IMPERIAL MINING GROUP LTD

CRATER LAKE SCANDIUM PROJECT, PRELIMINARY ECONOMIC ASSESSMENT

NATIONAL INSTRUMENT (NI) 43-101 REPORT

CRATER LAKE SCANDIUM PROJECT, PRELIMINARY ECONOMIC ASSESSMENT

PRELIMINARY ECONOMIC ASSESSMENT (NATIONAL INSTRUMENT [NI] 43-101 REPORT)

IMPERIAL MINING GROUP LTD

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Standards of Disclosure for Mineral Projects Technical Report for Imperial Mining Group Ltd by WSP Canada Inc. (WSP). The quality of information, conclusions, and estimates contained herein are consistent with the quality of effort involved in WSP's services. The information, conclusions, and estimates contained herein are based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Imperial Mining Group Ltd subject to the terms and conditions of its contract with WSP and relevant securities legislation. The contract permits Imperial Mining to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Imperial Mining Group Ltd. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

ABBREVIATIONS AND ACRONYMS

Acronyms	Term
AA	Atomic Absorption
ABA	Acid-Base Accounting
Acronyms	Term
AGP	AGP Mining Consultants
ARD	Acid Rock Drainage
ATV	All-terrain vehicle
AWG	Acidified water glass
CA	Core angle
CCB	Climate Change Branch
Ce	Cerium
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves
CIP	Carbon-in-Pulp
Co	Cobalt
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
COV	Coefficient of variation
CRM	Certified reference material
CSS	Closed Side Setting
Cu	Copper
DDH	Diamond drill hole
DFLR	Department of Fisheries and Land Resources
DFO	Department of Fisheries and Oceans
DMAE	Department of Municipal Affairs and Environment
DMS	Dense media separation
DNR	Department of Natural Resources
DTM	Digital Terrain Models
DTW	Department of Transportation and Works
Dy	Dysprosium
Dy2O3	Dysprosium oxide
EA	Environmental Assessment
EAD	Environmental Assessment Division
ECCC	Environment and Climate Change Canada
EEM	Environmental Effects Monitoring
EIS	Environmental Impact Statement
EMPA	Electron microprobe analysis
EPR	Environmental Preview Report
Er	Erbium
ESA	Endangered Species Act
ESA	Environmental Site Assessment
Eu	Europium
FAL	Freshwater Aquatic Life
FDP	Final Discharge Point
FEL	Front-End Loader
G&A	General and administration
GAC	Granular Activated Carbon
Gd	Gadolinium
GESTIM	Gestion des titres miniers (the MERN's online claim management system)
GHG	Greenhouse Gas
GPS	Global positioning system
GSC	Geological Survey of Canada
HADD	Harmful Alteration and Disruption or Destruction
HG	High Grade
HLS	Heavy liquid separation

Acronyms	Term
Ho	Holmium
HPC	High pressure caustic
HREE	Heavy rare earth element
HREO	Heavy rare earth oxide
IAA	Impact Assessment Act
IAA	Impact Assessment Agency
ICP	Inductively coupled plasma
ICP-MS	Inductively coupled plasma/mass spectrometry
ID2	Inverse distance squared
IEC	International Electrotechnical Commission
INTSYN	Intermediate syenite
ISO	International Organization for Standardization
IX	Ion exchange
JORC	Joint Ore Reserves Committee
La	Lanthanum
La2O3	Lanthanum oxide
LCT	Locked cycle test
LG	Low Grade
LIMS	Laboratory Information Management System
LIMS	Low-intensity magnetic separation
LMREE	Light-medium rare earth element
LOM	Life-of-Mine
LREE	Light rare earth element
LREO	Light rare earth oxide
Lu	Lutecium
MD&A	Management Discussion and Analysis
MDMER	Metal and Diamond Mining Effluent Regulations
MEND	Mine Environment Neutral Drainage
MERN	Ministère de l'Énergie et des Ressources Naturelles du Québec (Québec's Ministry of Energy and Natural Resources)
MGGA	Management of Greenhouse Gas Act
ML	Metal Leaching
MLA	Mineral liberation analysis
MRE	Mineral resource estimate
MSc	Master of Science
NAD	North American Datum
Nb	Niobium
Nd	Neodymium
Nd2O3	Neodymium oxide
Ni	Nickel
NI 43-101	National Instrument 43-101 (Canadian Securities Administrators) (Regulation 43-101 in Quebec)
NL	Newfoundland and Labrador
NLDMAE	Newfoundland and Labrador Department of Municipal Affairs and Environment
NN	Nearest neighbour
NP	Neutralization Potential
NPV	Net Present Value
NRCan	Natural Resources Canada
NSR	Net Smelter Return
NTS	National Topographic System
OB	Overburden
OK	Ordinary kriging
OLFESYN	Olivine ferro-syenite
ON	Ontario
PAG	Potentially Acid Generating
PAO	Provincial Archaeology Office

Acronyms	Term
PEA	Preliminary Economic Assessment
PEG	Pegmatitic dyke
PFS	Prefeasibility study
PLS	Primary leach solution
POM	Intermediate porphyry
PPD	Pollution Prevention Division
PPE	Personal Protective Equipment
Pr	Praseodymium
Pr2O3	Praseodymium oxide
PXFESYN	Pyroxene ferro-syenite
QA	Quality assurance
QA/QC	Quality Assurance / Quality Control
QC	Quebec
QC	Quality control
QEMSCAN	Quantitative evaluation of minerals by scanning electron microscopy
QFP	Quartz Feldspar Porphyry
QP	Qualified person (as defined in National Instrument 43-101)
RAA	Revenue Administration Act
RAR	Return Air Raise
RC	Reverse Circulation
RCP	Rehabilitation and Closure Plan
REE	Rare earth element
REO	Rare earth oxide
RMM	Rambler Ming Mine
RMR	Rock Mass Rating
ROM	Run of Mine
RQD	Rock quality designation
SAR	Species at Risk
SARA	Species at Risk Act
Sc	Scandium
Sc2O3	Scandium oxide
SCC	Standards Council of Canada
SD	Standard deviation
SEM	Scanning electron microscope
SG	Specific gravity
SI units	International System of Units
SIGEOM	Système d'information géominière (the MERN’s online spatial reference geomining information system)
Sm	Samarium
SOCC	Species of Conservation Concern
SPT	Sprucy Pond Trend
SRK	SRK Consulting Limited
SX	Solvent extraction
SYN	Syenite
Tb	Terbium
Tb4O7	Terbium oxide
Ti	Titanium
TIC	Total Inorganic Carbon
Tm	Thulium
TREO	Total rare earth oxide
USD:CAD	American-Canadian exchange rate
UTM	Universal Transverse Mercator
VMS	Volcanic Massive Sulphide

Acronyms	Term
WHIMS	Wet high-intensity magnetic separation
WMF	Waste Management Facility
WNS	White-nose Syndrome
WRMD	Water Resources Management Division
WSP	WSP Canada Inc.
XRD	X-Ray diffraction
XRT	X-Ray transmission
XRT	X-Ray Transmission
Y	Yttrium
Yb	Ytterbium

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1 EXECUTIVE SUMMARY

1.1 Introduction

In September 2021, Imperial Mining Group Ltd. (IPG) commissioned WSP Canada Inc to perform a Preliminary Economic Assessment of the Crater Lake Project in Québec (the “2022 PEA”) in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43-101.

The 2022 PEA was performed by WSP, except for Items 4 to 12, Item 14 and Item 23 which were prepared by InnovExplo as direct input from the 2021 Mineral Resource Estimate (the “2021 MRE”).

This 2022 PEA is preliminary in nature. There are no “Measured Resources”. In addition to Indicated resources, the mine plan presented in this study includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such, there is no certainty that this 2022 PEA based on these mineral resources will be realized.

The effective date of this 2022 PEA is July 26, 2022.

This PEA follows the work done previously by InnovExplo in the 2021 MRE.

Pierre Guay, Vice President of Exploration for Imperial Mining Group ltd. (“Imperial”), retained InnovExplo Inc. (“InnovExplo”) to prepare a technical report (the “Technical Report”) to present and support the results of a mineral resource estimate (the “2021 MRE”) for the Crater Lake Project (the “Project” or the “Property”) in accordance with National Instrument 43-101 Respecting Standards of Disclosure for Mineral Projects (“NI 43-101”) and Form 43-101F1.

The effective date of the 2021 MRE done by InnovExplo is September 17, 2021.

The Project is wholly owned by Imperial. It consists of the TGZ target at an advanced exploration stage, with a mineral resource estimate, and several target areas at an early exploration stage.

Imperial is a Canadian-based exploration and development company focused on advancing its Quebec properties for copper-zinc, gold, and technology metals. The corporate headquarters is at 410 Saint-Nicolas, Suite 236, Montreal, Quebec, H2Y 2P5. Imperial is a public company trading on the Toronto Stock Exchange (TSX) under the symbol “IPG”.

WSP is one of the world's leading engineering professional services consulting firms, based in Montréal, Québec.

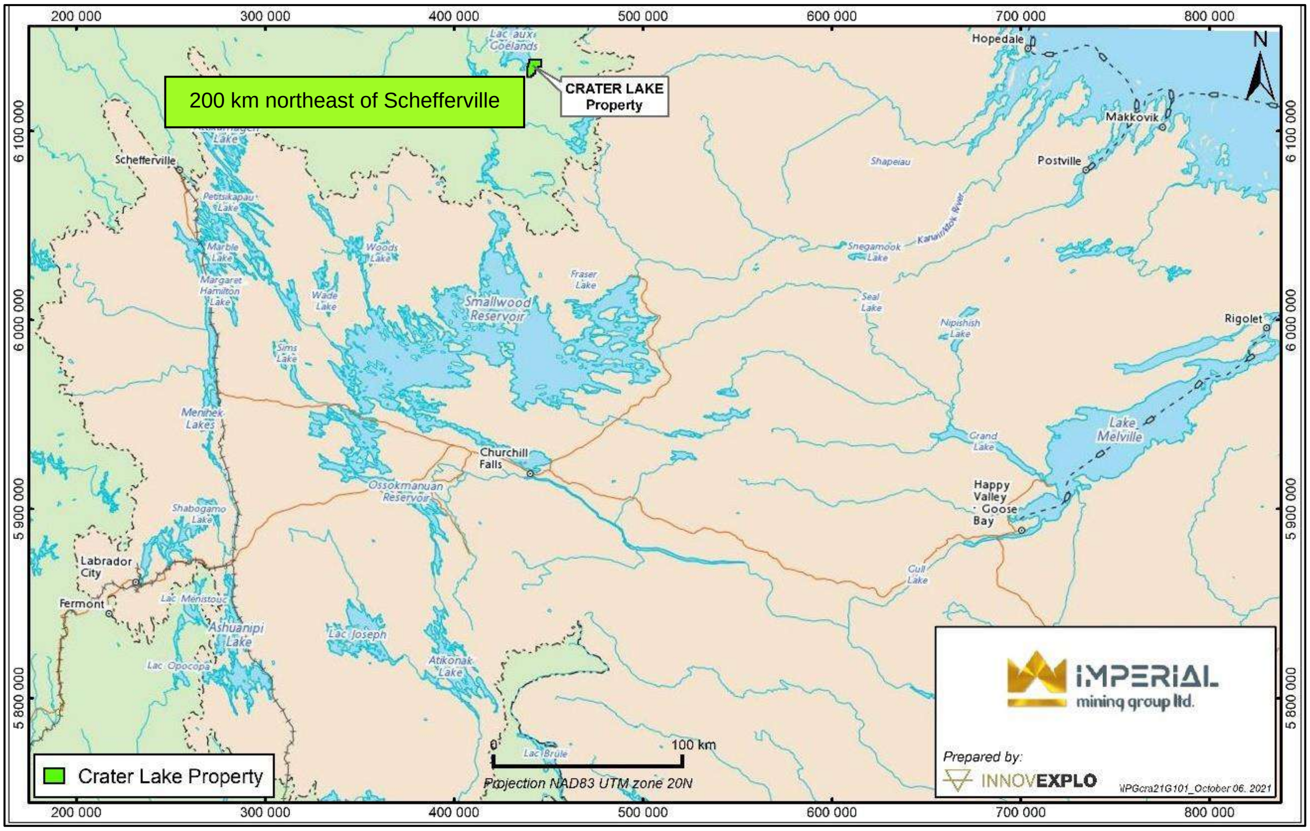
InnovExplo is an independent mining and exploration consulting firm based in Val-d’Or, Québec.

1.2 Property Description and Location

The Property is located near the Quebec-Labrador provincial border, approximately 200 km northeast of the city of Schefferville, Quebec, 190 km southwest of Nain, Newfoundland and Labrador (“NL”), and 300 km northwest of Happy Valley–Goose Bay, NL (Figure 1-1).

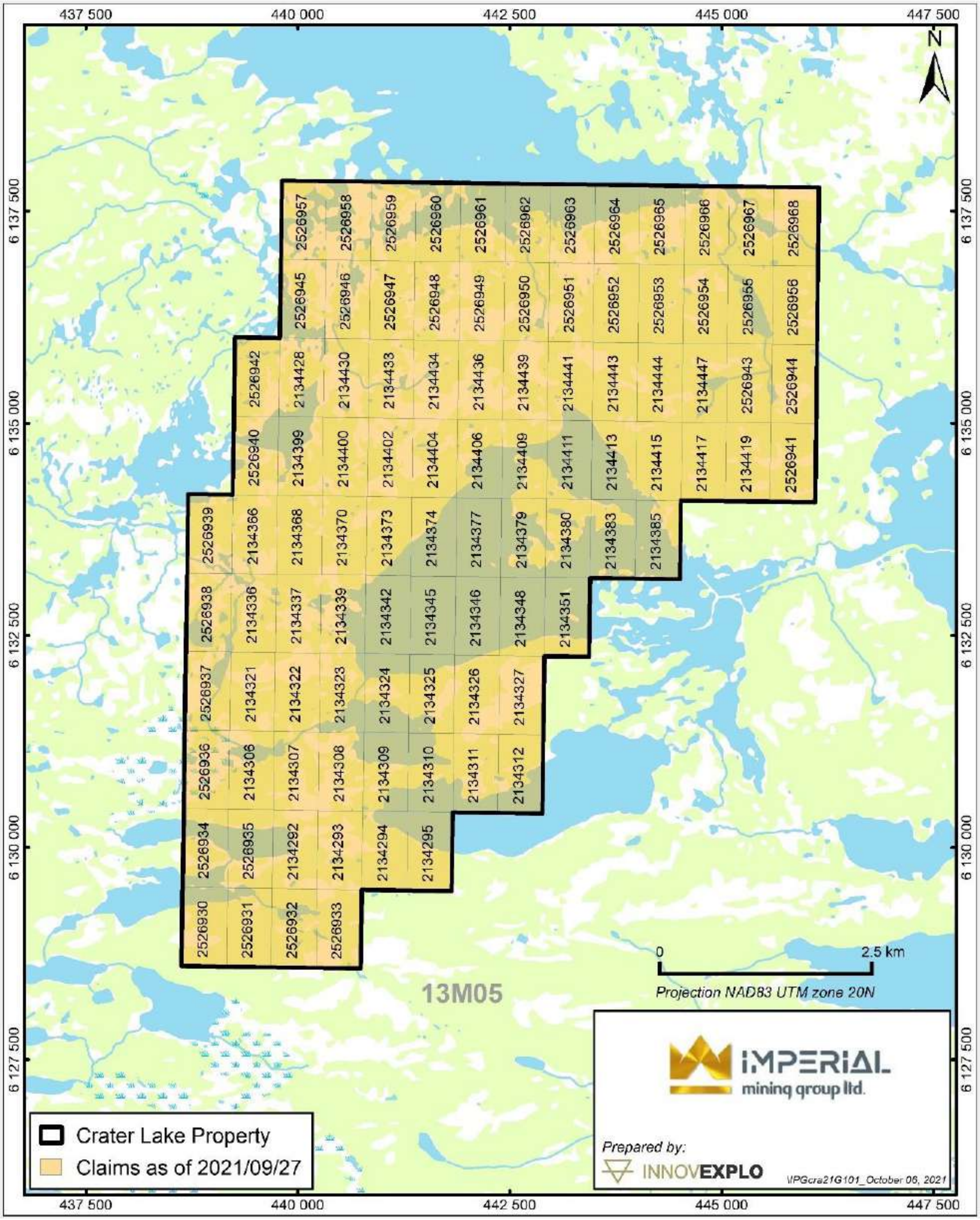
The nearest mine is Vale’s nickel-copper-cobalt mine at Voisey’s Bay, roughly 155 km to the northeast, on the coast of Labrador.

Figure 1-1: Project Location



The Property comprises 96 mineral claims and covers approximately 47.0 km^2 (Figure 1-2).

Figure 1-2: Mineral Claims



1.3 History

Prior to 1979, there were no known exploration activities on the Property.

Details of the historical work on the Property in Figure 1-1 and summarized in Table 1-1.

Table 1-1: Historical Ownership and Work on the Crater Lake Property

Year	Organization	Work
1979	Geological Survey of Canada	Airborne gamma-ray spectrometry
1980		Lac Chapiteau and Lac Ramusio map sheets completed at 1:50,000 scale
1996	Major General Resources Ltd. and Donner Resources Ltd	Surface geological and geochemical programs on the Lac Chapiteau Property
2007	Freewest Resources Canada Inc.	Field exploration (6 samples collected)
2008	Freewest Resources Canada Inc. (Freewest) and Quest	Freewest's uranium property assets, including the current Property, were transferred to Quest Uranium Corporation (Quest Uranium)
2009	Geological Survey of Canada	Open File including ten maps at 1:250,000 scale covering portions of western Labrador, north of Churchill Reservoir, and adjoining parts of Quebec
	Newfoundland and Labrador, Department of Natural Resources	Re-analysis of historical lake-sediment and lake-water geochemistry surveys (1978 to 2005) for additional elements and released in a new Open File
	Quest Rare Minerals Inc. (Quest)	Prospecting and sampling
		Helicopter-borne high-resolution magnetic and radiometric survey
2010		Petrographic study of 14 thin sections from samples collected in 2009
		Glacial till survey (1,222 samples)
		Models from airborne data as the starting point for modelling 4 standalone lines of ground magnetics
		Drilling program (8 DDHs): 1,170.15 m drilled and 663 samples
2010-2012	Quebec MERN, McGill University and Quest	Joint project to complete a Master's thesis to characterize the syenite intrusion and associated REE mineralization at Crater Lake. Thesis submitted in October 2012.
2011	Quest	Surface exploration program (prospecting, mapping and sampling). 101 stations and 199 collected samples.
		Drilling program: 6 DDHs (1,894 m and 1,171 samples)
2012		Surface exploration (prospecting, mapping, geochemical till survey). Additional mapping and channel sampling in selected areas. 261 stations, 231 grab samples, and 80 samples from 11 channels.
		Drilling program: 11 DDHs (2,498 m and 1,395 samples)
		Ground magnetics survey over two grids on the property to further investigate airborne geophysical anomalies

Year	Organization	Work
2013		A broader ground magnetic survey to cover the entire circular geophysical anomaly.
2014		Drilling program: 7 DDHs (1,446 and 879 samples).
2015-2017	Peak Mining	Peak Mining did not conduct any exploration work or drilling on the Property.
2018-2022	Imperial Mining Group	Exploration work including drilling and channel samples on the Property leading to this 2022 PEA.

1.4 Geological Setting and Mineralization

The syenite intrusion of Crater Lake is located in the Churchill Province. It intrudes or is coeval with the southeast end of the Mistastin Batholith, which covers an area of approximately 5,000 km2. The dominant lithologies of the batholith are granite and quartz monzonite with pyroxene, which are cut by younger biotite-hornblende granite, which is in turn intruded by a smaller olivine syenite, the Crater Lake syenite. Uranium-lead dating of three zircons places the age of the batholith at approximately 1.4 Ga (Petrella 2011).

The Crater Lake syenites are interpreted to be a late differentiate product of the Mistastin Batholith. The dominant exposed lithology is coarse- to medium-grained, massive syenite, which is mainly composed of perthitic K-feldspar and 1 to 10% by volume of interstitial ferromagnesian minerals (Petrella 2012). A magnetic and melanocratic unit, ferro-syenite, which commonly contains greater than 50% by volume of ferromagnesian minerals, occurs as large continuous to discontinuous subvertical and conical bodies, sills, narrow dikes and inclusions in the felsic syenites. Three large ferro-syenite bodies have been found on the property: TGZ, Boulder Lake and STG. Petrella (2012) interpreted the narrow ferro-syenite dikes as having formed by fractional crystallization of ferromagnesian minerals, leaving behind a residual magma that produced the felsic syenites. Assay results from surface samples and from 2014-2021 drill core indicate that the different types of ferro-syenite are the main host to the scandium and REE mineralization at Crater Lake.

At Crater Lake, scandium was enriched in the residual liquid of the parent Mistastin granite magma following extensive fractionation of feldspar, in which scandium is incompatible. This residual liquid became the Crater Lake quartz monzonite magma, which was enriched in scandium and iron. Ring faults developed as a result of caldera collapse, and the magma and minerals were emplaced as a slurry into these faults. The ferro-syenite formed by in situ fractionation of hedenbergite crystals, magnetite and hastingsite, and their physical segregation with the previously crystallized minerals. The extremely high FeO/FeO+MgO content of the quartz monzonite liquid resulted in high partition coefficients for scandium in the hedenbergite and hastingsite, allowing scandium to be incorporated into these minerals at exceptionally high concentrations under magmatic conditions. The physical segregation of hedenbergite and hastingsite in ferro-syenite cumulate rocks through gravitational settling and/or flow differentiation spatially concentrated the Sc-bearing minerals within the intrusion, resulting in the first known scandium deposit hosted by syenite. (Beland, 2021).

The REE mineralization is contained in small primary idiomorphic zircon and hydroxyapatite crystals (identified by XRD analysis). The latter locally form aggregates that were wholly or partly replaced by britholite-(Ce). Hydroxyapatite commonly occur as inclusions in pyroxene, amphibole and, less commonly, fayalite.

1.5 Exploration Status

Since the acquisition of the Property in 2017, Imperial Mining performed the following exploration programs.

2018 geophysic program: Geophysical modelling provided a better understanding of the 3D geometry of the Crater Lake intrusive complex and the vertical and lateral extent of the known areas of scandium mineralization on the Property.

2018 field exploration program: Detailed prospecting and geological mapping over three highly prospective scandium targets (the TGZ, STG and North Target areas). The prospecting and mapping program was followed by mechanical stripping and channel sampling. Scandium-rich outcrops and boulders in the vicinity of the TGZ and STG targets confirmed that both zones correspond to a similar scandium-rich target discovered in 2014 at the Boulder Scandium Zone. A total of 39 grab samples and 41 channel samples were collected. An additional 24 historical core samples were selected for a mineralogical study to be completed at McGill University in Montreal, Quebec.

2019 drilling program: A drilling program totalling 1,014 m in five (5) holes was completed on the TGZ target to evaluate the scandium potential of a high-intensity magnetic anomaly. Drilling took place 600 m north of a historical drill hole that had returned scandium grades of up to 506 g/t Sc2O3 over 19 m along the western side of the Crater Lake intrusion along the same magnetic trend. The program intercepted wide intervals of scandium and TREO+Y mineralization at the TGZ target. Highlights were CL19035 intersecting 314 g/t Sc2O3 and 0.371% TREO+Y over 95.5 m and CL19032 intersecting 341 g/t Sc2O3 and 0.421% TREO+Y over 47.9 m.

2020 geophysical program: A detailed ground magnetic survey was completed on the western half of the Property. The survey covered 130 line-km at a line spacing of 50 m. The survey better defined the scandium-bearing ferro-syenite rock units and fault structures controlling the concentration of scandium mineralization on the Property. Several new magnetic bodies were identified to the east and south of the STG target.

2020 field exploration program: A prospecting and mapping program was conducted over 38.2 km of unexplored terrain on the Property. 8 grab samples, 304 historical core samples and 17 new channel samples were selected for a detailed mineralogical and geochemical study at McGill University in Montreal, Quebec. Furthermore, strongly magnetic, boulders of ferro-syenite were found in the Hilltop target area and 300 m northeast of the STG target.

2020 drilling program: A drilling program totalling 676 m in four (4) holes was completed to test the scandium potential of high-intensity magnetic anomalies in the STG, TGZ and Northern target areas. CL20037 confirmed the lateral continuity of the TGZ target by intersecting intervals grading up to 253 g/t Sc2O3 over 29.14 m. CL20038 tested a ferro-syenite unit that was previously intersected parallel and east of the main TGZ target. The hole intersected multiple, narrow scandium-bearing ferrosyenite intervals grading up to 244 g/t Sc2O3 and 0.71% TREO+Y over 2.6 m and 192 g/t Sc2O3 and 0.50% TREO+Y over 3.6 m.

2020-2021 winter drilling program: A 14-hole drilling program totalling 2,049 m was completed on the TGZ target. The mineralization has been traced by drilling over 300 m in total strike length down to a vertical depth of up to 200 m. Highlights were CL21048 intersecting 298 g/t Sc2O3 and 0.355% TREO+Y over 111.93 m and hole CL21052 intersecting 299 g/t Sc2O3 and 0.342% TREO+Y over 99.8 m.

2021 field exploration program: A total of 23 channel samples representing a cumulative length of 23.4 m were collected at the STG Target olivine ferrosyenite, located 2.0 km south of the TGZ Target.

2021 summer drilling program: A two-hole drilling program for 345 m was undertaken to undercut channel sampling and geophysical results over the STG Target, located 2.0 km south of the TGZ Target. Hole CL21054 intersected an olivine ferrosyenite grading 252 g/t Sc2O3 and 0.366% TREO+Y over 115.8 m. The hole undercut a previous channel sample assay of 283 g/t Sc2O3 and 0.361% TREO+Y over 7.0 m.

1.6 Mineral Resource Estimate

The mineral resource estimate for the Crater Lake Project (the "2021 MRE") was prepared by Marina Iund, P.Geo., Paul Daigle, P.Geo., and Carl Pelletier, P.Geo., using all available information.

The studied area covers the mineralized domains collectively known as the TGZ target.

The 2021 MRE was established for scandium, lanthanum, praseodymium, neodymium, terbium and dysprosium. Other REEs were not included in the estimate.

The resource area has a NE-SW strike length of 500 m, a width of 120 m, and a vertical extent of 250 m below the surface. The 2021 MRE was based on a compilation of recent diamond drill holes ("DDH") completed by the issuer.

The Surpac database contains all 25 DDH, which corresponds to all the holes drilled on the Project. The holes cover the strike length of the Project at a regular drill spacing of 50 m.

The geological model was modelled in Leapfrog. The main lithologies of the deposit are massive syenite ("SYN") intruded by olivine ferro-syenite ("OLFESYN"), four (4) pyroxene ferro-syenites ("PXFESYN") and an intermediate syenite ("INTSYN"). Later pegmatitic dykes ("PEG") and intermediate porphyries ("POM") cut all units. The OLFFESYN, PXFESYN and INTSYN solids were used as mineralized domains. These domains are subvertical with an NE-SW strike.

The authors believe that the current mineral resource estimate can be classified as Indicated and Inferred mineral resources based on geological and grade continuity, data density, search ellipse criteria, drill hole spacing and interpolation parameters. The authors also believe that the requirement of "reasonable prospects for eventual economic extraction" has been met by having a cut-off grade based on reasonable inputs amenable to a potential open-pit extraction scenario.

The 2021 MRE is considered reliable and based on quality data and geological knowledge. The estimate follows CIM Definition Standards.

Table 1-2 displays the results of the 2021 MRE for the Project at the official 110.8 CA$/t NSR cut-off.

Table 1-2: 2021 Crater Lake Project Mineral Resource Estimate for an Open Pit Scenario

Category	NSR Cut-off (CA$/t)	Tonnage (Mt)	NSR Total (CA$/t)	Sc_2O_3 (g/t)	Dy_2O_3 (g/t)	La_2O_3 (g/t)	Nd_2O_3 (g/t)	Pr_2O_3 (g/t)	Tb_4O_7 (g/t)
Indicated	110.8	7.3	413	282.01	65.72	605.82	595.78	160.41	11.65
Inferred	110.8	13.2	386	264.24	62.24	568.63	573.04	154.02	11.13

Notes to accompany the Mineral Resource Estimate:

1. The independent and qualified persons for the mineral resource estimate, as defined by NI 43 101, are Marina Iund, P.Geo. (InnovExplo Inc.), Paul Daigle, P.Geo. (InnovExplo Inc. associate) and Carl Pelletier, P.Geo. (InnovExplo Inc.). The effective date of the estimate is September 17, 2021.
2. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. The mineral resource estimate follows current CIM Definition Standards.
3. The results are presented in situ and undiluted and considered to have reasonable prospects of economic viability.
4. The estimate encompasses three mineralized domains using the grade of the adjacent material when assayed or a value of zero when not assayed.
5. High-grade capping supported by statistical analysis was done on raw assay data before compositing: La_2O_3 (3690 g/t), Pr_2O_3 (1380 g/t), Nd_2O_3 (2100 g/t), Dy_2O_3 (215 g/t). No capping was applied to Sc_2O_3 and Tb_4O_7.
6. The estimate was completed using a sub-block model in GEOVIA SURPAC 2021 with user block size of 5m x 5m x 5m and minimum block size of 1.25m x 1.25m x1.25m. Grades interpolation was obtained by ID2 using hard boundaries.
7. Bulk density values were applied by lithology (g/cm3): INTSYN, OLFESYN = 3.13; PXFESYN = 2.91; SYN = 2.7; POMSYN = 2.77; PEG = 2.63 and OB = 2.0.
8. The mineral resource estimate is classified as indicated and inferred. The Indicated mineral resource category is defined with a minimum of three (3) drill holes in areas where the drill spacing is less than 60 m, and reasonable geological and grade continuity have been demonstrated. The Inferred category is defined with a minimum of two (2) drill holes in areas where the drill spacing is less than 120 m, and reasonable geological and grade continuity have been demonstrated. Clipping boundaries were used for classification based on those criteria.
9. The mineral resource estimate is pit-constrained with a bedrock slope angle of 45° and an overburden slope angle of 30°. It is reported at a NSR cut-off of 110.8 CA$/t .The NSR cut-off was calculated using the following parameters: processing cost = CA$14.89; transportation cost (concentrate transportation from mine site to processing plant): CA$7.19; refining and selling costs = CA$ 88.71; Sc_2O_3 price = US$1,500/kg; La_2O_3 price = US$0.6/kg; Pr_2O_3 price = US$29/kg; Nd_2O_3 price = US$29/kg; Tb_4O_7 price = US$386/kg; Dy_2O_3 price = US$124/kg; USD:CAD exchange rate = 1.25; scandium recovery to high grade scandium oxide product = 76.0%; REE recovery to mixed REE carbonate = 63.0%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
10. The number of metric tons was rounded to the nearest thousand, following the recommendations in NI 43 101 and any discrepancies in the totals are due to rounding effects.
11. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

1.7 Mineral Reserve Estimate

This technical report is a Preliminary Economic Assessment and does not require a chapter on Reserves.

1.8 Mining Methods

1.8.1 Open Pit Mining

The deposit is mined as open-pit with trucks and shovels. The mine life is 25 years. It is assumed that only rock materials will need drilling and blasting, therefore overburden material will not require drilling and blasting. Overburden and rock material will be hauled with 47-tonnes truck mainly on double lane traffic in-pit and ex-pit. Rock and overburden material will be loaded by 5.8 m³ bucket capacity hydraulic excavators and 6.4 m³ loaders.

1.8.2 Geotechnical Considerations

No geotechnical study was available for this study. No geotechnical study was part of the mandate of this study. Hence, the geotechnical parameters used in this PEA are conservative at a 45° overall slope angle considering that the pit is shallow. The overall slope angle takes into consideration the ramp and the berm widths.

1.8.3 Hydrogeological Considerations

No hydrogeological study was available for this study. No hydrogeological study was part of the mandate of this study. Hence, dewatering costs were not estimated in this study.

1.8.4 Mine Design

An open pit mine design that follows the economic pit shell and a mine life constraint of 25 years has been developed as well as a waste rock stockpile and an overburden stockpile. Mine operations cover drilling, blasting, loading, and hauling to meet the mine production schedule.

1.8.5 Mine Production

Prior to mine the Crater Lake Deposit, striping is required to remove topsoil and overburden. The total overburden material is 2.4 Mt. Total waste rock material is 18.9 Mt. Total Mineralized Material to be hauled to Mill is 10.6 Mt diluted at 5% with grades of 268.3 g/t Sc_2O_3 , 62.4 g/t Dy_2O_3, 583.0 g/t La_2O_3, 567.0 g/t Nd_2O_3, 152.8 g/t Pr_2O_3 and 11.1 g/t Tb_4O_7.

The mine production schedule follows the tonnages coming from three push-back phases: starter pit, middle pit, and ultimate pit.

1.9 Metallurgical Testwork

Testwork was performed from 2018 to 2022 by SGS Mineral Service in Canada and ANZAPLAN in Germany.

After mineralogical examination it was determined that the processing would have to occur in two steps.

1. Beneficiation
2. Hydrometallurgical processing

Beneficiation consists of low intensity and high intensity magnetic separation. To optimize the magnetic separationprocess, the optimal particle size was determined to be less than 0.15 mm and yielded up to 87.5% Scandium recovery and 68% Total Rare Earth Elements (TREE) recovery. Beneficiation using flotation and gravity separation were less efficient.

Hydrometallurgical processing involved two MET samples. Both samples underwent complex multiple stage leaching including high pressure caustic leaching and two stages of solvent extraction producing scandium oxide and TREE oxides separately. The scandium leach recoveries were 91% for MET1 and 84% for MET2. The TREE leach recoveries were 94% and 83% for MET 1 and MET 2, respectively

Future testwork is planned to include optimization of the hydrometallurgy process and Al-2%Sc Master alloy development as well as piloting the mineral beneficiation and hydrometallurgy processes.

1.10 Recovery Methods

Based on the mineral processing and hydrometallurgical testing performed to date scandium can best be recovered using magnetic separation to produce a scandium rich mineral concentrate and by treating the concentrate through caustic leaching and solvent extraction to produce a high-grade scandium oxide (Sc_2O_3) and a mixed REE hydroxide product. The scandium oxide or scandia will be processed together with alumina to produce Al-2%Sc master alloy in a direct electrolysis process similar to the Hall-Héroult electrolytic method used for the production of primary aluminum.

The processing operations will be split in two locations. The beneficiation plant, for the production of Sc/REE rich concentrate, will be located at Crater Lake near the open pit mine about 200 km NE of Schefferville, QC. The mineral concentrate produced at the mine site will be transported to the hydrometallurgical plant located in Sept-Îles , QC, to produce mixed REE hydroxide product, scandia, and Al-2% master alloy. The descriptions of the various areas encompassing the processing plants, including overall design criteria and block flow diagrams, are provided in chapter 17.

The beneficiation plant will be designed for an annual throughput capacity of 426,100 t/a. Due to the harsh climate at Crater Lake, the plant will operate for six months out of the year, from May to October. The plant daily feed rate will thus be 2380 t/d of ore with availability of 92%. The main unit operations of the beneficiation plant will include run-of-mine crushing, crushed ore stockpiling and reclaiming, grinding/classification, magnetic separation, tailings thickening and filtration, concentrate thickening and filtration, concentrate drying, roasting, cooling, and final dry low intensity magnetic separation. The overall scandium recovery through magnetic separation will be 90.2% with an overall weight recovery of 51.3%. The plant is therefore expected to produce 218,540 t/a of Sc/REE rich mineral concentrate.

The hydrometallurgical plant will be designed for an annual feed rate of 218,540 t of mineral concentrate and will operate year-round with a daily throughput of 600 t/d and availability of 92%. The main unit operations of the hydrometallurgical plant include concentrate loading and pulping, high pressure and temperature caustic leaching, hydrochloric acid leaching, scandium recovery systems through solvent extraction, mixed REE recovery by precipitation, Al-2%Sc master alloy production through electrolysis, as well as hydrochloric acid recovery system, caustic recovery system, final residue neutralisation and filtration. The overall scandium recovery through hydrometallurgy will be 81% and will thus produce 87.3 t/a of Sc_2O_3, with a mixed REE hydroxide product estimated to be 1,865 t/a. The Sc_2O_3 will be processed together with alumina to produce 2,562 t/a of Al-2%Sc master alloy.

1.11 Infrastructure

The project includes three different sites: Crater Lake Project Site, Emeril Rail loading station and Sept-Îles site.

The Project envisions the construction of the following key infrastructure items:

1.11.1 Crater Lake Site Infrastructure

Crater Lake’s site infrastructure includes the following:

- Access, haulage, and service roads
- Open pit mine
- Waste rock and overburden stockpiles
- Camp complex and services
- Multiservice building (truck shop, warehouse, dry and mine offices)
- Surface water management ponds, ditches, and pumping stations
- Tailings storage facility (TSF)
- Final effluent water treatment plant (WTP)
- Mineralized material cruising station and storage dome
- Process plant
- Concentrate storage area
- Diesel generators power plant and storage
- Air strip with terminal building
- Wind turbine
- Explosive and magazine storage

1.11.2 Sept-Îles Site Infrastructure

Sept-Îles' site infrastructure includes the following:

- Access, haulage, and service roads
- Hydrometallurgical and master alloy transformation plants
- Surface water management ponds, ditches, and pumping stations
- Filtered tailings storage facility
- Final effluent water treatment plant
- Gatehouse
- Warehouse and reagent storage area
- Railway connection and concentrate unloading area
- Concentrate storage area
- Electrical substation

1.11.3 Emeril Rail Loading Station

The Emeril Rail Loading Station infrastructure includes the following:

- Access road
- Concentrate storage area
- Rail loading facility
- Railway connection

1.11.4 Logistics

The concentrate produced during the summer will be stored in a dome at Crater Lake until winter. Concentrate will be hauled in bulk carrier trucks 152km over a single lane width winter road from Crater Lake to the end of the Orma Lake Road, four months of the year, from the start of December to the end of March. With the winter road being mainly an overland route, passing lanes will be established every 9.5km along the route such that convoys of four trucks each will meet at these passing lanes every 30 minutes.

The 150km Orma Lake Road to Churchill Falls will be refurbished for use during the winter months. The bridges and culverts along the existing Orma Lake Road will be retained.

At Churchill Falls the bulk carrier trucks will travel 199km to offload into a storage dome to be constructed at a newly established siding at the existing Emeril Railway station.

With a 501km one way distance, each round trip will take 19.7 hours per truck requiring two operators being assigned each truck to avoid doubling up of the truck requirements.

The project includes an emergency building midway between Churchill Falls and the Crater Lake site.

At the Emeril station, the IPG owned railway wagons will be loaded from the storage dome for rail transport to the Sept-Îles' hydrometallurgical facility as a year-round operation.

Supplies will be delivered from the port of Happy Valley along the same winter road route. Being a longer 638km one way distance, the round trip for the supply trucks will be 22.6 hours, again requiring two operators each truck to avoid the doubling up on the truck requirements.

The trucking contractor will need to have a fleet of about 60 trucks with 120 operators plus supervision. Front End Loaders with operators will be needed at each of the dome storage locations.

The Sept-Îles' Site includes an unloading arrangement that will be operated by the hydrometallurgical facility workforce.

1.12 Environmental

The project will require several approvals, permits and authorizations to initiate the construction phase, operate and close the project. In addition, Imperial Mining Group Ltd. will be required to comply with any other terms and conditions associated with the decree and authorization issued by the provincial and federal authorities. A list of permits and authorization is provided in Table 20-1.

The closure costs are estimated to a total of 74.4 M$ for Crater Lake Site and 65.1 M$ for the Sept-Îles Site, including 15% contingency. This cost includes site rehabilitation and restoration as well as the post-restoration monitoring for 5 to 10 years of both sites.

1.13 Capital and Operating Costs

The capital and operating cost details are presented in Table 1-3.

Table 1-3: Crater Lake Scandium Project Economic Summary, Base Case Scenario

CAPITAL COSTS	MCA$	($/t milled)
Direct Costs		
Mine Equipment, Crater Lake	$13.7	$1.29
Mill Plant Construction, Crater Lake	$63.7	$6.02
Hydrometallurgical Facility Construction, Sept-Iles	$160.1	$15.13
Electrical Distribution, At Both Crater Lake and Sept-Îles Sites	$14.1	$1.33
Infrastructure, At Both Crater Lake and Sept-Îles Sites	$113.5	$10.73
Tailings Storage Facilities including Water Management, At Both Crater Lake and Sept-Îles Sites	$69.3	$6.55
Initial Winter Road plus Orma Lake Road Rehabilitation	$46.6	$4.40
Between Sites Concentrate Handling Infrastructure	$27.5	$2.60
Crater Lake Camp (200 Person Capacity)	$26.5	$2.50
Pre-Production Mining Licenses	$0.2	$0.02
Pre-Production Mine and Mill Expenses to Capital	$67.9	$6.42
Subtotal Direct Costs	***$602.9***	***$57.00***
Owner's Costs	$6.8	$0.64
Indirect Costs (19% of Directs) *	$102.0	$9.64
Contingency (25% of Directs and Indirects) *	$159.2	$15.05
Total Capital Cost (All In, Taxes Extra)	**$870.1**	**$82.33**
OPERATING COSTS	**MCA$**	**($/t milled)**
Mine Crater Lake	$248.5	$23.49
Mill Crater Lake	$430.9	$40.73
Power Plant Crater Lake	$36.0	$3.40
Surface Mobile Equipment Crater Lake	$27.5	$2.60
Water Management Crater Lake	$192.4	$18.19
Lodging Crater Lake	$140.1	$13.25
Transportation + Domes Between Sites	$1,387.6	$131.18
Hydromet Facility Sc_2O_3 at Sept-Iles	$58.5	$5.53

Hydromet Facility Al + 2% Sc Master Alloy at Sept-Iles	$1,139.8	$107.75
Surface Mobile Equipment at Sept-Iles	$24.8	$2.35
Water Management at Sept-Iles	$108.6	$10.27
less Mine and Mill Pre-Production Expenses to Capital	($67.9)	($6.42)
Total Operating Cost	**$3,726.8**	**$352.32**
Selling General & Administrative Costs	$83.6	$7.90
Royalties (one time buy out)	$2.0	$0.20
Sustaining Capital Costs + Restoration	$207.7	$19.63
Total Operating Cost (All in)	**$4,020.1**	**$380.05**

1.14 Economic Analysis

A life of Mine (LOM) cashflow model was constructed based on the LOM production schedule for the Crater Lake Project using a Discounted Cashflow approach. The analysis generated positive results with a post-tax IRR of 32.8%, a post-tax NPV at 10% of 1,721 million and a post-tax payback of 3.0 years after the commencement of Master Alloy production at the Hydrometallurgical facility. Key outcomes of the economic evaluation for 100% of the Project, before any financing costs, are presented in Table 1-4. All costs are estimated in Canadian dollars (CA$) and referenced as '$', unless otherwise stated. Discounting commences with commercial production.

Table 1-4: Crater Lake Scandium Project Economic Summary, Base Case Scenario

Item	Unit	Value
Production		
Project life (from start of construction to closure)	years	26
Mine life	years	25
Total potential mill feed tonnage	M t	10.6
Average feed grade, Sc2O3	g/t	268
Mill recoveries Sc2O3 (Avg)	%	90%
Hydromet recoveries Sc2O3 (Avg)	%	81%
Payable	%	100%
Commodity Prices		
Sc2O3	US$/kg	1500
	CAN$/kg	1875
Exchange rate	CAN$: US$	1.25
Project Costs		
Average mining cost - OP	$/t milled, OP	23.49
Average milling Cost	$/t milled	40.73
Average hydromet cost	$/t milled	113.29
Average General & Administrative cost	$/t milled	43.63
Average concentrate shipment cost	$/t milled	131.18
Project Economics		
Gross revenue	$M	15,200
Total selling cost estimate	$M	86
Total operating cost estimate	$M	3,727
Total sustaining plus other capital cost estimate	$M	208
Total capital cost estimate	$M	871
Taxes	$M	4,050

Item	Unit	Value
Pre-tax cashflow	$M	10,309
After-tax cashflow	$M	6,259
Discount rate	%	10%
Pre-tax Net Present Value @ 10%	$M	2,971
Pre-tax Internal Rate of Return	%	42.9%
Pre-tax payback period	years	2.5
After-tax Net Present Value @ 10%	$M	1,721
After-tax Internal Rate of Return	%	32.8%
After-tax payback period	years	3.0

A pre-tax sensitivity analysis was conducted on the economic model to test changes in key economic assumptions. The model was found to be most sensitive to fluctuations in metal prices and exchange rates.

1.15 Conclusions and Recommendations

The project generates a positive post-tax IRR of 32.8% and a post-tax NPV at 10% of 1,721 million.

- WSP recommends Imperial Mining Group implement other post-study recommendations that will allow them to proceed to the next stage of study.
- A FEL2, Prefeasibility Study (PFS) that further defines the transportation by rail option is seen to be the next step.
- Resizing of the hydrometallurgical facility resulting from added concentrate deliveries related to a year-round access road opportunity would potentially improve the overall results.

The 2021 MRE for the TGZ target at the Crater Lake Project, at a 110.80 CA$/t NSR cut-off grade, comprises:

- Indicated Resource of 7,315,500 t grading 282 g/t Sc2O3, 66 g/t Dy2O3, 606 g/t La2O3, 596 g/t Nd2O3, 160 g/t Pr2O3, 12 g/t Tb4O7 equivalent to a 413 CA$/t NSR.
- Inferred Resource of 13,158,400 t grading 264 g/t Sc2O3, 62 g/t Dy2O3, 569 g/t La2O3, 573 g/t Nd2O3, 154 g/t Pr2O3, 11 g/t Tb4O7 equivalent to a 386 CA$/t NSR.

The following work program is recommended:

- Complete a 4,000 m drilling program on the southern portion of the TGZ target (Sections 0N to 350N) and at depth (sections 450N and 500N). The goal of this program is to investigate the extensions of the TGZ mineralized domains.
- Perform a Lidar topographic survey to cover the entire Property.
- Perform the planned additional metallurgical testwork (solvent extraction flowsheet development and optimization; development of Al-2%Sc master alloy production technology; and processing pilot program).

2 INTRODUCTION

2.1 Overview or Terms of Reference

This report supports the disclosure in Imperial Mining Group's press released dated June 13, 2022, titled 'IMPERIAL MINING RECEIVES POSITIVE RESULTS FOR THE PRELIMINARY ECONOMIC ASSESSMENT (PEA) FOR CRATER LAKE'. The overall study effective date of the press release is June 6, 2022.

The Mineral Resources are reported in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43-101.

2.2 Report Responsibility, Qualified Persons

This technical report was prepared by qualified persons cited below with their respective responsibility. The professional certification numbers of the QPs are presented in the certification form of each QP (Table 2-1).

Table 2-1: Qualified Persons and Responsibility

Qualified Person	Responsibility
Marina Iund	Items 4 to 13 and 23 as well as co-author of and share responsibility for items 1 to 3, 14 and 25 to 27
Paul Daigle	Co-author of and share responsibility for Items 14, 25 and 26
Carl Pelletier	Co-author of and share responsibility for Items 14, 25 and 26
Mireno Dhe Paganon	Items 17, 21.5, 21.6.2 and portions of Items 1, 3, 24, 25, 26 and 27 that are based on those Items.
Simon Latulippe	Items 18.4, 18.5, 18.6, and 20
William Richard McBride	Items 18.3.3, 18.7, 19, 21, 22 and portions of Items 1, 3, 24, 25, 26 and 27 that are based on those Items.
Zakaria Ould Moctar	Items 16 and portions of Items 1, 2, 3, 24, 25, 26 and 27 that are based on those Items.
Ewald Pengel	Item 13 and portions of Items 1 that are based on those Items.
Eric Poirier	Items 18.1, 18.2, 18.3.1, 18.3.2, 21.1.4 and portions of Items 1, 2, 3, 25, 26 and 27 that are based on those Items.

2.3 Site Visits

Mr. Daigle visited the Property on one occasion (from September 30 to October 1, 2014). During the visits, he reviewed selected drill core and inspected the core storage facility. He also collected drill core samples and surveyed drill hole collars for independent validation.

Ms. Iund visited the Property on one occasion (from May 7 to 9, 2021). During the visits, she reviewed selected drill core and inspected the core storage facility. She also collected drill core samples and surveyed drill hole collars for independent validation.

2.4 Effective Date

The effective date of the 2022 PEA prepared by WSP is July 26, 2022.

The effective date of the 2021 MRE prepared by InnovExplo is September 17, 2021.

2.5 Sources of Information

The documents listed in Item 3 and Item 27 were used to support this Technical Report. Excerpts or summaries from documents authored by other consultants are indicated in the text.

The 2017 NI 43-101 Technical Report (Daigle, 2017) was extensively used in the preparation of Item 4 through Item 6. A complete list of references is provided in Item 27.

The authors' assessment of the Project was based on published material and the data, professional opinions and unpublished material submitted by the issuer. The authors reviewed all relevant data provided by the issuer and/or by its agents.

The author also consulted other sources of information, mainly the Government of Quebec's online claim management and assessment work databases (GESTIM and SIGEOM, respectively) as well as Imperial's technical reports, annual information forms, MD&A reports and press releases published on SEDAR (www.sedar.com).

The authors reviewed and appraised the information used to prepare this Technical Report and believe that such information is valid and appropriate considering the status of the project and the purpose for which this Technical Report is prepared. The authors have thoroughly researched and documented the conclusions and recommendations made in this Technical Report.

2.6 Currency, Units of Measure

The abbreviations and acronyms used in this report are provided immediately before the Table of Contents. Units are included in Table 2-2. All currency amounts are stated in Canadian Dollars ($, CA$, CAD) or US dollars (US$, USD). Quantities are stated in metric units, as per standard Canadian and international practice, including metric tons (tonnes, t) and kilograms (kg) for weight, kilometres (km) or metres (m) for distance, hectares (ha) for area, percentage (%) for grades, and gram per metric ton (g/t) for minor metal grades. Wherever applicable, imperial units have been converted to the International System of Units (SI units) for consistency (Table 2-3).

Table 2-2: List of Units

Symbol	Unit
%	Percent
$, CA$, CAD	Canadian dollar
$/t	Dollars per metric ton
°	Angular degree
°C	Degree Celsius
cm	Centimeter
g	Gram
g/cm^3	Gram per cubic centimetre
g/t	Gram per metric ton (tonne)
ha	hectare
kg	Kilogram
km	Kilometre
km^2	Square kilometre
M	Million
m	Metre
mm	Millimeter
Ga	billion years
ppm	Parts per million
t	Metric tonne (1,000 kg)
T	Tesla
tpy	Metric tonnes per year
US$	American dollar
wt	Wet tonne
y	Year (365 days)

Table 2-3: Conversion Factors for Measurements

Imperial Unit	Multiplied by	Metric Unit
1 inch	25.4	mm
1 foot	0.3048	m
1 acre	0.405	ha
1 pound (avdp)	0.4535	kg
1 ton (short)	0.9072	t

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) who prepared this report relied on information provided by experts who are not QPs. The relevant QPs believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to this technical report.

William Richard McBride, P.Eng., relied on an Ernst and Young (EY) Market Analysis for scandium prepared for Imperial Mining Group (IPG). It is noted in item 19 that the contemplated production plans for IPG would produce 88.5 tons of scandium per year, well below the 1,970 tons forecast by EY for the industry requirements for the year 2040.

William Richard McBride, P.Eng., relied on metal selling prices from IPG for scandium oxide, master alloy Al-Sc2%, and REE oxides. It is noted in item 19 that these prices when compared to the USGS previous 5 year's spot price listing are generally conservative.

Mireno Dhe Paganon, P.Eng., relied on IPG for equipment recommendations and capital cost estimates for caustic leach, solvent extraction and Al-Sc2% master allloy electrolysis processes.

Ewald Pengel, P. Eng., relied on IPG for support in some of the interpretation of hydrometallurgical testwork data.

4 PROPERTY DESCRIPTION AND LOCATION

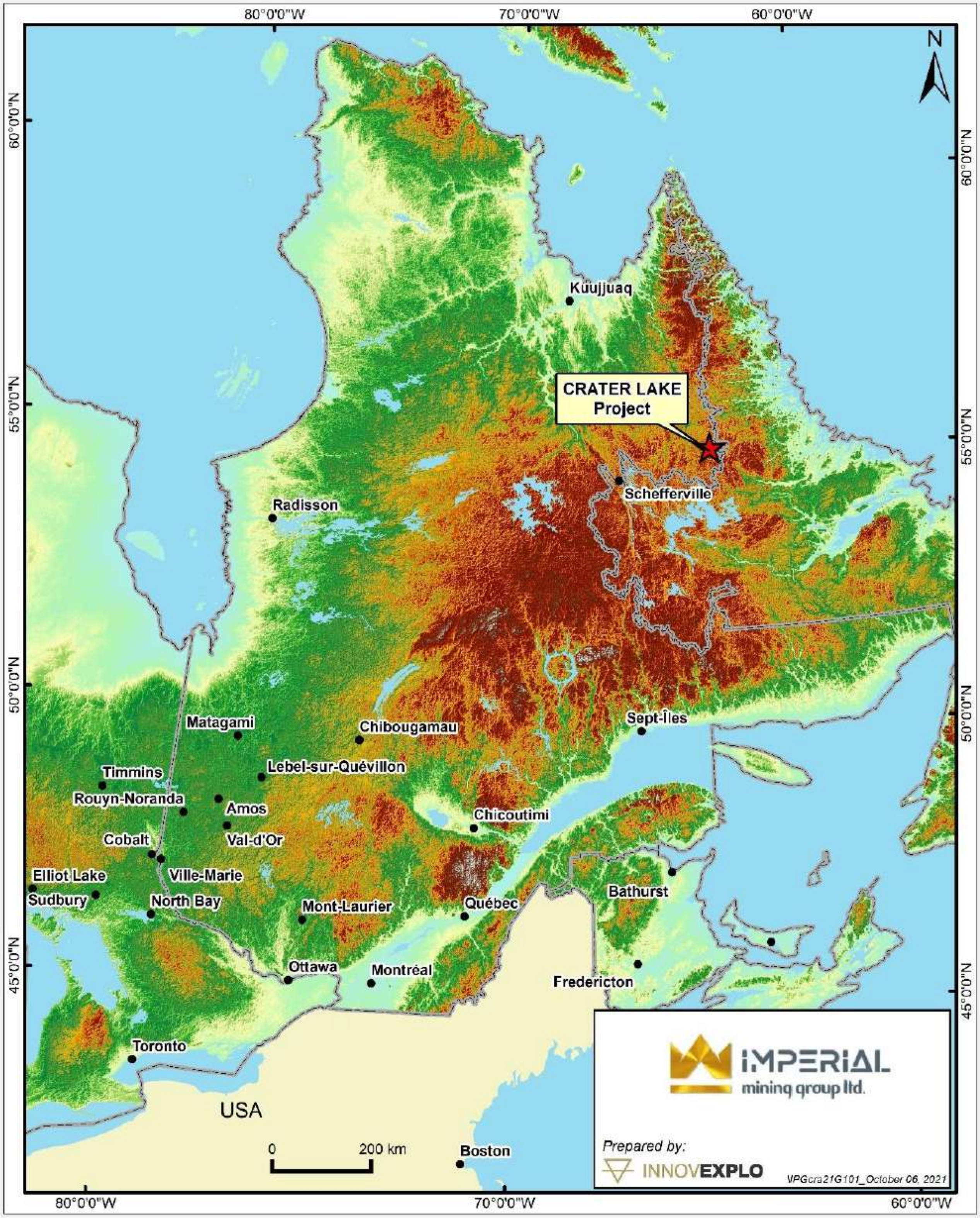
4.1 Location

The Property is located near the Quebec-Labrador provincial border, approximately 200 km northeast of the city of Schefferville, Quebec, 190 km southwest of Nain, Newfoundland, and Labrador (“NL”), and 300 km northwest of Happy Valley–Goose Bay, NL (Figure 4-1).

The Property lies within 1:50,000 scale NTS map sheet 013M05 (Lac Chapiteau) at the approximate latitude and longitude of 55°20' North and 63°54' West (UTM coordinates: 441600E, 6133600N, NAD 83, Zone 20). The Property is in the administrative region of Côte-Nord, governed by the Kativik Regional Government and the Province of Quebec.

The Property is situated approximately 15 km southeast of Lac des Goélands, Quebec, and approximately 66 km southwest of Mistastin Lake, NL, two of the larger lakes in the region.

Figure 4-1: Location of the Crater Lake Property



4.2 Mineral Titles Status

The issuer supplied all maps and tables, and a list of mineral titles comprising the Property. InnovExplo verified the status of all mineral titles using GESTIM, the Government of Quebec's online claim management system (gestim.mines.gouv.qc.ca: most recently viewed September 17, 2021).

The Property is made up of two contiguous mineral claim blocks: Crater Lake and Crater Lake Extension. The Property comprises 96 mineral claims and covers approximately 47.0 km2.

The Crater Lake claim block (the initial Crater Lake property) was acquired in December 2017. It consists of 57 contiguous claims owned 100% by Imperial, covering a total area of 27.9 km2. A 2% net smelter return ("NSR") royalty applies to these claims (see Item 4.4).

In 2018, Imperial acquired the Crater Lake Extension claim package, consisting of 39 mining claims covering a total area of approximately 19.1 km2. These 39 claims are not subject to any royalties and are 100% owned by Imperial.

All claims are current. There are no known outstanding issues at the time of writing.

The claim map is shown in Figure 4-2. A list of the claims is presented in Table 4-1.

Figure 4-2: Mining Title Map for the Crater Lake Property

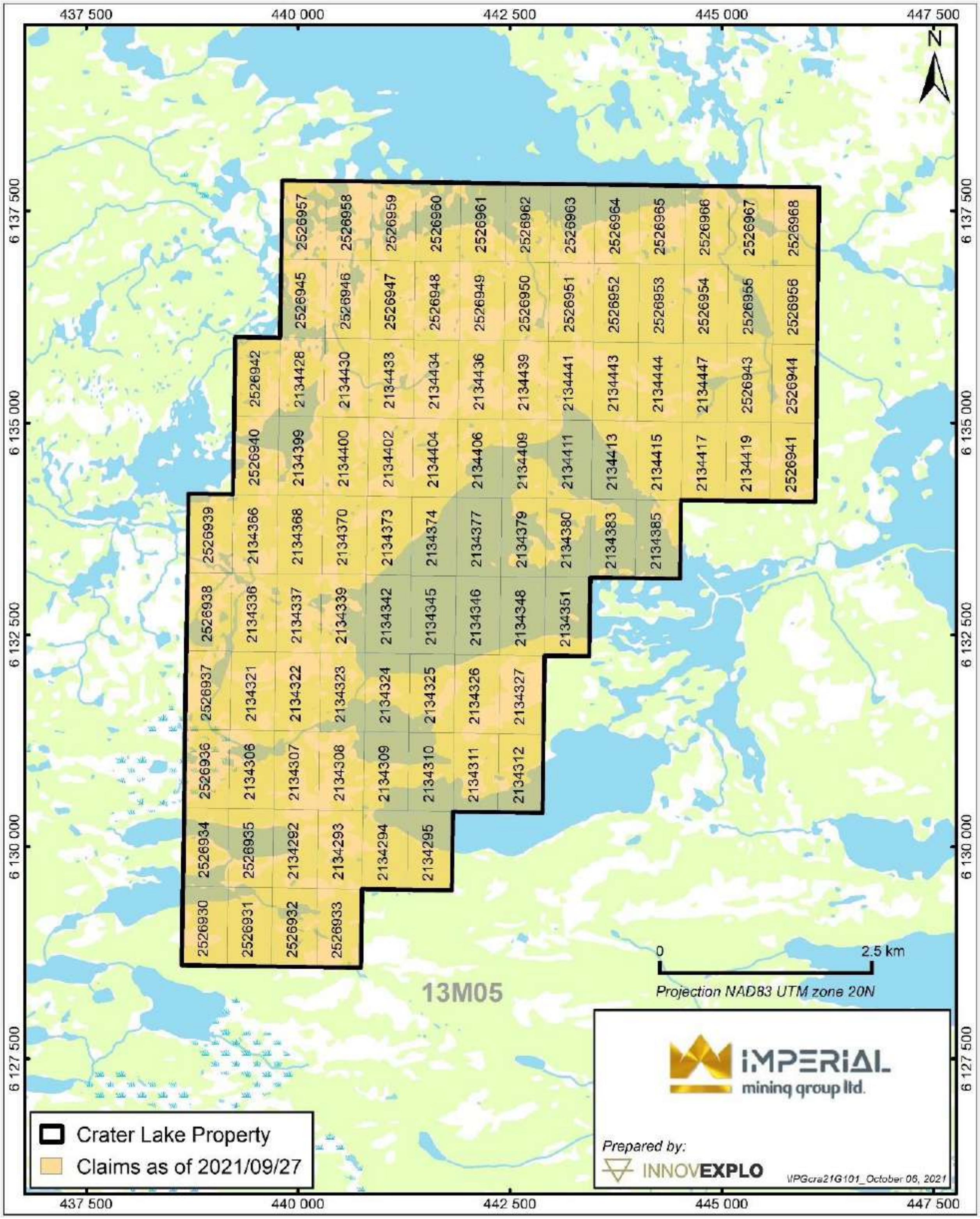


Table 4-1: List of Claims for the Crater Lake Property

Claim No.	Expiry Date	Area (ha)	Claim No.	Expiry Date	Area (ha)
Initial Crater Lake Claims			**Crater Lake Extension Claims**		
2134292	10/29/2022	49.05	2526930	11/11/2021	49.07
2134293	10/29/2022	49.05	2526931	11/11/2021	49.07
2134294	10/29/2022	49.05	2526932	11/11/2021	49.07
2134295	10/29/2022	49.05	2526933	11/11/2021	49.07
2134306	10/29/2022	49.04	2526934	11/11/2021	49.06
2134307	10/29/2022	49.04	2526935	11/11/2021	49.06
2134308	10/29/2022	49.04	2526936	11/11/2021	49.04
2134309	10/29/2022	49.04	2526937	11/11/2021	49.03
2134310	10/29/2022	49.04	2526938	11/11/2021	49.02
2134311	10/29/2022	49.04	2526939	11/11/2021	49.01
2134312	10/29/2022	49.04	2526940	11/11/2021	49.00
2134321	10/29/2022	49.03	2526941	11/11/2021	49.00
2134322	10/29/2022	49.03	2526942	11/11/2021	48.99
2134323	10/29/2022	49.03	2526943	11/11/2021	48.99
2134324	10/29/2022	49.03	2526944	11/11/2021	48.99
2134325	10/29/2022	49.03	2526945	11/11/2021	48.98
2134326	10/29/2022	49.03	2526946	11/11/2021	48.98
2134327	10/29/2022	49.03	2526947	11/11/2021	48.98
2134336	10/29/2022	49.02	2526948	11/11/2021	48.98
2134337	10/29/2022	49.02	2526949	11/11/2021	48.98
2134339	10/29/2022	49.02	2526950	11/11/2021	48.98
2134342	10/29/2022	49.02	2526951	11/11/2021	48.98
2134345	10/29/2022	49.02	2526952	11/11/2021	48.98
2134346	10/29/2022	49.02	2526953	11/11/2021	48.98
2134348	10/29/2022	49.02	2526954	11/11/2021	48.98
2134351	10/29/2022	49.02	2526955	11/11/2021	48.98
2134366	10/29/2022	49.01	2526956	11/11/2021	48.98
2134368	10/29/2022	49.01	2526957	11/11/2021	48.97
2134370	10/29/2022	49.01	2526958	11/11/2021	48.97
2134373	10/29/2022	49.01	2526959	11/11/2021	48.97
2134374	10/29/2022	49.01	2526960	11/11/2021	48.97
2134377	10/29/2022	49.01	2526961	11/11/2021	48.97
2134379	10/29/2022	49.01	2526962	11/11/2021	48.97
2134380	10/29/2022	49.01	2526963	11/11/2021	48.97
2134383	10/29/2022	49.01	2526964	11/11/2021	48.97
2134385	10/29/2022	49.01	2526965	11/11/2021	48.97
2134399	10/29/2022	49.00	2526966	11/11/2021	48.97
2134400	10/29/2022	49.00	2526967	11/11/2021	48.97
2134402	10/29/2022	49.00	2526968	11/11/2021	48.97
2134404	10/29/2022	49.00	Total Claims: 39		
2134406	10/29/2022	49.00			
2134409	10/29/2022	49.00			
2134411	10/29/2022	49.00			
2134413	10/29/2022	49.00			

Claim No.	Expiry Date	Area (ha)	Claim No.	Expiry Date	Area (ha)
Initial Crater Lake Claims			**Crater Lake Extension Claims**		
2134415	10/29/2022	49.00			
2134417	10/29/2022	49.00			
2134419	10/29/2022	49.00			
2134428	10/29/2022	48.99			
2134430	10/29/2022	48.99			
2134433	10/29/2022	48.99			
2134434	10/29/2022	48.99			
2134436	10/29/2022	48.99			
2134439	10/29/2022	48.99			
2134441	10/29/2022	48.99			
2134443	10/29/2022	48.99			
2134444	10/29/2022	48.99			
213447	10/29/2022	48.99			
Total Claims: 57					

4.3 Property Agreements

On December 28, 2017, Imperial completed the acquisition of a 100% interest in the Crater Lake claim block from Peak Mining Corporation ("Peak Mining") in consideration of 7,500,000 Imperial shares (the "Crater Lake Acquisition").

