NI 43-101 Technical Report

Preliminary Economic Assessment (PEA) #3 of the

Tamarack North Project – Tamarack, Minnesota

Submitted to: Talon Metals Corp.

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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) and is operated by Talon.

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. The 2018 Tamarack Earn-in Agreement came into effect on March 31, 2019 (the Kennecott Agreement Effective Date) and Talon is now the operator of the Tamarack Project.

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:

- **DRA Americas Inc. (DRA):** Mining methods, hydrometallurgical processing, project infrastructure, market studies and contracts, capital and operating costs, and economic analysis;
- **Foth Infrastructure & Environment (Foth):** Environmental studies, permitting, and social or community impacts;
- **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, and mineral resource estimate;
- **Metpro Management Inc. (Metpro):** Mineral processing, metallurgical testing, and recovery methods;
- **Paterson & Cooke Canada Inc. (Paterson & Cooke):** Paste backfill methods;
- **SLR Consulting (Canada) Ltd. (SLR):** Tailings/waste rock co-disposal methods.

1.2 Location and Ownership

The Tamarack Project is located in north-central Minnesota, approximately 89 kilometres (km) (55 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 20,348 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, wetlands, 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.

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% to 17.56%.

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. The Tamarack Earn-in Agreement came into effect on the Kennecott Agreement Effective Date.

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

- The payment of US$6M in cash to Kennecott – this has been completed;
- The issuance of US$1.5M worth of common shares in Talon to Kennecott – this has been completed;

- Within three years of the Kennecott Agreement Effective Date, Talon either spending US$10M in exploration expenditures on the Tamarack Project, or delivering a Pre-Feasibility Study (PFS) in accordance with NI 43-101, whichever comes first; and
- Also within three years of the Kennecott Agreement Effective Date, Talon paying Kennecott the additional sum in cash of US$5M.
- Provided Talon earned a 51% interest in the Tamarack Project, Talon will then have the right to further increase its interest in the Tamarack Project to 60% by:
    - Completing a Feasibility Study on the Tamarack Project within seven years of the Kennecott Agreement Effective Date; and
    - Paying Kennecott the additional sum of US$10M in cash on or before the seventh anniversary date of the Kennecott Agreement Effective Date.

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 Section 20. Since the review and permitting processes have an influence on environmental considerations, Section 20 addresses associated topics, including:

- Summary of results of baseline studies and anticipated additional studies needed for environmental review and permitting;
- Plans for mine waste management, site monitoring, and water management;
- Social and community relations; and,
- Mine closure.

Throughout the regulatory approval processes, Talon is required to demonstrate that the Tamarack North Project can avoid or mitigate potential environmental impacts in accordance with regulatory requirements and stakeholder considerations. That demonstration relies in part on the baseline studies and additional studies and analyses noted in Section 20.

Baseline studies initiated in 2006 focus on hydrology and wetlands in the region. A description of baseline studies conducted to date is provided in Table 20-1. The studies have not identified any environmental issue that could materially impact the ability to mine the resource. Substantial baseline data collection and studies have been completed to date, focusing on hydrology in the region and wetland habitat at the site.

On-going environmental baseline studies have (and continue) to document the following:

- Hydrogeological understanding of the Tamarack North Project area: Stratigraphy and geology in the project area are important to characterize, especially as they relate to water flows and interactions between surface and groundwater.
- Hydrological understanding of the watershed: Surface water monitoring stations have been located on significant water bodies, with data collection focusing on quantity (flows, levels) and water quality (field parameters and laboratory analytes). The data have been collected quarterly since 2006 and data collection continues.
- Geochemistry: Understanding the geochemistry of the ore and waste rock is critical to water management and environmental impact assessment. Geochemical testing has identified and confirmed mineralogical understanding of the ore body. Additional geochemical testing will be needed to optimize methods of water management, waste management, mine backfill approaches, and reclamation alternatives.
- Wetlands, vegetation, and potential presence of rare, threatened, and endangered (RTE) plant species: Studies supporting exploration activities and general infrastructure siting are summarized in Section 20. These resources at the site are consistent with the surrounding region. Vegetative communities include Pine Plantation, Northern Wet-Mesic-Hardwood Forest, and Northern Alder Swamp. Wetlands, lakes, and streams are common in the area, which is rural with agricultural and natural areas. Studies have thus far not identified any listed vegetation species.

As the project moves forward in design and plan for operations, additional environmental studies will be needed. These studies will support the environmental impact analysis specific to the proposed facility. Anticipated future studies include:

- Geology and Minerals – a report describing the resource, host rock, and regional geology. Building on the exploration drilling data, geologic characterization, geophysical testing, mineralogical characterization, and geotechnical characterization of the resource, host rock, and intrusive complex. This information will assist in the underground mine stability analysis and the analysis of groundwater flow in the mine during and after operations.
- Geochemistry and Waste Characterization – performed in accordance with Minnesota rules, with guidance from Minnesota Department of Natural Resources (MDNR).
- Additional hydrological studies to build on current data. This might include additional analytics and surface water monitoring locations and additional groundwater data to characterize the Quaternary system and Precambrian bedrock. Wetlands hydrologic study may also be needed to understand the groundwater surface water interactions.
- Wetlands – formal delineations have a five-year validity. Wetland assessments in hand will need updating. Once the site plan has been confirmed, a Level III assessment will be conducted to support permitting and environmental impact assessment.
- Vegetation, biota, and habitat studies – these studies will likely need updating and revalidation in and around the site area with emphasis on identifying potential listed species. This includes examining terrestrial and aquatic biota.
- Cultural resource studies – tribal, archaeological, and historical resources at the site and in the area will be documented and evaluated according to state and federal requirements. This topic is of great interest to stakeholders including tribes and the local communities. Social and community outreach is currently preliminary and will be developed to engage interested stakeholders.
- Aesthetic resource studies – visual and noise resources will be examined for potential impact on wildlife, the surrounding communities, and the activities common in the area.

Mine waste including tailings and waste rock will be managed in engineered facilities, minimizing potential environmental impacts in accordance with state and federal regulations. An innovative co-disposed filtered tailings facility (CFTF) will manage the low-sulphide (LS) tailings using waste rock for construction and structural stability. The tailings will have low water content, thereby minimizing water management issues and facilitating closure. Materials with the potential to react and produce contaminants will be managed in areas where drainage water will be collected and either used in the operation or treated before discharging to the environment. In concept, high-sulphide (HS) tailings and a portion of the LS tailings will be blended with cement and backfilled to the underground mine excavation, reducing the capacity needs of the CFTF and preventing subsidence.

A rigorous monitoring program will be implemented, building on baseline data. Once the facility is constructed, monitoring will demonstrate permit compliance and identify unanticipated impacts. Monitoring data will be submitted to the agencies regularly and will be accessible to the public, providing transparency.

Water management during operations and post closure will be accomplished in accordance with requirements using the most current tools and industry practices. Managing water on site, conserving mill water, and treating excess water to stringent standards prior to discharge are components of water management. During final reclamation, the CFTF will be fitted with an engineered cover system to prevent potential migration of contaminants into the surface and groundwater systems. Backfilling the mine will be completed to durably prevent migration of contaminants.

The project will undergo an environmental review, likely resulting in preparation of a federal-state Environmental Impact Statement (EIS). Significant permits and approvals will be needed including a Permit to Mine, Section 404 Wetland Permit, an Air Permit, a National Pollutant Discharge Elimination System (NPDES) permit, and others listed in Table 20-2. Project permit applications will be prepared once the project design and operation basis have been established. EIS development and permitting include closure plans and analyses to assure satisfactory long-term environmental conditions. A detailed closure plan will be developed in future studies.

1.4 Geology and Mineralization

The Tamarack Intrusive Complex (TIC) is an ultramafic to mafic intrusive complex that hosts Ni-Cu-Co sulphide mineralization with associated platinum (Pt), palladium (Pd) (PGEs) and gold (Au). The TIC is a multi-magmatic phase intrusion that consists of a minimum of two pulses: The fine grained ortho-cumulate olivine (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

Midcontinent 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 mineral resource and identified exploration targets, is over 7 km long and is the focus of this PEA.

The nickel (Ni)-copper (Cu)-cobalt (Co) sulphide mineralization with associated PGEs and Au formed 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 (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, four lesser-defined mineral zones, namely the 480 Zone, 221 Zone, 164 Zone and the CGO Bend Zone.

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 Zone	Peridotite and Feldspathic Peridotite	MZ/FGO	Disseminated and net textured sulphides
	CGO Bend	Feldspathic Peridotite	CGO	Disseminated sulphides
		Peridotite footwall (basal FGO mineralization)	FGO	Disseminated sulphides, 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 magnetics and electromagnetics (AEM) (fixed wing and helicopter based);
- Ground magnetics;
- Surface electromagnetics (EM);
- Surface gravity;

- Magnetotellurics (MT);
- Induced polarization (IP);
- Seismic;
- Mise-à-la-masse (MALM);
- Magnetometric resistivity (MMR);
- Downhole electromagnetics (DHEM).

Kennecott conducted extensive drilling at the Tamarack North Project since 2002. This drilling has comprised 260 diamond drill holes totalling 112,394.22 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

The qualified person (QP) of the mineral resource estimate 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.

It is the QP's opinion that the sampling process is representative of the mineralization at Tamarack North and that the sample preparation and QA/CQ procedures used, and the sample chain of custody were found to be consistent with Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Mineral Exploration Best Practice Guidelines (November 2018).

1.7 Data Validation

Golder compared recent assay data (2017, 2020) from the Kennecott database to the original assay certificates from ALS Chemex for the entire sample population used for resource estimation. Minor errors were identified during this review that were found to not be material to the mineral resource estimate.

During the 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 material changes to the drilling, logging, sampling, or chain of custody procedures since the 2014 site visit; therefore, it is the QP's opinion that the Tamarack North Project drill hole database has been prepared in accordance to CIM Estimation of mineral resources and Mineral Reserves Best Practise Guidelines (November 2018) and is of suitable quality to support the mineral resource estimate in this PEA.

1.8 Mineral Processing and Metallurgical Testing

The flotation flowsheet and conditions that were established in the 2016/2017 program were further optimized using a life-of-mine (LOM) composite that represented the entire 8.02 Mt of mineralized material that was reported in the March 2020 PEA. The head grade of this LOM composite was 1.69% Ni and 0.95% Cu. The primary focus of the program was to produce Ni and Cu concentrates that provide marketing optionality. The program considered three possible scenarios for the flotation concentrates:

- The **Ni Concentrate Scenario** would include shipping both Ni and Cu concentrates to smelters for processing.

- The **Ni Powder Scenario** would include shipping Cu concentrate to a smelter for processing, and transferring Ni concentrate to a co-located facility for production of Ni powder.
- The **Ni Sulphate Scenario** would still ship the Cu concentrate to smelters, but the Ni concentrate would be converted to Ni sulphates in a hydrometallurgical facility.

The flotation program on the LOM composite aimed to produce a Ni concentrate of at least 10.5% Ni to ensure marketability to a smelter. The simplified flowsheet that was developed for the March 2020 PEA was confirmed. The flowsheet comprises a bulk rougher, followed by bulk cleaning of the bulk rougher concentrate and Cu/Ni separation. A desulphurization stage is treating the bulk rougher tailings to produce high-sulphur and low-sulphur tailings streams. The high-sulphur tailings will be placed underground in the form of paste backfill.

The metallurgical projections that were developed for the March 2020 PEA were validated and adjustments were made for the LOM composite to take into account the addition of the 138 Zone mineralization, which displays a distinctively different metallurgical response.

A locked cycle flotation test (LCT) was completed on the LOM composite and the results are presented in Table 1-2.

Table 1-2: Simplified Circuit Mass Balance

Product	Weight	Assays, %					% Distribution				
	%	Cu	Ni	S	Fe	MgO	Cu	Ni	S	Fe	MgO
Cu Conc	2.2	29.9	1.13	32.5	32.5	0.80	71.6	1.6	12.9	4.8	0.1
Ni Conc	11.9	1.22	10.7	28.6	40.6	4.66	15.9	83.2	61.8	32.8	2.4
Bulk 1st Clnr Scav Tails	9.3	0.40	0.74	6.42	17.5	22.7	4.0	4.5	10.8	11.0	9.0
Bulk Scavenger Tails	76.6	0.10	0.21	1.05	10.0	27.1	8.5	10.7	14.6	51.4	88.6
Combined	100.0	0.92	1.53	5.54	14.8	23.4	100.0	100.0	100.0	100.0	100.0

A scoping level hydrometallurgical program was completed to evaluate the amenability of the Ni concentrate to leaching and downstream processing. The program included pressure oxidation (POX) and Albion leach tests, followed by neuralization tests to remove most impurities. Metal extraction rates in the POX tests were over 99% for Ni and Co and 88% for Cu. The extraction values were similar for the Albion test at 97% to 99%. Ni and Co losses in the neutralization stages were minimal at 1.1% and 0.9%, respectively. Most of the Cu was recovered into product streams that would be combined with the Cu flotation concentrate to maximize revenue. Downstream tests will continue for the next 12 months to produce a sample of a battery grade Ni sulphate.

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 (COG) 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 January 8, 2021. Mr. Brian Thomas is an independent QP pursuant to NI 43-101.

Table 1-3: Tamarack North Project Mineral Resource Estimate (January 8, 2021)

Domain	Classification	%Ni Cut-Off	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
Upper SMSU	Indicated Resource	0.5	1,462	1.32	0.78	0.04	0.17	0.11	0.11	1.81
Lower SMSU	Indicated Resource	0.5	2,340	2.08	1.10	0.05	0.55	0.34	0.25	2.87
MSU	Indicated Resource	0.5	124	5.72	2.36	0.12	0.60	0.46	0.23	7.23
Total	**Indicated Resource**	0.5	**3,926**	**1.91**	**1.02**	**0.05**	**0.41**	**0.26**	**0.20**	**2.62**
Upper SMSU	Inferred Resource	0.5	2,652	0.76	0.47	0.02	0.25	0.14	0.12	1.10
Lower SMSU	Inferred Resource	0.5	115	0.86	0.51	0.02	0.57	0.36	0.24	1.34
MSU	Inferred Resource	0.5	443	5.93	2.52	0.12	0.70	0.52	0.26	7.53
138	Inferred Resource	0.5	3,953	0.82	0.63	0.02	0.21	0.12	0.14	1.21
Total	**Inferred Resource**	0.5	**7,163**	**1.11**	**0.68**	**0.03**	**0.26**	**0.16**	**0.14**	**1.57**

- All resources reported at a 0.5% Ni 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 $25.00/$8.00 + Pt [g/t]/31.103 x $1,000/$8.00/22.04 + Pd [g/t]/31.103 x $1,000/$8.00/22.04 + Au [g/t]/31.103 x $1,300/$8.00/22.04. No adjustments were made for recovery or payability in the calculation of NiEq.

The mineral resources are derived from a Datamine-constructed block model (block sizes = 5 m by 5 m by 5 m for the SMSU and the 138 Zone; with 2.5 m x 2.5 m x 2.5 m sub-blocks for the MSU) of three mineral domains and are reported above a Ni cut-off of 0.5%. All domains were "unfolded" and had top cuts applied to restrict outlier values (Pt, 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 (SMSU, 138) and 1.0 m (MSU) composited drill holes. Specific Gravity (SG) estimates were based on laboratory 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.

The QP 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 mineral resource estimate may be materially impacted by the following:

- Changes in the break-even COG, as a result of changes in mining costs, processing recoveries, or metal prices;
- Changes in geological knowledge/interpretation, as a result of new exploration data.

1.10 Mining Methods

The underground mine will use underground mining methods with the objective of utilizing the best available technologies (BATs) that are as efficient, practical, and as environmentally responsible as possible. The mining methods and infrastructure for both the Ni Concentrate and Ni Sulphate Scenarios will be the same. Both bulk and selective mining methods (long hole stoping and drift and fill respectively) will be used. All stopes will be filled with a blend of development waste rock, where possible and paste backfill. Paste fill will be produced in a dedicated paste plant on surface adjacent to the mill and comprise 100% of the HS tailings as well as a portion of the LS tailings.

The primary access for personnel and materials will be a decline ramp from surface, which will also serve as the main fresh air intake for the mine and the main conduit for mine services. A multi-purpose shaft will be driven in close proximity to the mill, which will be the main exhaust for the mine. Additionally, it will house a vertical conveyor, which will serve as the primary materials handling system, secondary egress and redundant services, where required.

The underground mine will use a full battery/electric fleet containing no diesel powered equipment. Lateral development, including the main decline, and drift and fill mining will be predominantly done using a continuous miner. A jumbo will be utilized for traditional drill/blast mining, where the ground conditions and logistics prohibit the continuous miner from mining effectively.

The proposed mine plan includes the combined production from of each the mineral domains in the resource model (i.e., 138, MSU, Upper SMSU and Lower SMSU). The planned production rate is approximately 3,600 tonnes per day (tpd) (1.3 million tonnes per annum (Mtpa)). A summary of the mine plan is shown below:

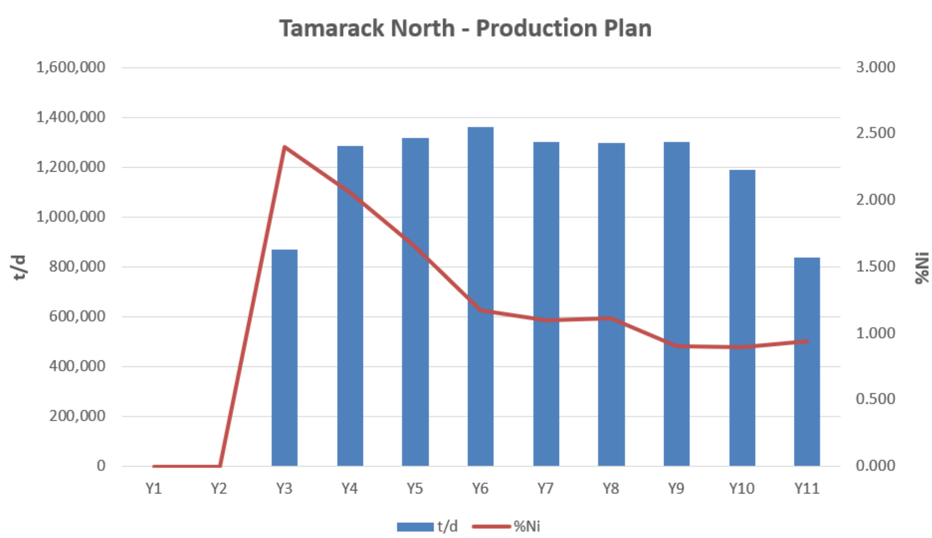


Figure 1-1: Mine Production Plan

Major infrastructure including maintenance and storage will be located on surface. An allowance for minor maintenance and temporary storage underground has been made in the mine plan and will be centralized in close proximity to the main ramp.

1.11 Recovery Methods

The process plant design is based on an average daily mill feed rate of 3,600 tpd for all three scenarios. The average LOM head grade is 1.34% Ni and 0.74% Cu. The plant feed characteristics and metallurgical performance are summarized in Table 1-4.

Table 1-4: Plant Feed Characteristics and Metallurgical Performance

Criteria	Units	Value		Source
		Expected Range	Design	
Solids SG	t/m^3	2.60 – 3.75	2.90	D
Run of Mine (ROM) Bulk Density	t/m^3	1.60 – 2.00	1.80	B
LOM Mill Ni Head Grade	% Ni	0.52 – 6.03	1.34	D
LOM Mill Cu Head Grade	% Cu	0.24 – 2.41	0.74	D
Mill Treatment Capacity	ktpa		1,314	C/D
Ni Recovery to Ni Concentrate	%		81.5	E/C
Ni Concentrate Grade	% Ni		10.2	E/C
Ni Concentrate Production	ktpa		141.9	E/C
Overall Cu Recovery	%		84.7	E/C
Cu Recovery to Cu Concentrate	%		69.3	E/C
Cu Concentrate Grade	% Cu		28.5	E/C
Cu Concentrate Production	ktpa		23.6	E/C

The metallurgical process consists of bulk rougher followed by two stages of cleaning of the rougher concentrate. The 2nd cleaner 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 the form of wet filter cake for the Ni Concentrate Scenario.

For the Ni Sulphate and Ni Powder Scenarios, only the Cu concentrate is shipped to a smelter. For the Ni Sulphate Scenario, the Ni concentrate is subjected to hydrometallurgical treatment comprising POX leach, neutralization, Cu removal, Ni/Co solvent extraction (SX), Co SX, and magnesium (Mg) removal. For the Ni Powder Scenario, the Ni concentrate is transferred to a co-located facility for the production of Ni powder.

The bulk rougher tailings are treated in a desulphurization stage to produce a low-mass HS stream and high-mass non-acid-generating (NAG) tailings. In concept, HS tailings will be placed underground in form of cemented paste backfill together with a portion of the LS tailings. The balance of the LS tailings will be placed in a CFTF. During the start of the mining activities, minimal HS tailings may report to the CFTF when voids are not available. In such situations, HS tailings will be distributed within LS tailings and covered in the 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 for the concentrator. The hydrometallurgical circuit equipment has also been used in numerous commercial operations and includes equipment such as POX autoclaves, mixing tanks, SX mixers and settlers, thickeners, and belt filters. Initial dewatering is generally performed in high-rate thickeners followed by filter presses or belt filters.

The concentrator 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 Depramin C, and pH modifier lime. Flocculants will be employed to assist in the dewatering of the concentrates and tailings streams.

The hydrometallurgical plant will also employ typical reagents such as oxygen, ferric chloride, pH modifiers (limestone, lime), sodium hydrogen sulphide (NaHS), SX diluent and extractant, and flocculants.

The total connected power for the concentrator is 9.8 MW, with 85% drawn. The total connected power of the hydrometallurgical plant is 6.0 MW with 85% 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 90 acres.

The CFTF will require approximately 75 acres. The remainder of the site layout area comprises decline and ore bin, mine and mill services building, communication, and concentrator facilities, hydromet plant, paste backfill plant temporary development rock storage, water treatment plant, mine offices, warehouse, and workshops, vehicle washing bays, security gatehouse and parking areas.

1.13 Capital Costs

The total estimated capital cost for either the Ni Powder Scenario or the Ni Concentrate Scenario is US$394.99M of which US$315.80M is the initial cost required during the first three years, including the first production year. The total estimated capital cost of the Ni Sulphate Scenario is US$646.44M, of which US$552.61M is the initial cost required during the first three years, including the first production year. The amounts include indirect costs and contingency.

Table 1-5: Tamarack North Project Capital Expenditure (CAPEX) Summary

US$M	Ni Powder Scenario or Ni Concentrate Scenario			Ni Sulphate Scenario		
Area	**Initial Cost (US$M)**	**Sustaining Cost (US$M)**	**Total Cost (US$M)**	**Initial Cost (US$M)**	**Sustaining Cost (US$M)**	**Total Cost (US$M)**
Mine	130.15	70.32	200.47	130.15	70.32	200.47
Process and Surface Facilities	167.51	22.01	189.51	390.56	50.41	440.97
Closure Costs other than CFTF	-	10.00	10.00	-	10.00	10.00
Salvage Value of Mill	-	(5.00)	(5.00)	-	(5.00)	(5.00)
Sub Total	**297.66**	**97.33**	**394.99**	**520.71**	**125.73**	**646.44**
Working Capital	18.15	(18.15)	-	31.90	(31.90)	-
Total	**315.80**	**79.18**	**394.99**	**552.61**	**93.83**	**646.44**

*May not total due to rounding

1.14 Operating Costs

The average operating cost per tonne milled for the nine year mine life is US$48.15/t of mill feed in the Ni Powder Scenario, US$75.99/t of mill feed in the Ni Sulphate Scenario and US$56.54/t of mill feed in the Ni Concentrate Scenario, all of which is detailed in the table that follows.

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

Cost Category	**Operating Cost (US$/t of Mill Feed)**		
	Ni Powder Scenario	**Ni Sulphate Scenario**	**Ni Concentrate Scenario**
Mining	$27.49	$27.49	$27.49
Processing (milling/concentrating)	$14.25	$14.25	$14.25
Hydrometallurgical Refining	-	$26.68	-
Product Handling, Transportation, Losses, and Insurance	$1.90	$2.22	$10.29
CFTF	$0.75	$0.75	$0.75
G&A	$3.76	$4.60	$3.76
Total OPEX *	**$48.15**	**$75.99**	**$56.54**

** May not total due to rounding*

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.

The financial model is based on the results of the PEA which is preliminary in nature and includes inferred resources that are considered too speculative geologically to have the economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The objective of the study is to determine the viability of the proposed facilities to mine and process the Tamarack North Project mineralized material. In order to do this, the cash flow arising from the base case was forecast, enabling a computation of NPV and IRR. The sensitivity of this NPV and IRR to changes in the base case assumptions is then examined.

Three scenarios, as detailed in Section 19 “Market Studies and Contracts”, were modelled:

Table 1-7: PEA Scenarios

	Scenario	Description
1	**Nickel Powder Scenario**	Nickel concentrates produced at site and thereafter, used to produce refined nickel powder by a third party for the EV market
2	**Nickel Sulphate Scenario**	Nickel concentrates from the project are refined at site in a hydrometallurgical process to produce nickel sulphates which are sold to the EV market
3	**Nickel Concentrate Scenario**	Nickel concentrates produced at site are transported and sold to a smelter, who in turn transports it to a refinery to produce LME grade nickel primarily for the stainless steel market

“Base Case”, “Low” and “Incentive” metal prices are presented in Table 1-8 and are in “real” (i.e. without inflation) dollars.

Table 1-8: 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	$3.50
Co	US$/lb	15.00	25.00	$30.00
Pt	US$/oz	$1,000	$1,000	$1,000
Pd	US$/oz	$1,000	$1,000	$1,000
Au	US$/oz	$1,300	$1,300	$1,300

The base case cash flow, 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 1-9. Results are also shown at comparative discount rates of 8% and 10% and on a pre-tax basis.

Table 1-9: Base Case NPV for all Scenarios in Million US$ and IRR

	Discount rate	Base Case Pricing		
		Nickel Powder Scenario	Nickel Sulphate Scenario	Nickel Concentrate Scenario
Pre-tax	**7%**	688	711	629
NPV in	**8%**	646	660	589
$ millions	**10%**	570	568	518
Pre-tax IRR		56.0%	37.6%	52.6%
After-tax	**7%**	**567**	**569**	**520**
NPV in	**8%**	530	524	485
$ millions	**10%**	463	443	423
After-tax IRR		**48.3%**	**31.9%**	**45.6%**
Initial CAPEX and working capital in $ millions		**316**	**553**	**316**

After-tax NPV at a 7% discount rate, initial CAPEX including working capital and after-tax IRR at base case pricing are illustrated in Figure 1-2 below.

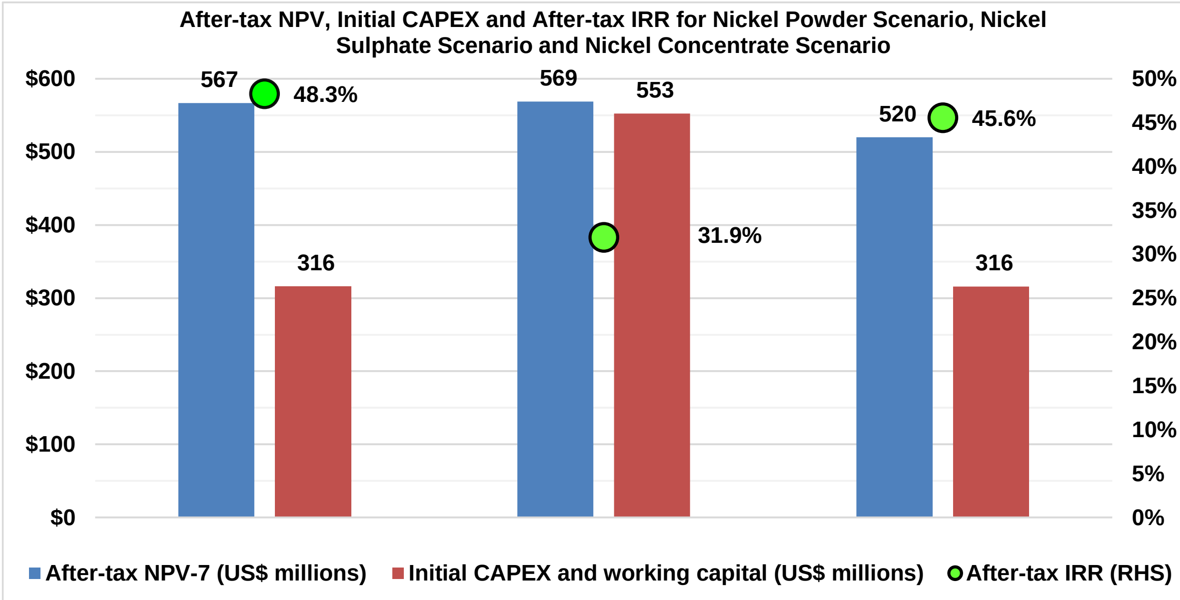


Figure 1-2: After-tax NPV, Initial CAPEX and Working Capital, and After-tax IRR for all Scenarios

The sensitivities of the after-tax and pre-tax NPV and IRR as well as other measures for all scenarios were tested using alternate metal price assumptions and discount rates as shown in Table 1-10.

Table 1-10: After-Tax and Pre-tax NPV Sensitivity Analysis and Additional Metrics

		Nickel Powder Scenario			Nickel Sulphate Scenario			Nickel Concentrate Scenario		
	Discount rate	Metal Price Case			Metal Price Case			Metal Price Case		
		Low	Base	Incentive	Low	Base	Incentive	Low	Base	Incentive
Pre-tax NPV	**7%**	496	688	917	478	711	970	439	629	854
US$ millions	**8%**	463	646	863	438	660	906	409	589	803
	10%	404	570	767	367	568	790	355	518	712
Pre-tax IRR		45.0%	56.0%	67.4%	29.2%	37.6%	45.7%	41.5%	52.6%	64.2%
After-tax NPV	**7%**	**415**	**567**	**744**	**387**	**569**	**769**	**369**	**520**	**695**
US$ millions	**8%**	386	530	698	351	524	714	342	485	651
	10%	333	463	616	286	443	615	293	423	573
After-tax IRR		**39.3%**	**48.3%**	**57.7%**	**25.1%**	**31.9%**	**38.6%**	**36.4%**	**45.6%**	**55.1%**
EBITDA margin		64%	**68%**	70%	60%	**64%**	66%	60%	**64%**	67%
EBIT margin		43%	**50%**	55%	34%	**41%**	47%	39%	**46%**	52%
Payback from start of production (pre-tax, undiscounted)		1.6	**1.4**	1.2	2.2	**1.8**	1.6	1.7	**1.4**	1.2
Payback from start of production (after-tax, undiscounted)		1.8	**1.5**	1.3	2.4	**2.1**	1.8	1.9	**1.6**	1.4

1.16 Conclusions

The PEA demonstrates a high after-tax IRR, low All-in Sustaining Cost (AISC), low capital intensity and a quick payback for the Tamarack Nickel Project. The PEA also clearly demonstrates that the Tamarack Nickel Project has the optionality to produce either Ni sulphates or concentrates for refined Ni powders to be used for the EV market or a Ni concentrate for the stainless steel market, with all contemplated scenarios having robust economics.

1.17 Recommendations

During 2021 Talon should primarily focus on resource expansion and definition to collect data required to complete a PFS and a Feasibility Study. It is recommended that between 25,000 and 30,000 m of drilling be completed for this purpose. Talon's in-house team of experienced specialists operate their own drilling and geophysical equipment efficiently and at low cost. It is therefore believed that this is achievable during 2021.

Talon has a comprehensive geotechnical logging program in place and should therefore continue with laboratory testing of drill core, collecting down hole data using acoustic televiewer (ATV) and full wave sonic technology, as well as in-situ stress measurement testing. Hydrological work should be conducted as appropriate for each level of study. It is recommended to install multilevel vibrating wire piezometers in selected historical drill holes and to conduct additional aquifer property testing within the glacial till and bedrock aquifers.

Geo-metallurgical testing programs should continue and should be based on the predicted LOM feed. Ni concentrates produced from geo-metallurgical testing programs should be used to complete the second phase of hydrometallurgical testing to produce Ni sulphates. Ni concentrates should also be used to develop a flowsheet that produce both Ni and iron (Fe) powders for use in battery precursor and battery cathode (see Section 19: Market Studies and Contracts). Waste products from geo-metallurgical testing should be used to continue environmental test work.

Detailed study recommendations are noted in Section 26.

2 INTRODUCTION

DRA was retained by Talon to participate in this independent PEA, in collaboration with various consulting companies, including DRA, Foth, Golder, Metpro, SLR, 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. More specifically, each consultant contributed to the completion of the component PEA sections as follows:

- **DRA**: Mining methods, hydrometallurgical processing, project infrastructure, market studies and contracts, capital and operating costs, and economic analysis;
- **Foth**: Environmental studies, permitting, and social or community impacts;
- **Golder**: Property description and location, accessibility, climate and physiography, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation, data verification, adjacent properties, and mineral resource estimate;
- **Metpro**: Mineral processing, metallurgical testing, and recovery methods;
- **Paterson & Cooke**: Paste backfill methods;
- **SLR**: Tailings / waste rock co-disposal methods.

This PEA demonstrates a conceptual mine development plan based on BATs. 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 PFS as articulated in Section 26 (Recommendations) of this document.

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 January 8, 2021. 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, including hydrometallurgical test work performed in 2020, has been compiled by Mr. Oliver Peters, P. Eng. Mr. Peters is the Principal Metallurgist and President of Metpro. This work is an update of previous metallurgical work completed on the Tamarack North Project by both

Metpro and Hatch Ltd. The recently performed hydrometallurgical test work has also been reviewed by Dr. Volodymyr Liskovych, P. Eng.

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 and by Kennecott. This PEA is based on the following data and pre-existing reports:

- PEA of the Tamarack North Project published in December 2018;
- Updated PEA of the Tamarack North Project published in March 2020;
- 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;
- Talon internal reports;
- Kennecott internal reports;
- Kennecott database of surface drill holes that included:
  - Ni, Cu, Co, Pt, Pd, Au, lithology sample/assay data;
  - Sample SG;
  - 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)

The QPs listed in Table 2-1 are responsible for the preparation of this PEA. Certificates are also contained herein. The following QPs have completed property site visits:

- Brian Thomas, P. Geo., completed site visit on July 16, 2014;
- Daniel Gagnon, P. Eng., completed site visit on October 5, 2017;
- Andrea Martin, P.E., completed site visits on April 18, 2014, January 23, 2019, and January 27, 2020.

Table 2-1: List of Responsible QPs

Name	Title, Company	Responsible for Section
Leslie Correia, Pr.Eng	Engineering Manager, Paterson & Cooke Canada Inc.	portions of 1 and 16
Tim Fletcher, P. Eng.	Senior 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.	19, 22, and portions of 1, 18, 21, 25, and 26
André-François Gravel, P. Eng.	Senior Mining Engineer, DRA Americas Inc.	portions of 1 and 16
Volodymyr Liskovych, P. Eng.	Principal Process Engineer, DRA Americas Inc.	13, 17, and portions of 1, 21, 25, 26, and 27 (hydromet aspects)
Andrea Martin, P.E.	Lead Environmental Engineer, Foth Infrastructure & Environment	20, and portions of 1, 3, and 26
Oliver Peters, P. Eng.	Principal Metallurgist and President, Metpro Management Inc.	13, 17, and portions of 1, 21, 25, 26, and 27 (mineral processing aspects)
David Ritchie, P. Eng.	Mine Waste Engineering Manager, SLR Consulting (Canada) Ltd.	portions of 1, 3 and 18
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

For all sections, Talon participated in the preparation of the report under the supervision of the QPs named above.

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
AISC	All in Sustaining Cost
Al	Aluminium
Al_2O_3	Alumina, aluminum oxide
AMT	Audio-frequency magnetotellurics
AP	Acid Potential
ARD	Acid rock drainage
As	Arsenic
ATV	Acoustic televiewer
Au	Gold
Avg	Average
Azm	Azimuth
B.Sc	Bachelor of Science
BATs	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)
BTS	Brazilian tensile strength
BWi	Bond Work Index
Ca	Calcium
$CaCO_3$	Calcium carbonate
CAPEX	Capital expenditure
CCD	Counter-current decantation
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/s	Centimetres per second
cm^3	Cubic centimetre
CMC	Carboxy methyl cellulose
Co	Cobalt
COG	Cut-off Grade
Cpy	Chalcopyrite
Cr	Chromium
CRM	Certified reference material
CSAMT	Controlled source audio-frequency magnetotellurics
.csv	Comma-separate values file (electronic file format)
Cu	Copper
$CuSO_4$	Copper sulphate
CV	Coefficient of Variation
DDR	Dip Direction
DHEM	Downhole Electromagnetic
DMS	Dense media separation
dmt	Dry metric tonnes
DRA	DRA Americas Inc.
E	East
EAW	Environmental Assessment Worksheet
EDA	Exploratory data analysis
EIS	Environmental Impact Statement
EM	Electromagnetic
EMIT	Electromagnetic Imaging Technology
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement, and Construction Management
EPS	Enhanced Production Scheduler
EV	Electric Vehicle
Fe	Iron
FGO	Fine grained ortho-cumulate olivine
FKL	Frontier Kemper Lakeshore
Fo	Forsterite
Foth	Foth Infrastructure & Environment
ft	Feet
FW	Footwall
G&A	General and Administrative
g	Gram
g/cc	Gram per cubic centimetre
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
GPa	Gigapascal
gpm	Gallons per minute
GPS	Global positioning system
HELP	Hydrogeologic Evaluation of Landfill Performance
Hg	Mercury
HLS	Heavy liquid separation
HPAL	High Pressure Acid Leach
HQ	Hole (outside diameter): 96 mm; core (inside diameter): 63.5 mm
HS	High-sulphide
HW	Hanging wall
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
IFRS	International Financial Reporting Standards
In	Indium
IP	Induced polarization
IPD	Inverse power distance
IRR	Internal rate of return
ISO	International Organization for Standardization
ITH	In-the-hole
JCR	Joint condition rating
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
LG	Low-grade
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
MAWP	Maximum allowable working pressure
MCR	Mid Continent Rift
MDH	Minnesota Department of Health
MDNR	Minnesota Department of Natural Resources
MEPA	Minnesota Environmental Policy Act
Metpro	Metpro Management Inc.
Mg	Magnesium
mg/L	Milligrams per litre
MgO	Magnesium oxide, magnesia
mGal	Milligal
MGS	Minnesota Geological Survey
MHP	Mixed hydroxide precipitate
MIBC	Methyl isobutyl carbinol
mL	Millilitre
ML	Metal leaching
mm	Millimetre
MMR	Magnetometric resistivity
MMS	Mixed massive sulphide
Mn	Manganese
Mo	Molybdenum
MOU	Memorandum of Understanding
MPa	Megapascal
MPCA	Minnesota Pollution Control Agency
MPUC	MN Public Utility Commission
MRV	Minnesota River Valley
MSO	Mineable Shape Optimizer
MSU	Massive sulphide unit
MT	Magnetotelluric
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
NaHS	Sodium hydrogen sulphide
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
NPC	Net present cost
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
OEM	Original Equipment Manufacturer
OK	Ordinary Kriging
OPEX	Operating expenditure
OTV	Optical televiewer
oz	Ounce (troy ounce - 31.1035 grams)
P. E.	Professional Engineer (US designation)
P. Eng.	Professional Engineer (Canadian designation)
P. Geo.	Professional Geologist
Pr. Eng.	Professional Engineer (South African designation)
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
POX	Pressure oxidation
PPI	Producer price inflation index
ppm	Parts per million
psi	Pounds per square inch
Pt	Platinum
Q'	Rock mass quality
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
RIM	Radio Imaging Method
RMR	Rock mass rating
ROD	Record of Decision
ROFR	Right of first refusal
ROM	Run of mine
RQD	Rock quality designation
RTE	Rare, Threatened, and Endangered
S	South
S	Sulphur
Sb	Antimony
SCR	Solid core recovery
SDD	Scoping Decision Document
SDS	State Disposal System
SE	Southeast
Se	Selenium
SEAW	Scoping Environmental Assessment Worksheet
SED	Metasediments
SEM	Sequential excavation method
SG	Specific gravity
SHPO	State Historic Preservation Office
Sigci	Uniaxial Compressive Strength
Sigt	Tensile Strength
SIPX	Sodium isopropyl xanthate
SLR	SLR Consulting (Canada) Ltd.
SMSU	Semi-massive sulphide unit
SO	Stope optimizer
SPLP	Synthetic Precipitation Leaching Procedure
SSTS	Subsurface Sewage Treatment System
STB	Surface Transportation Board
STP	Step data
.stp	Step file (electronic file format)
SW	Southwest
SWPPP	Stormwater Pollution Prevention Plans
SX	Solvent extraction
T, t	Tonnes
t/m^3	Tonnes per cubic metre
Talon	Talon Metals Corp.
TCS	Triaxial compressive strength
TCR	Total core recovery
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 (Chapter 11), Unfolded coordinate system (Chapter 14)
UIC	Underground Injection Control
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
VWP	Vibrating wire piezometer
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, Foth, Golder, Metpro, SLR, and Paterson & Cooke for Talon. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to DRA, Foth, Golder, Metpro, SLR, 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.

For Environmental Studies, Permitting and Community Impact (Section 20) and associated sub-sections, Foth has relied upon information it has developed as contracted by Kennecott Exploration Company and information provided by Talon for baseline data related to site hydrogeology, hydrology, geochemistry, wetlands, vegetative communities, and protected species. For Permitting Requirements (Section 20.7), Foth 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 historical metallurgical programs. Metpro managed the 2019/2020 metallurgical test programs at XPS in Sudbury and SGS in Lakefield on behalf of Talon to further improve the metallurgical results for the Tamarack mineralization. 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.

SLR has relied upon backfill / waste tonnages provided by Paterson & Cooke, mine plan tonnages and geochemical characterization of the tailings and mine rock provide by Talon, and site selection guidance by Talon.

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 20,348 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).

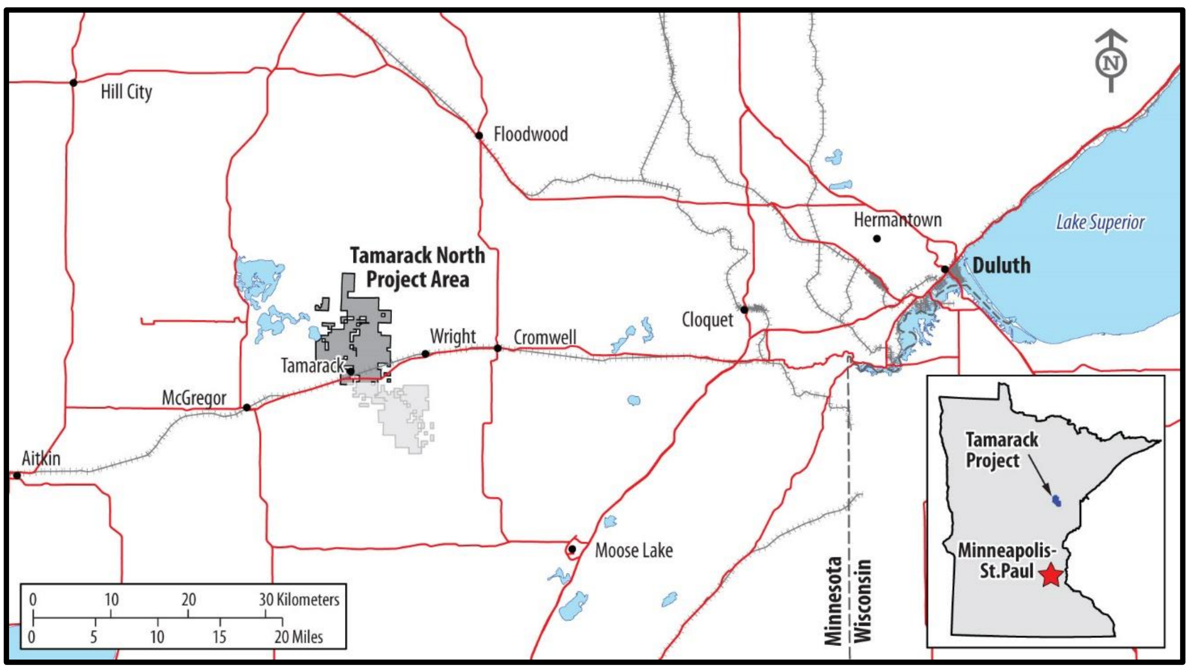


Figure 4-1: Location of the Tamarack North Project

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. Talon is presently the operator of 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 (the Kennecott Agreement came into effect on the Kennecott Agreement Effective Date). 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 several 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 the original 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
TOTAL	**$**	**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% NSR 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%, and eventually diluted to 17.56%.

4.2.3 2018 Tamarack Earn-in Agreement

On November 7, 2018, Talon and Kennecott entered into the 2018 Tamarack Earn-in Agreement. The 2018 Tamarack Earn-in Agreement came into effect on the Kennecott Agreement Effective Date. Pursuant to the 2018 Tamarack Earn-in Agreement, 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 has taken over operatorship of the Tamarack Project (with certain Kennecott employees being seconded to Talon) and has the right to increase its interest in the Tamarack Project to 51% by:

- The payment of US$6M in cash to Kennecott – this has been completed;
- The issuance of US$1.5M worth of common shares in Talon to Kennecott – this has been completed;
- Within three years of the Kennecott Agreement Effective Date, Talon either spending US$10M in exploration expenditures on the Tamarack Project or delivering a PFS in accordance with NI 43-101, whichever comes first; and
- Also within three years of the Kennecott Agreement Effective Date, Talon paying Kennecott the additional sum in cash of US$5M.
- Provided Talon has earned a 51% interest in the Tamarack Project, Talon will then have the right to further increase its interest in the Tamarack Project to 60% by:
    - Completing a Feasibility Study (as defined under the 2018 Tamarack Earn-in Agreement) on the Tamarack Project within seven years of the Kennecott Agreement Effective Date; and
    - Paying Kennecott an additional sum of US$10M in cash on or before the seventh anniversary date of the Kennecott Agreement Effective Date.

4.2.4 The New MVA

Upon Talon earning 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 fund 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 (subject to any additional dilution that may apply).

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 and surface rights owned or controlled by Kennecott and Talon are summarized in Figure 4-2. 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. The Tamarack land package has been reduced from 28,334 acres (2018) in order to save costs and shed non-essential land holdings.

Table 4-1: Summary of Tamarack North Project Interests

Type	Number	Acreage
Minnesota State Leases	40	18,730
Private Mineral Leases	1	38
Fee Minerals and Surface Interests	18	1,580
Total	**59**	**20,348**

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.

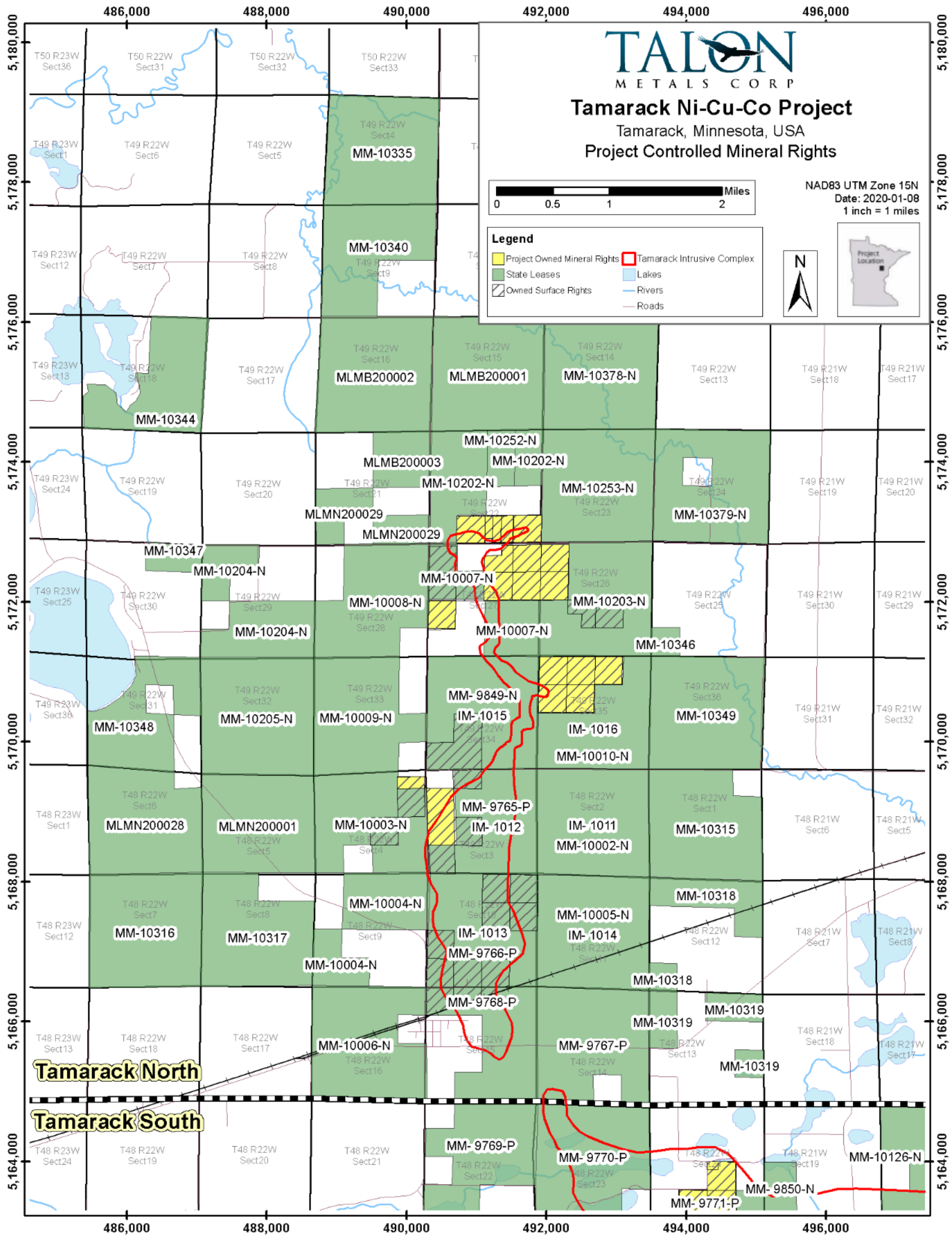


Figure 4-2: Tamarack North Project Mineral and Surface Rights

4.2.6.2 Minnesota State Leases

State Leases to Explore, Mine and Remove Metallic Minerals (State Leases) are issued by the MDNR 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 tonne of raw ore increases. This escalation of the royalty rate begins at a net return value per tonne of US$75.01. It rises to the maximum of 20% if such net return value exceeds US$444/t 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
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

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
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
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

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
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
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

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
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
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
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

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
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
MM 10378-N	3/4/2011	50 years	3.95%	0.55%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 14: W/2NW/4, SE/4NW/4, NE/4SW/4, SW/4SW/4, SE/4SW/4 Minerals, mineral rights and surface Sec. 14: 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 Sec. 14: 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	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 24: 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

State Lease Number	Start Date	Term	Base Royalty	Additional Royalty	Royalty Escalator Applies	Lands	Acreage
MLMB200001	3/3/2016	50	3.95%	0.75%	Yes	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 15: 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 Sec. 15: 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	**Township 49 North, Range 22 West, Aitkin County, Minnesota** Sec. 16: W1/2-NE1/4, NW1/4, S1/2, E1/2-NE1/4 Mineral and mineral rights	640
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 a surface use agreement covering privately owned surface interests (Private Agreement). There is currently one Private Agreement, which covers approximately 38 acres of surface use within the Tamarack North Project area. Table 4-3 provides further information on the Private Agreement.

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 Agreement

Type of Agreement	Term	Annual Fee (US$)	Lands	Acreage
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 1,580 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, NE/4 NW/4	240 (Surface and Mineral)

Township	Range	Section	Acreage
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. 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)
49 North	22 West	Sec. 27: SWNW excepting certain lands	37.96 (Surface Only)
49 North	22 West	Sec. 27: NWSW excepting certain lands Sec. 27: SENW excepting certain lands	78.18 (Surface and Mineral) (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 MDNR 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 2018 Tamarack Earn-in Agreement are detailed in Table 4-4.

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, is responsible for making application for the required permits and approvals for exploration. 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, MDNR, State Historic Preservation Office (SHPO), Minnesota Department of Health (MDH), Minnesota Pollution Control Agency (MPCA), Aitkin 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**
MDNR	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
MDNR	Burning Permit
MDNR	Permit to Work in Public Waters, including Public Waters Wetlands
MDNR	Water Appropriation Permit
MDNR	Wetland Conservation Act approvals for activities impacting certain wetlands
MDNR	Threatened and Endangered Species Review
Local	
Agency	**Permit/Approval**
City of Tamarack	Zoning and Building Permits
County	Conditional Use Permit
County	Zoning Permits

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 RTE species and vegetative community surveys.

Initiated in 2007/2008, Kennecott monitored 23 surface water locations and 12 ground water wells. As of 2014, Kennecott operated 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 was completed by Foth, of De Pere, Wisconsin.

Since taking over operations of the Tamarack project, Talon has continued the environmental baseline program initiated by Kennecott. Field parameters are collected from all 19 surface water monitoring sites in quarters two and four, groundwater measurements are read quarterly and samples for laboratory analysis and field parameters are collected annually.

Talon has started data collection for future bedrock hydrology studies, with the installation of two nested vibrating wire piezometers (VWPs). Future environmental work will include but not be limited to planning the Tamarack project's waste characterization program, continuing the bedrock hydrology/hydrogeology studies and initial air quality monitoring.

4.4.2 Environmental Liabilities

Talon has advised the QP 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. The QP has not independently verified this information as described in Item 3 of this report.

4.4.3 Significant Risk Factors

Talon has advised the QP 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. The QP has not independently verified this information as described in Item 3 of this report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & 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, wetlands and forested areas. The town of Tamarack (population 83, 2019 State of Minnesota Demographics Center), 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 (mASL). The Tamarack Project's field office is located in the city of 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 4-2). The parties have sufficient rights to allow for mining operations and supporting infrastructure in the area of mining interest.

6 HISTORY

6.1 Discovery

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 MDNR-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 MDNR 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).

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.

Prior to 2002, the Tamarack area was subject to only very limited exploration efforts and there had 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).

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).

6.2 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.

6.3 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 6-1, Figure 6-1 and Figure 6-2) 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 (LG) disseminated sulphide mineralization N of the Tamarack Zone.

Between 2003 and 2004 drilling was limited to a few holes (Table 6-1) with the first multi-hole program of 13 holes carried out in the winter of 2007, when the first significant intersection of disseminated sulphide mineralization was made with drill hole 07L031 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 08L042. During the subsequent delineation of the SMSU Zone in the same year, the MSU was first intersected in drill hole 08TK0049.

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 program 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 12TK0138 (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 6-1: 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 Bend Zones
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.

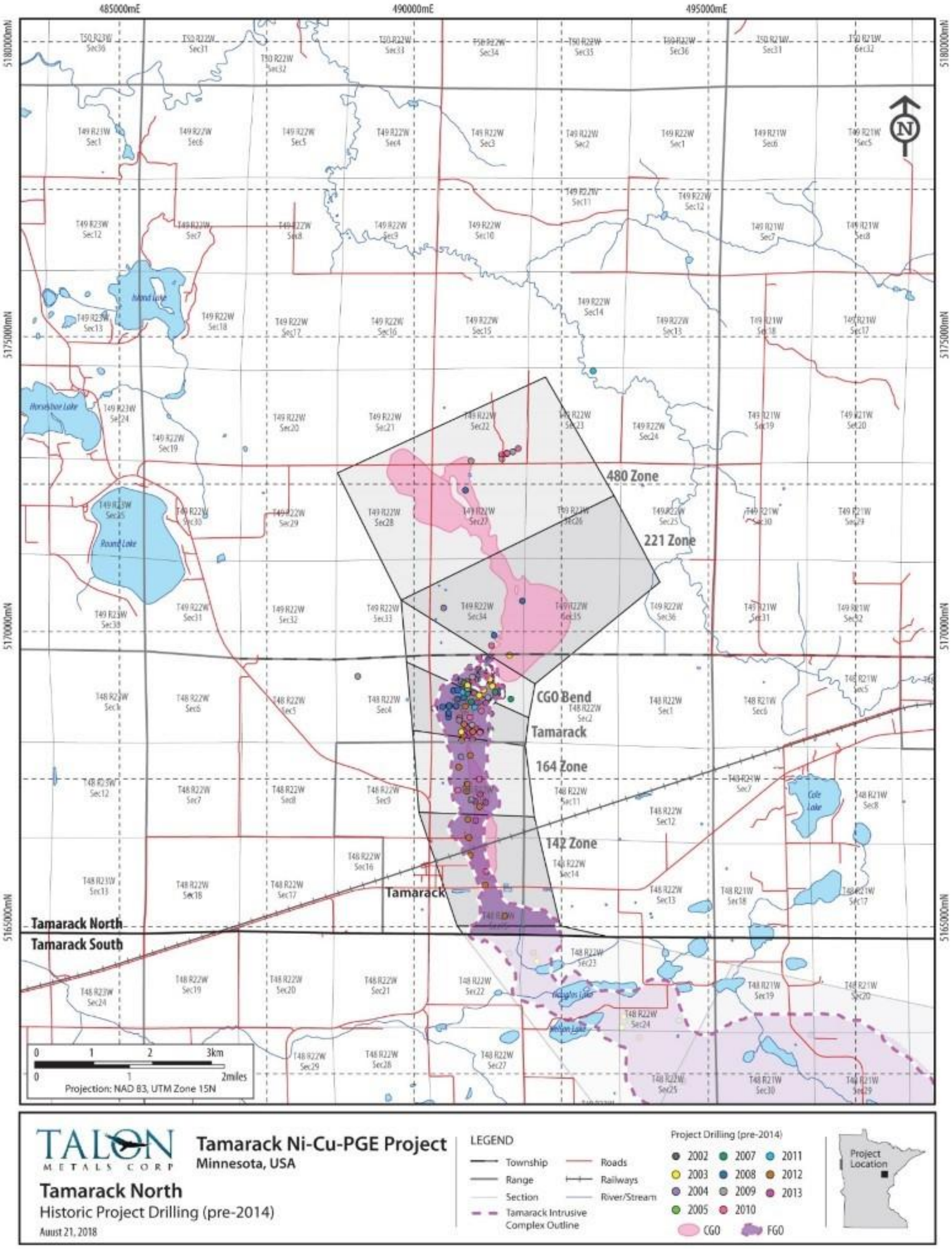


Figure 6-1: Plan View Showing the Locations of the Holes Drilled between 2002 and 2013 at Tamarack North

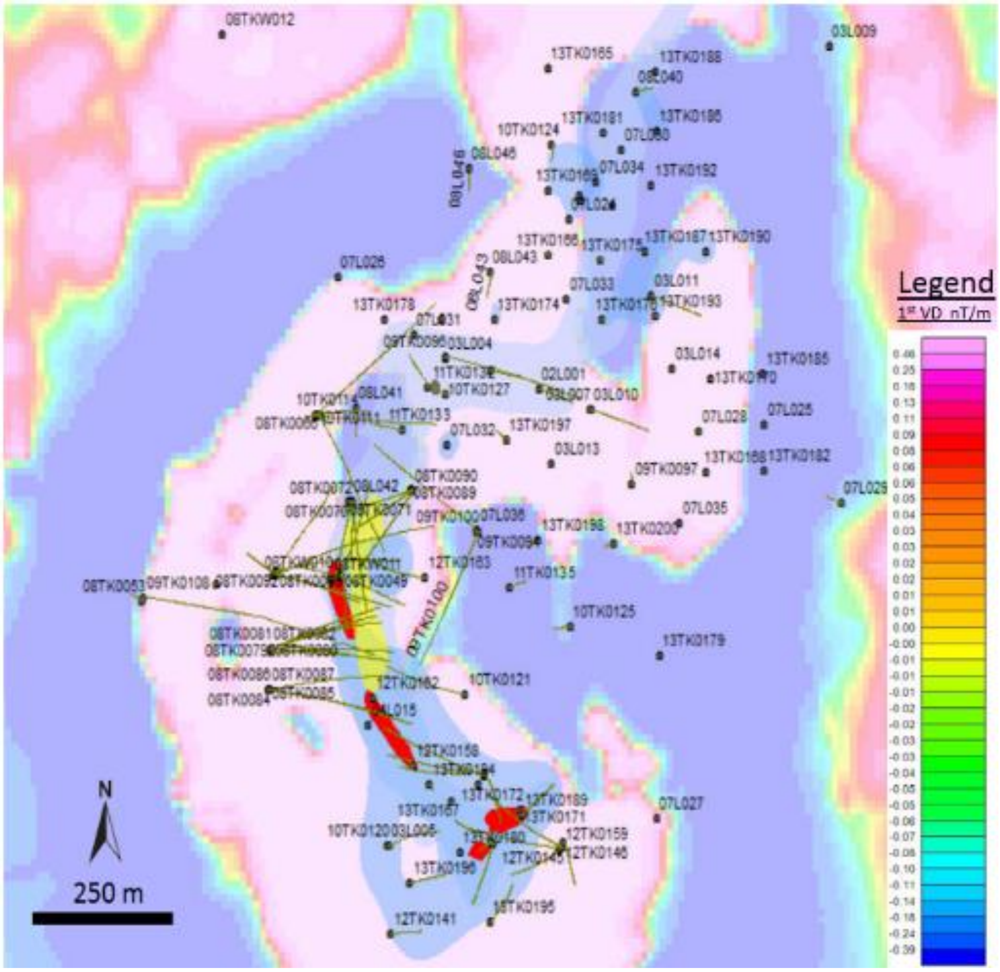


Figure 6-2: Enlarged Map Showing Localities of Drill Holes, in the Tamarack North Project (background 1VD magnetic image). Modified from Kennecott Internal Report and Survey Data, 2013

6.4 Kennecott-Talon Drilling Programs (2014-2018)

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.

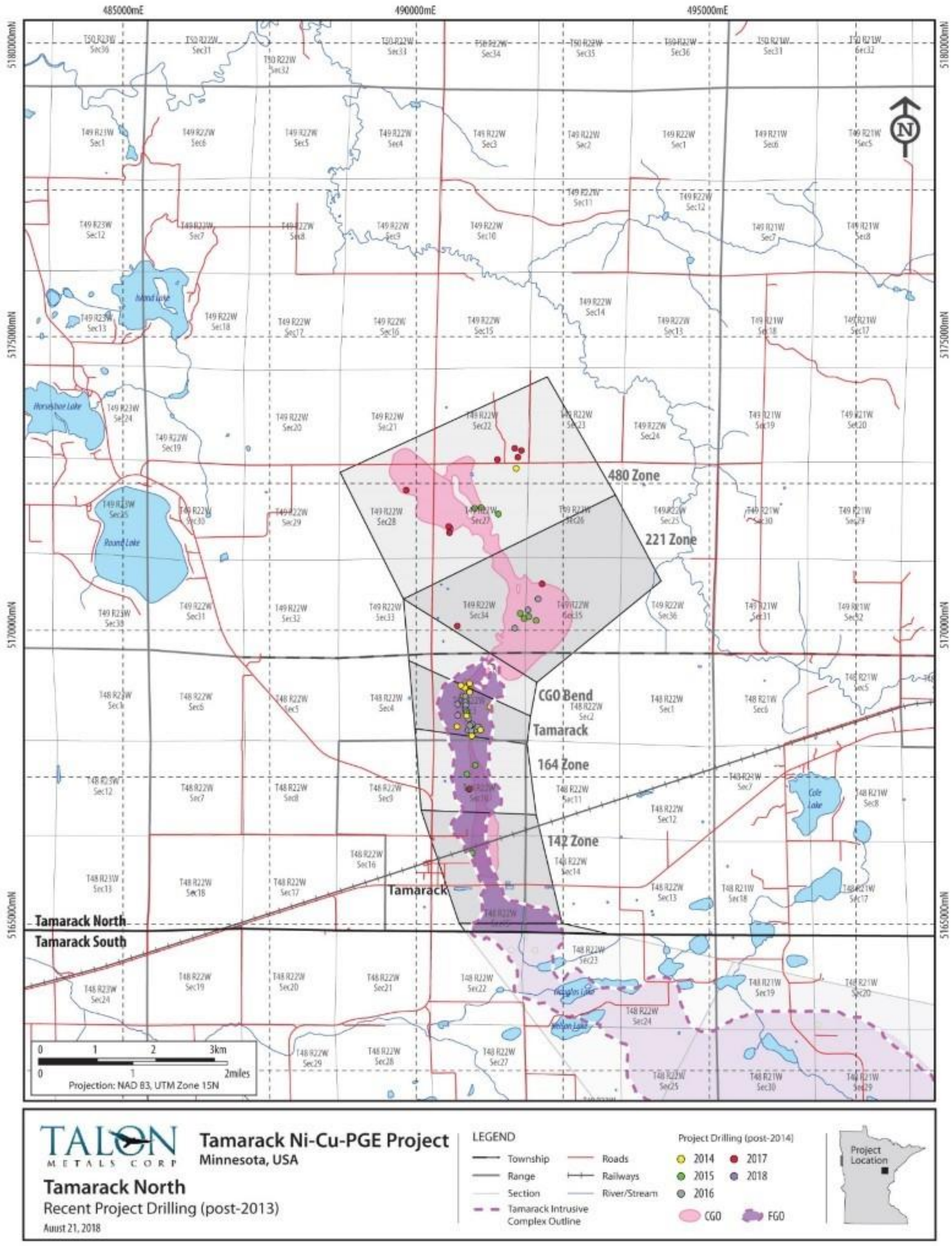


Figure 6-3: Plan View Showing the Locations of the Holes Drilled between 2014 and 2019 at Tamarack North

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 6-3 above).

The 2015 drilling season saw 10 new holes drilled, one historic hole deepened, and two holes pre-collared through overburden (OB) (Table 6-2 notes). 12LV0143 was deepened due to a reinterpreted borehole electromagnetic (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 sedimentary (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 6-3).

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 6-3 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-modeled 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 Zone 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.

No drilling was completed in 2019.

Table 6-2: 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.

6.5 Mineral Resource Estimates

On October 6, 2014 Talon published a maiden NI 43-101 report and mineral resource statement estimate (effective date August 29, 2014) for the Tamarack North Project as shown in Table 6-3 for the 2014 resource statement). The following resource estimates summarized in Table 6-3 and Table 6-4 are prepared in accordance with NI 43-101 but are historic in nature. The QP has not completed sufficient work for them to be considered as being current.

Table 6-3: 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**

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 mineral resource statement estimate was publicly disclosed in a press release (effective dated April 3, 2015) 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" resulting from an increase in the MSU mineralization (see Table 6-4). A technical report was not published at the time, as the increase was determined as to be not being material to the overall project tonnage.

Table 6-4: 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	0.013	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**	**3,386**	**1.63**	**0.94**	**0.04**	**0.31**	**0.19**	**0.17**	**2.11**

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 mineral resource statement estimate was publicly disclosed in a technical report and published in the release (effective dated February 15, 2018) entitled "Talon Metals Files Updated National Instrument 43-101 Technical Report on the Tamarack North Project" resulting from an increase in the MSU mineralization (see Table 6-5).

Table 6-5: Tamarack North Project Maiden Resource Statement (Effective Date February 15, 2018)

Domain	Mineral Resource Classification	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
SMSU	Indicated Resource	3,639	1.83	0.99	0.05	0.42	0.26	0.2	2.45
Total	**Indicated Mineral Resource**	**3,639**	**1.83**	**0.99**	**0.05**	**0.42**	**0.26**	**0.2**	**2.45**
SMSU	Inferred Mineral Resource	1,107	0.9	0.55	0.03	0.22	0.14	0.12	1.25
MSU	Inferred Mineral Resource	570	5.86	2.46	0.12	0.68	0.51	0.25	7.24
138 Zone	Inferred Mineral Resource	2,705	0.95	0.74	0.03	0.23	0.13	0.16	1.38
Total	**Inferred Mineral 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 2014, 2015 and 2018 mineral resource estimates are no longer current and the QP has not completed sufficient work to consider either the 2014, 2015 or 2018 mineral resource estimates as current and therefore, they should not be relied upon.

The 2014, 2015 and 2018 estimates were completed in accordance with NI 43-101 and following the requirements of Form 43-101F1. The mineral resource estimates followed the CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (November 2003) and were classified according to CIM Definition Standards for Mineral Resources & Mineral Reserves (May 2014).

The mineral resource estimates were derived using a geostatistical block modeling approach based on linear interpolation of the drill hole assay data available at the time of reporting. For more information, the reader may refer to the 2014 Technical Report filed on Sedar.com and Referenced in Item 27.

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-Co sulphide mineralization with associated 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 completely buried beneath approximately 30 to 60 m of Quaternary age glacial and fluvial sediments 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 Penok.ean 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).

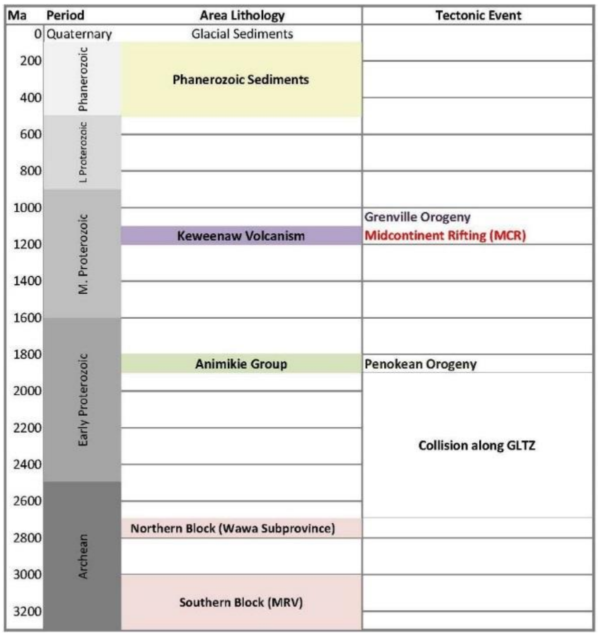


Figure 7-1: Major Depositional Periods and Structural Events Affecting Geological Emplacement and history of the TIC - Modified After Lundin Mining Corporation (2013)

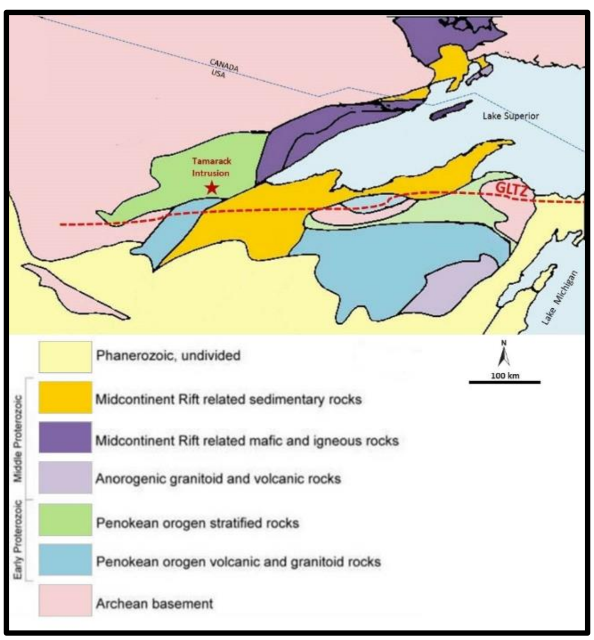


Figure 7-2: Regional Geological and Tectonic Setting for the TIC. The Great Lakes Tectonic Zone (GLTZ) Structure Represents an Inferred Position Due to Younger, Overlying Lithology - Modified from Khirkham (1995) and Lundin Mining Corporation (2013)

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. 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).

