NI 43-101 Technical Report

Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota

Submitted to: Talon Metals Corp.

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Prepared by Qualified Persons:

Leslie Correia, Pr.Eng

Silvia Del Carpio, P. Eng., MBA

Tim Fletcher, P. Eng.

Daniel Gagnon, P. Eng.

Kebreab Habte, P. Eng.

Oliver Peters, P. Eng.

Tom Radue, P. Eng.

Brian Thomas, P. Geo.

TOTAL SOLUTIONS OFFERING

TABLE OF CONTENTS

1. EXECUTIVE SUMMARY

1.1 Introduction

The Tamarack Project, located in Minnesota, USA, comprises the Tamarack North Project and the Tamarack South Project (refer Figure 7-5).

The Tamarack Project is currently 17.56% owned by Talon Metals Corp. (Talon), and 82.44% owned by Kennecott Exploration Company (Kennecott).

On November 7, 2018, Talon and Kennecott entered into an agreement (the 2018 Tamarack Earn-in Agreement) pursuant to which Talon has the right, subject to certain funding and reporting obligations, to increase its interest in the Tamarack Project to a maximum 60% interest and become the manager/operator of the Tamarack Project. The 2018 Tamarack Earn-in Agreement is subject to approval by the Talon shareholders of a financing required to be completed by Talon in connection with the Tamarack Project, such shareholder approval to be sought at a meeting to be held on or before January 31, 2019.

Talon has commissioned a team of consultants to complete a Preliminary Economic Assessment (PEA) in accordance with National Instrument 43-101 (NI 43-101) guidelines for the Tamarack North Project.

The following consultants contributed to completing the component PEA sections:

- **Barr Engineering (Barr):** Environmental studies, permitting, and social or community impacts;
- **DRA Americas Inc. (DRA):** Overall study management, mining methods, project infrastructure, market studies and contracts, capital and operating costs, and economic analysis;
- **Golder Associates Ltd. (Golder):** Property description and location, accessibility, climate and physiography, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, adjacent properties, Mineral Resource estimate, and tailings/waste rock co-disposal methods;

- **Metpro Management Inc. (Metpro):** Mineral processing, metallurgical testing, and recovery methods;
- **Paterson & Cooke Canada Inc. (Paterson & Cooke):** Paste backfill methods.

This PEA demonstrates a conceptual mine development plan based on Best Available Technologies (BAT). These technologies have been incorporated because of Talon's mandate to consistently follow BAT Principles and Practices. It is important to note that all design work is conceptual at the PEA stage and that additional drilling, testing, studies and engineering work will be necessary to complete a Pre-Feasibility Study (PFS) as articulated in Section 26 (Recommendations) of this document.

1.2 Location and Ownership

The Tamarack Project is located in north-central Minnesota, approximately 100 kilometres (km) (62 miles) west (W) of Duluth and 210 km (130 miles) north (N) of Minneapolis, in Aitkin County. The Tamarack North Project which this report represents, covers approximately 20,320 acres. The town of Tamarack (population 88, 2016 US Census Bureau) lies within the boundaries of the Tamarack Project (though away from the known mineralization) at an elevation of 386 metres (m) (1,266 feet (ft)) above sea level. The project area is characterized by farms, plantations and forested areas.

On June 25, 2014, Talon's wholly-owned, indirect subsidiary, Talon Nickel (USA) LLC (collectively, Talon), entered into an exploration and option agreement (the 2014 Tamarack Earn-in Agreement) with Kennecott (part of the Rio Tinto Group), pursuant to which Talon, subject to certain funding conditions, received the right to acquire a 30% interest in the Tamarack Project, which comprises both the Tamarack North Project and the Tamarack South Project.

On November 25, 2015, Kennecott and Talon amended the 2014 Tamarack Earn-in Agreement to provide that, subject to certain funding conditions, Talon would earn an 18.45% interest in the Tamarack Project.

On January 11, 2018, Talon and Kennecott entered into a mining venture agreement (the Original MVA). Pursuant to the Original MVA, Talon elected not to financially participate in the 2018 winter exploration program at the Tamarack Project. Consequently, Talon's interest in the Tamarack Project was diluted below 18.45%.

On November 7, 2018, Talon and Kennecott entered into the 2018 Tamarack Earn-in Agreement pursuant to which Talon has the right to increase its interest in the Tamarack Project to a maximum 60% interest and become the manager/operator of the Tamarack Project.

Pursuant to the 2018 Tamarack Earn-in Agreement, Talon initially has the right to increase its interest in the Tamarack Project to 51% by:

- Paying Kennecott US$6M cash and issuing US$1.5M worth of common shares in Talon to Kennecott on the effective date of the 2018 Tamarack Earn-in Agreement; and
- Within three years of the effective date of the 2018 Tamarack Earn-in Agreement, by Talon (a) incurring US$10M in exploration expenditures on the Tamarack Project, or (b) delivering a PFS in accordance with NI 43-101, whichever comes first; and
- Also within three years of the effective date of the 2018 Tamarack Earn-in Agreement, Talon paying Kennecott the additional sum in cash of US$5M.

In the event Talon successfully earns a 51% interest in the Tamarack Project, Talon will then have the right, within seven years of the effective date of the 2018 Tamarack Earn-in Agreement, to further increase its interest in the Tamarack Project to 60% by:

- Completing a Feasibility Study in accordance with NI 43-101; and
- Paying Kennecott an additional cash payment US$10M.

Upon Talon earning a 60% interest in the Tamarack Project, the parties have agreed to enter into a new mining venture agreement (the New MVA) under which Talon would assume the role of Manager of the Tamarack Project, and the parties would each be required to fund their *pro rata* share of expenditures in respect of the Tamarack Project or be diluted.

Section 4 of this PEA contains further details regarding Talon's interest in the Tamarack Project.

1.3 Environmental Considerations and Permitting

The Tamarack North Project will be subject to state and federal environmental review and permitting processes, which are described in Sections 20.6 and 20.7. Throughout the processes, Talon will demonstrate that the Tamarack North Project will avoid or mitigate potential impacts to the environment in accordance with regulatory requirements. Additional data collection beyond the baseline studies completed to date will be completed to support these processes.

These demonstrations will be supported by baseline studies to characterize existing physical and biological conditions at the site layout area (refer Section 18.3) conducted since 2006. A description of baseline studies conducted to date is provided in Table 20-1. Additional studies will be required to support further project siting, design, and environmental review and permitting efforts.

BAT have been implemented in the handling of mine waste, most notably:

- Development rock (from the shaft, levels, ramps, cross-cuts and drifts);
- Tailings that are produced because of producing the Ni and Cu concentrates.

The first priority was to determine if a High-Sulphide (HS) tailings stream could be produced. Metallurgical testing has proven that this is possible. Consequently, a Low-Sulphide (LS) tailings stream can be produced separately (refer Section 17.3.2).

A paste backfill study was commissioned to determine how much of the HS tailings and LS tailings can be mixed with cement and stored in mined out, underground voids. The results of this study showed that 100% of HS tailings and 37% of LS tailings can be blended with cement and cured underground (refer Section 16.2).

A number of studies were commissioned to investigate the use of BAT in regard to development rock and the remaining LS tailings (refer Section 18.6). These studies lead to the development of an innovative Co-disposed development rock and Filtered Tailings Facility (CFTF) which offers significant environmental and operating advantages over separate tailings storage and development rock storage facilities, including:

- Reduced risk of failure due to the implementation of dry-stacking instead of a traditional tailings dam;
- A major reduction in the waste facility footprint. Surface land required for the storage of LS tailings that are not placed underground with cement, will be reduced by approximately 40% versus traditional methods as a portion of the development rock void space will be utilized as LS tailings storage space;
- Improved tailings stability and reduced dusting compared to a standalone filtered tailings facility without co-disposal with development rock;
- At closure, the CFTF will be covered with a closure cover system. This will limit the amount of infiltration into the CFTF post closure;
- A significant reduction in fresh water requirements. In fact, 86.7% of water required by the processing plant will be recycled water of which 30% will be as a direct result of using a CFTF.

Section 18.6 contains a more detailed discussion of the application of the development rock, the fine grained ortho-cumulative olivine (FGO) and sedimentary (SED) from the shaft and levels as well as the remaining LS tailings.

In order to minimize the Tamarack North Project footprint three different mine access methods were considered (refer Section 16.8.1). As a result, mine access will be by a small diameter mine shaft, which reduces the surface expression of the excavation area by 99.9% compared to a box-cut and ramp access method. Consequently, the total surface area required for all facilities and the CFTF is limited to approximately 53 acres.

By implementing these BATs, Talon is addressing environmental sensitivities, such as:

- Potential mitigation for lost habitat of state and federal protected species;
- Potential wetland impacts and the need for wetland impact mitigation;
- Potential generation of acid rock drainage (ARD) and metal leaching (ML);
- Potential impacts to surface and ground water quality;
- Potential drawdown of surface water levels and flows.

Wetland delineation and evaluation studies in accordance with federal and local guidelines and manuals occurred in 2008 and 2009, covering the site layout area (refer Section 18.3). A 120 acre study area was initially evaluated and then expanded to a 580 acre study area.

Based on the results from these studies, the conceptual site layout (refer Section 18.3) has been partially placed on upland (23 acres) to minimize the impact on wetlands (30 acres). Section 20.2 contains a breakdown by area and wetland type.

A survey of a 322 acre study area of vegetative communities occurred in 2008 encompassing the potential site layout area. Flora was inventoried onsite and vegetative communities and habitats were mapped by type within the study area. The area where the conceptual site layout is located (refer Section 18.3) was delineated as Fallow Farm Fields/Young Pine Plantation. Satellite imagery dated 1991 suggests that much of this vegetative community had previously been farmed for many years. The vegetative communities that occur in the study area are characteristic of much of northeastern Minnesota, including Aitkin County. No unusual or uncommon natural vegetative communities were identified within the study area. Two invasive plant species (reed canary grass and narrow-leaved cattail) were abundant within several of the habitat types.

A survey for Rate Threatened and Endangered (RTE) species occurred in 2008. The survey study area covered the site layout area, except for a farm residence and adjacent buildings. The Minnesota Department of Natural Resources (MNDNR) maintains a geographic database of documented occurrences of threatened, endangered, and special concern species in Minnesota. A database search for RTE species that have been known to occur within several miles of the study area was conducted. This information and Minnesota's entire published list (MNDNR Division of Ecological Resources 2008) of RTE species were utilized while conducting the RTE field investigation within the study area in August 2008. The site was carefully surveyed using a series of thorough meander transects within all natural vegetative communities and other habitat types.

No federally listed or state listed threatened, endangered, special concern plant species or other rare natural features were documented within the study area. Because all habitat types documented within the study area are relatively common in Aitkin County and the associated ecoregion, the presence of RTE species would be unlikely.

The Tamarack North Project is expected to potentially have a negative water balance (net water demand) during the first two years of production, followed by potentially a positive water balance over the following five years of production (refer Section 18.7). Further geotechnical and hydrogeological work is needed to assess the impact of methods that may be implemented to restrict underground water infiltration into excavated voids.

Further work is also required to evaluate potential water sources. Trade-off studies of Water Treatment Plant options should be conducted during the PFS.

1.4 Geology and Mineralization

The Tamarack Intrusive Complex (TIC) is an ultramafic to mafic intrusive complex that hosts Ni-Cu sulphide mineralization with associated Cobalt (Co), Pt, Pd (PGEs) and Gold (Au). The TIC is a multi-magmatic phase intrusion, that consists of a minimum of two pulses: The FGO and the coarse grained ortho-cumulative (CGO) intrusion of the TIC (dated at 1105 Ma+/-1.2 Ma, Goldner 2011). The FGO and CGO intrusions are related to the early evolution of the approximately 1.1 Ga Mid Continent Rift (MCR) and have intruded into slates and greywackes of the Thomson Formation of the Animikie Group, which formed as a foreland basin during the Paleoproterozoic Penokean Orogen (approximately 1.85 Ga, Goldner 2011). The TIC is completely buried beneath approximately 35 m to 55 m of Quaternary age glacial and fluvial sediments. The TIC is consistent with other earlier intrusions associated with the MCR that are often characterized by more primitive melts.

The geometry of the TIC, as outlined by a well-defined aeromagnetic anomaly, consists of a curved, elongated intrusion striking north-south (NS) to southeast (SE) over 18 km. The configuration has been likened to a tadpole shape with its elongated, northern tail up to 1 km wide and large, 4 km wide, ovoid shaped body in the south (S) (Figure 7-5). The northern portion of the TIC (the Tamarack North Project), which hosts the currently defined resource and identified exploration targets, is over 7 km long and is the focus of this PEA.

The Ni-Cu sulphide mineralization with associated PGEs and Au form as the result of segregation and concentration of liquid sulphide from mafic or ultramafic magma and the partitioning of chalcophile elements into the sulphide from the silica melt (Naldrett, 1999). The various mineralized zones at the Tamarack North Project occur within different host lithologies, exhibit different types of mineralization styles, and display varying sulphide concentrations and tenors. These mineralized zones range from massive sulphides hosted by altered sediments in the massive sulphide unit (MSU), to net textured and disseminated sulphide mineralization hosted by the CGO in the semi-massive sulphide unit (SMSU), to a more predominantly disseminated sulphide mineralization as well as layers of net textured sulphide mineralization, in the 138 Zone (see Table 1-1). Mineralization in the 138 Zone, where interlayered disseminated and net textured mineralization occurs is also referred to as mixed zone (MZ) mineralization. All these mineralization types are typical of many sulphide ore bodies around the world. The current known mineral zones of the Tamarack North Project (SMSU, MSU and 138 Zone) that are the basis of the Mineral Resource estimate in this PEA are referred to collectively as the "Tamarack Zone". Also located within the Tamarack North Project are currently, two lesser defined mineral zones, namely the 480 and 164 Zones.

Table 1-1: Key Geological and Mineralization Relationships of the Tamarack North Project

Area	Mineral Zone	Host Lithology	Project Specific Lithology	Mineralization Type
Tamarack Zone	SMSU	Feldspathic Peridotite	CGO	Net textured and disseminated sulphides
	MSU	Meta-Sediments/ Peridotite (basal FGO mineralization)	Sediments	Massive sulphides
	138	Peridotite and Feldspathic Peridotite	MZ/FGO	Disseminated and net textured sulphides
	CGO Bend	Feldspathic Peridotite	CGO	Disseminated sulphides
		Peridotite footwall (basal FGO mineralization)	FGO	MMS and MSU
Other	221 Zone	Feldspathic Peridotite	CGO	Disseminated sulphides with ripped up clasts of massive sulphides
	480 Zone	Peridotite	FGO	Disseminated sulphides
	164 Zone	Peridotite	FGO	Blebby sulphides, sulphides veins

1.5 Exploration Programs

The TIC and associated mineralization were discovered as part of a regional program by Kennecott initiated in 1991. The focus on Ni and Cu sulphide mineralization was intensified in 1999 based on a model proposed by Dr. A. J. Naldrett of the potential for smaller feeder conduits associated with continental rift volcanism and mafic intrusions to host Ni sulphide deposits similar to Norilsk and Voisey's Bay.

Disseminated mineralization was first intersected at the Tamarack Project in 2002, and the first significant mineralization of massive and net-textured sulphides was intersected in 2008 at the Tamarack North Project.

To date, exploration has included a wide range of geophysical surveys including: airborne magnetic and electromagnetic (EM-MEGATEM and AeroTEM), ground magnetic, surface electromagnetic (EM) and magneto-telluric (MT), induced polarization (IP), gravity, seismic, Mise-à-la-masse (MALM) and downhole electromagnetic (DHEM). Kennecott conducted extensive drilling at the Tamarack North Project since 2002. This drilling has comprised 246 diamond drill holes totalling 102,358 m with holes between 33.5 m and over 1,224 m depth for an average hole depth of 428 m.

1.6 Sample Preparation, Quality Assurance (QA)/ Quality Control (QC) and Security

Golder reviewed Kennecott's sampling and QA/QC protocols along with the chain of custody of samples. Kennecott samples core continuously through the mineralization, and their sampling and logging procedures are consistent with industry standards and the assay methods are appropriate for the base metal sulphide mineralization found at the Tamarack North Project.

Their QA/QC program is based on insertion of certified reference materials (CRM), including a variety of standards, blanks and duplicate samples, used to monitor the precision and accuracy of their primary assay lab, and to prevent inaccurate data from being accepted into their assay database. The Kennecott QA/QC protocol is consistent with industry best practises.

Kennecott uses a system of metal seals to secure pails used to ship samples from the core shack to the assay lab ensuring that they have not been tampered with. Samples are prepared and stored in a secure facility and are monitored each step of the way to the lab. Golder is confident that the samples accurately reflect the mineralization and that there is little opportunity for samples to be tampered with. All procedures were found to meet or exceed industry standard practices.

1.7 Data Validation

Golder compared updated assay data (2017) from the Kennecott database to the original assay certificates from ALS Chemex for the entire sample population used for resource estimation. No errors were identified during this review.

During the qualified person (QP) site visit in 2014, Brian Thomas of Golder, surveyed four drill hole collars and then compared the coordinates to those provided by Kennecott. All collars were found to be consistent with the Kennecott collar coordinates, within the accuracy of the handheld global positioning system (GPS).

Golder, in 2014, conducted verification sampling of drill core from each of the three mineral domains. A total of nine samples were taken along with three additional CRM samples, including two standards and one blank. Assay values from the verification sample program were consistent with results obtained by Kennecott.

There have been no changes to the drilling, logging, sampling, or chain of custody procedures since the 2014 site visit; therefore, Golder has concluded that the Tamarack North Project drill hole database is of suitable quality to support the Mineral Resource estimate in this PEA.

1.8 Mineral Processing and Metallurgical Testing

Metallurgical testing of the Tamarack North Project was carried out in three main programs:

- The 2006 - 2010 program evaluated high-grade mineralization of SMSU hosted in CGO and low-grade mineralization from the CGO Zone;
- The 2012 - 2013 program focussed only on low-grade CGO mineralization;
- In 2016/2017 a total of seven domain composites were subjected to a metallurgical test program. Samples were selected from:
  - The MSU;
  - High-grade mineralization from the SMSU hosted in CGO;
  - Low-grade mineralization from the Lower and Upper 138 Zone;

- Low-grade mineralization from the CGO;
- Low-grade mineralization from the Upper CGO;
- Mixed massive sulphide (MMS) mineralization and an FGO interval above the MMS mineralization in the CGO Bend.

Head assays from all three phases of test work indicated no problematic concentrations of deleterious material, such as talc and chlorite, in the MSU and SMSU composites.

All samples were submitted to SGS Minerals Services for mineralogical and/or metallurgical testing.

In all cases the goal was to develop a process flowsheet that ultimately produces separate saleable Cu and Ni concentrates.

Test program results prior to the 2016/2017 program are summarized in the First Independent Technical Report on the Tamarack North Project with an effective date of August 29, 2014.

The primary objectives of the 2016/2017 test program were to:

- Obtain a flowsheet and test conditions suitable to treat the full range of MSU, SMSU, and disseminated mineral domains;
- Define expected recoveries over a wide spectrum of feed grades;
- Understand if there will be any synergies by blending low-grade domains with high-grade domains.

A total of 77 open circuit tests and seven locked cycle tests (LCTs) were carried out. The LCT results were used to develop metallurgical regression curves that can be used to project metal recoveries into the Cu and Ni concentrates. The head grades of the seven composites ranged between 0.31% to 2.80% Cu and 0.45% to 6.39% Ni.

Bond ball mill grindability tests produced work indices between 11.3 kilowatt-hours per tonne (kWh/t) for the MSU composite and 21.1 kWh/t for the CGO composite.

The test program culminated in a flowsheet and conditions that improved the flotation response of the disseminated composites compared to previous metallurgical programs. Furthermore, preliminary testing suggested that blending of MSU/SMSU and disseminated material responded better in the Cu-Ni separation circuit than the sum of the individual responses.

The MSU and SMSU composites produced high-grade Ni and Cu concentrates as well as very high recoveries, but the concentrates of the disseminated composites required blending with the higher-grade products to render reasonable concentrate grades. The Cu concentrates of the MSU and SMSU composites yielded grades of 31.6% Cu and 29.3% Cu at Cu recoveries of 91.4% and 84.0%, respectively. The Ni concentrates of the two high-grade composites graded 14.1% Ni to 17.1% Ni at 87.9% to 91.9% Ni recoveries, respectively. An additional 6.6% and 11.7% of the payable Cu units were recovered into the Ni concentrates of the MSU and SMSU composites. The total payable Cu recoveries were therefore 98% and 95.7% for the MSU and SMSU composites respectively. For the Ni concentrate, the Cu:Ni ratio of 0.03 for the MSU composite and 0.06 for the SMSU composite were well below the typical smelter requirement of less than 0.2 Cu:Ni. For the Cu concentrate, the Ni grades in the Cu concentrate were 1.53% Ni for the MSU composite and 0.95% Ni for the SMSU composite. The goal is to produce a Cu concentrate with <0.7% Ni. Flotation conditions and grind size in the Cu/Ni separation circuit have not been optimized. It is anticipated that optimized conditions to be developed in the next phase of testing will lower the Ni concentration in the Cu concentrate.

Levels of deleterious elements in the MSU and SMSU composites were consistently low. Magnesium (Mg) concentrations in Ni concentrate of MSU and SMSU composites were 0.22% magnesium oxide (MgO) and 3.20% MgO, respectively. Ni smelters generally desire Mg contents below 5.0% MgO in Ni concentrates and, thus, these results are satisfactory.

Iron (Fe):MgO ratios were 13.4:1 for the SMSU composite and 212:1 for the MSU composite.

Table 1-2 below summarizes the results of the 2016/2017 metallurgical testing program for the MSU and the SMSU composites.

Table 1-2: Summary of the 2016/2017 Metallurgical Testing Results for the MSU and SMSU Composites

Mineral Zone	Description	Assay (%)					Fe:MgO	Recovery (%)		
		Cu	Ni	Fe	MgO	S	Ratio	Cu %	Ni %	S %
MSU	Head (reconstituted)	2.75	6.31			25.8				
	Cu Concentrate	31.6	1.53	33.9		35.4		91.4	1.9	10.9
	Ni Concentrate	0.54	17.1	46.6	0.22	35.7	212	6.6	91.9	47.0
	Total Recovery							**98.0**	**93.8**	**57.9**
SMSU	Head (reconstituted)	1.51	3.11			13.6				
	Cu Concentrate	29.3	0.95	32.4		32.4		84.0	1.3	10.4
	Ni Concentrate	0.91	14.1	42.9	3.20	30.7	13.4	11.7	87.9	44.0
	Total Recovery							**95.7**	**89.2**	**54.4**

The Upper 138 Zone composite was the worst performer of the disseminated composites with only 51.7% Cu recovery into a Cu concentrate grading 14.5% Cu. The Ni concentrate of the LCT with the Upper 138 Zone composite graded 5.88% Ni at a low Ni recovery of 46.3%. The metallurgical performance of the other four disseminated composites fell between the results of the Upper 138 Zone and the SMSU composites. The Ni concentrates of the disseminated composites contained up to 14.6% MgO and alternative gangue depressants should be evaluated during the next phase of testing.

None of the low-grade material is included in the mine plan that is presented in this study.

1.9 Mineral Resource Estimate

Caution to readers: *In this Section, all estimates and descriptions related to Mineral Resource Estimates are forward-looking information. There are many material factors that could cause actual results to differ from the conclusions, forecasts or projections set out in this item. Some of the material factors include differences from the assumptions regarding the following: estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, commodity prices or product value, mining and processing methods and general and administrative (G&A) costs. The material factors or assumptions that were applied in drawing the conclusions, forecasts and projections set forth in this Item are summarized in other Items of this report.*

This Resource Estimate has been prepared by Mr. Brian Thomas (B.Sc, P.Geo), Senior Resource Geologist at Golder and is summarized in Table 1-3 below. The effective date of the resource estimate is February 15, 2018. Mr. Brian Thomas is an independent QP pursuant to NI 43-101.

Table 1-3: Tamarack North Project Mineral Resource Estimate (February 15, 2018)

Domain	Resource Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	*Calc NiEq (%)
SMSU	Indicated Resource	3,639	1.83	0.99	0.05	0.42	0.26	0.2	2.45
Total	**Indicated Resource**	**3,639**	**1.83**	**0.99**	**0.05**	**0.42**	**0.26**	**0.2**	**2.45**
SMSU	Inferred Resource	1,107	0.9	0.55	0.03	0.22	0.14	0.12	1.25
MSU	Inferred Resource	570	5.86	2.46	0.12	0.68	0.51	0.25	7.24
138 Zone	Inferred Resource	2,705	0.95	0.74	0.03	0.23	0.13	0.16	1.38
Total	**Inferred Resource**	**4,382**	**1.58**	**0.92**	**0.04**	**0.29**	**0.18**	**0.16**	**2.11**

All resources reported at a 0.83% NiEq cut-off.
No modifying factors have been applied to the estimates.
Tonnage estimates are rounded to the nearest 1,000 tonnes.
Metallurgical recovery factored in to the reporting cut-off.
**Where used in this Mineral Resource estimate, NiEq% = Ni%+ Cu% x $3.00/$8.00 + Co% x $12.00/$8.00 + Pt [g/t]/31.103 x $1,300/$8.00/22.04 + Pd [g/t]/31.103 x $700/$8.00/22.04 + Au [g/t]/31.103 x $1,200/$8.00/22.04*

The Mineral Resources are derived from a Datamine constructed block model (block sizes = 7.5 m by 7.5 m by 7.5 m for the SMSU and the 138 Zone; 3 m x 3 m x 1.5 m for the MSU) of three mineral domains and are reported above an equivalent nickel (NiEq) cut-off of 0.83%. All domains were "unfolded" and had top cuts applied to restrict outlier values (platinum (Pt), palladium (Pd) and Au). The three domains (Figure 14-1) utilized either Ordinary Kriging (OK) or inverse distance cubed (ID^3) methodology to interpolate grades (Ni, Cu, Co, Pt, Pd and Au) from 1.5 m composited drill holes. Density values were based on specific gravity (SG) measurements taken from whole core and where absent, regression formulas. The resources reported are based on a "blocks above cut-off" basis and were then examined visually by Golder and found to have good continuity.

Golder is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or any other potential factors that could materially impact the Tamarack North Project Mineral Resource estimate provided in this PEA. The resource is located in designated wetlands but this is not expected to affect future permitting.

1.10 Mining Methods

Three mining access methods were studied:

- Ramp Box Cut Method;
- Ramp Freeze Wall Method;
- Shaft Method.

Access through a vertical shaft was chosen as further discussed in Section 16.8.1.

The Tamarack deposit will be mined using underground mining methods. Mine development and operation costs assume contractor rates. Different underground mining methods will be utilized for the SMSU (consisting of an Upper and Lower SMSU) and the MSU.

The Upper and Lower SMSU will utilize transverse long hole open stoping with a delayed cemented paste backfill sequence.

The MSU zone will utilize overhand, transverse drift-and-fill with a delayed cemented paste backfill sequence.

Paste backfill will be used for the backfilling requirements of the Tamarack North Project for ground stability, increased ore recovery, and to minimize the amount of tailings stored on surface. The paste plant, which will be constructed on surface, will return 100% of the HS tailings back underground, as well as 37% of the LS tailings, which will eliminate the need to store these materials at the surface.

The planned production rate for the Tamarack North Project is 1,390 tonnes per day (tpd) of ore, which was shown to be sustainable for this type of deposit.

A mine maintenance and service area will be excavated at the first mine level for basic maintenance and service of underground equipment. Major components will be brought to surface for repair at contractor maintenance shops or sent to mine equipment supplier shops.

Based on a production rate of 1,390 tpd of ore approximately 220 people will be required for the underground operation.

1.11 Recovery Methods

The process plant design is based on an average daily mill feed rate of 1,390 tpd and an average Life of Mine (LOM) head grade of 3.09% Ni and 1.42% Cu. The plant feed characteristics and metallurgical performance is summarized in Table 1-4.

Table 1-4: Plant Feed Characteristics and Metallurgical Performance

Criteria	Units	Value		Source
		Expected/Avg.	Design	
Solids Density	t/m^3	2.90 – 3.75	3.08	D
Bulk Density	t/m^3	1.60 – 2.00	1.80	B
LOM Mill Head Grade	% Ni	1.98 – 5.97	3.09	D
LOM Mill Head Grade	% Cu	1.03 - 2.55	1.42	D
Mill Treatment Capacity	ktpa		507.3	C/D
Ni Recovery to Ni Concentrate	% Ni		85.0	E/C
Ni Concentrate Grade	% Ni		14.5	E/C
Ni Concentrate Production	ktpa		91.8	E/C
Overall Cu Recovery	% Cu		94.5	E/C
Recovery to Cu Concentrate	% Cu		84.4	E/C
Cu Concentrate Grade	% Cu		28.9	E/C
Cu Concentrate Production	ktpa		21.1	E/C

The metallurgical process consists of bulk rougher and scavenger flotation followed by separate cleaning of the rougher and scavenger concentrates. The upgraded rougher concentrate is subjected to Cu/Ni separation. The process generates separate Cu and Ni concentrates, which will be shipped to different smelters via rail in form of wet filter cake.

The bulk scavenger tailings are treated in a desulphurization stage to produce a low-mass HS stream and high-mass non-acid-generating (NAG) tailings. All the HS tailings will be placed underground in form of cemented paste backfill together with 37% of the LS tailings. The balance of the LS tailings will be placed in a CFTF.

The equipment that was selected for the processing plant represents well established technology such as a jaw and cone crusher, ball mill, tank flotation cells, and stirred media mills. Initial dewatering is performed in high-rate thickeners followed by filter presses for the two concentrates and a belt filter for the LS tailings stream.

The plant will employ a standard reagent suite consisting of sulphide collectors sodium isopropyl xanthate (SIPX) and potassium amyl xanthate (PAX), frother methyl isobutyl carbinol (MIBC), gangue depressant carboxy methyl cellulose (CMC), dispersant sodium metasilicate, and potential of hydrogen (pH) modifier lime. Flocculants will be employed to assist in the dewatering of the concentrates and tailings streams.

The total connected power is 4.0 megawatt (MW) with 80% drawn. It is assumed at this time that electrical power will be supplied through the electrical grid.

1.12 Project Infrastructure

The existing local transportation infrastructure is excellent. The site is accessible via an existing road which connects to the Minnesota State highway network.

The active Burlington Northern Santa Fe (BNSF) Railway passes by the town of Tamarack approximately 2.5 km S of the site layout area and connects to an extensive network of rail lines throughout the United States (US) and Canada, including access to the Duluth port.

The city of Duluth lies on the westernmost point of Lake Superior, and provides worldwide shipping access via the Great Lakes, St. Lawrence Seaway, and Atlantic Ocean shipping routes. For the benefit of the Tamarack Project, Kennecott has secured surface rights adjacent to the BNSF railway line to allow for the construction of a railroad siding near the site layout area, should this be required.

The Great River Energy Transmission Line crosses through the Tamarack North Project. The line connects through substations close to the nearby towns of Wright and Cromwell.

A conceptual site layout is shown in Section 18.3, comprising approximately 53 acres.

The CFTF will require approximately 25 acres. The hoist room, headframe, ore bins, conveyors, mineral processing facility and concentrates loadout as well as temporary development rock storage, water treatment facilities, workshops, vehicle washing bays, offices and parking areas comprise the remainder of the site layout area.

1.13 Capital Costs

Capital costs for the Tamarack North Project were estimated by DRA for the mine, process and surface facilities, and by Golder for the CFTF.

All cost estimates have been forecast in US dollars using constant, second quarter 2018 dollars, (i.e. in “real” dollars), without provision for inflation or escalation.

The total estimated capital cost is US$182.51M and is summarized in Table 1-5, of which US$174.31M is the initial cost required during the first 2.5 years prior to the start of production. The amounts include indirect costs and contingency. Contingency varies by line item and averages 20% for the initial cost of the mine and 23.5% for the initial cost of the process and surface facilities and totals US$29.38M.

Table 1-5: Tamarack North Project Capex Summary

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Mine	$72.44M	$21.38M	$93.83M
Process and Surface Facilities	$90.85M	$1.57M	$92.43M
Closure Costs	-	$6.25M	$6.25M
Salvage value of mill	-	($10.00M)	($10.00M)
Working capital	$11.01M	($11.01M)	-
Total*	**$174.31M**	**$8.20M**	**$182.51**

*May not total due to rounding

1.14 Operating Costs

The average operating costs per tonne of ore milled for the seven year life of the Tamarack North Project at the processing plant design capacity of 1,390 tpd are summarized in Table 1-6 below. In future studies, the processing costs may be reduced with a simplification of the process flowsheet.

Table 1-6: Operating Costs in US$/t of Mill Feed

Cost Category	Operating Cost (US$/t of ore milled)
Mining	$63.94
Processing	$18.87
Product Handling	$22.92
CFTF	$2.50
General & Administrative	$10.00
Total Opex	**$118.23**

1.15 Economic Analysis

DRA has prepared its assessment of the Tamarack North Project on the basis of a financial model, from which net present value (NPV), internal rate of return (IRR), payback and other measures can be determined. NPV and IRR can assist in the determination of the economic value and viability of a project.

Base case metal prices were based on analyst consensus long-term prices as well as current markets, forecasts and reports in the public domain. Alternate pricing scenarios were also considered.

Table 1-7: Base Case Metal Prices

	Unit	Base case
Ni	US$/lb	$8.00
Cu	US$/lb	$3.00
Co	US$/lb	$30.00
Pt	US$/oz	$1,100
Pd	US$/oz	$800
Au	US$/oz	$1,200

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the PEA will be realized.

The following table summarizes the base case LOM cash flow.

Table 1-8: Summary of Base Case Life of Mine Cash Flow

	LOM total (US$)	US$/tonne milled	US$/lb payable Ni
Payable nickel revenue	$1,020,771,336	$427.27	$8.00
Payable by-product revenue	$259,262,112	$108.58	$2.03
Total payable revenue	**$1,279,433,449**	**$535.85**	**$10.03**
Treatment and refining charges	$257,390,301	$107.80	$2.02
Insurance and losses	$1,635,269	$0.68	$0.01
Net smelter return	**$1,020,407,879**	**$427.37**	**$8.00**
Government and private royalties	$130,178,817	$54.52	$1.02
Transportation costs	$53,099,810	$22.24	$0.42
Net smelter return after royalties and transportation costs	**$837,129,251**	**$350.61**	**$6.56**
On-site costs			
Mining costs	$152,662,064	$63.94	$1.20
Processing costs	$45,055,324	$18.87	$0.35
Co-disposed Filtered Tailings Facility	$5,969,174	$2.50	$0.05
General & administrative costs	$23,876,695	$10.00	$0.19
Total on-site costs	**$227,563,258**	**$95.31**	**$1.78**
Net operating margin	**$609,565,994**	**$255.30**	**$4.78**
Capital expenditures	$182,505,450	$76.44	$1.43
Working capital	-	-	-
Net cash flow, before tax	**$427,060,543**	**$178.86**	**$3.35**
Corporate tax	$74,054,709	$31.02	$0.58
Net cash flow, after tax	**$353,005,834**	**$147.85**	**$2.77**

The following table provides the calculation of “C1 cash costs” and total costs.

Table 1-9: C1 Cash Cost and Total Cost

	LOM total (US$)	US$/tonne milled	US$/lb payable Ni
On-site costs	$227,563,258	$95.31	$1.78
Treatment and refining charges	$257,390,301	$107.80	$2.02
Insurance and losses	$1,635,269	$0.68	$0.01
Transportation costs	$53,099,810	$22.24	$0.42
Less: By-product revenue	($259,262,112)	($108.58)	($2.03)
C1 cash cost	**$280,426,526**	**$117.45**	**$2.20**
Government and private royalties	$130,178,817	$54.52	$1.02
C1 cash cost plus royalties	**$410,605,343**	**$171.97**	**$3.22**
Capital expenditures	$182,505,450	$76.44	$1.43
Total cost including CAPEX	**$593,110,793**	**$248.41**	**$4.65**

The base case cash flow, which is in real dollars, was evaluated by determining the after-tax NPV at a discount rate of 7.0% and the after-tax IRR as shown in Table 1-10. Results are also shown at comparative discount rates of 8% and 10% and on a pre-tax basis.

Table 1-10: Base Case NPV in Million US$ at Various Discount Rates and IRR

	Base case NPV discounted at			
	7%	**8%**	**10%**	**IRR**
Pre-tax	261	244	212	44.6%
After-tax	210	195	168	38.8%

The undiscounted pre-tax payback period is 1.9 years from the production start date in the third quarter of year one which along with other payback measures is included in the table that follows.

Table 1-11: Payback Period in Years from Production Start Date

	Undiscounted	Discounted
Pre-tax	1.9	2.1
After-tax	2.1	2.4

1.16 Conclusions

The present mine plan includes only a portion of the upper SMSU, the lower SMSU and the MSU.

The PEA is positive under a nickel (Ni) and copper (Cu) price scenario of $6.75/lb and $2.75/lb, respectively (28% after-tax IRR) with a base case IRR that ranks amongst the best globally (39% after-tax IRR). There are several opportunities to increase the Tamarack North Project NPV and therefore the following is recommended:

- Define a flowsheet and conditions capable of treating all of the MSU, SMSU, and 138 Zone mineralization while at the same time simplifying the present flowsheet;
- Increase the MSU mineral resource by exploring the open MSU extensions in the Tamarack Zone, the CGO Bend and potential MSU mineralization in the 164 Zone through geophysical and drilling methods;
- Using ore sorting to preconcentrate MSU by separating sediment/MSU and CGO/MSU midlings;
- Determine the optimal stope sizes in the SMSU;
- Update the production schedule to maximize early cash flows while maintaining a consistent plant feed.

1.17 Recommendations

A PFS should be completed once the extent of the mineralization, that will be accessed through the same infrastructure, has been delineated and a simplified flowsheet has been developed. Detailed study recommendations are noted in Section 26.

2. INTRODUCTION

DRA was retained by Talon to lead a consortium of consulting groups and to compile this independent PEA. The consortium consisted of: DRA, Golder, Barr, Metpro, and Paterson & Cooke. The purpose of this PEA is to support the disclosure of the results of a PEA for the Tamarack North Project, in accordance with NI 43-101 guidelines.

Three independent NI 43-101 Mineral Resource estimates for the Tamarack North Project have been prepared to date, each by Mr. Brian Thomas (B.Sc., P. Geo.), Senior Resource Geologist at Golder. The effective date of the most recent Mineral Resource estimate is February 15, 2018. All three Mineral Resource estimates were reviewed by Mr. Paul Palmer (P.Geo., P.Eng.). Mr. Brian Thomas completed a site visit to the Tamarack North Project on July 16, 2014. Mr. Thomas is a QP as defined by NI 43-101 guidelines.

A summary of the metallurgical test work completed on the Tamarack North Project has been compiled by Mr. Oliver Peters, P. Eng. Mr. Peters is the President and Chief Executive Officer (CEO) of Metpro. This work is an update of a previous summary of the metallurgical work completed on the Tamarack North Project by Mr. Manochehr Oliazadeh Khorakchy, P.Eng. of Hatch Ltd.

Talon is a TSX-listed company (symbol TLO) focused on the exploration and development of the Tamarack Project (which comprises the Tamarack North Project and the Tamarack South Project). Talon has a well-qualified exploration and mine management team with extensive experience in project management.

The PEA summarized in this Technical Report is considered by DRA to meet the requirements of a “Preliminary Economic Assessment” as defined in NI 43-101. The economic analysis contained in this PEA is preliminary in nature.

2.1 Sources of Information

The sources of information utilized in the preparation of this PEA were provided by Talon under the direction of Mr. James McDonald (P.Geo.), and by Kennecott under the direction of Mr. Robert Rush. This PEA is based on the following data and pre-existing reports:

- The 2014 Tamarack Earn-in Agreement (and all amendments thereto);
- The Original MVA;
- The 2018 Tamarack Earn-in Agreement;
- The New MVA;
- The Amended MVA;
- Tamarack Magmatic Nickel Copper Sulfide Due Diligence (Talon) report;
- Kennecott internal reports;
- Kennecott database of surface drill holes that included:
  - Ni, Cu, Co, Pt, Pd, Au, lithology sample/assay data;
  - Sample bulk density;
  - Drill hole collar survey data and down-hole survey data; and
  - QA/QC summary data and graphs.
- Assay certificates from ALS Chemex;
- Metal price assumptions based on an average of forecast long-term prices provided by major financial institutions located in North America and Europe.

Further sources of information utilized by the authors are listed in Section 27.

2.2 Qualified Persons (QPs)

This PEA was prepared by the QPs listed in Table 2-1. Certificates are also contained herein. The following QPs have completed property site visits:

- Brian Thomas, P. Geo., completed site visit on July 16, 2014;
- Tom Radue, P. Eng., completed site visit on August 5, 2018;
- Daniel Gagnon, P. Eng., completed site visit on October 5, 2017.

Table 2-1: List of Contributors

Name	Title, Company	Responsible for Section
Leslie Correia, Pr.Eng	Engineering Manager, Paterson & Cooke Canada Inc.	portion of 16
Silvia Del Carpio, P. Eng., MBA	Financial Analyst, DRA Americas Inc.	Sections 19 and 22
Tim Fletcher, P. Eng.	Project Manager, DRA Americas Inc.	2, portions of 1, 3, 21, 25, 26, and 27, and overall report compilation
Daniel Gagnon, P. Eng.	Vice President Mining and Geology, DRA Americas Inc.	18, and portions of 1, 3, 16, 21, 25, 26, and 27
Kebreab Habte, P. Eng.	Senior Geotechnical Engineer, Golder Associates Ltd.	portion of 18
Oliver Peters, P. Eng.	Consulting Metallurgist, Metpro Management Inc.	13, 17, and portions of 1, 21, 25, 26, and 27
Tom Radue, P.Eng.	Senior Geotechnical Engineer and Vice President, Barr Engineering	20, and portions of 1, 3, and 26
Brian Thomas, P. Geo.	Senior Resource Geologist, Golder Associates Ltd.	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, and portions of 1, 3, 25, 26, and 27

2.3 Units of Measure and Abbreviations

All units of measure used in this Technical Report are in the metric system, unless stated otherwise. Currencies outlined in the report are in US dollars unless otherwise stated.

The following symbols and abbreviations are used in this PEA.

<	Less than
>	Greater than
#	number
%	Percent
°	Degree
°C	Degrees Celsius
3D	Three dimensional
µm	Micron
ABA	Acid Base Accounting
ACT	Ace Core Orientation Tool
AEM	Airborne Electromagnetic

AERA	Air Emission Risk Analysis
Ag	Silver
Al	Aluminium
Al_2O_3	Alumina, aluminum oxide
AMT	Audio-frequency magneto-tellurics
AP	Acid Potential
ARD	Acid rock drainage
As	Arsenic
Au	Gold
Avg	Average
Azm	Azimuth
B.Sc	Bachelor of Science
Barr	Barr Engineering
BAT	Best Available Technologies
BH	Borehole
BHEM	Borehole electromagnetic
Bi	Bismuth
Bouguer	Regional earth gravity anomaly identified by height and bedrock corrections
BNSF	Burlington Northern Santa Fe (railway company)
BWi	Bond Work Index
$CaCO_3$	Calcium carbonate
CAPEX	Capital expenditure
Cd	Cadmium
CEO	Chief Executive Officer
cfm	cubic feet per minute
CFR	Code of Federal Regulations
CFTF	Co-disposed Filtered Tailings Facility
CGO	Coarse grained ortho-cumulate olivine
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
cm	Centimetre
cm^3	Cubic centimetre
CMC	Carboxy methyl cellulose
Co	Cobalt
Cpy	Chalcopyrite
Cr	Chromium
CRM	Certified reference material
CSAMT	Controlled source audio-frequency magneto-tellurics
.csv	Comma-separate values file (electronic file format)

Cu	Copper
$CuSO_4$	Copper sulphate
DHEM	Downhole Electromagnetic
DMS	Dense media separation
dmt	Dry metric tonnes
DNR	Department of Natural Resources
DRA	DRA Americas Inc.
E	Eastdhem
EAW	Environmental Assessment Worksheet
EDA	Exploratory data analysis
EIS	Environmental Impact Statement
EM	Electromagnetic
EMIT	Electromagnetic Imaging Technology
EPA	Environmental Protection Agency
EPS	Enhanced Production Scheduler
EV	Electric Vehicle
Fe	Iron
FGO	Fine grained ortho-cumulate olivine
Fo	Forsterite
ft	Feet
G&A	General and Administrative
g	Gram
g/cc	Gram per centimetre cube
g/t	Grams per tonne
Ga	Giga-annum (one billion years)
GCL	Geosynthetic Clay Liner
GLTZ	Great Lakes Tectonic Zone
Golder	Golder Associates Ltd
GOMS	Goals, Operators, Methods, Selection rules
gpm	gallons per minute
GPS	Global positioning system
HELP	Hydrogeologic Evaluation of Landfill Performance
Hg	Mercury
HLS	Heavy liquid separation
HQ	Hole (outside diameter): 96 mm; core (inside diameter): 63.5 mm
HS	High-sulphide
ICP	Inductively coupled plasma
ICP-AES	Inductively coupled plasma atomic emission spectroscopy

ICP-MS	Inductively coupled plasma mass spectroscopy
ID	Inverse distance
ID^2	Inverse distance squared
ID^3	Inverse distance cubed
In	Indium
IP	Induced polarization
IPD	Inverse power distance
IRR	Internal rate of return
ISO	International Organization for Standardization
ITH	in-the-hole
Kennecott	Kennecott Exploration Company
kg	Kilogram
kg/m^2	Kilograms per square metre
km	Kilometre
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hours per tonne
lb	Pound(s)
LCT	Locked cycle test
LHD	load-haul-dump
Li	Lithium
LLDPE	Linear low-density polyethylene
LME	London Metal Exchange
LOM	Life of Mine
LS	Low-sulphide
LV	Low voltage
M	Million
m	Metre
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metre per hour
Ma	Mega-annum
MALM	Mise-à-la-masse (test method)
mASL	Metres above sea level
MCR	Mid Continent Rift
MDH	Minnesota Department of Health
MEPA	Minnesota Environmental Policy Act
Metpro	Metpro Management Inc.

Mg	Magnesium
MgO	Magnesium oxide, magnesia
mGal	milligal
MGS	Minnesota Geological Survey
MIBC	Methyl isobutyl carbinol
mL	millilitre
ML	Metal leaching
mm	Millimetre
MMR	Magnetometric resistivity
MMS	Mixed massive sulphide
MNDNR	Minnesota Department of Natural Resources
Mo	Molybdenum
MOU	Memorandum of Understanding
MPCA	Minnesota Pollution Control Agency
MRV	Minnesota River Valley
MSO	Mineable Shape Optimizer
MSU	Massive sulphide unit
MT	Magneto-telluric
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVA	Mining Venture Agreement
MVI	Magnetization Vector Inversion
MW	Megawatt
MZ	Mixed zone
MZNO	Mixed zone olivine
n/a, N/A	Not applicable
N	North
NAAQS	National Ambient Air Quality Standards
NAG	Non-Acid Generating
NB	Nominal Bore
NE	Northeast
NEPA	National Environmental Protection Act
NESHAPS	New Source Performance Standards and National Emission Standards for Hazardous Air Pollutants
NI 43-101	National Instrument 43-101
Ni	Nickel
NiEq	Equivalent nickel
NN	Nearest neighbour

NP	Neutralization Potential
NPDES	National Pollutant Discharge Elimination System
NPR	Neutralization potential to acid potential
NPV	Net present value
NQ	Hole (outside diameter): 75.7 mm; core (inside diameter): 47.6 mmm
NRIA	Net Revenue Inflation Adjustment
NS	North-South
NSR	Net smelter return
NW	Northwest
NYMEX	New York Mercantile Exchange
OB	Overburden
OK	Ordinary Kriging
OPEX	Operating expense
oz	Ounce (troy ounce - 31.1035 grams)
P. Eng.	Professional Engineer
P. Geo.	Professional Geologist
PAG	Potentially Acid Generating
PAX	Potassium amyl xanthate
Paterson & Cooke	Paterson & Cooke Canada Inc.
Pb	Lead
Pd	Palladium
PEA	Preliminary Economic Assessment
PEM	Privacy enhanced mail (electronic file format)
PFS	Pre-Feasibility Study
PGE	Platinum group element
PGM	Platinum group metal
pH	potential of hydrogen (measure of acidity)
PLS	Pregnant leach solution
Pn	Pentlandite
Po	Pyrrhotite
PPI	Producer price inflation index
ppm	Parts per million
psi	pounds per square inch
Pt	Platinum
QA	Quality assurance
QC	Quality control
QCu	Density-weighted copper grade
QCo	Density-weighted cobalt grade

QEMSCAN	Quantitative Evaluation of Materials by Scanning Electron Microscope
QNi	Density-weighted nickel grade
QP	Qualified Person
Re	Rhenium
RGU	Responsible Government Unit
RMR	Rock mass rating
RTE	Rare, Threatened and Endangered
ROD	Record of Decision
ROFR	Right of first refusal
ROM	Run of mine
RTE	Rare, threatened, and endangered
S	South
S	Sulphur
Sb	Antimony
SDD	Scoping Decision Document
SDS	State Disposal System
SE	Southeast
Se	Selenium
SED	Sedimentary
SG	Specific gravity
SHPO	State Historic Preservation Office
SIPX	Sodium isopropyl xanthate
SMSU	Semi-massive sulphide unit
SPLP	Synthetic Precipitation Leaching Procedure
STP	Step data
.stp	Step file (electronic file format)
SW	Southwest
SWPPP	Stormwater Pollution Prevention Plans
SX	Solvent extraction
t/m^3	Tonnes per cubic metre
Talon	Talon Metals Corp.
TDEM	Time domain electromagnetic
Te	Tellurium
TEM	Transient electromagnetic
TIC	Tamarack Intrusive Complex
Tl	Thallium
tpa	Tonnes per annum
tph	Tonnes per hour

tpd	Tonnes per day
TSF	Tailings Storage Facility
U-Pb	Uranium-Lead
UCS	Uniaxial compressive strength
US	United States
US$	United States Dollars
USACE	US Army Corps of Engineers
UTEM	University of Toronto Electromagnetic System
UTM	Universal Transverse Mercator (coordinate system)
VOXI	Cloud based 3-D Inversion Service (Geosoft)
VPmg	3D modeling and inversion program for gravity, gravity-gradient, TMI and magnetic gradient data
W	West
w/w	By weight
WCA	Wetlands Conservation Act
wmt	Wet metric tonne
Zn	Zinc

3. RELIANCE ON OTHER EXPERTS

This PEA has been prepared by DRA, Golder, Barr, Metpro, and Paterson & Cooke for Talon. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to DRA, Golder, Barr, Metpro, and Paterson & Cooke at the time of report preparation;
- Assumptions, conditions, and qualifications as set forth in this report; and
- Data, reports, and other information supplied by Talon and other third-party sources.

In Sections 4.2 (Property Ownership), 4.3 (Permitting) and 4.4 (Environmental) of this PEA, the QPs have relied upon, and believe there is a reasonable basis for this reliance on, information provided by Talon regarding mineral tenure, surface rights, ownership details, the 2014 Earn-in Agreement, the Original MVA, the 2018 Earn-in Agreement, and other agreements relating to the Tamarack North Project, royalties, environmental obligations, permitting requirements and applicable legislation relevant to the Tamarack North Project. The QPs have not independently verified the information in these sections and have fully relied upon, and disclaim responsibility for, information provided by Talon in these sections.

DRA has relied upon data and documentation from Talon in respect of Market Studies (Section 19) and Economic Analysis (Section 22) of this PEA. DRA believes that information supplied by Talon is reasonable but DRA has not verified this data.

For Environmental Studies, Permitting and Community Impact (Section 20) and associated sub-sections, Barr has relied upon information provided by Talon (as generated by third party sources) for baseline data related to site hydrogeology, hydrology, geochemistry, wetlands, vegetative communities, and protected species. Barr used information from these studies and has not performed a detailed independent review of study methods or results. Barr has fully relied on the provided information for these sections and disclaims responsibility for information provided by Talon in these sections as it relates to the aforementioned studies. For Permitting Requirements (Section 20.7), Barr relied upon

Federal, State, and local regulations as well as institutional knowledge gained from developing similar mining projects in northern Minnesota.

Metpro has relied upon data and documentation from Talon with regards to the metallurgical results that were obtained in a recent test program at SGS Lakefield as well as historic metallurgical data. This information was used in the generation of Mineral Processing and Metallurgical Testing (Section 13) and of the process design criteria. The process design criteria provided critical input for Recovery Methods (Section 17) and Capital and Operating Costs (Section 21) of this PEA.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Tamarack Project located in north-central Minnesota is approximately 100 km (62 miles) W of Duluth and 210 km (130 miles) N of Minneapolis, in Aitkin County Figure 4-1). The Tamarack North Project which this report represents, covers approximately 20,320 acres. The boundary between the Tamarack North Project and the Tamarack South Project is located approximately along the 5165000 N Universal Transverse Mercator (UTM) line. More specifically, it occurs along the southern extremity of State Mineral Leases MM-10006-N, MM-9768-P, and MM-9767-P (Figure 4-2). The current Tamarack North Project mineralization is centred at approximately 490750 E/5168700 N NAD 83 15 N. The town of Tamarack, which gives the project its name, lies in the southern portion of the Tamarack North Project area (though away from the known mineralization).

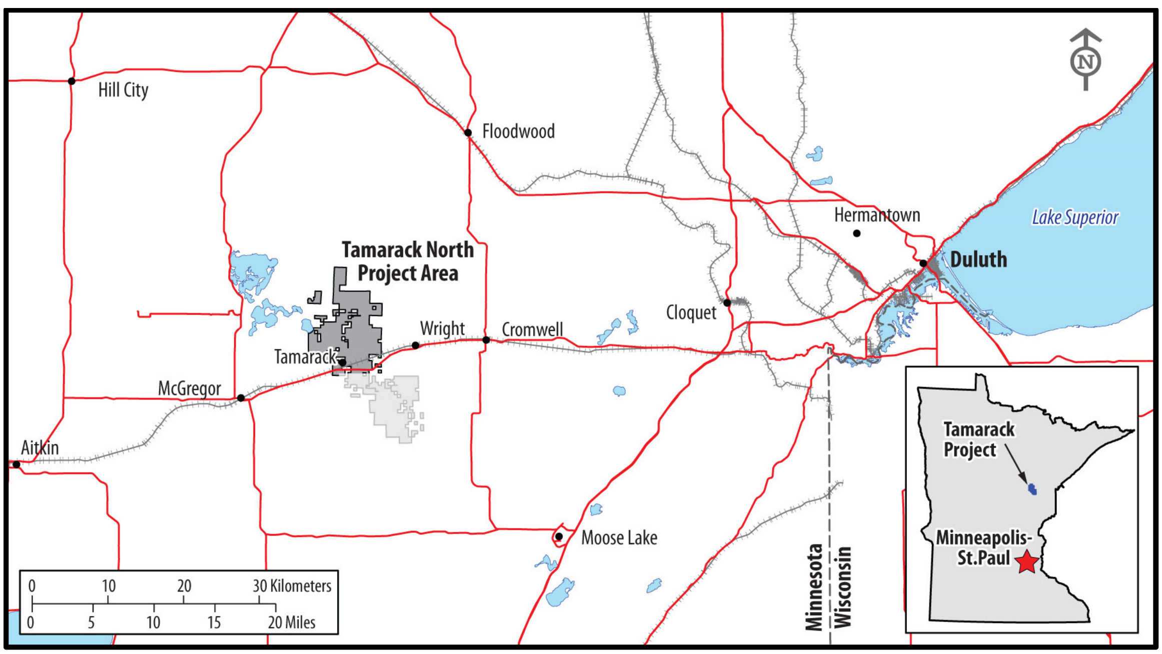


Figure 4-1: Location of the Tamarack North Project

4.2 Property Ownership

Both Kennecott and Talon hold interests in the Tamarack Project, which comprises the Tamarack North Project and the Tamarack South Project. As of the date of this PEA, Talon holds a 17.56% interest, and Kennecott holds an 82.44% interest, in the Tamarack Project.

On November 7, 2018, Talon and Kennecott entered into the 2018 Tamarack Earn-in Agreement pursuant to which Talon has the right to increase its interest in the Tamarack Project to a maximum 60% interest and become the manager/operator of the Tamarack Project. The 2018 Tamarack Earn-in Agreement is subject to approval by the Talon shareholders of a financing required to be completed by Talon in connection with the Tamarack Project, such shareholder approval to be sought at a meeting to be held on or before January 31, 2019. The 2018 Tamarack Earn-in Agreement is described in Section 4.2.3 below.

Prior to the 2018 Tamarack Earn-in Agreement, the relationship between Talon and Kennecott was governed by a number of other agreements (2014 Tamarack Earn-in Agreement, Original MVA, etc.), which are further described below.

4.2.1 2014 Tamarack Earn-in Agreement

On June 25, 2014, Talon entered into the 2014 Tamarack Earn-in Agreement with Kennecott, part of the Rio Tinto Group, pursuant to which Talon was granted the right to acquire an interest in the Tamarack Project.

Pursuant to the original terms of the 2014 Tamarack Earn-in Agreement, Talon had the right to acquire a 30% interest in the Tamarack Project over a three-year period (the Earn-in Period) by making US$7.5M in installment payments to Kennecott, and incurring US$30M in exploration expenditures (the Tamarack Earn-in Conditions). In addition, Talon agreed to make certain land option payments on behalf of Kennecott, which were payable over the Earn-in Period (and, when payable, were to be included as part of the Tamarack Earn-in Conditions).

On March 26, 2015, Kennecott and Talon amended the 2014 Tamarack Earn-in Agreement (the 2014 Tamarack Earn-in First Amending Agreement) to defer one of the option payments (the Deferred Option Payment) and delay further cash calls from being made by Kennecott.

On November 25, 2015, Kennecott and Talon entered into a further agreement to amend the 2014 Tamarack Earn-in Agreement (the 2014 Tamarack Earn-in Second Amending Agreement), to provide, among other things:

- That upon receipt by Kennecott from Talon of the sum of US$15M (which was in addition to previous amounts paid to Kennecott of US$10.52M), Talon would earn an 18.45% interest in the Tamarack Project and Talon would have no further funding requirements to earn its interest in the Tamarack Project;
- Once Kennecott had spent the funds advanced by Talon on exploration activities in respect of the Tamarack Project, subject to certain self-funding rights by Kennecott during such period, Kennecott would have 180 days to elect whether to: (a) proceed with an 81.55/18.45 joint venture with respect to the Tamarack Project in accordance with the terms of a MVA, with Kennecott owning an 81.55% participating interest, and Talon owning an 18.45% participating interest; or (b) grant Talon the right to purchase Kennecott's interest in the Tamarack Project for a total purchase price of US$114M (the Tamarack Purchase Option). In the event Kennecott granted Talon the Tamarack Purchase Option, and Talon elected to proceed with the Tamarack Purchase Option, Talon would have up to 18 months to close the transaction, provided it made an upfront non-refundable payment of US$14M; and
- Until Kennecott made its decision as to whether to grant Talon the Tamarack Purchase Option, Talon would be responsible for certain costs to keep the Tamarack Project in good standing based on its 18.45% interest. If Talon failed to make any of such payments, its interest in the Tamarack Project would be diluted in accordance with the terms of the Tamarack Earn-in Agreement.

On January 4, 2016, Talon made the US$15M payment to Kennecott (the Final 2014 Earn-in Payment) and earned an 18.45% interest in the Tamarack Project.

The total amount paid by Talon to Kennecott to earn its 18.45% interest in the Tamarack Project was US$25,520,800, broken down as follows:

Option payments	$ 1,000,000
Exploration	21,200,000
Land purchases	3,320,800
	$ 25,520,800

On December 16, 2016, Talon entered into a third amending agreement with Kennecott (the 2014 Tamarack Earn-in Third Amending Agreement) in respect of the 2014 Tamarack Earn-in Agreement.

Pursuant to the 2014 Tamarack Earn-in Third Amending Agreement, Talon and Kennecott agreed to co-fund a 2016/2017 winter exploration program at the Tamarack Project in the approximate amount of US$3.5M, with Talon funding its proportionate share of 18.45% thereof. The 2014 Tamarack Earn-in Third Amending Agreement also provided that Kennecott could elect at any time up to and including September 25, 2017 to grant Talon the Tamarack Purchase Option or proceed with the Original MVA (the Kennecott Decision Deadline).

On the Kennecott Decision Deadline, Talon received notification from Kennecott that it had decided to grant Talon the Tamarack Purchase Option on the terms of the 2014 Tamarack Earn-in Agreement. Pursuant to the 2014 Tamarack Earn-in Agreement, Talon had until November 6, 2017 to advise Kennecott as to whether or not it would exercise the Tamarack Purchase Option.

On November 1, 2017, Talon entered into a fourth amending agreement with Kennecott (the 2014 Tamarack Earn-in Fourth Amending Agreement) in respect of the 2014 Tamarack Earn-in Agreement. Pursuant to the 2014 Tamarack Earn-in Fourth Amending Agreement, Kennecott agreed to grant Talon an extension until December 31, 2017 to make its election as to whether it would exercise the Tamarack Purchase Option. In return for the granting of such extension by Kennecott, Talon agreed to grant Kennecott a 0.5% net smelter return (NSR) in the event Talon elected to exercise the Tamarack Purchase Option.

On November 16, 2017, Talon advised Kennecott that it had elected not to exercise the Tamarack Purchase Option. Consequently, under the terms of the 2014 Tamarack Earn-in Agreement, in February 2018 the parties were required to proceed to execute and deliver and operate under the Original MVA.

4.2.2 Original Mining Venture Agreement (Original MVA)

On January 11, 2018, Talon entered into a fifth amending agreement with Kennecott (the 2014 Tamarack Earn-in Fifth Amending Agreement) in respect of the 2014 Tamarack Earn-in Agreement. Pursuant to the 2014 Tamarack Earn-in Fifth Amending Agreement, Talon and Kennecott agreed to accelerate the timeframe for entering into the Original MVA, such that the parties would enter into the agreement with immediate effect (on January 11, 2018), rather than in February 2018.

Some notable characteristics of the Original MVA include the following:

- Kennecott was appointed Manager of the Tamarack Project, with a number of explicit duties and obligations articulated under the Original MVA;
- Talon and Kennecott agreed to establish a management committee to determine overall policies, objectives, procedures, methods and actions under the Original MVA, and to provide general oversight and direction to the manager who was vested with full power and authority to carry out day-to-day management under the Original MVA. The management committee consisted of two members appointed by Talon and two members appointed by Kennecott;

- Upon formation of the Original MVA, and beginning with the first program and budget under the Original MVA, each proposed program and budget had to provide for an annual expenditure of at least US$6.15M until the completion of a Feasibility Study (as defined under the Original MVA). The failure of either party to fund its share of each proposed program and budget was to result in dilution (and in certain circumstances accelerated dilution) in accordance with the terms of the Original MVA;
- In the event either party's participating interest in the Tamarack Project diluted below 10%, such party's interest would be converted into a 1% NSRs royalty; and
- In the event of a proposed transfer of either party's interest in the Tamarack Project to a third party, the other party had a right of first refusal (ROFR). In the event the non-transferring party elected not to exercise its ROFR, the non-transferring party had a tag-along right, while the transferring party had a drag-along right.

On January 11, 2018, pursuant to the terms of the Original MVA, Talon elected to not financially participate in the 2018 winter exploration program at the Tamarack Project. Consequently, Talon's interest in the Tamarack Project was diluted below 18.45%.

4.2.3 **2018 Tamarack Earn-in Agreement**

On November 7, 2018, Talon and Kennecott entered into the 2018 Tamarack Earn-in Agreement, pursuant to which Talon received the right to increase its interest in the Tamarack Project up to a maximum 60% interest. Under the 2018 Tamarack Earn-in Agreement, the Original MVA is in abeyance.

Pursuant to the 2018 Tamarack Earn-in Agreement, Talon initially has the right to increase its interest in the Tamarack Project to 51% by:

- Paying Kennecott US$6M cash and issuing US$1.5M worth of common shares in Talon to Kennecott on the effective date of the 2018 Tamarack Earn-in Agreement; and

- Within three years of the effective date of the 2018 Tamarack Earn-in Agreement, by Talon (a) incurring US$10M in exploration expenditures on the Tamarack Project, or (b) delivering a PFS in accordance with NI 43-101, whichever comes first; and
- Also within three years of the effective date of the 2018 Tamarack Earn-in Agreement, Talon paying Kennecott the additional sum in cash of US$5M.

In the event Talon successfully earns a 51% interest in the Tamarack Project, Talon will then have the right, within seven years of the effective date of the 2018 Tamarack Earn-in Agreement, to further increase its interest in the Tamarack Project to 60% by:

- Completing a Feasibility Study in accordance with NI 43-101; and
- Paying Kennecott an additional cash payment US$10M.

4.2.4 The New MVA

In the event Talon earns a 60% interest in the Tamarack Project, Talon and Kennecott have agreed to enter into a new mining venture agreement (the New MVA).

Some notable characteristics of the New MVA include the following:

- Talon will be appointed Manager of the Tamarack Project, with a number of explicit duties and obligations articulated under the New MVA;
- Each party will be required to find its *pro rata* share of expenditures or be diluted;
- Talon and Kennecott will establish a management committee to determine overall policies, objectives, procedures, methods and actions under the New MVA, and to provide general oversight and direction to the Manager who will be vested with full power and authority to carry out the day-to-day management under the New MVA. The management committee will consist of two members appointed by Talon and two members appointed by Kennecott;
- In the event either party's participating interest in the Tamarack Project dilutes below 10%, such party's interest will be converted into a 1% NSR;
- In the event of a proposed transfer of either party's interest in the Tamarack Project to a third party, the other party will have a ROFR.

4.2.5 Other Potential Agreements

In addition to the 2018 Tamarack Earn-in Agreement and the New MVA, Talon and Kennecott have contemplated two potential scenarios that would necessitate the entering into of alternative forms of MVAs.

First, in the event Talon does not earn a 51% interest in the Tamarack Project, the Original MVA will come back into force (excluding the requirement for an annual expenditure of at least US$6.15M until the completion of a Feasibility Study), with Kennecott once again taking on the role of the manager of the Tamarack Project, and Talon commencing with a 17.56% interest in the Tamarack Project.

Second, in the event Talon earns a 51% interest in the Tamarack Project, but does not earn a 60% interest in the Tamarack Project, the parties have agreed to enter into an amended mining venture agreement (Amended MVA) pursuant to which Talon will continue to be the Manager of the Tamarack Project, and will be required to free-carry Kennecott through to the completion of a Feasibility Study (as defined under the Amended MVA). Under the Amended MVA, and beginning with the first program and budget under the Amended MVA, each proposed program and budget by Talon must provide for an annual expenditure of at least US$6.15M until the completion of a Feasibility Study (as defined under the Amended MVA), failing which Talon will be subject to dilution.

4.2.6 Mineral Tenure

4.2.6.1 Introduction

Land in Minnesota is held by a combination of private, state and federal ownership. In addition, surface estate owner(s) may be the same or different to the mineral estate owner(s) (i.e., mineral interest may be severed from surface interest and form its own property ownership right).

The Tamarack North Project comprises:

- Minnesota State Leases (many of which also include the surface rights);
- Private Mineral Leases, Surface Use Agreements and Options to Purchase; and
- Fee Mineral and Surface Interests owned outright by Kennecott.

These various interests are summarized in Table 4-1. The mineral rights owned or controlled by Kennecott and Talon are summarized in Figure 4-2, and the surface rights owned or controlled by Kennecott and Talon are shown in Figure 4-3. All Tamarack North Project mineral and surface interests are held in Kennecott's own name and are currently subject to the 2018 Tamarack Earn-in Agreement.

Table 4-1: Summary of Tamarack North Project Interests

Type	Number	Acreage
Minnesota State Leases	40	18,730
Private Mineral Leases	3	154
Fee Minerals and Surface Interests	17	1,436
Total	**60**	**20,320**

It is noted that all locations for mineral leases and other property locations are described in the US Public Land Survey System in Township, Range, Section and Section subdivisions.

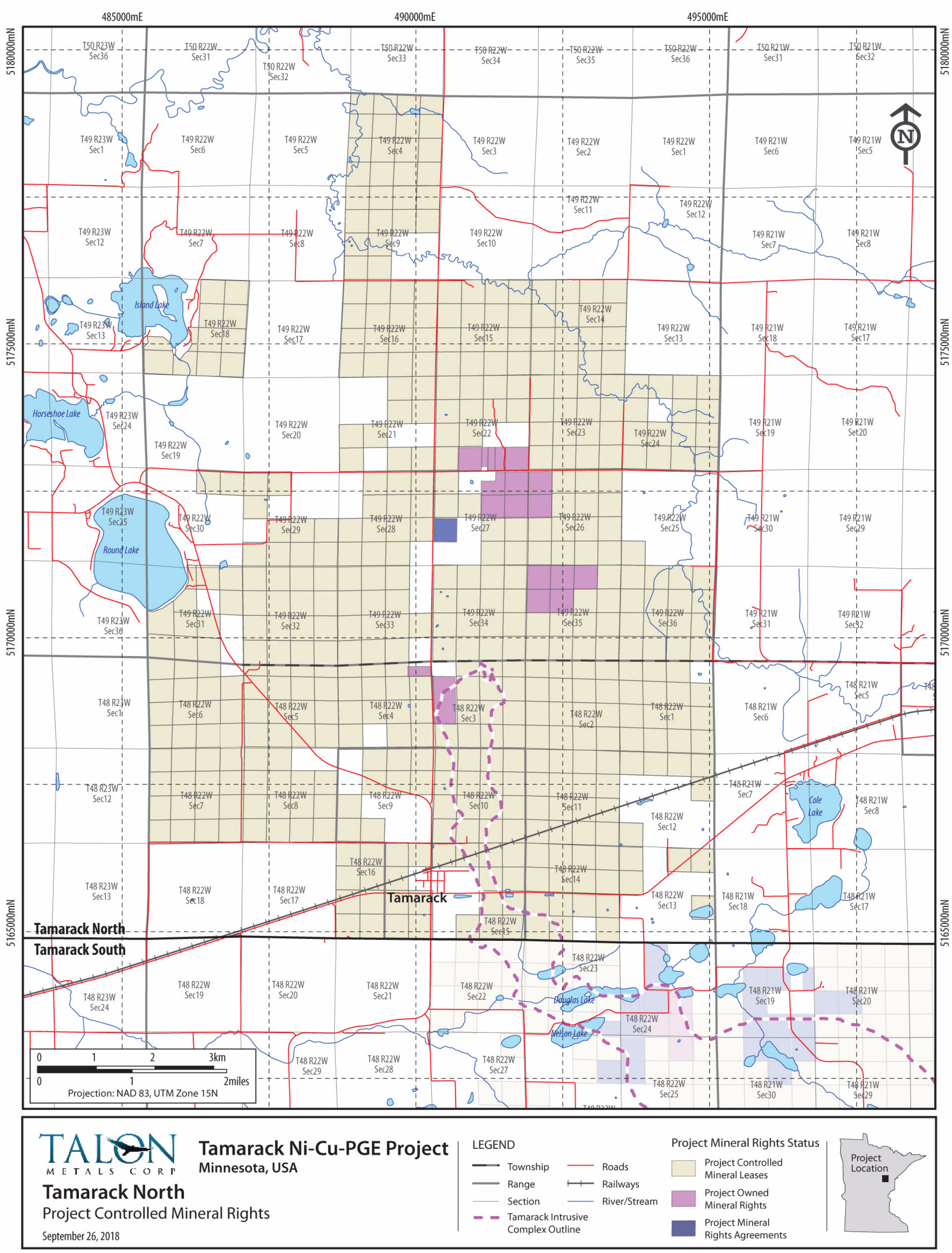


Figure 4-2: Tamarack North Project Mineral Rights

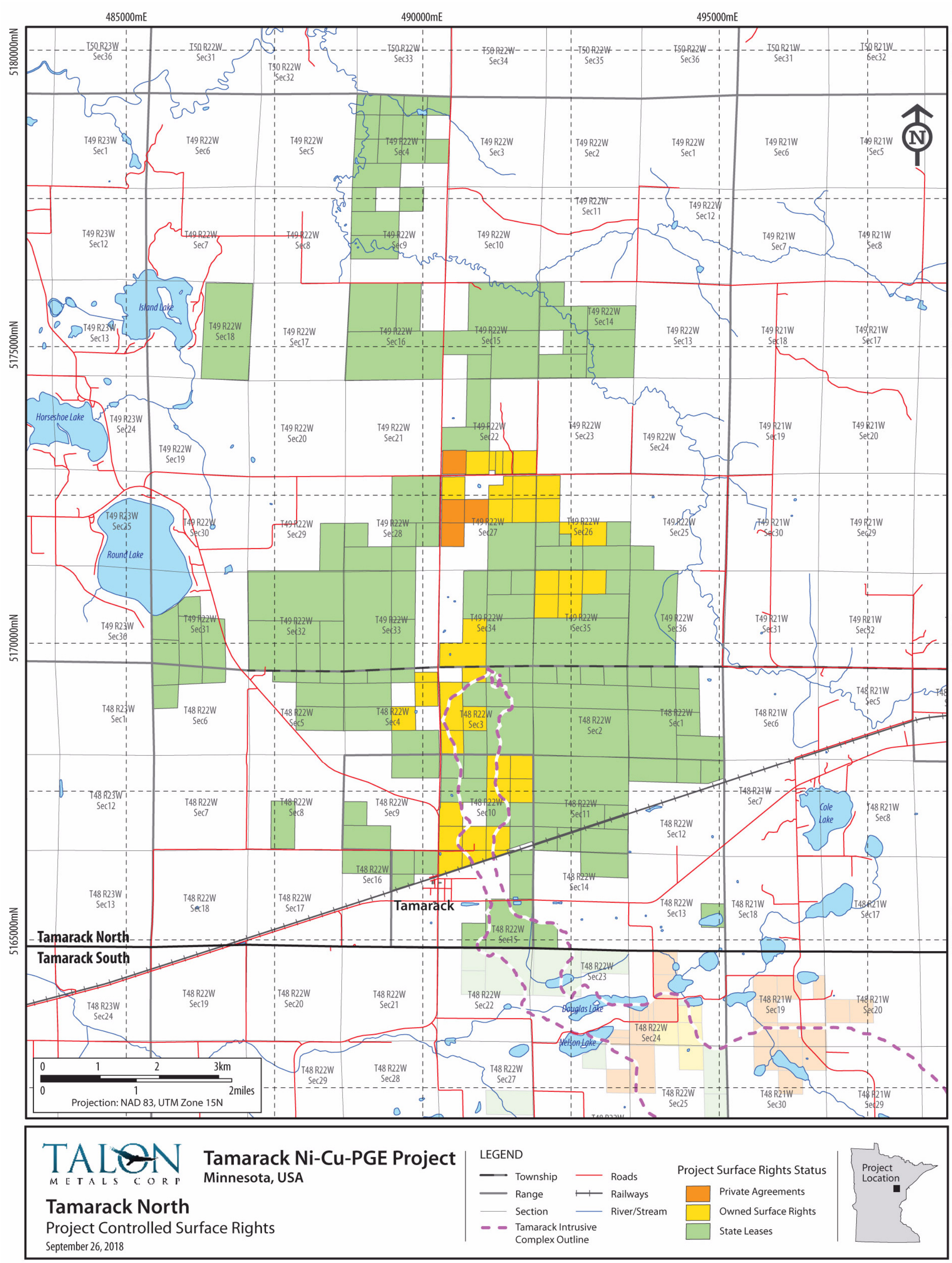


Figure 4-3: Tamarack North Project Surface Rights Status.

4.2.6.2 Minnesota State Leases

State Leases to Explore, Mine and Remove Metallic Minerals (State Leases) are issued by the MNDNR and may be held for up to 50 years. “Metallic Minerals” are defined in the State Leases as “any mineral substances of a metalliferous nature, except Fe ores and taconite ores”. State Leases allow a mining company to engage in mineral exploration and mineral development located on the State-owned property, subject to compliance with all laws and issued permits.

The Tamarack North Project comprises 40 State Leases, covering an area of approximately 18,730 acres (Table 4-2 contains further details of State Leases). The State Leases are issued on standard lease forms and generally contain uniform terms and conditions.

In order to keep the State Leases in good standing, certain quarterly and/or annual payments must be made to the State and/or County. Rental payments must be made to the State, and are paid quarterly in arrears on each February 20, May 20, August 20 and November 20 for the previous calendar quarter. The quantum of such rental payments are as follows:

- Initially, US$1.50 per acre for the unexpired portion of the then current year and US$1.50 per acre for each of the two succeeding years;
- US$5 per acre for the next three calendar years, payable quarterly;
- US$15 per acre for the next five calendar years, payable quarterly; and
- US$30 per acre per calendar year for the duration of the lease.

A county tax is also levied on the State Leases, with the current amount being US$0.40 per acre, payable on May 15 of each year.

An operating mining company must also pay a production royalty. The base royalty consists of a base rate (3.95%) and in some cases an additional royalty (applicable only to those leases acquired through state bids or negotiations with the State). Details are included in Table 4-2. State leases also contain a royalty escalation clause that increases the base royalty as the net return value per ton of raw ore increases. This escalation of the royalty rate begins at a net return value per ton of US$75.01. It rises to the maximum of 20% if such net return value exceeds US$444 per ton of raw ore.

The State of Minnesota has an option to cancel a mineral lease after the end of the 20th year if, by that time, a lessee is not actively engaged in mining ore under the lease from the mining unit, a mine within the same government township as the mining unit or an adjacent government township and has not paid at least US$100,000 to the State in earned royalty under a state lease in any one calendar year. The State must exercise that option within the 21st year of the lease. If the State does not cancel within the 21st year, the lessee has until the end of the 35th calendar year to meet the conditions. If the lessee has not met the conditions by the end of the 35th year, the State has another window to cancel the lease during the 36th calendar year of the lease.