The property acquisition agreement states:

— Peak Mining hereby agrees to sell, assign and transfer to Imperial, and Imperial hereby agrees to purchase and acquire from Peak Mining, an undivided 100% right, title and interest in and to the property, subject only to the royalties, in consideration of the purchaser issuing to Peak Mining 7,500,000 common shares in the capital of Imperial at a deemed price of $0.16 per share.

— Imperial assumes from Peak Mining, their rights and obligations under the Quest Rare Minerals Ltd. (Quest) Royalty Agreement, including for greater certainty Imperial's assumption of all obligations of Peak Mining as "Payor" under the Quest Royalty Agreement.

4.3.1 Imperial, Peak Mining, and NQ Exploration Agreement

On July 11, 2017, Peak Mining signed a letter of intent with NQ Exploration Inc. ("NQ Exploration") for the acquisition of the Crater Lake claim block through a new public company and wholly-owned subsidiary (Imperial). The new subsidiary was to be created for NQ Exploration's Quebec-based properties (the Carheil and Brouillan projects; not the subject of this report) and the Crater Lake property.

On September 11, 2017, NQ Exploration announced the execution of:

- The purchase and sale agreements as well as the arrangement agreement with Imperial, a wholly-owned subsidiary of NQ Exploration, which will be spun out as a separate public company that will own a 100% interest in two other exploration projects (the Opawica and La Ronciere Gold projects; not the subject of this report), subject to the Option.
- A share exchange agreement with AM Resources SAS, an arm's-length Colombian-based private coal mining exploration company, for the reverse take-over of NQ Exploration.

Concurrent with the closing of the above two agreements, Imperial acquired the Crater Lake claim block from Peak Mining.

4.4 Royalties

Quest retains a 2% NSR royalty in the Crater Lake claim block from the acquisition and transfer of the mining rights from Peak Mining on December 28, 2017. Those royalties are retained from the original acquisition and transfer of the Property between Peak Mining and Quest on July 11, 2017 (Item 4.3.1). The royalty may be purchased at any time by the payor for an aggregate of $2,000,000 or in two transactions, each for 50% of the royalty in exchange for the sum of $1,000,000. Nothing herein shall prevent the payor from simultaneously completing the two transactions, being 100% of the royalty in exchange for the sum of $2,000,000. In 2018, Quest filed for bankruptcy. At the time of the report, the status of the Quest’s royalty was not clarified.

4.5 Permits

Imperial has complete surface access to the Property. However, any new work programs will require that the appropriate permits and processes be completed under the MERN guidelines.

The author is not aware of any environmental liabilities on the Property.

4.6 Other Important Risk Factors

InnovExplo is not aware of any other significant factors or risks that could affect access, title, or the right or ability to estimate the mineral resources on the Property.

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

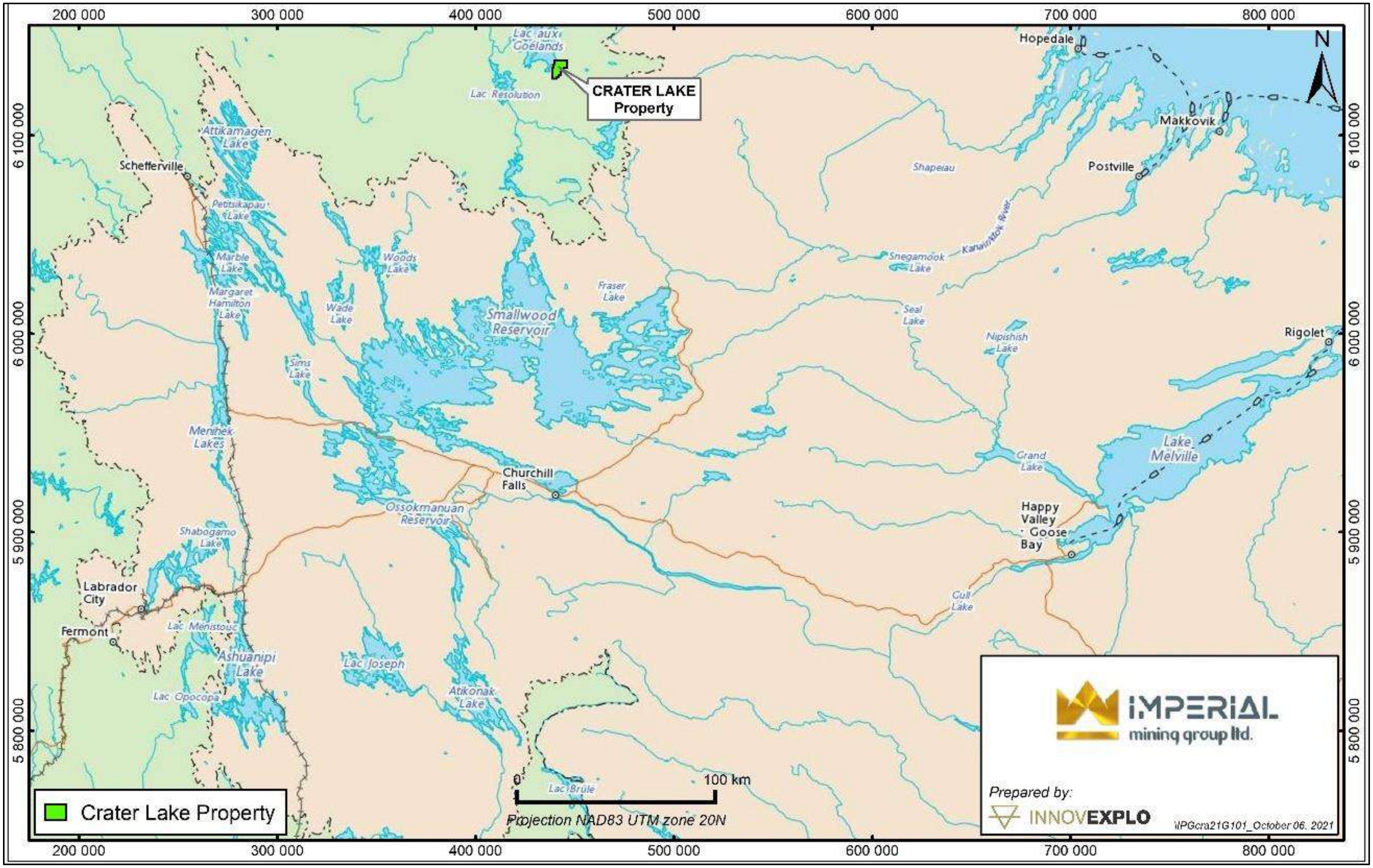
5.1 Accessibility

Access to the project area (Figure 5-1) is restricted to fixed-wing aircraft or helicopters. Due to the lack of an airstrip at the camp, fixed-wing aircraft are equipped with floats or skis, depending on the time of year.

Aircraft are chartered from Schefferville, QC (200 km southwest), Nain, NL (190 km northeast) or Happy Valley–Goose Bay, NL (350 km southeast). There are several regularly scheduled flights to Schefferville and Goose Bay from most major cities in eastern Canada.

Fixed-wing flights from Schefferville are typically 60 minutes, and flights from Goose Bay are typically 90 minutes. Supplying for the Project is done from both Schefferville and Happy Valley–Goose Bay with support from Quest's Strange Lake Camp.

Figure 5-1: Access to the Crater Lake Property



5.2 Climate

This region of northern Quebec is characterized by a cool subarctic climatic zone (Köppen climate classification) where summers are short and cool, and winters are long and cold with heavy snowfall. Specifically, the project is located within the Kingurutik-Fraser Rivers ecoregion of the Taiga Shield ecozone (Marsgall and Schut 1999). The ground is covered in snow for six to eight months of the year.

The closest historical weather data is taken from the Border A station from 1965 to 1979 (1965 to 1990 for annual rainfall; website: www.worldclimate.com), as displayed in Table 5-1.

Table 5-1: Climatic Data for the Project Area

Weather Type	Borden A
Minimum mean annual temperature (°C)	-10.4
Maximum mean annual temperature (°C)	-1.0
Average minimum January temperature (°C)	-27.3
Average maximum January temperature (°C)	-17.4
Average minimum July temperature (°C)	5.7
Average maximum July temperature (°C)	16.2
Average rainfall (mm)	666.0
Average snowfall (cm)	350.0

Exploration activities may be conducted during the summer and autumn months (June to November) and during winter to early spring (January to April).

5.3 Local Resources

There are no local resources in or around the Property. Local labour may be hired out of Schefferville, Nain or Goose Bay; however, most skilled and professional labour must be sourced elsewhere.

The nearest mine is Vale's nickel-copper-cobalt mine at Voisey's Bay, roughly 155 km to the northeast, on the coast of Labrador.

5.4 Infrastructure

There is no developed infrastructure in or around the Property. The nearest development infrastructure is in the town of Schefferville and Nain. Nain is a coastal community that also serves as the local supply and service centre for Voisey's Bay mine. Nain has no road access, but it is serviced by regular, year-round flights from Happy Valley–Goose Bay and by coastal freighters during the summer months. Schefferville and neighbouring communities of Matimekush (pop. 850) and Naskapi (pop. 900) act as local service and supply centers for several iron mines and hydro-electric dams in the area. They are serviced year-round by passenger and freight train service and have regularly scheduled flights to Quebec City and Sept-Îles , QC, and Wabush, NL.

The nearest seaport is in Nain, 200 km east of the Property, and the nearest railhead is in Schefferville, 200 km southwest of the Property, with access to the seaport of Sept-Îles on the Bay of St. Lawrence.

There is no source of electricity on or near the Property. Power must be generated on-site. The nearest sources of electricity are in Schefferville, supplied by the hydro-electric generating stations of Menehek (200 km southwest) and Churchill Falls (210 km south).

Water sources are abundant on and adjacent to the Property.

5.5 Physiography

The Property is situated to the west of a major watershed that runs along the border between Quebec and Labrador. The terrain is glacially scoured with moderate rolling hills and lakes and elevation ranging from 450 to 700 m above sea level. Larger hills are present in the northwest part of the property.

Eskers and boulder fields are common throughout the Property. The exposure and lack of vegetation (short growing season) promote stunted and thinly spaced vegetation often confined to sheltered valleys and enclaves. The vegetation on the Property consists mainly of tamarack trees, shrubs, and caribou moss.

Lakes, rivers, or bogs cover approximately 30% of the Property.

6 HISTORY

The following is a summary of previously completed work in the Project area. This summary is taken from Daigle (2017).

Prior to 1979, there were no known exploration activities on the Property.

Details of the historical work on the Property are presented below and summarized in Table 6-1.

Table 6-1: Historical ownership and work on the Crater Lake Property

Year	Organization	Contractor	Work	Results
1979	Geological Survey of Canada	-	Airborne gamma-ray spectrometry	Geophysical Series Map 36313G
1980		-	Lac Chapiteau and Lac Ramusio map sheets completed at 1:50,000 scale	Geophysical Series Map 6204G
1996	Major General Resources Ltd. and Donner Resources Ltd	-	Surface geological and geochemical programs on the Lac Chapiteau Property	Limited potential to host base metal mineralization
2007	Freewest Resources Canada Inc.	-	Field exploration (6 samples collected)	No reports available
2008	Freewest Resources Canada Inc. (Freewest) and Quest	-	Freewest's uranium property assets, including the current Property, were transferred to Quest Uranium Corporation (Quest Uranium)	In April 2010, Quest Uranium changed its name to Quest Rare Minerals Inc.
2009	Geological Survey of Canada	-	Open File including ten maps at 1:250,000 scale covering portions of western Labrador, north of Churchill Reservoir, and adjoining parts of Quebec	GSC Open File 6532 jointly released by the GSC, Geol Survey of Newfoundland, and the Direction Générale de Géologie du Québec
	Newfoundland and Labrador, Department of Natural Resources	-	Re-analysis of historical lake-sediment and lake-water geochemistry surveys (1978 to 2005) for additional elements and released in a new Open File	Open File LAB/1465
	Quest Rare Minerals Inc. (Quest)	-	Prospecting and sampling	"Discovery Outcrop": grab sample with 0.10% Sc, 0.29% Nb, 0.31% TREO
		MPX Geophysics Ltd. (MPX)	Helicopter-borne high-resolution magnetic and radiometric survey	
2010		Applied Petrographic Services Inc.	Petrographic study of 14 thin sections from samples collected in 2009	Description and observations in an internal report
		Vista Geoscience Ltd.	Glacial till survey (1,222 samples)	REE anomalies over the margins and down-ice of the circular magnetic anomalies. Most of the anomalies reflect short down-ice transport distances with till deposition at topographic barriers
		PGW Consulting Geophysicists	Models from airborne data as the starting point for modelling 4 standalone lines of ground	

Year	Organization	Contractor	Work	Results
			magnetics	
		-	Drilling program (8 DDHs): 1,170.15 m drilled and 663 samples	ML10002: 0.0284%Sc over 6.50 m and 0.0506 %Sc over 18.95 m
2010-2012	Quebec MERN, McGill University and Quest	-	Joint project to complete a Master's thesis to characterize the syenite intrusion and associated REE mineralization at Crater Lake. Thesis submitted in October 2012.	The thesis (Petrella, 2012) concluded that the Crater Lake syenite intrudes the Mistastin Batholith and consists primarily of coarse-grained syenite and lesser mafic syenite; the centre of the circular intrusion consists of medium-grained syenite with lesser mafic syenite. REE mineralization includes allanite and gittinsite.
2011	Quest	Exploration Sans Frontière	Surface exploration program (prospecting, mapping and sampling). 101 stations and 199 collected samples.	Of the 199-surface samples, 40 returned values greater than 0.50% TREO
		-	Drilling program: 6 DDHs (1,894 m and 1,171 samples)	ML11009: 0.252% TREO over 344.58m (entire hole) and several thin high grade intervals in ML11010
2012		Exploration Sans Frontière	Surface exploration (prospecting, mapping, geochemical till survey). Additional mapping and channel sampling in selected areas. 261 stations, 231 grab samples, and 80 samples from 11 channels.	Till sampling survey highlighted property-scale anomalies over the margins and down-ice of the circular magnetic anomalies. 14 channel samples returned values of > 0.5% TREO, and 13 surface samples returned values of> 0.5% TREO
		-	Drilling program: 11 DDHs (2,498 m and 1,395 samples)	No significant results
		Abitibi Geophysics	Ground magnetics survey over two grids on the property to further investigate airborne geophysical anomalies	Several dyke-like structures and two NE-SW-trending magnetic highs were identified in the northeastern part of the property
2013			A broader ground magnetic survey to cover the entire circular geophysical anomaly.	The ground magnetic data correlate very well with the less detailed airborne magnetic data
2014		-	Drilling program: 7 DDHs (1,446 and 879 samples).	ML14026:0.0262% Sc and 1.176 TREO + Y% over 167.83 m and 0.0351% Sc and 1.7206 TREO + Y% over 27.63 m ML14028: 0.0235% Sc and 1.08 TREO + Y% over 199.69 m and 0.0280% Sc and 1.4065 TREO + Y% over 77.92 m
2015-2017	Peak Mining	-	Peak Mining did not conduct any exploration work or drilling on the Property	

6.1 1979-1980: Geological Survey of Canada

In 1979, an airborne gamma-ray spectrometry survey was run in the Mistastin Lake area, including the Property area (Geophysical Series Map 36313G).

In 1980, the Lac Chapiteau and Lac Ramusio map sheets were covered as part of an airborne magnetic survey at 1:50,000 scale, including the project area (Geophysical Series Map 6204G).

6.2 1996: Major General Resources Ltd. and Donner Resources Ltd

A reconnaissance geology and geochemistry program was carried out on the Lac Chapiteau property to evaluate the area for potential Voisey's Bay-style Ni-Cu-Co mineralization. The result of this program identified the area as having limited potential to host base metal mineralization (Wares and Leriche, 1996).

6.3 2007-2009: Freewest Resources Canada Inc. and Quest Rare Minerals Inc.

In 2007, as part of a regional evaluation program, Freewest Resources Canada Inc. ("Freewest") collected six (6) samples in the area of what is now the Property. There are no reports available on this program.

In January 2008, Quest Uranium Corporation ("Quest Uranium") was formed. Part of Freewest's uranium property assets, including the Property, were transferred to this company. In April 2010, Quest Uranium changed its name to Quest Rare Minerals Inc.

6.4 2009-2012: Federal and Provincial Government Work and McGill University

6.4.1 2009: Geological Survey of Canada

In 2009, the area was covered as part of a joint Open File release by the GSC, the Geological Survey of Newfoundland and Labrador, and the Direction Générale de Géologie du Québec. This release compiles 10 maps covering a portion of western Labrador, north of the Churchill Reservoir, and adjoining parts of Quebec. Results are available as 1:250 000 scale full-coloured maps in pdf format. Eight (8) of these are radiometric maps, the result of the new 2009 airborne survey (Open File 6532).

6.4.2 2009: Newfoundland and Labrador, Department of Natural Resources

In 2009, the Geological Survey of Newfoundland and Labrador released lake-sediment and lake-water geochemical data collected from historical surveys. These surveys were conducted in Labrador by the Geological Survey of Newfoundland and Labrador from 1978 to 2005. Most of the data had been released previously in various Open File reports. However, as new analytical methods became available, some samples were re-analyzed for additional elements. Some of these data had not been released previously (Open File LAB/1465).

6.4.3 2009-2012: MERN and McGill University

As part of a joint project between Quest, McGill University and MERN, a Master's thesis was undertaken to characterize the syenite intrusion and associated rare earth element ("REE") mineralization at Crater Lake. The thesis was submitted in October 2012.

This work concluded that the Crater Lake syenite (under the name of Misery Lake in the thesis) intrudes the Mistastin Batholith and consists primarily of coarse-grained syenite and lesser mafic syenite; the center of the circular intrusion consists of medium-grained syenite with lesser mafic syenite. REE mineralization includes allanite and gittinsite (Petrella, 2012).

6.5 2009-2014: Exploration and Drilling Activities (Quest)

6.5.1 2009 Geophysics

Quest retained MPX Geophysics Ltd. (MPX), to conduct a helicopter-borne high resolution magnetic and radiometric survey. The survey area was flown at a nominal mean terrain clearance of 70 m. The survey block was flown along north-south (0°Az) flight lines separated by 400 m line spacings, and east-west (90°Az) tie lines at a line separation of 400 m (MPX, 2009).

6.5.2 2010 Petrography, Geochemistry, and Geophysics

Petrography

Quest contracted Applied Petrographic Services Inc. to complete a petrographic study on 14 thin sections taken from samples collected in 2009. Descriptions and observations were provided in an internal report.

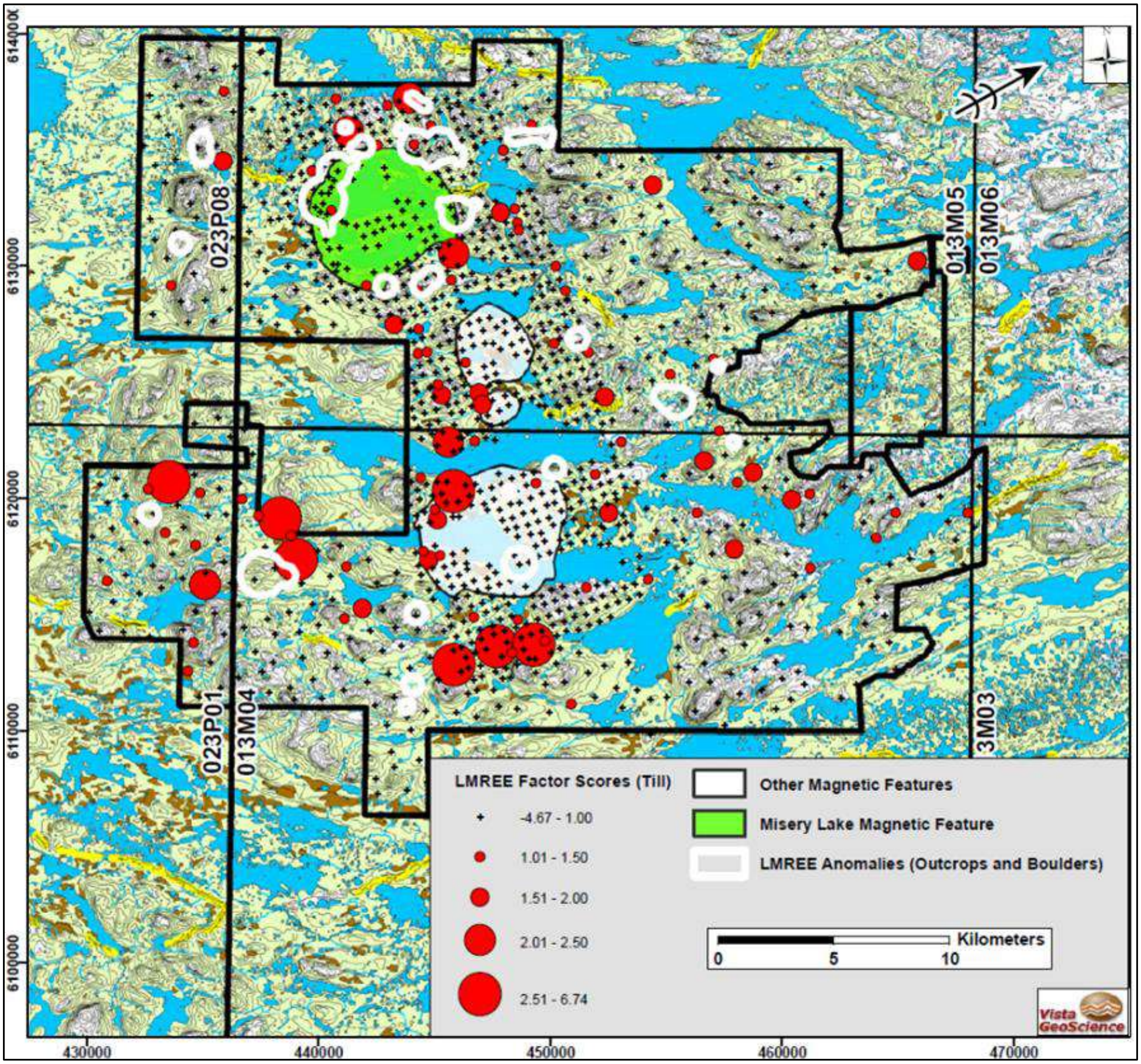
Till survey

Between July and August 2010, a till survey was carried out by Vista Geoscience (“Vista”) on behalf of Quest. A total of 1,222 samples of sandy till were collected, each 25-50 cm deep (Seneshen, 2011).

The survey revealed REE anomalies over the margins of and down-ice from the circular magnetic anomalies. Previous exploration by Quest and its contractors showed glacial transport distances of at least 7 km at Crater Lake. Most of the anomalies reflect short down-ice transport distances with till deposition at topographic barriers.

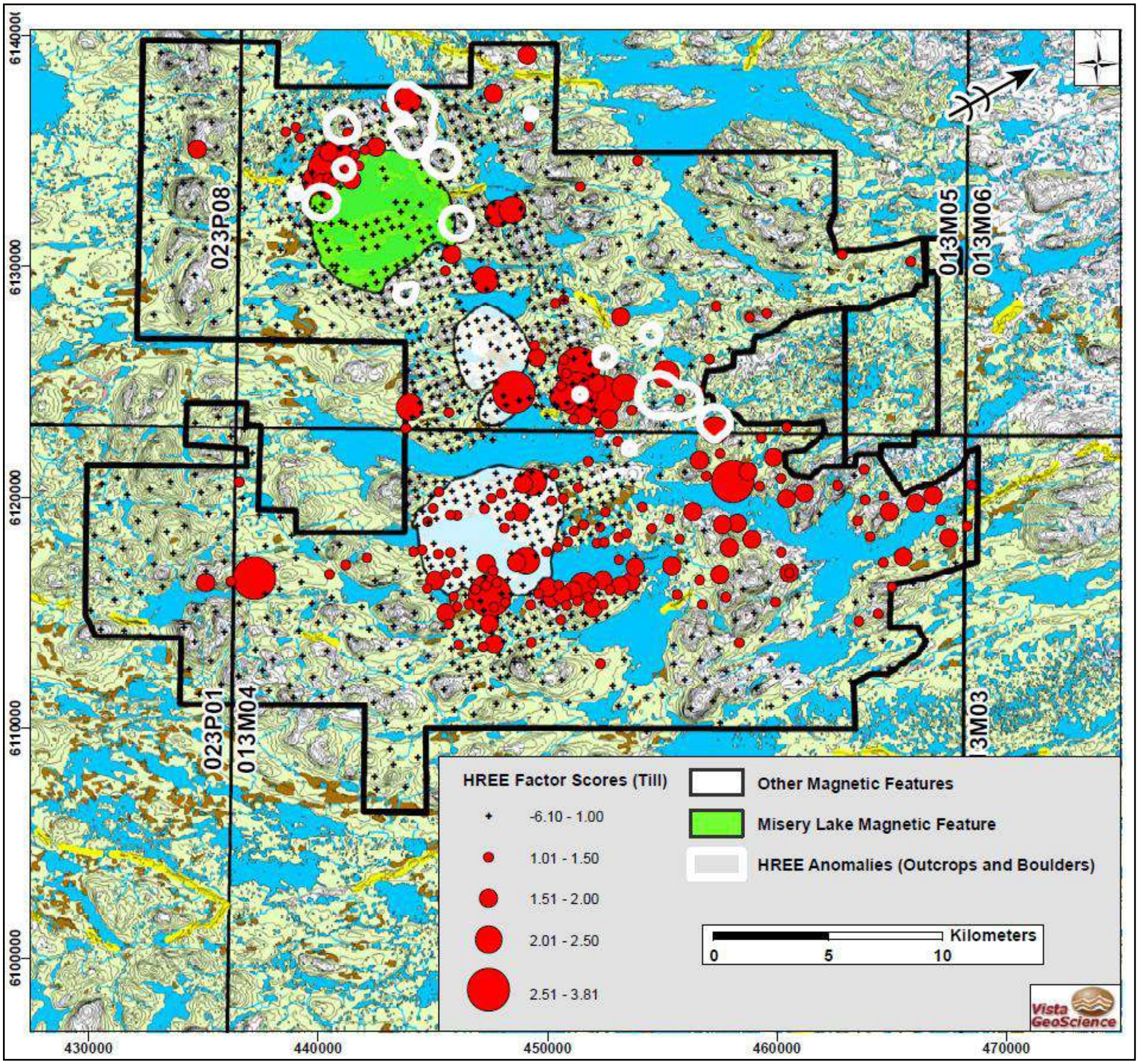
Figure 6-1 and Figure 6-2 display the light-medium REE (“LMREE”) and heavy REE (“HREE”) results and anomalies. Note that the LMREE results include europium and gadolinium.

Figure 6-1: LMREE Results for Till Samples from 2010



Source: Vista (2011)

Figure 6-2: HREE Results for Till Samples from 2010



Source: Vista (2011)

Geophysics

PGW Consulting Geophysicists ("PGW") was retained by Quest to interpret the airborne geophysical data from four (4) standalone lines of ground magnetic data. The lines were completed independently of each other over the outer response of the Crater Lake magnetic ring.

6.5.3 2010-2012 Drilling Programs

In September 2010, an eight (8)-hole drilling program tested magnetic anomalies from the 2009 airborne magnetic survey. A total of 1,170 m was drilled, and 663 samples were collected. The main unit encountered was syenite. No significant assay results were obtained.

In September and November 2011, a six (6)-hole drilling program continued testing the strong magnetic responses seen in the previous airborne geophysical surveys. A total of 1,894 m was drilled, and 1,171 samples were collected (Quest, 2013).

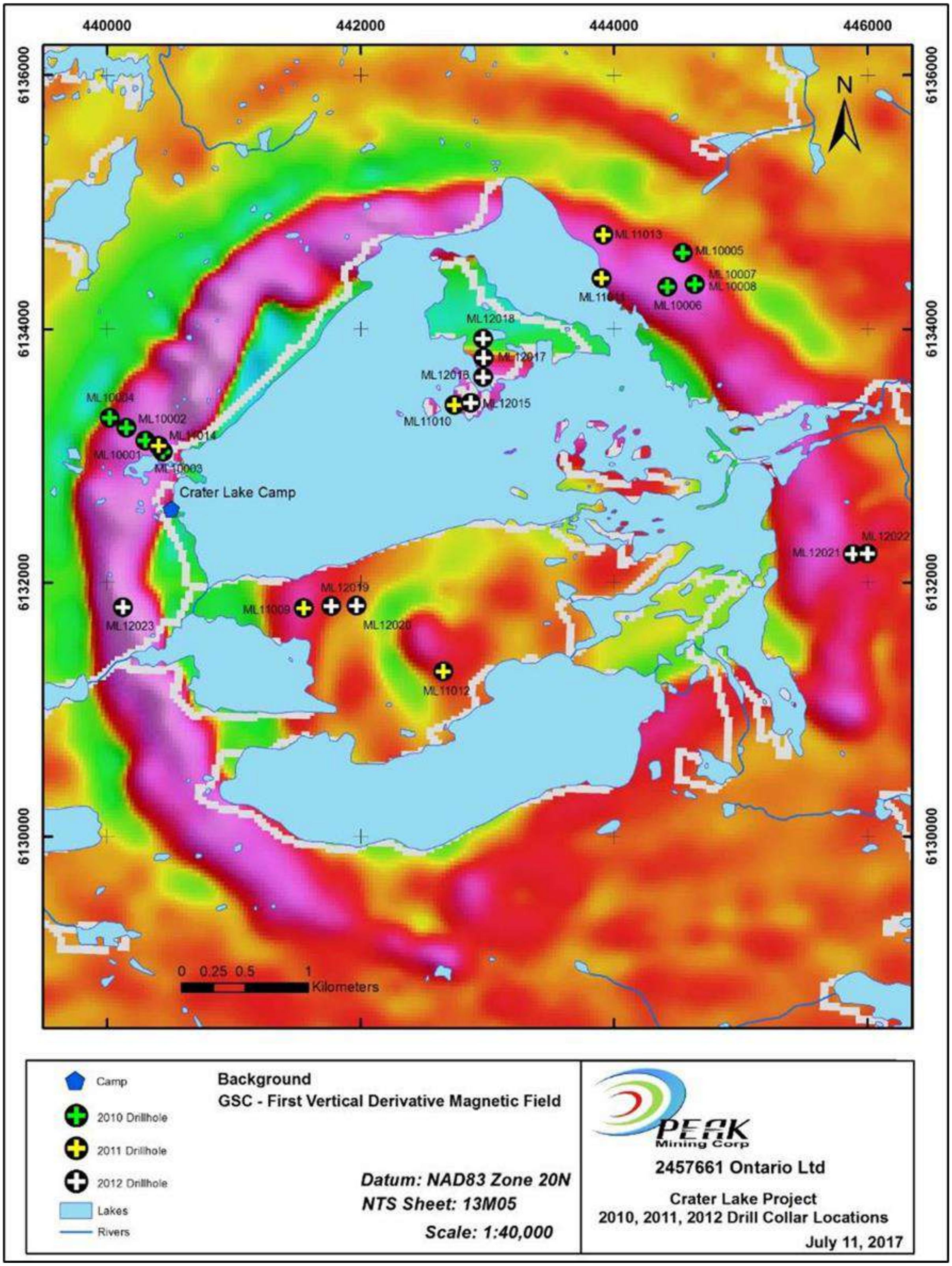
In September and October 2012, 2,498 m were drilled in 11 holes. All holes in the Crater Lake Intrusion intersected variably textured, medium-grained syenite. Two holes were drilled outside the Crater Lake Intrusion, testing weak circular magnetic features south of the Crater Lake magnetic ring feature (Quest, 2012).

Table 6-2 summarizes the 2010 to 2012 drilling programs. Figure 6-3 shows the collar position relative to the modelled ground magnetics data.

Table 6-2: Summary of the Crater Lake 2010-2012 Drilling Programs

Year	No. of Drill Holes	No. of Metres (m)	No. of Samples
2010	8	1,170	663
2011	6	1,894	1,171
2012	11	2,498	1,395
TOTAL	**25**	**5,532**	**3,229**

Figure 6-3: Collar Locations on a Background of Modelled Ground Magnetics Data



Source: Peak Mining (2017)

6.5.4 2011-2012 Surface Exploration

From July to August 2011, Quest conducted a surface exploration program to follow up the results from the 2010 geochemical till survey conducted on the Property. A limited mapping and prospecting program was completed with a total of 101 stations; 199 samples were collected and submitted for assay, of which 40 returned grades greater than 0.50% TREO.

Between August and October 2012, geologists from Quest and prospectors from Exploration Sans Frontière conducted a surface exploration program. The work focused on areas of historical work that included prospecting, mapping and a geochemical till survey. The till survey highlighted property-scale anomalies over the margins of and down-ice from the circular magnetic anomalies.

Selected areas were chosen for more detailed work that included outcrop stripping and channel sampling. The 2011 program highlighted individual samples that returned elevated REE values, and these were followed up. A total of 261 geological stations were sampled, yielding 231 samples, 80 of which were cut from 11 different channel locations.

Fourteen (14) of the channel samples returned values greater than 0.5% TREO, and 13 surface samples returned values greater than 0.5% TREO.

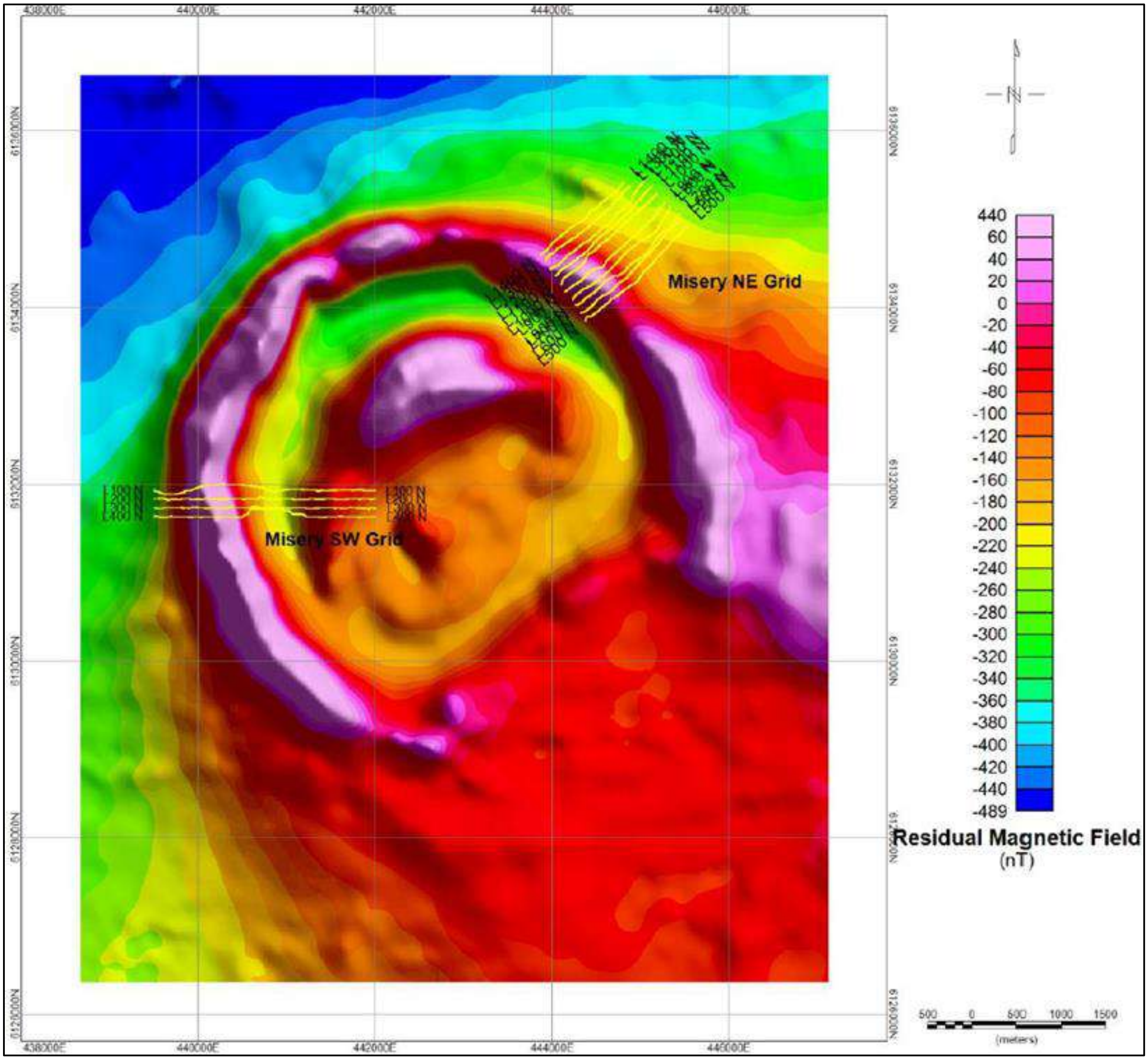
6.5.5 2012-2013 Geophysics

In October 2012, Abitibi Geophysics was contracted by Quest to conduct a small ground magnetics survey to characterize the large circular airborne magnetic feature. The aim was to identify any internal differentiation and to delineate potential domains of REE mineralization related to the intrusion.

Two grids were laid out on the northeast and southwest sides of the magnetic anomaly. A total of 24.75 line-km was surveyed at a station separation of 25 m. The locations of the two grids are shown in Figure 6-4.

The survey identified several dyke-like structures and two NE-SW trending magnetic highs in the northeastern part of the property. It was found that the two grids correlate well with the previous airborne magnetics survey (Abitibi Geophysics, 2012).

Figure 6-4: Residual Magnetic Field, 2012 Geophysical Survey, Showing Northeast and Southwest Grids

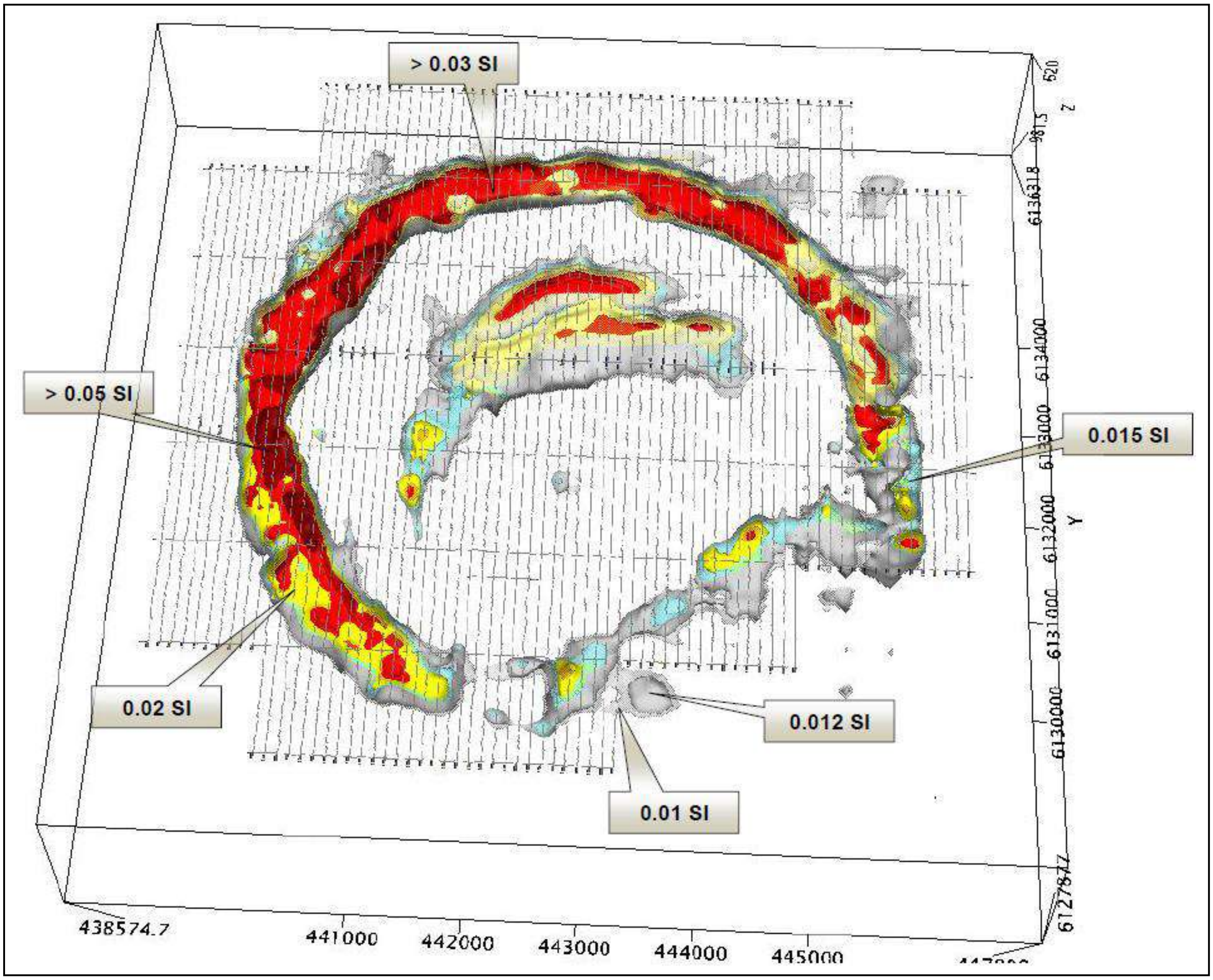


Source: Abitibi Geophysics (2012)

During the winter of 2013, Quest retained Abitibi Geophysics to conduct a property-wide ground magnetics survey (Abitibi Geophysics, 2013). The data from 470.5 line-km were used to build an unconstrained 3D subsurface magnetic susceptibility model of the Property and several 2D models. The resulting 3D models and maps were used to plan the 2014 exploration and drilling program. The results of this interpretation are shown in Figure 6-5.

Overall, the ground magnetic geophysical survey correlates very well with the less detailed airborne magnetic survey. Several previously unidentified anomalies were discovered as a result of the survey.

Figure 6-5: 3D Rendering of the Unconstrained Magnetic Anomaly, Perspective view Looking North



Note: North-South lines are at 100-m separation
Source: Abitibi Geophysics (2013)

6.5.6 2014 Drilling Program

During the winter of 2014, a total of 1,446 m was drilled in 7 holes. Several previously untested exploration targets were chosen based on the 2013 geophysical survey and previous surface geochemistry data. The holes were sampled along their lengths for a total of 879 collected samples. Downhole magnetic susceptibility data were collected upon the completion of each drill hole (Quest, 2014).

Table 6-3 summarizes the best composite drill intersections. Figure 6-6 presents the collar locations.

Table 6-3: Composited 2014 Drilling Results

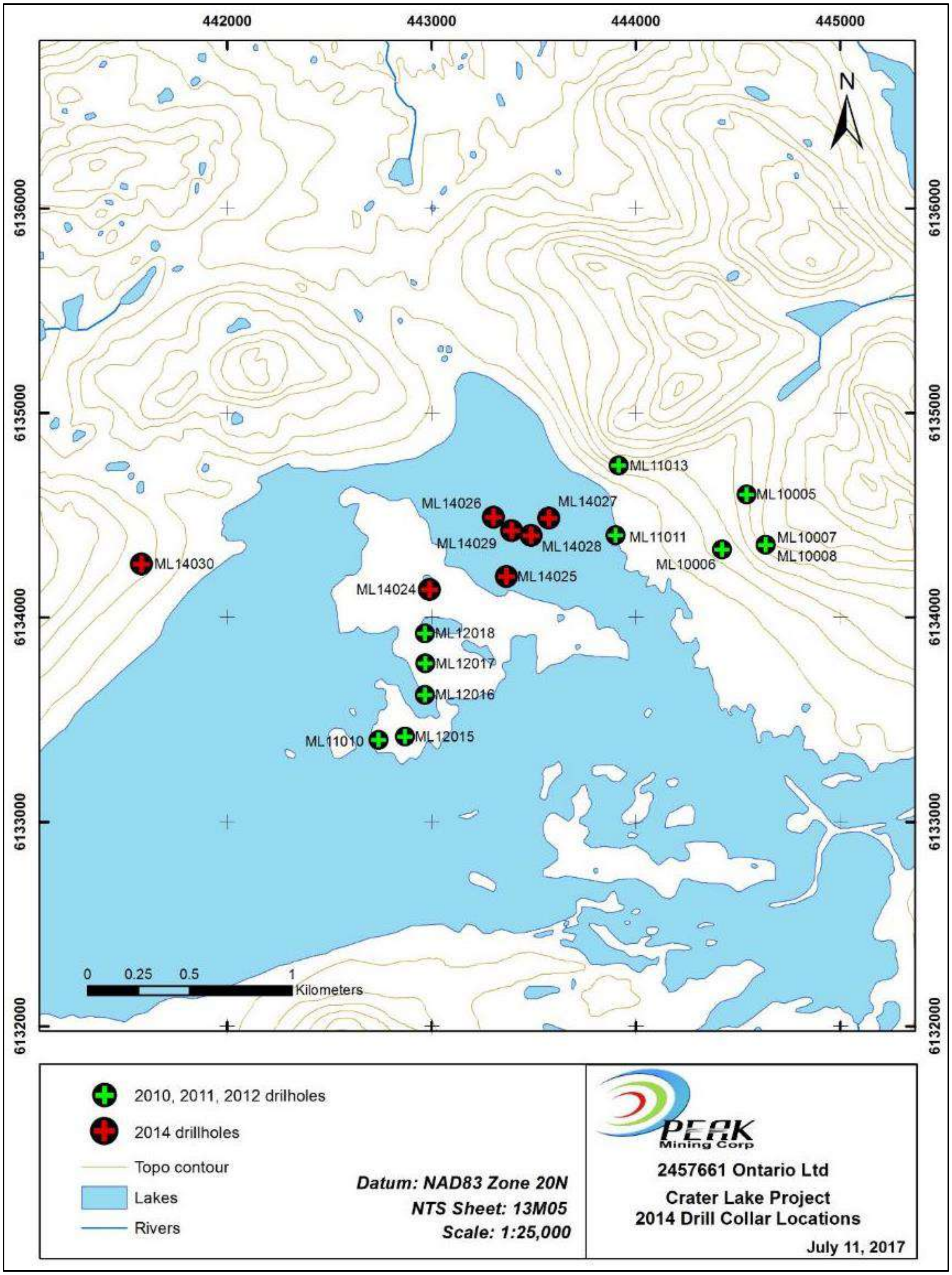
Hole ID	From (m)	To (m)	Thickness (m)	TREO+Y[1] (wt.%)	LREO[2] (wt.%)	HREO+Y[3] (wt.%)	HREO+Y/ TREO+Y	Sc_2O_3 %
ML14026	14.77	182.60	167.83	1.1760	1.0013	0.1747	14.86	0.0262
including	14.77	42.40	27.63	1.7206	1.4686	0.2521	14.65	0.0351
including	14.77	77.55	62.78	1.4779	1.2607	0.2172	14.70	0.0304
ML14028	13.22	212.91	199.69	1.0800	0.9178	0.1621	15.01	0.0235
including	13.22	91.14	77.92	1.4065	1.1977	0.2088	14.85	0.0280
ML14029	13.35	93.40	80.05	1.3353	1.1362	0.1991	14.91	0.0286
ML14030	177.00	183.04	6.04	1.1442	0.9632	0.1810	15.82	0.0319

Notes :

1. Total Rare Earth Oxides (TREO+Y) include: La_2O_3, CeO_2, Pr_6O_{11}, Nd_2O_3, Sm_2O_3, Eu_2O_3, Gd_2O_3, Tb_4O_7, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3 and Y_2O_3
2. Heavy Rare Earth Oxides (HREO+Y) include: Eu2O3, Gd_2O_3, Tb_4O_7, Dy_2O_3, Ho_2O_3, Er_2O_3, Tm_2O_3, Yb_2O_3, Lu_2O_3, Y_2O_3
3. Light Rare Earth Oxides (LREO) include: La_2O_3, CeO_2, Pr_6O_{11}, Nd_2O_3, Sm_2O_3

Source: Peak Mining (2017)

Figure 6-6: Location of the 2014 Drill Hole Collars



7 GEOLOGICAL SETTING AND MINERALIZATION

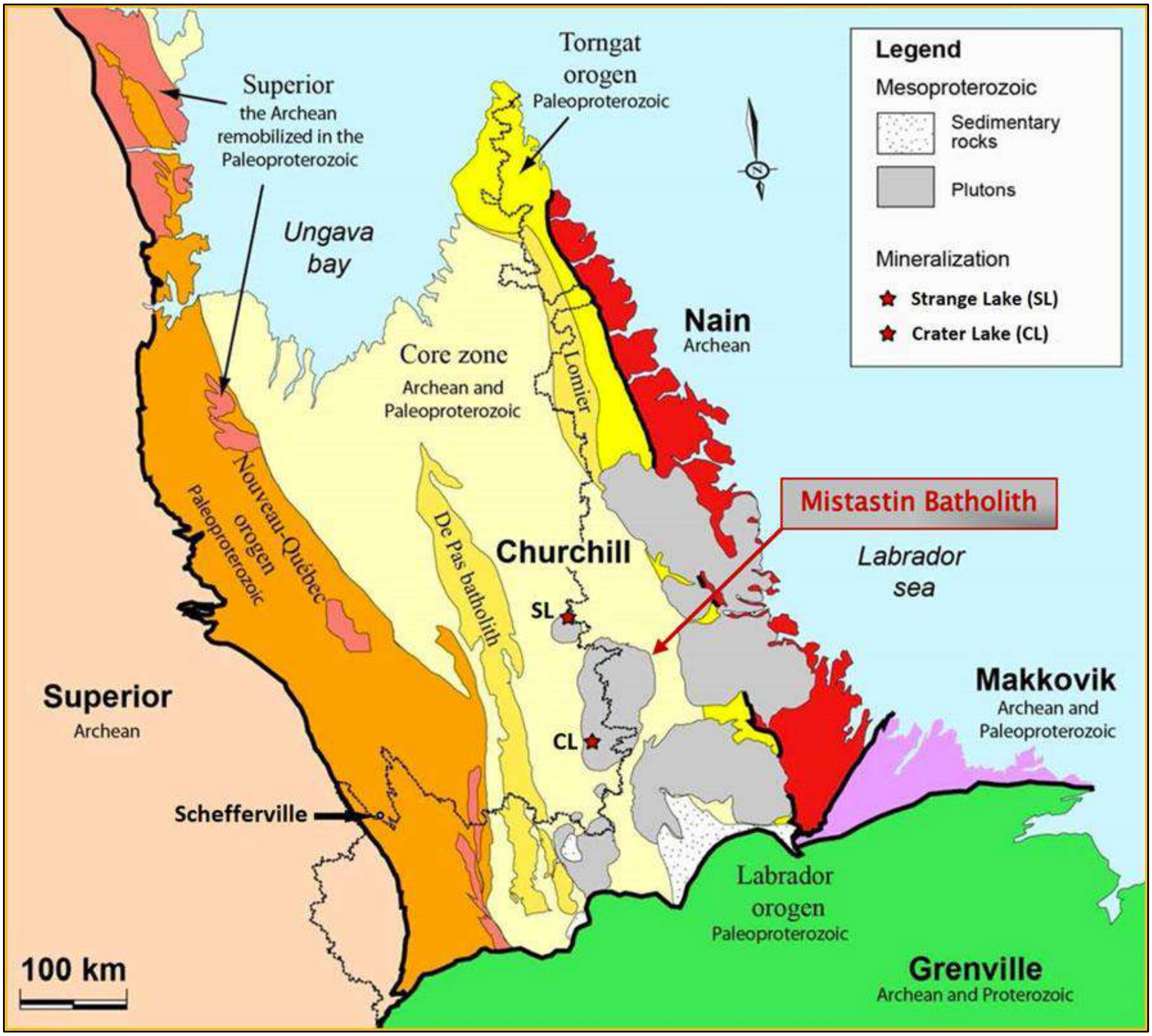
The following geological summary is taken from Daigle (2017).

7.1 Regional Geology

The region is underlain by five structural provinces: Nain, Superior, Churchill, Makkovik, and Grenville. Together, they record a crustal history ranging from about 3.8 to 0.6 Ga. The Nain and Superior Archean provinces are bounded by the younger Archean and Paleoproterozoic Churchill and Makkovik provinces, which in turn are truncated by the early Proterozoic Grenville Province (Figure 7-1).

The Churchill Province is subdivided into three parts. The western part consists of low-grade sedimentary and volcanic rocks in a west-verging fold and thrust belt (the Labrador Trough). The central part comprises predominantly reworked Archean rocks, which are juxtaposed against the Labrador Trough along mylonitic shear zones. The eastern part consists mainly of anorthosite and gabbro of the Rae Province (Swinden et al. 1991).

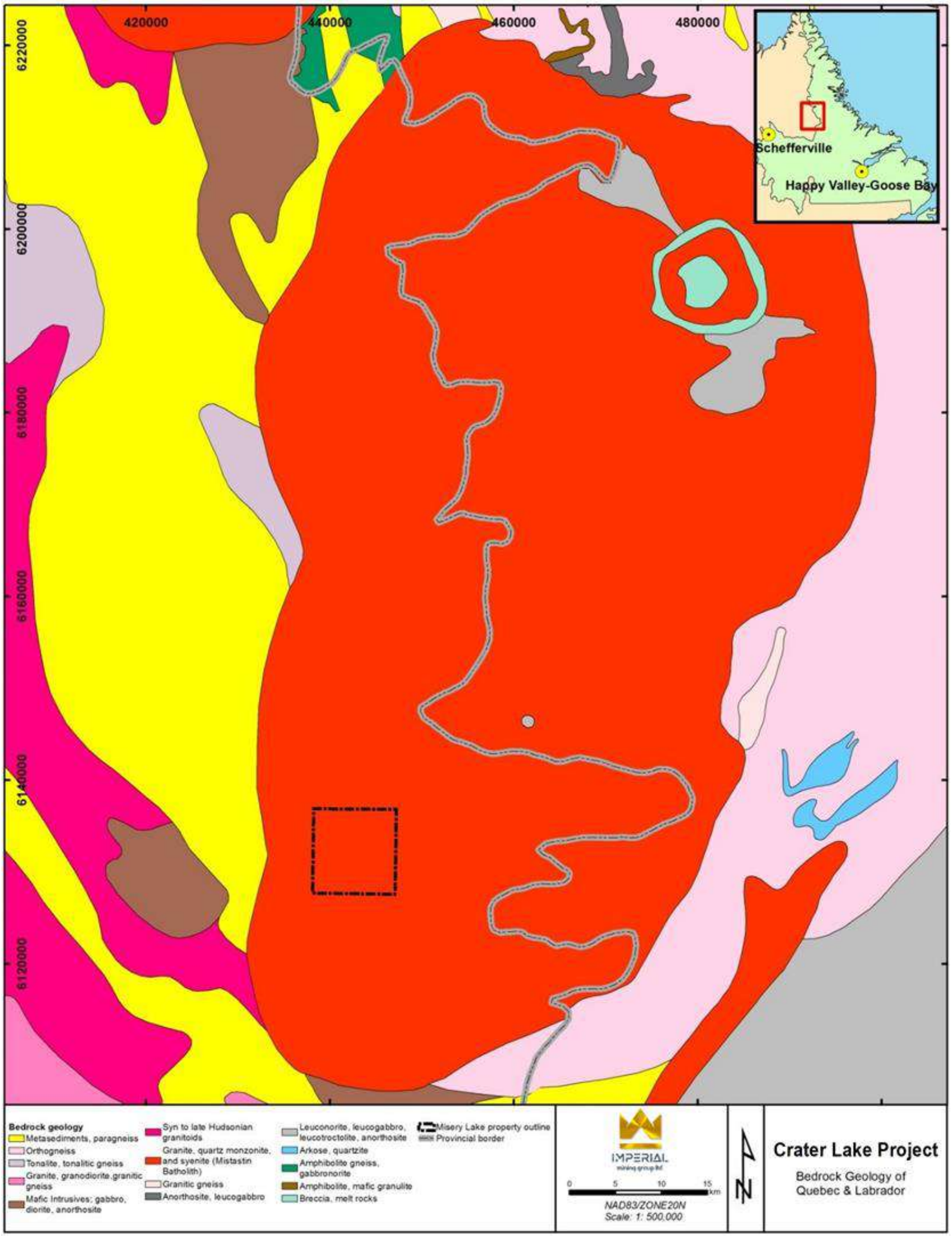
Figure 7-1: Geological Map of the Churchill Province Showing the Location of the Crater Lake and Strange Lake Deposits



Source: Hammouche et. al. (2012)

The syenite intrusion of Crater Lake is located in the Churchill Province. It intrudes or is coeval with the southeast end of the Mistastin Batholith (Figure 7-2), which covers an area of approximately 5,000 km^2. The dominant lithologies of the batholith are granite and quartz monzonite with pyroxene, which are cut by younger biotite-hornblende granite, which is in turn intruded by a smaller olivine syenite, the Crater Lake syenite. Uranium-lead dating of three zircons places the age of the batholith at approximately 1.4 Ga (Petrella 2011).

Figure 7-2: Regional Geology of the Crater Lake Property Area



Source: Quest (2014)

7.2 Property Geology

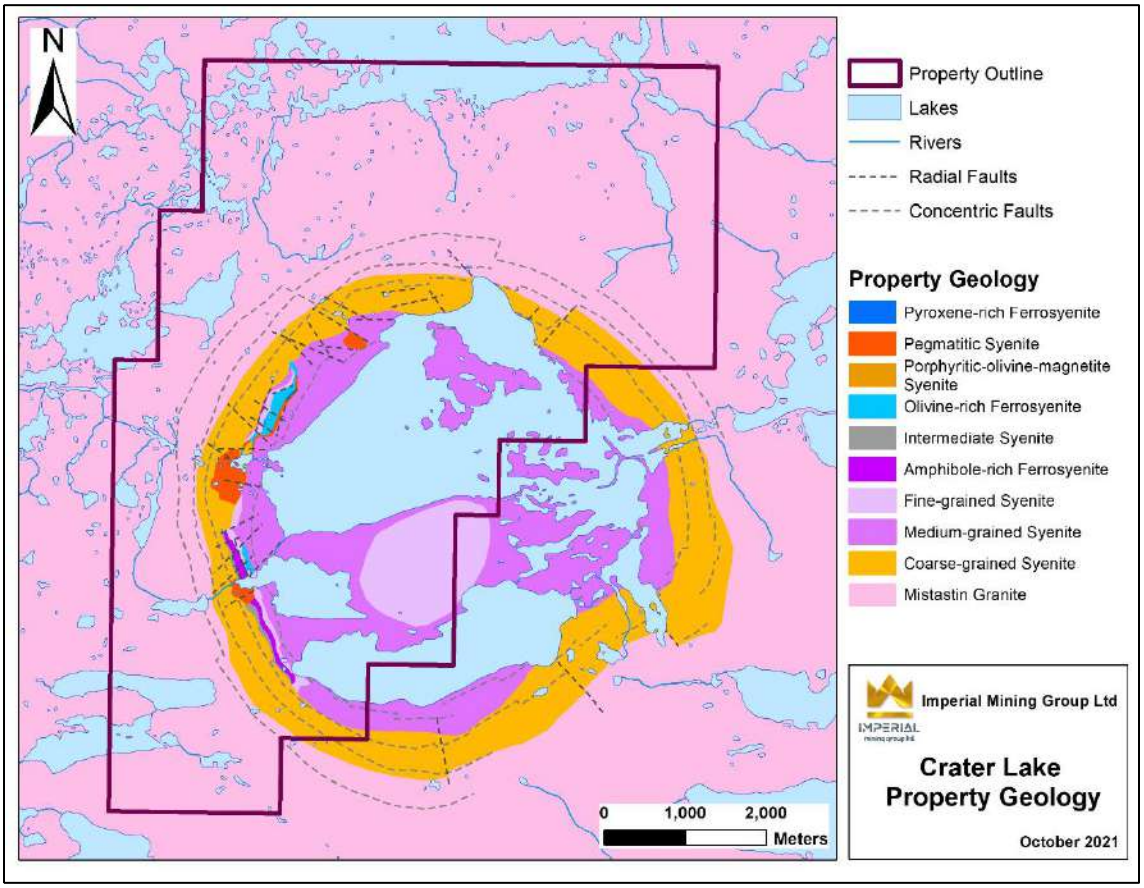
The Crater Lake intrusion displays a gradational contact with its host, the Mistastin rapakivi granite (Figure 7-3). Both have an A-type affinity and similar trace element composition. The Crater Lake syenites are therefore interpreted to be a late differentiate product of the Mistastin Batholith. The dominant exposed lithology (much of the intrusion is covered by a lake) is coarse- to medium-grained, massive syenite, which is mainly composed of perthitic K-feldspar and 1 to 10% by volume of interstitial ferromagnesian minerals, namely fayalite (iron chrysolite, Fe2SiO4), hedenbergite, ferro-pargasite and annite (iron-rich biotite), accompanied by accessory quartz, iron oxides (magnetite, titanium-rich magnetite, and ilmenite), zircon, fluorite, apatite and britholite (Petrella 2012). A magnetic and melanocratic unit, ferro-syenite, which commonly contains greater than 50% by volume of ferromagnesian minerals, including cumulate fayalite, hedenbergite and ferro-pargasite, occurs as large continuous to discontinuous subvertical and conical bodies, sills, narrow dikes and inclusions in the felsic syenites. The large ferro-syenite bodies are elongated and concordant to subconcordant to the main contact between the Crater Lake syenite and the Mistastin granite intrusions. These large bodies can reach up to 700 m long, up to 120 m wide, and are open at depth. Three large ferro-syenite bodies have been found on the property: TGZ, Boulder Lake and STG. Petrella (2012) interpreted the narrow ferro-syenite dikes as having formed by fractional crystallization of ferromagnesian minerals, leaving behind a residual magma that produced the felsic syenites. With continued fractional crystallization, the felsic syenites became more enriched in alkali and silica, and only became saturated with ferromagnesian at a very late stage, which explains the interstitial crystallization of the latter in the perthite-dominated syenite.

