regrinding in concentrate cleaning and determining how to tailor the production of concentrates for specific markets. Additional work is also needed on lower sulphur-bearing samples.

## 25.4 Recovery Methods

Concentrator design work completed to date confirms that the Turnagain mineralisation can be treated in a 90,000 t/d HPGR plus ball mill circuit followed by froth flotation to produce a high-grade nickel concentrate.

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

## 25.5 Tailings & Water Management

Results of the work completed to date suggest that the current waste and water management concept is practicable and should be carried forward to the next level of design (pre-feasibility study).

The conceptual design presented in this report can likely be optimised as the design basis and operating criteria are further refined.

## 25.6 Power Supply

With construction of the 287 kV powerline to Tatogga Lake there is increased certainty of power supply to the Turnagain Project and increased confidence in associated capital cost estimates.

## 25.7 Economic Analysis

The PEA case is based on the long-term nickel and cobalt prices of US$7.50/lb Ni and US$22.30/lb Co as provided by Wood Mackenzie, and returns a pre-tax IRR of 6.3% and pre-tax NPV @ 8% of US$(269) M. The after-tax NPV @ 8% is US$(443) M and the after-tax IRR is 4.9%. The project returns are highly sensitive to a variety of factors and these results should be interpreted in the context of the sensitivity analysis.

## 25.8 Opportunities & Risks

There are certain risks that may affect the viability of the project going forward that should be studied and addressed. Opportunities also exist that could have a positive impact on the project going forward. These risks and opportunities are outlined below.

### 25.8.1 Risks

Potential project risks are outlined in the subsections below.

#### 25.8.1.1 Geology & Mineral Resources

- Further studies related to lithological and geometallurgical characteristics may require refined domain models that restrict the amount of recoverable resource and decrease the size of the mineral resources.
- COVID-19 poses a risk to timelines and availability of personnel needed for project advancement and completion.
- There is no guarantee that further drilling will result in additional resources or increased classification.
- The optimised pit that constrains the resources (which defines the ‘reasonable prospects of eventual economic extraction’) traverses the river. This option has not been ruled out; however, it is not the selected option in this design.
- Lower commodity prices will change size and grade of the potential targets.

- Further work may disprove previous models and therefore result in condemnation of targets and potential negative economic outcomes.

#### 25.8.1.2 Mining

- Pit geotechnical and hydrogeological parameters are preliminary and shallower pit slopes could negatively impact resources particularly in the vicinity of the Turnagain River. However, the risk of significantly increasing strip ratios is not considered to be high.
- The mine production schedule described in this study includes inferred resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that these data will be realised.
- 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.

#### 25.8.1.3 Recovery Methods

- Long-term stockpiling of lower grade material may expose material to metallurgical degradation and may negatively impact mineral processing metal recoveries.
- The algorithm used to estimate overall nickel recovery in the concentrator has been validated against few samples of ore containing less than 0.5% sulphur grade, implying a lower confidence level in projected recoveries for the later years in the mine life (after Year 16 of operation, some years below 0.5% S grade). This further implies that the process design requirements of the recovery circuit may require changes as further metallurgical testwork is completed.
- The HPGR comminution circuit design for the processing plant requires extensive bulk material transport infrastructure (conveyors, bins, etc.). The cold climate of the Turnagain operation may lead to issues in material handling at transfer points due to frozen material.
- The impact of the Turnagain ore on HPGR equipment wear (e.g., HPGR rolls and discharge splitter) has yet to be determined through HPGR pilot testwork. There is potential that HPGR equipment wear rates may be higher than estimated in this study, which may adversely impact project economics through increased operating expenditure.

#### 25.8.1.4 Project Infrastructure

- Geotechnical parameters for the waste dump and low-grade stockpile designs and configurations need to be confirmed, although there is flexibility in the site layout to accommodate some adjustments.

#### 25.8.1.5 Environmental Baseline Studies, Permitting & Social or Community Impact

- The environmental impact assessment process requires multiple consecutive years of environmental baseline studies and assessment of the results, as well as extensive public, stakeholder and Indigenous Group consultation processes. The EIA process has not yet commenced in a formal manner, and the project timeline should reflect the probable duration of this process, including the uncertainty associated with the regulatory and consultation aspects of the process.

- Environmental aspects associated with infrastructure and ancillary aspects of the project, such as the access road, transmission line, camp and airstrip facilities, may need to be included in the scope of the EIA and an assessment certificate issued prior to work commencing.

### 25.8.2 Opportunities

Potential project opportunities are outlined in the subsections below.