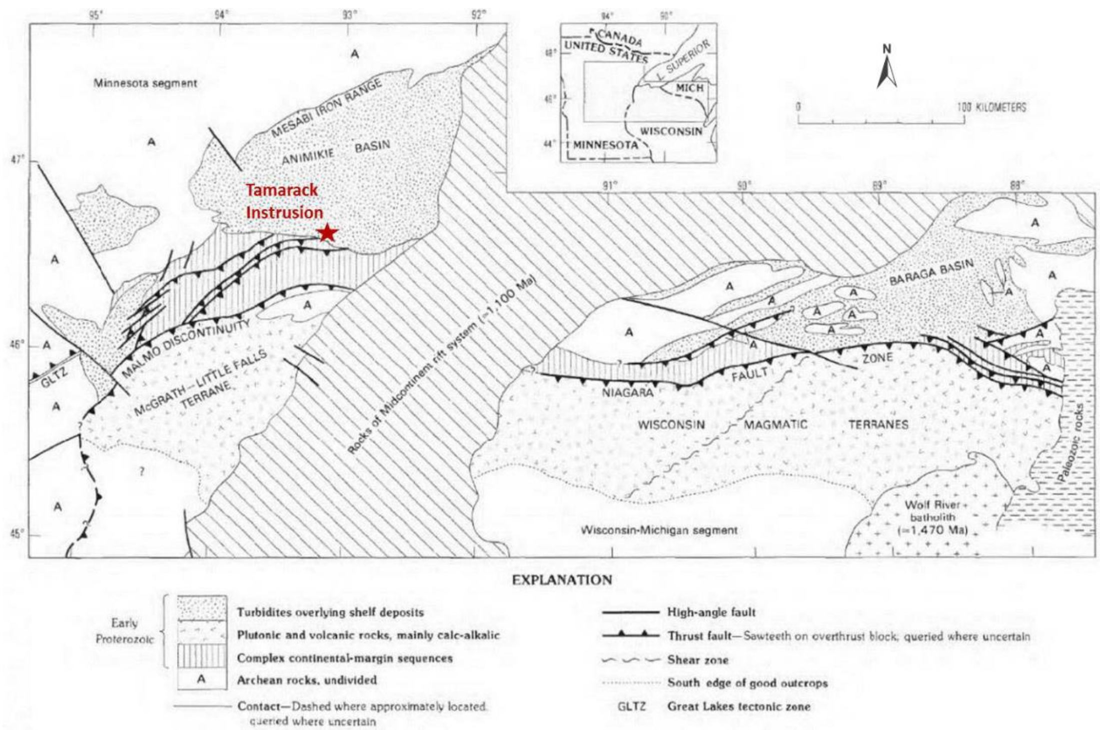


Figure 7-3: Location of TIC in Relation to MCR and Southern Boundary of the Animikie Basin with Tectonic Imbrication and Foredeep Development of the Penokean Orogen. Interpretation Based on Regional Geophysics and Results of Test-Drilling by Southwick et al., 1991

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 unconformity of the crystalline basement or potentially 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 2,000 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 2,000 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 falling within the same time frame, occur distally, and have more primitive melt signatures. They are interpreted to represent the early evolution of the MCR.

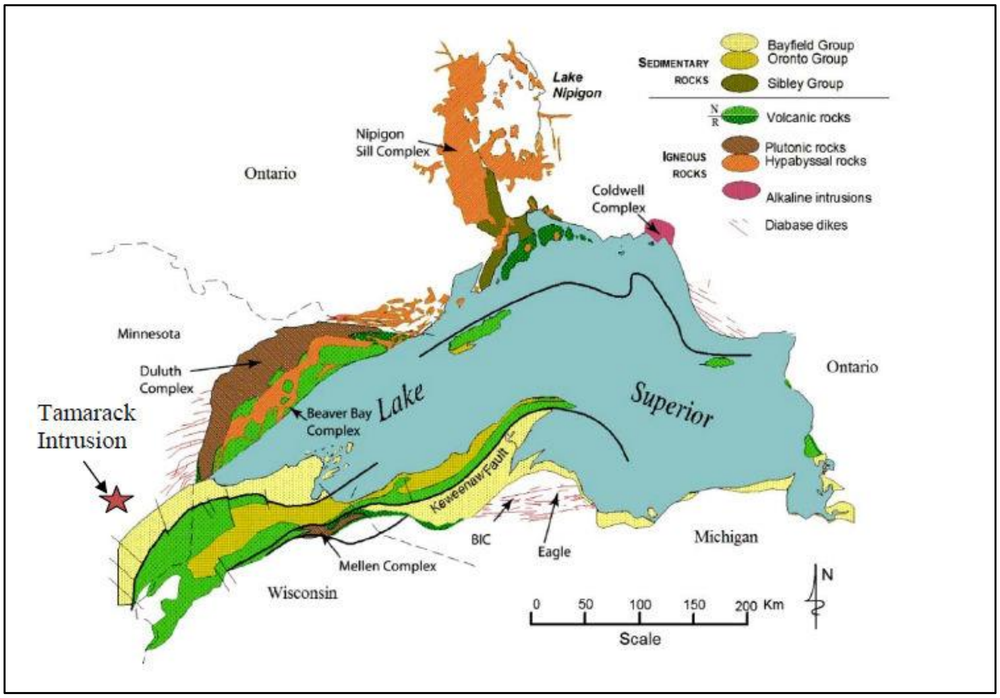


Figure 7-4: Map Showing Locality of TIC and Geology of Lake Superior Region with Location of Other Intrusive Components of the MCR (Goldner 2011, modified from Miller et al., 1995)

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 NE 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.

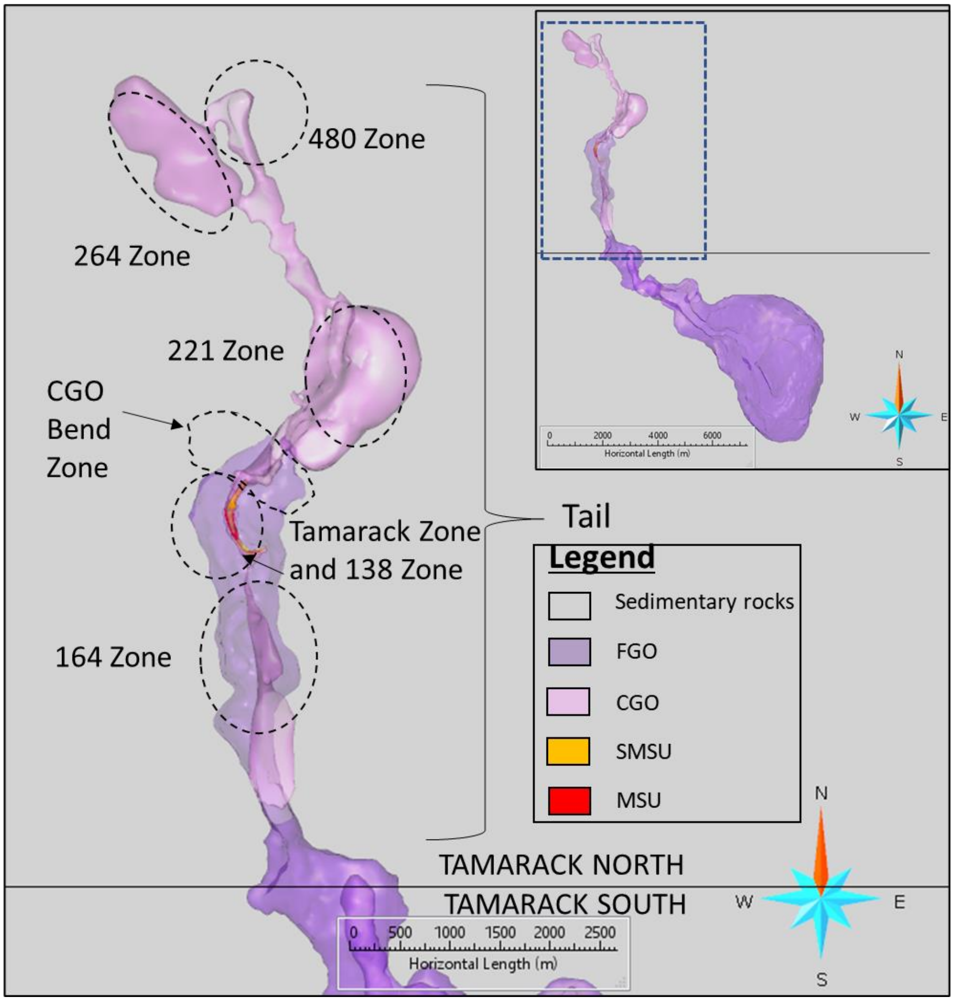


Figure 7-5: Interpreted Bedrock Geology Map Showing 18 km Long Strike of TIC with Long Narrow Intrusion that Hosts Currently Defined Mineralization Termed "Tail" forming Tamarack North Project (Kennecott Aeromagnetic Survey, Modified by Talon, 2017)

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 and partial melting. Observations from core at Tamarack North indicate that SED and structural fabrics have largely been obliterated by the contact 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 distinguishable 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 extends in a 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).

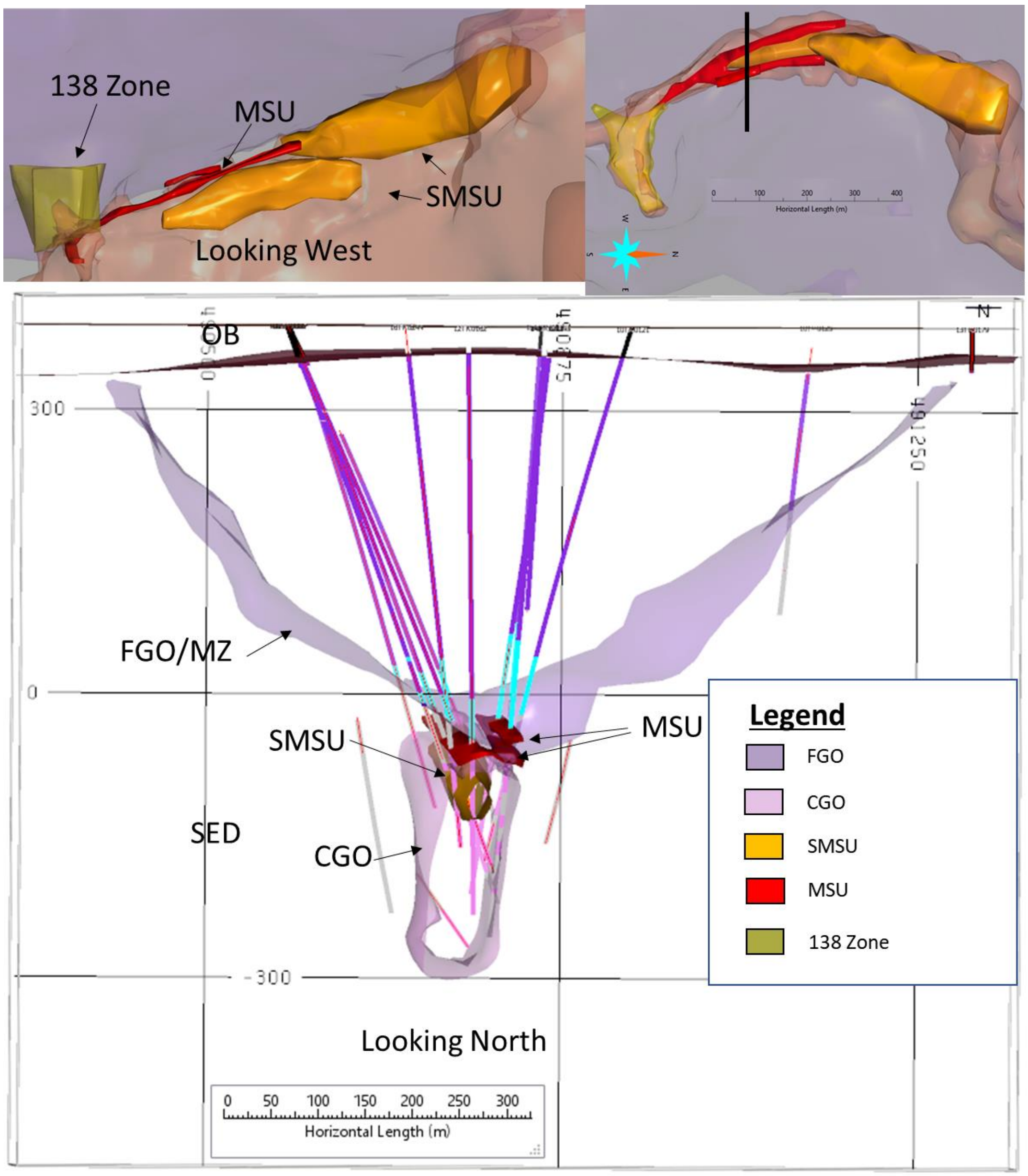


Figure 7-6: Plan, Long Section (S-N) and Cross Section Showing Main Components of Tamarack North Project including CGO at Base Intruding Dyke-Like Beneath FGO in Shape of a Keel. MZ intrusive occurs near interface of the two intrusions. Mineralization in SMSU occurs at top of the CGO, MSU occurs in what 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 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 mixed massive sulphide (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 is a chonolithic intrusion that 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. To the N, in the 480 Zone, the FGO intrusion appears to have a more complex plumbing system and does not appear to have been as affected by erosion. 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 vs 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, SED and MSU 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 vs 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 vs 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.

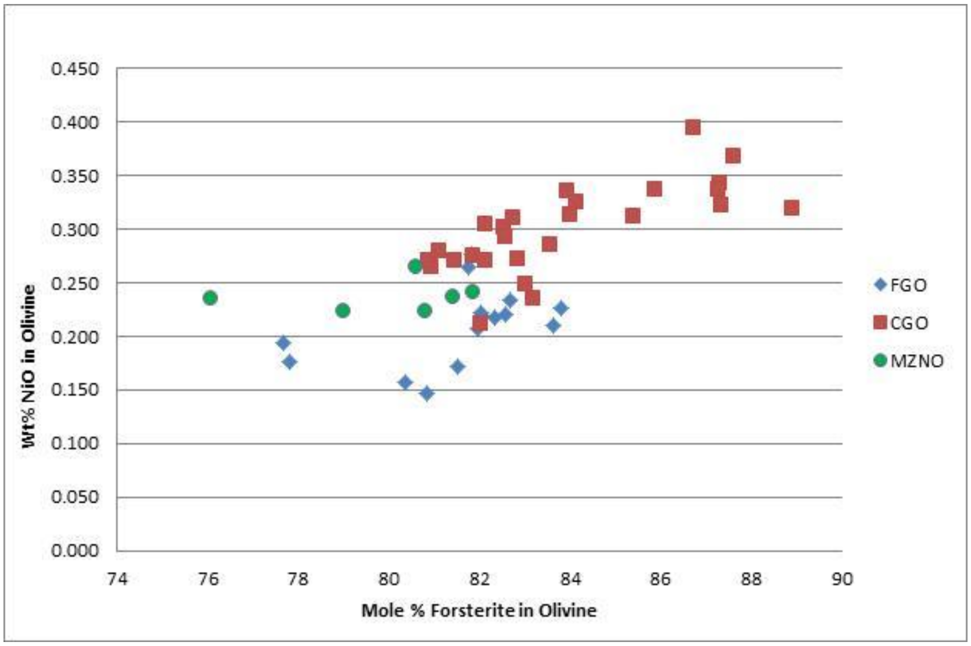


Figure 7-7: Plot of Ni in Olivine vs Fo Content of Olivine. FGO defines a Continuous Trend with Lower Ni Content than in CGO. FGO Olivine Defines a Narrow % Fo Range (82-84% Fo) Compared to CGO (81 89% Fo). Olivine from MZ falls between the two trends. (Data from Goldner, 2011)

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 historically been used at Tamarack to describe specific mineralized materials. These deposits have different mineralization styles, with different metal tenors, genetic implications and different resource potential.

7.2.5.1 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 (FW) 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 x 100 m for the northern basin (Figure 7-8).

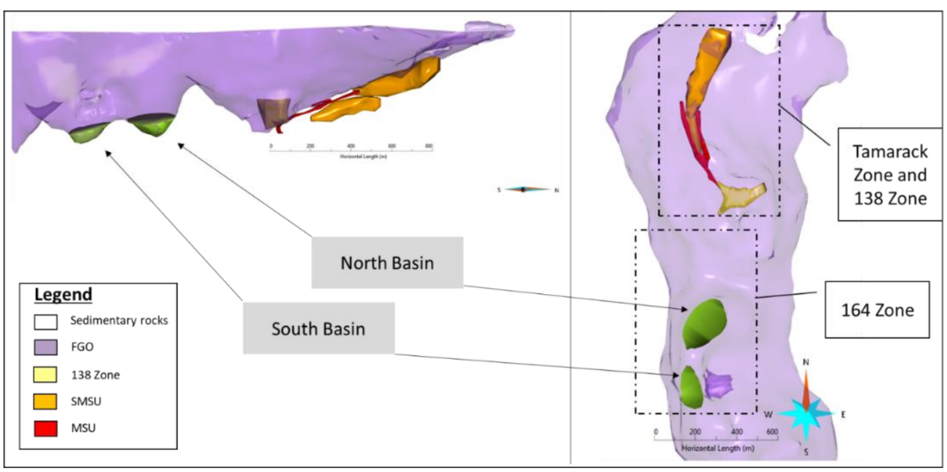


Figure 7-8: 164 Zone, Showing Emplacement of Interpreted Local Basin at Base of FGO. Results from 3D Interpolation of Integrated Magnetic and Gravity Modelling

A surface EM survey in the 164 zone has identified a string of EM anomalies at a depth of 500 to 600 m (Figure 7-9). The surface EM was also deployed over previously define resource areas to calibrate the survey. The same methodology was then used in areas outside of the resource area.

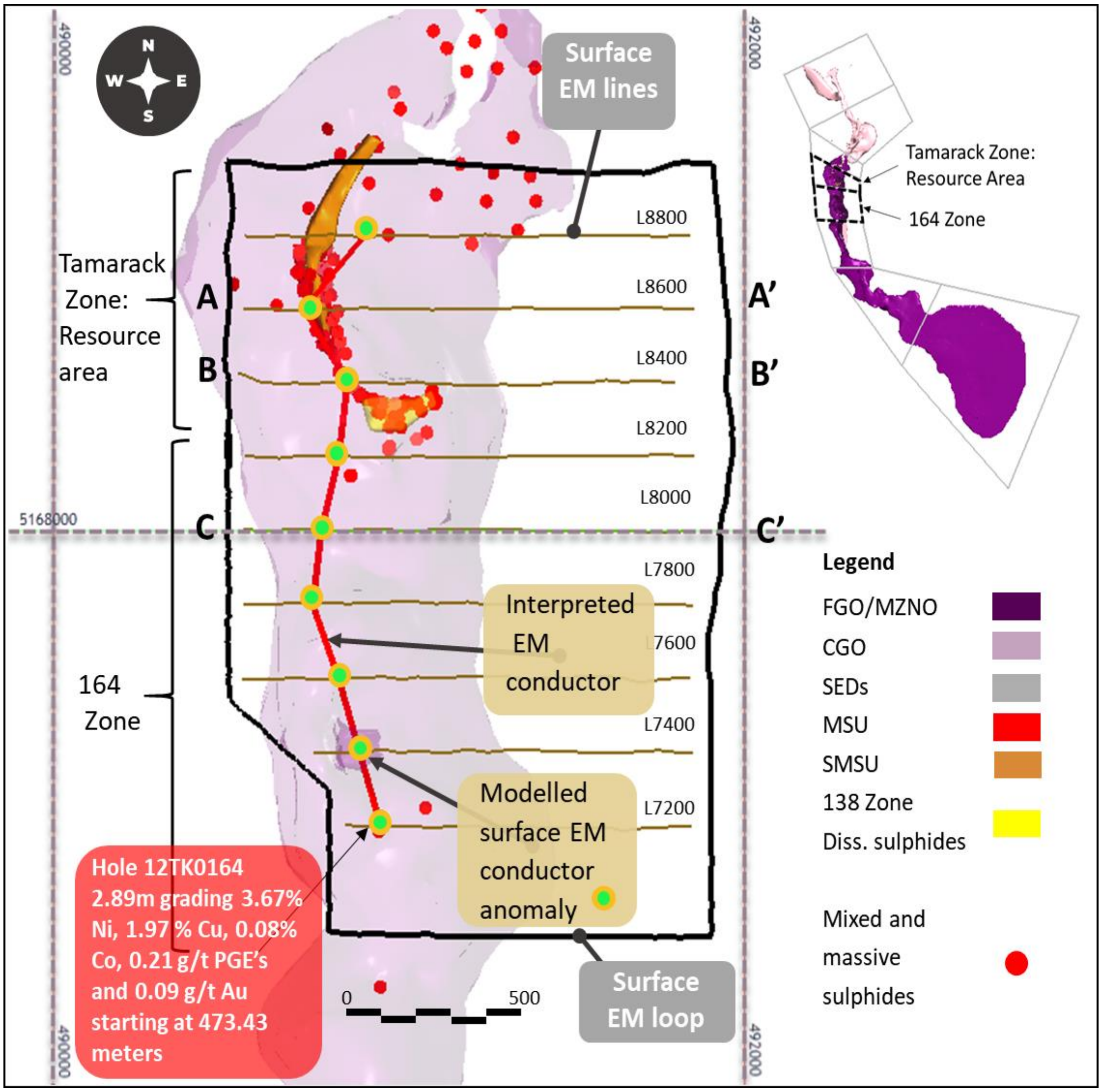


Figure 7-9: Plan view map of the Tamarack Zone (resource area) and 164 Zone (1 km S of the resource area) showing the location of the surface EM survey. The solid red line shows the location of the interpreted conductive anomalies

The surface EM appears to connect the Tamarack resource area to historical hole 12TK0164, an area of ~1.0 km of strike length within the TIC.

7.2.5.2 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.

7.2.5.3 The SMSU

The SMSU 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 modeled. 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 is a core of interstitial net textured sulphides (50% sulphides) (Figure 7-10). 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.

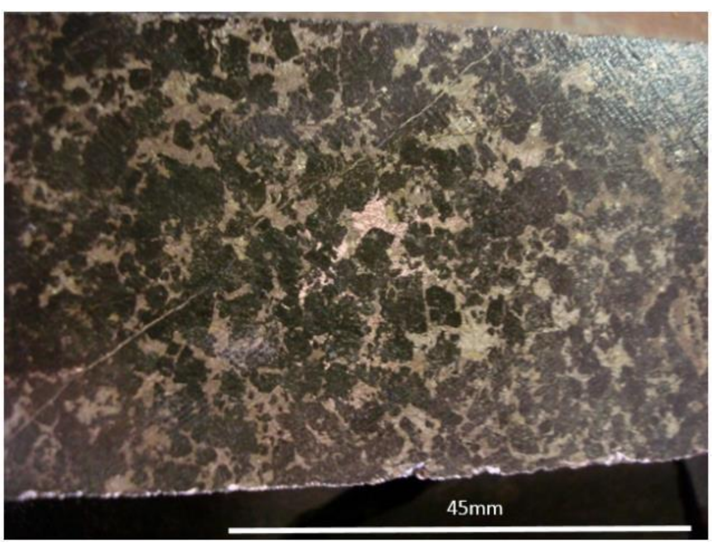


Figure 7-10: SMSU (net textured) Sulphide from Tamarack Drill Core

7.2.5.4 The MSU

MSU-type mineralization is defined as containing 80-90% sulphide (Figure 7-11). 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 7-11: MSU from Tamarack Drill Hole 12TK0158

7.2.5.5 The CGO Bend Area

<u>The CGO Bend East and West</u>

The CGO Bend consists of disseminated and basal FGO MSU-MMS mineralization, observed in the CGO E and W (Figure 7-12) and signifies where CGO forms a dog leg bend immediately N of the Tamarack Zone. The CGO Bend sulphide mineralization consist of a thick sheet of disseminated sulphide 1 to 38 m, with a FW 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 to 900 m, at a depth of 150 m to 250 m and a weak plunge to the S at 15°. The sheet like mineralization in the CGO E has a span of 500 m (E to W) by 900 m (N to S), whereas the CGO W has a span of 400 m (E to W) by 500 m (N to S). The sulphides are disseminated to blebby to massive in texture. In some instances a vertical increase of sulphides has been observed with depth. 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 7-12) part of a much larger interval of 26.27 m of 0.79% Ni, 0.41% Cu, 0.03% Co, 0.26 g/t PGE's and 0.12 g/t Au.

The potential for the mineralization is also supported by prominent DHEM conductors (Figure 7-12) and a recent low-frequency time domain electromagnetic (TDEM) survey over the eastern trend (Figure 7-12). The 2016 exploration program 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). These 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).

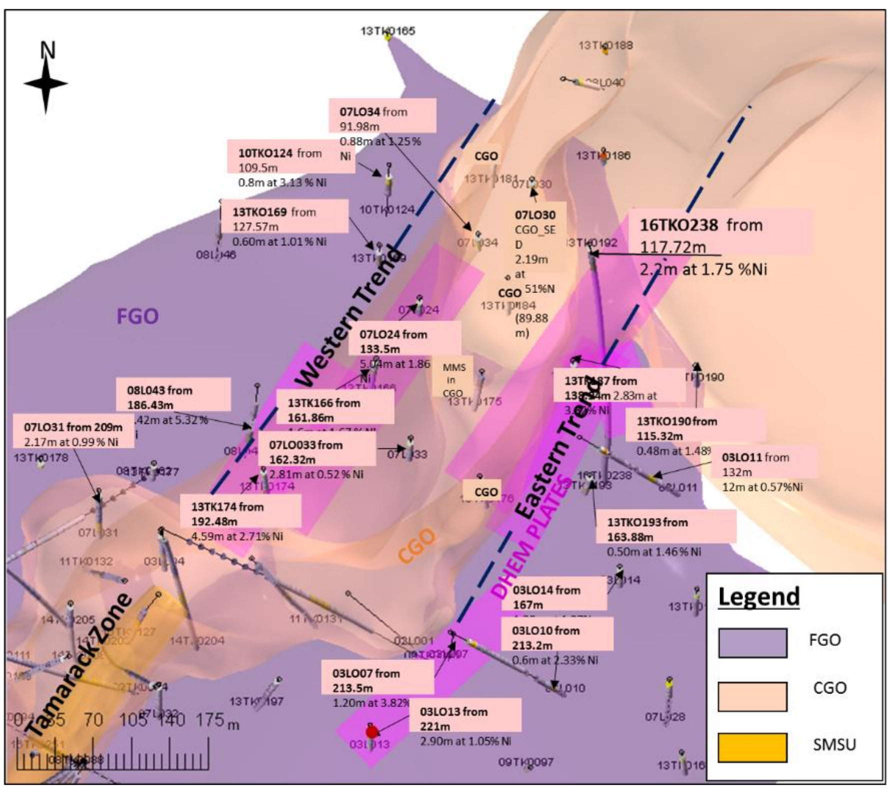


Figure 7-12: Plan View Showing CGO Bend up-Dip of the Tamarack Zone with Locality of Drill Hole 16TK0238 Towards N of CGO Bend Eastern Trend. Also shown are other historical drill hole intercepts and interpreted DHEM conductors which support potential for continuity of mineralization at FGO base both to E and W of CGO

7.2.5.6 The 264 Zone

The 264 Zone limited drilling has identified geology similar to the Tamarack Zone with the distinction that the CGO is over plating the FGO. Both appear to be a sill-like intrusion within the country rock meta-sedimentary. The mineralization is observed at the base of the FGO in the form of mixed and massive sulphides. Drill hole 18TK0264 intersected 0.25 m of 9.95% Ni, 5.74% Cu, 0.16% Co, 2.46 g/t PGEs and 0.32 g/t Au starting at 539.09 m. Bore hole EM shows an EM anomaly with a northwestern strike and shallow plunge to the N-W at ~-15°.

7.2.5.7 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 visually resembling the 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 tested the extent of the FGO and mineralization in the area. The interpretation of the results in the area have defined the relatively limited extent of mineralization, however the FGO-like intrusion that is extending E would require additional geophysical surveys to define a suitable target.

7.2.5.8 The 221 Zone

Drilling a bulging pattern in the first vertical derivative along the CGO intrusive has identified signification mineralization near the base of the intrusion. Hole 15TK0221 intersected massive sulphide mineralization with 0.3 m of 2.0% Ni, 0.56% Cu, 0.53 g/t PGE's and 0.51 g/t Au starting at 682.6 m. The mineralization is located near the FW contact of the CGO. In other holes the mineralization is hosted in the country rock sediments just blow the CGO FW contact, hole 15TK0229 intersected 9.88 m of 2.35% Ni 1.40% Cu, 0.77g/t PGE's and 0.17g/t Au (3.04% NiEq) including a 1.63 m basal zone of high-grade massive sulphide mineralization assaying 9.33% Ni, 5.14% Cu, 3.65 g/t PGE's and 0.71 g/t Au (12.01%NiEq). Further work is required to test the extend of the mineralization.

7.2.5.9 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 mm nuggets and veinlets in the weathered profile and persists into the serpentinized upper part of the FGO (Goldner, 2011).

7.2.5.10 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 in places 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.

7.2.6 Current Models for Formation of the Ni-Cu-Co Sulphide Mineralization in the Tamarack North Project and Extended 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, 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 a mantle source with some samples suggesting Proterozoic or Archean crust. 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 chemical signature, however it shows a crustal SED contamination. The MZ has been interpreted 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 and other changes in physical and chemical conditions in the magma likely contributed to sulphide accumulation.

Ni-Cu-Co sulphide deposits are economically important because they present favourable economics compared to the mining and processing of Ni laterite deposits.

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 deposits 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 four currently lesser defined mineral zones, namely the 480 Zone, the 221 Zone, the 164 Zone, and the CGO Bend.

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 Zone	Peridotite and Feldspathic Peridotite	MZ/FGO	Disseminated and net textured sulphides
	CGO Bend East and West	Feldspathic Peridotite	CGO	Disseminated sulphides
		Peridotite FW (basal FGO mineralization)	FGO	Disseminated, Mixed and massive sulphides
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 1979 and 1983 and the follow-up drill-testing by 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 Mineral Exploration

The TIC and associated mineralization were 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 magnetics and AEM (fixed wing and helicopter based);
- Ground magnetics;
- Surface EM;
- Surface gravity;
- MT;
- IP;
- Seismic;
- MALM;
- MMR; and
- BHEM.

From 2015 to 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 have enhanced the understanding of the Tamarack Project, and improved focus on existing targets.

Drilling in the main target areas of the Tamarack North Project has included 254 diamond drill holes totalling 109,907.4 m with holes between 33.5 m and over 1,224 m depth for an average hole depth of 431 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 borehole (BH) geophysical surveys have been conducted by Kennecott at the Tamarack Project since 2001 (Figure 9-1). AEM and magnetic surveys have included airborne MEGATEM (2001) and AeroTEM (2007, 2008, 2009).

Ground EM surveys were conducted using the Geonics EM-37 (2002), Crone Pulse EM (2003, 2012, and 2016), Lamontange UTEM-3 (2006), and the SJ Geophysics Volterra system (2019 and 2020).

A test line to evaluate different surface transient electromagnetic (TEM) systems was surveyed in 2012. The systems tested included:

- the UTEM-3 system;
- the Crone system using a SQUID sensor;
- the Crone system using a CRA95 coil sensor; and
- the electromagnetic imaging technology (EMIT) SMARTEM system using a SQUID sensor.

In addition, different BHEM systems were evaluated. These included:

- Crone Geophysics with a fluxgate sensor and a coil sensor;
- UTEM-4; and

- EMIT SMARTEM system with fluxgate sensor.

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.

Other surface geophysical surveys included: DC Resistivity/IP (2008), MALM (2008 and 2010), Gradient & Dipole IP/Resistivity (2010), and gravity (2001, 2002, 2011, 2015, and 2016).

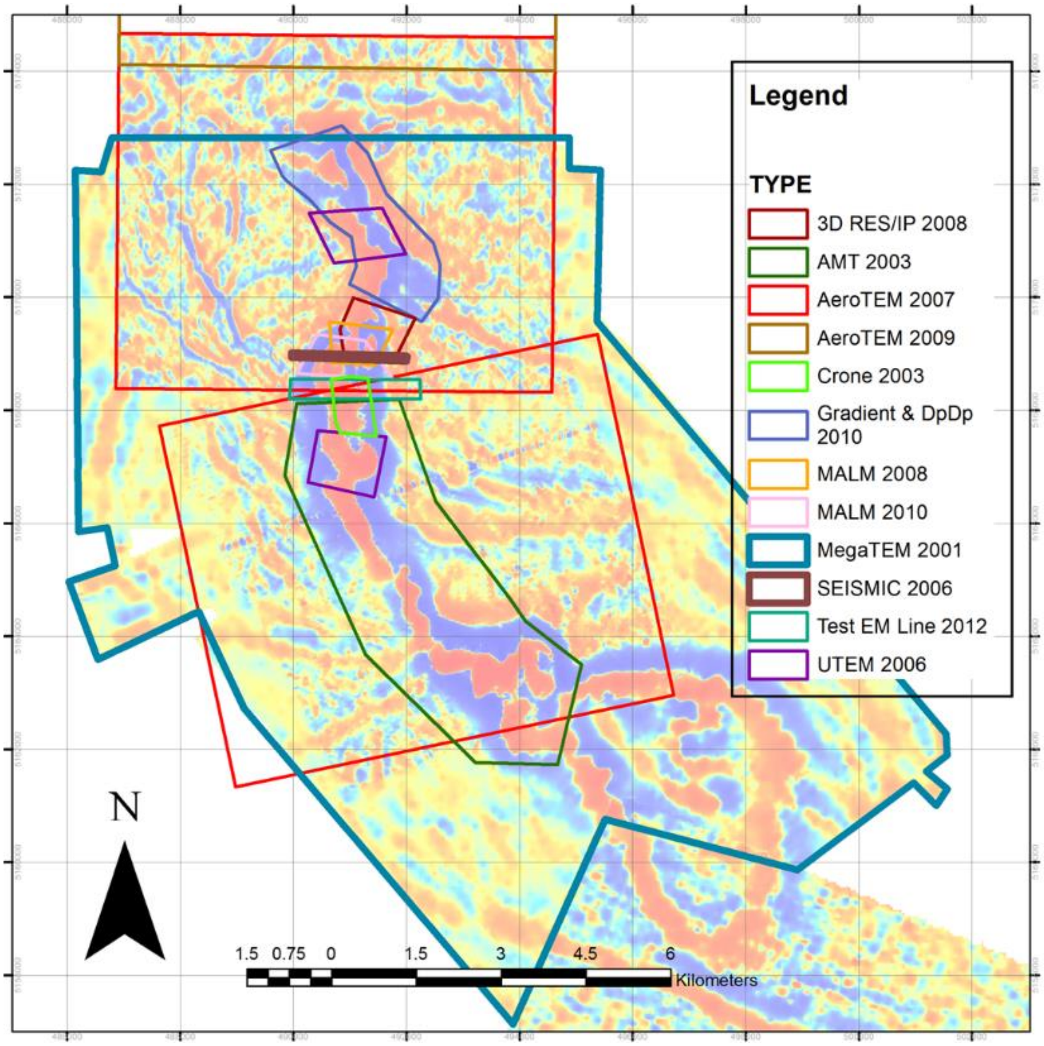


Figure 9-1: Map Showing Localities of Various Geophysical Surveys Conducted Over the entire TIC (composite magnetic TMI image background) Modified from Kennecott Internal Report and Survey Data, 2013

9.2.1.1 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 AEM 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 nuisance conductivity however due its smaller footprint it is less affected by power lines. The higher resolution (50 m line spacing vs 200 m line spacing for MEGATEM) AeroTEM surveys mapped with increased detail shallow conductivity within the 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 AEM systems over the known mineralization is mostly due to near-surface (top 300 m) conductivity within the FGO unit. Direct detection of economic mineralization from the air has yet to be confirmed at Tamarack.

9.2.1.2 Ground Surveys

Electrical and EM Surveys

A variety of ground electrical and EM surveys have been conducted on the property. Surveys included EM 37 (2002), Crone TEM/TDEM (2003 and 2016), audio-frequency magnetotellurics (AMT) (2003), Controlled source audio-frequency magnetotellurics (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 9-2). 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 lower frequency data successfully penetrated through the nuisance conductivity and highlighted conductors at the base of the FGO that were confirmed from drill intersections to be sulphides. These conductors also correspond with modeled BHEM plates.

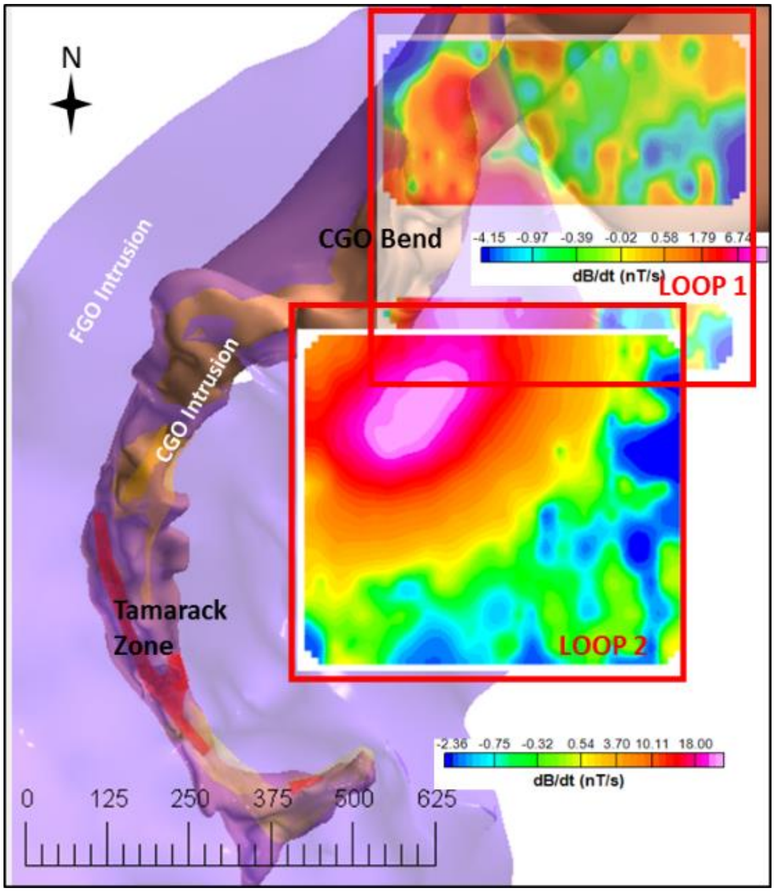


Figure 9-2: Colour Shaded Grids of Ch 20 Crone TEM Z Component for Loop 1 and 2 of TDEM Survey in CGO Bend Zone, Showing Anomalous Conductivity at Depth to the E of the CGO

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.

TDEM Survey, December 2019

A high-power low-frequency large loop TDEM was conducted along the trend of the known mineralization in December of 2019. This survey was conducted by SJ Geophysics out of Vancouver, British Columbia, using their proprietary Volterrra EM system. A 1,500 by 2,000 m loop was deployed around the known mineralization and extending more than one km to the S as shown in Figure 9-3, and 12 line km of data was collected using an inside-loop configuration.

This survey identified a long linear current anomaly along the known mineralization, that progressed much further S than what was expected. It is believed that the currents generated within the known mineralization are finding a path through the geology in the S to a distant ground. This path through the geology may be through mineralization and will be followed up on in 2020.

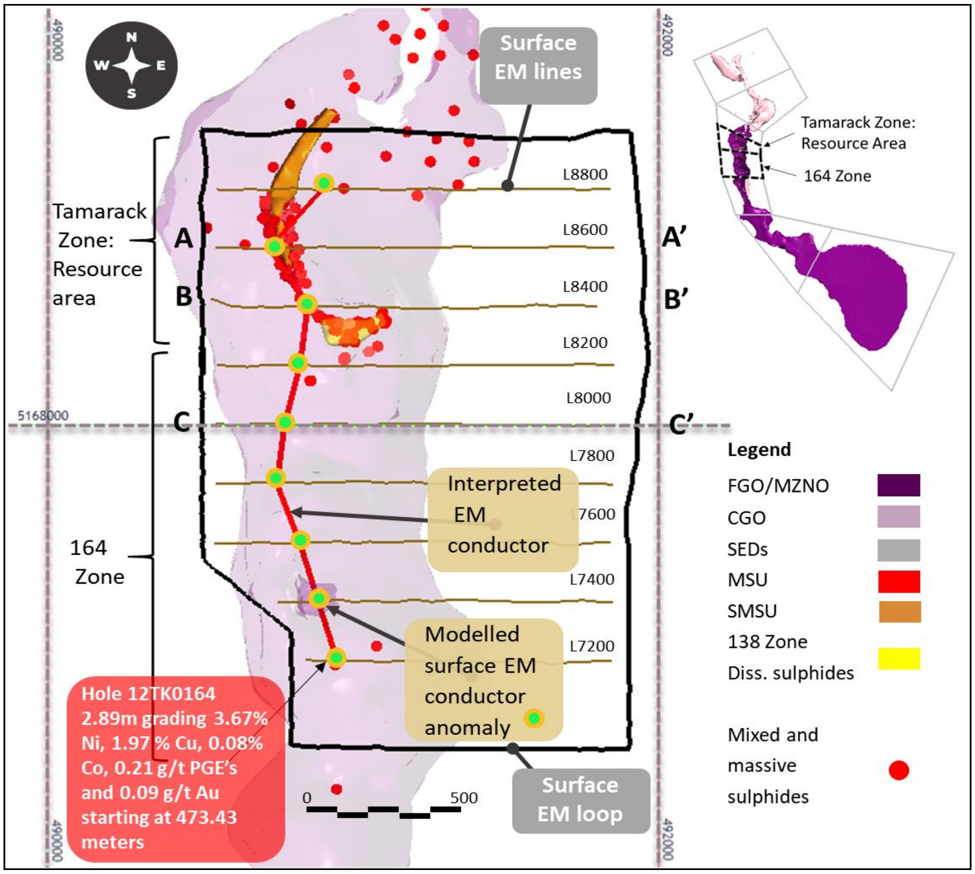


Figure 9-3: Plan view map of the Tamarack Zone (resource area) and 164 Zone (1 km S of the resource area) showing the location of the Surface EM survey. The solid red line shows the location of the interpreted conductive anomalies

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.

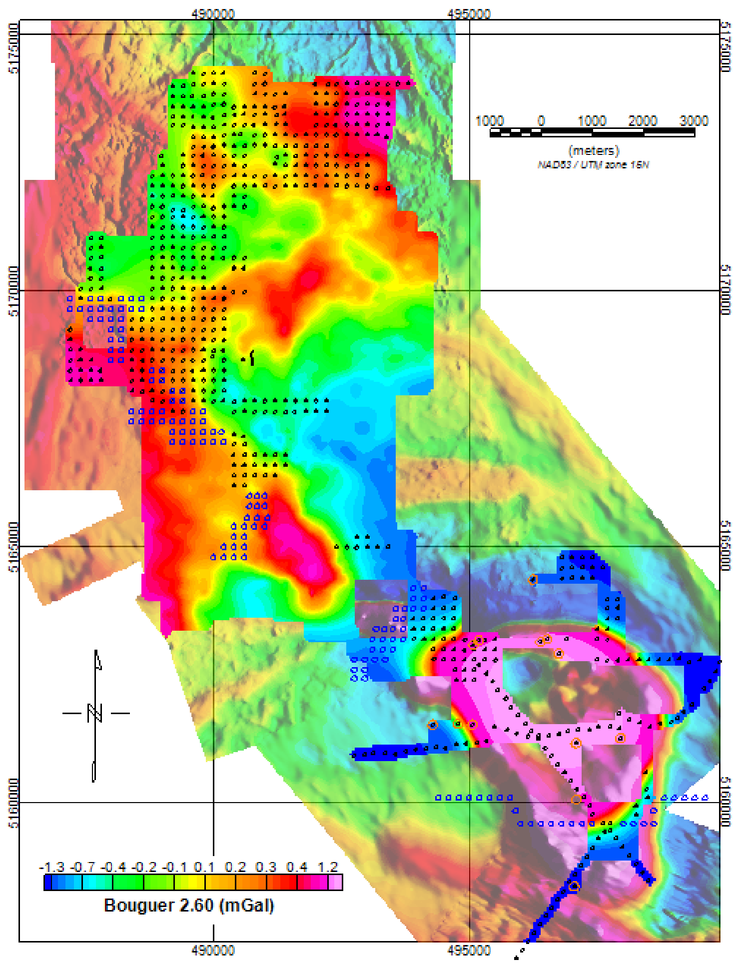


Figure 9-4: Bouguer (2.6 g/cc) Gravity Grid Combining 2011, 2015 and 2016 Surveys with Second Order Trend. Removed. Dots show locations of new data acquired in 2016 (Kennecott Gravity Survey, 2001, 2002, 2011, 2015 and 2016)

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.

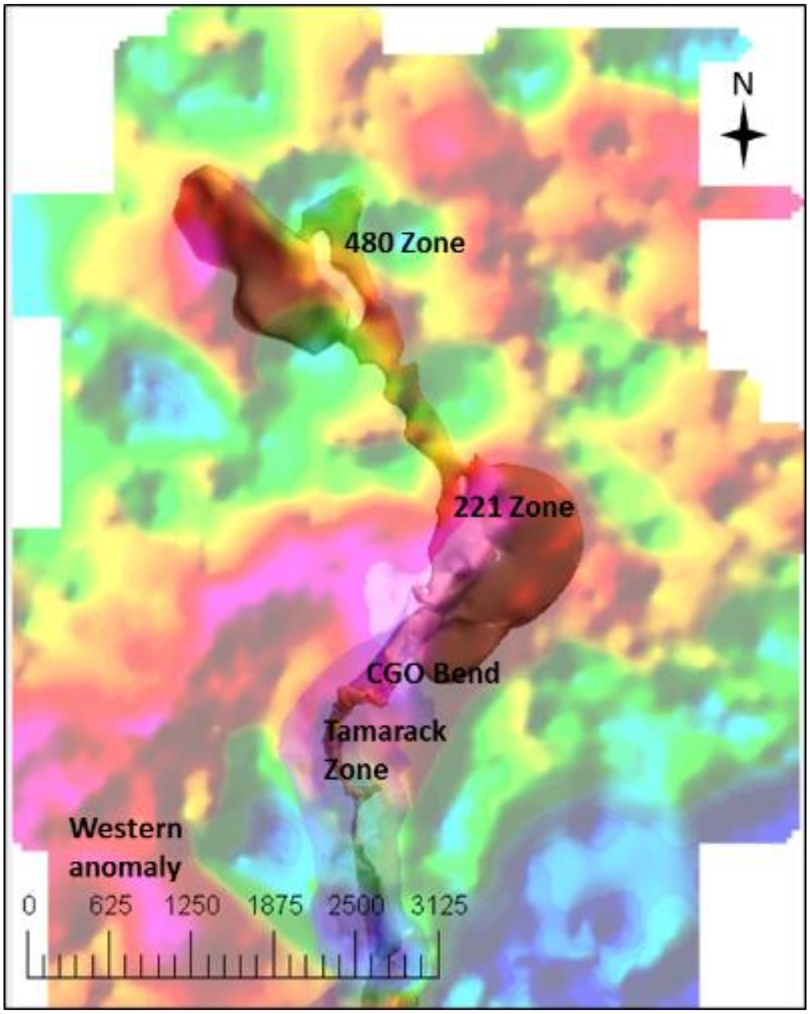


Figure 9-5: Unconstrained Bouguer (2.6 g/cc) Gravity Grid of Northern Tamarack with Modeled CGO Showing the 221 to CGO Bend Anomaly, the 480 Anomalies and the Western Anomaly (Kennecott Gravity Survey 2016)

Seismic Reflection (2006)

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

BHEM Surveys

To date, approximately 192 of the 265 holes at Tamarack North have been surveyed with the Crone BHEM system.

The off-time data was delivered as a Pulse-EM (PEM) format, while the step response was given in the Crone Step (STP) format. The BHEM surveys are very successful in locating sulphides in and near the drill holes. The careful interpretation of the step response data has proven to be very successful in delineating and expanding the MSU in the Tamarack Zone.

MT Survey

An 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. In 2020, a review was conducted of the Tamarack MT dataset by Bill Doerner of SourceOne Geophysical. The review of the data quality was positive, and three lines of data were inverted, yielding interesting results. While the data quality was acceptable, the station spacing was deemed to be too sparse for the target geometries. A recommended station spacing of 100 m over the deposit trend, which can be expanded to 400 m at the ends of each survey line was given in the study.

Radio Imaging

In early 2020, Talon completed seven panels of Radio Imaging (RIM) Survey during the winter program. The final results are pending for these panels, however preliminary results are encouraging. The panel, which comprises a section between holes 16TK0248 and 12TK0153, was completed and interpreted, highlighting the potential use of the technique on the Tamarack project. Drill hole 12TK0153 had intersected two intervals of the MSU, 12.19 m and 2.5 m respectively, whereas hole 16TK0248 has intersected disseminated sulphides but no massive sulphides. Figure 9-6 shows a cross-hole section between hole 16TK0248 and 12TK0153 and the signature of the MSU resource envelop. The resolution quality is in the order that it can map the MSU within 1 m error. This is visible by the interval of 2 m with no sulphide in the MSU interval of hole 12TK0153. The data also demonstrates the nature of the disseminated sulphide mineralization of the 138 zone where moderate attenuation of the signal compares to the near-complete attenuation in the MSU.

The results from the RIM survey are very encouraging and could potentially enable future drilling programs to delineate the edges of the Tamarack mineralization with high precision.

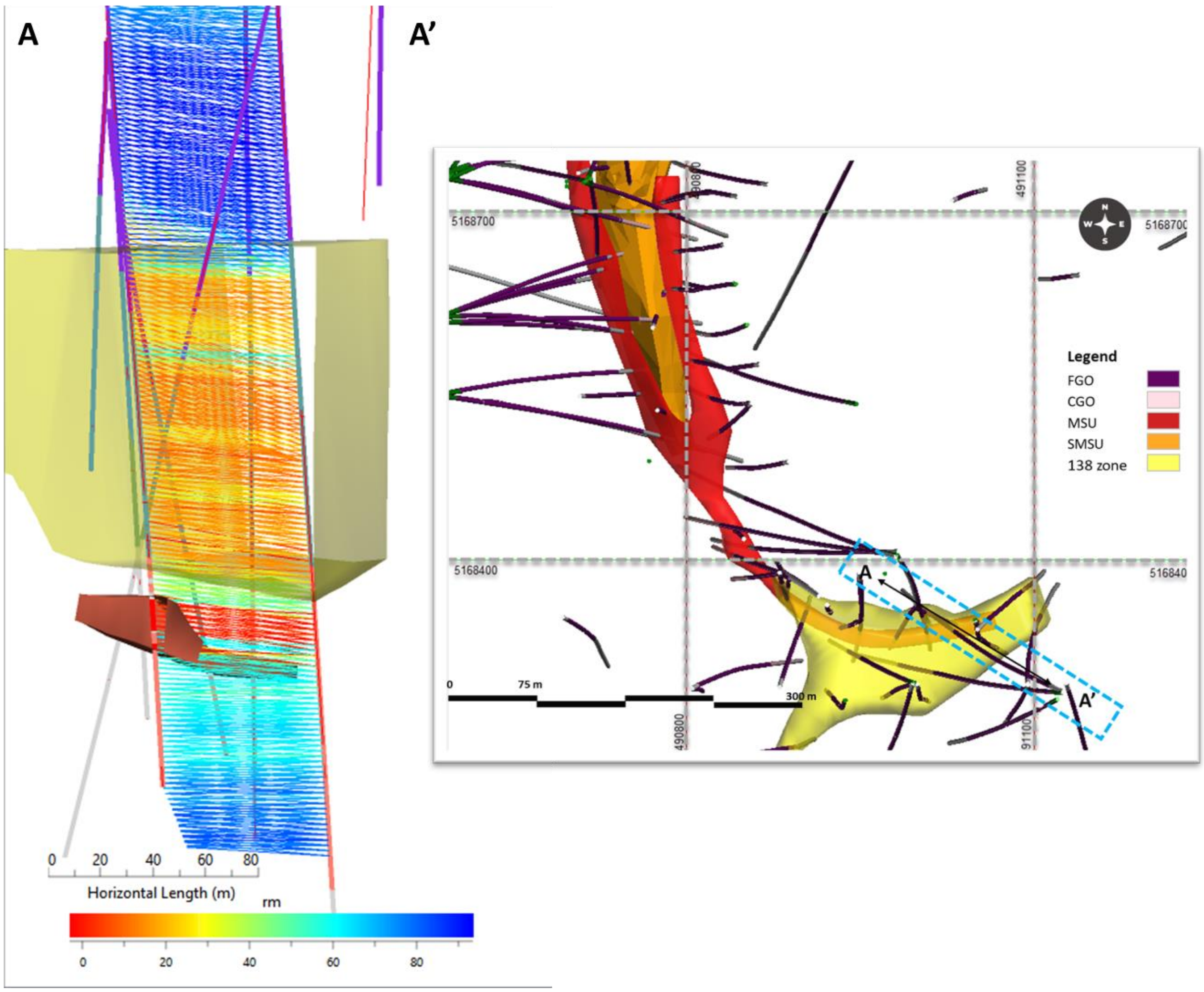


Figure 9-6: Cross hole interpretation of the RIM between holes 16TK0248 (left) and hole 12TK0153 (right). Rays that exhibit maximum attenuation appear red on this image, while weaker attenuation appears as rays coloured from yellow to blue. Two intercepts of massive sulphide mineralization in hole 12TK0153 that are 12.2 m and 2.5 m in thickness appear clearly in this image as two distinct zones of maximum attenuation, while the known semi-massive sulphides of the 138 Zone appear as a cloud of yellow, partially attenuated rays

10 DRILLING

10.1 Talon Drilling Programs (2020 - Present)

In November of 2019, Talon became the operator of the Tamarack Project and planned a winter drilling program targeting massive sulfides in the southern portion of massive sulfide body. These drill holes were planned utilizing open historic drill holes and directional drilling to optimize time and cost to hit the targets. A total of eight holes were drilled in and around the MSU in order to bring the southern portion of the MSU into indicated from inferred by getting the drill spacing down to 25 m.

Table 10-1: Breakdown of 2020 Drilling Conducted by Kennecott-Talon Joint Venture

Year	Number of Holes	Meters	Targets
2020	8	2,573	Tamarack Zone Southern MSU
TOTAL	**8**	**2,573**	

10.2 Resource Drill Holes

The number of total drill holes in the Tamarack North Project 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 10-2. 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 10-2 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 10-2 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 10-2).

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 (DDR) was used to estimate the true width of intersections (Table 10-2).

The estimated true width may be subject to change with additional drilling oriented across the deposit. Figure 14-17 to Figure 14-20 show drill hole cross-sections of the respective orebodies.

Table 10-2: 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**

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	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**
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**

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	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**
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**

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 (%)
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**
								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**
MSU	12TK0153A	490982	5168405	388	615.09	160	-82	**555**	**566.36**	**11.4**	**10.5**	**7.10**	**2.98**	**0.14**	**0.62**	**0.49**	**0.16**	**8.67**

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 (%)
MSU	12TK0153C	490982	5168405	388	618.13	164	-82	**578.45**	**585.59**	**7.1**	**6.6**	**8.31**	**3.26**	**0.16**	**0.84**	**0.65**	**0.43**	**10.15**
MSU	16TK0233C	490914	5168369	388	562.66	300	-85	**500.45**	**506.18**	**5.7**	**5.3**	**4.74**	**1.93**	**0.10**	**0.38**	**0.38**	**0.21**	**5.81**
MSU	16TK0233E	490914	5168369	388	562.36	302	-86	**513.11**	**523.65**	**10.5**	**9.7**	**5.69**	**2.34**	**0.12**	**0.56**	**0.60**	**0.27**	**7.02**
MSU	20TK0265	490949	5168389.28	388	584	174	-83	**543.78**	**546.8**	**3.0**	**2.8**	**4.23**	**2.17**	**0.09**	**0.51**	**0.41**	**0.29**	**5.42**
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*
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*

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	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*
138	12TK0153A	490982	5168405	388	615.09	160	-82	*424.12*	*471*	*46.9*	*46.9*	*0.58*	*0.38*	*0.02*	*0.11*	*0.06*	*0.09*	*0.81*
138	12TK0153B	490982	5168405	388	600.47	159	-83	*426*	*458*	*32*	*32*	*0.58*	*0.39*	*0.02*	*0.10*	*0.06*	*0.09*	*0.80*
138	12TK0153C	490982	5168405	388	618.13	164	-82	*426*	*459*	*33*	*33*	*0.58*	*0.43*	*0.02*	*0.11*	*0.06*	*0.08*	*0.83*
138	20TK0265	490949	5168389.28	388	584	174	-83	*422*	*482*	*60*	*60*	*0.58*	*0.38*	*0.02*	*0.11*	*0.07*	*0.07*	*0.80*

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 DDR of 25/170 for the calculation of estimated true thickness.
Note: *Uses an assumed Dip and DDR of 25/130 for the calculation of estimated true thickness.
Note: The 138 orebody assumed a Dip and DDR of 0/0 for the calculation of estimated true thickness.
Note: Estimated true thickness calculated via Datamine® "TRUETHK" Process.

10.3 Drill Hole and Core Logging Procedures

10.3.1 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 mm (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 DHEM survey.

10.3.2 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.

10.3.3 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 boxes 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 in boxes;

- Detailed geotechnical logging (logging interval based on geotechnical characteristics that are at least 25 cm long and up to 3.05 m at maximum). Standard logging and testing includes:
  - Total core recovery (TCR) and Solid Core Recovery (SCR)
  - Rock Quality Designation (RQD or L10)
  - Natural fracture count
  - Open vein count
  - IRS Hardness (Rock strength estimation)
  - Weathering Index
  - Alteration Index
  - Rock structure and texture
  - Joint set number (Jn)
  - Joint Roughness (Ja)
  - Joint Condition Rating (JCR)
  - Defect feature details and orientation
  - Pont structure details and orientation
  - Point load testing (axial and diametral every 6 m)
  - Laboratory sampling for uniaxial compressive strength (UCS), triaxial compressive strength (TCS) and Brazilian tensile strength (BTS) testing every 20 m
- 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).

10.3.4 Surveying

All collars are professionally surveyed to sub-meter 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 preferred sample length in mineralized zones. The following procedures are adhered to:

- Core is picked up at the drill site by Talon 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: Prior to being photographed the core is transferred to cardboard core boxes where the tags are stapled to the inside wall of the appropriate rows. Duplicate, blank, and standard sample tags are inserted and displayed on the core boxes for photographing;
- Core photography is conducted after the sample mark-up is completed on V-rails (definition and some reconnaissance holes);
- Boxed core (definition and reconnaissance holes) is photographed and was reintroduced in 2012 after being discontinued in 2008;
- In "definition" and "reconnaissance" categorized holes, a 20 cm sample is cut from the core for the purposes of BTS, TCS 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). 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;
- A density measurement via the hydrostatic-gravimetric method is performed every 20 m on a 10 cm sample in the core shack. Dry and wet weights for three density standards are recorded per use of the instrument setup. The scale is also calibrated using calibration weights at this time;
- Core sawing is conducted after core marking, sample tagging, and photography 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 (the 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 remaining 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 is 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/Talon;
- 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/Talon 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/Talon 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 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 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/Talon has adopted a number of rules of variance that are acceptable vs 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.

Project	Sample Type	Drillhole ID	Despatch #	Despatch Comments	Number of samples	Date samples despatched to lab	Despatched by	Lab	LabJobNo	# samples received by lab	Date Lab received samples
Tamarack	Drillcore	12TK0153A	T0001	153A mineralized	33	hand delivered	MG	ALS Thunder Bay	TB20031117	33	2/8/2020
Tamarack	Drillcore	16TK0233C	T0002	233C mineralized	11	3/12/2020	MG, JD, BK	ALS Thunder Bay	TB20066201	11	3/20/2020
Tamarack	Drillcore	16TK0233B, 16TK0233C	T0003	233B, 233C intrusive	124	3/12/2020	MG, JD, BK	ALS Thunder Bay	TB20066202	124	3/20/2020
Tamarack	Drillcore	20TK0265	T0004	265 intrusive	65	3/20/2020	JD, BK	ALS Thunder Bay	TB20069188	65	3/24/2020
Tamarack	Drillcore	20TK0265	T0005	265 mineralized	16	3/20/2020	JD, BK	ALS Thunder Bay	TB20069187	16	3/24/2020
Tamarack	Drillcore	12TK0153C	T0006	153C intrusive	82	3/24/2020	JD, BK	ALS Thunder Bay	TB20070864	82	3/26/2020
Tamarack	Drillcore	12TK0153C	T0007	153C mineralized	12	3/24/2020	JD, BK	ALS Thunder Bay	TB20070861	12	3/26/2020
Tamarack	Drillcore	16TK0233E	T0008	233E mineralized	16	3/24/2020	JD, BK	ALS Thunder Bay	TB20070862	16	3/26/2020
Tamarack	Drillcore	16TK0233D	T0009	233D intrusive	74	4/29/2020	JD, BK	ALS Thunder Bay	TB20097353	74	5/7/2020
Tamarack	Drillcore	12TK0153A	T0010	153A intrusive	81	4/29/2020	JD, BK	ALS Thunder Bay	TB20097354	81	5/7/2020
Tamarack	Drillcore	16TK0233E	T0011	233E intrusive	68	5/18/2020	JD	ALS Thunder Bay	TB20107838	68	5/20/2020
								ALS Thunder Bay			

Despatch #	Date Assays Finalized	QC Status	Reassay (y/n)	Date Reassay Finalized	QC Final	Standard ID	Failed Sample ID	Element (s)	QAQC Performed by	QAQC Date	QAQC Comments	QC Comments
T0001	2/27/2020	Pass	No		Pass	TAM26, TAM28, TAM-M			MG	4/13/2020	QAQC'd by MG, re-QAQC'd by BK/JD due to AcQ issues	
T0002	4/2/2020	Pass	No		Pass	TAM-M			JD, BK	4/13/2020		
T0003	4/8/2020	Fail	Yes	5/14/2020	Pass	TAM26, TAM29-B	101010, 101040	Cu, (Pd and Pt)	JD, BK	4/17/2020	Resubmitted standard #101040; STD was opened by customs; QAQC'd passed after re-assay; Cu on 101010 failed again no change in value;	
T0004	4/11/2020	Pass	No		Pass	TAM-28			JD, BK	4/13/2020	Co assays in sample 100540 were above 3SD; since values were provisional the QAQC still passed	
T0005	4/2/2020	Pass	No		Pass	TAM26			JD, BK	4/13/2020		
T0006	4/19/2020	Pass	No		Pass	TAM28, TAM29-B			JD, BK	4/20/2020	Duplicate parent sample was erroneously entered as 100611 switched to 100609 to fix mistype	
T0007	4/3/2020	Fail	Yes	4/29/2020	Pass	TAM-M	100660	Au, Pd, Pt	JD, BK	4/29/2020	Re-assay comments: Au is good, Pd is good, Pt is good after re-assay	
T0008	4/3/2020	Pass	No		Pass	TAM-M			JD, BK	4/13/2020		
T0009	5/20/2020	Pass	No		Pass	TAM28, TAM29-B			BK	5/20/2020	All values passed QC and were within 2 std dev	#101040 stanc
T0010	5/26/2020	Pass	No		Pass	TAM27, TAM-M, TAM29-B			JD, BK	5/26/2020	#101453 mistype standard as EA-M - actual TAM-M; #101440 Cu is at 2 SD line;	
T0011	6/8/2020	Pass	No			TAM28, TAM29-B			JD, BK	6/8/2020	#100790 was "outside" 3SD for Cu, but the SD values were off; sent to Paul to fix	
											Co is low for std sample #100090; [illegible]	

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/Talon are outlined in Section 11.1 above.

It is the QP's opinion that the sampling process is representative of the mineralization at Tamarack North and that the sample preparation and QA/CQ procedures used, and the sample chain of custody were found to be consistent with CIM Mineral Exploration Best Practice Guidelines (November 2018).

The QP recommends that Talon 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.

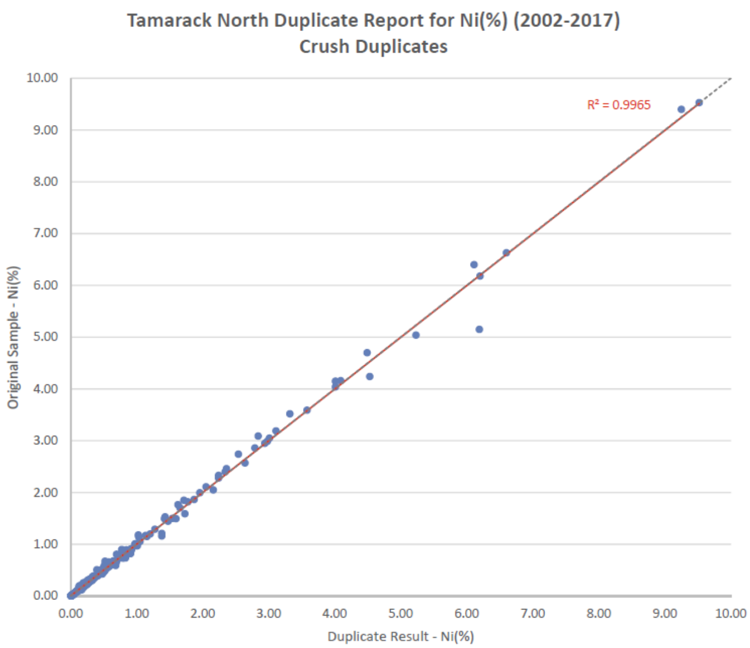


Figure 11-3: Comparison of Original vs Duplicate Coarse Reject Ni (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

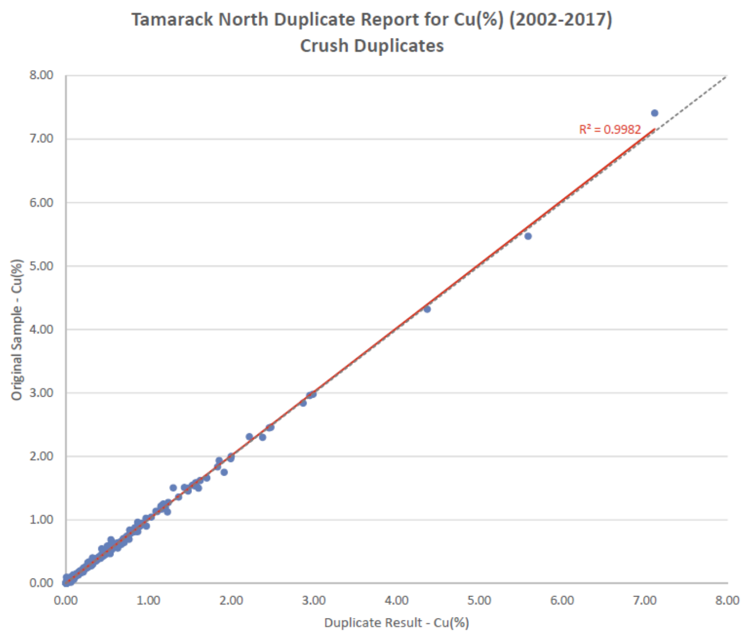


Figure 11-4: Comparison of Original vs Duplicate Coarse Reject Cu (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

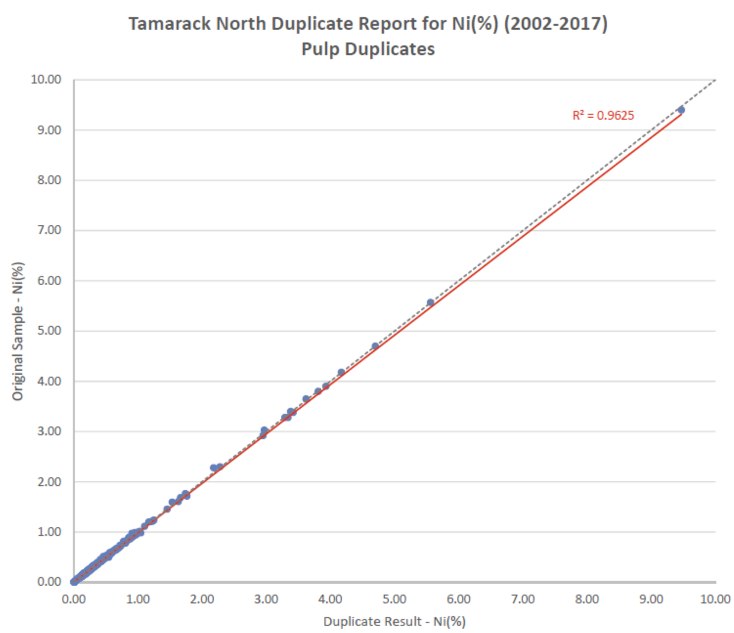


Figure 11-5: Comparison of Original vs Duplicate Pulps Ni (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

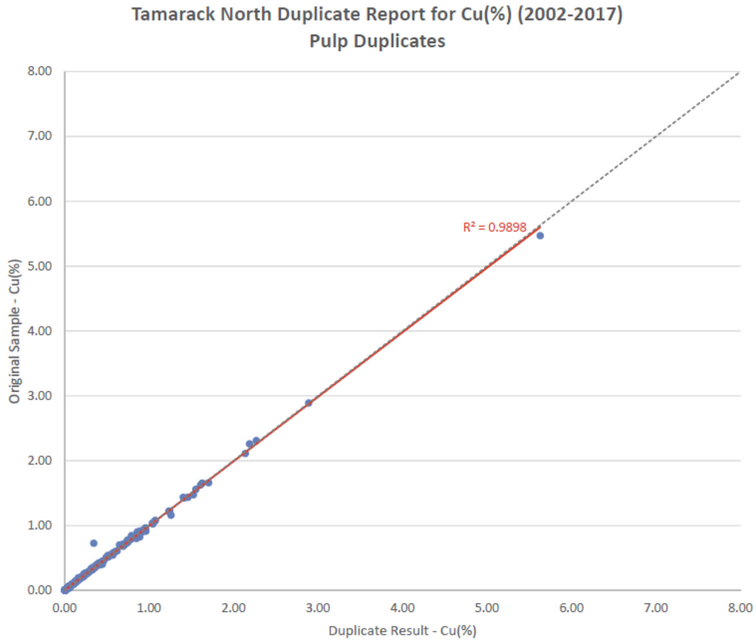


Figure 11-6: Comparison of Original vs Duplicate Pulps Cu (%) values for Tamarack North Drill Hole Samples between 2002 and 2017

12 DATA VERIFICATION

12.1 Golder

The QP completed a number of data verification checks in 2014, 2017, and 2020 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, 2017 and 2020) and a site visit by the QP (2014) to check drill hole collars, logging procedures, sample chain of custody and collection of independent samples for metal verification. In addition, a number of verifications were completed for the mineral resource estimate which is outlined in Section 14.