Several major radial and concentric faults are observed in the field and drill core and have also been interpreted from magnetic data and satellite images. Most of these subvertical and (less commonly) subhorizontal structures are concentrated inwards from the contact with the Mistastin granite to within the first 800 m of the Crater Lake intrusion. Major faults are characterized by a very intense potassic alteration with local concentrations of biotite, chlorite, epidote, and magnetite. Imperial’s geologists do not yet know if these faults played a role in the ferro-syenite emplacement.

The Crater Lake intrusion was interpreted by Petrella et al. (2014) to be a ring dyke complex due to the concentric lithological zonation of quartz monzonite and felsic syenite, the steep dip of the bodies toward the center of the intrusion, the presence of numerous intrusion-scale discontinuous concentric faults (interpreted from the magnetic data), and the occurrence of several late radial faults (occupied by pegmatites), all of which are characteristic features of ring complexes (e.g. Woolley, 2001; Coumans and Stix, 2016). Consistent with this interpretation, some of the Crater Lake felsic syenites feature a trachytic texture developed through the alignment of feldspar laths, indicative of flow before cooling.

There is a strong correlation between the location of known ferro-syenites and strong magnetic susceptibility. Indeed, a 3D magnetic susceptibility model (commissioned by Imperial) of the intrusion, from an iteratively reweighted inversion of data from a recent GPS-integrated ground magnetic survey, suggests that the ferro-syenite is a subvertical ring dyke with some local sill-like lateral extensions. So far, this model is supported by very thick intersections of ferro-syenite in several drill holes, and the steeply dipping layering in this unit (Beland, 2021).

Figure 7-3: Crater Lake Intrusion Geology



7.3 Geological Units: Intersected from 2014 to 2021

Prior to the start of the 2021 drilling campaign, Imperial geologists reinterpreted previous drilling results and reclassified some previously intersected units to better reflect the mineralogy of the Crater Lake lithologies. The following summarizes the units that were intersected during the drilling programs between winter 2014 through winter 2021.

7.3.1 Medium-grained Syenite

Medium-grained syenite is the main unit throughout the central part of the Crater Lake intrusion. Predominately grey to pale pink-orange in colour, the mineralogy consists of approximately 70 to 90% perthitic K-feldspars, with the remainder of the unit comprising ferromagnesian minerals, including iron-amphibole and minor fayalite and titanium-rich magnetite (Petrella, 2011). Trace interstitial quartz is rare. Zircon, fluorite, carbonate, and pyrite can also occur at trace levels. Feldspar grains are mostly subhedral laths 1 to 1.5 cm long, but can reach up to 2.5 cm. The mafic minerals are interstitial and are 5 mm in size. The unit is typically massive.

Relatively narrow (2 to 25 cm wide) mafic-rich sections occur throughout this unit. These bands or cumulates are 5-mm to 15-mm subhedral amphibole grains with interstitial magnetite and olivine. Minor REE mineralization can occur in these accumulations as interstitial cerium-britholite (Petrella, 2011). These mafic bands/cumulates often have sharp contacts and low core angles (less than 25°).

Potassic alteration is common throughout medium-grained syenite and results in a patchy appearance. Feldspar grains often exhibit pink colour. Amphibole commonly displays partial replacement by aegirine.

7.3.2 Fine-grained Syenite

The fine-grained syenite is composed of K-feldspar and amphibole crystals <1 to 4 mm in size. Feldspars are subhedral and make up approximately 90% of the unit. The remainder is interstitial amphibole, magnetite, and olivine. Mafic minerals are concentrated near the upper and lower contacts of this unit in some drill holes. This unit often displays a weak preferred orientation defined by K-feldspar laths and can range from approximately 10 to 35°. Alteration can occur as pink potassic overprinting. The alteration can occur parallel to the fabric of the unit.

7.3.3 Coarse-grained Syenite

The mineralogy of the coarse-grained syenite is similar to the fine-grained syenite, differing only in grain size. Coarse-grained syenite is composed of approximately 90% subhedral K-feldspar. The remainder is interstitial mafic minerals. A minor amount of disseminated pyrite occurs locally. Feldspar grains range in size from 1 to 5 cm, but in local megacrystic sections, they can exceed 10 cm. Zonation is observed in some feldspar grains, especially megacrysts. Interstitial mafic minerals are usually 1 to 5 cm in size. In megacrystic sections, subhedral amphibole, if present, can exceed 5 cm. Potassic overprinting is common, with a pink to orange colour. Amphibole is often replaced by aegirine. Complete replacement of the larger amphibole grains by epidote has been observed in several drill holes. The unit is weakly magnetic in areas with interstitial mafic minerals.

7.3.4 Variably Textured Syenite

As the name suggests, the variably textured syenite exhibits textural similarities with several other units. These textures, including medium-grained syenite, fine- and coarse-grained syenites and often pegmatite, are commonly distributed in a chaotic arrangement. The size of each section can range from several centimetres to a metre in length and appear to have no order. Contacts between each section can be sharp, irregular, or gradational.

7.3.5 POM Syenite

The POM syenite ("POMSYN") consists of a grey to light grey, medium-grained syenite with olivine and magnetite phenocrysts 5 to 10 mm in size. The feldspar laths are interlocking. The rims of the magnetite and olivine phenocrysts are partially altered to biotite and pyroxene, respectively. This unit is often found as inclusions in the ferro-syenite and the coarse-grained syenite. The size of these inclusions ranges from a few centimetres to less than 2 m in size.

7.3.6 Blebby Syenite

The blebby syenite consists of a medium- to coarse-grained grey syenite with interlocking, medium-grained feldspar and interstitial pyroxene, amphibole, olivine and biotite. Pyroxene occurs as blebs, either rimmed or partially altered by amphibole. Specks of olivine are also observed.

7.3.7 Trachytic Syenite

The trachytic syenite consists of a dark grey to grey, very fine- to fine-grained, foliated (trachytic) interlocking feldspar groundmass with up to 15% fine-grained specks of anhedral olivine and pyroxene.

7.3.8 Ferro-syenite

Different types of ferro-syenite units were identified during the 2014 through 2021 drilling programs.

Olivine ferro-syenite ("OLFESYN")

The olivine-rich ferro-syenite consists of a dark green to dark grey, mafic cumulates, fine- to medium-grained olivine-rich unit with up to 40% olivine, up to 15 % pyroxene, up to 10% amphibole, and 10% magnetite. Mafic minerals form a net-like (interstitial) texture in the medium-grained (0.5-1 cm) feldspar groundmass. The latter also displays coarse-grained (1-3 cm) feldspar fragments that are locally digested. The mafic minerals are fine- to medium-grained, irregular and anhedral with the exception of black needle-like amphibole and amphibole clots. This unit is moderately to highly magnetic.

Pyroxene Ferro-syenite (“PXFESYN”)

The pyroxene-rich ferro-syenite is a dark green-grey, medium- to fine-grained unit composed of up to 40% pyroxene, up to 15% magnetite, 10% olivine and up to 10% amphibole. The mafic cumulates (pyroxene, magnetite, olivine and amphibole) form a net-like (interstitial) texture in the medium-grained (0.5-1 cm) feldspar groundmass. This unit is highly magnetic. The PXFESYN is distinctly olivine-poor compared to the OLFESYN and seems to have more of a cumulate texture composed of pyroxene and magnetite.

Amphibole Ferro-syenite (“AMPFESYN”)

The amphibole-rich ferro-syenite is a black to dark grey, coarse- to medium-grained unit composed of up to 50% amphibole, up to 15% magnetite, 5% olivine and up to 5% pyroxene. The mafic cumulates (amphibole, magnetite, olivine, and pyroxene) form a net-like (interstitial) texture in the medium-grained (0.5-1 cm) feldspar groundmass. This unit is highly magnetic. The AMPFESYN unit is olivine- and pyroxene-poor compared to the PXFESYN and seems to have more of a cumulate texture of amphibole and magnetite.

7.4 Mineralization

Assay results from surface samples and from 2014-2021 drill core indicate that the different types of ferro-syenite are the main host to the scandium and REE mineralization.

At Crater Lake, scandium was enriched in the residual liquid of the parent Mistastin granite magma following extensive fractionation of feldspar, in which scandium is incompatible. This residual liquid became the Crater Lake quartz monzonite magma, which was enriched in scandium and iron. Fluorapatite, zircon, fayalite, and the cores of zoned hedenbergite crystals saturated in this magma chamber. Ring faults developed as a result of caldera collapse, and the magma and minerals were emplaced as a slurry into these faults. The ferro-syenite formed by in situ fractionation of unzoned hedenbergite crystals, magnetite and hastingsite, and their physical segregation with the previously crystallized minerals. The extremely high FeO/FeO+MgO content of the quartz monzonite liquid resulted in high partition coefficients for scandium in the hedenbergite and hastingsite, allowing scandium to be incorporated into these minerals at exceptionally high concentrations under magmatic conditions. The physical segregation of hedenbergite and hastingsite in ferro-syenite cumulate rocks through gravitational settling and/or flow differentiation spatially concentrated the Sc-bearing minerals within the intrusion, resulting in the first known scandium deposit hosted by syenite. (Beland, 2021).

The REE mineralization is contained in small primary idiomorphic zircon and hydroxyapatite crystals (identified by XRD analysis). The latter locally form aggregates that were wholly or partly replaced by britholite-(Ce). Two types of hydroxyapatite and one type of britholite-(Ce) have been identified. The first type of hydroxyapatite is magmatic and occurs as euhedral to subhedral, unzoned, transparent crystals that do not show evidence of having been altered. This type of apatite is very frequently observed in the other rock types of the intrusion. The second type of hydroxyapatite also occurs as primary, magmatic crystals but is compositionally zoned, with its core similar in composition to unzoned hydroxyapatite 1. This indicates that hydroxyapatite 2 continued to crystallize after hydroxyapatite 1. Crystals of hydroxyapatite 2 are commonly replaced in their outer parts by britholite-(Ce). Both types of hydroxyapatite commonly occur as inclusions in pyroxene, amphibole and, less commonly, fayalite.

7.4.1 Scandium

Scandium is a silvery-white transition metal, often classified as a REE along with yttrium and the 15 lanthanides. High-grade, large tonnage, easily mineable scandium deposits with favourable metallurgy and location are scarce, making it a commodity that is difficult to obtain in commercial quantities. Scandium is often found in trace amounts in REE deposits and occurrences, but it has only been mined as a by-product in a few uranium and REE mines globally, such as Zhovti Vody in Ukraine and Bayan Obo in China.

Two projects hosted in nickel laterite deposits in Australia have NI 43-101 or JORC-compliant resources that include scandium as one of the major products. They are presented here for comparative purposes, despite the different geological context. The Nyngan Project (Scandium International Mining Corp.) has mineral reserves of 1.4Mt at 409 g/t and M+I mineral resources of 16.92 Mt at 235 g/t Sc at a cut-off grade of 100 g/t (Rangott et al., 2016). The Platina Scandium Project (Platina Resources Limited) has mineral reserves of 4.02 Mt at 570 g/t Sc (cut-off grade of 400 g/t) and mineral resources of 35.6 Mt at 405 g/t Sc (cut-off grade of 300 g/t) (Platina Resources Ltd, 2018).

7.4.2 Nomenclature

The nomenclature for REE and associated metals is shown in Table 7-1. References to total rare earth oxide (TREO) include yttrium oxide unless otherwise stated.

Table 7-1: List of Elements and Oxides Associated with REE Mineralization

Element	Element Symbol	Common Oxide	Category
Light Rare Earth Oxides (LREO)			**Total Rare Earth Oxides (TREO)**
Lanthanum	La	La_2O_3	
Cerium	Ce	Ce_2O_3	
Praseodymium	Pr	Pr_2O_3	
Neodymium	Nd	Nd_2O_3	
Samarium	Sm	Sm_2O_3	
Heavy Rare Earth Oxides (HREO)			
Europium	Eu	Eu_2O_3	
Gadolinium	Gd	Gd_2O_3	
Terbium	Tb	Tb_4O_7	
Dysprosium	Dy	Dy_2O_3	
Holmium	Ho	Ho_2O_3	
Erbium	Er	Er_2O_3	
Thulium	Tm	Tm_2O_3	
Ytterbium	Yb	Yb_2O_3	
Lutetium	Lu	Lu_2O_3	
Yttrium	Y	Y_2O_3	
Niobium	Nb	Nb_2O_5	
Scandium	Sc	Sc_2O_3	

8 DEPOSIT TYPE

The following is taken from Quest (2014) and Daigle (2014):

"The Crater Lake Deposit is a large, scandium- and REE-bearing alkali igneous intrusive complex. Carbonatite and alkaline intrusive complexes (as well as their weathering products) are the primary sources of REE. Apart from REE, these rock types can also host deposits of niobium, phosphate, titanium, vermiculite, barite, fluorite, copper, calcite, and zirconium. Although these types of deposits are found throughout the world, only six are currently being mined for REE: five carbonatites (Bayan Obo, Daluxiang, Maoniuping, and Weishan deposits in China, and the Mountain Pass deposit in the USA) and one peralkaline intrusion-related deposit (as a byproduct at the Lovozero deposit, Russia).

Carbonatite and alkaline intrusive complexes are derived from partial melts of mantle material. Neodymium isotopic data of these deposits consistently indicate that the REE are derived from these parent magmas. These deposits and their associated rock types usually occur within stable cratonic settings, generally associated with intracontinental rift and fault systems. Extended periods of fractional crystallization of the magma in these settings led to enrichment in REE and other incompatible elements. In alkaline intrusive complexes, mineralization of REE occur as primary phases in magmatic layering or as later-stage dykes and veins (Verplanck et al., 2014).

REE deposits pose particular environmental challenges due to the associated uranium and thorium. There is also uncertainty surrounding the toxicity of the elements themselves. Acid mine drainage is typically not an issue due to the alkali nature of the rock types and minerals. Uranium has the potential for recovery as a byproduct, but thorium remains a waste product that requires management. Additionally, in some deposits, fluorine and beryllium can pose environmental challenges (Verplanck et al., 2014)."

9 EXPLORATION

9.1 Geophysical Surveys

9.1.1 2018 Geophysical Data Modelling

In February 2018, geophysical modelling provided a better understanding of the 3D geometry of the Crater Lake intrusive complex and the vertical and lateral extent of the known areas of scandium mineralization on the Property.

9.1.2 2020 Magnetic Ground Survey

In July and August 2020, Abitibi Geophysics of Val d'Or, Quebec, completed a detailed ground magnetic survey on the western half of the Property. The survey covered 130 line-km at a line spacing of 50 m.

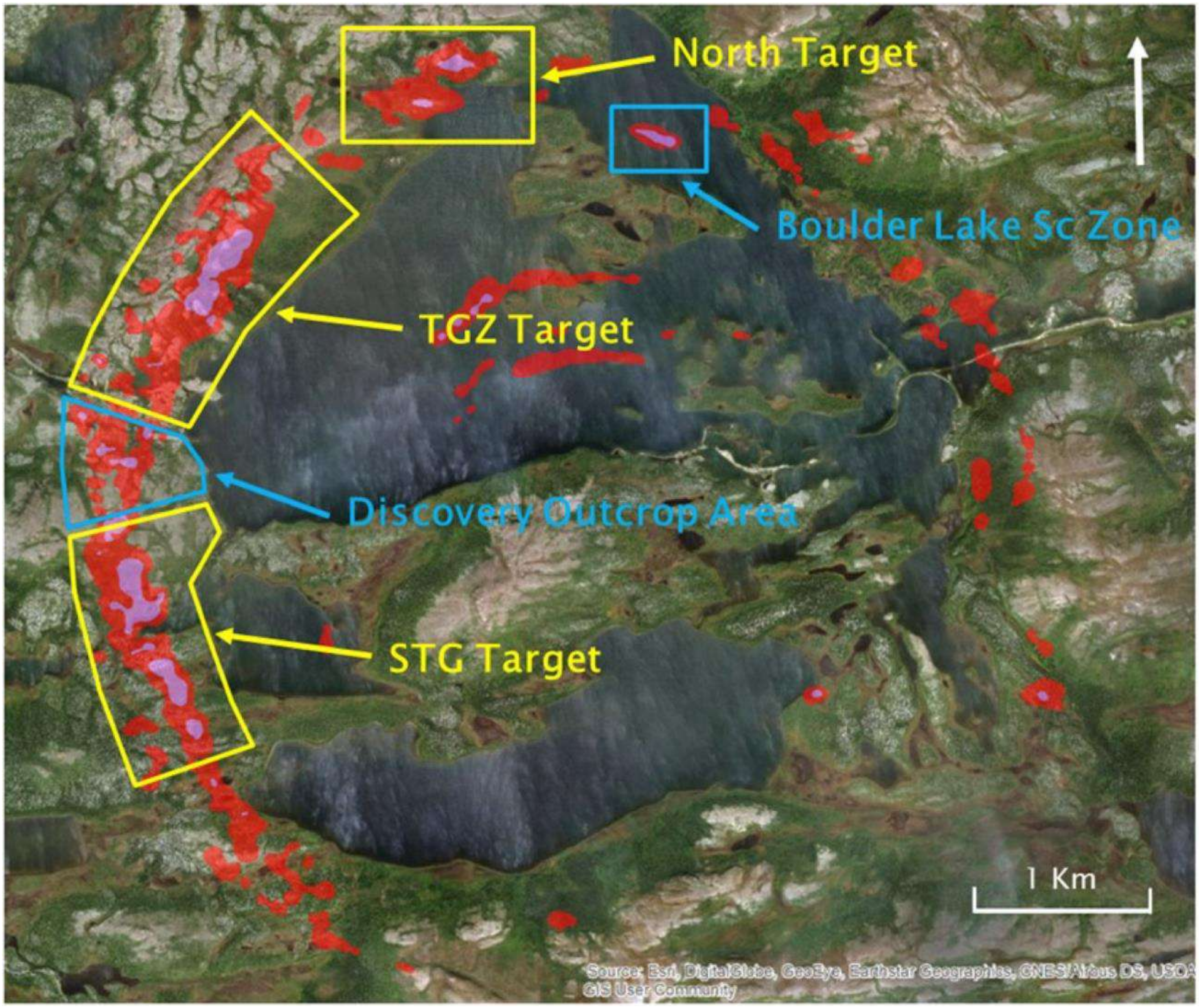
The survey better defined the scandium-bearing ferro-syenite rock units and fault structures controlling the concentration of scandium mineralization on the Property. Several new magnetic bodies were identified to the east and south of the STG target.

9.2 Surface Program

9.2.1 2018 Summer Exploration Program

Imperial's 2018 summer field program consisted of detailed prospecting and geological mapping over three highly prospective scandium targets: the TGZ, STG, and North Target areas (Figure 9-1). The prospecting and mapping program was followed by mechanical stripping and channel sampling. Scandium-rich outcrops and boulders in the vicinity of the TGZ and STG targets confirmed that both zones correspond to a similar scandium-rich target discovered in 2014 at the Boulder Scandium Zone. A total of 39 grab samples and 41 channel samples were collected. An additional 24 historical core samples were selected for a mineralogical study to be completed at McGill University in Montreal, Quebec. The best results from the program are illustrated in Table 9-1. Figure 9-2 displays the location of the field work results on the STG target.

Figure 9-1: Crater Lake Exploration Targets Over Ground Magnetic Map

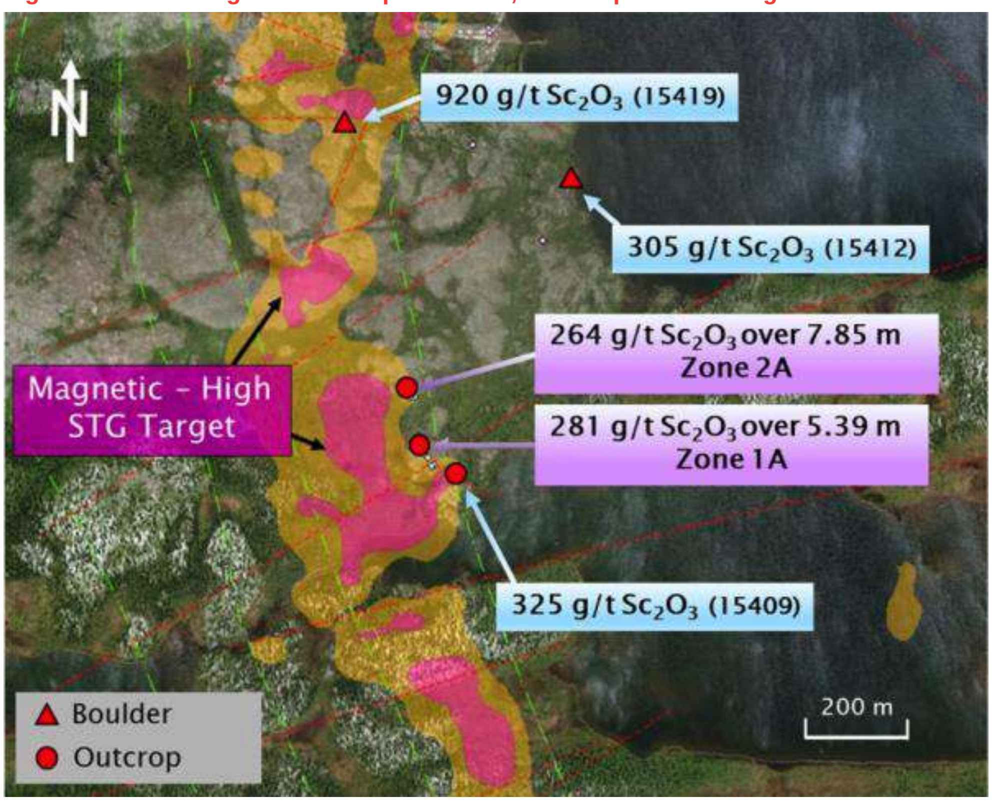


Source: Imperial (2018)

Table 9-1: Best Scandium Results on the Crater Lake Property, 2018 Exploration Program

Sample #	Easting	Northing	Sample Type	Target	Channel (m)	Rock Type	Sc2O3 (g/t)	TREO+Y (%)
15351	440672	6134128	Boulder	TGZ		Syenite	305	4.874
15352	440199	6133071	Boulder	TGZ		Syenite	57	8.296
15356	440222	6133196	Boulder	TGZ		Syenite	701	0.123
15402	440662	6132249	Boulder	STG		Fe-syenite	250	1.319
15403	440665	6132256	Boulder	STG		Fe-syenite	301	1.372
15407	440362	6131621	Outcrop	STG		Int-syenite	239	0.311
15409	440400	6131610	Outcrop	STG		Fe-syenite	325	0.329
15411	440662	6132249	Boulder	STG		Fe-syenite	308	1.379
15412	440665	6132256	Boulder	STG		Fe-syenite	305	1.140
15419	440168	6132371	Boulder	STG		Pyroxenite	920	1.010
15446	440359	6131619	Channel	STG, 1A	0.80	Intermediate-syenite	294	0.358
15501	440359	6131619	Channel	STG, 1A	0.80	Intermediate-syenite	298	0.362
15507	440.349	6131639	Channel	STG, 1B	1.00	Intermediate-syenite	305	0.358
15508	440349	6131639	Channel	STG, 1B	0.96	Intermediate-syenite	317	0.367

Figure 9-2: STG Target Grab Sample Results, 2018 Exploration Program



Source: Imperial (2018)

9.2.2 2020 Summer Exploration Program

A prospecting and mapping program was conducted over 38.2 km of unexplored terrain on the Property. 8 grab samples, 304 historical core samples and 17 new channel samples were selected for a detailed mineralogical and geochemical study at McGill University in Montreal, Quebec.

Furthermore, strongly magnetic, boulders of ferro-syenite were found in the Hilltop target area and 300 m northeast of the STG target.

9.2.3 2021 Summer Exploration Program

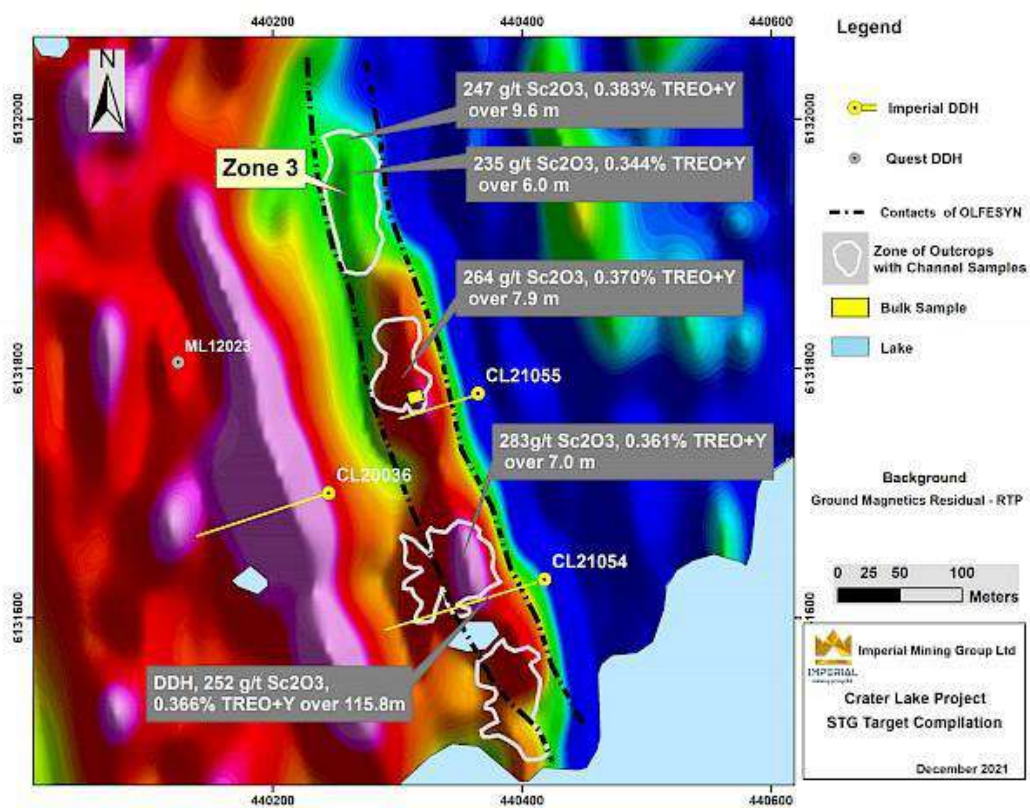
During the summer of 2021, a total of 23 channel samples representing a cumulative length of 23.4 m were collected at the STG target olivine ferrosyenite, located 2.0 km south of the TGZ target. Assay results are presented in Table 9-2 and Figure 9-3.

Table 9-2: Assay Results from the 2021 Channel Sampling Program

Channel #	Interval (m)	Sc (g/t)	Sc2O3 (g/t)	TREO+Y (%)
Zone 3A	9.60	161	247	0.383
Zone 3B	4.80	153	235	0.339
Zone 3C	6.00	153	235	0.344
Zone 3H	3.00	178	273	0.364

Note: 1 ppm of Sc metal equals 1.5338 ppm Sc_2O_3; TREO+Y includes oxides of La, Ce, Pr, Nd, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, and Lu plus Y.

Figure 9-3: Crater Lake 2021 Summer Channel Sampling Program



10 DRILLING

This item summarizes the issuer's 2019, 2020 and 2021 drilling campaigns (collectively, the "2019-2021 Program").

10.1 Drilling Methodology

The drilling was performed by Avataa Rouillier of Val-d'Or, Quebec, in 2019 and 2020, and by Cartwright Drilling of Happy Valley–Goose Bay, NL, in 2021. Collar locations were determined using a handheld GPS.

The drill was lined up using a Suunto compass. The downhole dip and azimuth were surveyed using a Reflex EZCOMII Shot tool. Surveys were taken at least every 30 m downhole. Prior to testing, at least 6 m of drill rods (2 rods) were removed from the hole to limit the chances of magnetic interference by the steel drill rods. Drilling contractors handled the instruments, and survey information was transcribed and provided in paper format to Imperial's geologists.

At the drill rig, the drill helpers placed core into core boxes and marked off each 3-m drill run using a labelled wooden block.

10.2 Collar Surveys

Casings were left in place with an identification tag. Collar locations were surveyed by Corriveau J.L. & Assoc. Inc of Val-d'Or after the drilling campaigns were completed.

10.3 Logging Procedures

The drill core was delivered by ATV or snowmobile and, when necessary, by helicopter, to the core shack area by drillers or by Imperial's staff, where it was cleaned of drilling additives and mud. An Imperial geologist quickly reviewed the core, checking for zones of mineralization and damaged or mislabelled core boxes. After fitting the core back together, the meterage was marked on the core and the RQD was estimated.

All data were recorded by the geologist using MX Deposit logging software. Input included descriptions of all aspects of significance: rock type, mineralization, alteration, structure, textures of interest. Photographs of selected portions of the core were taken and uploaded into the drilling software.

After samples were marked on the core, the core boxes were transferred to the core-saw shack and sawed by a technician. At this time, any thin section chips or core samples for specific gravity (SG) measurements were also cut. All thin-section cuts or SG samples were placed in a labelled sample bag and set aside. Once all sample intervals were sawed, the core technician placed one-half of the core in a labelled sample bag. The sampler stapled the sample tags to the core box underneath the half-core and re-wrote the sample interval's marks and the sample numbers on the remaining half with a red grease pencil. Bagged samples were loaded into rice bags to a total weight per bag between 10 and 20 kg. Each rice bag was labelled with the sample intervals and contact information (laboratory and company). The shipment data was entered into the shipment database.

Finally, overview photos of all core boxes were taken and uploaded into the logging software. The boxes were then transferred to the long-term core farm or temporarily placed in cross-piles.

10.4 Drill Programs

10.4.1 2019 Drill Program

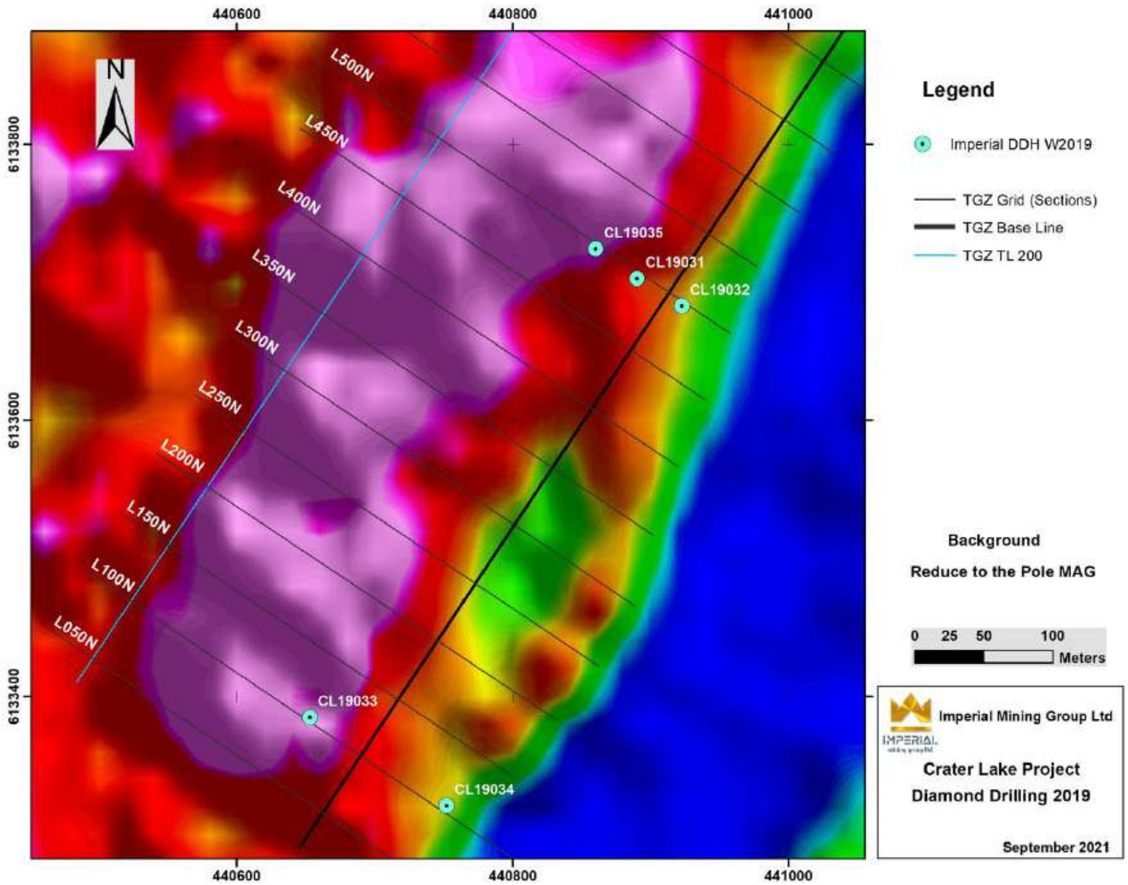
A winter drilling program totalling 1,014 m in five (5) holes was completed in late April 2019 on the TGZ target to evaluate the scandium potential of a high-intensity magnetic anomaly (Figure 10-1). Drilling took place 600 m north of a historical drill hole that had returned scandium grades of up to 506 g/t Sc_2O_3 over 19 m along the western side of the Crater Lake intrusion along the same magnetic trend. The best assay results are shown in Table 10-1. Figure 10-2 illustrates some of those results.

Table 10-1: Significant Assay Results from the 2019 Drilling Program

Hole #	From (m)	To (m)	Interval (m)	Sc (g/t)	Sc2O3* (g/t)	TREO+Y (%)
CL19031	115.80	148.75	33.0	207	318	0.340
	190.95	208.45	17.5	192	295	0.335
CL19032	145.15	220.00	47.9	251	341	0.421
CL19033	4.85	39.85	35	181	278	0.412
	63.75	177.65	113.9	202	310	0.370
CL19035	13.35	108.80	95.5	205	314	0.371

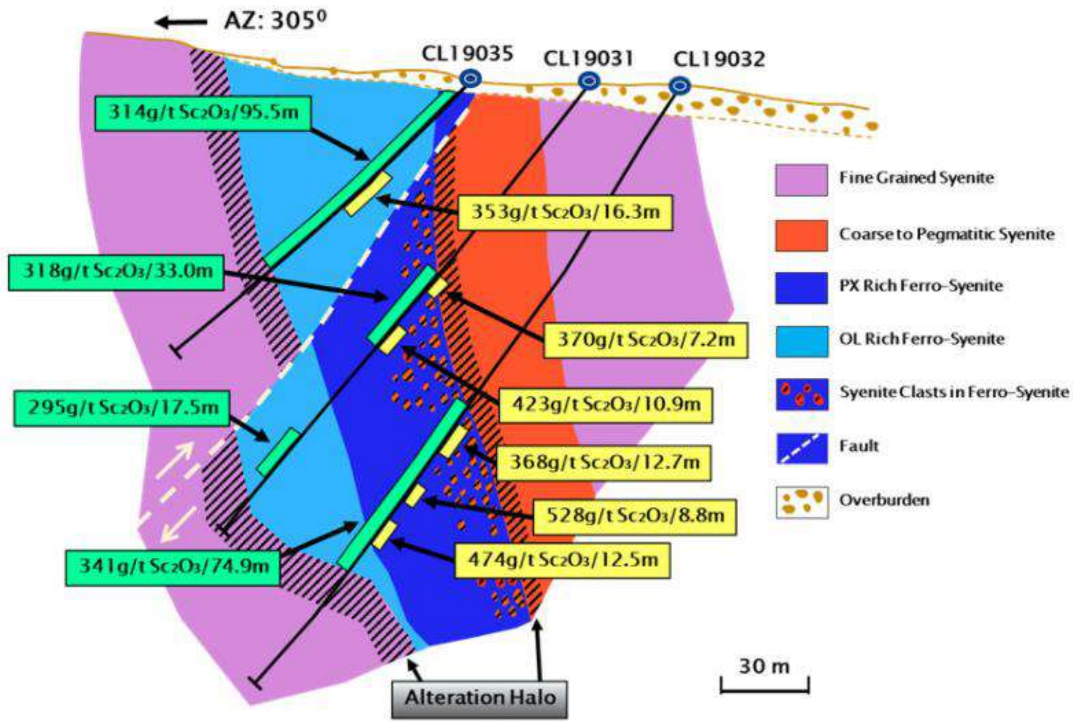
Note:
* 1 ppm of Sc metal equals 1.5338 ppm Sc_2O_3

Figure 10-1: Crater Lake 2019 Drilling Program



Source: Imperial (2021)

Figure 10-2: DDH Cross Section 500N, TGZ Target, 2019 Drilling Program



Source: Imperial (2019)

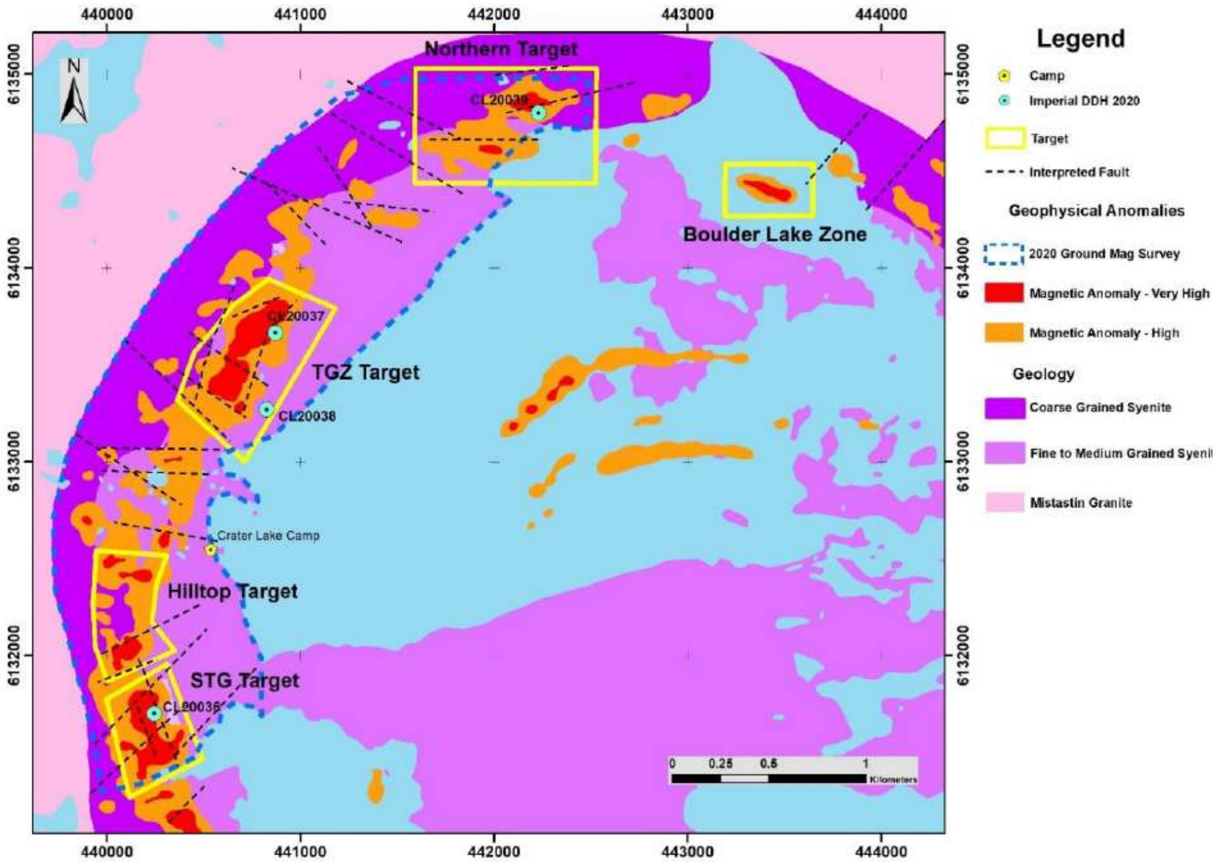
10.4.2 2020 Drill Program

A drilling program totalling 676 m in four (4) holes was completed between August 14, 2020, and 31, 2020. The drilling program was designed to test the scandium potential of high-intensity magnetic anomalies in the STG, TGZ, and Northern target areas (Figure 10-3).

Results from the 2020 drill program include:

- CL20036: Tested a strong magnetic anomaly parallel and west of the STG target (previous surface channel grading 289 g/t Sc_2O_3 and 0.364% REE over 7.04 m). The magnetic anomaly was explained by a 90 m interval of amphibole-rich ferro-syenite. No significant Sc or REE mineralization was intersected.
- CL20037: Tested the lateral continuity of the TGZ target. The mineralized intervals (total true length of 110 m) show excellent lateral and vertical continuity of the favourable TGZ target horizon. The best Sc and REE intervals from this hole are presented in Table 10-2.
- CL20038: Tested a ferro-syenite unit that was previously intersected parallel and east of the main TGZ target. The hole intersected multiple, narrow scandium-bearing ferrosyenite intervals grading up to 244 g/t Sc_2O_3 and 0.71% TREO+Y over 2.6 m and 192 g/t Sc_2O_3 and 0.50% TREO+Y over 3.6 m.
- CL20039: Tested a strong, 350-m-long by 100-m-wide magnetic anomaly over the Northern target. The geophysical anomaly was explained by the intersection of a coarse-grained syenite with small concentrations of pyroxene and amphibole. The hole did not yield any significant scandium or REE assays.

Figure 10-3: Crater Lake 2020 Drilling Program



Source: Imperial (2020)

Table 10-2: Significant Assay Results from the TGZ Target, 2020 Drilling Program

Hole #	From (m)	To (m)	Interval (m)	Sc (g/t)	Sc_2O_3 (g/t)	TRE+Y (%)	Magnet REO (%)
CL20037	127.81	156.95	29.14	165	253	0.305	24.3
	173.05	190.70	17.65	194	298	0.332	24.4
	207.82	218.52	10.70	152	233	0.389	24.4
	225.97	247.66	21.69	177	271	0.419	23.9

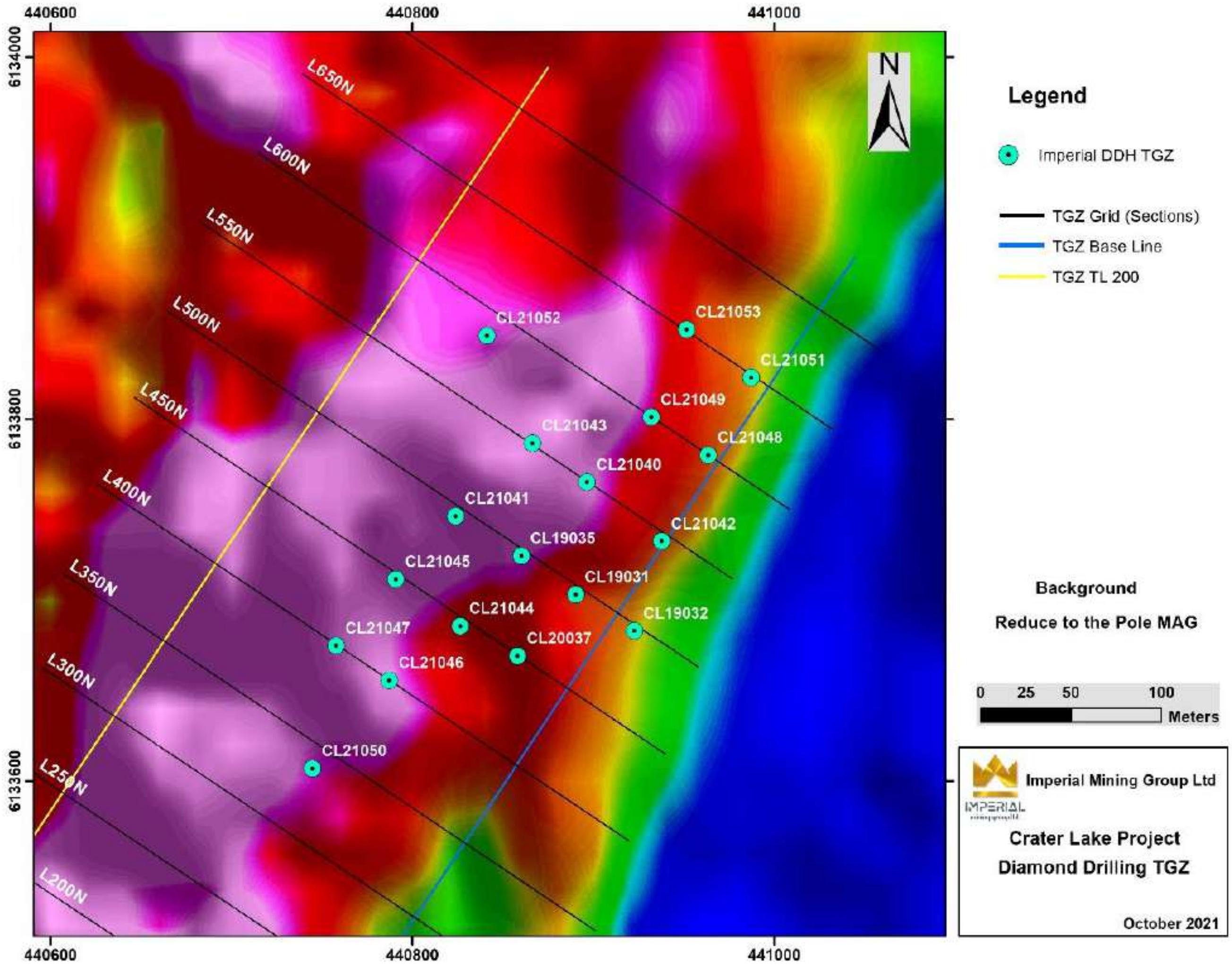
Note: 1 ppm of Sc metal equals 1.5338 ppm Sc2O3;.TREO+Y includes oxides of La, Ce, Pr, Nd, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb and Lu plus Y. The Magnet REOs include the total of Nd, Pr, Dy and Tb oxides as a percentage of the TREO+Y

10.4.3 2020-2021 Winter Drilling Program

A 14-hole drilling program totalling 2,049 m was completed on May 9, 2021, on the TGZ target (Figure 10-4). The objective of this drilling program was to outline mineral resources on 50- to 100-m centres.

All holes intersected the target mafic intrusive host rock and indicated that the TGZ target dips between 83° W to 70° E and strikes NNE. The true thickness of the mineralized zone varies between 55 and 135 m. Drilling has defined the mineralization on 50-m sections between sections 350N and 650N. The mineralization has been traced by drilling over 300 m in total strike length down to a vertical depth of up to 200 m. The best assay results are reported in Table 10-3. Figure 10-5 and Figure 10-6 illustrate some of those results.

Figure 10-4: Crater Lake 2021 Drilling Program



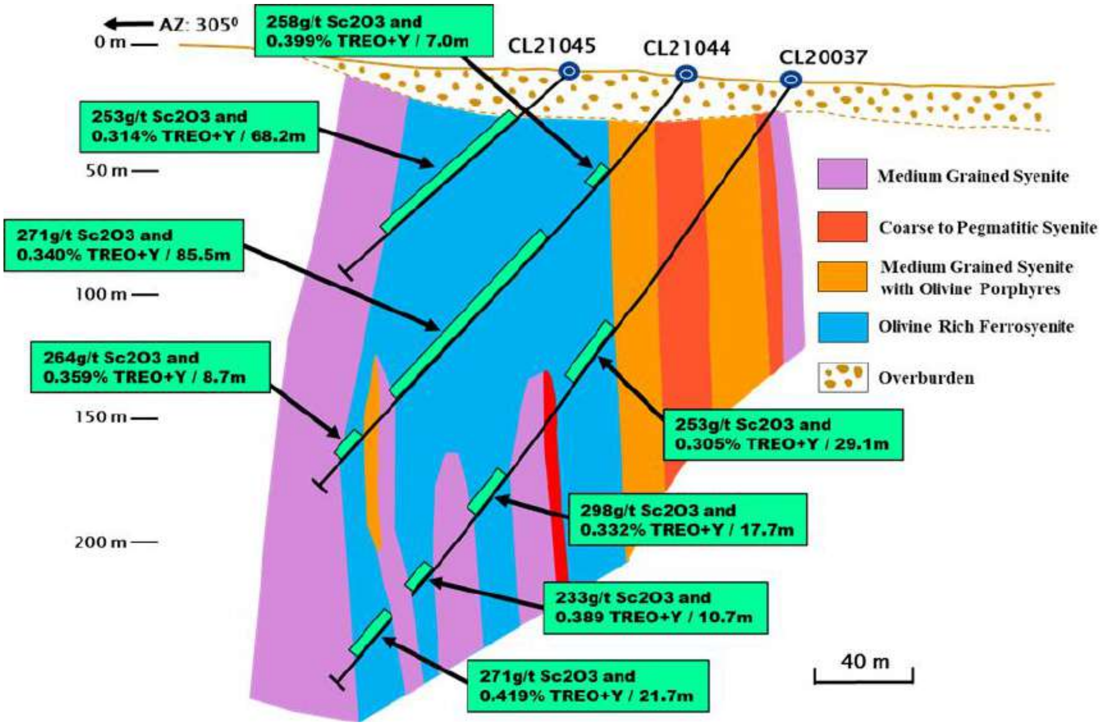
Source: Imperial (2021)

Table 10-3: Significant Assay Results from the 2021 Drilling Program

Hole #	From (m)	To (m)	Interval (m)	Sc (g/t)	Sc_2O_3 (g/t)	TRE+Y (%)
CL21040	20.85	101.5	80.2	187	287	0.320
CL21041	9.9	28.47	18.57	228	350	0.420
CL21042	46.95	81.38	34.43	198	304	0.380
	111.34	203.86	92.5	190	291	0.320
CL21043	9.4	32.40	23.0	199	305	0.390
CL21044	46.85	132.33	85.48	177	271	0.3396
	160.17	168.84	8.67	172	264	0.3590
CL21045	18.70	86.90	68.20	165	253	0.3141
CL21046	107.15	160.60	53.45	172	264	0.3258
CL21047	14.90	31.40	16.50	130	199	0.2684
	36.85	61.48	24.63	129	198	0.2633
	69.07	98.40	29.33	181	278	0.3310
CL21048	50.57	162.50	111.93	194	298	0.3547
CL21049	38.00	115.43	77.43	204	313	0.3441
CL21050	45.63	53.30	7.67	170	261	0.3305
	91.74	111.92	20.18	183	281	0.3873
CL21051	90.45	98.65	8.20	177	271	0.3197
	104.64	121.80	17.16	182	279	0.3218
	131.00	156.71	25.71	208	319	0.3481
CL21052	55.95	155.75	99.80	195	299	0.3417
CL21053	44.56	53.28	8.72	188	288	0.3720
	58.40	63.67	5.27	196	301	0.4239

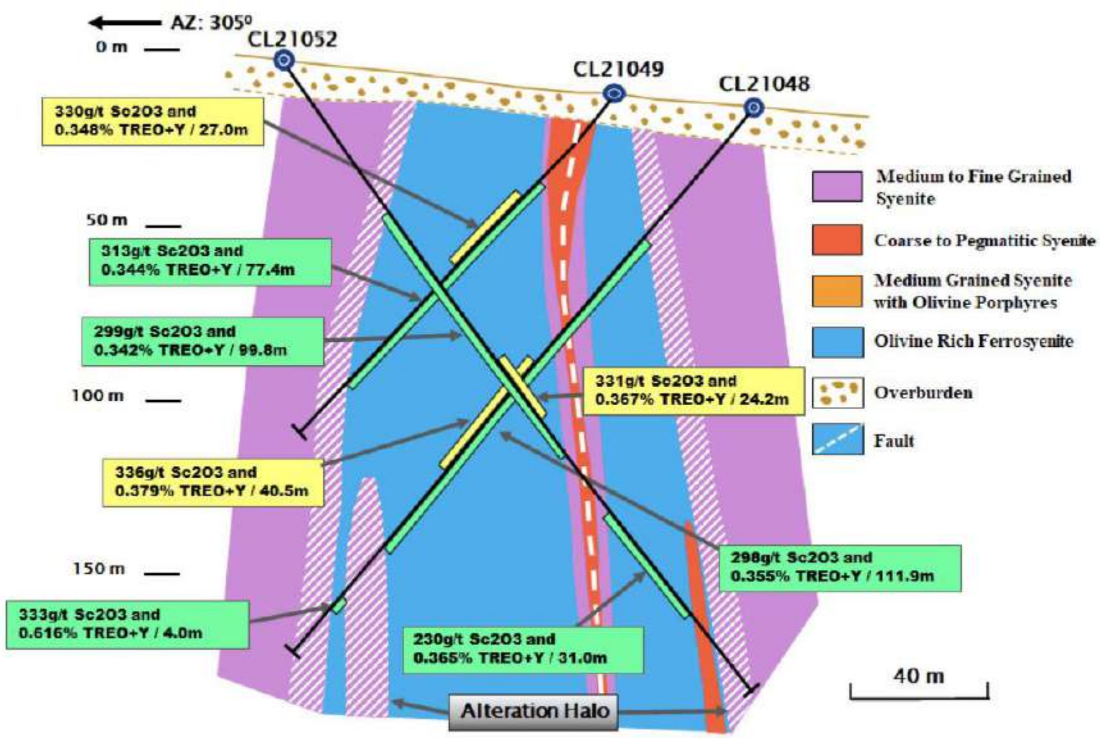
Note: 1 ppm of Sc metal equals 1.5338 ppm Sc2O3; TREO+Y includes oxides of La, Ce, Pr, Nd, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, and Lu plus Y.

Figure 10-5: DDH Cross Section 450N, TGZ Target, 2021 Drilling Program



Source: Imperial (2021)

Figure 10-6: DDH Cross Section 600N, TGZ Target, 2021 Drilling Program



Source: Imperial (2021)

10.4.4 2021 Summer Drilling Program

During the summer of 2021, a two-hole drilling program for 345 m was undertaken to undercut channel sampling and geophysical results (Figure 10-7) over the STG target, located 2.0 km south of the TGZ target.

Hole CL21054 intersected an olivine ferrosyenite grading 252 g/t Sc_2O_3 and 0.366% TREO+Y over 115.8 m. The hole undercut a previous channel sample assay of 283 g/t Sc_2O_3 and 0.361% TREO+Y over 7.0 m (Figure 10-8).

Hole CL21055 was planned as an undercut to a previous surface channel sampling results that returned 264 g/t Sc_2O_3 over 7.85 m. The hole intersected non-mineralized felsic syenite. A reduction of the magnetic intensity over the drill area and the observed drillhole geology is interpreted to have resulted in the olivine ferrosyenite at surface having been dyked-out by the younger unmineralized intrusion. More drilling is planned to better understand the geology in this area.

Figure 10-7: Crater Lake 2021 Summer Drilling Program

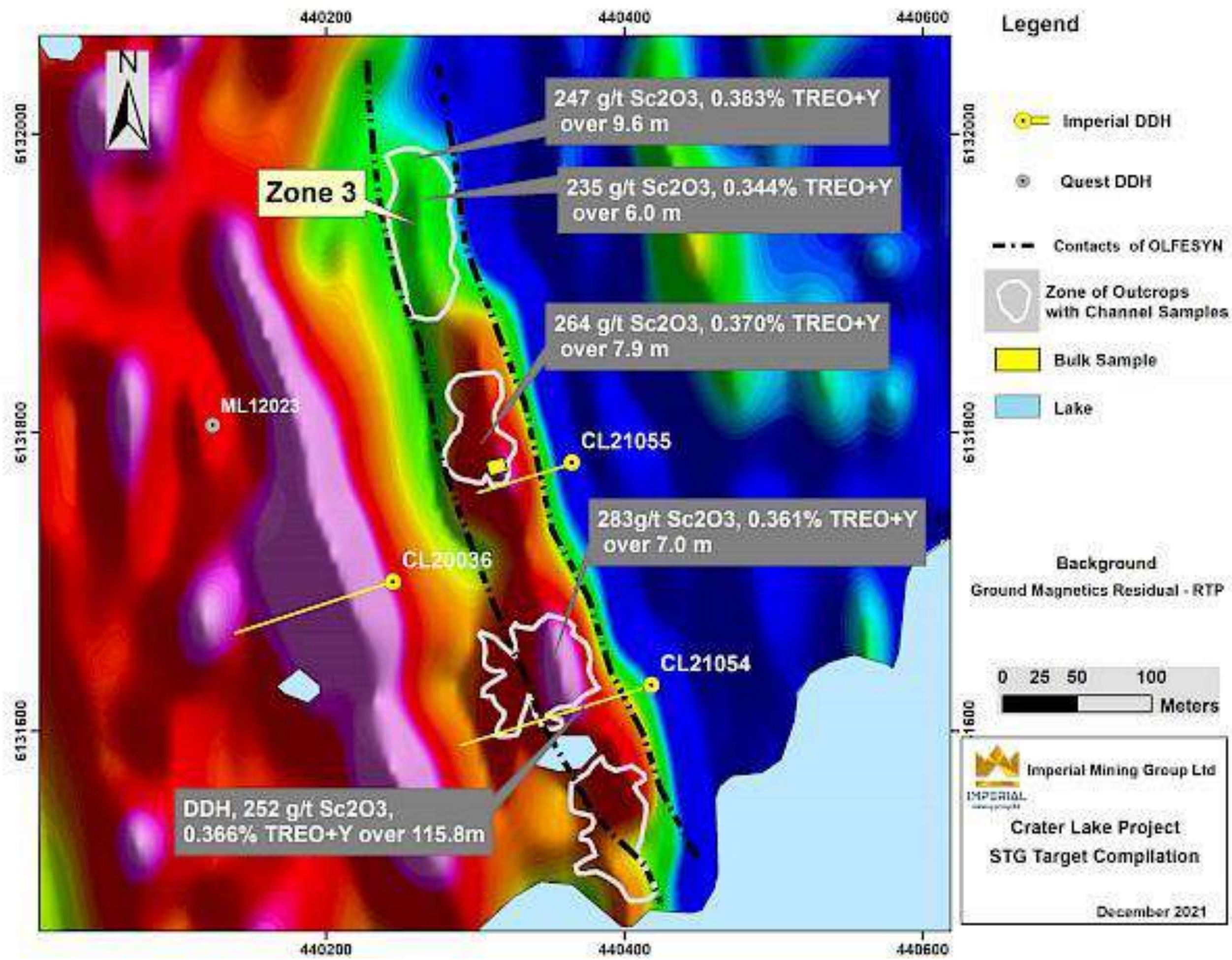
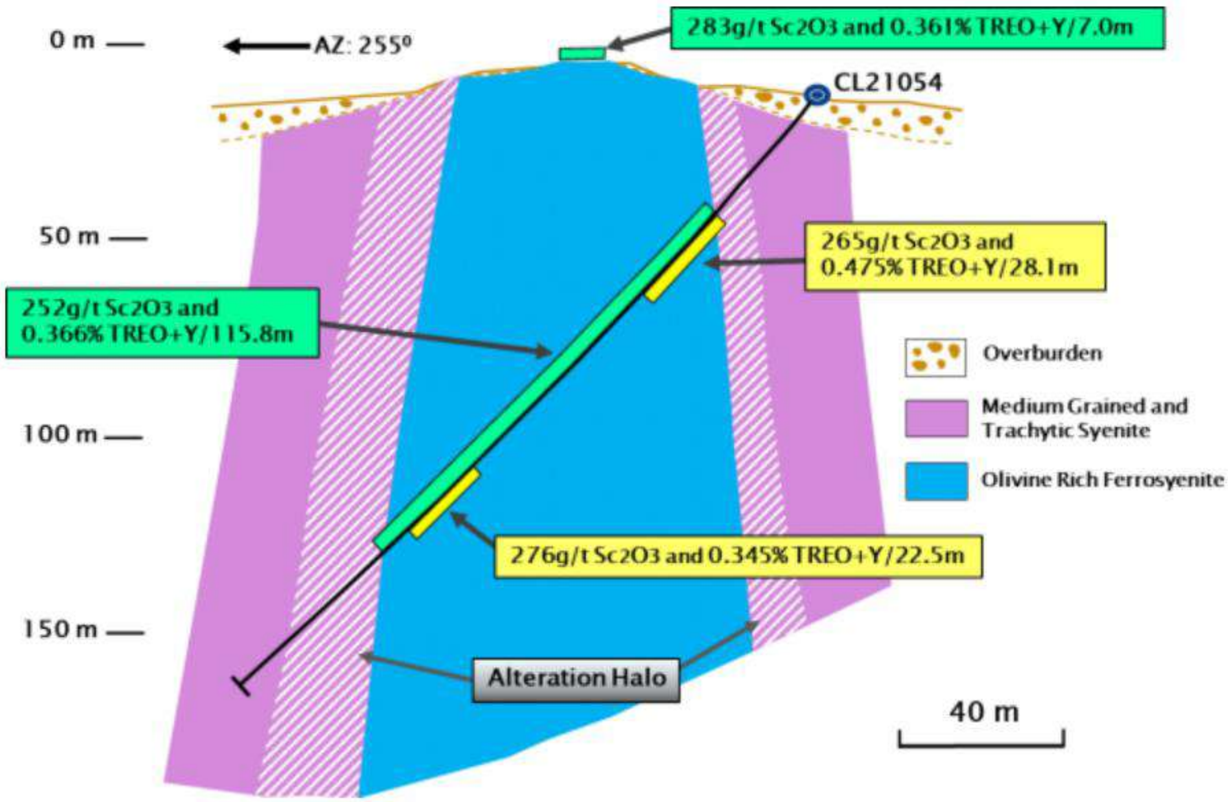


Figure 10-8: DDH Cross Section, STG Target, 2021 Drilling Program



11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This Item describes the issuer's sample preparation, analysis, and security procedures for the 2019-2021 diamond drilling campaigns (the "2019-2021 Program"). The issuer's geology team provided the information discussed below. InnovExplo reviewed and validated the information for the 2019-2021 Program, including the QA/QC procedures and results.