#### 25.8.2.1 Geology & Mineral Resources

- An intelligent, systematic exploration program could provide an excellent opportunity for successfully uncovering new discoveries.
- An increased understanding and derivation of alternative theories may result in further discovery and significant expansion for the project.
- Higher commodity prices will change size and grade of the potential targets.
- Potential for expansion and classification upgrade of resources contained within the ultimate pit and in the immediate proximity.
- Additional resources in the Hatzl and Cliff areas, with the latter offering potential for additional platinum and palladium values.
- Enhanced geometallurgical knowledge of the mineralisation will aid in detailing and segregating higher grade recoverable resources.
- Further metallurgical improvements are possible, particularly in relation to geological modelling and recovery.
- The opportunity of full project optimisation, taking into account the geometallurgical characteristics of the mineralisation as noted above.
- Opportunity to employ in-country resources and personnel due to COVID-19.

#### 25.8.2.2 Mining

- Re-routing a portion of the Turnagain River to facilitate access to mineralisation in the eastern Horsetrail Zone which is currently limited to the north by the natural river alignment.
- Use of automated mine production equipment offers potential productivity improvements and reduced operating costs.
- Use of diesel-electric, hybrid, or all-electric haul trucks may offer potential for reduced operating costs and carbon footprint.
- While light-duty vehicles, such as the support fleet, can feasibly be replaced by battery electric alternatives soon arriving on the market, heavier hauling trucks exceeding 100 tonnes of hauling capacity may require a more energy dense storage technology. Given that such vehicles have an electric drive already in place, the use of hydrogen fuel cells with batteries

as a hybrid enables the electrification of these vehicles to maintain their payload capacity while not sacrificing range significantly. Currently there are initiatives to engineer and produce hydrogen fuel cell haul trucks with a greater hauling capacity than 100 tonnes. Release dates are anticipated starting from 2021.

- Electrified trolley assist could possibly be applied to some sections of the haul routes to reduce haul truck fuel consumption.
- Application of in-pit and/or semi-mobile crushing and conveying may offer potential for reduced pit haulage fuel consumption and mine operating unit costs.

#### 25.8.2.3 Metallurgical Testing

- Enhanced geometallurgical knowledge of the mineralisation as an aid to pit optimisation and to facilitate targeting of higher value zones earlier in the mine life.
- Further metallurgical improvements, particularly related to geological modelling and recovery.

#### 25.8.2.4 Recovery Methods

- Potential to achieve higher payment for concentrates via other process technologies (roasting, hydrometallurgical refining, etc.).
- Additional preliminary engineering studies have indicated potential opportunities to improve project economics using an alternate comminution circuit design approach that may allow up to 35% higher single-train capacity.
- Opportunities linked to recent developments in comminution technology and flowsheet design can be explored in further studies. Configurations that may become attractive in the near future, and may be an object for future trade-offs:
  - HPGR-HPGR-BM-BM circuit to increase HPGR utilisation and overall comminution energy efficiency
  - HPGR-HPGR-stirred mill allowing even greater comminution energy efficiency
- Other mineral processing options, such as gravity separation, may provide an opportunity to improve recovery or may be used as a pre-concentration stage to reduce comminution circuit energy consumption. The Pt-Pd values in the nickel concentrate are not expected to be at payable levels. A gravity circuit on nickel concentrate could create a high-grade payable platinum-palladium concentrate.
- Expediting process plant expansion phasing will increase mineral processing capacity in early years, and may improve project economics.
- Woodgrove & Eriez flotation reactors may offer an opportunity to reduce plant footprint and energy usage in the recovery circuit if future testwork demonstrates their effectiveness in processing Turnagain ore.
- Modifications to flotation chemistry may offer opportunities to improve recovery.
- Additional metallurgical testwork has indicated potential improvements to project economics with a revised flotation strategy including separate high-grade and low-grade flotation circuits (roughers in series producing high- and low-grade rougher concentrates for separate

cleaning, allowing for customised reagent usage and possible low-grade rougher concentrate regrinding).

#### 25.8.2.5 Project Infrastructure & Tailings Management Facility

- Utilise run-of-mine waste rock as TMF buttress material to reduce total site footprint and potentially reduce TMF construction costs.
- Shared access development costs with the potential development of the Kutcho Creek Project further to the west.

#### 25.8.2.6 Power Supply

- Potential to negotiate the BC Hydro connection in the context of helping the government achieve its climate goals (avoid LNG, implement non-diesel mining fleet)
- Potential elimination of Tariff Supplement 37, which would significantly reduce projected BC Hydro interconnection fees.

#### 25.8.2.7 Environmental Baseline Studies, Permitting & Social or Community Impact

- Potential to provide economic opportunities for First Nations with infrastructure ownership and operation as well as contracting opportunities,
- Potential to monetise the sequestration of carbon in the tailings facility,
- Based on the diesel and electricity consumption for mining activities, the carbon intensity calculated for the operations is averaged to 74,428 $tCO_2e$/year. Most of the emissions are direct emissions (scope 1) and are coming from the diesel consumption of the mining fleet. There is an opportunity to reduce the carbon intensity to 23,080 $tCO_2e$/year through electrification of the mining fleet and shift the majority of the fleet-emissions to indirect emissions (scope 2) associated with the generation of electrical power by BC Hydro.