Table 4-2: Tamarack North Project State Lease Details

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 9765-P	9/7/2000	50 years	3.95%	N/A	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 3: Lot 3, NE/4SW/4, SW/4SW/4 Minerals and mineral rights Sec. 3: Lots 1-2, S/2NE/4, SE/4NW/4, SE/4SW/4, SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	482.26
MM 9766-P	9/7/2000	50 years	3.95%	N/A	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 10: NE/4NW/4, S/2NW/4, NW/4SE/4 Minerals, mineral rights and surface Sec. 10: SW/4, NE/4 Minerals and mineral rights Sec. 10: NW/4NW/4, NE/4SE/4, S/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	640

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 9767-P	9/7/2000	50 years	3.95%	N/A	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 14: N/2NE/4 Minerals, mineral rights and surface Sec. 14: N/2SE/4, SE/4SE/4, S/2NE/4, NW/4, NE/4SW/4, NW/4SW/4 except 2.58 acres for highway right-of-way, E/2SE/4SW/4 Minerals and mineral rights Sec. 14: SW/4SW/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	577.42
MM 9768-P	11/9/2005	50 years	3.95%	N/A	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 15: SW/4NE/4, NE/4NW/4 except 3.17 acres for railroad right-of-way, NW/4NW/4 except 2.14 acres for railroad right-of-way Minerals and mineral rights Sec. 15: NE/4NE/4 except 0.80 acres for railroad right-of-way, NW/4NE/4 except 3.17 acres for railroad right-of-way, SE/4NE/4, SE/4SW/4, SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	430.72
MM 9849-N	9/6/2001	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 34: NE/4NE/4, E/2NW/4 Minerals, mineral rights and surface Sec. 34: W/2NW/4, NW/4NE/4, SW/4 Minerals and mineral rights Sec. 34: S/2NE/4, SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	640.00
MM 10002-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 2: Lots 1-4, S/2NE/4, S/2NW/4, S/2 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	605.04
MM 10003-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 4: SW/4NE/4, SE/4NE/4, SW/4SW/4, N/2SE/4 Minerals and mineral rights Sec. 4: Lots 2-4, S/2NW/4, N/2SW/4, S/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	505.85

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 10004-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 9: S/2NE/4, NE/4NW/4 Minerals and mineral rights Sec. 9: N/2NE/4; SE/4NW/4, that part commencing at NW corner, thence S along W line of SE/4NW/4 206 ft to Round Lake Road the point of beginning, thence S along same W line a distance of 427 ft, thence deflect left 73˚ a distance of 612.5 ft, thence deflect left 87˚ 10 minutes a distance of 400 ft to centre of Round Lake Road, thence deflect left 92˚ along said road a distance of 762 ft to point of beginning; W/2SW/4; SE/4SW/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	326.50
MM 10005-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 11: All Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	640.00
MM 10006-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 16: N/2NE/4, SW/4NE/4, W/2, SE/4 Minerals and mineral rights	600.00
MM 10007-N	6/5/2003	50 years	3.95%	0.40%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 27: W/2NW/4, SE/4 Minerals and mineral rights Sec. 27: SE/4NW/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	280.00
MM 10008-N	6/5/2003	50 years	3.95%	0.40%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 28: NE/4, NE/4SE/4, SW/4SE/4 Minerals, mineral rights and surface Sec. 28: E/2NW/4, NE/4SW/4 Minerals and mineral rights Sec. 28: W/2SW/4, SE/4SW/4, NW/4SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	520.00
MM 10009-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 33: N/2NE/4SE/4 Minerals and mineral rights Sec. 33: W/2NE/4, W/2, W/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	500.00

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 10010-N	6/5/2003	50 years	3.95%	0.30%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 35: E/2NE/4, SW/4NE/4, SW/4, NE/4SE/4 except coal and iron, NW/4SE/4 except coal and iron, SW/4SE/4 except coal and iron, SE/4SE/4 except coal and iron Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	440.00
MM 10202-N	6/21/2008	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 22: N/2SW/4 Minerals, mineral rights and surface Sec. 22: NW/4, SW/4SW/4, E/2NE/4 Minerals and mineral rights	360.00
MM 10203-N	6/21/2008	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 26: E/2NE/4, W/2NE/4, E/2NW/4, NE/4SW/4, NW/4SE/4 Minerals and mineral rights Sec. 26: W/2SW/4, SE/4SW/4, NE/4SE/4, S/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	560
MM 10204-N	6/21/2008	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 29: SW/4NW/4, E/2SW/4, SW/4SW/4, W/2SE/4, undivided ½ interest in N/2NW/4 Minerals and mineral rights Sec. 29: E/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	400.00
MM 10205-N	6/21/2008	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 32: E/2SE/4 Minerals, mineral rights and surface Sec. 32: N/2, SW/4, W/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	640.00
MM 10252-N	9/30/2009	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 22: W/2NE/4 Minerals and mineral rights, except coal and iron	80.00
MM 10253-N	9/30/2009	50 years	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 23: All Minerals and mineral rights, except coal and iron	640.00

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 10315	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 1: SE/4NE/4, NE/4SE/4 Minerals and mineral rights Sec. 1: Lots 2-4, SW/4NE/4, S/2NW/4, SW/4, W/2SE/4, SE/4SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	588.30
MM 10316	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 7: Lots 1-4, E/2, E/2NW/4, E/2SW/4 Minerals and mineral rights	626.07
MM 10317	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 8: E/2SW/4 Minerals, mineral rights and surface Sec. 8: S/2NE/4, NW/4, W/2SW/4, SE/4 Minerals and mineral rights	560.00
MM 10318	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 12: NW/4NE/4, N/2NW/4 Minerals, mineral rights and surface Sec. 12: SE/4NE/4, SW/4SW/4 Minerals and mineral rights Sec. 12: NE/4NE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	240.00
MM 10319	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 13: N/2NE/4, W/2NW/4 Minerals and mineral rights Sec. 13: NE/4SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	200.00
MM 10335	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 4: Lots 3-4, SW/4NW/4, NW/4SW/4, NE/4SE/4 Minerals, mineral rights and surface Sec. 4: SE/4NE/4, SE/4SE/4, SW/4SE/4 Minerals and mineral rights Sec. 4: Lots 1-2, SW/4NE/4, SE/4NW/4, NE/4SW/4, S/2SW/4, NW/4SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	610.96

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 10340	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 9: NE/4NE/4, SW/4NE/4 except the north 100 ft, SE/4NE/4 except the N 100 ft, NE/4NW/4, S/2SW/4 Minerals and mineral rights Sec. 9: NW/4NE/4, SW/4NE/4 the N 100 ft, SE/4NE/4 the N 100 ft, W/2NW/4, SE/4NW/4, N/2SW/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	480.00
MM 10344	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 18: Lots 3-6, N/2NE/4, SE/4NE/4, E/2SE/4 Minerals and mineral rights Sec. 18: SW/4NE/4, W/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	438.97
MM 10346	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 25: SW/4SW/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	40.00
MM 10347	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 30: N/2NE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	80.00
MM 10348	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 31: Lot 1, SE/4NE/4, undivided ½ interest in NE/4NE/4, undivided ½ interest in NW/4NE/4 Minerals and mineral rights Sec. 31: Lots 2-4, E/2SW/4, W/2SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	430.36
MM 10349	2/26/2010	50 years	3.95%	0.611%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 36: W/2 Minerals, mineral rights and surface Sec. 36: E/2 Minerals and mineral rights	640.00

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MM 10378-N	3/4/2011	50 years	3.95%	0.55%	Yes	**<u>Township 49 North, Range 22 West, Aitkin County, Minnesota</u>** <u>Sec. 14</u>: W/2NW/4, SE/4NW/4, NE/4SW/4, SW/4SW/4, SE/4SW/4 Minerals, mineral rights and surface <u>Sec. 14</u>: NW/4SW/4, NE/4NE/4 except the N 2 rods and the E 2 rods, NW/4NE/4, NE/4NW/4 Minerals and mineral rights <u>Sec. 14</u>: NE/4NE/4 the N 2 rods, NE/4NE/4 the E 2 rods except the N 2 rods, S/2NE/4, SE/4 Minerals and mineral rights, including the interest in the surface thereof owned by the State, if any	640.00
MM 10379-N	3/4/2011	50 years	3.95%	0.55%	Yes	**<u>Township 49 North, Range 22 West, Aitkin County, Minnesota</u>** <u>Sec. 24</u>: W/2NE/4, SE/4NE/4, S/2SW/4, E/2SE/4, W/2SE/4, NE/4NE/4, NE/4NW/4, undivided ¾ interest in NW/4NW/4, undivided ¾ interest in SW/4NW/4, undivided ¾ interest in NE/4SW/4, undivided ¾ interest in NW/4SW/4 Minerals and mineral rights	600.00
MLMB200001	3/3/2016	50	3.95%	0.75%	Yes	**<u>Township 49 North, Range 22 West, Aitkin County, Minnesota</u>** <u>Sec. 15</u>: undivided $^{1}/_{3}$ interest in NE1/4-NW1/4, undivided $^{2}/_{3}$ interest in NW1/4-NW1/4, undivided $^{1}/_{3}$ interest in NW1/4-NW1/4, undivided $^{2}/_{3}$ interest in SW1/4-NW1/4, undivided $^{1}/_{3}$ interest in SW1/4-NW1/4, undivided $^{1}/_{3}$ interest in SE1/4-NW1/4, undivided $^{1}/_{3}$ interest in NE1/4-SW1/4, undivided $^{1}/_{3}$ interest in NW1/4-SW1/4, undivided $^{2}/_{3}$ interest in SW1/4-SW1/4, undivided $^{1}/_{3}$ interest in SW1/4-SW1/4, undivided $^{1}/_{3}$ interest in SE1/4-SW1/4, undivided $^{1}/_{3}$ interest in NE1/4-SE1/4, undivided $^{1}/_{3}$ interest in NW1/4-SE1/4 SE1/4-SE1/4, undivided $^{1}/_{3}$ interest in SE1/4-SE1/4 Mineral and mineral rights <u>Sec. 15</u>: undivided $^{2}/_{3}$ interest in NE1/4-NW1/4, undivided $^{2}/_{3}$ interest in SE1/4-NW1/4, undivided $^{2}/_{3}$ interest in NE1/4-SW1/4, undivided $^{2}/_{3}$ interest in NW1/4-SW1/4, undivided $^{2}/_{3}$ interest in SE1/4-SW1/4, undivided $^{2}/_{3}$ interest in NE1/4-SE1/4, undivided $^{2}/_{3}$ interest in NW1/4-SE1/4, undivided $^{2}/_{3}$ interest in SW1/4-SE1/4, undivided $^{2}/_{3}$ interest in SE1/4-SE1/4 Mineral, mineral rights, and surface rights	640
MLMB200002	3/3/2016	50	3.95%	0.75%	Yes	**<u>Township 49 North, Range 22 West, Aitkin County, Minnesota</u>** <u>Sec. 16</u>: W1/2-NE1/4, NW1/4, S1/2, E1/2-NE1/4 Mineral and mineral rights	640

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MLMB200003	3/3/2016	50	3.95%	0.75%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 21: NE1/4 Mineral and mineral rights	160
MLMN200001	2/24/2017	50	3.95%	0.50%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 5: Lot Four, SW1/4, S1/2-SE1/4 - Mineral and mineral rights Sec. 5: Lot One, Lot Two, S1/2-NE1/4, Lot Three, N1/2-SE1/4 Mineral, mineral rights and surface rights	556.31
MLMN200028	2/24/2017	50	3.95%	0.50%	Yes	**Township 48 North, Range 22 West, Aitkin County, Minnesota** Sec. 6: S1/2-NE1/4, SE1/4-NW1/4, E1/2-SW1/4, Lot Six, Lot Seven, SE1/4 Mineral and mineral rights Sec. 6: Lot Two, Lot Three, Lot Four, Lot Five Mineral, mineral rights, and surface rights	581.71
MLMN200029	2/24/2017	50	3.95%	0.50%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 21: undivided ½ interest NE1/4-SW1/4, undivided ½ interest NW1/4-SW1/4, undivided ½ interest SW1/4-SW1/4, undivided ½ interest SE1/4-SW1/4, undivided ¾ interest SE1/4-SE1/4 Mineral and mineral rights	110

4.2.6.3 Private Mineral Leases, Surface Use Agreements and Options to Purchase

In addition to the State Leases, the parties hold mineral leases, surface use agreements and options to purchase, covering privately owned surface and mineral interests (Private Agreements). There are three Private Agreements, which cover approximately 154 acres of surface and/or mineral interests within the Tamarack North Project area. The provisions and terms of each Private Agreement are specific to each Private Agreement. Certain Private Agreements include royalties payable if and when the Tamarack North Project begins production on lands covered by such Private Agreement. The royalties range from a 2% to 3.9% NSR and include certain buy-back rights. Table 4-3 provides further information on the Private Agreements.

Kennecott has also entered into easement agreements with certain property owners which allow the parties to install and monitor groundwater monitoring wells for a nominal annual fee.

Table 4-3: Summary of Private Agreements

Type of Agreement	Term	Annual Fee (US$)	Lands	Acreage
Lease and Option Agreement	July 1/15 to July 1/19	5,000	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 27: SWNW excepting certain lands *Surface Only*	37.96
Lease and Option Agreement	July 1/15 to July 1/19	5,000	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 27: NWSW excepting certain lands *Surface and Mineral* Sec. 27: SENW excepting certain lands *Surface Only*	78.18
Lease and Option Agreement	May 1/18 to May 1/22	5,000	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 22: SWSW *Surface Only*	38.2

4.2.6.4 Fee and Mineral Surface Interests

The parties also own fee surface and/or mineral interests which cover approximately 1,436 acres of land within the Tamarack North Project area. Details of the fee surface and mineral interests are detailed in Table 4-4. In certain instances, as part of the purchase price paid for the mineral rights, Kennecott agreed (in its previous capacity of Manager under the Original MVA) to pay a royalty to the previous mineral rights owner. The royalties range from a 2% NSR to a 3.9% NSR. There are also buy-back rights on certain of these royalties.

Table 4-4: Summary of Fee Mineral and Surface Interests

Township	Range	Section	Acreage
48 North	22 West	Sec. 3: NW/4 SW/4, SW/4 NW/4 except Parcel Nos. 8 and 9	80 (Surface and Mineral)
49 North	22 West	Sec. 22: SE/4SW/4	40 (Surface and Mineral)
48 North	22 West	Sec. 3: Government Lot 3	26.54 (Surface Only)
49 North	22 West	Sec 35: NW/4, NW/4 NE/4	200 (Surface and Mineral)
48 North	22 West	Sec. 3: SW/4 SW/4 except parcel no. 7	40 (Surface Only)
48 North	22 West	Sec. 3: NE/4 SW/4	40 (Surface Only)
49 North	22 West	Sec. 22: SE/4 SE/4 except Parcel No. 28	36 (Surface and Mineral)
49 North	22 West	Sec. 22: SW/4 SE/4 excepting certain lands	36.5 (Part Surface and Minerals, Part Surface Only)
48 North	22 West	Sec. 10: NW/4 SW/4 except Parcel No.6, Highway Plat No. 10; NE/4	198 (Surface Only)
48 North	22 West	Sec. 4: SE/4 NE/4	38.18 (Surface Only)
48 North	22 West	Sec. 4: NW/4 SE/4	40 (Surface Only)
48 North	22 West	Sec. 10: S/2 SW/4, SW/4 SE/4 Sec. 15: NE/4 NW/4 excepting certain lands	177.92 (Surface Only)
49 North	22 West	Sec. 26: W/2NW/4 Sec. 26: N/2 NE/4 SW/4, SE/4 NE/4 SW/4, NW/4 SE/4 Sec. 27: NE less 10 acres in the NW corner	300 (Surface and Minerals) (Surface) (Surface and Mineral)
49 North	22 West	Sec. 22: The E 400 ft of the W 750 ft of the SW/4 SE/4	11.57 (Surface Only)
49 North	22 West	Sec. 34: NE/4SW/4, SE/4SW/4, SW/4SW/4 excepting certain lands	118.01 (Surface Only)
48 North	22 West	Sec. 4: The South 561' of Lot 1	16.51 (Surface and Mineral)
49 North	22 West	Sec. 27: NWNW excepting certain lands	36.49 (Surface Only)

4.2.7 Surface Rights

The State Leases also grant the parties the right to use surface lands owned by the State of Minnesota within the leased land.

From a legal standpoint, where surface rights are owned by third parties, the State Leases provide that written notice to the owner of the surface estate must be provided at least 20 days in advance of surface activities and contemplate compensation payable by lessees to surface owners for any disturbance of the surface estate. Many states also address the rights of surface owners in case law, and although the Minnesota Supreme Court has not specifically opined on the issue, the general rule is that mineral rights carry with them the right to use as much of the surface as reasonably necessary to reach and remove the minerals, unless otherwise restricted by the mineral severance deed. Guidance provided by the MNDNR takes this approach.

Notwithstanding the above, to date, Kennecott's approach (initially as sole owner of the Tamarack North Project and then in its capacity as Manager under the Original MVA) for surface access over areas that it is interested in drilling has been to negotiate with the applicable surface land owner a surface use agreement. Also, in certain cases, Kennecott (initially as sole owner of the Tamarack North Project and then in its capacity as Manager under the Original MVA) negotiated an option to purchase the surface lands.

In the case of Private Agreements where there has been no severance of the surface and mineral estates, surface use is provided as part of the mineral lease. Where the mineral and surface estates are severed and where surface rights are held privately, surface access has typically been negotiated with the surface owner.

The surface rights held under the MVA are detailed in Figure 4-3.

4.2.8 Tax Forfeiture and Leasing of Mineral Rights

The Minnesota Severed Mineral Interests Law (Forfeiture Law) requires owners of severed mineral interests (i.e., mineral rights that are owned separately from the surface interest) to register their interests with the office of the county recorder.

Severed mineral interests are taxed. If the mineral interest owner does not file the severed mineral interest statement within the deadline provided by the law, the mineral interest forfeits to the State after notice and an opportunity for a hearing.

The owner, to avoid forfeiture, must prove to the court that the taxes were timely paid and that the county records specified the true ownership, or, in the alternative, that procedures affecting the title of the interest had been timely initiated and pursued by the true owner during the time when the interest should have been registered. To the extent the owner fails to prove this, the forfeiture to the State is deemed to be absolute. Additionally, if the owner of record fails to show up to the hearing, the forfeiture to the State is also deemed to be absolute.

The State may lease mineral rights prior to the completion of the forfeiture procedures, provided that the leased rights are limited to exploration activities, exploratory boring, trenching, test pitting, test shafts and drifts, and related activities. A lessee under such a lease may not mine the leased mineral rights until the forfeiture procedures are completed.

The State may have obtained interests in certain of the mineral rights leased under one or more of the State Leases pursuant to the Forfeiture Law and the forfeiture procedures may not have been completed for all the lands covered by these State Leases (forfeiture procedures are not required to have been completed until a lessee is looking to mine a property).

Until the forfeiture procedures have been completed, there is a remote risk that the owner of a mineral interest that the State has leased for the Tamarack North Project will demonstrate at a required hearing that the owner was in compliance with the registration and taxation requirements as detailed above. In such a case, the mineral rights would revert to this original owner. However, the State Leases that compose the area where the Mineral Resources are contained are not at risk of reversion to an original owner under Forfeiture Law.

4.3 Permitting for Exploration

The Tamarack North Project is currently in the exploration phase. It is understood that Kennecott (previously as Operator under the 2014 Earn-in Agreement, and then in its capacity as Manager under the Original MVA) had all the required permits and approvals for exploration operations. Going forward, Talon, in its capacity as operator under the 2018 Tamarack Earn-in Agreement, will be responsible for making application for the required permits and approvals for exploration operations. Federal, state, and local entities all have regulatory authority over various elements of the Tamarack North Project. Key agencies involved with project permitting will include the US Army Corps of Engineers (USACE), US Fish and Wildlife Service, MNDNR, State Historic Preservation Office (SHPO), Minnesota Department of Health (MDH), Minnesota Pollution Control Agency (MPCA), Aitken County, Carlton County, and City of Tamarack. Information on permits and approvals required for pursuing exploration operations at the Tamarack North Project is provided in Table 4-5 below.

Table 4-5: Summary of Current and Potential Exploration Permits/Approvals

Federal	
Agency	**Permit/Approval**
USACE	Clean Water Act – Section 404 Permit
SHPO	National Historic Preservation Act – Section 106
US Fish & Wildlife Service	Endangered Species Act Compliance – Section 7
State	
Agency	**Permit/Approval**
MNDNR	Exploration Plan
MDH	Explorer's License and Designated Responsible Individual; Exploratory Boring Notification
MDH	Temporary and Permanent Sealing Reports
MPCA	NPDES/SDS Construction Storm Water Permit (General Permit)
MPCA	NPDES/SDS Industrial & Storm Water Discharge Permit (General Permit)
MPCA	Storm Water Pollution Prevention Plan
MNDNR	Burning Permit
MNDNR	Permit to Work in Public Waters, including Public Waters Wetlands
MNDNR	Water Appropriation Permit
MNDNR	Wetland Conservation Act approvals for activities impacting certain wetlands
MNDNR	Threatened and Endangered Species Review
Local	
Agency	**Permit/Approval**
City of Tamarack	Zoning and Building Permits
County	Conditional Use Permit
County	Zoning Permits

If additional exploration becomes necessary to support project development, either amendments to current exploration permits or acquisition of new permits for exploration would be required.

4.4 Environmental

4.4.1 Baseline Work

Kennecott (initially as owner, and then in its capacity of Manager under the Original MVA) initiated baseline studies to support future environmental review and permitting of a potential mine at the Tamarack North Project. Work to date has included surface water and groundwater monitoring; wetland delineation and evaluation surveys; and rare, threatened and endangered species and vegetative community surveys.

Initiated in 2007/2008, Kennecott monitored 23 surface water locations and 12 ground water wells. As of 2014, Kennecott operates the regular, quarterly, monitoring of 19 surface water monitoring locations (18 streams/ditches and one lake) and 12 groundwater monitoring wells. Kennecott has also completed a limited amount (14 samples from six rock units) of static short-term acid-base accounting and leaching tests on various rock types. Independent oversight and sign-off of the sampling and analysis is completed by Foth Infrastructure and Environment LLC, of De Pere, Wisconsin.

4.4.2 Environmental Liabilities

Talon has advised Golder that it is not aware of the property having any environmental liabilities. A review of the MPCA's "What's in my Neighbourhood" database was completed for the property by Talon, and no contaminated site records were identified.

4.5 Significant Risk Factors

Talon has advised Golder that it is not aware of any significant factors or risks which may affect access, title, or the right or ability to perform work on the Tamarack North Project.

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

5.1 Introduction

The Tamarack Project is located in north-central Minnesota, approximately 100 km (62 miles) W of Duluth and 210 km (130 miles) N of Minneapolis, in Aitkin County (

Figure 4-1). The area is characterized by farms, plantations and forested areas. The town of Tamarack (population 88, 2016 US Census Bureau), which gives the project its name, lies within the boundaries of the Tamarack North Project (though away from the known mineralization) at an elevation of 386 m above sea level. The Tamarack Project's field office is located in Tamarack. Other small towns in the area are Wright (10 km east (E) from Tamarack) and McGregor (15 km W from Tamarack).

5.2 Accessibility

Access to the Tamarack North Project is via paved state and county highways and roads. From the city of Duluth, the Tamarack North Project can be accessed by Interstate 35 S for 32 km and then onto State Highway 210 W for 61 km to the town of Tamarack. The Tamarack North Project is easily accessible from Tamarack by paved road, with the current known mineralization located approximately 500 m laterally from a paved all-weather road.

5.3 Physiography

The Tamarack North Project transitions between the Minnesota/Wisconsin Upland Till Plain and the Glacial Lakes Upham and Aitkin ecoregion as defined by the Environmental Protection Agency (EPA) (Level III and IV Ecoregions of Minnesota, June 2015). The topography is level to gently rolling as is typical of old glacial lake plains. The soils are dominated by clay-silt to silty-sand Culver associated moraine deposits or by silty sand to sandy silt with clay interpreted as reworked pre-existing lake and stream sediments. Peat bogs are also found overlying the glacial till in the area (Jennings and Kostka, 2014). Relief is minimal, and where found is generally a result of small till moraines. As a result of the flat to gentle relief, poor drainage has allowed the area to be dominated by lowland conifers surrounding sedge meadows and marshland. Areas of higher relief will support aspen-birch and upland conifers.

5.4 Climate

The climate of Minnesota is typical of a continental climate, with hot summers and cold winters. Minnesota's location in the Upper Midwest allows it to experience some of the widest variety of weather in the US, with each of the four seasons having its own distinct characteristics. The annual average temperature at the Tamarack North Project is 5°C. The temperature averages a high of -7°C and a low of -18°C in January and a high of 26°C and a low of 13°C in July. Annual rainfall averages approximately 764 mm. Annual snowfall averages 142 centimetres (cm). (Tamarack Weather Averages, November 2017). Exploration operations at the Tamarack North Project can be conducted throughout the whole year (subject to any permitting restrictions) and future mining activities could be conducted on a year-round basis.

5.5 Local Resources

The mining support industries and industrial infrastructure in Minnesota are well developed and of a high standard, though most of the mining in the State occurs in the Mesabi Iron Range approximately 150 km to the northeast (NE). Any exploration and mining efforts will be well served by an extensive talent pool located throughout the area.

5.6 Sufficiency of Surface Rights

The Tamarack North Project has an extensive package of surface rights previously secured by Kennecott (previously as Operator under the 2014 Earn-in Agreement, and then as Manager under the Original MVA) (Figure 4-3). The parties have sufficient rights to allow for mining operations and supporting infrastructure in the area of mining interest.

6. HISTORY

Prior to 2002, the Tamarack area was subject to only very limited exploration efforts and there has been no prior mineral production from the Tamarack North Project. The relatively thick post mineral, glacial fluvial sediment cover and nearly complete lack of bedrock exposure severely hampered any early exploration (the nearest known bedrock exposure to the Tamarack North Project is located approximately 15 km to the SE of the deposit).

Starting in 1972, the Minnesota Geological Survey (MGS) oversaw a 12-year program to collect high-resolution airborne magnetic data over the entire State, including the Tamarack area. The program was paid for by a penny per pack tax on cigarettes sold in the State. This program ran concurrently to an MNDNR-sponsored program of regional lake sediment sampling. As part of the follow up to the airborne surveys, the State carried out a program of scientific drilling to try to identify the bedrock source of selected magnetic anomalies. Information from MNDNR staff involved with the program indicates that the magnetic anomalies were prioritized by the presence of anomalous lake sediment geochemistry. This is reported as being the case for the TIC, with two local lakes being anomalous in Ni, Cu and chromium (Cr).

In the summer of 2000, Kennecott leased mineral title in Aitkin County from the State of Minnesota covering areas of the Tamarack North Project. There were no apparent non-ferrous leases in this area previous to Kennecott's initial leasing (Historic State Nonferrous Metallic Mineral Leases, October 2017). Additional mineral title has been added to Kennecott's land position in the area since then as detailed in Section 4 of this PEA.

Kennecott began exploration on the Tamarack North Project in 2001 when Kennecott flew an airborne MEGATEM and magnetic survey covering most of the TIC. Ground EM and gravity surveys were also carried out to refine anomalies identified in the airborne survey.

In the winter of 2002, Kennecott began drilling at the Tamarack North Project (see Section 9 for further details of exploration work conducted by Kennecott). Drilling has occurred continuously on site since 2002 except for the years 2005 and 2006 (see Section 10 for further details of the drilling programs conducted by Kennecott).

On October 6, 2014 Talon published a maiden NI 43-101 compliant report and resource statement (effective date August 29, 2014) for the Tamarack North Project (see Table 6-1 for the 2014 resource statement).

Table 6-1: 2014 Tamarack North Project Maiden Resource Statement (Effective Date August 29, 2014)

Domain	Mineral Resource Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
SMSU	Indicated Mineral Resource	3,751	1.81	1.00	0.05	0.41	0.25	0.19	2.35
SMSU	Inferred Mineral Resource	949	1.12	0.62	0.03	0.25	0.16	0.14	1.47
MSU	Inferred Mineral Resource	158	5.25	2.47	0.11	0.66	0.44	0.22	6.42
138 Zone	Inferred Mineral Resource	2,012	0.95	0.78	0.03	0.23	0.14	0.17	1.33
TOTAL	**Indicated Mineral Resource**	**3,751**	**1.81**	**1.00**	**0.05**	**0.41**	**0.25**	**0.19**	**2.35**
TOTAL	**Inferred Mineral Resource**	**3,119**	**1.22**	**0.82**	**0.03**	**0.26**	**0.16**	**0.16**	**1.63**

Notes:
All resources reported above a 0.9% NiEq cut-off.
Mining recovery and dilution factors have not been applied to the estimates.
Tonnage estimates are rounded down to the nearest 1,000 tonnes.
Estimates do not include metallurgical recovery.
**Where used in this Mineral Resource estimate, NiEq% = Ni%+ Cu% x 2.91/9.20 + Co% x 14/9.20 + Pt [g/t]/31.103 x 1,400/9.2/22.04 + Pd [g/t]/31.103 x 600/9.2/22.04 + Au [g/t]/31.103 x 1,300/9.2/22.04*

An updated resource statement was published via a press release (effective date April 3, 2015) resulting from an increase in the MSU mineralization (see Table 6-2). No report was published, as the increase was deemed to be not material to the overall project tonnage.

Table 6-2: Tamarack North Project Updated Mineral Resource Estimate (Effective Date April 3, 2015)

Domain	Mineral Resource Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
SMSU	Indicated Mineral Resource	3,751	1.81	1.00	0.05	0.41	0.25	0.19	2.35
SMSU	Inferred Mineral Resource	949	1.12	0.62	0.03	0.25	0.16	0.14	1.47
MSU	Inferred Mineral Resource	422	6.00	2.48	.013	0.78	0.53	0.26	7.26
138 Zone	Inferred Mineral Resource	2,012	0.95	0.78	0.03	0.23	0.14	0.17	1.33
Total	**Indicated Mineral Resource**	**3,751**	**1.81**	**1.00**	**0.05**	**0.41**	**0.25**	**0.19**	**2.35**
Total	**Inferred Mineral Resource**	**3,383**	**1.63**	**0.94**	**0.04**	**0.31**	**0.19**	**0.17**	**2.11**

Notes:
All resources reported above a 0.9% NiEq cut-off.
Mining recovery and dilution factors have not been applied to the estimates.
Tonnage estimates are rounded down to the nearest 1,000 tonnes.
Estimates do not include metallurgical recovery.
**Where used in this Mineral Resource estimate, NiEq% = Ni%+ Cu% x 2.91/9.20 + Co% x 14/9.20 + Pt [g/t]/31.103 x 1,400/9.2/22.04 + Pd [g/t]/31.103 x 600/9.2/22.04 + Au [g/t]/31.103 x 1,300/9.2/22.04*

A detailed chronology of business agreements, decisions, and developments between Kennecott and Talon with respect to the Tamarack Project is contained in Section 4.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting; Introduction

The TIC is an ultramafic to mafic intrusive, hosting Ni-Cu sulphide mineralization with associated Co, PGEs and Au. The intrusion of the TIC (minimum age of 1105 Ma+/-1.2 Ma, Goldner 2011) is related to the early evolution of the approximately 1.1 Ga Mesoproterozoic MCR and has intruded into slates and greywackes of the Thomson Formation of the Animikie Group which formed as a foreland basin during the Paleoproterozoic Penokean Orogen (approximately 1.85 Ga, Goldner 2011). The TIC is completely buried beneath approximately 30 to 60 m of Quaternary age glacial and fluvial sediments.

The lack of outcrop has limited the understanding of the TIC in its regional geological context relative to its location in the deformed southern margin of the Animikie Basin. The TIC is intruding part of the Penokean accreted terrain, based on the age of the CGO intrusion (Goldner, 2011). The closest known portion of the accreted Penokean magmatic Arc terrane is located well to the S and E of the TIC. The TIC intrudes deformed sediments deposited in part in foreland basin in front of the accreted terrane, which likely was in turn dissected by subsequent rifting associated with the MCR and thus has contributed to a complex geological and structural setting. The regional geological setting is described below within the context of the major depositional periods and tectonic events (Figure 7-1 and Figure 7-2).

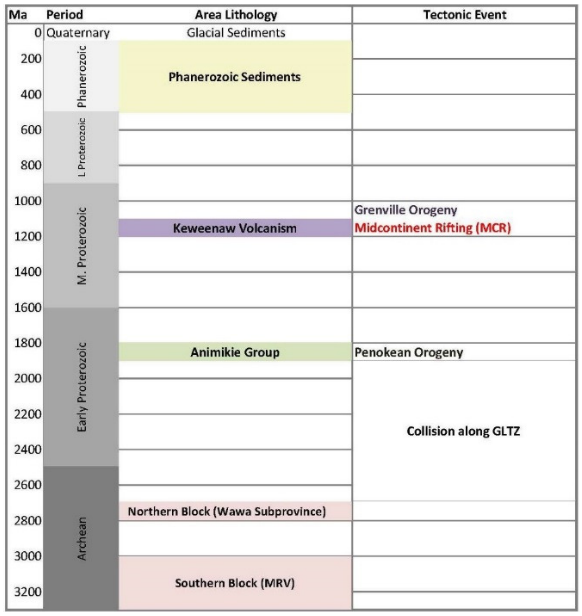


Figure 7-1: Major Depositional Periods and Structural Events Affecting Geological Emplacement and History of the TIC - Modified After Lundin Mining Corporation (2013)

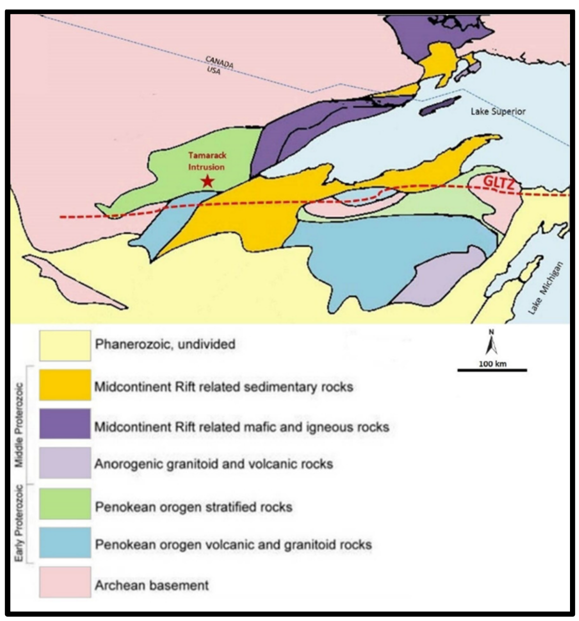


Figure 7-2: Regional Geological and Tectonic Setting for the TIC.
The GLTZ Structure Represents an Inferred Position Due to Younger, Overlying Lithology - Modified from Khirkham (1995) and Lundin Mining Corporation (2013)

7.1.1 Archean Stratigraphy and the Great Lakes Tectonic Zone (GLTZ)

Archean basement and supra-crustal rocks underlie the Paleoproterozoic Animikie SED Basin. The nearest outcrop of Archean basement rocks is located 35 km to the S of the TIC in the McGrath gneiss dome. In western Minnesota, the Archean is divided into an older, southern block referred to as the Minnesota River Valley (MRV) Terrane and the northern Wawa Sub-province of the Archean Superior Craton (Figure 7-1).

The southern Paleoarchean MRV Terrane comprises 3.3 Ga gneiss, migmatite and amphibolite of predominantly Middle Archean age, intruded by Late Archean granitoids.

The northern Wawa sub-province comprises late Archean (2.6-2.7 Ga) supra-crustal rocks intruded by a variety of intrusions. Wawa Sub-province rocks are believed to form the basement beneath the southern part of the Animikie Basin at Tamarack.

A broad E-W striking regional structural zone marks the boundary between the MRV Terrane and the Wawa Sub-province and is referred to as the GLTZ (Figure 7-2). The GLTZ can be inferred eastward from western Minnesota into northern Michigan and perhaps into Ontario, Canada. Kinematic analysis in the only known outcrop of the GLTZ S of Marquette, Michigan suggests the GLTZ at this location dips steeply southward, and that vergence was to the northwest (NW), indicative of an oblique collision that brought the Paleoarchean rocks over the younger Archean rocks of the Wawa Sub-province (Sims et al., 1993). The collision along the GLTZ is believed to have occurred between 2692-2686 Ma (Schneider et al., 2002).

The GLTZ appears to have played a direct role in localizing later Paleoproterozoic sedimentation and volcanism. Possible structures related to the GLTZ, may have localized other Paleoproterozoic SED basins and later MCR related intrusions in the region (Owen et al., 2013). Although the exact location of GLTZ beneath the Animikie Basin is uncertain, it has been interpreted by Holm et al. (2007) to occur just S of the TIC. Based on this interpretation it may be possible that it played a role in the localization of the Tamarack intrusion.

7.1.2 Paleoproterozoic; the Animikie Basin and the Penokean Orogen

The depositional and tectonic history of the Penokean Orogen is dated at around 1.85 Ga and in Minnesota consists of two main components. One is a fold and thrust belt representing an accreted terrain to the S while the other is a foreland basin (Animikie Basin) formed to the N as a result of a collision between the continental margin of the Archean Superior Province Craton and the Pembine-Wausau oceanic arc (Southwick et al., 1988, 1991; Schulz and Cannon, 2007) (Figure 7-3).

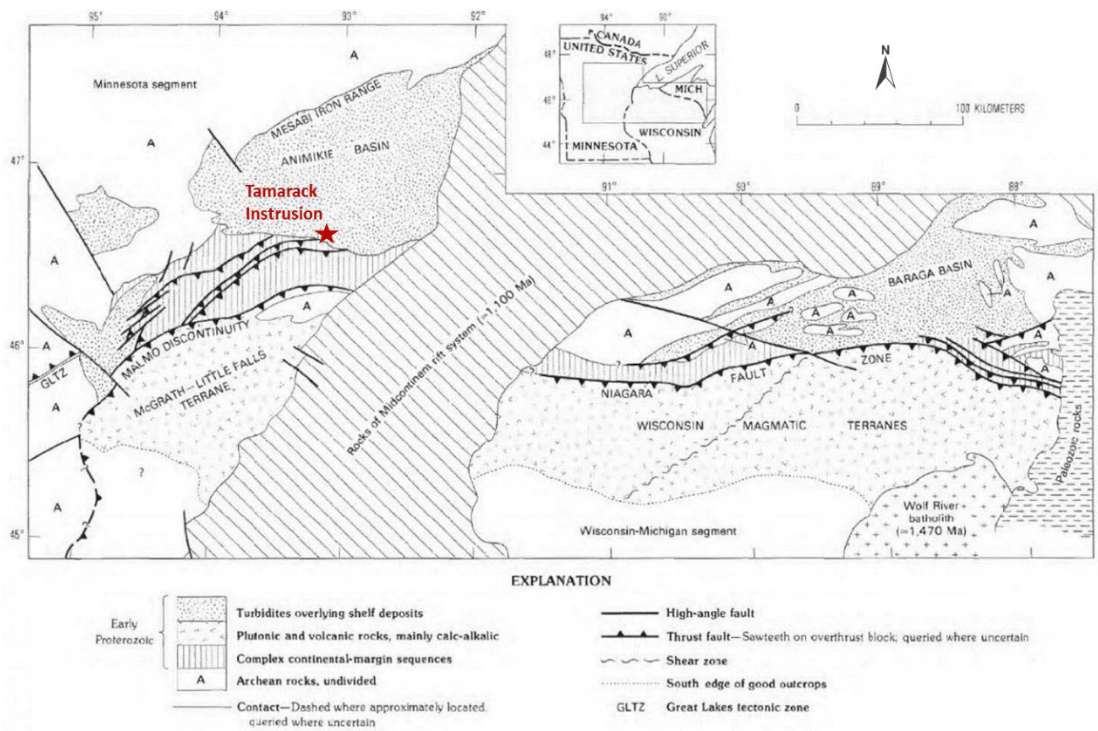


Figure 7-3: Location of TIC in Relation to MCR and Southern Boundary of the Animikie Basin with Tectonic Imbrication and Foredeep Development of the Penokean Orogen. Interpretation Based on Regional Geophysics and Results of Test-Drilling by Southwick et al., 1991

In east-central Minnesota, the Animikie Group sediments which are weakly to moderately folded and metamorphosed, unconformably overlie the more intensely deformed North Range Group and Mille Lacs Group and the Archean basement. The Animikie Group sediments include the basal quartzite and conglomerate of the Pokegama Formation; the Biwabik banded Fe formation and inter-bedded argillite, siltstone and sandstone of the Virginia Formation which are exposed in the Fe ore mines of the Mesaba Iron Range along the northern margin of the Animikie Basin. In the N of the basin these sediments are only weakly metamorphosed, but metamorphism and deformation increase towards the S where similar sediments have a well-developed axial planer foliation and are folded into N verging upright folds which become increasingly tighter and possibly overturned along the S margin of the basin. These more deformed and metamorphosed sediments are referred to as the Thomson Formation and have been interpreted to be the deformed equivalents of the Virginia Formation (Severson et al, 2003). Boerboom (2009) has subdivided the Thomson Formation into Upper and Lower sequences. The Lower sequence comprises carbonaceous siltstone and mudstone that is locally sulphide rich; and a proposed source for the sulphide in the TIC. The Upper Thomson consists of turbidite-like siltstone and sandstone.

At the Tamarack North Project, the host rocks to the TIC are the Upper Thomson Formation. The Lower Thomson Formation which sub-crops to the S of Tamarack North Project, dips towards the N (beneath the Upper Thomson Formation), and is interpreted to underlie the TIC at depth. A prominent seismic reflector under the TIC deposit at a depth of 4.6 to 4.8 km may represent the base of the Lower Thomson Formation in the TIC area (Goldner 2011).

7.1.3 Mesoproterozoic Mid-Continental Rift (MCR)

The Mesoproterozoic MCR is represented by a large igneous province that formed from intra-continental rifting at approximately 1.1 Ga (Hutchinson et al., 1990) resulting from a mantle plume. The MCR extends along a 2000 km arcuate path from the Lake Superior region to the southwest (SW) as far as Kansas and to the SE beneath Lower Michigan (Hinze et al., 1997). Although only exposed in the Lake Superior area, the extent of the MCR beneath younger cover can be interpreted from its pronounced gravity and aeromagnetic signature.

In the Lake Superior region, the Keweenaw Flood Basalt province represents the exposed portion of the MCR system. Seismic data indicates the rift below Lake Superior is filled with more than 25 km of volcanic rocks buried beneath a total thickness of up to 8 km of rift sediments (Bornhorst et al., 1994).

The Keweenaw Flood Basalt province was formed over a period of approximately 23 Ma (Miller and Vervoort, 1996) and shows various magnetic polarity reversals. Volcanism occurred in distinct phases, with an earlier phase dominated by low alumina basalts (<15% Al_2O_3) that include both olivine and pyroxene phyric picrites. These may have been derived from primitive magmas tapping a deep mantle source. The later volcanic phases are dominated by high alumina basalts (>15% Al_2O_3) with Mid Ocean Ridge Basalt like chemistry. The evolution of the MCR closely resembles that of other large igneous provinces such as the North Atlantic Igneous Province and the Siberian Traps. In the North Atlantic Igneous Province, picritic volcanic rock, associated with an early phase of "plateau like" flood basalts, are spread out over an area of 2000 km (Larsen et al., 2000).

In addition to the extrusive rocks, a large volume of intrusive rocks was emplaced and include the Duluth Complex, the Mellen Complex, the Coldwell Complex, the Beaver Bay Complex and the Nipigon Sill Complex, in addition to numerous dyke swarms and sills that may have acted as feeders for lava flows along the flanks of the rift. The TIC is one of the numerous smaller satellite intrusions which also include Eagle; Echo Lake; Bovine Intrusive Complex intrusions in upper Michigan; the Coldwell Complex near Marathon, Ontario; the Seagull Lake; Kitto, and Disraeli Lake intrusions in the Lake Nipigon area; and the Crystal Lake Gabbro in the Thunder Bay area (Goldner 2011, Figure 7-4). Many of these smaller intrusions, relative to the MCR volcanics, are older (3-15 Ma), occur distally, and have more primitive melt signatures. They are interpreted to represent the early evolution of the MCR.

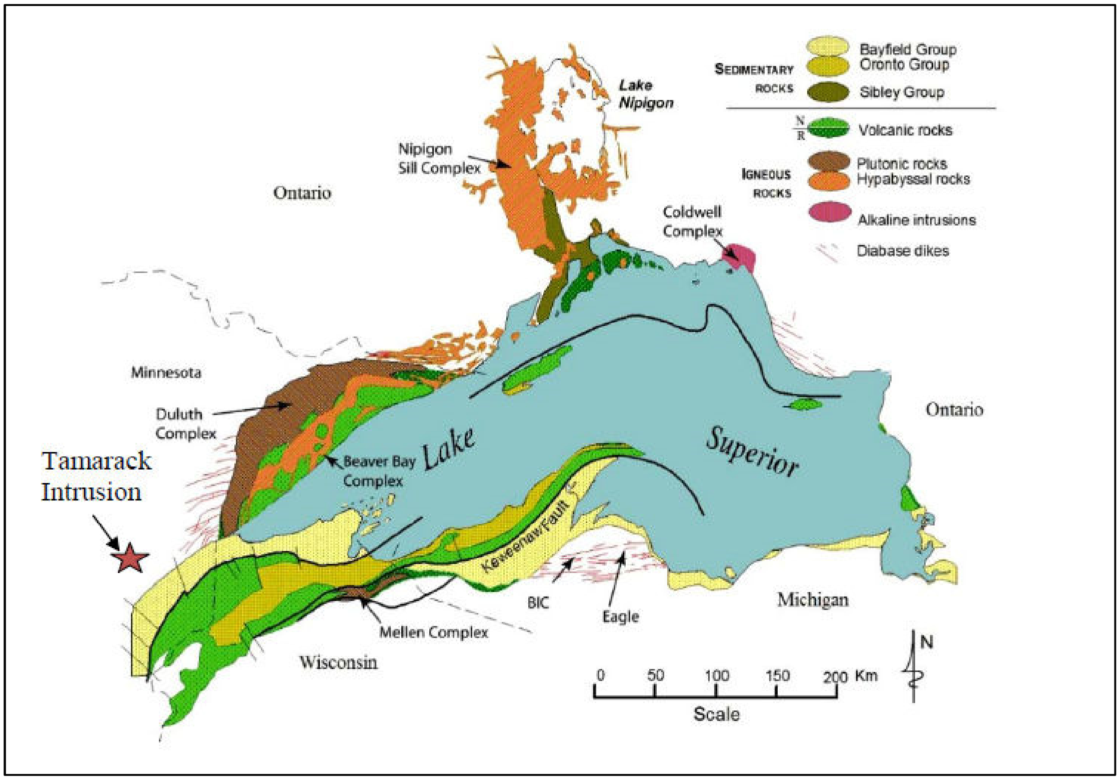


Figure 7-4: Map Showing Locality of TIC and Geology of Lake Superior Region with Location of Other Intrusive Components of the MCR (Goldner 2011, modified from Miller et al., 1995)

The MCR was terminated by a compressional tectonic phase resulting in the inversion of original, graben bounding, normal faults, into reverse faults. The compressional event has been interpreted to possibly be the result of the Grenville Orogeny which may have started as early as 1080 Ma and was probably completed by 1040 Ma (Bornhorst et al., 1994). The orogeny resulted in rotation of blocks towards the rift axis with local sediments derived from the erosion of uplifted horst blocks (e.g.: Hinckley Sandstone formation in Minnesota). There is currently no evidence to suggest that the TIC has been affected by this rotational event.

7.1.4 **Cretaceous**

Cretaceous sediments that include fluvial conglomerates and sandstones, overlain by transgressive tidal flats deposits (including lignite layers) and progressively deeper marine sediments representing a transgression, are preserved in western and central Minnesota. These sediments often overlie a well-developed paleo-lateritic weathering profile. At

Tamarack, Cretaceous siltstone and sandstone unconformably overlie parts of the TIC in the N and a layer of up to 30 m thick of Kaolinitic mudstone occurs in the NE of the TIC and is similar to other deposits that have been mined in the MRV for manufacturing brick and tiles.

7.1.5 Quaternary

Thick glacial-lacustrine deposits cover most of the Tamarack area as they do other large areas of Minnesota. The deposits are a complex sequence of lobes representing multiple advances and retreats from the last Pleistocene glaciation which spanned a period from 10,000 to 100,000 years ago. Fluvial reworked glacial sediments and varved clay layers occur between various lobe layers. Varved clay layers underlie widespread peat bogs in the Tamarack area and are believed to have been deposited in Glacial Lake Upham which covered much of northeastern Aitkin County.

7.2 Property Geology

7.2.1 Introduction

The TIC consists of a multistage magmatic event composed of mafic to ultramafic body that is associated with the early evolution of the MCR (with the youngest intrusion dated at 1105 Ma +/- 1.2 Ma, Goldner, 2011). This age is significantly older than other Duluth Complex Intrusions which consistently date at 1099 Ma. The TIC is consistent with other earlier intrusions associated with the MCR that are often characterized by more primitive melts.

The TIC has intruded into Thomson Formation siltstones and sandstones of the Animikie Group and is preserved beneath remnant shallow Cretaceous fluvial and tidal sediments and Quaternary glacial sediments which unconformably overlie the intrusive. The geometry of the TIC, as outlined by the well-defined aeromagnetic anomaly (Figure 7-5), consists of a curved, elongated intrusion striking N-S to S-E over 18 km. The configuration has been likened to a tadpole shape with its elongated, northern tail up to 1 km wide and large, 4 km wide, ovoid shaped body in the S (Figure 7-5). The northern portion of the TIC (the Tamarack North Project), which hosts the currently defined resource and identified exploration targets, is over 7 km long and is the focus of this PEA.

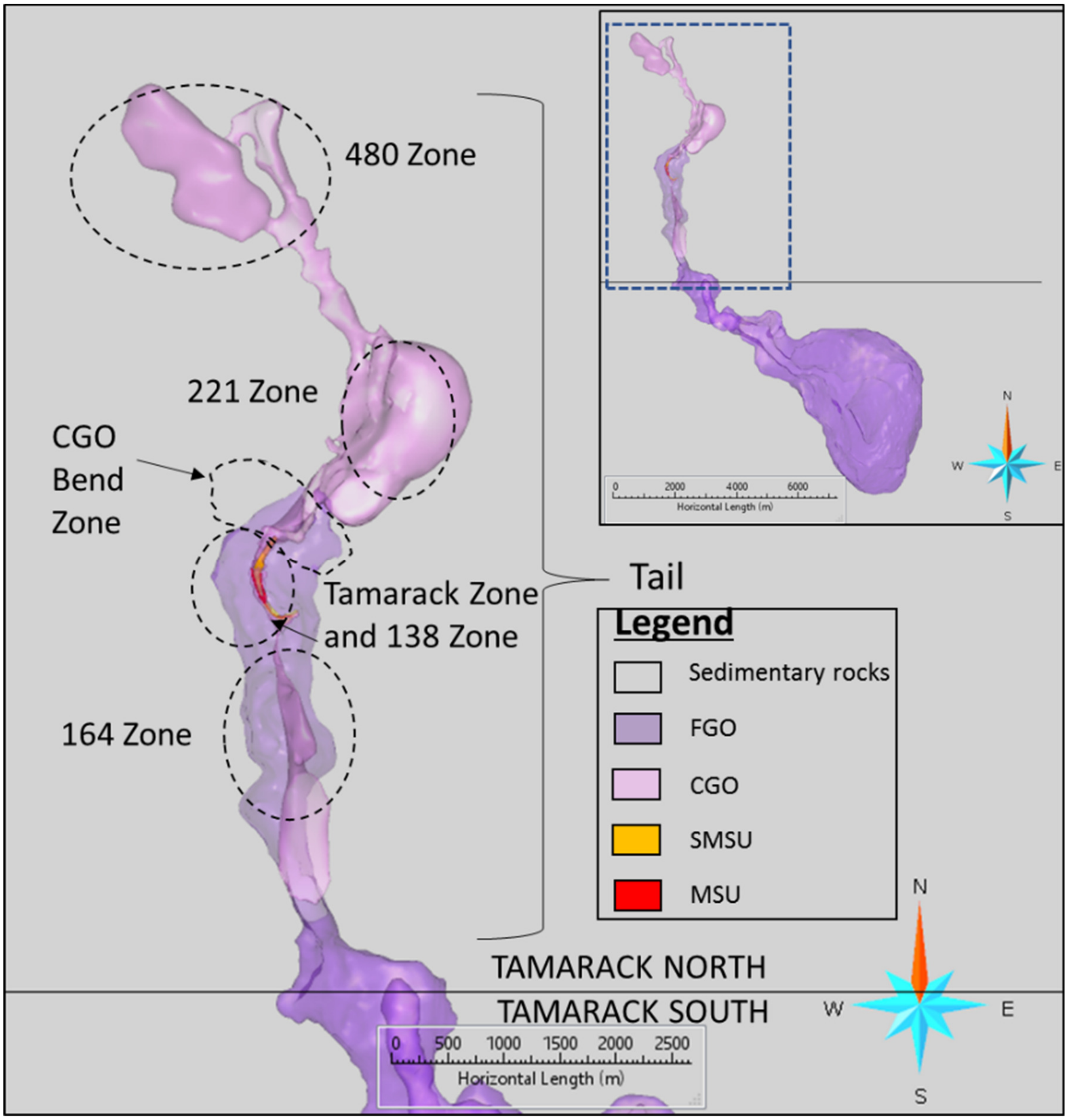


Figure 7-5: Interpreted Bedrock Geology Map Showing 18 km Long Strike of TIC with Long Narrow Intrusion that Hosts Currently Defined Mineralization Termed "Tail" forming Tamarack North Project (Kennecott Aeromagnetic Survey, Modified by Talon, 2017)

7.2.2 Paleoproterozoic (Thomson Formation)

The TIC is intruded into a folded and metamorphosed (greenschist facies) sequence of siltstone and sandstone turbiditic sediments of the Upper Thomson Formation that dip shallowly towards the N. Contact metamorphism peripheral to the TIC ranges from granoblastic to spotted hornfels. Observations from core at Tamarack North indicate that SED and structural fabrics have largely been obliterated by the metamorphism.

7.2.3 Overview of the Tamarack North Project

The Tamarack North Project has been interpreted to consist of at least two and possibly three separate phases of intrusions based on contact relationships, textural, and geochemical differences. The two main intrusive phases include an FGO that forms the wider, upper part of the intrusion in the mid and southern part of the tail; and a coarse grained, intrusive phase of CGO interpreted to have intruded dyke-like along structures and underplated the base of the FGO in the form of a keel that sub-crops as a result of pre-Cretaceous erosion in the N of the 'tail' area. N of the Tamarack Zone, the CGO intrusive extend in curvilinear shape with a N-S orientation. The intrusive nature of the CGO is variant from dyke to sills. The recent 3D inversion geological model using Magnetic and Gravity surveys best exemplifies the CGO intrusion nature (see Figure 7-5). In some areas (i.e. 221 Zone), the CGO appears to over-plate an FGO-like intrusive.

Associated with the contact between these two intrusions is also a hybrid phase, the MZ. The MZ geochemical signature resembles the FGO, however its mineralogy is slightly different with possible country rock contamination associated with possible sediment assimilation by FGO magma. It is interpreted that the MZ represents a contaminated FGO by thermal erosion of the country rock sediments, thus in the geological model both lithologies have been combined into single one, the FGO (Figure 7-6).

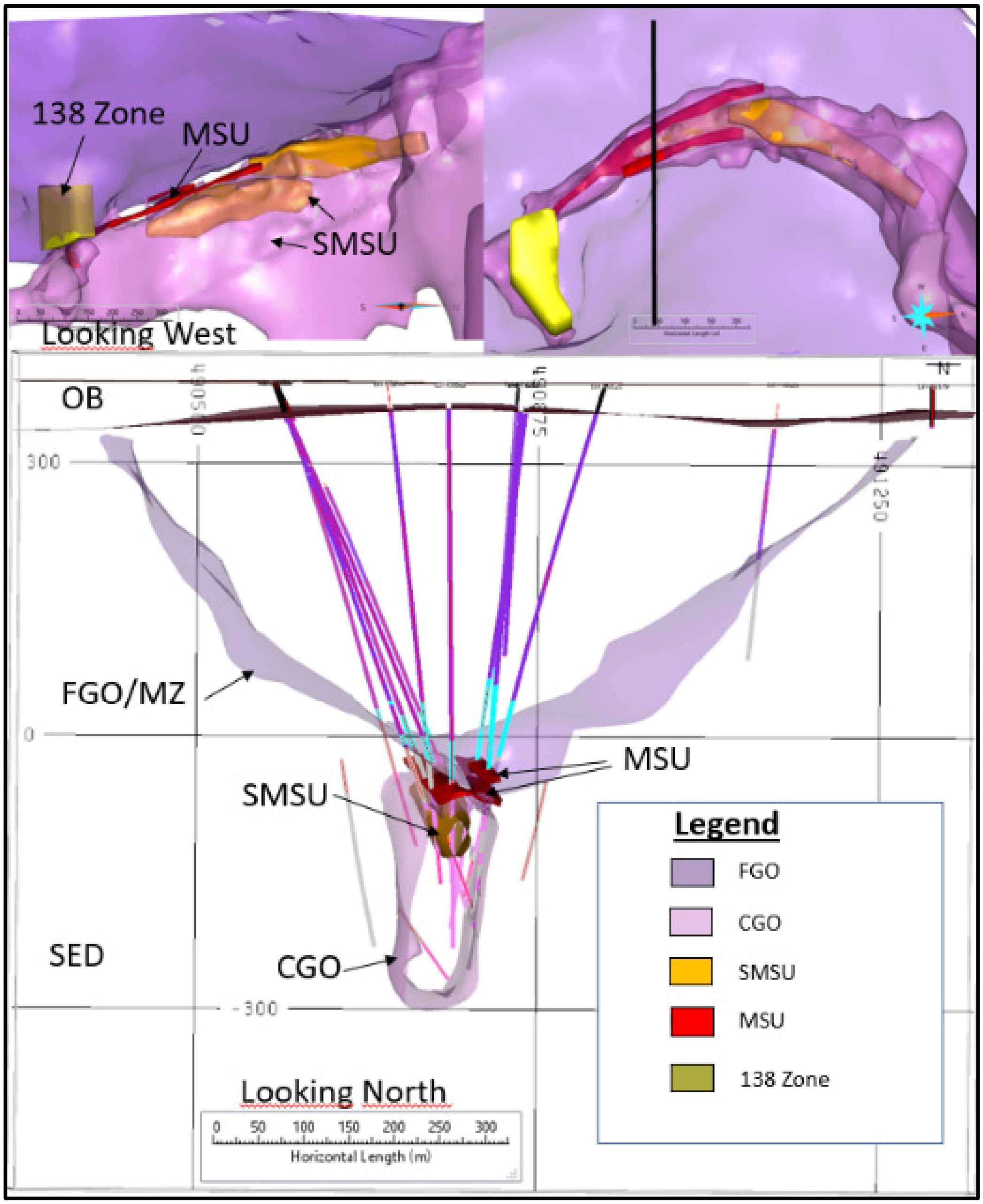


Figure 7-6: Plan, Long Section (S-N) and Cross Section Showing Main Components of Tamarack North Project including CGO at Base Intruding Dyke-Like Beneath FGO in Shape of a Keel. MZ intrusive occurs near interface of the two intrusions. Mineralization in SMSU occurs at top of the CGO, MSU occurs in what is interpreted as a wedge of remnant wall rock. In 138 Zone to the S of this section matrix and disseminated mineralization occurs in the MZ. Horizontal gridlines are metres above sea level (mASL).

Sulphide mineralization occurs within various lithological settings but is primarily associated near the FGO/CGO contact, within the 138 Zone and along the CGO/Sediment contact (Figure 7-6). More specifically, these zones are the SMSU (occurring in the upper part of the CGO near the FGO contact); the MSU (hosted within sediment but proximal to the wall rock contact of the FGO and CGO); and the 138 Zone (occurs S of the SMSU and within a large zone of MZ).

Other less developed exploration targets with defined mineralization include the shallow mineralization within the 480 Zone towards the northern part of the 'tail', the 164 style mineralization in the 164 Zone towards the southern end of the 'tail', widespread disseminated to MMS mineralization developed at shallow depths in the FGO, N of the SMSU mineralization, and a disseminated sulphide mineralization hosted in the CGO extending N of the SMSU, both known as the CGO Bend Zone.

The TIC consists of a tilted intrusion with dip to the S and E based on the magmatic layering observed in the FGO. The FGO is eroded progressively towards the N exposing the CGO N of the Tamarack North Project (Figure 7-5). Evidence for this apparent dip being the result of tectonic block rotation however has not been conclusively proven.

7.2.4 Intrusion Types

The different intrusions of the Tamarack North Project include:

- **FGO**: The FGO forms an elongated, S plunging, gutter shaped intrusion primarily in the centre and S portions of the Tamarack North Project that is progressively eroded to the N (although it appears to be preserved in the 480 Zone). The FGO intrusion is approximately 1 km wide at its erosional surface and up to 475 m thick. The intrusion is composed primarily of dunite/peridotite with FGO. The olivine (forsterite (Fo) at 70-86%, Goldner, 2011) decreases in modal amount downward towards the basal contact. The FGO intrusion is magmatically layered and defined by specific geochemical markers. The Magmatic layering dips to the S at 8˚ to 12˚. The magmatic layering is observed in Geochemical profile which consists of, from base to top, a Basal FGO, Mid-Lower FGO, FGO cumulate, Intermediate FGO and upper FGO. In the northern part of the FGO intrusion, the contact zone with sediments (country rock) is marked by a FGO and MZ lithology (MZNO). The Ni content of olivine is relatively low as plotted on a Ni vs Fo plot (Figure 7-7). Mineralization can occur as disseminated, MMS or blebby sulphides near or at the base of the FGO. When comparing Ni content of olivine versus the Mg number, we can determine that the FGO was sulphur saturated and likely provided the metals to form the mineralization within the FGO-MZNO/CGO;

- **CGO:** The CGO intrusion (age dated at 1105 Ma +/- 1.2 Ma) is currently interpreted as a separate, younger intrusive. In the Tamarack Zone, the CGO underplates and eroded the base of the FGO complex (described as the Keel). In the Tamarack Zone, the CGO has a dyke like behavior. The SMSU defined mineralization in the Tamarack North Project is contained within and near the top of the CGO. The CGO underplates the FGO and observation of chilling against the FGO, coupled with xenolith of FGO-like within CGO, Magnetic field reversal corresponding to CGO magnetic polarity overprinting in part the Magnetic signature of the FGO, indicates that the CGO post-dates the FGO. N of the Tamarack Zone, the CGO intrusive sills out into the country rock. Within the 221 Zone and 480 Zone the CGO appears to over-plate the FGO intrusive. The CGO is, lithologically, a feldspathic peridotite (60-30 modal percent olivine) with olivine gabbro present at the contact with enclosing sediments. The olivine's are substantially coarser in grain than those of the FGO, reaching as much as 1 cm in diameter. They also define a higher Ni trend on a plot of Ni content versus Fo in olivine (Figure 7-7). Although the CGO is chilled against the FGO in the N, further S the contact between the CGO and FGO bodies is commonly marked by what has been logged as a MZ. In this unit, the two distinctive intrusive types (FGO-CGO) do not show any obvious chill zone, and FGO and CGO occur together with smaller olivines occurring in the interstices between coarser olivine. When comparing Ni content of olivine versus the Mg number, we can determine that the CGO was sulphur under-saturated, never reach saturation within the study area, and did not provide significant metals to sulphides;

- **MZ**: MZ lithology is the least understood of the TIC. Models suggested included:
  - The MZ represents the contaminated lower portion of the FGO by country rock (meta-SED rocks) due to thermal erosion;
  - Separate intermediate phase intrusion between the FGO and CGO; and
  - A zone of mixing between the CGO and FGO.

  MZ is characterized by a bimodal population of CGO and FGO with Ni vs Fo plotting intermediate between CGO and FGO (Figure 7-7). MZ's often host varying amounts of disseminated sulphide mineralization that, within the 138 Zone, is significantly concentrated to form a Mineral Resource.

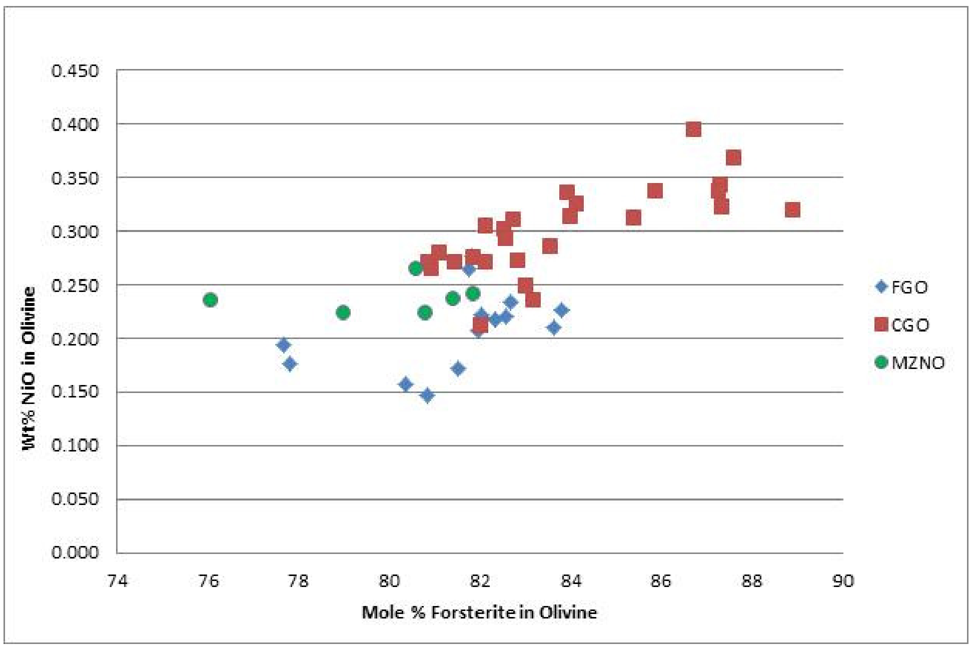


Figure 7-7: Plot of Ni in Olivine vs Fo Content of Olivine. FGO defines a Continuous Trend with Lower Ni Content than in CGO. FGO Olivine Defines a Narrow % Fo Range (82-84% Fo) Compared to CGO (81-89% Fo). Olivine from MZ falls between the two trends. (Data from Goldner, 2011).

7.2.5 Mineralization

The Ni-Cu-Co-PGE mineralization at the Tamarack North Project, occurs as various types ranging from disseminated to net textured to massive sulphides. Sulphide mineralogy is dominantly pyrrhotite (Po), pentlandite (Pn), chalcopyrite (Cpy), with minor cubanite. Pn occurs as coarse grains and as intergrowths with Po.

Although some of the mineralization names at the Tamarack North Project are used to describe mineralization lithologically in terms of sulphide concentration, they have been used by Kennecott to describe specific ore bodies. These ore bodies have different mineralization styles, with different metal tenors, genetic implications and different resource potential.

1) The 164 Zone

The mineralization type within the 164 Zone (Figure 7-8), which is located around 1.5 km S of the 138 Zone typically occurs as variable massive sulphide veins and pods < 2 m thick with blebby disseminated mineralization occurring at the base of FGO intrusion on the wall-rock contact (500 m depth), and often within hornfelsed and partially melted sediments near the chilled contact with the FGO. Mineralization is generally low tenor and has been interpreted as early cumulate mineralization associated with the base of the FGO. In the 164 Zone, the base of the FGO is more complex. Thick intervals of variable textured gabbro, magmatic breccia, and thin sills or dykes occur within the partially melted meta-sediment where coarse blebby disseminated mineralization occurs in variable textured gabbro with granophyric patches.

Recent geophysical modeling, using magnetic and gravity surveys has enabled interpretation of the footwall contact between FGO and country rock sediments. The work was completed by Mira Geoscience and identified the possible location of the Keel of the FGO where it is the loci of sulphide mineralization in the Tamarack Zone. Along the Keel, potential basin, local depression in the FGO base has been identified. Historical and current drilling has only covered the flank of the FGO sediments identifying blebby sulphide (mentioned above). The area remains open with regard to the basin which has a local dimension of 100 m x 200 m x 100 m for the southern basin and 170 m x 270 m x100 m for the northern basin (Figure 7-8).

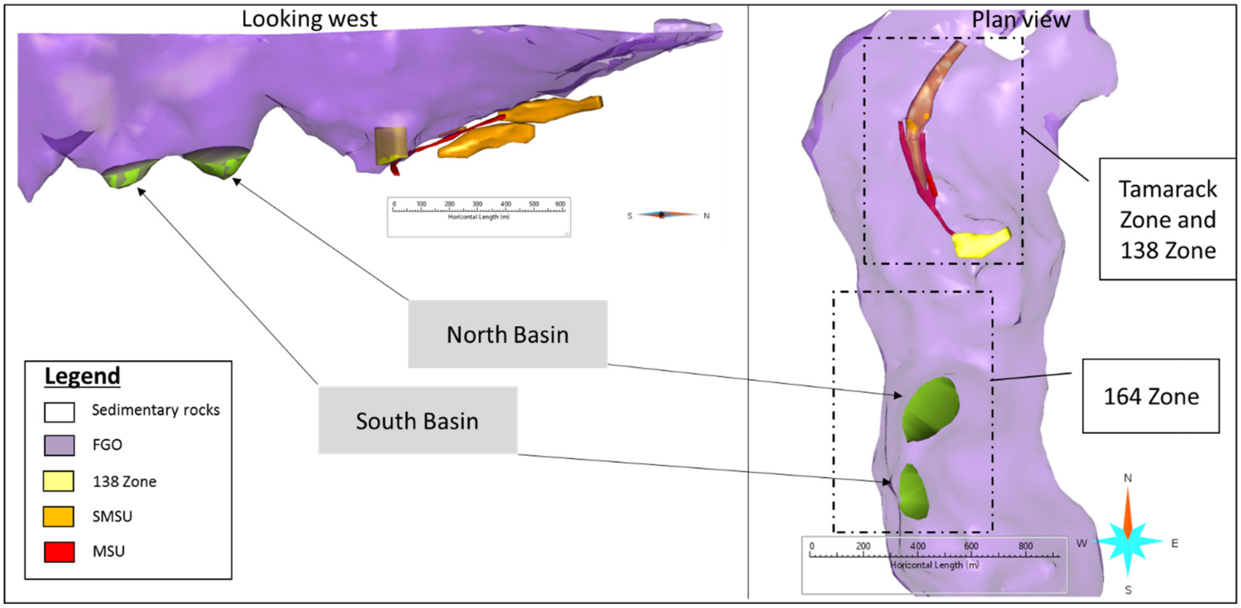


Figure 7-8: 164 Zone, Showing Emplacement of Interpreted Local Basin at Base of FGO. Results from 3D Interpolation of Integrated Magnetic and Gravity Modeling

2) The 138 Zone

A wide range of disseminated to net-textured and patchy net-textured sulphides typically occur in the 138 Zone. This type of mineralization is referred to as MZ mineralization. In the 138 Zone, MZ type sulphides appear to form a wedge-like zone of 200 m length, 120 m to 160 m height and a width of approximately 50 to 90 m, starting at ~350 m depth. The mineralization is hosted in FGO and contaminated FGO, i.e. in MZNO and FGO lithologies.

3) The SMSU Zone

The SMSU Zone forms the bulk of the defined Mineral Resource and occurs in the upper part of the CGO intrusion as an elongated boudin-aged tubular-shaped zone at the top of the CGO (Figure 7-6). Two SMSUs (Upper and Lower) have been modelled. The Upper SMSU body dimensions are 400 m long, 40 m to 80 m wide and 40 to 70 m vertically at a depth of 300 m to 325 m. The Lower SMSU body dimensions are 350 m long, 40 m to 65 m wide and 40 to 70 m vertically at a depth of 445 m to 485 m. Within the SMSU Zone is a core of interstitial net textured sulphides (50% sulphides) (Figure 7-9). Surrounding the net textured sulphides are disseminated sulphides forming a peripheral halo decreasing towards the CGO margins. This halo has been shown to have elevated Cu and PGE tenors that could be used in targeting SMSU extensions. The SMSU appears spatially associated with the presence of the MSU, emplaced approximately 50 m below the MSU. SMSU has only been observed in the CGO when MSU is present at the base of the FGO-Country rock above.

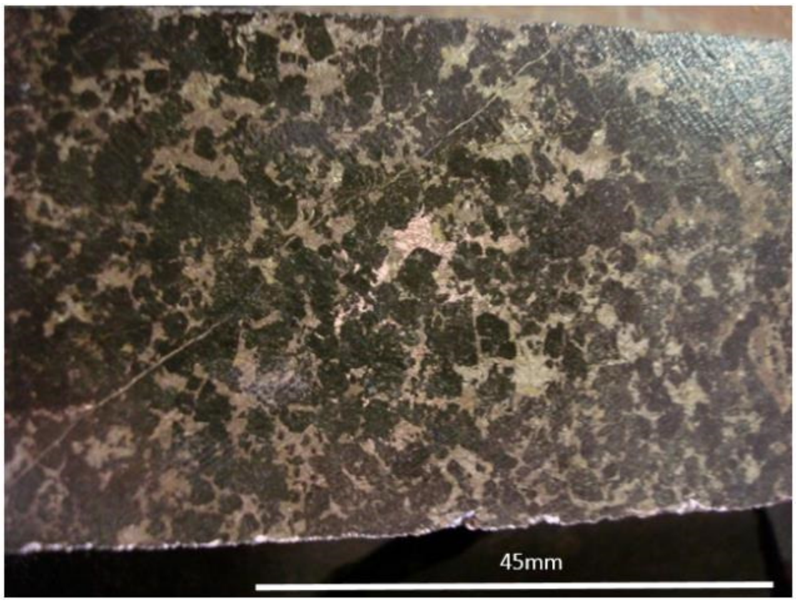


Figure 7-9: SMSU (net textured) Sulphide from Tamarack Drill Core

4) The MSU Zone

MSU-type mineralization is defined as containing 80-90% sulphide (Figure 7-10). The MSU also refers to a mineralized body hosted by intensely metamorphosed and partially melted meta-sediments occurring as fragments or wedges of country rock at the base of the FGO with typical dimensions of 10 to 30 m wide by 0.5 m to 18 m thick. The MSU has a strike length of 550 m at a depth of 275 m (N) to 550 m (S). Close to moderately spaced drilling (35 m to 100 m) to test these massive sulphides suggests that they form southward plunging, pipe-like zones. The zone has been drill intersected intermittently over 550 m from the SMSU to the 138 Zone. Texturally these massive sulphides occur in intensely metamorphosed sediments.

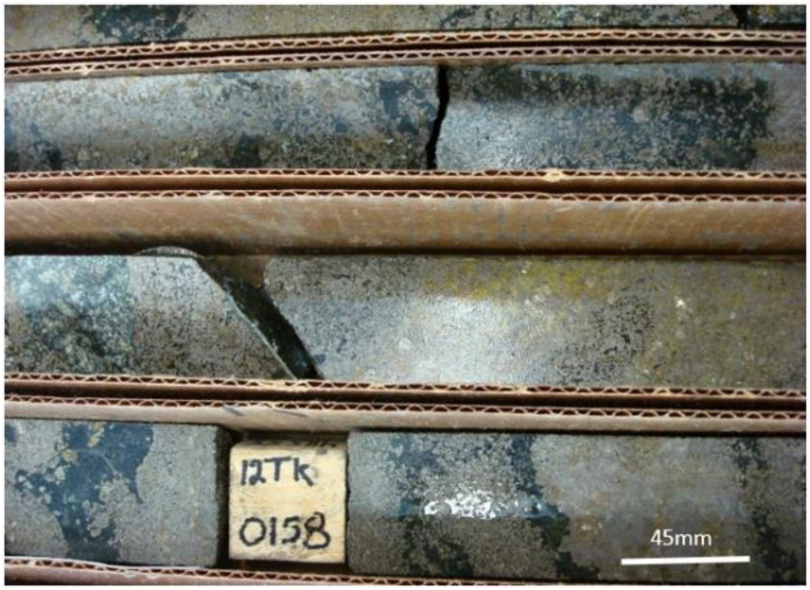


Figure 7-10: MSU from Tamarack Drill Hole 12TK0158

5) The CGO Bend

The CGO Bend Zone consists of basal FGO MSU-MMS mineralization, and signifies where CGO forms a dog leg bend immediately N of the Tamarack Zone. The CGO Bend sulphide mineralization is a footwall accumulation of primary sulphides in the FGO Keel and basin that vary in thickness from 0.2 m to 2.3 m, strike length of ~500 m, at an average depth of 150 m depth and a weak plunge to the S at 10˚. The sulphides are blebby to massive in texture. Historic drill hole 13TK0187, which graded 3.82% Ni and 1.62% Cu, 0.63 grams per tonne (g/t) PGE and 0.36 g/t Au over 2.33 m from a depth of 138.94 m was drilled in the northern section of the eastern CGO Bend (Figure 7-11).

The potential for the mineralization is also supported by prominent DHEM conductors (Figure 7-11) and a recent low-frequency time domain electromagnetic (TDEM) survey over the eastern trend (Figure 7-11). A recent exploration program has demonstrated that the CGO Bend basal FGO MSU/MMS extends 115 m further N with hole 238 with 2.2 m (from depth 117.72 m) at 1.75%Ni, 0.89% Cu (Press release, December 13, 2016). The new results show an exploration potential along the FGO base of 600 m in strike and 200 m in width at shallow depth (115 m in the N to 225 m in the S) (Figure 7-11).

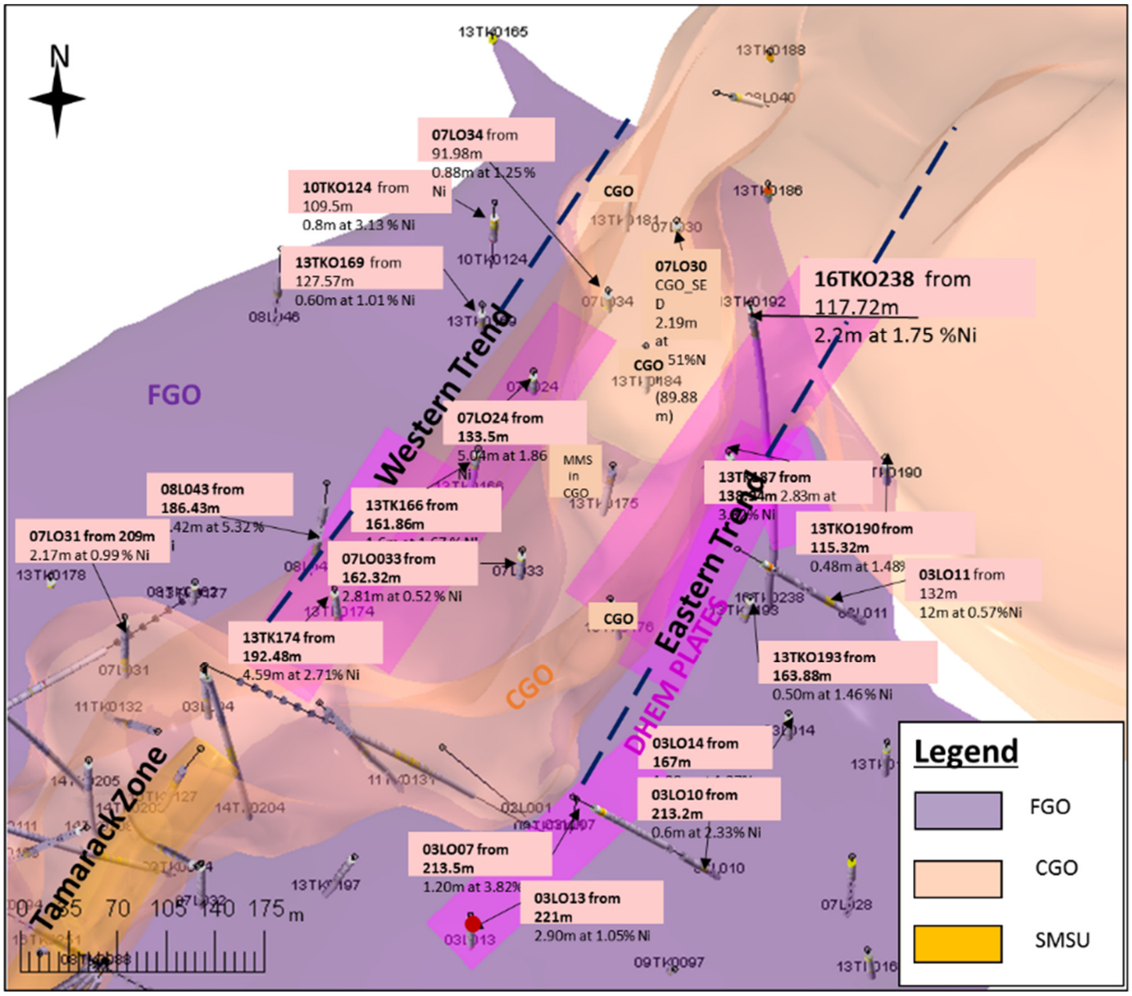


Figure 7-11: Plan View Showing CGO Bend up-Dip of the Tamarack Zone with Locality of Drill Hole 16TK0238 Towards N of CGO Bend Eastern Trend. Also shown are other historical drill hole intercepts and interpreted DHEM conductors which support potential for continuity of mineralization at FGO base both to E and W of CGO

6) The 480 Zone

Drilling in a narrow linear, E-W trending, positive magnetic anomaly at the northern portion of the Tamarack North Project, referred to as the 480 Zone, has intersected disseminated and net textured sulphide mineralization at a relatively shallow depth. The host olivine cumulates visually resemble olivine cumulates of the FGO intrusion to the S and include intervals of quartz xenolith rich magmatic breccia similar to those in the 164 Zone. The 2017 drilling program has tested the extent of the FGO and mineralization in the area. The interpretation of the results in the area has defined the relatively limited extent of mineralization, however the FGO-like intrusion that is extending E would require additional geophysical survey to define a suitable target.