11.1 Core Handling, Sampling and Security

The drill core is boxed and sealed at the drill rigs and delivered daily by road or helicopter to the logging facility where a technician takes over the core handling. Drill core is logged and sampled by experienced geologists or by a geologist-in-training under the supervision of a qualified geologist. A geologist marks the samples by placing a unique identification tag at the end of each core sample interval. Core sample lengths vary from 0.15 to 2 m, and sample contacts respect lithological contacts and/or changes in the appearance of mineralization or alteration (type and/or strength). The technician saws each marked sample in half. One half of the core is washed with clean water and placed in a plastic bag along with a detached portion of the unique bar-coded sample tag, and the other half of the core is returned to the core box with the remaining tag portion stapled in place. The core boxes are stockpiled or stored in outdoor core racks for future reference. Individually bagged samples were placed in security-sealed rice bags along with the list of samples for delivery to the assay laboratory.

QA/QC sample tags are also placed in the core boxes. Once core sampling is complete, the sampling technician adds the corresponding barren ("blanks") and standard samples (certified reference materials or "CRMs") to the shipments. For each shipment of 100 samples, no less than two (2) blanks and four (4) CRMs are included with the core samples.

11.2 Laboratory Accreditation and Certification

The International Organization for Standardization ("ISO") and the International Electrotechnical Commission ("IEC") form the specialized system for worldwide standardization. ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories sets out the criteria for laboratories wishing to demonstrate that they are technically competent, operating an effective quality system, and able to generate technically valid calibration and test results. The standard forms the basis for the accreditation of competence of laboratories by accreditation bodies.

For the 2019-2021 Program, samples were sent and prepared at Activation Laboratories Ltd. ("Actlabs") in Ancaster, Ontario for assaying. Actlabs received ISO/IEC 17025 accreditation through the Standards Council of Canada ("SCC"). Actlabs is a commercial laboratory independent of the issuer and has no interest in the Project.

11.3 Laboratory Preparation and Assays

Samples were analyzed for REE at the Actlabs laboratory in Ancaster, Ontario. Procedures used were Inductively Coupled Plasma ("ICP") for major elements and Inductively Coupled Plasma/ Mass Spectrometry ("ICP-MS") for trace elements.

The methodology is described as follows:

- Samples are sorted, bar-coded and logged into the Actlabs Lims program. They are then placed in the sample drying room.
- Samples are crushed to 80% passing 10 mesh (2.00 mm) and split using a Jones riffle splitter. A 250-g or 500-g split is pulverized to 95% passing 200 mesh (0.07mm). Only 50 g of the 500 g is used for the analysis itself (code RX-1: 500). The remaining 450 g is returned as pulp to the issuer's office, along with the reject from the original sample.
- Assay results are provided in Excel spreadsheets and the official certificate (sealed and signed) as a PDF.
- The pulverized pulp is placed in kraft sample bags, and the un-pulverized portions returned to their original sample bags.
- The remainder of the crushed samples (the rejects) and the pulps are stored at Actlabs facility.

11.4 Quality Assurance and Quality Control (QA/QC)

The issuer's QA/QC program for drill core includes the insertion of blanks and standards in the sample stream of core samples. About 6% of the samples were control samples in the sampling and assaying process. Four (4) standard and two (2) blank samples of barren rock were added to each group of 100 samples as an analytical check for the laboratory batches. In addition, the issuer's QA/QC includes field duplicate samples that comprised 1% of the core selected as quarter core sample duplicates for comparison with the original core sample.

Imperial's geologists were responsible for the QA/QC and database compilation. Upon receiving the analytical results, the geologists extracted the results for blanks and standards to compare against the expected values. If QA/QC acceptability was achieved for the analytical batch, the data were entered into the project's database; if not, the laboratory was contacted to review and address the issue, including retesting the batch if required.

The discussion below details the results of the blanks and standards used in the issuer's QA/QC program.

11.4.1 Certified Reference Materials (Standards)

Accuracy is monitored by inserting CRMs at a ratio of four (4) for every 100 samples (1:25). The standards were supplied by OREAS, Sudbury, Ontario. The definition of a QC failure is when the assay result for a standard falls outside three standard deviations ("3SD"). Gross outliers are excluded from the standard deviation calculation.

Between 2019 and 2021, three (3) different CRMs were used, two (2) for Sc only and one (1) for Sc and REE. Of the 82 CRMs inserted, four (4) returned results outside 3SD. From those four (4) fails, two (2) were identified as gross outliers, and the issuer took actions to explain the cause of the abnormal values. One (1) was a case of the incorrect standard recorded in the database (O460 instead of O464), and the other was an inversion between two samples (710032 and 710033). The database was corrected, and the gross outliers were removed from the QA statistics when they were identified as QC failures.

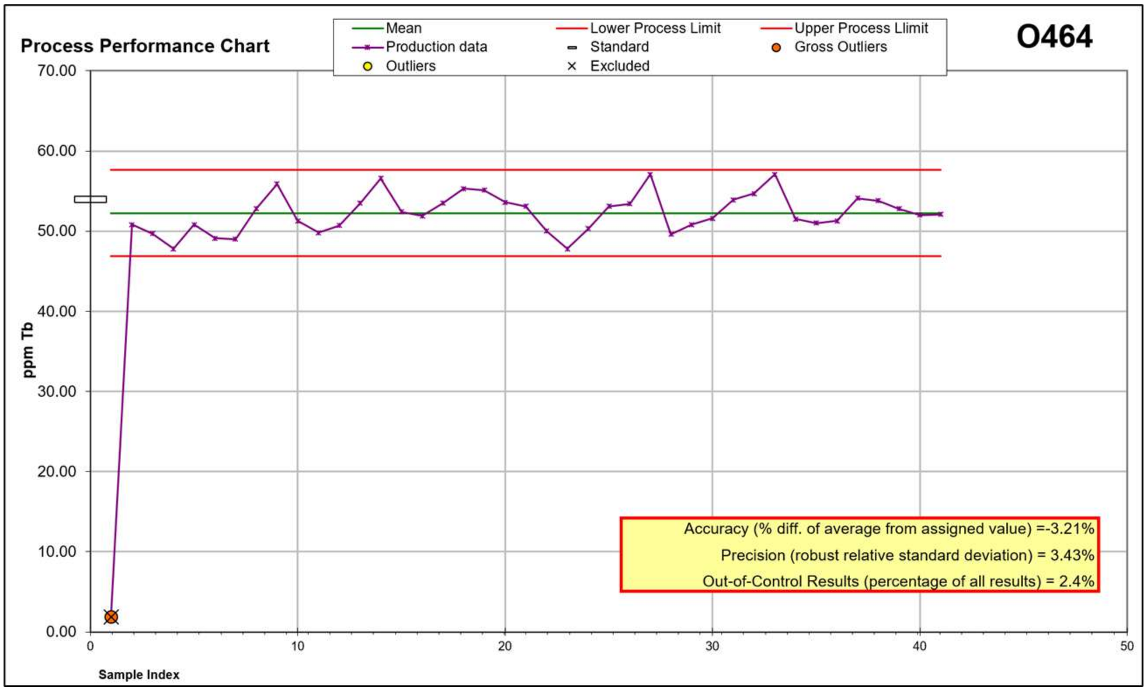
The overall success rate was 98% (Table 11-1). Overall, outliers did not show a persistent analytical bias (either below or above the 3SD limit). They were close to the 3SD threshold and appeared to be isolated errors, as other standards and blanks processed from the same batches passed. Consequently, no batch re-runs were performed.

Figure 11-1 shows an example of a control chart for the standard OREAS 464 assayed by Actlabs. A similar control chart was prepared for each CRM to visualize the analytical concentration value over time.

Table 11-1: Results of Standards Used Between 2019 to 2021 on the Project

CRM	No. of Assays	Metal	CRM value for Peroxide Fusion ICP (g/t)	CRM value for Acid Digestion (g/t)	Average (g/t)	Accuracy (g/t)	Precision (%)	Outliers	Gross Outliers	Percent passing QC
Oreas 180	17	Sc		41.5	42.8	3.2	3.6	0	0	100
Oreas 460	24	Sc		27.9	29.8	6.7	1.8	0	1	100
Oreas 464	41	Sc		141.0	155.2	10.1	1.8	2	1	95
		Dy	178		175	-1.7	3.5	0	1	100
		La	11700		11837.5	1.2	2.6	1	1	96
		Nd	9940		9594	-3.5	2.6	1	1	96
		Pr	2597		2539	-2.2	3.2	0	1	100
		Tb	54		52.27	-3.2	3.4	1	1	96

Figure 11-1: **Control Chart for Standard OREAS 464 Assayed by Actlabs**



For the scandium, the results exhibit a positive bias in terms of accuracy with an average of +7.7% and a precision of around 2.2% for representative standards. The accuracy bias can be explained by the difference of analysis methods used by OREAS and by Actlabs. OREAS standard value was defined with acid digestion while Actlabs used ICP to analyze samples. As ICP is a more precise analysis method than acid digestion, we can expect a more complete evaluation of the scandium content.

For the REE, the results exhibit a slight negative bias in terms of accuracy with an average of -1.9% and a precision around 3.1% for representative standards.

Both parameters meet standard industry criteria.

11.4.2 Blank Samples

Contamination is monitored by the routine insertion of a barren sample (blank) which goes through the same sample preparation and analytical procedures as the core samples.

A total of 49 blanks were inserted in the batches from the 2019 to 2021 drilling programs. The blanks were supplied by OREAS. The blank material consists of ornamental silica pebbles. The source deposit is situated in Carboniferous sedimentary rocks of the Maritimes Basin in New Brunswick. Blank material contents in Sc and La were defined by OREAS using aqua regia digest analysis method. Dy, Nd, Pr and Tb contents were not analyzed. A general guideline for success during a contamination QC program is a rate of 90% of blank assay results not exceeding the acceptance limits of three times (3x) the detection limit. The detection limit was 1 g/t for Sc and 10 g/t for La with the aqua regia digest analysis method.

For the 2019-2021 Program, no sample returned grades higher than 3x the detection limit for Sc and La (Table 11-2, Figure 11-2, and Figure 11-3). For Dy, Nd, Pr, and Tb, no values are available for the blank material; therefore, it is impossible to define a precise detection limit for those elements. However, we can note that the blank material's Dy, Nd, Pr, and Tb results from Actlabs are all homogenous and very low grade.

Table 11-2: Results of Blanks Used between 2019 and 2021

Metal	Acceptance Limit (g/t)	Quantity Inserted	Quantity Failed	Percent Passing QC
Scandium	1	49	0	100
Lanthanum	10	49	0	100

Figure 11-2: Time Series Plot for Blank Samples Assayed by Actlabs between 2019 and 2021 (Sc)

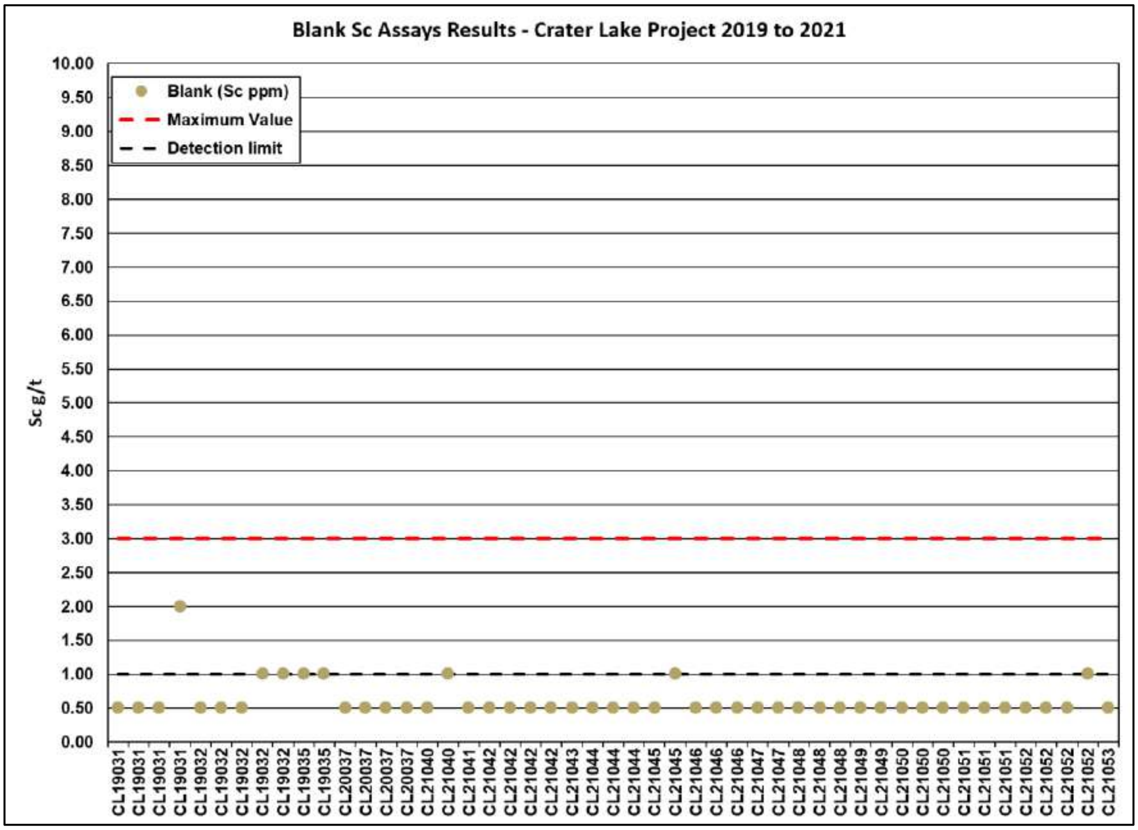
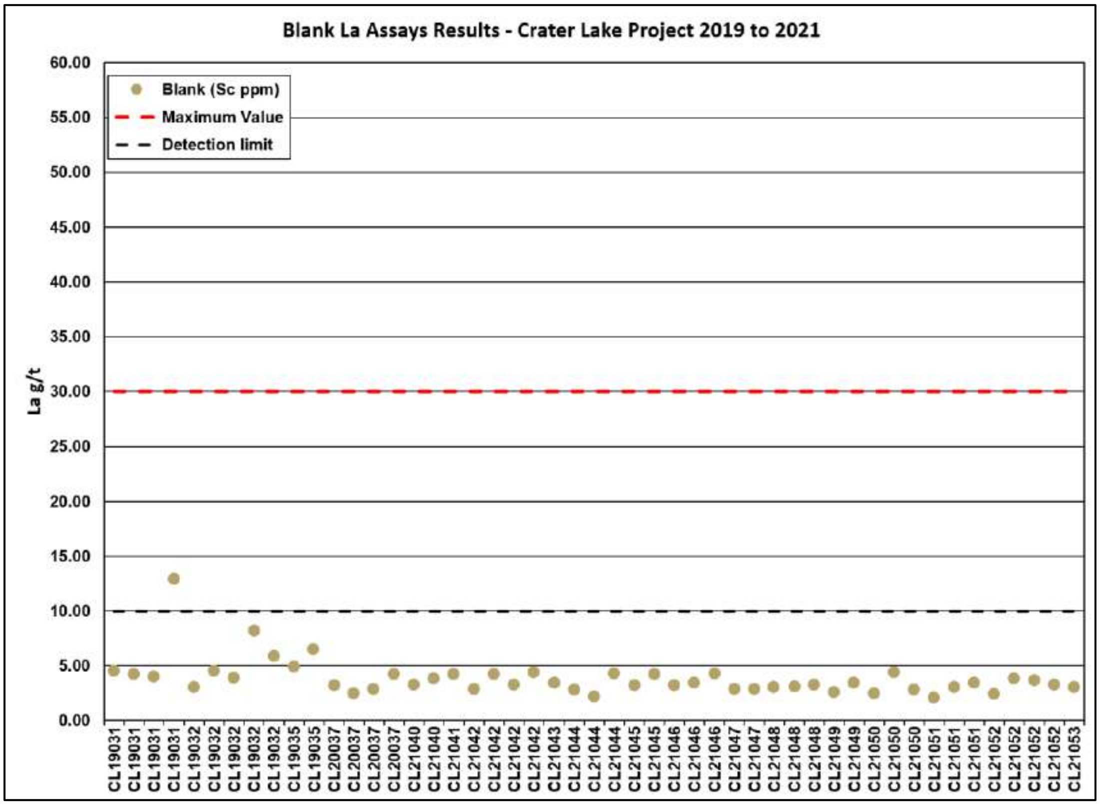


Figure 11-3: Time Series Plot for Blank Samples Assayed by Actlabs between 2009 and 2021 (La)



11.4.3 Field duplicates

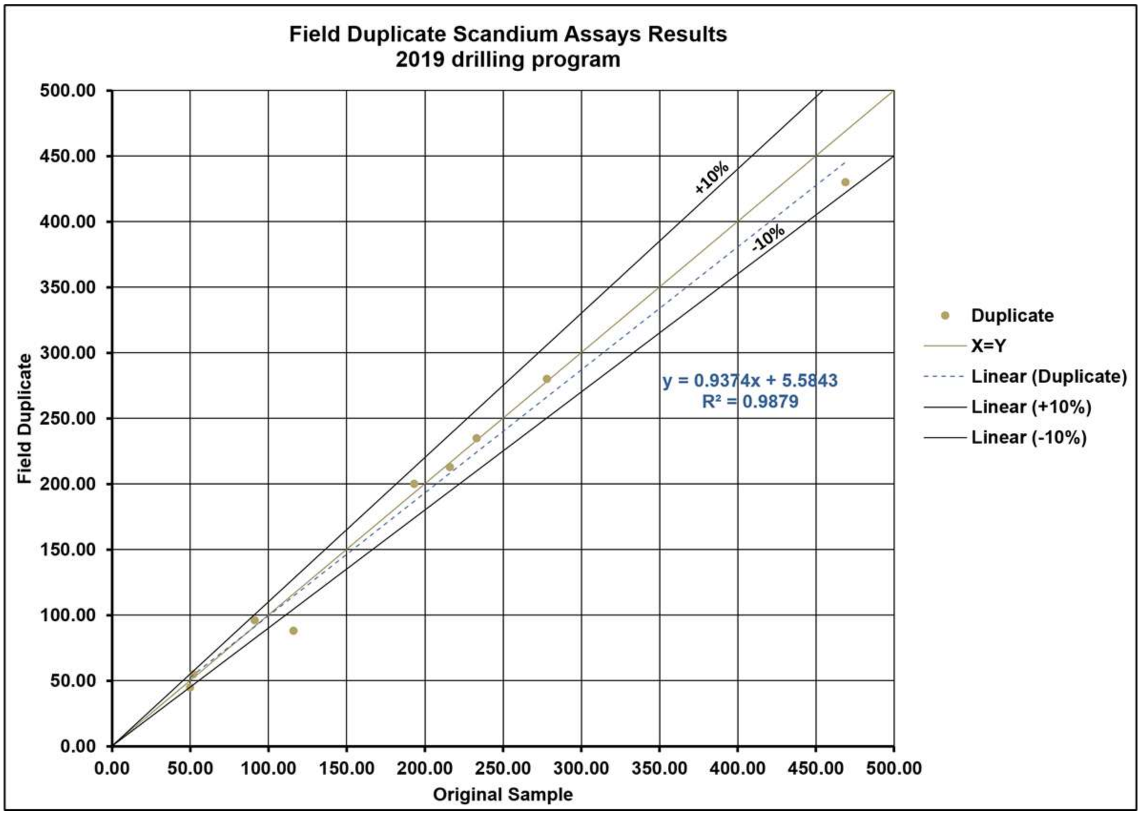
The 2019 Program included quarter-core duplicate samples to assess the presence of a "nugget effect" or heterogeneity of mineralization within individual intervals of sampled drill core. The issuer inserted nine (9) quarter-core duplicates into the sample stream. The difference between the original analysis and the quarter-core duplicate analysis is presented in Table 11-3. Figure 11-4 shows the scatter plots for Sc.

Results show a good precision with R^2=0.98 for Sc and an average of R^2=0.98 for La, Pr, Nd, Tb and Dy. Results also show a good accuracy monitored by the linear regression line for all studied metals (between the 10% tolerance limit). This good repeatability shows that Sc and REE distribution in the core seems homogenous.

Table 11-3: Results of Field Duplicates used During the 2019 Drilling Program

Metal	Coefficient of determination R^2 (%)
Sc	98
La	87
Pr	97
Nd	98
Tb	99
Dy	99

Figure 11-4: Linear Graph Comparing Original and Field Duplicate Samples Analyzed in 2019 (Sc)



11.5 Conclusion

The authors are of the opinion that the sample preparation, security, analysis, and QA/QC protocols performed by the issuer followed generally accepted industry standards, and that the data is valid and of sufficient quality for a mineral resource estimation.

12 DATA VERIFICATION

12.1 Crater Lake Scandium Verification

This item covers the data verification of the diamond drill hole databases supplied by the issuer (the "Imperial Database"). The database close-out date for this Technical Report is July 09, 2021.

The author's data verification included visits to the Property, drill sites and core logging facilities, as well as an independent review of the data for selected drill holes (surveyor certificates, assay certificates, QA/QC program and results, downhole surveys, lithologies, alteration and structures).

12.2 Site visit

The author, Marina Iund, visited the Project and the issuer's core shack from May 7 to May 9, 2021. She was accompanied by Pierre Guay, the issuer's VP Exploration. The site visit focused on the TGZ mineralized domains. Onsite data verification included a general visual inspection of the property and a review of drill collar location coordinates. At the core shack, the author examined selected mineralized core intervals, reviewed the QA/QC program and the descriptions of lithologies, alteration and mineralization. She also performed independent check assays on selected intercepts.

12.3 Core Review

The core boxes are stored on pallets. The core boxes were found to be in good order and properly labelled, and the sample tags were still present. The wooden blocks at the beginning and end of each drill run were still in the boxes, and they matched the indicated footage on each box. The author validated the sample numbers and confirmed the presence of mineralization in the reference half-core samples (Figure 12-1).

Figure 12-1: Photographs Taken during the Drill Core Review

Notes:
A) Mineralization from hole CL21040; B) Sawing facility; C) Core shack; D) Proper labelling of the drill core boxes and sample tag stapled in core box; E) Standard; F) Blank; G) quarter splits sample collected in hole CL21040

The author selected representative mineralized intercepts and collected five (5) samples for independent assaying. The samples are quarter splits, sawed by the issuer's contractor (Figure 12-1G). The samples were placed in plastic bags, sealed with plastic zip ties and packed in rice bags for transport to the independent assaying laboratory. Marina Iund transported the samples to the AGAT laboratory facility in Quebec City.

The results of the independent re-assaying show a general correlation between the original and re-assayed scandium and REE values. All five (5) mineralized samples yielded subeconomic to economic values for the intercepts (Table 12-1).

The author believes the field duplicates from the independent resampling program are reliable and consistent with the database.

Table 12-1: Results of InnovExplo's Independent Sampling

Sample Type	Hole ID	From (m)	To (m)	Sample ID	Sc (g/t)	Dy (g/t)	La (g/t)	Nd (g/t)	Pr (g/t)	Tb (g/t)
Original (Imperial)	CL21047	17.52	19	711946	199	71	673	639	171	12
	CL21041	23	24.5	711328	251	76	752	699	193	13
	CL21040	80.5	82	711278	202	60	525	548	142	10
	CL19035	35.5	37	710735	210	60	535	524	141	10
	CL20037	151.5	152.37	711033	198	59	490	513	135	10
Field Duplicate (InnovExplo)	CL21047	17.52	19	K504275	261	62	523	513	137	11
	CL21041	23	24.5	K504276	318	73	668	701	182	13
	CL21040	80.5	82	K504277	254	51	278	294	79	9
	CL19035	35.5	37	K504278	193	47	424	448	117	8
	CL20037	151.5	152.37	K504279	249	54	460	517	133	10
Difference					24%	-15%	-29%	-25%	-25%	-8%
					21%	-4%	-13%	0%	-6%	0%
					20%	-19%	-89%	-86%	-80%	-15%
					-9%	-28%	-26%	-17%	-21%	-21%
					20%	-10%	-7%	1%	-2%	-6%

12.4 Databases

12.4.1 Drill Hole Locations

The drill hole collars from the 2019-2021 diamond drilling campaigns (the "2019-2021 Program") were surveyed by Corriveau J.L. & Assoc. Inc. using a GPS base station.

The author confirmed the coordinates of eight (8) selected surface holes using a handheld GPS (Figure 12-2 and Table 12-2), then compared them to the database. All results had acceptable precision.

The collar locations in the Imperial Database are considered adequate and reliable.

Figure 12-2: Examples of Onsite Collar Location Verifications

Notes: A) CL19032 collar; B) CL21043 collar; C) CL21048 collar.

Table 12-2: Original Collar Survey Data Compared to InnovExplo's Checks

Hole ID	Original coordinates		InnovExplo coordinates		Difference (m)	
	Easting	Northing	Easting	Northing	Easting	Northing
CL19032	440922	6133684	440923	6133683	-1	1
CL19035	440859	6133724	440860	6133725	-1	-1
CL21043	440869	6133788	440866	6133787	3	1
CL21045	440792	6133712	440791	6133712	1	0
CL21046	440789	6133657	440787	6133656	2	1
CL21047	440760	6133677	440758	6133675	2	2
CL21048	440964	6133780	440964	6133780	0	0
CL21052	440841	6133848	440841	6133846	0	2

12.4.2 Downhole Survey

Downhole surveys were conducted using a Reflex EZCOMII Shot tool. Single-shot survey measurements were taken every 30m as well as at the end of the hole. The downhole survey information was verified for 70% of the holes used in the 2021 MRE.

Minor errors of the type normally encountered in a project database were identified and corrected.

12.4.3 Assays

The author had access to the assay certificates for the 2019-2021 Program. The assays in the database were compared to the original certificates provided by the laboratory. The verified holes represent 70% of the Imperial Database.

Minor errors of the type normally encountered in a project database were identified and corrected.

12.4.4 Conclusions

The author believes that the data verification process demonstrates the validity of the data and the protocols for the Project. The author considers the database for the Project to be valid and of sufficient quality to be used for the mineral resource estimate herein.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The process flowsheet developed for the Crater Lake Scandium Project is based on metallurgical development programs completed at various laboratories between 2018 and 2022. The process flowsheet consists of crushing and milling, magnetic separation, high-pressure caustic leach, followed by hydrochloric acid leach of the caustic leach residue, solvent extraction and co-electrowinning of Al and Sc from alumina (Al_2O_3) and scandium oxide (Sc_2O_3) to produce Al-2%Sc master alloy. A mixed REE hydroxide will be produced as a co-product.

Mineral processing and metallurgical testing were performed to evaluate the potential to recover scandium and rare earth elements from the deposit. The scandium mineral concentrate beneficiation testwork and the hydrometallurgical development programs are presented below.

13.1 Introduction

Between 2018 and 2022, metallurgical testing programs focused on the development of an extraction flowsheet for the Crater Lake Sc/REE mineralized samples were completed:

- Phase 1 Metallurgical Development Program:
  - Mineralogy and scoping-level evaluation of several mineral processing technologies for producing a Sc/REE mineral concentrate was completed at SGS Mineral Services ("SGS"), Lakefield, Ontario, Canada in 2018 and 2019.
- Phase 2 Metallurgical Development Program:

  Following Phase 1 scoping level physical separation program, SGS continued the physical separation testwork on the Master Composite in Phase 2 Metallurgical Development Program from October 2018 to March 2019. The Phase 2 program included magnetic separation, and gravity separation. The goal was to build on the results of the Phase 1 test program to develop a preliminary flowsheet for scandium beneficiation, with a particular focus on the optimization of magnetic separation.
- Mineralogy and bench-scale mineral processing program focused on the magnetic separation and flotation of two drill core bulk MET1 and MET2 samples from the TG Zone to generate a Sc/REE mineral concentrate was coordinated by MPlan International (MPlan) and the testwork was performed at ANZAPLAN, Germany in 2019 and 2020. MPlan, based in Toronto, Ontario, is a joint venture of the metallurgical service group Dorfner ANZAPLAN GmbH (Germany) and the mining consultancy Micon International Limited.
- Phase 3 Metallurgical Development Program:
  - Hydrometallurgical program focused on developing a Sc/REE extraction flowsheet from the Crater Lake mineral concentrate generated from Phase 2 Development Program. The Phase 3 hydrometallurgical program was coordinated by MPlan and completed at ANZAPLAN in Germany (2020 to 2021).

The Tables and Figures in this Item are extracted from the above-mentioned reports.

The results of the various metallurgical flowsheet development programs are summarized in several reports.

WSP was not involved with supervision or coordination of any of the testwork programs. WSP assumes that the information in the reports is reasonable and accurate.

13.2 Sample Representativeness

The following samples from the Crater Lake Scandium/REE deposit were used for the metallurgical development programs:

- Master Composite which consists of core samples from Quest Rare Mineral's 2014 drilling program was used for the mineralogical investigation and all the mineral processing test programs completed at SGS, Lakefield, Ontario.
- MET1 sample which represents a pyroxene-rich ferro-syenite with decimetric to metric clasts of pegmatitic or coarse-grained felsic syenite.
- MET2 sample which is an olivine-rich ferro-syenite with several centimetric clasts of fine-grained felsic syenite.

MET1 and MET2 samples are two (2) drill core bulk samples collected from the TG Zone during the 2019 Imperial Mining Group's drilling program. MET1 and MET2 which represent different mineralization types encountered in the zone were used for all metallurgical development programs completed at ANZAPLAN.

WSP was not involved with sample collection.

13.3 Phase 1 Metallurgical Development Program – SGS, Lakefield, Ontario

A mineralogical investigation was conducted on 20 kg of drill core samples from Quest Rare Mineral's 2014 drilling program (the "Master Composite") and several core pieces at SGS Lakefield from May to July 2018. The program was conducted in conjunction with a scoping-level mineral beneficiation test program that was also performed at SGS.

The results of the test programs are summarized in the following two reports:

- "An Investigation into A MINERALOGICAL INVESTIGATION OF ONE COMPOSITE SAMPLE AND SEVERAL CORE PIECES FOR THE OCCURENCE OF SCANDIUM FROM THE CRATER LAKE DEPOSIT, SGS Project 16669-001A – Final Report, July 12, 2018".
- "An Investigation into SCOPING LEVEL TESTS FOR THE RECOVERY OF SCANDIUM AND BY-PRODUCTS METALS FROM THE CRATER LAKE PROJECT, SGS Project 16669-001A – Final Report, February 26, 2019".

13.3.1 SGS Mineralogical Study

The main objective of the mineralogical investigation was to determine the overall mineral assemblage, the elemental deportment of the scandium, and for the Master Composite, determine the liberation characteristics of the various minerals phases to aid with the mineral beneficiation program.

The mineralogical test program was based on QEMSCAN, X-Ray Diffraction (XRD) and electron microprobe analysis (EMPA). The 20 kg drill core samples were crushed to 100% passing 10 mesh (1.7 mm), blended, and homogenized to form a Master Composite. A 1 kg split of the Master Composite was taken for mineralogical analysis, while the remaining sample was used for the scoping level physical separation programs.

A portion of the 1 kg sample for mineralogical analysis was stage ground to a P_{80} of 0.60 mm and subjected to a full screen analysis using Tyler screens. The grounded sample was classified into four size fractions: +0.6 mm, −0.6+0.425 mm, −0.425+0.15 mm, and −0.15 mm. Subsamples of each fraction were submitted for chemical analyses. The analyses included whole rock analysis (WRA) for the major elements including silica, aluminum, iron, magnesium, calcium, potassium, titanium, manganese, chromium, phosphorus, and scandium by X-ray Fluorescence (XRF).

The chemical analysis of size fractions along with the weight and elemental distribution for selected elements are summarized in Table 13-1 and Table 13-2. The data illustrates that the scandium concentration is consistently at 0.02%, and weight distributions are relatively even in each size fraction.

Table 13-1: Assay Grades for Selected Size Fractions of Crater Lake Master Composite Sample

Sample ID	Wt%	Assay, %										
		SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	ZrO_2	Sc
+0.600 mm	24.5	41.6	4.3	36.3	1.6	10.3	1.6	1.3	2.1	0.4	0.78	0.02
−0.600+0.425 mm	19.8	40.3	3.9	38.5	1.6	10.0	1.5	1.2	2.3	0.3	0.72	0.02
−0.425+0.150 mm	29.3	39.6	4.1	38.4	1.5	9.6	1.5	1.2	2.6	0.3	0.84	0.02
−0.150 mm	26.4	38.0	4.1	36.0	1.5	9.7	1.5	1.2	2.8	0.9	2.31	0.02
Head (Calc)	**100**	**39.8**	**4.11**	**37.3**	**1.52**	**9.89**	**1.52**	**1.22**	**2.47**	**0.48**	**1.19**	**0.02**

Table 13-2: Mass and Elemental Distributions for Selected Size Fractions of Crater Lake Master Composite

Sample ID	Wt%	Distribution, %										
		SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	ZrO_2	Sc
+0.600 mm	24.5	25.6	25.8	23.9	25.6	25.6	25.7	26.2	20.4	18.9	16.08	24.5
−0.600+0.425 mm	19.8	20.0	18.6	20.4	20.1	20.0	19.0	18.7	18.0	14.0	11.96	19.8
−0.425+0.150 mm	29.3	29.1	29.2	30.1	28.8	28.5	28.7	29.6	31.2	19.4	20.66	29.3
−0.150 mm	26.4	25.2	26.4	25.5	25.5	26.0	26.5	25.4	30.4	47.7	51.3	26.4
Head (Calc)	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**	**100**

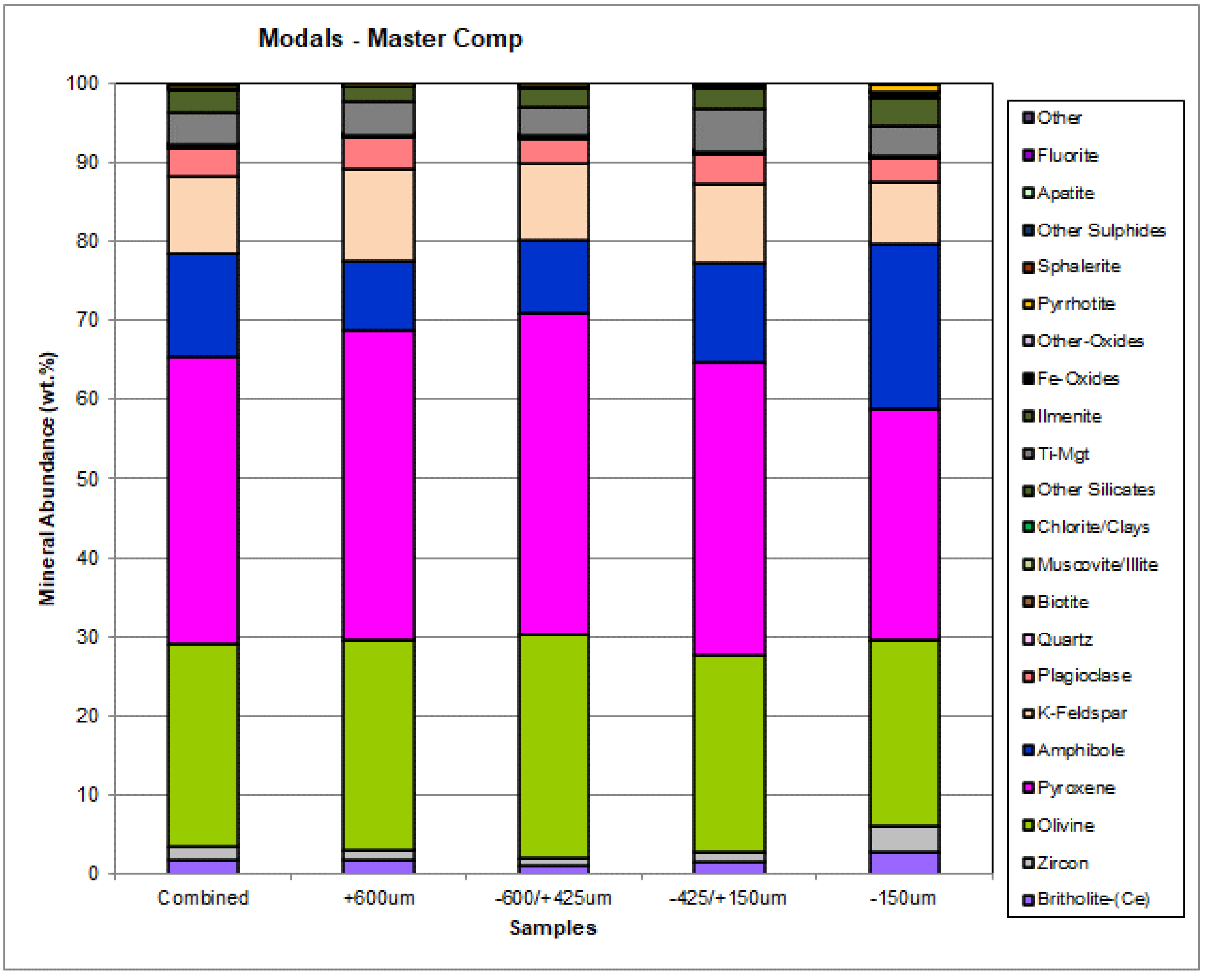
Subsamples of each fraction were taken to prepare graphite-impregnated polished epoxy grain mounts for mineralogical analyses using the QEMSCAN Particle Mineral Analysis (PMA). Additionally, an aliquot from the Master Composite was also submitted for qualitative XRD analysis to aid with the QEMSCAN investigation. EMPA was completed on polished sections of the sample at McGill University, Montreal, Quebec.

Mineral Assemblage and Distribution

The mineralogical study showed that the Master Composite sample consists mainly of pyroxene with moderate levels of olivine and amphibole. Minerals in minor abundance include plagioclase, micas, potassium feldspars, and ilmenite.

The modal mineralogy showing mineral assemblage and abundance in the Master Composite sample is presented in Figure 13-1.

Figure 13-1: Modal Mineralogy for Crater Lake Master Composite Sample by Size Fractions



The mineral distribution based on QEMSCAN and EMPA is presented in Table 13-3. The EMPA and QEMSCAN deportment data revealed that about 82% of the scandium is hosted primarily by pyroxene and to a lesser extent (18%) in the amphiboles. The scandium grade in pyroxene and amphibole is 0.05% and 0.03% respectively while iron concentrations between the various silicates and oxides are variable. The iron distribution illustrates that approximately 52.2% of iron is hosted by olivine, 29.3% by the pyroxene, 13.1% in the amphiboles, 4.2% from the Ti-magnetite, and 2.7% from the ilmenite.

Table 13-3: Master Composite Mineral Distribution

Mineral	Percentage	Comments
Pyroxene	36.2	Contains 0.05% Sc
Olivine	25.6	
Amphibole	13.1	Contains 0.03 % Sc
K-feldspar	9.7	
Ti-magnetite	4.3	Contains 0.02% Nb
Ilmenite	2.7	Contains 0.39% Nb
Britholite	1.8	
Zircon	1.8	

The EMPA showed that 100% of the scandium is hosted in two paramagnetic minerals (pyroxene and amphibole) which should be amenable to magnetic separation technique in producing a scandium mineral concentrate.

Mineral Liberation and Association

The liberation and association characteristics of pyroxene, amphibole, olivine feldspars (K-feldspar, and plagioclase combined), Ti-oxides (Ti-magnetite and ilmenite combined), britholite-(Ce), and zircon were examined for the four different size fractions using QEMSCAN analysis. A particle is classified as free if more than or equal to 95% of its area is of a mineral of interest is exposed. A particle is deemed to be liberated if more than or equal to 80% of its total area represents a particular mineral. The non-liberated particles are classified according to association characteristics, where binary association groups refer to particle area percent greater than or equal to 95% of the two minerals or mineral groups. The complex groups refer to particles with ternary, quaternary, and greater mineral associations including the mineral of interest.

The liberation of the main minerals is not very good at the target grind size of P_{80} of 0.6 mm, except for the feldspars with about 80% as free and liberated combined.

In order to achieve more than 80% liberation of the Sc-bearing minerals, the target grind size should be at least P_{80} of 0.15 mm.

13.3.2 Phase 1 SGS Physical Separation Scoping Level Testwork

Under the Phase 1 Metallurgical Development Program, SGS evaluated a series of scoping-level mineral processing techniques, including Davis Tube low-intensity magnetic separation ("LIMS"), Mozley Table (gravity separation), and wet high-intensity magnetic separation ("WHIMS") to identify processes that can be used to produce a Sc/REE mineral concentrate.

A number of small test charges from the Master Composite sample were stage ground to different particles sizes for the physical separation test program. The grind sizes initially investigated included 100% passing 20 mesh (0.850 mm), 35 mesh (0.500 mm), and 48 mesh (0.300 mm). The ground material was subjected to tests involving Davis Tube (low-intensity magnetic separation, LIMS), Mozley Table (gravity separation), and wet high-intensity magnetic separation (WHIMS).

SGS came to the following conclusions based on the results of the scoping level physical separation tests:

- Among the grind sizes of 100% passing 20 mesh (0.850 mm), 35 mesh (0.500 mm), 48 mesh (0.300 mm), and 150 mesh (0.105 mm), the finest grind size of 0.105 mm was found to perform best for the evaluated mineral processing techniques.
- Magnetic separation was identified as the primary process technology for the recovery and concentration of scandium.
- The Davis Tube was found to recover a fairly small amount of ferromagnetic material, likely titanomagnetite, achieving a titanium recovery of 40.8% with only 3.9% of the scandium.
- WHIMS recovered the scandium-bearing minerals, pyroxene and amphiboles, but with poor selectivity versus olivine in the ore. The finest grind that was tested (0.105 mm) attained a scandium recovery of 78% in 57% of the mass.
- Gravity separation on a Mozley Table recovered the majority of the zirconium (69%) in the sample but achieved poor selectivity. A substantial amount of scandium (26%) was lost to the gravity concentrate.

Further testing of low-intensity magnetic separation (LIMS) together with wet high-intensity magnetic separation (WHIMS) was recommended by SGS.

13.4 Phase 2 Metallurgical Development Program

In Phase 2 of the development program, further evaluation of mineral separation techniques and development of the physical separation flowsheet were completed at SGS, Lakefield, Ontario, Canada and ANZAPLAN, Hirschau, Germany.

13.4.1 Phase 2 SGS Physical Processing Program

Following Phase 1 scoping level physical separation program, SGS continued the physical separation testwork on the Master Composite in Phase 2 Metallurgical Development Program from October 2018 to March 2019. The Phase 2 program included magnetic separation, and gravity separation. The goal was to build on the results of the Phase 1 test program to develop a preliminary flowsheet for scandium beneficiation, with a particular focus on the optimization of magnetic separation.

The target for the concentrate produced was the recovery of greater than 80% of the scandium in less than 50% of the feed mass. A series of magnetic separation tests was conducted as part of the program, along with a handful of gravity separation tests on a Wilfley Table and a Mozley Table.

The mineral processing development work completed at SGS under the Phase 2 Metallurgical Program is summarized in the following report:

- "AN INVESTIGATION INTO THE MAGNETIC SEPARATION OF SAMPLE FROM THE CRATER LAKE PROJECT, SGS Project 16669-02 – Final Report, March 7, 2019".

Low intensity Magnetic Separation Testwork

SGS performed magnetic separation testwork based on the mineralogical analyses and results of Phase 1 scoping level physical separation tests which indicated that magnetic separation was the preferred method for scandium beneficiation. The results of the initial testwork indicated that finer grinding (to 100% passing 0.105 mm) led to improved scandium beneficiation performance in magnetic separation tests. The sample mineralogy suggested that this improved performance was due to an increase in pyroxene and amphibole liberation at the finer grind size. The objective of the program was to selectively recover the Sc-bearing pyroxene and amphiboles into a mineral concentrate.

Davis Tube Tests

Three Davis Tube tests were conducted as part of the current test program to evaluate the impact of finer feed particle sizes. The grind sizes tested were −48 mesh, −65 mesh, and −100 mesh. Three (3) subsamples of the Master Composite, weighing about 200 g each, were taken and stage-pulverized to the target grind size prior to Davis Tube testing. From each subsample, 20 g was split for the Davis Tube test, while the remainder was submitted for particle size analysis.

The Davis Tube tests were conducted at standard conditions, designed to mimic the performance of low intensity magnetic separation (LIMS) operating at 800 Gauss.

The assays for selected size fractions and elemental distribution of the Davis Tube tests are summarized in Table 13-4 and Table 13-5.

Based on the paramagnetic nature of amphibole and pyroxene (the main hosts of scandium), the Davis Tube tests were not expected to recover scandium, but rather to provide an indication of the possibility of rejecting highly magnetic impurity minerals (such as titanomagnetite) through LIMS.

Scandium losses to the Davis Tube magnetic concentrate were fairly low in each of the three tests, decreasing from 3.6% in the coarsest -48 mesh feed to 0.8% in the finest -100 mesh feed. The decrease in scandium losses was a function of both decreasing mass yield to the magnetic concentrate with finer particle size, as well as the decreasing grade of scandium in the magnetic concentrate with finer particle size.

Table 13-4: Phase 2 SGS Davis Tube Assays for Selected Size Fractions

Test	Grind Size K_{80}	Product	Wt %	Assays, %; g/t, Sc							
				Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O
D-1 −0.30 mm	0.218 mm	Mags	**8.6**	89	11.6	15.6	1.40	64.0	0.67	4.57	0.53
		Non-Mags	91.4	223	1.52	42.0	4.30	34.6	1.61	10.3	1.57
		Head (Calc.)	100	211	2.4	39.7	4.05	37.13	1.53	9.81	1.48
D-2 −0.21 mm	0.147 mm	Mags	**7.1**	62							
		Non-Mags	92.9	225	1.57	42.2	4.32	34.6	1.67	10.3	1.62
		Head (Calc.)	100	213							
D-3 −0.15 mm	0.101 mm	Mags	**5.8**	31	14.5	6.43	0.68	75.6	0.29	1.77	0.24
		Non-Mags	94.2	228	1.67	41.6	4.13	34.9	1.65	10.3	1.58
		Head (Calc.)	100	217	2.42	39.5	3.93	37.3	1.57	9.80	1.50

Table 13-5: Phase 2 SGS Davis Tube Elemental Distribution for Selected Size Fractions

Test	Grind Size K80	Product	Wt %	Distribution, %							
				Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O
D-1 −0.30 mm	0.218 mm	Mags	**8.6**	3.6	41.8	3.4	3.0	14.8	3.8	4.0	3.1
		Non-Mags	91.4	96.4	58.2	96.6	97.0	85.2	96.2	96.0	96.9
		Head (Calc.)	100	100	100	100	100	100	100	100	100
D-2 −0.21 mm	0.147 mm	Mags	**7.1**	2.0							
		Non-Mags	92.9	98.0							
		Head (Calc.)	100	100							
D-3 −0.15 mm	0.101 mm	Mags	**5.8**	0.8	35.0	0.9	1.0	11.8	1.1	1.1	0.9
		Non-Mags	94.2	99.2	65.0	99.1	99.0	88.2	98.9	98.9	99.1
		Head (Calc.)	100	100	100	100	100	100	100	100	100

Whole Rock Analysis (WRA) results were not available for the D-2 test (-0.21 mm) magnetic concentrate. However, a number of conclusions was drawn based on the WRA results of D-1 and D-3.

— A significant amount of titanium reported to the magnetic concentrate in both of these tests, though the proportion of titanium rejected decreased from ≈42% in D-1 (-0.30 mm) to ≈35% in D-3 (-0.15 mm).

— Selectivity in the Davis Tube was observed to increase with decreasing particle size, based on the higher titanium and iron grades in the D-3 magnetic concentrate than in the D-1 magnetic concentrate.

Based on the titanium deportment study that was completed as part of the mineralogical analysis on the sample in the -0.150 mm fraction, the titanium was found to be present mainly in ilmenite (57.7%) and titano-magnetite (16.4%), with the majority of the remainder present in amphibole and pyroxene. The deportment of 35% of the feed titanium to the D-3 magnetic concentrate indicates that the Davis Tube was successful in rejecting most of the titano-magnetite, as well as other titanium-bearing minerals. Based on the higher nominal magnetic susceptibility of ilmenite as compared to amphibole and pyroxene, it is expected that a proportion of the ilmenite present in the feed reported to the Davis Tube magnetic concentrate.

Based on the low scandium loss (0.8%) to the D-3 Davis Tube magnetic concentrate, a feed grind size of 100% passing 0.150 mm was selected for the LIMS tests.

Low Intensity Magnetic Separation (LIMS)

Based on the results of the Davis Tube tests, about 14 kg sample of the Master Composite was stage ground to 100% passing 100 mesh (0.150 mm). The ground sample was then processed though an Eriez L-8 low intensity magnetic separator (LIMS) in two passes, the first pass at 800 Gauss and the second pass at 1000 Gauss. The magnetic concentrate from each pass and the non-magnetic product from the second pass were collected, filtered, dried, and subsamples of each were submitted for scandium and WRA.

The results of the LIMS test are presented in Table 13-6 to Table 13-9. In general, the results of the LIMS test were similar to the results of the D-3 David Tube test at the same grind size. Titanium and iron rejection at around 35% and 12%, respectively to the LIMS combined magnetic concentrate were almost identical to those in the D-3 magnetic concentrate. Scandium losses to the LIMS combined magnetic concentrate were slightly higher (1.5%), due to slight increase in the concentrate mass yield and scandium grade.

The results indicated that LIMS can be used as an important pre-concentration step in the Crater Lake scandium beneficiation flowsheet to reject highly magnetic impurity minerals with minimal scandium losses.

Table 13-6: Phase 2 SGS LIMS Test Product Assays

Product	Weight		Assays, %; g/t						
	g	%	Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO
1st Pass Mags	858.0	6.4	44	13.0	8.07	0.95	70.7	0.34	2.17
2nd Pass Mags	47.3	0.4	140	6.25	25.2	2.57	48.1	0.99	6.45
Non-Mags	12579.1	93.3	229	1.63	42.4	4.31	34.3	1.57	10.2
Head (Calculated)	13484.4	100	217	2.37	40.2	4.09	36.7	1.49	9.68
Head (Analysis)			205	2.40	40.0	4.10	37.2	1.59	9.79

Table 13-7: Phase 2 SGS LIMS Test Elemental Distribution

Product	Weight	Distributions, %						
	%	Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO
1st Pass Mags	6.4	1.3	34.9	1.3	1.5	12.3	1.5	1.4
2nd Pass Mags	0.4	0.2	0.9	0.2	0.2	0.5	0.2	0.2
Non-Mags	93.3	98.5	64.2	98.5	98.3	87.3	98.3	98.3
Head (Calculated)	100	100	100	100	100	100	100	100

Table 13-8: Phase 2 SGS LIMS Combined Product Assays

Product	Weight		Assays, %						
	g	%	Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO
1st Pass Mags	858	6.4	44	13.0	8.07	0.95	70.7	0.34	2.17
1st & 2nd Pass Mags	905	6.7	49	12.6	8.97	1.03	69.5	0.37	2.39
Non-Mags	12579	93.3	229	1.63	42.4	4.31	34.3	1.57	10.2

Table 13-9: Phase 2 SGS LIMS Combined Product Elemental Distribution

Product	Weight		Distributions, %						
	g	%	Sc	TiO_2	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO
1st Pass Mags	858	6.4	1.3	34.9	1.3	1.5	12.3	1.5	1.4
1st & 2nd Pass Mags	905	6.7	1.5	35.8	1.5	1.7	12.7	1.7	1.7
Non-Mags	12579	93.3	98.5	64.2	98.5	98.3	87.3	98.3	98.3

Wet High Intensity Magnetic Separation (WHIMS)

A series of wet high intensity magnetic separation tests were completed to investigate the potential to separate the scandium-bearing amphibole and pyroxene from the major gangue minerals (ilmenite, feldspars, and olivine) remaining in the LIMS non-magnetic product. An Outotec SLon-100 laboratory scale Vertically Pulsating High Gradient Magnetic Separator (VPHGMS) was used for the WHIMS testwork.

Based on nominal magnetic susceptibility ranges of the minerals present in the LIMS non-magnetic product, it was believed that the ilmenite and olivine (fayalite) may be captured into a separate magnetic concentrate at a lower magnetic intensity than amphibole and pyroxene. However, magnetic susceptibilities vary between each of the minerals in the pyroxene and amphibole groups; the significant iron content of the pyroxenes and amphiboles in the Master Composite (as identified by the Phase I mineralogy study) may result in the magnetic susceptibilities of these minerals being too similar to those of ilmenite and olivine (fayalite) for effective separation.

Three tests were completed on SLon-100 to evaluate the performance of WHIMS at different grind sizes of −100 mesh (−0.15 mm), −150 mesh (−0.106 mm), and −200 mesh (−0.075 mm) in tests SLon #1, SLon #2, and SLon #3, respectively. Each of these tests included 5 passes through the SLon-100, with the first pass at a magnetic intensity of 2000 Gauss. The magnetic intensity was increased by 2000 Gauss in each subsequent pass up to 10000 Gauss in the fifth pass. The non-magnetic product from each pass served as the feed to the subsequent pass at a higher magnetic intensity. At the conclusion of each test, the five magnetic concentrates and non-magnetic product were filtered, dried, and submitted for scandium and WRA.

The results of the three SLon tests are presented in Table 13-10. The tests showed fair upgrade of the scandium in the feed, primarily in the magnetic concentrates generated at 6,000 Gauss and 8,000 Gauss. The results showed that the combined 4,000 Gauss – 8,000 Gauss magnetic concentrate achieved the highest scandium grade with stage-recoveries of over 80%.

The best results particularly in terms of scandium grade, were obtained from test SLon #1. The SLon #1 combined 4000 Gauss to 8000 Gauss magnetic concentrate graded 282 g/t Sc with 84.9% scandium stage-recovery in 70.5% of the feed mass. Scandium grade in the combined 4000 Gauss to 8000 Gauss magnetic concentrate was observed to decrease with decreasing particle size from SLon #1 to SLon #3.

Tests SLon #1 and SLon #2 delivered a non-magnetic product containing very little scandium (below the detection limit of 25 g/t). Visually, this non-magnetic product was observed to be a lighter brown, sandy colour as opposed to the black-coloured magnetic concentrates. Scandium losses to the non-magnetic product were 1 to 1.5% in 10 to 15% of the feed mass. Again, better performance was observed at the coarser grind size in test SLon #1.

While the results of these tests fell short of the project targets, they did provide an indication of the optimum feed grind size (−100 mesh) for scandium beneficiation in WHIMS.

Table 13-10: SLon #1 to 3 WHIMS Test Results

Test	Grind Size	Product	Mass	Assay, % (Sc g/t)				Distribution, %			
			%	Sc	SiO_2	Fe_2O_3	TiO_2	Sc	SiO_2	Fe_2O_3	TiO_2
Slon #1	−100 mesh	2,000 Gauss Mags	7.1	172	33.1	48.8	3.99	5.2	5.6	10.5	19.4
		2,000 - 4000 Mags	18.4	220	36.0	45.5	2.98	17.2	15.7	25.3	37.6
		2,000 - 6,000 Mags	48.1	261	38.3	41.1	2.08	53.6	43.7	59.7	68.5
		2,000 - 8,000 Mags	77.6	272	39.4	38.6	1.74	90.1	72.3	90.5	92.4
		2,000 - 10,000 Mags	85.5	270	39.6	37.8	1.67	98.5	80.1	97.8	97.8
		4,000 - 8,000 Mags	**70.5**	**282**	**40.0**	**37.5**	**1.51**	**84.9**	**66.7**	**80.0**	**73.0**
		Non-Mags	14.5	25	58.0	4.94	0.22	1.5	19.9	2.2	2.2
Slon #2	−150 mesh	2,000 Gauss Mags	8.0	186	34.3	48.0	3.55	6.2	6.7	10.7	16.5
		2,000 - 4000 Mags	24.0	221	36.4	45.3	2.94	22.2	21.4	30.2	41.0
		2,000 - 6,000 Mags	53.2	252	38.0	41.9	2.28	56.1	49.5	61.8	70.6
		2,000 - 8,000 Mags	81.9	263	38.8	40.2	1.97	89.9	77.6	91.2	93.6
		2,000 - 10,000 Mags	90.1	263	38.9	39.7	1.90	99.0	85.7	99.1	99.5
		4,000 - 8,000 Mags	**73.9**	**271**	**39.2**	**39.3**	**1.79**	**83.7**	**70.9**	**80.5**	**77.1**
		Non-Mags	9.9	25	59.2	3.2	0.09	1.0	14.3	0.9	0.5
Slon #3	−200 mesh	2,000 Gauss Mags	6.0	181	34.0	49.1	3.51	4.7	5.1	7.8	10.9
		2,000 - 4000 Mags	29.1	214	36.5	46.1	2.96	26.8	26.5	35.2	44.4
		2,000 - 6,000 Mags	61.7	238	37.6	43.5	2.45	63.4	58.0	70.5	78.0
		2,000 - 8,000 Mags	81.8	250	38.3	41.8	2.18	88.2	78.3	89.7	91.9
		2,000 - 10,000 Mags	86.5	251	38.4	41.4	2.13	93.8	83.0	94.1	94.9
		4,000 - 8,000 Mags	**75.8**	**255**	**38.6**	**41.2**	**2.07**	**83.5**	**73.2**	**82.0**	**81.0**
		Non-Mags	13.5	107	50.5	16.7	0.74	6.2	17.0	5.9	5.1

The global LIMS + WHIMS result for SLon #1, the WHIMS test that showed the best scandium upgrade is presented in Table 13-11. The combined SLon #1 4000 Gauss to 8000 Gauss magnetic concentrate recovered 83.6% of the feed scandium in 65.7% of the feed mass, and graded 282 g/t Sc. The addition of the 10000 Gauss magnetic concentrate led to a slight decrease in scandium grade to 279 g/t Sc, but increased scandium recovery to 91.8% in 73.1% of the feed mass.

A large proportion of the titanium (≈39%) was rejected to the combined LIMS magnetic concentrate, with an additional 12% reporting to the SLon #1 2000 Gauss magnetic concentrate. The SLon #1 non-magnetic product consisted primarily of feldspars, with less than 25 g/t Sc.

Table 13-11: LIMS - WHIMS Global Metallurgical Balance and Combined Products - SLon #1 Test

	Mass	Assay, % (Sc g/t)				Distribution, %			
	%	Sc	SiO_2	Fe_2O_3	TiO_2	Sc	SiO_2	Fe_2O_3	TiO_2
Products									
LIMS 800 Gauss Mags	6.4	44	8.07	70.7	13	1.3	1.3	12.7	37.4
LIMS 1000 Gauss Mags	0.4	140	25.2	48.1	6.25	0.2	0.2	0.5	1.0
SLon #1 2000 Gauss Mags	6.6	172	33.1	48.8	3.99	5.1	5.5	9.1	11.9
SLon #1 4000 Gauss Mags	10.5	250	37.8	43.5	2.35	11.9	9.9	12.9	11.2
SLon #1 6000 Gauss Mags	27.7	287	39.8	38.3	1.52	35.9	27.6	29.9	19.1
Slon #1 8000 Gauss Mags	27.5	290	41.0	34.5	1.18	36.0	28.2	26.7	14.7
Slon #1 10000 Gauss Mags	7.4	245	41.7	30.5	1.00	8.2	7.7	6.4	3.4
SLon #1 Non-Mags	13.5	<25	58.0	4.94	0.22	1.5	19.5	1.9	1.3
Head (Calc)	100.0	222	40.0	35.5	2.21	100.0	100.0	100.0	100.0
Combined Products									
LIMS Mag	6.7	50	9.08	69.4	12.6	1.5	1.5	13.3	38.7
SLon #1 2000 Gauss Mag	6.6	172	33.1	48.8	3.99	5.1	5.5	9.1	11.9
SLon #1 4000-8000 Gauss Mag	65.7	282	40.0	37.5	1.51	83.6	65.7	69.4	44.8
SLon #1 4000-10000 Gauss Mag	73.2	279	40.2	36.8	1.46	91.8	73.4	75.8	48.1
SLon #1 Non-Mags	13.5	<25	58.0	4.9	0.22	1.5	19.5	1.9	1.3

Three additional WHIMS tests were completed (SLon #4 to #6) to evaluate the impact of SLon-100 operating parameters on WHIMS selectivity for recovering scandium. The key parameters considered were the magnetic matrix size, the wash water flow rate, the pulsation rate, and the feed density.

Additional passes at different magnetic intensities were also added to further fractionate the magnetic particles collected.

The test results showed that variation in SLon-100 operating parameters had a significant impact on the results of the WHIMS tests, most noticeably in terms of the mass yield to the magnetic concentrate at each intensity.