12.1.1 Database Verification

The QP 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 2017, the QP 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. In 2020, an additional 157 samples having a Ni grade above 0.4%, representing 1,978 assays were checked identifying minor issues with three samples and eight Pt, Pd and Au assays. These issues were reviewed and found to be not material to the 2020 mineral resource estimate and are recommended to be corrected in the assay database. 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 2020 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
2014-2018	19	533	3,198	0	2017
2019-2020	7	157	1,978	8	2020

Only a small selection of all the drill holes at Tamarack North Project were validated against the original data. A total of 63 unique drill holes (2,781 samples), representing 6.7% of the total available assay data, was reviewed. The validated samples consisted of those used in the mineral resource estimate that were above the chosen COGs used at the time. 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

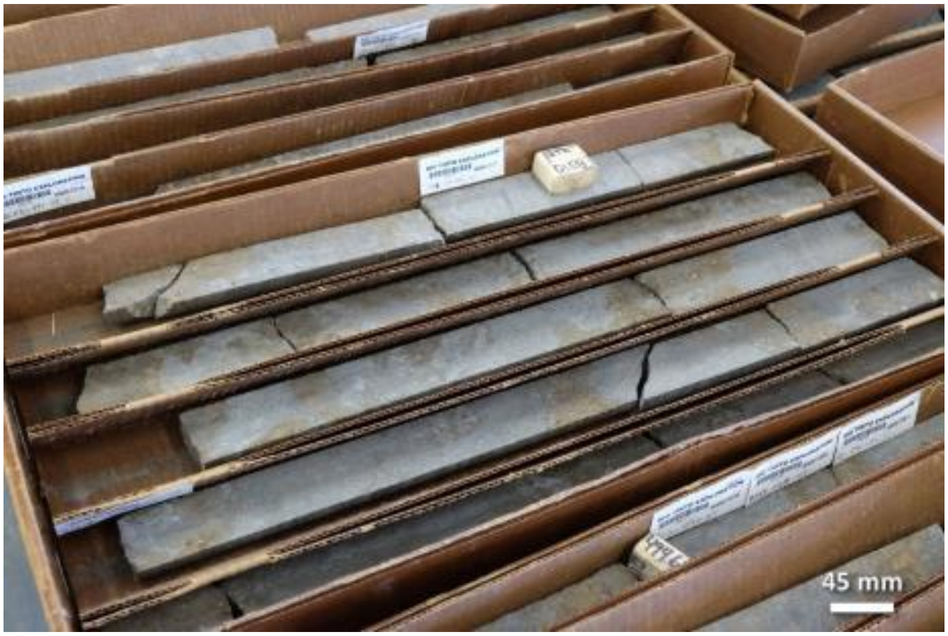


Figure 12-3: Example of Core from the MSU

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 did collect 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

Golder No.	Kennecott No.	Cu (%)		Ni (%)		Co (%)		Au ppm (g/t)		Pt ppm (g/t)		Pd ppm (g/t)		SG	
		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

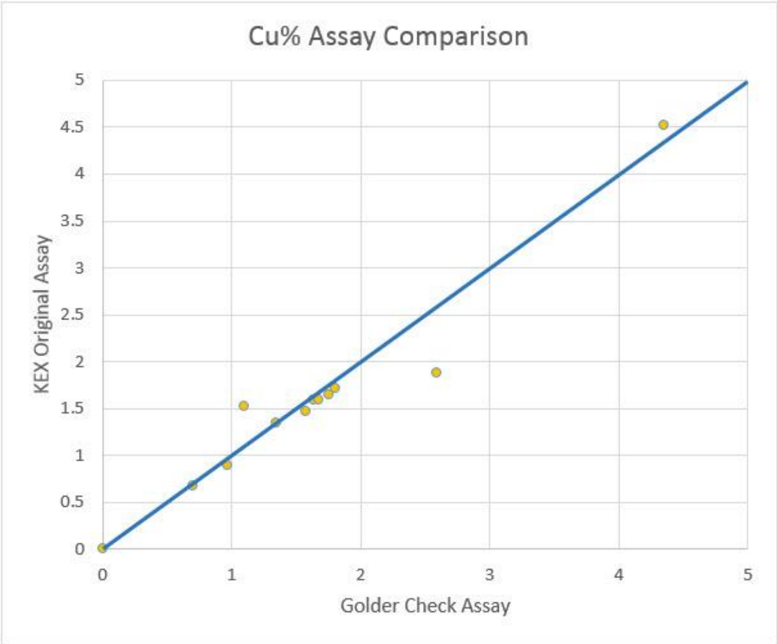


Figure 12-4: Validation Check of Cu Assays

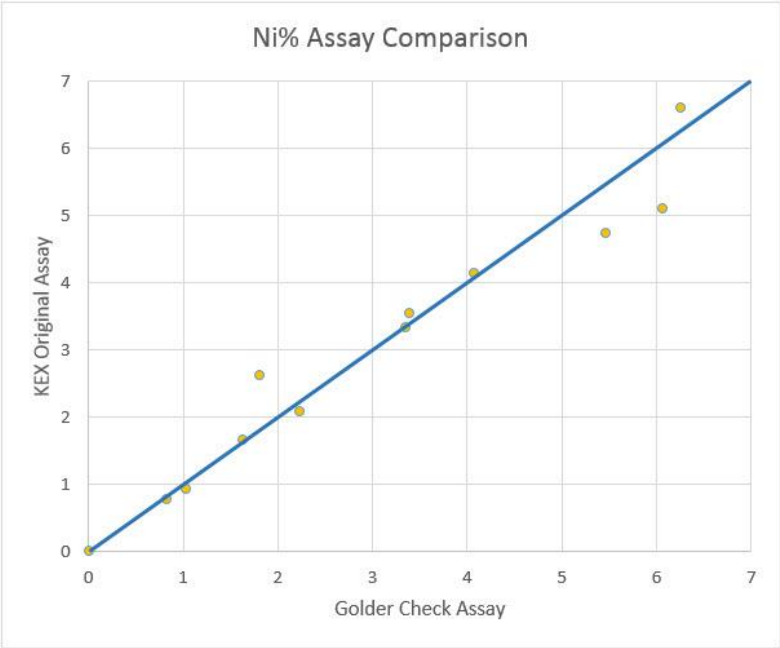


Figure 12-5: Validation Check of Ni Assays

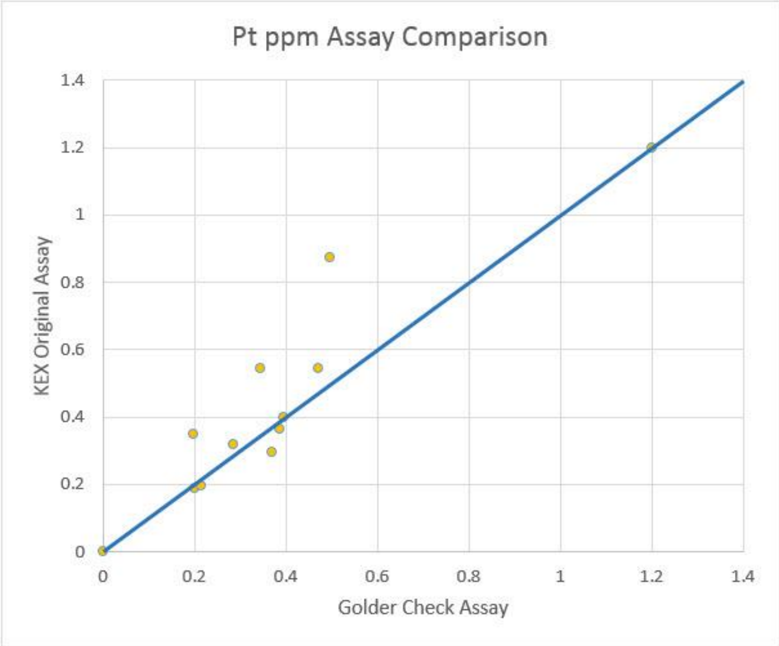


Figure 12-6: Validation Check of Pt Assays

On completion of the data validation, site visit and verification sampling, it is the QP's opinion that the Tamarack North Project drill hole database has been prepared in accordance to CIM Estimation of Mineral Resources and Mineral Reserves Best Practise Guidelines and is of suitable quality to support the mineral resource estimate in this PEA.

The QP recommends that SG measurements are completed from sample pulps where data is currently only available from field measurements.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

A total of 40 composites were evaluated in metallurgical test programs between 2006 and 2020. The test programs were completed at the facilities of SGS Minerals Services in Lakefield, Ontario, Canada and XPS in Sudbury, Ontario, Canada. The head grades of the composites ranged between 0.30 and 6.39% Ni and 0.20 and 2.80% Cu and included samples from the MSU, SMSU, CGO, 138 Zone, and 238 Zone.

13.1 Historic Metallurgical Data

13.1.1 Comminution Tests

Bond ball mill grindability (BWi) tests were carried out on seven domain composites in the 2016/2017 metallurgical program, to determine the energy requirements for ball milling. The tests were performed at a screen size of 106 µm (150 mesh). This screen size is representative of a mill discharge product of approximately P_{80} = 75 µm. The ball mill work indices range between 11.3 kWh/t for an MSU composite and 21.1 kWh/t for a CGO composite and the results for all seven composites are presented in Table 13-1.

Table 13-1: Tamarack Bond Ball Mill Grindability Tests Results

Composite	BWI in kWh/t
MSU	11.3
SMSU	15.1
Lower MZNO	21.0
CGO	21.1
Main North	20.1
Upper MZNO	15.0
238 Zone	18.7

No other types of crushing or grinding tests were completed to date. These tests will be included in the next phase of testing to produce data for more accurate sizing of the crushing and grinding circuit.

13.1.2 Mineralogy Characterization – Domain Composites

A chemical and mineralogical characterization was completed on the seven composites that were evaluated in the 2016/2017 metallurgical program.

A minor element scan identified Fe, Mg, and aluminum (Al) as the most abundant elements in the seven 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 7.3% in the CGO composite. The concentrations of talc were low in all composites and ranged between 0.14% in the SMSU and 0.91% in the CGO composite.

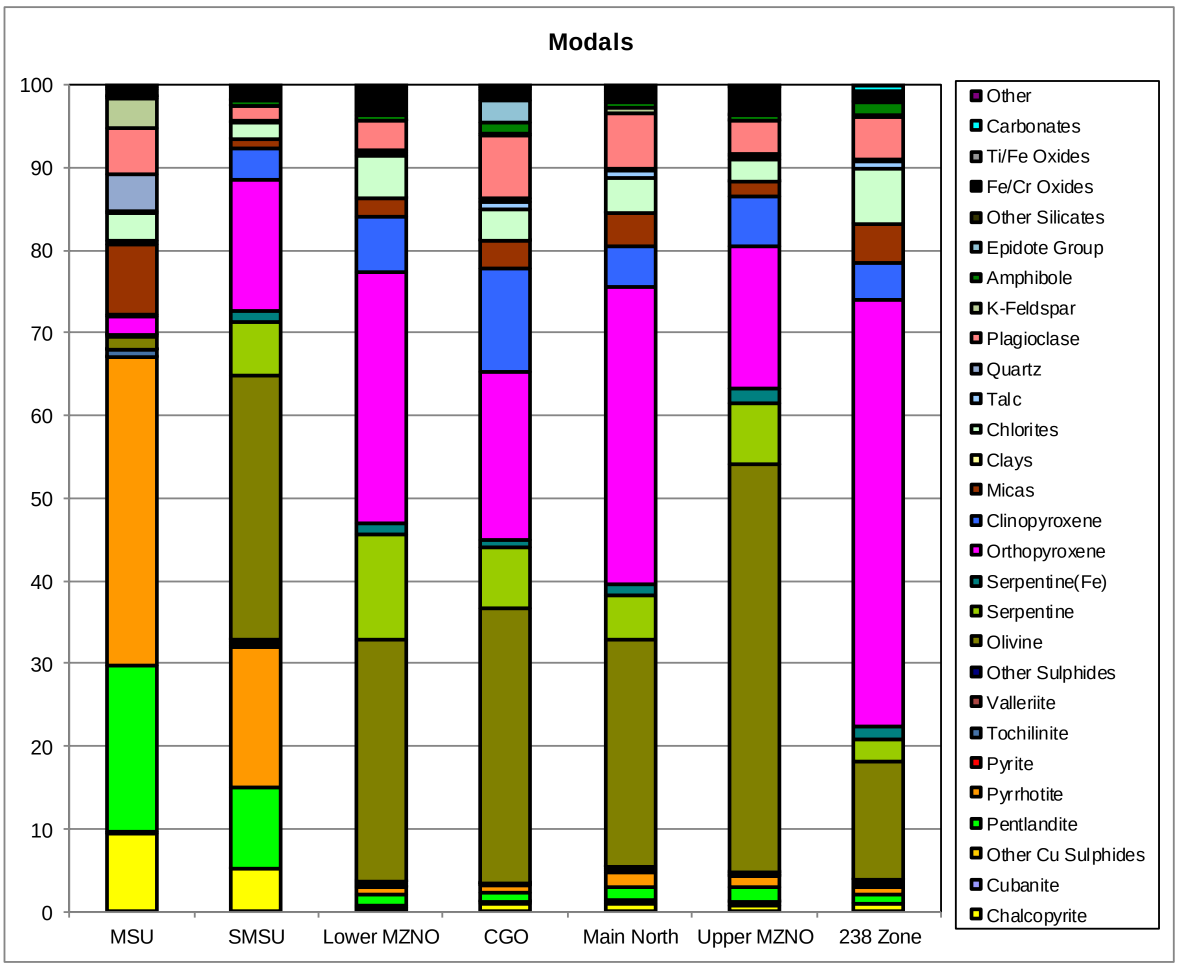
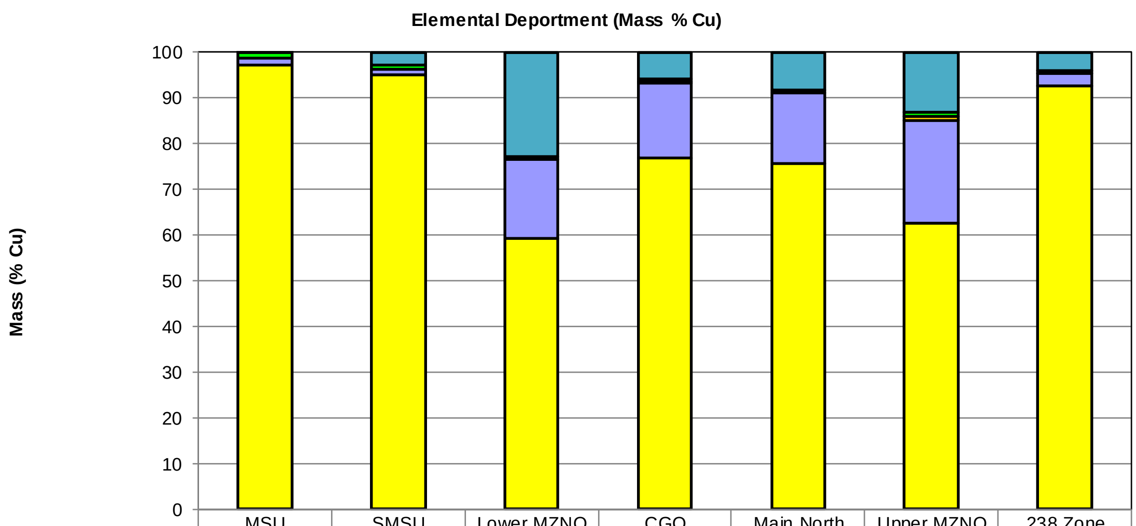


Figure 13-1: Modals of Tamarack Composites

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 CGO and Main North disseminated composites, the Cu deportment into Cpy was only 75.6% to 77.0%. Between 15.5% and 16.3% of the Cu was associated with cubanite and 5.9% to 8.2% 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 MZNO	CGO	Main North	Upper MZNO	238 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

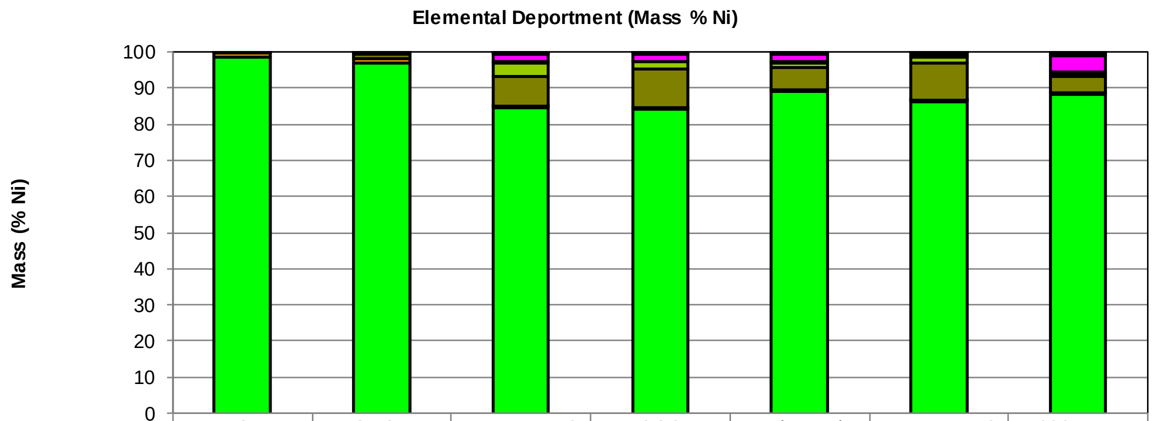
Figure 13-2: Elemental Deportment of Cu

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 Quantitative Evaluation of Materials by Scanning Electron Microscope (QEMSCAN) analyses is presented in Figure 13-3. While 98.1% and 96.0% of the Ni was associated with Pn in the MSU and SMSU composites respectively, the values decreased to 84.3% in the CGO composites. Up to 10.4% of the Ni units in the CGO composite were associated with olivine, which renders them unrecoverable by means of sulphide flotation. The increased deportment of Ni into non-sulphide gangue minerals is the primary reason for the sharp decrease in Ni rougher recovery for the lower grade samples. While mineralogical analysis was conducted on very few samples, Ni sulphide chemical analysis identified a consistent 0.1% Ni head grade associated with non-sulphide gangue minerals in low-grade composites, which is not recoverable by means of sulphide flotation. In a sample with a 0.5% Ni head grade, the Ni content in non-sulphide gangue minerals represents approximately 20% of the entire Ni values in the sample.

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

Element	MSU	SMSU	Lower MNZO	CGO	Main North	Upper MZNO	238
%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 MZNO	CGO	Main North	Upper MZNO	238 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. These values decreased to 66.0% and 72.7% in the CGO and Main North 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 CGO and Main North composites with values of 58.1% and 69.5%, respectively.

13.1.3 Metallurgical Optimization

A flowsheet optimization program was carried out at XPS in Sudbury, Ontario and SGS in Lakefield, Ontario using an LOM composite that attempted to replicate the combined SMSU, MSU, and 138 Zone. The objective of the metallurgical program was to develop a flowsheet and suitable conditions to produce a Ni concentrate that could be marketed to smelters, converted into Ni powder, or treated in a downstream hydrometallurgical processing facility. The minimum grade target for the Ni concentrate was established at 10.5% Ni to ensure good marketability for the pyrometallurgical option. Further, the process development aimed to produce a saleable Cu concentrate grading at least 25% Cu.

At the beginning of the optimization program, two composites were generated for mineralogical characterization. One composite attempted to be representative of the combined SMSU and MSU mineralization and the second composite aimed to represent only the 138 Zone mineralization. The two composites were subjected to mineralogical characterization by QEMSCAN. The modal mineralogy of the two composites is presented in Figure 13-4. Total sulphides accounted for 7.94% and 20.1% of the total sample mass in the 138 Zone and SMSU/MSU composites, respectively. Pn was the only Ni mineral, while Cpy and cubanite were identified as the Cu minerals. No electron probe analysis was performed on the two composites to quantify the deportment of Ni into Po, but it was assumed to be in line with the mineralogical analysis of the domain composites.

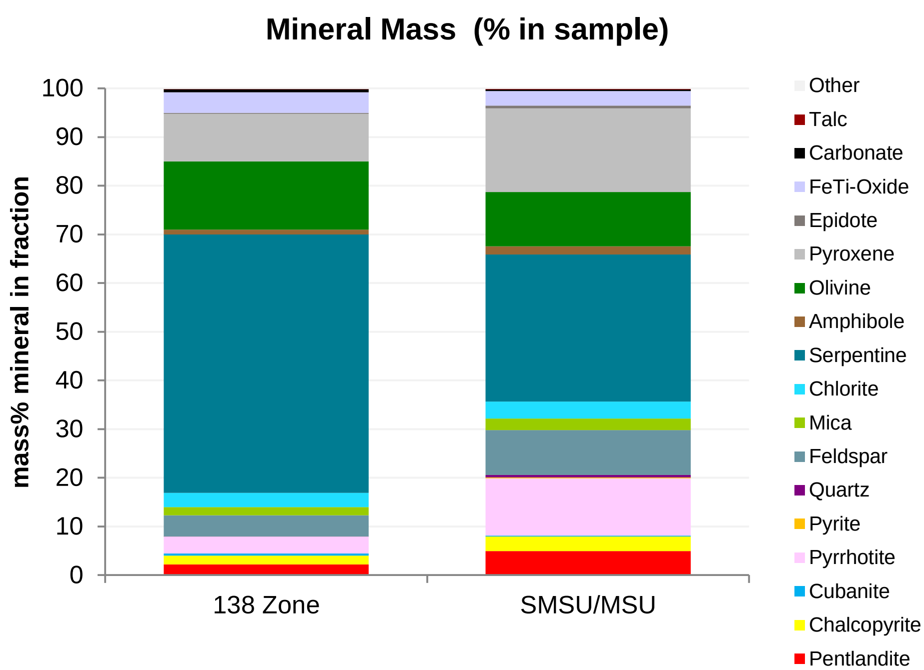


Figure 13-4: Modal Mineralogy of 138 Zone and SMSU/MSU Composites

The average mineral grain size of Pn was 25 µm in the 138 Zone composite and 42 µm in the SMSU/MSU composite. The average Cpy grain size was smaller at 15 µm and 37 µm in the 138 Zone composite and SMSU/MSU composite, respectively. The average mineral grain sizes of Po was comparable with Cpy.

To determine the mineralogical differences of material making up the two composites, the mineralization was further broken down into a high-grade and low-grade composite for the 138 Zone, MSU, as well as an Upper SMSU and Lower SMSU composite for the SMSU/MSU mineralization. The modal mineralogy of the five composites is depicted in Figure 13-5.

The total sulphide content in the MSU composite was 26.7% compared to 4.29% and 6.82% in the Lower and Upper SMSU composites, respectively. The main difference between the Lower and Upper SMSU composites was the higher Po content of 4.03% in the Upper SMSU compared to only 1.74% in the Lower SMSU. The concentration of Pn was identical in both composites at 2.00%. The two SMSU composites yielded higher cubanite concentrations of 0.25% and 0.18% in the Lower and Upper SMSU, respectively, compared to only 0.08% in the MSU composite. Cubanite is frequently linked to decreased Cu recoveries and lower Cu concentrate grades.

The two 138 Zone composites yielded serpentine concentrations of 53.4% to 54.7%, which may affect the selectivity of the flotation process.

Talc concentrations were low in all five composites at 0.02% to 0.06%.

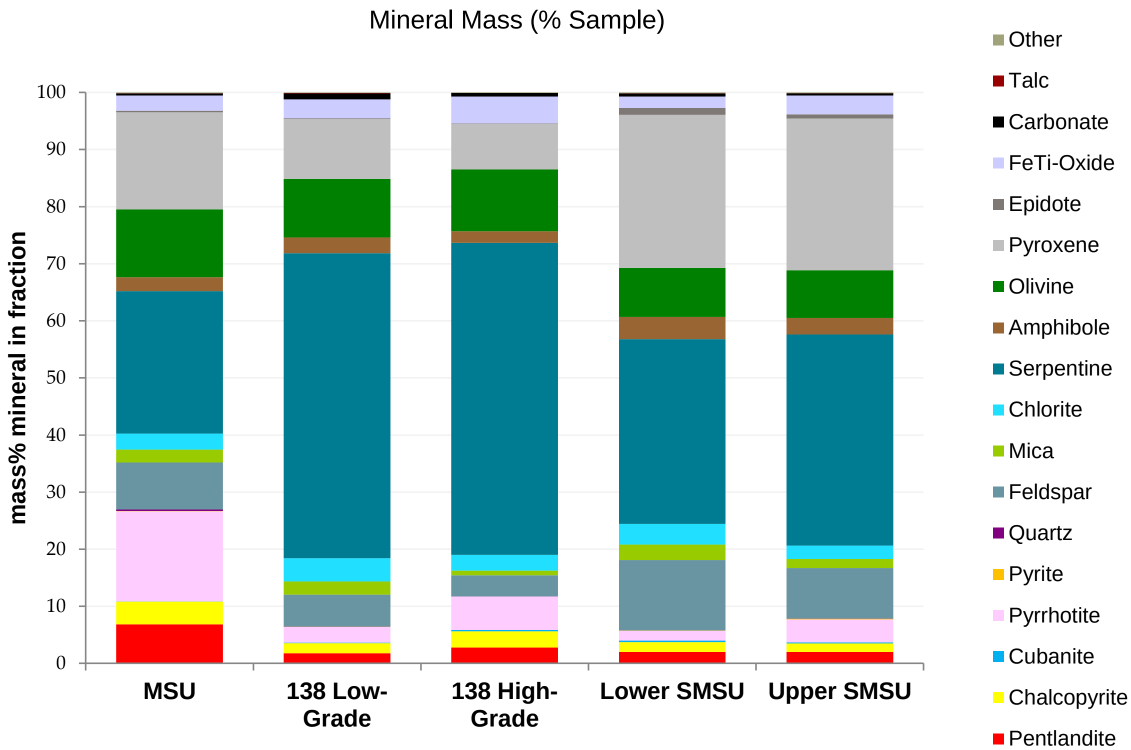


Figure 13-5: Modal Mineralogy of SMSU, MSU, and 138 Zone Composites

Prior to determining the composite recipe, it was necessary to establish if the inclusion of the 138 Zone mineralization in an LOM composite results in metallurgical challenges that would potentially prevent the ability to produce acceptable concentrates. Hence, one MSU/SMSU composite representing 5.32 Mt of the Tamarack mineralization and one MSU/SMSU/138 Zone composite representing the entire 8.02 Mt of the Tamarack mineralized material were generated and subjected to flotation testing. It should be noted that these tonnage numbers reflect the previous March 2020 PEA resources and do not include the additional tonnes that are reported in this PEA as a result of additional drilling that was performed between the March 2020 PEA and February 2021 PEA. At the beginning of the metallurgical optimization program, the mineralized material of 8.02 Mt was the most current information.

In order to determine the impact of the 138 Zone on the metallurgical performance, a total of 10 rougher kinetics tests were carried out on the two composites. Side-by-side tests on the two composites were completed on the LOM with and without the 138 Zone composites to investigate the impact of primary grind size, flotation time, reagent dosage, and reagent addition points. The results revealed that despite a more challenging mineralogy, the flotation selectivity in the rougher was not negatively impacted by the 138 Zone mineralization. One baseline cleaner test performed on each of the two composites confirmed a consistent selectivity in the cleaner stages and, therefore, a decision was made to include the 138 Zone mineralization in the LOM composite.

Seven batch cleaner flotation tests were carried out on the LOM with 138 Zone composite to establish suitable cleaner flotation conditions that maximize the Ni, Cu, and Co recovery into a 2nd cleaner concentrate while minimizing the entrainment of gangue minerals.

Due to scheduling conflicts a decision was made to perform the remaining tests at SGS in Lakefield, Ontario. A total of two rougher and eight batch cleaner tests were carried out by SGS to replicate the initial work performed by XPS and to develop suitable conditions for the Cu/Ni separation circuit.

At the end of the test program, a LCT was carried out on the LOM with 138 Zone composite to simulate a continuous operation of the circuit. The flowsheet that was used in the LCT is depicted in Figure 13-6. The mineralized material was ground to a size of P_{80} = 100 µm and then subjected to bulk rougher and bulk scavenger flotation. The combined product was upgraded in two stages of bulk cleaning. The bulk 1st cleaner tailings were subjected to a scavenger stage to minimize metal loses to the cleaner tailings. The bulk 1st cleaner scavenger concentrate and the bulk 2nd cleaner tailings were reground in a ball mill to improve mineral liberation before being combined with the bulk rougher and bulk scavenger concentrate of the next cycle. The bulk 2nd cleaner concentrate constituted the Cu/Ni separation circuit feed and was reground to P_{80} ~ 25 µm to improve liberation of the Ni and Cu minerals. This target grind size was established based on the results of the QEMSCAN analysis. The Cu/Ni separation circuit consisted of a standard Cu rougher and two stages of cleaning. Lime was added to maintain a pH of 12.0 and no further depressants or collectors were required.

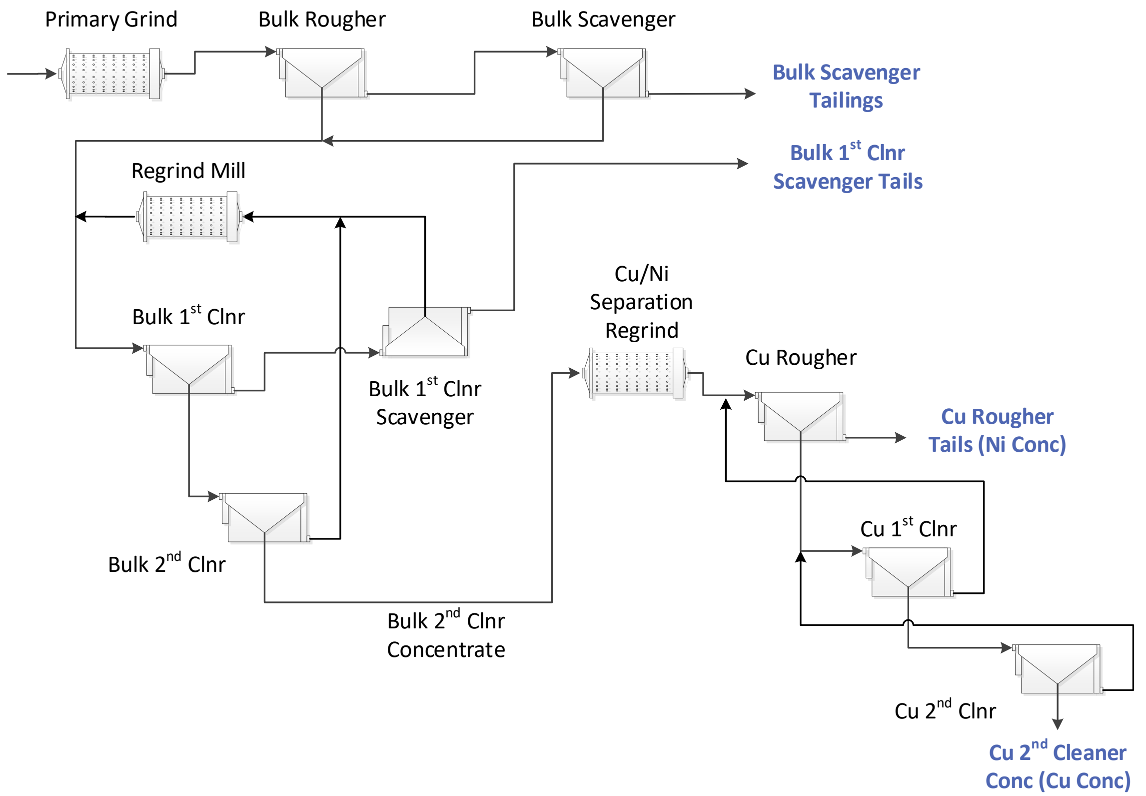


Figure 13-6: Flowsheet of LCT

The results of the LCT are presented in Table 13-3. A total of 83.2% of the Ni and 15.9% of the Cu were recovered into a Ni concentrate grading 10.7% Ni and 1.22% Cu. The Cu concentrate contained 71.6% of the Cu at a grade of 29.9% Cu. Less than 2% of the Ni reported to the Cu concentrate at a grade of 1.13% Ni. Mineralogical analysis of the Cu concentrate revealed that almost 50% of the Pn reporting to this product was free or liberated. This suggests that 50% of the Ni units were recovered to the Cu concentrate through entrainment, which is difficult to control in small laboratory scale tests. It is postulated that the Ni recovery into the Cu concentrate will be substantially lower in a commercial scale continuous operation due to better control of entrainment.

The Tamarack mineralization hosts a range of Mg bearing minerals and recovery into the Ni concentrate must be minimized. The proposed process conditions include depressants for the Mg minerals in the cleaning stage but carry over of Mg minerals into the Ni concentrate was still significant for the disseminated domains. The MgO concentration in the Ni concentrate of the LOM with 138 Zone composite was 4.66% MgO and, therefore, just below the typical smelter penalty threshold value of 5.0% MgO. Non-sulphide gangue rejection optimization is planned for future test programs to further reduce the gangue content in the Ni concentrate.

Table 13-3: Process Mass Balance

Product	Weight	Assays, %					% Distribution				
	%	Cu	Ni	S	Fe	MgO	Cu	Ni	S	Fe	MgO
Cu Conc	2.2	29.9	1.13	32.5	32.5	0.80	71.6	1.6	12.9	4.8	0.1
Ni Conc	11.9	1.22	10.7	28.6	40.6	4.66	15.9	83.2	61.8	32.8	2.4
Bulk 1st Clnr Scav Tails	9.3	0.40	0.74	6.42	17.5	22.7	4.0	4.5	10.8	11.0	9.0
Bulk Scavenger Tails	76.6	0.10	0.21	1.05	10.0	27.1	8.5	10.7	14.6	51.4	88.6
Combined	100.0	0.92	1.53	5.54	14.8	23.4	100.0	100.0	100.0	100.0	100.0

13.1.4 Metallurgical Analysis

A thorough analysis of all current and historical flotation tests was carried out to develop regression models for the flotation performance as a function of the Ni and Cu head grades.

The process variables of over 240 rougher, batch cleaner, and locked cycle flotation tests were reviewed, and tests with suitable conditions such as grind size and reagent regime were selected to develop refined grade and recovery projections. Further, samples well below the cut-of grade were not included in the analysis. Of the over 240 flotation tests, the rougher flotation conditions of approximately 60 tests were deemed suitable for the Tamarack mineralization in terms of primary grind size, reagent suite and dosages, and flotation time.

The bulk rougher flotation test results suggest that a natural pH and a primary grind size P_{80} of 100 to 130 µm should be targeted to achieve high Ni and Cu recoveries into a bulk rougher concentrate. Sufficient flotation time and collector addition is instrumental in achieving high Pn recovery into the bulk rougher concentrate.

The Ni rougher recoveries are plotted against the Ni head grade in Figure 13-7. The Ni rougher recovery model employs two different regression curves for head grades above and below 1.0% Ni:

< 1.0% Ni   Ni Rec = 21.757*ln(Ni Head) + 88.027

> 1.0% Ni   Ni Rec = 1.3936*Ni Head + 86.88

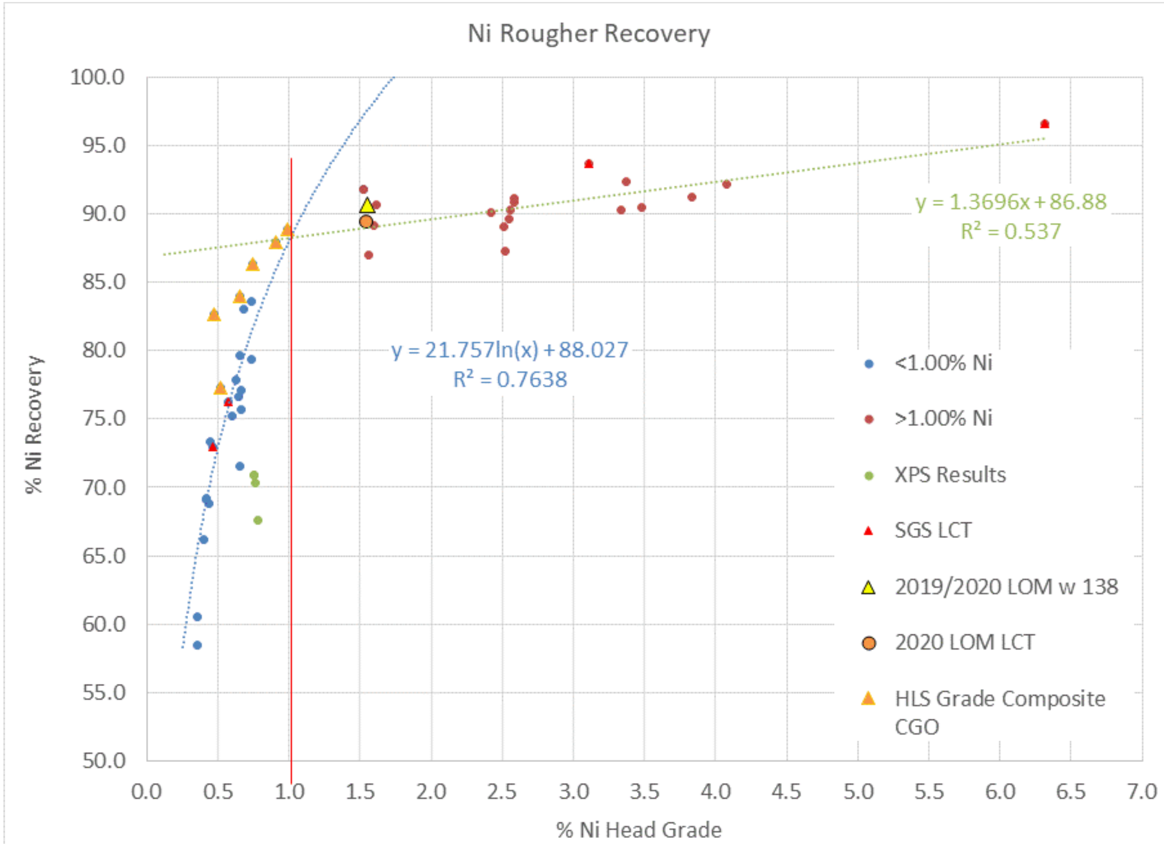


Figure 13-7: Ni Rougher Recovery vs Ni Head Grade

The Cu rougher recovery vs Cu head grade results are depicted in Figure 13-8. While the inclusion of the 138 Zone did not have a fundamental impact on the flotation selectivity or Ni recovery, the Cu flotation performance is negatively impacted by the 138 Zone mineralization. This inferior flotation performance was expected based on the mineralogical characterization of this domain and previous laboratory scale results.

Cu displays more variation compared to Ni but follows the same overall trend. At higher Cu head grades the variation in Cu recoveries was relatively small for a given head grade but increased noticeably at lower head grades. It is postulated that varying valleriite contents in the composites that were included in the development of the regression curve for Cu may be a primary reason for the increased variation in results. A complete Cu deportment study would be required for each composite to determine if more accurate projections could be made when taking into account the different Cu mineral species.

The applicable Cu rougher recovery regression curves for the LOM with 138 Zone are as follows:

< 0.30% Cu Cu Rec = 7.979*ln(Cu Head) +100.67

> 0.30% Cu Cu Rec = 0.8661* Cu Head + 90.75

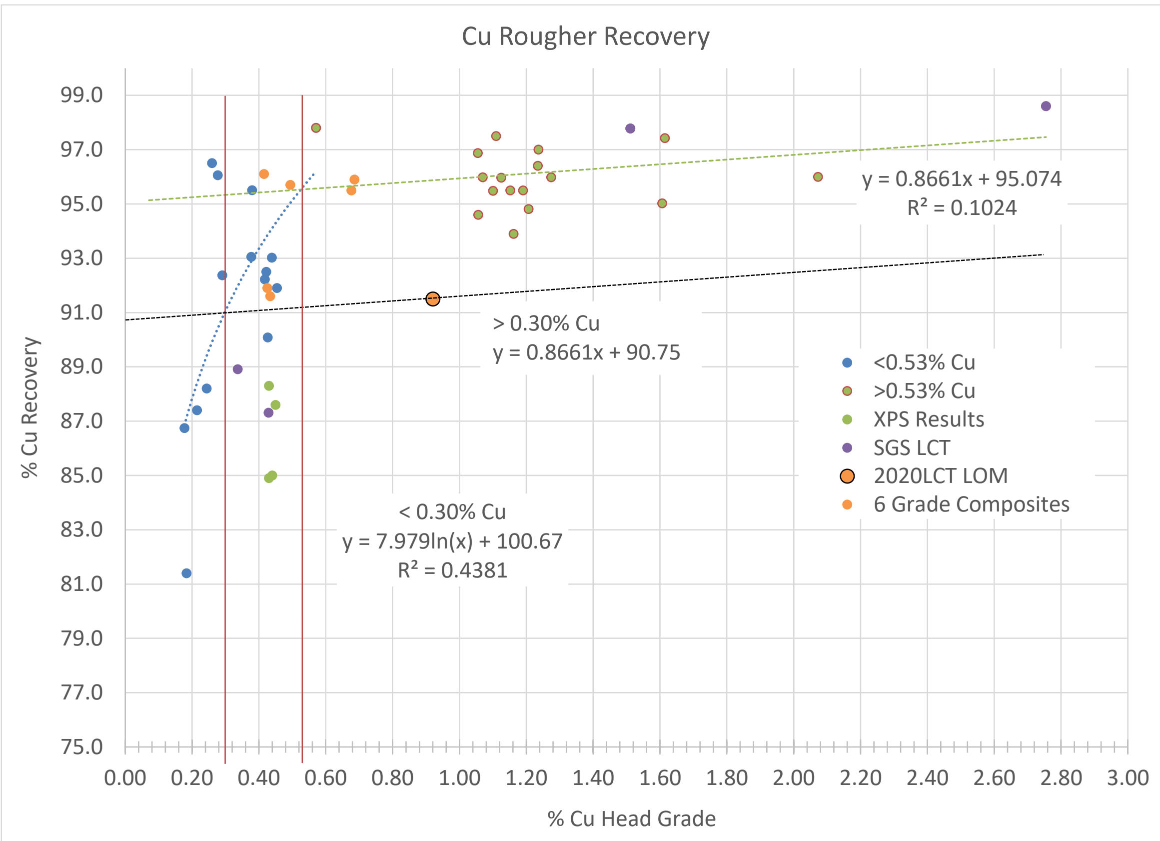


Figure 13-8: Cu Rougher Recovery vs Cu Head Grade

13.2 Cleaner Flotation Performance

Bulk cleaner concentrates of over 20% combined Cu and Ni were achieved for most composites with a single stage of cleaning. Regrinding of the bulk rougher concentrate resulted in elevated Ni losses, but conditions were not optimized.

Open circuit cleaner tests underestimate the overall metal recovery since intermediate concentrates and tailings are treated as final tailings. In a commercial operation, these intermediate products are cycled within the circuit, and the majority of the contained metal units eventually report to a final concentrate. On a laboratory scale, LCTs simulate the operation of a commercial plant by circulating all intermediate streams from one cycle to the next. LCTs are the only laboratory scale tests that provides a good assessment of the closed-circuit performance that is to be expected during continuous operation. Only seven LCTs were carried out to-date. The closed-circuit stage recoveries for Ni and Cu are presented in Figure 13-9 and Figure 13-10, respectively.

The three data points highlighted in blue were derived from LCTs on the SMSU, MSU, and LOM with 138 Zone composite. The red data points were recent open circuit tests on six low-grade composites with modelled closed-circuit performance. The cleaner Ni stage-recovery gradually decreased with lower head grades. However, owing to the simplicity of the flowsheet, stage-recoveries remain over 90% even for low-grade samples.

The cleaner Cu stage-recovery followed a similar trend, but with increased data scatter at the lower head grades.

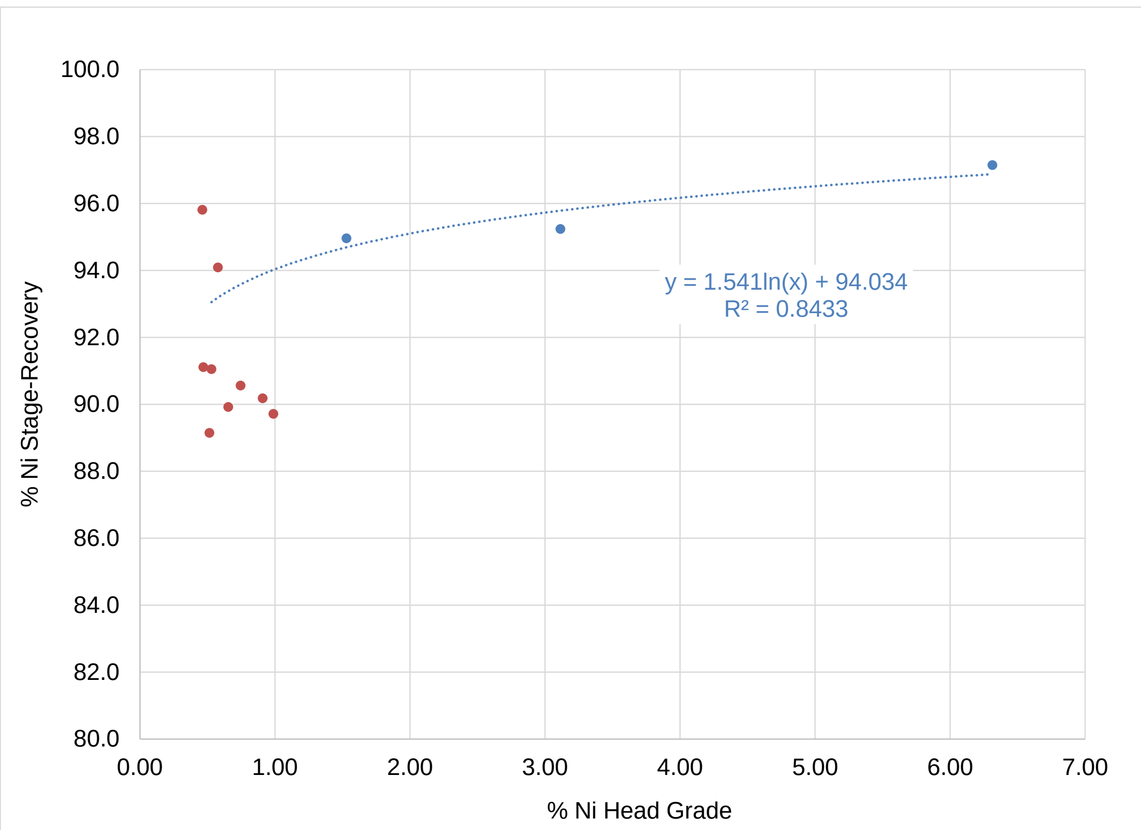


Figure 13-9: Closed Circuit Ni Stage-Recovery as a Function of Ni Head Grade

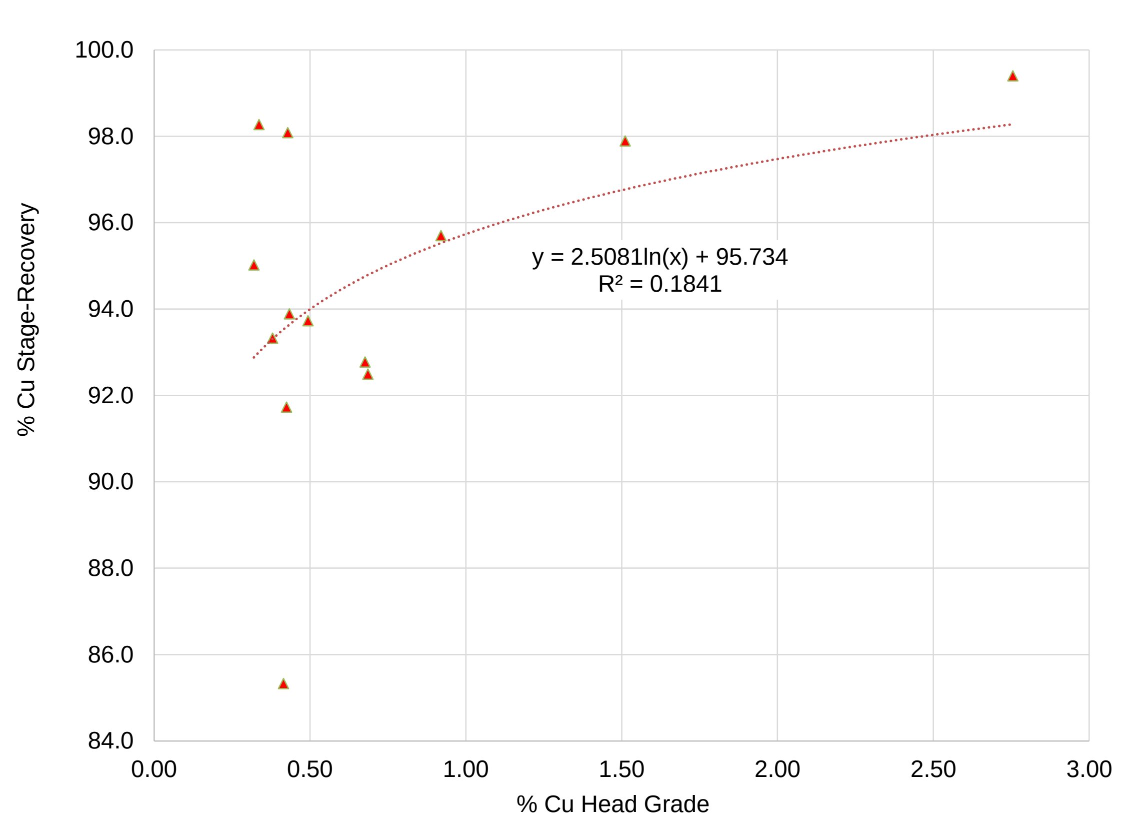


Figure 13-10: Closed Circuit Cu Stage-Recovery as a Function of Cu Head Grade

While several cleaner flotation tests employed Cu/Ni separation stages, they were also operated in open circuit. The only closed-circuit tests were the LCTs performed in 2016/2017 and the LCT on the LOM with 138 Zone composite in 2020. Hence, these results were chosen to project the deportment of Ni and Cu into the two concentrates. The Ni and Cu concentrate grades as a function of their respective head grades are depicted in Figure 13-11 and Figure 13-12, respectively.

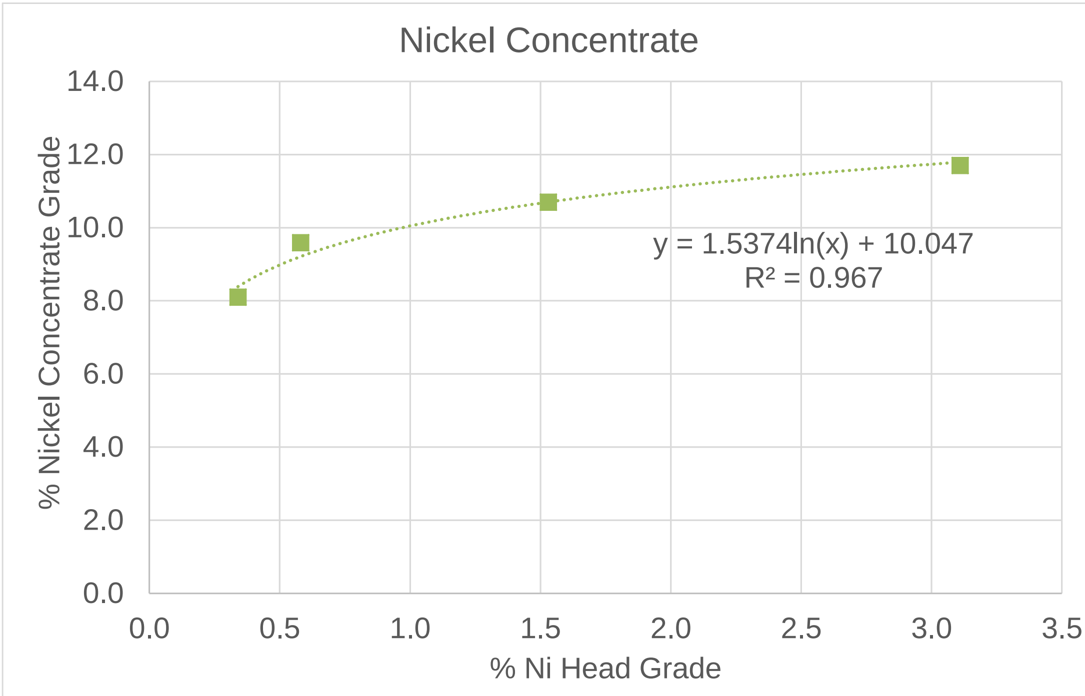


Figure 13-11: Ni Concentrate Grade vs Ni Head Grade

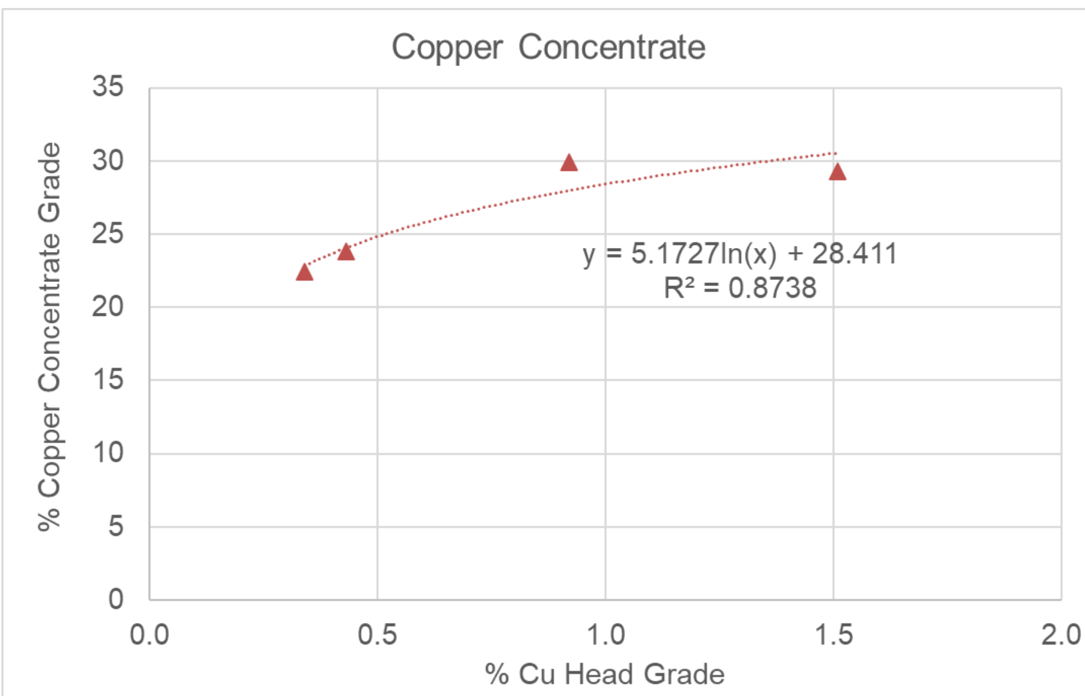


Figure 13-12: Cu Concentrate Grade vs Cu Head Grade

13.2.1 Concentrate Characterization

The final Cu and Ni concentrates of the LOM with 138 Zone composite LCT were submitted for chemical analysis to identify potential credit and penalty elements.

Credit elements in the two concentrates are presented in Table 13-4. The Cu concentrate contained 5.88 g/t Au, which will be payable above a deduction of typically 1 g/t. The platinum grade of 2.42 g/t in the Ni concentrate is likely another payable element.

Table 13-4: Ni Concentrate – Credit Elements

Sample ID	Mass	Co %	Pt (g/t)	Pd (g/t)	Au (g/t)	% Distribution			
						Co	Pt	Pd	Au
Ni Concentrate	11.9	0.240	2.42	1.26	0.28	64.1	82.5	69.3	18.7
Cu Concentrate	2.2	0.020	1.31	1.14	5.88	1.0	8.3	11.6	72.6
Feed	100	0.045	0.349	0.216	0.178	100	100	100	100

The analysis of the individual products of the LCT produced an average MgO content in the Cu and Ni concentrate of 0.80% MgO and 4.66% MgO, respectively. The complete results of the detailed concentrate analysis are presented in Table 13-5. Both concentrates reveal low levels of deleterious elements for Cu and Ni smelters and no penalty payments are expected.

Table 13-5: Detailed Concentrate Analysis

Sample ID	Assays (g/t)																
	Au	Pt	Pd	F	Cl	Hg	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Cr	Fe	K
Ni Concentrate	0.28	2.42	1.26	9	170	< 0.3	13	3,410	14	37.4	0.06	5.1	4,270	< 2	325	405,000	474
Cu Concentrate	5.88	1.31	1.14	5	41	< 0.3	35	439	< 10	4.5	< 0.03	5.7	1,020	12	18	320,000	< 70
Sample ID	**Li**	**Mn**	**Mo**	**Na**	**P**	**Pb**	**Sb**	**Se**	**Si**	**Sn**	**Sr**	**Ti**	**Tl**	**U**	**V**	**Y**	**Zn**
Ni Concentrate	< 8	258	12.4	932	< 200	55.7	2.1	93	32,200	< 2	9.45	399	< 0.4	< 0.4	20	< 1	192
Cu Concentrate	< 8	57.1	2.5	194	< 200	185	1.3	97	4,100	12	1.5	36.8	< 0.4	< 0.4	4	< 1	555

13.3 Life-of-Mine Mass Balance

The findings of the metallurgical analysis were used to develop of the LOM mass balance for the head grade of the February 2021 PEA.

The results of the LCT on the LOM with 138 Zone composites and the regression curves were employed to develop a high-level mass balance. This manual mass balance was used as a starting point to generate a full circuit mass balance using the Outotec HSC modelling software. The HSC mass balance is presented in Table 13-6.

The anticipated Ni concentrate grade is 10.2% Ni at 81.5% Ni recovery. A total of 15.4% of the Cu units in the mill feed is also expected to report to the Ni concentrate. The Cu concentrate is projected to contain 69.3% of the total Cu units at a grade of 28.5% Cu and 1.10% Ni.

Table 13-6: Process Mass Balance Ni Sulphate Scenario

Stream	Mass Rec %	Assay (%)			Recovery (%)		
		Cu	Ni	S	Cu	Ni	S
Bulk Rougher Feed	100.0	0.74	1.34	5.10	100.0	100.0	100.0
Bulk Rougher Conc	21.8	2.98	5.32	19.7	87.6	86.6	84.0
Bulk Rougher Tails	78.2	0.12	0.23	1.04	12.7	13.4	16.0
Mags	7.1	0.17	0.48	3.86	1.6	2.5	5.3
High Sulphur Thickener UF	16.3	0.30	0.51	5.09	6.6	6.2	16.3
Non-Mags	71.1	0.12	0.20	0.76	11.0	10.9	10.7
Low S Thickener Underflow	71.1	0.12	0.20	0.76	11.0	10.9	10.7
Low-Sulphur Tailings Filter Cake	71.1	0.12	0.20	0.76	11.0	10.9	10.7
Regrind Ball Mill Discharge	19.0	0.81	2.20	19.3	20.6	31.2	71.9
Bulk Cleaner 1 Conc	17.9	3.81	7.17	27.5	91.9	95.6	96.3
Bulk Cleaner 1 Tails	22.9	0.54	1.30	13.3	16.6	22.2	59.6
Bulk Cleaner Scav Conc	13.7	0.64	1.81	18.1	11.8	18.5	48.7
Bulk Cleaner Scav Tails	9.2	0.40	0.53	6.0	5.0	3.7	10.9
Bulk Cleaner 2 Conc	12.6	4.93	8.85	29.7	83.5	82.9	73.1
Bulk Cleaner 2 Tails	5.3	1.20	3.20	22.3	8.6	12.7	23.2
Cu-Ni Sep Regrind Mill Discharge	12.6	4.94	8.85	29.7	83.7	83.0	73.1
Cu Rougher Conc	2.8	20.9	3.59	31.3	78.4	7.4	17.1
Ni Concentrate	10.8	1.04	10.2	29.3	15.2	81.5	61.6
Cu Cleaner Conc	2.4	25.1	1.94	31.6	81.8	3.5	15.0
Cu Cleaner Tails	1.0	7.29	8.23	29.4	9.6	6.0	5.6
Cu Reclnr Conc (Cu Conc)	1.8	28.5	1.10	32.4	69.3	1.5	11.4
Cu Reclnr Tails	0.6	15.5	4.45	29.4	12.7	2.0	3.5
Cu Conc Thickener UF	1.8	28.5	1.10	32.4	69.3	1.5	11.4
Cu Conc Filter Cake	1.8	28.5	1.10	32.4	69.3	1.5	11.4
Ni Conc Thickener UF	10.8	1.06	10.2	29.3	15.4	81.5	61.6
Ni Conc Filter Cake	10.8	1.06	10.2	29.3	15.4	81.5	61.6

13.4 Analysis and Recommendations

A comprehensive review of the current and historical test results formed the basis for updated overall Ni and Cu recovery projections. The equations to determine the Ni and Cu recovery into the two concentrates as a function of the head grade are presented below:

Ni in Ni Concentrate

>1.0% Ni % Ni Recovery = (1.3936x+86.88)*(1.541ln(x) +94.034)/100*0.98

0.40 – 1.0% Ni % Ni Recovery = (21.757ln(x)+88.027)*(1.541ln(x) +94.034)/100*0.98

Ni in Cu Concentrate

>1.0% Ni % Ni Recovery = (1.3936x+86.88)*(1.541ln(x) +94.034)/100*0.02

0.40 – 1.0% Ni % Ni Recovery = (21.757ln(x)+88.027)*(1.541ln(x) +94.034)/100*0.02

Cu in Cu Concentrate

>0.30% Cu % Cu Recovery = (0.8661y+90.75)*(2.5081ln(x)+95.734)/100*0.85

<0.30% Cu % Cu Recovery = ((7.979ln(x)+100.67) *(2.5081ln(x)+95.734)/100*0.85

Cu in Ni Concentrate

>0.30% Cu % Cu Recovery = (0.8661y+90.75)*(2.5081ln(x)+95.734)/100*0.15

<0.30% Cu % Cu Recovery = ((7.979ln(x)+100.67) *(2.5081ln(x)+95.734)/100*0.15

During the next phase of testing, it will be paramount to further refine the regression curves with additional LCTs using domain samples and composites that reflect the actual mill feed grades and domain blends over the projected mine life. The production of a geo-metallurgical model to assess the suitability of the samples tested and a full variability test program throughout the deposit will have to be conducted during the PFS phase.

Ni smelters generally prefer a Ni concentrate with a minimum Fe:MgO ratio of 5:1. The Ni concentrate generated from the Tamarack LOM composite produced a good Fe:MgO ratio of 8.7:1, which is 75% higher than the desired minimum Fe:MgO ratio.

The reagent regime developed for the Tamarack mineralization is presented in Table 13-7. Given the significant cost of the proposed reagent regime, a dosage optimization will be carried out during the next phase of optimization testing.

Also, collector dosages will have to be optimized separately for each mineral domain as a function of head grades and the composition of non-sulphide gangue minerals because some minerals act as reagent robbers.

Table 13-7: Reagent Dosages

Reagent	Consumption of Mill Feed (g/t)
SIPX	215
PAX	230
MIBC	150
Depramin C	30
Lime	2,150
Flocculant	30

13.5 Hydrometallurgical Testing

The Ni concentrate that was generated in the LCT using the LOM with 138 Zone composite was utilized for a scoping level hydrometallurgical testing. The test program included leaching and impurity removal tests to establish metal extraction rates during leaching and metal losses in the impurity removal stages.

Two leach technologies were investigated, namely POX and the Albion process. The main differences between the two technologies are that the POX process employs a pressurized vessel (an autoclave) with a relatively short retention time, whereas the Albion process operates at atmospheric pressure with longer retention times and a fine regrind of the Ni concentrate prior to leaching.

Five POX tests were completed to evaluate the impact of temperature, regrind size and the addition of chlorides on metal extraction rates. Further, one Albion test was carried out employing conditions recommended by the Glencore technical team.

The results are presented in Table 13-8. Tests POX2, POX3, and POX4 produced the best overall results of the five POX tests. Test POX4 included a fine regrind of the Ni concentrate to P_{80} ~ 8 µm, and the results matched those of test POX2 for Ni and Co but yielded a 7% lower Cu extraction. Further, the precipitate of this test displayed very poor filtration properties, which suggests that solid-liquid separation would be challenging in a commercial operation. Test POX3 eliminated chlorides but was operated at a higher temperature of

225°C. Cu and Co recoveries matched or exceeded the results of test POX2, but Ni recoveries were 3% lower. Since test POX2 produced amongst the best overall extraction rates, required a lower temperature, and has the lowest sulphide oxidation rate (translating into oxygen savings), a decision was made to proceed with the conditions of test POX2 for the POX approach.

The test using the Albion process yielded good extraction rates of 97 to 99% for Ni, Co, and Cu. The leach liquor contained one of the highest Fe concentrations of all leach tests at 28,100 mg/L. A high Fe content in the leach liquor can lead to elevated metal losses during the impurity removal stage.

Table 13-8: POX and Albion Leach Conditions and Results

Test ID	POX1	POX2	POX3	POX4	POX5	Albion
Feed	As-Received Concentrate Sample	As-Received Concentrate Sample	As-Received Concentrate Sample	Fine Grind Feed Concentrate Sample	As-Received Concentrate Sample	Fine Grind Feed Concentrate Sample
Temperature (°C)	150	150	225	150	180	95
Oxygen Overpressure (kPa)	689	689	689	689	689	N/A
Initial Pulp Density (%)	10	10	10	10	10	25
Chloride Target (g/L)	0	2	0	0	0	0
Final PLS Acidity (g/L)	29	29	67	35	58	20
Sulphide Oxidation (%)*	70	69	99	99	98	65

* based on sulphide accountability

Extraction (%)	POX1	POX2	POX3	POX4	POX5	Albion
Ni	88	99	96	100	97	98
Co	89	99	99	99	96	97
Cu	53	88	100	81	97	99
Zn	93	91	91	90	90	96
Fe	40	12	7	45	44	89
Mg	81	79	86	82	84	86
Al	72	64	61	66	80	85
Cr	22	19	41	61	43	58
Mn	93	89	92	91	92	96
Ca	87	84	94	85	91	93
Na	24	32	60	42	50	26

Final PLS (mg/L)	POX1	POX2	POX3	POX4	POX5	Albion
Ni	10,000	11,800	12,500	11,100	12,000	5,410
Co	232	261	275	249	246	116
Cu	622	1,070	1,390	966	1,280	1,600
Zn	30	30	34	20	20	23
Fe	17,000	5,020	3,500	20,200	20,300	28,100
Mg	2,500	2,490	3,120	2,440	2,800	1,380
Al	251	235	240	237	284	144
Cr	10	7	20	73	17	34
Mn	54	36	39	36	36	63
Ca	411	430	537	227	473	202
Na	13	18	36	23	30	5

Final Residue (g/t)	POX1	POX2	POX3	POX4	POX5	Albion
Ni	15,500	641	4,550	627	6,080	5,610
Co	391	15	21	22	158	169
Cu	7,480	1,440	69	3,910	729	778
Zn	40	40	40	40	40	40
Fe	345,000	397,000	573,000	434,000	474,000	145,000
Mg	7,850	6,440	6,180	9,270	9,600	9,580
Al	1,270	1,300	1,870	2,030	1,280	1,060
Cr	450	298	355	815	419	1,030
Mn	48	46	45	64	59	99
Ca	831	816	462	702	807	635
Na	560	389	324	701	516	796

The autoclave discharge was subjected to a neutralization stage to remove most of the impurities such as Fe, Cr, and Al from the leach solution. The test revealed that a pH of 4.75 is required to precipitate the beforementioned elements to below detection limits. While Ni and Co precipitation were minimal at this pH, over 97% of the Cu co-precipitated into the residue. It was postulated that the high Fe concentration in the leach liquor resulted in the high Cu losses and, therefore, a two-stage neutralization approach was evaluated.

A target pH of 3.25 was established for the primary neutralization to maximize precipitation of Fe, Al, and Cr while minimizing co-precipitation of Cu. The results of the primary neutralization test are presented in Table 13-9. Only 3.7% of the Cu units reported to the precipitate together with 99.8% of Fe, 97.7% of Cr, and 72.7% of Al. This residue represents a final tailings stream.

Table 13-9: Results of Primary Neutralization

Sample & Quant.	Assay Units	Feed Solution	Final Filtrate	Residue	Precip. %
Ni	mg/L, g/t	10,900	9,360	51	0.0
Co	mg/L, g/t	235	201	<4	0.2
Cu	mg/L, g/t	951	723	370	3.7
Zn	mg/L, g/t	20	18	<20	7.9
Fe	mg/L, g/t	9,000	23.7	146,000	99.8
Mg	mg/L, g/t	2,650	2,700	915	2.6
Al	mg/L, g/t	231	24.6	930	72.7
Cr	mg/L, g/t	4.9	<0.2	135	97.7
Mn	mg/L, g/t	33.1	31.6	2.2	0.5
Ca	mg/L, g/t	423.0	573	167,000	94.8
Na	mg/L, g/t	21	17	62	22.3

The leach liquor of the primary neutralization stage was then contacted with limestone to increase the pH to 5.0. The results of the secondary neutralization are presented in Table 13-10. The secondary neutralization stage reduced the Fe, Al, and Cr content to below detection limit. Almost 85% of the Cu co-precipitated at a grade of 18.2% Cu. This product would be combined with the Cu concentrate prior to transport to a Cu smelter.

Table 13-10: Results of Secondary Neutralization

Sample & Quant.	Assay Units	Feed Solution	Final Filtrate	Residue	Precip. %
Ni	mg/L, g/t	9,360	9,870	30,100	1.1
Co	mg/L, g/t	201	215	434	0.7
Cu	mg/L, g/t	723	17.2	182,000	84.5
Zn	mg/L, g/t	18	17	684	12.2
Fe	mg/L, g/t	23.7	<0.2	7,080	99.0
Mg	mg/L, g/t	2,700	2,960	3,790	0.5
Al	mg/L, g/t	24.6	<0.2	7,120	95.2
Cr	mg/L, g/t	<0.2	<0.2	<20	22.5
Mn	mg/L, g/t	31.6	33.1	19.8	0.2
Ca	mg/L, g/t	573	715	114,000	34.2
Na	mg/L, g/t	17	21	32	0.5

Cu precipitation, shake-out-tests, and mixed hydroxide precipitation (MHP) tests were ongoing at the time the design basis was frozen for the PEA. The hydrometallurgical test program completed to date provided a proof of concept for the leaching and neutralization stages and produced low overall metal losses to the leach residue and neutralization tailings. Also, the leach liquor after the secondary neutralization stage contained low concentrations of deleterious elements. While remaining impurities such as manganese (Mn), Zn, and calcium (Ca) may be co-loaded with Ni and Co, subsequent impurity removal stages should work well given the much lower concentrations compared to leach liquors from other mineral sources such as laterites.

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 COG 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 Associates Limited. The estimate is based on assay data from drill programs completed by Kennecott and Talon between 2008 and 2020. The Tamarack North Project mineralization consists of three distinct geological domains as previously discussed in Section 7.2.5 of this report, 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 1.6.87.0 (Datamine).