# 26.0 RECOMMENDATIONS

## 26.1 Introduction

This report has shown that the Turnagain property is a potentially viable project at the base case parameters based on the current NI 43-101 resource. It is recommended that the project advance to the pre-feasibility study stage. Recommendations by study area for the next phase are summarised in the sections below.

## 26.2 Geology & Mineral Resources

To advance the Turnagain Project to the pre-feasibility study stage and further evaluate the potential adjacent deposits, the following work program is recommended:

- drill 25 holes for a total of 8,650 m focussed on upgrading inferred resources within the design pit to indicated resources and for geotechnical purposes
- carry out geometallurgical study and geometallurgical domain modelling to support an updated resource estimation and improved metallurgical modelling
- explore for significant Hatzl and Cliff deposits for potential to bring into resources

## 26.3 Mining

### 26.3.1 Plant Throughput & Metal Recoveries

Plant throughput and metal recoveries are applied without incorporating potential variations within or between the deposit geological domains. The geological model used in this PEA is a significant re-interpolation of data when compared to previous studies. Further geometallurgical testing, geological interpreting and geological modelling will be necessary to confirm the universal applicability of the parameters used in this report. It is recommended that future mine planning studies be performed incorporating updated geological, geometallurgical and domain data.

### 26.3.2 Increased Production Rate with Mine Resource Expansion

The Hatzl Zone has been excluded from the mining plan in this PEA study. The geological model outcrops in the zone, indicating there is potential for increasing the near-surface mineable resource. The inclusion of the Hatzl deposit would increase the size of the mine.

Attractive mineralisation appears to extend below the Turnagain River from the Horsetrail zone and the current pit shell configurations are limited by the river boundary. It is recommended that trade-off studies be performed to determine the viability of re-routing the river to the east to facilitate recovery of mineralisation both deeper in the Horsetrail pit and to the east of the current planned shell crest.

Fleet selections should be further refined to assess the potential application of automation, trolley assist, and pit crusher conveyor systems.

The proposed life of mine is 37 years. It is a low strip ratio operation, indicating there is an opportunity to increase the production capacity and throughput rate to better take advantage of economies of scale and lower unit costs. Lower operating costs could generate higher revenues and potentially increase the mineable resource. It is therefore recommended that production rate scenario trade-off studies be performed for further optimization. These scenario runs are to incorporate updated geological modelling, updated metallurgical testing, geometallurgical domain assessment, and updated scenario costing.

### 26.3.3 Geotechnical & 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 decreased, thereby reducing the total accessible resource and to some degree increasing the strip ratio. Conversely, any increase to wall slope angles would increase the volume of accessible resource.

Groundwater sources must be studied in order to implement a dewatering plan and determine a safe slope angle for the south pit wall, particularly with regard to the proximity of Turnagain River. Even relatively minor pit slope issues on the south wall could have potentially significant consequences. 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 are recommended to determine the degree of dewatering necessary to keep the pit dry under operating conditions.

Hydrogeological and geotechnical studies are recommended to determine the degree of dewatering necessary to keep the pit dry under operating conditions, and asses pit slope and waste dump design parameters.

## 26.4 Metallurgical Testing

To further advance the project the following work is recommended:

- Carry out testwork on low sulphur feed material. The majority of the samples tested to date have contained sulphur that is much higher than now planned.
- Enhance the current understanding of geometallurgical drivers and improve the geometallurgical link to the resource model. For example, potentially correlate lithotypes, alterations, etc. with throughput, grind energy and recovery.
- Test gravity separation on nickel concentrate samples to determine if a payable PGE concentrate can be produced.

## 26.5 Recovery Methods

The following aspects of mineral processing design will need to be explored in subsequent phases of this project:

- Explore the potential for regrinding part of the rougher concentrate to achieve higher recovery, potentially up to 4%.

- Explore the potential to produce two concentrates targeting two different markets.
- Conduct a grade versus recovery test work program (there might be opportunities to use a different reagent scheme if the focus is recovery and not grade, such as in this case (18% Ni grade).
- Conduct further comminution and HPGR pilot testwork to validate and refine the design of the comminution flowsheet. Comminution testwork recommendations for future phases of study are as follows:
  - single-stage open-circuit HPGR tests at different specific pressing force settings (~800 kg): ~$13,000 preliminary estimate cost for the testwork program
  - second-stage open-circuit HPGR tests at different specific pressing force settings (use first-stage HPGR product): ~$6,000 preliminary estimate cost for the testwork program
  - locked cycle HPGR tests with edge recycle at one recommended specific pressing force setting (700 kg): ~$13,000 preliminary estimate cost for the testwork program
  - stirred mill testing for second-stage fine grinding (optional) (~20 kg composite from subsample of HPGR testwork product): ~$5,000 preliminary estimate cost for the testwork program
  - due to the lack of certainty in power cost, stirred mill testwork can be explored to further provide comminution alternatives