7) Mineralization in the Weathered Laterite Zone

A weathered lateritic profile is irregularly preserved in the northeastern part of Tamarack North Project beneath Cretaceous and Quaternary cover and has concentrated Ni, Cu, Cr, and Fe. The weathered profile is up to 10 m thick, at 35 m depth and consists typically of a 0.5 m pisolithic, limontic hard cap, underlain by massive greenish saprolite, and saprock with remnant igneous textures. Native Cu up to 2% (visual estimation) can be observed as 1 to 3 millimetre (mm) nuggets and veinlets in the weathered profile and persists into the serpentinized upper part of the FGO (Goldner, 2011).

7.2.6 Quaternary and Cretaceous Cover and Weathering Profile

The Tamarack North Project does not outcrop at surface as it underlies 20 to 50 m of Quaternary glacial and fluvial sediments and in the N of the Tamarack North Project along the E part of the intrusion. Cretaceous siltstone and mudstone are preserved and unconformably overlie the preserved paleo-weathered lateritic profile of the FGO.

In the Tamarack North Project, the lateritic weathering profile is variably preserved. This is seen particularly in the E where up to 10 m thick saprock with remnant igneous textures and massive greenish saprolite covered with a pisolitic limonitic duri-crust can be found. Native Cu occurring as nuggets and veinlets can also be observed.

Serpentinization of olivine cumulates occurs over considerable thicknesses in the FGO below the weathered lateritic profile and is believed to be due to supergene alteration processes related to pre-Cretaceous weathering. Magnetite generated by the serpentinization process in the upper layers of the FGO is the main cause for the strong positive magnetic anomaly associated with parts of the Tamarack North Project.

Quaternary glacial-lacustrine deposits between 20 to 50 m cover the TIC with thicknesses increasing towards the S. The deposits are a complex arrangement of glacial and interglacial fluvial sands and silt and clay from lake sediments.

7.2.7 Current Models for Formation of the Ni-Cu-Co Sulphide Mineralization in the Tamarack North Project and Mineralization Area

The Tamarack North Project area contains two intrusions, the FGO rich intrusion and a CGO rich intrusion. Based on the geochemistry, both intrusions are derived from the same high-Mg olivine tholeiitic parental magma (Goldner, 2011).

Based on data available at the time Goldner (2011) proposed that the CGO was emplaced before the FGO intrusion. There are no uranium-lead (U-Pb) zircon age dates for the FGO intrusion, however contact relationships and paleomagnetic correlations with MCR volcanic rocks may indicate that the FGO is older than the CGO. The FGO is believed to be the primary source of the sulphide mineralization at Tamarack. The FGO intrusion is an open system magma conduit (termed a chonolith) that likely followed a zone of structural weakness in the meta-SED Animikie basin. The FGO magma likely intruded along a rift associated structure to produce the dyke-like CGO and the FGO sill-like body.

The low Ni content of olivine in the FGO coupled with the Ni, Cu, and PGE-depleted geochemistry of the upper part of the intrusion indicate that the magma achieved sulphide saturation well-before the crystallization of large amounts of olivine. In the TIC area, the FGO intrusion has the geometry of an elongate lopolithic sill. The FGO magma either carried sulphide formed at a greater depth in the plumbing system or it formed in-situ from the overlying open system magma column as the FGO intruded the Animikie Group SED rocks.

Sulphur Isotope studies indicate that the sulfur originates from Proterozoic and Archean crust as well as mantle contributions from the magma. As the flow rate of magma within the FGO intrusion decreased, the dense immiscible magmatic sulphide started to settle and coalesce towards the base of the intrusion. Sulphide that reached the basal contact, flowed toward topographic lows on the chamber floor and was able to accumulate in pools forming massive sulphide. Crystallization of olivine in the overlying FGO magma column resulted in trapping sulphides as disseminations and blebs. These sulphide textures occur in the ultramafic rocks above the keel of the intrusion and on the flanking sides of the N-S trending lopolithic sheet. The most important control on the loci of massive sulphide deposition is at the base of the FGO or along the keel of the FGO where, for example, the Tamarack Zone mineralization occurs.

The second phase of magmatic intrusion occurred at 1105 +-1.2 Ma (U-Pb age date on zircon) to form the CGO intrusion. The CGO intruded along a similar or perhaps, the same structure as the FGO, with a dyke-like configuration. The high Ni content of CGO and the normal Ni abundance levels in the un-mineralized CGO indicate that the magma did not reach sulphide saturation. The existing sulphide is in disequilibrium with the melts that formed the ultramafic rocks of the CGO, and so the CGO magma contributed negligible sulphide to the mineral zones at the Tamarack Project. As a result, the CGO did not form the mineral zones found within it.

The evidence suggests that the CGO intruded the country rock directly below the keel of the FGO in the Tamarack Zone. The CGO magma eroded the base of the FGO as well as portions of the basal accumulation of previously solidified magmatic sulphide mineralization at the base of the FGO, which represented a proto-ore for the CGO mineral zone. The eroded basal sulphide melted and digested by the CGO magma to form the SMSU. The remnant massive sulphides are preserved on the flanks of the FGO keel current as the MSU and the primary massive sulphide mineralization from the FGO keel was likely re-assimilated and re-concentrated by the CGO to form the SMSU which is hosted in the CGO directly below the FGO keel. The mineral zone in the CGO has a zoned composition grading from Ni-rich massive sulphides at the core to more Cu- and PGE-rich mineralization at the flanks. It appears that the nexus of CGO-related mineralization occurs where the CGO is proximal to the keel of the FGO. Whereas in areas where the CGO has not intruded at the Keel of the FGO, sulphide pool at the base of FGO may remain in their primary undisturbed location.

The MZ contact relationship with the FGO is gradual and likely shows a gradation textural change to the FGO. The MZ chemical composition resemble the FGO signature however it shows a more crustal SED contamination. We interpret the MZ to represent the contamination of FGO with country rocks sediments by thermal erosion.

8. DEPOSIT TYPES

The Tamarack North Project hosts magmatic Ni-Cu-Co-PGE sulphide mineralization. These deposits form as the result of segregation and concentration of liquid sulphide from mafic or ultramafic magma and the partitioning of chalcophile elements into the sulphide from the silica melt (Naldrett, 1999).

In order to sufficiently concentrate metals in a system, a number of basic factors are believed to be necessary including:

- A tectonic rift setting with upwelling mantle and deep-seated structures necessary to generate partial melting of primitive magmas;
- Large volumes of magma flowing through an open system to achieve a high R factor (ratio of melt to sulphide);
- Mid-level external sulphur source from crustal assimilation of sulphur rich rocks to maintain sulphur saturation and continued partitioning with a rising magma;
- Physical and chemical conditions for sulphide accumulation such as cumulate settling, changes in flow velocity, magma mixing etc.

Ni-Cu-Co sulphide deposits are economically important because they present favourable economics compared to the mining and processing of Ni laterite deposits. This is due to their relatively high-grade and comparatively low capital cost requirements.

The various mineralized zones at the Tamarack North Project occur within different host lithologies, exhibit different types of mineralization styles, and display varying sulphide concentrations and tenors. These mineralized zones range from massive sulphides hosted by altered sediments in the MSU, to net textured and disseminated sulphide mineralization hosted by the CGO in the SMSU; to a more predominantly disseminated sulphide mineralization as well as layers of net textured sulphide mineralization, in the 138 Zone (Table 8-1). Mineralization in the 138 Zone, where interlayered disseminated and net textured mineralization occurs is referred to as MZ mineralization. All these mineralization types are typical of many magmatic sulphide ore bodies around the world. The current known mineral zones of the Tamarack North Project (SMSU, MSU and 138 Zone) that are the basis of this resource statement are referred to as the Tamarack Zone. Also located within the Tamarack North Project are two currently lesser defined mineral zones, namely the 480 and the 164 Zone.

Table 8-1: Tamarack North Project – Key Geological and Mineralization Relationships

Area	Mineral Zone	Host Lithology	Project Specific Lithology	Mineralization Type
Tamarack Zone	SMSU	Feldspathic Peridotite	CGO	Net textured and disseminated sulphides
	MSU	Meta-Sediments/ Peridotite (basal FGO mineralization)	Sediments	Massive sulphides
	138	Peridotite and Feldspathic Peridotite	MZ/FGO	Disseminated and net textured sulphides
	CGO Bend	Feldspathic Peridotite	CGO	Disseminated sulphides
		Peridotite footwall (basal FGO mineralization)	FGO	MMS and MSU
Other	221 Zone	Feldspathic Peridotite	CGO	Disseminated sulphides with ripped up clasts of massive sulphides
	480 Zone	Peridotite	FGO	Disseminated sulphides
	164 Zone	Peridotite	FGO	Blebby sulphides, sulphides veins

9. EXPLORATION

9.1 Historical Investigations

The TIC was initially targeted from the Minnesota State airborne magnetic survey flown between 1972 and 1983 and the follow-up drill-testing by the MGS in 1984 of two holes, with peridotite intersected in AB-6 which was drilled on an anomaly N of the town of Tamarack.

9.2 Exploration by Current Owners

The TIC and associated mineralization was discovered as part of a regional program initiated by Kennecott in 2000. The focus on Ni and Cu sulphide mineralization was initiated in response to a 1999 model proposed by Dr. A.J. Naldrett of the potential for smaller feeder conduits associated with continental rift volcanism and mafic intrusions to host Ni sulphide deposits similar to Norilsk and Voisey's Bay. This model (Dynamic Conduit Model) challenged previously held models that Ni sulphide deposits were only associated with large layered complexes.

Exploration by Kennecott continued at the Tamarack Project concurrently with their testing of other targets since 2014. Disseminated mineralization was first intersected at the Tamarack Project in 2002, and the first significant mineralization of massive and semi massive sulphide was intersected in 2008.

To date, exploration has included a wide range of geophysical surveys including; airborne magnetic and EM (EM-MEGATEM and AeroTEM), ground magnetic and EM, IP, gravity, seismic, MALM and downhole EM. Recently (2015 and 2016) a number of new geophysical surveys were conducted. These included Gravity, MT and TDEM surveys. New inversions and 3D modeling were also conducted using current and pre-existing geophysical data. This new geophysical data and data products has enhanced the understanding of the Tamarack Project, improved focus on existing targets.

Drilling in the main target areas of the Tamarack North Project has included 242 diamond drill holes totalling 100,692 m.

9.2.1 Geophysics

The Tamarack Project is covered by Minnesota government regional magnetic and gravity surveys. The magnetic data in particular is recent of good quality and has played a key role in the recognition of the TIC and the targeting of early drilling.

A wide variety of airborne, ground, and DHEM geophysical surveys have been conducted by Kennecott at the Tamarack Project since 2001 (Figure 9-1). Airborne EM and magnetic surveys have included airborne MEGATEM (2001) and AeroTEM (2007, 2008, 2009).

Ground geophysical surveys included EM 37 (2002), Crone TEM/TDEM (2003 and 2016), Audio-Frequency Magneto-Tellurics (AMT) (2003), Seismic Reflection (2006), Controlled Source Audio-Frequency Magneto-Tellurics (CSAMT) (2006), University of Toronto Electromagnetic System (UTEM) (2006), 3D RES/IP (2008), MALM (2008 and 2010), Gradient & Dipole IP/Resistivity (2010), gravity surveys (2001, 2002, 2011, 2015, and 2016), and MT (2016).

A test line to evaluate different surface transient electromagnetic (TEM) systems was surveyed with the UTEM, the Crone system with SQUID sensor and with CRA95 coil sensor, the Electromagnetic Imaging Technology (EMIT) system with SQUID sensor, all in 2012. In addition, different borehole (BH) TEM systems were evaluated. These included Crone Geophysics with a fluxgate sensor and a coil sensor, Lamontagne Geophysics with the UTEM and Discovery Geophysics with the EMIT system with fluxgate sensor. Borehole electromagnetic (BHEM) was first tested in 2003 and has been used since as an important tool for the detection and delineation of sulphide bodies in and near drill holes. Most holes since 2007 and all holes drilled since 2011 have been surveyed with Crone BHEM.

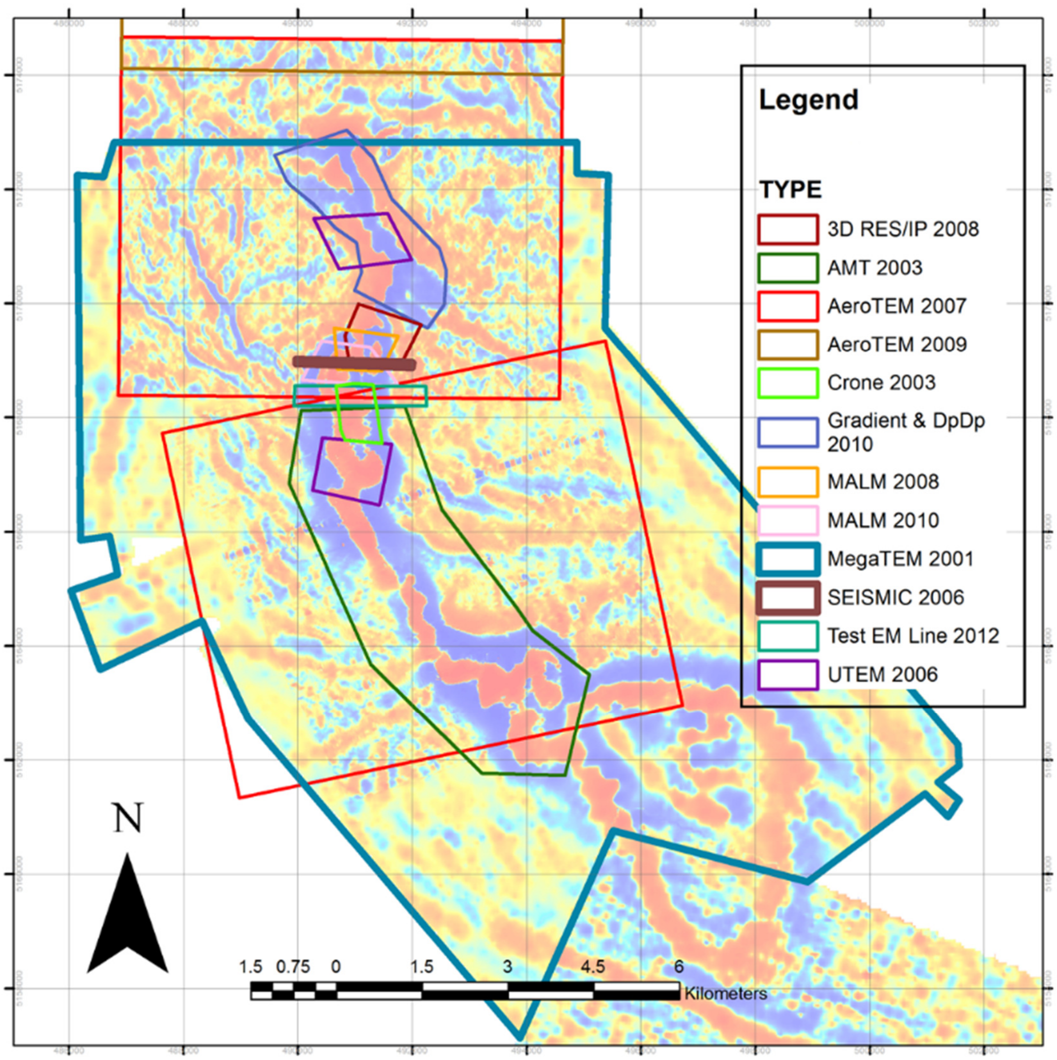


Figure 9-1: Map Showing Localities of Various Geophysical Surveys Conducted Over the entire TIC (composite magnetic TMI image background) Modified from Kennecott Internal Report and Survey Data, 2013.

Airborne Surveys (Magnetic and TEM)

The MEGATEM survey in 2001 identified a conductive anomaly that led to the drilling of the first hole of the program. The hole intersected disseminated mineralization hosted within a gabbro. The survey was strongly affected by the numerous power lines in the area. Subsequent airborne EM surveying was conducted using the AeroTEM system which has a smaller footprint than the more powerful but extended MEGATEM system and hence less sensitivity to nearby power lines (Figure 9-1).

The AeroTEM system operates at lower power and higher frequency than the MEGATEM system. As such there is potentially less penetration through conductive overburden (OB) however it does have the capability of measurements in the "on" time of the transmitted pulse which provides increased sensitivity to very conductive targets. As well, due its smaller footprint it can be less affected by powerlines. The higher resolution (50 m line spacing vs 200 m line spacing for MEGATEM) AeroTEM surveys mapped with increased detail the conductive shallow FGO unit which, at the time, was felt to be spatially related to potentially deeper mineralization. Based on Kennecott's subsequent work it appears that the response from both airborne electromagnetic (AEM) systems is mostly due to the near-surface (top 300 m) conductive FGO unit and that direct detection of mineralization from the air has not yet been achieved.

Magnetic 3D Magnetization Vector Inversion (MVI) Processing

In July 2015, a 3D MVI was performed over the 221 Zone on Tamarack magnetic data using VOXI, Geosoft's cloud based inversion. This MVI-VOXI processing was extended in 2016 to include all the TIC. Due to model size required to produce a resolution of 25 m x 25 m x 12.5 m cells the data was divided into seven separate blocks (Figure 9-2).

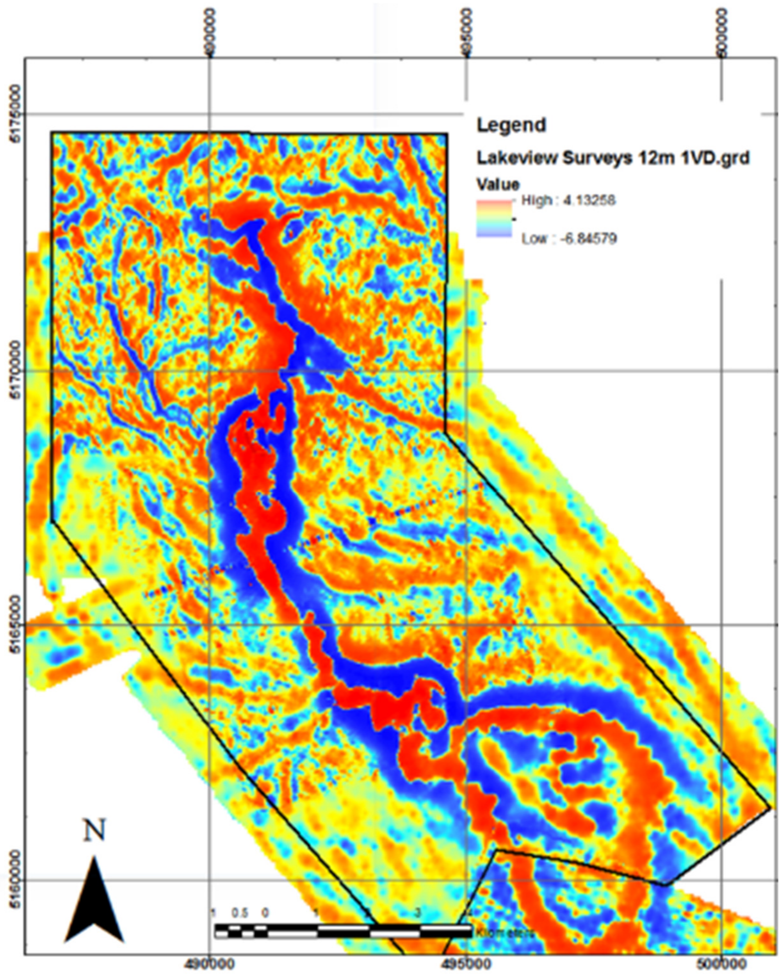


Figure 9-2: Merged First Derivative Magnetic Intensity with MVI Inversion Block. Airborne Magnetic survey (2001, 2007 and 2009). Scale bar is in km

Ground Surveys

Electrical and EM Surveys

A variety of ground electrical and EM have been conducted on the property. Surveys included EM 37 (2002), Crone TEM/TDEM (2003 and 2016), AMT (2003), CSAMT (2006), UTEM (2006), 3D RES/IP (2008), MALM (2008 and 2010), Gradient & Dipole-Dipole IP/Resistivity (2010), and MT (2016).

TDEM Survey, September 2016

A new high power low frequency TDEM was conducted along the eastern CGO Bend by Crone Geophysics in September 2016 (Figure 9-3). The fixed in-loop survey was testing potential thicker zones of base of FGO massive sulphide in the 40 m to 240 m depth range. The survey used two coincident 600 m x 600 m loops to increase the transmitted power. The survey successfully highlighted shallow conductors at the base of the FGO that are interpreted from drill intersections to be sulphides. These conductors also correspond with modelled DHEM plates.

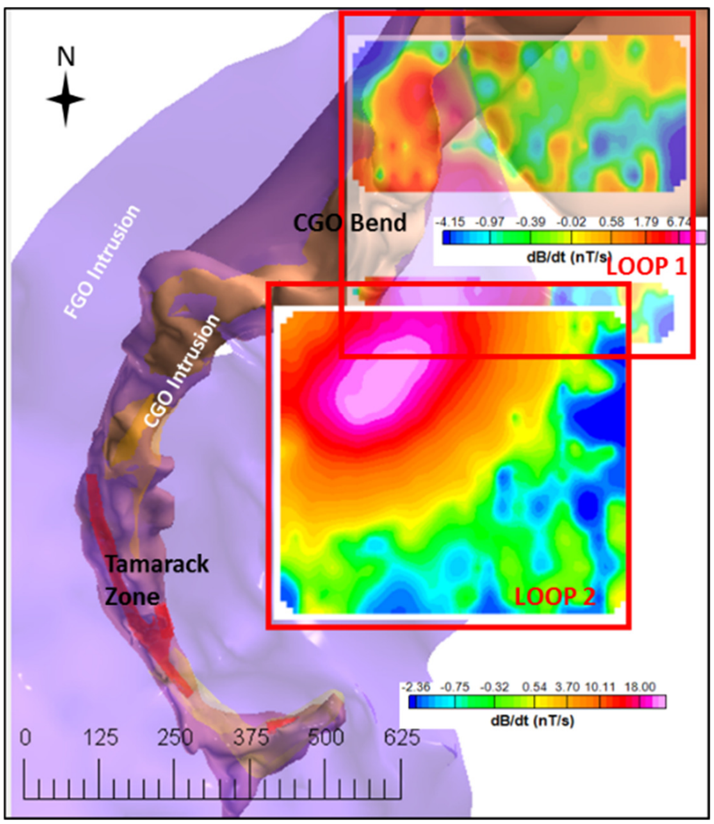


Figure 9-3: Colour Shaded Grids of Ch 20 Crone TEM Z Component for Loop 1 and 2 of TDEM Survey in CGO Bend Zone, Showing Anomalous Conductivity at Depth to the E of the CGO.

Kennecott completed detailed gravity surveying over both the Tamarack North and South properties in 2001, 2002 and 2011 to add to the available Minnesota State data. The new data did not change the larger picture much but provided more detail over the TIC.

Gravity Surveys

Gravity surveys conducted in 2015 and 2016 over the entire TIC have added considerable definition primarily to the Tamarack North Project area (Figure 9-4). These surveys were conducted in a number of phases and have been integrated with the older surveys. The 2015 ground survey consisted of 453 stations at a 200 m spacing and was conducted by Eastern Geophysics. The survey was initially targeted on the high density intrusive drilled in 15TK0221. The 2016 survey (Eastern Geophysics) with a total of 865 ground stations both expanded on and infilled gaps within the existing data. Survey data was integrated with previous data and unconstrained and constrained 3D VPmg inversions models were produced.

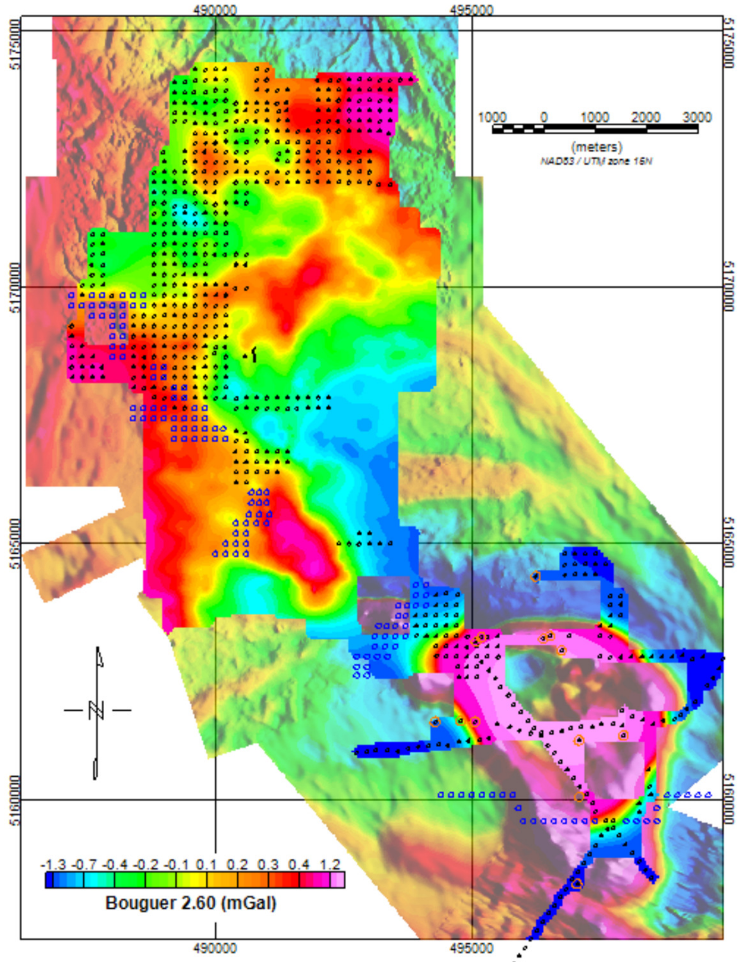


Figure 9-4: Bouguer (2.6 g/cc) Gravity Grid Combining 2011, 2015 and 2016 Surveys with Second Order Trend. Removed. Dots show locations of new data acquired in 2016 (Kennecott Gravity Survey, 2001, 2002, 2011, 2015 and 2016)

Figure 9-5 shows the dominant anomalies located in the 221 Zone S to the CGO Bend as well as the 480 Zone and W of the Tamarack Zone.

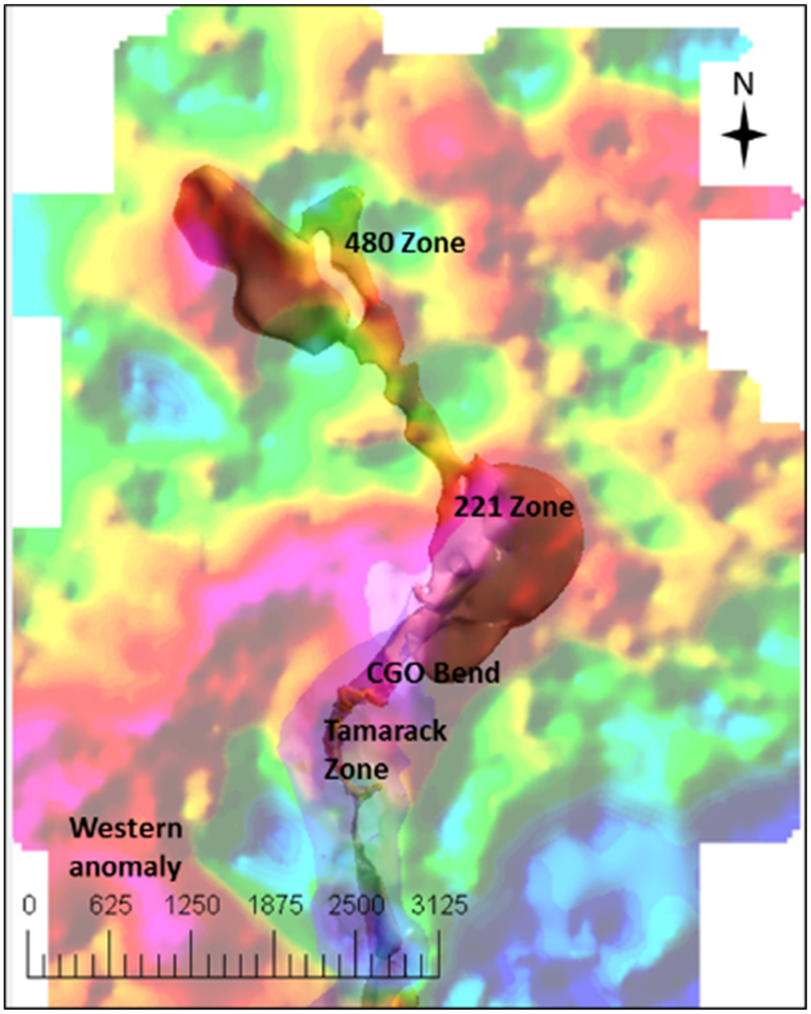


Figure 9-5: Unconstrained Bouguer (2.6 g/cc) Gravity Grid of Northern Tamarack with Modelled CGO Showing the 221 to CGO Bend Anomaly, the 480 Anomalies and the Western Anomaly (Kennecott Gravity survey 2016)

Seismic Reflection (2006)

Seismic reflection surveys were carried out on one test line and two survey lines.

DHEM Surveys

To date, approximately 166 of the 246 holes at Tamarack North have been surveyed with the Crone DHEM system.

Data for all holes was presented as privacy enhanced mail (PEM) files (off-time data with one on-time channel) and for many holes the step datas (STPs) were provided as well. The response from the DHEM surveys is dominated by the conductive FGO response which decays at late time and the response from the MSU and SMSU units which persists generally until late time. The DHEM surveys are very successful in locating sulphides in and near the drill holes. The use of STP has proven to be very successful in expanding the MSU in the Tamarack Zone.

MT Survey

A MT survey was completed in August 2016 by Quantec Geophysics, with 456 ground stations (including 52 repeats) over the Tamarack Project. Final 3D modeling was conducted subsequently. It was anticipated that the MT would provide an efficient way of extending known mineralization or identifying new large, deep conductive features. It was postulated that the lower frequency data should be able to separate the more conductive sulphides from the less conductive sulphide bearing FGO at depth. The unconstrained MT survey identified anomalies outside of the known mineralized zones. The effects of serpentization above the known mineralized zones (138 Zone, Tamarack Zone, and CGO Bend Zone) failed to provide sufficient resolution of the SMSU and MSU.

10. DRILLING

10.1 Historical Drilling

The historical drilling at the Tamarack Project is restricted to the two drill holes by the MGS that were targeted as follow-up on anomalies generated by the State Aeromagnetic Survey. These included AB-6 (1984) located N of the town of Tamarack which intersected peridotite and AB-5 (1984) which was drilled further S and intersected metamorphosed sediments. This drilling is not part of the current resource but contributes to the overall regional geological interpretation.

10.2 Kennecott Drilling Programs (2002-2013)

Kennecott has conducted extensive drilling at the Tamarack North Project since 2002. Prior to Talon's involvement, this drilling comprised 182 diamond drill holes (Table 10-1, Figure 10-1 and Figure 10-2) totalling 67,541 m with holes between 33.5 m and over 956 m depth for an average hole depth of 534 m. Drilling had been conducted in both summer and winter programs.

Drilling at the Tamarack North Project was initiated in the winter of 2002, with L02-01 intersecting broad zones of low-grade disseminated sulphide mineralization N of the Tamarack Zone.

Between 2003 and 2004 drilling was limited to a few holes (Table 10-1) with the first multi-hole programme of 13 holes carried out in the winter of 2007 when the first significant intersection of disseminated sulphide mineralization was made with drill hole 07L-031 N of the Tamarack Zone.

Drilling was stepped up in the summer and winter of 2008 with 51 drill holes after the first intersections of the SMSU in drill hole 08L-042. During the subsequent delineation of the SMSU Zone in the same year, the MSU was first intersected in drill hole 08TK-0049.

Drilling was reduced in 2009 to 15 holes following the economic downturn and mainly tested new targets while focusing on the 480 Zone to the N of the Tamarack North Project. Drilling in 2010 followed on from 2009 with 20 holes testing new targets with continued focus on the 480 Zone. Drilling in 2011 included five holes N of the Tamarack Zone.

In 2012, the programme was stepped up with 27 holes drilled to the S of the SMSU, with the first wide intersection of predominantly disseminated mineralization and interlayered net textured mineralization from drill hole 12TK-138 (in what was later to be called the 138 Zone).

39 holes were drilled during the 2013 campaign. The highlights included the defining of the 138 Zone, the first intercept of massive sulphide veins in meta-sediments in what is referred to as the 164 Zone (located approximately 1.5 km S of the 138 Zone), and further encountering of disseminated mineralization to the N of the Tamarack Zone.

Table 10-1: Breakdown of Drilling Conducted by Kennecott to 2013

Year	Number of Holes	Metres	Targets
2002	1	276	CGO Bend
2003	8	2,009	Tamarack, CGO Bend, 221 Zones
2004	3	915	Tamarack, 221 Zone, 164 Zones
2007	13	3,082	Tamarack and CGO BendZones
2008	51	19,286	Tamarack, CGO Bend, 221, 480 Zones
2009	15	5,215	Tamarack, 164, CGO Bend, 480 Zones
2010	20	7,347	Tamarack, 142, 164, CGO Bend, 221, 480 Zones
2011	5	1,857	Tamarack, CGO Bend, 480 Zones
2012	27	13,683	Tamarack, 164, 142 Zones
2013	39	13,378	Tamarack, CGO Bend, 142, 164 Zones
TOTAL	**182**	**67,048**	

Note: Due to pre-collared holes (OB) existing in one year and the full cored hole not drilled/completed till a following campaign, the hole completion date has been used as the qualifier for Year and Meterage drilled.

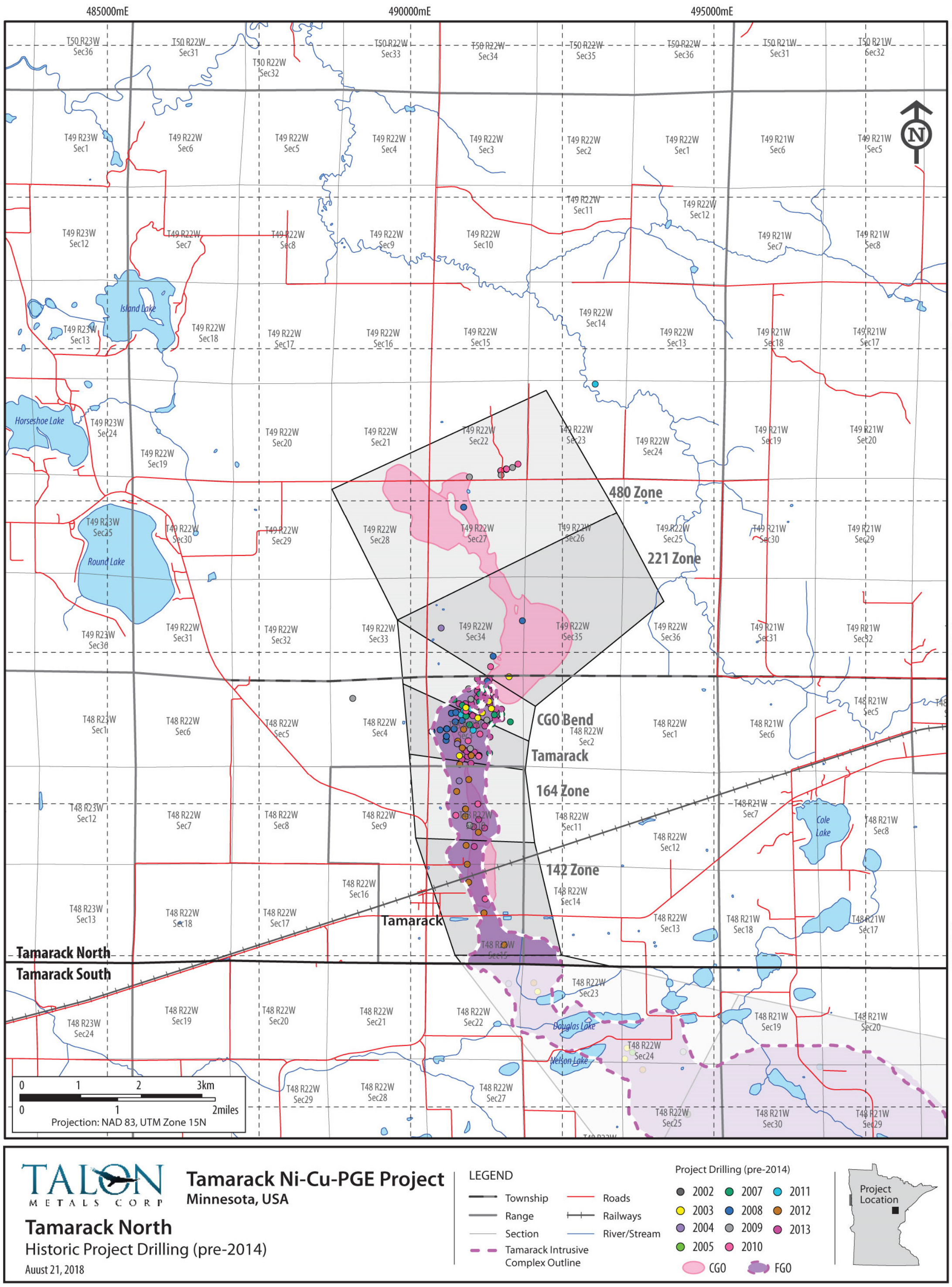


Figure 10-1: Plan View Showing the Locations of the Holes Drilled between 2002 and 2013 at Tamarack North.

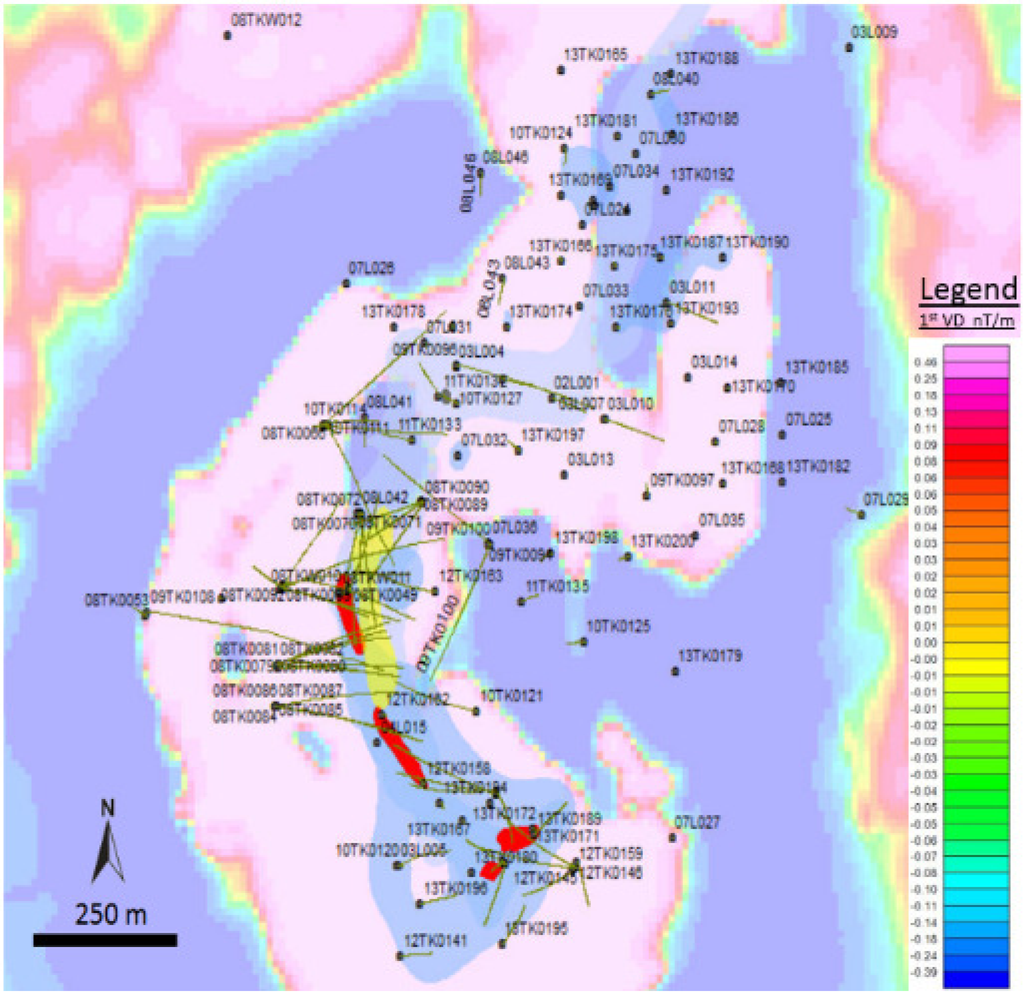


Figure 10-2: Enlarged Map Showing Localities of Drill Holes, in the Tamarack North Project (background 1VD magnetic image). Modified from Kennecott Internal Report and Survey Data, 2013

10.3 Kennecott-Talon Drilling Programs (2014-2017)

The drilling programs conducted by Kennecott (in its capacity as Operator under the 2014 Earn-in Agreement) were generally to be focused on the discovery of large tonnage economic Ni-Cu mineralization compliant with a Rio Tinto Tier One target (large, long-lived, low cost and upper quartile of worldwide commodity specific deposits). Subsequently however, the drilling targeted a wide range of purposes: 1) new targets based on current geologic models, 2) new targets based on geophysical characteristics but no lithologic knowledge, 3) extrapolation of existing mineralization, and 4) infill/delineation of existing mineralization.

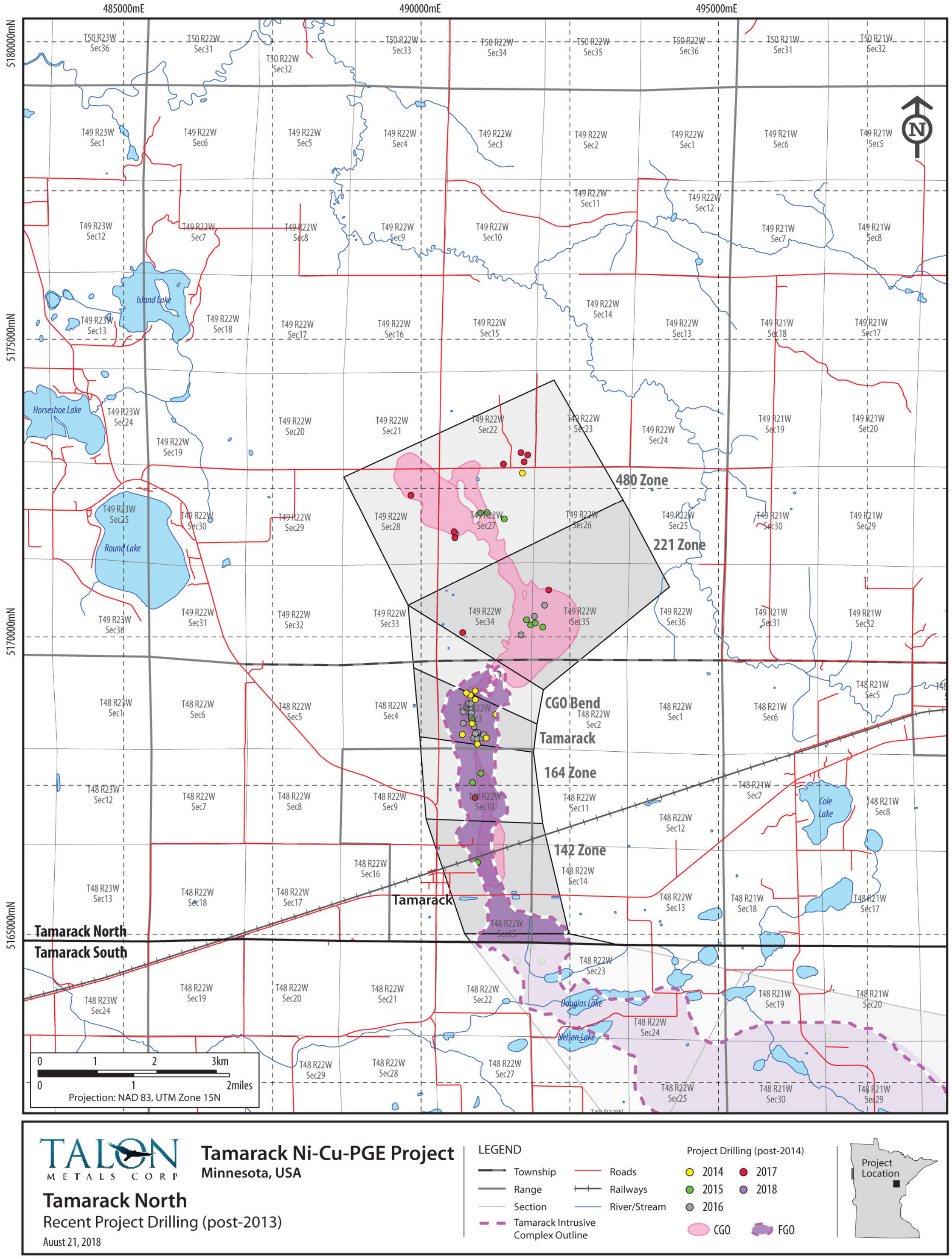


Figure 10-3: Plan View Showing the Locations of the Holes Drilled between 2014 and 2018 at Tamarack North

The 2014 drilling season saw 12 new holes drilled primarily concentrated in the Tamarack Zone. Extension of the MSU/SMSU was the primary focus. The continuation of the CGO intrusion between the Tamarack and 164 Zones was also tested. A single hole in the 480 Zone tested a magnetic low (Figure 10-3 above).

The 2015 drilling season saw 10 new holes drilled, one historic hole deepened, and two holes pre-collared through OB (Table 10-2 notes). 12LV0143 was deepened due to a reinterpreted BHEM suggesting the possibility of a CGO intrusion at depth. The 480 Zone was tested targeting further magnetic lows. Several holes in the 221 Zone tested newly discovered mineralization within a thin “FGO-Like” Brecciated intrusion that occurred at the contact between a thick overlying CGO intrusive and the host SED Thomson Formation. The remaining holes tested for a continuation of the CGO intrusion S of the Tamarack Zone within the 164 and 142 Zones. (Figure 10-3).

2016 drilling saw an aggressive campaign where 19 new holes were drilled, four new wedge (daughter) holes and the completion of one previously pre-collared hole (15TK0220). Further drilling testing the newly recognized, but thin mineralization at the base of the CGO intrusion continued in the 221 and CGO Bend Zones. Extending MSU and infilling both the existing MSU and SMSU mineralization completed the rest of the drilling.

The 2017 drilling program consisting of 12 holes was primarily focused to the N of the 221 Zone with the minor exception of one hole located to the far W of the 221 Zone and another in the 164 Zone (Figure 10-3 for locations). One hole consisted of a pre-collared depth (OB). Four holes were focused on extending previously identified (2009-2010) shallow mineralization within the 480 Zone. Two holes were in the previously untested western 480 Zone targeted a negative magnetic and a high gravity anomaly. Two holes located in the SW of the 480 Zone targeted negative magnetic and a low gravity anomaly. One hole located to the extreme N of the 221 Zone was targeted as a significant step-out of the existing thin, deep basal mineralization characteristic of the 221 Zone. Drill hole 17TK0261 targeted a high gravity anomaly approximately 670 m W of the Talon-modelled CGO intrusion. The final hole within the 164 Zone targeted a potential basal depression in the Talon-modelled FGO intrusion interpreted from gravity and magnetic data.

The 2018 campaign saw four holes drilled; one new hole in the 480 and three wedge holes in the 221 Zone. The 480 Zone hole followed up on a DHEM anomaly from previous drilling. The three wedge holes in the 221 Zone were 25 to 35 m step-outs from hole 15TK0229 looking for extensions of known MSU mineralization.

Table 10-2: Breakdown of Drilling Conducted by Kennecott-Talon Joint Venture

Year	Number of Holes	Metres	Targets
2014	12	7,298	Tamarack, CGO Bend and 480 Zones
2015	12	7,580	480, 221, Tamarack, 164, and 142 Zones
2016	24	13,596	Tamarack, CGO Bend, and 221 Zones
2017	12	5,456	480, 221, and 164 Zones
2018	4	1,383	480 and 221 Zones
TOTAL	**64**	**35,313**	

*Hole 12LV0143 was deepened by 494.5m in 2015.
Note: Due to pre-collared holes (OB) existing in one year and the full cored hole not drilled/completed till a following campaign, the hole completion date has been used as the qualifier for Year and Meterage drilled.

10.4 Resource Drill Holes

The number of total drill holes in the Tamarack North Project (246) and the number of drill holes that were included in the Mineral Resource estimate are different. Drill holes that had mineralized intercepts that were sufficient to meet the domain modeling cut-off and had sufficient continuity or weakly- to non-mineralized that helped define the limits of mineralization were included in the Mineral Resource estimate (see Section 14 for further details). The drill holes and the mineral intercepts that were used in the Mineral Resource are provided in Table 10-3 and Figure 14-1. Some of the remaining drill holes, occurring outside of the current Mineral Resource estimate (as defined in Section 14), do include relevant mineralization that could be included in an updated Mineral Resource estimate depending on results of future exploration programs.

Provided in Table 10-3 are the drill hole composited, mineralized intersections for the SMSU, MSU and 138 Zones from the Mineral Resource estimate provided in Section 14. The SMSU and MSU Zones consist of plunging pipe-like mineralization domains which do not have a tabular type geometry. The orientation of the drilling is mainly in the vertical to sub-vertical dip component, therefore there is some uncertainty regarding the relationship between drill hole intersection length and the true width of the deposit in some areas. Each drill hole listed in Table 10-3 includes the entire composited length used in the Mineral Resource estimate and may also include a selection of significant mineralization intervals within the composited length. If a drill hole intersection was composed entirely of significant mineralization the entire composited length was provided.

Golder has estimated the true width to be perpendicular to the plunge based on an average plunge of -25˚ and an average plunge direction of 170˚ for the SMSU and MSU Zones. There is a distinct curving of the MSU orebody below the 138 Zone. A plunge of -25˚ and plunge direction of 130˚ was used in those holes (Table 10-3).

Due to the strictly vertical nature of the drill holes in the 138 Zone there is a weak understanding of the plunge and plunge direction. Mineralization appears to be horizontal to sub-horizontal and therefore a dip of 0˚ and 0˚ dip direction was used to estimate the true width of intersections (Table 10-3).

The estimated true width may be subject to change with additional drilling oriented across the deposit. Figure 14-15 and Figure 14-16 show drill hole cross-sections of the respective orebodies.

Table 10-3: Drill Hole Composites Used in Mineral Resource for Each Mineralized Zone

Zone	Hole No.	Easting (m)	Northing (m)	Elev. (mASL)	Total Hole Length (m)	Azm	Dip	From (m)	To (m)	Sample Length (m)	Estimated True Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
Upper SMSU	08L042	490735	5168848	389	515.7	180	-80	**327.0**	**407.0**	**80.0**	**65.6**	**1.18**	**0.78**	**0.03**	**0.17**	**0.12**	**0.12**	**1.60**
Upper SMSU	08TK0048	490715	5168730	391	908.0	33	-79	**334.0**	**407.5**	**73.5**	**69.6**	**1.48**	**0.83**	**0.04**	**0.17**	**0.12**	**0.12**	**1.93**
								392.5	*397.0*	*4.5*	*4.3*	*4.04*	*1.31*	*0.10*	*0.42*	*0.27*	*0.11*	*4.84*
Upper SMSU	08TK0061	490673	5168988	389	634.3	146	-66	**395.5**	**397.0**	**1.5**	**1.0**	**0.12**	**0.01**	**0.01**	**0.01**	**0.00**	**0.00**	**0.14**
Upper SMSU	08TK0064	490672	5168987	389	492.9	96	-63	**367.5**	**409.5**	**42.0**	**32.5**	**0.68**	**0.44**	**0.02**	**0.18**	**0.10**	**0.10**	**0.95**
Upper SMSU	08TK0067	490735	5168847	389	590.4	168	-70	**372.0**	**415.5**	**43.5**	**32.0**	**0.43**	**0.29**	**0.01**	**0.10**	**0.07**	**0.06**	**0.60**
Upper SMSU	08TK0073	490846	5168867	390	550.5	251	-74	**327.5**	**386.0**	**58.5**	**50.9**	**0.40**	**0.26**	**0.01**	**0.07**	**0.05**	**0.05**	**0.56**
Upper SMSU	08TK0074	490846	5168867	389	531.9	250	-77	**323.5**	**398.5**	**75.0**	**65.7**	**1.44**	**0.86**	**0.04**	**0.15**	**0.10**	**0.12**	**1.90**
								332.5	*335.5*	*3.0*	*2.6*	*2.86*	*1.32*	*0.07*	*0.20*	*0.11*	*0.09*	*3.55*
Upper SMSU	08TK0089	490846	5168866	389	603.7	237	-76	**330.5**	**409.5**	**79.0**	**67.2**	**2.90**	**1.51**	**0.07**	**0.20**	**0.14**	**0.13**	**3.66**
								360.5	*390.5*	*30.0*	*25.4*	*4.10*	*2.01*	*0.10*	*0.21*	*0.15*	*0.15*	*5.10*
Upper SMSU	08TK0090	490848	5168866	390	534.0	217	-71	**355.3**	**415.0**	**59.7**	**47.0**	**0.75**	**0.58**	**0.02**	**0.14**	**0.08**	**0.11**	**1.06**
Upper SMSU	08TK0091	490596	5168734	390	526.7	79	-65	**391.1**	**411.5**	**20.5**	**15.8**	**0.73**	**0.43**	**0.02**	**0.12**	**0.08**	**0.07**	**0.98**
Upper SMSU	08TK0093	490598	5168729	390	545.0	64	-57	**393.5**	**411.5**	**18.0**	**13.7**	**0.62**	**0.50**	**0.02**	**0.36**	**0.20**	**0.15**	**0.98**
Upper SMSU	09TK0094	490970	5168799	389	509.6	310	-61	**352.5**	**429.0**	**76.5**	**72.5**	**0.56**	**0.35**	**0.02**	**0.08**	**0.05**	**0.06**	**0.76**
Upper SMSU	10TK0127	490909	5169024	389	599.9	282	-86	**304.0**	**353.5**	**49.5**	**45.6**	**0.73**	**0.46**	**0.02**	**0.23**	**0.13**	**0.12**	**1.03**
Upper SMSU	14TK0203	490910	5168938	388	651.7	326	-80	**326.5**	**352.0**	**25.5**	**24.4**	**0.44**	**0.22**	**0.02**	**0.28**	**0.16**	**0.10**	**0.65**
Upper SMSU	14TK0204	490909	5169083	388	557.2	141	-83	**304.5**	**335.0**	**30.5**	**26.2**	**0.66**	**0.49**	**0.02**	**0.29**	**0.17**	**0.18**	**1.00**
Upper SMSU	16TK0237	490839	5168769	389	502.3	268	-82	**342.4**	**381.5**	**39.1**	**35.0**	**0.83**	**0.69**	**0.02**	**0.14**	**0.09**	**0.14**	**1.20**
Upper SMSU	16TK0237A	490839	5168769	389	456.6	268	-82	**343.5**	**365.0**	**21.5**	**18.4**	**0.45**	**0.32**	**0.01**	**0.20**	**0.12**	**0.09**	**0.67**
Upper SMSU	16TK0241	490840	5168865	389	480.4	269	-84	**321.0**	**403.0**	**82.0**	**74.7**	**1.42**	**0.83**	**0.04**	**0.16**	**0.10**	**0.11**	**1.86**
Upper SMSU	16TK0242	490707	5168733	391	551.1	74	-85	**361.7**	**390.0**	**28.3**	**25.4**	**0.78**	**0.51**	**0.02**	**0.13**	**0.09**	**0.09**	**1.07**
Upper SMSU	16TK0251	490799	5168870	389	450.3	354	-84	**316.0**	**382.5**	**66.5**	**62.7**	**0.31**	**0.15**	**0.01**	**0.06**	**0.03**	**0.04**	**0.41**

Zone	Hole No.	Easting (m)	Northing (m)	Elev. (mASL)	Total Hole Length (m)	Azm	Dip	From (m)	To (m)	Sample Length (m)	Estimated True Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
Lower SMSU	08L042	490735	5168848	389	515.7	180	-80	**410.0**	**464.0**	**54.0**	**44.3**	**2.36**	**1.55**	**0.06**	**0.54**	**0.38**	**0.28**	**3.26**
								417.5	*428.0*	*10.5*	*8.6*	*4.53*	*2.48*	*0.10*	*0.48*	*0.41*	*0.14*	*5.80*
Lower SMSU	08TK0048	490715	5168730	391	908.0	33	-79	**407.5**	**479.5**	**72.0**	**68.7**	**2.35**	**1.48**	**0.05**	**0.63**	**0.39**	**0.32**	**3.25**
								418.0	*428.5*	*10.5*	*10.0*	*4.18*	*2.46*	*0.09*	*0.53*	*0.36*	*0.23*	*5.45*
Lower SMSU	08TK0049	490718	5168728	391	553.5	183	-80	**435.0**	**460.5**	**25.5**	**20.9**	**0.61**	**0.51**	**0.02**	**1.03**	**0.51**	**0.29**	**1.21**
Lower SMSU	08TK0058	490590	5168609	390	649.5	89	-71	**473.0**	**558.5**	**85.5**	**70.0**	**2.09**	**0.96**	**0.06**	**0.58**	**0.35**	**0.24**	**2.77**
								489.5	*513.5*	*24.0*	*19.6*	*3.44*	*1.34*	*0.09*	*0.42*	*0.28*	*0.13*	*4.24*
Lower SMSU	08TK0061	490673	5168988	389	634.3	146	-66	**445.0**	**493.0**	**48.0**	**31.9**	**0.88**	**0.67**	**0.02**	**0.67**	**0.39**	**0.31**	**1.44**
Lower SMSU	08TK0067	490735	5168847	389	590.4	168	-70	**423.0**	**506.5**	**83.5**	**62.0**	**2.43**	**1.20**	**0.06**	**0.56**	**0.33**	**0.24**	**3.20**
								448.5	*462.0*	*13.5*	*10.0*	*4.19*	*1.80*	*0.11*	*0.36*	*0.29*	*0.13*	*5.17*
Lower SMSU	08TK0075	490588	5168610	390	578.1	71	-68	**449.0**	**514.5**	**65.5**	**56.6**	**2.93**	**1.45**	**0.07**	**0.55**	**0.36**	**0.22**	**3.81**
								459.5	*485.0*	*25.5*	*21.9*	*3.97*	*1.78*	*0.10*	*0.35*	*0.30*	*0.17*	*4.95*
Lower SMSU	08TK0076	490593	5168728	390	553.8	101	-69	**448.5**	**493.5**	**45.0**	**34.1**	**0.96**	**0.72**	**0.03**	**0.76**	**0.40**	**0.32**	**1.57**
Lower SMSU	08TK0077	490592	5168729	390	558.1	100	-72	**449.0**	**482.0**	**33.0**	**26.9**	**0.46**	**0.29**	**0.01**	**0.46**	**0.27**	**0.17**	**0.77**
Lower SMSU	08TK0079	490589	5168605	390	582.8	90	-66	**458.7**	**525.5**	**66.8**	**54.2**	**2.24**	**1.13**	**0.06**	**0.39**	**0.27**	**0.18**	**2.92**
								476.0	*500.0*	*24.0*	*19.5*	*3.87*	*1.17*	*0.10*	*0.39*	*0.27*	*0.13*	*4.80*
Lower SMSU	08TK0081	490587	5168610	390	601.1	71	-69	**452.5**	**522.5**	**70.0**	**60.8**	**1.85**	**0.94**	**0.05**	**0.58**	**0.34**	**0.27**	**2.51**
								466.9	*487.5*	*20.7*	*17.9*	*3.39*	*1.34*	*0.09*	*0.33*	*0.30*	*0.13*	*4.17*
Lower SMSU	08TK0082[1]	490587	5168609	390	708.5	70	-73	**467.5**	**478.0**	**10.5**	**9.2**	**0.17**	**0.05**	**0.01**	**0.14**	**0.07**	**0.03**	**0.26**
Lower SMSU	08TK0083	490583	5168542	390	705.0	98	-67	**533.0**	**563.0**	**30.0**	**23.3**	**0.34**	**0.18**	**0.01**	**0.24**	**0.14**	**0.11**	**0.52**
Lower SMSU	08TK0086	490584	5168542	390	621.5	82	-68	**501.5**	**560.0**	**58.5**	**48.9**	**2.04**	**0.95**	**0.06**	**0.51**	**0.32**	**0.27**	**2.71**
Lower SMSU	08TK0089	490846	5168866	389	603.7	237	-76	**412.5**	**483.0**	**70.5**	**60.8**	**2.13**	**1.16**	**0.05**	**0.56**	**0.36**	**0.28**	**2.88**
								423.0	*430.5*	*7.5*	*6.5*	*4.28*	*2.17*	*0.10*	*0.41*	*0.39*	*0.13*	*5.42*
Lower SMSU	08TK0090	490848	5168866	390	534.0	217	-71	**419.5**	**461.5**	**42.0**	**33.4**	**1.20**	**0.80**	**0.03**	**0.51**	**0.29**	**0.27**	**1.77**

Zone	Hole No.	Easting (m)	Northing (m)	Elev. (mASL)	Total Hole Length (m)	Azm	Dip	From (m)	To (m)	Sample Length (m)	Estimated True Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
Lower SMSU	12TK0162	490775	5168529	388	620.9	230	-90	**475.0**	**518.0**	**43.0**	**38.9**	**0.62**	**0.47**	**0.02**	**0.59**	**0.36**	**0.23**	**1.06**
Lower SMSU	15TK0220	490843	5168638	389	538.9	276	-84	**458.6**	**468.2**	**9.7**	**9.0**	**0.25**	**0.16**	**0.01**	**0.24**	**0.14**	**0.08**	**0.42**
Lower SMSU	15TK0220A	490843	5168638	389	545.0	276	-84	**438.0**	**506.5**	**68.5**	**62.4**	**2.15**	**1.06**	**0.06**	**0.65**	**0.40**	**0.30**	**2.90**
								457.5	*469.5*	*12.0*	*10.9*	*3.49*	*1.34*	*0.09*	*0.42*	*0.30*	*0.19*	*4.31*
Lower SMSU	16TK0235	490845	5168713	389	539.2	282	-81	**436.0**	**463.5**	**27.5**	**24.2**	**0.51**	**0.39**	**0.02**	**0.44**	**0.25**	**0.18**	**0.85**
Lower SMSU	16TK0235A	490845	5168713	389	538.9	282	-81	**418.5**	**497.5**	**79.0**	**69.3**	**1.36**	**0.87**	**0.04**	**0.75**	**0.45**	**0.32**	**2.05**
								435.5	*441.5*	*6.0*	*5.3*	*3.42*	*1.73*	*0.09*	*0.85*	*0.50*	*0.24*	*4.51*
Lower SMSU	16TK0237	490839	5168769	389	502.3	268	-82	**407.0**	**429.5**	**22.5**	**20.2**	**1.36**	**0.71**	**0.03**	**0.44**	**0.30**	**0.21**	**1.89**
Lower SMSU	16TK0237A	490839	5168769	389	456.6	268	-82	**404.5**	**412.0**	**7.5**	**6.5**	**0.50**	**0.32**	**0.02**	**0.54**	**0.31**	**0.19**	**0.85**
Lower SMSU	16TK0242	490707	5168733	391	551.1	74	-85	**404.5**	**466.5**	**62.0**	**55.8**	**2.10**	**1.22**	**0.05**	**0.73**	**0.37**	**0.30**	**2.93**
								412.5	*430.5*	*18.0*	*16.2*	*3.70*	*1.71*	*0.10*	*0.31*	*0.27*	*0.15*	*4.63*
Lower SMSU	16TK0243	490864	5168569	388	605.9	260	-83	**478.0**	**503.5**	**25.5**	**23.5**	**0.70**	**0.40**	**0.02**	**0.64**	**0.37**	**0.28**	**1.14**
Lower SMSU	16TK0244	490708	5168541	389	554.4	88	-84	**493.5**	**510.0**	**16.5**	**14.8**	**0.32**	**0.25**	**0.01**	**0.22**	**0.35**	**0.13**	**0.56**
Lower SMSU	16TK0247	490833	5168672	389	480.1	253	-86	**442.0**	**466.0**	**24.0**	**21.6**	**0.40**	**0.29**	**0.01**	**0.47**	**0.27**	**0.18**	**0.71**
MSU	08TK0049	490718	5168728	391	553.5	183	-80	**396.0**	**408.0**	**12.0**	**9.8**	**6.03**	**3.30**	**0.11**	**0.67**	**0.59**	**0.33**	**7.74**
MSU	08TK0058	490590	5168609	390	649.5	89	-71	**448.8**	**452.2**	**3.3**	**2.7**	**4.96**	**2.56**	**0.08**	**0.52**	**0.45**	**0.46**	**6.31**
MSU	08TK0068	490733	5168847	389	516.3	194	-75	**378.4**	**382.2**	**3.7**	**2.9**	**3.63**	**1.36**	**0.09**	**0.31**	**0.30**	**0.08**	**4.41**
MSU	08TK0075	490588	5168610	390	578.1	71	-68	**420.5**	**423.7**	**3.1**	**2.7**	**5.15**	**2.11**	**0.10**	**0.44**	**0.35**	**0.09**	**6.26**
MSU	08TK0077	490592	5168729	390	558.1	100	-72	**396.4**	**409.9**	**13.6**	**11.0**	**5.82**	**2.68**	**0.13**	**0.51**	**0.44**	**0.22**	**7.25**
MSU	08TK0081	490587	5168610	390	601.1	71	-69	**421.1**	**431.6**	**10.5**	**9.1**	**5.05**	**3.03**	**0.09**	**0.96**	**0.52**	**0.28**	**6.68**
MSU	08TK0083	490583	5168542	390	705.0	98	-67	**497.5**	**507.8**	**10.3**	**8.0**	**7.01**	**2.89**	**0.14**	**1.32**	**0.70**	**0.30**	**8.78**
MSU	08TK0086	490584	5168542	390	621.5	82	-68	**468.0**	**469.5**	**1.5**	**1.3**	**0.02**	**0.01**	**0.00**	**0.00**	**0.00**	**0.00**	**0.02**
MSU	09TK0095	490983	5168407	389	663.9	265	-74	**512.9**	**516.6**	**3.7**	**3.4**	**4.75**	**2.23**	**0.10**	**1.06**	**0.53**	**0.33**	**6.13**
MSU *	12TK0153	490982	5168405	388	683.7	161	-82	**554.5**	**575.3**	**20.8**	**17.9**	**4.96**	**2.11**	**0.10**	**0.41**	**0.37**	**0.12**	**6.07**

Zone	Hole No.	Easting (m)	Northing (m)	Elev. (mASL)	Total Hole Length (m)	Azm	Dip	From (m)	To (m)	Sample Length (m)	Estimated True Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
								558.5	*568.1*	*9.5*	*8.2*	*7.18*	*3.38*	*0.14*	*0.52*	*0.53*	*0.11*	*8.86*
MSU	12TK0158	490850	5168418	388	594.7	58	-89	**482.9**	**495.7**	**12.8**	**11.6**	**5.86**	**2.28**	**0.13**	**1.28**	**0.58**	**0.40**	**7.37**
MSU	12TK0162	490775	5168529	388	620.9	230	-90	**439.1**	**443.0**	**3.9**	**3.5**	**2.64**	**1.15**	**0.06**	**0.13**	**0.23**	**0.13**	**3.26**
MSU *	13TK0171	491049	5168348	389	641.9	157	-90	**573.3**	**581.0**	**7.7**	**7.0**	**8.01**	**2.87**	**0.15**	**0.41**	**0.54**	**0.21**	**9.53**
MSU	14TK0211	490857	5168535	389	648.0	265	-85	**425.0**	**429.0**	**4.0**	**3.7**	**5.74**	**2.07**	**0.13**	**0.68**	**0.40**	**0.10**	**6.94**
								441.0	**456.9**	**15.9**	**14.7**	**7.14**	**2.43**	**0.17**	**0.81**	**0.68**	**0.37**	**8.67**
MSU	14TK0213	490857	5168535	389	618.0	216	-85	**435.7**	**443.4**	**7.7**	**6.9**	**5.09**	**2.22**	**0.10**	**0.91**	**0.47**	**0.31**	**6.42**
								455.1	**464.7**	**9.6**	**8.6**	**7.04**	**2.43**	**0.15**	**1.20**	**0.79**	**0.98**	**8.79**
MSU	15TK0220A	490843	5168638	389	545.0	276	-84	**411.0**	**415.1**	**4.1**	**3.7**	**2.01**	**1.24**	**0.05**	**0.50**	**0.53**	**1.16**	**2.99**
								414.0	*415.1*	*1.1*	*1.0*	*4.79*	*1.97*	*0.14*	*1.05*	*1.18*	*0.37*	*6.19*
MSU*	16TK0233A	490914	5168369	388	583.3	309	-84	**508.0**	**517.0**	**9.0**	**8.4**	**4.94**	**2.08**	**0.10**	**0.57**	**0.43**	**0.24**	**6.12**
								515.0	*516.0*	*1.0*	*0.9*	*9.06*	*3.37*	*0.19*	*0.23*	*0.76*	*0.14*	*10.79*
MSU*	16TK0234	490950	5168389	388	696.8	181	-85	**547.0**	**552.1**	**5.0**	**4.4**	**4.49**	**1.86**	**0.09**	**0.62**	**0.50**	**0.27**	**5.59**
MSU	16TK0235	490845	5168713	389	539.2	282	-81	**381.4**	**392.3**	**10.8**	**9.5**	**4.90**	**2.47**	**0.08**	**0.42**	**0.34**	**0.14**	**6.13**
MSU	16TK0235A	490845	5168713	389	538.9	281	-82	**379.5**	**390.7**	**11.2**	**9.8**	**4.73**	**2.38**	**0.09**	**0.32**	**0.28**	**0.10**	**5.89**
MSU	16TK0243	490864	5168569	388	605.9	260	-83	**418.0**	**428.5**	**10.5**	**9.7**	**5.88**	**2.32**	**0.14**	**0.51**	**0.42**	**0.09**	**7.16**
								435.3	**438.3**	**3.0**	**2.8**	**7.35**	**2.91**	**0.17**	**0.76**	**0.55**	**0.14**	**8.97**
MSU	16TK0244	490708	5168541	389	554.4	88	-84	**448.8**	**450.8**	**2.0**	**1.8**	**9.60**	**4.04**	**0.18**	**0.88**	**0.96**	**0.45**	**11.81**
MSU*	16TK0246	490881	5168290	388	611.4	10	-81	**529.0**	**533.4**	**4.4**	**4.0**	**5.13**	**2.12**	**0.12**	**0.69**	**0.48**	**0.29**	**6.39**
MSU	16TK0247	490833	5168672	389	480.1	253	-86	**398.0**	**403.0**	**5.0**	**4.5**	**3.26**	**2.59**	**0.04**	**0.16**	**0.30**	**0.28**	**4.43**
138	12TK0138	491125	5168286	389	731.5	274	-74	**431.5**	**564.0**	**132.5**	**128.8**	**1.06**	**0.99**	**0.03**	**0.71**	**0.18**	**0.21**	**1.71**
								510.1	*519.7*	*9.6*	*9.3*	*2.49*	*2.09*	*0.05*	*0.81*	*0.40*	*0.36*	*3.68*
138	12TK0146	491125	5168286	389	670.0	293	-75	**430.5**	**524.0**	**93.5**	**90.9**	**0.55**	**0.37**	**0.02**	**0.13**	**0.08**	**0.09**	**0.78**
								442.3	*455.5*	*13.2*	*12.8*	*1.03*	*0.85*	*0.03*	*0.19*	*0.12*	*0.24*	*1.51*

Zone	Hole No.	Easting (m)	Northing (m)	Elev. (mASL)	Total Hole Length (m)	Azm	Dip	From (m)	To (m)	Sample Length (m)	Estimated True Width (m)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
138	12TK0153	490982	5168405	388	683.7	161	-82	**423.0**	**534.0**	**111.0**	**110.1**	**0.46**	**0.31**	**0.02**	**0.11**	**0.07**	**0.07**	**0.65**
138	12TK0156	490996	5168294	388	703.8	293	-83	**417.3**	**533.8**	**116.5**	**115.8**	**0.88**	**0.65**	**0.03**	**0.22**	**0.12**	**0.14**	**1.26**
								495.5	*505.6*	*10.1*	*10.1*	*1.50*	*0.86*	*0.04*	*0.23*	*0.17*	*0.11*	*1.98*
138	12TK0160	490997	5168293	388	634.0	240	-86	**416.0**	**548.0**	**132.0**	**131.9**	**1.07**	**0.84**	**0.03**	**0.27**	**0.16**	**0.18**	**1.55**
								490.8	*504.9*	*14.1*	*14.0*	*2.08*	*1.24*	*0.05*	*0.39*	*0.22*	*0.17*	*2.78*
138	13TK0167	490922	5168361	388	635.8	240	-89	**415.5**	**509.3**	**93.8**	**93.8**	**0.31**	**0.14**	**0.01**	**0.12**	**0.06**	**0.05**	**0.43**
138	13TK0171	491049	5168348	389	641.9	157	-90	**416.0**	**531.0**	**115.0**	**115.0**	**0.65**	**0.45**	**0.02**	**0.17**	**0.10**	**0.11**	**0.93**
138	13TK0189	491051	5168340	389	652.7	47	-85	**415.3**	**524.1**	**108.9**	**108.1**	**0.39**	**0.21**	**0.02**	**0.12**	**0.07**	**0.06**	**0.54**
138	14TK0206	491095	5168293	388	786.0	356	-86	**417.0**	**526.0**	**109.0**	**108.3**	**0.46**	**0.31**	**0.02**	**0.16**	**0.09**	**0.08**	**0.67**
138	16TK0234	490950	5168389	388	696.8	181	-85	**419.0**	**530.0**	**111.0**	**109.5**	**0.44**	**0.24**	**0.02**	**0.10**	**0.06**	**0.05**	**0.59**
								508.4	*529.0*	*20.6*	*20.3*	*0.95*	*0.51*	*0.03*	*0.16*	*0.12*	*0.06*	*1.25*
138	16TK0245	490937	5168279	388	585.0	289	-88	**414.0**	**531.0**	**117.0**	**116.8**	**0.63**	**0.46**	**0.02**	**0.24**	**0.13**	**0.12**	**0.93**
138	16TK0246	490881	5168290	388	611.4	10	-81	**419.0**	**504.5**	**85.5**	**84.8**	**0.43**	**0.29**	**0.02**	**0.12**	**0.07**	**0.08**	**0.62**
138	16TK0248	491049	5168348	389	680.3	142	-87	**417.5**	**538.5**	**121.0**	**120.8**	**0.88**	**0.61**	**0.03**	**0.21**	**0.13**	**0.15**	**1.25**
								482.7	*486.0*	*3.4*	*3.3*	*2.08*	*0.68*	*0.05*	*0.25*	*0.10*	*0.10*	*2.51*
								519.0	*534.0*	*15.0*	*15.0*	*1.41*	*0.93*	*0.03*	*0.37*	*0.26*	*0.27*	*1.99*
138	16TK0250	490999	5168293	388	648.9	169	-88	**419.0**	**547.5**	**128.5**	**128.5**	**0.50**	**0.33**	**0.02**	**0.14**	**0.07**	**0.08**	**0.71**
								428.0	*437.0*	*9.0*	*9.0*	*1.19*	*0.87*	*0.03*	*0.18*	*0.12*	*0.16*	*1.66*

Note: Bold text indicates total hole composite used for Mineral Resource calculation.
Note: Italicized text indicates a significant intersection within the larger composite.
Note: Upper SMSU, Lower SMSU, and MSU (unless otherwise noted) assumed a Dip and Dip Direction of 25/170 for the calculation of estimated true thickness.
Note: * Uses an assumed Dip and Dip Direction of 25/130 for the calculation of estimated true thickness.
Note: The 138 orebody assumed a Dip and Dip Direction of 0/0 for the calculation of estimated true thickness.
Note: Estimated true thickness calculated via Datamine® "TRUETHK" Process.

10.5 Drill Hole and Core Logging Procedures

10.5.1 Drill Site Management

Drilling at the Tamarack North Project is challenged by the extensive wetlands. Drilling initially was restricted to winter months with frozen ground to minimize impacts to swamps and wetlands in the project area. In 2008, drilling was also initiated in the summer months using swamp mats for both access roads and drill platforms which have been very successful in minimizing the impact on the environment.

Kennecott has previously implemented and maintained strict environmental and safety protocols with regard to drilling which include: drilling contracts that ensure safety standards are not compromised, the use of swamp mats for drill platforms and access, and photographing the site before and after drilling and rehabilitation.

Diamond drilling diameters utilized at the Tamarack North Project have been primarily hole (outside diameter): 75.7 mm; core (inside diameter): 47.6 mmm (NQ) and hole (outside diameter): 96 mm; core (inside diameter): 63.5 mm (HQ) wireline. Sonic drilling has been used extensively to pre-collar holes through the overlying glacial sediments which are then completely cased off prior to commencing diamond core drilling. All casing depths and sizes are recorded in the Kennecott acQuire database.

Typical industry standard procedures are followed with all drilling and are outlined in the “Tamarack Core Processing Procedures Manual” including:

- All statutory permits and approvals received by appropriate regulatory bodies prior to drilling.
  (see http://www.dnr.state.mn.us/lands_minerals/metallic_nf/regulations.html)
- Drill collars initially located in the field using handheld GPS. Following completion of drilling each collar is either professionally surveyed or by differential GPS reading and collar position permanently marked with marker on cement cap. If permanent marker cannot be established because of ground conditions a certificate is issued by

surveyor. Collar positions are subsequently checked against high resolution satellite imagery.

- Closure of holes follow regulatory procedures as outlined by the MDH both for permanently abandoned holes, which are cemented from the base to surface with all casing removed, and temporarily abandoned holes, which are temporarily sealed according to regulations if there is a possibility of the hole being deepened or the hole is awaiting a downhole EM survey.

10.5.2 **Core Delivery and Logging**

Kennecott has previously defined and adopted clear procedures for core processing. A split-tube coring system has been adopted for all holes. Exploration holes are designated as either *reconnaissance* or as *resource* with each being treated somewhat differently. Resource core is transferred to V-rails directly from the core tube. Core is then transported a short distance to the core storage site via a customized, secure, v-rail enabled trailer. Core is only transferred to core boxes by the geologist after transport to the core storage site and after being marked-up and processed. This procedure minimizes breakage and ensures the core-orientation (by the Reflex Ace Core Orientation Tool (ACT)) that is used with each core-run is maintained. Reconnaissance designated core is primarily placed into boxes directly from the core tube although it can also be placed in the v-rail system at the discretion of the project manager.

10.5.3 **Geological Logging Procedures**

Geological summary logging is completed immediately on receiving the core while still in the V-rails and is intended to provide an overview of the key lithologies and features with accurate estimates of mineralization. The main unit lithologies are recorded with the codes; SED, FGO, CGO, MZ, SMSU, MSU, MMS etc. The logs are entered into the acQuire database and also prioritized for detailed logging.