The mineralogical study indicated that the scandium is hosted primarily by the pyroxene and to a lesser extent the amphiboles. In an ideal scenario, the magnetic separation could concentrate the high iron minerals (i.e., olivine) using a lower magnetic intensity, while the other iron bearing minerals (i.e. pyroxene and amphiboles) would report to a higher magnetic intensity product. However, the mineral distributions for each respective test product from one of the WHIMS tests showed that there is poor selectivity for the pyroxene/amphiboles against the olivine at the different magnetic intensities tested.

SGS Gravity Separation Testwork

One Mozley Table and one Wilfley Table tests were completed to simulate gravity separation. A Mozley Table is used as gravity separator to simulate spirals concentrators. The Mozley Table is considered a low G separator.

A Holman-Wilfley 800 laboratory-scale unit was used for the Wilfley test, with five launders positioned in such a way as to collect four concentrates and one tailings product. The test consisted of a single pass over the table. The Mozley Table was operated with the flat deck which is designed for the separation of -100 µm fines. The goal of this test was the stage-rejection of about 25-30% of the feed mass to the Mozley Table concentrate which was expected to consist primarily of ilmenite, olivine and zircon, with the slightly less dense amphibole, pyroxene, and feldspars reporting to the Mozley Table tailings.

The gravity separation tests did not show significant scandium upgrading. However, the iron and titanium were upgraded in the Mozley Table concentrate, suggesting that significant amounts of olivine and ilmenite did report to this product. But because of the low selectivity and the high scandium losses to the Mozley Table concentrate, it was decided not to proceed with additional magnetic separation testwork on the Mozley Table tailings.

Phase 2 SGS Physical Processing Conclusions and Recommendations

The following conclusions and recommendations were drawn based on the results of the Phase 2 physical processing testwork program completed on sample from the Crater Lake project at SGS:

- The optimum grind size for scandium beneficiation via magnetic separation was determined to be -100 mesh (-0.150 mm), based on the results of Davis Tube tests, LIMS, and WHIMS tests.
- LIMS can be used as an important pre-concentration step to reject the majority of the titano-magnetite in the sample, as well as a proportion of the ilmenite. Scandium losses to the LIMS magnetic concentrate were fairly low (≈1.5%).
- WHIMS tests using an Outotec SLon-100 resulted in a scandium upgrade at certain magnetic intensities. However, the selectivity of the WHIMS for scandium-bearing minerals, against olivine in particular, remained poor.
- The WHIMS operating parameters had a significant impact on the magnetic separation test results, particularly in terms of the mass yield to the magnetic concentrate and magnetic intensity at which different minerals were captured. However, the highest scandium upgrade was consistently achieved in the magnetic concentrate generated at a magnetic intensity of 0.8 Tesla.
- Gravity separation tests using a Wilfley Table and a Mozley Table displayed poor selectivity for scandium. Gravity separation may be a suitable technique to concentrate zircon from the sample.

13.4.2 Phase 2 MPlan Mineralogical and Physical Processing Programs

In continuation of the physical processing flowsheet development for the Project, Imperial Mining Group in September 2019 commissioned MPlan to undertake further metallurgical testwork on the mineralization from the recently discovered TGZ target. MPlan, based in Toronto, Ontario, is a joint venture of the metallurgical service group Dorfner ANZAPLAN GmbH (Germany) and the mining consultancy Micon International Limited.

The testwork was completed on two (2) drill core bulk samples collected from the TG Zone during the 2019 drilling program. The samples represent different mineralization types encountered in the zone, yielding a best assay of 474 g/t scandium oxide (Sc_2O_3) over 12.5 m in an interval grading 341 g/t Sc_2O_3 over 74.9 m.

Two approximately 100-kg of halved drill core samples of MET1 and MET2 from the TG zone, were used for this test program:

- The MET1 sample represents a pyroxene-rich ferro-syenite with decimetric to metric clasts of pegmatitic or coarse-grained felsic syenite.
- The MET2 sample represents an olivine-rich ferro-syenite with several centimetric clasts of fine-grained felsic syenite.

The objective of the current testwork was to reject the major portion of olivine from the samples by testing different separation methods, including magnetic separation, electrostatic separation, and flotation. The testwork program consists of two parts:

— Mineralogical characterization of both (MET1 and MET2) scandium-bearing samples
— Physical processing of the samples, targeting the production of a highly concentrated scandium mineral concentrate.

The testwork results are summarized in the following two reports:

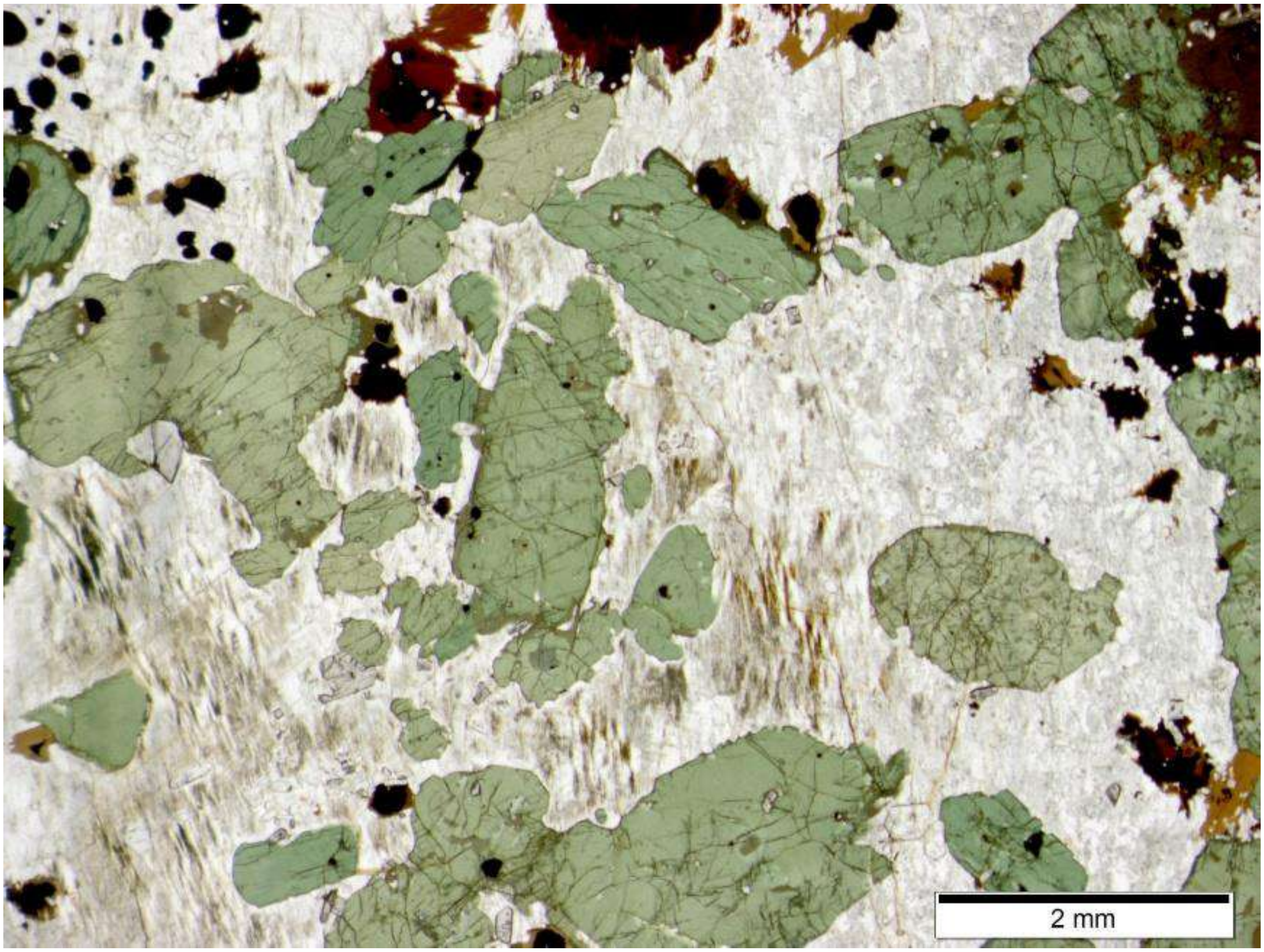
— Physical Processing of Scandium Bearing Mineral Samples Crater Lake Scandium Project Quebec, Canada Testwork Program 1 Report: Mineralogical Characterization of Scandium Bearing Mineral Samples, December 2, 2019; MPR-1955-Rev4
— Physical Processing of Scandium Bearing Mineral Samples Crater Lake Scandium Project Quebec, Canada Testwork Report Part 2: Physical Processing of Scandium Bearing Mineral Samples; February 21, 2020; MPR-1955-Rev4

MPLAN MET1 and MET2 Mineralogical Characterization

Polished sections, about 40 by 25 mm in size and about 25 µm thick, were prepared and petrographically analyzed. The samples were characterized by X-ray diffraction (“XRD”). Mineral liberation analysis (“MLA”), a quantitative analytical technique, was conducted using a scanning electron microscope (“SEM”). This program focused on scandium as well as the total rare earth elements (“TREE”) contained in the deposit. The mineralogical characterization showed that the MET1 sample represents a pyroxene with pegmatitic or felsic syenite with decimetric to metric clasts of pegmatitic or coarse-grained felsic syenite, which is interpreted to be the cumulate variety of the ferro-syenite. MET2 sample represents an olivine-rich ferro-syenite with centimetric clasts of fine-grained felsic syenite. It is interpreted to represent a more evolved variety. The studied thin sections from each sample are quite distinct, documenting the significant mineralogical and textural variability of these rocks. with felsic syenite.

The main mafic phases in both samples are yellow olivine (fayalite), green clinopyroxene (hedenbergite, Figure 13-2), brown and green amphibole (ferropargasite, see Figure 13-3), and opaque titanomagnetite and ilmenite. The main felsic phases are feldspars, micro-perthite and anorthoclase to albite, and especially red-brown biotite (annite) in the more evolved MET2 sample. The minor to accessory igneous and possibly late-magmatic-hydrothermal phases comprise zircon, apatite (always zoned and sometimes with britholite inclusions), various sulphides (mostly pyrrhotite, chalcopyrite, and sphalerite) and minor graphite. Texturally late alteration phases in fractures include calcite, mica, a series of unidentified Fe-rich silicates and a Ca-silicate.

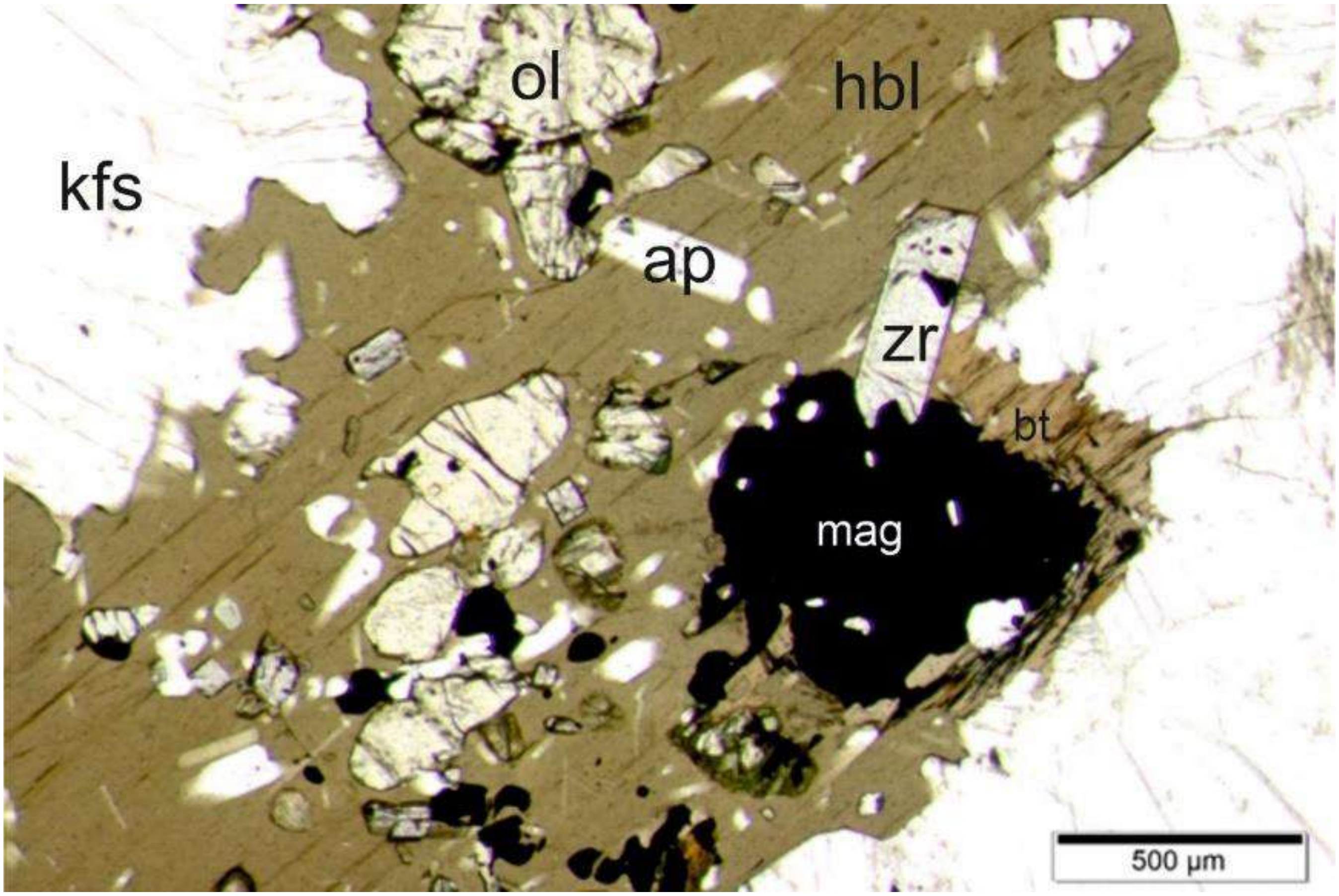
Figure 13-2: Polished Section Showing Anhedral Pleochroic Green Clinopyroxene Crystals in Perthitic K-feldspar



The scandium content of samples MET1 and MET2 was 238 mg/kg and 243 mg/kg, respectively. The total rare earth element (TREE) content in MET1 was 4334 mg/kg and 4030 mg/kg for MET2, respectively. Scandium was depleted in the fine fraction (−0.02 mm) in both samples, while TREEs were enriched in this fraction (Table 13-12).

Similar to the programs completed at SGS on the Master Composite, the XRD analyses for MET1 and MET2 showed the presence of pyroxene (hedenbergite) and amphibole (ferrohornblende/ ferropargasite), which are recognized as the Sc-bearing minerals. Different feldspars, namely albite, microcline and anorthoclase, fayalite, and mica (biotite) were identified as main components. Ilmenite, magnetite, and the REE-bearing phases (hydroxyl-)apatite and zircon were also identified in the samples.

Figure 13-3: Polished Section Showing Poikilitic Amphibole



Note: Includes inclusions of olivine, zircon, apatite and anhedral Ti-magnetite surrounded by aiotite at the contact to K-feldspar.

The two samples were, crushed, stage-milled and classified into to three size fractions, −0.02 mm, −0.10+.0.02 mm and −0.30+0.01 mm. The coarser fraction was used for electrostatic separation, and the finer fractions were used for magnetic, gravity separation, and flotation. The size fractions were assayed, and the results are summarised in Table 13-12.

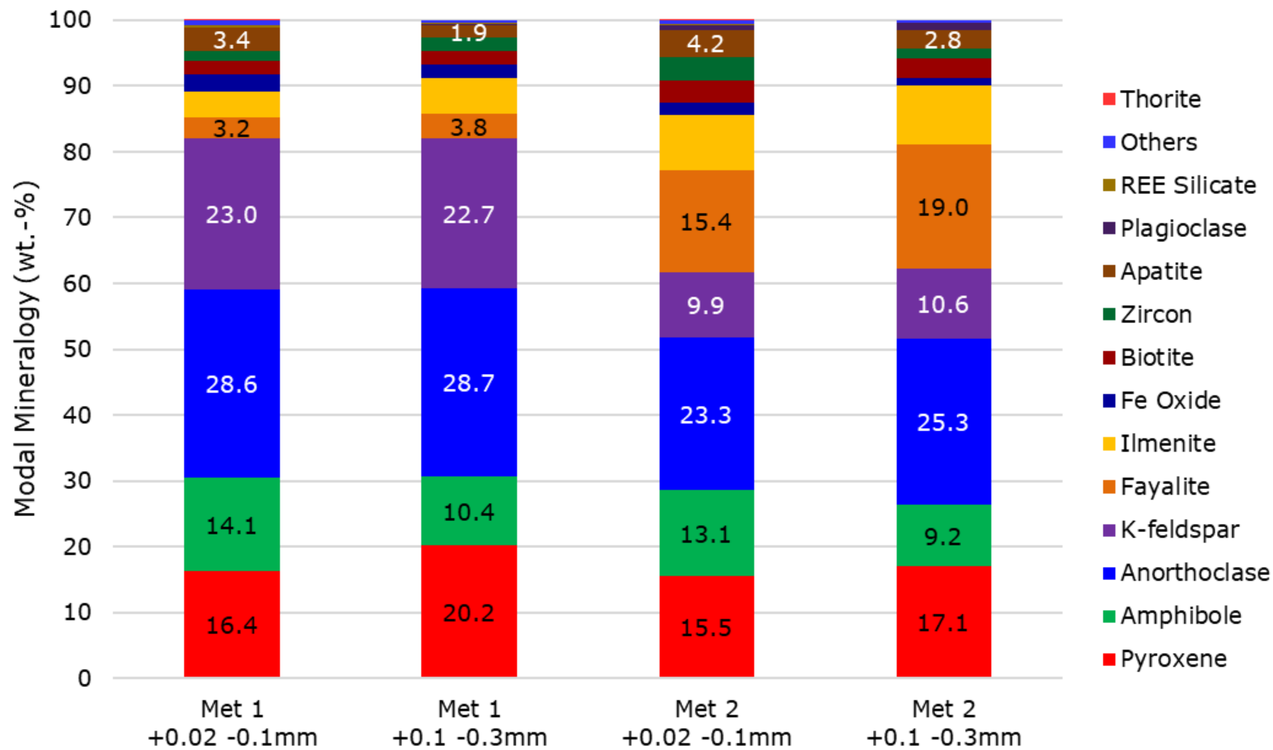
Table 13-12: MET1 and MET2 Chemical Analyses of Selected Elements

Sample	Size Factions	Weight	Chemical Analysis				Distribution, w%		
			Sc	TREE	SiO_2	Fe_2O_3	Sc	TREE	Fe_2O_3
		wt.-%	mg/kg	mg/kg	wt.-%	[wt.-%]	wt.-%	wt.-%	wt.-%
MET1	−0.30+0.10 mm	64.9%	240	3500	51.7	17.0	65.6%	52.5%	63.7%
	−0.10+0.02 mm	32.7%	240	5900	49.5	18.4	33.0%	44.6%	34.7%
	−0.02 mm	2.4%	140	5100	55.0	11.5	1.4%	2.9%	1.6%
	Head (calc.)	100%	238	4334	51.1	17.3	100%	100%	100%
MET2	−0.30+0.10 mm	62.5%	270	3900	44.6	26.8	69.4%	60.7%	63.4%
	−0.10+0.02 mm	35.9%	200	4100	42.4	26.0	29.5%	36.6%	34.7%
	−0.02 mm	1.7%	160	6500	46.6	19.3	1.1%	2.7%	1.8%
	Head (calc.)	100%	243	4031	43.8	26.4	100%	100%	100%

Based on MLA, the modal mineralogy of the samples was determined (Figure 13-4). Scandium-bearing minerals pyroxene and amphibole were found to account for approx. 30 wt. % in all samples, while pyroxene (15 to 20 wt.-%) dominates over amphibole (9 to 14 wt.-%). The most dominant gangue mineral in both MET1 and MET2 samples is anorthoclase (23 to 29 wt.-%), a Na-Ca-rich "ternary" K-feldspar. MET1 was found to contain K-feldspar (23 wt.-%) as the second most dominant phase; in MET2, fayalite (15 and 19 wt.-%), an iron-bearing olivine, is very prominent. It was determined that 82% of the scandium is contained in pyroxene and the remainder in amphiboles.

Mineral locking was determined for MET1 and MET2 size fractions −0.1+0.02 mm and −0.3+0.1 mm. The amount of unlocked minerals was low. In both samples, approximately 65% pyroxene and amphibole in fraction −0.1+0.02 mm and about 40 to 50 wt% in fraction −0.3+0.1 mm were unlocked, with the remainder of scandium-bearing pyroxene and amphibole being mostly intergrown with each other.

Figure 13-4: Modal Mineralogy of Fractions +0.02 -0.1 mm, and +0.1-0.3 mm of MET1 and MET2 Samples



MPLAN MET1 and MET2 Physical Processing Testwork

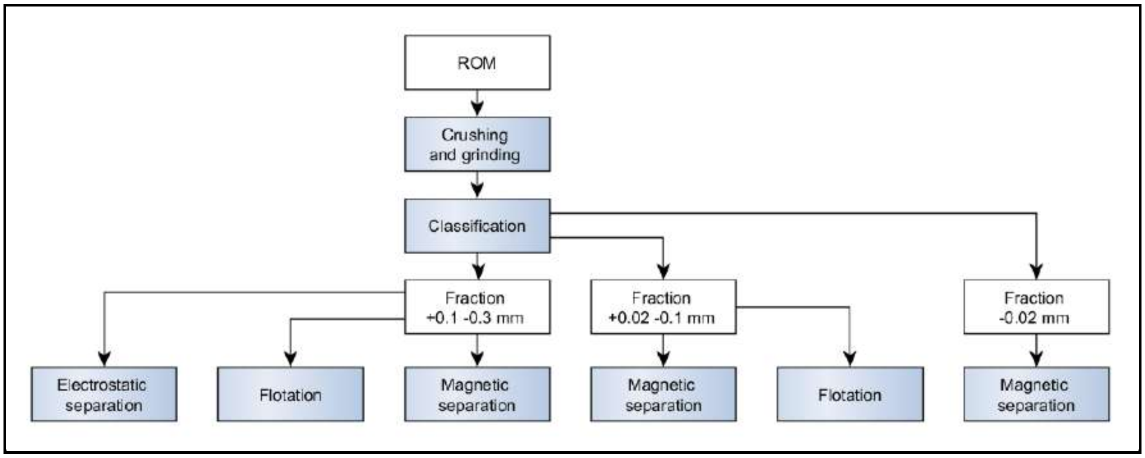
Various mineral processing technologies including sensor-based ore sorting, as well as magnetic, density and electrostatic separation techniques and flotation were evaluated under this program for the production of Sc-REE bearing mineral concentrate from MET1 and MET2 samples.

MET1 and MET2 samples were crushed in a jaw crusher and further grain size reduction of the samples was achieved by a double roll mill. The gap between both rolls was adjusted step by step after every passage of coarse material. The gap width was set between 1 mm at the beginning and 0.2 mm at the end of the comminution sequence. After each passage through the mill, the ground sample was screened to separate the product fractions. The oversized material was added to the next grinding step. Dry and wet screening was applied to classify various particle size fractions. The screening machine was equipped with removable screening decks. Screen cloths made of steel were used. For desliming of fraction −0.1 mm, a 1 inch diameter hydrocyclone was used and for desliming of fraction −0.15 mm, a 50 mm diameter hydrocyclone was deployed.

For sample MET1, beneficiation tests were carried out on different size fractions. Flotation, magnetic separation and electrostatic separation were applied to fraction −0.3+0.1 mm.

The physical processing steps applied to the samples are shown in Figure 13-5. The process was first developed for MET1 and then later adopted to MET2.

Figure 13-5: Separation Methods Applied to Size Fractions −0.3+0.1 mm, −0.1+0.02 mm and −0.02 mm



Electrostatic Separation

Minerals can be separated by electrostatic (high tension) method due to differences in surface conductivity. A pilot plant free fall electrostatic separator was used. The electrostatic field of 70 kV was generated by an electrostatic generator. The feed material was initially heated in some tests to temperatures up to 200°C and dosed with a preheated vibration feeder to feed the electrostatic separator. In one test, the material was activated by heating the sample and by the addition of diluted acid to the feed material prior to heating. In total, six electrostatic tests were completed.

Electrostatic separation was shown to be unselective regarding the scandium bearing minerals (hedenbergite and ferrohornblende).

Heavy Liquid Separation (HLS)

Fraction −6+0.8 mm was used for HLS testing. Prior to HLS tests, the sample material was washed to remove adherent fines to minimize contamination of the heavy liquid solution. HLS testwork was carried out using sodium metatungstate as heavy medium. The liquid SG was adjusted downwards or upwards by diluting with water or by boiling off excess water respectively. The density of the sodium metatungstate solution was adjusted to 2.85 g/cm³.

As a pre-concentration step, HLS was used to check the general suitability of dense media separation in lab scale. HLS was applied to fraction −6+0.8 mm of ore type MET1. For this fraction a scandium recovery of 90.6 wt.-% was achieved at an upgrade factor of 2.1. TREE recovery was 89.2 wt.-% at an upgrade factor of 2.0. Dense media separation (“DMS”) is mainly used as a first concentration method prior to flotation and/or magnetic separation.

Sensor-based Ore Sorting

Twenty-five (25) rock pieces from each ore type (MET1 and MET2) were used for bench scale testing. Each rock sample was assigned an identification number and tested with regards to sensor response in optical (colour) and X-ray transmission (“XRT”) analysis in cooperation with TOMRA Sorting, one of the most advanced providers of sensor-based sorting solutions in the world.

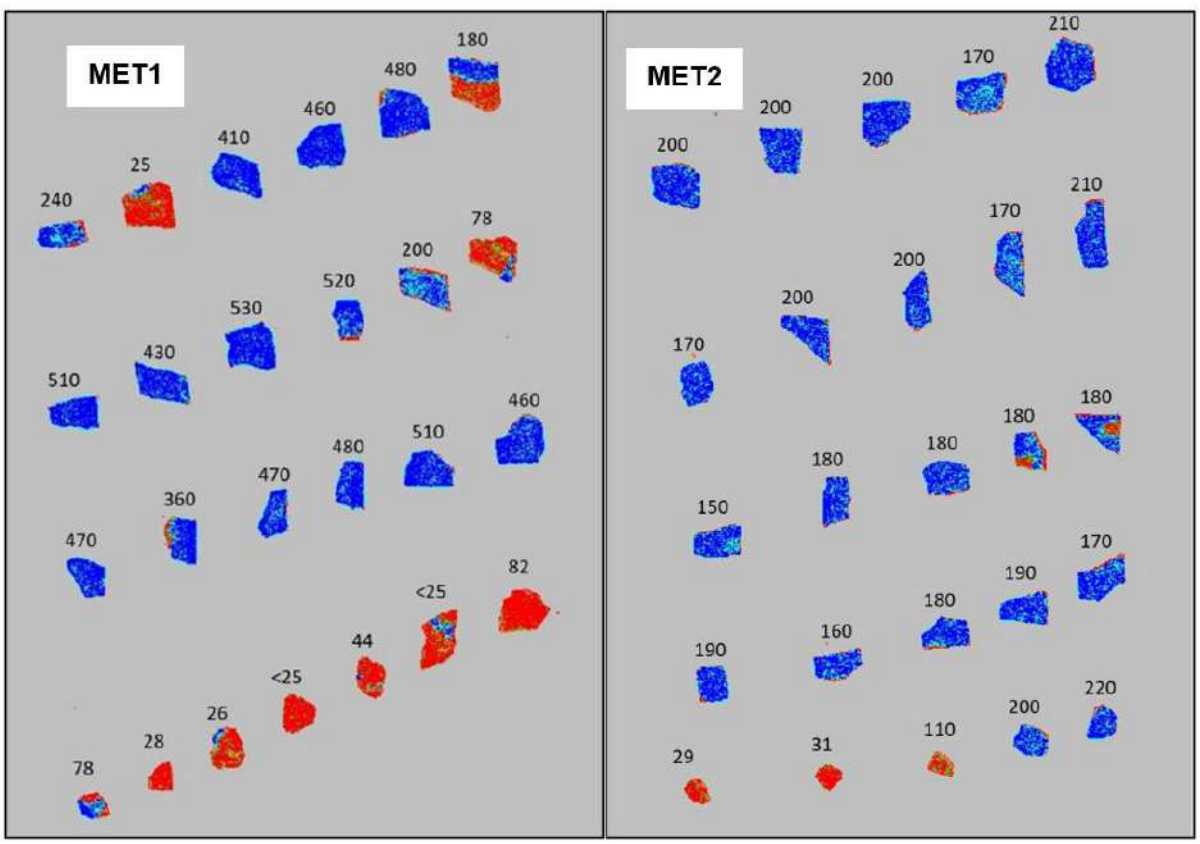
For the colour camera analyses, a high-resolution photo of each sample was taken. The COLOUR image of each sample was then analyzed using TOMRA’s image-processing software. This analyzing technique is restricted to surface features and requires clean surfaces making a washing step prior to colourmetric sorting mandatory. The material was discriminated with reference to brightness, colour and other geometric features such as size and shape.

For the XRT analysis, the samples were exposed to high energy X rays, thus allowing the sensor on the opposite side of the sample to detect transmitted X-rays. The X-ray sensor signal depends on the atomic density and material thickness and yields information about the chemical composition of the particles. This method does not require washing the mineral surface for it to be effective. Using TOMRA's image-processing software, changes in the intensity of X-ray passing through the samples are classified, either as high atomic density or low atomic density.

Good results were achieved with both sensors, however XRT showed an enhanced waste detection and therefore better potential for the scandium ore sorting. For classification, a cut-off grade of 100 mg/kg scandium was applied. By using XRT methodology all 38 product species were classified correctly by the sensor. Only one out of 12 waste pieces was erroneously referred to the product fraction. COLOUR sensor referred all the 38 product samples to the right group but missed 2 waste species recognizing them as product. An additional advantage of XRT sorting is that, in contrast to colourmetric sorting, no water is required to clean the surface of the particles as the bulk of the mineral sample is probed.

In order to present the correlation of XRT sensor response and scandium grade, images visualizing XRT sensor response are presented in Figure 13-6 with scandium grade (in ppm) indicated for the individual rock pieces. Blue colour represents high density, while red colour represents low density. The higher the blue portion in the XRT sensor response, the higher the scandium grade. XRT sorting perhaps has a good potential to be used as a stand-alone processing option or as a pre-concentration step at coarser particle size prior to further comminution and upgrading of the scandium grade by other mineral processing techniques.

Figure 13-6: XRT Inclusion Image of 25 Rock Pieces with Corresponding Scandium Grade in mg/kg



High Gradient Magnetic Separation

A two-step high gradient magnetic separation (HGMS) was conducted on the coarse fraction (−0.3+0.1 mm) of MET1 sample. The magnetic fraction from the first pass was fed to the HGMS a second time to further enrich the concentrate.

The HGMS concentrate collected after two passes, contained 71% of the mass and 96.8% of the scandium as well as 92% of TREE for MET1, while 98.4% of scandium together with 97% of TREE in 87% mass were recovered for MET2. Due to the high mass portion of the HGMS concentrates, the upgrade factors are 1.4 for scandium and 1.3 for TREE in MET1, and 1.15 and 1.13 respectively for scandium and TREE in MET2. The upgrade factors are lower in MET2 compared to the results achieved for MET1 sample, mainly due to their different mineralogical compositions. While sample MET1 contains a higher portion of non-magnetic feldspar, MET2 has an elevated portion of fayalite which are paramagnetic, and therefore, is separated together with the scandium-bearing minerals in the magnetic fraction.

Wet High Intensity Magnetic Separation (WHIMS)

The −0.10+0.02 mm size fractions of MET1 and MET2 samples were subjected to WHIMS at different magnetic field strength of 2000, 8000 and 12000 Gauss. For MET1 sample at 2000 Gauss, 94% of the scandium and 70% of the TREE were recovered in the magnetic fraction in 53% of the mass. At higher magnetic field strength, the scandium and TREE recoveries were higher: 95.8% at 8000 Gauss, and 96.1% at 12000 Gauss for scandium, and TREE recoveries of 95% at 8000 Gauss, and 98% at 12000 Gauss, but the concentrate mass was also elevated at 63% for 8000 Gauss, and 66% for 12000 Gauss field strength (Table 13-13).

The best enrichment of scandium was achieved at 2000 Gauss for both MET1 and MET2 samples. Scandium and TREE upgraded better in MET1 compared with MET2. The scandium upgrade factor for MET1 at 1.78 was superior to MET2 at 1.37 at the same magnetic strength of 2000 Gauss. Generally, across all the magnetic strength tested, scandium and rare earth elements upgraded better in the pyroxene-rich ferrosyenite (MET1) sample compared with the olivine-rich MET2 due to their different mineralogical compositions. While MET1 contains higher proportion of non-magnetic feldspar, MET2 has an elevated concentration of paramagnetic fayalite which reports to the magnetic concentrate with the Sc-bearing minerals.

Overall, the enrichment of the rare earth elements was inferior to scandium upgrade for both MET1 and MET2 samples at the same magnetic field strength. While 2000 Gauss was sufficient to recover most of the scandium-bearing minerals, a stronger magnetic field strength of 8000 Gauss was required to separate the TREE bearing mineral apatite into the WHIMS magnetic fraction.

Table 13-13: MET1 and MET2 −0.1+0.02 mm Size Fraction Response to WHIMS at Different Magnetic Field Strength

Samples	Magnetic Field Strenght, Gauss	Weight, wt.-%	Chemical Analysis			Distribution, wt.-%			Upgrade Factor		
			Sc, mg/kg	TREE, mg/kg	Fe_2O_3, wt-%	Sc	TREE	Fe_2O_3	Sc	TREE	Fe_2O_3
MET1 −0.1+0.02 mm	2000	52.9%	370	6000	26.8	94.1%	69.9%	95.2%	1.78	1.32	1.80
	8000	63.3%	330	8100	27.5	95.8%	95.0%	99.4%	1.58	1.50	1.57
	12000	66.5%	310	8100	26.6	96.1%	97.7%	99.6%	1.45	1.47	1.50
MET2 −0.1+0.02 mm	2000	70.1%	280	4200	35.3	95.9%	72.7%	97.1%	1.37	1.04	1.38
	8000	81.5%	240	4500	30.2	97.7%	91.2%	99.5%	1.20	1.12	1.22
	12000	83.7%	230	4200	31.7	97.9%	94.3%	99.7%	1.17	1.13	1.19

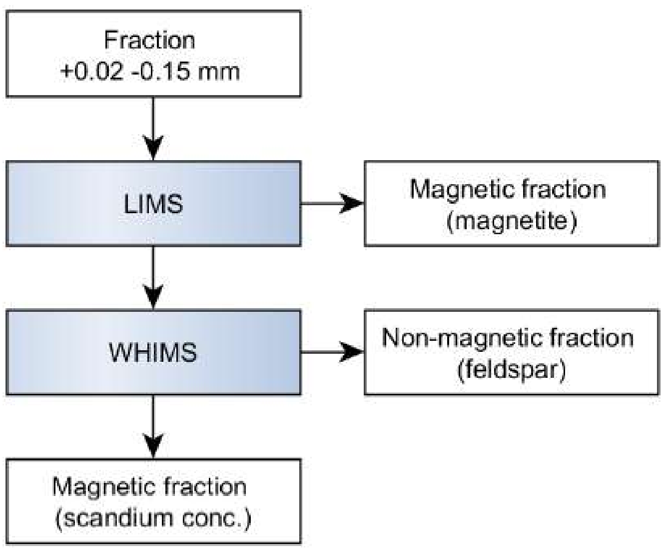
To simplify the flowsheet, one additional WHIMS test was performed on the −0.15+0.02 mm size fraction at 2000 Gauss instead of the −0.10+0.02 mm fraction that was used for the previous tests. During wet high intensity magnetic separation ferromagnetic particles stick to the matrix even after the magnet is switched off. This material could not be separated by normal washing procedure and had to be manually washed from the matrix after the test. For this particular test at −0.15+0.02 mm size fraction, the ferromagnetic material that stuck to the matrix (Mag Matrix) was collected separately for analysis.

In the magnetic fraction for this test, 80.4% of scandium, and 69.0% of TREE were recovered into 50.0% of the mass, resulting in a scandium upgrade factor of 1.66 and a TREE upgrade factor of 1.37. About 16% of scandium, 22% of TREE and 35.5% of mass were recovered into the ferromagnetic fraction (Mag Matrix), resulting in a scandium upgrade factor of 0.89 and a TREE upgrade factor of 1.17.

LIMS and WHIMS Combination

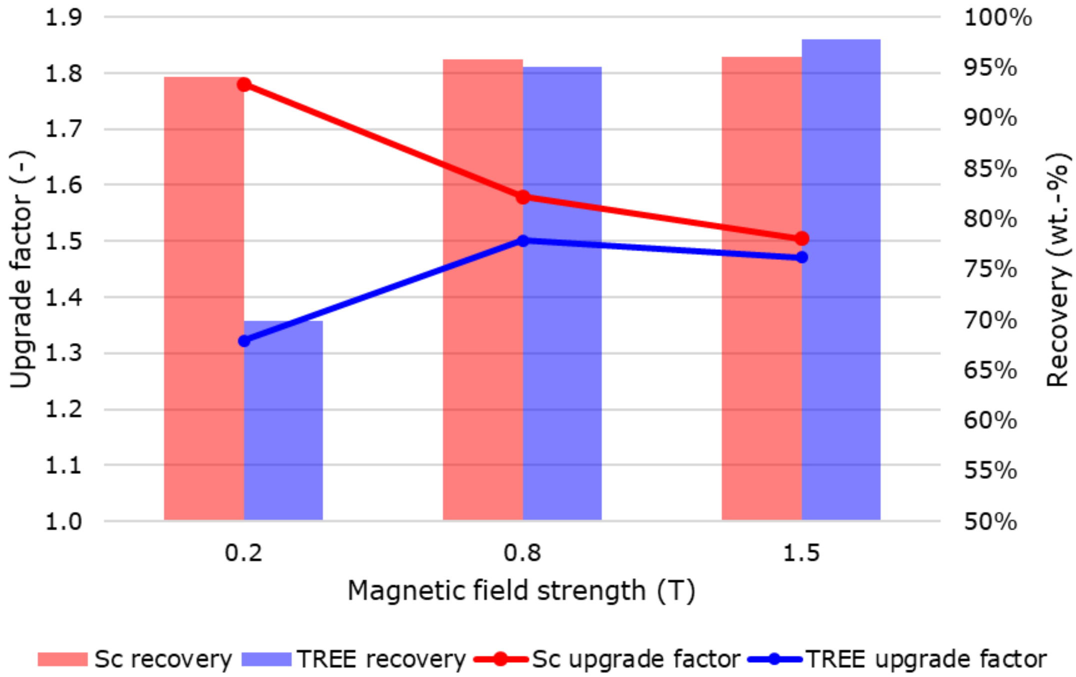
Since ferromagnetic minerals in the samples can be separated using low magnetic field strengths (LIMS), a combination of LIMS and WHIMS was used in subsequent tests, as shown in Figure 13-7.

Figure 13-7: Combination of LIMS and WHIMS



WHIMS tests on fraction −0.10+0.02 mm showed that a magnetic field strength of 2000 Gauss was sufficient to separate the scandium bearing minerals in the magnetic fraction, while feldspars, being the major gangue mineral, reported to the nonmagnetic fraction, resulting in a magnetic fraction containing 94 wt.-% of the scandium in 53 wt.-% of the feed mass for MET1. Scandium and TREE recoveries and their upgrade factors for tested magnetic field strengths are presented in Figure 13-8. While a magnetic field strength of 2000 Gauss was sufficient to recover scandium at a high upgrade factor, a higher magnetic field strength of 8000 Gauss was necessary to separate apatite, the main REE bearing mineral, together with the scandium bearing minerals to the magnetic fraction. Further increase of the magnetic field strength (15000 Gauss) did not result in an additional increase of recoveries nor a significant improvement to the upgrading of scandium and TREE.

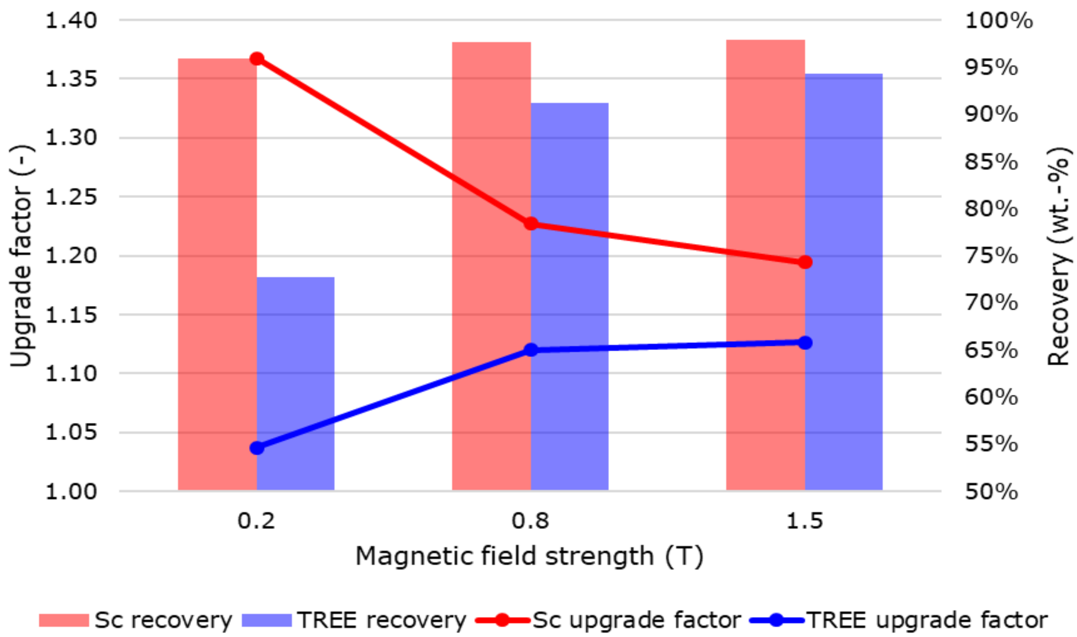
Figure 13-8: MET1 Response to WHIMS at Different Magnetic Field Strengths



Beneficiation tests for MET2 were based on the results of tests completed on MET-01 sample. For MET2, the tests were also carried out on different size fractions.

WHIMS tests showed that a magnetic field strength of 2000 Gauss was sufficient to separate the Scandium bearing minerals to a magnetic product from MET2 sample. In comparison to MET1, the feldspar content in MET2 was lower while the amount of fayalite present in MET2 was higher than in MET1. Since fayalite shows magnetic properties, in contrast to feldspar, a higher mass pull to the magnetic product fraction was achieved compared to MET1, resulting in a magnetic fraction containing 95.9 wt.-% of the scandium in 70.1 wt.-% of the feed mass. Scandium and TREE recoveries and upgrade factors at higher magnetic field strengths are presented in Figure 13-9. The general effects of WHIMS on MET1 are also true for MET2. While 2000 Gauss was sufficient to separate most of the scandium bearing minerals, 8000 Gauss was needed to separate the TREE bearing mineral apatite to the magnetic fraction as well.

Figure 13-9: MET2 Response to WHIMS at Different Magnetic Field Strengths



Flotation

Three (3) and five (5) litre laboratory flotation cells manufactured by HUMBOLDT WEDAG was used for the flotation tests. FA2 (tall fatty acid), Flotigam 4343, SM15, and Armac T collectors were the main flotation reagents that were evaluated. Flotation pH was adjusted with the addition of sulphuric acid and acidified water glass ("AWG"), or sodium silicate was used as the depressant.

Flotation kinetic tests showed that Flotinor SM15, a phosphoric acid ester as the main flotation collector reagent had the highest selectivity for the scandium bearing minerals for both MET1 and MET2 samples. In the first thirty seconds of a kinetic flotation test with SM15 collector, a concentrate of 49 wt.-% of the mass was separated containing 80 wt.-% of the scandium, resulting in a scandium upgrade factor of 1.63 for MET1. For MET2, 55.6 wt.-% of the mass containing 66.7 wt.-% of the scandium was recovered in the first thirty seconds. The scandium recovery was significantly reduced compared to the same flotation scheme applied MET1, which was due to a slightly different mineralogical composition and the reduced mineral liberation in MET2 compared to MET1.

Mineral Processing Options

In an attempt to simplify the flowsheet, fractions −0.15+0.02 mm and −0.02 mm were tested besides fractions −0.3+0.1 mm, −0.10+0.02 mm and −0.02 mm. LIMS, WHIMS and flotation were applied to fraction −0.15+0.02 mm while fraction −0.02 mm was subjected to wet high intensity magnetic separation only.

MET1 sample response to the following processing options, which combine different beneficiation methods, was evaluated:

- LIMS followed by WHIMS of −0.15+0.01 mm size fraction combined with the magnetic stream from WHIMS of −0.02mm fraction
- LIMS, followed by cleaner flotation of −0.15+0.01 mm size classification plus the magnetic product of −0.02 mm fraction.
- Flotation only of fraction −0.15+0.02 mm combined with the magnetic product of WHIMS for -0.02 mm fraction.

The mass pull and recoveries of scandium and TREE to mineral concentrate for the different processing options and the upgrade factors for scandium and TREE are presented in Table 13-14.

Table 13-14: Scandium, TREE Recoveries and Upgrade Factors for Process Options for Fraction −0.15 mm, MET1

Mineral Beneficiation Techniques	Recovery			Upgrade factor	
	Weight %	Sc %	TREE %	Sc	TREE
LIMS +WHIMS	38.8	87.5	67.9	2.3	1.8
LIMS + Flotation	39.6	81.9	70.4	2.1	1.8
Flotation+ WHIMS	45.6	82.8	87.9	1.8	1.9

The combination of LIMS and WHIMS (Option 1) represents the simplest of the three considered options as it involves magnetic separation only and does not require addition of any chemicals. It results in a scandium recovery of 87.5 wt.-% and an upgrade factor of 2.3. TREE recovery was 67.9 wt.-% at an upgrade factor of 1.8 for MET1. With this option, only magnetic separation at different magnetic field strengths is required. The slimes (−0.02 mm) fraction can be treated together with the nonmagnetic product from LIMS of −0.15+0.02 mm size classification in the wet high intensity magnetic circuit.

The option of combining LIMS with cleaner flotation for fraction −0.15+0.02 mm and WHIMS of fraction −0.02 mm (Option 2) involves both magnetic separation and flotation thus presenting a more complicated flowsheet requiring the addition of acid and flotation agents in the flotation step. It resulted in a scandium recovery of 81.9 wt.-% and upgrade factor of 2.1. TREE recovery at the identical upgrade factor of 1.8 was slightly higher (70.4 wt.-%) for Option 2 compared to magnetic separation only (Option 1). Two processing steps (LIMS and flotation) were applied to fraction −0.15+0.02 mm and a third processing step (WHIMS) to fraction −0.02 mm. Due to the higher scandium recovery and reduced complexity of magnetic separation only, the combination of LIMS and WHIMS (Option 1) is favourable compared to Option 2 which involves the integration of flotation after LIMS.

Option 3 in which cleaner flotation only was applied to fraction −0.15+0.02 mm and WHIMS was applied to the slime fraction −0.02 mm presents a combination exhibiting a reduced complexity of the flowsheet compared to Option 2, but still requires the use of acid and flotation reagents. Option 3 achieved relatively high scandium recovery of 82.8-wt. % and a significantly superior TREE recovery of 87.9 wt.-% in comparison to Options 1 and 2. The mass rejection for this option is lower, resulting in reduced upgrade factors of 1.8 for scandium and 1.9 for TREE.

For both options including flotation (Options 2 and 3), recovery rates are based on the cleaner concentrate of the third cleaner stage, without considering recirculation of the cleaner tailings.

Two processing options were considered for MET2 sample:

— LIMS of −0.15+0.02 mm fraction followed by WHIMS of the magnetic product from LIMS and slime fraction −0.02 mm.
— Cleaner flotation of fraction −0.15+0.02 mm combined with WHIMS of the slimes (-0.02 mm fraction).

The response of MET2 sample to different processing options are presented in Table 13-15. The combination of LIMS and WHIMS (Option 1) resulted in a scandium recovery of 77.8 wt.-% and upgrade factor of 1.5. TREE recovery was 57.5 wt.-% at an upgrade factor of 1.1.

In the Option 2, flotation only was applied to fraction −0.15+0.02 mm and WHIMS was used to process the slimes (−0.02 mm). Compared to the combination of LIMS and WHIMS only (Option 1), acid and flotation reagents have to be used. Flotation of fraction −0.15+0.02 mm and WHIMS processing of the slimes (fraction −0.02 mm) resulted in a similar scandium recovery of 74.1 wt.-% but at significantly increased TREE recovery of 83.3 wt.-%. The mass reduction for both options is similar, resulting in comparable upgrade factors of 1.5 for scandium but at significantly superior upgrade of 1.6 for TREE due to the increased TREE recovery with flotation.

Table 13-15: Scandium and TREE Recoveries and Upgrade Factors for Process Options for Fraction -0.15 mm, MET2

Mineral Beneficiation Techniques	Recovery			Upgrade factor	
	Weight %	Sc %	TREE %	Sc	TREE
LIMS +WHIMS	51.5	77.8	57.5	1.5	1.1
Flotation + WHIMS	50.9	74.1	83.3	1.5	1.6

In contrast to the flotation tests carried out for ore type MET1, AWG, a mixture of 80% sodium silicate and 20% oxalic acid was used in the flotation step as depressant for MET2 sample. The aim of using AWG was to depress fayalite. In the case of MET2 flotation with SM15 as collector, AWG was not effective in depressing fayalite, but was successful in limiting the amount of magnetite that deported to the Sc-REE mineral concentrate. Therefore, a combination of LIMS and flotation in case of MET2 was not necessary. AWG depressant could be tested on MET1 sample to depress magnetite instead of using LIMS as a separate processing step in order to reduce the complexity of the flowsheet and to increase the scandium recovery, in case flotation is selected as the preferred option to increase the yield of both TREE and scandium.

Recovery rates of the flotation process are based on the cleaner concentrate of the third cleaner stage, not considering recirculation of the cleaner tailings. Recirculation of cleaner tailings present a potential for the optimization of scandium and TREE recovery.

MPLAN MET1 and MET2 Physical Processing Testwork Summary

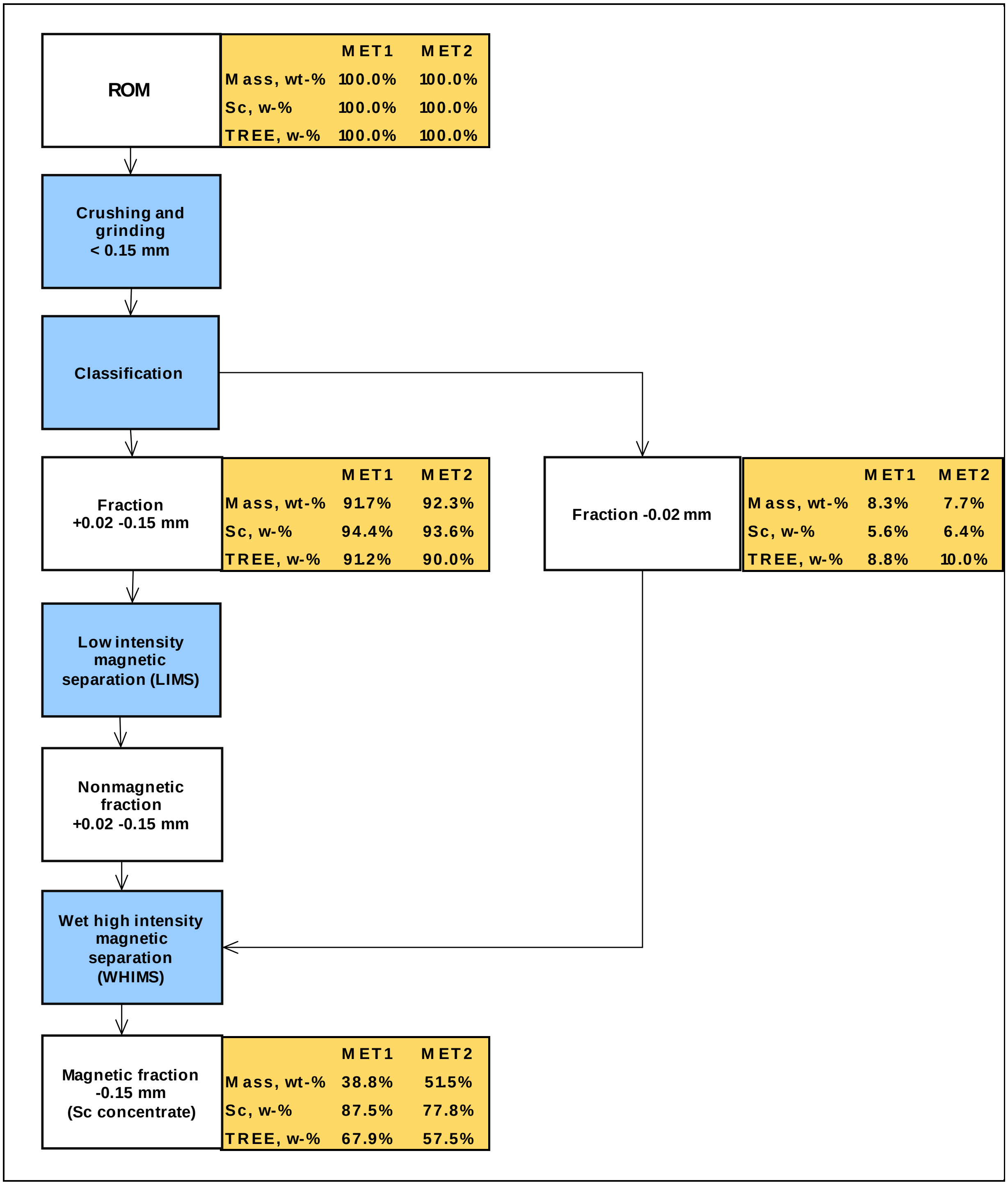
In summary, the mineral processing test program completed as part of Phase 2 metallurgical development by MPlan confirmed the following:

- Pyroxene (hedenbergite) and amphibole (ferrohornblende/ ferropargasite) are the only scandium bearing minerals in the mineralization. Different feldspars, namely albite, microcline, and anorthoclase, as well as fayalite and mica (biotite) were identified as main components. Ilmenite, magnetite and the REE bearing phases (hydroxyl-) apatite and zircon were also identified in the samples.
- A Sc mineral concentrate can be produced from Crater Lake mineralization by using simple low-cost magnetic separation techniques. A combination of LIMS and WHIMS, as presented in Figure 13-10, produced a mineral concentrate yielding 88% Sc recovery as well 69% recovery of TREE for MET1. A combination of LIMS and WHIMS on MET2 sample with differing mineralogy also yielded 78% Sc and 56% TREE.
- Additional testwork utilizing Sensor-based ore sorting and HLS methods confirm that XRT sensor-based sorting and DMS offer additional low-cost alternatives for inexpensively producing a mineral concentrate without the need of grinding, chemical reagents or extensive water consumption. DMS separation yielded recoveries of 90.6% Sc and 89.2% TREE in the mineral concentrate.

It was recommended to use a simple mineral processing flowsheet consisting of crushing and milling to −0.15 mm fraction, followed by LIMS to reject the ferromagnetic minerals and subjecting the LIMS nonmagnetic stream to WHIMS to recover the Sc and REE-bearing minerals into a mineral concentrate.

The recommended flowsheet with stage recoveries for mass, scandium, and total rare earth elements for MET1 and MET2 are presented in Figure 13-10.

Figure 13-10: Physical Separation Flowsheet



13.5 Phase 3 Metallurgical Development Program – Hydrometallurgy - MPlan

In July 2020, Imperial engaged MPlan to carry out the Phase 3 of the metallurgical development program. This program was focused on the development of an efficient hydrometallurgical process to recover a high-purity scandium oxide product from the scandium-rich mineral concentrates.

The Phase 3 Metallurgical Development program consists of:

- Bulk production of representative magnetic concentrate products from the MET1 and MET2 composite samples using the magnetic separation (LIMS – WHIMS Combination) flowsheet that was developed under Phase 2 program. The mineral concentrate was used in subsequent downstream hydrometallurgical flowsheet development.
- Development of a hydrometallurgical flowsheet for the extraction of scandium into a scandium oxide product and the rare earth elements (REE) into a bulk mixed REE concentrate.

The testwork results are summarized in the following three testwork reports:

- Preparation of a Scandium and Rare Earth Element Bearing Mineral Concentrate for Downstream Hydrometallurgical Testwork for the Crater Lake Scandium Project, Quebec, Canada, Phase 3 Testwork Report; MPlan International Limited, December 26, 2020
- Hydrometallurgical Processing of Scandium Bearing Mineral Samples Crater Lake Scandium Project, Quebec, Canada, Testwork Report Part 2: Hydrometallurgical Processing Scandium Bearing Mineral Samples; MPlan International Limited, July 21, 2020.
- Hydrometallurgical Processing of Scandium Bearing Mienral Samples Crater Lake Scandium Project, Quebec, Canada, Testwork Report Part 2: Preconcentration and Ion Exchange/Solvent Extration Tests; MPlan International Limited, November 15, 2021

13.5.1 MPlan Bulk Mineral Processing

Two mineralized MET1 and MET2 composite samples, weighing 100 kg each, were submitted for bulk processing to produce Sc/REE mineral concentrate for downstream hydrometallurgical test program. MET1 composite graded 180 g/t scandium and 2555 g/t TREE, whereas the MET2 composite graded 150 g/t scandium and 3157 g/t TREE. Jaw and roll crushing to 100% passing 3 mm was completed prior to blending and splitting into 10 kg test charges. A single confirmatory test charge of each composite was ball mill ground to 100% passing 150 microns subjected to magnetic separation, first by LIMS and then WHIMS of the LIMS non-magnetics. Once the response of the composites was confirmed, the remaining 90 kg of each sample was processed through the same scheme.

A summary of the head assay results including whole rock analysis and an ICP scan for the two samples used in this program is presented in Table 13-16. Composite head grades from MET1 and MET2 samples evaluated in the Phase 2 program are also provided for comparison.

Table 13-16: Head Assays for MET1 and MET2 Samples Used in Phase 2 Program and Split Used in Phase 3 Development Program

Analyte	Unit	MET 1			MET 2		
		10 kg Split	90 kg Split	Phase 2 Comp	10 kg Split	90 kg Split	Phase 2 Comp
Sc	[mg/kg]	170	180	240	160	150	240
SiO_2	[wt.-%]	50.1	50.2	51.1	46.4	46.2	43.9
Al_2O_3	[wt.-%]	12.1	11.9	11.8	10.8	10.7	9.6
Fe_2O_3	[wt.-%]	18.2	18.4	17.3	22.5	23.2	26.4
TiO_2	[wt.-%]	1.6	1.6	1.5	2	2.1	2.3
K_2O	[wt.-%]	4.3	4.2	4.3	3.6	3.5	3.1
Na_2O	[wt.-%]	4	4.1	4.1	3.5	3.5	3.2
CaO	[wt.-%]	5.7	5.8	6.2	6	6	6.4
MgO	[wt.-%]	1.2	1.2	0.9	1.6	1.6	1.8
MnO	[wt.-%]	0.58	0.6	0.53	0.73	0.76	0.85
BaO	[wt.-%]	0.08	0.09	0.06	0.11	0.09	0.1
P_2O_5	[wt.-%]	0.96	0.84	0.77	1.38	1.34	1.24
ZrO_2	[wt.-%]	0.82	0.79	0.87	0.98	0.96	1
LOI	[wt.-%]	0.6	0.7	0.62	1.1	1.2	0.21
Sc	[mg/kg]	170	180	240	160	150	240
Y	[mg/kg]	260	240	370	290	290	320
La	[mg/kg]	480	440	700	550	560	570
Ce	[mg/kg]	860	700	1,500	940	860	1,600
Pr	[mg/kg]	190	170	240	200	200	210
Nd	[mg/kg]	770	680	850	860	850	760
Sm	[mg/kg]	94	96	130	110	120	83
Eu	[mg/kg]	<5	<5	<25	<5	<5	<25
Gd	[mg/kg]	91	89	110	100	110	110
Tb	[mg/kg]	14	15	<25	16	15	<25
Dy	[mg/kg]	59	55	84	65	67	74
Ho	[mg/kg]	<5	<5	<25	<5	<5	<25
Er	[mg/kg]	35	33	47	40	41	43
Tm	[mg/kg]	<5	<5	<25	<5	<5	<25
Yb	[mg/kg]	31	29	46	35	35	40
Lu	[mg/kg]	7	8	<25	9	9	<25
TREE+Y	[mg/kg]	2891	2555	4077	3215	3157	3810

The LIMS testwork completed on –0.15 mm grind particle size of MET1 and MET2 samples indicated that an iron rich fraction that is low in scandium could be generated. The iron rich LIMS magnetic stream represents approximately 7% of the initial sample mass. WHIMS testwork at 8000 Gauss confirmed that high scandium recoveries could be achieved to a magnetic concentrate with up to 50% of the mass rejected to the non-magnetics.

The results of the magnetic separation tests for the 10 kg test charges of composites MET1 and MET2 are presented in Table 13-17 and Table 13-18.

Table 13-17: Magnetic Separation Test Results for MET1, 10 kg Test

MET1 10 kg	Weight recovery	Chemical Analysis					Distribution		
		Sc	TREE	SiO_2	Al_2O_3	Fe_2O_3	Sc	TREE	Fe_2O_3
	(wt-%)	(mg/kg)	(mg/kg)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)
LIMS	7.3	78	2,500	9.1	2.7	74.1	3.2	7.5	30
WHIMS 1 (2000 Gauss)	25.5	430	3,200	42.7	6.6	30.8	60.9	33.6	43.4
WHIMS 2 (8000 Gauss)	17.5	310	3,600	46.9	8.8	21.2	30.1	26	20.5
Non Mag (WHIMS)	49.8	21	1,600	62.1	18	2.2	5.8	32.9	6.1
Head (Cal)	100	180	2,422	50.6	12.4	18.1	100	100	100
LIMS + WHIMS	50.2	337	3,237	39.3	6.8	33.8	94.2	67.1	93.9
WHIMS 1 +2	**42.9**	**381**	**3,363**	**44.4**	**7.5**	**26.9**	**91**	**59.6**	**63.9**

Table 13-18: Magnetic Separation Test Results for MET2, 10 kg Test

MET2 10 kg	Weight recovery	Chemical Analysis					Distribution		
		Sc	TREE	SiO_2	Al_2O_3	Fe_2O_3	Sc	TREE	Fe_2O_3
	(wt-%)	(mg/kg)	(mg/kg)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)
LIMS	6.9	92	2,800	7.4	2.1	75.3	3.9	7.8	20.4
WHIMS 1 (2,000 Gauss)	32.2	230	2,000	38.7	5.1	38.5	45.2	25.7	48.5
WHIMS 2 (8,000 Gauss)	28.2	260	3,600	45.5	9	23.3	44.7	40.5	25.7
Non Mag (WHIMS)	32.7	31	2,000	60	17.1	4.3	6.2	26.1	5.4
Head (Cal)	100	164	2,506	45.4	9.9	25.6	100	100	100
LIMS + WHIMS	67.3	228	2,752	38.3	6.4	35.9	93.8	73.9	94.6
WHIMS 1 +2	60.4	244	2,747	41.9	6.9	31.4	89.9	66.1	74.1

Low intensity magnetic separation yielded comparable mass recoveries for both composites of approximately 7.0%. The main component of the LIMS magnetic concentrate appears to be iron minerals, with Fe_2O_3 grades of 74.1% and 75.3% for MET1 and MET2 respectively. Scandium grade in LIMS concentrates was well below the head grade, whereas the TREE grade was close to that observed for the head samples.