14.2 Drill Hole Data

A total of 297 diamond drill holes were provided by Talon containing 41,689 assay intervals having a total core length of 129,147 m. All drill hole data was provided as of June 1, 2020.

The 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.

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 14-1.

Table 14-1: Default Grades for Absent Data

Metal	Default Value	# Records Set to Default
Ni	0.001%	152
Cu	0.001%	331
Co	0.001%	130
Pt	0.001 ppm	5,308
Pd	0.001 ppm	1,024
Au	0.001 ppm	1,278
S	0.005%	200

It is the QP's opinion that the drill hole database is of suitable quality to support the Mineral Resource estimate in this Technical Report.

14.3 Geological Interpretation

14.3.1 Mineralization

Mineral domain models were interpreted from drill hole grade and lithology data for the SMSU, MSU, and 138 zones using a 0.4% Ni cut-off. New drilling was available in the MSU and 138 domains. No new holes were drilled in the SMSU, but model limits were increased to 5169600N, and the domain modeling COG was changed from 0.83% NiEq to 0.4% Ni, resulting in the Upper SMSU being significantly extended up to the top of bedrock surface. No changes to the lower portion of the SMSU were required, as there were only minor discontinuous samples above the modeling cut-off that were outside of the previous 2018 model. The MSU and 138 domains were updated to account for the new infill drilling and the 138 was also adjusted for the 0.4% Ni modeling cut-off. Figure 14-1 and Figure 14-2 provide an overview of the updated 2020 mineral domain models and extended model limits.

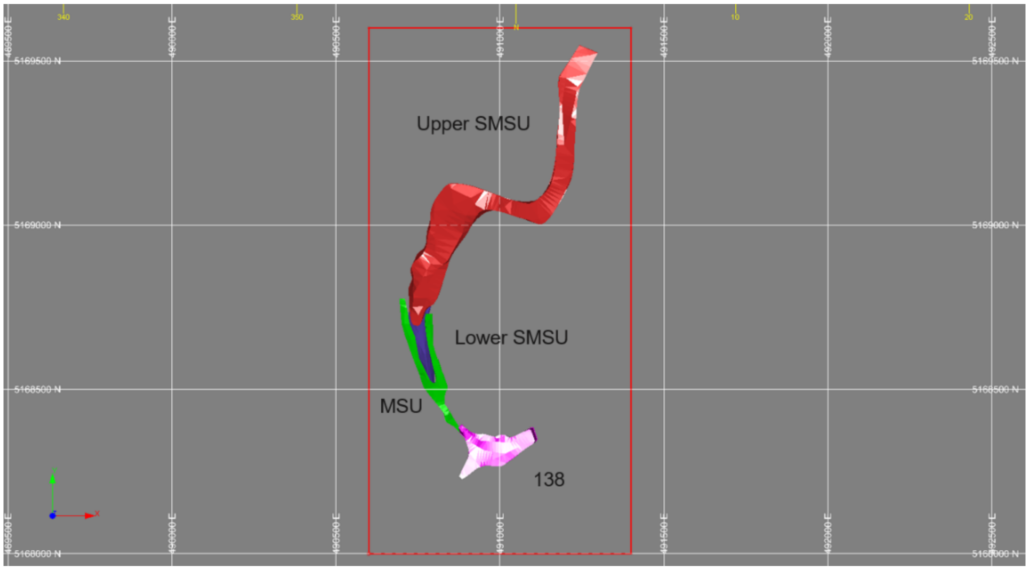


Figure 14-1: Plan View of Mineral Domains Tamarack North Project (red outline indicates model boundary limits)

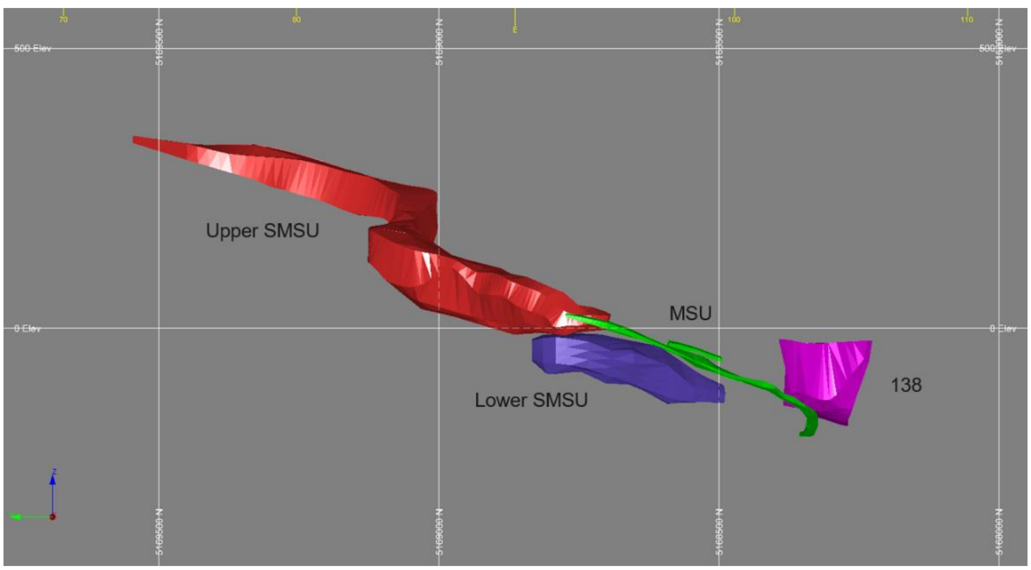


Figure 14-2: Oblique View of Mineral Domains Tamarack North Project (Facing N-W)

All mineral domain wireframe volumes were constructed by snapping points to the drill holes on the hanging wall (HW) and FW contacts, above the %Ni modeling COG. These points, along with boundary strings, were used to construct HW and FW surfaces, which were then linked to create 3D solid volumes.

A low-grade mineral envelope was interpreted to account for minor discontinuous mineralization outside of the main mineral domains, as shown in green in Figure 14-3.

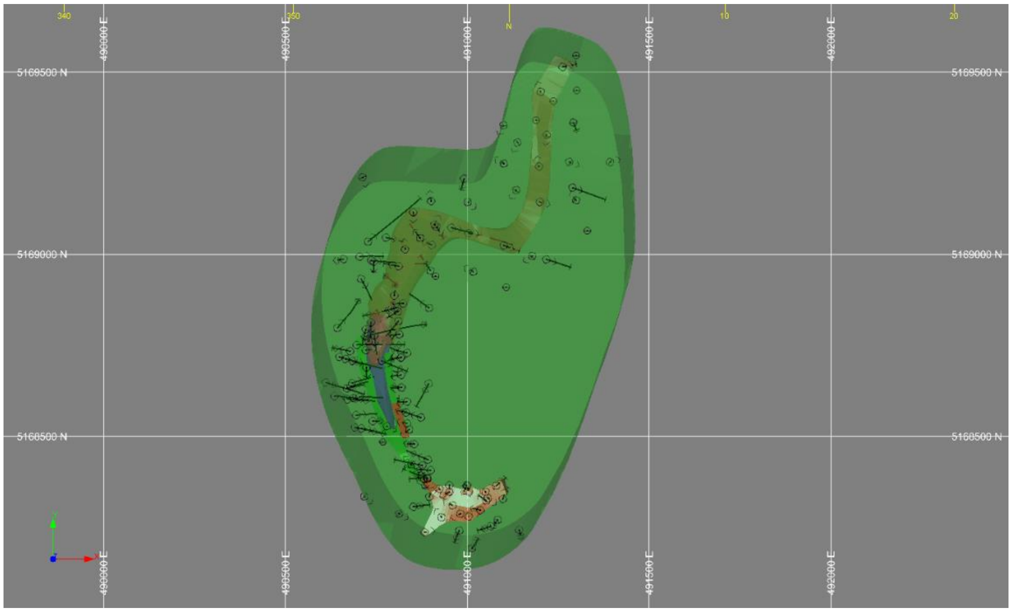


Figure 14-3: Plan View of low-grade Halo (shown in green)

Raw sample intervals were captured inside each mineral domain wireframe and verified visually to confirm the accuracy of the domain assignment process. Table 14-2 provides the sample break down by domain. It is noted that some holes intersect multiple domains.

Table 14-2: Summary of Captured Samples Tamarack North Project

Domain	Number of Samples
Upper SMSU	1,686
Lower SMSU	828
Total SMSU	**2,514**
MSU	227
138 Zone	1,412
Total	**4,153**

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 14-3 provides a summary of the descriptive statistics for the raw sample populations captured from within each mineral domain.

Table 14-3: Descriptive Statistics of the Tamarack North Project Sample Population

Domain	Field	Samples	Minimum	Maximum	Mean	Standard Deviation	Skewness	Coefficient of Variation
Upper SMSU	Ni (%)	1,686	0.06	7.00	0.59	0.69	3.24	1.15
Upper SMSU	Cu (%)	1,686	0.00	2.40	0.34	0.39	2.06	1.15
Upper SMSU	Co (%)	1,686	0.00	0.17	0.02	0.02	3.23	0.81
Upper SMSU	Pt (ppm)	1,686	0.00	1.35	0.14	0.18	2.66	1.31
Upper SMSU	Pd (ppm)	1,686	0.00	0.72	0.08	0.10	2.49	1.26
Upper SMSU	Au (ppm)	1,685	0.00	0.57	0.07	0.08	1.78	1.09
Lower SMSU	Ni (%)	828	0.12	5.06	1.68	1.28	0.68	0.76
Lower SMSU	Cu (%)	828	0.01	2.98	0.94	0.59	0.7	0.63
Lower SMSU	Co (ppm)	828	0.01	0.13	0.04	0.03	0.72	0.71
Lower SMSU	Pt (ppm)	828	0.01	5.41	0.58	0.41	2.95	0.72
Lower SMSU	Pd (ppm)	828	0.00	1.51	0.35	0.19	1.24	0.54
Lower SMSU	Au (ppm)	828	0.00	1.27	0.25	0.17	1.18	0.66
MSU	Ni (%)	227	0.02	10.15	5.33	2.54	-0.63	0.48
MSU	Cu (%)	227	0.01	5.75	2.30	1.09	-0.47	0.47
MSU	Co (ppm)	227	0.00	0.22	0.11	0.05	-0.47	0.49
MSU	Pt (ppm)	227	0.00	4.65	0.64	0.55	2.72	0.86
MSU	Pd (ppm)	227	0.00	1.18	0.47	0.24	-0.04	0.52
MSU	Au (ppm)	227	0.00	5.03	0.27	0.41	8.16	1.53
138	Ni (%)	1,412	0.12	10.05	0.57	0.62	7.57	1.08
138	Cu (%)	1,412	0.00	7.56	0.39	0.48	5.42	1.22
138	Co (ppm)	1,412	0.01	0.20	0.02	0.01	7.26	0.57
138	Pt (ppm)	1,412	0.00	112.00	0.18	1.66	66.78	9.14
138	Pd (ppm)	1,412	0.00	4.88	0.09	0.12	19.01	1.3
138	Au (ppm)	1,412	0.00	23.00	0.11	0.58	38.68	5.39

Figure 14-4 to Figure 14-7 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 domains, normal in the MSU and positively skewed in the 138 Zone.

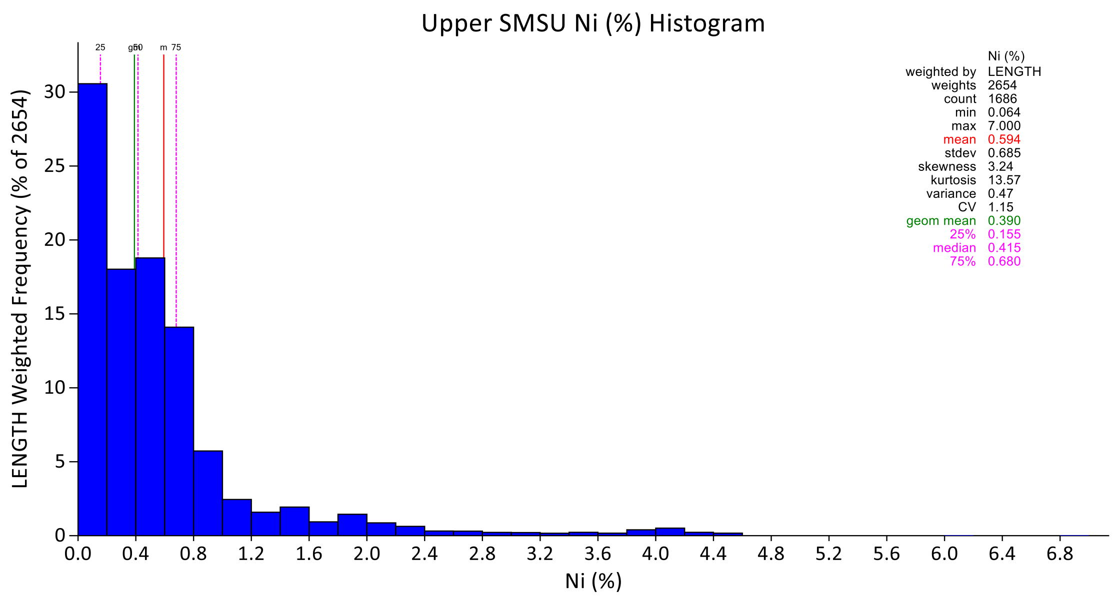


Figure 14-4: Histogram of %Ni for the Upper SMSU

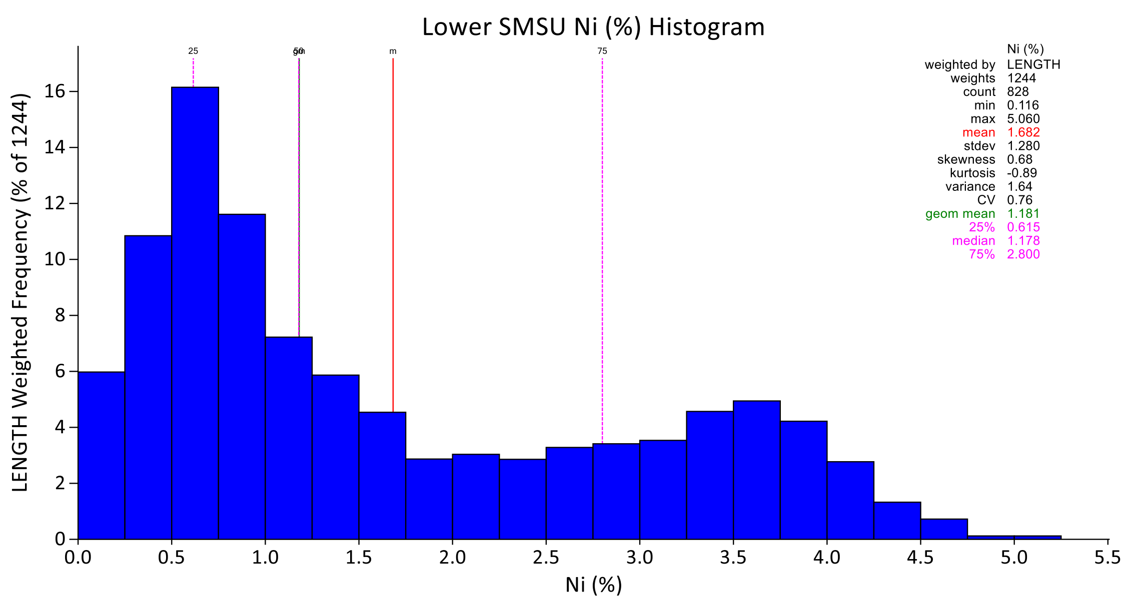


Figure 14-5: Histogram of %Ni for the Lower SMSU

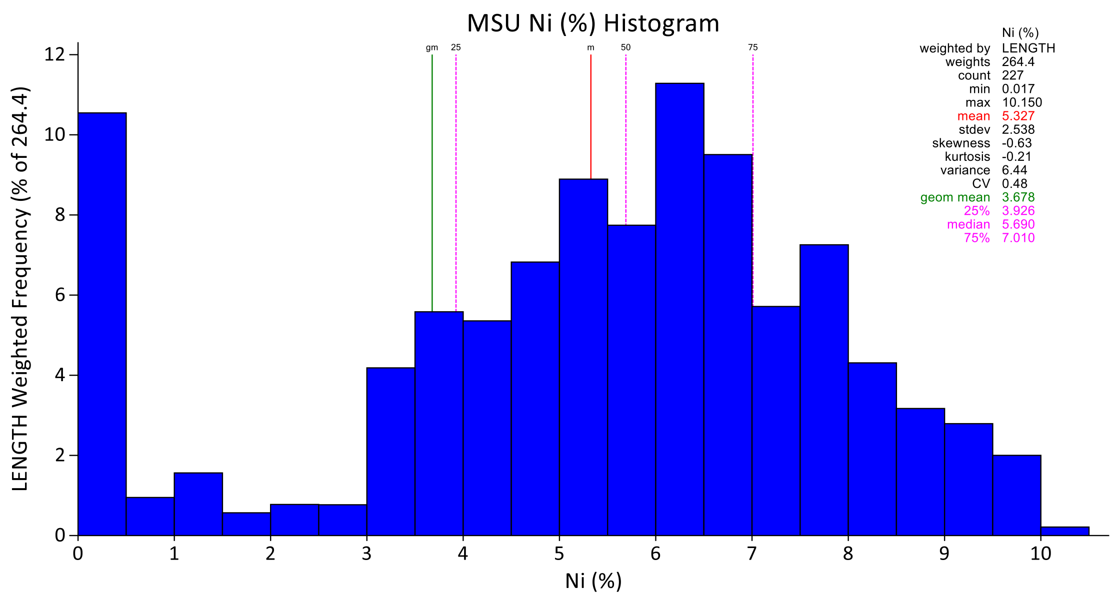


Figure 14-6: Histogram of %Ni for MSU

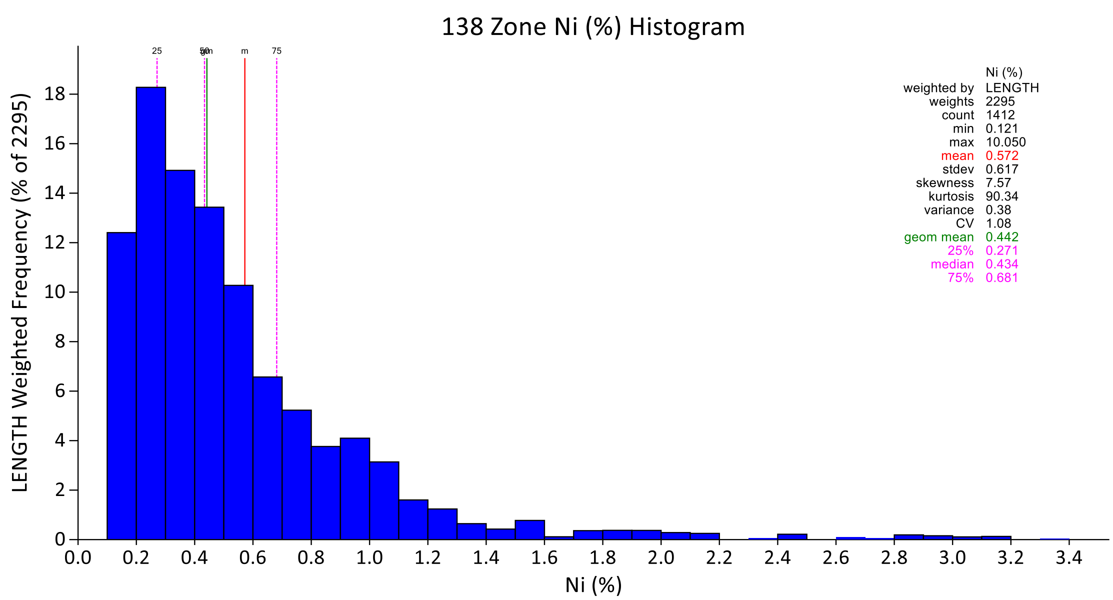


Figure 14-7: Histogram of %Ni for 138 Zone

14.4.2 Correlations

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

Table 14-4: Correlation Matrix of the Lower SMSU Domain

	Ni (%)	Cu (%)	Co (%)	Pt (ppm)	Pd (ppm)	Au (ppm)	S (%)	SG
Ni (%)	1							
Cu (%)	0.88	1						
Co (%)	0.99	0.83	1					
Pt (ppm)	-0.12	0.08	-0.17	1				
Pd (ppm)	0.03	0.2	-0.04	0.77	1			
Au (ppm)	-0.09	0.17	-0.17	0.71	0.7	1		
S (%)	0.99	0.84	1	-0.16	-0.03	-0.16	1	
SG	0.83	0.68	0.86	-0.21	-0.1	-0.29	0.86	1

Ni demonstrated a strong correlation with Cu, Co, and sulphur and a good correlation with measured density values. Ni did not demonstrate a correlation with PGM. A strong correlation between Ni, Cu, Co, and sulphur was shown across all five mineral domains. The correlation between sulphur and SG was used as the basis to calculate SG for absent intervals in the Lower SMSU domain, as described in Section 14.4.3. These correlations were also used to make assumptions that Co and SG have similar spatial continuity to Ni, as described in Section 14.6.2.

In the Upper SMSU, Ni was found to have the similar correlations with Cu, Co, sulphur but did not correlate well with SG; therefore, SG values were not calculated.

14.4.3 Specific Gravity (SG)

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

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

The QP elected to only use the SG measurements obtained from lab measurements and did not use the field measurements. Calculated SG values were substituted, where no lab measured data was available, based on polynomial regression formulas defined for each mineral domain. An SG 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 SG, Ni, and %S in the Upper SMSU, so no regression was used, and density was estimated using OK with the available lab measured data. No lab measured SG data were available for the 138. An SG was calculated for the 138 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 and then estimated using OK. The SG data from field measurements was later used to validate the model. The regression formulas used for each domain are listed below:

- SG (Lower SMSU) = 2.75988 + Sulphur (%) x 0.03808
- SG (MSU) = 2.77825 + Ni x 0.16578
- SG (138 Zone) = 2.76785 + Ni x 0.09198

Based on reasonably good correlations with the SG data, base metal grades (Ni, Cu and Co) were weighted by SG for estimation purposes for the Lower SMSU and MSU domains. New grade fields QNi, QCu, and QCo were calculated by multiplying the metal grade by measured SG, where available, and calculated SG in the absence of measured data. Grades in the Upper SMSU and 138 Zones were not weighted by SG.

X-Ok the Lower SMSU domain, and Ni and SG for the MSU domain, as shown in Figure 14-8 to Figure 14-9.

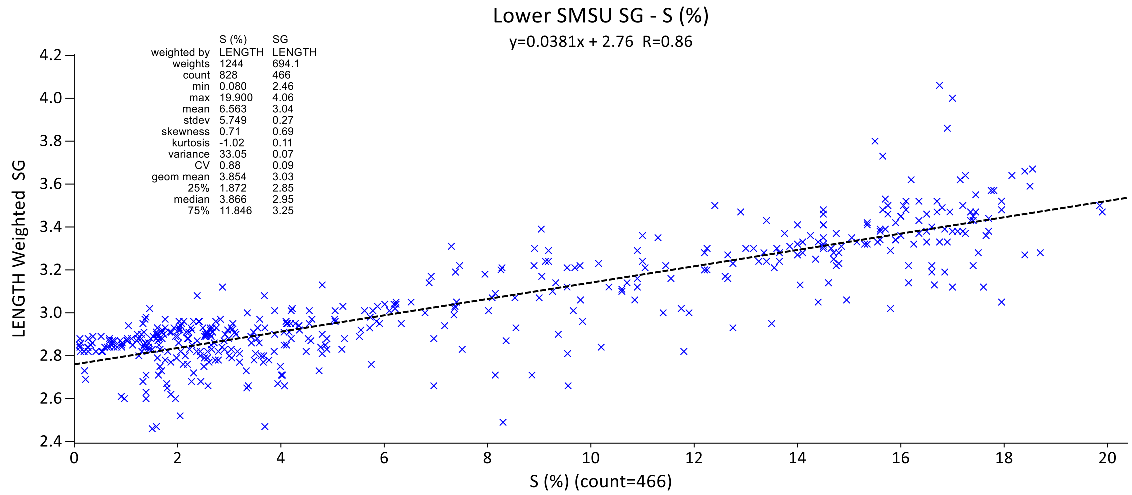


Figure 14-8: Scatter Plot of %S vs SG in the Lower SMSU

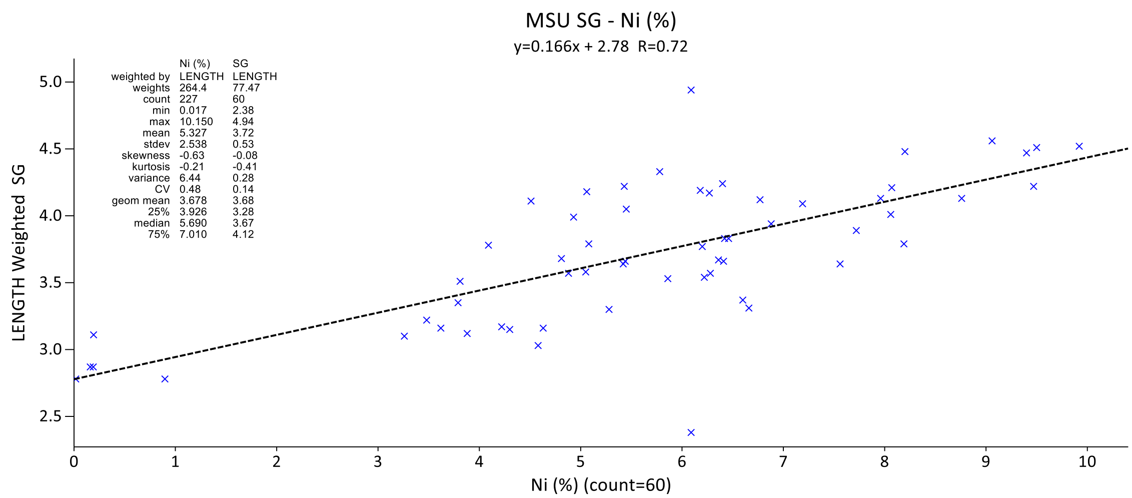


Figure 14-9: Scatter Plot of %Ni vs SG in MSU

14.4.4 Outliers

High-grade outlier data has the potential to bias block model grades if they are not handled by top cutting or, otherwise, restricting their influence through other estimation criteria. X-Y scatter plots were generated to assess the sample population for outlier values. A minor number of high-grade outliers were identified in the Pt, Pd, and Au populations of each domain and top-cut, as summarized in Table 14-5. Minor top cuts were also performed on the Ni and Cu grades in the 138 domain. 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	Ni (%)	Cu (%)	Co (%)	Pt (ppm)	Pd (ppm)	Au (ppm)
Upper SMSU	-	-	-	1.10	0.60	0.46
Lower SMSU	-	-	-	2.00	1.00	0.80
MSU	-	5.00	-	1.70	1.00	0.76
138	5.00	4.00	0.12	1.00	0.80	0.70
LG	4.00	3.00	0.12	1.00	0.60	0.40

A comparison of capped and uncapped mean grades and coefficient of variations (CVs) are summarized in Table 14-6.

Table 14-6: Comparison of Cut vs Non-Cut Mean Grades and CVs

Domain	Metal	Mean Uncapped	Mean Capped	Uncapped CV	Capped CV
Upper SMSU	Pt (ppm)	0.135	0.135	1.31	1.29
Upper SMSU	Pd (ppm)	0.080	0.080	1.26	1.25
Upper SMSU	Au (ppm)	0.073	0.073	1.09	1.08
Lower SMSU	Pt (ppm)	0.575	0.569	0.71	0.65
Lower SMSU	Pd (ppm)	0.347	0.345	0.54	0.52
Lower SMSU	Au (ppm)	0.254	0.252	0.66	0.65
MSU	Cu (%)	2.300	2.297	0.47	0.47
MSU	Pt (ppm)	0.639	0.607	0.86	0.70
MSU	Pd (ppm)	0.468	0.467	0.52	0.51
MSU	Au (ppm)	0.269	0.237	1.53	0.77
138	Ni (%)	0.570	0.560	1.07	0.89
138	Cu (%)	0.390	0.386	1.22	1.12
138	Co (%)	0.019	0.019	0.56	0.49
138	Pt (ppm)	0.180	0.153	9.13	0.83
138	Pd (ppm)	0.092	0.090	1.29	0.89
138	Au (ppm)	0.106	0.092	5.39	0.93

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. Histograms of raw sample length were generated for each domain to determine the most common (modal) sample length used at the Tamarack North Project which was the basis used to determine the composite length.

Samples captured within the Upper SMSU 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. 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 length of the sample data remained unchanged. The mean composite grades were found to be unchanged from the cut composite grades and there was no loss of sample length during the process, as summarized in Table 14-7.

Table 14-7: Comparison of Composite vs Raw Sample Lengths

Domain	Raw Sample Length	Composite Sample Length
Upper SMSU	2,653.83	2,653.83
Lower SMSU	1,245.98	1,245.98
MSU	264.35	264.35
138	2,334.11	2,334.11
LG	29,405.32	29,405.32

14.6 Resource Estimation

14.6.1 Unfolding

All mineral domains, with the exception of the LG domain, were unfolded for the purpose of grade estimation. The "Unfold" process within Datamine RM™ was used to transform the composite sample data from Cartesian coordinates into an unfolded coordinate system (UCS), as defined by the geometry of the FW and HW contacts of each mineral domain model. This transformation essentially removes bends, pinches, and swells, 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 FW (white) and HW (red) contacts of each mineral domain were constructed and tagged in cross-section view, as shown in the Upper SMSU example in Figure 14-10. 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 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-DDR of the zone and the Z-axis is assigned to UCSC representing the along strike direction of the zone.

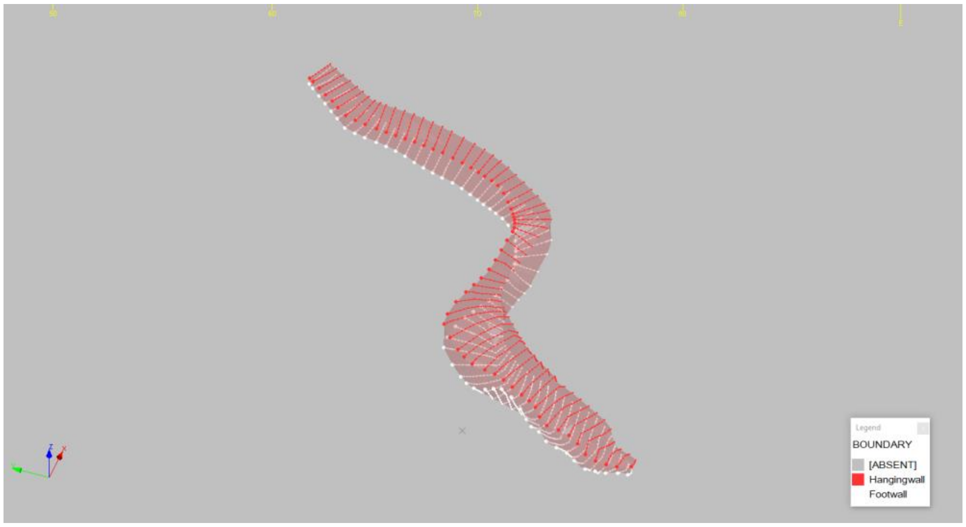


Figure 14-10: Example of Unfold Strings for the Upper SMSU Domain (facing N-E)

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 14-11 for the Upper SMSU domain.

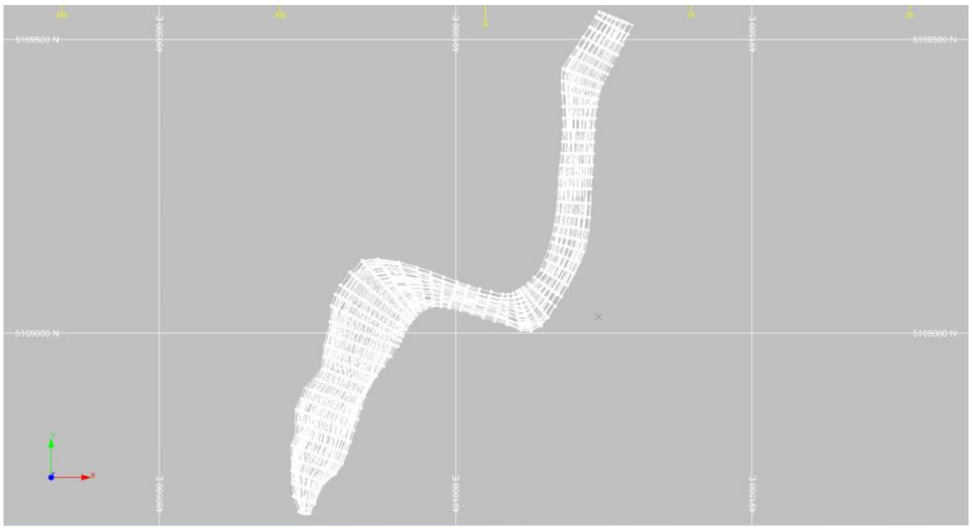


Figure 14-11: Example of Quadrilateral Strings for the Upper SMSU Domain

Visual inspection of the Nearest Neighbor (NN) models confirmed that the unfolding process had worked as expected for all zones and all of the samples were confirmed to be properly unfolded and used during the estimation process.

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 14-8. Variograms were not generated for the MSU domain due to insufficient data across the width of the deposit.

Table 14-8: 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 14-12. Summaries of all the variogram models are provided in Table 14-9.

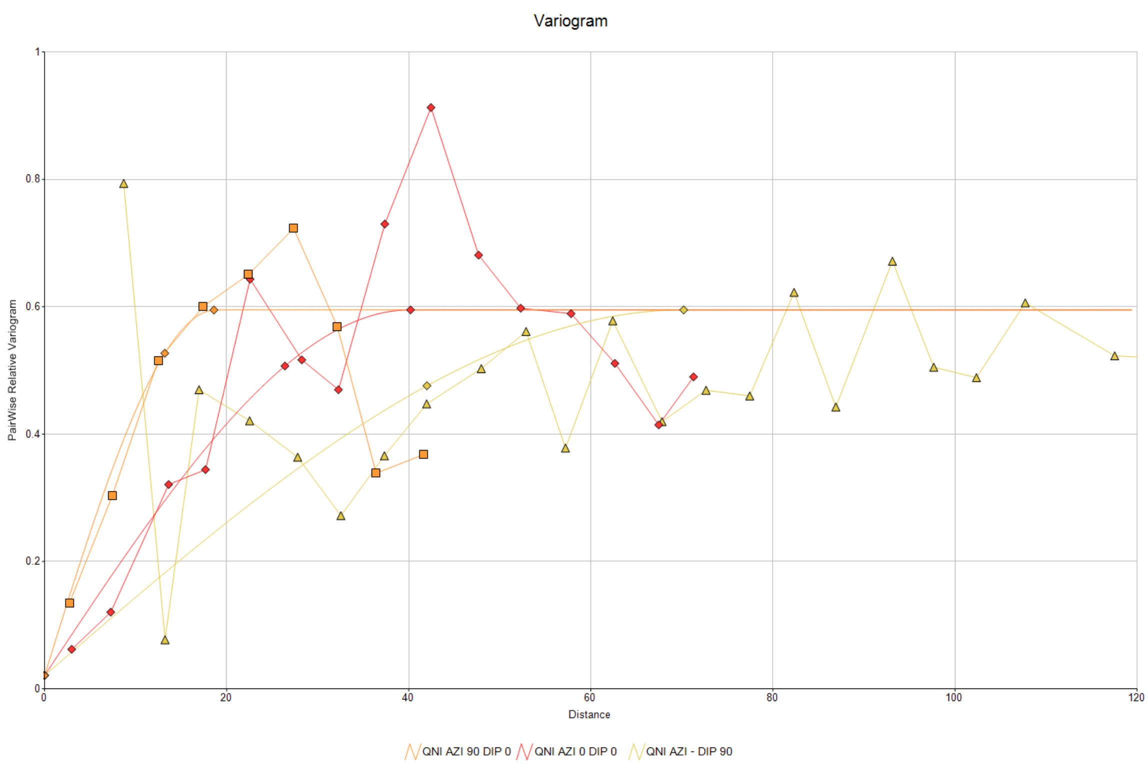


Figure 14-12: Lower SMSU %Ni Variogram Model

Table 14-9: Tamarack Grade Variogram Models (Unfolded)

Mineral Domain	Element	Nugget	1st Structure				2nd Structure			
			X-Range	Y-Range	Z-Range	Variance	X-Range	Y-Range	Z-Range	Variance
Upper SMSU	Ni	0.021	6.4	9.9	34.8	0.143	20.5	39.6	79.9	0.392
	Cu	0.053	12.0	11.3	45.1	0.227	20.0	59.7	80.1	0.296
	Pt	0.073	9.0	21.7	32.6	0.163	27.4	60.7	79.5	0.270
	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
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	37.0	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
138	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	20.0	45.7	50.2	0.470
	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	17.0	17.6	0.040	20.2	45.3	50.0	0.259

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

A 3D block model was defined as summarized in Table 14-10. Figure 14-2 displays the location of the model outlined in red.

Table 14-10: Block Model Definition

Direction	Minimum (m)	Maximum (m)	Range (m)	Block Size (m)	No. of Blocks
Easting (X)	490,600	491,400	800	5	160
Northing (Y)	5,168,000	5,169,600	1,600	5	320
Elevation (Z)	-400	200	600	5	160

All mineral domain solids were filled with blocks using the parameters described in Table 14-8. 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.

14.6.4 Estimation Methodology

OK was the interpolation method chosen to estimate grades in the Upper SMSU, Lower SMSU, and 138. This method assigns weights to the samples based on the modeled spatial continuity 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 for comparative purposes, but not chosen for resource reporting. ID^2 was used to estimate the LG domain.

Base metals (Ni, Cu, Co) were SG weighted for the Lower SMSU and MSU zones based on the observed correlations previously discussed. The 138 and Upper SMSU were not SG weighted, due to insufficient density data (138) and poor correlations with SG (Upper SMSU). Density values in the 138 were estimated from calculated values based on a regression formula, as discussed in Section 14.4.3. SG 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 g/cm^3. All domains utilized a nested search strategy, along with unfolding and top-cutting as summarized in Figure 14-11.

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 14-11: Summary of Estimation Methodology

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

Nested, anisotropic searches were performed for all domains using the modelled second structure variogram ranges for each element as the approximate search distances for each of the three axes, orthogonal to the unfolded plane of the deposit. The search parameters for all elements are summarized in Table 14-12. 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 approximately 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 a sample restriction with a maximum of six samples per hole. Un-estimated blocks were flagged in the model and then estimated without hole restrictions, along with expanded search distances.

Table 14-12: 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
Upper SMSU	10	30	60	4	12	2	4	12	4	2	12	6
Lower SMSU	10	20	35	4	12	2	4	12	4	2	12	6
MSU	4	10	20	4	12	2	4	12	3	4	12	6
138 Zone	15	30	60	4	12	2	4	12	4	2	12	6
LG	40	40	20	4	12	2	4	12	4	2	12	6

14.7 Mineral Resource Classification

Resource categories were assigned to broad regions of the block model based on the confidence of the estimates as they related to the geological understanding, continuity of mineralization relative to the style of mineralization, data quality, and drill hole density. A combination of drill hole density and the search volume used to estimate the grade of the block was used as an additional 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 was classified as “Inferred Mineral Resource”. No Measured Mineral Resources were outlined from the block model, as it is the QP’s opinion that the drill spacing and orientation of drilling was insufficient to adequately define the volume and extent of mineralization to meet the definition of the Measured Resource category.

Figure 14-13 to Figure 14-16 outline the mineral resource categories assigned to each mineral domain, where cyan (light blue) areas are Indicated Resources and orange areas are Inferred Resources. The 138 Zone was classified entirely as Inferred Resources due to limited metallurgical information and greater than 25 m drill spacing. Regions that were not assigned a resource category were flagged in the model as “potential” exploration target and were excluded from the 2020 MRE.

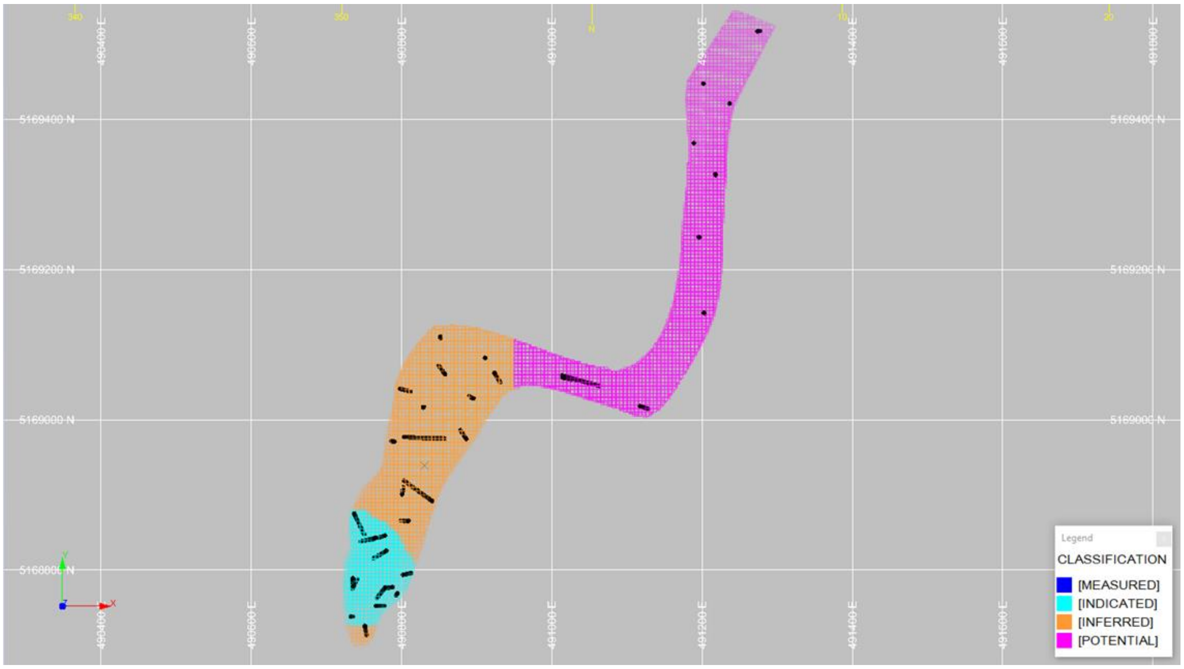


Figure 14-13: Upper SMSU Resource Classification (Plan view)

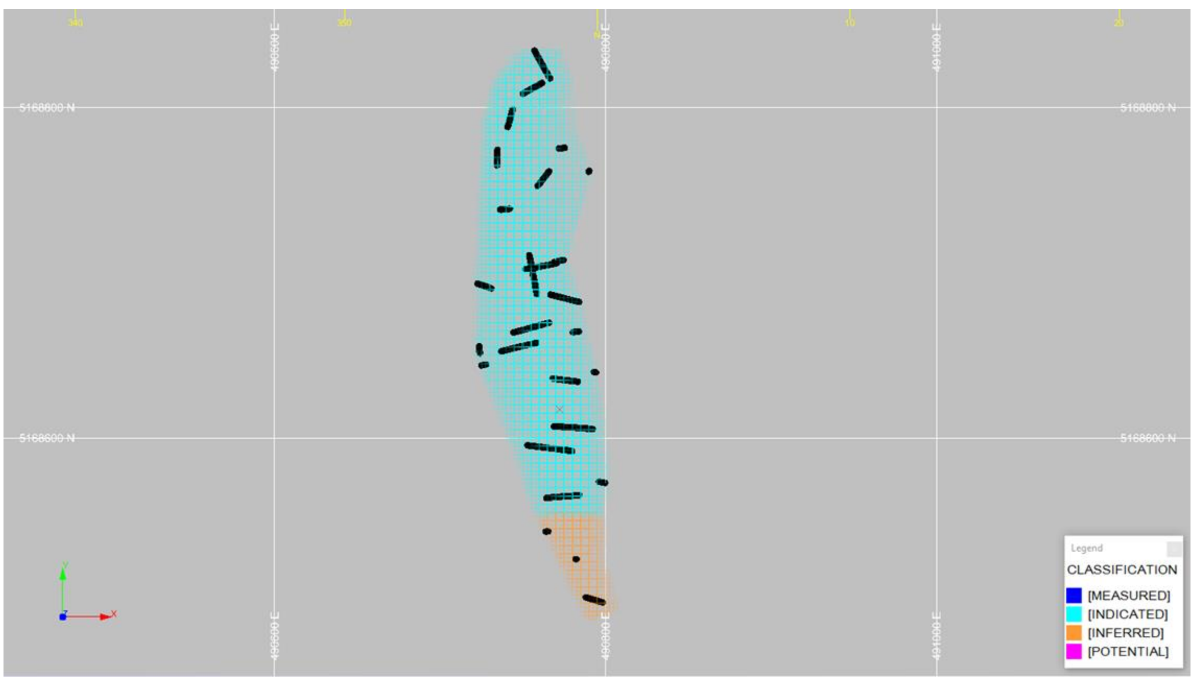


Figure 14-14: Lower SMSU Resource Classification

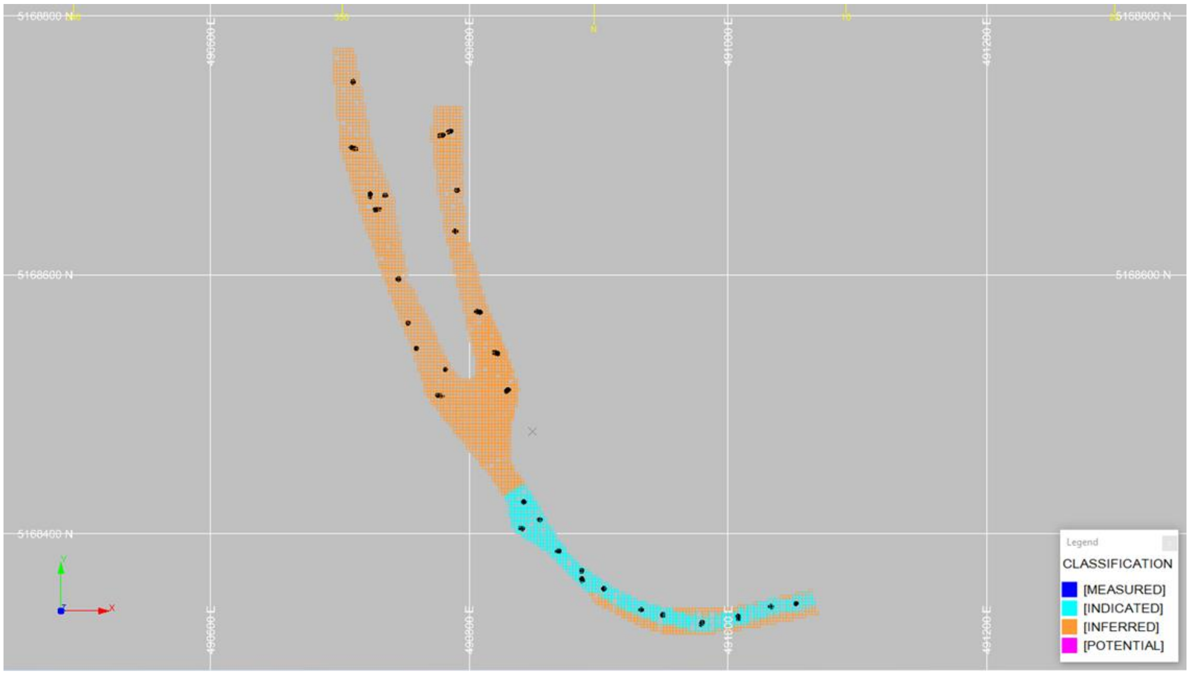


Figure 14-15: MSU Resource Classification

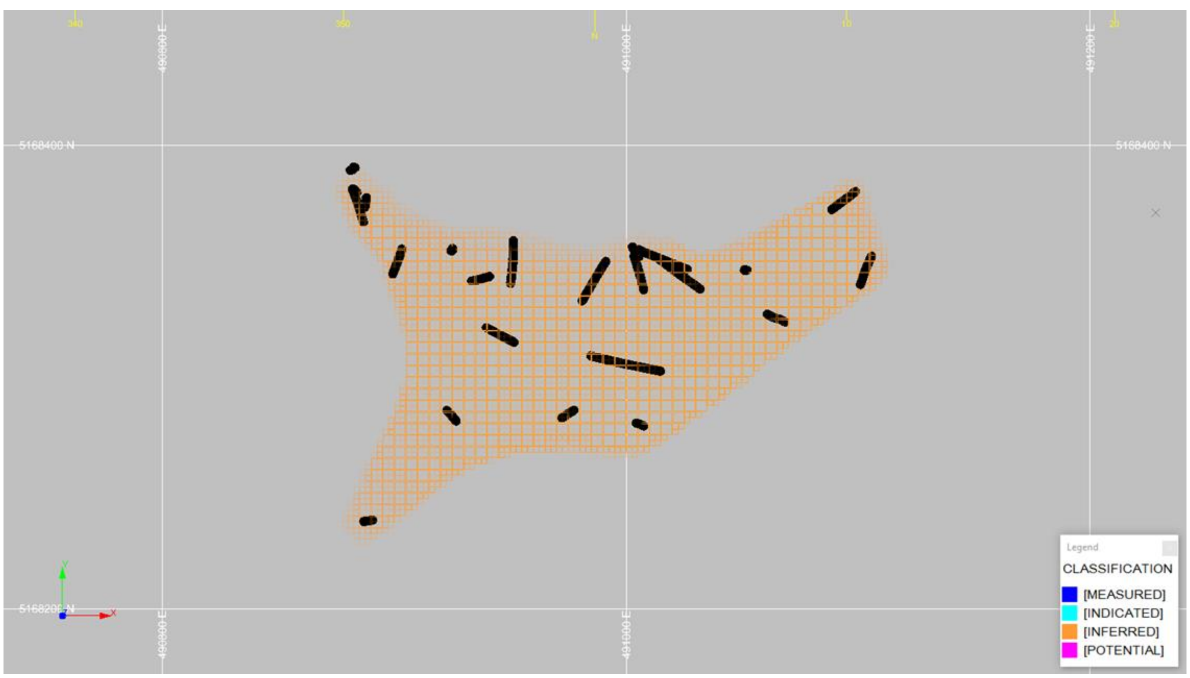


Figure 14-16: 138 Resource Classification

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 supported the categories assigned to the model. The majority of blocks in the Upper SMSU, Lower SMSU, and 138 were estimated within the first search volume, while the MSU was 54%, as listed in Table 14-13.

Table 14-13: Proportion of Model Tonnes by Search Volume

Domain	% 1st	% 2nd
Upper SMSU	89%	11%
Lower SMSU	97%	3%
MSU	54%	37%
138	99%	1%

14.8 Block Model Validation

The model validation process included a visual comparison of block model and composite grades in plan and section, along with a global comparison of mean grades and an evaluation of smoothing ratios. Block grades were visually compared to the drill hole composite data in all domains to ensure agreement and no material grade bias issues were identified, as demonstrated in Figure 14-17 to Figure 14-20.

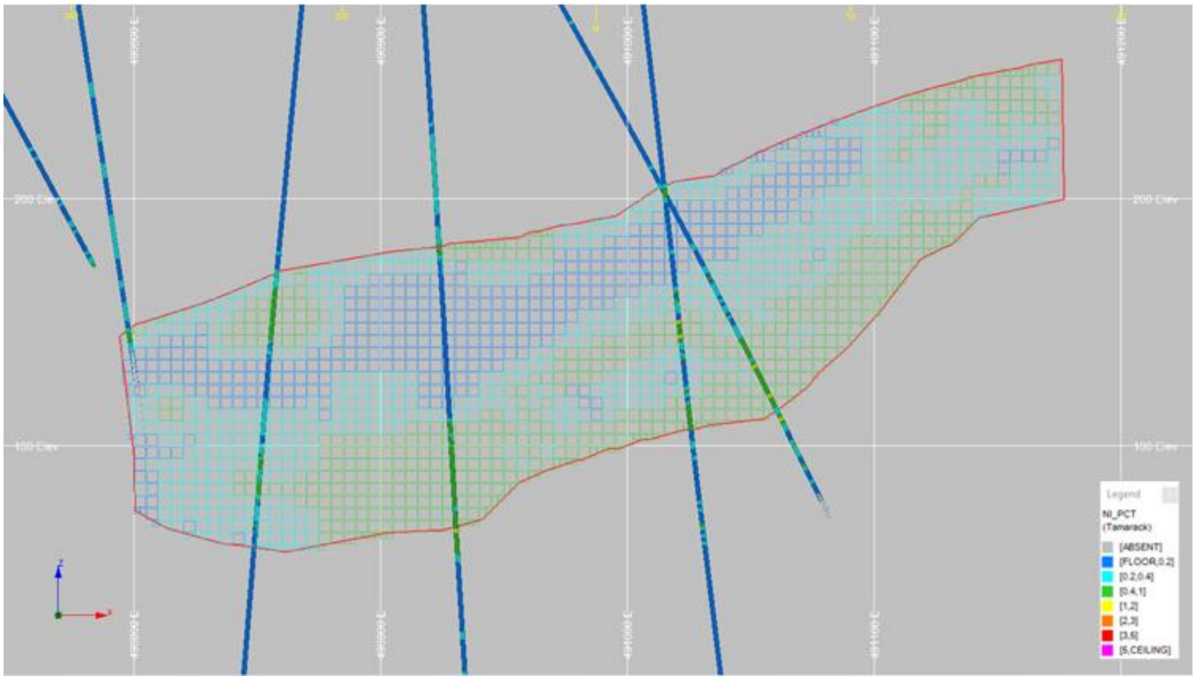


Figure 14-17: Comparison of Block Grades and Composite Samples in the Upper SMSU (5169060 N)

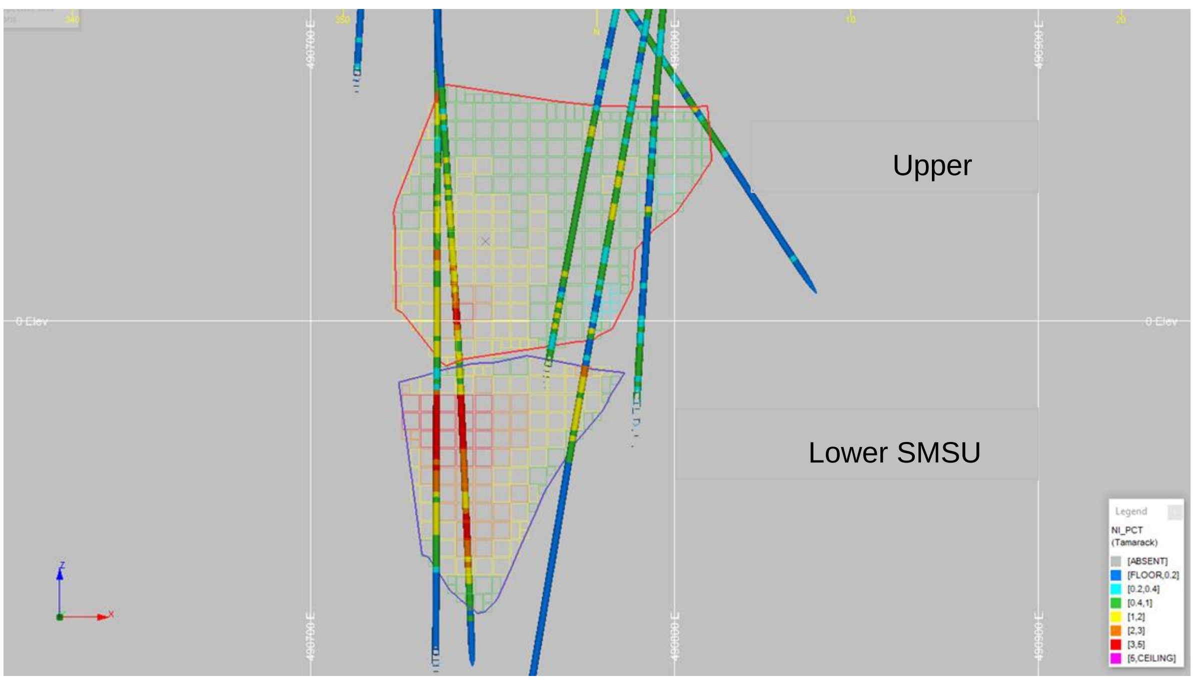


Figure 14-18: Comparison of Block Grades and Composite Samples in the Upper SMSU and Lower SMSU (5168780 N)

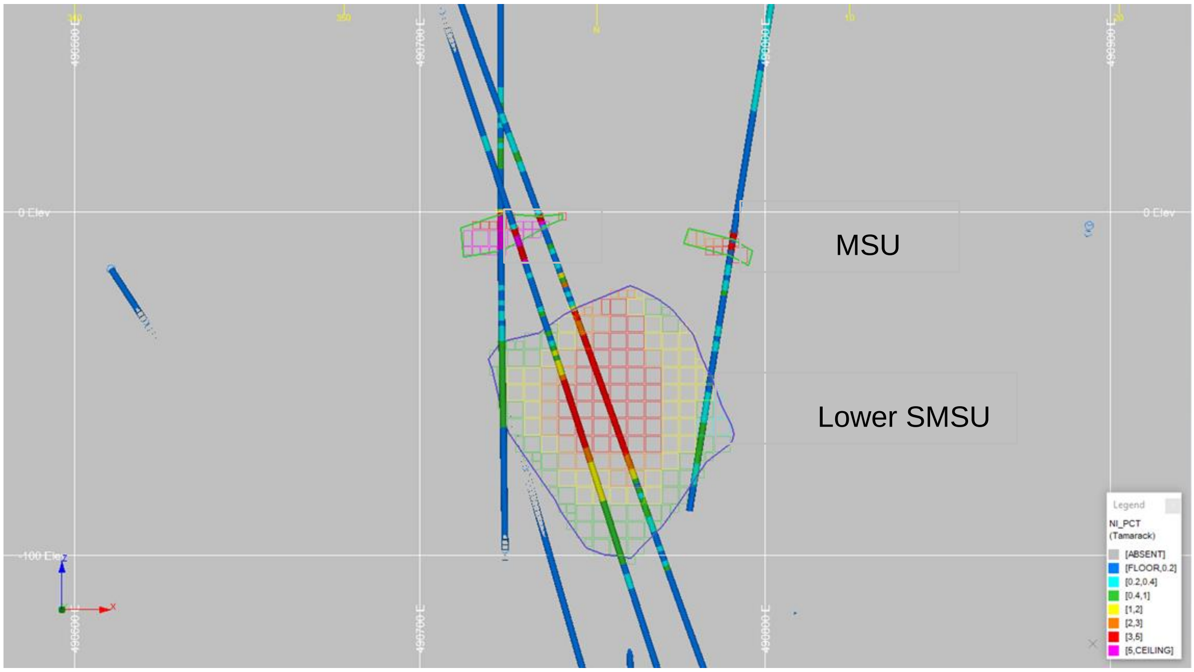


Figure 14-19: Comparison of Block Grades and Composite Samples in the MSU and Lower SMSU (5168660 N)

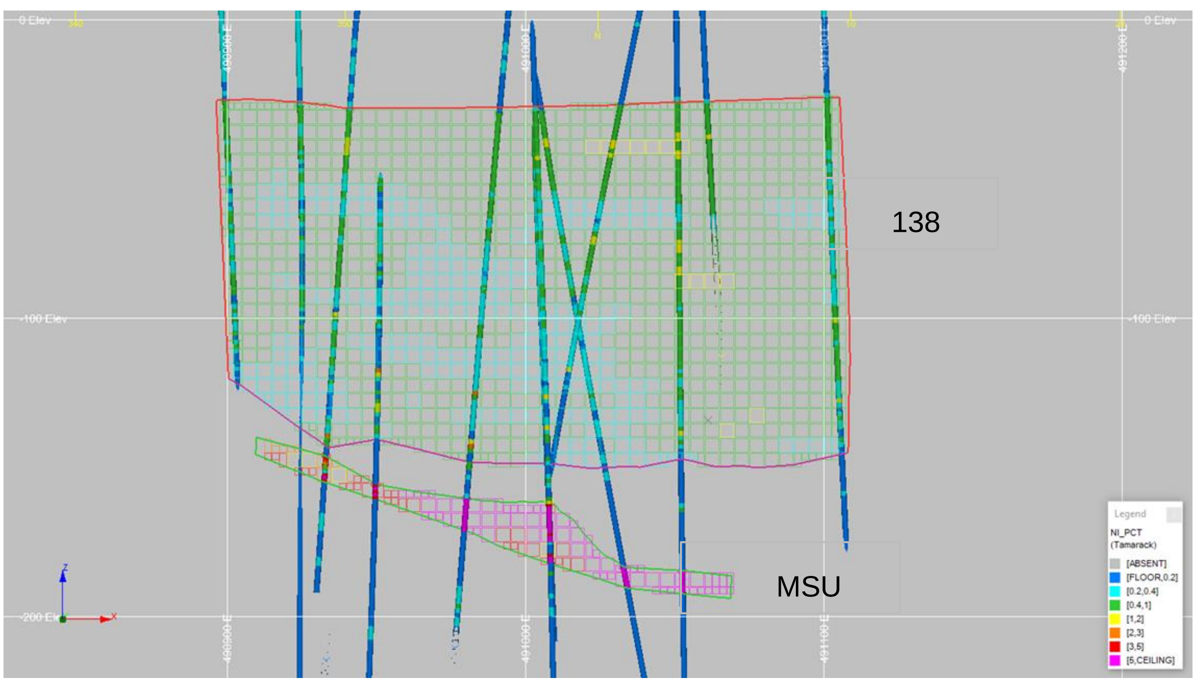


Figure 14-20: Comparison of Block Grades and Composite Samples in the MSU and 138 Domains (5168340 N)

Global statistical comparisons between the composite samples, NN estimates and the final estimates (OK or 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. The results summarized in Table 14-14 indicate that no significant grade bias was found in the block model.

Table 14-14: Validation Comparison of Global Mean Grades

Metal	Source	Upper SMSU (mean)	Lower SMSU (mean)	MSU (mean)	138 (mean)	LG (mean)
Ni (%)	Composites	0.59	1.68	5.33	0.56	0.14
	NN Model	0.50	1.93	5.94	0.69	0.12
	Final Model	0.50	1.93	5.88	0.69	0.12
Cu (%)	Composites	0.34	0.93	2.30	0.39	0.05
	NN Model	0.27	1.03	2.49	0.51	0.05
	Final Model	0.27	1.03	2.48	0.51	0.04
Co (%)	Composites	0.02	0.04	0.11	0.02	0.01
	NN Model	0.02	0.05	0.12	0.02	0.01
	Final Model	0.02	0.05	0.12	0.02	0.01
Pt (ppm)	Composites	0.14	0.57	0.61	0.15	0.03
	NN Model	0.14	0.55	0.69	0.19	0.03
	Final Model	0.14	0.54	0.68	0.18	0.03
Pd (ppm)	Composites	0.08	0.35	0.47	0.09	0.02
	NN Model	0.08	0.34	0.52	0.11	0.02
	Final Model	0.08	0.33	0.51	0.11	0.02
Au (ppm)	Composites	0.07	0.25	0.24	0.09	0.02
	NN Model	0.07	0.24	0.26	0.12	0.01
	Final Model	0.07	0.24	0.26	0.12	0.01

Smoothing (i.e. spreading, blending, averaging) of estimated grades can occur due to processes such as compositing samples, linear interpolation methods such as OK and ID, along with various other estimation parameters, including 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., 5 m^3. It is common to see higher smoothing than expected, which is an issue when reporting resources above a mining cut-off, as the overly smoothed distribution results in resource tonnages being overestimated and grades being underestimated.

Smoothing ratios were calculated for estimated model grades in the Upper SMSU and Lower SMSU and 138 domains, as shown in Table 14-15. Smoothing ratios are 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 parameters, including the variogram model, block size, and F-Function.

A smoothing ratio of 1 represents 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 could require corrective actions, as the proportion of tonnes and grade above the selective mining cut-off may not be representative of the deposit. 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

Metal	Upper SMSU	Lower SMSU	138
Ni (%)	1.21	1.13	1.02
Cu (%)	1.18	1.15	1.47
Co (%)	1.23	1.11	0.98
Pt (ppm)	1.32	1.33	0.99
Pd (ppm)	1.30	1.50	1.15
Au (ppm)	1.45	1.50	1.59

The smoothing ratio assessment indicated a low degree of smoothing within accepted tolerances with a few minor exceptions mainly seen in the precious metals. As a result, no variance correction was applied to the block model.

14.9 Cut-off Grade (COG)

The break-even reporting COG determined by Talon for this MRE was 0.5% Ni. Table 14-16 lists the long-term- metal prices and recovery assumptions used for the Ni cut-off calculation that were determined by Talon. Talon excluded Co, Pt, Pd, and Au as payable metals for the purpose of calculating the COG.

Table 14-16: Summary of Metal Price and Recovery Assumptions

Metal	Recovery	Price ($US)
Ni	82%	$8.00 / lb
Cu	70%	$3.00 / lb
Co	50%	$25.00 / lb
Pt	50%	$1,000 / oz
Pd	50%	$1,000 / oz
Au	50%	$1,300 / oz

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 costs (OPEX) based on the previous PEA Technical Report were estimated for bulk underground mining as summarized in Table 14-17 and appear to be within industry norms.

Table 14-17: Summary of Mining Cost Assumptions

OPEX	(US$/tonne)
Mining	$22.25
Processing	$13.63
G&A	$4.70
TOTAL	**$40.58**

14.10 Assessment of Mining Continuity

Grade shells were created from the block model to evaluate mining continuity at the 0.5% Ni COG. The grade shells demonstrated good continuity at the break-even COG as shown in Figure 14-21. The grade shells outline blocks above the COG and do not consider any mining constraints or modifying factors and were not used to constrain the mineral resource estimate.

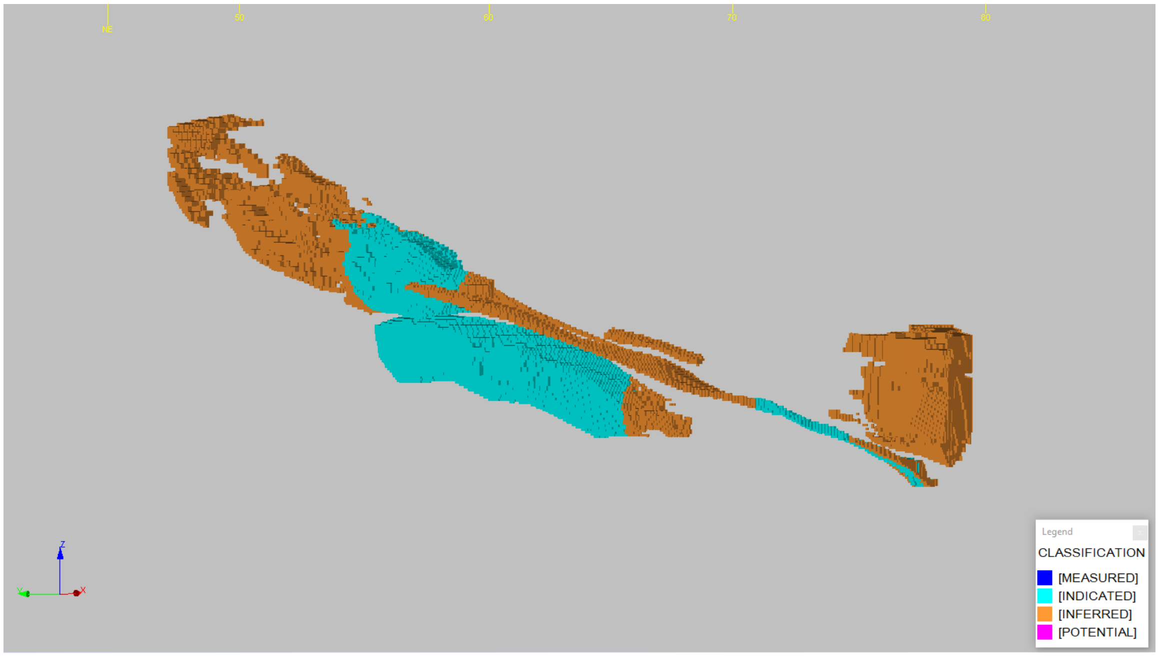


Figure 14-21: Grade Shell at 0.5% Ni Demonstrating Good Mining Continuity

14.11 Mineral Resource Statements

The Mineral Resource estimate for the Tamarack North Project has been estimated in conformity with November 2019 CIM "Estimation of Mineral Resource and Mineral Reserves Best Practice" 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.

This Mineral Resource Estimate was completed by Brian Thomas, P.Geo., an independent QP, as defined in NI 43-101. The effective date is January 8, 2021.

The Mineral Resources estimate is reported at a 0.5% Ni cut-off as summarized in Table 14-18.

Table 14-18: Mineral Resource Estimate for the Tamarack North Project, Effective Date January 8, 2021

Domain	Classification	%Ni Cut-Off	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)	Au (g/t)	NiEq (%)
Upper SMSU	Indicated Resource	0.5	1,462	1.32	0.78	0.04	0.17	0.11	0.11	1.81
Lower SMSU	Indicated Resource	0.5	2,340	2.08	1.10	0.05	0.55	0.34	0.25	2.87
MSU	Indicated Resource	0.5	124	5.72	2.36	0.12	0.60	0.46	0.23	7.23
Total	**Indicated Resource**	0.5	**3,926**	**1.91**	**1.02**	**0.05**	**0.41**	**0.26**	**0.20**	**2.62**
Upper SMSU	Inferred Resource	0.5	2,652	0.76	0.47	0.02	0.25	0.14	0.12	1.10
Lower SMSU	Inferred Resource	0.5	115	0.86	0.51	0.02	0.57	0.36	0.24	1.34
MSU	Inferred Resource	0.5	443	5.93	2.52	0.12	0.70	0.52	0.26	7.53
138	Inferred Resource	0.5	3,953	0.82	0.63	0.02	0.21	0.12	0.14	1.21
Total	**Inferred Resource**	0.5	**7,163**	**1.11**	**0.68**	**0.03**	**0.26**	**0.16**	**0.14**	**1.57**

- All resources reported at a 0.5% Ni cut-off.
- No modifying factors have been applied to the estimates.
- Tonnage estimates are rounded to the nearest 1,000 tonnes.
- Metallurgical recovery factored into the reporting cut-off.
- Where used in this Mineral Resource estimate, NiEq% = Ni%+ Cu% x $3.00/$8.00 + Co% x $25.00/$8.00 + Pt [g/t]/31.103 x $1,000/$8.00/22.04 + Pd [g/t]/31.103 x $1000/$8.00/22.04 + Au [g/t]/31.103 x $1,300/$8.00/22.04 No adjustments were made for recovery or payability in the calculation of NiEq.

Table 14-19 summarizes the changes from the 2018 reported resource estimates for tonnage and Ni and Cu.