- Conduct further flotation testwork using products of HPGR and comminution testwork.
- Conduct thickening and filtration testwork for concentrate. Tailings settling tests should be carried out to investigate tailings thickener as a means of reducing pumping costs associated with tailings reclaim.
- Optimise flotation equipment selection and sizing:
  - investigate recent flotation equipment developed (more modern) as opposed to tank cells (e.g., Eriez stack cells, Woodgrove SFRs)
  - investigate coarser flotation testing Eriez hydrofloat options

## 26.6 Project Infrastructure & Tailings Management Facility

Further optimisation required in the next phase of study includes the following:

- tailings thickener to save power pumping reclaim water back from TMF and have smaller tailings lines
- Evaluate the potential use of mine waste rock for TMF construction
- PFS design for earthworks and roads to improve the capital cost estimate accuracy

## 26.7 Power Line

In order to address risks associated with the transmission line and to avoid unnecessary delays in approval processes, key recommendations/next steps include the following:

- request BC Hydro conduct a System Impact Study to confirm available transmission capacity and connection methods

- review transmission line cost estimating assumptions and updating transmission line cost estimates, as necessary (estimates were based on actual construction information on the cost of the NTL from Terrace to Bob Quinn, and the transmission line extension to Tatogga Lake)
- assess potential transmission line routes for cases with connection to BC Hydro's grid; particularly, the section along Highway 37 beside Stikine Provincial Park
- engage with First Nations
- consult with stakeholders and engage with the public
- keep a watching brief on:
  - the progress of BC Hydro's Site C project, and public statements regarding the project's anticipated effect on electricity rate schedules
  - progress of other projects proposing to connect to the NTL, as this may impact the cost of interconnection for the proposed Turnagain mine
  - discussions regarding Tariff Supplement 37, and provide input to BC Hydro/Province of BC through channels such as the Mining Association of BC
  - the economics of large-scale batteries for industrial load shaping
  - the status of alternative LNG terminal projects and the commodity cost of LNG
- Optimisation in the next phase of study:
  - confirm the most economic configuration for the site power grid (i.e., main substation at the concentrator plant and 25 kV overhead line distribution to the other areas)
  - investigate how to improve the power factor, especially by reviewing options for large drives such as for the ball mills
  - consider diesel (rather than electric) mining equipment (shovels and drills) to reduce load swings on the site power grid

## 26.8 Environmental Baseline Studies, Permitting & Social or Community Impact

With initiation of the EIA process, the scope of environmental baseline studies will be defined, and the consultation and engagement process requirements will be established. Based on these inputs and realistic timelines for the EIA processes considering recent experience for mines in BC, a permitting strategy, timeline and work plan must be developed. This strategy must integrate with the planned progress of the design, and include an allowance for an assessment of alternatives and reviews by interested parties and stakeholders.

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# ATTACHMENT 1: QUALIFIED PERSON CERTIFICATES

# CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project" prepared for Giga Metals Corporation (the "Issuer") dated November 18, 2020, with an effective date of October 28, 2020 (the "Technical Report").

I, Ian Thompson, do hereby certify that:

1. I am Senior Mining Engineer with the firm of Hatch Ltd. with an office located at 1066 West Hastings Street, Suite 400, Vancouver, BC, Canada.
2. I am a graduate of UBC, where in 1989 I obtained a Bachelor of Applied Science degree in Mining and Mineral Process Engineering.
3. I have practiced my current profession continuously since 1989. My principal experience is in the area of open pit mining.
4. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia (Registration #27598).
5. I have personally inspected the subject property on October 9 to 10, 2018.
6. I have read the definition of a qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.1, 1.2, 1.3, 1.8, 1.9, 1.11, 1.13, 2, 3, 4.4, 5.1, 5.4, 15, 16, 21 (except for 21.1.2, 21.1.3.1, 21.1.4, 21.2.3, 21.2.6), 24, 25.2, 25.8.1.2, 25.8.2.2, 26.1, 26.3 and 27, and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Ian Thompson"*

Ian S. Thompson, P.Eng.
Senior Mining Engineer

## CERTIFICATE OF QUALIFIED PERSON

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I, Persio Pellegrini Rosario, do hereby certify that:

1. I am the Director of Comminution with the firm of Hatch Ltd. with an office located at 1066 West Hastings Street, Suite 400, Vancouver, BC, Canada.
2. I am a graduate of the University of British Columbia, where, in 2003 and 2010, respectively, I obtained the Master in Applied Sciences (MASc) and the Doctor in Philosophy (PhD) degrees in Mineral Processing through the Mining and Mineral Processing Department.
3. I have practiced my current profession continuously since 2003. My principal experience is in the areas of base and precious metals mineral processing.
4. I am a Professional Engineer registered with the registered with the Engineers and Geoscientists of British Columbia.
5. I have not personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.7, 17, 21.1.2, 21.2.3, 25.4, 25.8.1.3, 25.8.2.4, 26.5, and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Persio Pellegrini Rosario"*