Prioritization of core is determined during the summary logging. High priority core is processed, and logged as soon as possible. Lower priority core is retained and stored in V-rails until it can be processed and logged. Core processing and logging procedures include:

- Reference orientation line marking (based on Reflex ACT);
- Measurement conversion and run depth marking (Imperial to Metric);
- Run recovery logging and marking (core loss record);
- Core photography both on rails and boxes;
- Detailed geotechnical logging (logging interval based on geological domains and varied with detail required typically 3.05 m to 6 m). Standard logging and testing includes:
    - IRS Hardness (Rock strength estimation);
    - L10 (RQD);
    - Micro Defects;
    - Alteration Intensity;
    - Joint and fracture count and categorization;
    - Open and cemented joint set number;
    - Point load testing (every 20 m);
    - UCS (uniaxial compressive strength) Sampling;
    - Geotechnical Major Structures (Interval structure logging).
- Detailed Geological Logging: Detailed geological logging is an important process for recording and understanding the geology and mineralization. Kennecott has adopted the system of logging into the acQuire database with specific custom fields and drop-down lists to ensure consistency. The logging includes a lithology log, an alteration log, a mineralization log, a point structure log, a linear structure log (where structure orientations and dips are measured); and a magnetic susceptibility log with a handheld magnetometer (discontinued temporarily in 2008 but subsequently resumed).

10.5.4 Surveying

All collars are professionally surveyed to sub-metre accuracy after completion of the drill hole. Down-hole deviation surveys are conducted on all holes at the Tamarack North Project and include two independent surveys conducted on the hole completion, which include:

- A multi-shot survey with a magnetic tool (Flexit) provided by the drill contractor (survey shots conducted at least 10 m intervals);
- A multi-shot gyroscopic survey conducted by a down-hole survey contractor (survey shots conducted at a minimum of 20 m intervals).

The Flexit tool is susceptible to poor azimuth accuracy in the presence of strongly magnetic lithologies, such as those found at the Tamarack North Project. However, the dip readings are not affected by in hole magnetics and provide a reliable source of dip measurements as the hole progresses. Multi-shot gyroscopic surveys are not affected by magnetics and provide accurate downhole deviation.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Core Sampling and Chain of Custody

Standardized core sampling procedures were introduced by Kennecott in January 2007 and have been incorporated for all the sampling at the Tamarack North Project with only minor modifications made subsequently. The Tamarack North Project has adopted the use of split-tube coring as a means of minimizing core breakage and facilitating the recording of geotechnical and oriented core data (Kennecott Internal Doc, 2016). It is standard practice to sample all core irrespective of lithology type or sulphide content, although sulphide intervals are prioritized. Core is sampled on a minimum of 0.5 m intervals to a maximum of 3 m, with 1.5 m being the most common sample length. The following procedures are adhered to:

- Core is picked up at the drill site by Kennecott staff and returned to the secure core logging facility in the town of Tamarack (Figure 11-1).

Figure 11-1: Photo of Kennecott/Talon Core Processing Facility Tamarack, Minnesota

- Once at the core processing facility, the core is "quick-logged" for major lithological units and sulphide mineralization, and entered directly into the acQuire system database. Further detailed lithological logging will occur later in the process chain once geotechnical logging processes have occurred;
- Sample interval marking: Duplicate sample tags are inserted and displayed on the V-rails for photographing. Once photographed the core is transferred to cardboard core boxes where the tags are stapled to the inside wall of the appropriate rows;
- Core photography is conducted after the sample mark-up is completed on V-rails (definition and some reconnaissance holes);
- Boxed core (reconnaissance holes) is also photographed and was reintroduced in 2012 after being discontinued in 2008;
- In *"definition"* categorized holes, a 15 cm sample is cut from the core for the purposes of density and UCS measurements approximately every 20 m. Preference is given to core representative of the dominant lithology in the 20 m interval at the discretion of the geologist (i.e. at changes in lithology). A density measurement via the hydrostatic-gravimetric method is performed with the sample in the core shack. Dry and wet weights for three density standards are recorded every 20th primary density sample. The scale is also calibrated using calibration weights at this time. The UCS sample is labelled "UCS" with a unique sample tag associated with it, photographed (as part of the regular core photo process) and ultimately placed in a unique sample bag (with tag) until despatched to an appropriate testing laboratory;
- In *"reconnaissance"* holes, UCS sampling does not occur; however, density measurements on 10 cm lengths of core are carried out following the same parameters as identified above in "*definition"* categorized holes;
- Core sawing is conducted after core marking and sample tagging has occurred. Core is consistently cut 1 cm to the right of the orientation line. Both halves are returned to the box;

- Sample packaging: half-core samples (half without the orientation line) are packed, after air drying, in individual plastic bags with the sample ticket inserted inside the bag and the sample number written in permanent marker on the outside. The core is secured, and stored locally, out of the elements, until such time as it can be transported to the State core library in Hibbing, Minnesota;
- The QC protocol is documented by Kennecott and was generally followed at the Tamarack North Project since the start of the program (reportedly modified to the present procedure in early 2008). Current QC samples include:
  - Blanks: inserted at the beginning of every batch, at every 30th sample, at changes in lithology, and specifically, prior to and after highly mineralized samples. Blanks used have included commercially derived Silica Sand; GABBRO-1 (unmineralized half core from hole 07L039); GABBRO-2 (unmineralized half core from 07L038 since July 2008); GABBRO-128 (unmineralized half core from hole 10TK0128); and GABBRO-18 (unmineralized half core from hole 04L018);
  - Standards: a matrix-matched standard (corresponding to the sulphide content of the flanking samples) is inserted into the sample stream every 30 samples to monitor sample accuracy. A corresponding standard is also inserted at the beginning of significant changes in mineralization. The standards were prepared from coarse rejects of the Eagle Deposit (Michigan) (EA type) and Tamarack North Project (TAM type) drill holes and are certified by an independent subject matter expert after Round Robin testing at accredited laboratories;
  - Duplicates: Field, Coarse Reject, ®and Pulp duplicates are routinely used to monitor sampling and assay precision according to the following protocols:
    - Field Duplicates include two quartered core lengths submitted consecutively every 30 samples and are offset from the standards by 10 samples;
    - Coarse Reject Duplicates are splits from the coarse reject material that are inserted every 20 samples by the lab at the request of Kennecott. See Figure 11-3 and Figure 11-4;

  - Pulp Duplicates are randomly generated and assayed by ALS Minerals as an internal process at a rate of one every 30 samples. See Figure 11-5 and Figure 11-6;
  - Check assays from a secondary laboratory were not utilized by Kennecott to confirm the quality of the ALS Minerals values. However, the quality of the ALS Minerals values is monitored using acQuire® protocols for evaluating standards and blanks.
- Sample batches are packed in collapsible plastic bins for shipping. Sample consignments are limited to 200 samples and are grouped in batches of the same rock types and using the same assay methods. A dispatch form is created, with one copy being sealed in the container and the other emailed to the lab. The container is sealed with randomly selected, security tags that are listed in the Chain of Custody Sheet. Access to the samples cannot occur without breaking a seal;
- Samples are shipped to the ALS Minerals lab in Thunder Bay, Ontario, Canada via Manitoulin Transport for sample preparation;
- The Chain of Custody Sheet will be signed upon receipt at the lab in Thunder Bay, confirming that they are not damaged or tampered with. These forms are scanned and emailed to Kennecott.

ALS Minerals is independent to Kennecott and Talon and is one of the world's largest and most diversified testing services providers, with over 120 labs and offices in the Minerals Division. ALS Thunder Bay and Vancouver laboratories are accredited by the Canadian Association for Laboratory Accreditation and Standards Council of Canada (http://www.alsglobal.com/).

11.2 Sample Preparation and Assay Protocols

Sample preparation at ALS Minerals in Thunder Bay includes the following procedure:

- Samples are logged into the ALS Minerals database (LOG-21);
- Samples are weighed upon receipt then dried overnight (DRY-21);
- Entire sample is crushed to 70% -2 mm or better (CRU-31);

- 1000 g is split off using a rotary splitter or a Boyd crusher/rotary splitter combination (SPL-22);
- Entire 1000 g is pulverized to better than 85% passing 75 micron (µm) (PUL-32);
- Assay aliquots are taken from each sample and packaged for shipment to ALS Vancouver where the samples are digested and analyzed;
- Vacuum seal master pulp and all master pulp material is returned to Kennecott and stored at the Tamarack Project site;
- Crushers, splitters and pulverizers are washed with barren material at the start of each batch and as necessary within batches. Between-sample washes (WSH-21 and WSH-22) are used at the request of Kennecott for high-grade sample batches;
- Crushing QC tests are conducted every 20th to 40th sample;
- Pulverizing QC tests are conducted every 20th to 40th sample.

Sample analyses are conducted at the ALS Minerals Vancouver laboratory. The methodology for mineralized material at the Tamarack North Project is reported as follows:

- Ni, Cu, and Co grades are first analyzed by a 4-acid digestion and inductively coupled plasma atomic emission spectroscopy (ICP-AES) and inductively coupled plasma mass spectroscopy (ICP-MS) (ME-MS61). Grades reporting greater than 0.25% Ni and/or 0.1% Cu, using ME-MS61, trigger a sodium peroxide fusion with ICP-AES finish (ICP81);
- Pt, Pd and Au are initially analyzed by a 50 g fire assay with an ICP-MS finish (platinum group metal (PGM)-MS24). Any samples reporting greater than 1 g/t Pt or Pd trigger an over-limit analysis by ICP-AES finish (PGM-ICP27) and any samples reporting greater than 1 g/t Au trigger an over-limit analysis by AAS (Au-AA26);
- Total sulphur is analyzed by Leco Furnace (S-IR08).

The methodology for non-mineralized samples is reported as follows:

- Ni, Cu, and Co grades are first analyzed by a 4-acid digestion and mixed ICP-AES and ICP-MS (ME-MS61). Grades reporting greater than 0.25% Ni and/or 0.1% Cu, using ME-MS61, trigger a sodium peroxide fusion with ICP-AES finish (ICP81);
- Pt, Pd and Au are initially analyzed by a 50 g fire assay with an ICP-MS finish (PGM-MS24).

The methodology for litho-geochemical characterization of samples is reported as follows:

- ALS Minerals Code ME-ICP06 – Whole rock package for 13 oxides plus loss on ignition (ALS Minerals Code OA-GRA05) and total (ALS Minerals TOT-ICP06) – lithium (Li) metaborate or tetraborate fusion/ICP-AES finish;
- ALS Minerals Code ME-MS81 – Resistive trace 30 elements by Li meta-borate fusion and ICP-MS finish.
- ALS Minerals Code ME-4ACD81 – Eight (8) base metals plus Li and Sc by 4-acid digestion with an ICP-AES finish (silver (Ag), cadmium (Cd), Co, Cu, molybdenum (Mo), Ni, Pb, and Zinc (Zn)).
- ALS Minerals Code ME-MS42 – Nine (9) volatile trace elements by aqua regia digest with an ICP-MS finish (arsenic (As), bismuth (Bi), mercury (Hg), indium (In), rhenium (Re), Antimony (Sb), selenium (Se), tellurium (Te), thallium (Tl)).
- ALS Minerals Code ME-IR08 – Total sulphur and total carbon analyzed by combustion furnace.

The methodology for density measurements is reported as follows:

- ALS Minerals Code OA-GRA08 – SG is determined by the weighing a sample in air and in water, and it is reported as a ratio between the density of the sample and the density of water.

11.3 Assay Data Handling

After receiving assay results for each despatch, QA/QC standards, blanks and duplicate data are immediately processed (GOMS acQuire) to confirm that results are consistent with expected ranges and values. The values reported for ALS Minerals internal standards are also monitored. Kennecott has adopted a number of rules of variance that are acceptable versus those of exceedance. An internal QA/QC analysis manual is available for all users of the data. If established quality thresholds are exceeded then the sample is logged as a "Fail" and an investigation is initiated. Re-analysis, sample switch checks, and other means of investigation are acted upon to resolve exceedances. All actions are tracked and logged (See Figure 11-2). Assay data is only considered final within the acQuire system once they have passed all QA/QC checks. Talon only received assay data from Kennecott once the samples were designated as final within the acQuire system. Talon received the data via a secured web based transfer site as a comma-separate values (.csv) file.

LEGEND

	= loaded
	= pending QC
	= pending revision

Tamarack - Lakeview Assay Batch Tracking Sheet

Despatch	Workorder	Date Samples received	Date Assays Finalized	Project	Hole ID	QC Status	QC Final	Date assays loaded to database	Failure Rule	Standard ID	Sample ID for Failed Sample(s)	Elements	Date ALS Chemex Advised of Failure	Date re-runs received	Comments
E40370	VA08019642	18/02/2008	10/03/2008	Tamarack	08L042	Failed	Passed	02/06/2008	1	EA-02	40013265	Au, Pt, Pd			assays imported by Peter T
E40371	VA08028862	12/03/2008	08/04/2008	Tamarack	08L040	Passed		08/07/2008							assays imported by Peter T
E40372	VA08028863	12/03/2008	05/04/2008	Tamarack	08L045	Passed		08/07/2008							assays imported by Peter T
E40373	VA08033273	18/03/2008	17/04/2008	Tamarack	08L042	Passed		08/07/2008							assays imported by Peter T
E40374	VA08015798	25/03/2008	23/04/2008	Lakeview	08L044	Passed		24/07/2008							assays imported by Peter T
E40375	VA08049561	17/04/2008	22/05/2008	Lakeview	07L038	Passed		27/05/2008							assays imported by Peter T
E40376	VA08042717	07/04/2008	21/05/2008	Tamarack	08L041	Failed	Passed	11/06/2008	1	EA-01	40013135, 40013235	Cu, Nu	22/05/2008	06/06/2008	Values for re-runs pass; initial failure due to c
E40377	VA08049299	17/04/2008	22/05/2008	Lakeview	07L039	Passed		23/05/2008							assays imported by Peter T
E40378	VA08049292	17/04/2008	22/05/2008	Tamarack	07L037	Passed		27/05/2008							assays imported by Peter T
E40379	VA08049560	17/04/2008	22/05/2008	Tamarack	04L015	Failed	Passed	11/06/2008	1	EA-01	40014395	Au, Pt, Pd	23/05/2008	03/06/2008	Values for re-runs pass; cause of initial failur
E40380	VA08036667	18/02/2008	21/06/2008	Tamarack	08L042	Failed									check samples(?) - unknown assay method, r
E40381	TB08077084	12/06/2008	04/07/2008	Tamarack	08TK0048	Passed		03/07/2008							assays imported by Peter T
E40382	TB08083297	25/06/2008	15/07/2008	Tamarack	08TK0048 & 49	Failed	Passed	20/08/2008	1	EA-01	40015030	Au	16/07/2008	8/19/08	assays imported by Peter T - sample number
E40383	TB08088647	02/07/2008	23/07/2008	Tamarack	08TK0050	Failed	Passed	03/03/2008					30/07/2008		assays imported by Peter T.

Figure 11-2: Table of Failures and Corrections

11.4 Quality Assurance and Quality Control (QA/QC)

QA/QC programs are intended to monitor the accuracy and precision of the sampling and analysis process in order to quantify the reliability and accuracy of assay data. Typical QA/QC programs consist of a routine insertion of QC materials to measure laboratory performance. QC materials generally consist of CRM including standards and blanks (materials containing no economic minerals) as well as duplicate samples (duplicates).

The Tamarack North Project has shown QA programs consistent with industry standards. Written procedures, acceptable industry software, database organization, and data presentation all contribute to confidence in the current program. QC at the Tamarack North Project has evolved over the life of the project. The initial phase of the project saw duplicates, blanks and standards inserted at a rate of approximately 5% to 6%. With the maturity of the program and confidence in the laboratory the rate of insertion has been reduced to 3.5% to 4%. There is a consistent program of analyzing duplicates of pulps (lab), coarse rejects (lab) and core (field). Analysis of the coarse reject duplicate samples for Ni and Cu show a strong correlation and thus confirm proper sample splitting methodology carried out at the lab (see Figure 11-3 and Figure 11-4). Analysis of the pulp duplicate samples for Ni and Cu also show a strong correlation and thus confirm the lab precision (see Figure 11-5 and Figure 11-6).

The QA/QC standards, blanks and duplicate testing protocols applied by Kennecott are outlined in Section 11.1 above.

It is Golder's opinion that the sample preparation, security and analytical procedures used by Kennecott are consistent with industry standards and are appropriate for the Tamarack North Project. Golder has no material concerns with these processes.

Golder recommends that Kennecott prepare an annual report summarizing the QA/QC analysis of their CRM data and that they incorporate laboratory check assays, from a referee lab, into their protocol as a check against lab bias from their primary lab.

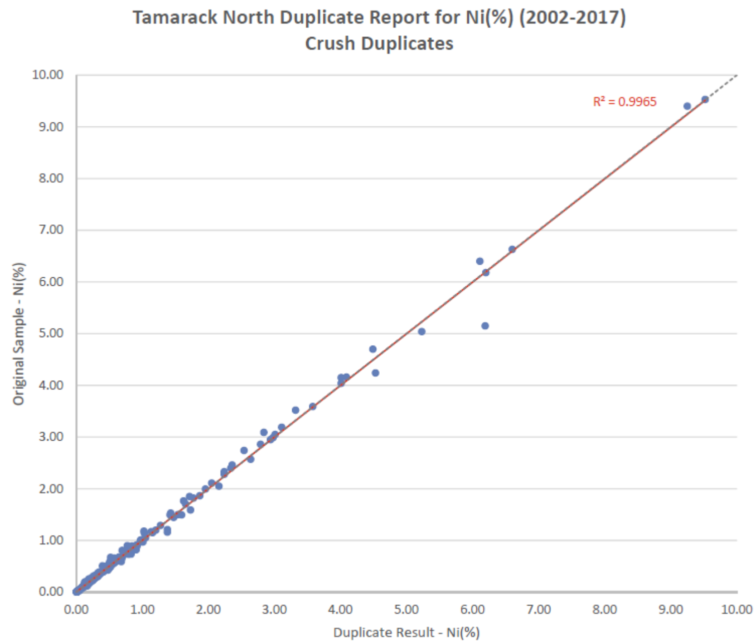


Figure 11-3: Comparison of Original vs Duplicate Coarse Reject Ni (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

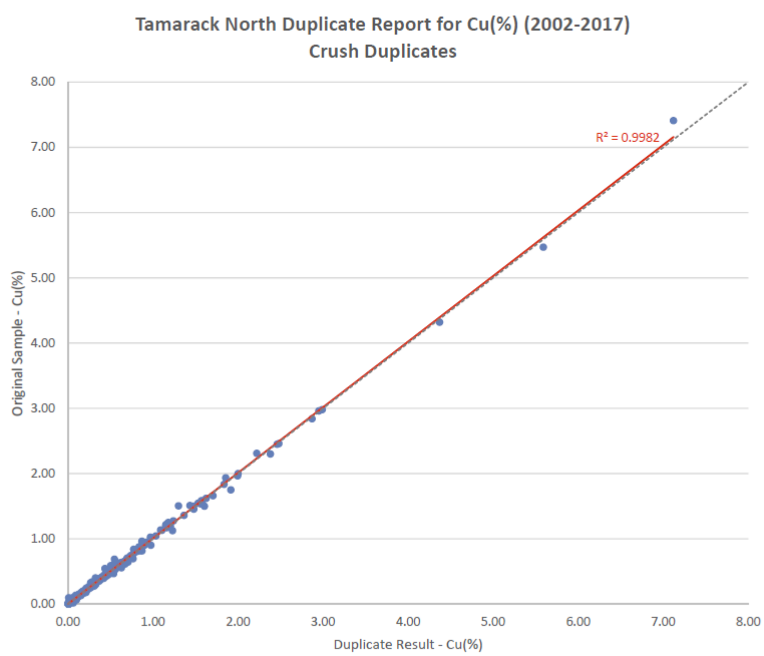


Figure 11-4: Comparison of Original vs Duplicate Coarse Reject Cu (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

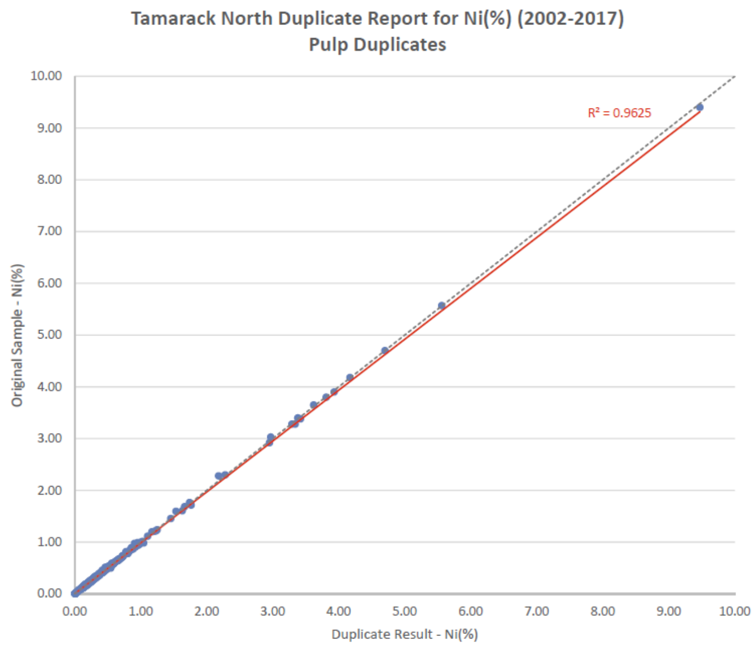


Figure 11-5: Comparison of Original vs Duplicate Pulps Ni (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

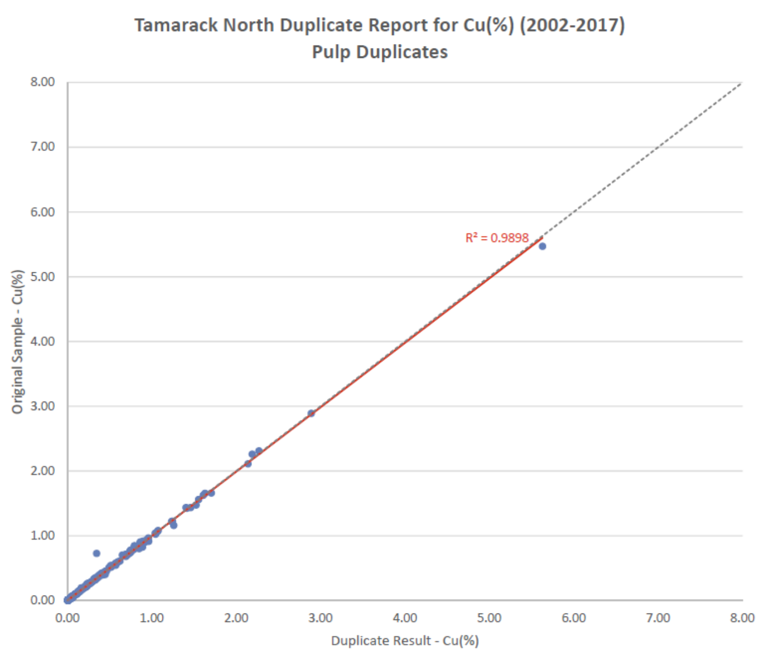


Figure 11-6: Comparison of Original vs Duplicate Pulps Cu (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

12. DATA VERIFICATION

12.1 Golder 2014

Golder completed a number of data verification checks in 2014 and 2017 while completing the Mineral Resource estimate for the Tamarack North Project. The verification work included a check of the drill hole database provided against original assay records (2014 and 2017) and a site visit by the QP (2014) to check drill hole collars, logging procedures, sample of custody and collection of independent samples for metal verification. In addition, Golder has completed a number of verifications of the Mineral Resource estimate which is outlined in Section 14.

12.1.1 Database Verification

Golder compared 2,091 sample assays for %Ni, %Cu, %Co, Pt parts per million (ppm), Pd ppm, Au ppm, from the supplied drill hole database to the original ALS Minerals certificates in the First Independent Technical Report on the Tamarack North Project with an effective date of August 29, 2014 (see Table 12-1). For the updated Mineral Resource estimate in this PEA, Golder reviewed a further 533 samples for %Ni, %Cu, %Co, Pt ppm, Pd ppm, Au ppm, from the supplied drill hole database (for holes drilled since the previous estimate) to the original ALS Minerals certificates. The database encompasses the entire set of drill holes at the Tamarack North Project. Samples found within the resource areas were preferentially chosen (2008 to 2016 drill programs – Tamarack North Project) as they are material to the validity of the Mineral Resource estimate. Assay certificates were available for all samples. A summary of the data validation is listed in Table 12-1.

Table 12-1: Drill Hole Sample Data Validation

Years of Drill Program	# of Holes	# of Samples	# of Assays	# of Errors	Check Year
2008-2013	37	2,091	25,983	0	2014
2014-2016	19	533	3,198	0	2017

Only a small selection of all the drill holes at Tamarack North Project were validated against the original data. A total of 48 unique drill holes (2,624 samples), representing 6.7% of the total available assay data, was reviewed. No errors were identified in any of the validated samples. No validation checks were completed on the remaining samples since most drill holes and samples were not included in the Mineral Resource estimate. It should be noted that certain assay values in ppm were expressed as percentages rounded to three decimal places in the database. Values below the detection limit were set to half of the detection limit instead of a zero value.

12.1.2 Site Visit

A site visit to the Tamarack North Project and Kennecott office, located in the town of Tamarack, Minnesota was carried out by Brian Thomas, P.Geo., QP for this Mineral Resource estimate and Technical Report, on July 16, 2014. No active drilling or core logging was ongoing at the time of the visit. The visit to the Tamarack North Project included:

- An overview tour of the exploration property;
- Inspection and GPS co-ordinate reading of drill collars 08TK0054, 08TK0058, 08TK0079 and 12TK0158 (Table 12-2);
- Visual inspection of physiography and general conditions.

Table 12-2: Validation Check of Drill Collars

Hole Number	Source	Easting	Northing	Elevation
08TK0054	Kennecott	490713	5168726	391
	Golder	490713	5168727	395
08TK0058	Kennecott	490590	5168609	390
	Golder	490588	5168610	391
08TK0079	Kennecott	490589	5168605	390
	Golder	490584	5168607	389
12TK0158	Kennecott	490850	5168418	388
	Golder	490850	5168419	390

All collar co-ordinates were found to closely match the Kennecott co-ordinates, generally within the accuracy of the GPS readings (±3 m).

The site visit to the Kennecott office and core logging facilities in Tamarack, Minnesota, included the following items:

- Review of logging and sampling procedures used on the drill holes;
- Review of core logs against the core available at time of visit;
- Review of Tamarack geological and mineralization characteristics with Kennecott staff;
- Collection of representative duplicate samples for analysis at an independent laboratory;
- Collection and review of all available data required for the Mineral Resource estimate;
- Review of QA/QC protocol; and
- Review of sampling and shipping protocol.

No significant issues were identified during the review of data collection procedures and sample chain of custody. The core logging matched the core well and all processes were found to meet or exceed industry standards.

A site visit was not completed for the updated Mineral Resource in this PEA, as there were no material changes to any of the procedures used by Kennecott for data collection.

12.1.3 **Independent Sampling**

As part of the 2014 sample verification program, nine core samples and three CRM samples were collected and transported back to Sudbury, Ontario, Canada where they were analyzed by Actlabs using sodium peroxide fusion with inductively coupled plasma (ICP) finish for base metals including Ni, Cu, and Co and fire assay with ICP finish for precious metals including Pt, Pd, and Au. Two Kennecott standards and one blank sample were also submitted to Actlabs to confirm their precision and accuracy. SG was also measured on the pulps. The Actlabs laboratory in Sudbury is certified International Organization for Standardization (ISO) 17025.

The objective of the samples collected was to represent the low, medium and high-grade mineralized samples of the three mineralized domains, and to confirm SG. Pictures of samples representing each mineral domain are displayed in Figure 12-1 to Figure 12-3.

Figure 12-1: Example of Core from the 138 Zone

Figure 12-2: Example of Core from the SMSU

Figure 12-3: Example of Core from the MSU

Golder samples 1310101-1310104 were from hole 12TK0138 (138), samples 1310105-1310107 (SMSU) were from hole 08TK0079, while samples 1310108-1310109 were from 12TK0158 (MSU). Sample 1310110 was a typical blank, and samples 1310111-1310112 were medium and high-grade standards. Generally, low to medium grade samples compared favourably as seen in Table 12-3 and Figure 12-4 to Figure 12-6. However, higher grade samples (Figure 12-5) incurred slightly more variation likely due to sample volume variance (Kennecott samples were ½ core while Golder used ¼ core) than due to analytical concerns. All assay results were found to fall within acceptable tolerances of the Kennecott results and no grade bias was evident.

The SG measured from sample pulps (Actlabs) showed some variance to the measurements taken from whole core by ALS Minerals (GRA08). SG measurements from ALS Minerals were only used for the MSU and SMSU domains. Kennecott does collects field SG measurements from select sections of core from all domains including the 138 Zone (see Section 11.1 for a description of the process). These values (10-15 cm) were not used by Golder in the resource model because there was concern regarding how representative they would be with respect to the larger assay sample interval (Golder used a density weighted assay estimation methodology in their model as described in Section 14).

Table 12-3: Sample Validation Check

		Cu (%)		Ni (%)		Co (%)		Au ppm (g/t)		Pt ppm (g/t)		Pd ppm (g/t)		Specific Gravity	
Golder No.	**Kennecott No.**	**Golder**	**Kennecott**	**Golder**	**Kennecott**	**Golder**	**Kennecott**	**Golder**	**Kennecott**	**Golder**	**Kennecott**	**Golder**	**Kennecott**	**Golder**	**Kennecott**
1310101	40064017	1.8	1.71	2.23	2.08	0.045	0.042	0.242	0.427	0.287	0.316	0.251	0.258	2.87	0
1310102	40064027	0.967	0.892	1.03	0.924	0.027	0.025	0.114	0.313	0.202	0.186	0.114	0.117	2.89	0
1310103	40064076	1.75	1.645	1.64	1.67	0.039	0.039	0.215	0.246	0.395	0.4	0.273	0.286	2.78	0
1310104	40064087	0.704	0.671	0.835	0.769	0.025	0.024	0.096	0.108	0.214	0.1945	0.139	0.137	2.78	0
1310105	40031592	1.1	1.525	1.81	2.62	0.044	0.058	0.15	0.227	0.197	0.348	0.312	0.469	2.92	3.29
1310106	40031612	1.64	1.59	4.08	4.15	0.097	0.1	0.182	0.101	0.471	0.543	0.371	0.338	3.28	3.38
1310107	40031616	1.58	1.475	3.4	3.54	0.09	0.096	0.141	0.142	0.371	0.293	0.352	0.339	3.37	3.45
1310108	40067371	1.67	1.595	6.07	5.11	0.125	0.107	0.385	0.249	0.346	0.543	0.61	0.504	3.44	0
1310109	40067377	2.59	1.88	5.47	4.73	0.121	0.102	0.33	0.445	0.497	0.872	0.651	0.483	3.37	0
1310110	blank	0.006	0	0.008	0	0.008	0	< 2	0	< 5	0	< 5	0	2.78	0
1310111	standard	1.35	1.35	3.35	3.34	0.087	0.0087	0.149	0.134	0.386	0.364	0.26	0.272	3.28	0
1310112	standard	4.35	4.52	6.26	6.607	0.162	0.179	0.227	0.265	1.2	1.2	0.794	0.778	4.18	0

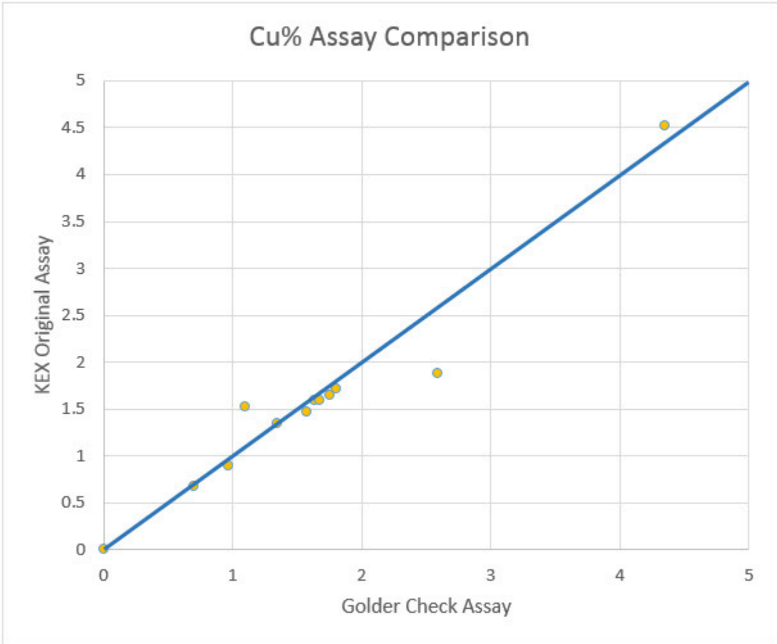


Figure 12-4: Validation Check of Cu Assays

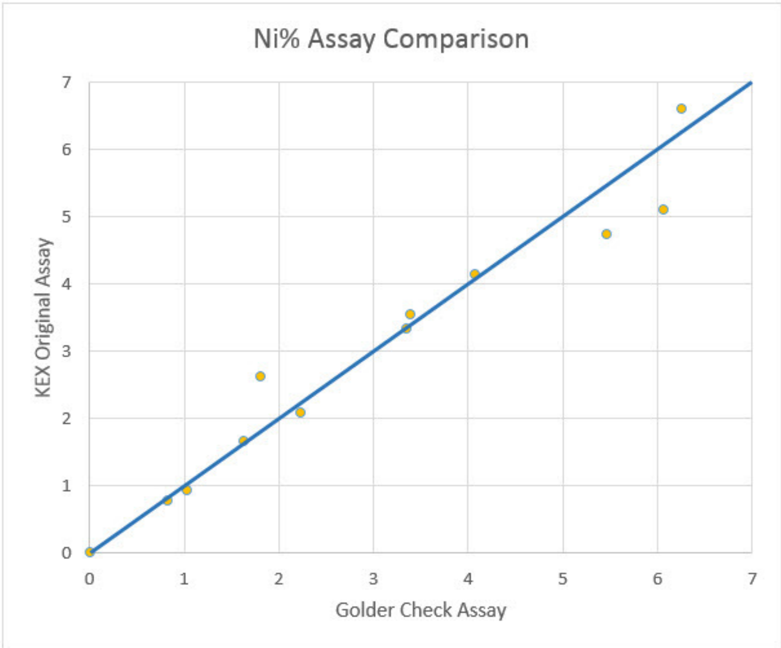


Figure 12-5: Validation Check of Ni Assays

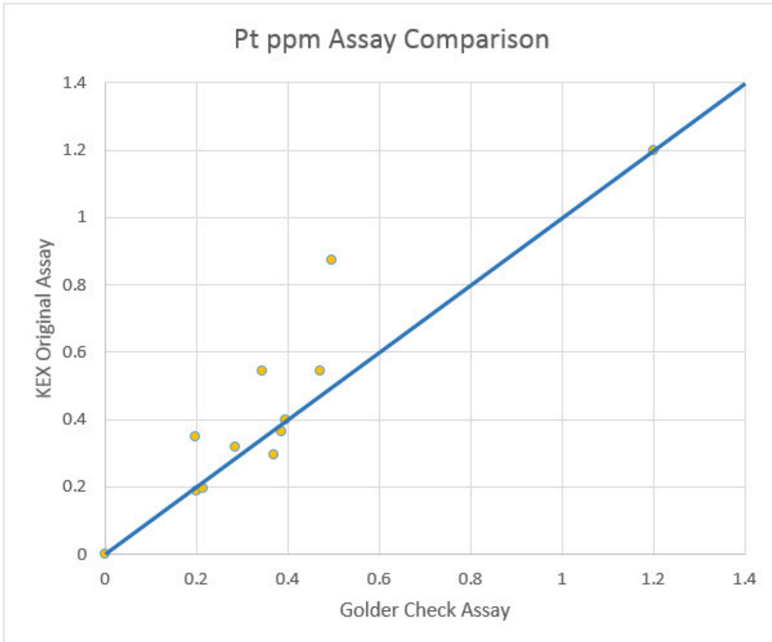


Figure 12-6: Validation Check of Pt Assays

On completion of the data validation, site visit and verification sampling, Golder concluded that the assay data is of suitable quality to support the Mineral Resource estimate. Golder recommends that SG measurements are completed from sample pulps where data is currently only available from field measurements.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Metallurgical Work

Results from metallurgical programs prior to the 2016/2017 test program are summarized in the First Independent Technical Report on the Tamarack North Project with an effective date of August 29, 2014. The QP for the metallurgical section of the report was Manochehr Oliazadeh Khorakchy, P.Eng. and a summary of the metallurgical section of the Technical Report is provided below.

Metallurgical testing of the Tamarack North Project was carried out in two programs: From 2006- 2010, samples consisting of high-grade mineralization from the SMSU hosted in CGO and low-grade CGO mineralization were submitted to SGS Minerals Services for mineralogical and metallurgical testing. The 2012-2013 program focussed only on low-grade mineralization in each of the intrusions.

Head assays from both phases of test work indicated that there were no problematic concentrations of deleterious material, such as As, talc and chlorite.

Mineralogy conducted by Quantitative Evaluation of Materials by Scanning Electron Microscope (QEMSCAN) on two master composites indicated that the dominant Cu sulphide was Cpy, with minor amounts of cubanite present. Pentlandite (Pn) was the dominant Ni sulphide with minor amounts of mackinawite. The dominant sulphide gangue mineral was Po, which needs to be rejected.

Bond ball mill grindability tests produced ball mill work indices (BWi) from 13.0 to 19.0 kWh/t (metric). The work index was found to increase as the sulphide to gangue ratio decreases.

Ni and Cu liberation analysis indicated that Ni and Cu were well liberated for a roughing stage, but that a regrind would likely be required to increase the concentrate grade. Rougher flotation tests were designed to investigate the effect of primary grind on rougher flotation recoveries. The optimum grade recovery relationships for both Ni and Cu were achieved at grinds having a P_{80} between 90 and 129 µm, the recovery of Ni was 89.2% to 90.7% while Cu was 93.9% to 95.5%. Initial rougher concentrate grades in excess of 20% Cu+Ni were readily achieved.

Batch cleaner flotation tests were carried out on all composites to establish the recoveries and grade of a final bulk Cu-Ni concentrate.

An initial investigation into the potential for producing separate, high-grade Cu and Ni concentrates from a bulk concentrate was also investigated and included regrinding of the rougher concentrate; however, no optimization work was commissioned at the time. Specifically, the impact of adding more collector and CMC to minimize metal losses in separate, high-grade Cu and Ni concentrates needed to be tested. The results also suggested that an additional cleaning step would be beneficial to help reject additional non-sulphide gangue. Both these steps were included in the 2016/2017 program.

Some open circuit cleaning tests employed a Cu-Ni separation stage following the bulk cleaner flotation stage. The Ni concentrate graded 21.5% Ni with a Cu:Ni ratio of 0.09. Ni recovery to this concentrate was 78.5%. The best results from the Cu separation tests produced a Cu concentrate grading 32.4% Cu with 0.72% Ni and a 71.4% Cu recovery to the Cu concentrate.

The results for the Cu-Ni separation tests were satisfactory for the Ni concentrate as the target of a Cu:Ni ratio of < 0.2 in the Ni concentrate was met, therefore production of a high-grade Ni concentrate with a Cu:Ni ratio of <0.2 is achievable. Producing a Cu concentrate that meets the target of <0.7% Ni in the Cu concentrate was not met. The best result achieved a Ni grade in the Cu concentrate of 0.72% Ni. The average Ni grade in the Cu concentrate of all the tests on samples from the SMSU was 1.2% Ni when a regrind and one stage of Cu cleaning was used.

An ICP scan carried out on a blend of Cu and Ni concentrates indicated that there were only low concentrations of impurity elements. These numbers are well below the smelter and refiner thresholds.

13.2 The 2016/2017 Test Program

In 2016/2017 a total of seven domain composites were subjected to a metallurgical test program. Samples were selected from:

- The MSU;
- High-grade mineralization from the SMSU hosted in the CGO;
- Low-grade mineralization from the Lower and Upper 138 Zone;
- Low-grade mineralization from the CGO;
- Low-grade mineralization from the Upper CGO;
- Low-grade MMS mineralization and FGO interval above the MMS mineralization in the CGO Bend.

The primary objectives of the 2016/2017 test program were to:

- Obtain a flowsheet and test conditions suitable to treat the full range of MSU, SMSU, and low-grade disseminated mineral domains;
- Define expected recoveries over a wide spectrum of feed grades;
- Understand if there will be any synergies by blending the low-grade domains with high-grade domains.

13.3 Sample Characterization

13.3.1 Chemical Characterization

Representative sub-samples of the seven domains were extracted during sample preparation. The sub-samples were subjected to a chemical analysis to determine the head grades of the composites. Pertinent results of the chemical analysis are presented in Table 13-1.

Table 13-1: Head Analysis of Tamarack Domains

Domain	Assays (%)			Assays (g/t)			
	Cu	Ni	S	Au	Pd	Pt	Ag
MSU	2.80	6.39	25.8	0.10	0.50	0.46	3.4
SMSU	1.59	3.17	13.7	0.17	0.25	0.26	3.5
CGO	0.34	0.45	1.31	0.19	0.34	0.56	2.2
Upper CGO	0.44	0.61	2.38	0.06	0.07	0.09	1.8
Upper 138 Zone	0.37	0.52	1.57	0.14	0.07	0.10	1.9
CGO Bend Zone	0.33	0.50	1.73	0.07	0.06	0.09	1.3
Lower 138 Zone	0.31	0.46	1.30	0.09	0.08	0.15	1.8

A minor element scan identified Fe, Mg, and aluminium (Al) as the most abundant elements in the four composites. No elevated concentrations of deleterious elements were identified in the samples.

The mineral abundance of the seven composites is depicted in Figure 13-1. Cpy, Pn, and Po represent almost 70% of the mass in the MSU composite and this value decreases to slightly over 30% in the SMSU composite. Olivine and pyroxenes were the most abundant non-sulphide gangue minerals in the SMSU and disseminated composites. Serpentine made up between 0.11% in the MSU composite and 12.7% in the Lower 138 Zone composite. The concentrations of talc were low in all seven composites and ranged between 0.14% in the SMSU and 0.91% in the CGO composite.

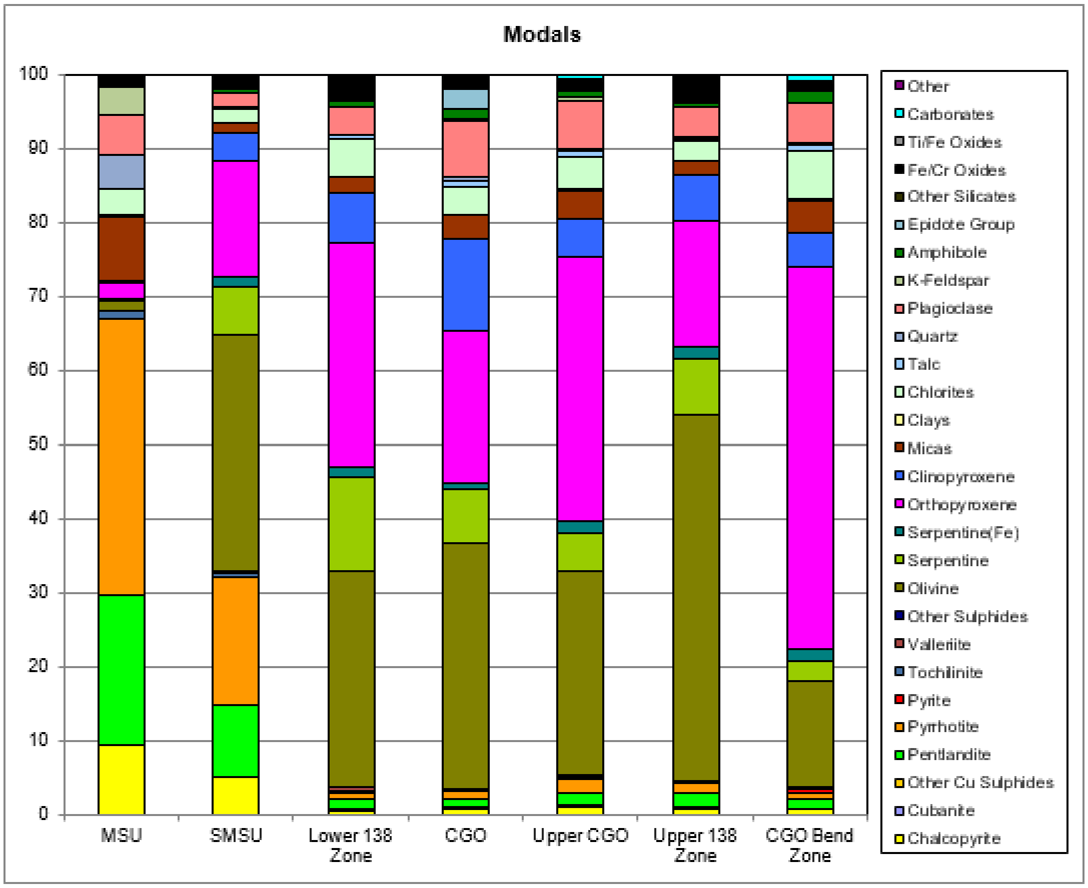
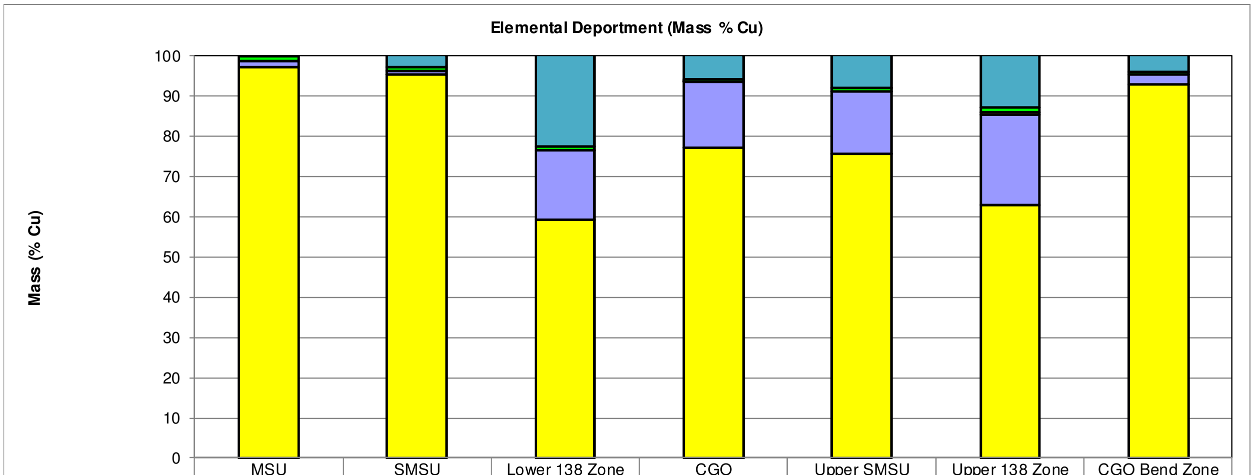


Figure 13-1: Modals of Tamarack Composites

The Cu deportment into the different Cu-bearing minerals is presented in Figure 13-2. In the MSU and SMSU composites almost all Cu units in the sample were associated with Cpy at 97.2% and 95.2%, respectively. Cubanite as the second most abundant Cu-Sulphide mineral contained between 1.4% and 1.0% of the Cu in the MSU and SMSU composites, respectively. Only 1.3% of the Cu reported to Pn and valleriite in the MSU composite, while this number increased to 3.8% in the SMSU composite.

In the five disseminated composites, the Cu deportment into Cpy was between 59.3% and 92.7%. Between 2.6% and 22.5% of the Cu was associated with cubanite and 4.0% to 22.7% with valleriite. Cubanite has a Cu content of only 23.4% compared to 34.6% in Cpy and, therefore, has negative implications on the Cu concentrate grade that can be achieved with this material. The deportment of Cu into valleriite will result in an overall lower recoverable percentage of Cu since the valleriite proves difficult to recover in the flotation process.



	MSU	SMSU	Lower 138 Zone	CGO	Upper SMSU	Upper 138 Zone	CGO Bend Zone
Valleriite	0.1	2.8	22.7	5.9	8.2	13.0	4.0
Pentlandite	1.2	1.0	0.8	0.7	0.8	0.9	0.7
Other Cu Sulphides	0.0	0.0	0.0	0.2	0.0	0.8	0.0
Cubanite	1.4	1.0	17.2	16.3	15.5	22.5	2.6
Chalcopyrite	97.2	95.2	59.3	77.0	75.6	62.7	92.7

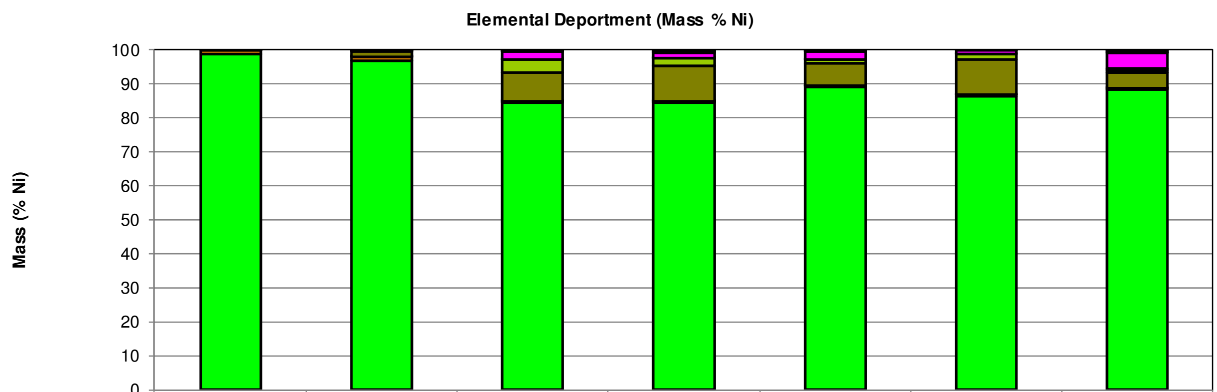
Figure 13-2: Elemental Deportment of Cu

Electron microprobe analysis was conducted on the seven composites to determine the chemical composition of specific minerals and to quantify the deportment of Ni into sulphide and non- sulphide gangue minerals. The concentrations of pertinent elements in Cpy, Pn, and Po are presented in Table 13-2. Further the elemental deportment of Ni as determined by microprobe and QEMSCAN analyses is presented in Figure 13-3. While 98.6% and 96.9% of the Ni was associated with Pn in the MSU and SMSU composites, respectively, the values decreased to as low as 84.3% in the disseminated composites. In those composites, up to 10.4% of the Ni units were associated with olivine.

Table 13-2: Concentrations of Pertinent Elements in Sulphide Minerals

Element	MSU	SMSU	Lower 138 Zone	CGO	Upper CGO	Upper 138 Zone	CGO Bend Zone
%Cu in Cpy	32.8	33.7	28.2	29.2	29.2	32.4	33.5
%Ni in Po	0.26	0.25	0.29	0.10	0.14	0.10	0.43
%Ni in Pn	33.9	34.8	32.3	31.3	31.8	25.9	32.9
%S in Cpy	34.7	34.9	35.0	34.5	34.8	34.5	34.7
%S in Po	39.2	39.2	38.5	39.1	39.0	38.7	38.6
%S in Pn	33.4	33.2	33.2	33.0	33.4	29.5	35.5

Note: Cpy = chalcopyrite, Pn = pentlandite, Po = pyrrhotite



	MSU	SMSU	Lower 138 Zone	CGO	Upper SMSU	Upper 138 Zone	CGO Bend Zone
Amphibole	0.00	0.01	0.07	0.15	0.07	0.04	0.19
Amphibole	0.00	0.01	0.07	0.15	0.07	0.04	0.19
Micas	0.11	0.03	0.36	0.57	0.49	0.19	0.81
Clinopyroxene	0.00	0.00	0.02	0.05	0.02	0.02	0.03
Orthopyroxene	0.01	0.19	2.35	1.71	2.20	0.97	4.57
Serpentine(Fe)	0.00	0.02	0.17	0.12	0.14	0.14	0.23
Serpentine	0.00	0.30	3.70	2.28	1.20	1.59	0.86
Olivine	0.04	1.44	8.49	10.44	6.38	10.43	4.74
Pyrite	0.00	0.00	0.00	0.00	0.01	0.00	0.02
Pyrrhotite	1.21	1.13	0.38	0.40	0.58	0.40	0.37
Pentlandite	98.62	96.89	84.47	84.27	88.91	86.22	88.18

Figure 13-3: Elemental Deportment of Ni

At a primary grind size of P_{80} ~ 100 µm free and liberated Cu-sulphides accounted for 85.8% in the MSU composite and 78.3% in the SMSU composite. This value decreased to 51.0% to 72.7% in the five disseminated composites.

Free and liberated Pn accounted for 87.2% in the MSU composite and 83.9% in the SMSU composite. Again, the degree of liberation was reduced in the disseminated composites with values of 58.1% to 71.0%.

13.4 Comminution Tests

Bond BWi tests were carried out on the seven composites, to determine energy requirements for ball milling. The tests were performed at a screen size of 150 µm (100 mesh), which is representative of a mill discharge product of approximately P_{80} = 75-100 µm.

The results of the BWi tests are presented in Table 13-3 and are further depicted in Figure 13-4. The BWi values ranged from 11.3 kWh/t for the MSU composite to 21.1 kWh/t for the CGO composite. While the MSU composite is considered soft, all disseminated composites except for the Upper 138 Zone composite were very hard. Less than 10% of the 6,100 samples tested at SGS Minerals produced BWi values higher than the three hardest disseminated composites as evidenced in the histogram that displays the frequency of test results for various hardness values.

No other crushing or grinding tests were completed as part of the current or past metallurgical test programs. These tests will be included in the next phase of testing as the results are required for proper sizing of the crushing and grinding circuit.

Table 13-3: Bond Ball Mill Grindability Test Results

Composite	BWi (kWh/t)
MSU	11.3
SMSU	15.1
Lower 138 Zone	21.0
CGO	21.1
Upper CGO	20.2
Upper 138 Zone	15.0
CGO Bend Zone	18.7

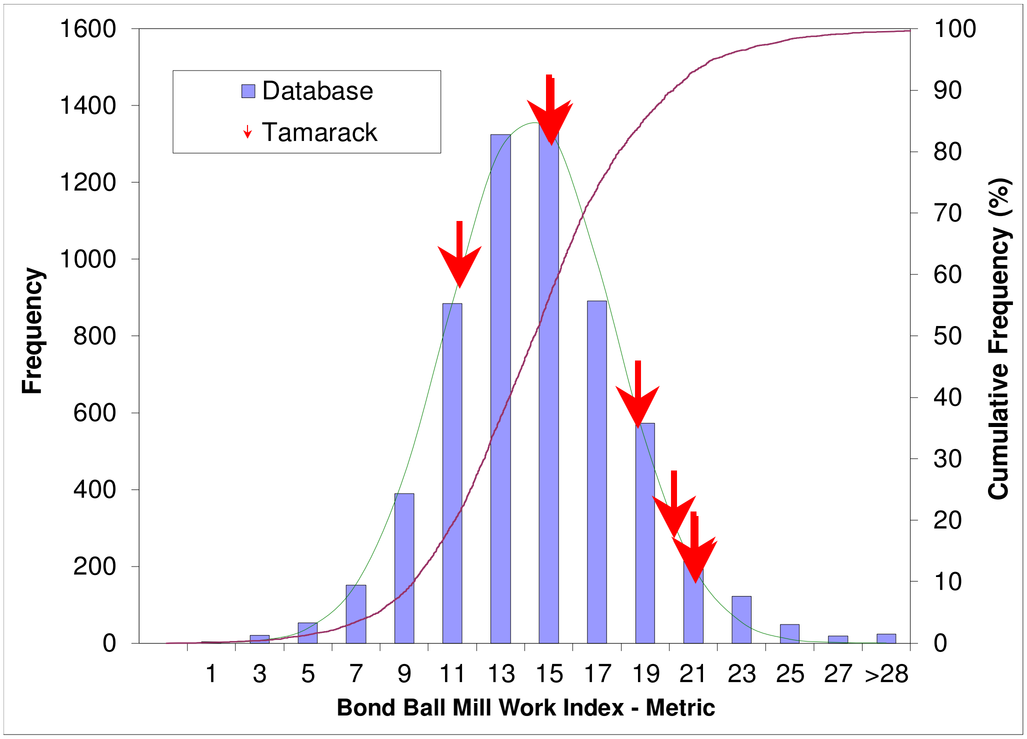


Figure 13-4: Bond Ball Mill Grindability Test Results Plot and SGS Database Histogram

13.5 **Flotation Test Program**

A total of 77 open circuit tests and seven LCTs were carried out in the latest metallurgical program to obtain a flowsheet and test conditions suitable to treat the MSU, SMSU, and disseminated mineral domains. The results of these tests are presented in the following sections.

13.5.1 **Flowsheet Development Tests**

A total of 67 rougher kinetic and open circuit cleaner tests were completed to develop the flowsheet starting with the rougher/scavenger stage through the various cleaning stages and finally Cu/Ni separation.

The first series of rougher kinetics tests evaluated the impact of primary grind sizes of P_{80}=70 µm and P_{80}=100 µm on the metallurgical performance of the seven composites. A comparison of the metallurgical results at the two primary grind sizes did not reveal a statistically significant difference between P_{80} ~ 70 µm and P_{80} ~ 100 µm. However, since the Cpy and Pn mineral grains were only moderately liberated in the disseminated composites, a decision was made to proceed with a primary grind size of P_{80}=70 µm.

Additional rougher kinetics tests were carried out on the Upper 138 Zone composite to improve rougher and scavenger flotation performance. Of the four gangue depressants/ dispersants tested (sodium silicate, guar gum, CMC, and sodium hexametaphosphate), CMC produced slightly better results. The addition of the sulphide activator copper sulphate ($CuSO_4$) failed to improve bulk scavenger flotation performance.

A total of 20 cleaner flotation tests were carried out on MSU, SMSU, Lower 138 Zone, CGO, and Upper 138 Zone composites to develop suitable conditions for the bulk cleaner and scavenger cleaner flotation circuits. Process variables that were investigated included: regrind, re-cleaning, pH modification, and flotation times. The challenge was to address the significantly different flotation performance of the MSU and SMSU composites compared to disseminated composites. While reagent dosages for the MSU and SMSU had to be minimized in the bulk rougher and cleaning stages to prevent the activation of Po, reagent robbing behaviour was observed for the disseminated composites, thus requiring higher reagent dosages.

The remaining 27 cleaner flotation tests investigated the full flowsheet including Cu/Ni separation. Process variables that were evaluated in the Cu/Ni separation circuit included: regrind (no regrind, coarse regrind and fine regrind), flotation times, and reagent dosages.

The Cu and Ni metals are recovered in the bulk rougher and bulk scavenger flotation stages. Flotation conditions are selective in these two flotation stages to recover the valuable minerals but minimize the recovery of iron-sulphides. The bulk scavenger tailings are then treated in a stand-alone desulphurization stage to separate the acid generating sulphides from the NAG non-sulphide gangue minerals using a comprehensive sulphide flotation strategy. Suitable conditions for a desulphurization stage were established to split the bulk scavenger tailings stream into a low-mass, HS tailings stream and a high-mass, LS tailings stream. A full optimization of the desulphurization process will be carried out during the pre-feasibility metallurgical test program (refer Section 26.4).

The HS tailings will be disposed of underground in form of a cemented paste backfill (refer Section 16.2) and the LS NAG tailings will be thickened and filtered for co-deposition with waste rock in a CFTF – refer Section 18.6.

The process development tests culminated in a given flowsheet and test conditions that were then validated in locked cycle flotation tests.

13.5.2 Locked Cycle Tests (LCTs)

At the end of the flowsheet development program, each of the seven composites was subjected to locked cycle testing. The LCT simulates closed circuit operation of a flowsheet by recycling the intermediate tailings into the following cycle to better simulate plant operating conditions. The flowsheet employed in the LCTs is presented in Figure 13-5. Note that the desulphurization stage was not included in the LCTs.

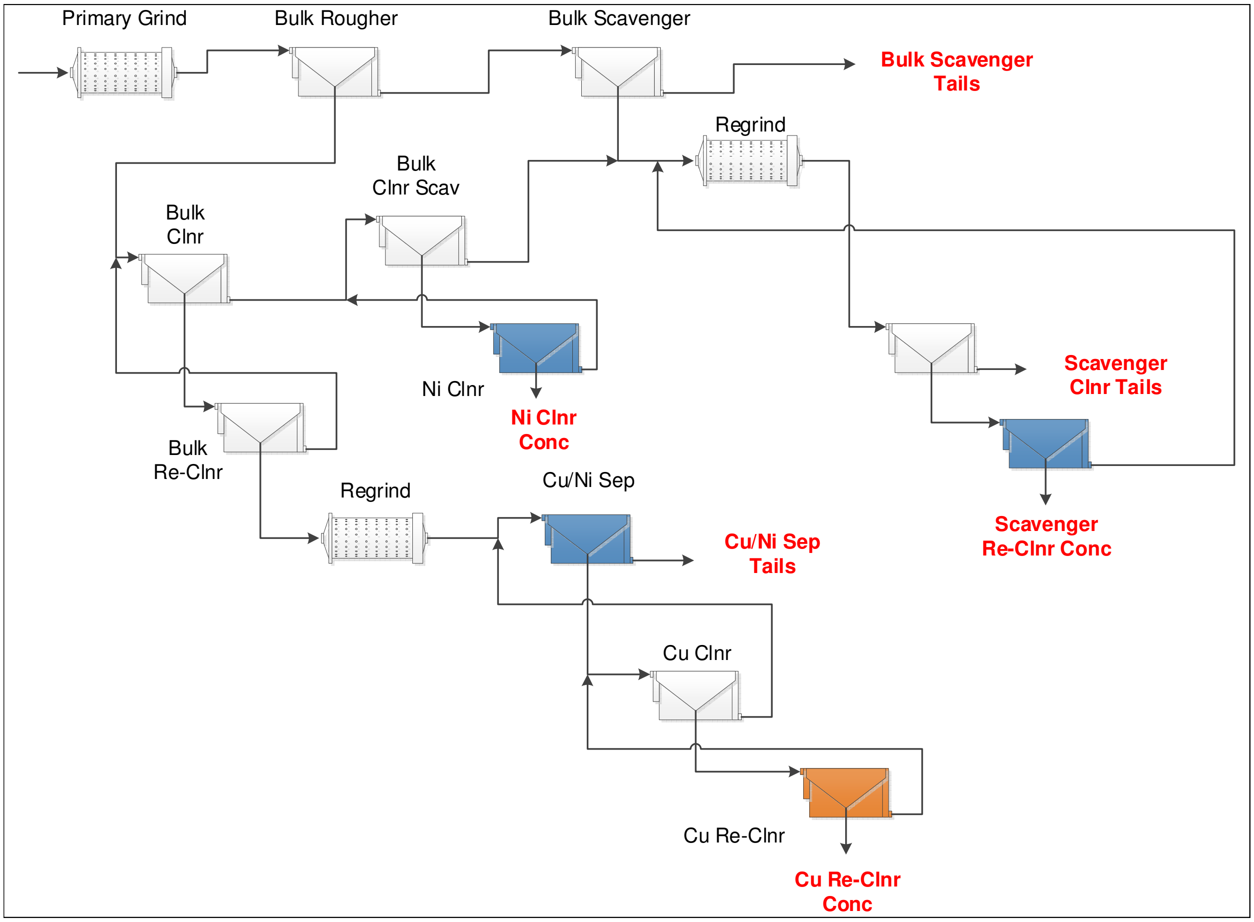


Figure 13-5: Locked Cycle Test Flowsheet

The MSU (LCT-1) and SMSU (LCT-2) samples are representative of the mine plan in this PEA (refer Section 16). The remainder of the samples are below the cut-off grade (0.83% NiEq – refer Section 14.9) and consequently not representative of either the mine model or the resource model in this PEA.

A summary of the mass balances for the seven LCTs is shown in Table 13-4. As expected, the MSU (LCT-1) and SMSU (LCT-2) produced good Ni and Cu concentrates with high metal recoveries. The remaining five composites proved more challenging, which was likely due to a combination of the low head grades in the samples and poor flotation selectivity in the final cleaning stages due to very low mass recoveries. Except for the CGO Bend Zone composite, all disseminated composites failed to produce a Cu concentrate grading at least 25% Cu. In addition to this, Ni recoveries into the Cu concentrates were high, resulting in Ni grades in the Cu concentrate of 1.30% to 6.72% Ni.

The highest Ni recoveries were achieved by including the scavenger recleaner concentrate in the Ni concentrate. However, since this product yields lower Ni concentrations, the resulting combined Ni concentrate graded lower. In the case of the MSU and SMSU composites, the combined Ni concentrates produced acceptable grades of 17.1% and 14.1% Ni respectively. The Ni concentrates of the disseminated composites yielded low-grades between 5.88% and 9.59% Ni. Excluding the scavenger recleaner concentrate raised Ni concentrate grades of the disseminated composites to between 7.7% and 13.5% Ni, albeit at up to 8.2% lower Ni recoveries.

To further illustrate the differences of the various domains with regards to metallurgical performance, the grade-recovery points for the Cu concentrates are depicted in Figure 13-6. The Ni concentrate data including and excluding the scavenger recleaner concentrate are presented in Figure 13-7 and Figure 13-8, respectively.

Table 13-4: Summary of Locked Cycle Tests – LCT-1 to LCT-7

Test & Comp	Sample ID	Weight %	Assay (%)			Distribution (%)		
			Cu %	Ni %	S %	Cu %	Ni %	S %
LCT-1 #75 MSU	Cu Re-Clnr Conc	8.0	31.6	1.53	35.4	91.4	1.9	10.9
	Cu/Ni Sep Tails	17.3	0.44	25.2	34.2	2.8	69.3	23.0
	Ni Clnr Conc	8.4	0.75	11.4	37.1	2.3	15.3	12.2
	Scav Reclnr Con	8.2	0.52	5.59	37.2	1.5	7.3	11.9
	Scav Clnr Tails	17.0	0.10	1.02	35.1	0.6	2.8	23.2
	Bulk Scav Tails	41.0	0.09	0.52	11.9	1.4	3.4	18.9
	Head (calculated)	100.0	2.75	6.31	25.8	100.0	100.0	100.0
	Head (direct)		2.80	6.39	25.8			
	Cu Concentrate	8.0	31.6	1.53	35.4	91.4	1.9	10.9
	Ni Concentrate	34.0	0.54	17.1	35.7	6.6	91.9	47.0
	Total Recovery					98.0	93.9	57.9
LCT-2 #76 SMSU	Cu Re-Clnr Conc	4.3	29.3	0.95	32.4	84.0	1.3	10.4
	Cu/Ni Sep Tails	10.0	1.07	21.5	30.7	7.1	69.1	22.6
	Ni Clnr Conc	3.8	1.07	11.1	31.9	2.7	13.4	8.8
	Scav Reclnr Con	5.6	0.53	2.99	30.2	2.0	5.4	12.5
	Scav Clnr Tails	10.6	0.29	1.31	26.7	2.1	4.5	20.8
	Bulk Scav Tails	65.7	0.05	0.30	5.13	2.2	6.3	24.9
	Head (calculated)	100.0	1.51	3.11	13.6	100.0	100.0	100.0
	Head (direct)		1.59	3.17	13.7			
	Cu Concentrate	4.3	29.3	0.95	32.4	84.0	1.3	10.4
	Ni Concentrate	19.4	0.91	14.1	30.7	11.7	87.9	44.0
	Total Recovery					95.7	89.3	54.3

Table 13-4: Summary of Locked Cycle Tests – LCT-1 to LCT-7 (continued)

Test & Comp	Sample ID	Weight %	Assay (%) Cu %	Ni %	S %	Distribution (%) Cu %	Ni %	S %
LCT-3 #77 Lower 138 Zone	Cu Re-Clnr Conc	0.8	21.2	4.22	29.7	52.8	7.1	18.1
	Cu/Ni Sep Tails	1.4	2.33	13.9	23.3	10.2	40.8	24.8
	Ni Clnr Conc	0.3	2.34	8.3	18.7	2.3	5.4	4.4
	Scav Reclnr Con	1.5	0.73	2.59	12.8	3.4	8.2	14.6
	Scav Clnr Tails	4.7	0.25	0.60	3.44	3.7	6.0	12.4
	Bulk Scav Tails	91.4	0.10	0.17	0.36	27.7	32.5	25.7
	Head (calculated)	100.0	0.32	0.47	1.29	100.0	100.0	100.0
	Head (direct)		0.31	0.46	1.30			
	Cu Concentrate	0.8	21.2	4.22	29.7	52.8	7.1	18.1
	Ni Concentrate	3.2	1.58	8.07	17.9	15.9	54.4	43.8
	Ni Conc without Scav ReClnr Conc	1.7	2.33	12.9	22.4	12.4	46.3	29.2
	Total Recovery					68.7	61.5	61.9
LCT-4 #78 CGO	Cu Re-Clnr Conc	1.0	22.4	4.64	30.4	66.8	10.1	22.7
	Cu/Ni Sep Tails	1.8	2.96	13.0	25.5	15.9	50.9	34.4
	Ni Clnr Conc	0.4	2.42	6.20	17.7	2.6	4.9	4.8
	Scav Reclnr Con	1.2	0.56	1.47	13.4	2.1	3.9	12.3
	Scav Clnr Tails	2.8	0.18	0.50	4.08	1.5	3.1	8.6
	Bulk Scav Tails	92.7	0.04	0.13	0.25	11.1	27.1	17.1
	Head (calculated)	100.0	0.34	0.46	1.34	100.0	100.0	100.0
	Head (direct)		0.34	0.45	1.31			
	Cu Concentrate	1.0	22.4	4.64	30.4	66.8	10.1	22.7
	Ni Concentrate	3.4	2.03	8.10	20.3	20.6	59.8	51.5
	Ni Conc without Scav ReClnr Conc	2.2	2.87	11.9	24.2	18.5	55.9	39.2
	Total Recovery					87.4	69.9	74.3
LCT-5 #79 Upper SMSU	Cu Re-Clnr Conc	1.4	23.8	2.91	31.5	76.1	6.9	19.9
	Cu/Ni Sep Tails	2.1	1.33	14.6	28.9	6.5	53.1	27.9
	Ni Clnr Conc	0.4	1.67	7.99	23.0	1.6	5.7	4.4
	Scav Reclnr Con	1.4	0.45	2.50	22.7	1.5	6.0	14.6
	Scav Clnr Tails	5.3	0.14	0.49	5.49	1.7	4.5	13.4
	Bulk Scav Tails	89.4	0.06	0.15	0.48	12.7	23.7	19.8
	Head (calculated)	100.0	0.43	0.58	2.17	100.0	100.0	100.0
	Head (direct)		0.44	0.61	2.38			
	Cu Concentrate	1.4	23.8	2.91	31.5	76.1	6.9	19.9
	Ni Concentrate	3.9	1.05	9.59	26.1	9.6	64.8	46.9
	Ni Conc without Scav ReClnr Conc	2.5	1.39	13.5	27.9	8.1	58.8	32.3
	Total Recovery					85.6	71.8	66.8
LCT-6 #80 Upper 138 Zone	Cu Re-Clnr Conc	1.3	14.5	6.72	28.1	51.7	16.9	23.8
	Cu/Ni Sep Tails	2.0	2.10	9.63	22.8	11.0	35.5	28.3
	Ni Clnr Conc	1.0	1.25	3.62	14.1	3.2	6.5	8.5
	Scav Reclnr Con	1.3	0.79	1.81	11.5	2.7	4.3	9.3
	Scav Clnr Tails	3.7	0.49	0.88	4.52	4.9	6.1	10.7
	Bulk Scav Tails	90.8	0.11	0.18	0.34	26.6	30.7	19.4
	Head (calculated)	100.0	0.38	0.53	1.58	100.0	100.0	100.0
	Head (direct)		0.37	0.52	1.57			
	Cu Concentrate	1.3	14.5	6.72	28.1	51.7	16.9	23.8
	Ni Concentrate	4.2	1.51	5.88	17.4	16.8	46.3	46.1
	Ni Conc without Scav ReClnr Conc	2.9	1.82	7.66	19.9	14.1	41.9	36.8
	Total Recovery					68.5	63.1	69.9
LCT-7 #81 CGO Bend Zone	Cu Re-Clnr Conc	1.0	26.8	1.30	30.1	82.0	2.7	16.4
	Cu/Ni Sep Tails	2.5	0.79	10.3	33.0	6.0	52.2	44.6
	Ni Clnr Conc	0.3	2.52	8.19	17.0	2.2	4.9	2.7
	Scav Reclnr Con	0.9	1.10	4.17	17.0	3.0	7.5	8.1
	Scav Clnr Tails	2.9	0.08	0.52	2.05	0.7	3.1	3.2
	Bulk Scav Tails	92.5	0.02	0.16	0.49	6.1	29.6	24.9
	Head (calculated)	100.0	0.33	0.49	1.83	100.0	100.0	100.0
	Head (direct)		0.33	0.50	1.73			
	Cu Concentrate	1.0	26.8	1.30	30.1	82.0	2.7	16.4
	Ni Concentrate	3.6	1.00	8.62	27.9	11.2	64.7	55.5
	Ni Conc without Scav ReClnr Conc	2.8	0.97	10.0	31.3	8.2	57.1	47.3
	Total Recovery					93.2	67.3	71.9

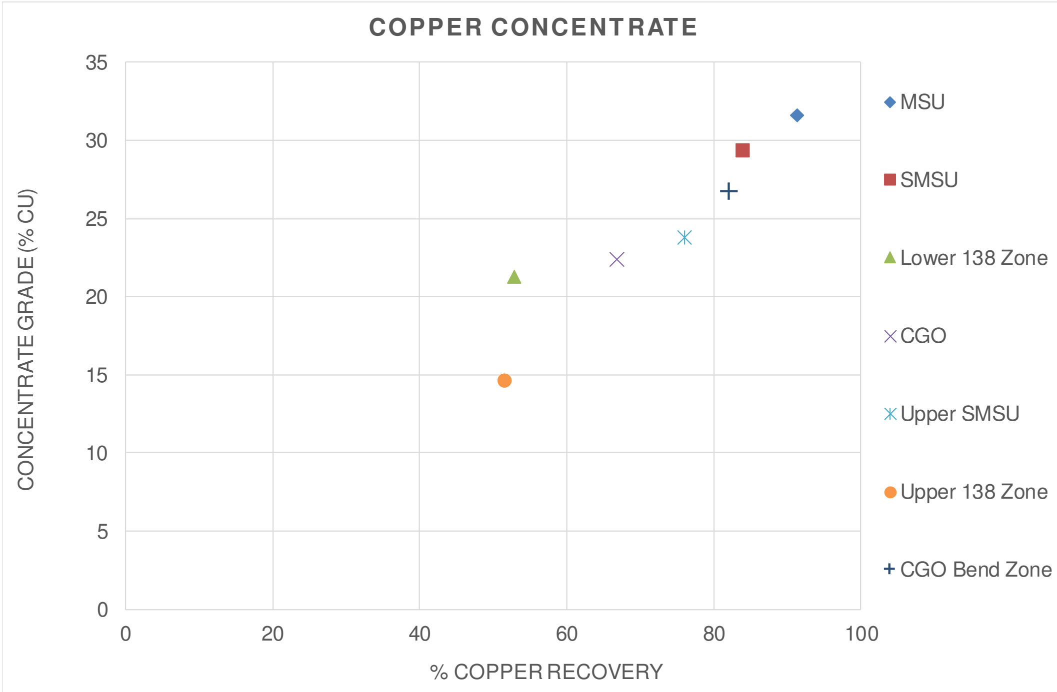


Figure 13-6: Cu Concentrate Grade-Recovery Points (LCT-1 to LCT-7)

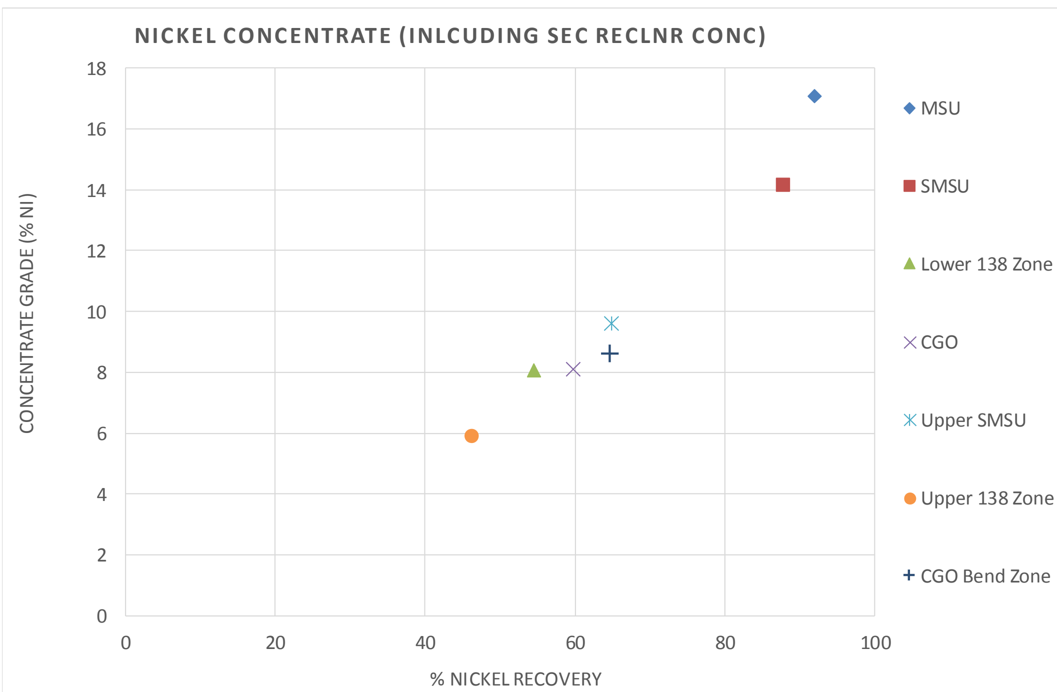


Figure 13-7: Ni Concentrate Grade-Recovery Points (LCT-1 to LCT-7) – Including Scavenger Recleaner Concentrate

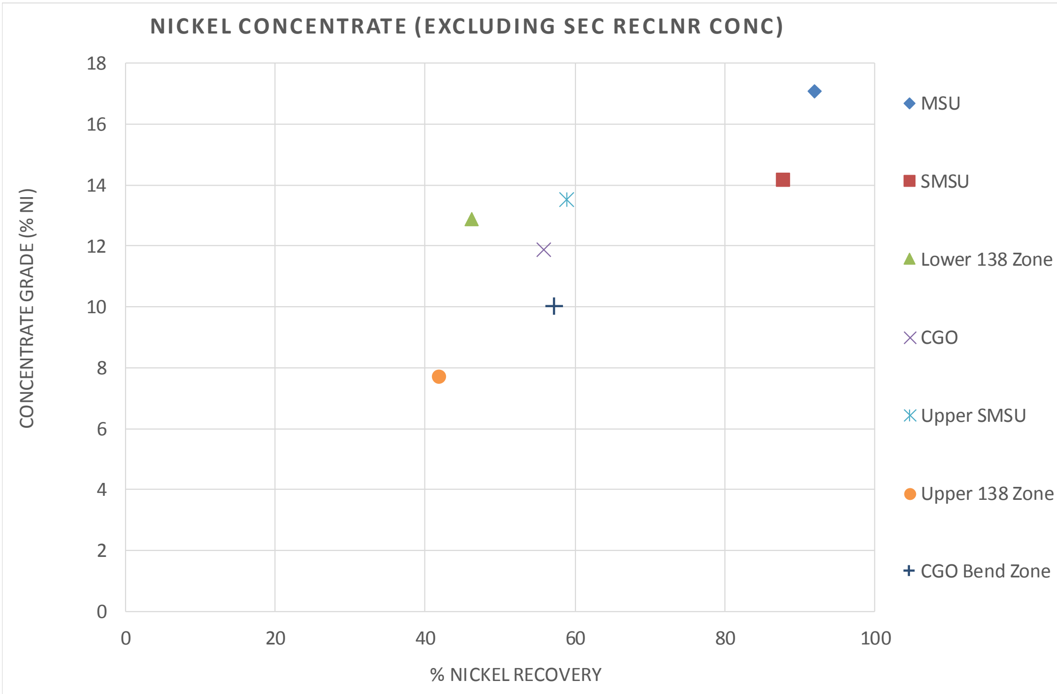


Figure 13-8: Ni Concentrate Grade-Recovery Points (LCT-1 to LCT-7) – Excluding Scavenger Recleaner Concentrate

13.5.3 Concentrate Characterization

The final Cu and Ni concentrates form the LCTs were submitted for chemical analysis to identify potential credit and penalty elements. A summary of pertinent elements is presented in Table 13-5.