Table 14-19: Comparison of 2018 and 2020 Mineral Resource Estimates

Domain	Classification	2018			2020			Difference		
		Tonnes (000)	Ni (%)	Cu (%)	Tonnes (000)	Ni (%)	Cu (%)	Tonnes (000)	Ni (%)	Cu (%)
Total	Indicated	3,639	1.83	0.99	3,926	1.91	1.02	287	0.08	0.03
Total	Inferred	4,382	1.58	0.92	7,163	1.11	0.68	2,781	-0.47	-0.24

The difference between the mineral resource estimate from 2018 reflects the change of the reporting COG from 0.83% NiEq to 0.5% Ni, as well as the increase of project limits to 5,169,600 N that resulted in a large volume increase in the Upper SMSU and 138 domains. Indicated resources increased marginally due to the conversion of some of the MSU from Inferred to Indicated Resource as a result of the 2020 winter drill hole program that provided more definition in the down-plunge region.

14.12 Risk Assessment

The QP 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. Portions of the resource are located 200 m to 600 m below designated wetlands, but this is not expected to affect future permitting.

The mineral resource estimate may be materially impacted by the following:

- Changes in the break-even COG, as a result of changes in mining costs, processing recoveries, or metal prices;
- Changes in geological knowledge/interpretation, as a result of new exploration data.

15 MINERAL RESERVE ESTIMATE

Not applicable.

16 MINING

The underground mine plan and design criteria are the same for each of the Ni Concentrate and Ni Sulphate Scenarios. The focus of this study was placed on evaluating opportunities that would achieve any or all of the following objectives:

1. Expand the mineable portion of the resource;
2. Accelerate time to production;
3. Reduce capital and operating costs;
4. Adopt processes that are aligned with Talon's objective of delivering Green Nickel™.

The following highlights the major design changes that have been incorporated into this PEA, and they are further described in this section:

	March 2020 PEA	February 2021 PEA
Primary Access	Shaft	Decline from Surface
Development	Drill / Blast	Road Header
Longhole Stope Sizes	7.5m W x 15m H x 15m L	15m W x 25m H x 30m L
Drift and Fill Size	3.0m W x 3.0m H	6.5m W x 5.0m H
Drift and Fill Excavation	Drill / Blast	Road Header
Mobile Equipment	Leased (OPEX)	Purchased (CAPEX/sustaining)
Material Handling	Hoisting (skips)	Vertical Conveyor
Main Infrastructure	Underground	Surface

16.1 Mining Methods

The Tamarack North deposit will be mined using both bulk and selective underground mining methods. The underground mine design framework and execution strategy is the same for both the Ni Concentrate and Ni Sulphate Scenarios.

Long hole mining has been selected as the primary mining method for all areas with the exception of a portion of the MSU, which is better suited to drift-and-fill due to the shape of the deposit.

The mine layout and different mining areas are shown in Figure 16-1 below.

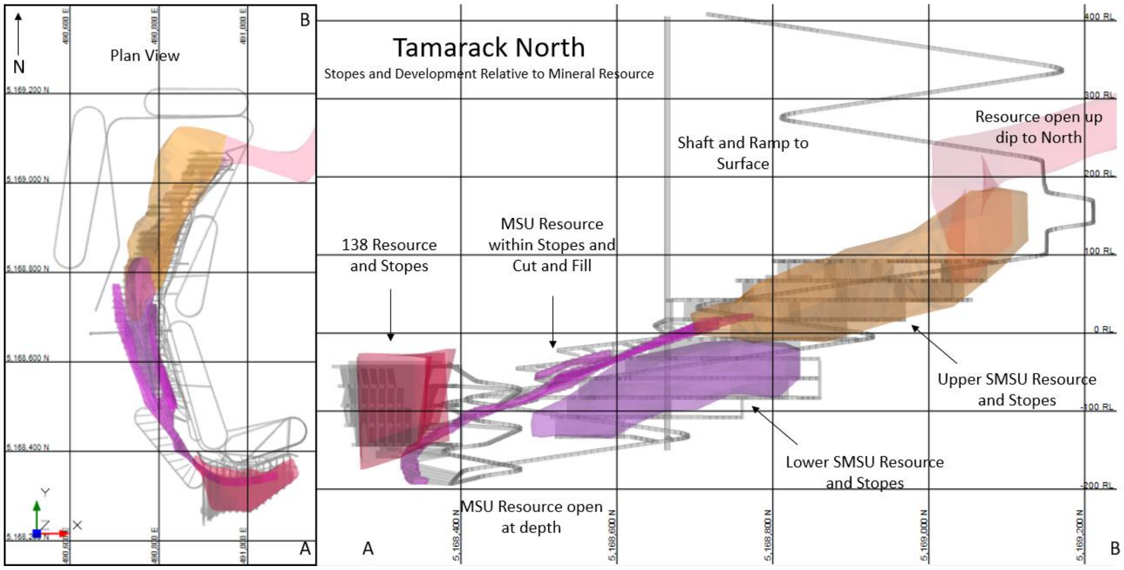


Figure 16-1: Mine Development and Production Areas

16.2 Bulk Mining

The primary mining method for each zone is transverse long hole open stoping with cemented paste backfill. Stopes are orientated E/W in the Upper SMSU and Lower SMSU and N/S in the 138 deposit. Figure 16-2 shows a typical layout for a primary-secondary transverse long hole open stopping method with paste backfill. The typical stope size is 15 m wide by 25 m high.

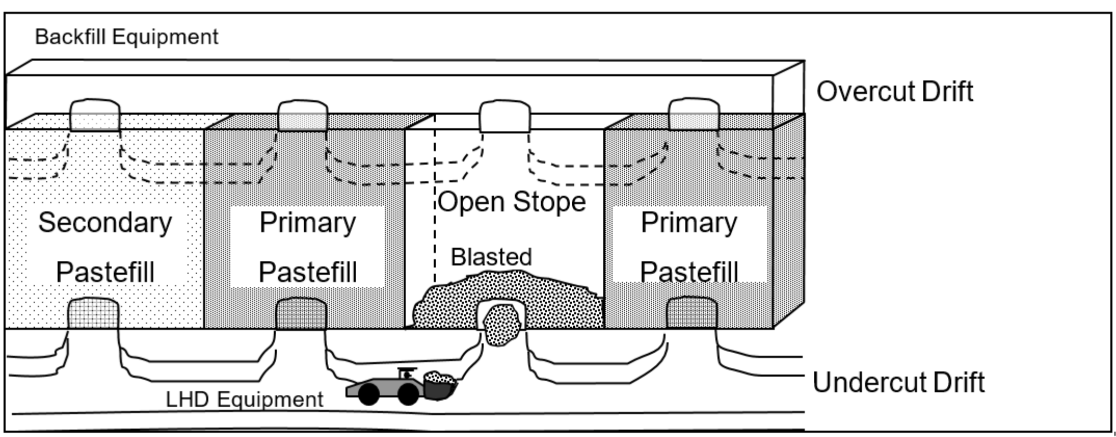


Figure 16-2: Long Section View of Primary-Secondary Transverse Long Hole Open Stoping Method with Paste Backfill

Long hole stopes will be mined in a bottom-up, primary/secondary sequence, with primary stopes designed four stope widths apart, and generally adhere to the following design basis:

- Accessed from a FW drive from waste cross cuts driven with a road header on 15 m spacing;
- Both upper and lower sill drives are driven with a road header from the FW drives, to the extents of the deposit using a road header. In the case where lower stopes have been mined and filled, the upper sill drift remaining from the bottom stope is utilized as the lower sill drift in subsequent levels;
- Stopes will be mined from HW to FW and generally be no longer than 30 m;
- Stopes will be mined in a “primary-secondary” sequence from the bottom levels and advancing upwards, trending up-dip along the deposit, as shown in Figure 16-3;

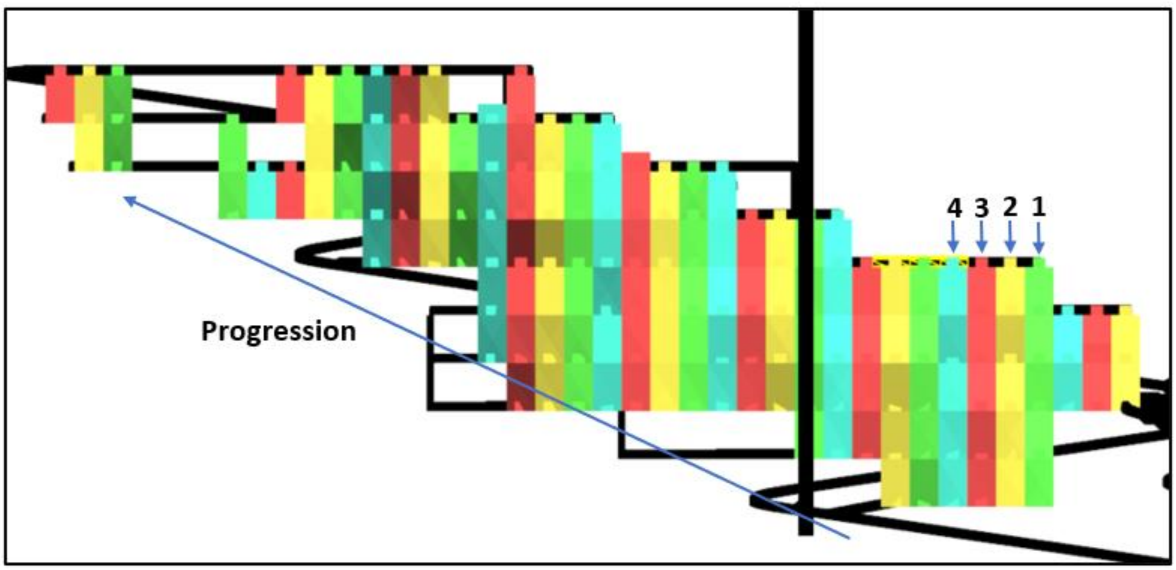


Figure 16-3: SMSU Stope Sequence

- A vertical slot will be driven from the top sill drive to the full width of the stope on the HW side. The slot is required to allow space for blast swell muck from the initial blast. Subsequent blasts will blast material into the open void toward the HW;
- After each blast, a remote-controlled load-haul-dump (LHD) will muck out the blasted material from the lower sill drift. Mineralized material will then be either loaded into trucks or transported directly to a central ore pass by LHD;

- Once the stope is fully mucked out, a fill fence is constructed at the FW contact of the stope in the lower sill drive. Cemented paste backfill, in addition to development waste rock, where available, is placed into the stope from the top sill;
- Once the paste backfill is cured the stope directly above it can be mined;
- Once three stopes are mined vertically in a column and the top stope in that series is cured, a secondary stope can be mined adjacent to the primary stope as shown in Figure 16-4;

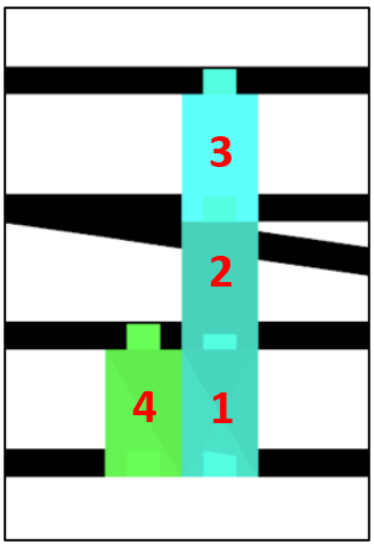


Figure 16-4: Secondary Stope Lag

- Similarly, once three stopes in a column are mined and cured from the secondary stopes, the tertiary stopes can be mined;
- Once the secondary stopes are mined out and backfilled, the stope above the initial primary can then be mined. This pattern continues throughout the orebody, advancing in a vertical chevron style pattern.

16.3 Drift and Fill Mining

Drift and fill mining will be used in the MSU as it requires a high degree of selectivity in order to minimize dilution and because the narrow shape isn't favorable for bulk mining. Drift and fill mining will be completed using a road header. Standard cuts will be 6.5 m wide by 5.0 m with cuts along the contact of the deposit being cut to the profile; this will allow full extraction of the high-grade areas, while minimizing dilution. Drift and fill mining will be progressed as follows:

- Unlike bulk stoping, drift and fill access for the MSU will be on the W side of the deposit in order to maximize recovery;
- Drift and fill mining will be completed on 5.0 m level spacing;
- Access to each level will be a single 4.5 m wide by 5.0 m high cross cut drifts from a dedicated access/FW drive and mining;
- Once the economic mineralized material is reached, the road header will drive parallel primary/secondary drifts, while maintaining access to each subsequent drift within the mineralized material in order to minimize waste development.
- Primary and secondary stopes will alternate by the standard width of the drifting (i.e. 6.5 m), where possible;
- Once a primary stope is mined out to the full width of the deposit, a fill fence will be constructed and a blend of development waste rock and paste fill will be used to tight-fill the drift;
- Once all primary drifts are developed and the paste fill is cured, secondary drifts will be mined.

Figure 16-5 shows an example of drift-and-fill stopes in the MSU.

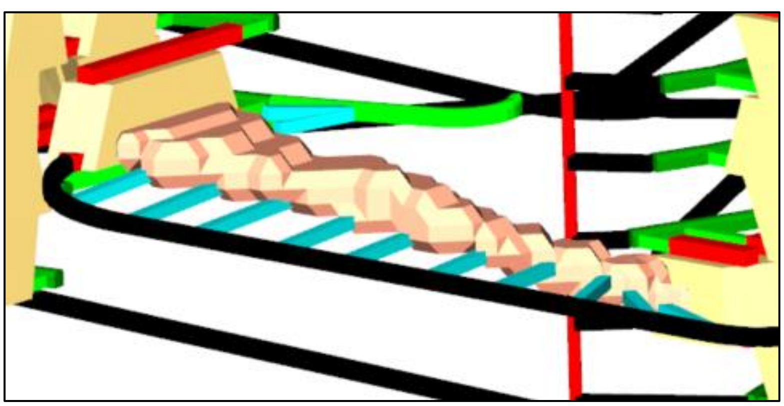


Figure 16-5: 3D View of Drift-and-Fill Stopes in the MSU

16.4 Underground Mine Plan and Schedule

16.4.1 Mine Design Criteria

Mine and operating criteria used for design and scheduling of the Tamarack deposit are given in Table 16-1.

Table 16-1: Mine Design Criteria

Parameter	Unit	Value
Main Ramp System		
Development Gradient	%	≤ 15
Excavated Width	M	6.50
Excavated Height	m	5.50
Road Bed Height	m	0.30
Concrete Road Bed	Yes/No	No
Shotcrete Thickness	m	(as needed)
Advance Rates (road header)	m/month	195 - 300
Advance Rates (traditional)	m/month	110 - 180
Development Overbreak	%	5
Re-muck Bays		
Length	m	12.00
Excavated Width	m	5.00
Excavated Height	m	5.50
Other Internal Development (typical)		
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
Advance Rates (road header)	m/month	195 - 300
Advance Rates (traditional)	m/month	110 - 180
Drift and Fill		
Excavated Width	m	6.50
Excavated Height	m	5.00
Road Bed Height	m	0.00
Concrete Road Bed	Yes/No	No
Total Advance Rates (road header)	m/month	200

Equivalent Production Rate	tpd	700-800
Transverse Long Hole Open Stoping		
Excavated Width (average)	m	15.00
Excavated Height (average)	m	25.00
Production Rate (maximum)	t/month/stope	30,000
Dilution	%	15
Recovery	%	85
Vent Shaft		
Diameter	m	6.00
Excavation Methodology		Blind Bore from Surface
Internal Raises		
Diameter	m	3.00 – 4.00
Excavation Methodology		Raise Bore/Alimak

16.5 Underground Development Mine Design

Due to shape and size, the Tamarack North deposit, with the exception of the MSU is well suited to be mined using bulk methods, such as long hole stope mining. The mine will be therefore designed across a series of levels at roughly equal intervals (25 m). Stopes will be roughly identical in size and consistently spaced. Levels will be joined by an internal ramp that connects to the main decline from surface, allowing easy access and egress from the mine.

Design of the Tamarack underground development was completed utilizing CAD software from Deswik. The Deswik.CAD product was used to complete the design utilizing floor centerlines (polylines) with appropriate attributes that describe the resulting tunnel solid (development) to be created. This includes an attribute for dimensions (height, width, arch) that will be used for creation of that solid. Tools within the Deswik.CAD will automatically convert floor centerlines directly into tunnel solids and wall outlines (polylines). Tunnel solids were also created at an appropriate length (mainly for economic consideration) to allow for an interrogation against the geological block model. These incremental solids and associated interrogation values (metal content, density, etc.) were subsequently turned into task activities in the mine schedule. The mine schedule will then apply resource allocation, further calculations, constraints, targets, or anything to aid in completing an economic sensitivity analysis. The mine schedule was completed with Deswik software utilizing both the Deswik Scheduler product and Deswik.IS (interactive scheduler) product.

16.6 Resource Mine Design Selection

The mine plan is based on the updated 2020 resource model. Prior to performing the development design, all block models were subject to a series of optimization scenarios using Deswik Stope Optimizer (SO). The purpose was to produce practical mining shapes used in conjunction with mine development design; together, they would contribute to an economic analysis.

The final parameters and/or assumptions used for the SO runs against the Tamarack block model are summarized in Table 16-2 below.

Table 16-2: Cut-off and SO Parameters

SO Parameter	Unit	Value	Variance
Resource Model		As Supplied by Talon	
Default Density	t/m^3	2.84	
Block Density	t/m^3	Variable per each block in the resource model	
Default Dip	Degrees	90	
Default Strike	Degrees	0	
Stope Height	m	25	
Maximum Waste Fraction	%		0 to 100
Maximum Length of Shape	m	15	
Minimum Dip Angle of Shape	Degrees	70	
Maximum Dip Angle of Shape	Degrees	110	
Maximum Strike Angle of Shape	Degrees		-45 to 45
Maximum Side Length Ratio		1.5	
Optimization Field	Field	NSR	

Figure 16-6 displays SO dimensional nomenclatures.

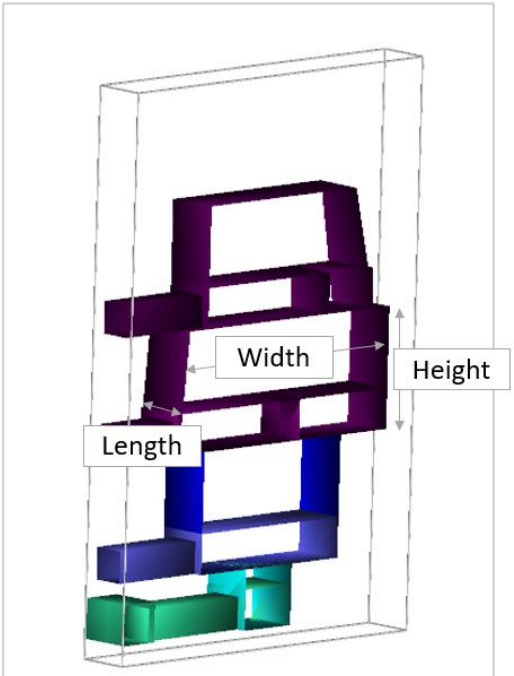


Figure 16-6: SO Framework

16.7 Cut off Value/NSR

When evaluating any resource block models, one of the largest determining factors of outcome is the COG that is selected to determine economic feasibility from an operating point of view. COGs are a minimum mineral grade associated with the value needed to "break-even" when mined. Both the mineral determined for the COG and the value of "break-even" are to be determined, prior to determining the COG. In this study, NSR was selected as the cut-off method. The NSR is the value of the concentrate delivered to the downstream user such as a smelter or hydromet facility, less the costs associated with delivering that product. These include:

- Mining Costs;
- Milling/Process Costs;
- CFTF Costs;
- G&A;
- Transportation;
- Royalties.

The NSR of each block in the resource model has been calculated separately for both the Ni Concentrate and Ni Sulphate Scenarios and differ based on payabilities and transportation costs. The stopes that are included in the mine plan are those where the average NSR of all blocks within the stope are profitable. The NSR for each block is unique as it is based on metallurgical recoveries of both Ni and Cu, shipping and handling costs by weight as well as government royalties, which vary with grade. Ni and Cu recoveries have been calculated for each block based on metallurgical recovery regression curves, which are as follows:

1. Ni Recovery (SMSU, MSU &138) > 1.00% Ni

- (1.3936 * %Ni Head + 86.88)/100*(0.0122*(%Ni Head)2 + 0.4154*%Ni Head + 94.011) /100*0.981

2. Ni Recovery (SMSU, MSU &138) < 1.00% Ni

- (24.373*ln(x)+88.472)*(0.0247x^2+0.3631x+93.871)/100

3. SMSU and MSU mineralization > 0.50% Cu

- SMSU/MSU Cu Recovery into Cu Conc = (0.8661 * % Cu Head + 95.074)/100 * 0.98 * 0.82
- SMSU/MSU Cu Recovery into Ni Conc = (0.8661 * % Cu Head + 95.074)/100 * 0.98 * 0.18

4. SMSU and MSU mineralization < 0.50% Cu

- SMSU/MSU Cu Recovery into Cu Conc = (7.979 * LN(% Cu Head) + 100.67)/100 * 0.98 * 0.82
- SMSU/MSU Cu Recovery into Ni Conc = (7.979 * LN(% Cu Head) + 100.67)/100 * 0.98 * 0.18

5. For all 138 Zone Cu Head Grades

- 138 Zone Cu Recovery into Cu Conc = (0.8661 * % Cu Head + 81.000)/100 * 0.90 * 0.82
- 138 Zone Cu Recovery into Ni Conc = (0.8661 * % Cu Head + 81.000)/100 * 0.90 * 0.18

It should be noted that only Ni and Cu revenue has been included in the NSR calculation, since the remainder of the metals (Co, Au, Pt, Pd only account for approximately 4% of total revenue). For a detailed description of assumed pay abilities, see Section 22.

The NSR per tonne was calculated using the prices below.

Table 16-3: Commodity Prices Used to Calculate NSR/tonne

	Unit	Base Case
Ni	US$/lb	$8.00
Cu	US$/lb	$3.00

16.8 Underground Development

16.8.1 Primary Access

The primary access for personnel and materials will be via a 6.5 m wide by 5.5 m high decline ramp from surface, which will be portaled next to the mill stockpile.

Past studies have considered both shaft and decline access options to mine the Tamarack North deposits. A shaft access has been favoured over a decline as it minimized surface impacts. Previous evaluation considered a large box cut that would be 337 m long x 155 m wide x 30 m deep box cut to access bedrock. This would result in the removal of 384,000 m^3 of material. The shaft, on the other hand, was sunk through a freeze wall down to bedrock and then unlined via traditional sinking methods to a depth of 540 m.

The mine access was revisited in this study since many of the key criteria had changed, resulting from a significantly larger resource, production throughput and overall LOM timeframe. This included the requirements for a larger shaft and a longer mine life, which would both require a more robust shaft design than previously considered.

Alternate excavation methods were considered to construct a portal and mine through the top 30 to 40 m of glacial till. It was determined that civil tunneling methods would offer a more appropriate means of access from an environmental impact standpoint, compared to a large box cut, therefore civil tunneling and excavation specialists from McMillan Jacobs, were retained to evaluate a practical methodology that would maximize safety and minimize surface impacts. Shaft and hoisting specialists from Stantec were retained to review the shaft and hoisting requirements to supplement this trade-off.

16.8.2 Ramp Access

The Tamarack North deposit is well suited for a decline access given that it is shallow and dips at approximately 20° along the entire strike length of the deposit. The benefits of a decline are significant and include:

- Ease of access in and out of the mine during normal operations and during an emergency;
- Early mineralized material potential prior to the completion of major infrastructure;
- Reduced overall underground exposure to personnel as infrastructure can be located to surface;
- Decouples production and primary access from a single entry-point.

McMillan Jacobs was retained to design a portal and access methodology that would mine through the surface till until sufficient bedrock coverage would be suitable for traditional lateral development methods. The primary objective was to minimize surface impacts.

For the portal construction, secant pile excavation support has been designed, as it provides the most robust and feasible form of excavation support in the widest range of conditions. Additionally, it will provide a barrier that will sufficiently mitigate most of the water seepage into the mine access. The portal excavation will be 61 m long, providing approximately 3 m of coverage over the tunnel. The width of the excavation has been designed to be 12 m, which allows sufficient clearance to install pre-support for the tunnel, prior to going underground.

Figure 16-7: Typical Secant Pile Excavation Support

The conceptual design for the tunnel assumes tunneling will be performed using sequential excavation method (SEM) in soils and drill and blast methods or grinding using a hardrock road header within rock. The excavation will utilize pre-support, consisting of a grouted pipe canopy that is constructed from the headwall inside of the box cut in advance of declining for the first 46 m of the tunnel. This will be followed by short excavation advances and then the application of support, which will include any combination of spiling, lattice girders and shotcrete.

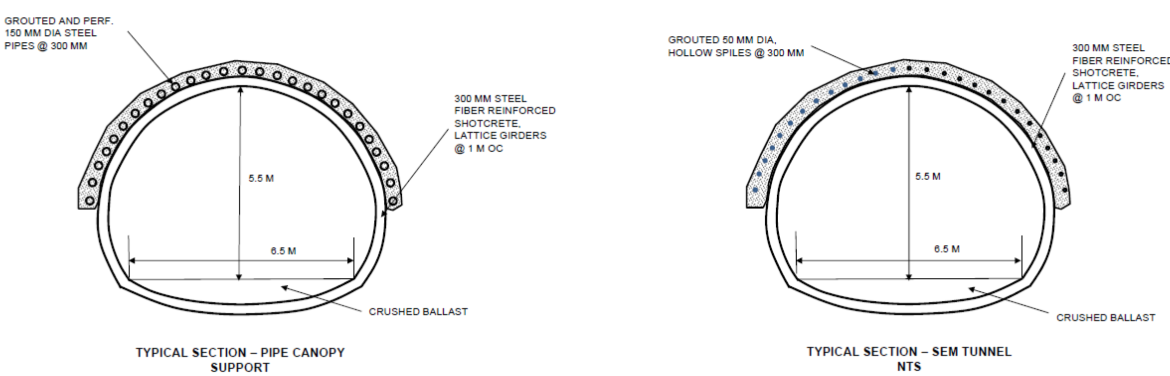


Figure 16-8: Typical Tunnel Sections

In the transition zone, where bedrock is intercepted, mixed mining methods will be used, included SEM, drill and blast and/or roadheader excavation. Once full bedrock is reached, the decline will advance using a road header and proceed with standard decline development.

The final design of the surface box cut will be 61 m long by 12 m wide by 10 m deep, which will result in the removal of only 3,700 m^3 of material, and a surface footprint of 732 m^2. This is approximately 2.5% of the footprint area and less than 1% of the material moved when compared to the previous box cut design.

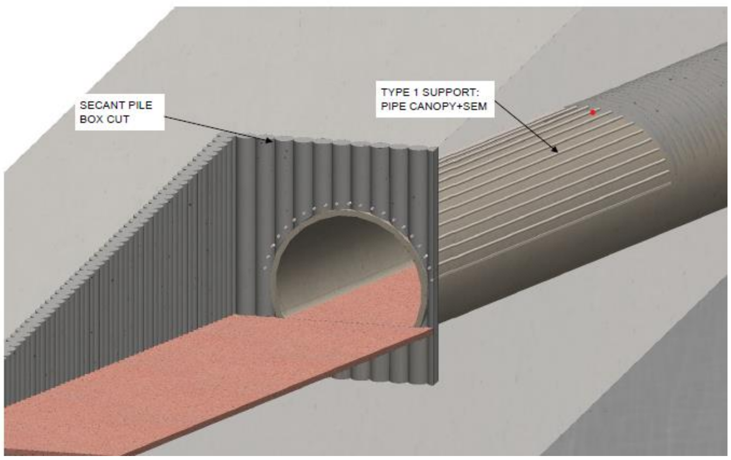


Figure 16-9: Box Cut Design

The capital cost for the portal and excavation including the development required until typical development crews could be mobilized is estimated at US$12.6M and will take approximately eight to 10 months to complete.

16.8.3 Shaft Method

The production shaft was designed to have a 6.0 m finished diameter, sunk to a depth of 528 m below collar. Unlike past mine scenarios, the production shaft will be concrete lined from collar to shaft bottom. This is due to both the functional life requirements of the shaft and the increase in size resulting from ventilation requirements. Stantec was retained to complete a shaft design, schedule and budget for a shaft that could support the LOM.

In order to assess the shaft requirements, both the main shaft and vent shaft requirements were evaluated. For a shaft scenario, the main shaft would comprise both skip hoisting and a personnel and materials cage, in addition to the main conduit for mine services and an auxiliary cage. The corresponding vent shaft for this scenario, would be a bald 5.0 m shaft that raise bored from surface once an underground access from the main shaft could be established. The shaft configuration is as shown below in Figure 16-10.

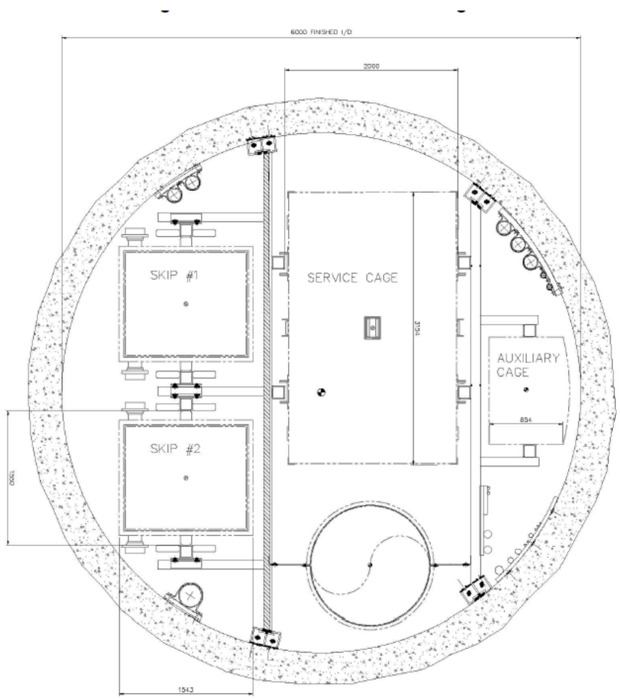


Figure 16-10: Shaft Cross Section

The key shaft and hoist plant parameters are shown below in Table 16-4.

Table 16-4: Shaft and Hoist Plant Parameters

Description	Value
Production Rate	4,000 tpd (Ultimate)
Shaft Diameter (finished)	6.0 m
Ventilation Flow	Down casting fresh air supply
Shaft Liner	Concrete, 12 inches thick
Shaft Depth	528 m
Number of Stations	3
Guide Type	Fixed
Surface Headframe	Enclosed steel structure c/w bins

The hoist requirements are outlined in Table 16-5 below:

Table 16-5: Hoist Requirements

Hoist Plant	Type	Drum Size	Payload	Rope Size
Production Hoist	Double Drum	12 foot	8 t	$1^3/_8$ inches
Service Hoist	Single Drum	12 foot	13 t	$1^3/_8$ inches
Auxiliary Hoist	Single Drum	6 foot	2 t	¾ inch

The shaft pre-sink would be through a freeze wall for the first 40 m until bedrock is reached, then through traditional shaft sinking methods to the final depth with intermittent stops to cut shaft stations.

Shaft sinking will advance at 1.0 m/day through the glacial till and a learning curve for the first 100 m of development in bedrock, after which a steady-state full development rate of 2.5 m/day will apply. It will take approximately two months to construct each of the three shaft stations. Each shaft station will be completed before advancing the shaft. Once shaft excavation is complete, the sinking gear will be replaced with staging that will be used to equip the shaft. The shaft will be equipped in a top-down sequence and will include the construction and installation of guides, steel sets, brackets, utilities, the loading pocket equipment and other permanent infrastructure around the shaft area. The total time to complete the shaft will be 38 months. At this point, off-shaft development that could access the mine zones can start. The total CAPEX for the shaft and associated infrastructure is estimated to be approximately $130M.

16.8.4 Selection of Primary Access

A decline from surface was selected as the primary mine access going forward. The key benefits to the decline include:

- Minimizes operational and safety risks associated with constructing and operating a shaft;
- Allows early access to mineralized material in the Upper SMSU extension;
- Accelerates time to first mineralized material by at least 18 months;
- Significantly lower capital cost;
- Greater operational flexibility (can employ multiple material flow streams);

- Major infrastructure (shops, services, etc) can be relocated to surface, reducing overall underground exposure and improving overall logistics;
- Greatly improves and allows early underground exploration potential.

16.8.5 Internal Development

Internal development is classified by development type and purpose. General assumptions pertaining to ground support, road bed material, etc, have been made and have been factored into the advance rate calculations and unit costs for each heading. All internal development is based on using a road header for 90% of the total development and 10% for traditional drill/blast using a jumbo. A 10% allowance has been added to development quantities to account for miscellaneous development, slashing, cut-outs, etc. The underground development heading sizes are as follows:

Table 16-6: Development Heading Properties

Development	Width	Height	Profile
Main Decline to Vent Shaft Access	6.5 m	5.5 m	Semi-Arch
Internal Decline	5.0 m	5.5 m	Semi-Arch
FW	5.0 m	5.5 m	Semi-Arch
Level Development	5.0 m	5.0 m	Semi-Arch
Waste Cross Cut	5.0 m	4.5 m	Square
Ore Cross Cut	6.5 m	5.0 m	Square
Drift and Fill Production Drives	6.5 m	5.0 m	Square

16.8.6 Vertical Development

16.8.6.1 Ore Passes

A 3.5 m ore pass will be driven in a central location in both the 138 and SMSU/MSU zones and be connected to each level by a finger raise. These ore passes will load out onto two separate conveyors that will join prior to the jaw crusher that will subsequently feed the vertical conveyor.

16.8.6.2 Vent Raises

The main exhaust will be through a vent shaft collared at surface and reaching a final depth of approximately 550 m below surface. conveying system, secondary egress and redundant services. The vent shaft will be blind bored from surface to its final depth and supported with shotcrete and bolts. Construction will be completed just-in-time with development as the decline access reaches the top access of the vent shaft. A total of two shaft accesses will be constructed: the first which will support the top half of the mine, and the second which will support the bottom half and the 138 Zone. A series of internal vent raises will be constructed to support operations in each of the mining areas. These will be typically constructed to allow for flow-through ventilation on each of the mine levels. An allowance has been made for an additional 300 m of ventilation raises that will be necessary to support the decline development that have not been detailed in the mine plan.

16.8.7 Mine Schedule and Analysis

The mine schedule was developed by linking production stopes as well and drift and fill headings in order to adhere to the geotechnical sequencing (primary/secondary sequencing) that was assumed for this study. These stopes were then linked to the development headings in order to generate a project critical path that would tie back to the decline portal. Basic productivity factors were then built into the model, including development, production and backfill rates as well as any planned delays and dependencies between tasks that are required to construct the mine. After the key criteria were programmed into the model, a series of priorities were incorporated in order to create a plan that was ideal from both a practical and financial standpoint.

16.8.7.1 Mine Production Rate

A series of production schedules were generated using the Deswik model ranging from 2,000 to 6,000 tpd then subsequently run through the financial model in order to assess the options that yielded the best financials for the project. The decision criteria for the production rate selection were as follows:

- Overall LOM (in years);
- Ability of the mine to maintain steady state production;
- Mine financials;
- Distribution of grade (mill feed).

The timing to first mineralized material should coincide with current indicators for Ni demand and an anticipated dip in future supply resulting from the expanding electric vehicle (EV) market. For this reason, it is believed that the ideal LOM for Tamarack is approximately 8-12 years. Based on the outputs from the model, the production rate selected for the Tamarack North Project is 3,600 tpd. This appeared to be the 'sweet spot' as it provided a suitable balance between production timing and the decision criteria list above. In scenarios with lower and higher production rates, the following challenges exist:

Lower Production Rates

- Production is extended longer than desired, yielding sub-optimal economics;
- Mine equipment and facilities are not optimized;
- A large amount of infrastructure is required relative to the throughput in order to mine from each of the zones to deliver a constant blend of material to the mill.

Higher Production Rates

- Exposed bottlenecks, such as mine sequence, lateral development and backfilling, lead to dips in production;
- Higher production rates require much more low-grade material to be mined early in the mine life in order to avoid dips;
- Higher production rates will inevitably require a step-change in infrastructure including the likelihood of an additional raise and/or decline to surface to support ventilation;
- Operational risk is associated with a fast ramp-up and maintaining a large number of active headings.

16.8.7.2 Pre-Production and Early Ramp-up

A pre-production period of 20 months is necessary to construct the portal and develop the decline down to the Upper SMSU where production will initiate. This allows approximately nine months to construct the portal and upper decline access down to bedrock and an additional 11 months to decline down to the Upper SMSU and upper vent access using a road header at 6.7 m/day. The decline development assumes that 10% of total development meters will be driven with a jumbo since it is expected that some areas may be unsuitable for road header development. The average decline advance rate has been de-rated to compensate for this. Production will begin once the vent shaft is connected to the main decline, producing a flow-through vent circuit. To accelerate time to first mineralized material, the vent shaft will be bored and lined from surface using blind boring so that the vent shaft is fully constructed by the time the decline and upper vent shaft access reaches it.

Once the SMSU is reached, a ramp up period of approximately two years is necessary to achieve steady-state mining of 3,600 tpd.

16.8.7.3 Mine Production Profile

The steady-state mine production rate averages approximately 3,600 tpd. The following figures outline the final mineralized material and waste production schedules.

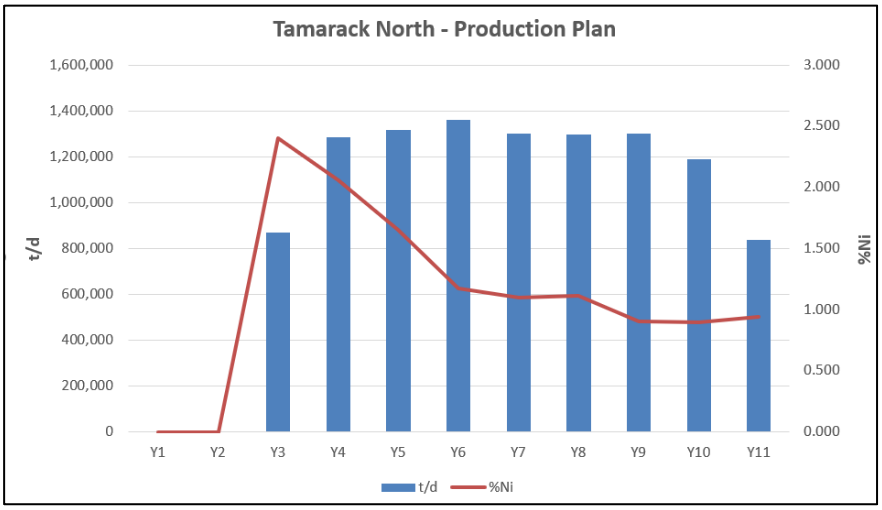


Figure 16-11: LOM Mineralized Material Production Plan

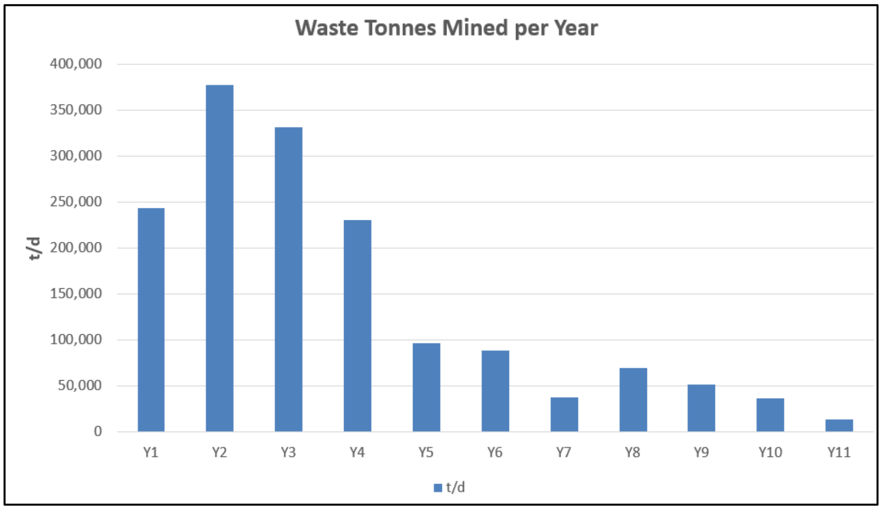


Figure 16-12: LOM Waste Production Plan

A summary of the mine plan is shown in Table 16-7 below:

Table 16-7: LOM Production Plan

Description	Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total/Average
Mineralized Material Tonnes	t	-	-	867,569	1,286,664	1,315,005	1,360,754	1,302,589	1,298,742	1,299,601	1,190,479	837,393	10,758,796
Mineralized Material tonnes per day	tpd	-	-	2,376.9	3,525.1	3,602.8	3,728.1	3,568.7	3,558.2	3,560.6	3,261.6	2,294.2	3,600 (avg)
Waste Development	t	242,948	377,426	331,531	230,514	96,189	88,786	37,495	69,789	51,807	36,287	13,410	1,576,182
Total Ni Tonnes	t	-	-	20,822	26,445	21,693	15,938	14,265	14,498	11,708	10,653	7,896	143,918
Ni Grade	%	-	-	2.40	2.06	1.65	1.17	1.10	1.12	0.90	0.90	0.94	1.34
Cu Tonnes	t	-	-	9,375	12,610	11,937	9,714	9,253	9,090	7,638	6,787	4,861	81,265
Cu Grade	%	-	-	1.08	0.98	0.91	0.71	0.71	0.70	0.59	0.57	0.58	0.76
Co Tonnes	t	-	-	488	654	538	429	393	399	334	308	229	3,773
Co Grade	%	-	-	0.056	0.051	0.041	0.032	0.030	0.031	0.026	0.026	0.027	0.035
Pt Ounces	g/t	-	-	13,102	16,404	13,640	12,238	11,165	9,559	8,740	6,578	3,806	95,232
Pt Grades	%	-	-	0.000	0.000	0.468	0.395	0.322	0.279	0.266	0.228	0.209	0.275
Pd Ounces	oz	-	-	8,792	10,589	8,654	7,423	6,853	5,952	5,235	4,004	2,375	59,877
Pd Grade	%	-	-	0.293	0.239	0.192	0.160	0.155	0.135	0.118	0.099	0.083	0.162
Au Ounces	oz	-	-	5,671	7,275	6,631	6,454	6,300	5,478	4,861	4,127	2,767	49,564
Au Grade	%	-	-	0.189	0.164	0.147	0.139	0.142	0.124	0.110	0.102	0.096	0.135

16.9 Geotechnical Analysis

16.9.1 Geotechnical Parameters

Initial geotechnical rock mass characterization was obtained from study work completed by Golder in 2008. Additional geotechnical logging, point load testing and laboratory sampling for future UCS testing as required was conducted on 30 selected holes from 2008 to 2016.

During the 2020 infill drilling program, detailed geotechnical logging, as well as a dedicated sampling program were conducted. ATV data was also collected in nine holes drilled during the 2020 infill drilling program, in addition to Optical Televiewer (OTV) data collected in four BHs in 2008 during a hydrophysical logging program.

Although geotechnical data is being collected continuously, including ATV and Full Wave Sonic BH logging, for selected infill drilling BHs, detailed statistical analyses and 3D modeling have not been completed to-date. Thus, rock mass characteristics and empirical stability assessments are based on study work completed previously.

Table 16-8 and Table 16-9 illustrate the rock mass characteristics (RMR'_{76} and Q') of the rock types that will be encountered during development and mining. The tables show that most rock types can be classified as “good” (RMR’ = 61 to 80) according to the Beniawski (1976) Rock Mass Rating (RMR) system. Serpentinized zones are classified as “poor” (RMR’ = 41 to 60) and are usually present in the contact between the igneous intrusion and the host sediment country rock. The extent and impact of this weak/altered zone will be quantified and mitigated with additional information collected in future drilling programs.

Table 16-8: Summary of RMR Data (Golder and Associates, 2008)

Lithology	RMR'_{76}				
	Average	Standard Deviation	Typical	Range	
				From	To
CGO	62	10	69	52	72
FGO/MZNO	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

Table 16-9: Summary of RMR Data (Golder and Associates, 2008)

Lithology	Q'					
	Average	Typical	Median by Discontinuity	Median by Domain	Range	
					From	To
CGO	18.7	11.1	14.5	8.2	1.7	75
FGO/MZNO	51.1	33.3	32.8	16	5	300
SED	18.4	5.6	5.6	7.8	1.3	50
SMSU	25.7	33.3	14.2	10.4	3.5	75
MSU	88.3	300	49.8	100	4.2	300
Serpentinized Zones	0.7	0.06	0.15	0.13	0.05	5.6

Detailed geotechnical logging was conducted on a total of 39 holes prior to the 2020 winter drilling program with a good distribution throughout the resource area. All infill BHs will be geotechnically logged in future study.

ATV logging had been conducted in 13 holes prior to the 2020 winter drilling program. This provides valuable information regarding orientation and in-situ conditions of small and large-scale geological structures encountered. ATV logging will be completed in selected infill holes to obtain a representative distribution of structural data throughout the resource area.

Laboratory rock property testing was conducted in 2008 and 2020. A total of 57 UCS, 25 BTS and 21 TCS tests were conducted during these two testing programs. Summary tables of all laboratory data available is illustrated in Table 16-10,Table 16-11 and Table 16-12. In addition to this data, more than 470 UCS samples have been collected and stored for future testing, as required.

Table 16-10: Summary of UCS Test Data

Lithology	UCS(MPa)				
	Average	Minimum	Maximum	Standard Deviation	Tests
CGO	99.7	42.0	146.1	25.5	18
FGO	83.9	50.3	109.8	17.0	15
MSU/MMS	96.3	61.1	142.9	32.5	8
SMSU	89.6	71.6	112.4	25.7	6
SED	183.7	136.7	222.2	34.3	8
MZNO	95.4	54.5	136.2	40.9	2
Lithology	**E(GPa)**				
	Average	**Minimum**	**Maximum**	**Standard Deviation**	**Tests**
CGO	45.0	10.8	69.7	14.3	18
FGO	30.1	11.9	46.0	10.5	15
MSU/MMS	42.4	30.0	52.9	7.8	8
SMSU	37.7	14.3	48.1	11.1	6
SED	42.4	37.2	48.1	4.1	8
MZNO	40.9	37.2	45.9	3.3	2
Lithology	**Poisson (v)**				
	Average	**Minimum**	**Maximum**	**Standard Deviation**	**Tests**
CGO	0.211	0.110	0.350	0.058	17
FGO	0.210	0.070	0.260	0.053	15
MSU/MMS	0.296	0.160	0.390	0.085	8
SMSU	0.210	0.180	0.230	0.019	6
SED	0.213	0.180	0.230	0.020	8
MZNO	0.207	0.180	0.230	0.021	2

Table 16-11: Summary of BTS Test Results

Lithology	BTS(MPa)				
	Average	Minimum	Maximum	Standard Deviation	Tests
CGO	12.3	12.3	12.3	-	1
FGO	10.3	10.0	10.5	0.3	2
MSU/MMS	6.4	4.2	9.6	1.6	15
SMSU	-	-	-	-	-
SED	17.3	16.1	18.7	1.0	4
MZNO	11.1	9.8	12.1	1.0	3

Table 16-12: Results of TCS Test Results

Lithology	TCS					
	Sigci (MPa)	mi	Sigt (MPa)	Cohesion (MPa)	Internal Friction Angle	Tests
MSU/MMS	140.32	9.193	15.26	42.02	28.87°	11
MZNO	156.59	9.366	16.71	32.07	42.23°	5

BH breakout and in-situ stress orientation data were collected as part of the ATV logging program. The orientation of the major principal stress has been determined to NNE-SSW in the majority if the BHs. A detailed analysis, which will include determining the true magnitude and orientation will be completed in future study.

In terms of large scale structures that might influence excavation stability, a possible duplex system of faults has been interpreted in the project area and regionally by aeromagnetic data. One fault has been confirmed within the 221 Zone drilling. The faults in the 221 zone was observed in the country rock metasedimentary sequence and sealed by quartz veins observed over 17 m. No faults have been observed in the Tamarack resource area, however offsets beds by small brittle fractures are observed in the country rock. The kinematics of the faults remains unknown and timing of faulting with respect to the intrusion of the TIC is also unknown. Future drilling and interpretation of areas where the structures are expected will form part of the development of a detailed structural model for the project area during the next phases.

16.9.2 Empirical Stope Stability Analysis

An empirical stability analysis on the proposed stope sizes was completed with geotechnical data currently available. The geotechnical parameters reported in the previous section were used to analyze stability of the typical stope sizes assumed for the mine design. The empirical method used for the stability analysis is the Potvin (1988) stability charts utilizing the stability number and the hydraulic radius of the back, HW, FW and vertical ends.

Separate stability analyses were conducted for the stopes in the SMSU and 138 Zones, as they have different orientations and are situated in different lithological units.

The parameters represented in Table 16-13 were used to complete the empirical stability analysis, which includes laboratory rock property testing, geotechnical logging data, joint set orientation and in-situ stress orientation from ATV and OTV data.

Table 16-13: Input Parameters for Empirical Stope Design

Input Parameter	SMSU	138 Zone (FGO)
Stope Orientation	E-W	N-S
Stope Height	24 m - 25 m	24 m - 25 m
Stope Back Width	14 m - 16 m	14 m - 16 m
Hydraulic Radius Back	14.2 – 15.7	14.2 – 15.7
Hydraulic Radius Vertical Ends	5.00 – 7.69	5.00 – 7.69
Hydraulic Radius HW	4.12 – 4.88	4.12 – 4.88
Hydraulic Radius FW	4.12 – 4.88	4.12 – 4.88
Stope Length	20 m - 30 m	20 m - 30 m
Average Depth	400 m	400 m
Sigma 1	29.7 MPa	29.7 MPa
Sigma 1 Orientation	NNE-SSW	NNE-SSW
Sigma 2	15.4 MPa	15.4 MPa
Sigma 2 Orientation	WNW-ESE	WNW-ESE
Sigma 3	11.0 MPa	11.0 MPa
Sigma 3 Orientation	OB	OB
UCS	83.8 MPa	89.6 MPa
Q'	10 - 40	10 - 100
Joint Set 1 Dip/DDR	85/163	74/55
Joint Set 2 Dip/DDR	21/227	39/164

The results of the empirical analysis for the SMSU indicate that the unsupported rock mass in the HW and the FW are generally expected to be stable, while the back and the vertical ends will generally be in the "Transition Zone", meaning some instability can be expected, however, which will either contribute to stope dilution (which is expected) or could be effectively mitigated with simple ground support. The results for the 138 Zone indicated that the vertical ends will generally be stable and the HW, FW and back rock masses are generally expected to be either "Stable" or in the "Transition Zone". Similarly, any area within transition can be mitigated with ground support, if required. Future data collection will reduce the range of the data and refine the plots.

The following activities are recommended to validate and refine the results above in future study:

- Geotechnical logging and statistical data analysis of all infill drill holes to refine the rock mass characteristics in the SMSU and 138 Zone;
- ATV (discontinuity orientation) and Full Wave Sonic (variation in rock strength) wireline logging to be conducted in specified BHs to obtain an adequate distribution of discontinuity orientation and continuous rock strength data;
- Laboratory testing of all lithologies to obtain an adequate confidence level in data (typically 80% confidence level for design) for numerical modelling;
- Conduct sufficient in-situ stress conditions (magnitude and orientation) testing, typically done by means of over-coring, to use in empirical and numerical stability analysis;
- Validate empirical design with numerical modelling, including stope stability with regards to extraction and backfill sequence, as well as regional mining induced stresses.

An empirical stope stability assessment approach is deemed sufficient at this project phase, as there is a high possibility that stope design, orientation and sequencing as well as mine layout optimization will occur during the next phases. A numerical modelling approach will be implemented during the PFS and Feasibility Study phases of the project, accompanied by in-situ stress measurement data, as this has a significant impact on stope stability.

16.10 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 could 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 2,161 m, or an average feature frequency of 1 per 216 m of length. 50% of these features were determined to be independent.

Additionally, hydraulic testing of discrete intervals using packers was 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 produce. A preliminary estimate of the inflows per independent bedrock features to a working mine was calculated to be 9.9 gpm.

The expected frequency of independent features was multiplied by the proposed development meters for the decline and mine footprint development respectively in order to calculate the groundwater inflows for each feature. The results are shown in Table 16-14 and Table 16-15 below and assume none of the water producing features are sealed.

Table 16-14: Potential Mine Water Production by Year for the Decline

Year of Mining	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Decline m	1,477.2	1,925.5	2,130.0	285.2	0.0	0.0	0.0
Cumulative Decline m	1,477.2	3,402.7	5,532.8	5,817.9	5,817.9	5,817.9	5,817.9
Estimated Number of Independent, Water Producing Features	3.4	7.9	12.8	13.5	13.5	13.5	13.5
Potential Production (from development) gpm	33.9	78.0	126.8	133.3	133.3	133.3	133.3
Potential Inflow (from decline) m^3/h	**7.7**	**17.7**	**28.8**	**30.3**	**30.3**	**30.3**	**30.3**
Year of Mining	**8**	**9**	**10**	**11**			
Total Decline m	0.0	0.0	0.0	0.0			
Cumulative Decline m	5,817.9	5,817.9	5,817.9	5,817.9			
Estimated Number of Independent, Water Producing Features	13.5	13.5	13.5	13.5			
Potential Inflow (from development) gpm	133.3	133.3	133.3	133.3			
Potential Inflow (from development) m^3/h	**30.3**	**30.3**	**30.3**	**30.3**			

Table 16-15: Potential Mine Water Production by Year for Footprint Development

Year of Mining	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Development m	0.0	1,333.1	3,550.0	5,923.0	4,655.9	2,991.0	2,971.0
Cumulative Development m	0.0	1,333.1	4,883.1	10,806.1	15,462.0	18,453.1	21,424.0
Estimated Number of Independent, Water Producing Features	0.0	3.1	11.3	25.0	35.8	42.7	49.6
Potential Inflow (from development) gpm	0.0	30.6	111.9	247.6	354.3	422.9	491.0
Potential Inflow (from development) m^3/h	**0.0**	**6.9**	**25.4**	**56.2**	**80.5**	**96.0**	**111.5**
Year of Mining	**8**	**9**	**10**	**11**			
Total Development m	2,940.9	1,230.6	2,065.1	1,267.4			
Cumulative Development m	24,364.9	25,595.5	27,660.6	28,928.0			
Estimated Number of Independent, Water Producing Features	56.4	59.2	64.0	67.0			
Potential Inflow (from development) gpm	558.4	586.6	633.9	662.9			
Potential Inflow (from development) m^3/h	**126.8**	**133.2**	**144.0**	**150.6**			

The following work/tests are recommended to refine/confirm the above results during the next stages of the project:

- Conduct additional spatially distributed hydrophysical logging;
- Conduct additional double packer testing to determine inflow expected from water producing features;
- Conduct additional spatial distributed pump testing in the shallow (till) and deep (fractured) aquifers with real time monitoring to determine the extent to which these aquifers are connected;
- Determine the influence the shallow (till) aquifer will have on groundwater inflow into the mining areas and what influence dewatering of the bedrock aquifer will have on the till aquifer;
- Compile a numerical groundwater model to integrate all data collected and refine estimated groundwater inflow into the decline and production development for the LOM.

16.11 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/day and a hydraulic conductivity of 2.4 ft/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 30.5 m) may be above 224 gpm (1,220 m^3/day). However, an average inflow of 68 gpm (374 m^3/day) is estimated for periods after dewatering effects have stabilized.

16.12 Mine Services

16.12.1 Mine Backfill System

Stopes will utilize paste backfill, mixed with development waste rock, when possible. Paste backfill has been selected as the primary fill method for long hole stoping and drift and fill stopes because of its physical properties, which provide a high degree of stability, generally allowing for increased mineral recovery and it will minimize the quantity of tailings stored on surface.

16.12.1.1 Paste Plant System Capacity

A paste plant will be constructed on surface, adjacent to the concentrator. For the Ni Concentrate and Ni Powder Scenarios, the tailings recipe will utilize two main tailings streams that are by-products of the flotation circuits in the concentrator: a LS tailings stream (NAG character to be confirmed in dynamic geochem tests), and a HS tailings stream containing 5.25% sulphur (~15% sulphides). For the Ni Sulphate Scenario, LS and HS tailings streams will also be produced, as well as tailings streams for leach residue, primary neutralization residue, and Mg residue.

For each scenario, all HS tailings, leach residue, primary neutralization residue, and Mg residue generated will be used in the paste backfill recipe, with LS tailings blended in to fill all the required voids. The impact of storing the HS tailings on surface will be higher than the LS tailings, due to the need for a lined impoundment and the socio-environmental concerns attached to storing sulphides on surface.

The steady-state system capacity for the plant will be 188 t/h (90 m^3/h). The system design is based on the following assumptions:

- The mass balances have been set to a 60% annual paste plant utilization. These flow rates are also achievable with the instantaneous production rate of all tailings;
- Mill availability is assumed to be 92%. The hourly tailings production rates incorporate this factor;
- A mineral replacement value of 90% was used;
- A solids concentration of 79.5% by mass was assumed;
- Binder addition rate was assumed to be 4%.

The system utilizations by year for the Ni Concentrate and Ni Sulphate Scenarios are shown below in Figure 16-13 and Figure 16-14 respectively.

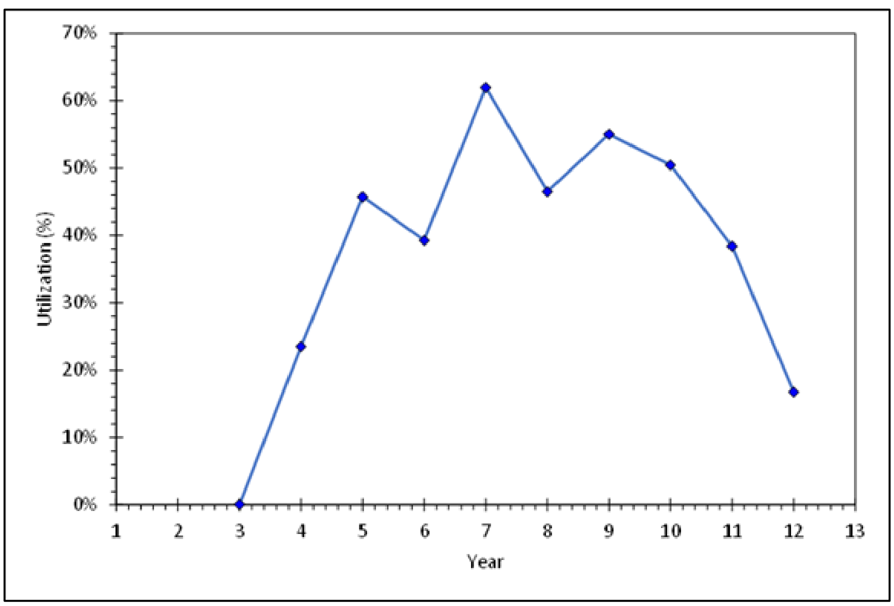


Figure 16-13: Backfill System Capacity at 90 m^3/h

16.12.1.2 Paste Delivery and Distribution

The paste distribution network is designed so that a single network can service all mine zones. Paste will be pumped from the paste plant to a pair of twin 8-inch BHs, cased with 6-inch diameter pipes located at the top of the deposit (the Upper SMSU). The paste will then gravity flow down the BHs and cascade down dip through the mine through inter-level BHs between levels.

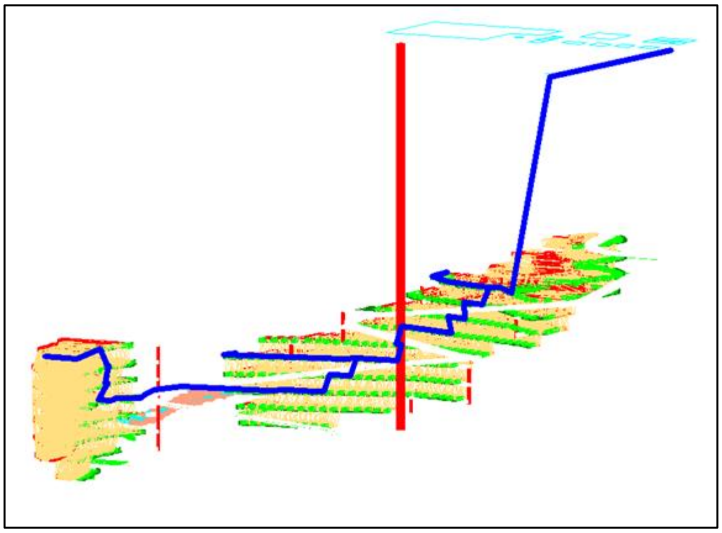


Figure 16-14: Paste Distribution System

16.12.2 Mine Service Water

Annual mine service water requirements are calculated for the LOM and are based on:

- Equipment usage;
- Dust suppression;
- General usage;
- Miscellaneous allowances.

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

- Equipment consumption is based on the manufacturers' specified water consumption multiplied by the estimated average equipment utilization per day;
- Muck pile wash down is based on four active areas;

- Dust suppression for the underground crusher and conveyor loading area are both assumed to be 5 m^3/h;
- An allowance of 10% is made for miscellaneous underground usage;
- A 5% allowance is made for leakage.

It is generally expected that the service water requirements will be fairly consistent on a day-to-day basis and no seasonal variations are expected. Based on these assumptions water steady-state consumption is expected to be achieved by year four and is estimated to be approximately 50 to 55 m^3/h (220 to 240 gpm). The estimated annual service requirements are shown in Figure 16-15 below:

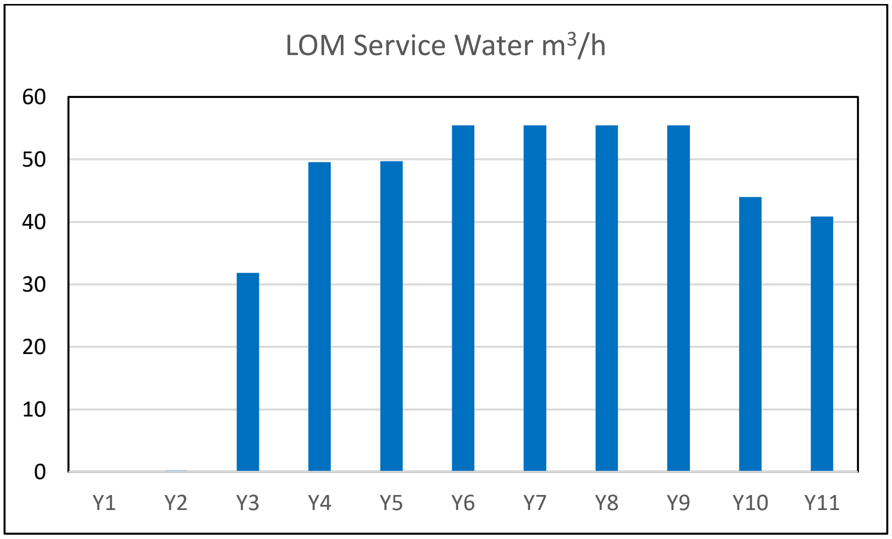


Figure 16-15: Annual Service Water Consumption

16.12.3 Underground Dewatering

Wastewater resulting from equipment usage, dust suppression and groundwater inflows will be captured in underground sumps, strategically positioned around the mine. The sumps will allow for the settling of solids, which will be periodically removed and disposed of in mined out stopes. Overflow water will be routed to a central sump located adjacent to the main pumping station, which will be located at a low point in the mine around the conveyor level. The main sump will have clean and dirty sides with the purpose of settling out solids prior to being pumped to surface. Clean water (with minimal fines) will then be pumped to surface up the ramp through intermediate pumping stations connected in series. Water will ultimately be pumped to and treated at the concentrator, then recycled for future use.

Pumping requirements are based on the combined water collected resulting from consumed service water and groundwater inflows. Tamarack can expect seasonal variations in groundwater inflows, therefore the peak seasonal inflows are assumed in calculating the pumping requirements. The total pumping requirements are based on the following assumptions:

- Groundwater inflows are based on an average inflow of 9.9 gpm per water bearing feature. An average of one water bearing feature per 216 m of drill data has been measured through past logging;
- It is assumed that groundwater inflows can be reduced by 20% by grouting;
- 6% of the total inflows will evaporate through moisture losses and report to the ventilation system;
- A contingency of 25% is added to the total groundwater inflows and 10% to equipment usage;
- Pumping calculations are based on 75% pump utilization.

The average instantaneous pumping requirements during steady state-mining are between 320 and 370 m^3/h, which accounts for peak groundwater inflows, which generally occur in mid-spring (April/May) each year. The estimated pumping requirements are shown in Figure 16-16.

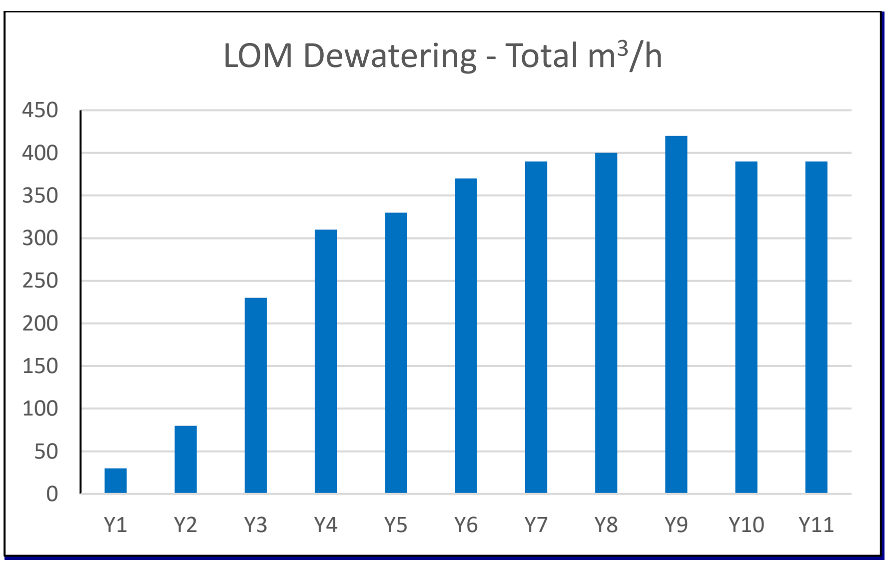


Figure 16-16: Mine Dewatering Requirements

16.12.4 Material Handling

16.12.4.1 Material Flow

Vertical conveyors are not as common as other material handling methods in underground hard rock applications, due to the functional hoisting limits, which are much less than skips. Additionally, they offer limited operational flexibility, compared to haul trucks, in that they are fixed in one location and typically cannot be expanded once in operation. Tamarack, however, is amenable to both of these criteria given that the overall strike length of the deposit is short enough that it can be easily centralized and that the deposit is relatively shallow compared to most underground mines. The vertical conveyor was then viewed as a reasonable option because it satisfied this criteria, and there is precedent in the industry with installations having been successfully used to support underground operations both in the USA and abroad.

The main benefits of a vertical conveyor when compared to other options, such as an inclined conveyor include:

- It has low relative capital costs;
- It can easily be automated;
- Maintenance is typically very low, as there are few moving parts;
- In-shaft maintenance is not needed;
- It can be almost completely isolated from people and other equipment.

A vertical conveying system was designed by Frontier Kemper Lakeshore (FKL), with support from Continental-Contitech, (the belt supplier) out of Germany. The system was designed at capacity of approximately 3,500 tpd and would be installed in the vent shaft. Two belt styles were considered: a Pocketlift® and Flexowell® belt style. The Flexowell® system was selected because it is lighter and therefore better suited for longer hoisting distances and yet still easily meets the functional requirements needed to support the underground operation. There currently exists global precedence at similar depths and throughput at other mines that have been designed and built by FKL and Continental-Contitech.

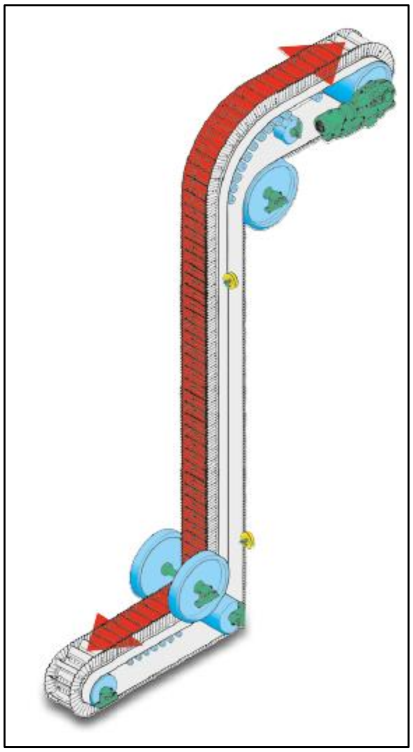


Figure 16-17: Flexowell® Vertical Conveyor

The vertical conveyor will be installed in the vent shaft. The head will sit on top of the shaft collar on surface and the tail end will be in a dedicated underground loading chamber. The belt will free-hang in the shaft and will not require any direct shaft access for maintenance. The belt is designed to a 10:1 safety factor. Mineralized material will be brought to the vertical conveyor by a central conveyor that will be fed by a central orepass for the Upper and Lower SMSU. A second conveyor will bring material from the 138 Zone, fed by a central orepass in the 138 Zone that will tie into the main conveyor. Prior to loading onto the vertical conveyor, mineralized material will be crushed to -6 inch by a jaw crusher, and tramp steel will be removed by a magnet/picking belt arrangement.

The vertical conveyor was ultimately selected as the preferred method due to the following reasons:

- Net present cost (NPC): An economic trade-off was completed comparing truck haulage to the vertical conveyor. Overall, the NPC of the vertical conveyor was $5.9M less than the trucking option;

- Operating costs: the annual steady-state operating costs are estimated to be approximately $1.3M lower for the vertical conveying option;
- The vertical conveyor offers greater operational efficiency than trucks. Should a truck be out of commission, it is typically not possible to 'make-up' lost tonnes, whereas the vertical conveyor is only scheduled to operate 18 hours per day and could easily make up additional tonnes, as required;
- Safety: The vertical conveyor can be better isolated from personnel and other equipment, thereby decoupling the primary access to the mine (the main decline from surface) from material flow;
- Maintenance: the maintenance requirements for the vertical conveyor are much less than a fleet of trucks and requires less mechanics and shop space;
- The underground truck fleet required for waste handling and internal haulage could subsidize the material flow to surface, if needed.

16.12.4.2 Waste Flow

Waste rock will be either placed into open stopes during the backfill cycle or hauled to surface using 50 t haul trucks and dumped at the surface stockpile or at the CFTF. A total of two 50 t haul trucks will be required to support the underground mine. These will be shared with the material flow circuit, where some internal haulage is required for material handling.

16.12.5 Mine Facilities

16.12.5.1 Mine Maintenance and Service Area

A central maintenance facility will be located on surface close to the portal. This will include a three-bay facility for heavy mobile equipment and a small bay for light vehicles and equipment, as well as a wash bay area. An underground service bay will be constructed in a central location that will be dedicated to light duty repair.

16.12.5.2 Warehousing and Storage

A central warehouse will be situated on surface that will facilitate storage of underground equipment and supplies. This will be in close proximity to the central maintenance facility. Underground storage will utilize abandoned remuck bays, as required. These will be used as short-term facilities for mine consumables, including bolts and screen. A dedicated maintenance storage bay will be adjacent to the underground shop.