Persio Pellegrini Rosario, MASc, PhD, P.Eng
Director, Comminution – Mining & Mineral Processing

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I, Evan Lewis Jones, do hereby certify that:

1. I am Regional Director, Environmental Services Group, with the firm of Hatch Ltd. with an office located at 1066 West Hastings Street, Suite 400, Vancouver, BC, Canada.
2. I am a graduate of the University of British Columbia, where, in 1985, I obtained a Bachelors of Applied Science through the Bio-Resource Engineering Department.
3. I am a graduate of the University of British Columbia, where, in 1990, I obtained a Masters of Applied Science through the Civil Engineering Department.
4. I have practiced my current profession continuously since 1985. My principal experience is in the areas of Environmental Engineering.
5. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia;
6. I have not personally inspected the subject property.
7. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
8. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
9. I am a contributing author of the Technical Report and am responsible for Sections 1.10, 4.3, 5.2, 5.3, 5.6, 20, 25.8.1.5, 25.8.2.7 and 26.8 and accept professional responsibility for these sections of the Technical Report.
10. I have had no prior involvement with the subject property.
11. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 9 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
12. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Evan Jones"*

Evan Jones, P.Eng., EP(CEA)
Regional Director, Environmental Services Group

HATCH

## CERTIFICATE OF QUALIFIED PERSON

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I, Gerald (Gerry) Schwab, do hereby certify that:

1. I am a Project Manager with the firm of Hatch Ltd. with an office located at 1066 West Hastings Street, Suite 400, Vancouver, BC, Canada.
2. I am a graduate of the University of British Columbia, where, in 1984 I obtained a BASc in Mechanical Engineering through the Engineering Department.
3. I have practiced my current profession continuously since 1984. My principal experience is in the areas of process plant design, and project management.
4. I am a Professional Engineer registered with the Engineers and Geoscientists of British Columbia;
5. I have not personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 5.5, 18,1, 18.5.5, 18.5.6, 18.5.7, 18.6, 18.7, 18.9, 18.10, 25.8.1.4, 25.8.2.5 and 26.6, and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Gerry Schwab"*

Gerry Schwab, P.Eng.

HATCH

## CERTIFICATE OF QUALIFIED PERSON

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I, Stefan Joseph Hlouschko, do hereby certify that:

1. I am a Consultant with the firm Hatch Ltd. with an office located at 2800 Speakman Drive, Mississauga, Ontario, Canada, L5K 2R7.
2. I am a graduate of Queen's University, Canada, where, in 2006 I obtained a Bachelor of Science in Engineering (Engineering Physics) degree through the Faculty of Applied Science. I am a graduate of Queen's University, Canada, where, in 2008 I obtained a Master of Science in Engineering (Mechanical) degree through the School of Graduate Studies.
3. My principal experience is in the areas of instrumentation and control engineering (7 years), and project assessment / management consulting (5 years).
4. I am a Professional Engineer registered with the Professional Engineers of Ontario.
5. I have not personally inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.12, 22, 25.7 and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Stefan J. Hlouschko"*

Stefan Joseph Hlouschko, P.Eng.

## CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project" prepared for Giga Metals Corporation (the "Issuer") dated November 18, 2020, with an effective date of October 28, 2020 (the "Technical Report").

I, Garth David Kirkham, do hereby certify that:

1. I am a consulting geoscientist with an office at 6331 Palace Place, Burnaby, British Columbia.
2. I am a graduate of the University of Alberta in 1983 with a B. Sc.
3. I have continuously practiced my profession since 1988. My principal experience is in the areas of geology and resource estimation having authored many NI43-101 technical reports including Bralorne, Table Mountain, Monument Bay, Minto, Kutcho Creek, Cerro Las Minitas and Cerro Blanco. I am currently the Geological and Resource Technical Expert for Group Ten Metals and Valore Metals developing the Stillwater West PGE-Ni and Kluane PGE-Ni-Cu, and the Pedra Branca PGE-Cr meta-volcanic, ultramafic deposits, respectively.
4. I am a Professional Geoscientist and a member in good standing of the Engineers and Geoscientists British Columbia.
5. I have visited the property on October 9 to 10, 2018.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101.
7. I am independent of Giga Metals Corporation as defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.4, 1.5, 4.1, 4.2, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.1, 25.8.1.1, 25.8.2.1 and 26.2 and accept professional responsibility for these sections of the Technical Report.
9. I have not had prior involvement in the Turnagain project.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Garth Kirkham"*

Garth Kirkham, P.Geo.
President, Kirkham Geosystems Ltd.