Mg is an important deleterious element in Ni concentrates as it will lead to smelter penalties above a typical limit of 5.0% MgO. The MSU and SMSU samples tested were both below this threshold at 0.22% and 3.20% MgO, respectively, as shown in Table 13-5.

The Fe:MgO ratio is also an important to consider and should ideally be above 4.5. The MSU and SMSU samples tested both met this requirement at 212:1 and 13.4:1, respectively.

Although the proposed process conditions include depressants for Mg minerals in the roughing and cleaning stages, carry over of Mg minerals into the Ni concentrate is still significant for the disseminated domains. However, as stated previously, these disseminated domains are below both the resource model and mine model cut-off grades in this PEA.

Further attention should be given to Co recoveries and grades in the Ni concentrate as these are at payable by-product concentrations.

Credits for Au, Pt, or Pd would likely be realized in all composites except for the SMSU, while Pd will only be at payable levels in the CGO mineralization. Au is at payable levels for the CGO and Upper CGO mineralization. Ag concentration in Ni concentrates were below the detection limit of 10 g/t.

Table 13-5: Ni Concentrate – Credit and Penalty Elements – Assays > minimum payable level indicated in bold font

Composite	Assays (%)		Assays (g/t)		
	Co	MgO	Au	Pt	Pd
MSU	0.35	0.22	0.14	**1.34**	1.19
SMSU	0.35	3.20	0.12	0.61	0.58
Lower 138 Zone	0.19	13.2	0.49	**3.21**	1.46
CGO	0.16	9.33	**1.49**	**12.9**	**5.41**
Upper CGO	0.24	7.30	**4.35**	**1.47**	0.96
Upper 138 Zone	0.13	14.6	0.42	**1.15**	0.76
CGO Bend Zone	0.25	7.90	0.34	**1.43**	0.89

Credit elements in the Cu concentrate are presented in Table 13-6. Au is above typical payable levels for all composites while Pt is likely payable for all composites except the SMSU. Pd is payable for the CGO.

Table 13-6: Cu Concentrate – Credit Elements –
Assays > minimum payable level indicated in bold font

Composite	Assays (g/t)			
	Au	Pt	Pd	Ag
MSU	**2.24**	**1.42**	0.25	< 10
SMSU	**2.74**	0.79	0.18	< 10
Lower 138 Zone	**6.47**	**2.56**	1.74	67
CGO	**12.3**	**6.81**	**9.38**	76
Upper CGO	**5.07**	**1.56**	0.79	51
Upper 138 Zone	**3.82**	**2.36**	0.96	57
CGO Bend Zone	**3.68**	**1.37**	0.90	< 10

13.5.4 Blend Tests

The SMSU, MSU, and disseminated domains could be blended to provide steady mill feed grade to ensure a stable operation. While it is quite common that the metallurgical performance of the blend is the sum of the performance of the individual domains, it is not applicable for all deposits.

To quantify the impact of blending on the metallurgical response of the Tamarack North Project mineralization, the MSU and SMSU composites were blended in a ratio of 1:1 (1 kilogram (kg) of MSU or SMSU with 1 kg of disseminated material) with the five disseminated composites to form a total of 10 composite blends. These blends were then subjected to batch cleaner tests using the optimized flowsheet and conditions.

The Cu/Ni separation performance of the low-grade composites improved significantly when blended with the MSU and SMSU composites. The results of the Cu/Ni separation response of the low-grade, MSU, SMSU, and blend composites are presented in Figure 13-9. The Ni concentration in the Cu concentrate ranged from 1.5% to over 3% for the low-grade composites. Once the low-grade composite was blended with MSU or SMSU composite, the Cu/Ni separation performance was in line with data obtained for the MSU and SMSU material. It is postulated that the inferior Cu/Ni separation response of the low-grade composites was the result of insufficient Cu units in the Cu/Ni separation stages to crowd out Pn. These results suggest that the high Ni grades observed for the low-grade composites in the LCTs may have been the result of a limitation with the flotation equipment rather than underlying metallurgical challenges.

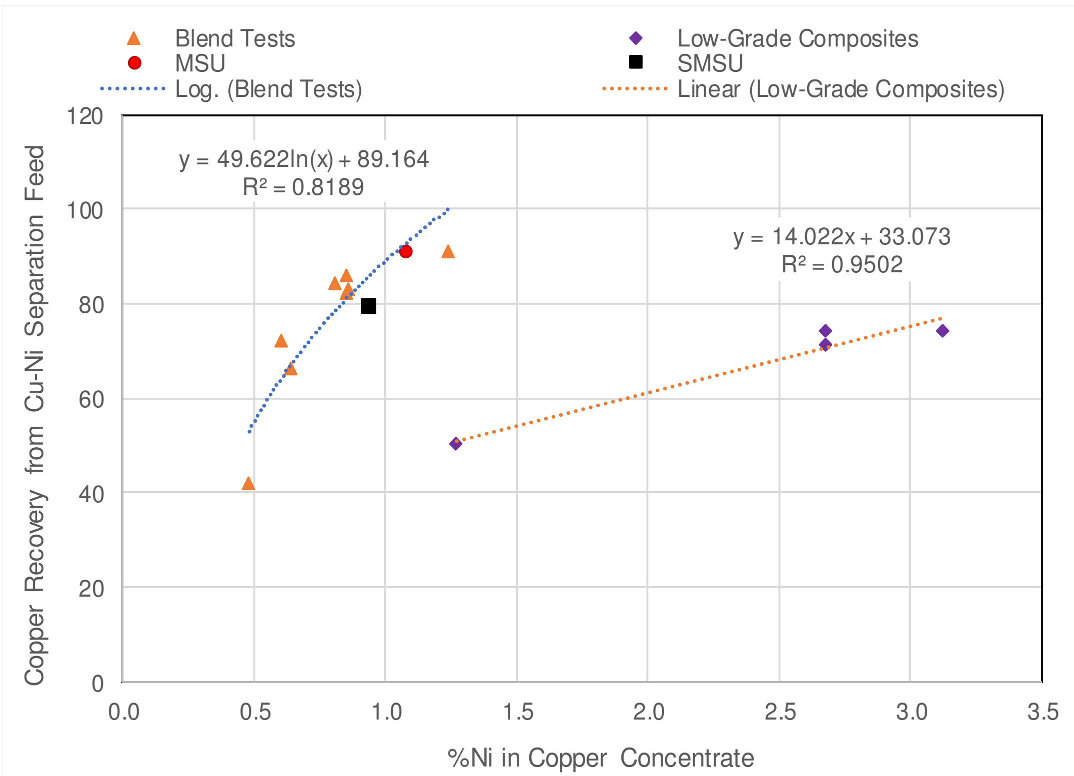


Figure 13-9: Impact of Blending Disseminated Composites with MSU and SMSU Composites on Cu/Ni Separation

13.6 Analysis and Recommendations

The primary objective of the 2016/2017 metallurgical test program at SGS Lakefield was to develop a flowsheet and test conditions suitable to treat all mineral domains encountered in the Tamarack North Project, with a focus on low-grade, disseminated sulphides.

The concentrate grades produced from the disseminated domains (all with feed grades below the cut-off grade applied in both the resource model and mine plan) were too low to be considered marketable if not blended with concentrates obtained from the MSU and

SMSU domains. Further, the Ni and Cu recoveries for the disseminated composites were significantly lower compared to the MSU and SMSU mineralization.

The Cu/Ni separation performance for the CGO, Lower 138 Zone, Upper 138 Zone composite remained poor until the end of the test program with high Ni grades in the Cu concentrate. However, their feed grades were extremely low at 0.3% to 0.4% Cu, and therefore the weight contribution to the Cu concentrate will be relatively low if blending ratios are not high in favour of these poorly performing composites.

Preliminary testing suggested that blending of MSU/SMSU and disseminated material responded better in the Cu/Ni separation circuit than the sum of the individual responses. Only a single 1:1 blending ratio (1 MSU or SMSU: 1 disseminated) was explored and different blending ratios may impact upon the metallurgical performance.

The LCT results were used to develop metallurgical regression curves that can be used to project metal recoveries into the Cu and Ni concentrates. The Ni recovery and concentrate grade projections are presented in Figure 13-10. The R^2 values of the regression curves including all test results were reasonably good for the projection of Ni recovery and Ni concentrate grade as a function of the Ni head grade. Eliminating the Upper 138 Zone composite as an outlier, the R^2 values of for Ni recovery and Ni concentrate grade improved significantly to 0.94 (from 0.85) and 0.99 (from 0.92), respectively, as evidenced in the lower section of the chart.

The Cu recovery and concentrate grade projections are presented in Figure 13-11. The Cu flotation performance of the disseminated composites was less consistent in terms of concentrate grades and Cu recovery into Cu concentrate, which is evidenced by the lower R^2 values of 0.54 and 0.50, respectively. Eliminating the Upper 138 Zone composite as an outlier had little impact on the Cu recovery trendline equation but improved the R^2 value of the Cu concentrate projection from 0.54 to 0.79.

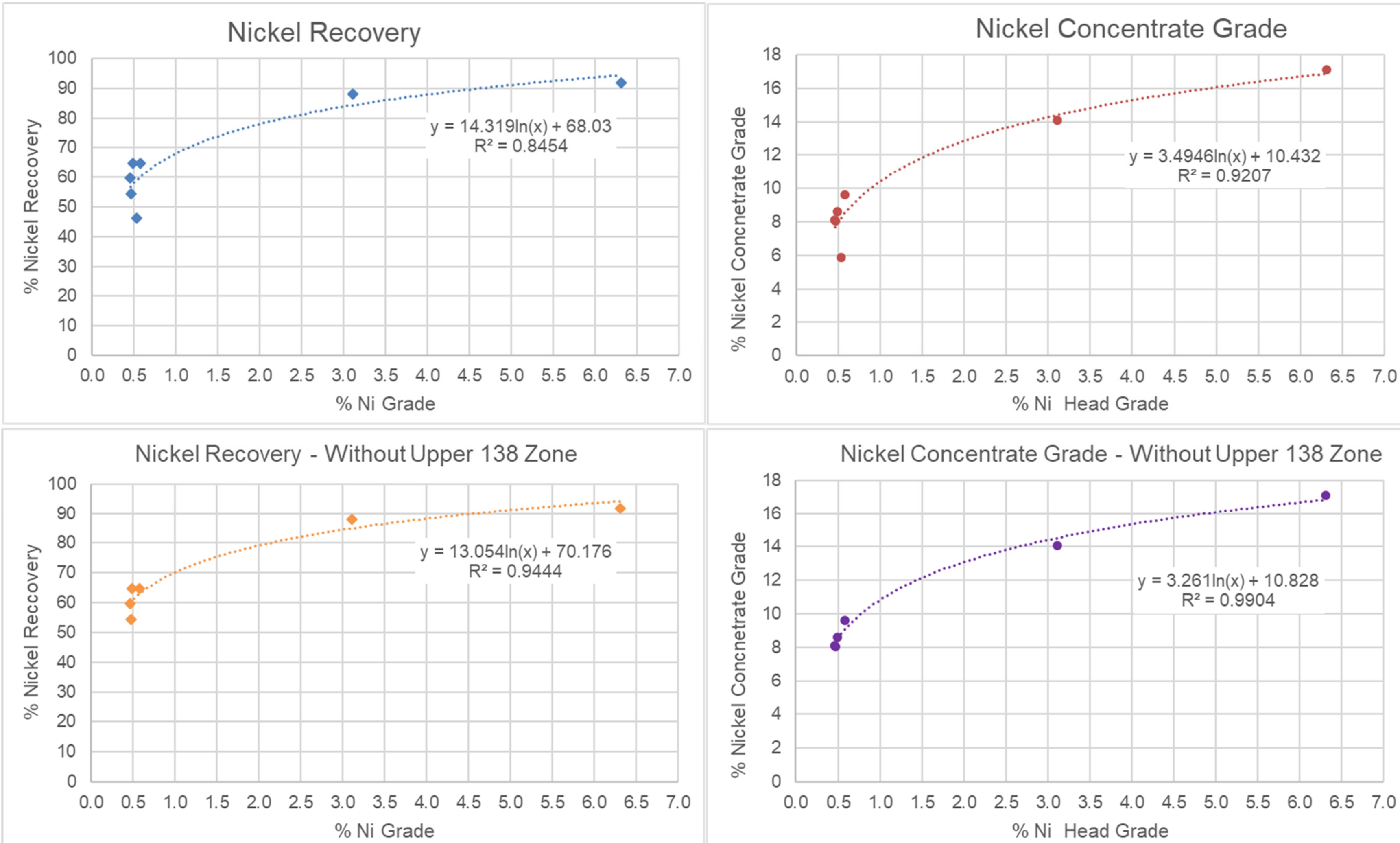


Figure 13-10: Metallurgical Projections for Ni Concentrates

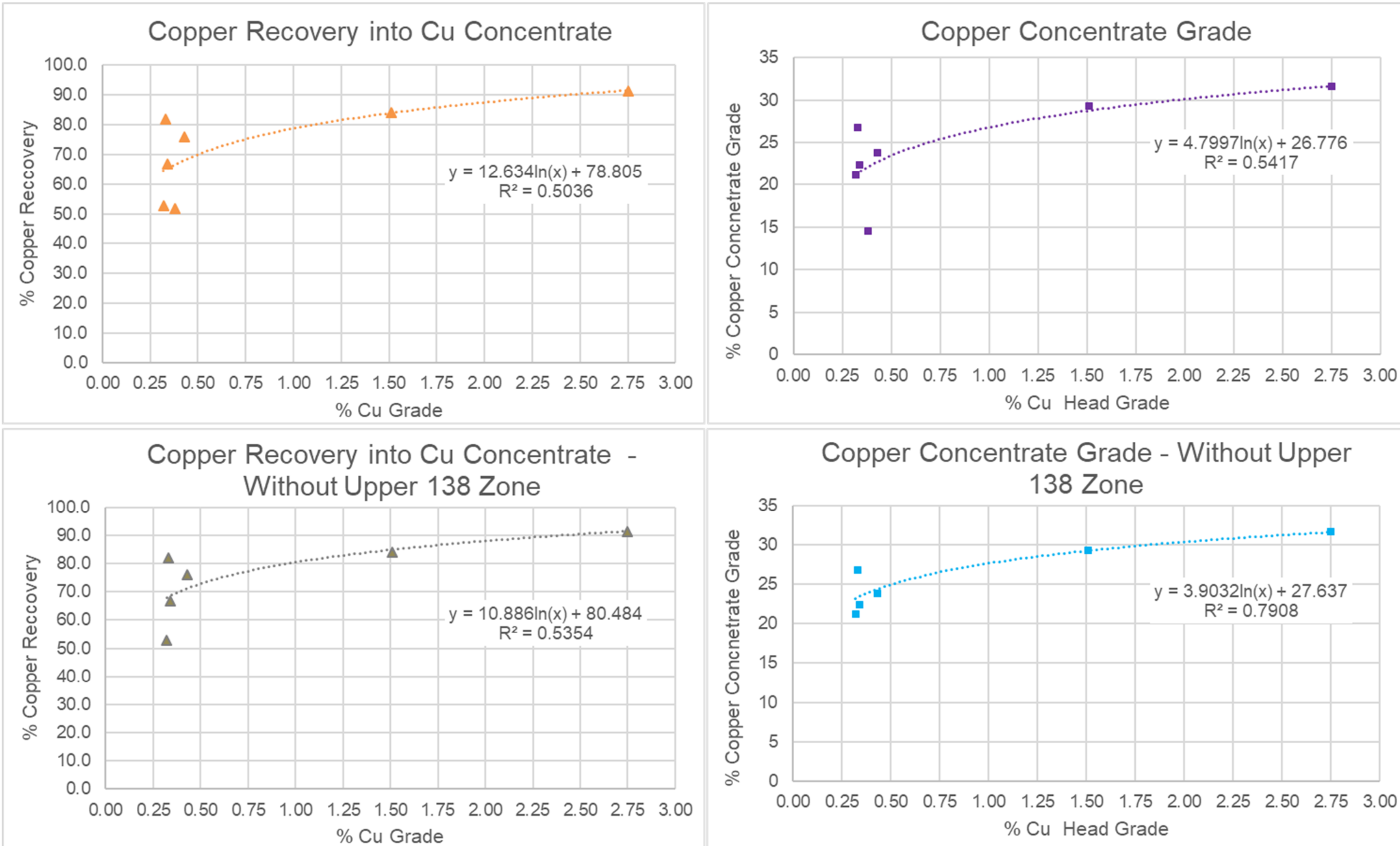


Figure 13-11: Metallurgical Projections for Cu Concentrates

The metallurgical projections of the Cu recovery into the Ni concentrate for low-grade samples are poor as shown in Figure 13-12. Between 11.2% and 20.8% of the contained Cu reported to the Ni concentrate at a comparable grade of 0.33-0.34% Cu. A mineralogical analysis of the Cu minerals in the Ni concentrate would be required to determine the nature of Cu losses and to develop a strategy to reduce these losses.

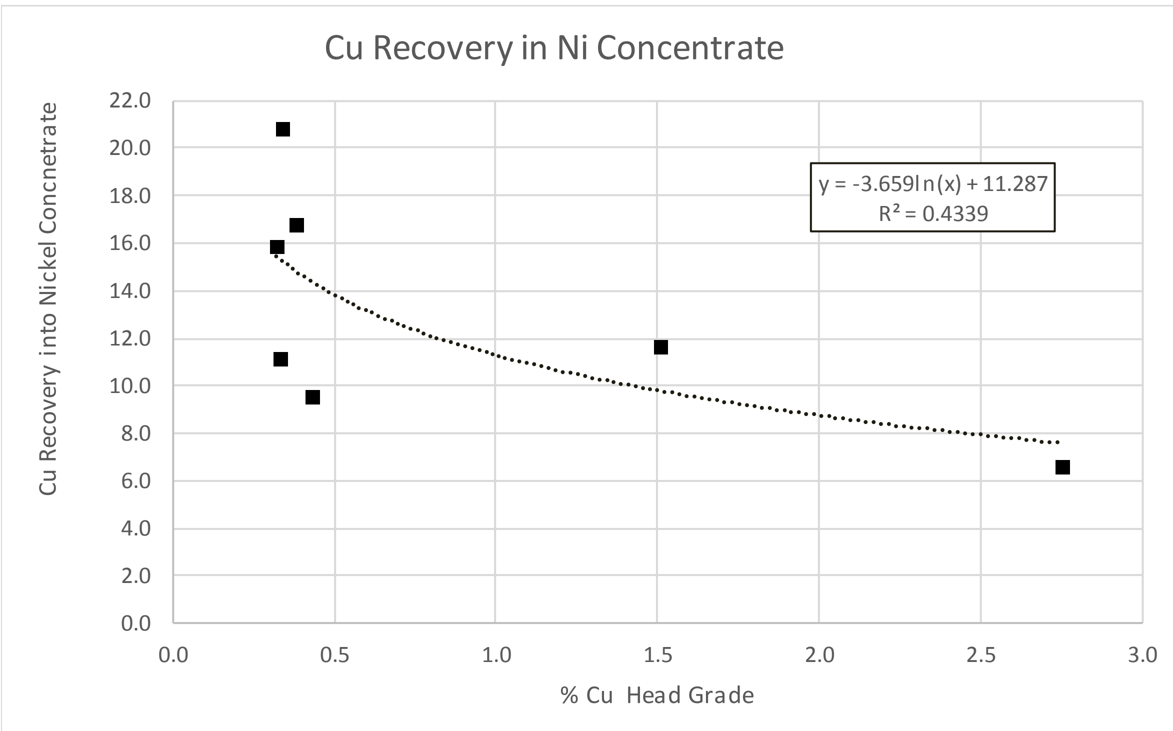


Figure 13-12: Metallurgical Projections for Cu Recovery into the Ni Concentrate

Levels of deleterious elements were low for the MSU and SMSU composites. Mg concentrations in the Ni concentrate of the MSU and SMSU composites were 0.22% MgO and 3.20% MgO, respectively and Fe:Ni ratios were 212:1 and 13.4:1 respectively. However, the Ni concentrates of the disseminated composites contained up to 14.6% MgO and alternative gangue depressants should be evaluated during the next phase of testing.

The MgO grades of the Ni concentrates of the seven domains are presented in Figure 13-13. The grades ranged from 0.22% MgO for the MSU composite to 14.6% MgO for the 138 Zone composite for very similar Ni head grades. The MgO grade of the Ni concentrate is driven primarily by the host rock mineralogy rather than the Ni head grade of the composite.

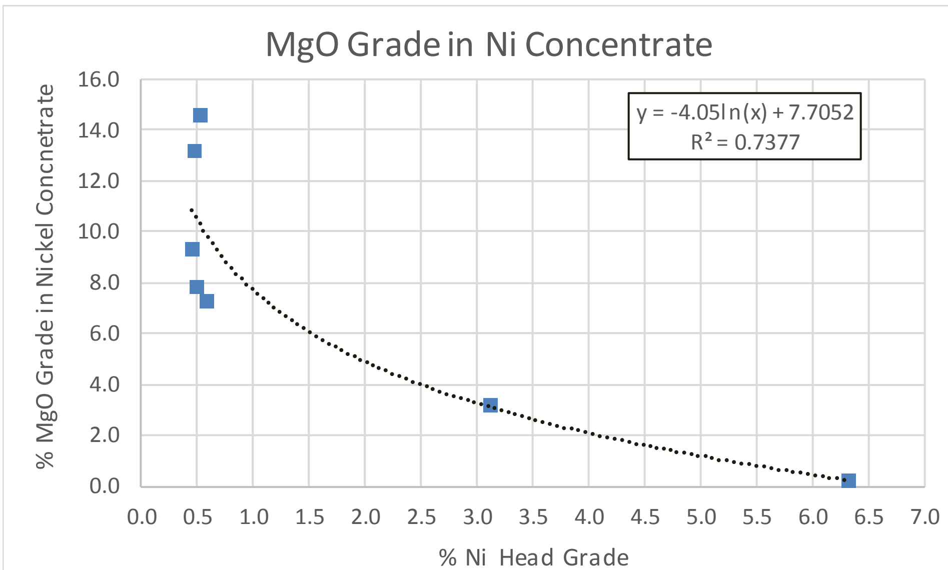


Figure 13-13: MgO Grade in Ni Concentrate

Ni smelters generally prefer a Ni concentrate with a minimum Fe:MgO ratio of 4.5:1. The Ni concentrates generated from the MSU and SMSU composites produced high Fe:MgO ratios of 212:1 and 13.4:1 respectively, and even the Upper CGO composite still yielded a ratio of 5.17:1. The remaining four disseminated composites produced Fe:MgO ratios between 2.11:1 for the Upper 138 zone composite and 4.06:1 for the CGO Bend Zone composite. Depending on the blending ratio of Ni concentrates from disseminated composites with Ni concentrate from the MSU and SMSU mineralization, an average Fe:MgO ratio of over 5:1 may be maintainable.

The reagent regime developed for the Tamarack North Project mineralization is presented in Table 13-7. It is noted that the dosage of 675 g/t CMC is considered very high and was driven by the low-grade disseminated samples tested in 2016/2017. It is postulated that the dosage could be reduced by at least 50% for the MSU and SMSU domains. Given the significant cost of the proposed reagent regime, a dosage optimization should be carried out during the next phase of testing.

Further, the collector dosage required for the disseminated composites was significantly higher than suggested by their sulphide head grades. This is a strong indication that collector “robbing” is taking place by some of the non-sulphide gangue minerals. Dosage levels vary for the different domains and must be established during the next phase of testing.

Table 13-7: Reagent Dosages

Reagent	Consumption of Mill Feed (g/t)
Sodium Isopropyl Xanthate (SIPX)	130
Potassium Amyl Xanthate (PAX)	330
Methyl Isobutyl Carbinol (MIBC)	125
Carboxy Methyl Cellulose (CMC)	675
Lime	730
Sodium Metasilicate	400

14. MINERAL RESOURCE ESTIMATE

14.1 Introduction

Caution to readers: *In this Item, all estimates and descriptions related to Mineral Resource estimates are forward-looking information. There are many material factors that could cause actual results to differ materially from the conclusions, forecasts or projections set out in this item. Some of the material factors include differences from the assumptions regarding the following: estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, commodity prices or product value, mining and processing methods and G&A costs. The material factors or assumptions that were applied in drawing the conclusions, forecasts and projections set forth in this Item are summarized in other Items of this report.*

The updated Mineral Resource estimate for the Tamarack North Project was completed by Mr. Brian Thomas, P.Geo., Senior Resource Geologist with Golder and senior peer review was provided by Mr. Paul Palmer, Principal, P.Geo., P.Eng. The estimate is based on assay data from drill programs completed by Kennecott between 2008 and 2016. The Tamarack North Project mineralization consists of three distinct geological domains as previously discussed in Section 7, including the SMSU hosted in CGO, the MSU hosted in meta-sediments, and the 138 Zone hosted in mixed FGO and CGO peridotites. Grade variables evaluated in this PEA include Ni, Cu, Co, Pt, Pd and Au as well as SG.

The software used for the updated Mineral Resource estimate in this PEA was Datamine Studio RM, release 1.2.47.0 (Datamine).

14.2 Drill Hole Data

A total of 242 diamond drill holes were provided by Talon (derived from Kennecott Database) regarding the Tamarack North Project, containing 37,265 assay intervals having a total core length of 100,692 m. All drill hole data was provided as of April 27, 2017.

The Tamarack North Project drill hole data was imported into Datamine from electronic .csv files and no interval errors were encountered during the process.

The drill hole file was reviewed in plan and section to validate the accuracy of the collar locations, hole orientations and down hole trace, and the assay data was analyzed for out of range values. The drill hole database was determined by Golder to be of suitable quality to support the updated Mineral Resource estimate in this PEA.

14.3 Geological Interpretation

14.3.1 Sample Selection

Four mineral envelopes were created to represent the MSU (green), SMSU (red) and 138 Zone (purple) occurring at the Tamarack North Project as illustrated in Figure 14-1. The SMSU was split into Upper and Lower segments based on observed grade distribution and domain analysis.

An approximate 0.83% NiEq cut-off was used to constrain the mineral envelopes in areas of continuous mineralization, however, some lower grade material was included where required to maintain geological continuity. NiEq is further explained in Section 14.9. Figure 14-1 illustrates the mineral domains and the samples within each. The Tamarack North Project resource estimate is based on the samples captured inside the domains as described in Table 14-1.

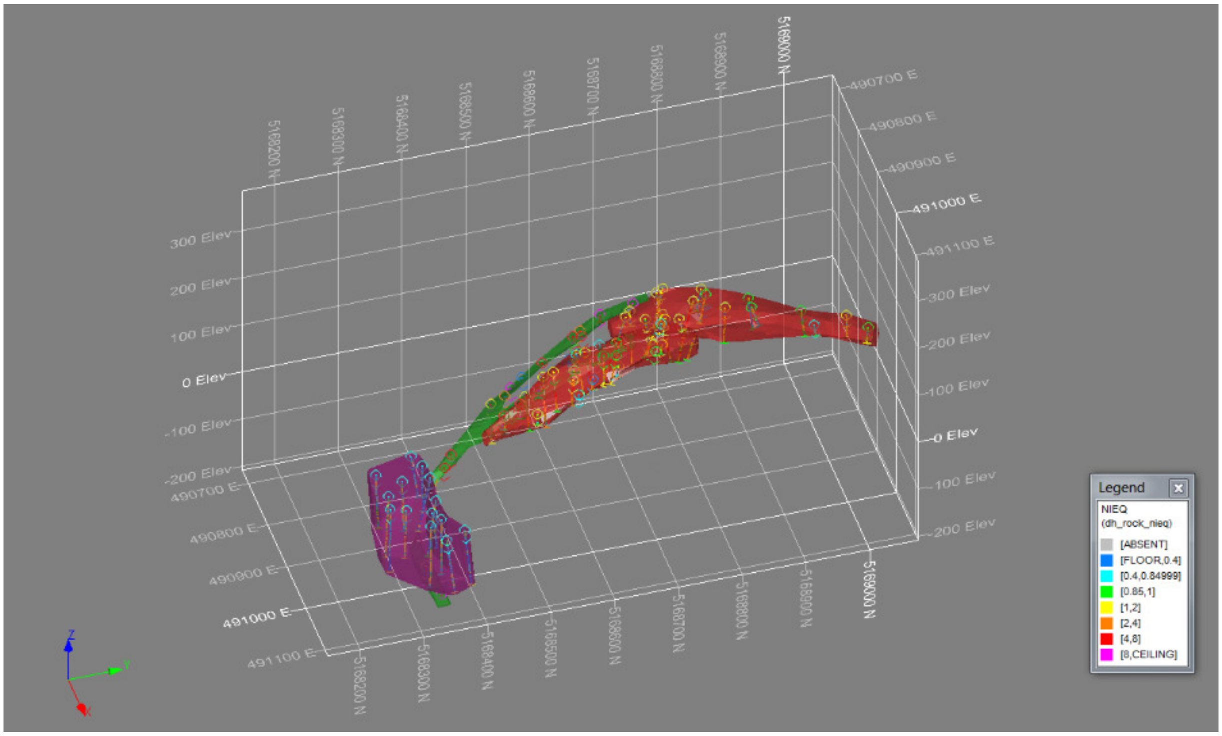


Figure 14-1: Oblique View of Mineral Domains Tamarack North Project (Facing N-W)

Raw sample intervals were captured inside each domain wireframe and verified visually to confirm the accuracy of the process. Table 14-1 provides the sample break down by domain. It is noted that some holes intersect multiple domains.

Table 14-1: Summary of Captured Samples Tamarack North Project

Domain	Number of Holes	Number of Samples	Total Sample Length (m)
Upper SMSU	20	643	971
Lower SMSU	27	828	1,246
Total SMSU	**38 ***	**1,471**	**2,217**
MSU	24	189	209
138 Zone	14	1,113	1,575
Total	**76**	**2,773**	**4,001**

* 9 of the holes drilled for SMSU intersect both Upper and Lower SMSU

14.4 Exploratory Data Analysis (EDA)

Descriptive statistics combined with a series of histograms and X-Y scatter plots were used to analyze the grade distribution of each sample population and to determine the presence of outliers and correlations between metals for each mineral domain.

14.4.1 Descriptive Statistics

Table 14-2 provides a summary of the descriptive statistics for the raw sample populations captured from within each mineral domain.

Table 14-2: Descriptive Statistics of the Tamarack North Project Sample Population

Domain	Field	Samples	Minimum	Maximum	Mean	Standard Deviation	Skewness	Coefficient of Variation
Upper SMSU	Ni (%)	643	0.11	4.49	1.02	0.91	1.85	0.89
Upper SMSU	Cu (%)	643	0.01	2.40	0.62	0.47	1.21	0.76
Upper SMSU	Co (%)	643	0.006	0.108	0.029	0.021	1.74	0.72
Upper SMSU	Pt (ppm)	643	0.003	0.863	0.155	0.13	2.08	0.82
Upper SMSU	Pd (ppm)	643	0.003	0.565	0.098	0.08	1.79	0.79
Upper SMSU	Au (ppm)	643	0.001	0.571	0.101	0.08	1.55	0.78
Lower SMSU	Ni (%)	828	0.12	5.06	1.68	1.28	0.68	0.76
Lower SMSU	Cu (%)	828	0.01	2.98	0.93	0.58	0.70	0.63
Lower SMSU	Co (%)	828	0.008	0.131	0.044	0.031	0.72	0.71
Lower SMSU	Pt (ppm)	828	0.006	5.410	0.575	0.41	2.94	0.72
Lower SMSU	Pd (ppm)	828	0.003	1.510	0.347	0.19	1.24	0.54
Lower SMSU	Au (ppm)	828	0.001	1.265	0.254	0.17	1.17	0.66
MSU	Ni (%)	189	0.017	10.15	5.53	2.30	-0.64	0.42
MSU	Cu (%)	189	0.005	5.75	2.41	0.99	-0.43	0.41
MSU	Co (%)	189	0.001	0.216	0.114	0.051	-0.42	0.44
MSU	Pt (ppm)	189	0.002	1.18	0.49	0.23	0.02	0.47
MSU	Pd (ppm)	189	0.0025	4.65	0.68	0.57	2.78	0.84
MSU	Au (ppm)	189	0.001	5.03	0.29	0.45	7.63	1.57
138 Zone	Ni (%)	1,113	0.115	9.89	0.64	0.62	6.65	0.96
138 Zone	Cu (%)	1,113	0.007	7.56	0.46	0.51	5.17	1.10
138 Zone	Co (%)	1,113	0.009	0.198	0.021	0.011	6.84	0.54
138 Zone	Pt (ppm)	1,113	0.00025	112	0.212	2.00	55.41	9.45
138 Zone	Pd (ppm)	1,113	0.0005	4.88	0.103	0.12	21.41	1.16
138 Zone	Au (ppm)	1,113	0.0005	1.48	0.109	0.10	4.44	0.96

Note: Sample statistics weighted by length for all domains.

Figure 14-2 to Figure 14-5 provide examples of the frequency distribution of the Ni sample populations of each domain. The Ni population was found to be weakly bi-modal in the SMSU, normal in the MSU and positively skewed in the 138 Zone.

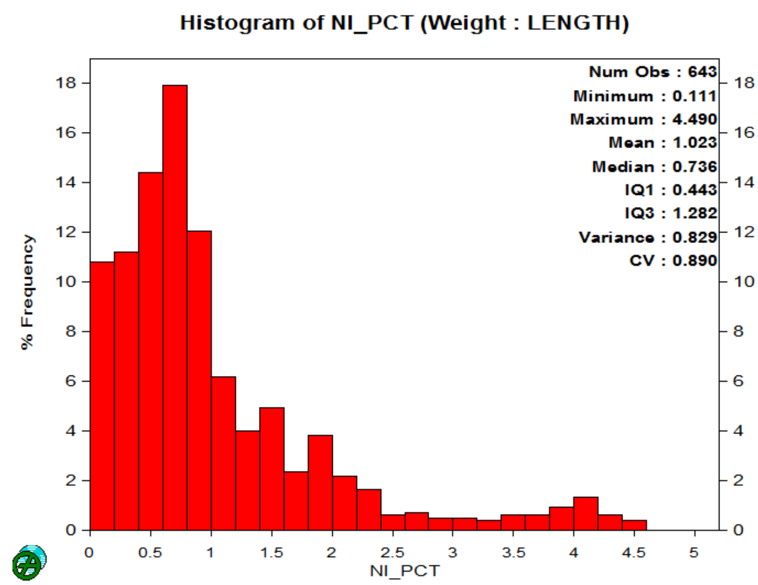


Figure 14-2: Histogram of %Ni for the Upper SMSU

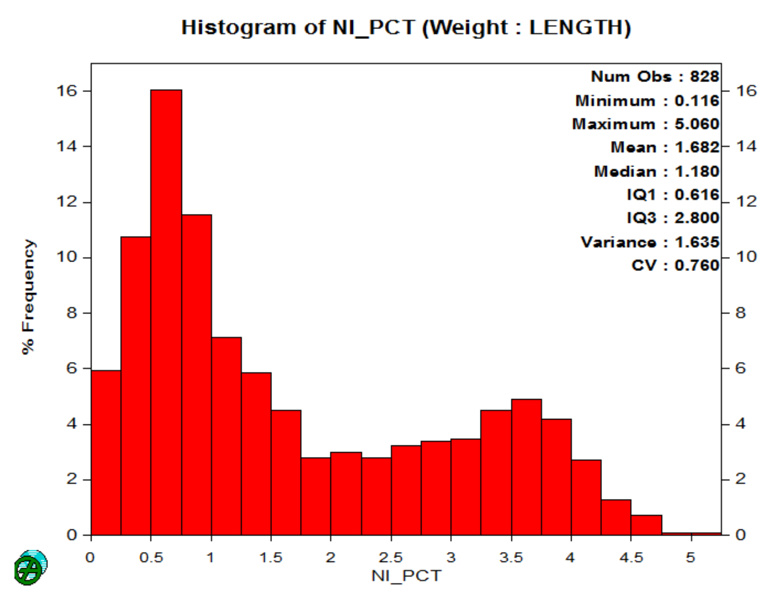


Figure 14-3: Histogram of %Ni for the Lower SMSU

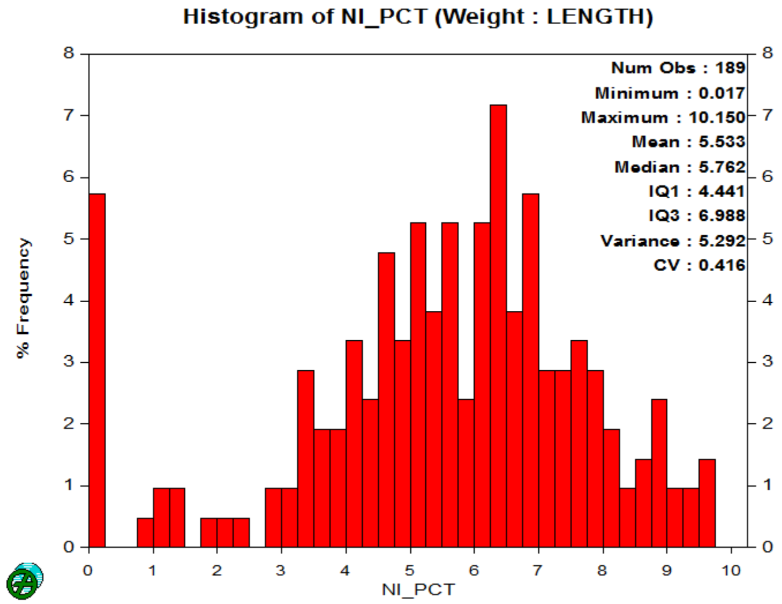


Figure 14-4: Histogram of %Ni for MSU

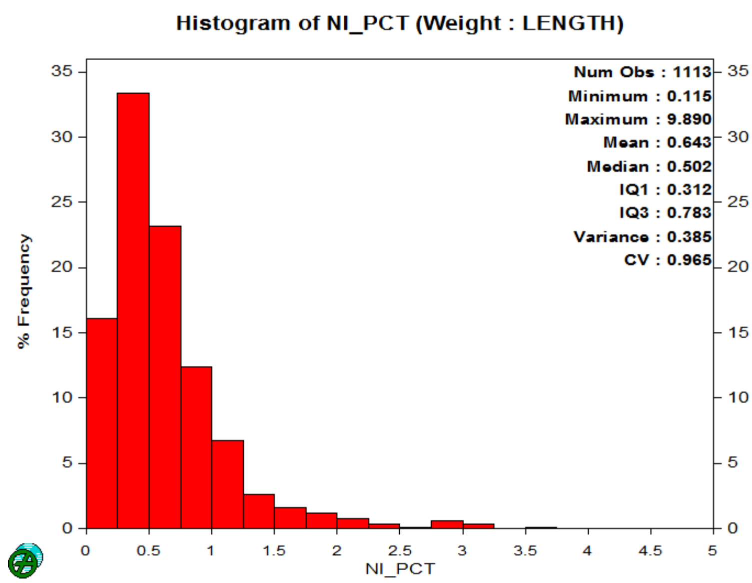


Figure 14-5: Histogram of %Ni for 138 Zone

Un-assayed intervals were assumed to be waste and assigned a metal value of one-half the detection limit for each metal as listed in Table 14-3. There was only one interval with absent metal assays for the entire captured sample population.

Table 14-3: Default Grades for Absent Data

Metal	Default Value
Ni	0.0025 %
Cu	0.0025 %
Co	0.001 %
Pt	0.0025 ppm
Pd	0.0025 ppm
Au	0.005 ppm

14.4.2 Correlations

A correlation matrix was generated for each domain, to determine the relationship between all metals and density values as illustrated for the Lower SMSU domain in Table 14-4.

Table 14-4: Correlation Matrix of the Lower SMSU

	Ni	Cu	Co	Pt	Pd	Au	S	Density
Ni	1							
Cu	0.8784	1						
Co	0.9865	0.8324	1					
Pt	-0.1219	0.0825	-0.1747	1				
Pd	0.0283	0.2011	-0.0378	0.7748	1			
Au	-0.0934	0.1673	-0.1688	0.7090	0.7049	1		
S	0.9877	0.8435	0.9970	-0.1648	-0.0278	-0.1605	1	
Density	0.8289	0.6797	0.8561	-0.2125	-0.0962	-0.2891	0.8600	1

In the Lower SMSU, Ni was found to have a strong correlation with Cu, Co, S, and a reasonably good correlation with measured density values. Cu was found to have a higher correlation with the PGMs than Ni. These are typical relationships generally associated with magmatic Ni sulphide deposits. The correlation between S and density was used as the basis to calculate density for absent intervals in the SMSU domain as described further in this section. These correlations were also used to make assumptions that Co and density have the similar spatially continuity as Ni as described in the variography section.

In the Upper SMSU, Ni was found to have the similar correlations with Cu, Co, S but was not very well correlated with density, so density values were not calculated. The raw lab measured density values were used to estimate density into the model as explained further in Section 14.6.4.

14.4.3 **Bulk Density**

Density data obtained from cut core (single piece taken from sample bag) lab measurements (ALS Minerals) was the main source of the data values in the supplied assay database. Field measurements were also taken on site from 10 cm core samples, taken approximately every 20 m, using the weight in air versus the weight in water method based on the following formula:

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

Golder elected to only use the density measurements obtained from lab measurements and did not use the field measurements. Calculated density values were substituted, where no lab measured data was available, based on polynomial regression formulas defined for each mineral domain. Density was assigned to absent drill hole intervals by polynomial regression for the MSU and Lower SMSU domains based on moderate to good correlations with Ni and Sulphur. There was a poor correlation between density and Ni and % Sulphur in the Upper SMSU so no regression was used and density was estimated using OK with the available lab measured data. No lab measured density data was available for the 138 Zone. Density was later assigned to the 138 Zone model based on a regression formula derived from the Lower SMSU domain, limited to the same Ni and Cu grade range as observed in the 138 Zone. Density data from field measurements was later used to validate the model. The regression formulas used for each domain are listed below:

- Density (Lower SMSU) = 2.75988 + Sulphur (%) x 0.03808;
- Density (MSU) = 2.79247 + Ni x 0.17519;
- Density (138 Zone) = 2.76785 + Ni x 0.09198 (applied to block model, not estimated).

Based on reasonably good correlations with the density data, Golder decided that it would be appropriate to weight the base metal grades (Ni, Cu and Co) by density for estimation purposes for the Lower SMSU and MSU domains. New grade fields density-weighted nickel grade (QNi), density-weighted copper grade (QCu), and density-weighted cobalt grade (QCo) were calculated by multiplying the metal grade by measured density, where available, and calculated density in the absence of measured data. Grades in the Upper SMSU and 138 Zones were not weighted by density.

X-Y scatter plots were generated to illustrate the relationship between Sulphur and density, for the Lower SMSU domain, and Ni and density for the MSU domain as shown in Figure 14-6 and Figure 14-7.

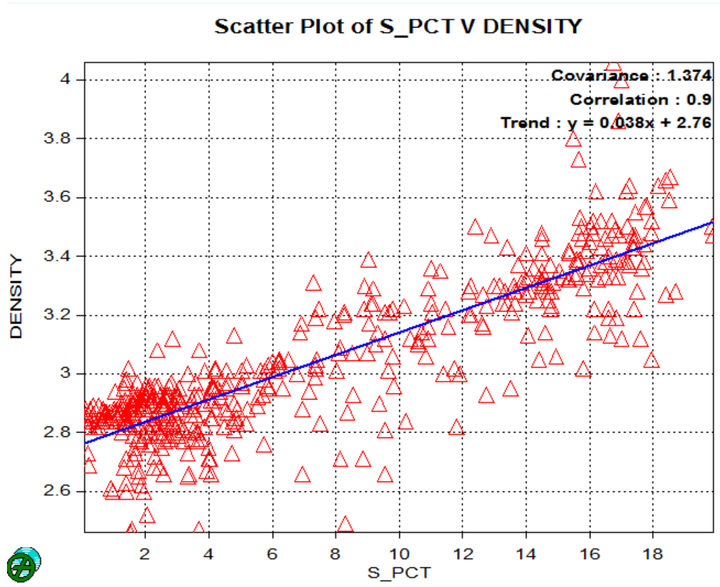


Figure 14-6: Scatter Plot of %S vs Density in the Lower SMSU

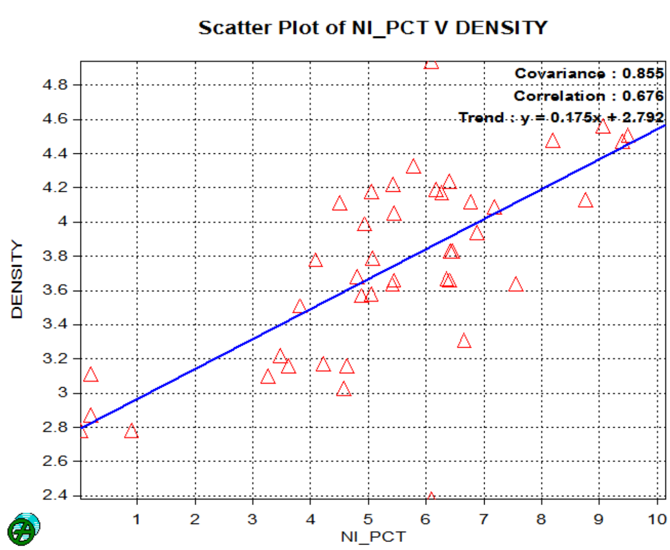


Figure 14-7: Scatter Plot of %Ni vs Density in MSU

14.4.4 **Outliers**

X-Y scatter plots were generated to assess the sample population for outlier values. High-grade outlier data has the potential to bias the block model grades if they are not handled by top cutting or otherwise restricting their influence through other estimation criteria. A minor number of high-grade outliers were identified in the Pt, Pd and Au populations of each domain and top-cut as indicated by the red lines shown in Figure 14-8, Figure 14-9 and Table 14-5. Minor top cuts were performed on the Ni and Cu grades in the 138 Zone.

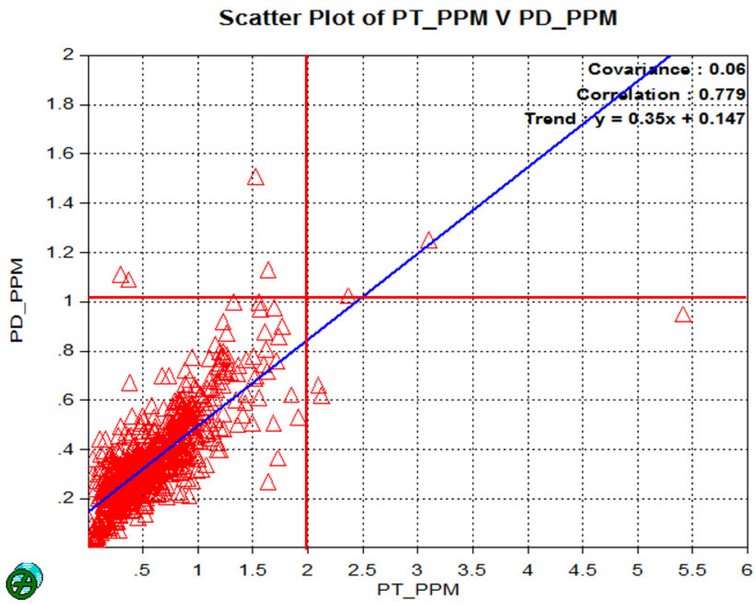


Figure 14-8: Scatter Plot of Pt vs Pd in the Lower SMSU

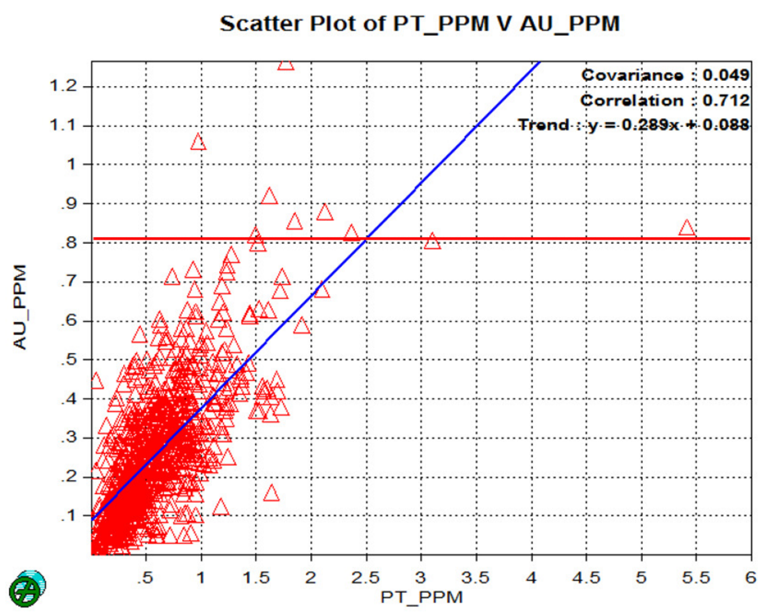


Figure 14-9: Scatter Plot of Au vs %Cu in the Lower SMSU

The identified PGM outliers were top-cut as listed in Table 14-5. Top cutting reduces the value of an outlier to a set maximum value, reducing the potential for bias in the block model.

Table 14-5: Summary of Top Cuts

Domain	Metal	Top Cut Value	# Samples Cut
Upper SMSU	Pt (ppm)	0.8	3
	Pd (ppm)	0.45	1
	Au (ppm)	0.4	4
Lower SMSU	Pt (ppm)	2	5
	Pd (ppm)	1	6
	Au (ppm)	0.8	8
MSU	Cu %	5.0	1
	Pt (ppm)	1.0	2
	Pd (ppm)	1.71	8
	Au (ppm)	0.76	3
138 Zone	Ni %	5	1
	Cu %	4	1
	Pt (ppm)	1	5
	Pd (ppm)	1	1
	Au (ppm)	0.8	5

14.5 Compositing

Compositing samples is a technique used to give each sample a relatively equal length weighting to reduce the potential for bias due to uneven sample lengths. A histogram of raw sample length was generated for each domain to determine the most common sample length used at the Tamarack North Project as illustrated in Figure 14-10 for the Lower SMSU.

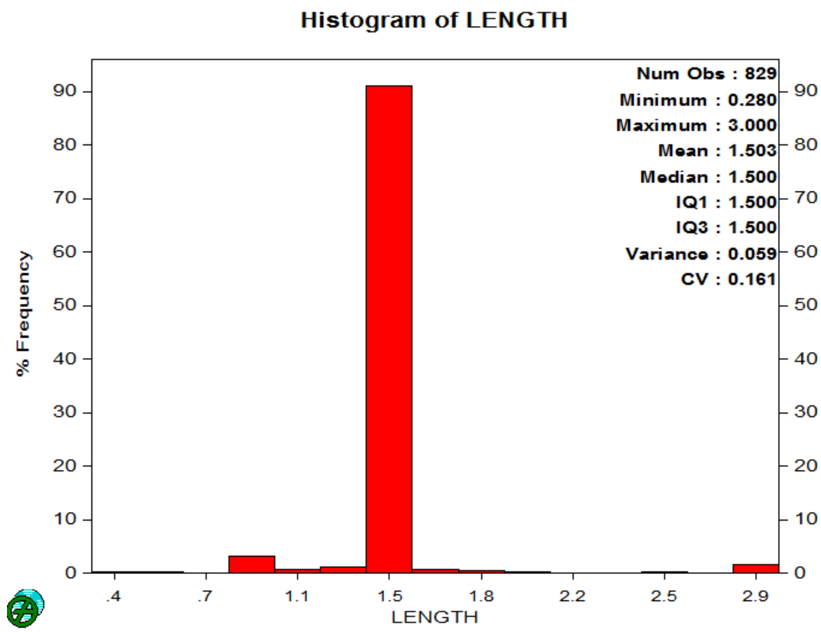


Figure 14-10: Histogram of Raw Sample Length (m) (Lower SMSU)

Samples captured within the Upper and Lower SMSU and 138 Zone domains were composited to an average length of 1.5 m and the samples in the MSU domain were composited to an average length of 1 m. These intervals were chosen because they were the most common sample lengths and provide a reasonable level of sample support. An option to use a variable composite length was chosen for all domains to prevent the loss of sample information and the creation of short composites that are generally formed along the contacts when using a fixed length.

Composite samples were validated visually in plan and section and a histogram of composite length was generated to confirm compositing was completed as expected. The histograms displayed a normal distribution around the chosen composite lengths and the total lengths of the composites, as well as the mean sample grades, which were found to match that of the raw captured samples.

14.6 Resource Estimation

14.6.1 Unfolding

The "Unfold" process within Studio 3 was used to transform the composite sample data from Cartesian coordinates into an UCS, as defined by the geometry of the footwall and hanging wall contacts of each mineral wireframe. This transformation essentially removes bends, pinches and swells in the mineral model, allowing for more robust variogram calculations and grade estimation. This was considered an appropriate process to employ given the variable orientations of each mineral wireframe.

Strings representing the footwall (white) and hanging wall (red) contacts of the deposit were constructed and tagged in cross-section view, as shown in Figure 14-11. These strings were then used to transform the composite samples into the UCS. The same unfold strings are used in the grade estimation process to unfold the blocks into the same transformed system as the composite samples. The process unfolds discretization points from the prototype model and estimates the grades for each in the UCS. The process then assigns the estimated grades back to the corresponding cell in the Cartesian model. In the UCS, the X-axis is assigned to UCSA which represents the across strike thickness of the zone, the Y-axis is assigned to UCSB representing the down-dip direction of the zone and the Z-axis is assigned to UCSC representing the along strike direction of the zone.

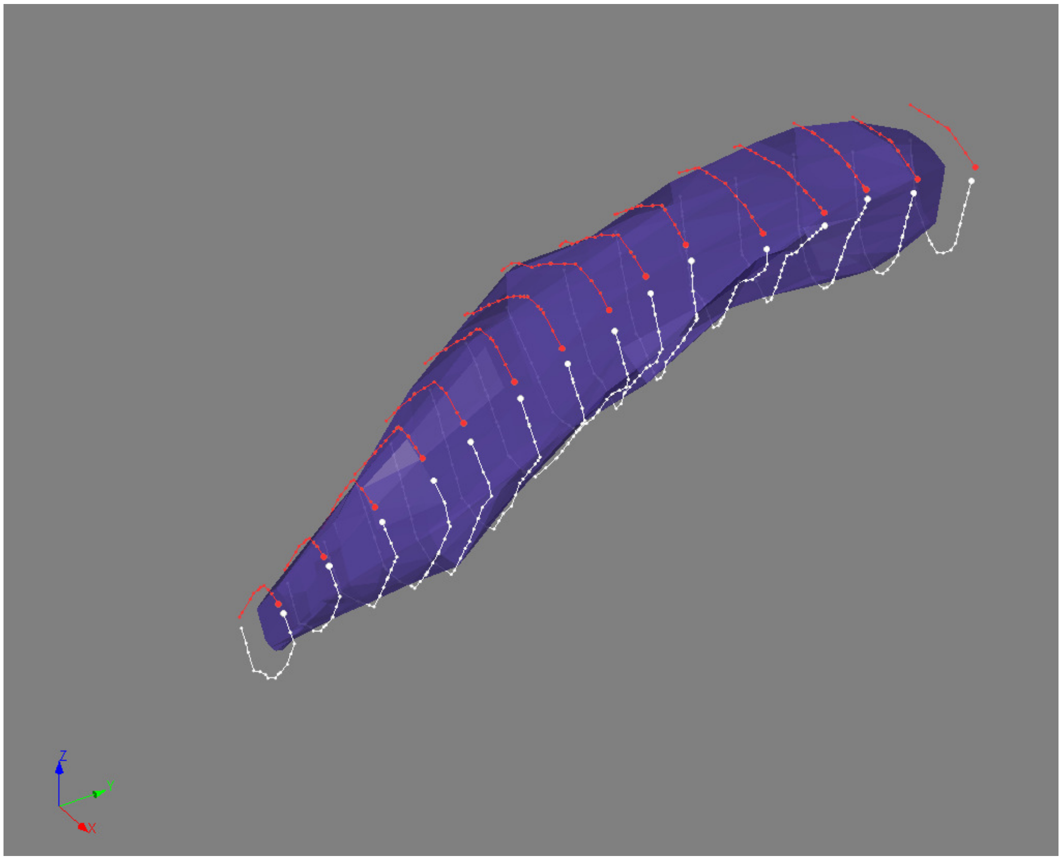


Figure 14-11: Lower SMSU Unfold Strings, Oblique View Facing NW

The unfolded samples were validated visually in unfold space for each zone. Quadrilateral strings created during the process were inspected to confirm that unfolding had performed as expected as shown in Figure 14-12 for the MSU domain.

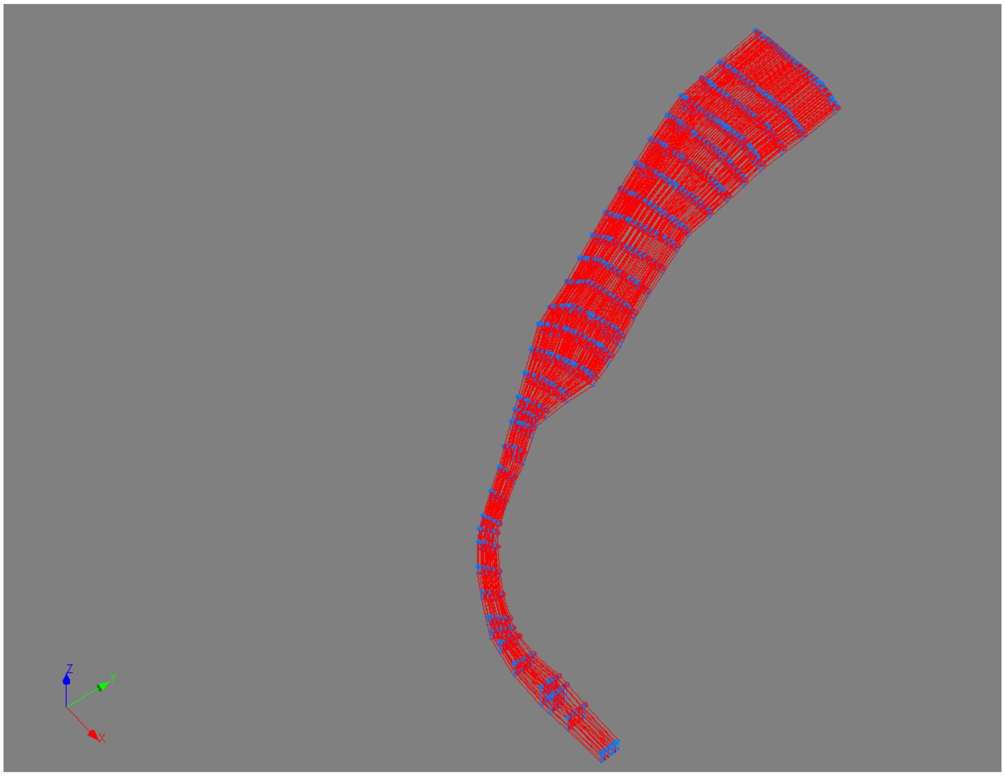


Figure 14-12: MSU Review of Quadrilateral Strings, Oblique View Facing NW

Visual inspection of the nearest neighbour (NN) model (as described later in Section 14.8), confirmed that the unfolding process had worked as expected for all zones.

14.6.2 Grade Variography

Experimental grade variograms were generated from the unfolded composite data for all model variables to assess the spatial variability for the purpose of assigning Kriging weights to the composite samples. Samples situated in the directions of preferred geological continuity receive higher Kriging weights resulting in a greater influence on the block estimate.

Pairwise relative experimental grade variograms were generated based on the parameters outlined in Table 14-6. Variograms were not generated for the MSU domain due to insufficient data across the width of the deposit.

Table 14-6: Grade Variogram Parameters

Elements	SMSU (Upper and Lower)	138 Zone
Rotations	0	0
Lag Distance (m)	20	30
Number of Lags	15	15
Sub-Lag Distance (m)	5	15
Number Lags to be Sub-Lagged	5	4
Regularization Angle (degrees)	22	22
Number of Azimuths	2	2
Cylindrical Search Radius	30	30

A set of two structure spherical variogram models were fitted to the variogram data. An example of the variogram model for Ni in the Lower SMSU is provided in Figure 14-13. Summaries of all the variogram models are provided in Table 14-7.

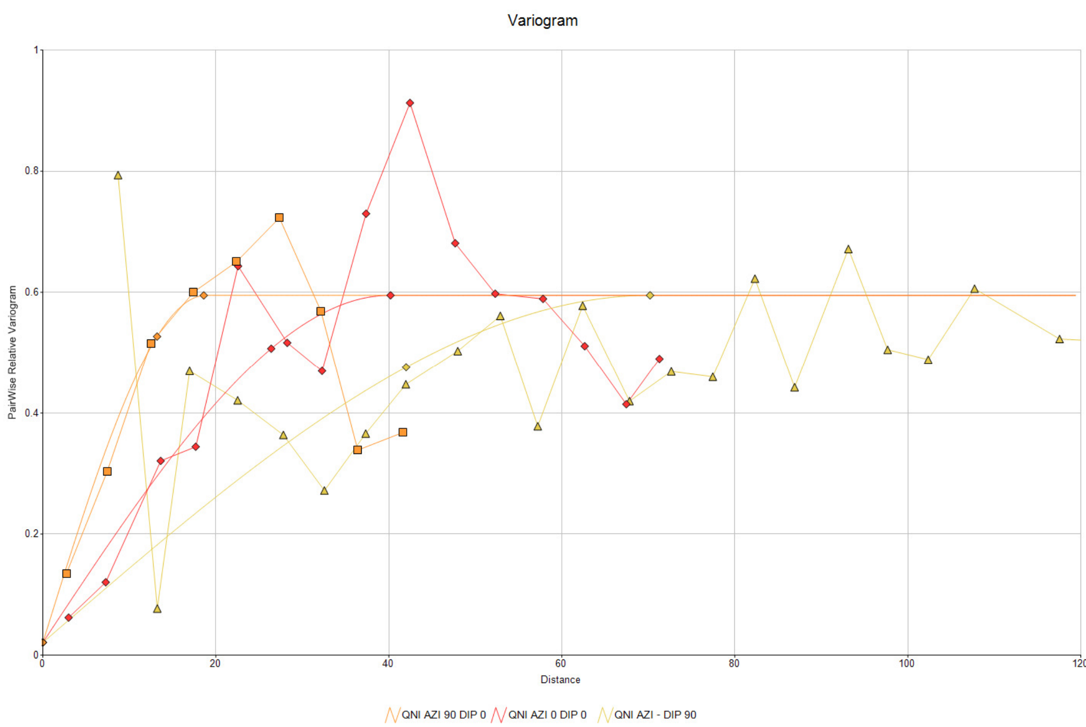


Figure 14-13: Lower SMSU %Ni Variogram Model

Table 14-7: Tamarack Grade Variography (Unfolded)

Mineral Domain	Element	Nugget	1st Structure				2nd Structure			
			X-Range	Y-Range	Z-Range	Variance	X-Range	Y-Range	Z-Range	Variance
Lower SMSU	QNi	0.021	12.9	26.3	42.2	0.006	18.8	40.1	70.2	0.568
	QCu	0.053	12.9	17.3	20.3	0.084	27.4	31.8	50.7	0.357
	Pt	0.073	2.4	18.3	44.6	0.135	26.3	50.9	79.7	0.211
	Pd	0.058	13.9	18.1	17.9	0.082	37	40.2	59.7	0.194
	Au	0.074	6.5	11.4	25.6	0.116	18.1	27.4	60.1	0.226
Upper SMSU	Ni	0.021	6.4	9.9	34.8	0.143	20.5	39.6	79.9	0.392
	Cu	0.053	12	11.3	45.1	0.227	20	59.7	80.1	0.296
	Pt	0.073	9	21.7	32.6	0.163	27.4	60.7	79.5	0.27
	Pd	0.075	10.9	17.5	27.4	0.173	25.9	59.7	79.7	0.259
	Au	0.074	5.4	18.5	38.8	0.303	20.1	60.3	79.9	0.129
138 Zone	Ni	0.056	9.7	23.5	16.8	0.003	20.2	45.8	50.1	0.317
	Cu	0.129	7.9	18.6	21.1	0.003	20	45.7	50.2	0.47
	Pt	0.088	7.5	21.1	12.1	0.001	20.6	46.4	49.6	0.266
	Pd	0.108	7.7	16.4	15.8	0.019	19.8	44.7	50.2	0.228
	Au	0.155	8.5	17	17.6	0.04	20.2	45.3	50	0.259

Notes:
In the UCS, X (vertical) is across the mineralization, Y is down-dip, and Z is along strike.
QNi (density-weighted nickel grade) and QCu (density-weighted copper grade) are density weighted variables.

The down-dip (Y-Range) and along strike (Z-Range) directions of the mineralization were determined to be the directions of greatest grade continuity. The second structure range of each axis was used as the basis to define the search ellipse dimensions used for interpolating grades into the Mineral Resource block model.

14.6.3 Block Model Definition

The Tamarack North Project prototype model covers an area of UTM NAD 83 grid coordinates from 490,650 E to 491,200 E, 5,168,150 N to 5,169,100 N, and -250 to 150 m elevation. Block shape and size is typically a function of the geometry of the deposit, density of sample data, and expected potential SMU. On this basis, a parent block size of 7.5 m (E-W) by 7.5 m (NS) by 7.5 m elevation was chosen for the SMSU and 138 Zone. The block model definition parameters are summarized in Table 14-8.

Table 14-8: SMSU and 138 Zone Block Model Prototype Summary

Origin			Block Size (m)			Number of Blocks		
X	Y	Z	X	Y	Z	X	Y	Z
490,650.0	5,168,150.0	-250.0	7.5	7.5	7.5	75	125	55

All mineral domain solids were filled with blocks using the parameters described in Table 14-8 except for the MSU domain. Cell splitting (2X) was used for improved definition of boundaries. All domain volumes were then compared to the filled model volumes to confirm there were no errors during the process.

The MSU model prototype was defined as described in Table 14-9.

Table 14-9: MSU Block Model Prototype Summary

Origin			Block Size (m)			Number of Blocks		
X	Y	Z	X	Y	Z	X	Y	Z
490,650.0	5,168,150.0	-250.0	3	3	1.5	183	316	267

14.6.4 Estimation Methodology

OK was the interpolation method chosen to estimate grades in the Upper and Lower SMSU and 138 Zone. This method assigns weights to the samples based on the modelled spatial continuity (variography) of the sample data. The MSU domain did not have sufficient data for variogram modeling, so the ID^3 interpolation method was chosen. This method assigns weights to samples based on the distance from the block centroid, with closer samples having a higher weighting. ID^3 was chosen over inverse distance squared (ID^2) due to the high-grade nature of the domain in order to prevent high-grades from spreading through areas of lower grade. ID^2 was also used in the SMSU and 138 Zone for comparative purposes, but not chosen for resource reporting.

Base metals (Ni, Cu) were density weighted for the Lower SMSU and MSU Zones based on observed correlations previously discussed. The 138 Zone and Upper SMSU were not density weighted due to insufficient density data. Density values in the 138 Zone were calculated from OK grade estimates based on a regression formula as discussed in Section 14.4.3. Density in the Upper SMSU was estimated from the raw lab determined values using OK, and missing values were assigned the NN value or a default of 2.89 tonnes per cubic metre (t/m^3). All domains utilized a nested search strategy, along with unfolding and top-cutting as summarized in Table 14-10.

NN interpolation was also used to estimate each domain for model validation purposes. NN estimates use the sample grade closest to the centroid of the block and represent de-clustered sample grades for use in block model validation.

Table 14-10: Summary of Estimation Methodology

Geological Domain	Interpolation Methods	SG Weighting of Base Metals	Nested Search	Unfolding	Top Cutting
Lower SMSU	OK, ID^2, NN	Yes	Yes	Yes	Yes
Upper SMSU	OK, ID^2, NN	No	Yes	Yes	Yes
MSU	ID^3, NN	Yes	Yes	Yes	Yes
138 Zone	OK, ID^2, NN	No	Yes	Yes	Yes

Nested, anisotropic searches were performed for all domains using the modelled second structure variogram ranges for each element as a guide for each of the three axes, orthogonal to the unfolded plane of the deposit. The search parameters for all elements are summarized in Table 14-11. It is noted that as with the variogram ranges, these search parameters are used in unfolded space during the interpolation process, where X is across the deposit, Y is down-dip, and Z is in the strike direction. The search radius of the first search was restricted to one-half the variogram range with the second search being the full variogram range and the third search being twice the variogram range. For the MSU domain the search ellipse was based on the relative geometry of the mineralization. Search strategies for each domain used an elliptical search with a minimum of four samples and a maximum of 12 samples, utilizing an octant restriction of at least three octants with a maximum of four samples per octant, as well as a maximum of six samples per hole. Unestimated blocks were flagged in the model and then estimated without octant or hole restrictions, along with expanded search distances. Search parameters are further summarized in Table 14-11.

Table 14-11: Summary of Search Parameters (Unfolded)

Element	1st Search					2nd Search			3rd Search			All
	X-Range	Y-Range	Z-Range	Min. Samples	Max. Samples	SVOL Factor 2	Min. Samples	Max. Samples	SVOL Factor 3	Min. Samples	Max. Samples	Max. per hole
Lower SMSU	10	20	35	4	12	2	4	12	4	2	12	6
Upper SMSU	10	30	40	4	12	2	4	12	4	2	12	6
MSU	4	10	20	6	12	2	6	12	3	6	12	4
138 Zone	10	22	25	4	12	2	4	12	4	2	12	6

14.7 Mineral Resource Classification

Resource classifications were assigned to broad regions of the block model based on QP confidence related to geological understanding and continuity of mineralization relative to the style of mineralization, along with data quality and density. A combination of drill hole density and the search volume used to estimate the grade of the block was used as an addition guide for outlining classification regions. Areas where the drill hole spacing is on average 25 m or less and most of the blocks were estimated in the first or second search volume are classified as “Indicated Mineral Resource”. Areas where the drill hole spacing is wider than 25 m and the majority of block was estimated in the second or third search volume are classified as “Inferred Mineral Resource”. No Measured Mineral Resource was outlined from the block model as it is Golder’s opinion that the drill spacing and orientation of drilling is insufficient to adequately define the volume and extent of mineralization to meet that classification. Figure 14-14 outlines the Mineral Resource classifications assigned to the SMSU, where green areas are Indicated Resources and blue areas are Inferred

Resources. The MSU and 138 Zone were classified entirely as Inferred Resources due more complex geology/geometry and greater than 25 m drill spacing.

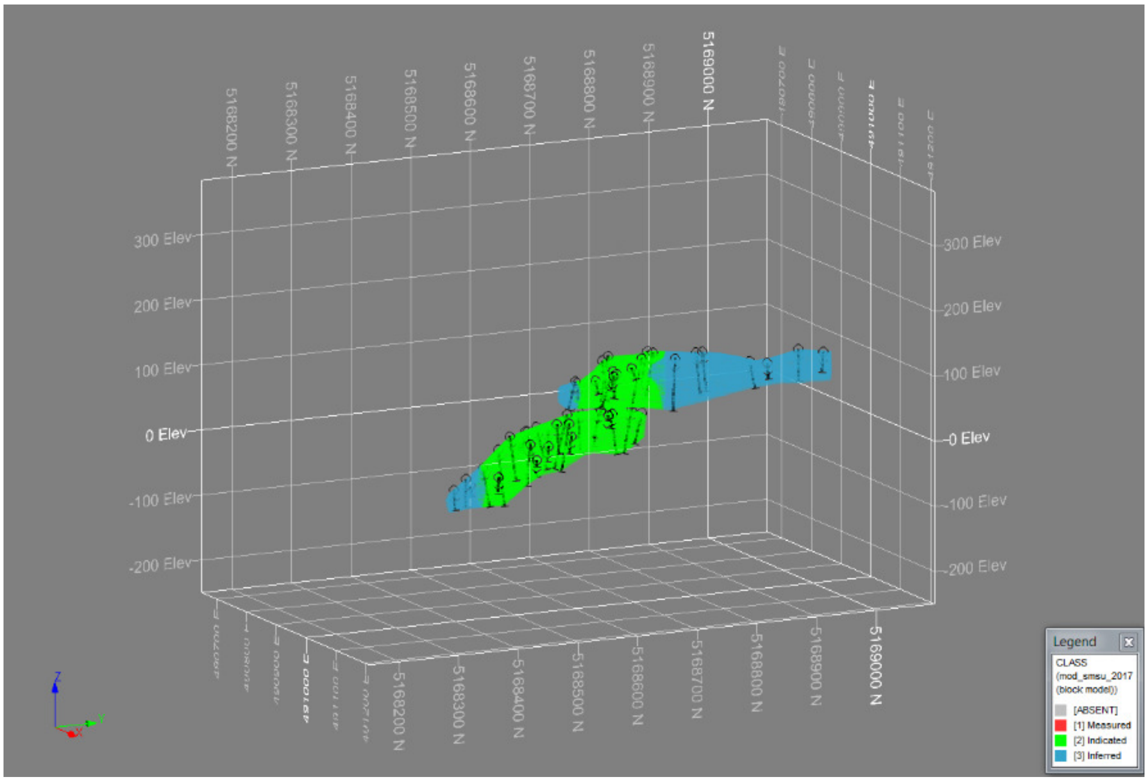


Figure 14-14: SMSU Resource Classification (Oblique View Facing NW)

Table 14-12 summarizes the data density statistics by classification and domain.

Table 14-12: Data Density Statistics

Domain	Mineral Resource Classification	Global Model Tonnage (t)	# of Holes	# of Composite Samples	Tonnes per Hole	Tonnes per Composite Samples
Lower SMSU	Indicated	2,431,358	24	772	101,307	3,149
	Inferred	171,415	3	60	57,138	2,857
Upper SMSU	Indicated	1,354,654	12	370	112,888	3,661
	Inferred	1,627,067	8	277	203,383	5,874
MSU	Inferred	571,612	24	188	23,817	3,040
138 Zone	Inferred	4,936,837	14	1,052	352,631	4,693

The number of blocks estimated in each of the search volumes was reviewed to ensure that the proportion of cells estimated for each was relatively consistent with the spacing of the drill hole data and the classification assigned to the model. 75% of the blocks in the Lower SMSU and 71% in the Upper SMSU were estimated within the first search volume while the MSU and 138 Zone were 5% and 28% respectively as listed in Table 14-13. All the 138 Zone resources are classified as Inferred Resource due to average drill spacing being greater than 25 m and in the case of the MSU, even though tonnes per composite and tonnes per hole are similar to Indicated Resource in the SMSU, there is much greater geological complexity and uncertainty of geometry which will require more detailed drilling to account for.

Table 14-13: Summary of Tonnes per Search Volume

Domain	% 1st	% 2nd
Lower SMSU	75%	24%
Upper SMSU	71%	28%
MSU	5%	47%
138 Zone	28%	70%

14.8 Block Model Validation

The model validation process included a visual comparison of block and composite grades in plan and section, along with a global comparison of mean grades and swath plots. Block grades were visually compared to the drill hole composite data in all domains to ensure agreement. No material grade bias issues were identified and the block grades compared well to the composite data as demonstrated in Figure 14-15 and Figure 14-16. The bimodal distribution observed in the SMSU domain was found to be well represented in the block model.