16.12.5.3 Explosives Storage

Explosives will be stored on surface in an area that is away from the main surface infrastructure area. Detonators and bulk explosives will be stored in separate buildings no less than 50 ft from one another. Explosives will be transported underground in underground charge trucks as required; there will be no explosives storage underground.

16.12.5.4 Refuge Stations

The underground mine will be serviced with a total of four self-contained portable refuge chambers, which will be used in an emergency, where immediate egress via either the ramp to surface or secondary egress hoist is not possible. Four stations will be required to provide sufficient refuge for the peak underground personnel per shift plus a contingency of 50%. Portable refuge stations have been selected because they offer flexibility in that they can easily be moved to active areas, as required. During steady-state operations, it is expected that a 16-person refuge chamber will be located on the conveyor level close to the vent shaft (which houses the emergency hoist), and three 12-person refuge chambers will be positioned in a central location in each of the 138, Lower SMSU and Upper SMSU mining areas.

16.12.5.5 Charging Stations

The Tamarack North mine will use a full fleet of battery/electric powered equipment. (No diesel equipment is assumed for this study). Current battery fleets are designed to require periodic battery swaps that are typically aligned to a shift length or half-shift length of time. Generally speaking, heavy use mobile equipment such as trucks and LHDs have a functional charge life between four and six hours, whereas low use equipment, such as utility vehicles or drills, which generally run on connected electric power aside from transport from one area to another, have a functional charge length of a full shift. A series of charging bays will be situated next to the underground shop that will support the heavy underground equipment that does not periodically return to surface, i.e. haul trucks, LHDs, drills and bolters. Service equipment, which includes, the grader, explosives trucks, mechanics trucks, light vehicles and shotcrete haulers and sprayers will utilize battery charge stations on surface.

16.12.6 Emergency Egress

The primary egress from the mine will be through the main decline to surface. A secondary egress system was designed by FKL. It will be a dedicated electric hoisting system located in the vent shaft and comprise a 4-person cage suspended by a small headframe on surface. The cage will be guided on rope guides in the vent shaft and only be used in an emergency and for periodic shaft inspections.

Figure 16-18: 4-Person Emergency Egress Hoist

Emergency interlevel egress will be via internal raises equipped with a pre-fabricated ladderway system. A ladder system (Laddertube) was designed for Talon by Safescape. This was selected because they are economical and can be installed quickly with minimal effort.

Figure 16-19: Inter-Level Escapeway

16.12.7 Mobile Equipment

16.12.7.1 Battery/Electric vs Diesel Powered Equipment

The equipment original equipment manufacturers (OEMs) are currently investing significantly into the development battery powered equipment to eventually eliminate diesel from underground operations. Talon Metals has a bold goal of becoming one of the most environmentally friendly underground mines constructed. Given recent advancements in battery technology, it is expected that battery powered equipment will be standard by the time the Tamarack North mine is in production. For that reason, a full fleet of battery/electric mobile equipment is assumed for this study. Fully mobile equipment, such as loaders or haul trucks will be battery powered, where stationary equipment, including drills or the road header will be tethered during operations, but driven on battery-powered drives that are used during transport.

16.12.7.2 Lateral Development Equipment

The Sandvik MH621 roadheader was selected as the basis for this study because it currently has the best proven track record for underground hard rock applications and a more complete set of operational data could be provided to support this study. It should be noted that the MH621 replaces the need for a jumbo and LHD during typical operations.

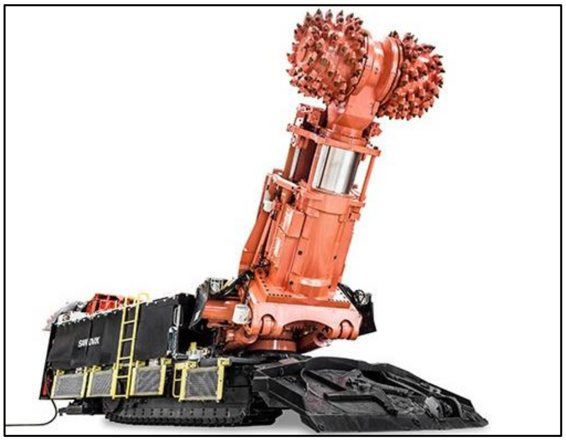


Figure 16-20: Sandvik MH621

In order to assess the viability of the MH621, Talon Metals provided Sandvik a comprehensive dataset of the expected rock properties, mine plan and design criteria for each geological domain. This included geological and geotechnical information as well as design and operational considerations. The two key factors that influence machine productivity are generally rock hardness and abrasivity. The rock hardness provided to Sandvik is based on historic lab testing and is considered robust. The abrasivity was determined based on the Cerchar abrasivity testing and was performed by a third party lab. Based on the abrasivity results and general rock hardness for each domain, Sandvik provided the following productivity estimates:

Indicative net cutting rate [NCR] and specific pick consumption [SPC] figures for a MH621, equipped with cutterhead type R425-TC72 and 22mm TC diameter picks

Lithology	UCS(MPa)			CAI	NCR [solid m³/nch]			SPC [picks / solid m³]
	Average	*Min*	*Max*	*Average*	*Average*	*Max*	*Min*	*Average*
CGO/SMSU	99,65	42	146,1	1,57	21	50	14	0,21
FGO	83,89	50,3	109,8	1,17	25	42	19	0,09
MSU/MMS	96,23	61,1	142,9	1,38	22	34	15	0,20

The volume of cut rock per effective net cutting time (counted in net cutting hours ... nch) defines the net cutting rate. The effective net cutting time is defined by the time the cutter head is in contact with the rock and actually cutting. Thus, supplementary actions of the cutter head like profiling and loading or any idling time during cutting operation does not contribute to the effective net cutting time.
For calculating machine productivity or excavation performance, please consider that the ratio of machine operating hours to net cutting hours is usually about 2:1 (ranging from 1.5:1 to 2.5:1). This difference in the times is caused by a number of delays and standstills (non-productive times) during cutting operation, when the machine is operated at the face.

Figure 16-21: Sandvik MH621 Anticipated Productivity

In addition to the estimated performance, Sandvik also provided general maintenance requirements, as well as capital and operating costs to support this study. Based on the net cutting rate determined by Sandvik, the estimated advance rates in the decline will be 1.7 times faster than that of face drilling with a jumbo. It should be noted that, past experience has shown that the road header will periodically encounter ground or situations that are not best suited for normal operations. In this case, Sandvik recommended retaining a jumbo and LHD to deal with challenging ground. It is assumed that 10% of total lateral development will therefore be driven with a jumbo using drill/blast cycles; this has been factored into the development costs and advance rates.

16.12.7.3 Equipment Fleet

The following table includes the full LOM equipment fleet requirement.

Table 16-16: Mining Equipment Fleet

Equipment	Size/Use	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Lateral Development												
Road Header	Sandvik MH621	1	1	2	2	2	2	2	2	2	1	1
Boom Jumbo	2 Boom	1	1	1	1	1	1	1	1	1	1	1
LHD	17.5 t	1	1	1	1	1	1	1	1	1	1	1
Haul Truck	50 t	1	1	1	1	1	1	1	1	1	1	1
Rock Bolter	Ground Support	1	2	3	3	3	3	3	2	2	2	2
Scissor Lift	5 m reach	1	2	3	3	3	3	3	3	3	2	2
Shotcrete Sprayer	Ground support	1	2	2	2	2	2	2	2	2	2	2
Shotcrete/Concrete Hauler	5 m^3 Capacity	1	2	2	2	2	2	2	2	2	2	2
Anfo Loader	Development	1	1	1	1	1	1	1	1	1	1	1
Production												
Long-Hole Drill	Production	0	0	1	3	3	3	3	3	3	3	3
LHD	17.5 t	0	0	1	4	4	4	4	4	4	4	4
Haul Truck	50 t	0	0	0	1	1	1	1	1	1	1	1
Anfo Loader	Production	0	1	3	3	3	3	3	3	3	3	3
General Services												
Road Grader	Road maintenance	1	1	1	1	1	1	1	1	1	1	1
Maintenance Truck	Flatbed w/crane	0	1	2	2	2	2	2	2	2	2	2
Boom Truck	Supplies delivery	0	1	2	2	2	2	2	2	2	2	2
Personnel Carriers	16 person	0	0	2	4	4	4	4	4	4	4	4
Supervisor Vehicle	Supervision	2	4	6	6	6	6	6	6	6	6	6

16.12.8 Staffing Requirements

Mine staffing includes both company employees and contract workers. Mine staffing is exclusive of labour that is specified for the surface operations, including the concentrator, hydromet plant, CFTF, and general administration. It is assumed that company employees will include:

- Administrative staff and safety;
- Engineering;
- Geology;
- Maintenance management.

Contract employees will include both engineering, procurement, and construction management (EPCM) and operational staff, including:

- EPCM leadership team;
- Contract supervision;
- Shift miners;
- Hourly maintenance employees.

The total number of company employees is estimated and based on past experience for similar operations. During steady-state mining, an average of 34 company employees will be required to support the underground operations.

Contract mining staffing is based on the following assumptions:

- EPCM and supervision are based on past experience for similar operations;
- Mining staff are based on one miner per piece of mobile equipment, three crews;
- Mobile equipment mechanics are based on one mechanic per the sum of unavailable equipment, i.e. if five pieces of equipment each have an availability of 80%, it is assumed that one mechanic would be required to service those five pieces of equipment (i.e. 5 X (1-0.8) = 1). A total of seven mechanics is therefore required per shift, or 21 total mechanics during steady-state mining.

A total of 121 contract miners are required for steady-state mining.

The total staffing requirements of approximately 300 people will therefore be required to support the underground operations during steady-state mining.

Figure 16-22: Mine Staffing Requirements

17 RECOVERY METHODS

This section describes the process design basis for the generation of the circuit mass and water balance, the metallurgical plant process design criteria, and the selection and sizing of the major processing equipment required to process the Tamarack mineralized material under the mine production schedule outlined in Section 16.

Three different processing routes are being considered at this stage, as follows:

- **Ni Concentrate Scenario:** This scenario represents a conventional processing approach for the Tamarack mineralization comprising a concentrator that produces Cu and Ni concentrates, which will be both shipped to smelters for further processing.
- **Ni Sulphate Scenario:** This scenario includes a hydrometallurgical plant to transform the Ni concentrate into a value-add product. In this scenario, only the Cu concentrate will be shipped to a smelter for further processing.
- **Ni Powder Scenario:** This scenario includes Ni concentrates produced from the project are used to produce refined Ni powder for the EV market. The Cu concentrate will be shipped to a smelter for further processing.

The flotation process design for all processing routes comprise bulk rougher flotation followed by cleaning of the bulk rougher concentrate. The upgraded rougher concentrate is subjected to Cu/Ni separation. The process generates separate Cu and Ni concentrates. Further, the bulk rougher tailings are treated in a desulphurization stage to produce a low-mass, high-sulphur tailings stream and a high-mass, low-sulphur tailings product. In the case of the Ni Concentrate and Ni Powder Scenarios, the Ni concentrate represents the final output of the Tamarack metallurgical facility.

The Ni Sulphate Scenario treats the Ni concentrate in a POX autoclave followed by two stages of neutralization, Cu removal, Ni and Co SX, and Mg precipitation. The extract (loaded organic) is scrubbed and stripped and then subjected to Co SX. The Ni is precipitated as Ni (II) sulphate hexahydrate in a crystallizer and the Co is precipitated with sodium hydrosulphide as Co sulphide.

The concentrator is identical for the three scenarios.

17.1 Key Process Design Criteria – Concentrator

To maximize the NPV of the project, the mine plan was developed to extract higher grade mineralized material in the early stage and then process the lower grade material towards the end of the mine life. This approach necessitates a different sizing approach for the concentrator to avoid equipment limitations due to more metal units in the mill feed during the early mine life and harder mineralized material towards the end of the mine life.

The process design criteria were generated based on an average daily mill feed rate of 3,600 tpd and a maximum annual head grade of 2.40% Ni and 1.08% Cu. The results of the metallurgical test program described in Section 13 were used to generate a circuit mass balance for the Tamarack flowsheet, which is presented in Table 17-1. The higher-than-average Cu and Ni head grades resulted in slightly larger equipment in the cleaning and concentrate dewatering circuits.

Table 17-1: Flotation Circuit Mass Balance for High-grade Mill Feed

Stream	Mass Rec %	Assay (%)			Recovery (%)		
		Cu	Ni	S	Cu	Ni	S
Bulk Rougher Feed	100.0	1.08	2.40	9.00	100.0	100.0	100.0
Bulk Rougher Conc	33.6	3.00	6.46	22.9	93.1	90.3	85.2
Bulk Rougher Tails	66.4	0.11	0.35	2.00	6.9	9.7	14.8
Mags	8.5	0.20	0.59	10.6	1.6	2.1	10.0
High Sulphur Thickener UF	20.7	0.25	0.75	10.2	4.8	6.4	23.5
Non-Mags	58.0	0.10	0.31	0.75	5.3	7.6	4.8
Low S Thickener Underflow	58.0	0.10	0.31	0.75	5.3	7.6	4.8
Low-Sulphur Tailings Filter Cake	58.0	0.10	0.31	0.75	5.3	7.6	4.8
Regrind Ball Mill Discharge	20.7	0.79	2.09	18.5	15.2	18.0	42.6
Bulk Cleaner 1 Conc	26.5	3.89	8.41	28.6	95.5	92.8	84.3
Bulk Cleaner 1 Tails	27.8	0.50	1.34	14.1	12.7	15.5	43.5
Bulk Cleaner Scav Conc	15.6	0.66	1.73	17.3	9.6	11.2	29.9
Bulk Cleaner Scav Tails	12.2	0.28	0.85	10.0	3.2	4.3	13.6
Bulk Cleaner 2 Conc	21.3	4.55	9.67	30.2	89.9	86.0	71.7
Bulk Cleaner 2 Tails	5.2	1.18	3.18	22.1	5.6	6.8	12.7
Cu-Ni Sep Regrind Mill Discharge	21.3	4.55	9.67	30.2	89.9	86.0	71.7
Cu Rougher Conc	4.1	20.7	3.82	31.3	79.4	6.6	14.4
Ni Concentrate	18.8	1.21	10.8	29.9	21.0	84.6	62.4
Cu Cleaner Conc	3.2	26.2	2.04	31.9	78.2	2.7	11.4
Cu Cleaner Tails	1.6	7.30	8.02	29.4	10.6	5.2	5.1
Cu Reclnr Conc (Cu Conc)	2.6	28.9	1.28	32.5	68.9	1.4	9.3
Cu Reclnr Tails	0.7	15.4	5.01	29.3	9.3	1.4	2.1
Cu Conc Thickener UF	2.6	29.0	1.28	32.5	69.2	1.4	9.3
Cu Conc Filter Cake	2.6	29.0	1.28	32.5	69.2	1.4	9.3
Ni Conc Thickener UF	18.8	1.21	10.8	29.9	21.0	84.6	62.4
Ni Conc Filter Cake	18.8	1.21	10.8	29.9	21.0	84.6	62.4

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

A – Talon Metals
B – Metpro recommendation
C – Metpro calculation
D – DRA Input
E – Metallurgical testing
F – Standard industry practise
G – Vendor recommendation

The mineralized mill feed material characteristics and the expected metallurgical performance over the life of mine are presented in Table 17-2.

Table 17-2: Plant Feed Characteristics and Metallurgical Performance

Criteria	Units	Value		Source
		Expected Range	Design	
Solids SG	t/m^3	2.60 – 3.75	2.90	D
ROM Bulk Density	t/m^3	1.60 – 2.00	1.80	B
LOM Mill Ni Head Grade	Ni	0.52 – 6.03	1.34	D
LOM Mill Cu Head Grade	Cu	0.24 – 2.41	0.74	D
Mill Treatment Capacity	ktpa		1,314	C/D
Ni Recovery to Ni Concentrate	%		81.5	E/C
Ni Concentrate Grade	% Ni		10.2	E/C
Ni Concentrate Production	ktpa		141.9	E/C
Overall Cu Recovery	%		84.7	E/C
Cu Recovery to Cu Concentrate	%		69.3	E/C
Cu Concentrate Grade	% Cu		28.5	E/C
Cu Concentrate Production	ktpa		23.7	E/C

The operating schedule of the processing plant is detailed in Table 17-3.

Table 17-3: Plant Design Operating Schedule

Criteria	Units	Value		Source
		Expected / Average	Design	
ROM Material Delivered to Mill	ktpa	1,314		C/D
Crusher Plant Operating Schedule				
Days per Week	days	7		B
Shifts per Day	shifts	3		B
Hours per Shift	h	8		B
Utilization	%	70		B
Operator Hours per Day	h	24		B
Operating Hours per Annum	h	6,132		B
Crusher Circuit Throughput	tph	214.3	267.9	B

Criteria	Units	Value		Source
		Expected / Average	Design	
Milling and Flotation Operating Schedule				
Days per Annum	days	365		B
Hours per Day	h	24		B
Utilization	%	92		B
Operating Hours per Annum	h	8,059		B
Average Mill Feed Rate	tph	163.0	195.6	C

17.2 Process Block Flow Diagram – Concentrator

The simplified process flowsheet for the crushing and grinding circuit, the flotation circuit, and the dewatering circuit is presented in Figure 17-1.

The crushing circuit comprises primary jaw crushing, secondary cone crushing, and ball mill grinding. The grinding circuit product is subjected to bulk rougher flotation, followed by cleaning of the bulk rougher concentrate and a Cu/Ni separation circuit to produce Cu and Ni concentrates. The concentrates will be thickened and filtered separately and shipped to different smelters via rail. The desulphurization flotation stage comprising a magnetic separator will recover most remaining sulphides into a HS tailings stream for use as paste backfill.

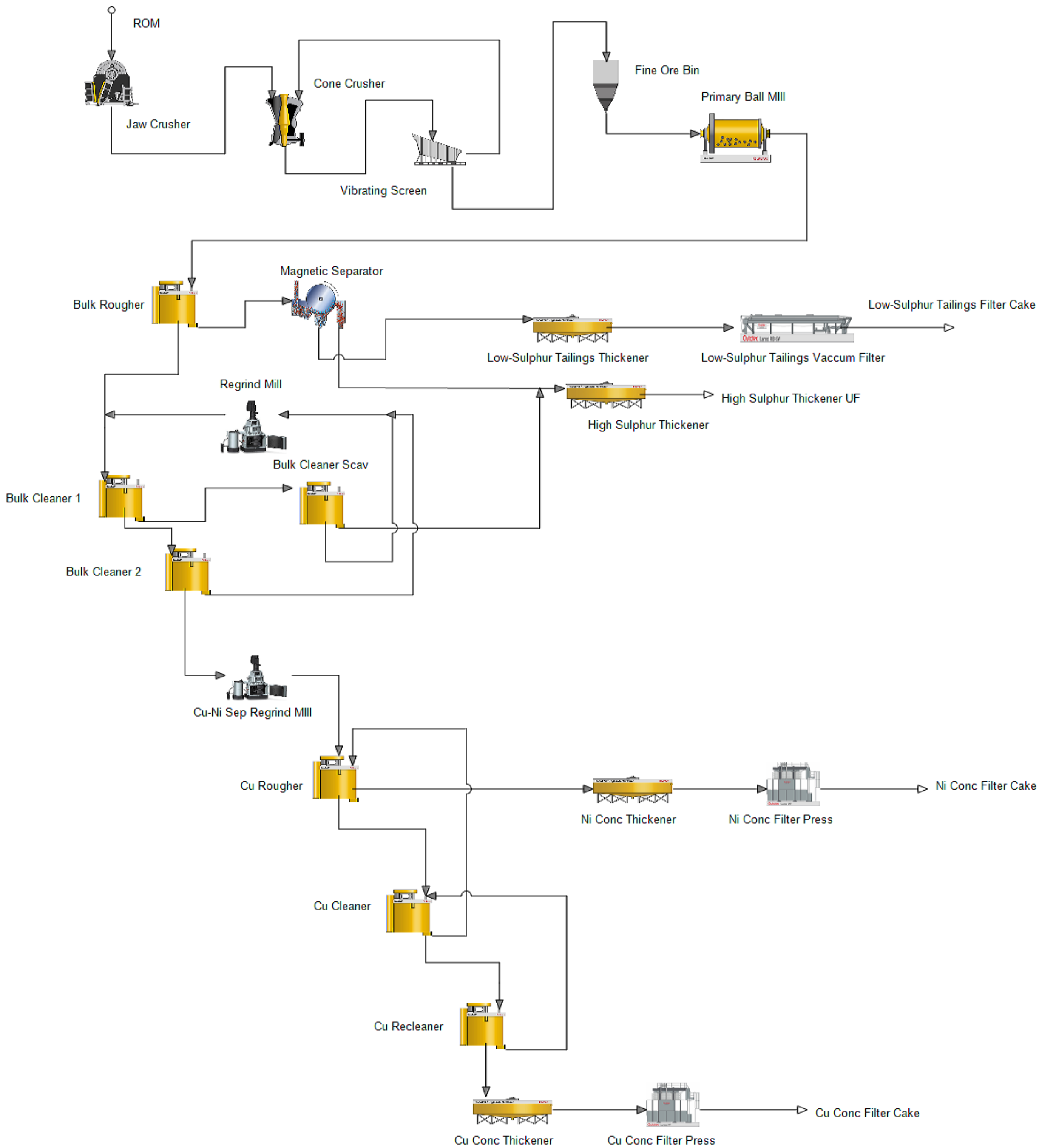


Figure 17-1: Process Flowsheet

17.3 Process Description – Concentrator

17.3.1 Crushing and Primary Grinding

Mineralized ROM material that was crushed underground is delivered to a ROM bin feeding the crushing and screening section of the plant. The ROM material with an F_{100} of 153 mm is reduced to a product size P_{80} of 17 mm in the two stages of crushing. The crushing circuit comprises a jaw crusher that is operated in open circuit followed by a cone crusher that is operated in closed circuit with a vibrating screen. The two-stage crushing circuit operates at approximately 70% utilization and a design factor of 25%, equating to a feed capacity of approximately 267.9 tph.

The cone crusher product is transferred to two fine ore bins with a combined capacity of 3,000 m^3 to decouple the crushing and grinding circuits due to the lower mechanical availability of the crushing circuit. The mineralized material is transferred from the fine ore bins to a 5.7 m x 8.7 m EGL ball mill that is operated in closed circuit with classifying hydrocyclones to generate a flotation circuit feed with a P_{80} of 100 µm.

17.3.2 Bulk Rougher and Magnetic Separation

The ball mill cyclone overflow gravitates to the bulk rougher flotation cells at a mass flow rate of 163.0 tph. The sulphide collectors SIPX and PAX and frother MIBC are added to the flotation feed box to recover the Cu and Ni values in the mineralized material. Bulk rougher flotation is carried out in tank cells with a volume of 4 x 160 m^3, corresponding to a retention time of approximately 72 minutes that is instrumental in achieving maximum metal recoveries. The pH of the slurry is lowered to 7.0 in the last tank cell using sulphuric acid (H_2SO_4) to maximize metal recovery. The bulk rougher concentrate is transferred to the bulk rougher cleaning circuit.

The bulk rougher tailings gravitate to a magnetic separator to further reduce the sulphide content in the bulk rougher flotation tailings. The slurry is treated in a low-intensity magnetic field strength of 0.115T. The non-mags represent the LS tailings stream and the magnetic fraction is combined with the bulk cleaner 1 scavenger tailings to form the high-sulphur tailings product.

17.3.3 Bulk Cleaner

The bulk rougher concentrate is transferred to the bulk cleaner 1 flotation stage, which comprises 3 x 50 m^3 tank cells. The combined retention time in the bulk cleaner 1 flotation cells is 14 minutes. A small dosage of Depramin C is added to the pump feed box to assist with the depression of non-sulphide gangue minerals.

The bulk cleaner 1 concentrate is subjected to one additional stage of cleaner flotation to further reduce the amount of gangue minerals reporting to the Cu/Ni separation circuit. The bulk cleaner 2 stage comprises 3 x 30 m^3 tank cells with a combined retention time of 12 minutes. The bulk cleaner 2 concentrate presents the Cu/Ni separation feed.

The bulk cleaner 1 flotation tailings gravitate towards the bulk cleaner 1 scavenger flotation cells to recover slow floating particles and middlings of sulphides and gangue minerals. The 3 x 50 m^3 tank cells correspond to a retention time of 24 minutes. The bulk cleaner 1 scavenger tailings are combined with the magnetic fraction of the magnetic separation stage before being subjected to dewatering.

The combined bulk cleaner 1 scavenger concentrate and the bulk cleaner 2 tailings are dewatered in a cyclone, and the cyclone underflow gravitates to a regrind in a VTM-250-WB Vertimill® to reduce the particle size from P_{80} = 100 µm to P_{80} = 60 µm. The purpose of the regrind mill is to improve liberation of the sulphide minerals. The mill discharge is returned to the hydrocyclone. The cyclone overflow is combined with the bulk rougher concentrate prior to processing in the bulk cleaner 1 flotation stage. A small dosage of SIPX is added to the bulk cleaner 1 feed box to promote the flotation of newly liberated sulphide minerals.

17.3.4 Cu/Ni Separation

The bulk cleaner 2 concentrate is directed to the feed pump box of the Cu/Ni separation regrind mill cyclone. The regrind mill is operated in closed-circuit to reduce the P_{80} in the bulk cleaner 2 concentrate from 60 µm to 25 µm prior to Cu/Ni separation. Regrinding is performed in a VTM-400-WB Vertimill® with steel grinding media.

The regrind mill cyclone overflow gravitates to the feed box of the Cu/Ni separation flotation cells. Lime is 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 are added at this stage. The separation is carried out in 2 x 30 m^3 tank cells with a combined retention time of 12 minutes.

The Cu/Ni separation tailings represent the Ni concentrate, which is pumped to the Ni concentrate thickener.

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

The Cu/Ni separation concentrate is transferred to the Cu cleaner flotation stage with 2 x 5 m^3 tank cells and a retention time of 10 minutes. The Cu cleaner tailings are transferred back to the Cu/Ni separation feed and the Cu cleaner concentrate is pumped to the Cu recleaner stage, which consists of 2 x 5 m^3 tank cells and a retention time of 10 minutes. The pH is controlled at 12.0 in both cleaning stages, and only MIBC is added to promote Cu flotation.

17.3.5 Ni and Cu Concentrate Thickening

The plant sizing was performed for a peak daily Ni and Cu concentrate production levels of 675 tpd and 93 tpd, respectively, are anticipated at a mill feed rate of 3,600 tpd. The concentrate production will decrease from those values depending on the stage of mine development and actual mill feed rates and grades. Both thickeners have been designed for the peak expected production demand, plus a 20% design factor for concentrate tonnage.

The two concentrates are transferred to two high rate thickeners. The thickened slurry is pumped to separate holding tanks. The solution from the overflow of the two thickeners is returned to the plant process water tank.

17.3.6 Ni and Cu Concentrate Filtration

The thickened Ni and Cu concentrate slurries are pumped from their stock tanks to pressure filters. The dewatered filter cakes are stockpiled. The concentrates are reclaimed from the stockpiles using a front-end loader, transferred into rail cars, and shipped to Ni and Cu smelters for further processing in the case of the Ni Concentrate Scenario. In the other two scenarios, the Ni concentrate filter cake will be reclaimed and processed in the co-located facilities. Filtrates from the pressure filters are pumped to the concentrate thickeners.

17.3.7 Tailings Thickening

Daily average LS and HS tailings production levels of 2,561 tpd and 587 tpd, respectively, are anticipated at a feed rate of 3,600 tpd. The actual production will fluctuate from those values depending on the stage of mine development and actual mill feed rates and grades.

The two tailings streams are transferred to two high rate thickeners. Both thickeners have been sized for the maximum annual production demand plus a 20% design factor for throughput. The thickened slurries are pumped to separate holding tanks. The overflow solution of the two thickeners is returned to the plant process water tank.

17.3.8 Tailings Filtration

The results of the hydraulic model, conducted by Paterson & Cooke in Sudbury, predict that sufficient head is available such that a paste mixture with 79.5% w/w can be distributed to all of the underground areas. HS tailings generated will be used in the paste backfill recipe. LS tailings will be primarily used in the paste backfill recipe blended with HS tailings to fill all the required voids. The balance of the LS tailings filter cake will be placed into a CFTF.

17.4 Energy, Water, and Process Materials Consumption - Concentrator

17.4.1 Energy

The total plant energy requirements from the major mechanical equipment list was established at 8,153 kW. Pumps and plant services were factored at 20% for a total connected power of 9,784 kW. The operational power draw is projected to be 85% of connected power or 8,316 kW. Power will be supplied by the electrical grid.

17.4.2 Water

The total water requirements of the grinding and flotation circuit are estimated at 543.0 m^3/h. This water requirement includes water addition in the grinding circuit, dilution water, and launder water. The mineralized material is assumed to contain a moisture content of at least 3% or 5.0 m^3/h.

All process water that is recovered in the dewatering circuits of the two concentrates and two tailings streams are returned to the process water tank. The water in the concentrate and tailings streams amounts to 35.3 m^3/h.

The total amount of reclaimed water is projected to be 426.0 m^3/h. Hence, the freshwater requirement to make up the water deficit is 117.0 m^3/h, which includes an allowance of 35.0 m^3/h of fresh water for gland water, potable water, reagent makeup, etc.

17.4.3 Process Consumables

Reagent types and dosages were established in the two metallurgical programs conducted at XPS Sudbury and SGS Lakefield in 2019 and 2020.

The grinding media, liner, and lifter consumption were calculated based on an estimated Bond abrasion index that corresponds to the 70th percentile of abrasiveness of more than 2,000 samples that were tested at SGS. This approach was taken since no Bond abrasion index data is presently available for the Tamarack mineralization.

The reagent consumption and grinding media wear rates are presented in Table 17-4 and Table 17-5 respectively.

Table 17-4: Reagent Consumption Rates

Reagent	Consumption (g/t)
SIPX	215
PAX	230
MIBC	150
Carboxy methyl cellulose (CMC)	30
Lime	2,150
Flocculant	30

Table 17-5: Grinding Media Consumption

Application	Consumption (kg/t of mill feed)
Primary Ball Mill Balls	0.99
Bulk Rougher Concentrate Vertimill Media	0.31
Cu/Ni Separation Vertimill Media	0.57

17.5 Major Equipment List - Concentrator

A list of major mechanical equipment of the concentrator is provided in Table 17-6, which served as the basis for the development of the capital cost estimate.

Table 17-6: Major Mechanical Equipment

Description	Equipment
Primary Crusher	Metso C120 jaw crusher
Secondary Crusher	Metso HP6 cone crusher
Crusher Closed-Circuit Screen	Inclined screen (31 m2 screening surface)
Fine Ore Bin	3,000 m^3
Primary Ball Mill	5.7 m x 8.7 m ball mill (4,200 kW)
Bulk Rougher Flotation Cells	4 x 160 m^3
Magnetic Separator	LIMS (0.115 T)
Bulk Cleaner Regrind Mill	Vertimill VTM-250-WB®
Bulk 1st Cleaner Flotation Cells	3 x 50 m3 Tank Cells
Bulk 1st Cleaner Scavenger Flotation Cells	3 x 50 m3 Tank Cells
Bulk 2nd Cleaner Flotation Cells	3 x 30 m3 Tank Cells
Cu/Ni Separation Circuit Regrind Mill	Vertimill VTM-400-WB
Cu Rougher	2 x 30 m3 Tank Cells
Cu Cleaner	2 x 5 m3 Tank Cells
Cu Recleaner	2 x 5 m3 Tank Cells
Ni Concentrate Thickener	20 m diameter, high-rate
Ni Concentrate Filter Press	Pressure Filter - 130 m^2
Cu Concentrate Thickener	7 m diameter, high-rate
Cu Concentrate Filter Press	Pressure Filter - 22 m^2
HS Tailings Thickener	18 m diameter, high-rate
LS Tailings Thickener	36 m diameter, high-rate
LS Tailings Vacuum Filter	200 m^2 Belt Filter

17.6 Key Process Design Criteria – Hydrometallurgical Plant for Ni Sulphate Production

The process design criteria were generated based on an average daily hydrometallurgical plant feed rate of 475 tpd and an average Ni concentrate head grade of 10.2% Ni, 0.23% Co, and 1.06% Cu. A hydrometallurgical test program provided the results for the leaching, neutralization, and purity removal stages for this LOM Ni concentrate. The SX circuits and the Mg precipitation circuit were designed using available engineering data of similar projects.

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

A – Talon Metals

B – Metpro recommendation

C – Metpro calculation

D – DRA Input

E – Metallurgical testing

F – Standard industry practise

G – Vendor recommendation

The average LOM mineralized mill feed material characteristics and expected metallurgical performance are presented in Table 17-7.

Table 17-7: Plant Feed Characteristics and Metallurgical Performance

Criteria	Units	Value		Source
		Expected Range	Design	
Solids SG	t/m^3	4.00 – 4.40	4.19	D
Bulk Density	t/m^3	2.50 – 2.75	2.62	B
Ni Concentrate Grade	% Ni	9.90 – 10.5	10.2	D
Cu Grade in Ni Concentrate	% Cu	0.90 – 1.32	1.06	D
Ni Recovery to Ni Sulphates	%		77.4	E/C
Ni Sulphate Production (as $NiSO_4$*$6H_2O$)	ktpa		61.9	E/C
Co Recovery to Co Sulphides	%		60.9	E/C
Co Sulphide Production	ktpa		0.320	E/C
Cu Recovery Cu Conc, Sec Neut & CuS	%		81.8	E/C
Cu Concentrate Grade	% Cu		27.4	E/C
Cu Concentrate Production	ktpa		29.0	E/C

The operating schedule of the processing plant is detailed in Table 17-8.

Table 17-8: Hydrometallurgical Plant Design Operating Schedule

Criteria	Units	Design	Source
Ni Concentrate to POX Circuit	ktpa	173.4	C/D
Days per Week	days	7	B
Shifts per Day	shifts	3	B
Hours per Shift	h	8	B
Utilization	%	85	B
Operator Hours per Day	h	24	B
Operating Hours per Annum	h	7,446	B

17.7 Process Block Flow Diagram – Hydrometallurgical Plant for Ni Sulphate Production

The simplified process flowsheet for the hydrometallurgical plant is presented in Figure 17-2.

The Ni concentrate is repulped and leached in an autoclave. The autoclave discharge is treated in two stages of neutralization followed by Cu removal. Ni and Co are removed with SX before the barren solution is treated to remove Mg. Ni and Co are separated by a second SX circuit. Ni is crystallized as Ni sulphate hexahydrate and Co is precipitated as Co sulphides.

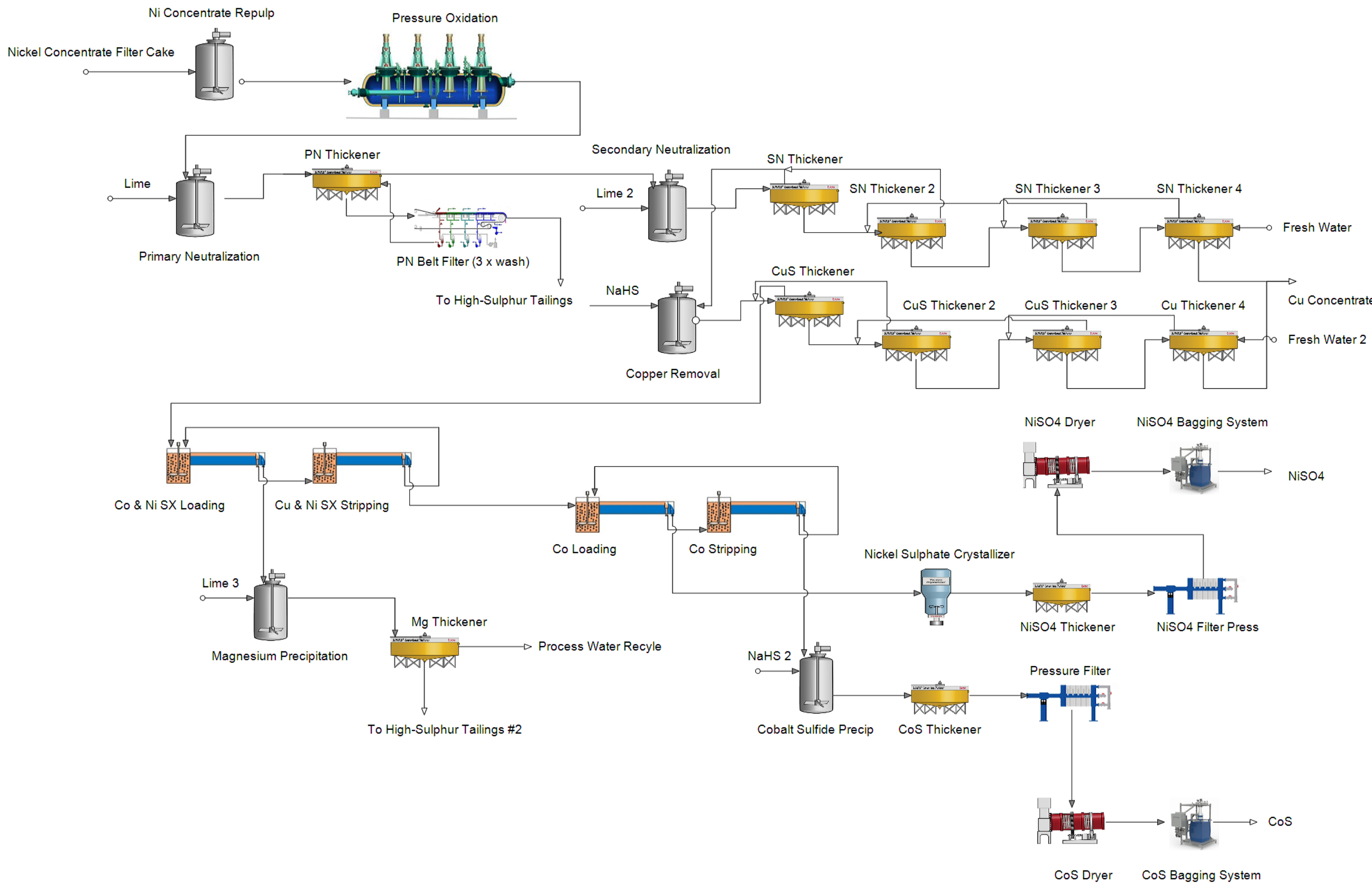


Figure 17-2: Process Flowsheet – Hydrometallurgical Plant

17.8 Process Description – Hydrometallurgical Plant

17.8.1 Pressure Oxidation

The hydrometallurgical plant has been designed to operate at approximately 85% utilization with an average feed rate of 19.8 tph of Ni concentrate. The Ni concentrate is repulped in a 4.5 m x 8.0 m (D x H) agitated tank at a solids concentration of 10% w/w. The slurry is heated to approximately 70°C using direct steam from the autoclave discharge vent before being pumped into the autoclave.

The 5 m x 23 m (D x L) autoclave operates at a temperature of 150°C with a maximum allowable working pressure (MAWP) of 580 kPa (absolute). The purpose of the autoclave is to oxidize the sulphides in the Ni concentrate to form soluble metal sulphides. The autoclave retention time is 90 minutes, and it operates at a pH of 1.2. Chloride ions are added to the autoclave as a catalyst at a concentration of 2 g/l of chlorides in the slurry.

The extraction rates of the POX leach for Ni, Co, and Cu are 99.5%, 99.4%, and 88.9%, respectively.

The autoclave discharge is transferred to a flash vessel, which reduces the slurry to atmospheric pressure. The steam that is generated in the flash vessel is recycled to pre-heat the Ni concentrate slurry in the pre-pulp tank.

The leach residue consists mainly of goethite (~50%), hematite (~25%), and elemental sulphur (~15%). Minor minerals include jarosite, talc, quartz, and diopside.

17.8.2 Primary, Secondary Neutralization, and Cu Removal

Besides the valuable elements (Ni, Co, and Cu), the leach residue contains impurities that must be removed prior to downstream processing of the leach liquor. These impurities include Fe (~413,000 mg/L), Al (~3,300 mg/L), Mn (~270mg/L), Cr (~210 mg/L), and Zn (~190 mg/L).

Most impurities are removed by raising the pH of the leach liquor using calcium carbonate ($CaCO_3$) (limestone). Initial neutralization tests aimed to achieve the impurity removal in a single stage, but co-precipitation of Cu was high and resulted in Cu losses of over 90%. Hence, impurity removal is performed in two stages with a target pH of 3.25 in the first stage and a target pH of 4.75 in the second stage.

Limestone is added to the autoclave discharge in the first of two agitated tanks (4.5 m D x 8.0 m H). The discharge of the first tank gravitates to the second agitated tank and then discharges into an 8 m diameter high-rate thickener. The thickener overflow is directed to the secondary neutralization stage and the thickener underflow is pumped onto a vacuum belt filter with a combined filtration area of 90 m^2. Three stages of washing are performed on the belt filter and washing is performed in counter current mode.

Almost all Fe and chromite are precipitated in the first neutralization stage with very low Ni, Co, and Cu losses of <0.1%, 0.2%, and 3.6% respectively.

The primary neutralization liquor is then treated in the secondary neutralization stage to further lower Fe and Al concentrations to below the detection limit. The secondary neutralization is performed in two 4.5 m D x 8.0 m H agitated tanks. Limestone is added to the first tank only to prevent short-circuiting. Approximately 60% to 70% of the Cu is co-precipitated.

The secondary neutralization slurry is transferred to a 11 m diameter high-rate thickener and the thickener overflow is pumped to the to the Cu removal circuit. The thickener underflow is transferred to a three-stage counter-current decantation (CCD) circuit consisting of three 2 m diameter thickeners. The underflow of the last thickener is combined with the Cu concentrate prior to thickening.

The secondary neutralization thickener overflow is transferred into the first of two 4 m D x 7.0 m H agitated tanks for Cu removal. NaHS is added at approximately 240% stoichiometric to precipitate any remaining Cu ions as Cu sulphide. The discharge of the second Cu removal tank is treated in a 11 m diameter high rate thickener followed by a 3-stage CCD circuit with 2 m diameter thickeners. The thickener underflow is combined with the Cu concentrate and the thickener overflow is transferred to the SX circuit.

17.8.3 Ni and Cu SX

The Cu-free thickener overflow is subjected to the first stage of SX in a mixer and settler using Versatic 10 as the extractant and Exxsol D80 as the diluent. The loaded organic is pumped to the stripping stage in another mixer and settler, while the raffinate is transferred to the Mg removal circuit. The striped organic is returned to the loading stage and the pregnant aqueous phase containing Ni and Co is treated in the Co SX circuit.

The pregnant solution is processed in a mixer and settler using Cyanex 272 as the extractant and Exxsol D80 as the diluent. Co is loaded onto Cyanex 272 while Ni remains in the aqueous phase, which effectively separates Ni and Co. The loaded Cyanex 272 is stripped in another mixer and settler to transfer Co into the aqueous phase for further treatment in the Co sulphide recovery circuit. The raffinate of the Ni and Co separation SX stage is transferred to the Ni sulphate recovery circuit.

17.8.4 Ni Sulphate Recovery Circuit

The aqueous phase after Co SX only contains Ni and trace amounts of other elements and is treated in a crystallizer to precipitate the Ni as Ni sulphate hexahydrate particles. The crystallizer discharge is thickened, and the thickener underflow is further dewatered in a pressure filter. The filter cake is dried, and the dry Ni sulphate hexahydrate is bagged in one tonne bulk bags.

17.8.5 Co Sulphide Recovery Circuit

The Co solution is treated with NaHS to precipitate the Co as a sulfide. The CoS is dewatered in a thickener and filter press. The filter cake is dried and the product is bagged.

17.8.6 Mg Removal

The barren leach solution is subjected to a final precipitation stage to remove the 2.8 g/L of Mg that remains in solution. The barren solution is pumped into the first of two 5.0 m D x 9.0 m H agitated tanks and the pH is raised to 9.0 using lime. The discharge of the second tank is transferred to a 11 m diameter high-rate thickener and the thickener overflow is returned to the front-end of the circuit to re-pulp Ni concentrate. The thickener underflow is pumped onto a 90 m^2 belt filter. The filtrate is returned to the thickener and the filter cake is combined with the high-sulphur tailings stream.

17.9 Energy, Water, and Process Materials Consumption – Hydrometallurgical Plant

17.9.1 Energy

The total hydrometallurgical plant energy requirements were estimated at a total connected power of 6,000 kW. The operational power draw is anticipated to be 85% of connected power or 5,100 kW. Power will be supplied by the electrical grid.

17.9.2 Water

The total water requirements of the hydrometallurgical circuit are estimated at 30 m^3/h or 1.67 m^3/t of Ni concentrate.

The process water that is recovered in the dewatering circuit of the Mg precipitation stage is circulated back to the process water tank at a rate of 208 m^3/h.

17.9.3 Process Consumables

Reagent types and dosages were established in the 2020 hydrometallurgical program up to the Cu precipitation stage. The balance of the reagent dosages was developed from similar operations. The reagent dosages are presented in Table 17-9.

Table 17-9: Reagent Consumption Rates

Reagent	Consumption (kg/t of Ni Concentrate)
Ferric Chloride	27.8
Oxygen	298
Limestone – Primary Neutralization	390
Limestone – Secondary Neutralization	10.0
NaHS	17.6
SX – Diluent	1.19
Extractant	0.12
Lime – Mg Precipitation	135
Flocculant	0.10

17.10 Major Equipment List – Hydrometallurgical Plant

A high-level list of major mechanical equipment is provided in Table 17-10, which served as the basis for the development of the capital cost estimate. Detailed equipment specifications were developed for the front end of the hydrometallurgical plant up to Cu sulphide precipitation based on testwork results. All other equipment sizing was determined by comparison of the proposed circuit with similar plants.

Table 17-10: Major Mechanical Equipment for Hydrometallurgical Plant

Description	**Specification – Hydrometallurgical Plant**
Ni Concentrate Repulp	Agitated tank
Leach Circuit	Autoclave (brick lined)
Primary Neutralization	Agitated tanks, belt filter with three stage wash
Secondary Neutralization	Agitated tanks, four-stage CCD
Cu Removal	Agitated tanks, four-stage CCD
SX – Ni and Co	Mixer & settlers (loading, stripping)
SX – Co	Mixer & settlers (loading, stripping)
Ni Sulphate Circuit	Crystallizer, thickener, dryer, bagging system
Co Sulphide Circuit	Agitated tanks, thickener, belt filter, dryer, bagging system
Mg Removal	Agitated tanks, thickener

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 powerhouse industrial facility will be provided for the distribution of power and backup power generation. The powerhouse 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.

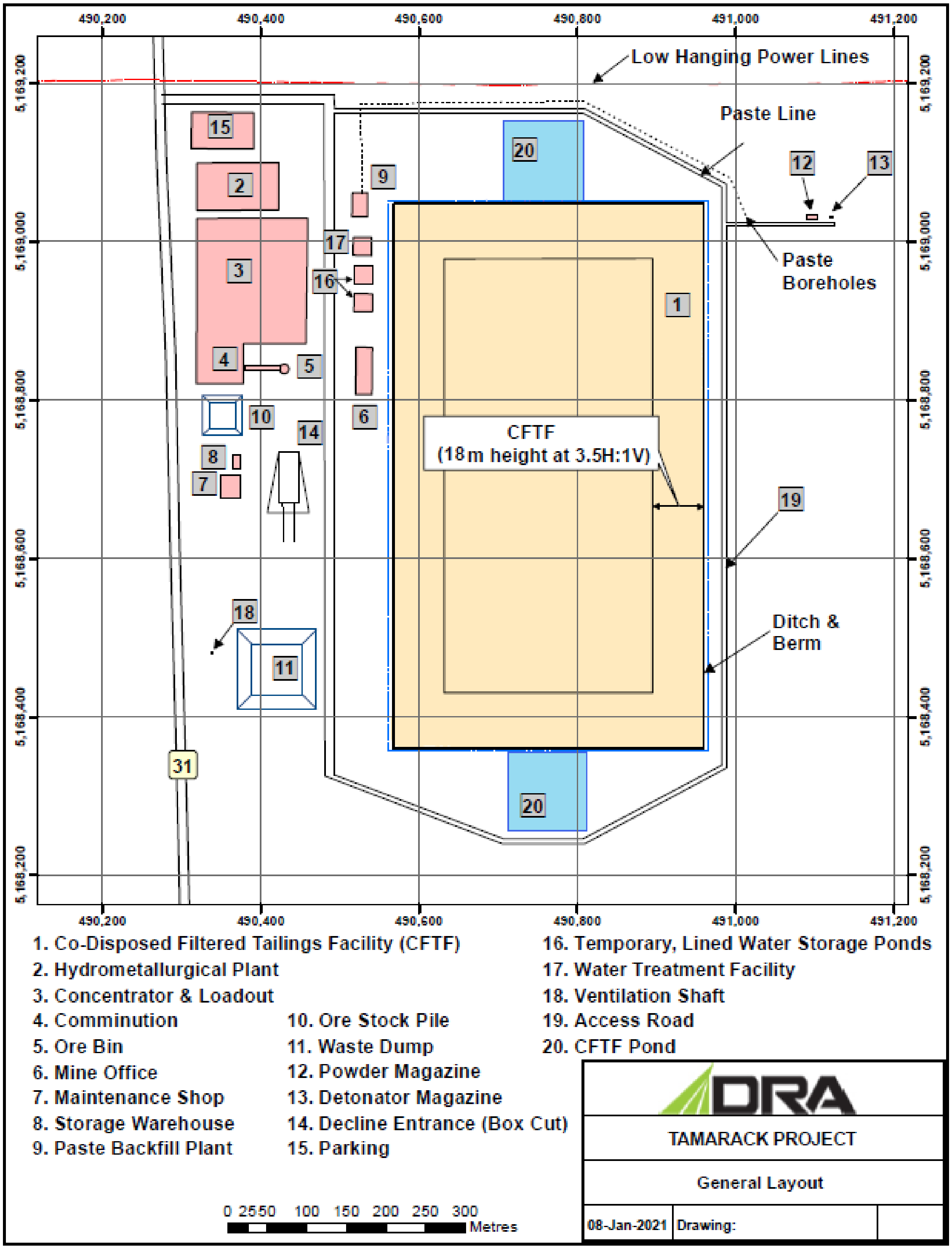


Figure 18-1: Concept Project Site Layout

18.4 Buildings and Facilities

18.4.1 Surface Portal

The main decline portal will be a small box cut 12 m wide x 61 m long x 8.5 m deep. It will serve as the primary entrance and egress from the mine and be situated in a location that minimizes the traffic on the surface footprint.

18.4.2 Mine Vent Shaft Surface Infrastructure

A multi-purpose shaft will be constructed on the main surface footprint. The shaft will house a vertical conveyor, in addition to emergency hoisting system and mine services such as power and water. An emergency hoisting system and small headframe will be constructed adjacent to the shaft collar. Additionally, the head discharge end of the vertical conveyor will be constructed on surface that will deliver mineralized material to the surface stockpile.

18.4.3 Ore Storage

An allowance is made for the mineralized material stockpile that will be adjacent to the mill.

18.4.4 Mine and Mill Services Building

The mine and mill services building will be located to ensure optimal and safe movement of personnel and equipment.

18.4.5 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.

18.4.6 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.

18.4.7 Hydromet (Hydrometallurgical) Plant

The hydromet building will house equipment for the production of a Ni sulphate product.

18.4.8 Paste Backfill Plant

The paste plant will be located adjacent to the mill. Paste fill will be pumped from the paste plant to the to a BH to the E of the main footprint area.

18.4.9 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.

18.4.10 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 in future project phases.

18.4.11 Vehicle Washing Bays

All vehicles leaving the main operations area will be washed before leaving. Underground vehicles will be washed at the end of shift, upon return to surface or prior to scheduled maintenance.

18.4.12 Mine Office, Warehouse and Workshops

The concentrator and mine are supported by administrative, supplies and maintenance functions housed in the mine office, warehouses (to store supplies used in the mining operation) and workshops (mechanical, electrical and instrumentation), including powder and detonator magazines which are located remote from the rest of the surface infrastructure. A parking area will be located near the warehouse and workshops.

18.4.13 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.

18.6 Co-disposed Filtered Tailings Facility (CFTF)

Talon commissioned the investigation of alternative options for the management and storage of LS tailings at surface (DRA 2018):

- Slurry tailings pumped, and deposited in a lined Tailings Storage Facility (TSF);
- High density slurry or paste tailings pumped and deposited in a lined TSF;
- Cemented paste tailings pumped and deposited in a lined TSF;
- Filtered tailings transported to a lined TSF by conveyor or by haul truck.

After assessing the alternatives and the design drivers for the project the project team determined that a CFTF was ideal for the project to optimize waste management footprints and address geochemical uncertainties (DRA 2018).

The LS tailings will be filtered to remove sufficient water to produce a soil-like consistency to allow LS tailings and development rock to be deposited together in the lined CFTF, as described below. Any precipitation falling on the facility or moisture that is released from the tailings will be collected in a lined ditch around the CFTF perimeter and directed to one of two CFTF collection ponds. Any collected water will be recycled back to the processing plant for re-use or treated by the Water Treatment Plant and recycled or discharged.

Co-disposal of filtered tailings and development rock offers significant environmental and operating advantages over separate tailings 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;
- Improved tailings stability and reduced dusting compared to a standalone filtered tailings facility without co-disposal with development rock;
- A reduced project footprint creating less disturbed area and reducing initial earthworks requirements;

- At closure, the CFTF will be covered with a composite closure cover system. This will limit the amount of infiltration into the CFTF post closure, potentially reducing long-term water treatment and post-closure care liabilities;
- A reduction in fresh water requirements.

The original tailings disposal alternatives evaluation and CFTF design concept were conducted by Golder (DRA, 2018). The updated CFTF design and facility description was provided by SLR Consulting. The following sections summarize its key criteria and features.

18.6.1 Mining and Processing

All three processing scenarios will produce 10.76 Mt of mined mineralized material (see Section 16.8 for an annual production schedule). The mine will generate up to 1.58 Mt of development rock. To date, static geochemical testing has been conducted on the various development rock and mineralized material types at the Tamarack North Project. However, no testing has been carried out on samples of tailings; 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 NAG, while the CGO was interpreted to be Potentially Acid Generating (PAG). 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. No further geochemical information has been collected, so this PEA update assumes that the ratio between NAG and PAG development rock is unchanged from the previous March 2020 PEA at 69% PAG. Therefore, approximately 0.56 Mt of development rock is classified as NAG while the remaining 1.25 Mt (all CGO) is classified as PAG.

The processing plant was designed to process up to 3,600 tpd (1.314 Mtpa) of mineralized material. The processing plant will generate two separate concentrate streams, namely Ni and Cu, and two tailings streams: HS tailings which comprise Po and other sulphides and low LS tailings which comprise mainly silicates. Over the LOM, the hydrometallurgical processes will generate a total of 11.18 Mt of tailings including residues from the primary neutralization circuit and the MgO removal circuit process residue, of which 9.10 Mt will be LS tailings and the remaining 2.08 Mt will be HS tailings as detailed in Table 18-1.

It is planned that approximately 56% of the total tailings (6.26 Mt) will be used for backfilling underground stopes. This includes HS tailings and close to one-quarter of LS tailings. The remaining 5.44 Mt of LS tailings are dewatered using a filtration plant to a solids content of approximately 85% and trucked to the CFTF for co-disposal with development rock.

18.6.2 Tailings and Development Rock Production

The CFTF is designed to provide adequate storage capacity for the PAG development rock, NAG development rock, and filtered LS tailings. The in-place dry densities of these waste streams are required to estimate the storage volume requirements of the CFTF as presented in Table 18-1. The specific and void ratio assumed for the various waste streams are based on preliminary information available from Talon as well as SLR's experience on similar projects.

Table 18-1: Assumed Geotechnical Properties and Volumes of Waste Streams for Co-disposal

Material Streams		Symbol	Assumed Porosity (n) or Void Ratio (e)	Dry Density t/m³	LOM Waste Totals Mt	Mm³
Development Waste Rock	NAG	A	n = 0.3	1.94	0.51	0.26
	PAG	B	n = 0.3	2.02	1.13	0.56
	PAG Voids	C	-	-		(0.19)
Total Milled Mineralized Material		D	-	-	10.76	
Tailings	LS	E	-	-	6.99	
	HS	F	-	-	1.60	
	Residue	G	-	-	2.14	
	Tailings Paste Backfill	H	-	-	4.85	
	Tailings to CFTF	I = E+F+G-H	e = 0.73	1.58	5.88	3.72
Required CFTF Capacity		**(A+B+I) - (0.5 C)**				**4.45**

Notes:

1. NAG/PAG waste rock fraction split assumed to be unchanged from Updated PEA (DRA 2020).
2. Portion of HS tailings to backfill is 92%. Tailings to the CFTF comprise 5.44 Mt LS, 0.13 Mt HS, and 0.32 Mt residues.

Based on the densities in Table 18-1, the volumes of NAG development rock, PAG development rock, LS tailings that will be sent to the CFTF for the Ni Sulphate Scenario is included in Table 18-1. The other two scenarios do not involve neutralization residues and would result in a lower deposited volume in the CFTF. 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. The full volume of NAG development rock will be used for construction of the perimeter wall. It was conservatively assumed that approximately 50% of the PAG development rock void space will be filled with filtered tailings. That is, approximately 93,000 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. The CFTF storage capacity is 4.45 Mm^3. Flexibility for variations in design assumptions can be provided by varying the height.

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. In addition to rock core testing one desulfurized tailings sample was subjected to the same suite of geochemical testing.

Two samples of both sediment and FGO were submitted for analysis. Three CGO samples were also analyzed; however, two of the CGO samples were re-classified as SMSU and included in the mine plan. The third CGO sample with sulphur content of 0.21% is considered representative of CGO development rock.

Review of the BH database shows that sulphide contents typically increase with proximity to the mineralized material zone. This is consistent with the preliminary ABA results with LS contents in units farthest from the mineralized material zone (SED and FGO) and increasing sulphide contents within the units closest to the mineralized material 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 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 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 value less than 1 is an indication of PAG;
- NPR value between 1 and 2 indicates an uncertain potential; and
- NPR value 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 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 by Foth (2008); however, a detailed assessment was not completed. A review of the data in comparison to average crustal abundances presented by 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 do 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 rock core 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 tailings sample was found to have an NPR of 4.34 and a total sulphur concentration of 0.93%. Further geochemical testing is needed but the tailings are generally assumed to be NAG or mildly PAG.

The current geochemical characterization is considered sufficient for the purpose of the PEA considering the amount of development rock anticipated over the LOM. Additional geochemical characterization is required as the Tamarack North Project progresses into future phases, 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, mineralized material, and tailings and will include additional static testing as well as long-term kinetic testing to further understand the acid rock drainage (ARD)/ metal leaching (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. A key operating criterion for CFTF operations will be the time to onset of acid production (or "fuse-time") of the PAG rock.

18.6.4 CFTF Conceptual Design and Design Criteria

The CFTF general arrangement plan is shown in Figure 18-2. The CFTF will be rectangular and will cover a footprint area of approximately 75 acres (303,300 m^2 or 3.26 million ft^2) with a height of 25 m (82 ft) for the Ni Sulphate Scenario and lower for the other two scenarios. Key CFTF components include:

- **Base Grade –** 0.5% slope from the middle of the facility N and S towards the collection ponds, to collect runoff water and seepage;
- **Base Liner System –** composite liner as explained in Section 18.6.5 below;

- **Perimeter Containment** – 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 3 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;
- **CFTF Collection Ponds** – at fixed locations to collect underdrainage and surface runoff for environmental protection;
- **Access Ramp** – at the NW 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 or co-mingled as dictated by the geochemical drivers – refer to Section 18.6.9 below.

The CFTF development can be staged into two parts to reduce runoff volumes collected and to defer capital cost. The N or the S half can be developed first with its accompanying base grading, liner, perimeter ditch and berms, and stormwater management pond. Construction of the CFTF up to its design height within one half of the facility is expected to be adequate for approximately the first four years of tailings production. Staging of the CFTF in this way is also ideal for progressive rehabilitation and minimizing runoff collection and treatment requirements. A 50% footprint CFTF for the Ni Sulphate Scenario is illustrated on Figure 18-2.

The design criteria for the CFTF is summarized in Table 18-2 below.

Table 18-2: Key CFTF Design Criteria

Parameter	Description
General	
Storage Capacity (minimum)	4.45M m^3 (approximately 25 m height)
Base Grade	Continuous to convey leachate by gravity to external pond
Base Liner	Composite liner with a leachate collection system
Perimeter Containment	Constructed progressively using NAG development rock
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	1.58 t/m^3
Moisture Content	17.6% gravimetric (weight water/weight dry soil)
Development Rock	
NAG Dry Density	1.94 t/m^3
PAG Dry Density	2.02 t/m^3
PAG Porosity	0.3
Voids to be filled with tailings (PAG)	50% approximate

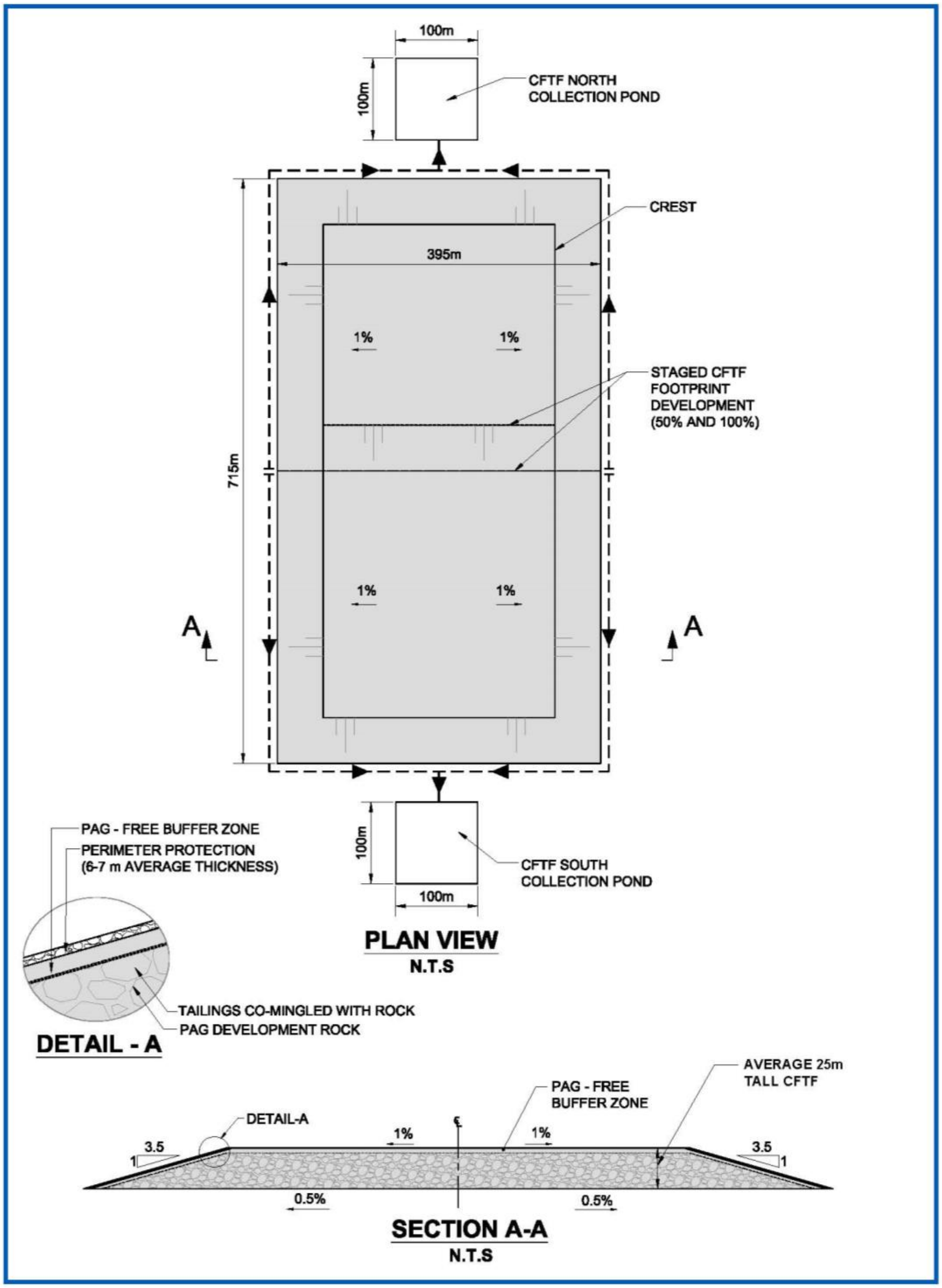


Figure 18-2: CFTF General Arrangement Plan

18.6.5 CFTF Base Grade and Base Liner

The base of the CFTF will be graded to provide a 0.5% slope to the N and S with a drainage divide across the middle of the facility. The base grading will allow runoff water collected by the perimeter ditch and seepage collected by the leachate collection system to flow by gravity to the CFTF collection 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 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 the crest of the perimeter berm.

A 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 soil protection layer provided over the leachate drainage layer reduces 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 prevent 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 perimeter NAG rock containment to enhance stability, erosion protection, and dust control. Enough NAG rock is present in the mine plan to provide an average horizontal perimeter thickness of 6 to 7 m with a 3.5H:1V exterior side slope. The wall will be constructed in 1.5 m high lifts in the upstream construction method.

The lower lifts of the perimeter rockfill will have a filter zone separating the NAG rock from tailings to prevent fines migration. For upper lifts of the perimeter rockfill a geotextile will be provided between the development rock and the tailings to act as a filter to allow seepage to flow into the perimeter ditch but preclude tailings migration.

NAG development rock production in the initial years of mining will exceed the needs for CFTF construction, but NAG needs will exceed the production in the later years of mining. Therefore, approximately 15% of the NAG rock will need to be temporarily stockpiled outside of the CFTF footprint before ultimately being incorporated into the CFTF.

18.6.7 Perimeter Berm and Ditch

A berm approximately 1 m high and a ditch will be provided at the perimeter of the CFTF approximately 3 m from the toe of the first perimeter wall lift. The berm will be used for anchoring the base liner system and also for supplementing the flow capacity of the perimeter ditch. The ditch will direct the one in 100 year, 24 hour storm runoff water from the exterior slopes of the perimeter wall into the CFTF collection ponds from where it will be pumped to the plant.

18.6.8 Access Ramp

A ramp will be provided on the NW 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 near the optimum water content, assumed to be 17.6% moisture by mass, using the filtration plant located within the process plant area. Filtering to optimum water content allows the tailings to be hauled, placed and compacted like a soil.

The filtered tailings will be trucked to the co-disposal area, 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 in the interior of the CFTF such that a buffer zone of NAG tailings will separate the PAG rock from the perimeter wall of the CFTF to mitigate the risk for ARD and ML. Filtered tailings will also be co-mingled with the PAG development rock by alternating truck-loads of tailings and PAG and working them together while spreading with a dozer. For design purposes it is assumed that half of the PAG rock pore space is filled with tailings. Future geochemical studies will examine the cost-benefit of further void filling to inhibit oxidation reactions.

Dust suppression covers will be installed on all trucks. Dust generation will be mitigated by having a perimeter wall height that exceeds the tailings height. ARD generation will be mitigated through regularly covering the PAG rock or co-mingled with NAG tailings. Future studies should examine the exposure (“fuse”) time required before acid production to confirm the operational requirements for co-mingling.

Tight process controls will be required for the filtering of tailings to ensure the design intent is met. However, during operations performance variability and plant upsets in the tailings filtering process should be expected. During such periods, off-spec tailings should be deposited in the middle of the CFTF and co-mingled with PAG so they do not affect the physical stability of the facility.

18.6.10 CFTF Closure Cover

At closure, the top of the co-disposal area will be regraded to have a 1% crown. This will create a stable post-closure landform that will shed precipitation runoff.

The perimeter wall and the top of the CFTF will be provided with a closure cover system that will consist of the following, from bottom to top:

- 552 g/m^2 non-woven cushion geotextile;
- 0.6 m thick liner bedding (soil);
- 3.5 kg/m^2 GCL;
- 1.5 mm linear low-density polyethylene (LLDPE) geomembrane liner;
- 0.3 m thick coarse aggregate drainage layer;
- 330 g/m^2 non-woven filter geotextile;
- 0.45 m thick soil for root penetration;
- 0.15 m thick topsoil;

- Vegetation with native grass species.

The closure cover will include swales 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.

The closure cover can be installed progressively to reduce dust generation, contact water runoff, and closure liabilities. The staging of the CFTF in two halves also allows for rehabilitation of the first half after the second half of the facility is opened for co-disposal.

18.6.11 CFTF Water Balance and Stormwater

During operation, all runoff and seepage from the CFTF facility will be captured, treated in the water treatment plant and/or used in the process plant. Some water loss through evaporation from the collection ponds is expected. Average annual runoff from the CFTF facility was estimated using the facility and ponds footprints (323,000 m^2), a runoff factor of 0.6, and precipitation data from US Climate Data – Duluth, Minnesota (**https://www.usclimatedata.com/climate/duluth/minnesota/united-states/usmn0208**). The facility development is assumed to occur in two stages with progressive reclamation and covering of the first stage occurring in tandem with development of the second stage. Once the first stage is covered and runoff water quality has reached acceptable levels, it will be detained for sedimentation and then discharged to the environment. Through this staging, runoff collection and treatment from the full CFTF footprint should be limited to a short period when transitioning between stages 1 and 2. Table 18-3 summarizes the average monthly inflows and outflows from the CFTF facility.

Table 18-3: Annual Average CFTF Flows

Inflow/Outflow	**Average Monthly Flows (m^3/month)**	
Open Footprint	50%	100%
CFTF Facility Runoff	6,088	12,175
Pond Evaporation	333	667
Net CFTF Outflow	**5,754**	**11,509**

Two stormwater retention ponds, both with a capacity of 25,000 m^3, referred to as the CFTF North and South Collection Ponds, have been included in the project. The CFTF Collection Ponds have the capacity to capture the 100-year 24-hour storm event runoff (156 mm of rainfall, 0.9 runoff factor) from the CFTF facility plus some additional capacity for regular operation.

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 and hydromet plant mass balance water quantities estimated in the course of process design;
- Quantities of water from the CFTF were advised by SLR as per Section 18.6;
- Quantities of mine water production were estimated as explained in Section 16.10;
- Quantities of mine water required were estimated as explained in Section 16.12.

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 concentrator and hydromet plant were calculated by estimating the total water requirement for all processes less water recycled from thickening and filtering, resulting in a net water requirement for the Ni Concentrate or Ni Powder Scenarios of 515 USgpm at plant capacity. Note that 78.5% of water required by the processing plant will be recycled;
- Net water requirement for the Ni Sulphate Scenario (concentrator plus hydromet plant) will demand about 1200 USgpm at plant capacity due to fresh water demand for the hydromet plant circuits. About 54% of concentrator plus hydromet plant water required will be recycled;
- Water collected from the CFTF is estimated at:
    - 37 USgpm from year 3 through year 4;
    - 55 USgpm for years 5 to 8; and
    - 37 USgpm for years 9 to 11.
- Water required by underground operations at plant capacity is estimated to be in a range from 805 to 1605 USgpm;

- Water expected to be produced from underground operations was calculated as explained in Section 16.10.

The net result per year is shown in Table 18-4 and Table 18-5 below, for the different plant processing scenarios.