WHIMS separation of the LIMS magnetic tailings was carried out in two passes, at 2000 Gauss and 8000 Gauss. Both passes resulted an upgrading of scandium to the magnetic fraction, with combined scandium recoveries of approximately 90%. For MET1, 61% of the scandium was recovered in the first pass at 2000 Gauss, and an additional 31% was recovered from the first pass tailings at the higher intensity of 8000 Gauss. The scandium recovery for MET2 was essentially evenly split between the two passes.

Mass rejected to non-magnetic tailings was 50% for MET1 and 33% for MET2. As a result, MET1 achieved a better upgrade ratio, reaching 381 g/t scandium for the combined WHIMS concentrate, compared to 244 g/t scandium for MET2. Only minor upgrading of TREE to the concentrate was observed for either composite.

Based on the positive results achieved with the 10 kg batch tests, the remaining 90 kg of sample was processed through a similar flowsheet consisting of a LIMS separation followed by a single pass through the WHIMS at 8,000 Gauss. Results of the 90 kg tests are presented in Table 13-19 and Table 13-20, respectively, for MET1 and MET2 samples.

Table 13-19: Magnetic Separation Test Results for MET1, 90 kg

MET1 90 kg	Weight recovery	Chemical Analysis					Distribution		
		Sc	TREE	SiO_2	Al_2O_3	Fe_2O_3	Sc	TREE	Fe_2O_3
	(wt-%)	(mg/kg)	(mg/kg)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)
LIMS	6.8	55	2400	8.55	2.18	69.7	2.2	6.9	26.1
WHIMS (8000 Gauss)	48.6	340	3500	45.3	8	25.5	95.5	72.2	68.4
Non-Mag (WHIMS)	44.7	9	1100	62.5	18	2.23	2.3	20.9	5.5
Head (Cal)	100	173	2354	50.5	12.1	18.1	100	100	100

Table 13-20: Magnetic Separation Test Results for MET2, 90 kg

MET2 90 kg	Weight recovery	Chemical Analysis					Distribution		
		Sc	TREE	SiO_2	Al_2O_3	Fe_2O_3	Sc	TREE	Fe_2O_3
	(wt-%)	(mg/kg)	(mg/kg)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)	(wt-%)
LIMS	6.6	71	2400	8.55	2.18	69.7	2.5	5.6	18.6
WHIMS (8000 Gauss)	62.7	280	3300	42.3	7.38	30.5	95	73.6	77.7
Non-Mag (WHIMS)	30.8	15	1900	61.1	17.8	2.98	2.5	20.8	3.7
Head (Calculated)	100	185	2810	45.9	10.2	24.6	100	100	100

The 90 kg tests achieved very similar mass recoveries and grades to the LIMS concentrate. For both samples, scandium and TREE recoveries to the single WHIMS concentrate were slightly better than those observed for the combined concentrate in the 10 kg tests. Improved recovery may be the result of longer running time (minimization of start/end effects). Compared to the Phase 2 test results, the upgrading of scandium to concentrate for MET1 was lower in the present program, however, the recovery was higher despite the lower initial head grade of the samples. This suggests that the two results may lie on the same grade-recovery curve. For composite MET2, the scandium recovery and gangue rejection realized in the present program was virtually identical to that of the Phase 2 program.

About 45 kg of MET1 and 62 kg of olivine-rich MET2 mineral concentrates were produced for feeding the Phase 3 hydrometallurgical development programs. About five kilograms of each mineral concentrate samples were used for the laboratory scale hydrometallurgical development programs at MPlan – ANZAPLAN which focused on the development of mineral decomposition methods, and bench scale evaluation of recovery methods for scandium and rare earth elements from primary leach solution (PLS). The remaining mineral concentrate samples are dedicated to bulk treatment and generation of PLS for development of the hydrometallurgical processing flowsheet and optimization. The hydrometallurgical development program which also includes olivine removal from the olivine-rich MET2 mineral concentrate to further reduce the concentrate mass is ongoing at SGS, Canada at the time of writing this report.

13.5.2 MPlan Hydrometallurgy Development Program – Mineral Decomposition Method

The bulk mineral concentrates prepared from MET1 and MET2 samples were used as feed for the Sc/REE hydrometallurgical flowsheet development program conducted at MPlan laboratory in Hirschau, Germany. The objective of the program was to develop an efficient hydrometallurgical process to recover a high-purity scandium oxide product and a mixed REE concentrate.

The following mineral decomposition methods were tested at bench scale to extract scandium from the mineral concentrates and ore samples:

- Acid bake of mineral concentrate with concentrated sulphuric acid at a temperature range of 250°C to 300°C for a period up to 180 minutes, followed by water leach of the acid bake calcine.
- Acid soaking of ore sample crushed to -6 mm in 20 wt-% sulphuric acid solution at ambient temperature and at 60°C for several weeks.
- High pressure acid leach of mineral concentrate in aqueous sulphuric acid solution at temperature greater than 150°C in an autoclave.
- Caustic (NaOH and/or Na_2CO_3) roasting of concentrate at 900°C, followed by water leach and wash at 90°C. The resulting residue from the roast – water leach stage was subsequently leached in mineral acid.
- High pressure caustic (“HPC”) leach of concentrate, followed by water wash and mineral acid leach of the solid residues from the caustic leach stage.

While apatite, the major REE-bearing mineral in the concentrates, was easily decomposed with most of the method listed above, yielding REE extraction in the range of 43% to 74% to primary leach solution (“PLS”), the decomposition of Sc-bearing minerals, ferro-hornblende and hedenbergite, was quite challenging, with Sc recovery to PLS less than 5%.

The Sc-bearing minerals remained largely unaffected by acidic decomposition methods. Scandium extraction to PLS for acid bake - water leach was very low at less than 5%. Only heap leaching and high-pressure acid leach are the two acid decomposition methods that showed Sc recoveries above 20 wt-% but below 50 wt.-%.

A caustic roast, followed by a mineral acid leach of the solid residue, also showed very poor Sc extraction between 18 wt-% and 25 wt-%, while REE extractions to PLS were also poor between 23 wt-% and 46 wt-%.

A high-pressure caustic leach, followed by a mineral acid leach of the solid residues, showed remarkable recovery of scandium and the REE from Crater Lake Sc/REE mineralization:

- The method showed scandium recovery to PLS of 91 wt-% for MET1, and 84% for MET2 mineral concentrates.
- The recovery of TREE including yttrium (TREE+Y) of 94 wt-% and 83 wt-% respectively for MET1 and MET2 samples.
- The high recoveries of Sc and TREE+Y from both samples show that the method has excellent efficacy in extracting Sc and REE from samples representing different mineralization.

Hydrochloric acid (HCl) is preferred to sulphuric acid as the lixiviant to avoid the formation of insoluble precipitates which may lead to low extraction of scandium and rare earth elements. The key test conditions for the high-pressure caustic cracking of Sc/REE mineral concentrate and subsequent hydrochloric acid leach of the caustic residues are presented in Table 13-21. The metallurgical balance for the hydrochloric acid leach is presented in Table 13-22.

Table 13-21: Test Conditions for HPC Leach and HCl Leach of HPC Residues

Test Conditions for High Pressure Caustic Leach						
Test ID	HPC2	HPC	HPC7	HPC8	HPC9	HPC10
Feed Sample	MET1	MET1	MET1	MET2	MET1	MET2
Leaching agent dose, kg/t conc.	0.40	0.80	1.38	1.38	1.00	1.38
Solid content, %	60	43	30	30	30	30
Leaching time, minutes	120	120	120	120	120	120
Leaching temperature, °C	250	180	220	220	220	220
Test Conditions for Hydrochloric Acid Leach of HPC Residues						
Test ID	HPC2-HCL	HPC5-HCL	HPC7-HCL	HPC8-HCL	HPC9-HCL	HPC10-HCL
Reagent, HCl	20%	20% l	20%	20%	20%	20% l
Leaching time, minutes	60	1200	60	60	60	60
Leaching temperature, °C	Ambient	Ambient	Ambient	Ambient	Ambient	Ambient
Solid content, %	10	10	10	10	10	10

Table 13-22: Elemental Extraction of Hydrochloric Acid Leach of HPC Residues

Element	HPC2-HCL		HPC5-HCL		HPC7-HCL		HPC8-HCL		HPC9-HCL		HPC10-HCL	
	MET1		MET1		MET1		MET2		MET1		MET2	
	Extraction		Extraction		Extraction		Extraction		Extraction		Extraction	
	Residue	PLS	Residue	PLS	Residue	PLS	Residue	PLS	Residue	PLS	Residue	PLS
	wt-%	wt-%	wt-%	wt-%	wt-%	wt-%	wt-%	w.-%	wt-%	wt-%	wt-%	wt-%
Na	2.2	97.8	4.0	96.0	3.8	96.2	11.7	88.3	5.1	94.9	5.1	94.9
Mg	21.9	78.1	26.6	73.4	98.6	1.4	99.4	0.6	99.1	0.9	14.4	85.6
Al	12.6	87.4	16.5	83.5	10.9	89.1	26.8	73.2	20.2	79.8	17.2	82.8
Si	94.3	5.7	96.6	3.4	91.6	8.4	97.1	2.9	98.6	1.4	98.8	1.2
P	19.3	80.7	13.4	86.6	21.7	78.3	47.7	52.3	35	65	10.7	89.3
K	13.5	86.5	22.2	77.8	25.1	74.9	62.2	37.8	32.5	67.5	46.8	53.2
Ca	22.3	77.7	26.9	73.1	99.6	0.4	99.9	0.1	99.9	0.1	12.4	87.6
Fe	15.4	84.6	20.8	79.2	8	92	27.6	72.4	12.2	87.8	12.1	87.9
Zr	97.6	2.4	99.1	0.9	97.7	2.3	99.6	0.4	98.1	1.9	99.5	0.5
Sc	30	70	33	67	9.3	90.7	16.3	83.7	11.5	88.5	16.4	83.6
TREE	10.3	89.7	12.7	87.3	6.6	93.4	16.9	83.1	9.7	90.3	8.8	91.2

13.5.3 MPlan Hydrometallurgy Development Program – Primary Leach Solution Treatment

The hydrochloric acid primary leach liquor containing scandium, REE, and other dissolved impurities that was generated from the mineral concentrate decomposition tests was used for the bench scale evaluation of recovery methods for scandium and rare earth elements. The scope of the program was to test several methods to recover scandium, specifically evaluation of different extractants for solvent extraction (SX) and ion exchange (IX). Precipitation of Sc from PLS and re-leach of the crude Sc precipitate to generate a more concentrated Sc-containing liquor was also evaluated as a method for separating Sc from metal impurities that co-extracted with scandium and REE during mineral decomposition - HCl leach.

Scandium Precipitation from PLS and Releach

The hydrochloric acid leach liquor contains not just scandium and REEs, but several other metals that are present in the minerals. These impurity elements may be separated from Sc and REEs by selective precipitation. This is accomplished by neutralization of the acid leach solution. Prior to Sc precipitation, Fe^{3+} has to be fully reduced to Fe^{2+} since $Fe(OH)_2$ does precipitate at a higher pH than scandium allowing for separation. For the reduction of iron (III) to iron (II) elemental iron powder is used. The liquor is heated to 90°C, then iron powder is added, and the reaction mixture is stirred for 60 minutes and the slurry is filtered to remove undissolved solids. The liquor pH was raised to 3.5 with calcium hydroxide and then to pH 6 with the addition of sodium carbonate. The resulting precipitate which contained about 95% of the Sc, and 69% of the REE was filtered and re-leached in a mineral acid.

Sulfuric acid and hydrochloric acid were the two mineral acids evaluated. HCl was found to be more effective for redissolution of Sc from the precipitated solids since it does not form gypsum with residual calcium in the solids. However, Sc recovery in redissolution did not exceed 76.2 wt.-%. Due to relatively low recovery (76%) of Sc, it was decided to conduct SX and IX tests directly on the HCl leach liquor generated from the high-pressure caustic leach and subsequent HCl leach testwork.

Solvent Extraction (SX) – Ion Exchange (IX)

Extraction shake-out tests for reagent screening were completed on four organic reagents diluted in kerosene. The organic reagents tested for extracting Sc from the hydrochloric acid leach solution include D2EHPA, TBP, Cyanex 272, and TOPO. D2EHPA, plus TBP as the modifier and Cyanex 272 are the reagents that have shown remarkable Sc extraction, while Cyanex 272 appears to have better selectivity for Sc over the REE.

Purolite MTS9300, a polystyrenic macroporous, iminodiacetic acid chelating resin with high capacity was the IX resin tested under this program. Two-stage scrubbing of the loaded resin was performed with 2 moles HCl solution and water. Ethylenediaminetetraacetic acid (EDTA) was used as the stripping agent. Initial evaluation showed that the loading capacity of MTS9300 IX resin was inferior to the three of the SX extraction reagents that were tested, hence further development work on IX was discontinued.

The loaded organic from the SX reagent screening tests was used in a series of scoping level scrubbing tests which focused on the removal of co-extracted impurities such as iron, titanium, magnesium, calcium, potassium and aluminum. Two moles of sulfuric acid, plus 5% hydrogen peroxide (2M H_2SO_4 + 5% H_2O_2), oxalic acid and 120 g/L NaCl salt solutions were the scrubbing reagents tested on D2EHPA, TOPO, Cyanex 272, and TBP. For the TBP reagent, 130 g/L magnesium nitrate ($Mg(NO_3)_2$) solution was tested in place of the NaCl salt solution. The data showed that the best scrubbing regime that provided adequate impurity removal without significant loss of scandium is sulphuric acid plus hydrogen peroxide at ambient temperature.

It is well known that scandium is difficult to strip from D2EHPA. Others have used different stripping agents, including ammonium bifluoride, phosphoric acid and sodium hydroxide solution to strip scandium from loaded organics. This project will focus on sodium hydroxide as the preferred stripping reagent; hence several concentrations of sodium hydroxide will be tested for stripping scandium in the hydrometallurgical development program at SGS, which is ongoing at the time of writing this report. The REE remaining in the SX raffinate will be precipitated as a mixed REE carbonate product with sodium carbonate. Optimization of the hydrometallurgical flowsheet will be completed in future development programs, prior to the commencement of the pilot program.

13.6 Flowsheet Selection for Crater Lake Scandium / REE Project

The process flowsheet selected for the production of high-grade scandium oxide and mixed REE products is based on the metallurgical development programs completed by SGS Canada, Inc, and MPlan International between 2018 and 2022. The flowsheet consists of crushing and milling of the run of mine (ROM) materials, followed by magnetic separation, using LIMS and WHIMS techniques to generate a Sc/REE-rich mineral concentrate. The mineral concentrate is processed through high-pressure caustic leach to liberate scandium from the Sc-bearing silicate minerals. The solid residue from the HPC process containing the Sc and REE is subsequently leached with hydrochloric acid solution to extract the Sc/REE content into a primary leach solution. Scandium is extracted from the PLS with solvent extraction, while REE remaining in the SX raffinate is precipitated as a mixed REE carbonate product. Caustic and hydrochloric acid are the two major reagents, both will be regenerated and recycled to minimize reagent costs.

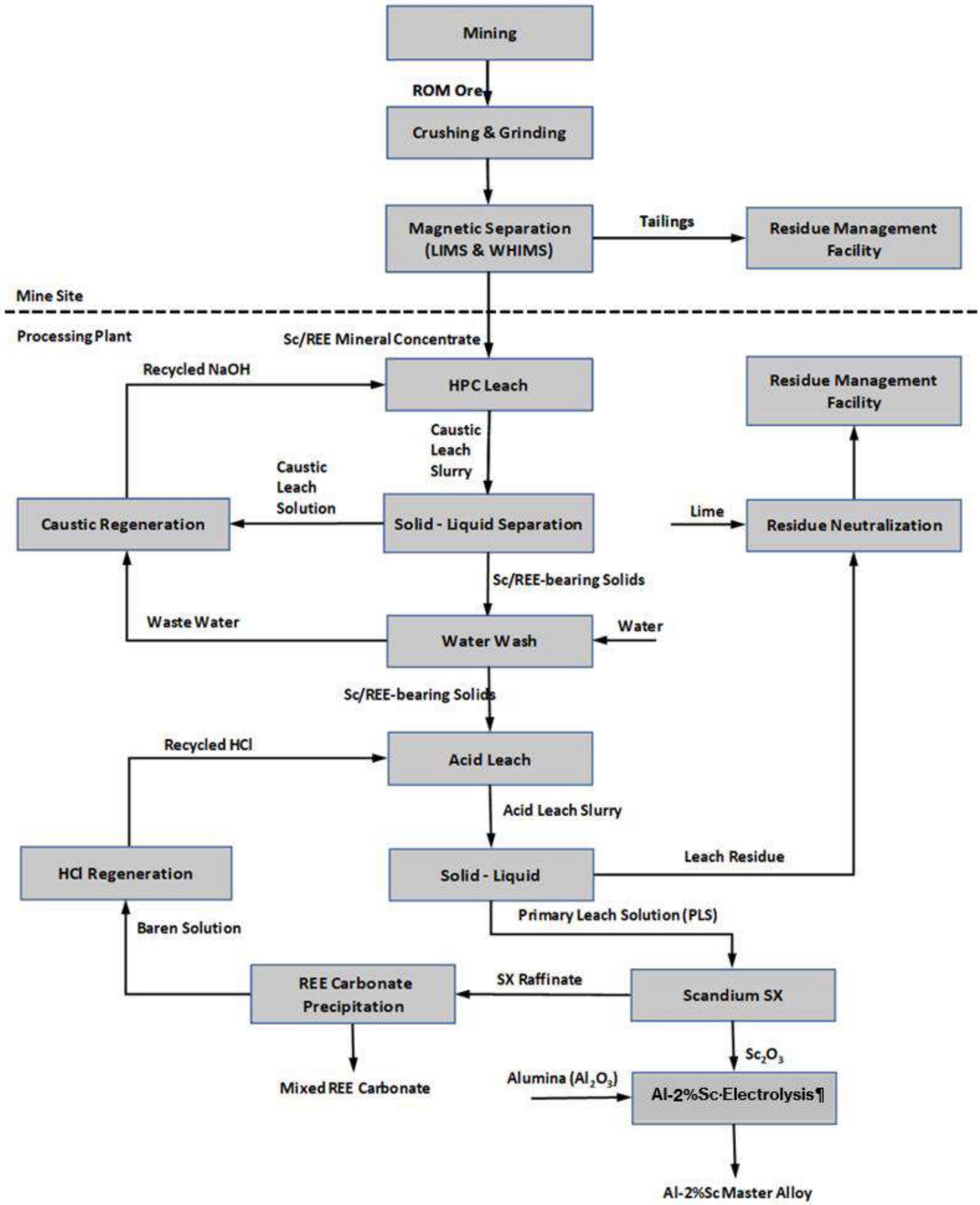
The scandium SX will use organophosphorus extractant (D2EHPA or Cyanex 272) to recover Sc into the organic phase. The Sc-loaded organic will be scrubbed with a solution of sulfuric acid plus hydrogen peroxide to remove impurity metals that co-extract with Sc. Scandium is stripped from the scrubbed organic with concentrated sodium hydroxide solution. The stripped scandium hydroxide is re-leached in mineral acid from where it is precipitated with oxalic acid. The scandium oxalate is calcined to produce a high-grade 99.9% scandium oxide product that can be used for the production of Al-2%Sc master alloy.

The Al-2%Sc master alloy production technology will be based on co-electrowinning of Al and Sc from alumina (Al_2O_3) and Sc_2O_3 to form a master alloy. This method offers higher recovery of scandium (90%-95%) and improved energy efficiency as it can be performed at a lower temperature, and it minimizes the formation of intermetallic with high Sc content in comparison to the aluminothermic method, which is the current state-of-the-art technology for Al-2%Sc master alloy production.

The development of Al-2%Sc master alloy flowsheet is yet to be developed for this project and will be completed at the next stage of project development.

The simplified process flowsheet for the production on Al-2%Sc master alloy and a mixed REE product is presented in Figure 13-11.

Figure 13-11: Simplified Process Flow Diagram for Crater Lake Sc/REE Project



13.7 Future Development Programs

Most of the processes in the flowsheet have been tested at bench scale. The mineral processing flowsheet, the high-pressure caustic leach and the hydrochloric acid leach for Sc dissolution have all been tested. Solvent extraction reagents, scrubbing, and Sc stripping conditions have been evaluated At the time of writing this report, work is ongoing on the bulk treatment of the Sc/REE mineral concentrates, including olivine removal, and hydrometallurgical development program focused on finalizing the Sc SX circuit and optimizing Sc solubilization processes. The Al-2%Sc master alloy development work focused on co-electrowinning of alumina and Sc_2O_3 is planned to start late 2022 or early 2023.

Imperial Mining Group in the fall of 2021 has collected 16 t of sample from Crater Lake site to feed its planned pilot programs. The pilot plant sample will feed a mineral processing pilot program which is planned to start in Q3/Q4 of 2022, will produce a Sc/REE-rich mineral concentrate to feed the hydrometallurgy pilot program, commencing in early to mid 2023.

Metallurgical development programs currently in progress or planned to start in the next six months are summarized in Table 13-23.

Table 13-23: Ongoing and Future Metallurgical Development Programs for Crater Lake Sc/REE Project

#	Program	Type	Start date
1	Hydrometallurgy optimization and SX development	Bench scale	In progress
2	Al-2%Sc Master alloy technology development	Bench scale	4th quarter, 2022 / 1st quarter, 2023
3	Mineral processing piloting	Pilot program	Late 2022 / early 2023
4	Hydrometallurgy flowsheet piloting	Pilot program	2nd quarter of 2023

14 MINERAL RESOURCE ESTIMATES

The mineral resource estimate for the Crater Lake Project (the "2021 MRE") was prepared by Marina Iund, P.Geo., Paul Daigle, P.Geo., and Carl Pelletier, P.Geo., using all available information.

The studied area covers the mineralized domains collectively known as the TGZ target.

The 2021 MRE was established for scandium, lanthanum, praseodymium, neodymium, terbium and dysprosium. Other REEs were not included in the estimate.

The effective date of the 2021 MRE is September 17, 2021.

14.1 Methodology

The resource area has a NE-SW strike length of 500 m, a width of 120 m, and a vertical extent of 250 m below the surface. The 2021 MRE was based on a compilation of recent diamond drill holes ("DDH") completed by the issuer.

The 2021 MRE was prepared using Leapfrog GEO 2021.1 ("Leapfrog") and GEOVIA Surpac 2021 ("Surpac") software. Leapfrog was used for the 3D geological modelling. Surpac was used for the estimation, which consisted of 3D block modelling and the inverse distance square ("ID2") interpolation method. Statistical, capping and variography studies were completed using Snowden Supervisor v8.13 and Microsoft Excel software.

The main steps in the methodology were as follows:

— Review and validation of the database
— Validation of the geological model and interpretation of the mineralized units
— Validation of the drill hole intercepts database, compositing database and capping values for the purposes of geostatistical analysis and variography
— Validation of the block model and grade interpolation
— Revision of the classification criteria and validation of the clipping areas for mineral resource classification
— Assessment of the resources with "reasonable prospects for economic extraction" and selection of appropriate cut-off grades and pit shell; and
— Generation of a mineral resource statement.

14.2 Drill Hole Database

The diamond drill hole database contains 25 DDH, which corresponds to all the holes drilled on the Project. The Surpac database contains all 25 DDH (Figure 14-1 and Figure 14-2).

The holes cover the strike length of the Project at a regular drill spacing of 50 m. A total of 2,414 intervals were sampled (1,244 samples in mineralized domains), representing 2,667 m of drilled core (1,426 m drilled in mineralized domains). The resource database includes scandium, lanthanum, praseodymium, neodymium, terbium and dysprosium assays, as well as lithological, alteration and structural descriptions.

In addition to the basic tables of raw data, the Surpac database includes several tables containing the calculated drill hole composites and wireframe solid intersections required for the statistical analysis and resource block modelling.

Figure 14-1: Surface Plan View of the Geological Model and the Validated DDH Used in the 2021 MRE

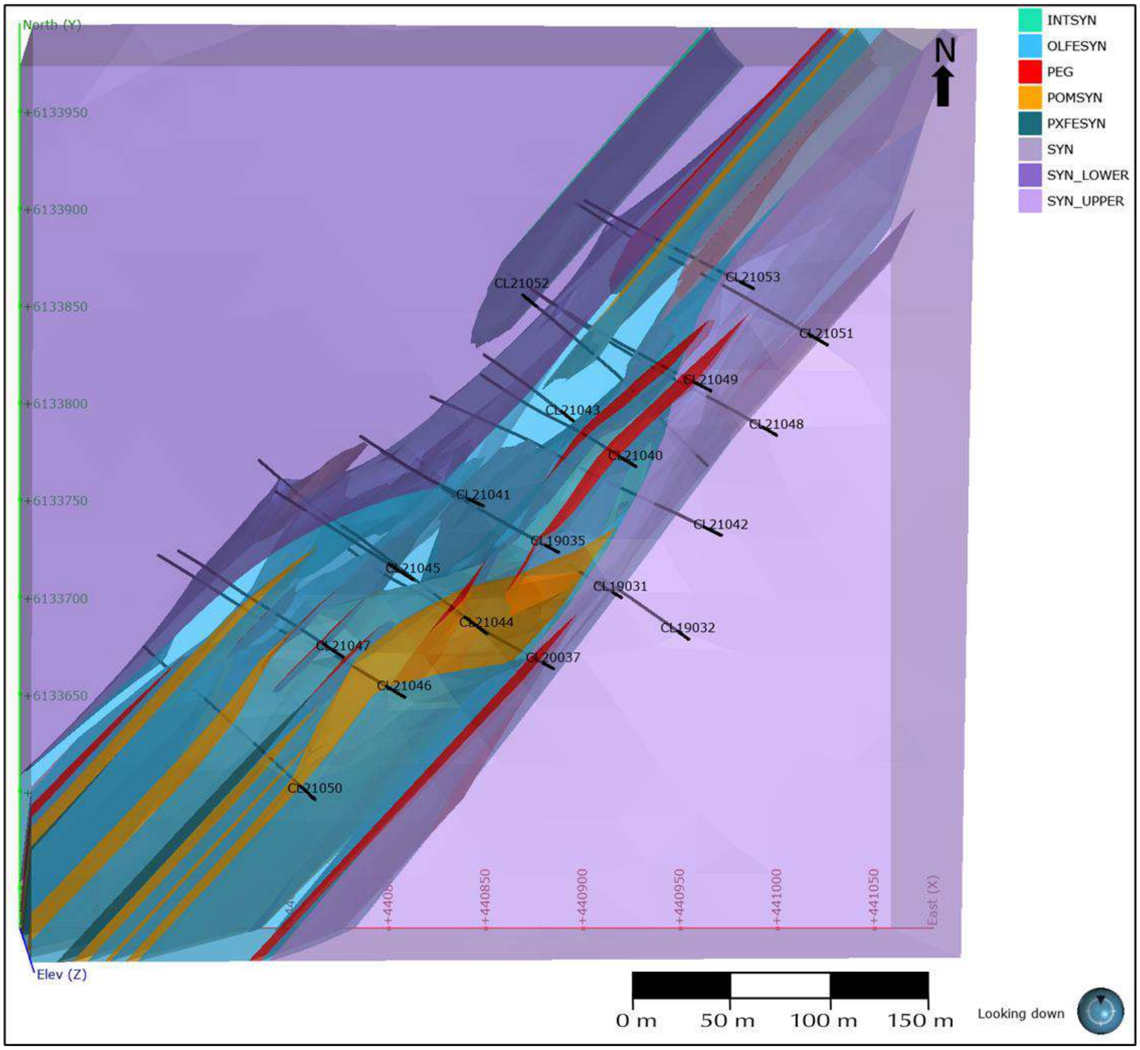
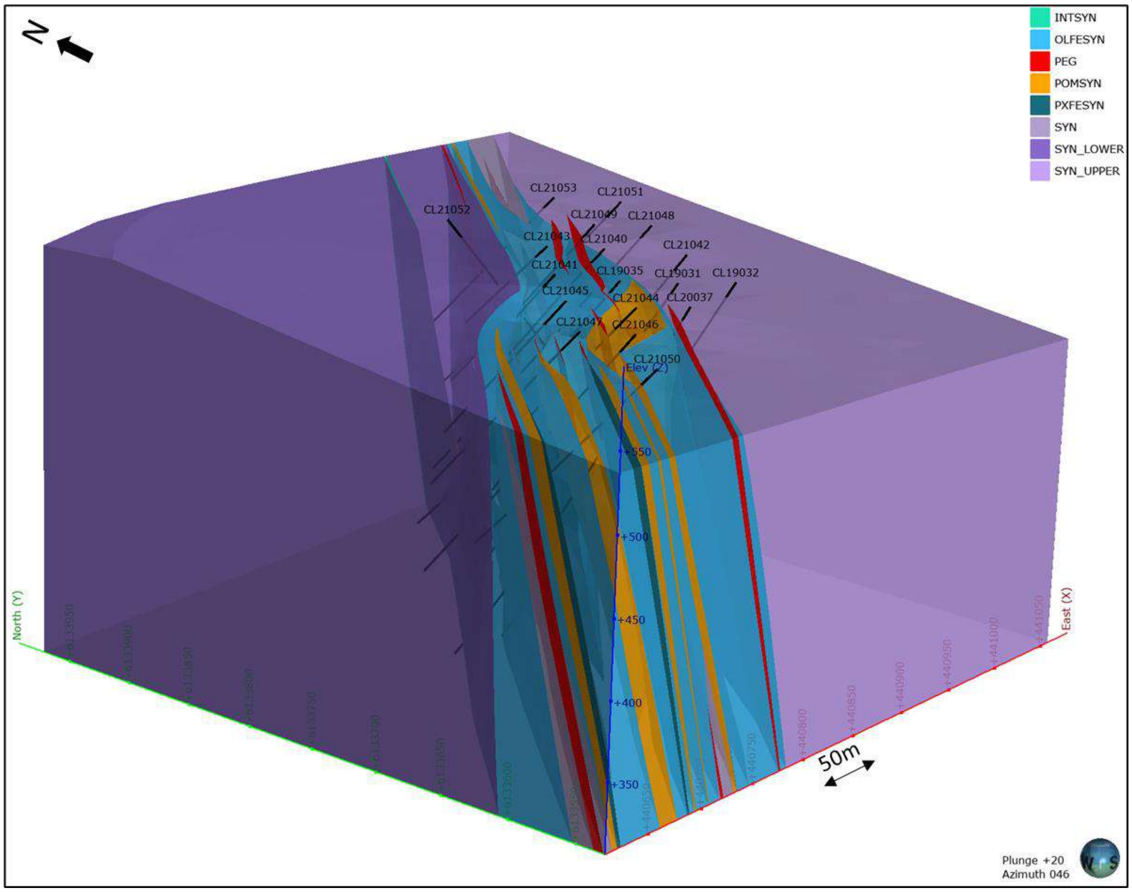


Figure 14-2: Isometric View of the Geological Model and the Validated DDH used in the 2021 MRE



14.3 Geological Model

The drill hole data were used to create the 2021 geological model (Figure 14-1 and Figure 14-2). All geological solids were modelled in Leapfrog and snapped to drill holes.

The main lithologies of the deposit are massive syenite ("SYN") intruded by olivine ferro-syenite ("OLFESYN"), four (4) pyroxene ferro-syenites ("PXFESYN") and an intermediate syenite ("INTSYN"). Later pegmatitic dykes ("PEG") and intermediate porphyries ("POM") cut all units.

The 2021 geological model was included in the Surpac block model to assign densities to the blocks. The OLFFESYN, PXFESYN and INTSYN solids were used as mineralized domains. These domains are subvertical with an NE-SW strike.

Two surfaces were created to define the topography and the overburden/bedrock contact. The surfaces were generated from surveyed drill hole collars and a Lidar topographic survey.

14.4 High-grade Capping

Basic univariate statistics were performed on the raw assay datasets for the OLFESYN domain. The datasets for other mineralized domains were too low to yield relevant results. The results obtained for OLFESYN were applied to the other domains. The following criteria were used to decide if capping was warranted:

— The coefficient of variation of the assay population is above 2.0.

— The quantity of metal contained in the top 10% highest grade samples is above 40%, and/or the quantity of metal in the top 1% highest grade samples is higher than 10%.

— The probability plot of the grade distribution shows abnormal breaks or scattered points outside the main distribution curve.
— The log-normal distribution of grades shows erratic grade bins or distanced values from the main population.

The capping threshold decided for all domains is consistent with the combination of three criteria:

— A break in the probability plot.
— A coefficient of variation below 2.0 after capping.
— The total metals of the top 1% highest grade samples is below 10% after capping.

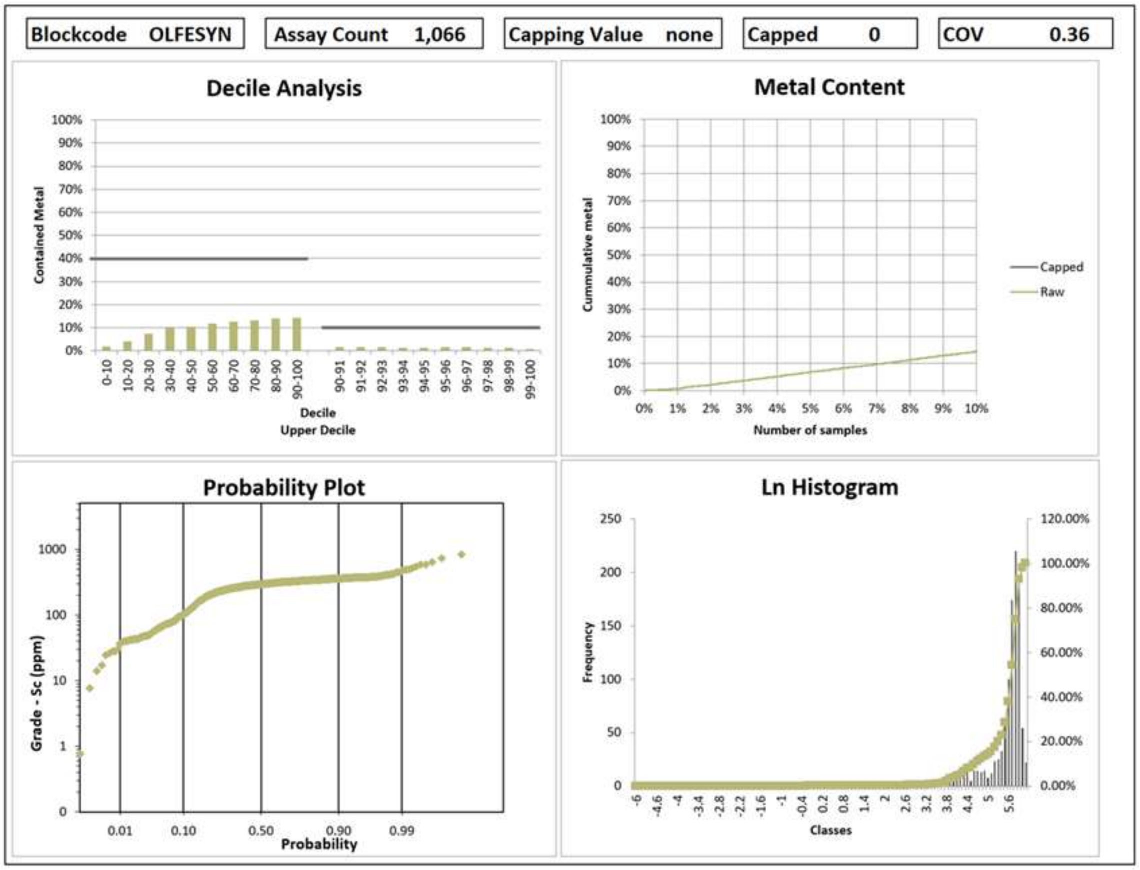
Capping was applied to raw assays. Table 14-1 presents a summary of the statistical analysis by metal.

Figure 14-3 shows an example of graphs supporting the capping threshold decisions.

Table 14-1: Summary of Univariate Statistics on Raw Assays

Oxide	No. of samples	Max (g/t)	Uncut Mean (g/t)	Uncut COV	High Grade Capping (g/t)	No. of Cut Samples	Cut Mean (g/t)	Cut COV	% Samples Cut	% Loss Metal Factor
Sc_2O_3	1064	825.18	268.48	0.36	none	0	268.48	0.36	0	0
Dy_2O_3	1064	573.90	65.93	0.4	215	1	65.59	0.34	0.2	0.5
La_2O_3	1064	15011.84	623.95	0.85	3690	2	609.96	0.42	0.2	2.2
Nd_2O_3	1064	11780.64	600.62	0.69	2100	2	589.22	0.35	0.2	1.9
Pr_2O_3	1064	3358.80	166.89	0.71	1380	3	165.08	0.47	0.2	1.1
Tb_4O_7	1064	92.10	11.54	0.41	none	0	11.54	0.41	0	0

Figure 14-3: Example of Graphs Supporting the No-capping Decision for the Sc_2O_3 Oxide in Olivine Ferro-syenite



14.5 Density

Densities are used to calculate tonnage from the estimated volumes in the resource grade block model.

From 2019 to 2021, the issuer submitted a total of 176 samples for specific bulk gravity ("SG") analysis. Table 14-2 presents a summary of the statistical analysis by lithological unit.

For each lithological unit, the median result was used to code the SG attribute in the block model. An arbitrary value of 2.00 g/cm^3 was assigned to overburden.

Table 14-2: Summary Statistics for SG by Lithology

Lithological Unit	Sample No.	Length (m)	Minimum (g/cm^3)	Maximum (g/cm^3)	COV	Mean (g/cm^3)	Median (g/cm^3)	Applied Value (g/cm^3)
INTSYN	1	0.17	3.15	3.15		3.15	3.15	3.13
OLFESYN	93	17.07	2.68	3.35	0.04	3.12	3.13	3.13
PEG	3	0.65	2.62	2.69	0.01	2.65	2.63	2.63
POMSYN	18	3.21	2.74	3.29	0.04	2.81	2.77	2.77
PXFESYN	4		2.77	3.27		2.93	2.91	2.91
SYN	10	2.16	2.65	2.75	0.01	2.70	2.70	2.70
SYN_LOWER	32	5.8	2.65	2.78	0.01	2.71	2.71	
SYN_UPPER	15	2.63	2.68	2.76	0.01	2.70	2.7	
OB	0	0						2.00

14.6 Compositing

In order to minimize any bias introduced by variations in sample lengths, the capped assays were composited within each mineralized zone. The thickness of the mineralized solids, the proposed block size, and the original sample length were taken into consideration when selecting the composite length.

The intervals defining each mineralized domain were composited to 1.5 m equal lengths with any tail length equally distributed within each lithology. A grade of 0.00 g/t was assigned to missing sample intervals. A total of 2,073 composites were generated.

Table 14-3 summarizes the basic statistics for the raw data and composites.

Table 14-3: Summary Statistics for the Raw Data and Composites

Oxide	Lithological Unit	Raw Assays				Composites			
		No. of Samples	Max Grade (g/t)	Mean Grade (g/t)	COV	No. of Comp.	Max Grade (g/t)	Mean Grade (g/t)	COV
Sc_2O_3	INTSYN	7	368.11	222.48	0.69	5	359.41	222.48	0.69
	OLFESYN	1066	825.18	277.49	0.31	870	825.18	277.49	0.27
	PXFESYN	176	1033.78	284.54	0.72	115	877.33	284.54	0.55
Dy_2O_3	INTSYN	7	123.95	82.51	0.49	5	120.36	82.51	0.49
	OLFESYN	1066	215.00	66.20	0.30	870	210.03	66.20	0.25
	PXFESYN	176	294.96	70.49	0.68	115	169.86	70.49	0.50
La_2O_3	INTSYN	7	1243.17	819.14	0.53	5	1206.72	819.14	0.53
	OLFESYN	1064	3690.00	611.68	0.35	867	3690.00	611.68	0.30
	PXFESYN	176	2967.18	646.12	0.67	115	1630.43	646.12	0.48
Nd_2O_3	INTSYN	7	1224.72	764.41	0.62	5	1188.47	764.41	0.61
	OLFESYN	1066	2100.00	596.11	0.30	870	2100.00	596.11	0.26
	PXFESYN	176	2892.67	633.00	0.73	115	1667.95	633.00	0.54
Pr_2O_3	INTSYN	7	335.88	211.02	0.60	5	325.48	211.02	0.60
	OLFESYN	1066	1380.00	161.01	0.42	870	1380.00	161.01	0.36
	PXFESYN	176	777.08	170.42	0.71	115	426.26	170.42	0.52
Tb_4O_7	INTSYN	7	21.17	13.81	0.54	5	20.56	13.81	0.54
	OLFESYN	1066	92.10	11.64	0.34	870	61.40	11.64	0.29
	PXFESYN	176	52.93	12.32	0.70	115	30.82	12.32	0.51

14.7 Block Model

A block model was established to enclose a sufficiently large volume to host an open pit. The model corresponds to a sub-blocked model in Surpac, rotated 42° clockwise (Y axis oriented at N042° azimuth).

The user block size was defined as 5m x 5m x 5m with a minimal sub-block size of 1.25m x 1.25m x 1.25m. Block dimensions reflect the sizes of mineralized domains and plausible mining methods. All blocks with more than 50% of their volume falling within a selected solid were assigned the corresponding solid block code.

Table 14-4 presents the properties of the block model.

Table 14-5 provides details about the naming convention for the corresponding Surpac solids as well as the rock codes and precedence assigned to each individual solid.

Table 14-4: Block Model Properties

Properties	Y (rows)	X (columns)	Z (levels)
Min. coordinates	6133550	440000	300
Max. coordinates	6135000	441100	600
User block size	5	5	5
Min. block size	1.25	1.25	1.25
Rotation	42	0	0

Table 14-5: Block Model Naming Convention and Rock Codes

Solid Name	Description	Rockcode	Precedence
Air (above topography)	air	1	1
OB	overburden	33	2
PEG	waste	50	5
POMSYN	waste	55	10
PXFESYN	mineralization	101	20
OLFESYN	mineralization	100	25
INTSYN	mineralization	102	30
SYN	waste	60	50
SYN_UPPER	waste	60	50
SYN_LOWER	waste	60	50

14.8 Variography and Search Ellipsoids

3D variography, carried out in Supervisor, yielded the best-fit model along an orientation that roughly corresponds to the strike and dip of the mineralized domains. The variography analysis was inconclusive for INTSYN and PXFESYN due to insufficient information. The variography was also inconclusive in defining the range for the scandium in the Z-axis due to the lack of information. A value of one-third (1/3) of the major axis was arbitrarily applied. Variography results obtained from OLFESYN were then applied to other mineralized domains.

The search ellipsoid was based on the variography study. The interpolation strategy counts two (2) cumulative passes. The first pass corresponds to half (0.5x) of the variography ranges and the second pass is 1x the ranges.

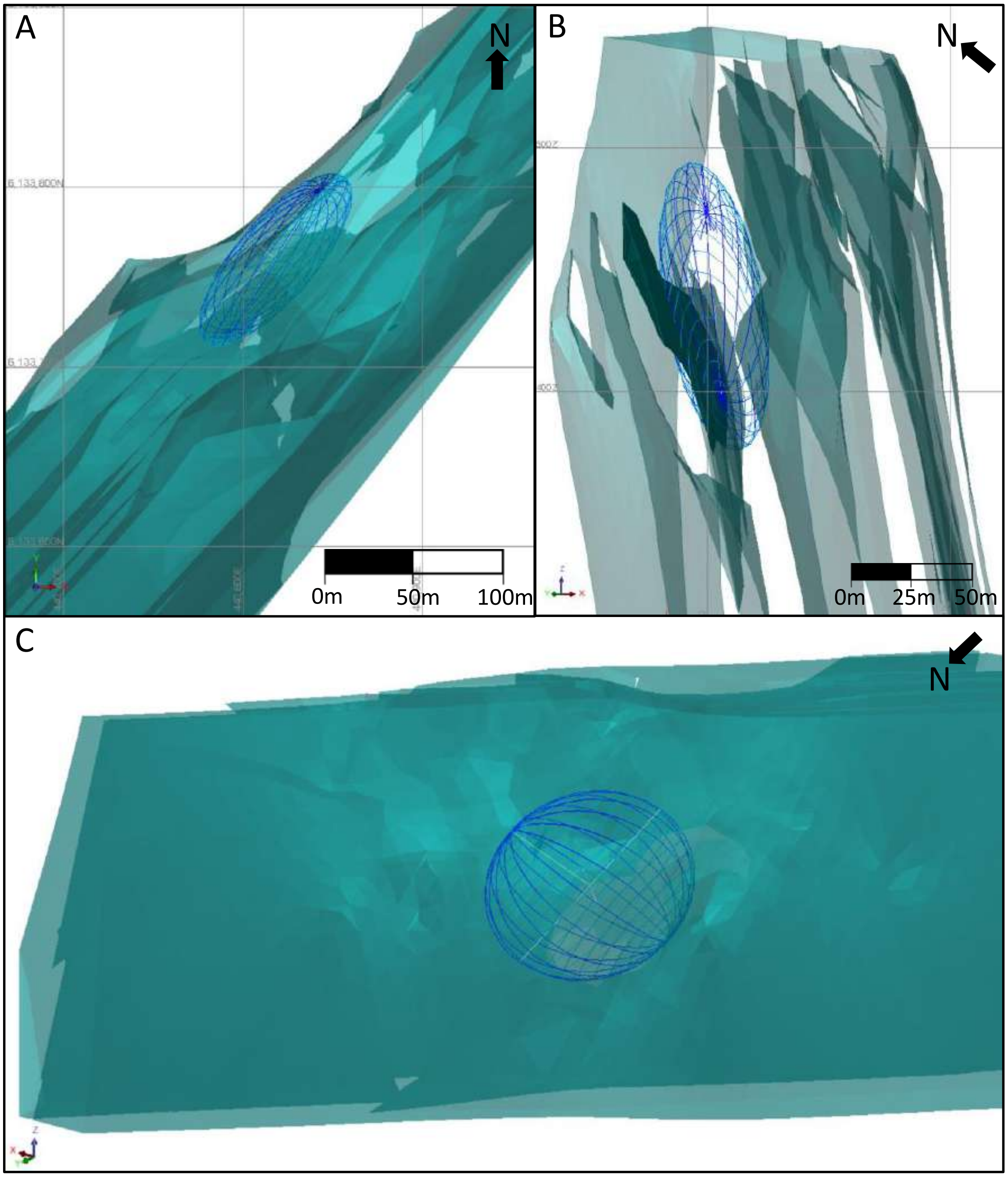
Table 14-6 summarizes the parameters of the ellipsoids used for interpolation.

Figure 14-4 illustrates the shape and range of the scandium search ellipsoids for the first pass.

Table 14-6: Variogram Model Parameters

Oxide	Surpac Coordinates			Model Type	Variogram Components			
	Z	X	Y		Nugget	Range X (m)	Range Y (m)	Range Z (m)
Sc_2O_3	52	-49	-75	Spherical	0.05	120	120	40
Pr_2O_3	34	29	-78	Spherical	0.1	160	140	55
Nd_2O_3	34	29	-78	Spherical	0.05	140	115	60
Tb_4O_7	34	29	-78	Spherical	0.2	145	125	55
Dy_2O_3	34	29	-78	Spherical	0.07	150	135	60
La_2O_3	32	39	-77	Spherical	0.1	155	100	55

Figure 14-4: Views of the Scandium Search Ellipsoids for the First Pass with the OLFESYN Wireframe



Notes: A) Plan view; B) Section view, looking NE; C) Isometric view, looking SW

14.9 Grade Interpolation

The variography study provided the parameters for interpolating the grade model using capped composites. The interpolation was run on point area workspaces extracted from the composite datasets (flagged by zone) in Surpac. A cumulative 2-pass search was used for the resource estimate. Pass 1 corresponds to half (0.5x) the variography ranges and Pass 2 is 1x the variography ranges for blocks not estimated during the first pass. The interpolation profiles were applied to each mineralized zone using hard boundaries.

Several models were produced using the nearest neighbour ("NN"), inverse distance square ("ID2") and ordinary kriging (OK) interpolation methods to choose the method that best respects the raw assay and composite grade distribution for each metal. Models were compared visually (on sections, plans and longitudinal views), statistically, and with swath plots. The focus was to limit the smoothing effect to preserve local grade variations while avoiding the smearing of high-grade values. The ID2 method was selected for the final resource estimate.

For Dy2O3, La2O3, Pr2O3, Nd2O3, and Tb2O3, high-grade values did not show continuous distributions. The interpolation distance for the high-grade values was restricted to avoid high-grade smearing and grade over-estimation in the block model. The top grades were defined by studying the grades frequency histogram for each element. The interpolation restriction distance was defined as one-third of the major variogram range. Table 14-7 presents the restriction parameters.

The strategy and parameters for the grade estimation are summarized in Table 14-8.

Table 14-7: High Grade Interpolation Restriction Parameters

Oxide	Top Grade (g/t)	Interpolation Restriction Distance (m)
Sc_2O_3	-	-
Pr_2O_3	300	27
Nd_2O_3	1000	23
Tb_4O_7	20	24
Dy_2O_3	80	25
La_2O_3	1000	26

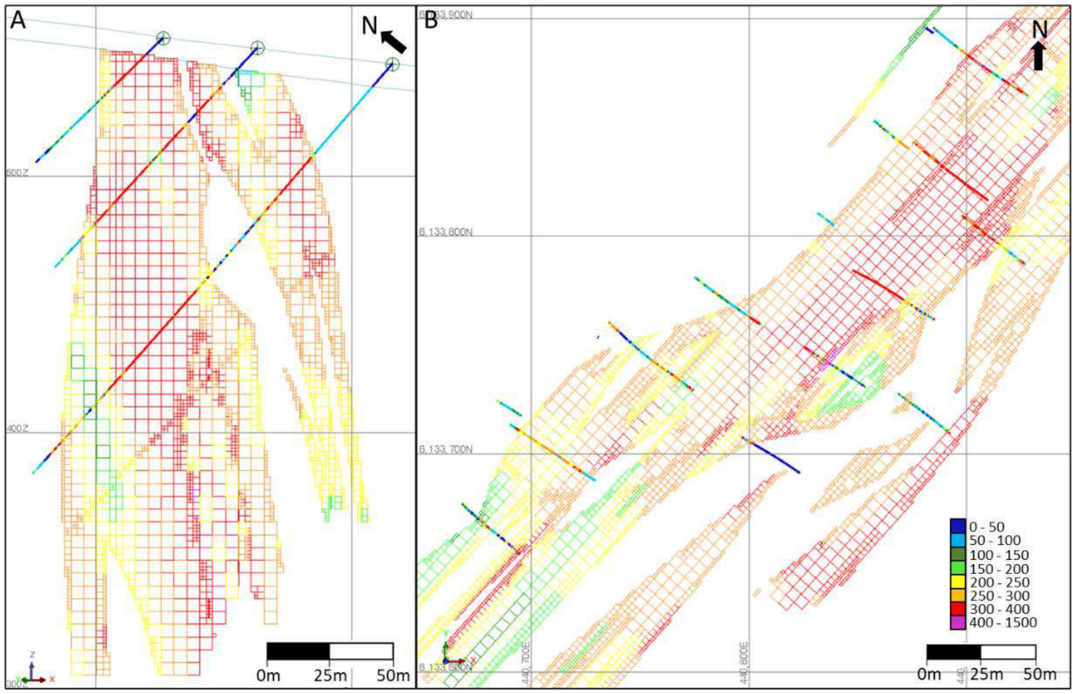
Table 14-8: Interpolation Strategy

Pass	Number of composites		
	Min	Max	Max per hole
1	9	18	4
2	5	18	4

14.10 Block Model Validation

Block model grades and composite grades were visually compared on sections, plans and longitudinal views for both densely and sparsely drilled areas. No significant differences were observed, and a generally good match was noted in the grade distribution without excessive smoothing in the block model. The process confirmed that the block model honours the drill hole composite data (Figure 14-5).

Figure 14-5: Scandium Grade Distribution



Notes: A) Section view, looking NE (+/- 25 m); B) Plan view (+/- 25 m).

The OK and NN models were used to check for local bias in the models. The OK model matches well with the ID2 model. The differences in the high-grade composite areas are within acceptable limits. The trend and local variation of the estimated ID2 and OK models were compared to the NN models and the composite data using swath plots in three directions (North, East, and Elevation). The ID2, NN and OK models show similar trends in grades, with the expected smoothing for each method compared to the composite data.

Table 14-9 compares the global block model mean for three (3) interpolation scenarios (OK, ID^2, and NN) and the composite grades for each metal. Generally, the comparison between composite and block grade distribution did not identify any significant issues.

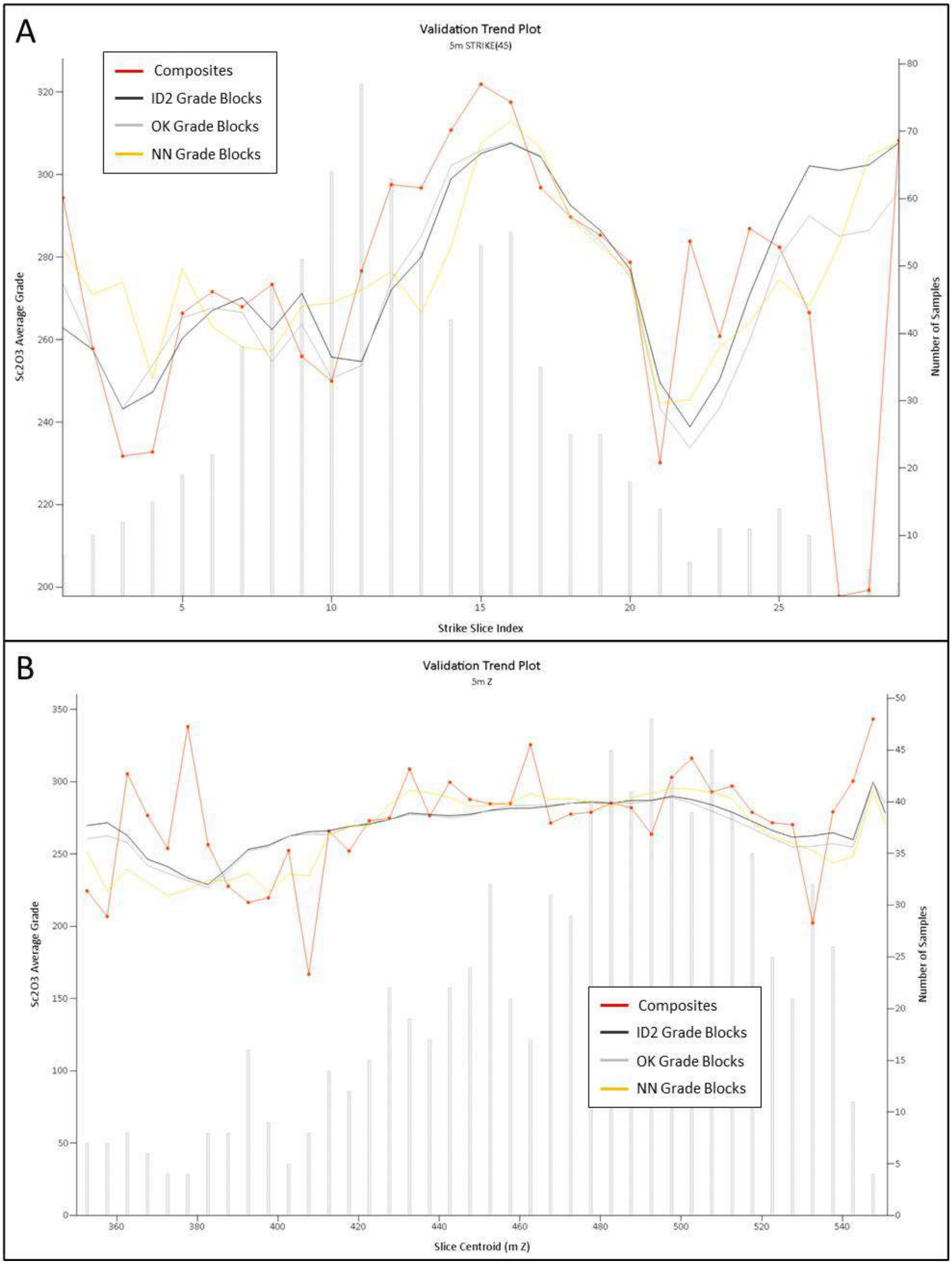
Figure 14-6 shows an example of the swath plot used to compare the block model grades to the composite grades. In general, the model correctly reflects the trends shown by the composites, with the expected smoothing effect.

Table 14-9: Comparison of Block Model and Composite Mean Grades

Oxide	Property	Composite	ID2 Model	OK Model	NN Model
Number		870	169547	169547	169547
Sc_2O_3	Mean (g/t)	277.573	276.036	273.987	276.180
	COV	0.283	0.147	0.162	0.318
Dy_2O_3	Mean (g/t)	66.460	65.700		70.924
	COV	0.271	0.158		0.452
Tb_4O_7	Mean (g/t)	11.684	11.691		12.775
	COV	0.301	0.193		0.598
La_2O_3	Mean (g/t)	615.710	613.790		
	COV	0.314	0.224		
Nd_2O_3	Mean (g/t)	598.914	596.474		
	COV	0.279	0.163		

Note: Blocks classified as Indicated only; No cut-off grade applied.

Figure 14-6: Swath Plots Comparing Interpolation Methods to the DDH Composites for Scandium



Notes: A) NW-SE cross section; B) Elevation.

14.11 Mineral Resource Classification

No Measured resources were defined.

Indicated resources were defined for blocks estimated in the first pass (minimum of 3 DDH) and at the boundaries, within 30 m of a drill hole, or at mi-distance to the last drill hole meeting the indicated criteria. Inferred resources were defined for the remaining interpolated blocks (minimum 2 DDH).

The resource category was assigned using clipping boundaries. In some cases, isolated blocks were upgraded or downgraded to homogenize the model with respect to the geological and grades continuity.

14.12 Cut-off Grade for Mineral Resources

Under CIM Definition Standards, mineral resources should have “reasonable prospects of eventual economic extraction”. Given the nature of the mineralization (polymetallic content, large zone width and widespread grade distribution), the cut-off grade of the Project is expressed as net smelter return (“NSR”) and the assumptions made for its calculation apply to a potential open pit scenario.

An NSR value was calculated for each element in the block model with the following formula: Metal price (CA$/t) x Block Value (g/t) x recovery (%) /10^3. An NSR total was then calculated using the following formula: NSR Total (CA$/t) = NSR Sc_2O_3 + NSR La_2O_3 + NSR Pr_2O_3 + NSR Nd_2O_3 + NSR Tb_4O_7 + NSR Dy_2O_3 - Concentrate Transportation Cost.

Detailed parameters used for each element are described in Table 14-10.

Table 14-10: Input Parameters used to Calculate the NSR Block Model Attributes

Attribute	Metal Price (CA$/kg)	Block Value (g/t)	Recovery (%)	Concentrate Transportation Cost (CA$/t ore milled)
NSR_Sc_2O_3	1875	Id2_Sc_2O_3	0.76	17.01
NSR_La_2O_3	0.6*	Id2_La_2O_3	0.63	17.01
NSR_Pr_2O_3	36*	Id2_Pr_2O_3	0.63	17.01
NSR_Nd_2O_3	37*	Id2_Nd_2O_3	0.63	17.01
NSR_Tb_4O_7	483*	Id2_Tb_4O_7	0.63	17.01
NSR_Dy_2O_3	155*	Id2_Dy_2O_3	0.63	17.01

Notes: * Prices were discounted by 70% as the Project assumes sales as a bulk TREO+Y concentrate

For the 2021 MRE, an NSR cut-off of 110.8 CA$/t has been selected based on the assumptions described in Table 14-11. The selection of reasonable prospective parameters, which assume that some or all of the estimated resources could potentially be extracted, is based on an open-pit mining scenario (470,000 tpy). This is also based on the assumption of onsite mineral concentrate production and transport of the concentrate from the mine site to a processing plant.