Knight Piésold
CONSULTING

# CERTIFICATE OF QUALIFIED PERSON

I, Daniel Friedman, P.Eng., do hereby certify that:

1. This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project" prepared for Giga Metals Corporation (the "Issuer") dated November 18, 2020, with an effective date of October 28, 2020 (the "Technical Report").
2. I am employed as a Specialist Civil Engineer of Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
3. I am a graduate of McGill University, Montreal, Canada, B.Eng. (Civil), 2003. I have practiced my profession continuously since 2004. My principal experience is in the areas of water and waste management for mining projects and hydrotechnical engineering.
4. I am a Professional Engineer (No. 32571) in good standing with Engineers and Geoscientists of British Columbia in the area of Civil Engineering.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I visited the Turnagain property on September 7, 2005, and June 16, 2009.
7. I am a contributing author of the Technical Report and am responsible for Sections 1.9.2, 18.2, 18.4, 18.5.1, 18.5.2, 18.5.3, 18.5.4, 21.1.4, 21.2.6 and 25.5 and accept professional responsibility for these sections of the Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have had no involvement with the property that is the subject of this Technical Report.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Effective Date: October 28, 2020
Signing Date: November 18, 2020

*"Daniel Friedman"*

______________________________________

Daniel Friedman, P.Eng.


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KERR WOOD LEIDAL
consulting engineers

## CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project" prepared for Giga Metals Corporation (the "Issuer") dated November 18, 2020, with an effective date of October 28, 2020 (the "Technical Report").

I, Ronald J. Monk, do hereby certify that:

1. I am a Principal and Energy Sector Leader with the firm of Kerr Wood Leidal Associates Ltd. with an office located at 200 – 4185A Still Creek Drive Burnaby, BC, Canada V5C 6G9.
2. I am a graduate of the University of British Columbia, where, in 1987 I obtained Bachelor of Applied Science (with Honours), in Civil Engineering through the Civil Engineering Department.
3. I am a graduate of the University of British Columbia, where, in 1992 I obtained a Master of Engineering, in Civil Engineering (Hydraulics and Construction Management) through the Civil Engineering Department.
4. I have practiced my current profession continuously since 1987. My principal experience relevant to this Technical Report is in the areas of power supply analysis, connection to power utilities, power utility rates and tariffs and the planning of power lines for mines.
5. I am a Professional Engineer registered with Engineers and Geoscientists BC.
6. I have not personally inspected the subject property.
7. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
8. I am independent of the Issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
9. I am a contributing author of the Technical Report and am responsible for Sections 1.9.1, 18.8, 21.1.3.1, 25.6, 25.8.2.6 and 26.7 and accept professional responsibility for these sections of the Technical Report.
10. I have prior involvement with the subject property regarding power supply alternatives and carbon reduction strategies since August 2011.
11. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 9 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
12. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Ronald J. Monk"*

Ronald J. Monk, M.Eng., P.Eng.
Principal & Energy Sector Leader

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project" prepared for Giga Metals Corporation (the "Issuer") dated November 18, 2020, with an effective date of October 28, 2020 (the "Technical Report").

I, Christopher John Martin, do hereby certify that:

1. I am President with the firm of Blue Coast Metallurgy Ltd with an office located at 1020 Herring Gull Way, Parksville, BC V9P 1R2.
2. I am a graduate of Camborne School of Mines, where, in 1984 I obtained a Bachelor of Science Degree in Mineral Processing Technology, and McGill University, where, in 1987 I obtained a Master of Engineering Degree in Metallurgical Engineering through the Department of Mining and Metallurgy.
3. I have practiced my current profession continuously since 1987. My principal experience is in the areas of comminution, and beneficiation of base and precious metal minerals, both in operations and project development.
4. I am a Chartered Engineer registered with the Engineering Council of the United Kingdom, and a Member of the Institution of Materials, Minerals and Mining.
5. I personally inspected the subject property on October 19, 2010.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the Issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Sections 1.6,13, 25.3, 25.8.2.3 and 26.4 and accept professional responsibility for these sections of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I have accepted responsibility contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*"Christopher John Martin"*

Christopher John Martin
President

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, “N.I. 43-101 Technical Report & Preliminary Economic Assessment for the Turnagain Project” prepared for Giga Metals Corporation (the “Issuer”) dated November 18, 2020, with an effective date of October 28, 2020 (the “Technical Report”).

I, Andrew Robert Mitchell, do hereby certify that:

1. I am Head of Nickel Research with the firm of Wood Mackenzie Ltd with an office located at Exchange Place 5 Semple Street, Edinburgh, EH3 8BL, United Kingdom.
2. I am a graduate of Camborne School of Mines, where, in 1988 I obtained Ph. D in pyrometallurgy through the same institution.
3. I have practiced my current profession continuously since 1995. My principal experience is in the areas of nickel market analysis.
4. I am a professional engineer registered with the Institute of Materials, Minerals and Mining.
5. I have personally not inspected the subject property.
6. I have read the definition of qualified person set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of National Instrument 43-101.
7. I, as a qualified person, am independent of the issuer, as that term is defined in Section 1.5 of National Instrument 43-101.
8. I am a contributing author of the Technical Report and am responsible for Section 19 and accept professional responsibility for this section of the Technical Report.
9. I have had no prior involvement with the subject property.
10. I have read National Instrument 43-101 and the sections of the Technical Report referenced in paragraph 8 of this Certificate and confirm that these sections have been prepared in accordance with National Instrument 43-101.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the section of the Technical Report for which I have accepted responsibility contains all scientific and technical information required to be disclosed to make the Technical Report not misleading.