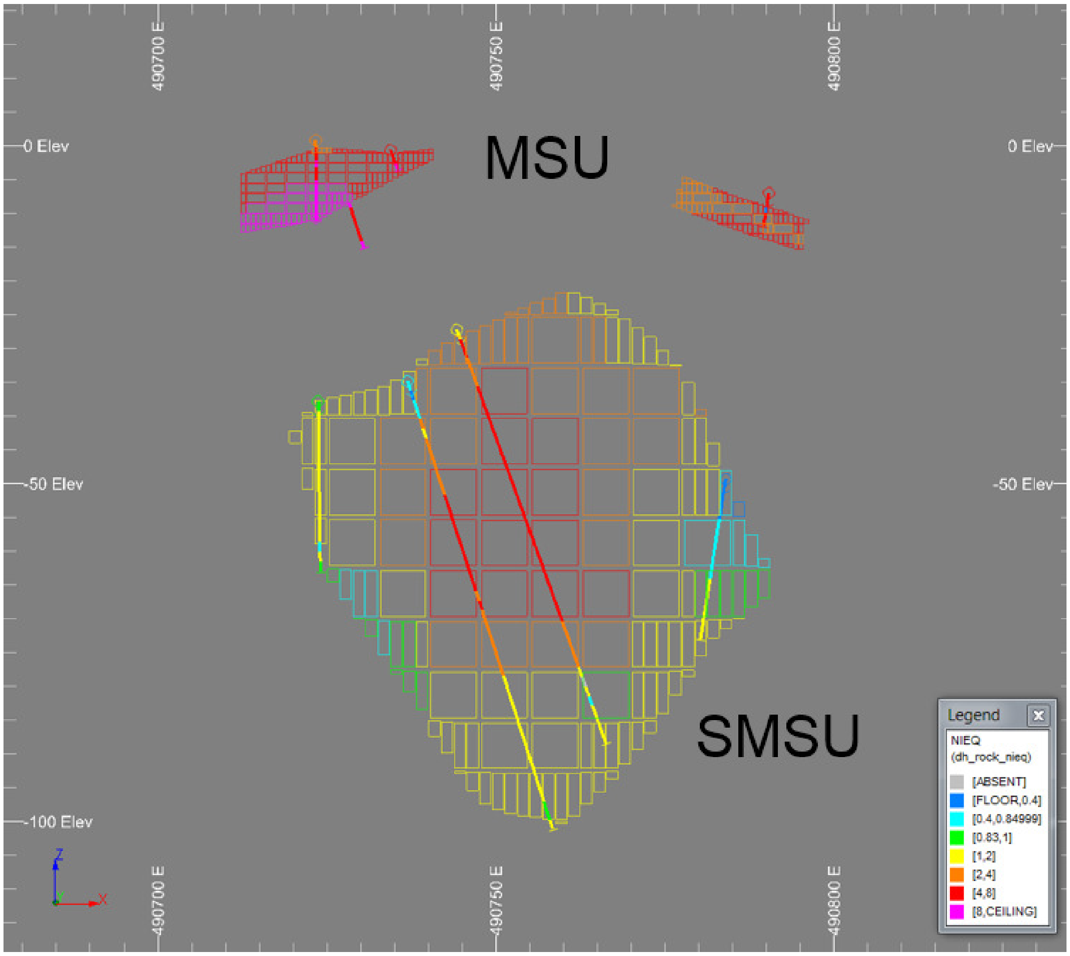


Figure 14-15: Lower SMSU and MSU Domains – E-W Section 5168660N (Facing N)

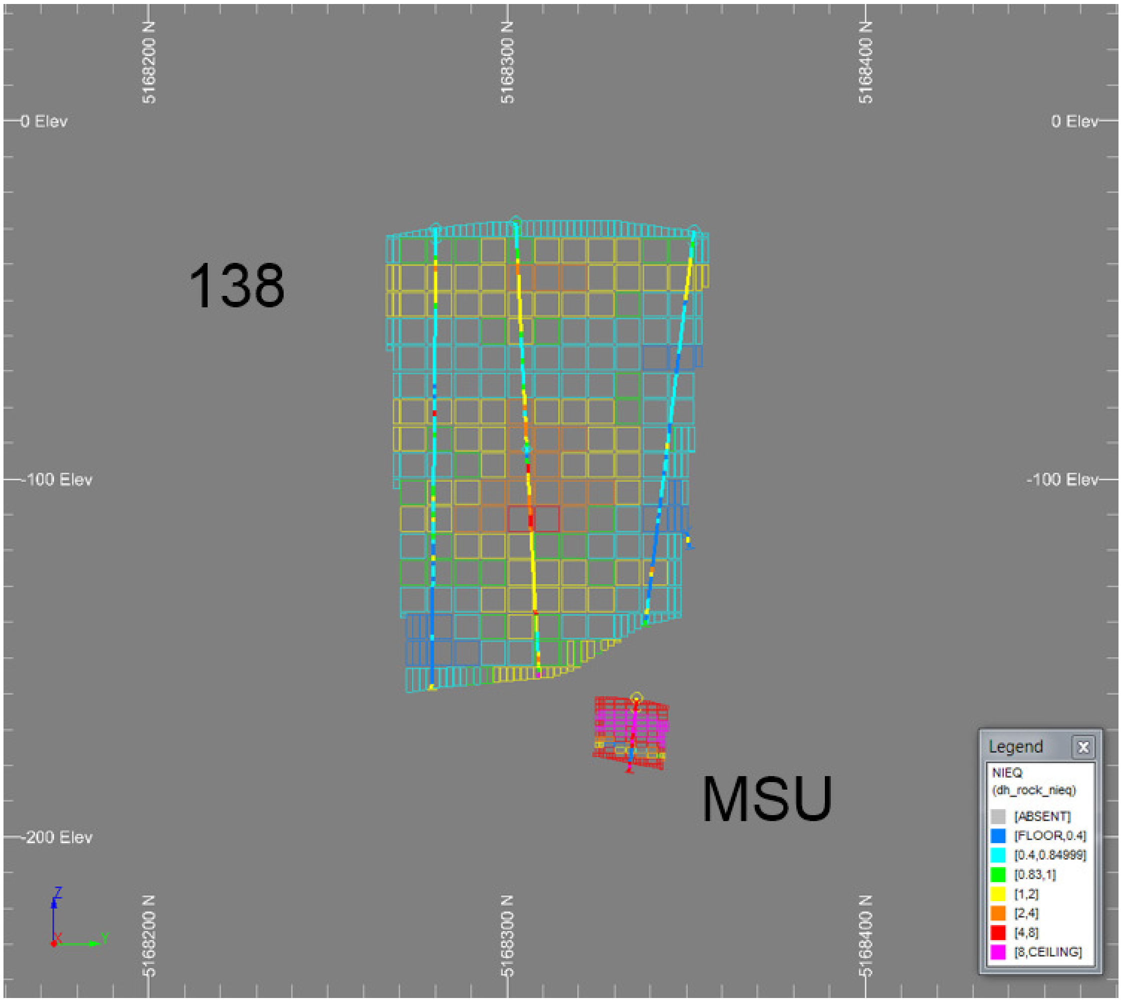


Figure 14-16: 138 Zone Domain NS Section 491000E (Facing W)

Global statistical comparisons between the composite samples, NN estimates and the final estimates (OK or inverse distance (ID)) for each metal were compared to assess global bias, where the NN model estimates represent de-clustered composite data. Clustering of the drill hole data can result in differences between the global means of the composites and NN estimates. Similar global means of the NN and OK estimates would suggest that there is no global grade bias in the model. The results summarized in Table 14-14 indicate that no significant grade bias was found in the block model.

Table 14-14: Statistical Comparison of Global Mean Grades

Field	Source	Lower SMSU	Upper SMSU	MSU	138 Zone
		Mean	Mean	Mean	Mean
Ni	Composites	1.68	1.02	5.53	0.63
	NN Model	1.96	1.07	5.80	0.71
	Final Model	1.91	1.05	5.85	0.70
Cu	Composites	0.93	0.62	2.41	0.46
	NN Model	1.04	0.63	2.44	0.52
	Final Model	1.01	0.62	2.46	0.52
Co	Composites	0.04	0.03	0.11	0.021
	NN Model	0.05	0.03	0.12	0.022
	Final Model	0.05	0.03	0.12	0.022
Pt	Composites	0.57	0.16	0.49	0.17
	NN Model	0.55	0.16	0.53	0.19
	Final Model	0.54	0.16	0.51	0.18
Pd	Composites	0.35	0.10	0.68	0.10
	NN Model	0.34	0.10	0.68	0.12
	Final Model	0.33	0.10	0.67	0.12
Au	Composites	0.25	0.10	0.29	0.11
	NN Model	0.24	0.10	0.27	0.12
	Final Model	0.24	0.10	0.25	0.12

A series of swath plots of Ni grades was generated from slices throughout each domain model and are presented in Figure 14-17 to Figure 14-19. The swath plots compare the model grades to the de-clustered composite grades to identify local grade bias in the model. Review of these swath plots did not identify any bias in the model that is material to the Mineral Resource estimate as there was general agreement between the de-clustered composites (NN model) and the final model grades.

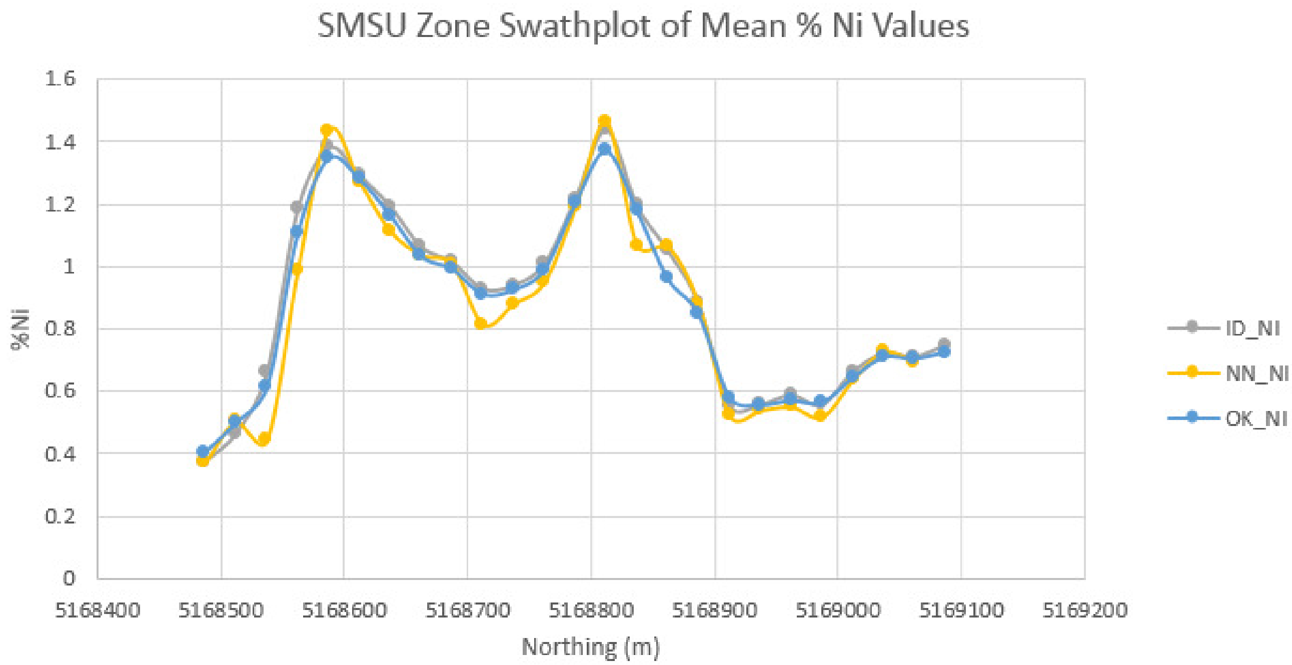


Figure 14-17: SMSU Zone Swath Plot of Mean % Ni Values for NN, inverse power distance (IPD) and OK

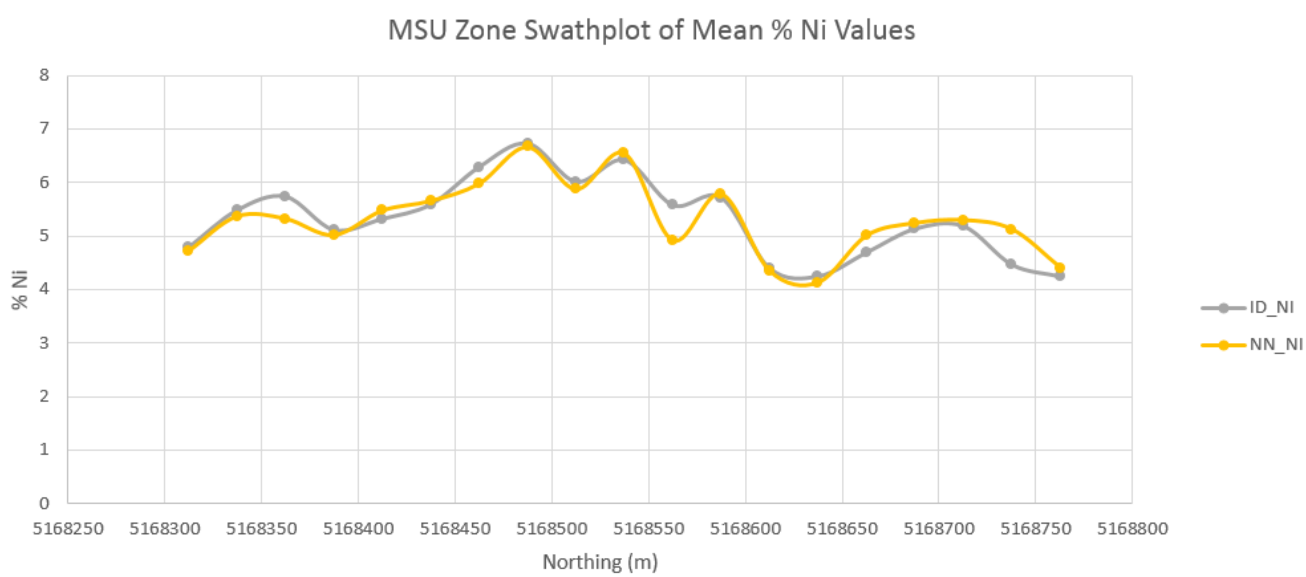


Figure 14-18: MSU Zone Swath Plot of Mean % Ni Values for NN and IPD

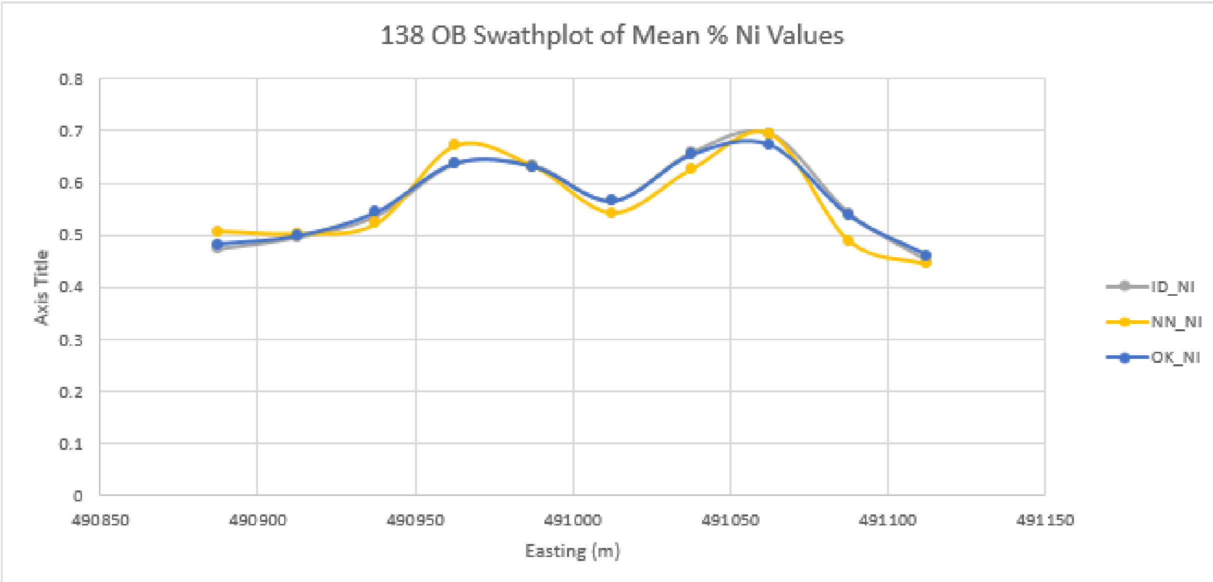


Figure 14-19: 138 Zone Swath Plot of Mean % Ni Values for NN, IPD and OK

14.8.1 **Smoothing Assessment**

Smoothing (i.e. spreading, blending, averaging) of estimated grades can occur due to estimation processes such as compositing samples, linear interpolation methods such as OK and ID, along with various other estimation parameters such search distances and the number of samples used in the estimate. A certain degree of smoothing is expected due to the change of support size from core sized samples to large mining blocks ex. 7.5 cubic metres (m^3) (SMSU, 138 Zones). However, it is also common to see higher smoothing than expected which is an issue when reporting resources above a mining cut-off as the overly smoothed distribution could result in resource tonnages being overestimated and grades being underestimated.

Smoothing ratios were calculated for %Ni in the SMSU and 138 Zone, as stated in Table 14-15, based on the ratio between the theoretical model variance and actual model variance, where the theoretical variance is calculated based on the sum of the variance inside the block and variance between blocks using such parameters as the variogram model, block size and F Function. A smoothing ratio of 1 would represent the ideal scenario where the expected variance equals the model variance and ratios between 0.8 to 1.2 are within acceptable tolerances and would not require any corrective actions. Ratios less than 0.8 are considered "under-smoothed" (low tonnes and high-grade) and over 1.2 are considered "over-smoothed" (high tonnes and low-grade) and would require corrective actions as the proportion of tonnes and grade above the selective mining cut-off would not be representative. Corrective actions would include options such as adjusting various estimation parameters or conducting a variance correction. Smoothing ratios were not calculated for the MSU as variograms were not modelled.

Table 14-15: Summary of Smoothing Ratios

Domain	Smoothing Ratio
Upper SMSU	1.12
Lower SMSU	1.14
138 Zone	2.02

The smoothing ratio assessment indicates a low degree of smoothing in the Upper and Lower SMSU and a moderate amount of smoothing in the 138 Zone. Smoothing in the SMSU was within acceptable tolerances and was therefore not corrected. A log normal smoothing correction was applied to the 138 Zone to correct the over-smoothed grade distribution. The correction results in an increase or decrease of grades relative to the mean grade to achieve the expected variance (i.e. grades below the mean are reduced, grades above the mean are increased).

14.9 Cut-off Grade

The cut-off grade, provided by Talon for this Mineral Resource estimate is a 0.83% NiEq. Table 14-16 lists the long-term metal prices and recovery assumptions used in the calculation of the NiEq cut-off that were provided by Talon.

Table 14-16: Talon Long Term Metal Price and Recovery Assumptions

Metal	Recovery	Price
Ni	66%	US$8.00/lb
Cu	85%	US$3.00/lb
Co	50%	US$12.00/lb
Pt	50%	US$1,300/oz
Pd	50%	US$700/oz
Au	50%	US$1,200/oz

Based on the above metal price assumptions, the NiEq resource values were defined using the following formula:

- NiEq% = Ni%+ Cu% x $3.00/$8.00 + Co% x $12.00/$8.00 + Pt [g/t]/31.103 x $1,300/$8.00/22.04 + Pd [g/t]/31.103 x $700/$8.00/22.04 + Au [g/t]/31.103 x $1,200/$8.00/22.04

Talon's long-term metal price assumptions are based on the average metal price forecast from a number of recognized financial institutions from North America and Europe.

Operating expense (OPEX) costs were estimated for bulk underground mining as summarized in Table 14-17 and appear to be within industry norms.

Table 14-17: Summary of OPEX Assumptions

OPEX	US$/tonne
Mining	$64.00
Milling	$22.00
General & Administrative (G&A)	$16.00
TOTAL	**$102.00**

14.10 Mineral Resource Statements

The Mineral Resource estimate for the Tamarack North Project is reported in accordance with NI 43-101 and has been estimated in conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines.

Mineral Resources are not mineral reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource will be converted into mineral reserve.

Inferred Mineral Resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as mineral reserves.

The Mineral Resource estimate was completed by Brian Thomas, P.Geo., an independent QP as defined in NI 43-101 with senior review provided by Paul Palmer, P.Geo., P.Eng. The effective date of this Mineral Resource estimate is February 15, 2018.

The Mineral Resources are reported at a NiEq cut-off of 0.83%, while other cut-offs are listed to demonstrate tonnage and grade sensitivities. The resources reported are based on a “blocks above cut-off” basis but were examined visually and found to have good continuity.

Table 14-18 reports the Indicated and Inferred Mineral Resources for the Tamarack North Project and Table 14-19 summarizes the sensitivities of other cut-offs.

Table 14-18: Tamarack North Project 2018 Mineral Resource Estimate

Domain	Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	Calc NiEq (%)
SMSU	Indicated	3,639	1.83	0.99	0.05	0.42	0.26	0.2	2.45
Total	**Indicated**	**3,639**	**1.83**	**0.99**	**0.05**	**0.42**	**0.26**	**0.2**	**2.45**
SMSU	Inferred	1,107	0.9	0.55	0.03	0.22	0.14	0.12	1.25
MSU	Inferred	570	5.86	2.46	0.12	0.68	0.51	0.25	7.24
138 Zone	Inferred	2,705	0.95	0.74	0.03	0.23	0.13	0.16	1.38
Total	**Inferred**	**4,382**	**1.58**	**0.92**	**0.04**	**0.29**	**0.18**	**0.16**	**2.11**

Notes:
All resources reported at a 0.83% NiEq cut-off.
No modifying factors been applied to the estimates.
Tonnage estimates are rounded to the nearest 1,000 tonnes.
Metallurgical recovery factored in to the reporting cut-off.

Table 14-19: Tamarack North Project 2018 Resource Sensitivities

NiEq Cut-Off (%)	Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
0.7	Indicated	3,711	1.81	0.98	0.05	0.42	0.26	0.20	2.43
0.7	Inferred	5,263	1.40	0.82	0.04	0.26	0.17	0.15	1.88
0.83	**Indicated**	**3,639**	**1.83**	**0.99**	**0.05**	**0.42**	**0.26**	**0.20**	**2.45**
0.83	**Inferred**	**4,382**	**1.58**	**0.92**	**0.04**	**0.29**	**0.18**	**0.16**	**2.11**
0.9	Indicated	3,588	1.85	1.00	0.05	0.42	0.26	0.20	2.48
0.9	Inferred	3,914	1.70	0.98	0.04	0.30	0.19	0.17	2.26
1	Indicated	3,470	1.89	1.02	0.05	0.43	0.27	0.21	2.53
1	Inferred	3,336	1.88	1.06	0.05	0.32	0.21	0.18	2.48

Notes:
No modifying factors been applied to the estimates.
Tonnage estimates are rounded to the nearest 1,000 tonnes.
Metallurgical recovery factored in to the reporting cut-off.
Bold represents the official resource.

Table 14-20 summarizes the changes from previously reported Mineral Resource estimates for tonnage and Ni and Cu.

Table 14-20: Summary of Resource Changes

Domain	Classification	2015			2018			Difference		
		Tonnes (000)	Ni (%)	Cu (%)	Tonnes (000)	Ni (%)	Cu (%)	Tonnes (000)	Ni (%)	Cu (%)
SMSU	Indicated	3,751	1.81	1.00	3,639	1.83	0.99	-112	0.02	-0.01
Total	**Indicated**	**3,751**	**1.81**	**1.00**	**3,639**	**1.83**	**0.99**	**-112**	**0.02**	**-0.01**
SMSU	Inferred	949	1.12	0.62	1,107	0.9	0.55	158	-0.22	-0.07
MSU	Inferred	422	6.00	2.48	570	5.86	2.46	148	-0.14	-0.02
138 Zone	Inferred	2,012	0.95	0.78	2,705	0.95	0.74	693	0	-0.04
Total	**Inferred**	**3,383**	**1.63**	**0.94**	**4,382**	**1.58**	**0.92**	**999**	**-0.05**	**-0.02**

The difference in the Mineral Resource estimate largely reflects the change in domain volumes resulting from new drill holes added to each mineral domain as well as a slightly lower reporting cut-off value. The MSU mineralization was infilled (by drilling) down plunge resulting in a large increase to reflect the additional continuity of the mineralization. New holes in the SMSU provided increased definition resulting in a slight reduction of tonnage and increased grade, whereas new drill holes in the 138 Zone expanded the footprint resulting in an increase of tonnage.

Golder is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or any other potential factors that could materially impact the Tamarack North Project Mineral Resource estimate provided in this PEA. The resource is located in designated wetlands but this is not expected to affect future permitting.

15. MINERAL RESERVE ESTIMATE

Not applicable.

16. MINING OPERATIONS

16.1 Mining Methods

The Tamarack deposit will be mined using underground mining methods.

Mine development and operation costs, for purposes of this PEA, assume contractor rates. Different underground mining methods will be utilized for the SMSU (consisting of an Upper and Lower SMSU) and the MSU, with each described in the following sections. The mine layout and different mining areas are shown in Figure 16-1 below.

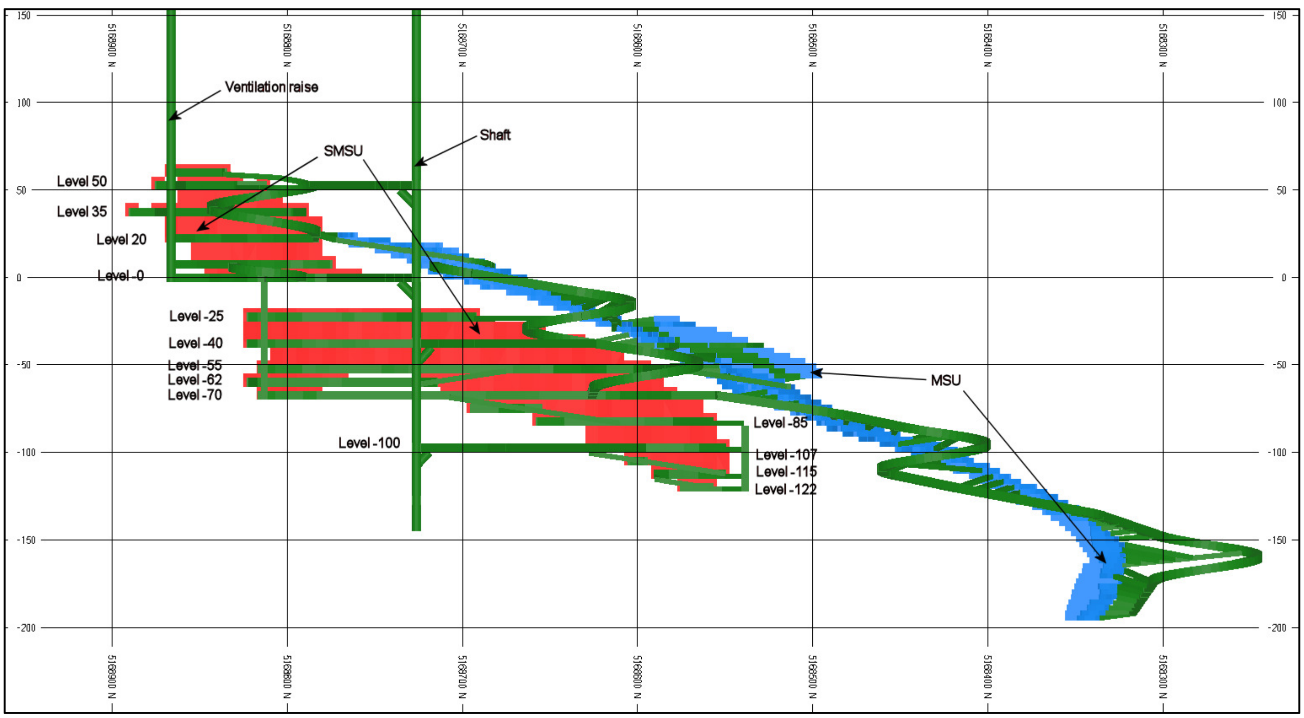


Figure 16-1: Mine Development and Production Areas

16.1.1 Mining Method for the Upper and Lower SMSU

The mining method proposed for the Upper and Lower SMSU is transverse long hole open stoping with cemented paste backfill. Figure 16-2 shows a typical layout for a primary-secondary transverse long hole open stoping method with paste backfill.

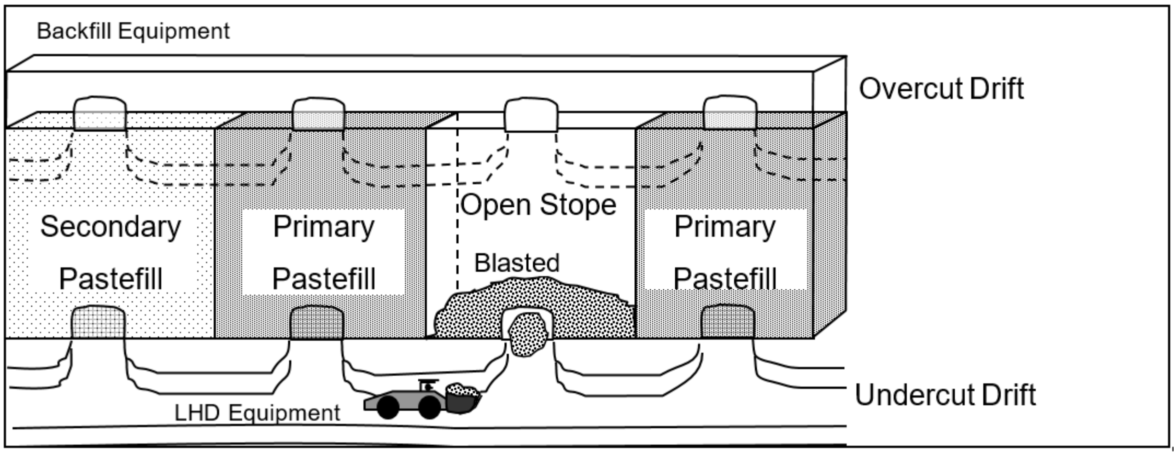


Figure 16-2: Long Section View of Primary-Secondary Transverse Long Hole Open Stoping Method with Paste Backfill

For the Upper and Lower SMSU ore bodies, mining will be progressed by means of the following steps:

- Levels [millilitre (mL) 50] and [mL0] will be developed away from the shaft to reach the Upper SMSU. Levels [mL-40] and [mL-100] will be developed to reach the Lower SMSU;
- A system of ramps will be developed from each level, parallel to each of the Upper and Lower SMSU ore bodies. Following this, crosscuts will be developed into the orebodies, an overcut (for drilling and blasting) and an undercut (for loading and mucking) for each stope;
- Stopes will be mined in a “primary-secondary” sequence from the bottom levels and advancing upwards;
- A vertical slot parallel with the SMSU is developed from where fan drilling will be used to blast the trough. Blast holes are then drilled from an overcut therefore the overcut will be silled out prior to blasting. After blasting, a remote-controlled load-haul-dump (LHD) vehicle will be used to remove the blasted material from the stope via the

undercut and into trucks for haulage to the re-muck bays, ore pass, ore-bin and skip. Once the stope is fully mucked out, cemented paste backfill will be placed from the overcut;

- Once the cemented paste backfill has cured in two primary stopes on a level, the secondary stope in between can be mined out. Once the secondary stopes have been mined out and backfilled, the stope above the initial primary can then be mined. This pattern continues throughout the ore body, advancing in a vertical chevron style pattern;
- In order to support this mining method, a cemented paste backfill (see Section 16.2 Paste Backfill System) of adequate strength is required. The hardness of the secondary stope backfill, could be less than that of the primary stopes. This will reduce cement use and thus reduce costs. In addition, any waste rock from development will be placed in secondary stopes to reduce cost and the need to hoist it to surface for storage.

Figure 16-3 shows the proposed Lower and Upper SMSU stopes, and includes a proposed stope development schedule by year.

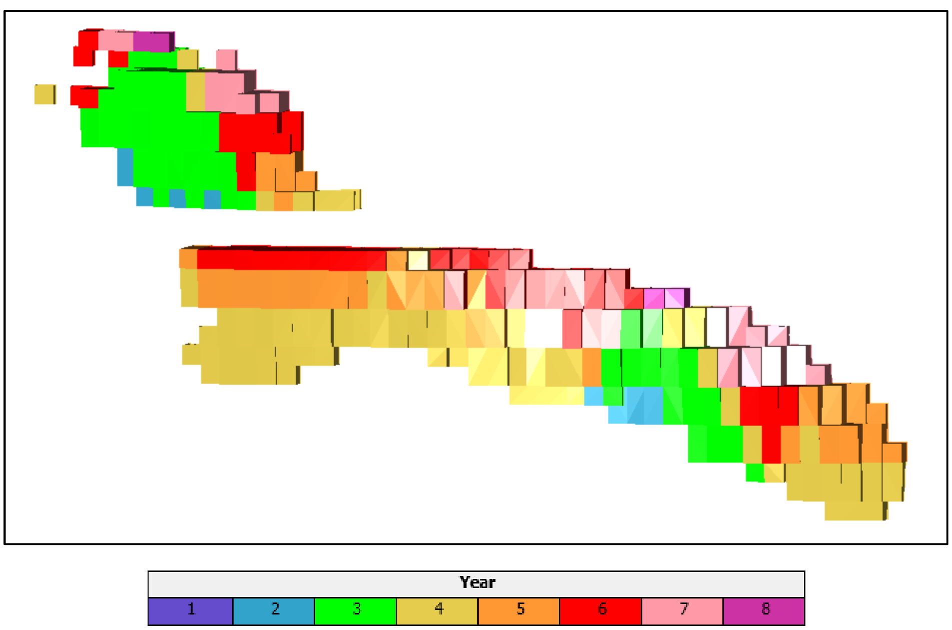


Figure 16-3: 3D View of Upper and Lower SMSU Stopes: Yearly Development Schedule

16.1.2 Mining Method for the MSU

The mining method proposed for the MSU zone is overhand, transverse drift-and-fill with a cemented paste backfill. Figure 16-4 shows a typical overhand, transverse drift-and-fill mining arrangement.

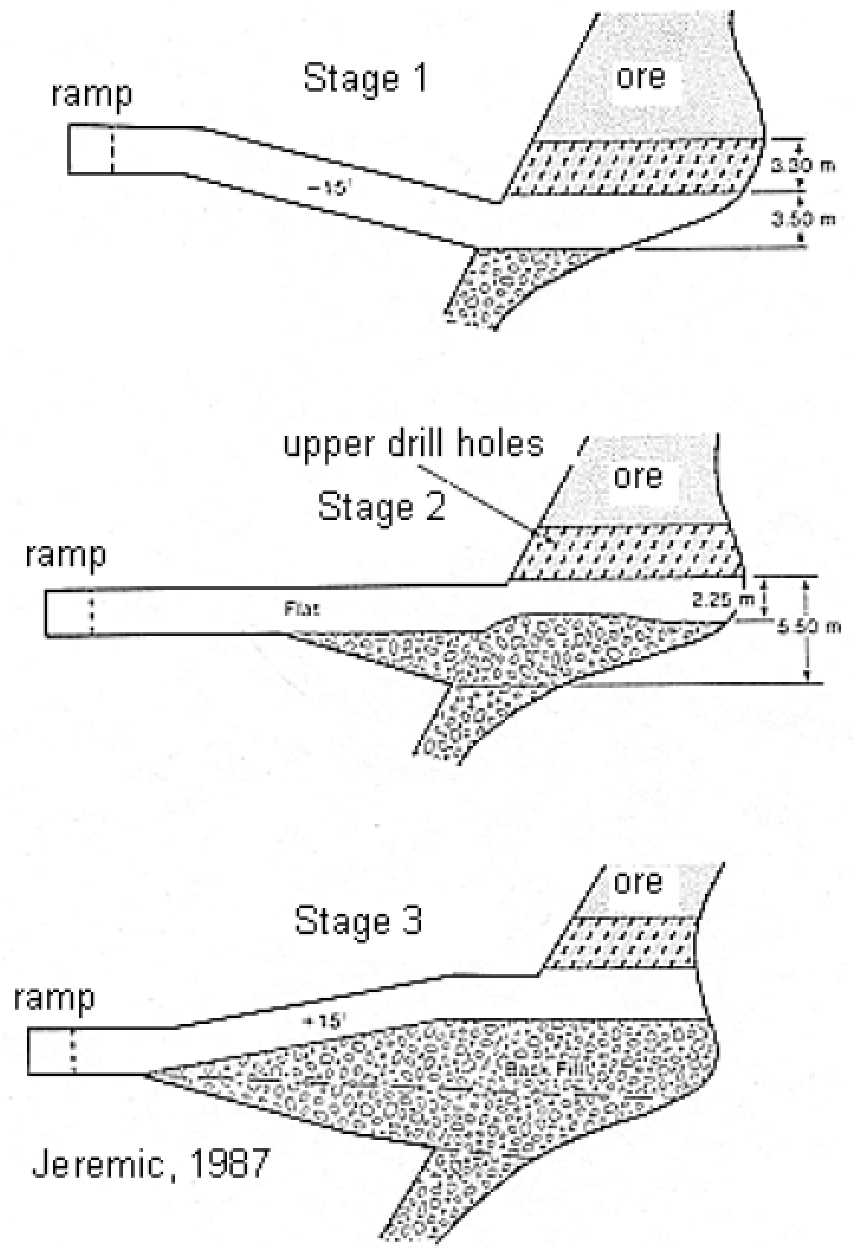


Figure 16-4:Illustrative Stages of a Transverse Drift and Fill Mining Method

Mining will be progressed by means of the following steps:

- From level [mL-40], a main access ramp parallel to the MSU ore body will be developed;
- From this main access ramp, short crosscuts will be developed towards the MSU ore body;
- The crosscut, a short, bottom, slashing ramp will be developed at a gradient not exceeding 20% downwards;
- Once the ore body is reached, the ramp will be levelled out and development continues at a flat gradient towards the hanging wall of the ore body in a transverse direction. This is the first/bottom stope. Excavation of each stope will be done with conventional drill-and-blast mining techniques;
- The bottom stope is then filled with cemented paste backfill up to the short crosscut, tight filling to the back of the stope;
- Depending on the vertical height of the MSU ore body a second short ramp will be developed from the same crosscut followed by a second stope above the first bottom stope. Each backfilled bottom stope will serve as the floor for the one above. Depending on the vertical height of the MSU ore body, a final third upwards slashing ramp with a maximum inclination of 20% will be developed (from the same crosscut), followed by a third stope above the second stope;
- As stopes will be "stacked" on top of and adjacent to one another, a cemented paste backfill (see Section 16.2 Paste Backfill System) of adequate strength will be required;
- Any remaining MSU, above the third stope, will be accessed from a different ramp and set of crosscuts, following the same mining methodology;
- Stopes will be mined simultaneously on a "primary-secondary" sequence, taking into account cemented paste backfill curing times. The transverse, overhand drift-and-fill mining method provides the necessary scheduling flexibility required for maximizing high-grade ore recoveries. This method allows for quick ramp up once development is in place.

Figure 16-5 shows the proposed drift-and-fill stopes for the MSU, and includes a yearly development schedule.

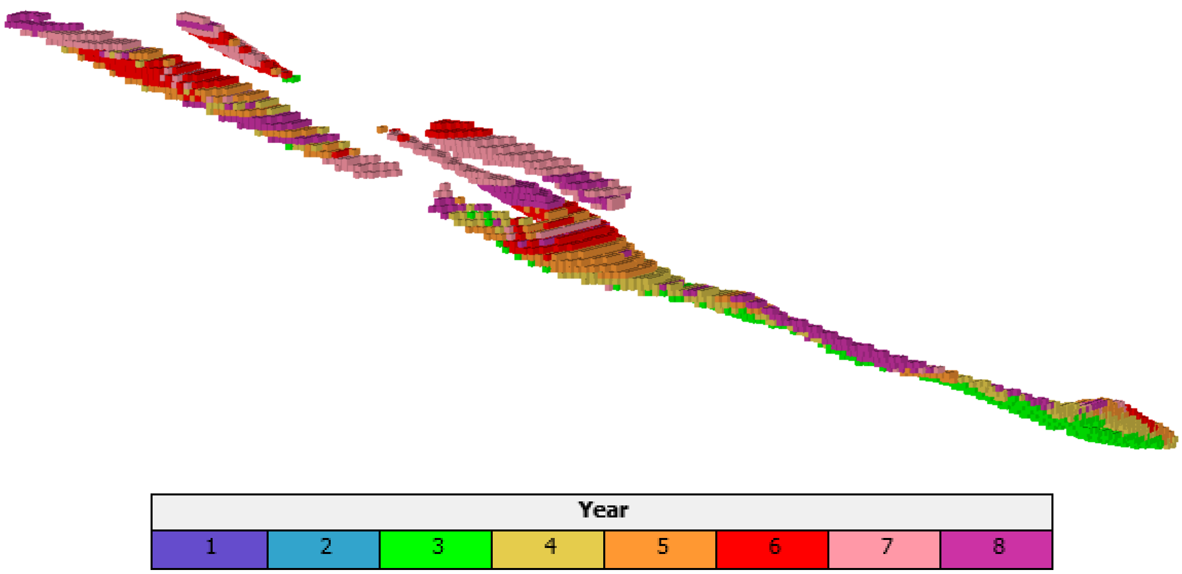


Figure 16-5: 3D View of MSU Drift-and-Fill Stopes Including Stope Development by Year

16.2 Paste Backfill System

Paste backfill will be used for the majority of the backfilling requirements for the Tamarack North Project for ground stability, increased ore recovery, and to minimize the quantity of tailings stored on surface. A strong backfill that includes the addition of cement is required for the primary MSU and SMSU stopes. The secondary long hole open stopes and drift-and-fill stopes can be filled with a backfill requiring less cement.

A paste plant will be constructed on surface, adjacent to the mill, that will dewater the two tailings streams produced in the mill and mix the solids with a slag-based binder and treated water (trim water) to achieve the final paste density. For more information on water treatment, refer Section 18.4.7.

The paste plant will return 100% of the HS tailings back underground, which will eliminate the need to store this material on surface. The HS tailings stream will be blended with the LS tailings stream. The blended material will have a sulphides content low enough to mitigate the potential for self-heating. A binder addition rate of 4% is assumed for this study to also mitigate any self-heating effects as well as ensure that the Unconfined Compressive Strength requirements are achieved. It is assumed for this PEA that backfill curing time will be 28 days for all stopes. Strength test work was not incorporated into this study. As there are more tailings produced than required for backfilling purposes, the excess, LS tailings that are not required to achieve the annual backfill requirements will be filtered to 85% solids and co-disposed with development rock for storage in a lined facility at surface as further described in Section 18. There will not be a requirement for a tailings dam.

An evaluation of the annual backfill requirements was completed and the paste plant nominal flow rate was set at 30 m^3/h. This results in an annual system utilization of approximately 51%, while still maintaining reasonable velocity in a 100 nominal bore (NB) (4 inch) pipeline. A nominal paste solids concentration (by mass) of 79.5% was used in hydraulic modelling of the underground.

A high-level hydraulic model of various deposition points was developed based on mine design and rheology from similar projects. The results of the hydraulic modelling predict that sufficient head is available such that, combined with a positive displacement paste pump, a paste with 79.5% solids (by mass) can be distributed to all of the SMSU zone, and the upper portion of the MSU zone. The furthest extents of the MSU zone can be reached using a surface pump, however a higher slump (lower density) paste is required. Rheology test work will be required in subsequent stages of the project to confirm the results of this preliminary hydraulic modelling, as well as to confirm the pumping requirements.

The underground paste distribution system will include two sets (four holes in total) of surface-to-underground BHs to supply paste to the underground workings. Each BH will be cased with a 100 NB unlined steel pipe. Each set of BHs will have one duty and one standby BH. One set of BHs will target the mL50 level shaft access, providing access to the majority of the SMSU area. The other set of BHs will be drilled to access the mL-40 shaft access, to distribute fill to the MSU zone.

Once underground, the paste fill will travel through a network of pipelines to reach the location where paste is needed. Two dedicated interlevel BHs (one duty and one standby) will be drilled to connect levels or bypass ramp development as required.

The mass balance is shown in Table 16-1 below:

Table 16-1: Mass balance of mine production and mill production

MINE PRODUCTION										
Year		Unit	Total	2	3	4	5	6	7	8
MSU		t	505,937	-	59,953	94,193	94,137	94,770	89,458	73,426
SMSU		t	1,055,162	20,622	277,707	241,127	238,863	203,044	70,427	3,373
Ore Development		t	826,570	65,267	156,040	165,123	156,816	187,046	86,443	9,835
		t								
		t								
Total Mine Production		t	2,387,670	85,889	493,700	500,443	489,816	484,860	246,328	86,634
Production Rate		t/day		239	1,371	1,390	1,361	1,347	684	241
BACKFILL VOLUME										
Year		Unit	Total	2	3	4	5	6	7	8
Total Voids Opened - Stope Development (From DRA)		m³	691,747	27,051	142,702	148,000	142,979	141,528	66,702	22,784
Void Replacement Factor- Paste				**90.0%**	**90.0%**	**90.0%**	**90.0%**	**90.0%**	**90.0%**	**90.0%**
Total Paste Volume		m³	622,572	24,346	128,432	133,200	128,681	127,375	60,032	20,506
		%voids	90.0%							
Total Backfill Volume		**m³**	**622,572**	**24,346**	**128,432**	**133,200**	**128,681**	**127,375**	**60,032**	**20,506**
TAILINGS AND WASTE ROCK										
Year		Unit	Total	2	3	4	5	6	7	8
Tailings and Waste Rock Available										
Total Mill Production		t	2,387,670	**85,889**	**493,700**	**500,443**	**489,816**	**484,860**	**246,328**	**86,634**
Ni Concentrate Produced	18.1%	dry t	432,168	15,546	89,360	90,580	88,657	87,760	44,585	15,681
Cu Concentrate Produced	4.2%	dry t	100,282	3,607	20,735	21,019	20,572	20,364	10,346	3,639
Tailings Available for Backfill - Low Sulphide	54.9%	dry t	1,310,831	47,153	271,041	274,743	268,909	266,188	135,234	47,562
Tailings Available for Backfill - High Sulphide	22.8%	dry t	544,389	19,583	112,563	114,101	111,678	110,548	56,163	19,752
Waste Rock Produced		dry t								
Tailings Required - Pastefill										
Total Backfill Volume		m³	622,572	24,346	128,432	133,200	128,681	127,375	60,032	20,506
Total Backfill Water		m³	255,755	10,001	52,760	54,719	52,863	52,326	24,661	8,424
Total Backfill Dry Tonnes		dry t	1,077,509	42,136	222,282	230,535	222,714	220,453	103,900	35,490
Binder	4.0%	dry t	43,100	1,685	8,891	9,221	8,909	8,818	4,156	1,420
Total HS Tailings Required		dry t	544,389	19,583	112,563	114,101	111,678	110,548	56,163	19,752
Total LS Tailings Required		dry t	490,020	20,868	100,827	107,212	102,127	101,087	43,581	14,318
HS Tailings Amount in Tailings			51%	46%	51%	49%	50%	50%	54%	56%
LS Tailings Amount in Tailings			45%	50%	45%	47%	46%	46%	42%	40%
Total Tailngs Required (H.S and L.S)		t	1,034,409	40,451	213,391	221,313	213,805	211,635	99,744	34,070
Total Binder Required		t	43,100	1,685	8,891	9,221	8,909	8,818	4,156	1,420
Total L.S Tailings Surplus/Defict (to TSF)		t	820,810	26,285	170,214	167,531	166,782	165,101	91,653	33,244

16.3 Geotechnical Parameters

All geotechnical data was obtained from previous study work completed by Golder over several phases from 2008 to 2014. No geotechnical work was completed as part of this PEA study phase.

Table 16-2 below shows a summary of the rock types that may be encountered during the development and stoping process.

Table 16-2: Summary of Rock Mass Rating (RMR) data (Golder and Associates, 2008)

Rock Type	RMR'$_{76}$				
	Average	Standard Deviation	Typical[1]	Range[2]	
				From	To
CGO	62	10	69	52	72
FGO	66	9	66	57	75
SED	65	8	59 to 74	57	73
SMSU	61	9	66	52	70
MSU	70	11	72 to 77	59	81
Serpentinized Zones	41	13	34	28	54
Checked by: RPB/JJT					

[1] Typical RMR'$_{76}$ is based on individual RMR'$_{76}$ parameter assessment
[2] Range is based on one standard deviation either side of the mean (average)

The table shows that most rock can be classified as "good" (RMR' = 61 to 80) according to the Beniawski (1976) RMR system. Serpentinized zones are classified as "poor" (RMR' = 41 to 60).

It is not expected that the dimensions chosen for the excavations should provide any major geotechnical complications.

The advance rates proposed should be achievable with "poor" ground not exceeding 15%.

It is anticipated that all development headings (walls and back) will be bolted and screened, and shotcrete added in areas of poor ground.

16.4 Hydrological Parameters – Bedrock

In 2008 interval-specific fractures were logged in four drill holes that intercepted the SMSU, MSU, CGO, FGO and Sediments (drill holes 08TK0048, 49,50 and 08TK0054) to determine the frequency of features that will be water producing. The geophysical techniques included caliper, full wave form sonic, fluid temperature, fluid resistivity, and optical BH imager logs. A total of 10 features with a hydraulic conductivity greater than $1x10^{-6}$ cm/s were found along the total bore length of 2161 m, or an average feature frequency of 1 per 216 m of length, however it was concluded that only 50% of these features would be independent.

In addition, hydraulic testing of discrete intervals using packers were performed to determine the potential volume of water that each of the independent features with a hydraulic conductivity greater than $1x10^{-6}$ cm/s could be produce. A preliminary estimate for the inflow from these independent bedrock features to a mine working was calculated at 9.9 gallons per minute (gpm).

The expected frequency of independent features was multiplied by the proposed development meters (refer Table 16-10) and the calculated water production for each feature. It was assumed that none of the water producing features will be sealed. The result is shown in Table 16-3 below.

Table 16-3: Potential Mine Water Production by Year

Production Year	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Cumulative development meters	4,660	9,953	14,774	17,678	20,608	22,689	22,905
Estimated number of independent, water producing features	11	23	34	41	48	53	53
Potential production (from development) US gpm	109	228	337	406	475	525	525

These preliminary calculations do not take into account sealing off of dilated fracture zones.

16.5 Hydrological Parameters –Surficial

In 2008 a pump test to evaluate the hydraulic characteristics of the water bearing sand was designed and implemented to the NE of the site layout area. The pump test required the design and installation of a pumping well and two monitoring wells. The pump test data were best matched to a theoretical model for a leaky confined aquifer. The test data yielded an average transmissivity of 165 ft squared per day and a hydraulic conductivity of 2.4 ft per day. The average value of hydraulic conductivity was input to an analytical groundwater inflow model to estimate the seepage of groundwater into an idealized excavation. Output from the model bounded the average value with reasonable upper and lower values.

The results of the model indicate initial average seepage into an idealized excavation (circular excavation with a radius of 100 ft (30.5 m) may be above 224 gpm (1,220 m^3 per day)). However, an average inflow of 68 gpm (374 m^3 per day) is estimated for periods after dewatering effects have stabilized.

16.6 Design and Operating Parameters

16.6.1 Production Rate

The production rate selected for the Tamarack North Project is 1,390 tpd of ore. This production rate is based on a combination of a modification of Taylor's formula (from Long's study done in 2009 the resultant rate is 1,150 tpd) as well as the vertical dropdown method which showed a 1,390 tpd production rate would be sustainable for this type of deposit.

The vertical advance rate is based on study work completed by McCarthy (1993). The study work used results from several mining operations to ascertain what vertical rate of development can be achieved successfully. For a mining operation with a production rate of less than 1 million tonnes per annum (Mtpa), a vertical advance rate of between 30 to 40 m per year should be used. Using 40 m per year, the tonnage rate should be around 1,250 tpd.

As there are two different mining areas, it was decided that the vertical advance rates calculated above, could be increased.

The production rate was subsequently used in the Enhanced Production Scheduler (EPS) software, which confirmed it was achievable.

16.6.2 Design and Operating Parameters

Mine design and operating criteria used for design and scheduling of the Tamarack deposit are given in Table 16-4.

Table 16-4: Mine Design Criteria

Parameter	Unit	Value
Main Ramp System		
Development Gradient	%	≤ 15
Excavated Width on Straight	m	5.00
Excavated Width on Bend	m	5.00
Bend Radius	m	≥ 30
Excavated Height	m	5.50
Road Bed Height	m	0.30
Concrete Road Bed	Yes/No	No
Shotcrete thickness	m	0.1
Final Width	m	4.80
Final Height	m	5.10
Advance Rates	m/month	120 - 150
Development overbreak	%	5
Re-muck Bays		
Length	m	12.00
Excavated Width	m	5.00
Excavated Height	m	5.50
Final Width	m	4.80
Final Height	m	5.10
Re-muck Bays Gradient	%	-15
Spacing of Re-muck Bays	m	120
Passing Bays		
Length	m	12.00

Excavated Width (overall width)	m	9.00
Excavated Height	m	5.50
Final Width	m	8.80
Final Height	m	5.10
Spacing of Passing Bays	m	360
Other Internal Development		
Development Gradient	%	0.00
Excavated Width	m	5.00
Excavated Height	m	5.00
Road Bed Height	m	0.30
Concrete Road Bed	Yes/No	No
Final Width	m	4.80
Final Height	m	4.60
Advance Rates	m/month	180 - 300
Drift and Fill		
Development Gradient	%	<20
Excavated Width	m	3.00
Excavated Height	m	3.00
Road Bed Height	m	0.00
Concrete Road Bed	Yes/No	No
Advance Rates	m/month	180 - 300
Equivalent Production Rate	tpd	170 to 280
Paste Backfill Curing Time	days	28
Dilution	%	5
Recovery	%	95
Transverse Long Hole Open Stoping		
Development Gradient	%	0.00
Excavated Width	m	7.50
Excavated Height	m	15.00
Production Rate (max)	t/month	20 000
Paste Backfill Curing Time	days	28
Dilution	%	15
Recovery	%	85
Vent Pass		
Circular/Rectangular		Circular
Diameter	m	5.20

16.7 Mine Design

Prior to performing any mine design, all block models were subject to a series of runs (estimates) using Mineable Shape Optimizer (MSO). The aim was to produce suitable mineable stopes, or targets, which would aid with the design of mine development. In simple terms, the MSO shapes would form, or contribute to, any potentially mineable reserve.

MSO relies on a series of user-input parameters. These parameters dictate the range within which MSO can generate a shape. If the parameters are too restrictive, MSO will struggle to provide many shapes; and conversely if the parameters are too loose MSO will generate many shapes but few might actually be practical from a mining point of view. Setting up various cases or runs within MSO, allows the user to inspect the resultant shapes and refine the parameters further to reach an optimal set. Several MSO runs were performed until a final, optimized set of runs were chosen for the mine design process.

16.7.1 Cut off Value/Net Smelter Return (NSR)

One of the main inputs into MSO is the cut-off value.

The cut-off used to run the MSO is different to the cut-off that was used for purposes of estimating the Mineral Resource (refer Section 14.9). Instead of applying a cut-off grade, a minimum NSR/tonne was used with separate costs per tonne for each of the SMSU and MSU:

- SMSU: US$117/tonne;
- MSU: US$157/tonne.

The NSR per tonne was calculated using the prices below:

Table 16-5: Commodity Prices Used to Calculate NSR/tonne

	Unit	Low
Ni	US$/lb	$6.75
Cu	US$/lb	$2.75
Co	US$/lb	$20.00
Pt	US$/oz	$1,100
Pd	US$/oz	$800
Au	US$/oz	$1,200

The metallurgical recovery formula was applied to the calculated result. Concentrate transportation charges, water treatment charges, royalties, smelter treatment charges and refining charges were deducted to calculate the NSR/tonne. This value was then used to determine inclusions or exclusion from the mine plan.

Due to the application of this cut-off formula most of the inferred mineral resource estimate tonnage in the SMSU and all of the tonnage in the 138 Zone have been excluded from the mine plan (refer to Figure 16-6 below).

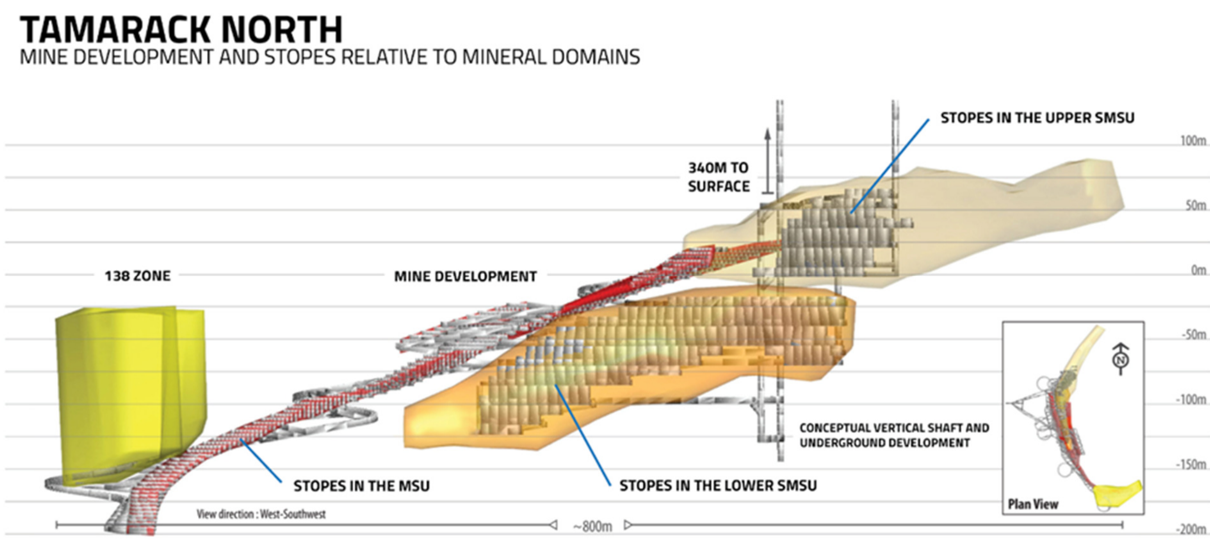


Figure 16-6: Long section (looking W) of the mine plan development and stopes in relation to the wireframes for resource domains

In addition, and as a result of the application of this cut-off formula, conservative metallurgical projections were applied. A metallurgical test program to simplify the flowsheet and reagent regime and to evaluate samples from the inferred mineral resource should be commissioned as fully described in Section 25.4. The simplified flowsheet will help to reduce the capital and operating costs for a given plant throughput and maximize the recovery of all sulphide minerals to minimize environmental liabilities. The objective of this metallurgical test program is to include most of the mineral resource estimate in the next iteration of the mine plan.

16.7.2 Other MSO Parameters

The final parameters and/or assumptions used for the Tamarack ore body are summarized in Table 16-6 below.

Table 16-6: Cut-off and MSO Parameters

MSO Parameter	Unit	SMSU		MSU	
		Value	Variance	Value	Variance
Resource Model		As supplied by Talon			
Default Density	t/m^3	2.85		2.85	
Block Density	t/m^3	Variable per each block in the resource model			
Slice Interval	M	0.5		0.5	
Default Dip	Degrees	90		90	
Default Strike	Degrees	0		0	
Section Spacing	M	7.5	5 to 10	3	3 to 6
Level Spacing	M	15	5 to 30	3	3 to 5
Maximum Waste Fraction	%	5	0 to 100	100	0 to 100
Minimum Width of Shape	M	5		5	
Maximum Length (Span) of Shape	M	100		100	
Minimum Waste Pillar Width	M	~0		~0	
Minimum Dip Angle of Shape	Degrees	85		85	
Maximum Dip Angle of Shape	Degrees	95		95	
Maximum Strike Angle of Shape	Degrees	45		45	
Maximum Angle Change	Degrees	20		20	
Maximum Side Length Ratio		1.5		1.5	

Figure 16-7 displays the basics of MSO, including nomenclature. While the user can specify dip and strike, the supplied orebody wireframes are utilized to control shape generation. If the wireframes exhibit sharp changes in either dip or strike, then the resultant MSO shapes will also exhibit such changes. This may lead to the generation of impractical mining shapes or stope blocks, however for this PEA, the MSO shapes generated are considered more than acceptable.

The MSO shapes that were developed are inclusive of internal dilution. Any external dilution that is expected has been excluded from the MSO process. Recovery and external dilution was considered in the results from the scheduling exercise.

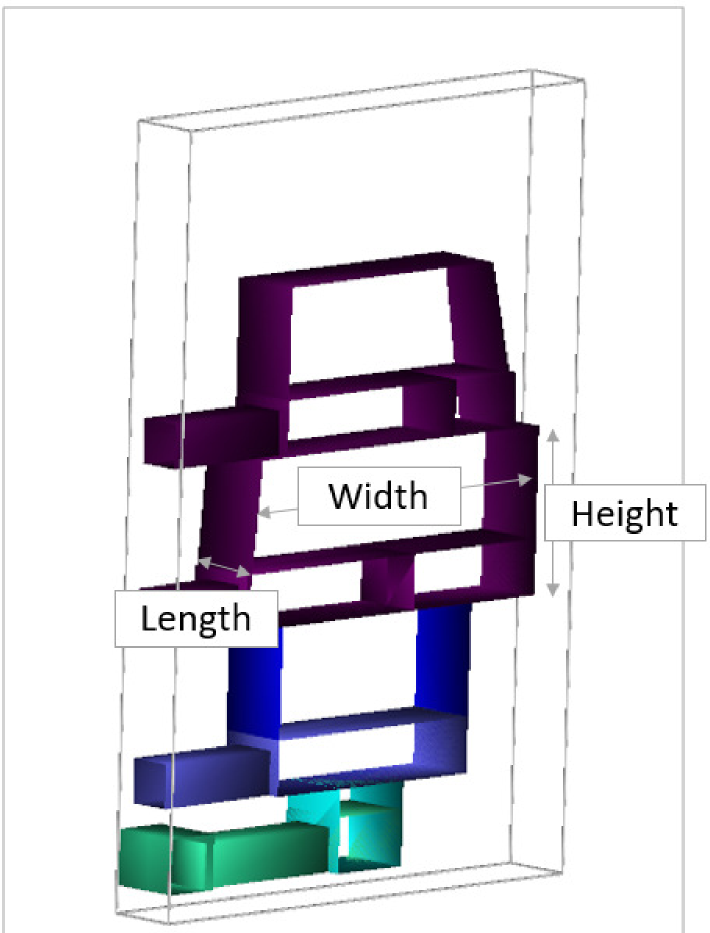


Figure 16-7: MSO Framework

Section spacing represents the size of blocks in the strike orientation. Various section length input parameters to the MSO were used and it was determined that 7.5 m section spacing fits the block model more accurately and minimized internal dilution. Level spacing is the height of the blocks created. As with the section spacing, various height input parameters were used and it was determined that 15 m would best fit the block model and minimize dilution. MSO allows the creation of sub-shapes. These shapes are created in areas where the level spacing height does not allow for a full stope. MSO was allowed to make a maximum of two sub-shapes thus allowing for shapes of either 15 m or 7.5 m in height.

When more detailed, manual mine design is completed, the stope sizes could increase in both length and width.

16.7.3 MSO Results

The results of the selected MSO stope shapes are summarized in Table 16-7 below. The results do not include external dilution.

Table 16-7: MSO Stope Tonnage

	MSO Stope Designs		
Zone	**Tonnes**	**Ni %**	**Cu %**
MSU	546,711	5.44	2.27
SMSU	1,744,881	2.67	1.31

16.8 Underground Development

The preferred MSO shapes for the SMSU and MSU zones were imported into Studio 5D Planner. The development design was then included to fit the MSO shapes.

16.8.1 Mine Access

The underground mine will be accessed via a 540 m deep, 5 m diameter, vertical shaft equipped with ore skips and a personnel cage. A second emergency egress will be possible via a ventilation raise.

This method of access to the deposit was selected after a trade-off study comparing ramp access from surface to shaft access. In both cases, 30 m of water-logged glacial till would have to be excavated to reach the bedrock. In the case of the ramp method, either a box cut would have to be excavated to provide access through the glacial till to the portal face, or a freeze wall installed to ramp into the glacial till. For the shaft, a freeze wall would also be required for excavation of the 30 m deep shaft collar. A summary of the three methods are given below.

Ramp Box Cut Method

In the case of access to the underground mine via a ramp from surface, a box-cut and portal would need to be excavated through 30 m of glacial till and 5 m into the rock formation. For a 5 m wide ramp at -15% grade in the glacial till, 384,000 m^3 of glacial till material would have to be removed to reach bedrock. From that point, 86,000 m^3 of rock would have to be removed from the ramp to reach the bottom of the orebody at 500 m depth. All this waste and till would need to be stored in a surface stockpile. The box-cut dimensions would be 337 m in length and width of up to 155 m at surface. The excavation footprint would be approximately 28,900 square metres (m^2). This method would require continuous pumping of the groundwater around the excavation to ensure stable walls and no water entering the mine. The total ramp length would be 3,100 m long from surface to the bottom of the deposit. Ore and waste would be hauled to surface using 40 t underground trucks. The capital cost for this option is estimated at US$22.2M (box-cut, ramp excavation and three haulage trucks), and the operating cost is estimated at US$2.63/t (truck haulage only). This method would take approximately 30 months to complete.

Ramp Freeze Wall Method

For the freeze wall method, the glacial till would be frozen to allow excavation of a 6.0 m diameter cemented tunnel (5 m final opening) directly into frozen glacial till at -15% grade. The freeze wall would essentially be 7.5 m diameter, with a 200 m long frozen OB pipe around the ramp to reach bedrock. This method would need stockpiling of 5,600 m^3 of glacial till and 86,000 m^3 of rock to reach the bottom of the deposit (500 m depth). The total ramp length would also be 3,100 m long from the surface to the bottom of the deposit. Ore and waste would be hauled to surface using 40 t underground trucks. The capital cost for this option is estimated at US$23.9M (freeze wall, ramp excavation and three haulage trucks), and the operating cost is estimated at US$2.63/t (truck haulage only). This method would take approximately 42 months to complete.

Shaft Method

For a shaft, a freeze wall of 7.5 m diameter, 30 m deep in the glacial till would be needed to construct a 6 m cemented shaft collar (5 m final opening). This method would produce 1000 m^3 (of glacial till) and 14,200 m^3 of development rock to reach the bottom of the shaft (540 m depth). The shaft is assumed to be not concrete lined, but rather only screened and shotcreted as presently there is no evidence of major water inflows in the ore body (refer Section 16.4). Ore and waste would be hoisted to surface via 5 t skips, and personnel, equipment and consumables via a service cage. The capital cost for this option is estimated at US$25.2M (freeze wall, collar, shaft excavation and equipping, headframe and hoists), and the operating cost is estimated at US$1.42/t (ore and waste hoisting). This method would take approximately 26 months to complete.

Selection of Shaft Method

Based on the cumulative capital cost, operating cost, construction period, surface footprint (especially for waste storage and impact on wetlands) noise production, and social acceptability, the shaft method was selected as the means to access the deposit.

16.8.2 Hoisting

The mine will be accessed via a 5 m diameter production shaft. The shaft will be equipped with 5 t skips and a cage for transport of workers, equipment and consumables. The headframe will have ore and waste bins. The ore bin will be connected to the process plant via conveyor. The waste bin will have a chute for loading into surface trucks for transport to the CFTF – refer Section 18.6. Two surface drum-type hoists will service the shaft, one for the skips and one for the service cage. Hoisting will be needed for 12 hours per day.

16.8.3 Internal Development

All internal development (levels and ramps) will be designed to be developed at 5.0 m wide and 5.0 m high, which permits use of 7 t LHDs and 20 t underground trucks.

16.8.4 Internal Ore Transport

An ore pass system will link all main levels to the skip loading station for hoisting ore to surface. With the small stope size in the SMSU and drift-and-fill method used in the MSU, the mined ore will be of a size not requiring an underground crusher. A grizzly will be installed over the ore pass and any oversize rock will be handled by mobile rock breaker.

16.8.5 Vent Raises

The ventilation system will comprise the production shaft (intake) as well as a 4 m diameter exhaust vent raise which will also be equipped with a ladderway for secondary emergency egress. Internal vent raises will be excavated between levels to ensure adequate ventilation of all levels. The main vent fan will be installed at the exhaust raise (pull system) and auxiliary fans will ensure proper ventilation on the active production areas. The estimated underground air requirement for Tamarack is 250,000 cubic feet per minute (cfm).

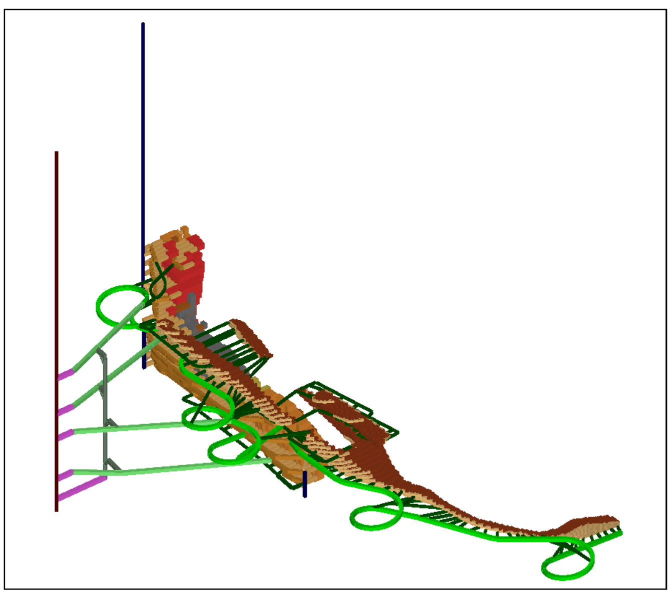


Figure 16-8: Tamarack Mine Development

16.9 Mine Services

16.9.1 Mine Maintenance and Service Area

A mine maintenance and service area will be excavated on level 0 m for basic maintenance and service of underground equipment. Major components will be brought to surface for repair at the contractor's own maintenance shops or sent to the mine equipment supplier shops. Prior to maintenance, all mine equipment will be washed in a dedicated wash bay underground next to the maintenance shop.

16.9.2 Underground Services

- The following mine services will be located underground at level 0 m:
  - Fully equipped lunch room/refuge station and portable toilets;
  - Main Electrical substation;
  - Explosives and detonator storage areas;
  - Storage for ground support material (bolts and screen);
  - Storage for equipment parts and consumables;
  - Fuelling station (fuel to be supplied in self-contained bulk fuel units such as SatStat to be hoisted from surface to underground).

16.9.3 Mine Water Requirements

The following assumptions were made for calculating the estimated underground mine water requirements:

- Pump and sump sizing will be based on maximum process and water inflow rates possible during one shift over LOM;
- At maximum production, there will be two Development Crews and two in-the-hole (ITH) Production Crews operating during one shift;
- Each Development Crew is outfitted with the following:
  - One 2-boom jumbo for face drilling and cable bolting;
  - One 1-boom bolter for installing ground support and services installation holes;

  - One Stoper/Jackleg;
  - Water hose and nozzle for dust suppression during LHD mucking (at the face, and in the re-muck bay).
- Water use by jumbos is estimated based on 100 minutes drilling time (2 drills) and 20 minutes reaming time (1 drill) per round;
- There will be two diamond drill operations;
- There will be one unit drilling BHs via raisebore pilot hole drilling (Process water consumption based on pilot hole drilling/flushing);
- Electric pressure washers are used for cleaning mobile equipment (4 US gpm at 3,000 pounds per square inch (psi));
- Water hoses are used for shotcrete machine cleaning and sump flushing at 35 US gpm;
- There will be one construction crew installing infrastructure and the only water consuming activity will be the shotcreting activities;
- Backfill Line flushing assumed to occur at start and end of each shift;
- Backfill decant water was assumed 24/7;
- Construction crews install infrastructure, including services, pouring concrete pads, and applying shotcrete (water consuming activities). For these calculations, assume the crews are performing shotcrete activities.

Based on these assumptions water consumption will be 188 gpm average with a peak consumption of 382 gpm.

16.10 Equipment Selection

As the mine will be developed and operated by contract mining, the following are estimated based on required development and production rates and typical contractor fleets:

Table 16-8: Mining Equipment Selection

Area & Equipment	Size/Capacity/Use	Quantity
Development		
Jumbo	2-boom	2
LHD	7 tonne	2
Truck	20 tonne	1
Rock bolter	Ground support	2
Scissor lift	5 m reach	2
Production		
Jumbo (MSU)	2-boom	1
Rock Bolter	Ground support	
Long-hole drill	Capability of 40m depth and 100mm diameter	1
LHD	7 tonne	2
Truck	20 tonne	1
Scissor lift	5 m reach	1
Services		
Scissor lift	5 m reach	1
Road grader	Road maintenance	1
Boom truck	Parts & consumables delivery	1
Fuel/Lube truck	Equipment fuel & lube	1
Maintenance truck	Flatbed w/crane & fuel/lube	1
Anfo Loader	For production	1
Service tractors	4-person for supervisors	4
Personnel carriers	10-person for production & dev	2

16.11 Staffing Requirements

Based on a production rate of 1,390 tpd of ore and 350 tpd of development, approximately 220 people will be required for the underground operation, including:

- Stope Miners;
- Development Miners;
- Equipment Operators;
- Hoist Operators;

- Mobile Fleet Operators;
- Support Miners;
- Diamond Drillers;
- Electricians;
- Mechanics;
- Maintenance Workers;
- Technicians;
- Managers;
- Superintendents;
- Foremen;
- Engineers;
- Geologists;
- Shift Bosses.

16.12 Production Schedule

The scheduling was conducted in both Studio 5D Planner and EPS. The scheduling is done in two stages.

Stage 1 is done in Studio 5D Planner where a sequence is assigned. The sequencing process is done for EPS (Stage 2) to identify which tasks are allowed to be done prior to others or, conversely, which tasks have to wait for others.

In EPS, the scheduling is done by applying resources to individual tasks. These resources are given production rates as set out in the mine design criteria.

A levelling process is undertaken where EPS considers the resources, tasks and possible targets and this results in a relatively smoothed schedule.

The final LOM production profile breakdown is shown in Table 16-9 Table 16-8below.

Table 16-9: Breakdown of LOM Production Profile

Domain	Resource Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq
SMSU	Indicated Resource	1,706	2.48	1.22	0.06	0.37	0.25	0.18	3.34
Total	Indicated Resource	1,706	2.48	1.22	0.06	0.37	0.25	0.18	3.34
SMSU	Inferred Resource	175	2.50	1.14	0.06	0.30	0.22	0.14	3.27
MSU	Inferred Resource	506	5.35	2.23	0.11	0.63	0.47	0.23	6.88
138 Zone	Inferred Resource	-	-	-	-	-	-	-	-
Total	Inferred Resource	681	4.62	1.95	0.10	0.54	0.40	0.21	5.96

The NiEq is calculated as follows:

*NiEq% = Ni%+ Cu% x $2.75/$6.75 + Co% x $20.00/$6.75 + Pt [g/t]/31.103 x $1,100/$6.75/22.04 + Pd [g/t]/31.103 x $800/$6.75/22.04 + Au [g/t]/31.103 x $1,200/$6.75/22.04

Table 16-10 below shows the final production schedule for the Tamarack resource.

Table 16-10: Tamarack Production Schedule

		PRE-PROD								
Description	**Units**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Total**
Total Tonnes	**t**	**46,753**	**337,617**	**676,245**	**667,948**	**552,102**	**535,920**	**286,166**	**88,086**	**3,190,836**
Ore Tonnes per Day	tpd		239	1,371	1,390	1,361	1,347	684	241	829
Ore Tonnes			**85,889**	**493,700**	**500,443**	**489,816**	**484,860**	**246,328**	**86,634**	**2,387,670**
MSU	t			59,953	94,193	94,137	94,770	89,458	73,426	505,937
SMSU Stope	t		20,622	277,707	241,127	238,863	203,044	70,427	3,373	1,055,162
Ore Development	t		65,267	156,040	165,123	156,816	187,046	86,443	9,835	826,570
Nickel										
Total	t		1,944	13,799	14,843	15,772	15,217	8,188	4,076	73,840
Overall Grade	%		2.26	2.80	2.97	3.22	3.14	3.32	4.71	3.09
Copper										
Total	t		995	6,300	7,066	7,283	6,943	3,702	1,720	34,008
Overall Grade	%		1.16	1.28	1.41	1.49	1.43	1.50	1.99	1.42
Cobalt										
Total	t		49	332	343	372	368	190	88	1,742
Overall Grade	%		0.06	0.07	0.07	0.08	0.08	0.08	0.10	0.07
Platinum										
Total	oz		880	5,206	8,042	7,003	6,401	3,015	1,776	32,321
Overall Grade	g/t		0.32	0.33	0.50	0.44	0.41	0.38	0.64	0.42
Palladium										
Total	oz		517	3,686	5,419	4,924	4,582	2,162	1,211	22,501
Overall Grade	g/t		0.19	0.23	0.34	0.31	0.29	0.27	0.43	0.29

Gold										
Total	oz		430	2,494	3,721	2,972	2,837	1,196	661	14,310
Overall Grade	g/t		0.16	0.16	0.23	0.19	0.18	0.15	0.24	0.19
Sulphur										
Total	t		7,928	56,044	59,900	64,031	61,656	32,899	16,162	298,621
Overall Grade	%		9.23	11.35	11.97	13.07	12.72	13.36	18.66	12.51
Waste										
Waste Development	t	46,753	251,728	182,545	167,505	62,286	51,060	39,838	1,452	803,167
Development Metres										
Shaft	m	502	34							536
Level Development	m	100	878	79						1,057
Raise Bore	m			350	50					400
Vent Raise	m					105				105
Ramp	m		903	714	40	57				1,713
SMSU Crosscut	m		1,133	531	1,916	725	740	286	16	5,346
SMSU Ore Drive	m		542	952	1,192	1,120	1,369	499	47	5,720
Ore Connection	m		349	1,490	841	354	262	646	14	3,955
MSU Crosscut	m			1,178	771	544	559	651	139	3,842
Other	m		220		10					230

17. RECOVERY METHODS

This section describes the process design basis that facilitated the generation of the circuit mass and water balance, the concentrator process design criteria, and selection and sizing of the major processing equipment required to treat the Tamarack North ore in accordance with the mine production schedule outlined in Section 16.

The metallurgical process consists of bulk rougher and scavenger flotation followed by separate cleaning of the rougher and scavenger concentrates. The upgraded rougher concentrate is subjected to Cu/Ni separation. The process generates separate Cu and Ni concentrates. Further, the bulk scavenger tailings are treated in a desulphurization stage to produce a low-mass HS stream and high-mass, potentially NAG tailings.

The process was designed utilizing results of a metallurgical test program that was conducted at SGS Lakefield in 2016 and 2017 on MSU, SMSU, CGO, and Main North samples. The LOM mill feed is projected to produce a Cu concentrate grading 28.9% Cu at 84.4% Cu recovery. The Ni concentrate is expected to grade 14.5% Ni at 85.0% Ni and 10.1% Cu recovery. It is expected that Ni in the Cu concentrate will be further reduced with additional testing, which will increase the Cu grade in the Cu concentrate. Also, if the proportion of Cu in the Ni concentrate can be reduced, the increased number of Cu units reporting to the Cu concentrate will likewise help to increase the final concentrate grade closer to 30% Cu.

17.1 Key Process Design Criteria

The process design criteria were generated based on an average daily mill feed rate of 1,390 tpd and an average LOM head grade of 3.09% Ni and 1.42% Cu. The results of the metallurgical test program were used to project the metallurgical results for this LOM head grade using regression curves.

The process design criteria were developed from a range of different sources, which are outlined below:

A – Talon Input
B – Metpro Recommendation
C – Metpro Calculation
D – Third Party Input
E – Metallurgical Testing
F – Standard Industry Practice
G – Vendor Recommendation

The mineralized mill feed material characteristics and expected metallurgical performance are presented in Table 17-1.

Table 17-1: Plant Feed Characteristics and Metallurgical Performance

Criteria	Units	Value		Source
		Expected/Avg.	Design	
Solids Density	t/m^3	2.90 – 3.75	3.08	D
Bulk Density	t/m^3	1.60 – 2.00	1.80	B
LOM Mill Head Grade	% Ni	1.98 – 5.97	3.09	D
LOM Mill Head Grade	% Cu	1.03 - 2.55	1.42	D
Mill Treatment Capacity	ktpa		507.3	C/D
Ni Recovery to Ni Concentrate	% Ni		85.0	E/C
Ni Concentrate Grade	% Ni		14.5	E/C
Ni Concentrate Production	ktpa		91.8	E/C
Overall Cu Recovery	% Cu		94.5	E/C
Recovery to Cu Concentrate	% Cu		84.4	E/C
Cu Concentrate Grade	% Cu		28.9	E/C
Cu Concentrate Production	ktpa		21.1	E/C

The operating schedule of the processing plant is detailed in Table 17-2.

Table 17-2: Plant Design Operating Schedule

Criteria	Units	Value		Source/ Responsibility
		Expected/Avg.	Design	
ROM Material Delivered to Mill	ktpa		507.3	C/D
Crusher Plant Operating Schedule				
Days per Week	days	7	7	B
Shifts per Day	shifts	3	3	B
Hours per Shift	h	8	8	B
Utilization	%	70	70	B
Operating Hours per Annum	h		6,132	B
Crusher Circuit Throughput	tph	82.7	103.4	B
Milling and Flotation Operating Schedule				
Days per Annum	days	365	365	B
Hours per Day	h	24	24	B
Utilization	%	92	92	B
Operating Hours per Annum	h		8,059	B
Mill Feed Rate	tph	63.0	75.5	C

17.2 Process Block Flow Diagram

The simplified process flowsheets for the crushing and grinding circuit, the flotation circuit, and the dewatering circuit are presented in Figure 17-1, Figure 17-2, and Figure 17-3, respectively.

The crushing circuit consists of primary jaw crushing and secondary cone crushing followed by ball mill grinding. The ball mill discharge is subjected to bulk rougher and scavenger flotation followed by cleaning of the bulk rougher and scavenger concentrates to produce separate Cu and Ni concentrates. The concentrates will be thickened, filtered and shipped to different smelters via rail. The desulphurization stage will recover most of the remaining sulphides into a HS tailings stream for paste backfill (refer Section 16.2). Some LS tailings will be disposed of as paste backfill and the remainder of the LS tailings will be co-disposed with waste rock in a CFTF – refer Section 18.6.