Table 14-11: Input Parameters Used to Calculate the Open-pit Cut-off Grade

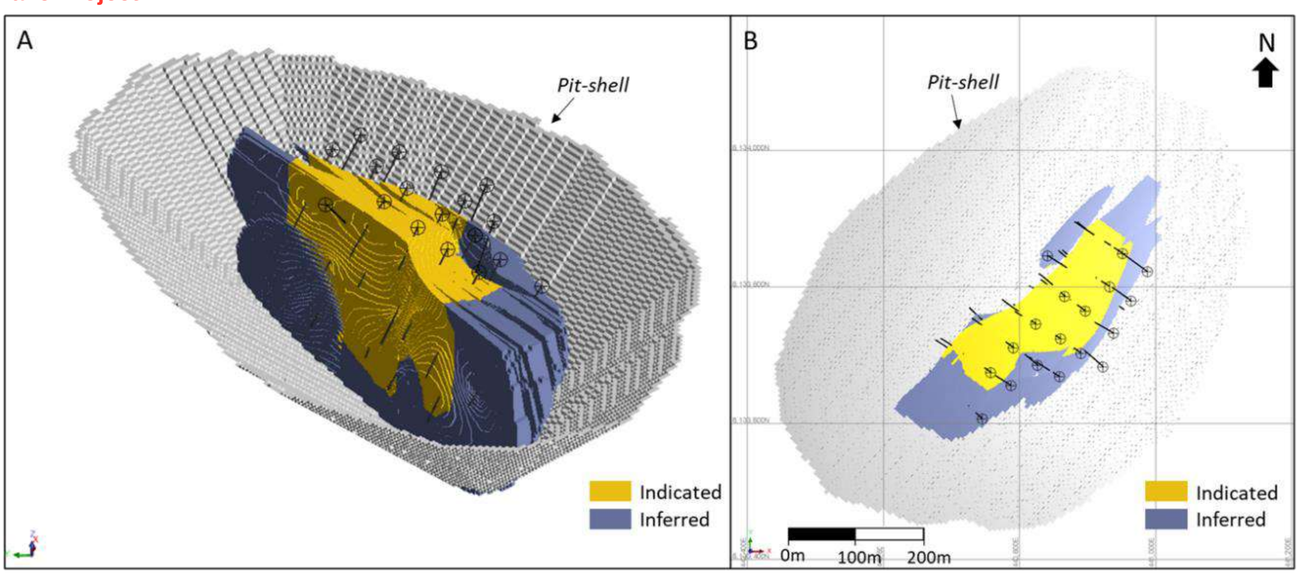
Parameters	Unit	Value
Sc_2O_3 price	US$/kg	1,500
La_2O_3 price*	US$/kg	0.6
Pr_2O_3 price*	US$/kg	29
Nd_2O_3 price*	US$/kg	29
Tb_4O_7 price*	US$/kg	386
Dy_2O_3 price*	US$/kg	124
Processing cost	CA$	14.89
Scandium recovery to high-grade scandium oxide product	%	76
REE recovery to mixed REE carbonate	%	63
Transportation cost (transport of concentrate from mine site to processing plant)	CA$	17.01
G&A	CA$	7.19
Refining and selling costs	CA$	88.71
USD:CAD exchange rate		1.25

Notes: *Prices were discounted by 70% as the Project assumes sales as a bulk TREO+Y concentrate

14.13 Mineral Resource Estimate

The authors believe that the current mineral resource estimate can be classified as Indicated and Inferred mineral resources based on geological and grade continuity, data density, search ellipse criteria, drill hole spacing and interpolation parameters. The authors also believe that the requirement of "reasonable prospects for eventual economic extraction" has been met by having a cut-off grade based on reasonable inputs amenable to a potential open-pit extraction scenario.

Figure 14-7: Isometric (A) and Plan View (B) showing the Pit-Shell and the Classified Mineral Resources of the Crater Lake Project



The 2021 MRE is considered reliable and based on quality data and geological knowledge. The estimate follows CIM Definition Standards.

Table 14-12 displays the results of the 2021 MRE for the Project at the official 110.8 CA$/t NSR cut-off.

Table 14-12: 2021 Crater Lake Project Mineral Resource Estimate for an Open Pit Scenario

Category	NSR Cut-off (CA$/t)	Tonnage (Mt)	NSR Total (CA$/t)	Sc_2O_3 (g/t)	Dy_2O_3 (g/t)	La_2O_3 (g/t)	Nd_2O_3 (g/t)	Pr_2O_3 (g/t)	Tb_4O_7 (g/t)
Indicated	110.8	7.3	413	282.01	65.72	605.82	595.78	160.41	11.65
Inferred	110.8	13.2	386	264.24	62.24	568.63	573.04	154.02	11.13

Notes to accompany the Mineral Resource Estimate:

4. The independent and qualified persons for the mineral resource estimate, as defined by NI 43 101, are Marina Iund, P.Geo. (InnovExplo Inc.), Paul Daigle, P.Geo. (InnovExplo Inc. associate) and Carl Pelletier, P.Geo. (InnovExplo Inc.). The effective date of the estimate is September 17, 2021.
5. These mineral resources are not mineral reserves, as they do not have demonstrated economic viability. The mineral resource estimate follows current CIM Definition Standards.
6. The results are presented in situ and undiluted and considered to have reasonable prospects of economic viability.
7. The estimate encompasses three mineralized domains using the grade of the adjacent material when assayed or a value of zero when not assayed.
8. High-grade capping supported by statistical analysis was done on raw assay data before compositing: La_2O_3 (3690 g/t), Pr_2O_3 (1380 g/t), Nd_2O_3 (2100 g/t), Dy_2O_3 (215 g/t). No capping was applied to Sc_2O_3 and Tb_4O_7.
9. The estimate was completed using a sub-block model in GEOVIA SURPAC 2021 with user block size of 5m x 5m x 5m and minimum block size of 1.25m x 1.25m x1.25m. Grades interpolation was obtained by ID2 using hard boundaries.
10. Bulk density values were applied by lithology (g/cm3): INTSYN, OLFESYN = 3.13; PXFESYN = 2.91; SYN = 2.7; POMSYN = 2.77; PEG = 2.63 and OB = 2.0.
11. The mineral resource estimate is classified as indicated and inferred. The Indicated mineral resource category is defined with a minimum of three (3) drill holes in areas where the drill spacing is less than 60 m, and reasonable geological and grade continuity have been demonstrated. The Inferred category is defined with a minimum of two (2) drill holes in areas where the drill spacing is less than 120 m, and reasonable geological and grade continuity have been demonstrated. Clipping boundaries were used for classification based on those criteria.
12. The mineral resource estimate is pit-constrained with a bedrock slope angle of 45° and an overburden slope angle of 30°. It is reported at a NSR cut-off of 110.8 CA$/t .The NSR cut-off was calculated using the following parameters: processing cost = CA$14.89; transportation cost (concentrate transportation from mine site to processing plant): CA$7.19; refining and selling costs = CA$ 88.71; Sc_2O_3 price = US$1,500/kg; La_2O_3 price = US$0.6/kg; Pr_2O_3 price = US$29/kg; Nd_2O_3 price = US$29/kg; Tb_4O_7 price = US$386/kg; Dy_2O_3 price = US$124/kg; USD:CAD exchange rate = 1.25; scandium recovery to high grade scandium oxide product = 76.0%; REE recovery to mixed REE carbonate = 63.0%. The cut-off grades should be re-evaluated in light of future prevailing market conditions (metal prices, exchange rates, mining costs etc.).
13. The number of metric tons was rounded to the nearest thousand, following the recommendations in NI 43 101 and any discrepancies in the totals are due to rounding effects.
14. The authors are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, or marketing issues, or any other relevant issue not reported in the Technical Report, that could materially affect the Mineral Resource Estimate.

Table 14-13 shows the NSR cut-off sensitivity analysis of the 2021 MRE. The homogeneity of the grade of the elements across the deposit as well as the high-grade nature of it makes the mineral resource low sensitive to variation of NSR cut-off.

The reader should be cautioned that the numbers provided in should not be interpreted as a mineral resource statement. The reported quantities and grade estimates at different NSR cut-off are presented in-situ and for the sole purpose of demonstrating the sensitivity of the resource model to the selection of a reporting NSR cut-off.

Table 14-13: Cut-off grade Sensitivity for the Crater Lake Project

Cut-off NSR ($/t)	Tonnage (t)	NSR total ($/t)	Sc_2O_3 (g/t)	Dy_2O_3 (g/t)	La_2O_3 (g/t)	Nd_2O_3 (g/t)	Pr_2O_3 (g/t)	Tb_4O_7 (g/t)
Indicated Resource								
90	7,316,375	413	281.98	65.71	605.77	595.73	160.39	11.65
110.8	**7,315,544**	**413**	**282.01**	**65.72**	**605.82**	**595.78**	**160.41**	**11.65**
130	7,312,414	413	282.09	65.73	605.96	595.94	160.45	11.65
150	7,308,770	413	282.18	65.75	606.07	596.12	160.5	11.65
170	7,301,881	413	282.34	65.78	606.2	596.33	160.55	11.66
Inferred Resource								
90	13,161,580	386	264.2	62.23	568.6	572.9	153.98	11.13
110.8	**13,158,383**	**386**	**264.24**	**62.24**	**568.63**	**573.04**	**154.02**	**11.13**
130	13,149,714	386	264.36	62.28	568.77	573.35	154.1	11.14
150	13,135,305	387	264.53	62.32	568.95	573.76	154.22	11.15
170	13,104,812	387	264.88	62.4	569.32	574.44	154.42	11.16

15 MINERAL RESERVE ESTIMATES

This Item is not required for a Preliminary Economic Assessment Technical Report.

16 MINING METHODS

16.1 Introduction

The mining methods for the Imperial mining group Crater Lake Scandium Project have been prepared by WSP employee Zakaria Moctar, Eng., who stands as the qualified person. All work related to pit optimization for the report was performed using Pseudoflow tool in Whittle Software and verified by Pseudoflow tool in Deswik. Whittle and Deswik are commercially available software. The pit design was performed by Deswik open pit tool. The mine planning was performed using Deswik Sched for Open Pit Metals. The deposit is suitable for an open pit as the mineralization is shallow and material quantities are significant.

Item 14 of this report, prepared by Innov Explo, indicates Crater Lake Mineral Resources breakdown as 7.3 Mt Indicated, and 13.1 Mt Inferred for a total of 20.4 Mt resource materials. Since this project is at a PEA level, both Indicated and Inferred resources can be considered for pit optimization. WSP performed a pit optimization with the same economic parameters used for Mineral Resources for comparison purposes as an internal validation prior to economic pit optimization. WSP was able to confirm that the Mineral Resources for Crater Lake are 7.3 Mt Indicated and 13.1 Mt Inferred for a total of 20.4 Mt in-pit resource materials. Considering a targeted mill throughput of 421,700 tonnes /year, the potential mine life is 48.3 years. However, for the purpose of the PEA study and financial considerations to comply with standard project durations, IPG agreed to limit the duration of the project to 25 years. Therefore, the final pit shell selection was driven by economic parameters, such as NPV and a mine life of up to 25 years.

From the final selected pit shell, a final pit design was created adhering to geotechnical parameters and mine design criteria for the Crater Lake deposit. For mine planning purposes three pushback phases were created: a Starter Pit, Middle Pit and Ultimate Pit.

For construction purposes, one year of pre-production was planned to remove a total of 1 Mt of waste rock, even though mineralized materials are found at the outcrop of the deposit and are easily accessible.

Mine production to feed the processing plant will begin in Project Year 1 and will average 426,100 t of mineralized material per year until Year 24 and decrease to 351,539 t at year 25. The life of the mine is 25 years. Mineralized materials will come from the Starter Pit in Year 1 and Year 2, then from the Middle Pit from Year 2 to Year 9, and finally from the Ultimate Pit from Year 9 to Year 25.

The average total mine production including overburden, waste rock and mineralized materials is 1.2 Mt/year with a peak of 2.7 Mt at Year 11 due to the transition from the Middle to the Ultimate pit. The mine schedule was prepared to smooth the total material movement as much as possible to maintain a relatively constant mine equipment fleet. Total overburden material is 2.3 Mt, excavated from Year 1 to Year 10. Total waste rock material and total mineralized material are respectively 18.9 Mt and 10.6 Mt spread over the mine life. The average strip ratio is 2.0 tonnes waste per tonne of mineralized material.

16.2 Data Acquisition

To ensure that correct inputs and reasonable assumptions have been used for mine planning, costs provided by IPG were validated by reviewing historical and current costs at similar mines. The proposed costs were communicated and reviewed by appropriate individuals in WSP before use.

16.2.1 Topographic Surface

The topographic surface was provided by IPG as a .dtm file, dated September 13, 2021. The topographic file used was named Topography.dtm. The topographic surface above the deposit culminates at elevation Z=575 in the northwest and gradually descends to level Z=525 in the southwest and continues to descend until it reaches the surface of Crater Lake.

16.2.2 Resource Block Model

WSP received the original block model from InnovExplo. The original block model file is named bm_craterlake_final.mdl (Surpac format). The block model is a sub-cell block with parent block sizes of 5 m x 5 m x 5m (X, Y, Z). Depending on the geology of the deposit, some parent blocks are subdivided into sub-cell blocks of 2.5 m x 2.5 m x 2.5m (X, Y, Z) or 1.25 m x 1.25 m x 1.25 m (X, Y, Z).

The resource model has numerous attributes such as density (density), rock type category (rockcode), NSR (nsr_tot), Scandium oxide Sc2O3 (id2_sco), and five rare earth elements: Dysprosium Dy_2O_3 (id2_dy2o), Lanthanum La_2O_3 (id2_la2o), Neodymium Nd_2O_3 (id2_nd2o), Praseodymium Pr_2O_3 (id2_pr2o), and Terbium Tb_4O_7 (id2_tb4o)

The attribute (class) identifies the classification (Measured, Indicated, Inferred, unclassified) for each block.

The detailed information regarding origin coordinates and rotation is presented in Table 16-1. The attributes summary of the block model, as provided in the original bock model file, named "bm_craterlake_final.mdl," is presented in Table 16-2.

Table 16-1: Block Model Local Coordinates

Type	Y (m)	X (m)	Z (m)
Minimum Coordinates	440,000	6,133,550	300
Maximum Coordinates	441,100	6,135,000	600
User Block Size	5	5	5
Min. Block Size	1.25	1.25	1.25
Rotation (Degrees)	42	0	0

Table 16-2: Block Model Attributes

Attribute Name	Type	Decimals	Background
class	Integer	-	99
class_temp	Integer	-	0
closest_dist	Real	3	-99
density	Float	2	2.7
id2_dy2o3	Float	2	0
id2_la2o3	Float	2	0
id2_nd2o3	Float	2	0
id2_pr2o3	Float	2	0
id2_sc2o3	Float	2	0
id2_tb4o7	Float	2	0
mining	Integer	-	0
nn_dy2o3	Float	2	0
nn_sc	Float	2	0
nn_tb4o7	Float	2	0
nsr_dy2o3	Float	2	0
nsr_ing	Float	2	0
nsr_la2o3	Float	2	0
nsr_nd2o3	Float	2	0
nsr_pr2o3	Float	2	0
nsr_sc2o3	Float	2	0
nsr_tb4o7	Float	2	0
nsr_total	Float	2	0
ok_sc_test3	Float	2	0
pass_dy2o3	Integer	-	0
pass_la2o3	Integer	-	0

Attribute Name	Type	Decimals	Background
pass_nd2o3	Integer	-	0
pass_nn	Integer	-	0
pass_pr2o3	Integer	-	0
pass_sc2o3	Integer	-	0
pass_scnn	Integer	-	0
pass_tb4o7	Integer	-	0
rockcode	Integer	-	99

16.2.3 Density

The specific gravity was originally coded in the block model for each lithological unit, and the median result used to code the bulk density. Table 16-3 presents the density for each rock category.

Table 16-3: Densities per Rock Category

Material Type	Layer/Code	Density
Overburden	Rockcode 33	2.00
Waste Rock	PEG - Rockcode 50	2.63
	POMSYN - Rockcode 55	2.77
	SYN - Rockcode 60	2.70
	UNNAMED -Rockcode 99	2.70
Mineralized Material Zones*	OLFESYN - Rockcode 100	3.13
	PXFESYN - Rockcode 101	2.91
	INTSYN - Rockcode 102	3.13

16.2.4 Swell Factor

Based on WSP's experience on similar projects, it was assumed that a swell factor of 1.3 (30%) will be applied to blasted rock materials.

16.2.5 Moisture Content

No moisture content was assumed in this PEA. However, it is recommended to use an appropriate moisture content for future engineering studies to add more precision for haul truck requirements since all mined materials are hauled in wet condition.

16.2.6 Summary of Parameters

Table 16-4 summarizes the parameters used in this study and the respective data sources used to inform the pit design and the construction of rock and overburden dumps.

Table 16-4: Summary of Parameters and Respective Data Sources

Parameter	Description	Data Source
Resource Block Models	Provided by InnovExplo	The model file is bm_craterlake_final.mdl
Planning Block Models	The mine model was cloned from the resource block model, then additional attributes were coded by WSP such as economic attributes for pit optimization and an updated NSR (NSR_COT), RCODE	bm_craterlake_final_all_fields_eco
Geotechnical Parameters	Geotechnical parameters are used to determine the slope angles and wall configurations required for guiding the pit optimization, the design of the pushbacks, and the final pit shell. WSP has considered the geotechnical parameters that were issued to IPG by WSP on the Technical note prepared by Hugo Fisette on February 3rd, 2022.	
Mining Costs	WSP reviewed mining costs provided by IPG and compared them with similar mines before use.	211-11382-00 DC-001_PitOptimization - Rev 3.xlsx
Process Costs	WSP has used the Process costs as an input from IPG.	211-11382-00 DC-001_PitOptimization - Rev 3.xlsx
General and Administrative Costs (G&A)	WSP has used the General and Administrative costs as an input from IPG.	211-11382-00 DC-001_PitOptimization - Rev 3.xlsx
Refining and Selling Prices	WSP has used the Refining and Selling Prices as an input from IPG.	211-11382-00 DC-001_PitOptimization - Rev 3.xlsx
Process Recoveries	WSP has used the Process Recoveries as an input from IPG.	211-11382-00 DC-001_PitOptimization - Rev 3.xlsx
Royalties	WSP did not consider any royalty for pit optimization. However, a 2% royalty was used in the financial model as agreed with IPG.	211-11382-00 DC-001_PitOptimization - Rev 4.xlsx
Site Layout	Organization and placement of site infrastructure – General Arrangement	211-11382-00-200-G-101_CL_In progress 20220120.pdf
Initial Topography	A digital terrain model representing the starting surface for mining, provided by IPG	Topography.dtm
Pit and Dump Design Criteria	Used to guide the design and construction of rock and overburden dumps	211-11382-00 DC-001_PitOptimization - Rev 4.xlsx

16.2.7 Geotechnical Pit Slope Parameters

WSP completed the PEA study without geotechnical or geomechanical data, besides RQD (Rock Quality Designation) values obtained during exploration core logging. The geotechnical design parameters used in the pit slope design are therefore based on industry standards and typical slopes corresponding to the lithological units were used.

WSP prepared a technical note on geotechnical review and recommendations and submitted the document to IPG on February 3, 2022.

The overburden material is mainly constituted of glacial till at variable thicknesses, ranging from 0 to approximately 10 meters in the pit area. Based on a typical friction angle of 40 degrees for the till, the overburden slope angle is limited to 30 degrees, which would give a Safety Factor of 1.5, adequate for long-term stability.

For the pit excavated in bedrock, the parameters are shown in Table 16-5.

Table 16-5: Slope Angles Applied to Pit Design

Item	Value
Bench Face Angle-BFA (º)	75
Maximum Bench Height (m)	20
Minimum Bench Width (m)	8.5
Maximum Inter-Ramp Angle-IRA (º)	55
Overall Slope Angle-OSA (º)	45

The bench configuration has been based on the modified Ritchie criterion, an empirical relationship between the bench height and the catch bench width, leading to a minimum allowable bench width of 8.5 metres for a 20-metre-high double bench configuration. A bench face angle of 75 degrees is recommended for the bedrock slopes. The aforementioned bench configuration leads to a maximum Inter-Ramp Angle (IRA) of 55 degrees, which reflects a relatively aggressive design approach. Consequently, the Overall Slope Angle (OSA) is to be limited to 45 degrees, by considering the insertion of the haulage ramps spiraling down the pit walls and by adding geotechnical berms if and where necessary.

16.3 Hydrogeology and Hydrology

The scope of this PEA does not cover hydrogeology and hydrology study.

16.4 Economic Pit shell Optimization

The pit optimization process consists of utilizing economic criteria including costs and revenues as well as geotechnical considerations to determine to what extent the deposit can be mined profitably.

The pit optimization was determined using Whittle with the Pseudoflow algorithm and verified by running Deswik with the Pseudoflow algorithm, using the same economic criteria (mining costs, processing costs, revenue per block).

In order to comply with NI 43-101 guidelines regarding the standards of disclosure for mineral projects; and considering that this project is at a PEA level, the calculation of revenue per block was applied to all resource categories, including Indicated and Inferred blocks. The resource block model does not contain any blocks classified as Measured Resources.

Table 16-6 summarizes the criteria used for pit optimization.

Table 16-6: Pit Optimization Criteria

SECTION	ITEM	UNIT	VALUE
1	**Reference Block Model (bm_craterlake_final.mdl in Surpac format)**		
	Parent Block Size (X, Y, Z)	(m, m, m)	(5, 5, 5)
	Sub-Cell Block Size - Child level 1 (X, Y, Z)	(m, m, m)	(2.5, 2.5, 2.5)
	Sub-Cell Block Size - Child level 2 (X, Y, Z)	(m, m, m)	(1.25,1.25,1.25)
2	**Density / Specific gravity**		
	Overburden - rockcode 33	in-situ metric tonnes/ m³	2.00
	Waste - rockcode 50 - PEG	in-situ metric tonnes/ m³	2.63
	Waste - rockcode 55 - POMSYN	in-situ metric tonnes/ m³	2.77
	Waste - rockcode 60 - SYN	in-situ metric tonnes/ m³	2.70
	Waste - rockcode 99	in-situ metric tonnes/ m³	2.70
	Ore - rockcode 100 - OLFESYN	in-situ metric tonnes/ m³	3.13
	Ore - rockcode 101 - PXFESYN	in-situ metric tonnes/ m³	2.91
	Ore - rockcode 102 - INTSYN	in-situ metric tonnes/ m³	3.13
3	**Geotechnical Parameters / Optimization Slopes for Whittle/Deswik or other software**		
	Overal Slope Angle for the entire deposit	°	45
4	**Mine design criteria**		
	Cut-off grade - not used in the Pit Optimization		
	Mining Recovery	%	95
	Mining Dilution	%	5
	Dilution Grade for Scandium (Sc2O3)	g/t	0
	Dilution Grade for Dysprosium (Dy2O3)	g/t	0
	Dilution Grade for Lanthanum (La2O3)	g/t	0
	Dilution Grade for Neodymium (Nd2O3)	g/t	0
	Dilution Grade for Praseodymium (Pr2O3)	g/t	0
	Dilution Grade for Terbium (Tb4O7)	g/t	0
5	**Operating costs/parameters**		
	Mining Cost (Ore)	CAD$/tonne of ore	6.78
	Mining Cost (Waste)	CAD$/tonne of ore	6.78
	Mining Cost (Overburden)	CAD$/tonne of ore	2
	Mining Rehabilitation Cost (Waste)	US$/t waste	0
	Dewatering Cost	U$/t (all material types)	0
	Discounted Rate	%	8
6	**Processing costs, G&A, etc.**		
MET 02	Processing cost	CAD$/t milled	15.32
MET 02	Transportation cost (concentrate) from mine site to processign plant	CAD$/t milled	21.95
MET 02	General and administration costs (G&A)	CAD$/t milled	5.96
	Total Processing costs for MET02	**CAD$/t milled**	**43.23**
7	**Recovery**		
	Recovery for Scandium (Sc2O3)	%	76
	Recovery for Dysprosium (Dy2O3)	%	63
	Recovery for Lanthanum (La2O3)	%	63
	Recovery for Neodymium (Nd2O3)	%	63
	Recovery for Praseodymium (Pr2O3)	%	63
	Recovery for Terbium (Tb4O7)	%	63
8	**Selling Prices**		
	Selling Price for Scandium (Sc2O3)	CAD$/kilogram	1875
	Selling Price for Dysprosium (Dy2O3)	CAD$/kilogram	203
	Selling Price for Lanthanum (La2O3)	CAD$/kilogram	0.5
	Selling Price for Neodymium (Nd2O3)	CAD$/kilogram	36
	Selling Price for Praseodymium (Pr2O3)	CAD$/kilogram	36
	Selling Price for Terbium (Tb4O7)	CAD$/kilogram	606
	Exchange Rate US/CAD		1.25
9	**Refining and Selling costs**		
	Refining and Selling Cost for Scandium (Sc2O3)	CAD$/t ore milled	88.71
	Refining and Selling Cost for Dysprosium (Dy2O3)	CAD$/t ore milled	108.19
	Refining and Selling Cost for Lanthanum (La2O3)	CAD$/t ore milled	108.19
	Refining and Selling Cost for Neodymium (Nd2O3)	CAD$/t ore milled	108.19
	Refining and Selling Cost for Praseodymium (Pr2O3)	CAD$/t ore milled	108.19
	Refining and Selling Cost for Terbium (Tb4O7)	CAD$/t ore milled	108.19

All financial figures used for the pit optimization are in Canadian dollars. WSP analysed the costs provided by IPG and carried out an internal verification to validate and reflect mining costs based on equipment, fuel, inflation and manpower costs in the northern Quebec region. The geotechnical parameters used are the same as those mentioned in Item 16.3 of this report.

WSP ran the pit optimization based on the same parameters used for Mineral Resources Estimates and found that the final pit shell with revenue factor 1 is identical to the pit shell provided by InnovExplo for Mineral Resources.

All parameter costs listed on Table 16-6 were used solely to generate economic pit shells and should not be confused with operating costs that will be used to develop mining cost estimates in Item 21.

By applying cost parameters and the above-mentioned reference selling prices of scandium oxide and the associated five rare earth elements the pit optimization exercise generated a series of 22 shells with revenue factors ranging from 0.113 to 1.5 (Table 16-7).

Table 16-7: Pit Optimization 22 Shells Revenue Factors

Pit	Rev. Factor	Rock* Tonnes	Mineralized** Tonnes	Strip Ratio g/t	SC2O Grade g/t	DY2O Grade g/t	LA2O Grade g/t	ND2O Grade g/t	TB4O Grade g/t	PR2O Grade g/t	NSRT Grade CAD/t
1	0.113	424,282	243,187	0.74	307.91	68.81	613.17	610.45	12.09	161.91	451.11
2	0.118	713,409	418,016	0.71	303.27	68.14	609.04	605.25	12.01	160.81	444.25
3	0.123	1,096,113	628,743	0.74	298.88	67.36	605.12	599.97	11.92	160.04	437.75
4	0.128	1,739,911	929,763	0.87	293.98	66.35	605.70	597.68	11.86	160.03	430.60
5	0.133	3,339,105	1,692,862	0.97	288.25	65.33	601.90	589.05	11.70	158.78	422.05
6	0.138	5,263,717	2,529,040	1.08	284.88	65.09	602.99	588.22	11.65	158.75	417.19
7	0.143	7,498,057	3,392,554	1.21	283.19	65.23	605.31	589.98	11.66	159.42	414.86
8	0.148	9,683,035	4,172,708	1.32	281.59	65.24	606.47	590.90	11.66	159.68	412.60
9	0.153	12,545,615	5,141,517	1.44	279.56	64.91	604.43	588.67	11.60	159.03	409.60
10	0.158	15,831,561	6,245,226	1.53	276.72	64.34	600.60	584.04	11.50	157.79	405.32
11	0.163	18,531,978	7,109,097	1.61	274.69	63.89	596.07	580.09	11.42	156.73	402.25
12	0.168	22,758,922	8,360,361	1.72	272.31	63.42	591.24	575.85	11.33	155.63	398.66
13	0.173	25,839,778	9,202,069	1.81	271.02	63.11	587.97	573.35	11.27	154.84	396.70
14	0.178	31,927,862	10,769,711	1.96	268.93	62.61	582.70	569.08	11.17	153.54	393.50
15	0.183	38,691,298	12,338,202	2.14	267.77	62.25	579.21	566.13	11.10	152.58	391.70
16	0.188	43,317,617	13,371,278	2.24	266.89	62.05	577.22	564.45	11.06	152.07	390.37
17	0.2	50,714,787	14,915,349	2.4	265.44	61.78	573.42	562.31	11.01	151.42	388.19
18	0.4	88,662,798	19,815,179	3.47	258.37	60.59	556.76	554.11	10.79	148.97	377.69
19	0.6	97,354,942	20,271,331	3.8	257.79	60.48	555.15	553.54	10.78	148.84	376.82
20	0.8	100,745,605	20,378,809	3.94	257.70	60.45	554.45	553.42	10.77	148.82	376.70
21	1	102,122,637	20,413,854	4	257.65	60.44	554.24	553.32	10.77	148.81	376.61
22	1.5	103,902,457	20,442,739	4.08	257.62	60.44	553.94	553.12	10.77	148.79	376.57

*Rock tonnes = Overburden + Waste Rock + Mineralized Material (Ind. +Inf.)
** Mineralized tonnes = Mineralized Material (Ind. +Inf.)

The pit shell of revenue factor 1, in this case Pit #21, can be compared to the resources shell for a comparison of Mineral Resources. Pit #21 contains 20.4 Mt of mineralized material which is equal to the Mineral Resources presented in Item 14 of this report. The strip ratio of Pit #21 is 4.0.

The selection of the optimal pit is driven by comparison of the highest net present value (NPV) generated for each individual pit shell associated to a revenue factor. At this stage of the study, the selection of the optimal pit can be based on the discounted cashflow (DCF) to simplify NPV calculations while assuming that capital costs are constants for each scenario. Thus, to select the optimal pit, WSP used the pit-by-pit graph tool in Whittle which calculates the discounted cashflow for each individual pit shell associated to a revenue factor.

Based on inputs from IPG, WSP ran the pit-by-pit graph tool assuming a production feed rate to the mill of 427,100 t/y, no mining constraint capacity (total production from pit including mineralized and waste materials) and a discount rate of 8%.

The pit-by-pit graph generates the DCF for each revenue factor and for three options: the base case, the worst case and the specified case.

The best case considers that smaller pit shells will be mined one after the other in ascending order of revenue factor, thus favorizing a mine production schedule that starts with low strip ratios. The best case generates the highest DCF value in theory, but is not necessarily practical, as it requires multiple low distance expansions that are not optimum for mine equipment sizing.

The worst case considers mining full benches one after the other from highest to lowest elevation, thus forcing the mining of all waste materials on any single bench before proceeding to the next lower bench. The worst case generates the lowest DCF value.

The specified case is an option where the mine production schedule follows a sequence of selected pushbacks that are operationally possible while respecting equipment sizing and safety distances between each pushback. The specified case optimizes the DCF but generates a DCF value between the worst case and the best case.

To run the pit-by-pit graph with the specified case option, the following pushbacks were selected, with a fixed Lead of 3 benches per year.

— Pushback 1: Pit 3, revenue factor = 0.123
— Pushback 2: Pit 7, revenue factor = 0.143
— Pushback 3: Pit 14, revenue factor = 0.178
— Pushback 4: Pit 21, revenue factor = 1

Pushback selections are driven by:

— Tonnage and mine life,
— Adequate and safe pit dimensions,
— Significant increase in tonnage versus revenue factor variation.

Table 16-8 summarizes pit shell optimization results utilizing an 8% discount rate.

Table 16-8: Various Pit Shell by Revenue Factor

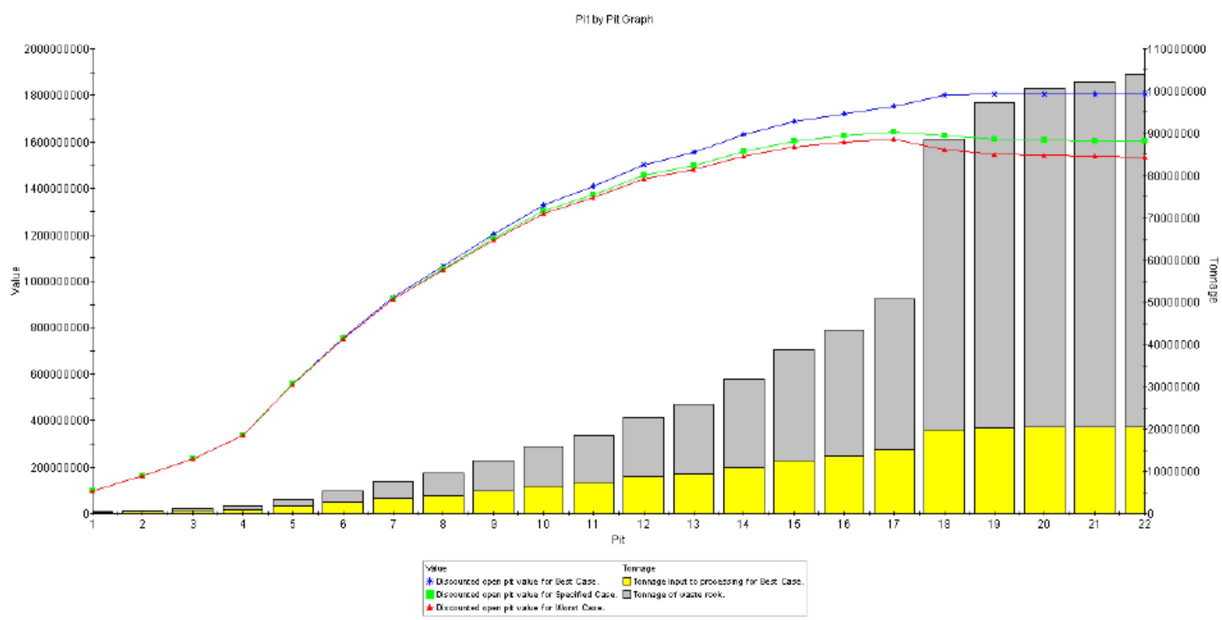
Final pit	Rev. Factor	Open pit cashflow best $ disc	Open pit cashflow specified $ disc	Open pit cashflow worst $ disc	tonne input best	Waste best tonne	Mine life years best	Mine life years specified	Mine life years worst
1	0.113	97,769,224	97,769,224	97,769,224	244,821	179,461	0.6	0.6	0.6
2	0.118	160,629,283	160,629,283	160,629,283	425,224	288,186	1.0	1.0	1.0
3	0.123	236,805,127	236,526,738	236,526,738	652,879	443,234	1.5	1.5	1.5
4	0.128	337,357,154	336,763,947	336,313,704	987,484	752,427	2.3	2.3	2.3
5	0.133	559,616,584	558,057,549	556,669,065	1,809,854	1,529,252	4.2	4.2	4.2
6	0.138	757,392,655	753,522,352	751,612,353	2,654,606	2,609,111	6.2	6.2	6.2
7	0.143	933,095,566	924,980,301	922,498,772	3,539,652	3,958,405	8.3	8.3	8.3
8	0.148	1,064,848,125	1,053,155,587	1,048,780,813	4,322,137	5,360,898	10.1	10.1	10.1
9	0.153	1,203,204,684	1,185,891,568	1,179,171,194	5,305,329	7,240,287	12.5	12.5	12.5
10	0.158	1,328,602,838	1,302,529,082	1,292,988,006	6,428,484	9,403,077	15.1	15.1	15.1
11	0.163	1,408,317,691	1,373,731,656	1,362,013,974	7,289,934	11,242,044	17.1	17.1	17.1
12	0.168	1,503,866,162	1,455,814,399	1,441,052,281	8,576,113	14,182,810	20.1	20.1	20.1
13	0.173	1,555,663,496	1,498,733,311	1,482,327,886	9,419,907	16,419,871	22.1	22.1	22.1
14	0.178	1,632,336,254	1,559,283,105	1,540,268,909	10,995,141	20,932,721	25.8	25.8	25.8
15	0.183	1,690,101,884	1,605,259,345	1,580,658,167	12,561,432	26,129,866	29.5	29.5	29.5
16	0.188	1,719,269,400	1,625,586,287	1,597,225,295	13,571,491	29,746,126	31.9	31.9	31.9
17	0.2	1,752,986,090	**1,644,949,836**	1,610,895,567	15,059,927	35,654,860	35.3	35.3	35.3
18	0.4	1,803,072,816	1,625,976,073	1,567,591,526	19,819,027	68,843,771	46.5	46.5	46.5
19	0.6	1,805,173,691	1,611,770,333	1,548,074,271	20,271,331	77,083,611	47.6	47.6	47.6
20	0.8	1,805,448,382	1,606,180,319	1,540,545,531	20,378,809	80,366,796	47.8	47.8	47.8
21	1	1,805,464,043	1,603,666,939	1,537,209,954	20,413,854	81,708,784	47.9	47.9	47.9
22	1.5	1,805,391,032	1,601,316,305	1,533,320,158	20,442,739	83,459,718	48.0	48.0	48.0

The optimization results show a higher DCF for Pit #17 at a revenue factor of 0.2 (red font) for the specified scenario.

Since the mine life must be limited to 25 years, as mentioned in Item 16.1 Introduction, Pit #14 (highlighted in green) was selected as the optimal pit with a revenue factor of 0.178.

Figure 16-1 presents the pit-by-pit graph generated by Whittle with a discounted cashflow (DCF) of 8%.

Figure 16-1: Discounted Cashflow per Pit Shell



16.5 Pit Design

Once the final pit shells have been determined, the next step is to design the operational pit, usually named ‘’pit design’’, for each pushback phase. The operational pit must be designed to meet industry and territorial safety standards to provide practical and safe access for mine personnel and equipment from surface to pit bottom. The pit design uses the pit shell as a guideline and includes ramps, smoothed pit walls and ensures that the pit can be safely mined using selected equipment. The following Item provides the parameters and methodology used to build the pit design and present the results, including pit design geometry and in-pit resources.

16.5.1 Mining Methods

The mining method selected for the Crater Lake project is a conventional drill and blast mine operation with truck and shovel excavation and transport of blasted material.

It was assumed in this PEA that overburden removal and hard rock mining will be performed by the Owner. Other scenarios involving contractors were not contemplated for this study, but such scenarios should be investigated during future studies on this project.

The overburden material is mainly constituted of glacial till at variable thicknesses, ranging from 0 m to approximately 10 m in the pit area. No drill and blast activities are planned for the removal of overburden removal.

The ore and waste rock material will be drilled and blasted in 10 m benches, then loaded into rigid-frame haul trucks with hydraulic shovels and wheel loaders.

16.5.2 Haul Road and Ramp Design

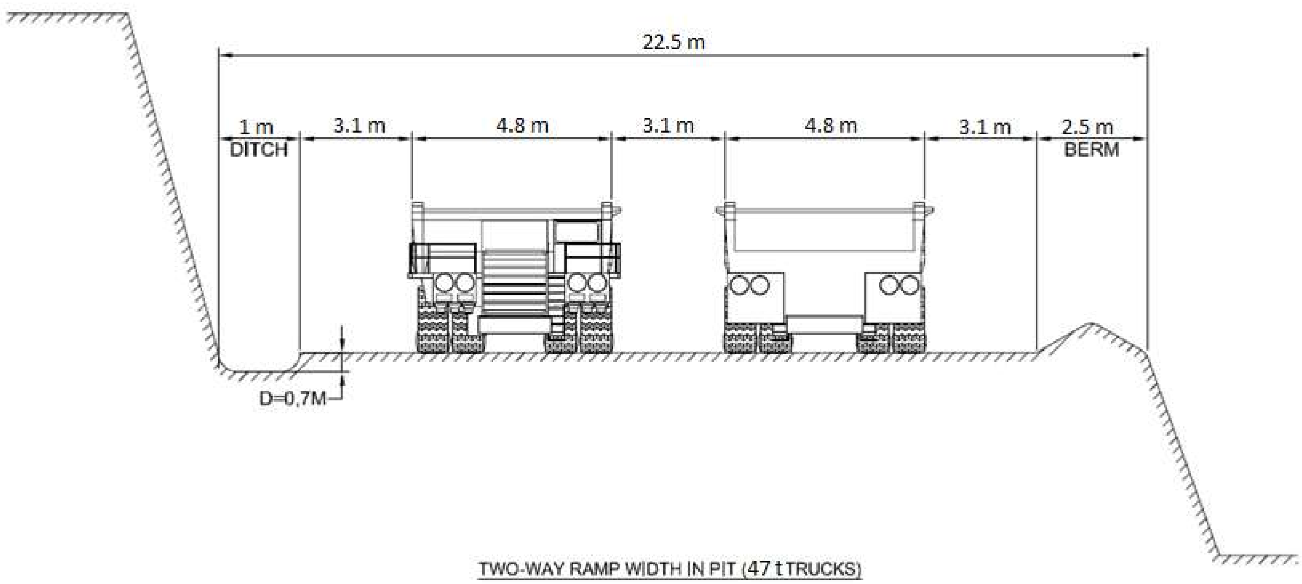
In-pit haul roads and ramps are designed with an overall width of 22.5 m on overburden and rock. All ramps and haul roads are designed for double-lane traffic, except for the last two benches in the pit bottom which are designed for single-lane traffic of 16.5 m width to increase ore recovery from the deepest benches. Haul traffic must be closely monitored for safe operations when mining the bottommost two benches.

In-Pit Dual-Lane Traffic on Rock

For in-pit dual-lane traffic on rock, a minimum width of 3.5 times the width of the largest truck was designed. The overall width of a 47-t mining truck is 4.8 m (CAT 772) which results in a running surface of 16.8 m assuming an operating width multiplier of 3.5 for dual-lane traffic. The allowance for a ditch (1 m) and a berm (2.5 m) increases the total road width to 20.3 m. The overall ramp width was design at 22.5 m to account for more flexibility for ditches and in the event the final truck selection is larger.

Figure 16-2 shows a typical in-pit dual-lane ramp on rock

Figure 16-2: Typical In-Pit Dual-Lane Ramp on Rock

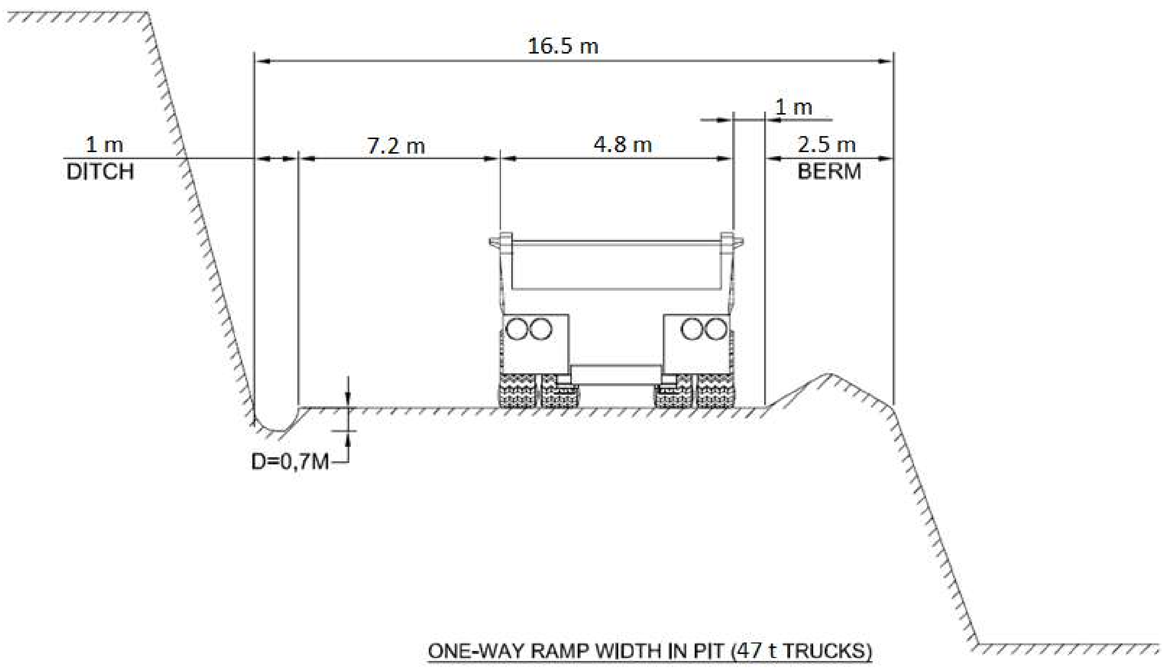


In-Pit Single-Lane Traffic on Rock

For in-pit single-lane traffic on the rock, a minimum width of two times the width of the largest truck was used. The overall width of a 47-t mining truck is 4.8 m (CAT 772) in a running surface of 9.6 m. The allowance for a ditch (1 m) and a berm (2.5 m) increases the total road width to 13 m. The overall ramp width was design at 16.5 m to account for more flexibility for ditches and in the event the final truck selection is larger.

Figure 16-3 shows a typical single-lane ramp on rock.

Figure 16-3: Typical In-Pit Single-Lane Ramp on the Rock



16.5.3 Mine Dilution and Mining Recovery

Due to the specification of large scale mine equipment, orebody shape, and ore contact with waste material, it is always challenging to perfectly separate ore material from waste material during mining operations. As a result, diluting waste rock is sometimes mined as ore material, particularly at the fringes of a deposit, and as such a dilution percentage is usually added to the ore stream to take account of this mixture. In addition, during mining operations some ore material is not mined. The "lost" ore will be moved from ore material and added to waste material for volumetric balance. To account for ore losses a mining recovery factor is estimated and applied.

WSP performed an ore loss and dilution calculation on level 510, based on geologic lenses provided by IPG, to reflect realistic mine recovery in the mine production schedule. The results show an average of 95% mine recovery and 5% dilution.

A dilution of 5% and a mine recovery of 95% are comparable to similar projects in this region. WSP recommends updating estimates of ore loss and mined dilution in subsequent engineering studies.

16.5.4 Final Pit Design

Based on the selected final pit shell and pit design criteria, a final pit design was created.

The final pit design was designed to allow for multiple ramp accesses to waste and ore. The highest level is at Z=575 at surface and the deepest level is at Z=380 at pit bottom, with a total pit depth of 195m. A dual-lane traffic ramp of 22.5 m width was designed from surface to elevation Z=400, then a single-lane traffic ramp of 16.5 m width was designed from elevation Z=400 to pit bottom. The main ramp at the pit exit serves the ore crusher in the north, the waste rock stockpile in the west and the overburden stockpile in the northwest. A maximum grade of 10% was considered for ramps in the pit design to optimize haul truck performance.

A 3D view, and a sectional view of the final pit design, are shown on Figure 16-4 and Figure 16-5, respectively.

Figure 16-4: Final Pit – 3D View

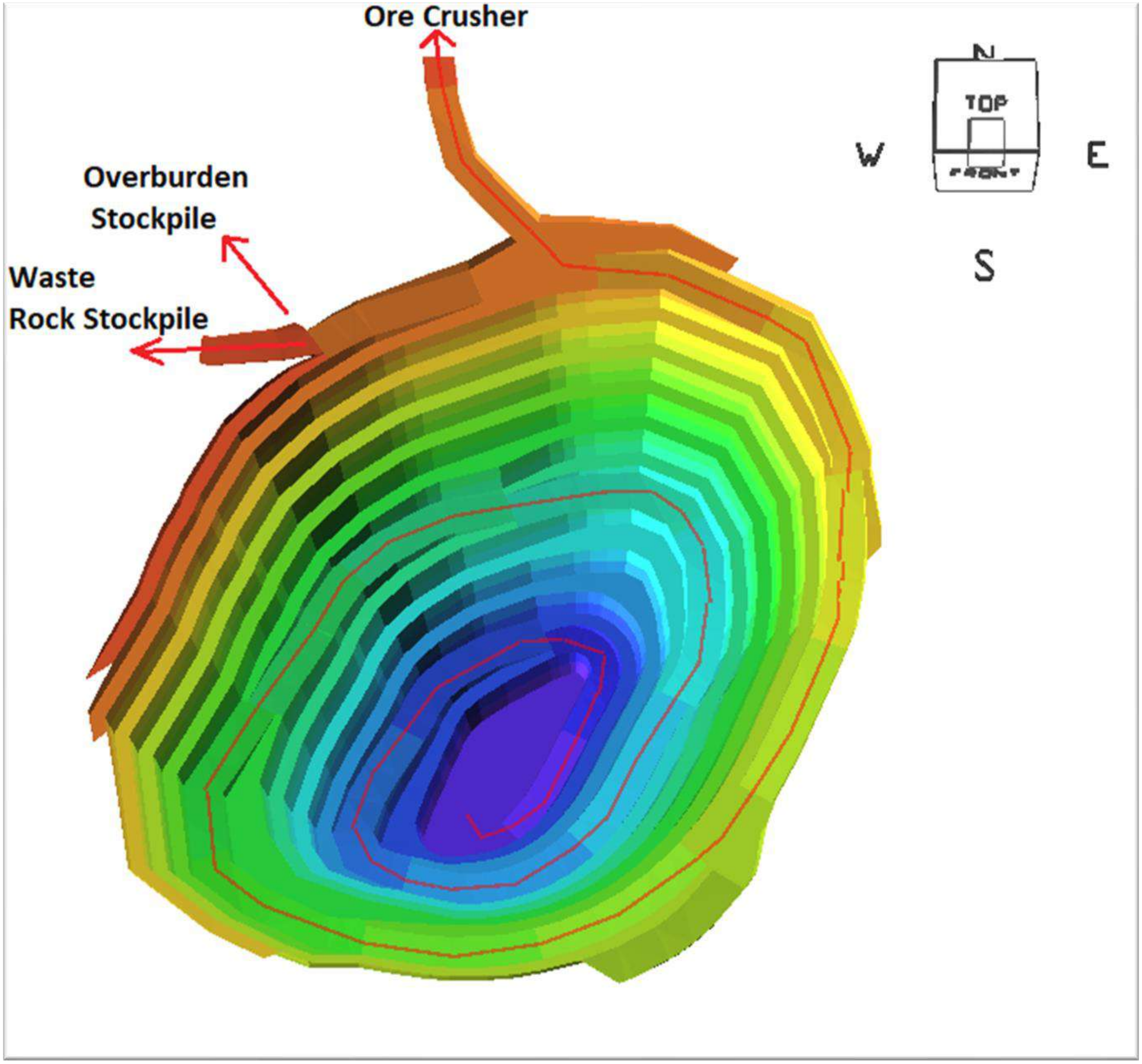
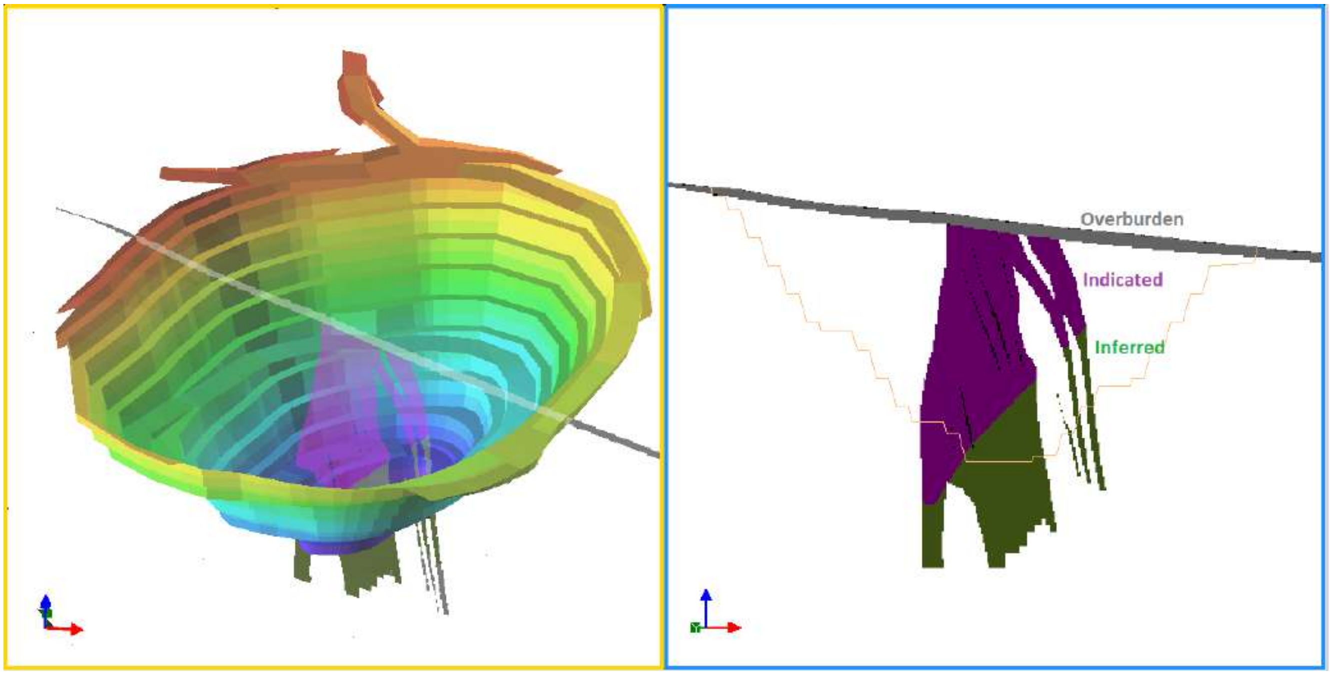


Figure 16-5: Final Pit – Sectional View



16.5.5 Pushback Design

In order to achieve mine planning tonnage targets for ore and waste, three pushbacks were designed for the Crater Lake pit.

The pushback design names are as follows:

- Pushback 1: Starter Pit
- Pushback 2: Middle Pit
- Pushback 3: Ultimate Pit (final pit design)

—

Pushback 1, pushback 2, and pushback 3 were designed to follow respectively pit shell #3, pit shell #7, and pit shell #14, as illustrated in Figure 16-5.

Figure 16-6 illustrates pit shell selections prior to the pushback design.

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Figure 16-6: Pushback – Pit Shell Selection

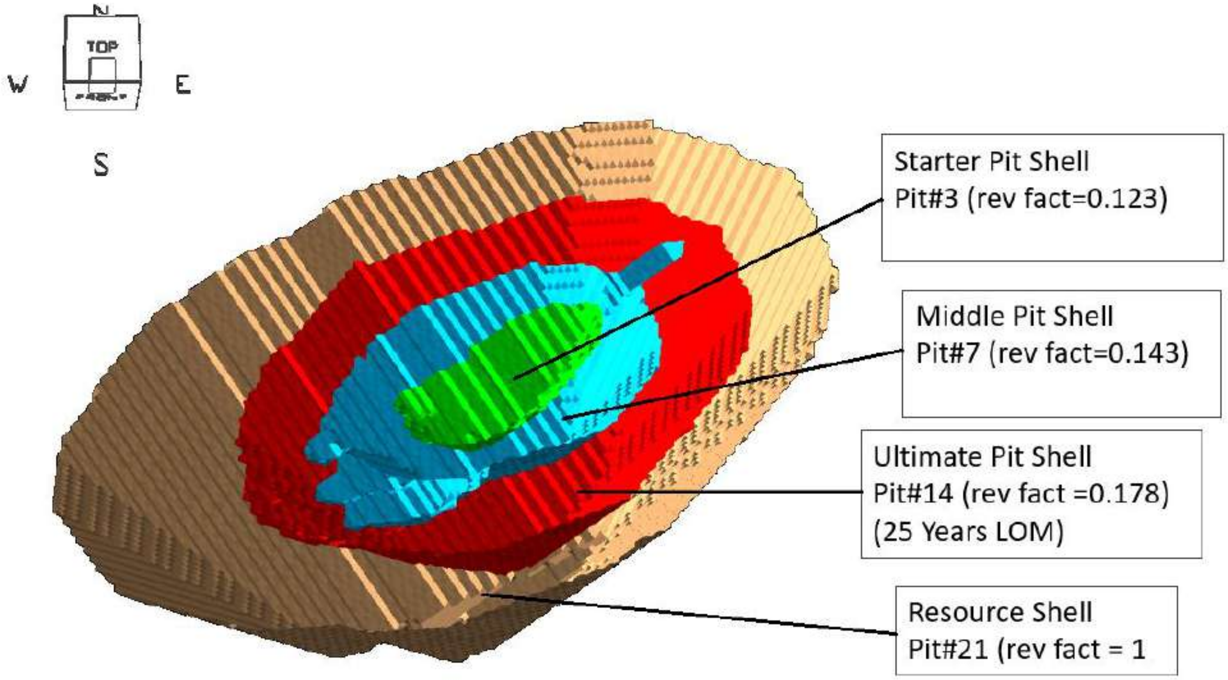


Figure 16-7 and Figure 16-8 illustrate pushback design in 3D and sectional view, respectively.

Figure 16-7: Pushback Design – 3D View

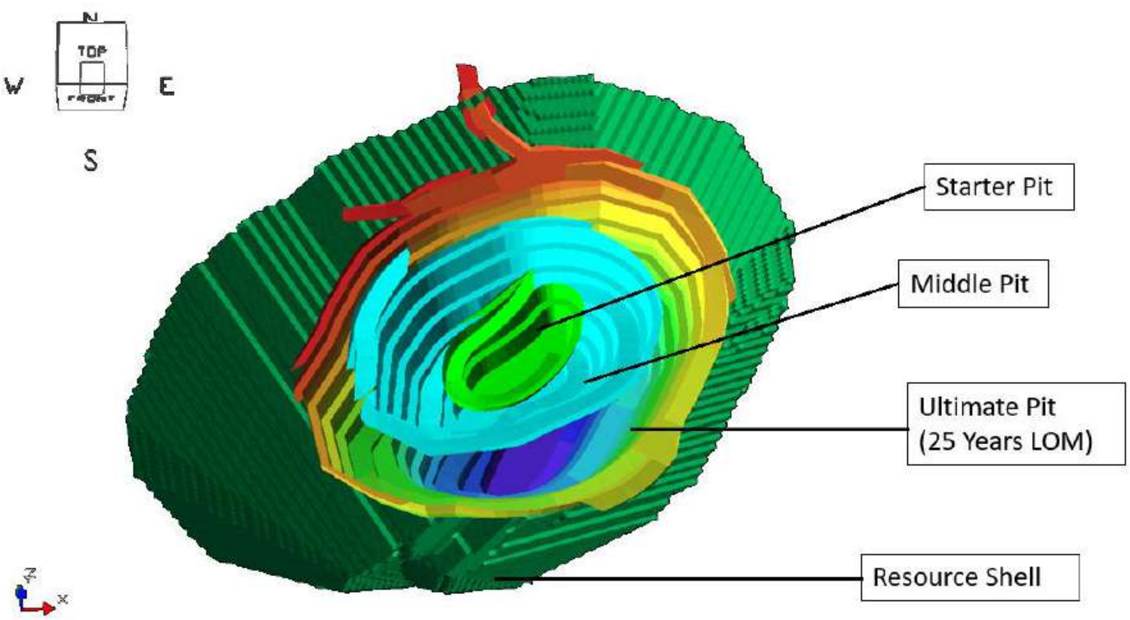
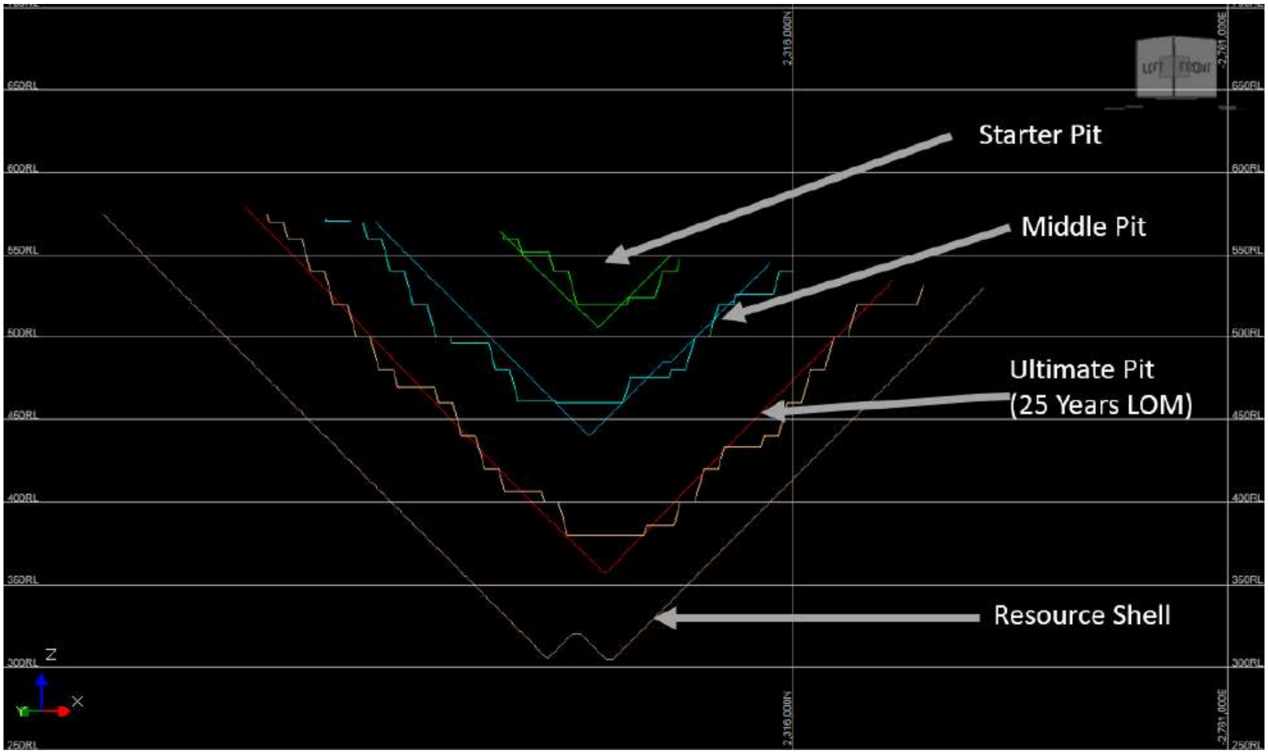


Figure 16-8: Solid Pushbacks – Sectional View



16.5.6 In-Pit Resources

Since Item 15 for Reserves of this report was not requested at this PEA level, the official in-situ in-pit resources statement was calculated based on the final pit design and the updated cut-off grade.

After creating the pit design surfaces for each pushback and the ultimate pit, a solid was created from the topography to the bottom of each pushback, then queried in Deswik Sched to calculate the reserves. Based on the economic parameters used in Table 16.6 Pit Optimisation Parameters, the updated cut-off grade is NSR= 110.27 CAD/tonne. The NSR calculation is based on the selling prices of the Scandium oxide and the associated revenues, and the cost of mining, processing, concentrate transportation from mine site to processing plant, general and administration.

Table 16-9 summarizes the in-situ in-pit resources per pushback.

Table 16-9: In-Situ In-Pit Resources per Pushback

	Tonnage	Sc_2O_3	Dy_2O_3	La_2O_3	Nd_2O_3	Pr_2O_3	Tb_4O_7
	M tonnes	g/t	g/t	g/t	g/t	g/t	g/t
Start Pit							
Ind	0.608	301.3	68.1	628.1	602.2	159.2	12.0
Inf	0.000	268.5	76.1	722.4	650.1	180.5	13.6
Ind + Inf	**0.608**	**301.3**	**68.1**	**628.1**	**602.2**	**159.2**	**12.0**
Middle Pit							
Ind	2.939	290.2	67.4	625.6	609.3	165.1	12.0
Inf	0.049	267.3	68.7	657.6	600.8	161.8	12.2
Ind + Inf	**2.988**	**289.8**	**67.4**	**626.2**	**609.2**	**165.0**	**12.0**
Ultimate Pit							
Ind	3.199	280.2	64.9	595.8	592.9	159.1	11.5
Inf	3.810	273.5	64.2	612.4	585.5	158.1	11.5
Ind + Inf	**7.009**	**276.5**	**64.5**	**604.8**	**588.9**	**158.5**	**11.5**
All Pushbacks							
Ind	6.718	286.6	66.3	611.6	601.2	161.8	11.8
Inf	3.887	273.2	64.2	613.1	585.3	157.9	11.5
Ind + Inf	**10.604**	**281.7**	**65.5**	**612.2**	**595.3**	**160.4**	**11.7**

For the rest of Item 16, the tonnages and grades will be presented with the addition of 5% dilution and 95% mine recovery, as mentioned in Table 16-14.