Signed and sealed this 18 day of November 2020

*“Andrew Robert Mitchell”*

Andrew Robert Mitchell, PhD.
Research Director, Head of Nickel Research

# ATTACHMENT 2: UNITS OF MEASURE, ACRONYMS & ABBREVIATIONS

## Units of Measure

| | |
|---|---|
| Above mean sea level | amsl |
| Ampere | A |
| Annum (year) | a |
| Billion years ago | Ga |
| Carbon dioxide equivalent | $C0_2e$ |
| Carbon dioxide equivalent (total) | $tCO_2e$ |
| Centimetre | cm |
| Cubic centimetre | $cm^3$ |
| Cubic feet per second | $ft^3/s$ or cfs |
| Cubic foot | $ft^3$ |
| Cubic inch | $in^3$ |
| Cubic metre | $m^3$ |
| Cubic yard | $yd^3$ |
| Day | d |
| Days per week | d/wk |
| Days per year (annum) | d/a |
| Dead weight tonnes | dwt |
| Degree | ° |
| Degrees Celsius | °C |
| Degrees Fahrenheit | °F |
| Diameter | ø |
| Dry metric ton | dmt |
| Foot | ft |
| Gallon | gal |
| Gallons per minute (US) | gpm |
| Gigawatt-hours | GWh |
| Gram | g |
| Grams per litre | g/L |
| Grams per tonne | g/t |
| Greater than | > |
| Hectare (10,000 $m^2$) | ha |
| Hertz | Hz |
| Horsepower | hp |
| Hour | h |
| Hours per day | h/d |
| Hours per week | h/wk |
| Hours per year | h/a |
| Inch | " |
| Kilo (thousand) | k |
| Kilogram | kg |
| Kilograms per cubic metre | $kg/m^3$ |
| Kilograms per hour | kg/h |
| Kilograms per square metre | $kg/m^2$ |
| Kilojoule | kJ |
| Kilometre | km |

| | |
|---|---|
| Kilometres per hour | km/h |
| Kilonewton | kN |
| Kilopascal | kPa |
| Kilovolt | kV |
| Kilovolt-ampere | kVA |
| Kilovolts | kV |
| Kilowatt | kW |
| Kilowatt hour | kWh |
| Kilowatt hours per tonne | kWh/t |
| Kilowatt hours per year | kWh/a |
| Less than | < |
| Litre | L |
| Litres per minute | L/m |
| Megabytes per second | Mb/s |
| Megapascal | MPa |
| Megavolt-ampere | MVA |
| Megawatt | MW |
| Megawatt-hours | MWh |
| Metre | m |
| Metres above sea level | masl |
| Metres per minute | m/min |
| Metres per second | m/s |
| Metric ton (tonne) | t |
| Micrometre (micron) | µm |
| Miles per hour | mph |
| Milliamperes | mA |
| Milligram | mg |
| Milligrams per litre | mg/L |
| Millilitre | mL |
| Millimetre | mm |
| Million | M |
| Million tonnes | Mt |
| Minute (plane angle) | ' |
| Minute (time) | min |
| Month | mo |
| Nanoteslas | nT |
| Newton | N |
| Newtons per metre | N/m |
| Ounce | oz |
| Parts per billion | ppb |
| Parts per million | ppm |
| Pascal (newtons per square metre) | Pa |
| Pascals per second | Pa/s |
| Percent | % |
| Phase (electrical) | Ph |
| Pound(s) | lb |
| Pounds per square inch | psi |
| Power factor | pF |
| Revolutions per minute | rpm |
| Second (plane angle) | " |
| Second (time) | s |
| Specific gravity | SG |

Square centimetre ........ $cm^2$
Square foot ........ $ft^2$
Square inch ........ $in^2$
Square kilometre ........ $km^2$
Square metre ........ $m^2$
Thousand tonnes ........ kt
Tonne (1,000 kg) ........ t
Tonnes per day ........ t/d
Tonnes per hour ........ t/h
Tonnes per year ........ t/a
Total dissolved solids ........ TDS
Total suspended solids ........ TSS
Volt ........ V
Week ........ wk
Weight/weight ........ w/w
Wet metric ton ........ wmt
Yard ........ yd
Year (annum) ........ a
Year (US) ........ y