Table 18-4: Net Water Balance by Year for the Ni Concentrate and Ni Powder Scenarios

Production Year	1	2	3	4	5	6	7	8	9	10	11	12
						gpm						
Processing Plant												
Water required for processing			(1,578)	(2,341)	(2,392)	(2,476)	(2,370)	(2,363)	(2,364)	(2,166)	(1,523)	
Water recycled from processing			1,238	1,837	1,877	1,942	1,859	1,854	1,855	1,699	1,195	
Water deficit (at the processing facility)			(340)	(504)	(515)	(533)	(510)	(509)	(509)	(467)	(328)	
Comingled Filtered Tailings Facility (CFTF)												
Water run-off from the CFTF			37	37	55	55	55	55	37	37	37	
Water deficit at surface			(303)	(467)	(460)	(478)	(455)	(453)	(472)	(430)	(291)	
Water used and produced at the mine (underground)												
Water required			(119)	(218)	(244)	(244)	(244)	(244)	(244)	(194)	(180)	
Potential cumulative water production			925	1,365	1,541	1,629	1,717	1,761	1,849	1,717	1,717	
Net water deficit/(surplus) underground			805	1,147	1,297	1,385	1,473	1,517	1,605	1,523	1,537	
Total			502	680	837	907	1,018	1,064	1,133	1,094	1,246	
Recycle in process plant			**78.5%**	**78.5%**	**78.5%**	**78.5%**	**78.5%**	**78.5%**	**78.5%**	**78.5%**	**78.5%**	

Table 18-5: Net Water Balance by Year for the Ni Sulphate Scenario

Production Year	1	2	3	4	5	6	7	8	9	10	11	12
							gpm					
Processing Plant (Concentrator & Hydromet)												
Water required for processing			(1,657)	(2,457)	(2,511)	(2,598)	(2,487)	(2,480)	(2,482)	(2,273)	(1,599)	
Water recycled from processing			888	1,317	1,346	1,393	1,333	1,329	1,330	1,218	857	
Water deficit (at the processing facility)			(769)	(1,140)	(1,165)	(1,206)	(1,154)	(1,151)	(1,152)	(1,055)	(742)	
Comingled Filtered Tailings Facility (CFTF)												
Water run-off from the CFTF			37	37	55	55	55	55	37	37	37	
Water deficit at surface			(732)	(1,103)	(1,110)	(1,150)	(1,099)	(1,095)	(1,115)	(1,018)	(705)	
Water used and produced at the mine (underground)												
Water required			(119)	(218)	(244)	(244)	(244)	(244)	(244)	(194)	(180)	
Potential cumulative water production			925	1,365	1,541	1,629	1,717	1,761	1,849	1,717	1,717	
Net water deficit/(surplus) underground			805	1,147	1,297	1,385	1,473	1,517	1,605	1,523	1,537	
Total			74	44	187	235	374	422	490	505	832	
Recycle in process plant (Concentrator & Hydromet)			**53.6%**	53.6%	53.6%	53.6%	53.6%	53.6%	53.6%	53.6%	53.6%	

Further work is necessary to better forecast mine water production and treatment.

18.7.2 Temporary Water Storage Ponds

An estimated 1.4M gallons of water will need to be stored in water storage ponds, which amounts to roughly the water storage capacity of two Olympic sized swimming pools.

Consideration in future project phases 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

Nickel Powder Scenario

The inefficiency of the present supply chain from mine to battery cathode was articulated during Tesla's Battery Day on September 22, 2020 (**Battery Day**).

Below is an illustration of the traditional cathode process:

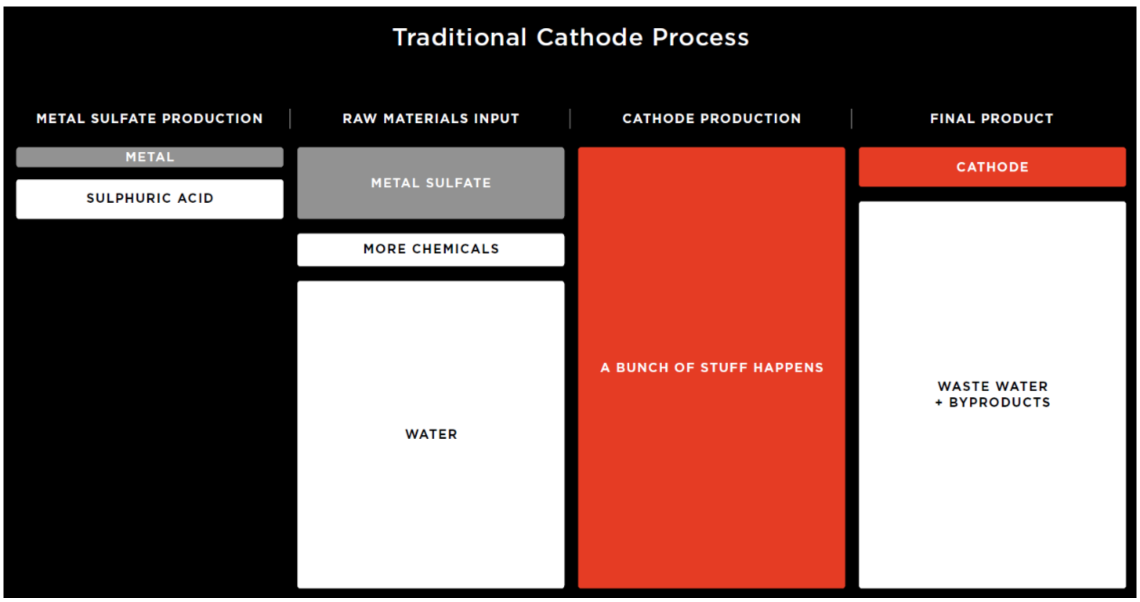


Figure 19-1: Traditional Cathode Process: Battery Day

Mined ore is typically crushed, concentrated, and shipped to a smelter, where it is converted to a Ni matte using a pyrometallurgical process. From there, the Ni matte is shipped to a hydrometallurgical refinery where Ni metal is produced. The metal is then shipped to a different hydrometallurgical facility that uses sulphuric acid to produce a metal sulphate (called a Ni sulphate). When making battery cathode material, the Ni sulphate is further processed through the addition of more chemicals and water, thus yielding additional by-products and wastewater.

"It's insanely complicated. It's a small world journey of, 'I am a nickel atom, what happens to me?' And it is crazy. You're going around the world three times, it's the equivalent of digging the ditch, filling the ditch and digging the ditch again, it's total madness basically."

Elon Musk, Battery Day, September 22, 2020

During Battery Day, Tesla proposed a new cathode process, which is illustrated below:

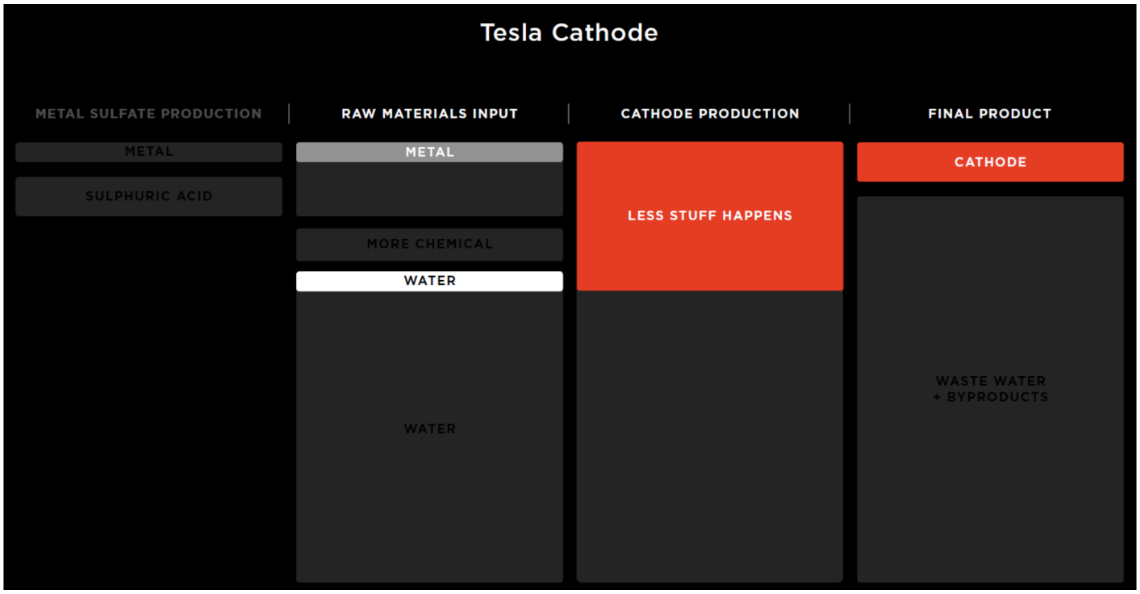


Figure 19-2: Proposed Tesla Cathode Process: Battery Day

Working from first principles, Tesla asked the question: *"How do we get from the nickel ore in the ground to the finished nickel product for a battery"* looking at the entire value chain. The solution is by directly consuming raw metal Ni powder, which dramatically simplifies the value chain, thereby eliminating intermediate steps and consequently unnecessary transportation. The same process could also potentially consume metal powder from recycled EV and grid storage batteries. The process enables both simpler processing and recycling.

During Battery Day, Tesla announced that they are going to start building their own cathode facility in North America, leveraging North American resources that exist for Ni and Li. By localizing the cathode supply chain and production, Tesla can significantly reduce miles traveled and the cost of the materials that end up in the cathode. To be clear, Tesla announced that cathode production would be part of the Tesla cell production plant.

"So it would just be basically raw materials coming from the mine".

Elon Musk, Battery Day, September 22, 2020

By 2026 the Tamarack Project is expected to be one of the only sources of domestically produced Ni in the USA.

The Ni powder market for EV battery cathode is one of three product markets that Talon is focussed on. Under the Ni Powder Scenario, Talon will produce Ni concentrates that are optimized for producing Ni powders, which could be used for EV battery cathode production.

As articulated during Battery Day, this approach has the potential to materially decrease the cost of Ni used in batteries, by eliminating intermediate processes and associated transportation, creating a win-win for both the battery cathode manufacturer and the mining company, leading to higher profitability for both parties. Further details are contained in Section 22 Economic Analysis of this PEA.

The process that is being considered for producing Ni powders is also being evaluated for producing refined Fe powders as a by-product which could potentially be used for Li Fe phosphate batteries.

When selling Ni concentrates to smelters, no credit or revenue is realized for Fe content, however a high Fe to Mg ratio is advantageous, which means the Ni concentrate producer (i.e., the mine) is not compensated for its Fe content. The Tamarack North Project concentrates have approximately four times as much Fe as Ni. The price of refined Fe powders is estimated at $1.50/lb (calculated as 5.50 $/kWh / 1000 x 600 Wh/kg / 2.2046) (source: Battery Day). From the same process Talon could therefore generate substantial additional revenues at very little additional cost. So far, Talon has not included any revenues derived from the sale of refined Fe powders in its economic forecasts, however flowsheet development is focussed on the production of both Ni and Fe powders to supply the Ni and Fe battery markets.

Nickel Sulphate Scenario

As shown in Figure 19-3 below, Talon has a unique opportunity in the United States to remove a myriad of steps and unnecessary transportation costs from the supply chain to produce Ni sulphates locally for use in battery cathode materials.

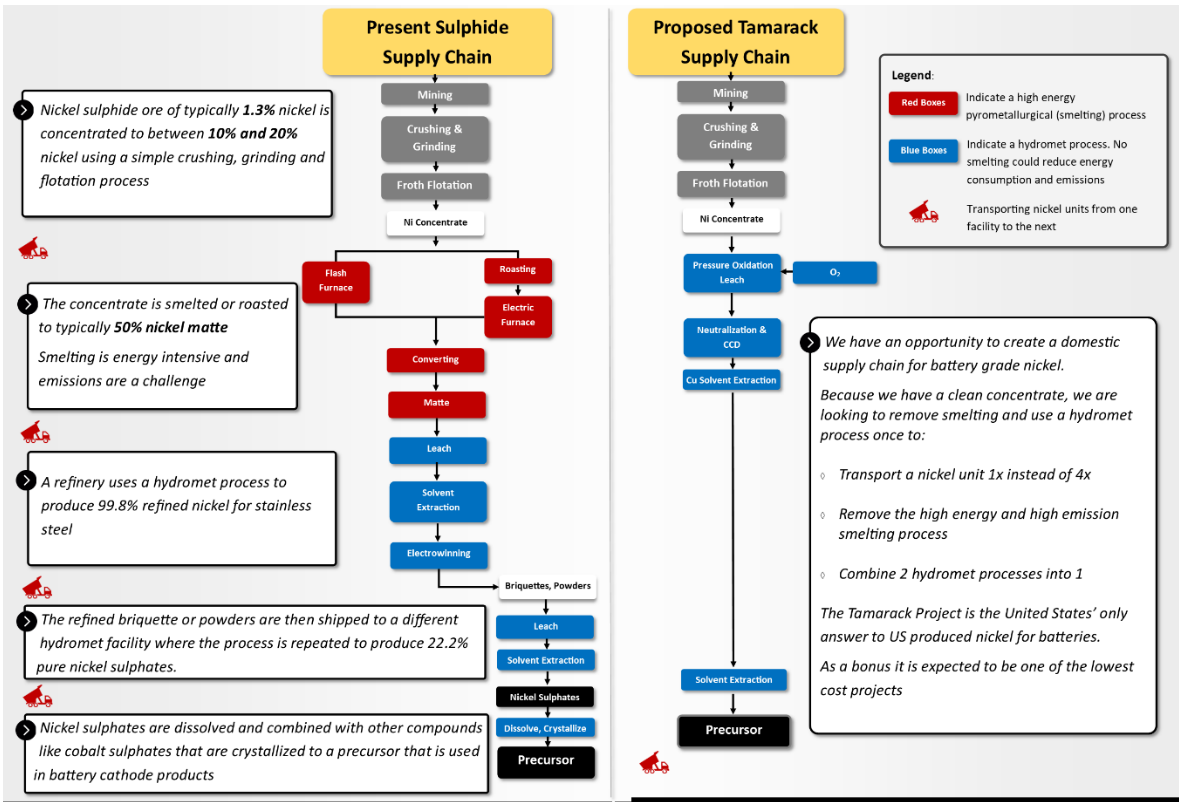


Figure 19-3: The Efficient Ni Sulphates Supply Chain Developed by Talon vs the Traditional (Present) Ni Sulphate Supply Chain

The present methodology of producing Ni sulphates is inefficient as it is an extension of the process that manufactures Ni for the stainless-steel market: The equivalent of digging the ditch, filling the ditch and digging the ditch again.

As explained in Section 13, Talon has successfully completed the first step of developing a flowsheet to produce Ni sulphates directly from Tamarack Ni concentrates at site. Talon will continue to develop the Ni Sulphate Scenario should there be a demand for this product when the Tamarack Project is in production, as Ni sulphate is currently the preferred Ni feedstock of battery manufacturers.

Nickel Concentrate Scenario

Wood Mackenzie forecasted a prolonged Ni concentrate deficit, as shown in Figure 19-4 below.

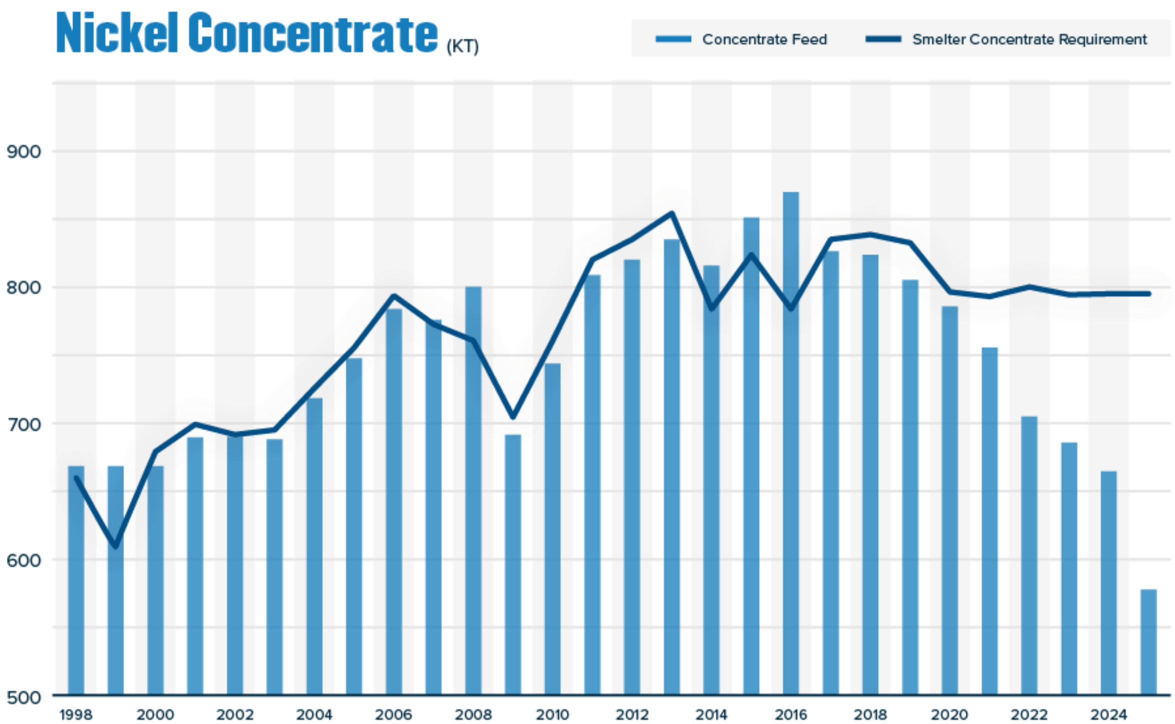


Figure 19-4: Forward-Looking Ni Concentrate Supply and Demand. (Wood Mackenzie, The Future of Nickel Production – Page 19, February 2015)

On a regional scale, the Tamarack Project is expected to play an important role in replacing declining Ni concentrates at Sudbury, Ontario, Canada smelters. Given that the Tamarack Project is on excellent infrastructure and is in proximity to North American Ni smelters, the Tamarack Project is expected to be an essential source of Ni concentrates for these North American smelters. Notwithstanding this, Talon has used the most recent publicly available economic terms for Ni concentrates in its economic analysis.

19.2 Premium for Green Nickel™

As outlined in Section 20 of this PEA, the Tamarack Project is ideally positioned to minimize its carbon footprint. Currently, most mining companies only consider their carbon footprint from ore to concentrate; however, going forward, it is expected that future customers, particularly within the EV market, will demand a full life cycle assessment (from mine to finished product) and all attempts to minimize carbon footprint be undertaken from mine to EV.

The Tamarack Project is endeavoring to have the lowest carbon footprint of all known Ni projects globally when producing Green Nickel^{TM}. This is expected to be possible because:

- Tamarack mineralization has high-grades compared to other projects currently in the exploration and development phases. This means more Ni will be produced per tonne of rock mined, therefore requiring less energy and less waste as well as a significantly smaller footprint than comparatively low-grade Ni sulphide projects or laterite Ni projects;
- Tamarack's electricity will be sourced from 100% renewable energy sources;
- Waste produced from the Tamarack Project will be dewatered and stored underground while remaining waste will be blended with development rock in a CFTF that will utilize pore space, thereby reducing overall mine footprint and avoiding the need for a tailings dam. Water captured will be recycled resulting in a substantial reduction of freshwater usage;
- The proposed CFTF at the Tamarack Project is also expected to be amenable to capture and store carbon (i.e., carbon sequestration). Testing is currently underway to determine quantities and deployment.

Note that a premium price for Green Nickel^{TM} has not been included in Talon's economic analysis; however, it is expected that there will be a bifurcation between Green Nickel^{TM} and other Ni products. It is therefore reasonably believed that Green Nickel^{TM} will ultimately command a premium price in the coming years.

19.3 Metal Prices

Base case metal prices are based on present London Metal Exchange (LME) prices and analyst consensus long-term "real" (i.e. without inflation) prices. These metals are sold on openly traded markets such as the 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 19-1: Assumed Real Metal Prices

Commodity	Unit	Low	Base Case	Incentive
Ni	US$/lb	$6.75	$8.00	$9.50
Cu	US$/lb	$2.75	$3.00	$3.50
Co	US$/lb	15.00	25.00	$30.00
Pt	US$/oz	$1,000	$1,000	$1,000
Pd	US$/oz	$1,000	$1,000	$1,000
Au	US$/oz	$1,300	$1,300	$1,300

Discounts to these prices have been assumed for both the Ni Powder and Ni Concentrate Scenarios, as both scenarios will require further processing to yield final products.

A premium of $1.25/lb to the LME Ni price has been applied for the Ni Sulphate Scenario as although it is expected to be produced directly from Ni concentrates at the Tamarack Project, Ni sulphate producers who use LME Ni as a feedstock must incur additional costs, over and above the LME Ni price and therefore are expected to charge a premium.

Incentive price is an estimated price believed to be required to incentivize new mines to be constructed. Selected incentive price is based on Talon research, however it may be higher or lower depending on numerous factors such as: inflation, future volume of demand for Ni, required return on capital and cost profile (both CAPEX and OPEX) of new projects that potentially could be constructed to meet a supply shortfall among other factors. Incentive price represents a possible price during periods of Ni demand growth such as due to the projected growth in the EV market.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

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 20.4 and Section 20.5. Throughout the regulatory 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 and analysis beyond the current environmental baseline studies will be completed to support these processes.

In accordance with Technical Report content expectations, this section addresses:

- Summary of results of environmental baseline studies and known environmental issues;
- Requirements and plans for waste and tailings disposal, site monitoring, and water management during operations and post mine closure;
- Project permitting requirements, status of applications, and requirements for performance or reclamation bonds;
- Potential social or community related requirements and plans for the project, status of any negotiations;
- Mine closure requirements and costs.

20.2 Environmental Studies – Summary of Results

Environmental baseline studies to characterize existing physical and biological conditions were initiated in 2006. A summary of environmental characterization from studies to date is provided in Table 20-1. Additional baseline and environmental engineering studies will be needed to support project siting, design, and environmental review and permitting efforts. There are no known environmental issues that could materially impact the ability to mine the resource.

Table 20-1: Environmental Studies – Summary of Results

Subject	Summary of Results
Hydrogeology	The hydrostratigraphic units in the Tamarack North Project area consist of unconsolidated glacial deposits with a typical thickness of >30 m 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 continue to be monitored regularly since data collection was initiated in 2008. The monitoring network includes 12 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 and 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 for purposes of completing a water balance (refer Section 18.7).
Hydrology	Surface water monitoring stations were established on a series of stream sections and lake sites surrounding the site and within the water basin. 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. Approximately 20 surface water monitoring locations were sampled with some adjustments in locations and stations since the program inception. Recent sampling includes measurement of flow three times a year, biannual measurement of field parameters and an annual collection of surface water samples at a subset of locations. The results from this data were used to complete three mine access trade off studies. (refer Section 16.8)
Geochemistry	Fourteen drill core samples have been analyzed. The samples were selected to comprise rock types, spatial distribution, and sulphur 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. An 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.

<table>
<tr><th>Subject</th><th>Summary of Results</th></tr>
<tr><td>Wetlands</td><td>
Multiple wetland delineation and evaluation studies have been conducted in and around the site since 2008. The studies were consistent with federal and local guidelines. Wetland boundaries were mapped and reviewed with local regulatory staff. A 120-acre study area was initially evaluated, then expanded to a 580-acre study area.
The conceptual site layout (refer Section 18.3) has been placed close to the ore body, straddling upland areas (57.37 acres) and wetland areas (178.64 acres). Alternative site layouts will be considered in accordance with regulatory requirements. Wetland surveys will need to be updated before the start of the formal environmental review process.
The wetland type by approximate area of the Project site is shown below:
<table>
<tr><th>Description</th><th>Acres</th></tr>
<tr><td>Upland</td><td>57.37</td></tr>
<tr><td>Sedge meadow</td><td>58.18</td></tr>
<tr><td>Alder thicket</td><td>29.36</td></tr>
<tr><td>Shrub carr</td><td>63.53</td></tr>
<tr><td>Deep marsh (in man-made pond)</td><td>2.38</td></tr>
<tr><td>Coniferous bog</td><td>25.04</td></tr>
<tr><td>Hardwood swamp</td><td>0.15</td></tr>
<tr><td>Total</td><td>236.01</td></tr>
</table>
</td></tr>
<tr><td>Vegetative Communities</td><td>
A survey of a 322-acre study area of vegetative communities was conducted in 2008 over most of 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 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 (Picea mariana). 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 (Phalaris arundinacea), 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 (2.8 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 (also man-made), 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 observed 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. RTE plant species or their potential
</td></tr>
</table>

<table>
<tr><th>Subject</th><th>Summary of Results</th></tr>
<tr><td></td><td>habitat was not observed. This survey will need to be updated before the start of the formal environmental review process with anticipated similar results.
The approximate area by Vegetative community type is shown below:
<table>
<tr><th>Description</th><th>Acres</th></tr>
<tr><td>Northern Poor Fen</td><td>153.5</td></tr>
<tr><td>Northern Wet-Mesic-Hardwood Forest</td><td>9</td></tr>
<tr><td>Northern Wet-Mesic Hardwood Forest/Northern Wet Ash Swamp</td><td>41</td></tr>
<tr><td>Northern Poor Conifer Swamp</td><td>38</td></tr>
<tr><td>Northern Alder Swamp</td><td>11.5</td></tr>
<tr><td>Man-made Pond</td><td>2</td></tr>
<tr><td>Mature Alder Thicket</td><td>1</td></tr>
<tr><td>Fallow Farm Fields/Pine Plantation</td><td>66</td></tr>
<tr><td>Total</td><td>322</td></tr>
</table></td></tr>
<tr><td>RTE Plant Species</td><td>A survey for RTE species was conducted in 2008. The survey study area covered much of the site area with the exception of a farm residence and adjacent buildings, and areas to the S and NW which were subsequently added to the site. The MDNR maintains a restricted geographic database of documented occurrences of threatened, endangered, and special concern species in Minnesota. An authorized database search was conducted for RTE species that have been known to occur within several miles of the study area. This information and Minnesota's entire published list (MDNR 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 is anticipated. These surveys will need to be updated before the start of formal environmental review process.</td></tr>
</table>

20.2.1 Requirements and Plans for Waste and Tailings Disposal

Waste generated from mining activities will be managed in a manner that protects the environment. Mine waste includes:

- Development rock generated from the excavating decline, levels, ramps, cross-cuts and drifts;
- Tailings generated in the production of Ni and Cu concentrates.

The approach to managing these wastes will follow industry practice and will address the mechanisms of potential environmental impacts. Collecting drainage and seepage water from stored material, mitigating dust, providing satisfactory reclamation for long term environmental protection are required in accordance with Minnesota nonferrous metallic mineral mining rules (Minnesota Rules, Chapter 6132).

The two waste streams will be managed as they are generated:

- Development rock will be generated over the full LOM, initially from rock excavated from the main access decline ramp originating from a portal on-site and then from on-going mine operations. A portion of the waste rock will be blended with paste fill and deposited into stopes as part of the backfill cycle. Where this isn't possible, the remainder will be trucked to surface through the main decline and stored on a lined Temporary Development Rock Storage Area (refer Section 18.4.10). This structure will be designed and managed such that all drainage water will be collected and either used in operations or treated to water quality limits before discharging to the environment. Dust will be minimized by a variety of practices and management techniques.
- Tailings will be generated once ore processing commences. Metallurgical testing has proven that HS tailings may be generated in addition to a LS tailings stream (refer Section 17.3.9). HS tailings and a portion of LS tailings will be used in the paste backfill recipe to fill stopes, with remaining LS tailings permanently managed in an engineered storage facility, the CFTF. 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 the HS tailings and a portion of LS tailings can be blended with cement and cured underground (refer Section 17.3.8).

The CFTF concept was developed to manage the aboveground waste storage design. A number of studies were commissioned to investigate the use of BAT to manage development rock and the remaining LS tailings (refer Section 18.6). These studies led to the development of an innovative CFTF, which offers significant environmental and operating advantages over separate tailings storage and development rock storage facilities. Advantages include:

- Reduced risk of failure as the facility stores minimal water;
- A major reduction in the waste facility footprint since stored water is minimal;

- 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 composite closure cover system. This will limit the amount of infiltration into the CFTF post-closure, reducing long-term water treatment needs and post-closure care liabilities;
- A significant reduction in fresh water requirements. Almost 90% of the water required by the processing plant will be recycled water;
- Potential to sequester carbon, thus reducing the project's overall carbon footprint.

Section 18.6 contains a more detailed discussion of the CFTF.

20.2.2 Other Environmental Considerations

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. Mine operations, therefore have the potential to impact water quality. Localized dewatering associated with near-surface mining, including construction of the portal or vent shaft collar, therefore has the potential to impact surface waters. Construction will employ water stoppage measures, which may include a grouted pipe canopy or freeze wall that would be built in advance of development. Employing one of these measures will limit water seepage into the mine workings during development and will later be replaced with ground support appropriate for conditions encountered during the working phase of mining operations.

In order to manage waste, 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;
- Potential sequestration of carbon.
- Site Monitoring

Monitoring will be required throughout operations and after mine closure. A groundwater and surface water monitoring network similar to the current baseline monitoring network is anticipated as a permit requirement. After mine closure, some or all of the monitoring network will remain, and monitoring will continue for an approved duration. Monitoring is overseen by the MDNR, with regular data submittals required. Changes in monitoring must be approved by the agency.

20.3 Water Management for Operations and Post Mine Closure

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 Section 18.7). Based on this water balance, the Tamarack North Project is expected to have a potentially negative water balance during the first three years of production, followed by potentially a positive water balance over the rest of the LOM (Section 18.7). Although it has been assumed that approximately 20% of underground water bearing features will be grouted, further hydrogeological work is needed to assess the practical requirements, in addition to the overall impacts of fracture sealing. Further work is required to assess potential water sources. Trade-off studies of Water Treatment Plant options will be conducted in future study.

During operations, water entering the site and potentially contacting reactive materials (called contact water) will need to be managed. Contact water sources include mine water inflow, water consumed during normal mine operations, precipitation within the contact area of the site, seepage through the Temporary Development Rock Storage Area and CFTF. The site is located in an area with relatively high precipitation including both rain and snow. During operations, all contact water will be collected and either used in ore processing, stored, or treated to permit limits prior to discharge to the environment.

Water management post mine closure will be greatly reduced compared to the management needs during operations. Sources of contact water will be removed with the completed reclamation; the mine excavation will be backfilled, and infrastructure and storage piles removed. The CFTF will be covered with a composite closure cover system that limits infiltration into the CFTF and associated drainage, reducing long-term water treatment needs.

20.4 Project Permitting Requirements

For nonferrous mining projects in Minnesota, permitting can proceed after or in parallel to environmental review. State-level and federal-level environmental review would be completed through an EIS process subject to the Minnesota Environmental Policy Act (MEPA) requirements for nonferrous mines, and the National Environmental Policy Act (NEPA).

NEPA/MEPA compliance may be achieved through one of the following paths. A Memorandum of Understanding (MOU) between the lead federal and state agencies could be entered to prepare a single, joint EIS that fulfills both federal and state lead agency requirements. Alternately, the lead federal agency and lead state agency may decide that a joint EIS is not appropriate and that each level of government would require its own EIS. In this scenario, two EIS documents would be prepared – one under NEPA and a second under MEPA.

The lead agencies would include the MDNR 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 one or more Native American Tribes.

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) and Adequacy Decision 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, nor have any permit applications been prepared.

20.4.1 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, timetable 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) under MEPA; the EAW provides a basis for preparation of a draft and subsequent final Scoping Decision Document (SDD). Under the Council on Environmental Quality NEPA guidelines, 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.

20.4.2 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 and by the lead federal agency in accordance with the agency's NEPA program. The Draft 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 and NEPA 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. It is expected that the RGU and lead federal agency would hold an informational meeting once the Draft EIS is released for public review.

20.4.3 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 and lead federal agency would discuss any responsible opposing views relating to scoped issues which were not adequately discussed in the Draft EIS, as appropriate, and would indicate the agency's responses to the views.

20.4.4 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.

20.5 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, significant permits are obtained through a process that includes a public comment period. Talon has not initiated permitting efforts to date.

Significant permits anticipated for the Tamarack North Project's include the Permit to Mine from the MDNR (Section 20.5.1), the NPDES / State Disposal System (SDS) Permits from the MPCA (refer to Section 20.5.2), the Air Permit from the MPCA (refer to Section 20.5.3), and Section 404 Permit from the USACE (Section 20.5.4).

Equally important are the local permitting and approvals. County and municipal units of government have building and zoning requirements to address. The local communities and their representatives will have opportunities to provide input, understand the Tamarack North Project, and negotiate on relevant issues. Talon has not defined social or community related requirements and plans for the Tamarack North Project. Formal negotiations and agreements with local communities for the Tamarack North Project have not been initiated. Community engagement is further described in Section 20.6.

Table 20-2: Potentially Applicable Permits and Approvals for the Tamarack North Project

Permit or Approval	Regulatory Agency	Regulatory Citation	Description
Scoping Environmental Assessment Worksheet (SEAW)	MDNR and Environmental Quality Board	MN Rules Chapter 4410.1000	Ch. 4410 rules require that any metallic mining project have an EIS prepared. The first step in the EIS process is to prepare a SEAW that will serve as the basis for the EIS scoping process. The MDNR will be designated as the RGU.
EIS	MDNR and Environmental Quality Board, USACE	MN Rules Chapter 4410.2000, 40 Code of Federal Regulations (CFR) Section 1500-1518	Completion of EIS is mandatory for development of a metallic mining facility. The EIS must include the requirements of Ch. 4410.2300. Information must include a description of the project and potential alternatives. The EIS must address the environmental, economic, employment and sociological impacts from the project and potential mitigation measures.
Environmental Assessment (EA)	USACE	40 CFR Section 1500-1518	NEPA is triggered by a proposal to any Federal agency for a major Federal action. CEQ broadly defines a major Federal action and includes: • adoption of official policy; • adoption of formal plans; • adoption of programs; • approval of specific projects. Data needs for an EA are similar to the SEAW required by the MDNR.
Section 106 Review. National Historic Preservation Act Compliance	USACE and SHPO	36 CFR Section 800	Section 106 of the NHPA requires federal agencies to account for the effects of their undertakings on historic properties and provide the Advisory Council on Historic Preservation a reasonable opportunity to comment.
Endangered Species Consultation	USFWS	Endangered Species Act, section 7(a)(1), (2)	Consultation by the USACE to determine ESA impacts of federal action on federally endangered species.
Permit to Mine – Non Ferrous Metallic Mining	MDNR	MN Rules Chapter 6132.1000 to 6132.5300	PTM application (Ch. 6132.1100) must be preceded by a mine waste conference per Ch. 6132.1000. Mine waste characterization must be included with the application along with information on the environmental setting, mining and reclamation plans and mining and reclamation maps. The application must include financial assurance and a plan for the first year of operation.
Dam Safety Permit	MDNR	MN Rules Chapter 6115.0300 to 6115.0520	Rules apply to structures that pose potential threat to public safety or property. Exemptions apply for dams less than 6 ft high and/or impoundments less than 15 acre-ft of water. Permits are required for new dams, to perform major maintenance, modify dam operation, or reconstruct a dam.

Permit or Approval	Regulatory Agency	Regulatory Citation	Description
Water Appropriation Permit	MDNR	MN Rules Chapter 6115.0600 to 6115.0810	Ch. 6115.0600 to 6115.0810 require a water use permit for withdrawal of more than 10,000 gallons per day or 1M gallons per year from waters of the state. Permit applications for metallic mining facilities must provide additional information per Ch. 6115.0720 including withdrawal plans, water use and storage, and disposal of waters of the state.
Public Waters Permit Program	MDNR and Aitkin County	MN Rules Chapter 6115.0160 to 6115.0280 MN Status 103G.245; Aitkin County	Applies to a facility that changes or diminishes the course, current, or cross section of public waters, entirely or partially within waters of the state, including filling, excavating or placing of materials in or on the beds of public waters. State permitting would also be required for work in delineated wetland areas.
Wetland Permits			
Section 404 of the Clean Water Act Permit	USACE	CWA 40 CFR 230; Section 404(b)(1)	Filling, excavating or placing materials into either waters of the state or waters of the US, will require wetlands permits. Depending on the classification of the wetland area, both state and federal jurisdictions could be triggered. For dredge and fill impacts to wetlands or waters of the US - USACE permit required. under the jurisdiction of the USACE.
Minnesota Wetland Conservation Act – Wetland Replacement Plan Approval	MDNR	MN Rules Chapters 8420 and 6132.5300	Wetland permitting necessary for impacts under a Permit to Mine require Wetland Replacement Plan as part of approval. This follow the WCA but is administered by the MDNR.
Section 401 Water Quality Certification or Waiver	MPCA	CWA 40 CFR 230: Section 401	For projects needing federal authorization with a discharge to the waters of the US. Required in conjunction with Section 404 Permit.
State Agency – MPCA Permits			
Individual NPDES and SDS Permits	MPCA	MN Rules Chapter 7001.1035	NPDES permit is required for wastewater discharge containing any pollutants to Waters of the US. The SDS may be applied to the project through the minimal seepage through lined facilities and the backfilled excavation.
NPDES Construction Storm Water Permit	MPCA	MN Rules Chapter 7090.2000 to 7090.2060	Likely a general permit will not be applicable. The permit requires implementation of Best Management Practices in accordance with a Storm Water Pollution Prevention Plan addressing the construction duration.
Air Permit	MPCA	MN Rules Chapter 7007	All facilities with sources of air emissions are required to obtain an air permit, unless it meets certain exemptions under Ch. 7007.0300. MPCA has several types of air permits that may apply depending on facility-wide emission estimates. A likely air permit type for this facility is a state synthetic minor permit.

Permit or Approval	Regulatory Agency	Regulatory Citation	Description
USEPA Requirements			
Underground Injection Control (UIC) Permit	USEPA	40 CFR 144.3. 146; MN Rules 4725.2050 for variance approval	Applicable to activities that could allow movement of fluid containing contaminants into an underground source of drinking water and that violate primary drinking water standards. If a Class V injection well is determined for mine backfill, a variance from the MN non-injection rule is needed from MDH.
Miscellaneous Requirements			
Permit for the Take of Threatened and Endangered Species Incidental to a Development Project	MDNR	MN Rules Chapter 6212.1800 to 6212.2300	A permit may be required if one removes, transports or sells any portion of a species designated as threatened, endangered or a species of special concern. A list of species is codified at Ch. 6134. Permits are available for certain conditions by the MDNR. Certain exemptions are also available, especially for situations where unknowing destruction takes place.
Hazardous Waste Generator License	MPCA	MN Rules Chapter 7045.0225	Hazardous waste generators must obtain a license for each generation site. Facilities generating more than 10 gallons of hazardous waste are subject to annual fees and reporting.
Aboveground Storage Tank Permit and Notification	MPCA	MN Rules Chapter 7001.4205	Facilities storing less than 1M gallons of industrial products need to notify MPCA of tanks storing 1,100 gallons or more.
Open Burning Permit	MDNR	MN Statute 88.16	Permission is required from the local MDNR forestry office or fire warden prior to starting any open fire.
Subsurface Sewage Treatment System (SSTS) Permit	Aitkin County	MN Rules Chapter 7082.500 and Aitkin County Individual Sewage Treatment System and Wastewater Ordinance No. 1	By state rule, Aitkin County has been delegated authority to issue licenses for SSTS in its jurisdiction. Counties are required to adopt ordinances in accordance with Ch. 7080 and 7081. Aitkin County has rules for obtaining a permit under its own ordinance.
Aquatic Vegetation Removal Permit	MDNR	MN Rules Chapter 6280	Permit is required for removal of emergent and submerged vegetation, but removal of the latter may be allowed if area is less than 2,500 square ft. Physical removal of floating leaf vegetation is allowed if channel is no greater than 15 ft wide.
License to Cross Public Lands and Waters	MDNR	MN Rules Chapter 6135	For installation of utility services (as defined in statute), across MDNR administered land and public waters.
Easement or Lease to Construct	MDNR	MN Statutes 84.63 and 84.631	If a road is constructed across state land, either a lease or easement must be issued by the state.
Road Across State Land	MDNR		Easements are issued for constructing and maintaining roads, while leases are issued for the long-term right to use or occupy land.

Permit or Approval	Regulatory Agency	Regulatory Citation	Description
Railroad Spur Installation	Surface Transportation Board (STB) and/or MN Department of Transportation	MN Rules Chapter 8830.2150 and 8830.9991	If a railroad spur is installed, approval may depend on a multifaceted jurisdictional test through the STB. An operating license may be needed if a railroad crossing includes necessity for a warning signal
Conditional Use Permit	Aitkin County	General Zoning Ordinance; Shoreland Management Ordinance; Floodplain Management Ordinance; Mining Ordinance	Pursuant to county ordinances for zoning, shoreland management, floodplain management, mining of metallic minerals and wetlands protection, a conditional use permit will be required.
Building Permits	Aitkin County	City Ordinance	Permits will be required for building construction, including compliance with various building codes.
Electrical Transmission	MN Public Utility Commission (MPUC)	MN Rules Chapter 7849 and 5350 MN Statutes 216B.42	If transmission upgrades are needed, coordination will be needed between transmission owners in the area. A site or route permit may be needed from the MPUC if it involves installation of power lines or substations at certain thresholds or re-routing of high transmission line to serve a single customer and will be located on property owned at least 80% by the customer.

20.5.1 Permit to Mine (MDNR)

Pursuant to Minnesota Rules, chapter 6132, a Permit to Mine will be required and signifies a legal approval issued by the commissioner of the MDNR to conduct a mining operation. The purpose of the MDNR 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 MDNR 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. Meeting these standards is 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 annually by the MDNR and can be adjusted at any time.

20.5.2 NPDES/SDS Permits (MPCA)

Permits with the intent to protect waters for uses such as drinking water, aquatic life, and recreation are required under the NPDES/SDS program (refer to Minnesota Statutes, Section 115.04 and Section 115.07), which is administered by the MPCA.

The NPDES program applies to wastewater and stormwater discharges from point sources into surface waters. Potential project discharges requiring permit coverage may include mine dewatering, wastewater, industrial stormwater, and construction stormwater. Pursuant to water quality standards of receiving and downstream waters, the individual NPDES/SDS permit establishes wastewater discharge effluent limitations and monitoring requirements. An anti-degradation analysis is required at the time of the application. The objective of the anti-degradation analysis is to demonstrate that the project will 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 avoided and minimized and only allowed for the purpose of important economic or social development.

Coverage for industrial stormwater discharges will likely be included with the individual NPDES/SDS permit. Additionally, a Construction Stormwater General Permit requires 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 must comply with the requirements of the permits by implementing best management practices described in the Stormwater Pollution Prevention Plans (SWPPPs).

The SDS program applies to the construction and operation of disposal systems, regardless of whether they discharge to surface waters and/or groundwater. A groundwater non-degradation analysis is required at the time of the application. The objective of the non-degradation analysis is to show that, to the maximum practicable extent, groundwater will be maintained at its natural quality. Where applicable, the analysis will document how a proposed change is justifiable for economic or social development and will not preclude appropriate present and future uses of the groundwater.

20.5.3 Air Permit (MPCA)

For most sizable mining facilities, an air permit will be needed before construction and operations can begin (40 CFR parts 52 and 70. Minnesota rules part 7007). State and federal programs have been established to protect air quality as it relates to human health and the environment. Applicability of federal and state air permitting rules will need to be evaluated for the project. The applicable rules depend on the quantity and type of pollutants emitted and the potentially affected air shed.

Production, design and operational details are incorporated into the permit and are the basis for the facility emission calculations. Regardless of the type of air permit needed, the environmental review and permit application must demonstrate how the facility will maintain compliance with applicable standards. Analyses include the Class I modeling evaluation of facility impacts on air quality 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, New Source Performance Standards, and National Emission Standards for Hazardous Air Pollutants (NESHAPS). Evaluations of Hg emissions, emission deposition on soils and local water bodies, and Air Emission Risk Analysis (AERA) will be documented.

Management approaches to airborne dust avoidance and mitigation are also included in the air permit application.

Air permits are specific to infrastructure, equipment selection, and operations descriptions. Changes in the design basis and selections necessitates a permit amendment evaluation that may require changes to the permit. Permit amendments can range from minor to major levels of effort and time.

20.5.4 Wetland Permitting

A permit from the USACE for the discharge of dredged or fill material to waters of the US is required under Section 404 of the Clean Water Act. Where project impacts to wetlands are unavoidable, compensation (i.e. the construction, restoration or enhancement of wetlands) is required as replacement for affected wetlands. In order to obtain the Section 404 Permit, a Section 401 Water Quality Certification is required from the MPCA.

The MDNR regulates impacts to wetlands and other waters listed on the state's Public Waters Inventory. The Minnesota Wetlands Conservation Act (WCA) requires a state permit for impacts to wetlands beyond those covered by USACE and/or public waters permitting. A Wetland Replacement Plan is required and would be 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 needs to be submitted to the USACE, MDNR, and Aitkin County under each entity's respective application process. Financial assurance could be part of the WCA permitting.

20.6 Social or Community Engagement

The primary agencies that Talon interacts with for the Tamarack Project include the MDNR, MPCA, MDH, USACE, and Aitkin County.

With the status of the Tamarack Project, Talon's current objective is to maintain open communication with agencies in order to keep regulators informed on project activities and future plans. The MDNR and Aitkin County managers have both attended site tours to learn about the Company and activities for moving the project forward. The MPCA has participated in Talon presentations and interactions with MDH and ACE primarily fall into communications regarding exploration work permits. There have been no issues to date with these agencies approving Talon's work permits. All primary agencies receive Talon's quarterly newsletter updates as an additional avenue to stay up to date on the project.

Talon maintains engagement with the community by attending local meetings, sending out a community newsletter, and maintaining an open-door policy for anyone to stop by and learn more about the project. This communication is proactive, providing information before, during, and after any increased activity or changes to the project. Talon also strives to participate in local events and support local organizations. This engagement will increase as the project reaches the PFS stage.

20.6.1 Discussion of Mine Closure

Talon's strategy is to engage with stakeholders with the end in mind. A robust closure plan that engages stakeholders is therefore ongoing. 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. A detailed closure plan will be developed in future study and will align with state PTM and reclamation requirements.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Nickel Powder Scenario and Nickel Concentrate Scenario

The total estimated capital cost of the Ni Powder Scenario or the Ni Concentrate Scenario is US$394.99M and is summarized in Table 21-1, of which US$315.80M is the initial cost required during the first three years, including the first production year. The amounts include indirect costs and contingency which are detailed in the various sections further below.

Table 21-1: Tamarack North Project CAPEX Summary - Ni Powder or Ni Concentrate Scenarios

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Mine	$130.15M	$70.32M	$200.47M
Process and Surface Facilities	$167.51M	$ 22.01M	$189.51M
Closure Costs other than CFTF	-	$10.00M	$10.00M
Salvage Value of Mill	-	($5.00M)	($5.00M)
Sub Total*	**$297.66M**	**$97.33M**	**$394.99M**
Working Capital	$18.15M	($18.15M)	-
Total*	**$315.80M**	**$79.18M**	**$394.99M**

** May not total due to rounding*

All costs are estimated in fourth quarter 2020 US dollars, without provision for inflation or escalation.

Nickel Sulphate Scenario

The total estimated capital cost of the Ni Sulphate Scenario is US$646.44M and is summarized in Table 21-2, of which US$552.49M is the initial cost required during the first three years, including the first production year. The amounts include indirect costs and contingency which are detailed in the various sections further below.

Table 21-2: Tamarack North Project CAPEX Summary - Ni Sulphate Scenario

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Mine	$130.15M	$70.32M	$200.47M
Process and Surface Facilities	$390.56M	$50.41M	$440.97M
Closure Costs other than CFTF	-	$10.00M	$10.00M
Salvage Value of Mill	-	($5.00M)	($5.00M)
Sub Total*	**$520.71M**	**$125.73M**	**$646.44M**
Working Capital	$31.90M	($31.90M)	-
Total*	**$552.61M**	**$93.83M**	**$646.44M**

** May not total due to rounding*

All costs are estimated in fourth quarter 2020 US dollars, without provision for inflation or escalation.

21.1.1 Mine Capital Costs

The estimated initial mine CAPEX comprises decline development, vent shaft development and associated infrastructure, mine surface facilities, mobile equipment, underground development and services.

It is assumed that the mine will be developed and operated with mine contractors, but equipment will be owner-supplied, therefore mobile equipment purchases are included in the CAPEX.

Sustaining CAPEX includes ongoing development as well as mobile equipment rebuild and replacement costs over the LOM.

Table 21-3: Mine CAPEX Summary – All Scenarios

Area	**Initial Cost (US$)**	**Sustaining Cost (US$)**	**Total Cost (US$)**
Primary Access	$21.15M	-	$21.15M
Underground Development	$ 45.27 M	$7.53M	$52.79M
Mobile Equipment	$28.79M	$49.26M	$78.05M
Fixed Equipment and Services	$18.62M	$9.04M	$27.66M
Sub-total	**$113.83M**	**$65.83M**	**$179.66M**
Indirect Costs	$9.01M	$3.67M	$12.68M
Contingency	$7.31M	$0.83M	$8.13M
Total *	**$130.15M**	**$70.32M**	**$200.47M**

** May not total due to rounding*

21.1.2 Process and Surface Facilities Capital Costs

The estimated process and surface facilities CAPEX comprises the process plant, plant infrastructure (concentrator building, electrical substation and distribution, reverse osmosis plant, water supply system, and fire protection), CFTF and paste backfill, and other surface facilities (administrative office, maintenance shops, mine change house, surface warehouses, garages, security, parking lots). The CFTF is built-up in two stages over the mine life (first phase during construction and expansion phase in year four), followed by closure in the final year of operation.

Table 21-4: Process and Surface Facilities CAPEX Summary – Ni Powder Scenario or Ni Concentrate Scenario

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Process Plant	$48.41M	$9.68M	$58.09M
Plant Infrastructure	$30.64M	-	$30.64M
CFTF and Paste Backfill	$21.85M	$9.72M	$31.58M
General Plant Services	$6.05M	-	$6.05M
Other Surface Facilities	$1.06M	-	$1.06M
Sub-total	**$108.01M**	**$19.41M**	**$127.42M**
Indirect Costs	$28.65M	-	$28.65M
Contingency	$30.84M	$2.60	$33.45M
Total *	**$167.51M**	**$22.01M**	**$189.51M**

** May not total due to rounding*

Table 21-5: Process and Surface Facilities CAPEX Summary – Ni Sulphate Scenario

Area	Initial Cost (US$)	Sustaining Cost (US$)	Total Cost (US$)
Process Plant	$194.06M	$38.81M	$232.87M
Plant Infrastructure	$29.62M	-	$29.62M
CFTF and Paste Backfill	$22.58M	$9.00M	$31.58M
General Plant Services	$5.72M	-	$5.72M
Other Surface Facilities	$0.79M	-	$0.79M
Sub-total	**$252.76M**	**$47.81M**	**$300.57M**
Indirect Costs	$63.64M	-	$63.64M
Contingency	$74.16M	$2.60M	$76.77M
Total *	**$390.56M**	**$50.41M**	**$440.97M**

** May not total due to rounding*

Other Capital Costs

Closure costs excluding the CFTF are estimated at US$10M and relate to the removal of the process plant and other surface infrastructure facilities as well as land reclamation and other general site closure costs. Closure costs associated with the CFTF, such as a closure cover on the CFTF facility and drainage equipment are included in Process and Surface Facilities Capital Costs in the amount of US$6.65M including contingency.

As the mill, equipment and facilities will have been in operation for approximately 9 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$5M at the end of the mine life.

21.2 Operating Costs

The OPEX for the Tamarack North Project at the processing plant design capacity of 3,600 tpd is summarized in Table 21-6 below.

Table 21-6: Operating Costs in US$/t of Mill Feed

Cost Category	Operating Cost (US$/t of Mill Feed)		
	Ni Powder Scenario	Ni Sulphate Scenario	Ni Concentrate Scenario
Mining	$27.49	$27.49	$27.49
Processing (milling/concentrating)	$14.25	$14.25	$14.25
Hydrometallurgical Refining	-	$26.68	-
Product Handling, Transportation, Losses, and Insurance	$1.90	$2.22	$10.29
CFTF	$0.75	$0.75	$0.75
G&A	$3.76	$4.60	$3.76
Total OPEX *	**$48.15**	**$75.99**	**$56.54**

** May not total due to rounding*

21.2.1 Mine Operating Costs

The estimated mine OPEX average over the mine life is $27.49/t of mill feed or material mined. This average cost includes direct mining costs, including contract mining labour (the CAPEX includes purchase of mobile mining equipment) in addition to backfill, stope development (in waste), ore flow, ventilation, and mine services costs. Costs were derived from guidance from OEMs, and recent contacting prices obtained by DRA from reputable mining contractors.

The updated mine OPEX costs for this PEA are considerably less than the previous study. This is largely attributed to the realized efficiencies of larger stopes (15 m W x 25 m H x 30 m L in this January 2021 PEA, vs 7.5 m W x 15 m H x 15 m L in the March 2020 PEA), efficiency gains for utilizing a road header for the development of the sill drifts, relocation of key infrastructure on surface resulting from the decline ramp and utilizing a vertical conveyor as opposed to skip hoisting as the primary ore flow circuit).

A summary of the mine OPEX split by mining method is shown in Table 21-7 below.

Table 21-7: Mine Operating Cost Summary

Mining Method	Tonnes Mined	Percentage of Total	Mine Operating Cost (US$/t of mill feed)	Direct Cost	Indirect Cost	Total Cost
Drift & Fill	779,382	7.2%	$39.95	$18.58M	$12.55M	$31.13M
Long hole Stoping	8,456,397	78.6%	$22.86	$57.08M	$136.20M	$193.28M
Mineralized Material Development	1,523,017	14.2%	$46.88	$46.86M	$24.53M	$71.39M
Totals	10,758,796	100.0%	$27.49 (weighted average)	$122.52M	$173.28M	$295.80M

Mine sustaining CAPEX is an additional $70.32M over the LOM, equivalent to $6.54/t mined or of mill feed. The two pre-production years and the first operating years were classified as initial CAPEX.

21.2.2 Process Plant Operating Costs

A breakdown of the processing operating costs is provided in Table 21-8 and further details for the basis of the estimates are provided in the following sections.

Table 21-8: Processing Operating Cost Breakdown

Cost Category	All Scenarios		
	Annual Cost US$/year	Unit Cost US$/t of Mill Feed	Percentage of Total
Labour	$6,615,000	$5.03	35.3
Electrical Power	$4,370,921	$3.33	23.3
Reagents	$2,753,487	$2.10	14.7
Grinding Media	$1,996,052	$1.52	10.7
Consumables	$2,360,319	$1.80	12.6
Spares & Miscellaneous	$629,418	$0.48	3.4
Total Processing Costs*	$18,725,198	$14.25	100.0

** May not total due to rounding*

A breakdown of the hydrometallurgical refining costs for the hydrometallurgical facility is provided in Table 21-9 and further details of the basis of the estimates are provided in the following sections.

Table 21-9: Hydrometallurgical Refining Operating Cost Breakdown

Cost Category	Annual Cost US$/year	Unit Cost US$/t of Hydromet Feed	Unit Cost US$/t of Mill Feed	Percentage of Total
Labour	$8,896,500	$71.67	$7.44	27.9%
Electrical Power	$3,153,600	$25.41	$2.64	9.9%
Reagents	$11,575,183	$93.25	$9.68	36.3%
Consumables	$5,906,321	$47.58	$4.94	18.5%
Spares & Miscellaneous	$2,362,528	$19.03	$1.98	7.4%
Total Processing Costs *	$31,894,132	$256.94	$26.68	100.0%

**May not total due to rounding*

<u>Methodology</u>

The operating costs are estimated based on the actual projected annual tonnes of Ni concentrate generated from the concentrator that feeds into the hydrometallurgical refinery. The operating cost estimates considered pertinent metallurgical results and mass balance outputs.

21.2.2.1 Labour

Staffing is established based on the resource requirements of plants with comparable size and unit operations. Staffing requirements were differentiated between mine and mill operation, maintenance, technical, and administration.

21.2.2.2 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 of the concentrator was determined by summation of the connected power of all major mechanical equipment plus 20% for pumps and plant services. The total power draw was estimated at 85% of the total connected power.

The total electrical power draw of the hydrometallurgical plant is estimated at 6.0 MW.

21.2.2.3 Reagents

The reagent dosages were established using the metallurgical data that was developed in recent test programs at XPS in Sudbury and SGS in Lakefield. The reagents costs were then calculated using recent prices from reputable North American reagent suppliers.

In cases where no test results were available such as the diluent and extractant consumption in the SX circuit, data from comparable plants were used to estimate the reagent costs.

21.2.2.4 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.

21.2.2.5 Consumables, Spares and Miscellaneous

The concentrator consumables, which include all items except reagents and grinding media, were calculated as 15% of the labour, electrical, reagents, and grinding media costs. The concentrator allowance for spares and miscellaneous items was determined as 4% of the labour, electrical, reagents, and grinding media costs.

The maintenance consumables for the hydrometallurgical plant were estimated as 25% of labour, electrical, and reagent costs. The allowance for spares and miscellaneous items for the hydrometallurgical plant was established at 10% of the labour, electrical, and reagent costs.

21.2.3 Product Handling, Transportation, Losses, and Insurance

This item consists of US$1.90/t of mineralized material for the Ni Powder Scenario, US$2.22/t of mineralized material for the Ni Sulphate Scenario and US$10.29/t of mineralized material for the Ni Concentrate Scenario, related to the handling of concentrates or Ni sulphate, transportation, losses and insurance costs, with the exception that Ni concentrates in the Ni Powder Scenario and Ni sulphate in the Ni Sulphate Scenario is sold on an FOB basis.

21.2.4 Co-disposed Filtered Tailings Facility (CFTF)

Operating expenses for the CFTF consist of labour, fuel, maintenance and perimeter embankment filters.

21.2.5 General and Administrative (G&A)

The G&A costs are estimated at US$4.5M per year for the Ni Powder and Ni Concentrate Scenarios and US$5.5M per year for the Ni Sulphate Scenario, and include rent, utilities, insurance, office, property taxes, mineral leases and salaries related to management, accounting, human resources, procurement, environment, safety and administration.

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 objective of the study is to determine the viability of the proposed facilities to mine and process the Tamarack North Project mineralized material. In order to do this, the cash flow arising from the base case was forecast, enabling a computation of NPV and IRR. The sensitivity of this NPV and IRR to changes in the base case assumptions is then examined.

Three scenarios, as detailed in Section 19 “Market Studies and Contracts”, were modelled:

	Scenario	Description
1	**Nickel Powder Scenario**	Nickel concentrates produced at site and thereafter used to produce refined nickel powder by a third party for the EV market
2	**Nickel Sulphate Scenario**	Nickel concentrates from the project are refined at site in a hydrometallurgical refinery to produce nickel sulphates which are sold to the EV market
3	**Nickel Concentrate Scenario**	Nickel concentrates produced at site are transported and sold to a smelter, who in turn transports a product to a refinery to produce LME-grade nickel primarily for the stainless steel market

Under all the three scenarios, the Cu concentrate produced from the project is shipped to a traditional Cu smelter. The CAPEX of the Ni Sulphate Scenario is greater than that of the Ni Powder and Ni Concentrate Scenarios, which corresponds with higher revenues and OPEX as a result of additional hydrometallurgical refining processing costs associated with the hydrometallurgical refinery. Hydrometallurgical refining yields a value-added product, namely Ni sulphate, that sells for a premium. The Cu produced by the hydrometallurgical refinery is combined with the Cu concentrate.

The financial model is based on the results of the PEA which is preliminary in nature and includes inferred resources that are considered too speculative geologically to have the economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

22.2 Economic, Taxation and Royalty Assumptions

22.2.1 Exchange Rates

All cost estimates are forecast in US dollars and metal prices are in US dollars, therefore no exchange rate is required. All results are expressed in US dollars.

22.2.2 Inflation Rates and Escalation

All cost estimates are prepared using constant, fourth quarter 2020 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 is determined based on many factors relating to risk and return including:

- location and country risk;
- characteristics of the project, such as:
  - access to infrastructure;
  - availability of labour;
  - expected position on the global cost curve;
  - project size;
  - project complexity such as risks associated with mining, processing and capital cost overruns.
- that the forecast is in “real” as opposed to nominal dollars;
- availability and cost of capital such as debt financing and off-take financing;
- corporate income tax rates; and
- the metal prices used in the forecast.

Based on the above, DRA has assumed 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 are discussed in Section 19. Price assumptions presented in Section 19 are replicated in Table 22-1 below, 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	$3.50
Co	US$/lb	15.00	25.00	$30.00
Pt	US$/oz	$1,000	$1,000	$1,000
Pd	US$/oz	$1,000	$1,000	$1,000
Au	US$/oz	$1,300	$1,300	$1,300

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 mineralized material. Since the mineralized material value used to determine the royalty is not updated annually, a Net Revenue Inflation Adjustment (NRIA) must be deducted from a mine's NSR per tonne (imperial tonne) before transportation costs to arrive at an adjusted NSR per tonne, which is then compared to a list of values per tonne and royalties provided in a table by the state of Minnesota.

The NRIA is calculated based on the US producer price inflation index (PPI) of the current period and the PPI for November 1994 ("Base Index") which had a value of 121.5, using the following formula:

- (PPI 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 2020 is approximately US$49.21. The Minnesota royalty rates mapped to present-day mineralized material value less the NRIA (i.e. a 1994 ore value) are shown in Table 22-2.

Table 22-2: Minnesota Royalty Rates to be Applied to Present-day NSR per tonne Less the NRIA

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%		

Under the Ni Powder Scenario, the average net revenue per (imperial) tonne of ore over the LOM is US$182/t before the NRIA and US$130/t after the NRIA, resulting in an average royalty of approximately 5.6%. Under the Ni Sulphate Scenario, the average net revenue per (imperial) tonne of ore over the LOM is US$197/t before the NRIA and US$145/t after the NRIA, resulting in an average royalty of approximately 6.5%. Under the Ni Concentrate Scenario, the average net revenue per (imperial) tonne of ore over the LOM is US$182/t before the NRIA and US$129/t after the NRIA, resulting in an average royalty of approximately 5.6%.

Private royalties of 1.86% on NSR were also included, which primarily reflects a 1.85% NSR royalty to a subsidiary of Triple Flag Mining Finance Bermuda Ltd. ("Triple Flag Royalty").

The Triple Flag Royalty is 3.5% of NSR and will be based on Talon's participating interest in the Tamarack Project, except (i) where Talon's interest reduces below 17.56%, in which case it will be paid assuming Talon's interest is unchanged at 17.56% or (ii) where Talon has vested at 51% and Talon's interest reduces below 51%, in which case it will be paid assuming Talon's interest is unchanged at 51%; or (iii) where Talon has vested at 60% and Talon's interest reduces below 60%, in which case it will be paid assuming Talon's interest is unchanged at 60%.

The royalty agreement contains a one-time put right pursuant to which the Royalty Holder has an option, exercisable within 10 calendar days of March 7, 2022, to cause Talon to repurchase the entire NSR royalty for a cash payment of US$8.6M ("the Royalty Put

Option”). The Royalty Put Option may be accelerated in a number of circumstances, including upon an event of default as defined under the Royalty Agreement. In the event the Royalty Holder does not exercise the one-time put right, Talon has a one-time option to reduce the percentage of the NSR royalty by 1.65% from 3.5% to 1.85% in exchange for cash in the amount of US$4.5M.

Given that the US$4.5M buy-down of the royalty is expected to occur in 2022 and is prior to the construction period of the Tamarack North Project and outside of the forecast period contemplated in the LOM model, a 1.85% royalty has been applied in the financial model. Were a 3.5% royalty included instead of the 1.85% royalty, the after-tax NPV would decrease by approximately 2.8% to 3.6% and the after-tax IRR would decrease by approximately 0.6% to 0.7%.

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. The combined federal and state tax rate is therefore 22.94%.

22.3 Technical Assumptions

22.3.1 Mine Production Schedule

The following graph illustrates the annual mining rate by mining method. The peak mining rate is 3,728 tpd in year four and the average during years 2 to 8 is 3,544 tpd.

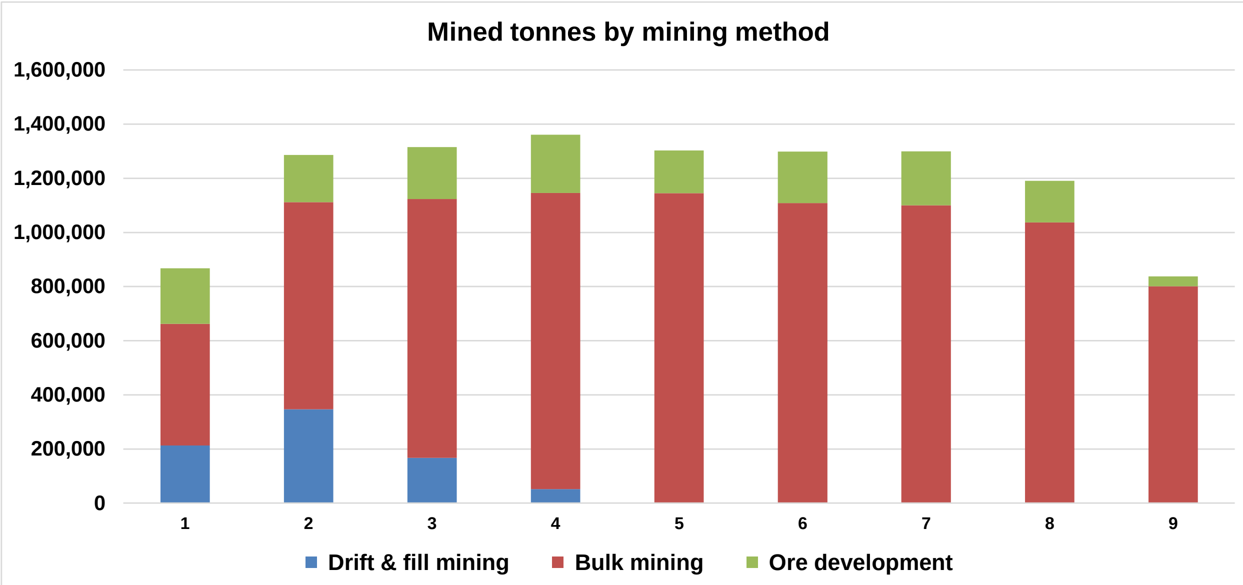


Figure 22-1: Mining Production Schedule (by mining method, tonnes and years)

22.3.2 Processing Schedule

Processing occurs concurrently with mining. The peak processing rate of 3,728 tpd occurs in year four which is consistent with the capacity of the concentrator of 3,600 tpd plus a 20% design factor. The average milling rate during years two to eight is 3,544 tpd.

Figure 22-2 illustrates the milled tonnes and grade profile over the LOM. The equivalent grades are calculated using the base case metal prices in Table 22-1.

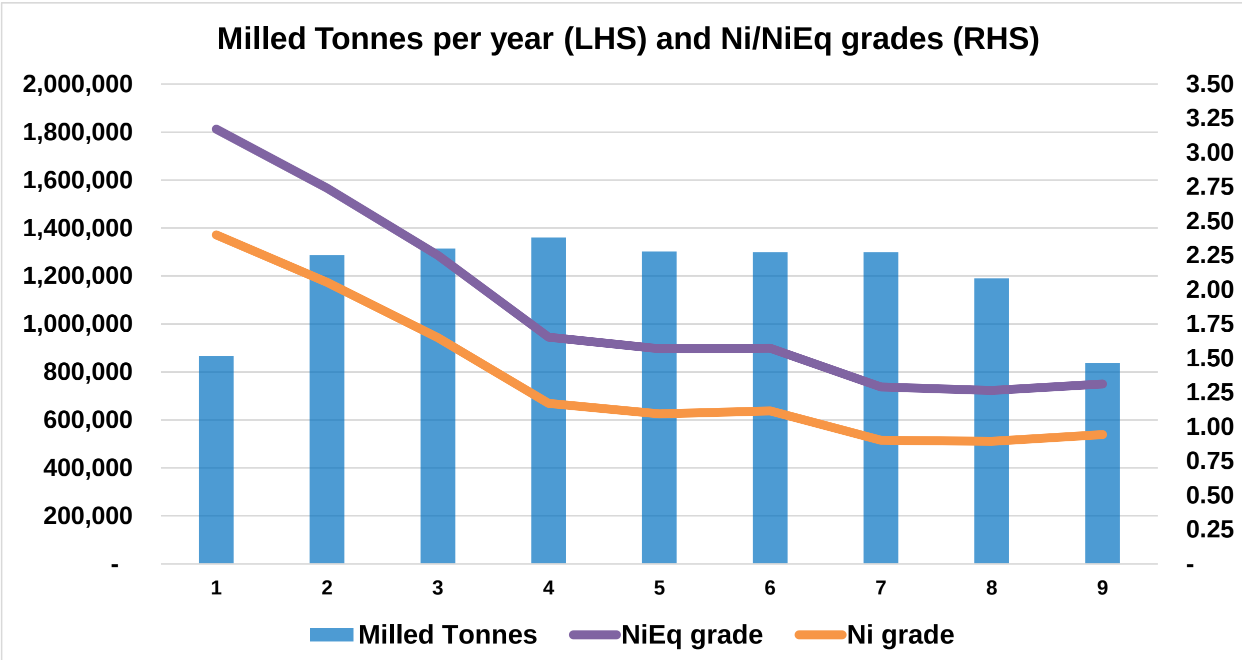


Figure 22-2: LOM Milled Tonnes and Grade Profile

22.3.3 Net Smelter Return (NSR)

In both the Ni Powder Scenario and the Ni Sulphate Scenario, the products sold, namely Ni concentrate and Ni sulphate respectively, are sold on an FOB basis into the EV battery supply chain. In the Ni Concentrate Scenario, Ni concentrate will be sold to a smelter.

The Cu concentrate will be sold directly to smelters or to traders in North America, Europe, and Asia in all scenarios.

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. The Ni sulphate produced under the Ni Sulphate Scenario is expected to be battery-grade material.

DRA has reviewed the smelter terms, the terms for the payment of metal, and the deductions for treatment and refining, and in the case of the Ni Sulphate Scenario has additionally reviewed the Ni sulphate premium for Ni contained in sulphate form. DRA is of the opinion that the smelter and product contract terms, as applied in the economic model, are reasonable.

Cu concentrate handling and transport costs were estimated to be US$82.00/wmt of Cu concentrate. As for the Ni concentrate, transportation costs were only assumed in the Ni Concentrate Scenario (US$72.00/wmt) because under the Ni Concentrate Scenario, Ni concentrate is shipped to a smelter. In the Ni Powder and Ni Sulphate Scenarios, the products are sold on an FOB basis. An additional charge of 0.16% was included to cover insurance and losses where concentrate is transported which includes the Cu concentrate in all scenarios and the Ni concentrate in the Ni Concentrate Scenario.