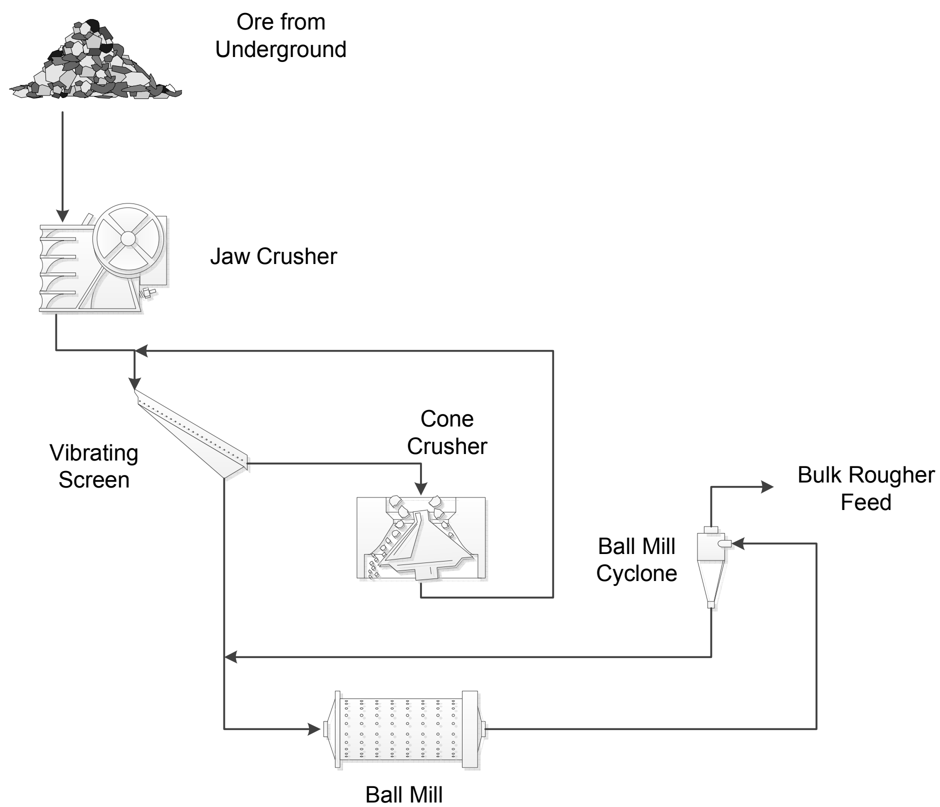


Figure 17-1: Crushing and Grinding Circuit

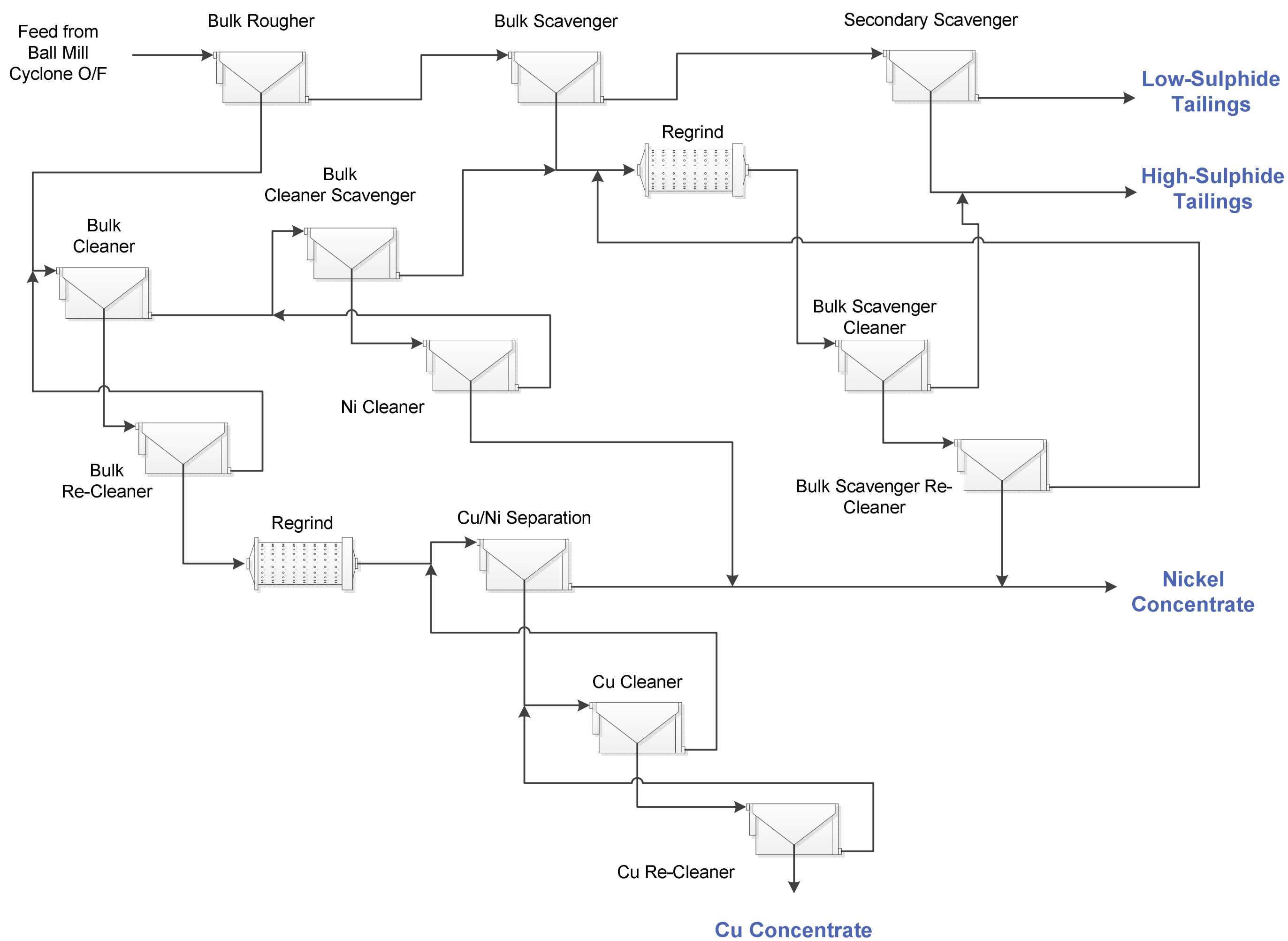


Figure 17-2: Flotation Circuit

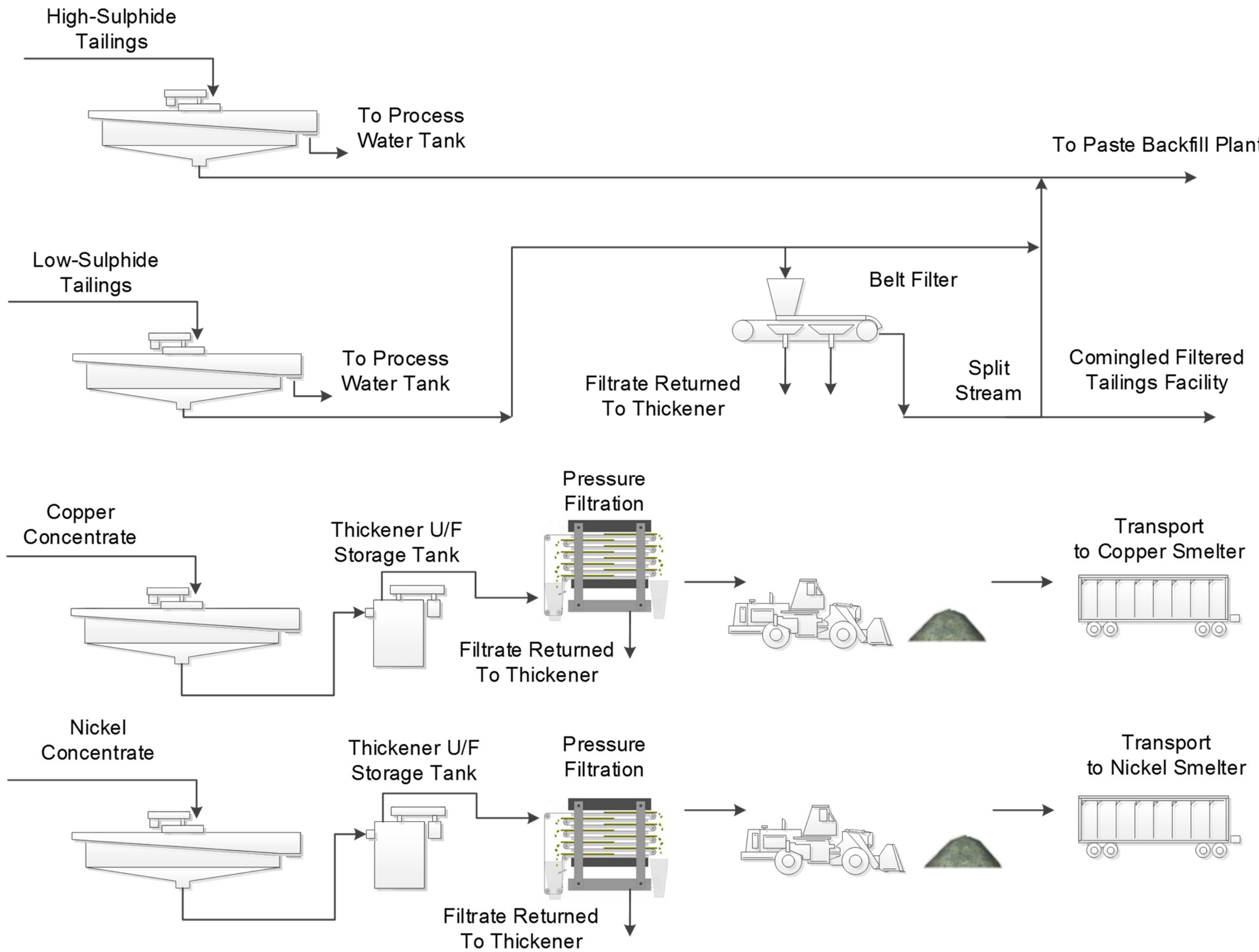


Figure 17-3: Dewatering Circuits

17.3 Process Description

17.3.1 Crushing and Primary Grinding

The crushing circuit will consist of a jaw crusher that will operate in open circuit followed by a cone crusher that will operate in closed circuit. The two-stage crushing circuit is designed to operate at ±70% utilization and a design factor of 25%, equating to a feed capacity of approximately 103 tonnes per hour (tph). Mineralized run of mine (ROM) material will be delivered to a ROM bin feeding the crushing and screening section of the plant. A grizzly with rock breaker installed underground will remove any oversize material greater 850 mm. The ROM material will be crushed to a product size P_{80} of 16 mm in the two stages of crushing. Classification of the cone crusher product will be performed on a vibrating screen.

The cone crusher product will be transferred to a 1,200 m^3 fine ore bin to decouple the crushing and grinding circuits due to lower availability of the crushing circuit. The ore will then be transferred from the fine ore bin to a 4.0 m x 5.8 m ball mill in closed circuit with a classification cyclone to generate a flotation circuit feed with a P_{80} of 70 µm.

17.3.2 Bulk Rougher, Bulk Scavenger, and Secondary Scavenger Flotation

The ball mill cyclone overflow with a grind size of P_{80} = 70 µm will gravitate to the bulk rougher flotation cells at a mass flow rate of 63.0 tph. The selective sulphide collector SIPX, MIBC, and gangue depressant CMC will be added to the flotation feed box to assist with recovery of Cu and Ni values. Lime will be added to maintain a pH of 9.0. Bulk rougher flotation will take place in three tank cells with a volume of 20 m^3 each. The bulk rougher concentrate will be transferred to the bulk rougher cleaning circuit.

Bulk rougher tailings will gravitate to the bulk scavenger flotation stage (3 cells of 20 m^3 each), which aims to recover any remaining Ni units with the aid of the stronger sulphide collector PAX. pH will be maintained at 9.0 during this stage of flotation. Bulk scavenger concentrate will be pumped to the bulk scavenger cleaning circuit. Bulk scavenger tailings will gravitate to the secondary bulk scavenger flotation stage.

The secondary bulk scavenger flotation circuit will recover most of the remaining sulphide minerals into a low-mass HS tailings stream and a high-mass LS tailings stream. Higher dosages of PAX and dispersant sodium metasilicate will be added to the secondary bulk scavenger flotation stage consisting of three 20 m^3 tank cells to achieve this objective. The HS tailings stream will be acid generating; 100% of this stream will be placed underground with cement as paste backfill (refer to Section 16.2). A portion of the LS tailings will also be placed underground with cement as paste backfill (refer to Section 16.2) and the remainder of the LS tailings, that cannot be placed underground, will be filtered and dry-stacked together with mine development rock at surface (refer to Section 18.7).

17.3.3 Bulk Cleaner and Ni Cleaner Flotation

Bulk rougher concentrate will be subjected to two stages of cleaning to reject non-sulphide gangue minerals and Po. Increased flotation selectivity will be achieved by increasing the cleaning circuit pH from 9.0 to 10.0 using lime. CMC will be added as a gangue depressant and SIPX to promote the flotation of Cpy and fast floating Pn. All reagents will be introduced into the pump box of the pump that transfers bulk rougher concentrate to the bulk cleaner flotation feed box.

Retention times in the bulk cleaner and bulk recleaner stages will be 10 minutes and 9 minutes, respectively. Bulk cleaner flotation will be performed with three 5.0 m^3 trough flotation cells, and bulk recleaner will utilize two 4.0 m^3 trough flotation cells.

The bulk recleaner tailings will be circulated back to the bulk cleaner stage, and the concentrate transferred to the Cu/Ni separation circuit.

The bulk cleaner tailings will gravitate to three 2.0 m^3 bulk cleaner scavenger cells to recover most of the remaining pentlandite into a secondary Ni concentrate. Lime will be added to maintain a pH of 10.0, and collector SIPX introduced in the feed box to promote pentlandite flotation.

The bulk cleaner scavenger concentrate will be pumped to a Ni cleaner to reject additional Po units. The increased separation efficiency will be achieved with the addition of lime to a pH of 10.5. SIPX will be added to promote pentlandite flotation. Ni cleaner flotation will be performed in three 1.0 m^3 trough flotation cells, with the tailings stream pumped back to the feed box of the bulk cleaner scavenger.

The bulk cleaner scavenger tailings will be combined with the bulk scavenger concentrate to be processed in a bulk scavenger cleaning circuit. A regrind to P_{80} = 20 microns will be performed on the combined bulk scavenger concentrate and bulk cleaner scavenger tailings.

17.3.4 Cu/Ni Separation

Bulk recleaner concentrate will be pumped to the feed pump box of the Cu/Ni separation regrind mill cyclone. The regrind mill will operate in closed circuit to reduce the P_{80} in the bulk recleaner concentrate from 65 µm to 35 µm prior to Cu/Ni separation. Regrinding will be performed in a VTM-150-WB Vertimill® or similar with steel grinding media.

The regrind mill cyclone overflow will gravitate to the feed box of the Cu/Ni separation flotation cells. Lime will be added to the feed box of the flotation cell to maintain a pH of 12.0, to promote the separation of Cu and Ni minerals. No further reagents will be added at this stage. Separation will be carried out in three 3.0 m^3 trough flotation cells.

Cu/Ni separation tailings represent the primary component of the Ni concentrate. The Cu/Ni separation tailings stream will be combined with the Ni cleaner concentrate and the bulk scavenger recleaner concentrate and pumped to the Ni concentrate thickener.

The Cu/Ni separation concentrate will still contain quantities of Ni and will thus be subjected to two additional cleaning stages to minimize recovery of Ni into Cu concentrate, given that no credit will be received for any Ni units in the Cu concentrate.

The Cu/Ni separation concentrate will be transferred to the Cu cleaner flotation stage with four 1.0 m^3 trough flotation cells. The Cu cleaner tailings will be transferred back to the Cu/Ni separation feed, and the Cu cleaner concentrate will be pumped to the Cu recleaner stage, which will consist of four 1.0 m^3 trough flotation cells. The pH will be controlled at 12.0 in both cleaning stages, and only MIBC will be added to promote Cu flotation.

17.3.5 Bulk Scavenger Cleaning Circuit

The purpose of the bulk scavenger cleaning circuit will be to recover most of the pentlandite from the bulk scavenger concentrate and the bulk cleaner scavenger tailings into a third Ni concentrate stream, and to reject non-sulphide gangue and Po into the tailings.

The bulk scavenger concentrate and the bulk cleaner scavenger tailings will be combined in the feed pump box of the bulk scavenger regrind mill cyclone. The cyclone underflow will gravitate into a VTM-200-WB Vertimill® with steel grinding media. The mill feed with an F_{80} of 45 μm will be ground to a product size of P_{80} = 20 μm.

The cyclone overflow will be directed to the feed box of the bulk scavenger cleaner flotation stage. A low dosage of CMC will be added to depress non-sulphide gangue minerals, and SIPX will be introduced to promote pentlandite flotation. pH will be controlled at 10.5 to depress Po. Flotation will be performed in four 5.0 m^3 trough flotation cells. Bulk scavenger cleaner tailings will be combined with secondary scavenger concentrate to form the HS tailings stream.

The bulk scavenger cleaner concentrate will be upgraded further in a bulk scavenger recleaner using three 3.0 m^3 trough flotation cells. Bulk scavenger recleaner concentrate will be combined with Cu/Ni separation tailings and Ni cleaner concentrate to form Ni concentrate. The three streams will be pumped to the Ni dewatering circuit. Bulk scavenger recleaner tailings will be circulated back to the regrind stage of the bulk scavenger cleaner flotation stage.

17.3.6 Ni and Cu Concentrate Thickening

Average Ni and Cu concentrate productions of roughly 252 tpd and 58 tpd respectively, are anticipated at the design mill feed rate of 1,390 tpd. Concentrate production will fluctuate from those values depending on the stage of mine development and actual mill feed rates and grades.

The two concentrates will be transferred to two high rate thickeners. Both thickeners will be designed for the average expected production demand plus a 20% design factor. The thickened slurry will be pumped to separate holding tanks. The overflow of the two thickeners will gravitate back to the plant process water tanks.

17.3.7 Ni and Cu Concentrate Filtration

Thickened Ni and Cu concentrate slurries will be pumped from their stock tanks to pressure filters, and the dewatered filter cake will be stockpiled. The concentrates will be shipped in specialized sealed containers designed to prevent loss of concentrate product and any potential for concentrate dust evolution.

Filtrates from the pressure filter will be recycled to the high rate thickeners to avoid collecting solids of concentrate during cake buildup and other solids such as filter cloth debris in the process water tanks.

17.3.8 Tailings Thickening

Average LS and HS tailings productions of 317 tpd and 764 tpd, respectively, are anticipated at the design mill feed rate of 1,390 tpd. The actual production will fluctuate from those values depending on the stage of mine development and actual mill feed rates and grades. Both thickeners will be designed for the average expected production demand plus a 20% design factor.

The two tailings streams will be transferred to two high rate thickeners, and the thickened slurry pumped to separate holding tanks. The overflow of the two thickeners will gravitate back to the plant process water tank.

17.3.9 Tailings Filtration

The maximum solids concentration of the paste for backfill is estimated at 79.5% by weight (w/w) by Paterson & Cooke. The thickener underflow of the HS tailings thickener will be placed into a small pond for temporary storage to eliminate self-heating risk. The HS tailings will be retrieved from the small pond and combined with LS tailings filter cake and binder to produce a paste with a solids content of 79.5% w/w.

Tailings from the LS tailings thickener will be dewatered to 15-20% moisture content (preliminary estimate pending geotechnical testing) using a belt filter (refer Section 18.6.9) before being transported by truck to the CFTF.

17.4 Energy, Water, and Process Materials Consumption

17.4.1 Energy

The total plant energy requirement from the major mechanical equipment list will be 2,600 kilowatt (kW). Pumps and plant services were factored for a total connected power of 3,750 kW. The operational power draw is anticipated to be approximately 85% of connected power, or 3,200 kW. Electrical power will be supplied by the electrical grid.

17.4.2 Water

The total water requirement of the grinding and flotation circuit is estimated at 774 gpm (175.8 tph). This water requirement includes water addition in the grinding circuit, dilution, and launder water.

All process water recovered in the dewatering circuits of the two concentrates and two tailings will be circulated back to process water tanks. The total amount of reclaimed water amounts to 671 gpm (152.4 tph). Hence, the fresh water requirements to make up the water deficit will be 103 gpm (23.4 tph), which includes an allowance of 22 gpm (5 tph) of fresh water for glands, potable, reagent makeup, etc. Refer to Section 18.7.1 for an estimated, annual project water balance.

17.4.3 Process Consumables

Reagent types and dosages were established during the 2016/2017 metallurgical program conducted at SGS Lakefield.

The grinding media, liner, and lifter consumption is calculated based on an estimated Bond abrasion work index that corresponds to the 50^{th} percentile of abrasiveness of more than 2,000 samples that were tests at SGS. This approach was taken since no Bond abrasion work index data is presently available for the Tamarack SMSU and MSU mineralization.

The reagent consumption and grinding media wear rates are presented in Table 17-3 and Table 17-4 respectively.

Table 17-3: Reagent Consumption Rates

Reagent	Consumption (g/t)
Sodium Isobutyl Xanthate (SIPX)	130
Potassium Amyl Xanthate (PAX)	330
Methyl Isobutyl Carbinol (MIBC)	125
Carboxy Methyl Cellulose (CMC)	338
Sodium Metasilicate	400
Lime	730
Flocculant	80

Table 17-4: Grinding Media Consumption Rates

Application	Consumption (kg/t of mill feed)
Primary Ball Mill Balls	1.64
Cu/Ni Separation Vertimill Media	0.66
Bulk Scavenger Vertimill Media	1.09

17.5 Major Equipment List

A list of major mechanical equipment is provided in Table 17-5, which served as the basis for the development of the capital cost estimate.

Table 17-5: Major Mechanical Equipment

Description	Specification
Primary Crusher	Metso C96 jaw crusher
Secondary Crusher	Metso HP200 cone crusher
Crusher Closed-Circuit Screen	1.9 m x 6.1 m inclined screen
Fine Ore Bin	1,200 m^3
Primary Ball Mill	4.0 m x 5.8 m ball mill
Bulk Rougher Flotation Cells	3 x 20 m^3 tank cells
Bulk Scavenger Flotation Cells	3 x 20 m^3 tank cells
Secondary Scavenger Flotation Cells	3 x 20 m^3 tank cells
Bulk Cleaner Flotation Cells	3 x 5.0 m^3 trough cells
Bulk Recleaner Flotation Cells	2 x 4.0 m^3 trough cells
Bulk Cleaner Scavenger Flotation Cells	3 x 2.0 m^3 trough cells
Ni Cleaner Flotation Cells	3 x 1.0 m^3 trough cells
Cu/Ni Separation Circuit Regrind Mill	Vertimill® - VTM-150-WB
Cu/Ni Separation Flotation Cells	3 x 3.0 m^3 trough cells
Cu Cleaner Flotation Cells	4 x 1.0 m^3 trough cells
Cu Recleaner Flotation Cells	4 x 1.0 m^3 trough cells
Bulk Scavenger Regrind Mill	Vertimill® - VTM-200-WB
Bulk Scavenger Cleaner Flotation Cells	4 x 5.0 m^3 trough cells
Bulk Scavenger Recleaner Flotation Cells	3 x 3.0 m^3 trough cells
Ni Concentrate Thickener	12 m diameter, high-rate
Ni Concentrate Pressure Filter	Pressure filter– Outotec PF 15-45
Cu Concentrate Thickener	6 m diameter, high-rate
Cu Concentrate Pressure Filter	Pressure filter – Outotec PF 12-12.5
High-Sulphide Tailings Thickener	12 m diameter, high-rate
Low-Sulphide Tailings Thickener	23 m diameter, high-rate
Low-Sulphide Tailings Vacuum Filter	93.5 m2 Vacuum Filter

18. INFRASTRUCTURE

18.1 Introduction

The existing local transportation infrastructure is excellent. The site is accessible via an existing road which connects to the Minnesota State highway network. The active BNSF Railway passes by the town of Tamarack and connects to an extensive network of rail lines throughout the US and Canada, including access to the Duluth port. The city of Duluth lies on the westernmost point of Lake Superior, and provides worldwide shipping access via the Great Lakes, St. Lawrence Seaway, and Atlantic Ocean shipping routes. For the benefit of the Tamarack Project, Kennecott has secured surface rights adjacent to the BNSF railway line to allow for the construction of a railroad siding near the project site, should this be required.

18.2 Site Access and Power

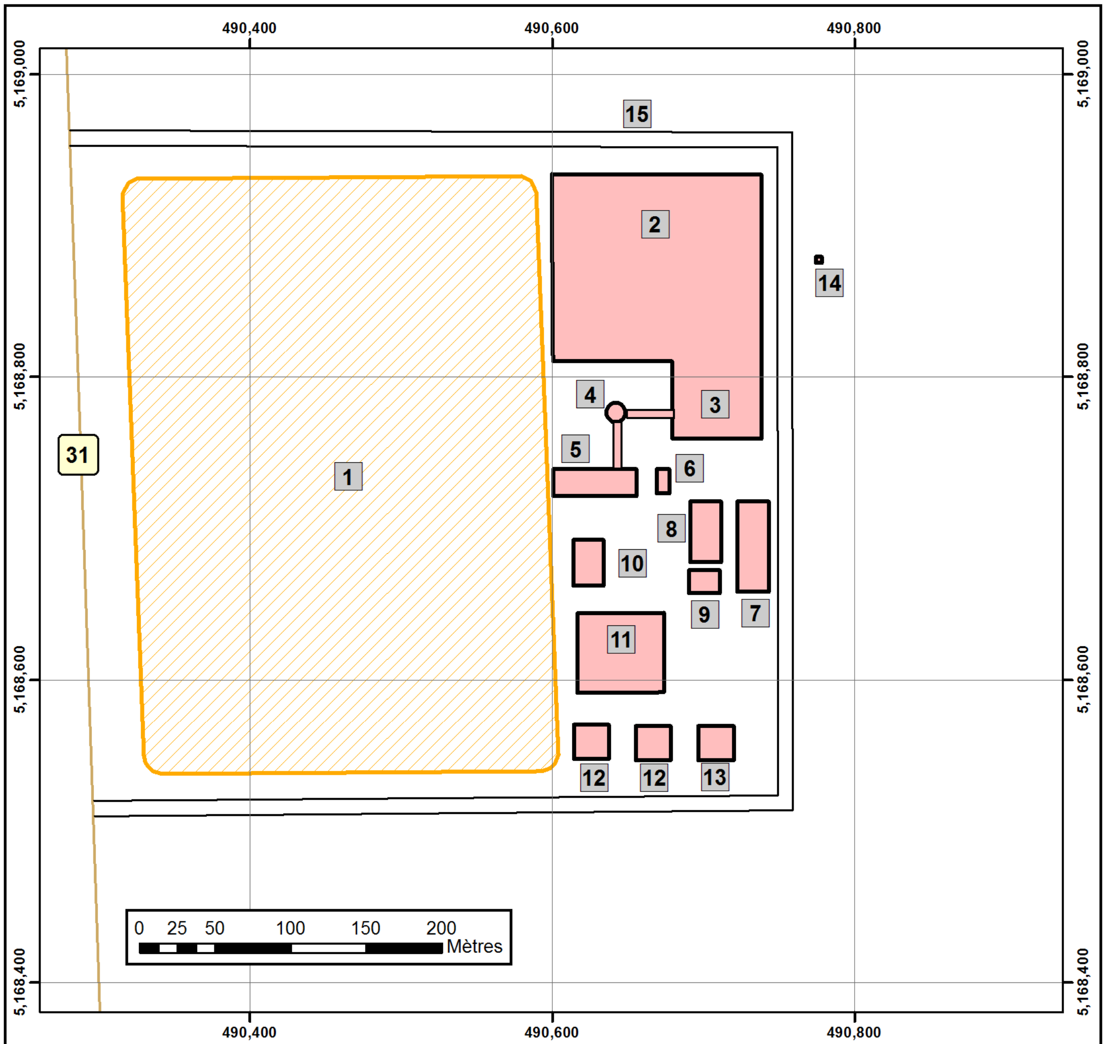
For the purposes of this PEA study, it is assumed that the site will be accessible by a two-lane road directly from paved County Highway 31.

A Great River Energy Transmission Line crosses the Tamarack North Project property. The line connects through substations close to the nearby towns of Wright (10 km away) and Cromwell (20 km away). A standby diesel generator will be used to supply emergency power if utility power is interrupted.

A Power House industrial facility will be provided for the distribution of power and backup power generation. The Power House will contain a step down facility from the main transmission line and a back up generator.

18.3 Site Layout Considerations and Concept

A conceptual site layout is presented in Figure 18-1.



1. Comingled Filtered Tailings Facility (CFTF)
2. Concentrator & Loadout
3. Comminution
4. Ore Bin
5. Production Shaft
6. Hoist Building
7. Mine Office
8. Warehouse, Shops & Maintenance
9. Wash Bay
10. Paste Backfill Plant
11. Parking
12. Temporary, lined water storage ponds
13. Water treatment Facility
14. Ventilation Shaft
15. Access Road

DRA		
TAMARACK PROJECT		
General Layout		
07-25-2018	Drawing:	

Figure 18-1: Concept Project Site Layout

18.4 Buildings and Facilities

18.4.1 Production Shaft, Hoist Building and Ore Bin

Ore will be supplied to the concentrator from the production shaft via conveyors and an ore bin, which will provide ore storage to ensure continuous operation of the concentrator, as ore hoisting is intermittent.

18.4.2 Mine and Mill Services Building

The mine and mill services building will be located to ensure optimal and safe movement of personnel and equipment.

18.4.3 Comminution

The Jaw and Cone crushers as well as the ball mill will be constructed and operated in an enclosed facility to reduce noise and contain dust while ensuring the safety of operating and maintenance personnel, especially during summer lightning storms.

18.4.4 Concentrator

The concentrator building will house equipment for the recovery of Ni and Cu concentrates and storage for these concentrates prior to shipping. A metallurgical laboratory will also be contained within the concentrator building.

18.4.5 Paste Backfill Plant

A paste backfill plant will return 100% of the HS tailings as well as a portion of the LS tailings, blended with cement, back to the underground void space, which will eliminate the need to store this material on surface. A building will be provided to house the paste backfill plant and associated facilities. A description of the paste backfill plant and distribution of the paste backfill material is provided in Section 16.2.

18.4.6 Water Treatment Plant

The investigation of Water Treatment Plant alternatives is outside of the scope of this PEA. Further work is therefore required to determine and study Water Treatment Plant options.

18.4.7 Temporary Development Rock Storage Area

An allowance is made for a Temporary Development Rock Storage Area, for the purpose of containing waste rock until such time as the permanent CFTF (per Section 18.6) is established. Further details of this facility will be developed during the PFS.

18.4.8 Vehicle Washing Bays

All vehicles leaving the main operations area will be washed before leaving.

18.4.9 Mine Office, Warehouse and Workshops

The concentrator and mine are supported by administrative, supplies and maintenance functions housed in the mine office, warehouse (to store supplies used in the mining operation) and workshops (mechanical, electrical and instrumentation). A parking area will be located near the warehouse and workshops.

18.4.10 Security Gatehouse

Site access and exit will be security controlled at all times.

18.5 Logistics

Raw materials and maintenance supplies will be shipped to the site by road. Ni and Cu concentrates will be despatched to customers by road to the port of Duluth. The concentrates will be shipped in specialized sealed containers designed to prevent loss of concentrate product and any potential for concentrate dust evolution. The specialized 20 ft containers will require minimal infrastructure to load at both the site and at the port or train loading areas. These envisaged containers are used extensively in South America for transporting Cu concentrates.

18.6 Co-disposed Filtered Tailings Facility (CFTF)

> ***Caution to Readers**: This Item contains forward-looking information related to the management and storage of tailings for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions: project development plan and production schedule, geochemical characterization of rock, water balance and water storage, and facilities design criteria.*

Talon commissioned the investigation of alternative options for the management and storage of LS tailings at surface:

- Slurry tailings consisting of transporting a low solids content, high water content slurry by pipeline and centrifugal pumps to a lined Tailings Storage Facility (TSF);
- Paste consisting of transporting a high solids content, reduced moisture content paste by pipeline and positive displacement pumps to a lined TSF;
- Cemented paste consisting of transporting a high solids content, reduced moisture content cemented paste by pipeline and positive displacement pumps to a lined TSF;
- Filtered tailings consisting of transporting high solids content, low moisture content tailings to a lined TSF by conveyor or by truck haul.

None of these tailings storage options investigated were acceptable to Talon as it required separate development rock and LS TSFs. An innovative approach was therefore developed which combines the disposal facilities for LS tailings and development rock into a single CFTF. Tailings will be thickened and filtered to remove most of the contained water (refer to Sections 17.3.8 and 17.3.9), allowing the solid LS tailings to be stacked. The filtered, LS tailings and development rock will be deposited together on a lined facility as described below. Any moisture that is released from the tailings, together with precipitation, will be collected in a lined ditch around the perimeter of the CFTF which then drains to temporary storage ponds at the SE corner of the facility with any collected water recycled back to the processing plant for re-use after treatment by the Water Treatment Plant.

Co-disposal of filtered tailings and development rock offers significant environmental and operating advantages over separate tailings storage and development rock storage facilities, including:

- Reduced risk of failure as the facility is not required to store water;
- A major reduction in the waste facility footprint: Surface land required for the storage of LS tailings that are not placed underground with cement, will be reduced by 40% versus traditional methods as a portion of the development rock void space will be utilized as storage space;
- Improved tailings stability and reduced dusting compared to standalone filtered tailings facility without co-disposal with development rock;
- At closure, the CFTF will be covered with a closure cover system. This will limit the amount of infiltration into the CFTF post closure, potentially reducing post closure care liabilities;
- A significant reduction in fresh water requirements. In fact, 86.7% of water required by the processing plant will be recycled water of which 30% will be as a direct result of using a CFTF.

A design and detailed description of the proposed CFTF was provided by Golder. The following sections summarize its key criteria and features.

18.6.1 Mining and Processing

The mine will produce 2.39 million tonnes (Mt) of ore (see Section 16.12) for an annual production schedule). Development of the mine will generate up to 0.86 Mt of development rock.

To date static testing has been conducted on the various development rock and ore types at the Tamarack North Project. To date, no testing has been carried out on samples of tailings; however this is planned for later stages of study.

The development rock types consist of Sediments, FGO and CGO (see Section 7.2.4 for descriptions of FGO and CGO).

The preliminary geochemical results for the sediment and FGO development rock suggest this material may be Non-Acid Generating (NAG), while the CGO was interpreted to be Potentially Acid Generating (PAG). However, two of the three samples collected of anticipated CGO development rock has since been determined to be ore based on the updated mine plan or ore resource model and may not be representative of the CGO development rock. Since limited geochemical testing has been performed to date, a conservative approach was used for this PEA and assumes that only the Sediments and FGO rock types will be NAG and therefore 0.37 Mt of development rock were classified as NAG while the remaining 0.48 Mt (all CGO) were classified as PAG.

The processing plant is designed to process 1,390 tpd (0.5 Mtpa) of ore. The processing plant will generate two separate concentrate streams, namely Ni and Cu. The processing plant will generate two tailings streams: HS tailings comprised of Po and other sulphides and low LS tailings comprised mainly of silicates (refer Sections 17.1 and 17.2). Over the life of the mine the processing plant will generate a total of 1.87 Mt of tailings, of which 1.31 Mt will be LS tailings and the remaining 0.55 Mt will be HS tailings as detailed in Section 16.2.

Approximately 56% of the tailings streams (1.04 Mt) are planned to be used for backfilling the underground stopes. This includes all of the HS tailings (0.55 Mt) and 0.49 Mt of the LS tailings. The remaining 0.82 Mt of the LS tailings are planned to be dewatered using a filtration plant to a 15% moisture content and trucked to the CFTF for co-disposal with the development rock.

18.6.2 Tailings and Development Rock Production Schedule

The CFTF is designed to provide storage capacity for 0.48 Mt of PAG development rock, 0.37 Mt of NAG development rock and 0.82 Mt of LS filtered tailings. The deposited dry densities of these waste streams are required to estimate the storage volume requirement of the CFTF. The deposited dry densities of these waste streams in turn depend on their SG and void ratio/porosity. The SG and void ratio assumed for the various waste streams based on preliminary information available from Talon and Golder's experience on similar materials are presented in Table 18-1.

Table 18-1: Assumed Geotechnical Properties of Waste Streams

Input	Unit	NAG Development Rock	PAG Development Rock	Filtered LS Tailings
Specific Gravity	-	2.77	2.89	2.94
Deposited Void Ratio	-	0.43	0.43	0.73
Deposited Porosity	-	0.30	0.30	0.42
Deposited Dry Density	t/m^3	1.94	2.03	1.70

The deposited dry density assumed for the NAG development rock, PAG development rock, and LS tailings are 1.94 t/m^3, 2.03 t/ m^3 and 1.70 t/ m^3, respectively. Based on these densities, the volumes of NAG development rock, PAG development rock, and LS tailings that will be sent to the CFTF will be approximately 0.19 Mm^3, 0.24 Mm^3, and 0.48 Mm^3, respectively as shown in Table 18-2. The NAG development rock will be used to construct the perimeter wall of the CFTF whereas the PAG development rock will be co-disposed with the filtered tailings within the CFTF. It is estimated that 0.19 Mm^3 of NAG development rock will be required for the perimeter wall. It was conservatively assumed that approximately 50% of the PAG development rock void will be filled with filtered tailings. That is, approximately 35,800 m^3 of the filtered tailings will be filling the void space of the PAG development rock which otherwise would have been filled with air and/or water. Therefore, the design storage capacity of the CFTF is 0.88 Mm^3.

Table 18-2: Volume of Tailings and Development Rock for Co-disposal

Year	Development Rock			LS Tailings	
	NAG (m^3)	PAG (m^3)	Total (m^3)	(t)	(m^3)
1	19,791	-	19,791	-	-
2	66,151	46,030	112,181	26,285	15,462
3	40,947	58,472	99,418	170,214	100,126
4	19,222	70,728	89,950	167,531	98,548
5	14,345	27,207	41,552	166,782	98,107
6	9,549	23,730	33,279	165,101	97,118
7	18,879	11,165	30,044	91,653	53,914
8	4,076	1,256	5,332	33,244	19,555
Total	**192,960**	**238,588**	**431,547**	**820,810**	**482,830**

18.6.3 Geochemical Characterization of Development Rock

A preliminary geochemical characterization program was completed on various lithologies in 2008 (Foth, 2008). Fourteen samples of rock core from six rock units were selected from the available exploration drill core and submitted for static testing including; Acid Base Accounting (ABA), NAG pH, Elemental analysis, and Synthetic Precipitation Leaching Procedure (SPLP). Of the fourteen samples, seven samples are considered representative of development rock as further explained below. No tailings samples were submitted for geochemical testing.

Two samples of both sediment and FGO were submitted for analysis. Three CGO samples were also analysed; however, the first of the three CGO samples tested from drill hole 08TK0048 over interval 383.61 m to 384.76 m has been re-classified as SMSU and included in the mine plan (1.645% Ni and 0.963% Cu were assayed from 383.5 m to 385 m). The second CGO sample with sulphur content of 2.01% from drill hole 08TK0049 over interval 445.5 m to 446.9 m has been reclassified as SMSU and included in the Mineral Resource estimate (0.613% Ni and 0.416% Cu were assayed from 445.5 m to 447 m). Although this drill hole is presently included in the Mineral Resource estimate (refer Section 14.6), it is excluded from the mine plan due to a higher cut-off grade applied to the mine plan (refer Section 16.7.1) than to the Mineral Resource estimate. If the metallurgical testing program proposed under Section 25.2 is successful, it is likely that this resource will be included in a future mine plan and therefore it will not constitute development rock. The third CGO sample with sulphur content of 0.21% is representative of CGO development rock.

Review of the BH database shows that sulphide contents typically increase with proximity to the ore zone. This is consistent with the preliminary ABA results with low sulphide contents in units farthest from the ore zone (SED and FGO) and increasing sulphide contents within the units closest to the ore zone (CGO transitioning to SMSU).

Acid Potential (AP) of the sediment samples are 1.9 and 2.2 (t $CaCO_3$/1000 t) while AP of the FGO samples ranged from 9.4 to 12.5. AP was 6.6 for the CGO sample representative of development rock and 62.8 for the CGO sample representative of CGO rock presently included in the Mineral Resource estimate, but not the mine plan

Neutralization Potential (NP) for the sediment samples were 42 and 28 (t $CaCO_3$/1000 t) and ranged from 68 to 338 for the FGO samples. NP was 64 for the CGO sample representative of development rock and 253 for the CGO sample representative of CGO rock presently included in the Mineral Resource estimate but not in the mine plan.

The ratio of neutralization potential to acid potential (NPR) is used to determine a sample's propensity to generate acid conditions as per guidelines presented in MEND (2009) and as follows:

- NPR values less than 1 is an indication of PAG;
- NPR values between 1 and 2 indicates an uncertain potential; and
- NPR values above 2 indicates NAG.

NPR values for the sediment samples were 22.4 and 12.8. NPR values ranged from 7.25 to 34.89 for the FGO samples. NPR was 9.75 for the CGO sample representative of development rock and 4.03 for the CGO sample representative of CGO material presently included in the Mineral Resource estimate but not in the mine plan.

Based on the ABA results to date, the FGO, sediment and CGO samples representative of development rock are considered NAG. Nonetheless, pending further testing, this PEA assumes that all CGO development rock will be classified as PAG and consequently will not be used in the perimeter wall, but will be stored inside the CFTF.

Elemental (whole rock) analysis results were presented in Foth (2008); however, a detailed assessment was not completed. A review of the data in comparison to average crustal abundances presented in Price (1997) shows that some elements in the FGO and CGO had concentrations higher than the average crustal abundance, including: Co, Cu, Ni and Zn. It should be noted, however, that higher concentrations in the solid-phase does not necessarily identify elements that will be released at elevated concentrations when the material comes in contact with water.

SPLP analysis was performed on all samples. SPLP is designed to determine the potential mobility of both organic and inorganic analytes present in liquids, soils, and wastes. The concentrations of most metals in the leachate solutions were close to or below limits of detection.

The current geochemical characterization is considered sufficient for the purpose of the PEA considering the amount of development rock anticipated over the life of the mine. Additional geochemical characterization is required as the Tamarack North Project progresses into PFS and should be completed in accordance with regulatory requirements and accepted best practice guidelines. The ongoing characterization will be based on the updated mine plan for the Tamarack North Project with sampling representative of the anticipated tonnages of development rock, ore and tailings and will include additional static testing as well as long-term kinetic testing to further understand the ARD/ML potential. The

ARD/ML potential inferred from the results of geochemical characterization will be used to identify rock units that may require special handling or treatment, in order to develop mine development management and mitigation strategies that minimize the Tamarack North Project's effect on the receiving environment during construction, operations, closure and into post-closure.

18.6.4 CFTF Concept Design and Design Criteria

The general arrangement plan of the CFTF is shown in Figure 18-1. The CFTF will be rectangular and will cover a footprint area of approximately 25 acres (101,000 m^2 or 1.08 million ft^2). The maximum facility height will be 16 m (53 ft), and its key component features will include:

- **Base Grade** – 1% slope from NW corner to SE corner, to collect runoff water and seepage;
- **Base Liner System** – composite liner as explained in Section 18.6.5 below;
- **Perimeter Wall** – to protect the interior of the CFTF, and will provide stability, erosion protection, and dust control – refer Section 18.6.6 below;
- **Perimeter Berm and Ditch** – at the CFTF perimeter, about 5 m from the toe of the first perimeter wall lift. The berm will be 1 m high, and will be used for anchoring the base liner system and for creating a perimeter ditch – refer to Section 18.6.7 below;
- **Access Ramp** – at the SE corner of the facility, to access the surface of the co-disposal area. The ramp will be 15 m wide and have a 10% slope. Refer to Section 18.6.8 below;
- **Co-Disposal Area** – Filtered tailings from the processing plant trucked to co-disposal area, placed in thin lifts and compacted. The tailings will be co-disposed with adjacent layers of development rock – refer to Section 18.6.9 below.

The design criteria for the CFTF is summarized in Table 18-3.

Table 18-3: Key CFTF Design Criteria

Parameter	Description
General	
Life of Mine	7 years
Storage Capacity	880,000 m^3
Base Grade	Continuous to collect leachate by gravity to external pond
Base Liner	Composite liner with a leachate collection system
Perimeter Wall	Constructed progressively using NAG development rock in 2 m lifts
Perimeter Berm	To direct 1 in 100-year, 24-hour storm runoff water from exterior slopes of perimeter wall to external pond
Closure Cover	Composite liner system with drainage layer and soil layer for vegetation growth
Filtered Tailings	
Dry Density	1.70 t/m^3
Moisture Content	15% w/w
Development Rock	
NAG Dry Density	1.94 t/m^3
PAG Dry Density	2.03 t/m^3
PAG Porosity	0.3
Voids to be Filled with Tailings (PAG)	50%

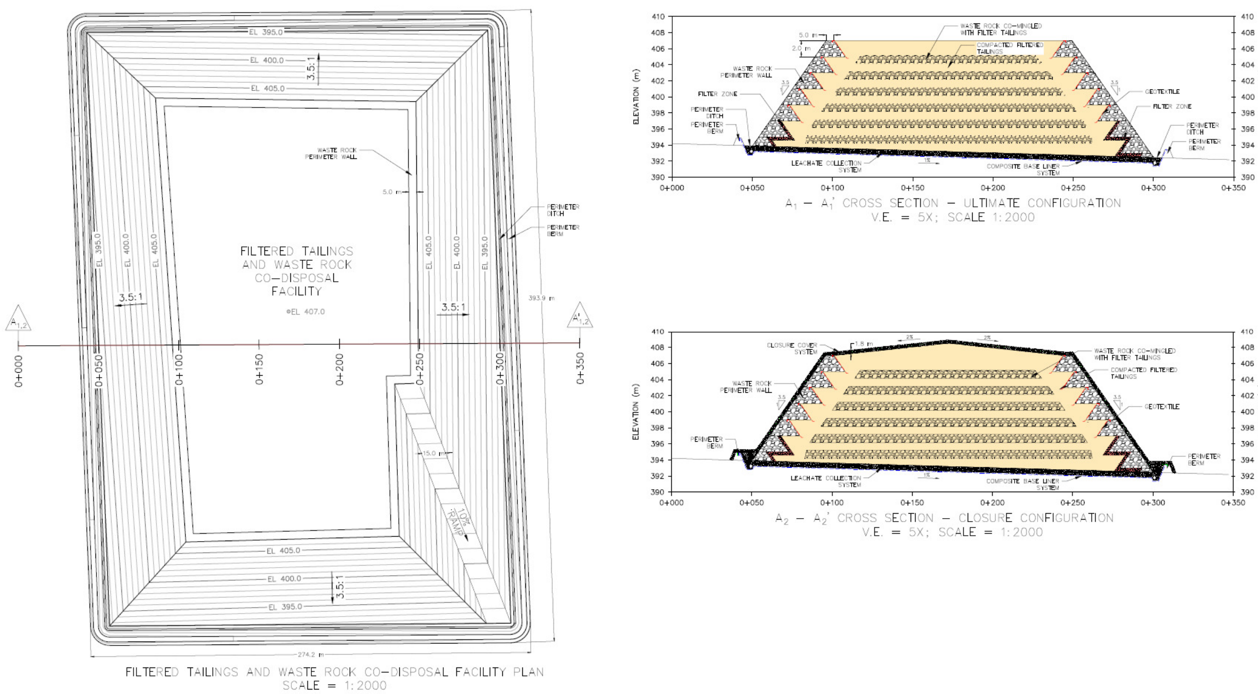


Figure 18-2: General Arrangement Plan of the CFTF

18.6.5 CFTF Base Grade and Base Liner

The base of the CFTF will be graded to provide a 1% slope to allow runoff water collected by the perimeter ditch and seepage collected by the leachate collection system to flow by gravity to the SE corner of the facility and eventually into one of the temporary, lined water storage ponds. The base grade will be prepared through a cut-to-fill operation and care will be taken to ensure an even base.

A composite liner will be provided over the finished base grade of the CFTF, which will consist of the following, from bottom to top:

- 3.5 kilograms per square metre (kg/m^2) Geosynthetic Clay Liner (GCL);
- 1.5 mm HDPE geomembrane liner;
- 0.3 m thick <6.4 mm aggregate leachate collection layer;

- 330 g/m^2 Non-woven filter geotextile;
- 0.3 m thick OB soil protection layer.

The geomembrane liner will serve as a hydraulic and a diffusion barrier against contaminant transport from the CFTF into the environment. The GCL will act as a back-up hydraulic and diffusion barrier in the unexpected event of deterioration of the geomembrane liner. Both the HDPE geomembrane liner and the GCL will be anchored on crest of the perimeter berm.

A filter geotextile will be provided over the leachate drainage layer to reduce the potential for clogging of the leachate drainage layer with fines from the overlying tailings and from the soil protection layer. The protection soil provided over the leachate drainage layer is to reduce the potential for the liner system to be damaged during the initial placement of the filtered tailings and co-disposing of filtered tailings and development rock.

A perimeter trench containing perforated pipes and coarse aggregate will be provided to convey the leachate collected from the base of the CFTF into the external pond. A thermal berm will be provided over a portion of the perimeter trench to protect the leachate pipeline from freezing during winter months. The thermal berm will be constructed using NAG development rock.

18.6.6 Perimeter Wall

The interior of the CFTF will be protected using the perimeter wall. The wall will be constructed using NAG development rock. The wall will be provided for stability, erosion protection and dust control. The wall will have a 5 m wide crest and 3.5H:1V interior and exterior side slopes. The wall will be constructed in 2 m high lifts in the upstream construction method.

The interior side slopes of the bottom two lifts of the perimeter wall will have a transition zone and a filter zone. For the remaining lifts of the perimeter wall, a geotextile will be provided between the development rock and the tailings to act as a filter. These zones will filter out the tailings from the seepage water that will flow through the perimeter wall into the perimeter ditch.

18.6.7 Perimeter Berm and Ditch

A 1 m high berm will be provided at the perimeter of the CFTF approximately 5 m from the toe of the first perimeter wall lift. The berm will be used for anchoring the base liner system and also for creating a perimeter ditch. The ditch will direct the 1 in 100 year, 24 hour storm runoff water from the exterior slopes of the perimeter wall into the temporary water storage ponds.

18.6.8 Access Ramp

A ramp will be provided on the SE end of the facility to access the top surface of the CFTF. The ramp will be 15 m wide and will have a 10% slope.

18.6.9 Co-disposal Area

Tailings will be dewatered to 15% moisture content (metallurgical moisture content, preliminary pending geotechnical testing) using the filtration plant, which will be located within the processing plant building. The filtered tailings will be trucked to the co-disposal area. The filtered tailings will be placed in thin lifts and compacted adjacent to layers or zones of NAG and PAG development rock. Conventional vibratory rollers will be used to compact the filtered tailings. The PAG development rock will be placed a minimum of 30 m from the perimeter wall of the CFTF to mitigate the risk for ARD and ML. Where possible filtered tailings will also be co-mingled with the PAG development rock. Dust suppression covers will be installed on all trucks. Dust will be contained by the perimeter wall height that will exceed the material contained in the co-disposal area.

18.6.10 CFTF Closure Cover

At closure, the top of the co-disposal area will be regraded to have a 2% crown. This will create a stable post-closure landform that will easily shed runoff water.

The perimeter wall and the top of the CFTF will be provided with a closure cover system which will consist of the following, from bottom to top:

- 552 g/m^2 Non-woven cushion geotextile;
- 0.6 m thick random OB liner bedding;
- 3.5 kg/m^2 GCL;
- 1.5 mm linear low-density polyethylene (LLDPE) geomembrane liner;
- 0.3 m thick <9.5 mm coarse aggregate drainage layer;
- 330 g/m^2 Non-woven filter geotextile;
- 0.45 m thick OB soil for root penetration;
- 0.15 m thick topsoil;
- Vegetation with native grass.

The closure cover will also include chutes and ditches to collect runoff water into sedimentation ponds prior to release to the environment. The objective is to allow the site to be left in a state where the only water produced is due to run-off.

18.6.11 CFTF Water Balance

The design of the temporary water ponds and Water Treatment Plant require an estimation of the contact water input from the CFTF.

The three main sources of contact water from the CFTF will be:

- Precipitation that penetrates the CFTF and reports to the leachate collection system over the LOM;
- Runoff water from the perimeter wall that reports to the perimeter ditch;
- Direct runoff water from the perimeter ditch.

The contact water (leachate and runoff) from the CFTF have been estimated using the Hydrogeologic Evaluation of Landfill Performance (HELP) model (Schroeder et al., 1997).

The contact water (direct runoff) from the perimeter ditch of the CFTF has been estimated using an assumed runoff coefficient for the ditch lining.

The HELP model integrates many input data, such as: precipitation, temperature, solar radiation, relative humidity and wind speed when predicting the contact water generation rate from the CFTF. The default climatic data available within the HELP model database for the Duluth weather station has been used in the analysis.

The CFTF cross-section used in the HELP model include the following layers: an 8 m thick layer of filtered tailings (half of the maximum design thickness of the CFTF to represent an average contact water generate rate), a 0.3 m thick soil protection layer, a 0.3 m thick granular leachate collection layer, and a 1.5 mm thick HDPE geomembrane liner. A 1% longitudinal slope was assumed for the granular leachate collection layer.

The preliminary material properties assumed for the various layers of the CFTF cross-section are summarized in Table 18-4. For all layers, the initial water content was taken as equal to or slightly greater than the field capacity water content, which is the water content at which the soil/tailings can no longer absorb/retain additional moisture under free drainage. This provides a conservative contact water generation rate as it represents a steady-state saturated flow, ignoring the time to reach the field capacity water content.

Table 18-4: HELP Model Input Parameters

Input Parameters	Unit	Tailings	Protection Layer	Drainage Layer	HDPE Liner
Thickness	m	8.0	0.3	0.3	0.0015
Total porosity	-	0.42	0.40	0.40	-
Field capacity	-	0.31	0.03	0.03	-
Wilting point	-	0.18	0.01	0.01	-
Hydraulic conductivity	cm/s	$1x10^{-5}$	0.3	1.4	$2x10^{-13}$

The average contact water generation rate from the CFTF is estimated at approximately 341 mm/year. The generation rate represents the sum of the runoff from the perimeter wall (327 mm/year) plus the leachate collected by the leachate collection layer at the base of CFTF (14 mm/year). The catchment area of the CFTF, excluding the perimeter ditch will be approximately 94,935 m^2. Therefore, on annual basis the external pond will receive approximately 31,044 m^3 of runoff water from the perimeter wall of the CFTF and approximately 1,329 m^3 of leachate from the leachate collection layer at the bottom on the CFTF.

The catchment area of the perimeter ditch will be approximately 6,341 m^2. Assuming a runoff coefficient of 0.9, the total annual runoff water that will generate from the perimeter ditch of the CFTF is approximately 4,302 m^3.

The total estimated contact water from the CFTF and perimeter ditch into the external water pond is summarized in Table 18-5 below.

Table 18-5: Annual Contact Water Volumes from CFTF to External Pond

Contact Water Source	Volume (m^3/year)
Runoff from CFTF Perimeter Wall	31,044
Leachate at CFTF Base	1,329
Runoff from CFTF Perimeter Ditch	4,302
TOTAL	**36,675**

18.7 Tamarack North Project Preliminary Water Balance

18.7.1 Water Sources and Water Storage

A preliminary water balance was developed, to account for major water streams through the mine, processing plant, tailings management area, and paste backfill facility.

The following data were used for the calculations:

- Concentrator mass balance water quantities estimated in the course of process design;
- Quantities of water from the CFTF were advised by Golder as shown above;
- Quantities of mine water production were estimated as explained in Section 16.4;
- Quantities of mine water required were estimated as explained in Section 16.9.3.

The calculations focused on maximizing the reuse of water, and excluded any water for domestic use on site. The results are summarized below:

- Net water required at surface for the processing plant was calculated by estimating the total water requirement for all processes less water recycled from thickening and filtering, resulting in a net water requirement of 103 gpm at plant capacity. Note that 86% of this water will be recycled;
- Water collected from the CFTF is estimated at 36,675 m^3 per year (20 gpm);
- Water required by underground operations at plant capacity is estimated at 188 gpm (refer Section 16.9.3 for a summary of the method of calculation);
- Water expected to be produced from underground operations were calculated as explained in Section 16.4.

The net result per year is shown in Table 18-6 below.

Table 18-6: Net Water Balance by Year

Production Year	1	2	3	4	5	6	7
	gpm						
Processing Plant							
Water required for processing	133	774	774	774	774	381	134
Water recycled from processing	115	671	671	671	671	330	116
Water deficit (at the processing facility)	(18)	(103)	(103)	(103)	(103)	(50)	(18)
Co-disposed Filtered Tailings Facility (CFTF)							
Water run-off from the CFTF	20	20	20	20	20	20	20
Water deficit at surface	2	(83)	(83)	(83)	(83)	(30)	2
Water used and produced at the mine (underground)							
Water required	(188)	(188)	(188)	(188)	(188)	(93)	(33)
Potential cumulative water production	109	228	337	406	475	525	525
Net water deficit/(surplus) underground	(79)	39	148	218	287	432	492
TOTAL	**(77)**	**(43)**	**66**	**135**	**204**	**402**	**494**

Further work is necessary to better forecast mine water production and treatment.

18.7.2 Temporary Water Storage Ponds

An estimated 1.4 million gallons of water will need to be stored in water storage ponds, which amounts to approximately the water storage capacity of 2 Olympic sized swimming pools.

During PFS consideration should be given to construction methods and the number of temporary water storage ponds that could initially serve as water collection ponds during construction.

All water pumped from the water storage ponds will be treated at the Water Treatment Plant from where it will be pumped to a process water tank for re-use.

19. MARKET STUDIES AND CONTRACTS

19.1 Market Analysis

For this PEA, it is assumed that separate Ni and Cu concentrates will be sold and shipped to smelters in North America. Treatment and refining charges, metal payability and settlement terms have been estimated based on confidential information received by Talon and input from market participants.

The average annual concentrate production, excluding ramp-up and ramp-down years, is forecast to be approximately 79,500 dry metric tonnes (dmt) of Ni concentrate and 18,300 dmt of Cu concentrate.

The estimated typical grade of the two concentrates are expected to be as follows:

Table 19-1: Composition of Ni and Cu Concentrates

	Ni Concentrate	Cu Concentrate
Moisture	8.0%	8.0%
Ni	14.5%	n/a
Cu	0.79%	28.9%
Au	n/a	2.23 g/t

The intention is to sell all concentrates under long-term contracts directly to smelters. Both the Ni and Cu concentrates are expected to be of clean quality with low levels of impurities.

19.2 Treatment Costs and Refining Costs

The Tamarack Ni and Cu concentrates will be sold directly to smelters or to traders in North America, Europe, and Asia. Based on metallurgical testing results to date, both the Ni and the Cu concentrates are of clean quality with low levels of impurities and good by-product credits. DRA has reviewed the smelter terms, the terms for the payment of metal, and the deductions for treatment and refining, and applied appropriate considerations in the economic model. Based on a review of publicly available information regarding smelter contract terms, DRA is of the opinion that the smelter contract terms, as applied in the economic model, are typical of the industry.

19.3 Transportation

Ni and Cu concentrates will be transported by rail using the railway line that crosses the property. Based on information provided by a national railway company, Talon has estimated all-in transportation costs by rail from the mine site to smelters as follows:

Table 19-2: All-in Transportation Costs

	Ni concentrate	Cu concentrate
All-in transportation cost	US$90/tonne	US$100/tonne

19.4 Metal Prices

Base case metal prices were based on analyst consensus long-term “real” (i.e. without inflation) prices as well as current markets, forecasts and reports in the public domain. The metals that will be sold are openly traded on terminal markets such as the London Metal Exchange (LME), the London Platinum and Palladium Market, the New York Mercantile Exchange (NYMEX) and the London Bullion Market.

The base case financial analysis of the Tamarack North Project uses the following estimated real metal prices. Alternative metal price scenarios were also considered.

Table 19-3: Assumed Real Metal Prices

	Unit	Low	Base case	Incentive
Ni	US$/lb	$6.75	$8.00	$9.50
Cu	US$/lb	$2.75	$3.00	$3.25
Co	US$/lb	$20.00	$30.00	$40.00
Pt	US$/oz	$1,100	$1,100	$1,100
Pd	US$/oz	$800	$800	$800
Au	US$/oz	$1,200	$1,200	$1,200

Incentive case metal prices were based on DRA's and Talon's estimation of real metal prices during the 2020's and 2030's (being the period when the Tamarack North Project is expected to be in operation) by referencing current markets, forecasts and reports in the public domain.

There is expected to be a shortage of supply of Ni and Co and to a lesser degree Cu due to an increase in demand from battery manufacturers and the automotive industry for electric vehicles (EV). There is forecast, in particular, to be a shortage of refined Ni also known as "Class 1" or LME-grade Ni which is precisely the Ni that the Tamarack North Project will produce. Refined Ni is used to make Ni sulphate, the preferred feedstock for batteries. Thus, without refined Ni, EVs cannot be produced.

As a result, new Ni and Co mines will need to be constructed to meet demand. Most Ni and Co projects, however, are not economic at current metal prices, so therefore, prices will need to rise beyond a price that covers operating costs ("marginal cost pricing") to a price that covers all of operating costs, capital costs and a reasonable return on capital invested ("incentive pricing") in order to "incent" the construction of new Ni/Co projects.

As it relates to Ni, Mining.com and the internationally accredited mining and metals consultancy Wood Mackenzie states:

> Finding enough Ni raw materials for battery-sulphate producers "is likely to be a considerable challenge post-2025" according to WoodMac and the firm has a price prediction to match the anticipated supply problem: The Ni market starts to need additional Ni from unidentified resources in 2023, we envisage prices reaching an annual average peak of US$28,700/t (US$13.00/lb) by 2022.

The low metal price case was selected based on a conservative estimate of long-term prices.

20. ENVIRONMENTAL CONSIDERATIONS AND PERMITTING

20.1 Introduction

The Tamarack North Project will be subject to state and federal level environmental review and permitting processes which are described in Section 20.6 and Section 20.7. Throughout the processes, Talon will illustrate that the Tamarack North Project will avoid or mitigate potential impacts to the environment in accordance with regulatory requirements. Additional data collection beyond the baseline studies completed to date will be completed to support these processes.

20.2 Baseline Studies

Baseline studies to characterize existing physical and biological conditions have been conducted since 2006. A description of baseline studies conducted to date is provided in Table 20-1. Additional baseline and environmental engineering studies will be required to support project siting, design, and environmental review and permitting efforts.

Table 20-1: Existing Baseline Studies

Project Component	Scope of Work
Hydrogeology	The hydrostratigraphic units in the Tamarack North Project area consist of unconsolidated glacial deposits with a typical thickness of >100 ft overlaying Precambrian bedrock. Onsite testing indicates that the hydraulic conductivity of the glacial deposits, which consist of a complex sequence of sand and gravel, clay, and silt, is generally significantly higher than the hydraulic conductivity of the bedrock. Groundwater elevations and groundwater quality have been monitored regularly since 2008. The monitoring network includes a total of twelve monitoring wells constructed in the unconsolidated glacial deposits. The monitoring program details have varied somewhat since 2008, but in recent years has included quarterly groundwater elevation measurements in each of the wells and semi-annual groundwater sample collection from a subset of eight monitoring wells. The groundwater flow direction is generally from E to W across the site area. There are no permanent monitoring wells constructed in the bedrock. Groundwater conditions in the bedrock have been assessed using BH geophysical techniques and packer testing at four exploration BHs. Discrete groundwater samples were collected from the bedrock during packer testing to characterize groundwater quality. This information was used to estimate the potential mine water inflows from the mine for purposes of completing a water balance (refer Section 18.7.2).

Hydrology	Surface water monitoring stations were established at a series of stream sections and lake sites. A surface water monitoring program commenced in 2008 and data was collected related to measurement of flow, field water quality measurements, and collection of surface water samples for analysis. A total of 21 surface water monitoring locations were sampled. The results from this data were used to complete three mine access trade off studies. (refer Section 16.8.1).
Geochemistry	Fourteen drill core samples have been analyzed. The samples were selected to comprise rock types, spatial distribution, and sulfur content at site. Rock types tested included: • fine-grained olivine orthocumulate (4) • coarse-grained olivine orthocumulate (2) • sedimentary units (2) • semi-massive sulfide units (3) • saprolite (2) • massive sulfide unit (1) Tests included static ABA tests targeting similar but slightly variable information on the potential for the rock samples to generate or neutralize acidity. Whole rock analysis of around 60 elements was conducted on all 14 samples. A SPLP leach was conducted on all 14 samples; major and trace metals were analyzed in the leachate. The results from these studies were used for purposes of designing an innovative CFTF: Refer Section 18.6.
Wetlands	Wetland delineation and evaluation studies in accordance with federal and local guidelines and manuals occurred in 2008 at the site layout area. Wetland boundaries were mapped and reviewed with local regulatory staff. A 120-acre study area was initially evaluated and then it was expanded to a 580-acre study area. Based on the results from these studies, the conceptual site layout (refer Section 18.3) has been partially placed on upland (23 acres) to minimize the impact on wetlands (30 acres). The breakdown by area of the Project site is shown below: **Description** — **Acres** Upland — 23.0 Sedge meadow — 17.2 Alder thicket — 5.0 Shrub carr — 4.8 Deep marsh (in man-made pond) — 2.4 **Total** — **52.4**
Vegetative Communities	A survey of a 322-acre study area of vegetative communities occurred in 2008 over the site layout area. Flora was inventoried onsite and vegetative communities and habitats were mapped by type within the study area. The area where the conceptual site layout is located (refer Section 18.3) was delineated as Fallow Farm Fields/Young Pine Plantation. Satellite imagery dated 1991 suggests that much of this vegetative community had previously been farmed. This community is now dominated by scattered, young red pine (*Pinus resinosa*), white pine (*Pinus strobus*), and black spruce. The herbaceous stratum is dominated by goldenrods (*Solidago spp.*), pearly everlasting (*Anaphalis margaritacea*), and a host of non-native species such as

	reed canary grass, redtop (*Agrostis gigantea*), smooth brome (*Bromus inermis*), ox-eye daisy *(Chrysanthemum leucanthemum*), clovers (*Trifolium spp.*), yarrow (*Achillea millefolium*), timothy (*Phleum pretense*), and tall buttercup (*Ranunculus acris*). Man-Made Pond (2.38 acres): A man-made pond exists within the western portion of the study area. Its herbaceous community is relatively diverse, with no species exhibiting complete dominance. The small farm pond, directly E of the farmstead lot, is somewhat older and has a more mature herbaceous community. Thick stands of narrow-leaved cattail (*Typha angustifolia*) dominate the littoral zone, and node pondweed (*Potamogeton nodosus*) covers much of the water surface. The vegetative communities that occur in the study area are characteristic of much of northeastern Minnesota, including Aitkin County. No unusual or uncommon natural vegetative communities were identified within the study area. Two invasive plant species (reed canary grass and narrow-leaved cattail) were abundant within several of the habitat types. No RTE plant species or their potential habitat was observed. The approximate area size by Vegetative community type is shown below:

Description	Acres
Pine Plantation	38.6
Homestead	7.1
Man-made Pond	2.8
Northern Wet-Mesic-Hardwood Forest	1.7
Northern Poor Fen	1.4
Northern Alder Swamp	0.8
Alder Thicket	0.03
Total	**52.4**

Rare, Threatened & Endangered Plant Species	A survey for Rate Threatened and Endangered (RTE) species occurred in 2008. The survey study area covered the site layout area, except for a farm residence and adjacent buildings. The MNDNR maintains a restricted geographic database of documented occurrences of threatened, endangered, and special concern species in Minnesota. An authorized database search for RTE species that have been known to occur within several miles of the study area was conducted. This information and Minnesota's entire published list (MNDNR Division of Ecological Resources 2008) of RTE species were utilized while conducting the RTE field investigation within the study area in August 2008. The site was carefully surveyed using a series of thorough meander transects within all natural vegetative communities and other habitat types. No federally listed or state listed threatened, endangered, special concern plant species or other rare natural features were documented within the study area. Because all habitat types documented within the study area are relatively common in Aitkin County and the associated ecoregion, the presence of RTE species would be unlikely.

20.3 Addressing Environmental Sensitivities Through Implementing BAT

BAT have been implemented in the handling of mine waste, most notably:

- Development rock (from the shaft, levels, ramps, cross-cuts and drifts);
- Tailings that are produced because of producing the Ni and Cu concentrates.

The first priority was to determine if a HS tailings stream could be produced. Metallurgical testing has proven that this is possible. Consequently, a LS tailings stream can be produced separately (refer Section 17.3.2).

A paste backfill study was commissioned to determine if and how much of the HS tailings and LS tailings can be mixed with cement and stored in mined out, underground voids. The results of this study showed that 100% of HS tailings and 37% of LS tailings can be blended with cement and cured underground (refer Section 16.2).

A number of studies were commissioned to investigate the use of BAT in regard to development rock and the remaining LS tailings (refer Section 18.6). These studies lead to the development of an innovative CFTF which offers significant environmental and operating advantages over separate tailings storage and development rock storage facilities, including:

- Reduced risk of failure due to the implementation of dry-stacking instead of a traditional tailings dam;
- A major reduction in the waste facility footprint: Surface land required for the storage of LS tailings that are not placed underground with cement, will be reduced by 40% versus traditional methods as portion of the development rock void space will be utilized as LS tailings storage space;
- Improved tailings stability and reduced dusting compared to standalone filtered tailings facility without co-disposal with development rock;
- At closure, the CFTF will be covered with a closure cover system. This will limit the amount of infiltration into the CFTF post closure;

- A significant reduction in fresh water requirements. In fact, 86.7% of water required by the processing plant will be recycled water of which 30% will be as a direct result of using a CFTF.

Refer Section 18.6 for a more detailed discussion of the application of the development rock, the FGO and SEDs from the shaft and levels and the remaining LS tailings.

In order to minimize the Tamarack North Project footprint three different mine access methods were considered (refer Section 16.8.1). As a result, mine access will be by a small diameter mine shaft, which reduces the surface expression of the excavation area by 99.9% compared to a box-cut and ramp access method. Consequently, the total surface area required for all facilities and the CFTF is limited to approximately 53 acres.

By implementing these BATs, Talon is addressing environmental sensitivities, such as:

- Potential mitigation for lost habitat of state and federal protected species;
- Potential wetland impacts and need for wetland impact mitigation;
- Potential generation of ARD and ML;
- Potential impacts to surface and ground water quality;
- Potential drawdown of surface water levels and flows.

20.4 Groundwater

Groundwater in the surficial aquifer in the region is generally located near the surface and of high quality. The groundwater is hydraulically connected to surface waters in the area, although the degree of connectivity has not been determined. Construction and operation of mine features have the potential to impact water quality. Dewatering associated with construction has the potential to impact surface waters. Construction will therefore employ techniques such as a freeze-wall during construction of the small diameter shaft, which will be cemented down to the bedrock.

20.5 Water Management

A water management plan will be developed, detailing a strategy for managing water in a manner consistent with environmental requirements related to both water quantity and water quality. The water management plan will be designed to avoid, minimize and mitigate adverse changes in surface water hydrology and confirm compliance with surface and groundwater water quality standards.

A preliminary annual water balance was developed for purposes of this PEA (refer Sections 16.4, 16.5 and 16.9.3).

Based on this water balance, the Tamarack North Project is expected to have a potentially negative water balance during the first two years of production, followed by potentially a positive water balance over the following five years of production (refer Section 18.7). Further geotechnical and hydrogeological work is needed to assess the impact of fracture sealing.

Further work is required to assess potential water sources. Trade-off studies of Water Treatment Plant options should be conducted during the PFS.

20.6 Environmental Review Process

State-level and federal-level environmental review would be completed through an Environmental Impact Statement (EIS) process subject to the Minnesota Environmental Policy Act (MEPA) and National Environmental Protection Act (NEPA) requirements for nonferrous mines.

Typically, a Memorandum of Understanding (MOU) between the lead agencies would be entered to prepare a single joint EIS. The lead agencies would include the Department of Natural Resources (DNR) as the MEPA Responsible Government Unit (RGU) (Minnesota Rules, part 4410.2000, subpart 2) and likely the USACE as the federal lead agency. Additional cooperating agencies may also be identified and could potentially include the US EPA, US Fish and Wildlife Service, and Fond du Lac Band of Lake Superior Chippewa.

The four major steps in the EIS environmental review process are:

- Scoping of the EIS;
- Preparation of the draft EIS;
- Preparation of the final EIS; and
- Documentation of the Record of Decision (ROD) regarding the adequacy of the EIS.

The EIS environmental review process invites participation from the public and interested stakeholders. A brief summary of each of the four major steps, as described in Minnesota Rules, chapter 4410, is provided in the subsequent sections. A similar process is required under federal rules (consistent with the NEPA) however there are differences related to scoping procedures, time frames, decision processes, etc. The environmental review process has not yet been initiated by Talon.

20.6.1 Scoping

The purpose of the scoping process is to reduce the scope and bulk of an EIS. During the scoping process, potentially significant issues relevant to the proposed project are identified. Additionally, the level of detail, content, potential alternatives to the proposed action (project), procedures for assessment of cumulative impacts, time table for preparation, and preparers of the EIS, as well as the permits for which information will be developed concurrently with the EIS, are determined during scoping. A Minnesota Environmental Assessment Worksheet (EAW) must be filed for all projects that require an EIS (Minnesota Rules, part 4410.2100, subpart 2); the EAW provides a basis for preparation of a draft and subsequent final Scoping Decision Document (SDD). Under the Council on Environmental Quality NEPA regulations, an agency has the discretion to accept comments on the EIS process from the publication of the Notice of Intent through the release of a final EIS. Typically, scoping comments are received prior to the release of a draft EIS and incorporated in the draft EIS, whereas comments on the draft EIS are received after its release and incorporated into the final EIS. Therefore, stakeholders may provide suggestions for modification of the scope and analysis throughout the EIS process.

20.6.2 Draft EIS

A Draft EIS would be prepared by the RGU consistent with Minnesota Rule, parts 4410.0200 through 4410.6500 and in accordance with the final SDD. The EIS would describe the proposed project, assess the potential environmental, economic and sociological impacts of the proposed project and consider reasonable alternatives or modifications to avoid adverse impacts. Minnesota Rules provide for robust evaluation of alternatives to the proposed action, including alternative size, configuration, location, etc. to avoid and minimize potential adverse impacts of the proposed action. The Draft EIS would be distributed and made available for review and comment by the public and other government agencies. The RGU would hold an informational meeting.

20.6.3 Final EIS

The Final EIS would ultimately identify the likely impacts of the Tamarack North Project as well as alternatives that may lessen or mitigate adverse impacts. It would respond to the comments on the Draft EIS consistent with the scoping decision. The RGU would discuss at appropriate points in the Final EIS any responsible opposing views relating to scoped issues which were not adequately discussed in the Draft EIS and would indicate the RGU's response to the views.

20.6.4 Record of Decision (ROD) and/or Adequacy Decision

The EIS process would conclude with a federal ROD and state Adequacy Decision that would explain each agency's decision, summarize the alternatives considered, and provide the plans for mitigation and monitoring, as necessary.

20.7 Permitting Requirements

After the environmental review process, the Tamarack North Project would be required to obtain applicable local, state, and federal permits. A preliminary list of permits that may be required for the Tamarack North Project is provided in Table 20-2. Permitting requirements may change if additional permitting requirements are identified within the environmental review process and/or as the Tamarack North Project siting and design progresses. Generally, final permitting requirements include a public comment period for members of the public and to provide input on the Tamarack North Project and its permits. Talon has not initiated permitting efforts to date.

The permitting requirements with the greatest potential to impact the Tamarack North Project's design, schedule, or cost include the Permit to Mine from the DNR (refer to Section 20.7.1), the National Pollutant Discharge Elimination System (NPDES)/ State Disposal System (SDS) Permits from the MPCA (refer to Section 20.7.2), the Air Permit from the MPCA (refer to Section 20.7.3), and Department of the Army Section 404 Permit from the USACE (refer to Section 20.7.4).

Dependent upon the final permitting requirements, there may be additional opportunities for members of the public to provide input on the Tamarack North Project and its permits (e.g., the county zoning permit may have a public hearing component). Talon has not identified any additional social or community related requirements and plans for the Tamarack North Project. Negotiations or agreements with local communities for the Tamarack North Project have not been initiated.

Table 20-2: Required Permits

Agency	Permit or Approval	Permit Rationale and Applicable Regulation
Federal		
USACE	Clean Water Act – Section 404 permit	For impacts to wetlands or waters under the jurisdiction of the USACE under the CWA, 40 CFR Part 230: Section 404(b)(1)
USACE	Tribal Government to Government Consultation	Required for impacts to tribal resources. USACE Tribal Consultation Policy, 1 Nov 2012.
US Fish & Wildlife Service	Endangered Species Act – Section 7 Compliance	Required for USACE to issue a Section 404 Permit. Applies to federally-listed species only. 50 CFR Part 402
SHPO	National Historic Preservation Act – Section 106 compliance	Required for USACE to issue a Section 404 Permit. NHPA Section 106-- 54 U.S.C. Section 306108.
State		
MNDNR	State Lease to Explore, Mine and Remove Nonferrous Metallic Minerals	Lease required to explore and develop nonferrous metallic minerals, including Ni. Minnesota Rules, chapter 6125.
MNDNR	Permit to Mine	Required to conduct a mining operation. Minnesota Rules, chapter 6132.
MNDNR	Dam Safety Permit (for temporary water storage ponds) – see Section 18.7.2.	Potentially needed for tailings basin dikes or similar structures. Permit is generally triggered if impoundment is greater than 6-ft high and has the capacity to store 15 acre-ft or more. Minnesota Rules, parts 6115.0300-6115.0520.
MNDNR	Work in Public Waters Permit	For projects that impact or modify wetlands, lakes, and other waters included on the State's Public Waters Inventory. Minnesota Rules, chapter 6115.
MNDNR	Threatened and Endangered Species Take Permit	Required if project has potential to take state-listed threatened or endangered species. Minnesota Rules, parts 6212.1800-6212.2300 and chapter 6134.
MNDNR	Water Appropriations permit	For projects that withdraw more than 10,000 gallons of water per day or 1 million gallons of water per year. Also for projects that divert or transport infested waters. Minnesota Rules, chapter 6115.
MNDNR	Minnesota Wetland Conservation Act Approval	Required for impacts to all wetlands that are not included on the State's Public Waters Inventory. Requires Wetland Replacement Plan as part of approval. Minnesota Rules, part 6132.5300.
MNDNR	Burning Permit (if needed for construction or land clearing)	If burning is proposed for land clearing or in advance of construction. Minnesota Statute 88.16.
MNDNR	Access Easement or Lease	Required to construct access road across State lands. Easements are issued for constructing and maintaining roads. Leases are issued for long-term right to use/occupy State land. Minnesota Statutes, 84.63, 84.631, and 85.015
MPCA	Section 401 Water Quality Certification	Required under the Clean Water Act for USACE to issue Section 404 Permit. Applies if project discharges from a point source to a USACE jurisdictional water.
MPCA	National Pollutant Discharge Elimination System and State Disposal System (NPDES/SDS) Permits	The NPDES Permit covers discharge of industrial waste water and stormwater from point sources into surface waters. The SDS permit covers construction and operation of waste water disposal systems discharging to surface waters and/or groundwater. A non-degradation analysis would be required to support this permit application. Minnesota Rules, part 7001.1035

Agency	Permit or Approval	Permit Rationale and Applicable Regulation
MPCA	NPDES/SDS General Construction Stormwater Permit	Required if construction will disturb more than one acre of land or if MPCA determines construction activities pose a risk to water resources. Minnesota Rules, parts 7090.2000-7090.2060.
MPCA	NPDES/SDS General Industrial Stormwater Permit	Applies to facilities where stormwater comes into contact with significant industrial materials that may result in polluted run-off. Minnesota Rules, Chapter 7090.3000-7090.3060.
MPCA	Solid Waste Permit	For disposal of solid waste, which generally includes solid, semisolid, and liquid wastes from industrial or mining facilities. A tailings basin may be considered a Type III disposal facility and could require this permit. Minnesota Rules Chapter 7035.
MPCA	Air Emissions Permit	Permit required for all facilities with sources of air emissions. Several types of air permits may apply, depending on facility-wide emissions estimates. Minnesota Rules, chapter 7007.
MPCA	General Storage Tank Permit for fuel tanks	Required for facilities that store more than 1,000,000 gallons and must obtain an Individual Permit. If storing less than 1,000,000 gallons but more than 1,100 gallons, must submit a notification to MPCA. Minnesota Rules, part 7001.4205.
MPCA	Hazardous Waste Generator License	For facilities that generate hazardous waste, a license is required. Minnesota Rules 7045.0225.
MPCA	Waste Tire Storage Permit (if needed)	Required if Project facilities accumulate more than 50 waste tires at any given time, typically associated with on-site equipment maintenance.
MDH	Permit for Non-Community Public Water Supply System	Required if the system is designed to serve at least 25 people, such as employees, on a regular basis. May require accompanying Wellhead Protection Plan.
MDH	Permit for Public On-site Sewage Disposal System	Required if on-site sewage disposal system is installed.
MDH	Radioactive Material Registration	Required for facilities intending to possess or use radioactive materials in such quantities that active control is required to assure safety.
Minnesota Department of Transportation/Surface Transportation Board	Railroad Spur Installation Approval	Approval for railroad spur installation may be needed, depending on Surface Transportation Board warrant analysis.
Local		
Aitkin County	Zoning Permit/Conditional Use Permit	May be required to acknowledge mine is an allowable use within zoned district(s).
Aitkin County	Building Permit	May be required for construction of buildings.
Aitkin County	Shoreland Permit	Required for work within shoreland areas of waters included on the State's Public Waters Inventory.
Aitkin County	Subsurface Sewage Treatment System Permit	Required to ensure septic systems effectively treat wastewater. Administered at local level according to MPCA regulations.
Town of Tamarack	Building Permit and/or Zoning Permit	If construction occurs within the incorporated area of the Town of Tamarack, local permitting requirements (e.g., a Building Permit or a Zoning Permit) may apply.
BNSF Railway	Design approval	For ancillary facilities (such as rail spurs) that connect to the main rail line.