## Abbreviations/Acronyms

Acid base accounting ........ ABA
Acid rock drainage ........ ARD
Alternating current ........ AC
Ammonium nitrate ........ AN
Ammonium nitrate and fuel oil ........ ANFO
Association for the Advancement of Cost Engineering ........ AACE
Autogenous grinding ........ AG
Average emission factor ........ AEF
Bed volume ........ BV
Blue Coast Metallurgy Ltd. ........ Blue Coast
British Columbia ........ BC
Canadian ........ CDN
Canadian dollars CAD or C$
Canadian Electrical Manufacturers Association ........ CEMA
Canadian Gas Association ........ CGA
Canadian Institute of Mining and Metallurgy ........ CIM
Canadian Metals Exploration Ltd ........ CME
Canadian Standards Association ........ CSA
Capital cost allowance ........ CCA
Closed-circuit television ........ CCTV
Coarse ore storage ........ COS
Cominco Engineering Services Ltd. ........ CESL
Commissioned land surveyor ........ CLS
Compound annual growth rate ........ CAGR
Degree of alteration ........ DOA
Democratic Republic of Congo ........ DRC
Differential global positioning system ........ DGPS
Differential pressure ........ DP
Direct current ........ DC
Distributed control system ........ DCS

Effective grinding length ........ EGL
Electric vehicle ........ EV
Electrical resistance ........ Ohm
Energy storage system ........ ESS
Engineering, procurement, and construction management ........ EPCM
Environmental impact assessment ........ EIA
Environmental impact statement ........ EIS
Environmental, social & governance ........ ESG
Fisheries and Oceans Canada ........ DFO
Flow coefficient ........ CV
Free on board ........ FOB
Front-end loader ........ FEL
Global positioning survey ........ GPS
Global positioning system ........ GPS
Goods and Sales Tax (Canada) ........ GST
Greenhouse gas ........ GHG
Hard Creek Nickel Corporation ........ HNC
Helicopter-borne electromagnetic ........ HBE
High density polyethylene ........ HDPE
High pressure ........ HP
High voltage ........ HV
High-pressure acid-leaching ........ HPAL
High-pressure grinding rolls ........ HPGRs
Horizontal-to-vertical ratio ........ H:V
Induced polarisation ........ IP
Input/output ........ I/O
Institute of Electrical and Electronics Engineers ........ IEEE
Instrumentation and control ........ I&C
Internal rate of return ........ IRR
International Standards Organization ........ ISO
Isopropyl xanthate collector ........ SIPX
Kerr Wood Leidal Ltd. ........ KWL
Kirkham Geosystems Ltd. ........ Kirkham
Knight Piesold Ltd. ........ KP
Kutcho Copper Corporation ........ Kutcho
Land and Resource Management Plan ........ LRMP
Light emitting diode ........ LED
Liquified natural gas ........ LNG
Load-haul-dump unit (scooptram) ........ LHD
Local area network ........ LAN
London Metal Exchange ........ LME
Low grade ........ LG
Low pressure ........ LP
Methyl isobutyl carbinol ........ MIBC
Mill head value ........ MHV
Motor control centre ........ MCC
National Building Code of Canada ........ NBC
National Electrical Manufacturers Association ........ NEMA
National Electrical Safety Code ........ NESC
National Inventory Report (Canada) ........ NIR
Net present value ........ NPV
Net smelter royalty ........ NSR

| | |
|---|---|
| Nickel pig iron | NPI |
| Nickel-cobalt-aluminum | NCA |
| Nickel-manganese-cobalt | NMC |
| Northwest Transmission Line | NTL |
| Not potentially acid-generating | NPAG |
| Occupational Safety and Health Administration | OSHA |
| Original equipment manufacturer | OEM |
| Overburden | OB |
| Overflow | O/F |
| Oversize | O/S |
| Papua New Guinea | PNG |
| Personal computer | PC |
| Phase (electrical) | Ph |
| Plant control system | PCS |
| Platinum Group Elements | PGEs |
| Polyvinyl chloride | PVC |
| Potentially acid generating | PAG |
| Power factor | pF |
| Preliminary economic assessment | PEA |
| Process flow diagram | PFD |
| Provincial Sales Tax | PST |
| Qualified person | QP |
| Relative humidity | RH |
| Rock mass rating | RMR |
| Rock quality designation | RQD |
| Run-of-mine | ROM |
| Semi-autogenous grinding | SAG |
| Supervisory control and data acquisition | SCADA |
| Tahltan Central Government | TCG |
| Tailings management facility | TMF |
| Traditional Land Use | TLU |
| Treatment and refining charges | TC/RC |
| Unconfined compressive strength | UCS |
| Underflow | U/F |
| Undersize | U/S |
| Uninterruptible power supply | UPS |
| Union Miniere Exploration and Mining Corporation Ltd. | UMEX |
| Valued ecosystem component | VEC |
| Variable frequency drive | VFD |
| Very high frequency | VHF |
| Very low frequency | VLF |
| Virtual private network | VPN |
| Volatile compound | VOC |
| Volt direct current | VDC |
| Water treatment plant | WTP |
| Wood Mackenzie | WM |
| Workplace Hazardous Material Information System | WHMIS |