Table 22-4: NSR and NSR After Royalties and Transportation – Payable Ni Basis

	Nickel Powder Scenario			Nickel Sulphate Scenario			Nickel Concentrate Scenario		
	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni
Payable nickel revenue	1,666.4	154.89	8.00	1,978.9	183.93	8.00	1,624.7	151.02	8.00
Nickel sulphate premium	-	-	-	309.2	28.74	1.25	-	-	-
Payable by-product revenue	524.5	48.75	2.52	598.2	55.60	2.42	561.3	52.17	2.76
Total payable revenue	2,190.9	203.64	10.52	2,886.2	268.27	11.67	2,186.1	203.19	10.76
Treatment and refining charges for copper concentrates	28.1	2.61	0.13	32.5	3.02	0.13	28.1	2.61	0.14
Insurance and losses	0.6	0.06	0.00	0.7	0.07	0.00	3.5	0.32	0.02
Net smelter return revenue	2,162.2	200.97	10.38	2,853.0	265.18	11.53	2,154.5	200.26	10.61
royalties	160.7	14.94	0.77	193.1	17.95	0.78	153.1	14.23	0.75
Product handling and transportation costs	19.8	1.84	0.10	23.1	2.15	0.09	107.2	9.97	0.53
Net smelter return revenue after royalties and transportation costs	1,981.7	184.19	9.51	2,636.7	245.08	10.66	1,894.2	176.06	9.33

Using the Base Case metal price assumptions, the contribution of each metal to the NSR over the LOM is shown in Figure 22-3.

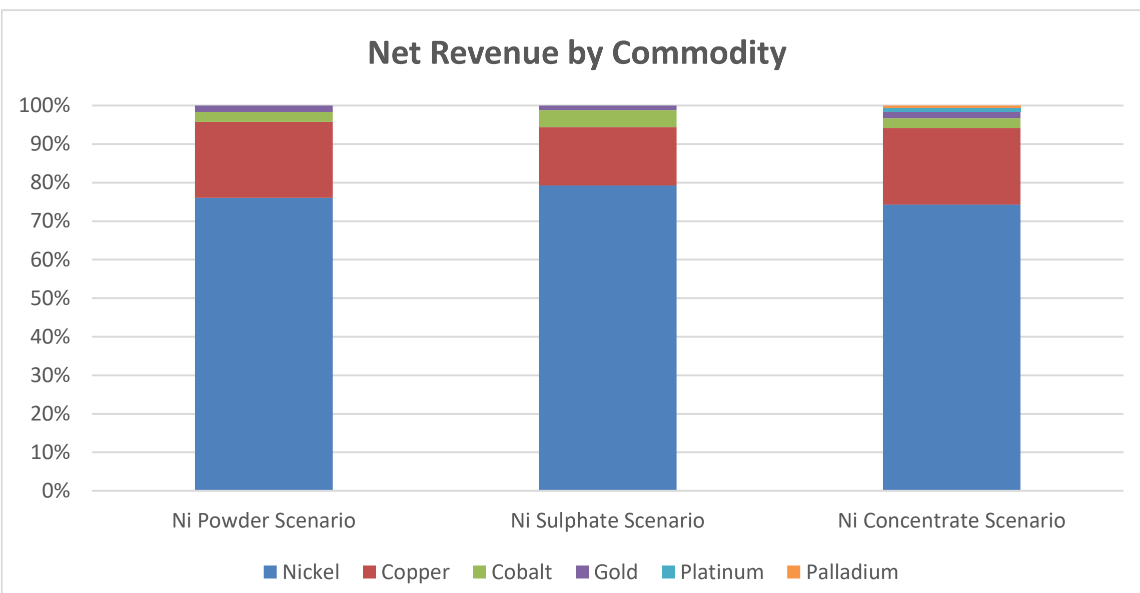


Figure 22-3: Contributions of Metals to NSR

22.3.4 Operating Costs

Direct (on-site) operating costs average $41.02/t milled over the LOM for the Ni Powder Scenario or the Ni Concentrate Scenario, and $66.26 for the Ni Sulphate Scenario as further detailed in Table 22-5.

Table 22-5: On-site Costs over LOM – Payable Ni Basis

	Nickel Powder Scenario			Nickel Sulphate Scenario			Nickel Concentrate Scenario		
	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni
On-Site Costs									
Mining Costs	295.8	27.49	1.42	295.8	27.49	1.20	295.8	27.49	1.46
Processing Costs	153.3	14.25	0.74	153.3	14.25	0.62	153.3	14.25	0.75
Hydromet Refining Costs	-	-	-	287.0	26.68	1.16	-	-	-
Co-Disposed Filtered Tailings Facility	8.0	0.75	0.04	8.0	0.75	0.03	8.0	0.75	0.04
General & Administrative Costs	40.5	3.76	0.19	49.5	4.60	0.20	40.5	3.76	0.20
Total On-Site Costs	**497.7**	**46.26**	**2.39**	**793.7**	**73.77**	**3.21**	**497.7**	**46.26**	**2.45**

Figure 22-4 provides a breakdown of operating costs by year over the LOM.

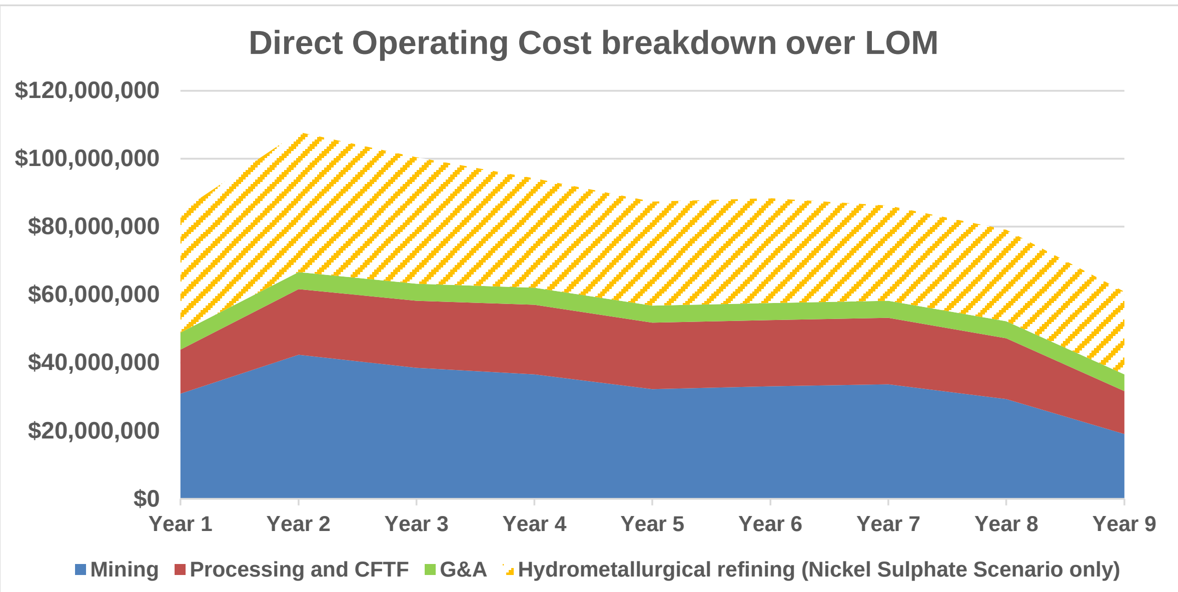


Figure 22-4: Direct Operating Cost Breakdown over LOM

22.3.5 Capital Costs

Initial cost for the Ni Powder Scenario or Ni Concentrate Scenario is estimated to be $315.80M while for the Ni Sulphate Scenario, initial cost is estimated to be $552.61M, reflecting the incremental cost of the hydrometallurgical refining facility. The higher sustaining cost of the Ni Sulphate Scenario also reflects the inclusion of the hydrometallurgical refining facility. Table 22-6 compares the initial and sustaining capital costs of the three scenarios.

Table 22-6: Capital Cost Summary

US$ millions	Nickel Powder Scenario or Nickel Concentrate Scenario			Nickel Sulphate Scenario		
	Initial Cost (US$)	**Sustaining Cost (US$)**	**Total Cost (US$)**	**Initial Cost (US$)**	**Sustaining Cost (US$)**	**Total Cost (US$)**
Mine	130.15	70.32	200.47	130.15	70.32	200.47
Process and Surface Facilities	167.51	22.01	189.51	390.56	50.41	440.97
Salvage Value of Mill	-	(5.00)	(5.00)	-	(5.00)	(5.00)
Closure Costs other than CFTF	-	10.00	10.00	-	10.00	10.00
Sub Total	**297.66**	**97.33**	**394.99**	**520.71**	**125.73**	**646.44**
Working Capital	18.15	(18.15)	-	31.90	(31.90)	-
Total	**315.80**	**79.18**	**394.99**	**552.61**	**93.83**	**646.44**

Figure 22-5 illustrates the CAPEX including working capital by year. Initial Cost is CAPEX during years -2, -1 and 1 and initial working capital.

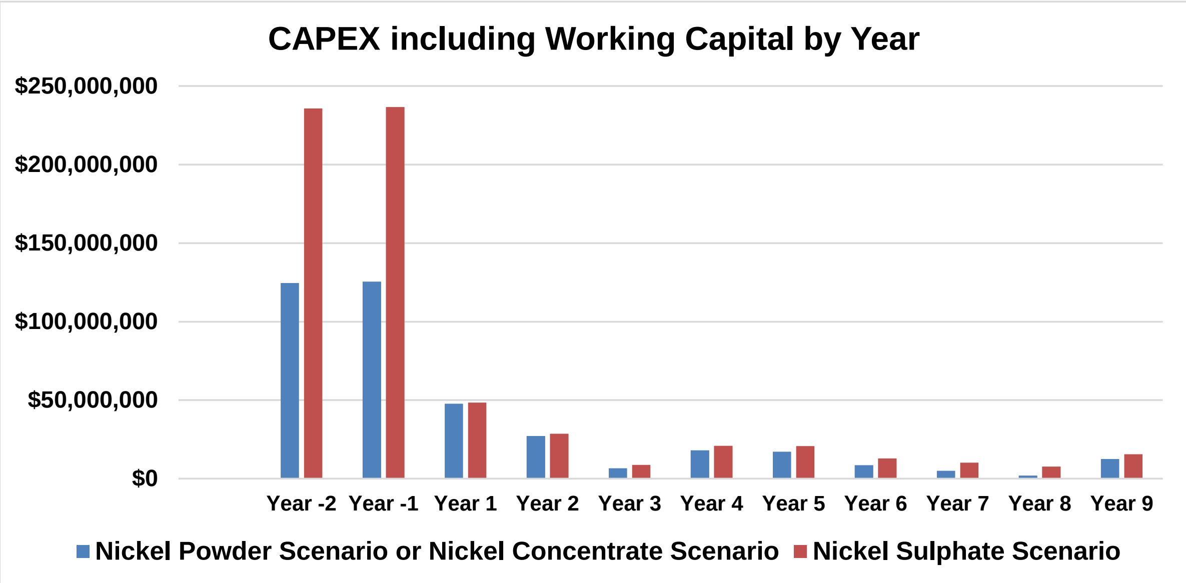


Figure 22-5: Capital Costs (US$) Including Working Capital by Year

22.3.6 Tamarack North Project Cash Flow Summary

The following table summarizes the base case LOM cash flow:

Table 22-7: Summary of Base Case LOM Cash Flow

	Nickel Powder Scenario			Nickel Sulphate Scenario			Nickel Concentrate Scenario		
	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni	LOM Total (US$M)	US$/tonne of Mill Feed	US$/lb of Payable Ni
Net smelter return revenue after royalties and transportation costs	**1,981.7**	**184.19**	**9.51**	**2,636.7**	**245.1**	**10.66**	**1,894.2**	**176.06**	**9.33**
On-Site Costs									
Mining Costs	295.8	27.49	1.42	296	27.49	1.20	295.8	27.49	1.46
Processing Costs	153.3	14.25	0.74	153	14.25	0.62	153.3	14.25	0.75
Hydromet Refining Costs	-			287	26.68	1.16	-		
Co-Disposed Filtered Tailings Facility	8.0	0.75	0.04	8	0.75	0.03	8.0	0.75	0.04
General & Administrative Costs	40.5	3.76	0.19	50	4.60	0.20	40.5	3.76	0.20
Total On-Site Costs	**497.7**	**46.26**	**2.39**	**793.7**	**73.77**	**3.21**	**497.7**	**46.26**	**2.45**
Net Operating Margin	**1,484.0**	**137.94**	**7.12**	**1,843.0**	**171.30**	**7.45**	**1,396.6**	**129.81**	**6.88**
Capital Expenditures	395.0	36.71	1.90	646.4	60.09	2.61	395.0	36.71	1.94
Working Capital	-	-	-	-	-	-	-	-	-
Net Cash Flow (before tax)	**1,089.0**	**101.22**	**5.23**	**1,196.6**	**111.22**	**4.84**	**1,001.6**	**93.09**	**4.93**
Corporate Tax	171.8	15.97	0.82	198.5	18.45	0.80	152.4	14.17	0.75
Net Cash Flow (after tax)	**917.3**	**85.26**	**4.40**	**998.1**	**92.77**	**4.03**	**849.2**	**78.93**	**4.18**

Table 22-8 provides the annual cash flow over LOM for the Ni Powder Scenario.

Table 22-9 provides the annual cash flow over LOM for the Ni Sulphate Scenario.

Table 22-10 provides the annual cash flow over LOM for the Ni Concentrate Scenario.

Table 22-8: Base Case LOM Annual Cash Flow – Ni Powder Scenario

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Diluted tonnes processed		10,758,796			867,569	1,286,664	1,315,005	1,360,754	1,302,589	1,298,742	1,299,601	1,190,479	837,393
Diluted grades													
Nickel	%	1.34			2.40	2.06	1.65	1.17	1.10	1.12	0.90	0.89	0.94
Copper	%	0.76			1.08	0.98	0.91	0.71	0.71	0.70	0.59	0.57	0.58
Platinum	g/t	0.27			0.47	0.40	0.32	0.28	0.27	0.23	0.21	0.17	0.14
Palladium	g/t	0.17			0.31	0.26	0.20	0.17	0.16	0.14	0.12	0.10	0.09
Gold	g/t	0.14			0.20	0.18	0.16	0.15	0.15	0.13	0.12	0.11	0.10
Cobalt	%	0.04			0.06	0.05	0.04	0.03	0.03	0.03	0.03	0.03	0.03
Recovery to concentrates and/or sulphate													
Nickel	%	82.1			84.34	83.68	82.86	81.78	81.59	81.64	78.89	78.75	79.86
Copper	%	86.9			87.95	87.65	87.41	86.70	86.68	86.64	86.15	86.07	86.11
Cobalt	%	64.1			64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10
NSR before transportation costs and royalties	$	2,162,804,149			306,585,499	390,672,226	326,190,917	243,200,960	220,588,145	221,906,475	176,308,780	159,171,236	118,179,912
Transportation, losses and insurance	$	20,419,698			2,241,385	3,055,625	2,923,785	2,467,779	2,352,644	2,315,543	2,000,921	1,786,614	1,275,403
NSR after transportation costs, insurance and losses, but before royalties	$	2,142,384,451			304,344,114	387,616,601	323,267,132	240,733,181	218,235,501	219,590,932	174,307,859	157,384,622	116,904,509
Minnesota royalty	$	120,884,522			27,881,781	28,168,316	18,259,685	10,432,915	9,194,190	9,251,294	6,876,130	6,208,539	4,611,672
Private royalty	$	39,798,492			5,653,718	7,200,648	6,005,245	4,472,035	4,054,101	4,079,281	3,238,070	2,923,691	2,171,703
NSR net of transportation and royalties	$	1,981,701,437			270,808,615	352,247,637	299,002,201	225,828,231	204,987,209	206,260,358	164,193,660	148,252,392	110,121,134
Mining	$	295,804,571			30,882,141	42,334,855	38,532,828	36,630,937	32,244,204	33,072,835	33,703,101	29,339,213	19,064,457
Dense media seperation		-			-	-	-	-	-	-	-	-	-
Processing	$	153,318,563			12,363,325	18,335,640	18,739,515	19,391,472	18,562,580	18,507,765	18,520,007	16,964,958	11,933,302
Hydrometallurgical refining		-			-	-	-	-	-	-	-	-	-
Filtered tailings facility	$	8,044,000			648,653	961,996	983,186	1,017,391	973,903	971,027	971,669	890,082	626,092
G&A	$	40,500,000			4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000
Net profit before tax, interest, CAPEX and working capital	$	1,484,034,303			222,414,495	286,115,145	236,246,673	164,288,431	148,706,523	149,208,731	106,498,882	96,558,139	73,997,284
Capital expenditures	$	-											
Mine Development	$	89,472,861	27,094,465	35,645,361	17,624,482	7,850,153	810,701	223,850	223,850	-	-	-	-
Mine Equipment and depreciable	$	110,998,261	13,865,495	6,117,291	29,802,600	18,886,232	5,077,233	10,920,824	15,741,581	7,157,391	3,429,615	-	-
Process and Surface Facilities CAPEX	$	170,835,210	83,631,975	77,520,995	242,056	484,112	726,168	968,224	1,210,280	1,452,336	1,694,392	1,936,448	968,224
Salvage value of mill and moveable equipment		(5,000,000)	-	-	-	-	-	-	-	-	-	-	- 5,000,000
CFTF		18,678,860	-	6,110,980	-	-	-	5,915,780	-	-	-	-	6,652,100
Closure costs other than CFTF	$	10,000,000	-	-	-	-	-	-	-	-	-	-	10,000,000
Total CAPEX	$	394,985,192	124,591,935	125,394,626	47,669,138	27,220,497	6,614,102	18,028,678	17,175,711	8,609,727	5,124,007	1,936,448	12,620,324
Working capital	$	-	-	18,147,795	- 1,614,672	- 844,241	- 303,932	- 1,314,778	192,735	160,788	- 1,500,131	(3,892,601)	(9,030,962)
Pre-tax cash flow	$	1,089,049,111	(124,591,935)	(143,542,421)	176,360,029	259,738,889	229,936,503	147,574,532	131,338,077	140,438,216	102,875,006	98,514,292	70,407,923
Income tax	$	171,789,497	-	-	29,014,054	36,346,136	29,688,677	17,292,747	17,419,874	17,757,009	10,106,319	10,934,166	3,230,515
After-tax cash flow	$	917,259,614	(124,591,935)	(143,542,421)	147,345,975	223,392,753	200,247,827	130,281,785	113,918,203	122,681,207	92,768,687	87,580,125	67,177,408
Cumulative cash flow	$		(124,591,935)	(268,134,356)	(120,788,380)	102,604,372	302,852,199	433,133,984	547,052,187	669,733,394	762,502,081	850,082,206	917,259,614
Funding requirement to positive cash flow	$		315,803,494										
Project after-tax NPV-7			567,000,000										
Project after-tax IRR			48.3%										

Table 22-9: Base Case LOM Annual Cash Flow – Ni Sulphate Scenario

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Diluted tonnes processed		10,758,796			867,569	1,286,664	1,315,005	1,360,754	1,302,589	1,298,742	1,299,601	1,190,479	837,393
Diluted grades													
Nickel	%	1.34			2.40	2.06	1.65	1.17	1.10	1.12	0.90	0.89	0.94
Copper	%	0.76			1.08	0.98	0.91	0.71	0.71	0.70	0.59	0.57	0.58
Platinum	g/t	0.27			0.47	0.40	0.32	0.28	0.27	0.23	0.21	0.17	0.14
Palladium	g/t	0.17			0.31	0.26	0.20	0.17	0.16	0.14	0.12	0.10	0.09
Gold	g/t	0.14			0.20	0.18	0.16	0.15	0.15	0.13	0.12	0.11	0.10
Cobalt	%	0.04			0.06	0.05	0.04	0.03	0.03	0.03	0.03	0.03	0.03
Recovery to concentrates and/or sulphate													
Nickel	%	78.0			84.34	83.68	82.86	81.78	81.59	81.64	78.89	78.75	79.86
Copper	%	84.5			87.95	87.65	87.41	86.70	86.68	86.64	86.15	86.07	86.11
Cobalt	%	64.1			64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10
NSR before transportation costs and royalties	$	2,853,680,593			407,996,822	519,163,378	430,666,095	319,675,185	289,045,655	291,525,908	231,043,796	208,982,414	155,581,340
Transportation, losses and insurance	$	23,834,397			2,639,889	3,589,760	3,428,018	2,874,782	2,740,261	2,696,158	2,318,927	2,068,920	1,477,683
NSR after transportation costs, insurance and losses, but before royalties	$	2,829,846,196			405,356,933	515,573,619	427,238,077	316,800,404	286,305,394	288,829,750	228,724,870	206,913,494	154,103,656
Minnesota royalty	$	150,134,033			38,435,764	39,221,367	23,173,535	11,711,373	10,002,822	10,108,237	6,971,167	6,162,990	4,346,778
Private royalty	$	42,980,299			6,376,863	8,236,095	6,715,489	4,813,716	4,287,677	4,318,433	3,282,824	2,902,242	2,046,961
NSR net of transportation and royalties	$	2,636,731,864			360,544,305	468,116,157	397,349,053	300,275,315	272,014,895	274,403,081	218,470,879	197,848,263	147,709,917
Mining	$	295,804,571			30,882,141	42,334,855	38,532,828	36,630,937	32,244,204	33,072,835	33,703,101	29,339,213	19,064,457
Dense media seperation		-			-	-	-	-	-	-	-	-	-
Processing	$	153,318,563			12,363,325	18,335,640	18,739,515	19,391,472	18,562,580	18,507,765	18,520,007	16,964,958	11,933,302
Hydrometallurgical refining		287,047,185			35,671,352	41,241,568	37,186,326	32,213,555	30,650,991	30,853,640	28,028,392	26,960,397	24,240,963
Filtered tailings facility	$	8,044,000			648,653	961,996	983,186	1,017,391	973,903	971,027	971,669	890,082	626,092
G&A	$	49,500,000			5,500,000	5,500,000	5,500,000	5,500,000	5,500,000	5,500,000	5,500,000	5,500,000	5,500,000
Net profit before tax, interest, CAPEX and working capital	$	1,843,017,545			275,478,834	359,742,096	296,407,198	205,521,960	184,083,218	185,497,814	131,747,709	118,193,612	86,345,104
Capital expenditures	$	-											
Mine Development	$	89,472,861	27,094,465	35,645,361	17,624,482	7,850,153	810,701	223,850	223,850	-	-	-	-
Mine Equipment and depreciable	$	110,998,261	13,865,495	6,117,291	29,802,600	18,886,232	5,077,233	10,920,824	15,741,581	7,157,391	3,429,615	-	-
Process and Surface Facilities CAPEX	$	422,293,558	194,796,630	188,685,650	970,282	1,940,564	2,910,846	3,881,128	4,851,410	5,821,692	6,791,974	7,762,256	3,881,128
Salvage value of mill and moveable equipment		(5,000,000)	-	-	-	-	-	-	-	-	-	-	- 5,000,000
CFTF		18,678,860	-	6,110,980	-	-	-	5,915,780	-	-	-	-	6,652,100
Closure costs other than CFTF	$	10,000,000	-	-	-	-	-	-	-	-	-	-	10,000,000
Total CAPEX	$	646,443,540	235,756,590	236,559,281	48,397,364	28,676,949	8,798,779	20,941,581	20,816,841	12,979,083	10,221,588	7,762,256	15,533,228
Working capital	$	-	-	31,899,552	- 4,806,037	- 1,858,051	- 1,547,125	- 1,705,419	243,397	- 545,524	- 1,767,130	(4,572,459)	(15,341,203)
Pre-tax cash flow	$	1,196,574,005	(235,756,590)	(268,458,833)	231,887,506	332,923,199	289,155,544	186,285,798	163,022,980	173,064,255	123,293,250	115,003,816	86,153,079
Income tax	$	198,504,590	-	-	35,007,822	44,287,728	35,862,393	19,240,809	20,170,085	20,976,011	10,787,416	12,707,041	(534,714)
After-tax cash flow	$	998,069,415	(235,756,590)	(268,458,833)	196,879,684	288,635,471	253,293,151	167,044,990	142,852,895	152,088,244	112,505,834	102,296,775	86,687,793
Cumulative cash flow	$		(235,756,590)	(504,215,423)	(307,335,738)	(18,700,268)	234,592,883	401,637,873	544,490,768	696,579,012	809,084,846	911,381,621	998,069,415
Funding requirement to positive cash flow	$		552,612,787										
Project after-tax NPV-7			569,000,000										
Project after-tax IRR			31.9%										

Table 22-10: Base Case LOM Annual Cash Flow – Ni Concentrate Scenario

	Unit	LOM Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Diluted tonnes processed		10,758,796			867,569	1,286,664	1,315,005	1,360,754	1,302,589	1,298,742	1,299,601	1,190,479	837,393
Diluted grades													
Nickel	%	1.34			2.40	2.06	1.65	1.17	1.10	1.12	0.90	0.89	0.94
Copper	%	0.76			1.08	0.98	0.91	0.71	0.71	0.70	0.59	0.57	0.58
Platinum	g/t	0.27			0.47	0.40	0.32	0.28	0.27	0.23	0.21	0.17	0.14
Palladium	g/t	0.17			0.31	0.26	0.20	0.17	0.16	0.14	0.12	0.10	0.09
Gold	g/t	0.14			0.20	0.18	0.16	0.15	0.15	0.13	0.12	0.11	0.10
Cobalt	%	0.04			0.06	0.05	0.04	0.03	0.03	0.03	0.03	0.03	0.03
Recovery to concentrates and/or sulphate													
Nickel	%	82.1			84.34	83.68	82.86	81.78	81.59	81.64	78.89	78.75	79.86
Copper	%	86.9			87.95	87.65	87.41	86.70	86.68	86.64	86.15	86.07	86.11
Cobalt	%	64.1			64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10	64.10
NSR before transportation costs and royalties	$	2,157,968,417			305,610,987	389,329,235	325,217,953	243,329,357	220,817,786	221,480,243	176,423,460	158,781,594	116,977,802
Transportation, losses and insurance	$	110,678,555			14,719,902	19,106,636	16,357,120	12,695,068	11,575,876	11,668,164	9,536,305	8,636,676	6,382,809
NSR after transportation costs, insurance and losses, but before royalties	$	2,047,289,862			290,891,084	370,222,599	308,860,834	230,634,289	209,241,910	209,812,079	166,887,155	150,144,918	110,594,993
Minnesota royalty	$	115,036,101			26,384,950	26,768,591	17,363,590	9,995,248	8,815,293	8,839,314	6,583,397	5,922,946	4,362,773
Private royalty	$	38,031,945			5,403,804	6,877,524	5,737,623	4,284,430	3,887,030	3,897,622	3,100,217	2,789,201	2,054,493
NSR net of transportation and royalties	$	1,894,221,816			259,102,330	336,576,484	285,759,620	216,354,611	196,539,587	197,075,143	157,203,541	141,432,771	104,177,727
Mining	$	295,804,571			30,882,141	42,334,855	38,532,828	36,630,937	32,244,204	33,072,835	33,703,101	29,339,213	19,064,457
Dense media seperation		-			-	-	-	-	-	-	-	-	-
Processing	$	153,318,563			12,363,325	18,335,640	18,739,515	19,391,472	18,562,580	18,507,765	18,520,007	16,964,958	11,933,302
Hydrometallurgical refining		-			-	-	-	-	-	-	-	-	-
Filtered tailings facility	$	8,044,000			648,653	961,996	983,186	1,017,391	973,903	971,027	971,669	890,082	626,092
G&A	$	40,500,000			4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000
Net profit before tax, interest, CAPEX and working capital	$	1,396,554,682			210,708,211	270,443,992	223,004,091	154,814,811	140,258,901	140,023,517	99,508,764	89,738,518	68,053,877
Capital expenditures	$	-											
Mine Development	$	89,472,861	27,094,465	35,645,361	17,624,482	7,850,153	810,701	223,850	223,850	-	-	-	-
Mine Equipment and depreciable	$	110,998,261	13,865,495	6,117,291	29,802,600	18,886,232	5,077,233	10,920,824	15,741,581	7,157,391	3,429,615	-	-
Process and Surface Facilities CAPEX	$	170,835,210	83,631,975	77,520,995	242,056	484,112	726,168	968,224	1,210,280	1,452,336	1,694,392	1,936,448	968,224
Salvage value of mill and moveable equipment		(5,000,000)	-	-	-	-	-	-	-	-	-	-	- 5,000,000
CFTF		18,678,860	-	6,110,980	-	-	-	5,915,780	-	-	-	-	6,652,100
Closure costs other than CFTF	$	10,000,000	-	-	-	-	-	-	-	-	-	-	10,000,000
Total CAPEX	$	394,985,192	124,591,935	125,394,626	47,669,138	27,220,497	6,614,102	18,028,678	17,175,711	8,609,727	5,124,007	1,936,448	12,620,324
Working capital	$	-	-	18,147,795	- 1,614,672	- 844,241	- 303,932	- 1,314,778	192,735	160,788	- 1,500,131	(3,892,601)	(9,030,962)
Pre-tax cash flow	$	1,001,569,490	(124,591,935)	(143,542,421)	164,653,745	244,067,736	216,693,922	138,100,911	122,890,455	131,253,002	95,884,888	91,694,671	64,464,516
Income tax	$	152,407,184	-	-	26,329,159	32,751,879	26,651,424	15,119,925	15,482,369	15,650,334	8,503,101	9,370,052	2,548,940
After-tax cash flow	$	849,162,306	(124,591,935)	(143,542,421)	138,324,586	211,315,857	190,042,498	122,980,987	107,408,086	115,602,667	87,381,787	82,324,619	61,915,576
Cumulative cash flow	$		(124,591,935)	(268,134,356)	(129,809,770)	81,506,087	271,548,585	394,529,571	501,937,657	617,540,324	704,922,112	787,246,730	849,162,306
Funding requirement to positive cash flow	$		315,803,494										
Project after-tax NPV-7			520,000,000										
Project after-tax IRR			45.6%										

22.3.7 C1 Cost and AISC

C1 cost or C1 cash cost, total cost and AISC are not IFRS (International Financial Reporting Standards) measures and, although calculated according to accepted industry practice, they may not be directly comparable to calculations carried out by other companies.

A) Nickel Powder Scenario

No benchmark has been set for expressing C1 cash cost or AISC for selling Ni powders to the EV industry in large quantities.

To be consistent with the Ni-to-stainless steel industry best practices, the following methodology has been used:

C1 Cash Cost: The cash cost of producing a pound of Ni in concentrate sold FOB at the mine gate less all by-product credits is shown in the following table.

Table 22-11: C1 Cash Cost of the Ni Powder Scenario

	Nickel Powder Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in concentrate**
On-site Cash Costs	497.7	46.26	1.91
Off-site Cost of By-products	48.5	4.51	0.19
Less: By-product Revenue	(524.5)	(48.75)	(2.01)
Net Cost of Producing Nickel in Concentrate at the Mine	**21.7**	**2.02**	**0.08**

Ni concentrates need to be refined to produce Ni powder for the EV industry, which will require additional capital and operating costs: For instance, if it costs $1.17/lb to convert Ni concentrate to a Ni powder, the C1 cash cost of producing Ni powder would be $1.25/lb of Ni. Talon is working towards developing a flowsheet to determine the actual cost of converting Ni concentrates to Ni powder at the mine site.

AISC: The AISC of producing a pound of Ni in concentrate sold FOB at the mine gate is the C1 cash cost plus royalties and sustaining CAPEX as shown in the following table.

Table 22-12: AISC of the Ni Powder Scenario

	Nickel Powder Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in concentrate**
C1 Cash Cost	21.7	2.02	0.08
Government and Private	160.7	14.94	0.62
Sustaining CAPEX	97.3	9.05	0.37
AISC of Producing Nickel in Concentrate at the Mine Gate	**279.7**	**26.00**	**1.07**

In the Ni Powder Scenario, Ni concentrates are sold at a discount to the LME Ni price, similar to the Ni Concentrate Scenario. However, the discount to the LME Ni price is expected to be smaller for the Ni Powder Scenario as compared to the Ni Concentrate Scenario, given the supply chain from mine to battery removes several processing and transportation steps (thereby creating a win-win for both the mining company and the battery company).

Should the facility that converts Ni concentrates to Ni powders for the EV industry be co-located at the mine site, transportation costs will be extremely low compared to the Ni Concentrate Scenario: Ni powders for the EV industry require 99.99%+ purity and therefore, almost no waste is transported. In contrast, Ni concentrates at approximately 12% by mass of valuable metals requires the transportation of 88% waste. As with the Ni Sulphate Scenario, the product under the Ni Powder Scenario is sold FOB at the mine gate.

Ni concentrates need to be refined to produce Ni powder for the EV market, which will require additional capital and operating cost: For instance, if it costs $1.17/lb to convert Ni concentrate to a Ni powder, the AISC of producing Ni powder would be $2.24/lb of Ni. Talon is working towards developing a flowsheet to determine the actual cost of converting Ni concentrates to Ni powder at the mine site.

B) Nickel Sulphate Scenario

Ni sulphates produced at site will be sold FOB at the mine gate and therefore, the C1 cash cost is calculated as shown in the following table.

Table 22-13: C1 Cash Cost of the Ni Sulphate Scenario

	Nickel Sulphate Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in Ni Sulphate**
On-site Cash Costs	506.7	47.09	2.05
On-site Cash Cost of Converting a Ni Concentrate to a Ni Sulphate	287.0	26.68	1.16
Off-site Cost of By-products	56.4	5.24	0.23
Less: By-product Revenue	(598.2)	(55.60)	(2.42)
Net Cost of Producing Nickel in Sulphate at the Mine Gate	**251.9**	**23.42**	**1.02**

On site cash costs are higher for the Ni Sulphate Scenario (before accounting for the cost of converting Ni concentrate to a Ni sulphate) because 95% of Ni in concentrates is expected to be recovered in the hydrometallurgical process, thereby reducing the denominator (pounds of Ni shipped) and increasing the overall cost per pound of Ni.

The higher by-product revenue compared to the Ni Powder Scenario is due to higher Co revenues, as the Ni Sulphate Scenario produces a Co sulphide, which due to its grade has high payabilities (compared to the lower payabilities assumed in the Ni Powder Scenario).

AISC: AISC is the C1 cash cost for Ni sulphates produced at site plus royalties and sustaining CAPEX and is shown in the following table.

Table 22-14: AISC of the Ni Sulphate Scenario

	Nickel Sulphate Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in Ni Sulphate**
C1 Cash Cost	251.9	23.42	1.02
Government and Private	193.1	17.95	0.78
Sustaining CAPEX	125.7	11.69	0.51
AISC of Producing Nickel in Sulphate at the Mine Gate	**570.8**	**53.05**	**2.31**

In the Ni Sulphate Scenario, Ni sulphates are sold as opposed to Ni concentrates. The traditional supply chain requires the production of LME grade Ni that is used as a feedstock to produce Ni sulphates. Ni sulphates are therefore typically sold at a premium to the LME Ni price. In the case of the Tamarack Nickel Project, Ni sulphates could be produced directly from Ni concentrates at site, thereby reducing the number of process and transportation steps. The premium to market price applies irrespective of processing route.

Royalties per pound are higher in the Ni Sulphate Scenario compared to the Ni Powder Scenario because under the Ni Sulphate Scenario, a value-added product (that results in a premium price) is sold.

AISC under the Ni Sulphate Scenario is higher than the Ni Concentrate Scenario because of the incremental CAPEX associated with the hydrometallurgical refinery and higher royalties because Ni sulphate is a value-added product that sells at a premium to LME Ni.

C) Nickel Concentrate Scenario

The Ni Concentrate Scenario contemplates the traditional Ni-to-stainless steel supply chain in which concentrate is sold to a smelter, which is in turn sold to a refinery to produce LME-grade Ni.

C1 Cost: The cost of producing a pound of LME-grade Ni per pound of Ni in concentrate sold CIF to the smelter less all by-product credits is shown in the following table.

Table 22-15: C1 Cost of the Ni Concentrate Scenario

	Nickel Concentrate Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in concentrate**
On-site Cash Costs	497.7	46.26	1.91
Less: Value of By-product in Concentrate	(767.0)	(71.29)	(2.95)
Net Cost of Producing Ni in Concentrate at the Mine Gate	(269.3)	(25.03)	(1.03)
Product Handling, Transportation, Insurance and Losses	110.7	10.29	0.43
Smelting, Refining, and Deductions by the	692.0	64.32	2.66
C1 Cost of a lb of Nickel in LME Grade Briquettes	**533.4**	**49.58**	**2.05**

Under the Ni Concentrate Scenario, by-product revenue is the highest because the by-products are calculated using the gross value of metal in the concentrate transported to the smelter.

Product handling, transportation insurance and losses in the Ni Concentrate Scenario are high because of the need to transport both the Ni and Cu concentrates from the mine to smelters.

The smelting, refining and deductions line item consists of both cash charges by the smelters/refiners such as treatment charges and refining charges, as well as deduction of metal units sent to the smelter but not paid to the mine.

AISC: The C1 cost of producing a pound of LME-grade Ni per pound of Ni in concentrate sold CIF to the smelter plus royalties and sustaining CAPEX as shown in the following table.

Table 22-16: AISC of the Ni Concentrate Scenario

	Nickel Concentrate Scenario		
Cost	**LOM Total (US$M)**	**US$/tonne of Mill Feed**	**US$/lb of Ni in concentrate**
C1 Cost	533.4	49.58	2.05
Government and Private Royalties	153.1	14.23	0.59
Sustaining CAPEX	97.3	9.05	0.37
AISC of a lb of Nickel in LME Grade Briquettes	**783.8**	**72.85**	**3.01**

In the Ni Concentrate Scenario, NI concentrates are sold at a discount to the LME Ni price. This discount to the LME Ni price is expected to be higher for the Ni Concentrate Scenario compared to the Ni Powder Scenario, as the traditional supply chain from mine to stainless steel requires more processing and transportation steps. Traditionally, transportation and insurance costs are incurred by the mining company and is therefore included in the cost calculation.

The C1 cost of $2.05/lb Ni to produce a Ni briquette calculated for this PEA is lower than the C1 cost of $2.67/lb Ni in the March 2020 PEA: There was a cost increase due to the addition of lower grade disseminated sulphide materials to the mine plan from both the 138 and Upper SMSU Domains, which was more than offset by the reduction in costs due to economies of scale realized from increasing the production rate from 2,000 tpd to 3,600 tpd, as well as major improvements to mine access, development and stoping costs and the application of the latest publicly available smelting and refining terms, which have improved since the March 2020 PEA.

22.3.8 Base Case Evaluation

The base case cash flow, 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 22-17. Results are also shown at comparative discount rates of 8% and 10% and on a pre-tax basis.

Table 22-17: Base Case NPV for all Scenarios at Various Discount Rates in Million US$ and IRR

		Base Case Pricing		
	Discount rate	Nickel Powder Scenario	Nickel Sulphate Scenario	Nickel Concentrate Scenario
Pre-tax	**7%**	688	711	629
NPV in	**8%**	646	660	589
$ millions	**10%**	570	568	518
Pre-tax IRR		56.0%	37.6%	52.6%
After-tax	**7%**	**567**	**569**	**520**
NPV in	**8%**	530	524	485
$ millions	**10%**	463	443	423
After-tax IRR		**48.3%**	**31.9%**	**45.6%**
Initial CAPEX and working capital in $ millions		**316**	**553**	**316**

After-tax NPV at a 7% discount rate, initial CAPEX including working capital and after-tax IRR at base case pricing are illustrated in Figure 22-6 below.

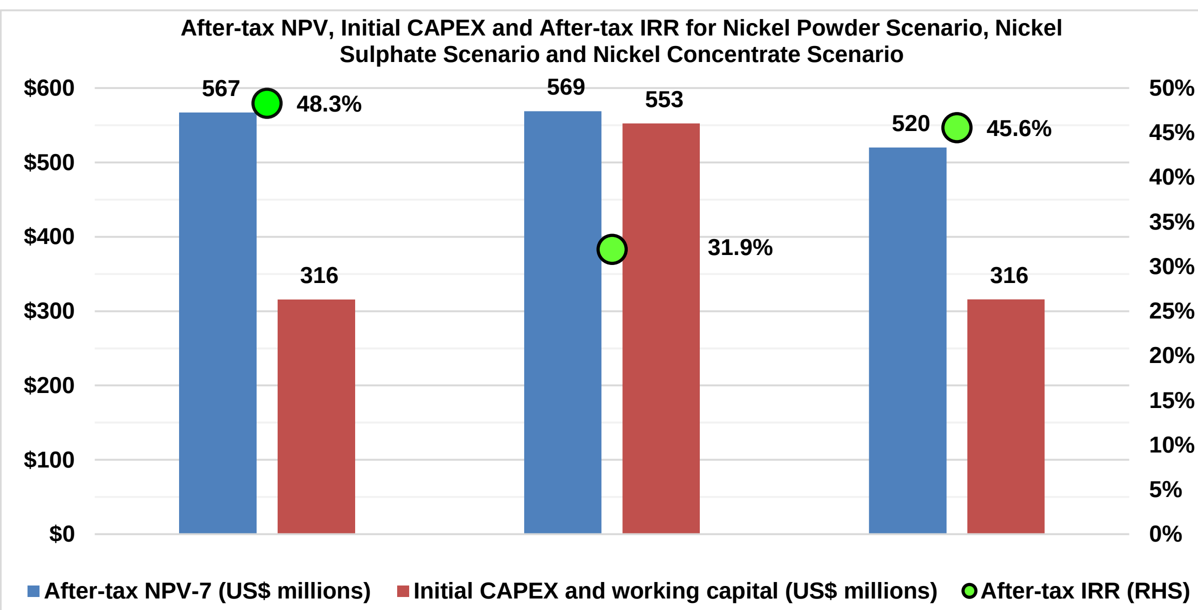


Figure 22-6: After-tax NPV, Initial CAPEX and Working Capital, and After-tax IRR for all Scenarios

The undiscounted pre-tax payback period is 2.2 years under the Ni Powder Scenario, 2.7 years under the Ni Sulphate Scenario and 2.3 years under the Ni Concentrate Scenario from the production start date. Payback measures on a pre-tax, after-tax, undiscounted and discounted basis are included in Table 22-18.

Table 22-18: Payback Period in Years from Production Start Date

Years from start of production		Base Case Pricing		
		Nickel Powder Scenario	Nickel Sulphate Scenario	Nickel Concentrate Scenario
Pre-tax	**Undiscounted**	1.4	1.8	1.4
	Discounted	1.5	2.1	1.6
After-tax	**Undiscounted**	1.5	2.1	1.6
	Discounted	1.7	2.4	1.8

22.4 Sensitivity and Risk Analysis

22.4.1 Metal Price Assumptions and Discount Rates

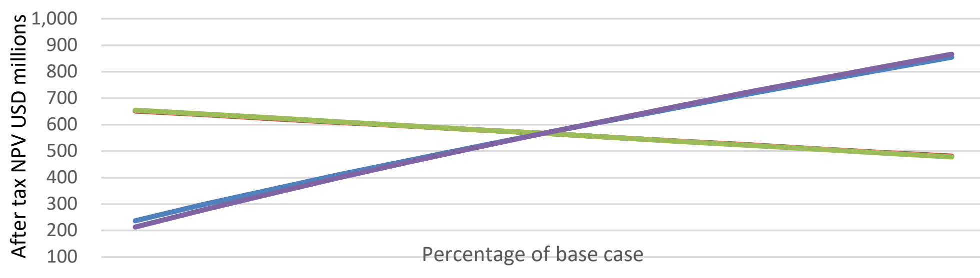
The sensitivities of the after-tax and pre-tax NPV and IRR as well as other measures were tested using alternate metal price assumptions and discount rates as shown in Table 22-19.

Table 22-19: 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

		Nickel Powder Scenario			Nickel Sulphate Scenario			Nickel Concentrate Scenario		
	Discount rate	Metal Price Case			Metal Price Case			Metal Price Case		
		Low	Base	Incentive	Low	Base	Incentive	Low	Base	Incentive
Pre-tax NPV	**7%**	496	688	917	478	711	970	439	629	854
US$ millions	**8%**	463	646	863	438	660	906	409	589	803
	10%	404	570	767	367	568	790	355	518	712
Pre-tax IRR		45.0%	56.0%	67.4%	29.2%	37.6%	45.7%	41.5%	52.6%	64.2%
After-tax NPV	**7%**	**415**	**567**	**744**	**387**	**569**	**769**	**369**	**520**	**695**
US$ millions	**8%**	386	530	698	351	524	714	342	485	651
	10%	333	463	616	286	443	615	293	423	573
After-tax IRR		**39.3%**	**48.3%**	**57.7%**	**25.1%**	**31.9%**	**38.6%**	**36.4%**	**45.6%**	**55.1%**
EBITDA margin		64%	**68%**	70%	60%	**64%**	66%	60%	**64%**	67%
EBIT margin		43%	**50%**	55%	34%	**41%**	47%	39%	**46%**	52%
Payback from start of production (pre-tax, undiscounted)		1.6	**1.4**	1.2	2.2	**1.8**	1.6	1.7	**1.4**	1.2
Payback from start of production (after-tax, undiscounted)		1.8	**1.5**	1.3	2.4	**2.1**	1.8	1.9	**1.6**	1.4

22.4.2 Capital, Operating Costs, Grade and Revenue Sensitivity

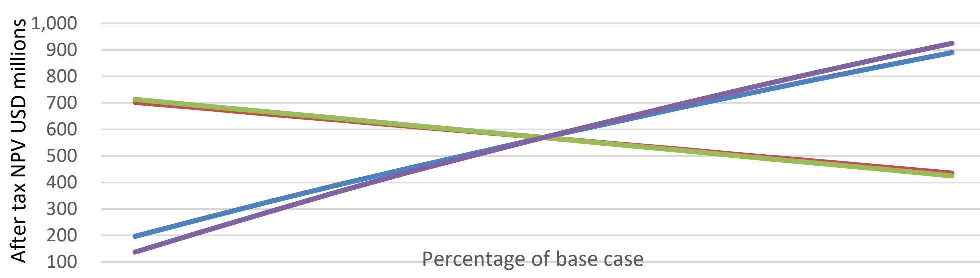
The sensitivity of the after-tax NPV under the three scenarios was tested assuming changes in metal prices, operating costs, head grade and capital costs in a range of 30% around the Base Case as shown in Figure 22-7, Figure 22-8 and Figure 22-9 for the Ni Powder, Ni Sulphate and Ni Concentrate Scenarios, respectively.



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	237	298	355	411	464	516	567	617	666	715	762	809	855
Operating costs	652	638	623	609	595	581	567	552	538	524	509	495	481
CAPEX	655	640	626	611	596	581	567	552	537	522	508	493	478
Grade	213	278	340	400	458	513	567	619	671	722	771	819	866

Metal prices Operating costs CAPEX Grade

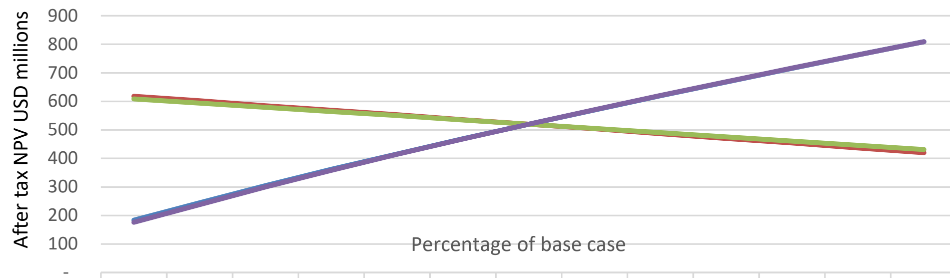
Figure 22-7: Sensitivity of Base Case After-Tax NPV to Changes in Metal Prices, Grade, Operating Costs and Capital Costs – Ni Powder Scenario



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	197	264	329	392	452	511	569	625	681	735	788	840	890
Operating costs	702	680	657	635	613	591	569	547	525	502	480	458	435
CAPEX	713	689	665	641	617	593	569	545	521	497	473	449	425
Grade	137	215	291	366	437	504	569	632	695	755	814	870	925

Metal prices Operating costs CAPEX Grade

Figure 22-8: Sensitivity of Base Case After-Tax NPV to Changes in Metal Prices, Grade, Operating Costs and Capital Costs – Ni Sulphate Scenario

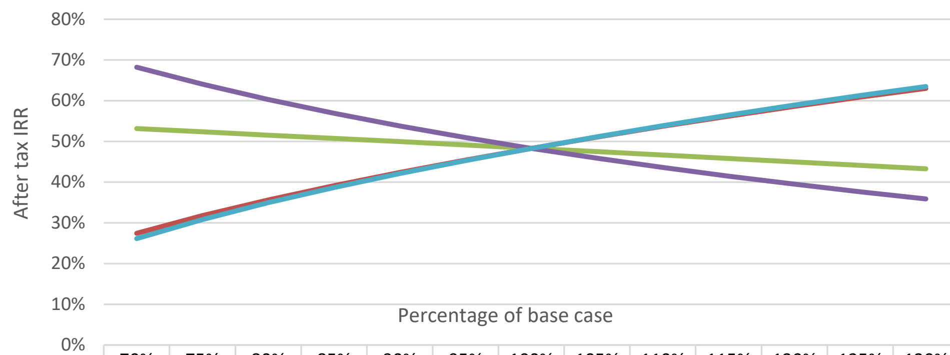


	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	184	245	305	362	416	469	520	570	620	668	716	763	809
Operating costs	618	602	585	569	553	536	520	504	487	470	454	437	420
CAPEX	609	594	579	564	550	535	520	505	490	476	461	446	431
Grade	176	238	300	358	415	468	520	571	621	669	717	763	809

Metal prices Operating costs CAPEX Grade

Figure 22-9: Sensitivity of Base Case After-Tax NPV to Changes in Metal Prices, Grade, Operating Costs and Capital Costs – Ni Concentrate Scenario

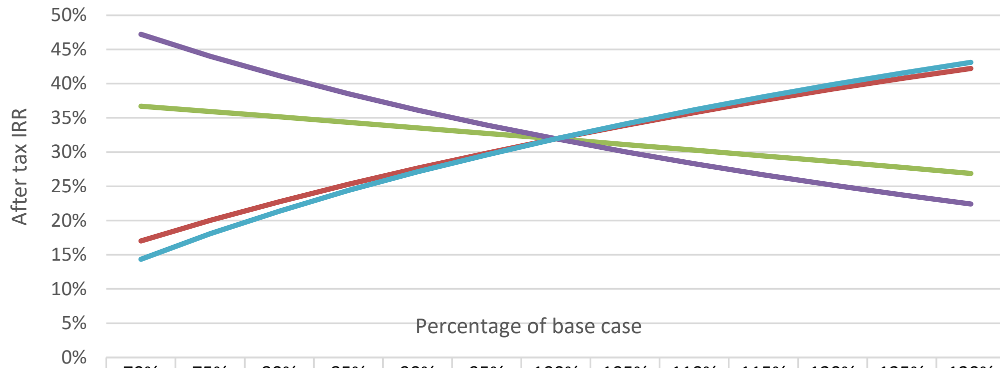
The sensitivity of the after-tax IRR under the three scenarios was tested assuming changes in metal prices, operating costs, head grade and capital costs in a range of 30% around the Base Case as shown in Figure 22-10, Figure 22-11 and Figure 22-12 for the Ni Powder, Ni Sulphate and Ni Concentrate Scenarios, respectively.



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	27%	32%	36%	39%	42%	45%	48.3%	51%	54%	56%	59%	61%	63%
Operating costs	53%	52%	52%	51%	50%	49%	48.3%	47%	47%	46%	45%	44%	43%
CAPEX	68%	64%	60%	57%	54%	51%	48.3%	46%	44%	41%	39%	38%	36%
Grade	26%	31%	35%	39%	42%	45%	48.3%	51%	54%	56%	59%	61%	63%

Metal prices Operating costs CAPEX Grade

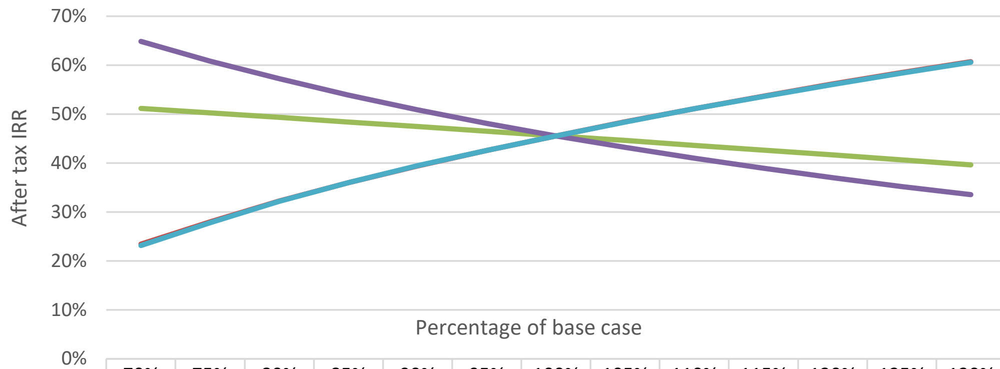
Figure 22-10: Sensitivity of Base Case After-Tax IRR to Changes in Metal Prices, Operating Costs, Grade and Capital Costs – Ni Powder Scenario



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	17%	20%	23%	25%	28%	30%	31.9%	34%	36%	38%	39%	41%	42%
Operating costs	37%	36%	35%	34%	34%	33%	31.9%	31%	30%	29%	29%	28%	27%
CAPEX	47%	44%	41%	39%	36%	34%	31.9%	30%	28%	27%	25%	24%	22%
Grade	14%	18%	21%	24%	27%	30%	31.9%	34%	36%	38%	40%	42%	43%

Metal prices Operating costs CAPEX Grade

Figure 22-11: Sensitivity of Base Case After-Tax IRR to Changes in Metal Prices, Operating Costs, Grade and Capital Costs – Ni Sulphate Scenario



	70%	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%	130%
Metal prices	23%	28%	32%	36%	39%	43%	45.6%	48%	51%	54%	56%	58%	61%
Operating costs	51%	50%	49%	48%	47%	47%	45.6%	45%	44%	43%	42%	41%	40%
CAPEX	65%	61%	57%	54%	51%	48%	45.6%	43%	41%	39%	37%	35%	34%
Grade	23%	28%	32%	36%	39%	43%	45.6%	48%	51%	54%	56%	58%	61%

Metal prices Operating costs CAPEX Grade

Figure 22-12: Sensitivity of Base Case After-Tax IRR to Changes in Metal Prices, Operating Costs, Grade and Capital Costs – Ni Concentrate Scenario

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 the QP's opinion that the information relating to geology, exploration and mineral resource estimation presented in this Technical Report is representative of the Tamarack North Project, and based on the verification and data analysis work completed, is of the opinion that the sample database is of suitable quality to support the basis of the mineral resource estimates and recommendations reached in this Technical Report.

The QP has taken reasonable steps to make the block model and Mineral Resource estimate representative of the project data, but notes that there are risks related to the accuracy of the estimates related to the following:

- The assumptions used by the QP to prepare the data for resource estimation;
- The accuracy of the interpretation of mineralization;
- Estimation parameters used by the QP;
- Assumptions and methodologies used to estimate SG;
- Orientation of drill holes;
- COG and related assumptions of commodity prices, mining costs and metallurgical recovery.

For these reasons, actual results may differ materially from the reported Mineral Resource estimates.

25.2 Mining Methods

The Tamarack North Project is amenable to underground mining methods at a rate of 3,600 tpd using drift-and-fill and longhole stoping with cemented paste backfill. Access will be from a decline from surface, and development and production performed by mining contractors. The expected mine life based on current deposit extent and Mineral Resources is up to 11 years, however the deposit is open and a longer mine life is believed to be possible.

25.3 Mineral Processing and Metallurgical Testing

A metallurgical test program on a revised LOM composite representing the entire 8.02 Mt of the March 2020 PEA mineralized material confirmed the robustness of the simplified flowsheet and conditions that were generated in the March 2020 PEA.

The results that were obtained in the 2019 / 2020 metallurgical results were in good agreement with the Ni and Cu recovery regression curves that were reported in the March 2020 PEA. Adjustments to the LOM regression curves were made to reflect the fact that the 138 Zone mineralization was included in the overall LOM composite compared to only SMSU and MSU mineralization in the March 2020 PEA.

The low Po to Pn ratio compared to many other Ni deposits simplifies the process circuit, and a Po rejection circuit is not required to achieve high Ni recoveries into a saleable Ni concentrate grading at least 10% Ni.

The low levels of deleterious elements in the Cu and Ni concentrates are not expected to trigger any penalty payments. The MgO content in the Ni concentrate of the LOM composite was just below the typical 5% threshold of smelters. Also, very limited work was completed on determining the most effective MgO depressant.

Credits for by-products will mostly derive from Cu and Co with potentially minor contribution from Au, Pt, and Pd.

A scoping level hydrometallurgical program demonstrated that both POX and Albion leach technologies are suitable to achieve very high metal extraction rates into the pregnant leach solution (PLS). The current flowsheet employs POX pending a capital and operating cost trade-off study for the Albion process. The two stage neutralization steps are successful in removing most impurities while minimizing the Ni and Co losses. Further, the residue of the secondary neutralization stage and the Cu removal circuit generate two product streams that can be combined with the Cu concentrate. This is significant, since payability of Cu in the Ni concentrate is very low. The two Cu bearing product streams from the hydrometallurgical plant reduce the Cu concentrate grade by only 1%.

While development of the complete hydrometallurgical circuit is still ongoing, the leach, neutralization, and impurity removal tests have demonstrated high extraction rates during leaching and low metal losses during impurity removal. These results are very encouraging since losses to the leach, neutralization, and impurity removal residues are the primary sources of metal losses in the hydrometallurgical process.

25.4 Recovery Methods

The flowsheet was designed for a nameplate capacity throughput of 3,600 tpd and an annual throughput of 1,314,000 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 81.5% and 74.7% respectively. The Ni and Cu concentrate grades are projected to be 10.2% Ni and 28.5% Cu.

The low feed rate facilitates a simple crushing and grinding circuit with two stages of crushing and a single stage of ball milling but a SAG mill circuit will be evaluated once additional comminution data will be available.

The hydrometallurgical process employs technologies that have been implemented on a commercial scale in numerous operations. The overall Ni recovery from mineralized material to Ni sulphates was established at 77.4%. A total of 60.9% of the Co contained in the mineralized material can be recovered into a Co sulphite product. The Cu recovery into the Cu concentrate improved from 69.3% for the Ni Concentrate Scenario (the balance of 15.4% of the Cu units reported to the Ni concentrate) to 81.8% for the Ni Sulphate Scenario.

25.5 Infrastructure

The conceptual site layout designed during the previous March 2020 PEA was modified and updated for the currently envisaged project definition. Notably, the updated CFTF design concept still 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, a significant portion of the area was utilized as farmland in 1992.

Preliminary water balances were completed for the processing scenarios contemplated, based on waste rock characterization, geophysical measurements, pump tests and estimates 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 Economics

At the assumed base case metal prices, key metrics of the PEA of the Tamarack North Project are summarized in the following table. All amounts are in US dollars.

Table 25-1: Key Metrics of the PEA

All amounts in US$	Scenario		
	Nickel Powder Scenario	Nickel Sulphate Scenario	Nickel Concentrate Scenario
After-Tax NPV(1), (2)	$567M	$569M	$520M
After-Tax IRR(1)	48.3%	31.9%	45.6%
Initial CAPEX and Working Capital	$316M	$553M	$316M
Payback Period, pre-tax(3)	1.4 years	1.8 years	1.4 years
Payback Period, after-tax(3)	1.5 years	2.1 years	1.6 years
Mine Life(3)	9 years	9 years	9 years

(1) Metal prices of $8.00/lb Ni, $3.00/lb Cu, $25.00/lb Co, $1,000/oz Pt, $1,000/oz Pd and $1,300/oz Au.
(2) Discount rate of 7%
(3) From the start of production

The PEA illustrates a high after-tax IRR, low AISC, low capital intensity and a quick payback for the Tamarack Nickel Project. The PEA also clearly demonstrates that the Tamarack Nickel Project has the optionality to produce either Ni sulphates or concentrates for refined Ni powders to be used for the EV market or a Ni concentrate for the stainless steel market, with all contemplated scenarios having robust economics.

The financial model is based on the results of the PEA which is preliminary in nature and includes inferred resources that are considered too speculative geologically to have the economic consideration applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

26 RECOMMENDATIONS

It was previously recommended that Talon should determine whether it should sell Ni concentrates or Ni sulphates by developing and contrasting the economic impact of these two scenarios. The Company's targeted completion date to achieve this milestone was the end of February 2021.

With this PEA, Talon has now achieved this objective.

Since that time, Tesla, Inc. has articulated that it has the goal of consuming raw Ni powder, which will dramatically simplify the Ni-to-battery supply chain. The Ni Powder Scenario contemplated by Talon in this PEA is based on selling Ni concentrates to a co-located refinery that produces high purity Ni powders. Talon believes, however, that the same refinery could potentially produce high purity Fe powders as a by-product. It is therefore recommended that going forward, the economic impact of a scenario where Talon produces both refined Ni and Fe powders is quantified.

If successful, Talon could potentially produce three products:

- Refined Ni powders for the Ni battery cathode market;
- Refined Fe powders for the lithium iron phosphate (LFP) market; and
- Cu concentrates.

Talon's next milestone (**Milestone 1**) should therefore be to develop a flowsheet in addition to CAPEX and OPEX estimates for the above scenario. A successful outcome could allow Talon to lower its cut-off grade and therefore, increase the amount of metal it could produce from the Tamarack Nickel Project, which would lead to a different resource envelope and consequently mine plan.

Once Talon has decided whether it should produce Ni concentrates, Ni sulphates or Ni and Fe powders, the Company will complete a PFS (**Milestone 2**).

In the meantime, going forward Talon should also focus on resource expansion and definition drilling to progress towards PFS and Feasibility Study assuming the Company proceeds with the Ni Powder Scenario contemplated in the February 2021 PEA. It is estimated that between 25,000 and 30,000 m of drilling will be required, mostly focused on expansion of the Tamarack Nickel Project's current resource area. Talon's in-house team of experienced specialists operate their own drilling and geophysical equipment efficiently and at low cost. It is therefore expected that this recommendation is achievable during 2021.

Talon also has a comprehensive geotechnical logging program in place and should therefore continue with laboratory testing of drill core, collecting down hole data using acoustic televiewer and full wave sonic technology, as well as in-situ stress measurement testing. Hydrological work should be conducted as appropriate for each level of study. Installation of multilevel vibrating wire piezometers in selected historical drill holes and additional aquifer property testing within the glacial till and bedrock aquifers are also recommended.

Geo-metallurgical testing programs should continue and should be based on the predicted LOM feed bearing in mind the specific requirements of each of the potential customers for the different products listed above. It is thus recommended that Talon maintain optionality and complete the second phase of hydrometallurgical testing for production of Ni sulphates. Waste products from geo-metallurgical testing programs should be used to continue environmental test work.

The following budget allocation is recommended:

Table 26-1: Milestone 1: Flowsheet Development including Mine Planning, Metallurgy and Engineering

Item	Description	Details	Amount (C$)	Amount (US$)
1.0	Mine planning, metallurgy, and engineering	Includes extensive metallurgical testing and trade-off studies in advance of a feasibility study as well as a final PEA to enable the Company to select one of the three markets listed above	$2,300,000	$1,800,000
		Total	**$2,300,000**	**$1,800,000**

The budget to collect data for a PFS and a Feasibility Study is shown below.

Table 26-2: Milestone 2: Resource Expansion and Data Collection for a PFS and a Feasibility Study

Item	Description	Details	Amount (C$)	Amount (US$)
1.0	Exploration	Includes geophysical, geological, assays, geotechnical and hydrogeological data collection and related site costs	$9,300,000	$7,300,000
2.0	Environmental baseline, permitting and community	Includes baseline data collection as well as extensive testing of tailings and waste rock to optimize the CFTF	$700,000	$550,000
3.0	Mineral leases and land	Payments required to keep all properties in good standing	$1,000,000	$800,000
		Total	**$11,000,000**	**$8,650,000**

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28 CERTIFICATES OF QUALIFIED PERSONS – NI 43-101

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Leslie Correia, Pr. Eng., do hereby certify that:

1) I am an Engineering Manager with Paterson & Cooke Canada Inc., with an office at 1351-C Kelly Lake Road, Unit #2, Sudbury, Ontario, Canada;
2) I am a graduate from University of Stellenbosch a Bachelor of Engineering in Chemical Engineering in 2005;
3) I am a member in good standing of the Engineering Council of South Africa (ECSA, Membership 20120236);
4) My relevant experience is 13 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 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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for portions of Section 16;
7) I have not visited the site;
8) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(Signed) Leslie Correia

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) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Tim Fletcher, P. Eng., do hereby certify that:

1) I am a Senior Project Manager with DRA Americas Inc., with an office at 2900 – 20 Queen Street West 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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 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) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(signed) Tim Fletcher

Tim Fletcher, P. Eng.
Senior Project Manager
DRA Americas Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Daniel M. Gagnon, P. Eng., do hereby certify that:

1) I am the VP Mining and Geology with Met-Chem, a division of DRA Americas Inc., with an office at Suite 600, 555 Rene-Levesque Blvd. W. Montreal, Quebec, Canada;
2) I am a graduate from Ecole Polytechnique de Montreal with a B. Eng. In Mining Engineering in 1995;
3) I am registered member of Ordre des Ingenieurs du Quebec (Membership Number 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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for Sections 19 and 22, and portions of Sections 1, 18, 21, 25, and 26;
7) I have visited the site on October 5, 2017;
8) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(signed) Daniel M. Gagnon

Daniel M. Gagnon, P. Eng.
VP Mining and Geology
DRA Americas Inc.

DRA

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Andre-Francois Gravel, P. Eng., do hereby certify that:

1) I am a Senior Mining Engineer with Met-Chem, a division of DRA Americas Inc., with an office at Suite 600, 555 Rene-Levesque Blvd. W. Montreal, Quebec, Canada;
2) I am a graduate from Ecole Polytechnique de Montreal with a Bachelor's degree in Mining Engineering in 2000;
3) I am registered member of Ordre des Ingenieurs du Quebec (Membership Number 125135);
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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for portions of Section 16;
7) I have not visited the site;
8) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(signed) André-François Gravel

Andre-Francois Gravel, P. Eng.
Senior Mining Engineer
DRA Americas Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Volodymyr Liskovych, PhD, P. Eng., do hereby certify that:

1) I am Process Engineer at DRA Americas Inc., with an office at 2900 – 20 Queen Street West Toronto, Ontario, Canada;
2) I am a graduate from the Zaporizhzhia State Engineering Academy (Zaporizhzhia, Ukraine) with a Metallurgical Engineer degree in 1996 and from the National Metallurgical Academy of Ukraine (Dnipro, Ukraine) with a PhD degree in Metallurgical Engineering in 2001;
3) I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 100157409);
4) I have worked as a Metallurgical Engineer continuously since my graduation from Zaporizhzhia State Engineering Academy;
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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for hydrometallurgical aspects of Sections 13 and 17, and portions of Sections 1, 21, 25, 26, and 27;
7) I have not visited the site;
8) I have 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; and
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 8th day of January 2021.

(signed) Volodymyr Liskovych

Volodymyr Liskovych, PhD, P. Eng.
Principal Process Engineer
DRA Americas Inc.

CERTIFICATE OF AUTHOR

To accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Andrea K. Martin, P.E., do hereby certify that:

1) I am a Lead Environmental Engineer with Foth Infrastructure & Environment, LLC, with an office at 2121 Innovation Court, Suite 300, De Pere, Wisconsin, USA;
2) I am a graduate from Michigan Technological University in Chemical Engineering in 1981 and University of Wisconsin Green Bay with a Master's Degree in Environmental Science & Policy;
3) I have current professional engineer licenses in Minnesota (License No. 48512), Michigan (License No. 6201052301), Illinois (License No. 062.045871), and Wisconsin (License No. 359942-6) and I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME) (No. 4173551RM);
4) I have worked as an Engineer in the Mining & Metals industry since 2005 and prior to that, 17 years for a large metal plate construction company;
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) #3 of the Tamarack North Project – Tamarack, Minnesota " dated January 8, 2021 and am responsible for Section 20, and portions of Sections 1, 3, and 26;
7) I have visited the site on April 18, 2014, January 23, 2019, and January 27, 2020;
8) My prior involvement with the property is related to the environmental baseline studies and other conceptual site studies;
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; and
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 8th day of January 2021.

I hereby certify that Section 20 and associated portions of Sections 1, 3, and 26 of this report were prepared by me and that I am a duly Licensed Professional Engineer under the laws of the state of Minnesota.

(signed) Andrea Martin

Andrea K. Martin, P.E.
Lead Environmental Engineer
Foth Infrastructure & Environment, LLC
Minnesota P.E. License No. 48512

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, Oliver Peters, P. Eng., do hereby certify that:

1) I am Principal Metallurgist and President with Metpro Management Inc., with a business address at 102 Milroy Drive, Peterborough, Ontario, Canada;
2) I am a graduate of the mineral processing program at the Technical University of Aachen, Germany in 1998;
3) I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 100078050);
4) I have worked as a Mineral Processing 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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for mineral processing aspects of Sections 13 and 17, and portions of Sections 1, 21, 25, 26, and 27;
7) I have not visited the site;
8) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(signed) Oliver M. Peters

Oliver M. Peters, P. Eng.
Principal Metallurgist and President
Metpro Management Inc.

CERTIFICATE OF AUTHOR

To Accompany the NI 43-101 Technical Report entitled **"Preliminary Economic Assessment (PEA) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of January 8, 2021.

I, David Ritchie, P. Eng., do hereby certify that:

1) I am a Geotechnical Engineer and Engineering Service Line Manager with SLR Consulting (Canada) Ltd., with an office at 36 King Street East, 4th Floor, Toronto, Ontario, Canada;
2) I am a graduate from Ryerson Polytechnic University in 1995 with a B. Eng in Civil Engineering, and from the University of Western Ontario in 2000 with an M. Eng. (Geotechnical);
3) I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 90488198);
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) #3 of the Tamarack North Project – Tamarack, Minnesota** " dated January 8, 2021 and am responsible for portions of Sections 3 and 18.6;
7) I have not visited the site;
8) My prior involvement with the property is limited to participation in previous technical reports and in the previous PEAs for 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; and
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 8th day of January 2021.

(signed) David Ritchie

David Ritchie, P. Eng.
Engineering Service Line Manager
SLR Consulting (Canada) Ltd.

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) #3 of the Tamarack North Project – Tamarack, Minnesota"** (the "Technical Report"), prepared for Talon Metals Corp. with an effective date of: January 8, 2021.

(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-six years of mine geology, mineral resource estimation and consulting experience in a variety of mineral projects nationally and internationally covering gold and base metal deposits including 9 years of direct nickel-copper, magmatic sulphide deposit experience with Vale in Sudbury, Ontario (formerly INCO LTD).

(d) My most recent personal inspection of the property described in the Technical Report occurred on July 16th, 2014 and 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 Mineral Resource estimate and technical report titled "**First Independent Technical Report on the Tamarack North Project, Tamarack, Minnesota**" with an effective date of August 29, 2014 and have completed an interim Mineral 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 as well as the technical report titled "**Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota**" with an effective date of December 14, 2018 and the technical report titled **"Updated Preliminary Economic Assessment (PEA) of the Tamarack North Project – Tamarack, Minnesota"** with an effective date of March 12, 2020.

(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; and

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(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 8th day of January 2021.

(signed and sealed) Brian Thomas

____

Brian Thomas, P. Geo.
Senior Resource Geologist
Golder Associates Ltd.

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