20.7.1 Permit to Mine (DNR)

Pursuant to Minnesota Rules, chapter 6132, a Permit to Mine would be required and signifies a legal approval issued by the commissioner of the Minnesota DNR to conduct a mining operation. The purpose of the DNR Permit to Mine program is to control possible adverse environmental effects of nonferrous metallic mineral mining, to preserve natural resources, and to encourage planning of future land utilization (Minnesota Rules, part 6132.0200). Therefore, it is DNR policy that mining activities be planned and executed in a manner to reduce environmental impacts, mitigate impacts where unavoidable, and reclaim the mining area to a condition that protects natural resources and minimizes the need for maintenance to the extent practicable.

The nonferrous mining rules set forth in Minnesota Rules, chapter 6132 include a detailed procedure for obtaining a Permit to Mine, including requirements for:

- Mine waste characterization (Minnesota Rules, part 6132.1000);
- The contents of a Permit to Mine application (Minnesota Rules, part 6132.1100);
- Financial assurance (Minnesota Rules, part 6132.1200); and
- Annual reporting (Minnesota Rules, part 6132.1300).

Reclamation standards are further defined in Minnesota Rules, part 6132.2000 through part 6132.3200 and include standards for siting, buffers, reactive mine waste, OB portion of pitwalls, storage pile design, tailings basins, heap and dump leaching facilities, vegetation, dust suppression, air overpressure and ground vibrations from blasting, subsidence, corrective action, and closure and post-closure maintenance. These standards are accomplished through the use of appropriate mining methods, proper mine waste management, and implementing passive reclamation procedures that maximize physical, chemical, and biological stabilization of areas disturbed by mining, along with the use of active treatment technologies when necessary.

The requirements for financial assurance are also determined in the Permit to Mine application process. Financial assurance is designed to address the cost for reclamation of the Tamarack North Project, should the mine be required to close for any reason at any time, and includes closure and post-closure maintenance activities. The financial assurance requirements are reviewed and can be adjusted on an annual basis.

20.7.2 NPDES/SDS Permits (MPCA)

Permits with the intent to protect waters for uses such as drinking water, aquatic life, and recreation would be required under the NPDES/SDS program (refer to Minnesota Statutes, Section 115.04), which is administered by the MPCA.

The NPDES/SDS program would apply to the construction and operation of wastewater disposal systems discharging to surface waters and/or groundwater. Pursuant to water quality standards of receiving and downstream waters, the permit would establish wastewater discharge effluent limitations and monitoring requirements. A non-degradation analysis would be required at the time of the application. The objective of the non-degradation analysis is to achieve and preserve the highest possible water quality in surface waters, such as lakes, streams, and wetlands, by maintaining and protecting existing uses. Where applicable, the analysis will document how degradation of high water quality is minimized whenever possible and only allowed for the purpose of important economic or social development.

The NPDES/SDS program would apply to wastewater and stormwater discharges from point sources into surface waters. Potential project discharges requiring permit coverage may include mine dewatering, wastewater, tailings basin discharges, industrial stormwater, and construction stormwater. The NPDES/SDS permit would establish the surface water quality limits for the Tamarack North Project, the type(s) and level of water treatment, and best management practices to be implemented for control of discharges. Coverage for industrial stormwater discharges could either be included with the NPDES/SDS permit or applied for separately under the Industrial Stormwater General Permit. Additionally, a Construction Stormwater General Permit would require implementation of best management practices and permanent stormwater management techniques specific to managing stormwater run-off from construction sites. Water management during construction and operations would be required to be consistent with the requirements of the permits and would implement best management practices as planned for in the Stormwater Pollution Prevention Plans (SWPPPs).

20.7.3 **Air Permit (MPCA)**

For most sizable mining facilities, an air permit will need to be acquired before construction and operations can begin (40 CFR parts 52 and 70. Minnesota rules part 7007). Applicability of federal and state air permitting rules will need to be evaluated. These programs have been established to protect air quality as it relates to human health and the environment. The applicable rules depend on the type of process emitting the pollutants, the quantity of emissions, the types of pollutants emitted, and the affected air shed.

The air permit would provide the basis for the facility to demonstrate compliance with air quality related standards and associated regulations. Production, design and operational details are incorporated into the permit and are the basis for the permit emission calculations. Changes from these requirements and design basis would necessitate a permit amendment evaluation that may require changes to the permit. Permit amendments can range from minor to major levels of effort and time.

Depending on the type of air permit needed, the facility may need to perform a number of analyses to demonstrate compliance with applicable standards. Some of these could potentially include Class I modeling evaluation of facility impacts on air quality related values at wilderness areas, national parks and other similar air sheds and Class II modeling to demonstrate compliance with National Ambient Air Quality Standards (NAAQS). Federal and state rules may also mandate Best Available Control Technology or New Source Performance Standards and National Emission Standards for Hazardous Air Pollutants (NESHAPS). Other requirements may also include airborne dust management, evaluations of Hg emissions, and emission deposition on local water bodies and Air Emission Risk Analysis (AERA).

20.7.4 Wetland Permitting

A permit from the USACE for the discharge of dredged or fill material to waters of the US would be required under Section 404 of the Clean Water Act. Where project impacts to wetlands would be unavoidable, compensation (i.e., the construction, restoration or enhancement of wetlands) would be required as replacement for affected wetlands. The MNDNR regulates impacts to wetlands and other waters that are included on the state's Public Waters Inventory list. The Minnesota Wetlands Conservation Act (WCA) also requires that a state permit be obtained for impacts to wetlands. A Wetland Replacement Plan would be required and incorporated into the mining and reclamation plans for the Tamarack North Project under the Permit to Mine. Aitkin County will also require compliance with its wetland ordinances.

Applications for wetland impacts and an associated Wetland Replacement Plan would be submitted to the USACE, MNDNR, and Aitkin County under each entity's respective application process. Financial assurance could be part of the WCA permitting.

20.8 Planned End Use and Sustainable Development

Talon's strategy is to engage with stakeholders with the end in mind. A robust closure plan that engages stakeholders will therefore be developed starting at the pre-feasibility stage. Developing an understanding of stakeholder concerns, needs, and preferences will help shape plans that will avoid adverse environmental impacts while at the same time achieving common end goals.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The total estimated capital cost is US$182.51M is summarized in Table 21-1, of which US$174.31M is the initial cost required during the first 2.5 years prior to the start of production. The amounts include indirect costs and contingency which are detailed in the various sections further below.

Table 21-1: Tamarack North Project Capex Summary

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Mine	$72.44M	$21.38M	$93.83M
Process and Surface Facilities	$90.85M	$1.57M	$92.43M
Closure Costs	-	$6.25M	$6.25M
Salvage value of mill	-	($10.00M)	($10.00M)
Working capital	$11.01M	($11.01M)	-
Total*	**$174.31M**	**$8.20M**	**$182.51M**

*May not total due to rounding

21.1.1 Mine Capital Costs

The estimated mine capex of US$72.4M comprises shaft sinking and equipping, hoist and headframe installation, mine surface facilities, underground development and services.

As the mine will be developed and operated with mine contractors, no mine development or production fleet purchase will be necessary as the contractor will supply their own fleet which is included in development and production unit costs.

Sustaining capex estimated at US$21.4M includes ramp and waste lateral development spread over the six-year mine life.

Table 21-2: Mine Capex Summary

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Shaft	$29.51M		$29.51M
Underground Development	$23.02M	$21.18M	$44.20M
Equipment and Services	$2.35M	$0.20M	$2.55M
Sub-total	**$54.88M**	**$21.38M**	**$76.26M**
Indirect Costs (@ 10%)	$5.49M		$5.49M
Contingency (@ 20%)	$12.07M		$12.07M
Total	**$72.44M**	**$21.38M**	**$93.83M**

*May not total due to rounding

21.1.2 Process Plant Capital Costs

The estimated process and surface facilities capex of US$92.43M comprises the process plant, plant infrastructure (concentrator building, electrical substation and distribution, reverse osmosis plant, water supply system, and fire protection), co-disposal facility and paste backfill, other surface facilities (administrative office, maintenance shop, mine change house, surface warehouse, garages, security, and parking lots). The co-disposal facility is gradually built-up over time so some of its cost is forecast to occur in year two of the mine life.

Table 21-3: Process and Surface Facilities Capex Summary

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Process Plant	$33.80		$33.80
Plant Infrastructure	$11.82		$11.82
Co-disposal Facility and Paste Backfill	$7.55	$1.57	$9.12
Other Surface Facilities	$1.54		$1.54
Indirect Costs (33.5%)	$18.85		$18.85
Contingency (@ 23%)	$17.30		$17.30
Total*	**$90.85**	**$1.57**	**$92.43**

*May not total due to rounding

21.2 Other Capital Costs

Closure costs are estimated at US$6.25M and relate to a closure cover on the co-disposal facility and drainage equipment, removal of the process plant and other surface infrastructure facilities as well as land reclamation.

As the mill, equipment and facilities will only have been in operation for six years, it is assumed that the mill and other components of the Tamarack North Project will be able to be sold on the secondary and scrap markets for US$10M at the end of the mine life.

21.3 Operating Costs

The OPEX for the Tamarack North Project at the processing plant design capacity of 1,390 tpd are summarized in Table 21-4 below. In future studies, the processing costs may be reduced with a simplification of the process flowsheet.

Table 21-4: Operating Costs in US$/t of Mill Feed

Cost Category	**Operating Cost (US$/t of ore)**
Mining	$63.94
Processing	$18.87
Product Handling	$22.92
Filtered Tailings Facility and Paste Backfill	$2.50
General & Administrative	$10.00
Total OPEX	**$118.23**

*May not total due to rounding

21.3.1 Mine Operating Costs

The estimated mine OPEX average over the mine life is US$63.94 per tonne of ore for MSU, SMSU stopes and ore development. This average cost includes contract mining, backfill, stope development (in waste), hoisting, ventilation, and mine services costs. The OPEX also includes Mine G&A costs for geology, mine engineering and mine management.

A summary of the mine OPEX split by mining method is shown in Table 21-5 below.

Table 21-5: Mine Operating Cost Summary

Item	Estimated OPEX (US$/t of ore)
MSU Stopes	$87.92
SMSU Stopes	$40.11
Ore Development	$79.67
Average Cost	**$63.94**

21.3.2 Process Plant Operating Costs

A breakdown of the processing costs is provided in Table 21-6 and further details for the basis of the estimates is provided in the following sections.

Table 21-6: Processing Operating Cost Break-Down

Cost Category	Annual Cost US$/year	Unit Cost US$/tonne of Feed	% of Process Operating Costs
Labour	$3,611,250	$7.12	37.6
Electrical Power	$1,681,920	$3.32	17.6
Reagents	$1,459,139	$2.88	15.2
Grinding Media	$1,294,222	$2.55	13.5
Consumables	$1,206,980	$2.38	12.6
Spares & Miscellaneous	$321,861	$0.63	3.5
Total Processing Costs	**$9,575,372**	**$18.87**	**100.0**

*May not total due to rounding

21.3.2.1 Methodology

The operating costs have been estimated based on the design capacity of 1,390 tpd. The operating costs estimate considered pertinent metallurgical results and mass balance outputs.

21.3.2.2 Labour

Staffing has been established based on the resource requirements of plants with comparable size and unit operations. Staffing requirements were differentiated between mill operation, maintenance, technical, and administration.

21.3.2.3 Electrical Power

An electricity cost of US$0.06/kWh was utilized for cost estimation purposes, which is consistent with publicly available posted rates for the region. The total connected power was determined by summation of the connected power of all major mechanical equipment plus 35% for pumps and plant services. The total power drawn was estimated at 80% of the total connected power.

21.3.2.4 Reagents

The reagent dosages were established using the metallurgical data that was developed in a recent test program at SGS Lakefield. The reagents costs were then calculated using recent prices from reputable North-American reagent suppliers.

21.3.2.5 Grinding Media

The grinding media consumption was determined using a Bond abrasion work index that was projected from other comminution data, anticipated grinding media load, mill dimensions, and anticipated grinding energy. The unit costs for the grinding media were obtained from North-American grinding media suppliers.

21.3.2.6 Consumables, Spares and Miscellaneous

The consumables, which include all items except reagents and grinding media, were calculated as 15% of the labour, electrical, reagents, and grinding media costs. The allowance for spares and miscellaneous items was determined as 4% of the labour, electrical, reagents, and grinding media costs.

22. ECONOMIC ANALYSIS

22.1 Basis of Evaluation

DRA has prepared its assessment of the Tamarack North Project on the basis of a financial model, from which NPV, IRR, payback and other measures can be determined. NPV and IRR can assist in the determination of the economic value and viability of a project.

The object of the study was to determine the viability of the proposed mine and mill to mine and process the Tamarack North Project ore. In order to do this, the cash flow arising from the base case has been forecast, enabling a computation of the NPV and IRR. The sensitivity of this NPV and IRR to changes in the base case assumptions is then examined.

22.2 Economic, Taxation and Royalty Assumptions

22.2.1 Exchange Rates

All cost estimates have been forecast in US dollars and metal prices are in US dollars, therefore no exchange rate was required. All results are expressed in US dollars.

22.2.2 Inflation Rates and Escalation

All cost estimates have been prepared using constant, second quarter 2018 dollars, i.e. in “real” dollars without provision for inflation or escalation.

22.2.3 Weighted Average Cost of Capital

The weighted average cost of capital was determined based on many factors including location, characteristics of the project such as access to infrastructure, expected position on the global cost curve, project size and complexity, that the forecast is in “real” as opposed to nominal dollars and the metal prices used in the forecast.

Based on the above, DRA has used a 7% discount rate as the weighted average cost of capital for the Tamarack North Project as the base case. Alternative results at different discount rates are provided for comparative purposes.

22.2.4 **Metal Prices**

Base case, Low and Incentive metal prices were discussed in Section 19. Price assumptions are presented in Table 22-1 and are in "real" (i.e. without inflation) dollars.

Table 22-1: Assumed Real Metal Prices

	Unit	Low	Base case	Incentive
Ni	US$/lb	$6.75	$8.00	$9.50
Cu	US$/lb	$2.75	$3.00	$3.25
Co	US$/lb	$20.00	$30.00	$40.00
Pt	US$/oz	$1,100	$1,100	$1,100
Pd	US$/oz	$800	$800	$800
Au	US$/oz	$1,200	$1,200	$1,200

22.2.5 **Royalty**

Royalties in Minnesota are complex and based on a sliding-scale that increases exponentially with an increase in the value of the ore. Since the ore value used to determine the royalty is not updated annually, a Net Revenue Inflation Adjustment (NRIA) is required to be deducted from a mine's NSR per ton (imperial ton) before transportation costs to arrive at an adjusted NSR per ton.

The NRIA is calculated based on the US producer price inflation index (PPI) of the current period and the index for November 1994 ("Base Index") which had a value of 121.5, using the following formula:

(Producer Price Inflation Index USA – 121.5) / 121.5 x 75

In order to forecast the NRIA, an estimated inflation rate of 2.25% was used. The NRIA for 2018 is approximately US$49.00. The Minnesota royalty rates mapped to ore value less the NRIA are shown in Table 22-2.

Table 22-2: Minnesota Royalty Rates to be Applied NSR per ton Less the NRIA (all figures expressed in US$)

Index	Royalty	Index	Royalty	Index	Royalty	Index	Royalty	Index	Royalty
$ 75.00	3.95%	$ 100.00	4.19%	$ 200.00	6.39%	$ 300.00	10.53%	$ 400.00	16.62%
$ 80.00	3.99%	$ 110.00	4.33%	$ 210.00	6.71%	$ 310.00	11.05%	$ 410.00	17.34%
$ 85.00	4.03%	$ 120.00	4.48%	$ 220.00	7.06%	$ 320.00	11.59%	$ 420.00	18.07%
$ 90.00	4.08%	$ 130.00	4.65%	$ 230.00	7.43%	$ 330.00	12.15%	$ 430.00	18.83%
$ 95.00	4.14%	$ 140.00	4.84%	$ 240.00	7.81%	$ 340.00	12.73%	$ 440.00	19.61%
		$ 150.00	5.05%	$ 250.00	8.21%	$ 350.00	13.33%	$ 444.01	20.00%
		$ 160.00	5.28%	$ 260.00	8.64%	$ 360.00	13.95%		
		$ 170.00	5.52%	$ 270.00	9.08%	$ 370.00	14.59%		
		$ 180.00	5.79%	$ 280.00	9.55%	$ 380.00	15.25%		
		$ 190.00	6.08%	$ 290.00	10.03%	$ 390.00	15.93%		

The average net revenue per imperial ton of ore over the LOM is US$388/ton before the NRIA and US$337/ton after the NRIA, resulting in an average royalty of approximately 12.7%.

22.2.6 Taxation

The forecast uses a federal corporate tax rate of 21% and a Minnesota Occupation Tax rate of 2.45%. The Minnesota tax is deductible against federal tax. Federal tax deductions related to depletion respecting limitations were considered in accordance with US tax law for mining companies.

22.3 Technical Assumptions

22.3.1 Mine Production Schedule

The following graph illustrates the annual mining rate of waste and ore described on Table 16-10. Ore is categorized as MSU and SMSU. Development material is also considered ore as it has an approximate diluted NiEq grade of 3.36%, compared to 6.85% for MSU ore and 3.29% for SMSU ore. The peak mining rate is 1,878 tpd in year two and the average during years two to five is 1,689 tpd.

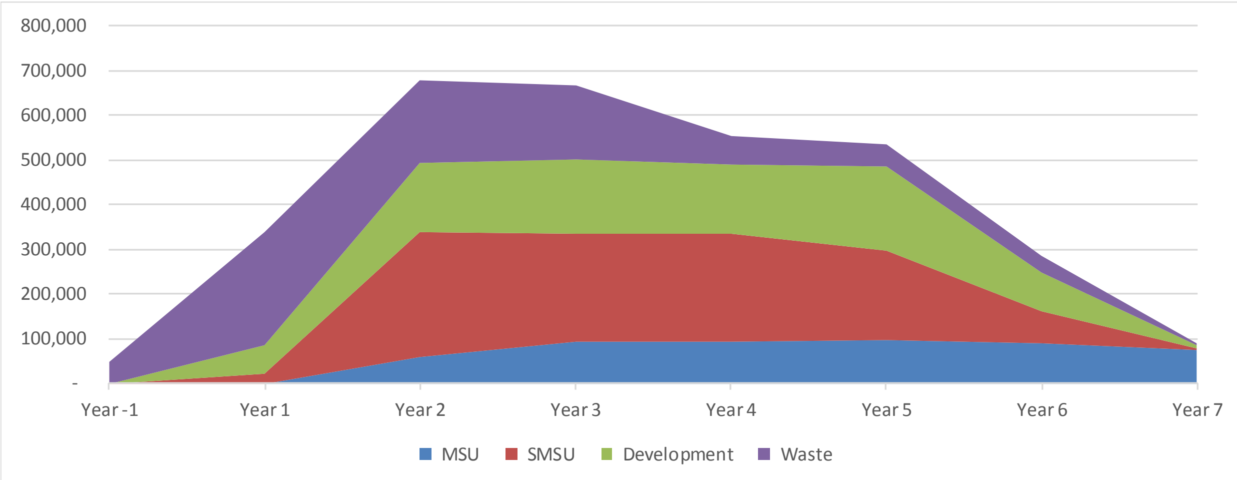


Figure 22-1: Mining Production Schedule

The LOM split of ore between MSU, SMSU and Development is 21%, 44% and 35%, respectively. Given MSU's high ore grade, it has a significant impact on profitability of the project.

22.3.2 Processing Schedule

Processing occurs concurrently with mining as per Figure 22-1, excluding waste. The peak processing rate is 1,390 tpd in year three and the average during years two to five is 1,367 tpd.

Figure 22-2 illustrates the grade profile over the LOM resulting from mining the MSU, SMSU and Development ore.

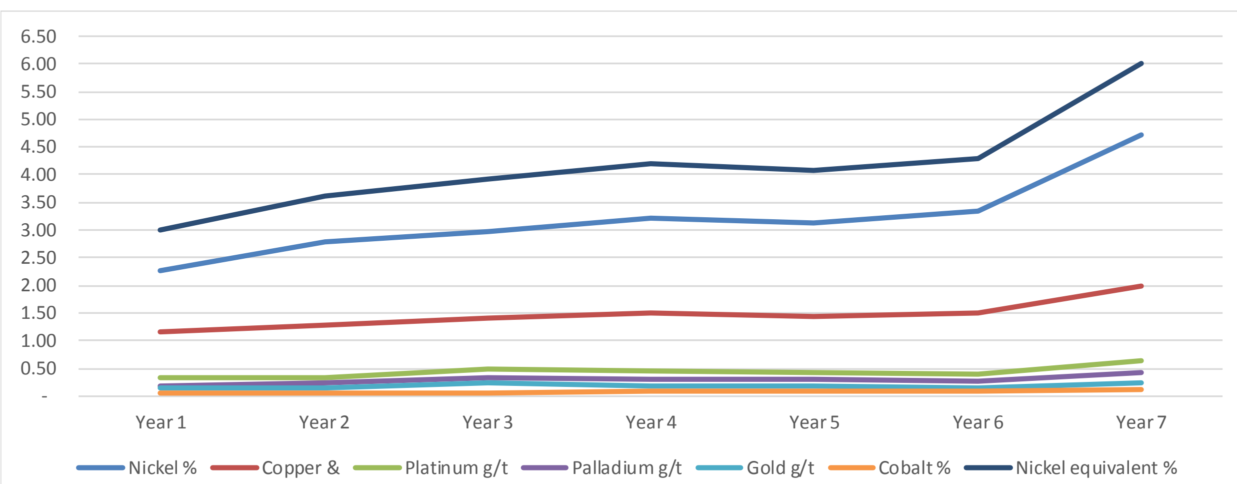


Figure 22-2: LOM Grade Profile

22.3.3 Net Smelter Return

The Tamarack North Project Ni and Cu concentrates will be sold directly to smelters or to traders in North America, Europe, and Asia. Based on metallurgical testing results to date, both the Ni and the Cu concentrates are of clean quality with low levels of impurities and good by-product credits. DRA has reviewed the smelter terms, the terms for the payment of metal, and the deductions for treatment and refining, applied in the economic model. DRA is of the opinion that the smelter contract terms, as applied in the economic model, are typical of the industry.

Concentrate transport costs were estimated to be US$90.00 per wet metric tonne (wmt) of Ni concentrate and US$100.00/wmt of Cu concentrate. An additional charge of 0.16% was included to cover insurance and losses.

Table 22-3: NSR and NSR After Royalties and Transportation

	LOM total (US$)	US$/tonne milled	US$/lb payable Ni
Payable Ni revenue	$1,020,171,336	$427.27	$8.00
Payable by-product revenue	$259,262,112	$108.58	$2.03
Total payable revenue	**$1,279,433,449**	**$535.85**	**$10.03**
Treatment and refining charges	$257,390,301	$107.80	$2.02
Insurance and losses	$1,635,269	$0.68	$0.01
Net smelter return	**$1,020,407,879**	**$427.37**	**$8.00**
Government and private royalties	$130,178,817	$54.52	$1.02
Transportation costs	$53,099,810	$22.24	$0.42
Net smelter return after royalties and transportation costs	**$837,129,251**	**$350.61**	**$6.56**

Using the base case metal price assumptions, the contribution of each metal to the NSR over the LOM is shown in Figure 22-3.

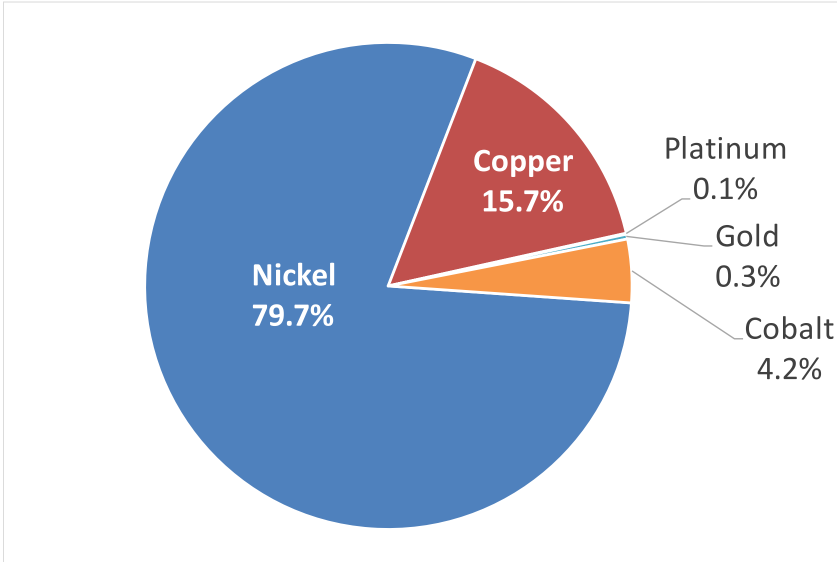


Figure 22-3: Contributions of Metals to NSR

22.3.4 Operating Costs

Direct operating costs average US$95.31/t milled over the LOM, including US$63.94/t for mining, US$18.87/t for processing, US$2.50/t for the CFTF and US$10.00/t for G&A costs. Figure 22-4 provides a breakdown of operating costs over the LOM.

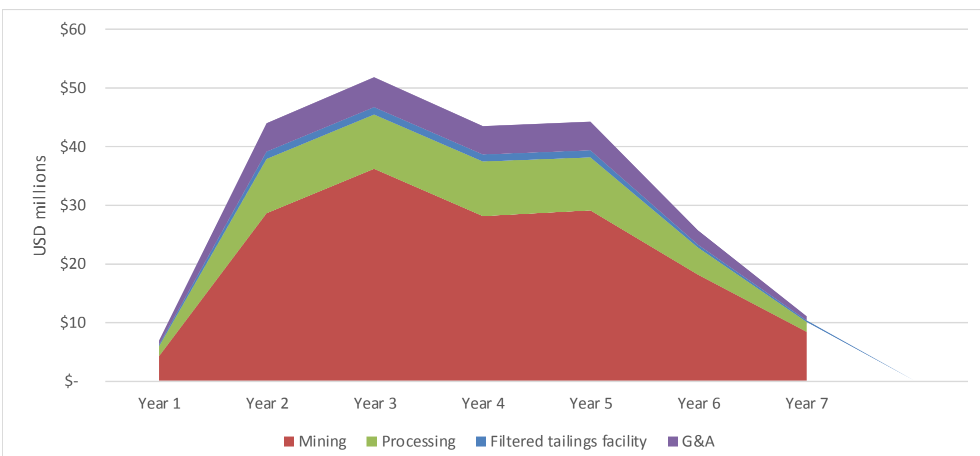


Figure 22-4: Direct Operating Cost Breakdown over LOM

22.3.5 Capital Costs

Pre-production capital costs are estimated to total US$174.31M including US$72.44M for mine and mine equipment and US$90.85M for process and surface facilities and US$11.01M for working capital. These costs include contingencies ranging from 20% to 25% as described in Section 21.

Sustaining capital costs during the remainder of the mine life total US$29.2M including closure costs. Salvage value of mill and other components is estimated to be US$10M given the short mine life of approximately six years.

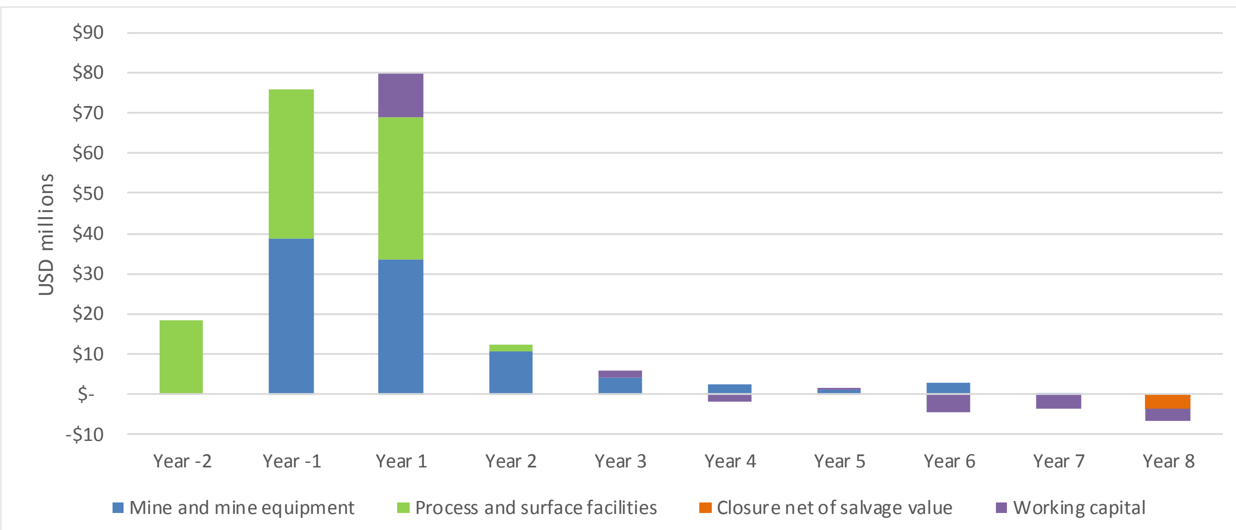


Figure 22-5: Capital Costs by Year

22.3.6 Tamarack North Project Cash Flow Summary

The following table summarizes the base case LOM cash flow:

Table 22-4: Summary of Base Case Life of Mine Cash Flow

	LOM total (US$)	US$/tonne milled	US$/lb payable Ni
Payable nickel revenue	$1,020,771,336	$427.27	$8.00
Payable by-product revenue	$259,262,112	$108.58	$2.03
Total payable revenue	**$1,279,433,449**	**$535.85**	**$10.03**
Treatment and refining charges	$257,390,301	$107.80	$2.02
Insurance and losses	$1,635,269	$0.68	$0.01
Net smelter return	**$1,020,407,879**	**$427.37**	**$8.00**
Government and private royalties	$130,178,817	$54.52	$1.02
Transportation costs	$53,099,810	$22.24	$0.42
Net smelter return after royalties and transportation costs	**$837,129,251**	**$350.61**	**$6.56**
On-site costs			
Mining costs	$152,662,064	$63.94	$1.20
Processing costs	$45,055,324	$18.87	$0.35
Co-disposed Filtered Tailings Facility	$5,969,174	$2.50	$0.05
General & administrative costs	$23,876,695	$10.00	$0.19
Total on-site costs	**$227,563,258**	$95.31	$1.78
Net operating margin	**$609,565,994**	**$255.30**	**$4.78**
Capital expenditures	$182,505,450	$76.44	$1.43
Working capital	-	-	-
Net cash flow, before tax	**$427,060,543**	**$178.86**	**$3.35**
Corporate tax	$74,054,709	$31.02	$0.58
Net cash flow, after tax	**$353,005,834**	**$147.85**	**$2.77**

Table 22-5 provides the calculation of "C1 cash costs" and total cost.

Table 22-5: C1 Cash Cost and Total Cost

	LOM total (US$)	US$/tonne milled	US$/lb payable Ni
On-site costs	$227,563,258	$95.31	$1.78
Treatment and refining charges	$257,390,301	$107.80	$2.02
Insurance and losses	$1,635,269	$0.68	$0.01
Transportation costs	$53,099,810	$22.24	$0.42
Less: By-product revenue	($259,262,112)	($108.58)	($2.03)
C1 cash cost	**$280,426,526**	**$117.45**	**$2.20**
Government and private royalties	$130,178,817	$54.52	$1.02
C1 cash cost plus royalties	**$410,605,343**	**$171.97**	**$3.22**
Capital expenditures	$182,505,450	$76.44	$1.43
Total cost including CAPEX	**$593,110,793**	**$248.41**	**$4.65**

Table 22-6 provides the annual cash flow over LOM.

Table 22-6: Base Case Life of Mine Annual Cash Flow

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Diluted tonnes mined and processed		2,387,670			85,889	493,700	500,443	489,816	484,860	246,328	86,634	-
Dilutes grades												
Nickel	%	3.09			2.26	2.80	2.97	3.22	3.14	3.32	4.71	-
Copper	%	1.42			1.16	1.28	1.41	1.49	1.43	1.50	1.99	-
Platinum	g/t	0.42			0.32	0.33	0.50	0.44	0.41	0.38	0.64	-
Palladium	g/t	0.29			0.19	0.23	0.34	0.31	0.29	0.27	0.43	-
Gold	g/t	0.19			0.16	0.16	0.23	0.19	0.18	0.15	0.24	-
Cobalt	%	0.07			0.06	0.07	0.07	0.08	0.08	0.08	0.10	-
Recovery to concentrates												
Nickel	%	85.0			81.15	83.79	84.53	85.56	85.24	85.96	90.31	-
Copper	%	94.5			93.32	93.89	94.48	94.78	94.56	94.85	96.48	-
Payable metal												
Nickel	lbs	127,521,417			3,199,755	23,450,953	25,448,252	27,371,035	26,308,462	14,276,462	7,466,497	-
Copper	lbs	66,820,290			1,927,649	12,284,816	13,870,788	14,345,273	13,641,798	7,297,969	3,451,997	-
Platinum	lbs	806			23	-	426	189	118	-	50	-
Palladium	oz	-			-	-	-	-	-	-	-	-
Gold	oz	3,562			109	587	1,076	722	689	225	153	-
Cobalt	oz	1,787,987			50,132	339,962	351,946	381,873	377,258	195,004	91,812	-
NSR before transportion costs and royalties	$	1,022,043,148			26,233,970	187,370,226	205,712,342	219,525,512	210,643,769	113,702,060	58,855,267	-
Transportation, losses and insurance	$	54,735,079			1,501,857	10,275,673	11,105,928	11,672,223	11,251,974	6,023,272	2,904,152	-
NSR after transportion costs, insurance and losses, but before royalties	$	967,308,068			24,732,113	177,094,553	194,606,414	207,853,289	199,391,795	107,678,789	55,951,115	-
Minnesota royalty	$	129,825,464			1,926,633	19,141,118	24,045,732	29,900,527	26,910,268	16,199,698	11,701,487	-
Private royalty	$	353,353			9,070	64,780	71,121	75,897	72,826	39,310	20,348	-
NSR net of transportation and royalties	$	837,129,251			22,796,410	157,888,655	170,489,561	177,876,865	172,408,701	91,439,780	44,229,280	-
Mining	$	152,662,064			4,281,391	28,560,633	36,106,354	28,252,872	28,995,322	18,045,513	8,419,978	-
Processing	$	45,055,324			1,620,726	9,316,110	9,443,368	9,242,821	9,149,311	4,648,212	1,634,777	-
Filtered tailings facility	$	5,969,174			214,723	1,234,249	1,251,109	1,224,539	1,212,150	615,820	216,584	-
G&A	$	23,876,695			858,890	4,936,995	5,004,435	4,898,156	4,848,601	2,463,281	866,336	-
Net profit before tax, interest, CAPEX and working capital	$	609,565,994			15,820,681	113,840,668	118,684,295	134,258,476	128,203,316	65,666,953	33,091,604	-
Capital expenditures	$	-										
Mine Development	$	67,423,916	-	16,909,310	29,331,556	10,652,327	3,983,111	2,398,803	1,177,336	2,907,280	64,194	-
Mine equipment	$	26,402,000	-	22,044,000	4,158,000	50,000	-	150,000	-	-	-	-
Process and Surface Facilities CAPEX	$	92,426,790	18,485,358	36,970,716	35,397,612	1,573,104	-	-	-	-	-	-
Other	$	- 3,747,256	-	-	-	-	-	-	-	-	-	(3,747,256)
Total capital expenditures	$	182,505,450	18,485,358	75,924,026	68,887,168	12,275,431	3,983,111	2,548,803	1,177,336	2,907,280	64,194	(3,747,256)
Working capital	$	-	-	-	11,011,997	-	1,939,320	- 2,046,719	146,749	- 4,608,139	- 3,658,788	(2,784,419)
Net cash flow after CAPEX and working capital	$	427,060,543	(18,485,358)	(75,924,026)	(64,078,484)	101,565,237	112,761,864	133,756,392	126,879,232	67,367,813	36,686,198	6,531,675
C1 cash cost per lb/Ni, net of credits	$/lb	$ 2.20			$ 2.45	$ 2.33	$ 2.39	$ 2.00	$ 2.10	$ 2.26	$ 2.00	
Income tax	$	74,054,709	-	-	-	11,950,014	14,108,616	18,725,739	18,608,357	8,263,968	3,045,709	(647,693)
After-tax cash flow	$	353,005,834	(18,485,358)	(75,924,026)	(64,078,484)	89,615,224	98,653,248	115,030,653	108,270,875	59,103,845	33,640,490	7,179,368
Cumulative cash flow	$		(18,485,358)	(94,409,384)	(158,487,868)	(68,872,645)	29,780,604	144,811,257	253,082,132	312,185,977	345,826,466	353,005,834
Funding requirement to positive cash flow	$		174,308,549									

22.3.7 Base Case Evaluation

The base case cash flow, which is in real dollars, was evaluated by determining the after-tax NPV at a discount rate of 7.0% and the after-tax IRR as shown in Table 22-7. Results are also shown at comparative discount rates of 8% and 10% and on a pre-tax basis.

Table 22-7: Base Case NPV in Million US$ at Various Discount Rates and IRR

	Base case NPV discounted at			
	7%	**8%**	**10%**	**IRR**
Pre-tax	261	244	212	44.6%
After-tax	210	195	168	38,8%

The undiscounted pre-tax payback period is 1.9 years from the production start date in the third quarter of year one which along with other payback measures is included in the table that follows:

Table 22-8: Payback Period in Years from Production Start Date

	Undiscounted	**Discounted**
Pre-tax	1.9	2.1
After-tax	2.1	2.4

22.4 Sensitivity and Risk Analysis

22.4.1 Metal Price Assumptions and Discount Rates

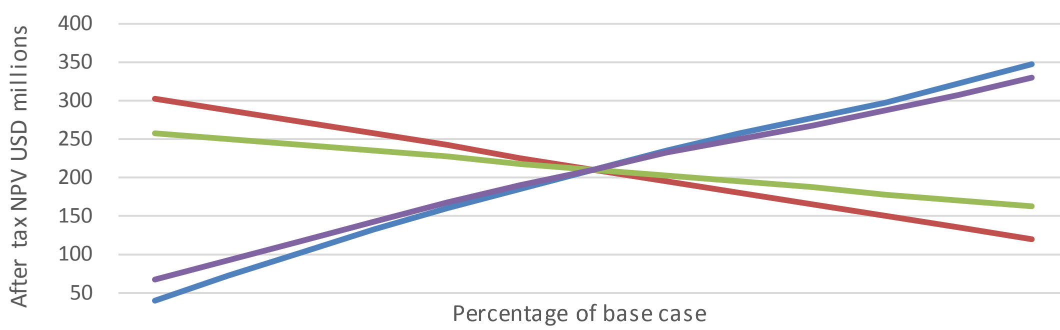
The sensitivity of the after-tax and pre-tax NPV and IRR as well as other measures was tested using alternate metal price assumptions and discount rates as shown in Table 22-9.

Table 22-9: After-Tax and Pre-tax NPV in Million US$ and After-Tax and Pre-tax IRR and Other Measures using Base Case and Alternate Metal Price Assumptions and Discount Rates

		After-tax			Pre-tax		
		Metal price scenario			Metal price scenario		
		Low	Base	Incentive	Low	Base	Incentive
Discount rate	NPV 7%	130	210	287	163	261	354
	NPV 8%	119	195	268	150	244	332
	NPV 10%	98	168	234	127	212	292
IRR		27.9%	38.8%	48.3%	32.2%	44.6%	55.3%
C1 Cash Cost per lb of payable Ni		$2.47	$2.20	$1.93	$2.47	$2.20	$1.93
Payback from start of production in years		2.6	2.1	1.8	2.5	1.9	1.6

22.4.2 Capital, Operating Costs, Grade and Revenue Sensitivity

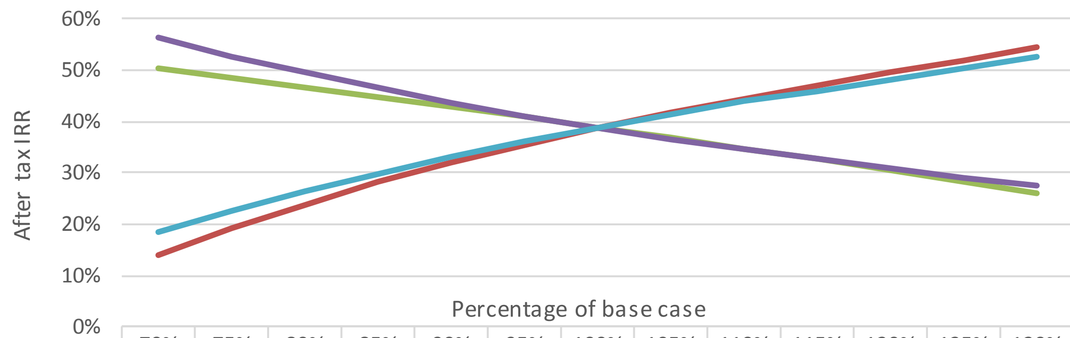
The sensitivity of the after-tax NPV was tested assuming changes in metal prices, operating costs, grade and capital costs in a range of 30% around the base case as shown in Figure 22-6.



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	40	72	102	131	159	185	210	234	256	276	297	321	347
Operating costs	301	286	271	256	241	225	210	195	180	165	149	134	119
CAPEX	258	250	242	234	226	218	210	202	194	186	178	170	162
Grade	67	93	118	143	166	189	210	231	250	268	286	306	329

Figure 22-6: Sensitivity of Base Case After-tax NPV to Changes in Metal Prices, Grade, Operating Costs and Capital Costs

The sensitivity of the after-tax IRR was tested assuming changes in metal prices, operating costs, grade and capital costs in a range of 30% around the base case as shown in Figure 22-7.



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	14%	19%	24%	28%	32%	35%	39%	42%	45%	47%	49%	52%	55%
Operating costs	50%	49%	47%	45%	43%	41%	39%	37%	35%	33%	30%	28%	26%
CAPEX	56%	53%	49%	46%	44%	41%	39%	37%	35%	33%	31%	29%	28%
Grade	18%	22%	26%	30%	33%	36%	39%	41%	44%	46%	48%	50%	53%

Figure 22-7: Sensitivity of Base Case After-tax IRR to Changes in Metal Prices, Operating Costs, Grade and Capital Costs

23. ADJACENT PROPERTIES

There are no adjacent properties considered material to the Tamarack North Project resources.

24. OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation necessary with respect to this report.

25. INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource

It is Golder's opinion that the sample preparation, security and analytical procedures used by Kennecott are consistent with industry standards and are appropriate for the Tamarack North Project. Golder has no material concerns with these processes.

On completion of the data validation, site visit and verification sampling, Golder is of the professional opinion that the quality of the assay data is of suitable quality to support the Mineral Resource estimate.

Golder is unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or any other potential factors that could materially impact the Tamarack North Project resource estimate provided in this technical report. The resource is located in designated wetlands but this is not expected to affect future permitting.

25.2 Mineral Processing and Metallurgical Testing

The MSU and SMSU mineralization has moderate grinding energy requirements. Furthermore, the MSU and SMSU composites produced high-grade, marketable Ni and Cu concentrates as well as very high recoveries. Levels of deleterious elements in the MSU and SMSU composites were consistently low.

Preliminary testing resulted in a split HS and LS tailings streams.

Testing of low-grade, disseminated sulphides comprised the majority of the met testing to date. The feed grades of these composites fall below the cut-off of both the resource model and the mine plan and are therefore not relevant for purposes of this PEA.

25.3 Mining Methods

The Tamarack North Project is amenable to underground mining at a rate of 1,390 tpd using cut-and-fill and transverse open stoping with cemented paste backfill. Access will be from a shaft, and development and production performed by mining contractors. Expected mine life based on current deposit extent and Mineral Resources is seven years, however the deposit is open and a longer mine life can be expected.

25.4 Recovery Methods

The flowsheet was designed for a nameplate capacity throughput of 1,390 tph (507,350 tonnes per annum (tpa)). Conventional mineral processing technologies were selected to produce one Ni and one Cu concentrate as well as one LS and one HS tailings stream. The LOM Ni and Cu concentrate recoveries are estimated at 85.0% and 94.5% respectively. The Ni and Cu concentrate grades are projected to be 14.5% Ni and 28.9% Cu. The low feed rate facilitates a simple crushing and grinding circuit with two stages of crushing and a single stage of ball milling. It should be noted that the flowsheet was designed to accommodate a low-grade, higher volume feed and therefore an optimized metallurgical flowsheet should be developed that will result in a less complex plant layout and improved operability, which will in turn have the potential to reduce CAPEX and OPEX.

25.5 Infrastructure

A preliminary, conceptual site layout was designed. Notably, an innovative CFTF was designed which negates the need for a tailings dam while reducing the site footprint and the potential wetland impact. Although a portion of the site was delineated as wetlands in 2008 and 2009, the total area was utilized as farmland in 1992.

A preliminary water balance was completed based on waste rock characterization, geophysical measurements, pump tests and an estimate of water requirements for the mine and the processing plant.

25.6 Environmental Considerations and Permitting

The process of environmental review and permitting for a mining project of this type is well understood. The existing environmental baseline information that has been gathered to date will need to be augmented to support the engineering design of the project and the environmental review and permitting processes. Based on the information available to date, there are no environmental aspects that should prevent the project development.

25.7 Capital and Operating Costs

There is an opportunity to reduce CAPEX and OPEX by using a simplified flowsheet as discussed in Section 25.4.

25.8 Economics

The results show that the after-tax NPV of US$210M and the after-tax IRR of 39% is robust and remains positive for the range of sensitivities evaluated. The sensitivity analysis examined the impact on NPV (at a 7% discount rate) of a 30% positive or negative change in metal prices, grade, operating costs and capital costs. The Tamarack North Project is most sensitive to changes in metal prices, followed next by changes in grade.

26. RECOMMENDATIONS

The Tamarack North Project should be advanced to the PFS stage. Recommendations by area are as follows.

26.1 Exploration of Open Areas

There are several opportunities to increase the project NPV through further exploration of the Tamarack Zone, the 138 Zone, the CGO Bend and the 164 Zone. Figure 26-1 below shows the locations of each of these areas.

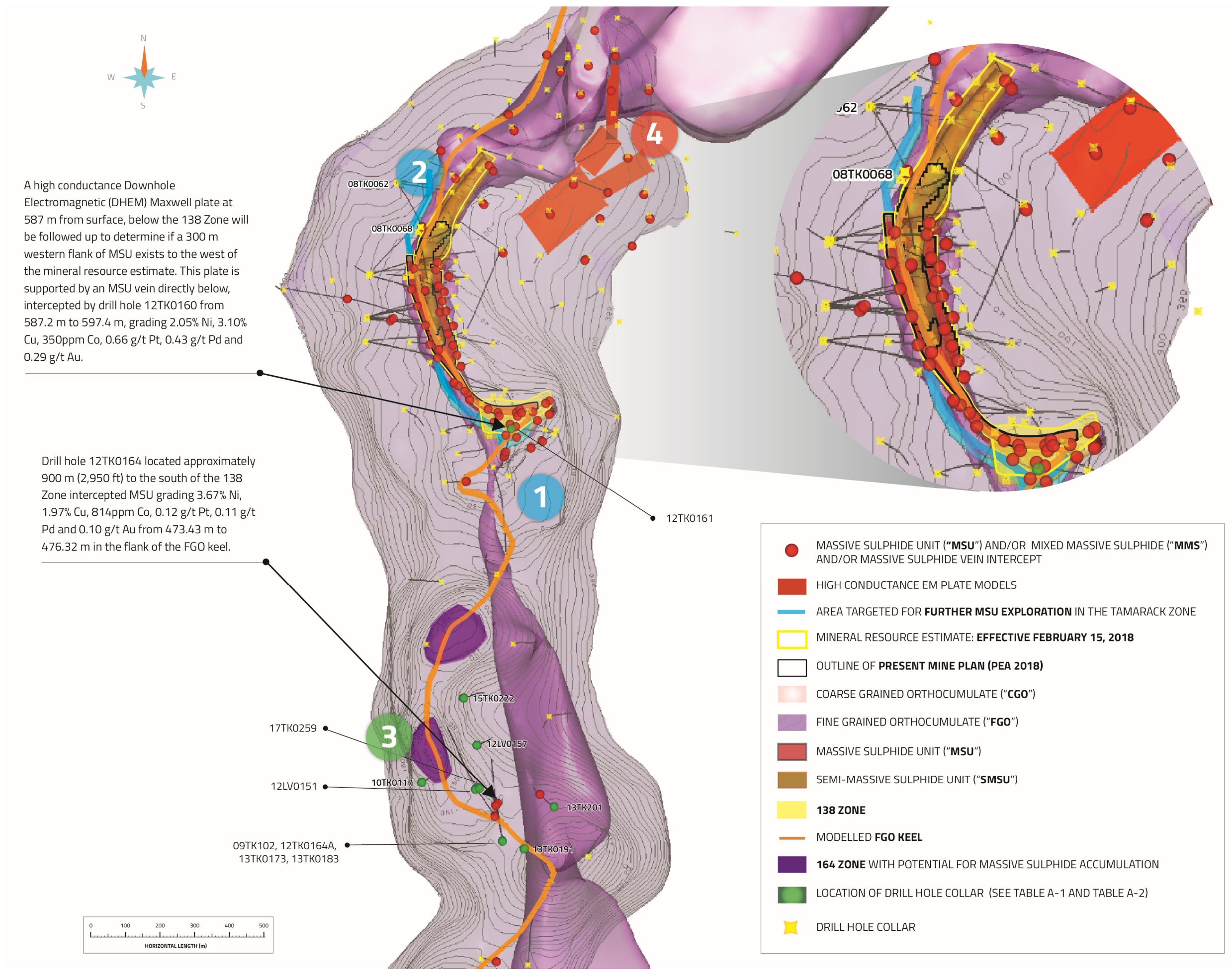


Figure 26-1: Plan View of the Tamarack and 138 Zones as well as Areas Targeted for Further Exploration

1. A high conductance Downhole Electromagnetic (DHEM) Maxwell plate at 587 m from surface, below the 138 Zone will be followed up to determine if a 300 m western flank of MSU exists to the W of the mineral resource estimate. This plate is supported by an MSU vein directly below, intercepted by drill hole 12TK0160 from 587.2 m to 597.4 m, grading 2.05% Ni, 3.10% Cu, 350 ppm Co, 0.66 g/t Pt, 0.43 g/t Pd and 0.29 g/t Au (Refer to Annex A and B);

2. An approximately 340 m (1,115 ft) gap in MSU between two MSU intercepts from drill hole 08TK0062 and drill hole 08TK0068 (Refer to Annex A and B) remains to be drilled following the modelling of a DHEM conductor;

3. At the Tamarack Zone and the 138 Zone, massive sulphide settling occurred along the FGO keel that resembles the hull of a boat where massive sulphide settling may have occurred. We have approximately 1 km (0.6 miles) of the keel with two areas that display a similar widening of the keel (modelled from gravity and magnetic surveys as well as contouring using drill holes) where massive sulphide settling may have occurred. Drill hole 12TK0164 located approximately 900 m (2,950 ft) to the S of the 138 Zone intercepted MSU grading 3.67% Ni, 1.97% Cu, 814 ppm Co, 0.12 g/t Pt, 0.11 g/t Pd and 0.10 g/t Au from 473.43 m to 476.32 m in the flank of the FGO keel (Refer to Annex A and B);

4. Surface EM is supported by drill intercepts of high-grade Ni-Cu-Co mineralization over an 78,000 m^2 (19 acre) area to the NE of the Tamarack Zone between 90 m (295 ft) and 195 m (640 ft) from surface.

These areas will be accessed through the same infrastructure. Furthermore, these areas will use the same processing facilities.

Exploration methods recommended include:

- *Surface EM:* Previous surface EM over the CGO Bend enabled Talon to model Maxwell plates that showed areas of high conductance which could possibly be the result of basal MSU at the FGO contact as evidenced by historical drilling;
- *Downhole tomography:* Forward modelling and a field test is recommended. If successful, this method could be used in conjunction with DHEM and MMR (see below) to map mineralization prior to drilling;
- *Magnetometric resistivity (MMR) surveys*: This method involves measuring magnetic fields arising from a current input into the ground. If successful, this method could map areas where MSU is continuous along strike;
- Re-entering of drill holes to collect DHEM data where historical intervals of data collection exceeded 5 meters;
- Combining the results of all geophysical methods to determine precise targets for drilling.

26.2 **Geology and Mineral Resources**

The geophysical survey methods described in Section 26.1 should be used to plan an infill drill program that will upgrade resources not already in the indicated category. This program should take into account the results of the metallurgical test program recommended under Section 26.4 below as infill drilling needs to be limited to resources that will form part of the mine plan.

With respect to sample preparation and QA/QC, Golder recommends the following:

- That the Operator prepare an annual report summarizing the QA/QC analysis of their CRM data and that they incorporate laboratory check assays, from a referee lab, into their protocol to confirm the quality of assay values from their primary lab;
- That SG measurements are completed from sample pulps where data is currently only available from field measurements.

26.3 Mining Methods

In anticipation of bringing the Tamarack North project to PFS stage, additional technical studies are recommended:

- Additional Rock Mechanics Studies;
- Additional hydrogeological assessment;
- Backfill testing.

26.4 Mineral Processing and Recovery Methods

A series of tests programs are recommended to support the PFS for the Tamarack North project.

26.4.1 Test Program to Determine an Optimal Feed Blend

A metallurgical test program is recommended that focuses on the mineral resources that may be included in a revised mine plan. Most of the mineral resource estimate tonnage in the SMSU and all of the tonnage in the 138 Zone have been excluded from this first PEA (refer Figure 26-2 below). One objective of the proposed metallurgical test program outlined below is to expand testing to include the total mineral resource estimate. A second objective is to simplify the flowsheet and reagent regime, thus reducing the capital and operating expenses for a given plant throughput. Both these initiatives could substantially increase the Tamarack Project NPV and should therefore be conducted as a matter of priority.

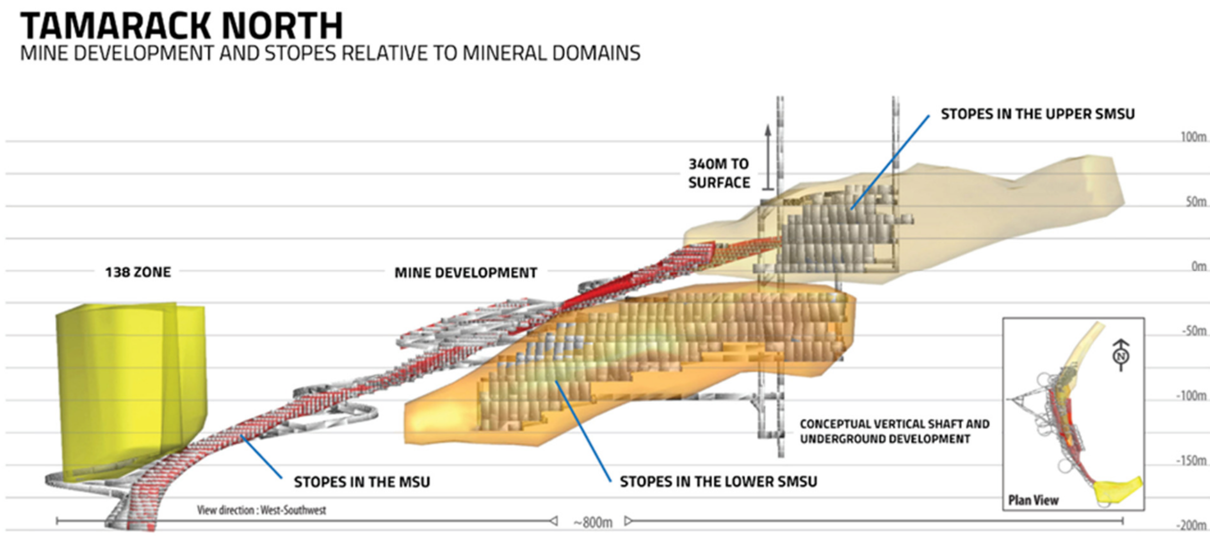


Figure 26-2: Long Section (looking W) of the PEA Mine Plan Development and Stopes in Relation to the Tamarack North Resource Estimate (effective February 15, 2018)

Talon is evaluating different approaches for the overall mine development. In order to determine if saleable Ni and Cu concentrates can be generated from the tonnage that is presently excluded from the mine plan, a phased metallurgical test program is planned. In the first phase five composites will be generated and characterized to identify the modals, mineral association, and liberation properties. The following five composites have been identified:

1) SMSU Indicated and MSU Inferred;
2) Upper SMSU Inferred only;
3) Lower SMSU Inferred only;
4) High-grade 138 Zone
5) Low-grade 138 Zone

The first composite represents the tonnage included in the current mine plan and will serve as baseline data. The two SMSU composites represent the tonnage that is excluded from the current mine plan. Since this material originates from two separate lenses, mineralogical characterization is performed on each of the two lenses to identify any mineralogical differences that may exist. In the case of the 138 Zone, a low-grade and high-grade composite will be generated since the two waste-grade samples that were evaluated in the 2016/2017 metallurgical test program displayed challenging metallurgy. Using two grade composites will help to identify metallurgical differences as a function of the head grade. Since the composites will be generated from sub-composites that have been established to represent different cut-off grades, additional mineralogy composites can be generated if deemed necessary based on the characterization of the five composites. At the beginning of the second phase of the PFS program, the results of the mineralogical analysis will then be used to determine suitable composites for the flotation test program. The second phase of metallurgical work will then focus on simplifying the existing flowsheet and on demonstrating that the presently excluded tonnage can be processed successfully.

In this way value from the present resource will be maximized and the flowsheet will be simplified. The final regression curve will be based on locked cycle testing using composites that reflect the actual mill feed grades over the projected mine life. The production of a geo-metallurgical model to assess the suitability of the samples tested will be constructed and a full variability test program throughout the deposit will be conducted during the PFS phase.

The reagent regime developed for the Tamarack North Project mineralization is presented in Table 13-7. It is noted that the dosage of 675 g/t CMC is considered very high and was driven by the low-grade disseminated samples tested in 2016/2017. It is postulated that the dosage could be reduced by at least 50% for the MSU and SMSU domains. Given the significant cost of the proposed reagent regime, a dosage optimization should be carried out during the next phase of testing.

Further, the collector dosage required for the disseminated composites was significantly higher than suggested by their sulphide head grades. This is a strong indication that collector “robbing” is taking place by some of the non-sulphide gangue minerals. Dosage levels vary for the different domains and must be established during the next phase of testing.

26.4.2 Comminution Testing

The primary grind size should be reconsidered to minimize the comminution energy in the primary grinding circuit. Pending the outcome of the test program to determine the optimal feed composition (refer Section 26.4.1 above), the plant layout should be re-designed to accommodate the potential addition of a re-grind mill in the future.

26.4.3 Product Testing

Further testing will be required to verify that a desulphurization strategy is feasible for the expected LOM mill feed grade and expected domain blends. In addition, chemical and physical characterization of the final Ni and Cu concentrates should be performed. Self-heating tests on Ni, Cu, and HS tailings should be conducted to quantify the self-heating potential of the product streams.

26.4.4 Pre-Concentration Tests

Approximately 64,000 tonnes of MSU are currently excluded from the mine plan due to dilution. Two preconcentration strategies should be explored:

- Sensor-based ore sorting;
- Heavy liquid separation (HLS) (to evaluate dense media separation (DMS)).

26.4.5 Metallurgical Optimization

Once the most favourable blend strategy and consequently mine production schedule has been modeled, the following studies are recommended to generate the metallurgical data required for a PFS:

- Optimize the flotation flowsheet and conditions to maximize the metal recovery into saleable concentrates;
- Develop grade-recovery relationships for Cu and Ni concentrates for economic sensitivity analysis purposes;
- If HLS proved successful, a bulk sample should be processed through a pilot scale DMS;

- Complete a full comminution test program including low energy impact tests, Bond ball and rod mill grindability tests, Bond abrasion test, and SMC tests;
- Carry out solid-liquid separation tests on concentrates and tailings;
- Perform jar tests to quantify the grinding energy requirement for the regrind mill(s).

26.4.6 Potential to Produce Ni and Co Sulphates for the Battery Market at Tamarack

In addition to conventional mineral processing technologies, treatment of the Ni concentrate in a hydrometallurgical process should be evaluated with the objective of producing Ni and Co sulphates. A breakdown of the various unit areas requiring investigation is provided below:

- Leach methods;
- Pregnant leach solution (PLS) neutralization and primary impurity removal;
- Cu Recovery;
- Impurity solvent extraction (SX);
- Ni and Co SX;
- Purification of strip solutions;
- Solid liquid separation of final products.

26.4.7 Optimization of Recovery Methods

The following recommendations are made with respect to recovery methods:

- Optimize comminution circuit once grinding test results are obtained. This includes trade off between conventional crushing/grinding circuit and SAG/ball mill grinding circuit;
- Incorporate the optimized process flowsheet into the plant layout;
- Incorporate column flotation into the Cu/Ni separation circuit; and
- Optimize tailings management once preliminary paste backfill data is available.

26.5 Data Required for Design of Surface Facilties

Further geotechnical logging and testing is required from surface through to bedrock to plan surface infrastructure construction, including soil and structural rock logging. This work will allow the assessing of key design parameters such as bearing capacities. A potential aggregate source solution should be investigated to assess chemical and physical suitability.

26.6 Environmental, Permitting, and Social Considerations

Additional baseline work remains to be completed for the Project. The following work is recommended to support various engineering design elements and the further assessment of likely impacts from the project that will be needed for a PFS:

- In-field confirmation and mapping of wetland types and boundaries;
- Further hydrogeological characterization of geologic units, with a focus on interconnections between bedrock, surficial deposits and surface water including wetlands;
- Waste characterization – additional static testing and initiation of kinetic tests – with a focus on the material that will be stored temporarily or permanently above ground;
- Wild rice surveys and sediment sampling;
- A trade-off study needs to be conducted to determine the optimal effluent treatment system;
- Water treatment options need to be studied in order to select a water treatment method.

In addition, a more detailed water balance for all phases of the project will be necessary in order to determine engineering controls and mitigation measures likely to be needed.

Finally, early development of a robust engagement plan including participating with local and tribal stakeholders prior to formal environmental review processes is also recommended.

26.7 Budget

The following table summarizes an estimate of the cost of each of the recommendations discussed above.

Table 26-1: Budget for Recommendations

		Amount (US$)
26.1	**Geology and Mineral Resources**	
a)	Geophysical survey methods: Surface EM, MMR	200,000
b)	Drilling, DHEM and downhole tomography	7,500,000
c)	Annual report summarizing QA/QC analysis	10,000
d)	Specific gravity measurements completed from sample pulps	Incl. in future drilling budget
26.2	**Mining Methods**	
a)	Additional rock mechanics studies	400,000
b)	Additional hydrogeological assessment	150,000
c)	Backfill testing	40,000
26.3	**Mineral Processing and Recovery Methods**	
26.3.1	**Test program to determine an optimal feed blend**	150,000
26.3.2	**Comminution Testing**	40,000
26.3.3	**Product Testing**	
a)	Desulphurization	10,000
b)	Chemical and physical characterization of final Ni and Cu concentrates	2,000
c)	Self-heating tests on Ni, Cu, and HS tailings	10,000
26.3.4	**Pre-Concentration Tests**	
a)	Ore sorting	25,000
b)	Heavy liquid separation	10,000
26.3.5	**Metallurgical Optimization**	
a)	Optimize flotation flowsheet and conditions to maximize metal recovery into saleable concentrates	15,000
b)	Develop grade-recovery relationships for Cu and Ni concentrates for economic sensitivity analysis purposes	15,000
c)	If HLS proved successful, a bulk sample should be processed through a pilot scale DMS	20,000
d)	Complete a full comminution test program including low energy impact tests, bond ball and rod mill grindability tests, bond abrasion test, and SMC tests	30,000
e)	Carry out solid-liquid separation tests on concentrates and tailings	15,000
f)	Perform jar tests to quantify the grinding energy requirement for the regrind mill(s)	10,000

26.3.6	**Potential to Produce Ni and Co Sulphates for the Battery Market**	
a)	Leach methods	70,000
b)	PLS neutralization and primary impurity removal	30,000
c)	Cu Recovery	20,000
d)	Impurity SX	30,000
e)	Ni and Co SX	70,000
f)	Purification of strip solutions	20,000
g)	Solid liquid separation of final products	5,000
h)	Miscellaneous	50,000
26.3.7	**Optimization of Recovery Methods**	
a)	Optimize comminution circuit once grinding test results are obtained. This includes trade off between conventional crushing/grinding circuit and SAG/ball mill grinding circuit	Internal
b)	Incorporate the optimized process flowsheet into the plant layout	Internal
c)	Incorporate column flotation into the Cu/Ni separation circuit	Internal
d)	Optimize tailings management once preliminary paste backfill data is available	Internal
26.4	**Data Required for Design of Surface Facilities**	
a)	Geotechnical logging and testing from surface through to bedrock to plan surface infrastructure construction, including soil and structural rock logging	180,000
26.5	**Environmental, Permitting, and Social Considerations**	
a)	In-field confirmation and mapping of wetland types and boundaries	15,000
b)	Further hydrogeological characterization of geologic units, with a focus on interconnections between bedrock, surficial deposits and surface water including wetlands	Internal
c)	Waste characterization – additional static testing and initiation of kinetic tests – with a focus on the material that will be stored temporarily or permanently above ground	150,000
d)	Wild rice surveys and sediment sampling	80,000
e)	Detailed water balance for all phases of the project	20,000
	Total (US$)	**9,392,000**

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28. LIST OF TABLES

29. LIST OF FIGURES

30. CERTIFICATES OF QUALIFIED PERSONS

DRA Americas
44 Victoria St
Suite 300
Toronto, Ontario
Canada
M5C 1Y2

CERTIFICATE OF AUTHOR

To accompany the NI 43-101 Technical Report entitled "Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack Minnesota", prepared for Talon Metals Corporation with an effective date of December 14, 2018 (the "Technical Report").

I, Leslie Correia, Pr.Eng., do hereby certify that:

1) I am Engineering Manager with Paterson & Cooke Canada Inc., with an office at 1351-C Kelly Lake Road, Unit #2, Sudbury, Ontario, P3E 5P5;

2) I am a graduate of the University of Stellenbosch (Bachelor of Engineering (Chemical), 2005;

3) I am a member in good standing of the Engineering Council of South Africa (ECSA), License #20130236;

4) My relevant experience is 12 years as an independent consultant. I have been responsible for hydraulic, process and mechanical design of slurry pump and pipeline systems, backfill plant and reticulation system design, capital and operation cost estimates and project management of mining projects worldwide;

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 have participated in the preparation of the report titled "Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack Minnesota" dated December 14, 2018 and am responsible for a portion in Section 16 related to paste backfill;

7) I have not visited the site;

8) I have had no prior involvement with the property that is the subject of the Technical Report;

9) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

10) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report;

11) I am independent of the issuer as defined in Section 1.5 of NI 43-101;

12) I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 14th day of December 2018.

Leslie Correia, Pr.Eng.
Engineering Manager
Paterson & Cooke Canada Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota",** prepared for Talon Metals Corporation with an effective date of December 14, 2018 (the "Technical Report").

I, Silvia Del Carpio, P. Eng., MBA, do hereby certify that:

1) I am a Financial Analyst with Met-Chem, a division of DRA Americas, with an office at suite 300, 555 René-Lévesque Blvd. West, Montréal, Canada;

2) I am a graduate from McGill University, Montreal, Quebec, Canada with B. Eng. in Materials Engineering in 2007;

3) I am a registered member of "*Professional Engineers Ontario*" (#100134350). I have worked for more than 10 years in the mining industry in various positions since my graduation from university.

   My experience for the purpose of the Technical Report is:

   - Review and interpretation of financial results of industrial projects in Europe, Asia and the Americas;
   - Generation of cash flows and valuation of mining projects;
   - Management and finance training obtained during a Master of Business Administration degree from London Business School in London, United Kingdom.

4) 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 by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

5) I have participated in the preparation of the report entitled titled "**Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota**" dated December 14, 2018 and am responsible for Sections 19 and 22 and portions of Sections 1, 25, 26, and 27;

6) I have not visited the site;

7) I have had no prior involvement with the property that is the subject of the Technical Report;

8) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

9) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report.;

10) I am independent of the issuer as defined in section 1.5 of NI 43-101;

11) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;

This 14th day of December 2018.

LICENSED PROFESSIONAL ENGINEER
S. C. DEL CARPIO
100134350
PROVINCE OF ONTARIO

Silvia Del Carpio, P. Eng., MBA
Financial Analyst
Met-Chem, a division of DRA Americas

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corporation with an effective date of December 14, 2018.

I, Tim Fletcher, P. Eng., do hereby certify that:

1) I am a Project Manager with DRA Americas Inc., with an office at 300-44 Victoria Street, Toronto, Ontario, Canada;

2) I am a graduate from University of Toronto, with a B.A.Sc. in Mechanical Engineering in 1992 and an M.A.Sc. in Metallurgical Engineering in 1995;

3) I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 90451964);

4) I have worked as an Engineer in the Mining & Metals industry continuously since my graduation from university;

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 by 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 have participated in the preparation of the report entitled " titled **Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota** " dated December 14, 2018 and am responsible for Section 2, portions of Sections 1, 3, 21, 25, 26, and 27, and overall report compilation;

7) I have not visited the site;

8) I have had no prior involvement with the property that is the subject of the Technical Report;

9) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

10) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report.;

11) I am independent of the issuer as defined in section 1.5 of NI 43-101;

12) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;

This 14th day of December 2018

Tim Fletcher, P. Eng.
Project Manager
DRA Americas Inc.

LICENSED PROFESSIONAL ENGINEER
T. E. FLETCHER
14-Dec-2018
PROVINCE OF ONTARIO

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota",** prepared for Talon Metals Corporation with an effective date of December 14, 2018 (the "Technical Report").

I, Daniel M. Gagnon, P. Eng., do hereby certify that:

1) I am VP Mining and Geology with Met-Chem, a division of DRA Americas, with an office at suite 600, 555 René-Lévesque Blvd. West, Montréal, Canada;
2) I am a graduate from "*École Polytechnique de Montréal*" with B. Eng. in Mining Engineering in 1995;
3) I am a registered member of "*Ordre des Ingénieurs du Québec*" (118521);
4) I have worked as a Mining Engineer continuously since my graduation from university;
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 by 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 have participated in the preparation of the report entitled " titled **Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota**" dated December 14, 2018 and am responsible for Section 16 and portions of Sections 1, 3, 21, 22, 25, 26, and 27;
7) I have visited the site on October 5, 2017;
8) I have had no prior involvement with the property that is the subject of the Technical Report;
9) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;
10) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report.;
11) I am independent of the issuer as defined in section 1.5 of NI 43-101;

12) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;

This 14th day of December 2018.

Daniel M. Gagnon, P. Eng.
VP Mining and Geology
Met-Chem, a division of DRA Americas

GOLDER

CERTIFICATE OF QUALIFIED PERSON KEBREAB BERHANE HABTE

I, Kebreab Berhane Habte P.Eng., state that:

(a) I am a Senior Geotechnical Engineer at:
Golder Associates Limited
6925 Century Avenue, Suite #100
Mississauga, Ontario, L5N 7K2

(b) This certificate applies to the technical report titled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** with an effective date of: Dec 14, 2018 **(the "Technical Report").**

(c) **I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My** qualifications as a qualified person are as follows. I am a graduate from University of Asmara, Eritrea in 1998 with Bachelor of Science degree in Soil and Water Conservation and a Masters of Civil Engineering degree from the University of KwaZulu-Natal, South Africa in 2004. I am a professional engineer in good standing with the Professional Engineers Ontario (PEO) in Canada (#100174660). I practiced my profession continuously for more than 17 years since my graduation from university. I have worked first as a Geotechnical Engineer and then as Senior Geotechnical Engineer in the field of mine waste management with Golder Associates Ltd. since 2005, in many parts of the world, including Africa, North America, South America and Europe.

(d) I did not visit the project site.

(e) I am responsible for a portion of Section 18 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(g) I had no prior involvement with the Project.

(h) I have read National Instrument 43-101. The part of the Technical Report for which I am responsible has been prepared in compliance with this Instrument.

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Mississauga, Ontario, Canada, this 13th day of December 2018.

LICENSED PROFESSIONAL ENGINEER
K. B. HABTE
100174660
13 Dec 2018
PROVINCE OF ONTARIO

Kebreab Berhane Habte, P.Eng.(ON)
Senior Geotechnical Engineer
Golder Associates Ltd.

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Metpro Management Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corporation with an effective date of December 14, 2018.

I, Oliver Peters, P.Eng., M.Sc., MBA, do hereby certify that:

1) I am President and Principal Metallurgist with Metpro Management Inc. with an office at 102 Milroy Drive, Peterborough, Ontario, Canada;

2) I am a graduate from RWTH Aachen with a M.Sc. in Mineral Processing in 1998 and an MBA from Athabasca University in 2007;

3) I am a registered member the Professional Engineers of Ontario (100078050);

4) I have worked as a Mineral Processing Engineer and Project Manager continuously since my graduation from university;

5) I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101I have participated in the preparation of the report entitled " titled **Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota** " dated December 14, 2018 and am responsible for Sections 13 and 17, and portions of 1, 21, 25, 26, and 27.

6) I have not visited the site;

7) I have had no prior involvement with the property that is the subject of the Technical Report;

8) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

9) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report.;

10) I am independent of the issuer as defined in section 1.5 of NI 43-101;

11) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;

Metpro Management Inc.

This 14th day of December 2018

Oliver Peters, P.Eng., M.Sc., MBA
President & Principal Metallurgist
Metpro Management Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota" (the "Technical Report"),** prepared for Talon Metals Corporation with an effective date of December 14, 2018.

I, Thomas J. Radue, do hereby certify that:

1) I am a consulting senior geotechnical engineer and Vice President at Barr Engineering Company with an office at 4300 MarketPointe Drive, Suite 200, Minneapolis, Minnesota 55435;

2) I am a graduate from the University of Wisconsin, with a Bachelor of Science Degree in Civil Engineering in 1982, and a Master of Science Degree in Civil and Environmental Engineering specializing in Geotechnical Engineering in 1985; and a 1999 graduate from the University of Minnesota with an MBA specializing in Strategy and Operations.

3) I am a Professional Engineer licensed in the State of Minnesota (P.E. License No. 20951), and further licensed in the States of MI, ND, TN, UT, and WI.

4) I have worked as a Consulting Engineer serving the Mining & Metals industry continuously since 1989.

5) I am a member of the Society for Mining, Metallurgy & Exploration (SME), and a member of the American Society of Civil Engineers (ASCE). ASCE does not have authority under statute but does require compliance with professional standards of competence and ethics (ASCE Code of Ethics) and has disciplinary powers to censure or expel for violation of the Code;

6) 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, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;

7) I have participated in the preparation of the report entitled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** dated December 14, 2018. I am responsible for contribution to and directing and coordinating other Barr consultants and compiling their inputs to the preparation of Section 20 (Environmental Considerations and Permitting), and portions of Sections 1 (Executive Summary), 3 (Reliance on other Experts), and 26

(Recommendations). I have been remunerated for preparing portions of this report on the basis of a fee for services;

8) I visited the site on August 5, 2018.

9) I have had no prior involvement with the property that is the subject of the Technical Report;

10) At the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

11) I have no personal knowledge, as of the date of the Technical Report, of any material fact or material change which is not reflected in this Technical Report;

12) I am independent of the issuer as defined in section 1.5 of NI 43-101;

13) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form;

This 14th day of December 2018.

Thomas J. Radue

Thomas J. Radue, P. Eng.
Senior Geotechnical Engineer
Barr Engineering Company

GOLDER

CERTIFICATE OF QUALIFIED PERSON BRIAN THOMAS

I, Brian Thomas P.Geo., state that:

(a) I am a Geologist at :
Golder Associates Limited
33 Mackenzie Street, Suite 100
Sudbury, Ontario, P3C 4Y1

(b) This certificate applies to the technical report titled **"Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** with an effective date of: Dec 14, 2018 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows. I am a graduate of Laurentian University with a B.Sc. in Geology from 1994, am a member in good standing of the Association of Professional Geoscientists of Ontario (#1366) and a member in good standing of the Engineers and Geoscientists of British Columbia (#38094). My relevant experience after graduation includes over twenty-four years of experience in mine geology and mineral resource evaluation of mineral projects nationally and internationally in a variety of commodities including 9 years of experience with Vale Nickel in Sudbury (formerly INCO LTD.)

(d) My most recent personal inspection of the property described in the Technical Report occurred on July 16th, 2014 and was for a duration of 1 day.

(e) I am responsible for Items 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23 and relevant portions of Items 1, 3, 25, 26, and 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I have previously participated in the Resource Estimate and First Independent Technical Report for the Tamarack North project with an Effective Date of August 29, 2014 and have completed an interim Resource Estimate of the MSU zone, with an Effective Date of April 3, 2015 publicly disclosed in the April 8, 2015 press release entitled "Talon Metals Announces 167% Increase in Tonnage for the Inferred Massive Sulphide Resource, and an Increase in Grade from 6.42% to 7.26% NiEQ in the Massive Sulphide Unit at Tamarack". I have also participated in the preparation of the technical report titled "**Second Independent Technical Report on the Tamarack North Project – Tamarack, Minnesota"** with an effective date of: March 26, 2018;

(h) I have read National Instrument 43-101. The part of the Technical Report for which I am responsible has been prepared in compliance with this Instrument.

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of Technical Report for which I am responsible, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario this 13th of December 2018.

PROFESSIONAL GEOSCIENTIST
BRIAN E. THOMAS
PRACTISING MEMBER
1366
ONTARIO

B. Thomas

Brian Thomas, P. Geo.
Senior Resource Geologist
Golder Associates Ltd.