Table 16-10 summarizes the diluted in-pit resources for each pushback. Dilution and Mine recovery were applied to tonnages. Dilution was applied to grades.

Table 16-10: Diluted In-Pit Resources per pushback

	Tonnage	Sc_2O_3	Dy_2O_3	La_2O_3	Nd_2O_3	Pr_2O_3	Tb_4O_7
	M tonnes	g/t	g/t	g/t	g/t	g/t	g/t
Start Pit							
Ind	0.607	286.9	64.8	598.2	573.5	151.6	11.4
Inf	0.000	255.7	72.5	688.0	619.2	171.9	13.0
Ind + Inf	**0.607**	**286.9**	**64.8**	**598.2**	**573.5**	**151.6**	**11.4**
Middle Pit							
Ind	2.932	276.4	64.2	595.9	580.3	157.2	11.4
Inf	0.049	254.6	65.4	626.3	572.2	154.1	11.6
Ind + Inf	**2.980**	**276.0**	**64.2**	**596.3**	**580.2**	**157.2**	**11.4**
Ultimate Pit							
Ind	3.191	266.8	61.9	567.5	564.6	151.5	10.9
Inf	3.801	260.4	61.1	583.2	557.6	150.5	11.0
Ind + Inf	**6.991**	**263.3**	**61.4**	**576.0**	**560.8**	**151.0**	**11.0**
All Pushbacks							
Ind	6.701	272.9	63.2	582.5	572.5	154.1	11.2
Inf	3.877	260.2	61.1	583.9	557.4	150.4	11.0
Ind + Inf	**10.578**	**268.3**	**62.4**	**583.0**	**567.0**	**152.8**	**11.1**

Table 16-11 presents a summary of diluted in-pit resources with associated waste materials.

Table 16-11: Diluted In-Pit Resources and waste materials per pushback

	Units	**Start Pit**	**Middle Pit**	**Ultimate Pit**	**All Pushbacks**
Ind +Inf	M tonnes	**0.607**	**2.980**	**6.991**	**10.578**
Overburden	M tonnes	0.272	0.844	1.251	2.367
Waste Rock	M tonnes	0.087	4.036	14.774	18.897
Total Waste	**M tonnes**	**0.360**	**4.879**	**16.025**	**21.264**
Strip Ratio		0.59	1.64	2.29	2.01
Total Material	**M tonnes**	**0.966**	**7.859**	**23.016**	**31.842**
Mine Life @ 426100 t/y	Years	1.4	7.0	16.4	24.8

16.6 Waste Dump Design

Two waste stockpiles were designer for Crater Lake Project.

The waste rock stockpile is located on the West side of the pit. It has a capacity of 10.5 Mm^3 and can handle the total waste rock requirements 18.8 M tonnes. The maximum height of the waste stockpile is at Z= 570, and the lowest level on the ground of the topography is at Z=502. The design of the dump is summarized in Table 16-12.

Table 16-12: Waste Rock Stockpile Design

Items	Unit	Parameters
Bench height	m	10
Face angle	°	33.7
Berm	m	20
Inter-ramp angle	°	21
Ramp Width	m	15

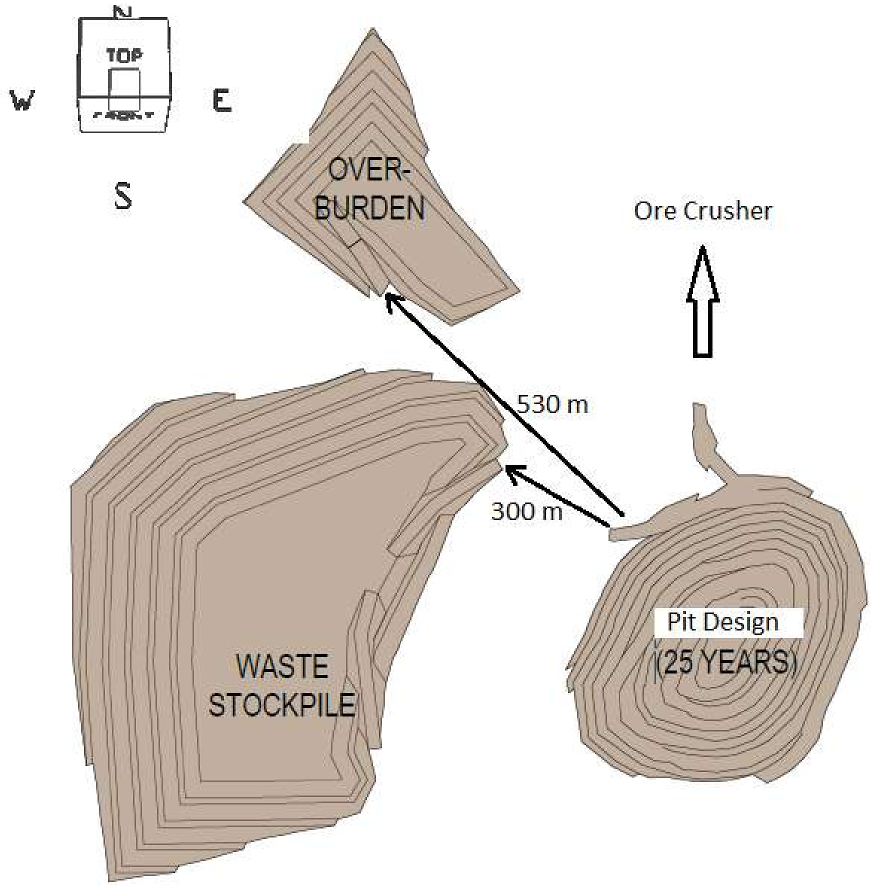
The waste overburden stockpile is located on the northwest side of the pit. It has a capacity of 1.2 Mm^3 and can handle the total overburden requirements 2.4 M tonnes. The maximum height of the waste stockpile is at Z= 580, and the lowest level on the ground of the topography is at Z=527. The design of the dump is summarized in Table 16-13.

Table 16-13: Overburden Stockpile Design

Items	Unit	Parameters
Bench height	m	10
Face angle	°	26.5
Berm	m	15
Inter-ramp angle	°	20
Ramp Width	m	25

Figure 16-9 shows an overall view of the waste stockpile and overburden stockpile design.

Figure 16-9: Solid Pushbacks - Sectional View



16.7 Mine Planning

The mine plan was developed with the target of feeding the mill at capacity while optimizing revenues, particularly during the first years of production, to have a positive impact on NPV. As such, the emphasis was on having quick access to rock ore material to generate cash revenues as soon as possible.

To maintain a relatively smooth strip ratio over time, overburden removal will not stop and will move from one pushback to the next adjacent pushback as soon as the previous pushback is free from overburden.

16.7.1 Mine Planning Parameters

The mine planning parameters are summarized as follows.

Work Schedule

It is assumed that the mine will operate 7 days a week, 24 hours per day, and 6 months a year with 2 shifts of 12 hours each day.

Mill Capacity

A constant yearly mill capacity of 426,100 tonnes was provided by IPG to WSP as the base case for this PEA. No ramp-up production at the mill was considered in this PEA.

16.7.2 Mine Production Schedule

The mine production schedule (Table 16-14) was completed on a yearly basis to meet mine planning parameters stated above, while smoothing the equipment fleet, particularly the truck haulage fleet. WSP ran the mine production schedule on a quarterly basis from the beginning to the end of mine life to ensure production targets are realistic and to avoid too much variation in the production equipment. The mine production schedule was realized with a dilution of 5% and a mine recovery of 95%.

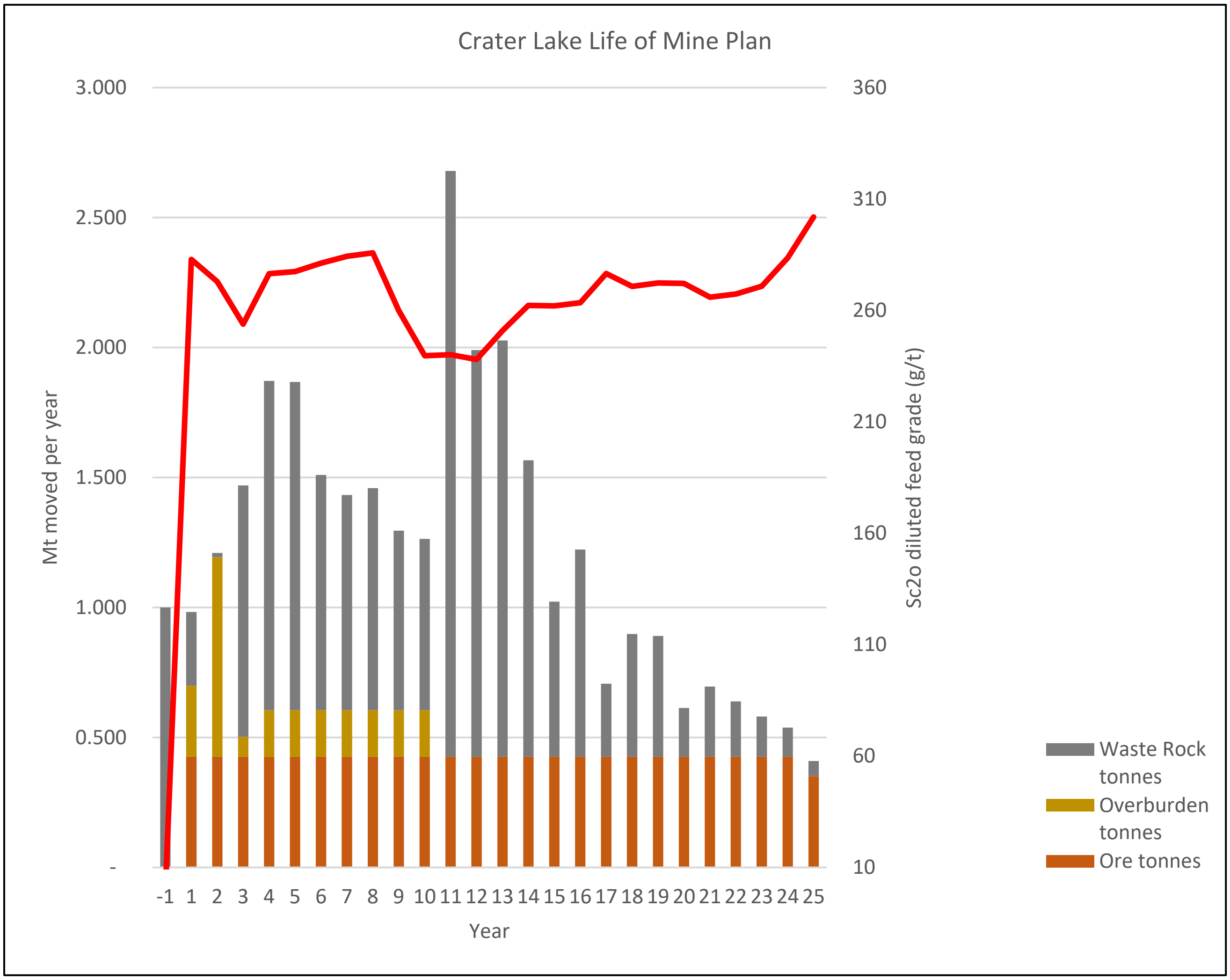
Table 16-14 summarizes the detailed mine production schedule by year with diluted tonnes and grades. Figure 16-10 presents the mine production schedule per year with diluted tonnes and grades.

Table 16-14: Mine Production Schedule

Description	Units	Pre-Prod	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Overburden	Mt	-	0.272	0.767	0.076	0.179	0.179	0.179	0.179	0.179	0.179	0.179	-	-
Waste Rock	Mt	1.000	0.284	0.016	0.967	1.267	1.263	0.905	0.828	0.854	0.691	0.659	2.253	1.564
Mineralized Material (diluted)	**Mt**	**-**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**
Strip ratio				1.8	2.4	3.4	3.4	2.5	2.4	2.4	2.0	2.0	5.3	3.7
Total material	**Mt**	1.000	0.983	1.210	1.470	1.872	1.868	1.510	1.433	1.459	1.296	1.264	2.679	1.991
Sc_2O_3 (diluted)	**g/t**	-	282.9	272.9	253.9	276.5	277.4	281.2	284.3	285.8	259.9	239.7	240.2	238.0
Dy_2O_3 (diluted)	**g/t**	-	64.1	63.3	60.5	63.9	64.5	66.6	65.4	65.2	61.6	58.7	58.5	59.5
La_2O_3 (diluted)	**g/t**	-	599.7	585.3	583.4	591.1	602.6	616.0	595.9	596.5	574.5	571.2	582.2	580.0
Nd_2O_3 (diluted)	**g/t**	-	566.8	568.2	548.4	574.0	583.4	601.4	588.8	589.2	568.2	544.8	544.0	546.5
Pr_2O_3 (diluted)	**g/t**	-	149.9	152.9	150.8	154.3	158.5	164.7	158.3	158.1	151.7	147.1	147.1	150.8
Tb_4O_7 (diluted)	**g/t**	-	11.3	11.2	10.8	11.4	11.5	11.9	11.6	11.5	11.0	10.7	10.7	10.8

Description	Units	Year13	Year14	Year15	Year16	Year17	Year18	Year19	Year20	Year21	Year22	Year23	Year24	Year25	Total
Overburden	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-	**2.367**
Waste Rock	Mt	1.601	1.140	0.596	0.797	0.280	0.472	0.464	0.187	0.270	0.213	0.154	0.112	0.058	**18.897**
Mineralized Material (diluted)	**Mt**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.426**	**0.352**	**10.578**
Strip ratio		3.8	2.7	1.4	1.9	0.7	1.1	1.1	0.4	0.6	0.5	0.4	0.3	0.2	**2.0**
Total material	**Mt**	2.027	1.566	1.023	1.223	0.707	0.898	0.890	0.613	0.696	0.639	0.580	0.538	0.409	**31.842**
Sc_2O_3 (diluted)	**g/t**	250.9	262.3	262.0	263.4	276.5	270.8	272.3	272.1	265.9	267.4	270.8	283.5	301.9	268.3
Dy_2O_3 (diluted)	**g/t**	61.5	63.0	62.1	62.6	64.3	64.0	63.2	62.5	61.0	61.5	60.1	60.4	62.3	62.4
La_2O_3 (diluted)	**g/t**	593.3	589.5	577.6	586.8	588.6	591.4	589.2	576.1	564.4	572.0	550.3	549.7	564.7	583.0
Nd_2O_3 (diluted)	**g/t**	561.5	570.0	560.9	567.4	579.3	581.6	575.0	566.7	555.2	558.8	549.3	550.9	576.4	567.0
Pr_2O_3 (diluted)	**g/t**	152.5	153.3	150.4	151.8	154.9	155.0	153.0	152.1	148.9	150.7	148.5	149.7	154.2	152.8
Tb_4O_7 (diluted)	**g/t**	11.0	11.3	11.0	11.1	11.3	11.4	11.2	11.1	10.8	10.9	10.6	10.7	11.1	11.1

Figure 16-10: Crater Lake Life-of-Mine Plan



In Year 11, total mine production peaks at 2.6 Mt. This peak is caused by the increasing strip ratio from 2 to 5.3 in Year 11, resulting in an increased quantity of 2.2 Mt of waste rock required to strip the third pushback (Ultimate Pit). In practice, this situation can be mitigated by starting waste rock removal from the Ultimate Pit earlier in years 9 and 10 to smooth the mine fleet equipment capacity.

Table 16-15 summarizes the mine production schedule for each pushback.

Table 16-15: Detailed Production Schedule by Pushback

Start Pit	Units	Pre-Prod	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Overburden	Mt	-	0.272	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	Mt	-	0.071	0.016	-	-	-	-	-	-	-	-	-	-
Mineralized Material (diluted)	Mt	-	0.426	0.180	-	-	-	-	-	-	-	-	-	-
Strip ratio		-	0.8	0.1	-	-	-	-	-	-	-	-	-	-
Total material	**Mt**	-	0.770	0.197	-	-	-	-	-	-	-	-	-	-
Sc_2O_3 (diluted)	**g/t**	-	282.9	296.5	-	-	-	-	-	-	-	-	-	-
Dy_2O_3 (diluted)	**g/t**	-	64.1	66.6	-	-	-	-	-	-	-	-	-	-
La_2O_3 (diluted)	**g/t**	-	599.7	594.8	-	-	-	-	-	-	-	-	-	-
Nd_2O_3 (diluted)	**g/t**	-	566.8	589.4	-	-	-	-	-	-	-	-	-	-
Pr_2O_3 (diluted)	**g/t**	-	149.9	155.5	-	-	-	-	-	-	-	-	-	-
Tb_4O_7 (diluted)	**g/t**	-	11.3	11.7	-	-	-	-	-	-	-	-	-	-

Middle Pit	Units	Pre-Prod	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Overburden	Mt	-	-	0.767	0.076	-	-	-	-	-	-	-	-	-
Waste Rock	Mt	1.000	0.213	-	0.967	0.608	0.604	0.246	0.169	0.195	0.032	-	-	-
Mineralized Material (diluted)	Mt	-	-	0.246	0.426	0.426	0.426	0.426	0.426	0.426	0.178	-	-	-
Strip ratio		-	-	3.1	2.4	1.4	1.4	0.6	0.4	0.5	0.2	-	-	-
Total material	**Mt**	1.000	0.213	1.013	1.470	1.034	1.030	0.673	0.595	0.621	0.210	-	-	-
Sc_2O_3 (diluted)	**g/t**	-	-	255.6	253.9	276.5	277.4	281.2	284.3	285.8	296.7	-	-	-
Dy_2O_3 (diluted)	**g/t**	-	-	60.9	60.5	63.9	64.5	66.6	65.4	65.2	66.9	-	-	-
La_2O_3 (diluted)	**g/t**	-	-	578.3	583.4	591.1	602.6	616.0	595.9	596.5	603.4	-	-	-
Nd_2O_3 (diluted)	**g/t**	-	-	552.6	548.4	574.0	583.4	601.4	588.8	589.2	608.4	-	-	-
Pr_2O_3 (diluted)	**g/t**	-	-	151.0	150.8	154.3	158.5	164.7	158.3	158.1	161.6	-	-	-
Tb_4O_7 (diluted)	**g/t**	-	-	10.9	10.8	11.4	11.5	11.9	11.6	11.5	11.8	-	-	-

Ultimate Pit Total	Units	Pre-Prod	Year1	Year2	Year3	Year4	Year5	Year6	Year7	Year8	Year9	Year10	Year11	Year12
Overburden	Mt	-	-	-	-	0.179	0.179	0.179	0.179	0.179	0.179	0.179	-	-
Waste Rock	Mt	-	-	-	-	0.659	0.659	0.659	0.659	0.659	0.659	0.659	2.253	1.564
Mineralized Material (diluted)	Mt	-	-	-	-	-	-	-	-	-	0.248	0.426	0.426	0.426
Strip ratio		-	-	-	-	-	-	-	-	-	3.4	2.0	5.3	3.7
Total material	**Mt**	-	-	-	-	0.838	0.838	0.838	0.838	0.838	1.086	1.264	2.679	1.991
Sc_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	233.6	239.7	240.2	238.0
Dy_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	57.9	58.7	58.5	59.5
La_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	553.9	571.2	582.2	580.0
Nd_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	539.4	544.8	544.0	546.5
Pr_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	144.6	147.1	147.1	150.8
Tb_4O_7 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	10.5	10.7	10.7	10.8

Start Pit	Units	Year13	Year14	Year15	Year16	Year17	Year18	Year19	Year20	Year21	Year22	Year23	Year24	Year25
Overburden	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Mineralized Material (diluted)	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Strip ratio		-	-	-	-	-	-	-	-	-	-	-	-	-
Total material	**Mt**	-	-	-	-	-	-	-	-	-	-	-	-	-
Sc_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Dy_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
La_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Nd_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Pr_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Tb_4O_7 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-

Middle Pit	Units	Year13	Year14	Year15	Year16	Year17	Year18	Year19	Year20	Year21	Year22	Year23	Year24	Year25
Overburden	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Mineralized Material (diluted)	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Strip ratio		-	-	-	-	-	-	-	-	-	-	-	-	-
Total material	**Mt**	-	-	-	-	-	-	-	-	-	-	-	-	-
Sc_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Dy_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
La_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Nd_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Pr_2O_3 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-
Tb_4O_7 (diluted)	**g/t**	-	-	-	-	-	-	-	-	-	-	-	-	-

Ultimate Pit	Units	Year13	Year14	Year15	Year16	Year17	Year18	Year19	Year20	Year21	Year22	Year23	Year24	Year25
Overburden	Mt	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	Mt	1.601	1.140	0.596	0.797	0.280	0.472	0.464	0.187	0.270	0.213	0.154	0.112	0.058
Mineralized Material (diluted)	Mt	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.426	0.352
Strip ratio		4	2.7	1.4	1.9	0.7	1.1	1.1	0.4	0.6	0.5	0.4	0.3	0.2
Total material	**Mt**	2.027	1.566	1.023	1.223	0.707	0.898	0.890	0.613	0.696	0.639	0.580	0.538	0.409
Sc_2O_3 (diluted)	**g/t**	250.9	262.3	262.0	263.4	276.5	270.8	272.3	272.1	265.9	267.4	270.8	283.5	301.9
Dy_2O_3 (diluted)	**g/t**	61.5	63.0	62.1	62.6	64.3	64.0	63.2	62.5	61.0	61.5	60.1	60.4	62.3
La_2O_3 (diluted)	**g/t**	593.3	589.5	577.6	586.8	588.6	591.4	589.2	576.1	564.4	572.0	550.3	549.7	564.7
Nd_2O_3 (diluted)	**g/t**	561.5	570.0	560.9	567.4	579.3	581.6	575.0	566.7	555.2	558.8	549.3	550.9	576.4
Pr_2O_3 (diluted)	**g/t**	152.5	153.3	150.4	151.8	154.9	155.0	153.0	152.1	148.9	150.7	148.5	149.7	154.2
Tb_4O_7 (diluted)	**g/t**	11.0	11.3	11.0	11.1	11.3	11.4	11.2	11.1	10.8	10.9	10.6	10.7	11.1

16.8 Equipment Fleet

This Item presents the equipment fleet required to meet the production schedule.

The operating costs are calculated from an Owner-operated scenario. Table 16-16, Table 16-17, and Table 16-18 summarize the major mining equipment selected for the project, the support equipment, and the auxiliary fleet. The mining equipment is selected to match the mine production schedule presented in Table 16-14.

Figure 16-11 shows the Owner's major production fleet over the life of mine for hard rock material.

Table 16-16: Major Mining Equipment Fleet

Major Mining Equipment Fleet	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Hydraulic Excavator - CAT395	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader - CAT 988K	1	1	1	1	1	1	1	1	1	1	1	1	1
Rigid Truck - CAT 772G	3	2	3	4	4	4	4	3	3	3	3	7	5
Drills - EPIROC FlexiRoc D60-SF	1	1	1	1	1	1	1	1	1	1	1	1	1
Major Mining Equipment Fleet	**Year13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Year 20**	**Year 21**	**Year 22**	**Year 23**	**Year 24**	**Year 25**
Hydraulic Excavator - CAT395	1	1	1	1	1	1	1	1	1	1	1	1	1
Loader - CAT 988K	1	1	1	1	1	1	1	1	1	1	1	1	1
Rigid Truck - CAT 772G	5	4	2	3	2	2	2	2	2	2	1	1	1
Drills - EPIROC FlexiRoc D60-SF	1	1	1	1	1	1	1	1	1	1	1	1	1

Figure 16-11: Production Trucks and Shovel Requirements in Hard Rock

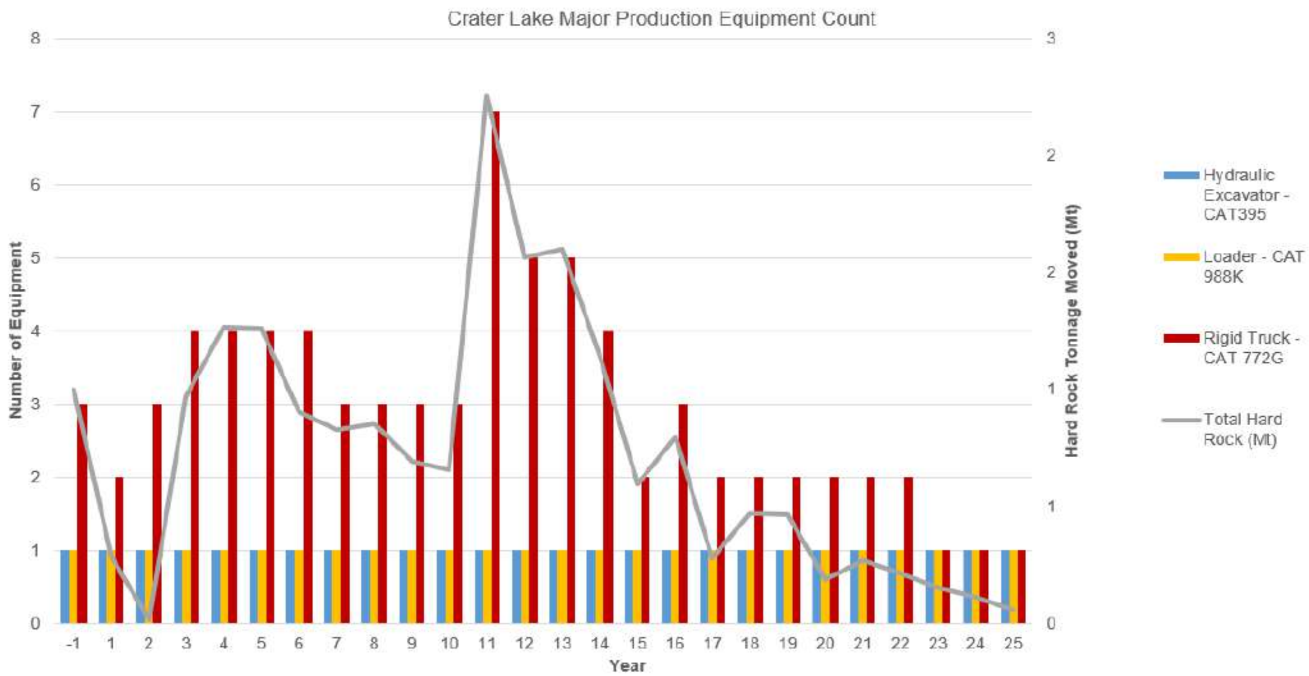


Table 16-17: Support Equipment Fleet

Support Equipment Fleet	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Dozer (Cat D8)	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor grader (Cat 14M)	1	1	1	1	1	1	1	1	1	1	1	1	1
Small loader (Cat 972)	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Excavator (Cat352)	1	1	1	1	1	1	1	1	1	1	1	1	1
Articulated Truck - Cat 745	1	1	1	1	1	1	1	1	1	1	1	1	1

Support Equipment Fleet	Year13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Dozer (Cat D8)	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor grader (Cat 14M)	1	1	1	1	1	1	1	1	1	1	1	1	1
Small loader (Cat 972)	1	1	1	1	1	1	1	1	1	1	0	0	0
Small Excavator (Cat352)	1	1	1	1	1	1	1	1	1	1	1	1	1
Articulated Truck - Cat 745	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16-18: Auxiliary Equipment Fleet

Auxiliary Equipment Fleet	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Water Truck (Cat 745)	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor (Cat CS76B)	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractor	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck (Mechanical)	1	1	1	1	1	1	1	1	1	1	1	1	1
Personal carrier	1	1	1	1	1	1	1	1	1	1	1	1	1
Tyre Handler	1	1	1	1	1	1	1	1	1	1	1	1	1
50t Crane	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Lube truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Flat Bed truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Tower	1	1	1	1	1	1	1	1	1	1	1	1	1
Pick-up truck	8	8	8	8	8	8	8	8	8	8	8	8	8
Boom truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Auxiliary Equipment Fleet	**Year13**	**Year 14**	**Year 15**	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Year 20**	**Year 21**	**Year 22**	**Year 23**	**Year 24**	**Year 25**
Water Truck (Cat 745)	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor (Cat CS76B)	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractor	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck (Mechanical)	1	1	1	1	1	1	1	1	1	1	1	1	1
Personal carrier	1	1	1	1	1	1	1	1	1	1	1	1	1
Tyre Handler	1	1	1	1	1	1	1	1	1	1	1	1	1
50t Crane	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Lube truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Flat Bed truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Tower	1	1	1	1	1	1	1	1	1	1	1	1	1
Pick-up truck	8	8	8	8	8	8	8	6	6	6	6	3	3
Boom truck	1	1	1	1	1	1	1	1	1	1	1	1	1

16.8.1 Loading

One hydraulic excavator (CAT 395, 5.8-m³ bucket capacity) and one loader (CAT 988K, 5.3- to 6.4-m³ bucket capacity) were selected to load 47-t trucks (CAT 772G) for both soft and hard rocks. The excavators are used to reshape the final overburden walls, scale final rock walls, move pumps and pipes, dig ditches, and rehandle material from time to time. For overburden removal, the loader will be mainly used to load 40-t articulated trucks (CAT 745). The loader will also be used for ore stockpile rehandling in front of the crusher when the main production shovels cannot feed the mill to capacity.

16.8.2 Hauling

CAT 772G 47-t trucks have been selected to haul both overburden and blasted rock materials. From the pre-production Year to Year 2, approximately three 47-t trucks will be required, then this number increases to four. The period from year 11 to Year 13 will require on average of six 47-t trucks then this number decreases to three. However, an effort can be made in further studies for this project to smooth the equipment fleet. From Year 15 to year 22, only two 47-t trucks will be required, then one 47-t truck will be required from Year 23 to the end of the mine in Year 25.

16.8.3 Drilling and Blasting

For the mine life of this project, only one drill (Epiroc Flexiroc D60 FS, 4-6 " to 66'') will be required. A spare drill for pre-splitting is enough to cover the mass drilling during idle time due to mass drill maintenance.

16.9 Auxiliary Fleet

An auxiliary fleet of mobile equipment is required to support mine operations. The list of equipment is based on experience from similar operations. The auxiliary equipment will not be used full time, thus reduced operator requirements have been allocated. For example, two pieces of auxiliary equipment may require only one operator as the operator will use one or the other depending on the situation.

16.10 Manpower Requirements

Table 16-19 summarizes the manpower requirements for the mine operation.

Table 16-19: Manpower Requirements

Manpower - Mine Operations	Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operators													
Production Operators (4 shifts) - Hauling/Loading	14	14	14	14	14	14	14	14	14	14	14	14	14
Auxiliary Operators (day shift)	13	13	13	13	13	13	13	13	13	13	13	13	13
Pumping team	0	0	0	0	0	0	0	0	0	0	0	0	0
Blasting team	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Management													
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Assistant Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Pit Foremen	4	4	4	4	4	4	4	4	4	4	4	4	4
Drilling Foremen	2	2	2	2	2	2	2	2	2	2	2	2	2
Labourers	4	8	8	8	8	8	8	8	8	8	8	8	8
Fleet Management	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Trainers	2	2	2	2	2	2	2	2	2	2	2	2	2
Clerks	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Maintenance													
Maintenance Planners	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Foremen	2	4	4	4	4	4	4	4	4	4	4	4	4
Mechanics	7	7	7	7	7	7	7	7	7	7	7	7	7
Electricians	2	2	2	2	2	2	2	2	2	2	2	2	2
Welders	2	2	2	2	2	2	2	2	2	2	2	2	2
Machinists	2	2	2	2	2	2	2	2	2	2	2	2	2
Labourers	2	4	4	4	4	4	4	4	4	4	4	4	4
Fuel/lube Technicians	2	4	4	4	4	4	4	4	4	4	4	4	4
Technical services													
Superintendent Technical Services	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Engineers	1	2	2	2	2	2	2	2	2	2	2	2	2
Mining Engineers	1	2	2	2	2	2	2	2	2	2	2	2	2
Mine Technicians	2	2	2	2	2	2	2	2	2	2	2	2	2
Chief Surveyors	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyors	2	2	2	2	2	2	2	2	2	2	2	2	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologists	1	2	2	2	2	2	2	2	2	2	2	2	2
Geologists	1	2	2	2	2	2	2	2	2	2	2	2	2
Senior Grade Control Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Grade Control Technicians	1	2	2	2	2	2	2	2	2	2	2	2	2
Total Manpower - Mine Operations	**84**	**99**	**99**	**99**	**99**	**99**	**99**	**99**	**99**	**99**	**99**	**99**	**99**

Manpower - Mine Operations	Year13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25
Operators													
Production Operators (4 shifts) - Hauling/Loading	14	14	14	14	14	13	13	13	13	13	11	11	11
Auxiliary Operators (day shift)	13	13	13	13	13	12	12	12	12	12	10	10	10
Pumping team	0	0	0	0	0	0	0	0	0	0	0	0	0
Blasting team	4	4	4	4	4	4	4	3	3	3	3	3	3
Mine Management													
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Asistant Superintendent	1	1	0	0	0	0	0	0	0	0	0	0	0
Pit Foremen	4	4	4	4	4	4	4	4	4	4	4	4	4
Drilling Foremen	2	2	2	2	2	2	2	2	2	2	2	2	2
Labourers	8	8	5	5	5	5	5	5	5	5	5	5	5
Fleet Management	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Trainers	2	2	2	2	2	2	2	2	2	2	2	2	2
Clerks	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Maintenance													
Maintenance Planners	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Foremen	4	4	2	2	2	2	2	2	2	2	2	2	2
Mechanics	7	7	7	7	7	5	5	5	5	5	4	4	4
Electricians	2	2	2	2	2	2	2	2	2	2	1	1	1
Welders	2	2	2	2	2	2	2	2	2	2	2	2	2
Machinists	2	2	2	2	2	2	2	2	2	2	2	2	2
Labourers	4	4	4	4	4	4	4	4	4	4	4	4	4
Fuel/lube Technicians	4	4	4	2	2	2	2	2	2	2	2	2	2
Technical services													
Superintendent Technical Services	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Engineers	2	2	2	2	2	2	2	1	1	1	1	1	1
Mining Engineers	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Technicians	2	2	2	2	2	2	2	2	2	2	2	2	2
Chief Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyors	2	2	2	2	2	2	2	2	2	2	2	2	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Geologists	2	2	2	2	2	2	2	1	1	1	1	1	1
Geologists	2	2	2	2	2	2	2	2	2	2	2	2	2
Senior Grade Control Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Grade Control Technicians	2	2	2	2	2	2	2	2	2	2	2	2	2
Total Manpower - Mine Operations	**99**	**99**	**93**	**91**	**91**	**87**	**87**	**84**	**84**	**84**	**78**	**78**	**78**

16.11 Dewatering Design

The scope of this PEA does not cover hydrogeology and hydrology study.

17 RECOVERY METHODS

The mineral processing and metallurgical development programs have shown that scandium can best be recovered using magnetic separation to produce a scandium rich mineral concentrate and by treating the concentrate through caustic leaching, hydrochloric acid leaching of the caustic residue and solvent extraction to produce a high-grade Scandium Oxide (Sc_2O_3) and a mixed REE product. The Sc_2O_3 will be processed together with alumina to produce Al-2%Sc master alloy in a direct electrolysis process similar to the Hall – Heroult electrolytic method used for the production of primary aluminum.

The process design has been split in two locations. The Beneficiation Plant for the production of Sc rich concentrate will be located at Crater Lake near the open pit mine located about 200 km NE of Schefferville, QC. The Sc/REE mineral concentrate produced at the mine site will be transported to the Hydrometallurgical Plant located in Sept-Îles , QC, for the production of mixed REE product, Sc_2O_3, and Al-2% master alloy. The descriptions of the various areas encompassing the processing plants are provided below. This information serves as the basis for the development of the capital and operating cost estimates presented in Item 21.

17.1 Beneficiation Plant

The beneficiation plant will be designed for an annual throughput capacity of 426,100 t/a. However, due to the harsh climate at Crater Lake, the plant will operate for only six months out of the year, from May to October. The plant daily feed rate will be 2380 t/d of ore with availability of 92%. The overall scandium recovery through magnetic separation will be 90.2% with an overall weight recovery of 51.3%. The plant is therefore expected to produce 218,541 t/a of Sc/REE rich mineral concentrate.

17.1.1 Process Design Criteria

The main unit operations of the beneficiation plant will include crushing, crushed ore stockpiling/reclaiming, grinding/classification, magnetic separation, tailings thickening/filtration, concentrate thickening/filtration, concentrate drying/roasting/cooling, and final dry low intensity magnetic separation. The key process design criteria are listed in Table 17-1.

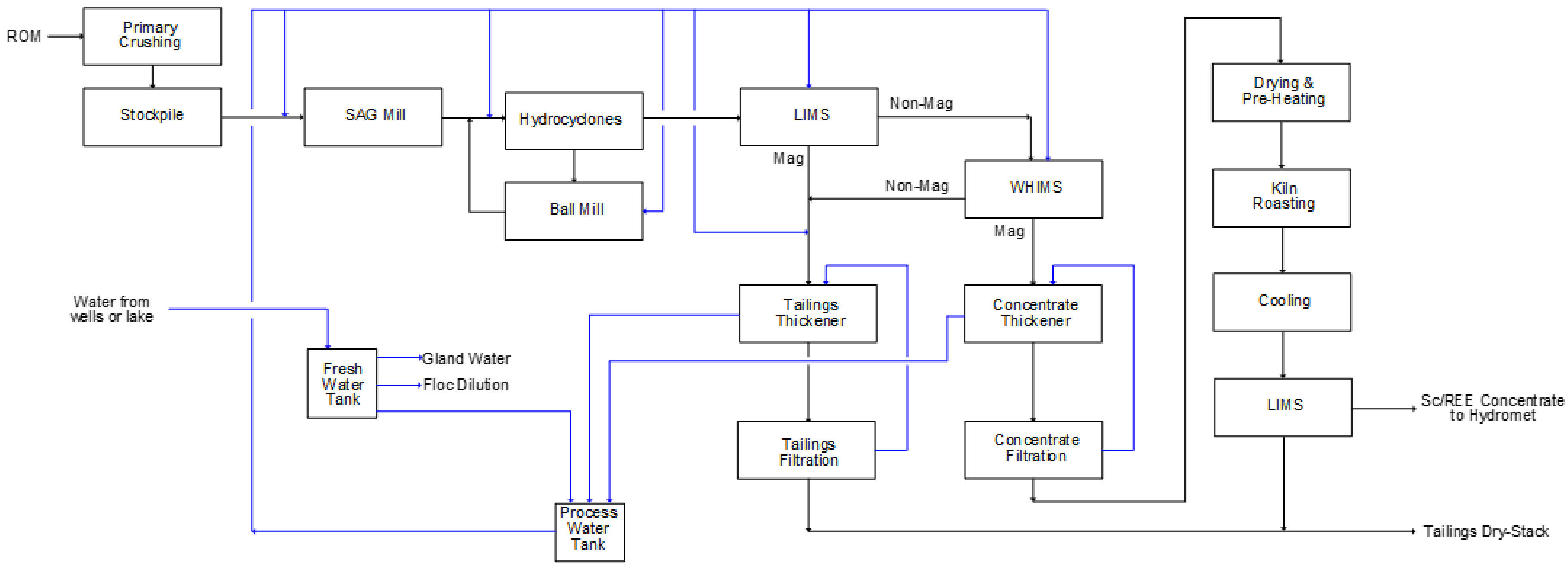
Table 17-1: Beneficiation Process Design Basis

PARAMETER	UNITS	VALUE
Total Ore Processing Rate	t/year	426,100
Daily Ore Processing Rate	t/day	2380
Scandium Oxide (Sc_2O_3) Ore Grade	g/t	282
Total Rare Earth Oxide (TREO+Y) Ore Grade	g/t	3242
Scandium Recovery to Concentrate	%	90
TREO+Y Recovery to Concentrate	%	65
Overall Concentrate Weight Recovery	%	51
Overall Concentrate Sc_2O_3 Grade	g/t	496
Overall Concentrate TREO+Y Grade	g/t	4100
Crusher Throughput Rate	t/h	132
Crusher Availability	%	75
Mill Throughput Rate	t/h	108
Mill Availability	%	92
Total Concentrate Production	t/year	218,540
Total Tailings	t/year	207,560

17.1.2 Beneficiation Flowsheet

A block flow diagram of the major unit operations of the beneficiation plant is described in Figure 17-1.

Figure 17-1: Beneficiation Plant Block Flow Diagram



17.1.3 Crushing

Mining trucks will deliver ore from the open pit mine to a single dump point located above the crusher. A hopper with static grizzly followed by a vibrating grizzly feeder will feed the ore to a jaw crusher while allowing fine material to bypass the jaw crusher. The crusher will be sized with a utilization of 75% to allow for maintenance and for delays in mine truck hauling of ore. A hydraulic rock breaker will be installed adjacent to the crusher to break up any rock bridging in the hopper grizzly. The primary crushing facility will allow for a maximum rock size of 1 m and the mine blasting plan will take this limitation into account. The jaw crusher is expected to produce a crushed ore with particle size P_{80} of approximately 125 mm. Crushed ore and the bypassed fines from the grizzly feeder will be combined on a common conveyor for stockpiling.

17.1.4 Stockpiling, Reclaiming

The crushed ore from the primary crushing facility will be conveyed to a crushed ore stockpile that will be covered to keep it dry and prevent any dust from being picked up by the wind. The stockpile will have a live storage capacity of approximately twelve (12) hours to compensate for any delays in ore deliveries from the mine or major blockages and breakdowns in the crusher. In the event of longer delays, a dozer will manipulate the stockpile in order to utilize the full amount of crushed ore present. Three (3) belt feeders will be located in a ventilated tunnel underneath the crushed ore stockpile. Two of the belt feeders should be capable of handling the full plant capacity. The belt feeders will discharge onto a reclaim conveyor that will feed crushed ore to the concentrator at a rate of 108 t/h. The placement of the belt feeders should allow for control of particle size feeding the plant.

17.1.5 Grinding, Classification

The grinding area will consist of a conventional two stage SAG mill and Ball mill circuit. The SAG mill will be open circuit and the ball mill will be in closed circuit with hydrocyclone classifiers to produce a final product P80 of approximately 100 microns. The grinding power requirements and size of the mills were based on similar historical projects as the grindability characteristics of the ore were not available at this stage of the project development. The grindability tests will be completed at the next stage of the project development.

17.1.6 Magnetic Separation

The grinding circuit product will be fed to a primary stage low intensity magnetic separation (LIMS). A single low intensity counter rotating drum unit 1200-mm diameter with field strength of up to 1,200 Gauss should be capable of handling the plant throughput. The magnetic fraction will be collected in a launder under the magnetic separator and report to the tailings dewatering and filtration circuit.

The non-magnetic fraction from the primary stage will be fed to the secondary stage high intensity magnetic separation. A single vertical pulsating high gradient magnetic separator (VPHGMS) with field strength of up to 10,000 Gauss (1 Tesla) will be considered for this second stage of magnetic separation. This design uses a combination of magnetic force, pulsating fluid action and gravity to process the material. The magnetic fraction recovered from this stage will be collected and pumped to the concentrate dewatering circuit, while the non-magnetic fraction will be pumped to the tailings dewatering circuit.

17.1.7 Tailings Dewatering

The magnetics fraction from the LIMS and non-magnetics fraction from the VPHGMS will be pumped to the tailings dewatering circuit. This will consist of a thickener and filter press. The thickener underflow will be fed to the filter press via a filter feed tank with typical residence time of 5 to 10 hours. The filter cake from the filter press will discharge onto a conveyor belt for dry stacking to the tailings storage facility (TSF). The filtrate solution from the filter press will be recirculated back to the thickener. The thickener overflow with typical clarity <100 milligrams of suspended solids per litre will be collected in the process water tank for re-use as dilution water to the various unit operations of the mill.

17.1.8 Concentrate Dewatering

The magnetics fraction from the VPHGMS will be pumped to the concentrate dewatering circuit. This will consist of a thickener and filter press. The thickener underflow will be fed to the filter press via a filter feed tank with similar residence time. The filter cake from the filter press will discharge onto a conveyor belt for further processing through drying and roasting. The filtrate solution from the filter press will be recirculated back to the thickener. The thickener overflow with typical clarity <100 milligrams of suspended solids per litre will be collected in the process water tank for re-use as dilution water to the various unit operations of the mill.

17.1.9 Drying, Roasting, Cooling

The concentrate filter cake will be dried and roasted in preparation for the final gangue mineral separation step.

17.1.10 Magnetic Separation - Dry

Roasted concentrate from the silo will be fed to low intensity magnetic separators (LIMS) via conveyors. Dry magnetic separation requires free flowing material to perform efficiently. Due to the fineness of the concentrate particles, dry magnetic separation will require multiple stages with smallest available pole pitch (distance between magnet poles) for reasonable separation efficiency. The pole pitch will be selected according to the particles size to allow for particle rotation, re-stratification of material bed and thus release of non-magnetic entrapped particles. Three dry drum magnetic separators, 1,200-mm diameter will be used for this application. The magnetic fraction will be rejected to tailings, while the non-magnetic fraction will be conveyed to the concentrate storage facility.

17.1.11 Utilities and Chemicals

Power

The beneficiation plant will have an estimated 4392 kW of installed power. Approximately 50% of the power requirement will be for the grinding process. Electrical power to the operations will be supplied from diesel generators.

Water

Process water will be required and will be largely recirculated within the plant. The overflow from the tailings and concentrate thickeners, as well as some make-up water from the lake will be fed to a process water tank which will provide water to the grinding and magnetic separation circuits. The plant will also be equipped with a freshwater tank whose bottom section will serve as the fire water source and its upper section will provide fresh gland water for pump seals, flocculant reagent preparation, and process water make-up. The plant water pipes and fittings will be in carbon steel, with pressure and temperature ratings according to ANSI B16.5 Class 150.

Compressed Air

Compressed air will be required for general plant air and instrumentation air. The compressed air system will include water cooled compressors, vertical air receivers, one instrument air dryer, and the related auxiliary equipment (oil separator and filters). The compressed air will be supplied at a nominal pressure of 100 psig for both instrument air and plant air distribution networks. The instrument air will be purified according to ISO 8573.1 Class 1-1-1 quality requirements (dew point of -70 ºC), whereas the plant air will be purified according to ISO 8573.1 Class 1-6-1.

Chemicals

The beneficiation plant will be equipped with a flocculant reagent preparation and distribution system. Flocculant is assumed to be received in 25 kg bags. No other chemicals are required for the magnetic separation process. Diesel will be required for the roasting process.

17.2 Hydrometallurgical Plant

The hydrometallurgical plant will be designed for an annual feed rate of 218,540 t of mineral concentrate and will operate 24 hours a day, all year-round with a daily throughput of 600 t, and a plant availability of 92%. The overall scandium recovery through hydrometallurgy will be 81% and will thus produce 87.3 t/a of Sc2O3, with a mixed REE product estimated to be 1,865 t/a. The Sc2O3 will be processed together with alumina to produce 2,562 t/a of Al-2%Sc Master Alloy.

17.2.1 Process Design Criteria

The main unit operations of the hydrometallurgical plant include concentrate loading and pulping, high pressure caustic leaching, hydrochloric acid leaching, scandium recovery systems through solvent extraction, mixed REE recovery by precipitation, HCl recovery system, caustic recovery system, residue neutralisation and filtration, Al-2%Sc Master Alloy through electrolysis.

The main process design criteria are listed in Table 17-2.

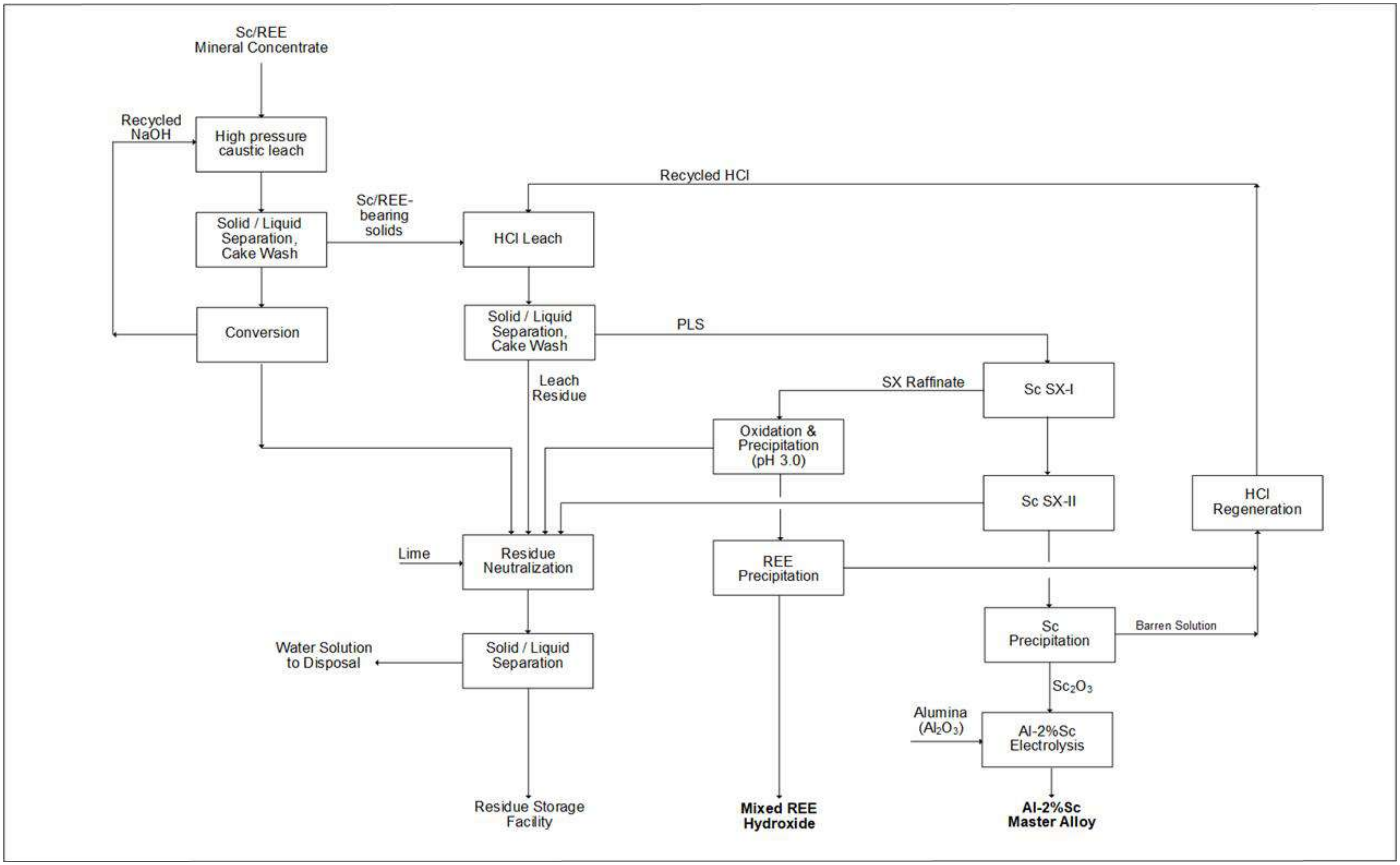
Table 17-2: Hydrometallurgy Main Process Design Basis

PARAMETER	UNITS	VALUE
Total Ore Processing Rate	t/year	218,540
Daily Ore Processing Rate	t/day	600
Scandium Oxide (Sc_2O_3) Feed Grade	g/t	496
Total Rare Earth Oxide (TREO+Y) Feed Grade	g/t	4100
Caustic Leach Feed Rate	t/h	27
Scandium Recovery	%	81
Total Reagents used	t/year	140,375
Total Solids Residue	t/year	295,264
Total Liquid in Residue	t/year	61,626
Total Sc_2O_3 Production	t/year	87.3
Total Mixed REE Production	t/year	1,865
Total Al-2%Sc Master Alloy Production	t/year	2,562

17.2.2 Hydrometallurgical Flowsheet

A block flow diagram of the major unit operations of the hydrometallurgical plant is described in Figure 17-2.

Figure 17-2: Hydrometallurgical Plant Block Flow Diagram



17.2.3 High Pressure Caustic Leach

Concentrate Transfer and Storage

Concentrate from Crater Lake will be transported by pneumatic conveyor to a storage silo. The storage silo will have a storage capacity of about 1 week. The concentrate will then be transported from the silo to the pulping area by pneumatic conveyor.

Pulping

The concentrate will be mixed with circulating liquids such as filtrate and wash filtrate from the 1st stage autoclave residue filtration in an agitated tank. Sodium hydroxide (NaOH) will be added and adjusted in ratio to the incoming concentrate feed.

Autoclave Feed and Pre-Heating

From the agitated pulper tank, the slurry is pumped to an agitated autoclave feed tank. The autoclave feed take will be equipped with a vibrating screen to reject oversized particles and trash material. The purpose of the feed tank is to provide buffer capacity of about 3 hours before the autoclave to minimize disturbances in autoclave operation. Slurry density and pH will be monitored, and if necessary controlled in the autoclave feed tank by the addition of process water and NaOH solution. From the autoclave feed tank, slurry is pumped to the autoclave through a direct contact tray type pre-heater where slurry is heated with autoclave flash steam. Here the slurry is heated to maximum 165 °C and 6 bar(g) pressure by using high pressure flash steam from the flash vessel. Slurry from the pre-heater is pumped to the autoclave using a positive displacement pump.

Autoclave Pressure Leaching and Flashing

In the autoclave, Sc and REE are leached under high pressure and temperature. The autoclave consists of six mechanically agitated compartments and operates at max 220°C and approx. 26 bar(g). Autoclave temperature is controlled by direct high pressure steam injection. Pressurized air is provided by a compressor system and injected to the vessel gas phase above slurry surface. Autoclave pressure is controlled by pressure control valve systems.

Slurry from the autoclave is released by pressure difference through two-stage flashing. The purpose of the two-stage pressure-controlled flashing is to decrease pressure and temperature from the autoclave operating conditions to atmospheric pressure and temperature, in a controlled manner. Quenching water feed lines are connected to autoclave flashing area pipelines to provide optional cooling down of the slurry and prevent cavitation due to elevations and pressure loss in pipelines.

In the high-pressure flash vessel, the pressure will be reduced to about 7 bar(g) by a pressure reduction valve. Simultaneously the temperature will drop from 220 °C to approx. 165 °C. Vapour generated in the flashing is fed to the pre-heater to heat the autoclave feed slurry. The slurry flows by pressure difference to the atmospheric flash vessel.

In the atmospheric flash vessel, pressure is reduced from 7 bar(g) to atmospheric pressure by a pressure reduction valve. The temperature will also drop to approximately 100 °C. Vapour generated in the atmospheric flashing stage will be fed to the off-gas scrubber. The slurry will then be pumped to leaching area filter feed tank.

Gas Scrubber

Leaching gases, i.e., off-gases from the pre-heater, autoclave and the second flashing stage, are collected and treated in a venturi gas scrubber. Solids carryover in the off-gas are washed out by spraying water through nozzles to the venturi throat. The wash water is collected in the scrubber internal water reservoir and circulated back to the nozzles. Make up / freshwater is added to the reservoir and is controlled by the cooling demand (temperature measurement) of the liquid in the gas scrubber circuit. Bleed from the wash water circulation is pumped out regularly to remove solids and dissolved impurities from the wash circulation. Scrubber water is fed to the filtering area and used for cake washing.

Caustic Leach Thickening / Filtration

After pressure leaching, the slurry will be fed to the caustic leach thickener and filtration package for solid-liquid separation. The filtration cycle will include cake washing. The filter cake will be conveyed to the HCl leach circuit at a solids rate of 24 t/h. Filtrate and wash water is fed back to the thickener. Leach solution residue from thickener overflow consists mainly of sodium silicate Na_2SiO_3 + H2O, sodium aluminate $2NaAlO_2$ + H_2O and other gangue minerals.

Conversion

Sodium hydroxide (NaOH) is regenerated from pressure leach sodium silicate (Na_2SiO_3) waste solution with the addition of lime. Lime will be fed to an agitated tank in ratio to the sodium silicate content of the caustic leach waste solution. Slurry from the conversion reactor will be fed to a filter press via a filter feed tank. The filter cake from the filter press will discharge onto a conveyor belt for further processing through residue neutralization. The filtrate solution will contain the recovered NaOH and will be recirculated back to the autoclave feed tank.

17.2.4 Hydrochloric Acid Leach

Leach Solution Preparation and Leaching

Filter cakes from the caustic leaching filter will be fed to the HCl feed pulper via screw conveyor to a covered leach reactor vessel with agitator. Thirty five percent hydrochloric acid solution and regenerated HCL will be fed to the incoming filter cake mass. The HCl leach residence time is one hour at ambient temperature.

Leach Filtration and Cake Washing

The resulting slurry will be pumped to the HCl leach residue filtration package. The pregnant leach solution will be separated from the solid residue material. The filter cake will be washed with water before conveying it to residue neutralization.

Iron Reduction

The PLS will be pumped to a covered agitated vessel for iron reduction from ferric to ferrous according to the reduction reaction $2Fe^{3+} + Fe \rightarrow 3Fe^{2+}$. This will limit the co-extraction of iron with scandium by the solvent extraction organic. Iron filings or powder will be added in ratio to the iron concentration of the incoming liquor and heated to 90 °C. The resulting solution will be pumped at a rate of 45 m3/h to the solvent extraction circuit for Sc and REE recovery.

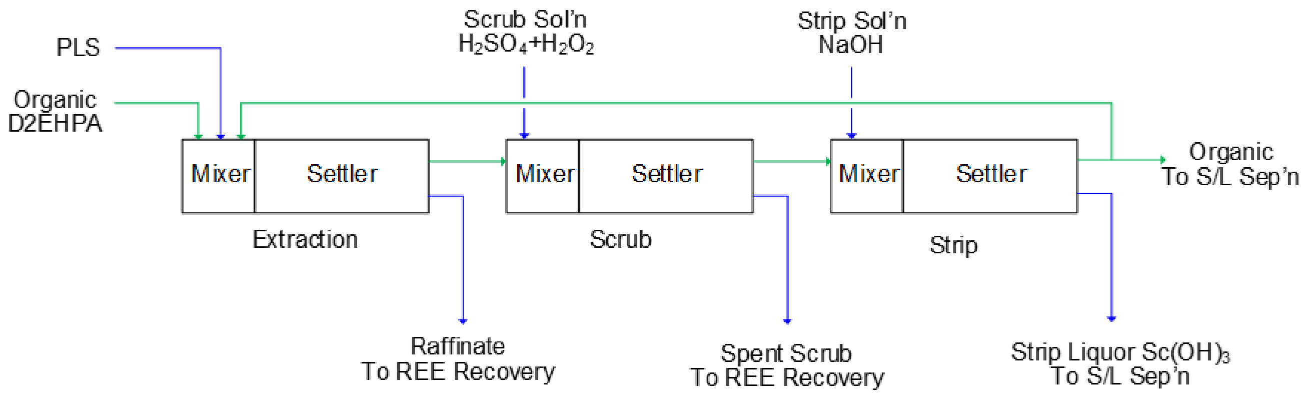
17.2.5 Scandium Recovery systems

Solvent Extraction SX-I

The use of solvent extraction as a means of scandium in solution is central to the process. Figure 17-3 summarizes the solvent extract SX-I process. The unit operation will consist of three stages:

- SX-I Extraction: The organic reagent selected to extract Sc from the leach solution is Di (2-Ethylhexyl) Phosphoric Acid (D2EHPA), modified with tributyl phosphate (TBP) in an organic diluent Shellsol D90. The extraction circuit produces a Sc loaded organic with some impurity metals and a Sc depleted raffinate. The raffinate will be sent for further treatment to recover REEs.
- SX-I Scrubbing: Sulfuric acid (H_2SO_4) and hydrogen peroxide (H_2O_2) will be used as the scrub solution to primarily remove co-extracted titanium from the loaded organic phase. The scrubbed organic containing Sc will be advanced to stripping.
- SX-I Stripping: Sodium hydroxide (NaOH) will be used to strip scandium from the loaded organic. It is normally conducted under conditions in order to produce a strip liquor containing a high concentration of $Sc(OH)_3$ solids. The expected Sc recovery to the strip liquor is 97%. The $Sc(OH)_3$ is the product of the SX-I circuit and will proceed to filtration. The Sc-depleted organic will be recycled to the extraction stage.

Figure 17-3: Solvent Extraction Process SX-I



The extraction, scrubbing, and stripping process will take place in mixer-settlers operating in series such that any one mixer-settler can be taken off-line for maintenance. Due to the high vapour pressure of the organic solution, the entire system will need to be closed and operated under slight positive pressure with an inert blanket and overpressure relief valves. Gases vented from this process will be sent to a scrubbing system. This process area will also need retainment dykes to contain and pump out spills to an oil/water separator within the effluent treatment system.

Optimization of the hydrometallurgical flowsheet will continue in future development programs, prior to the pilot program, as well as sizing and quantity of mixer settlers required for each stage.

Scandium Hydroxide Slurry Filtration

The aqueous stream leaving SX-I will pass through a filtration stage to recover the $Sc(OH)_3$ solids. A vacuum belt filter is considered for this step however a centrifuge can also be used.

As for the organic phase, part of the stream from the stripping stage will be diverted for treatment with diatomaceous earth where impurities that have accumulated in the organic phase can be adsorbed. The resulting slurry can be pumped to a centrifuge decanter which can separate the solids, organics and aqueous solution that might have been entrained.

$Sc(OH)_3$ Precipitate Releach

A solution of H_2SO_4 will be used to releach the $Sc(OH)_3$ solids in preparation for the second solvent extraction process (SX-II) where co-extracted impurities such as Zr and Nb are removed. At this stage of the process, the $Sc(OH)_3$ filter cake solids rate will be approximately 30 kg/h. This will be fed directly to a leach reactor via screw conveyor. Dilution water and H_2SO_4 solution will be added in ratio to the incoming filter cake mass to leach the $Sc(OH)_3$ solids. The releach aqueous solution will be directed to SX-II for removal of Zr and Nb.

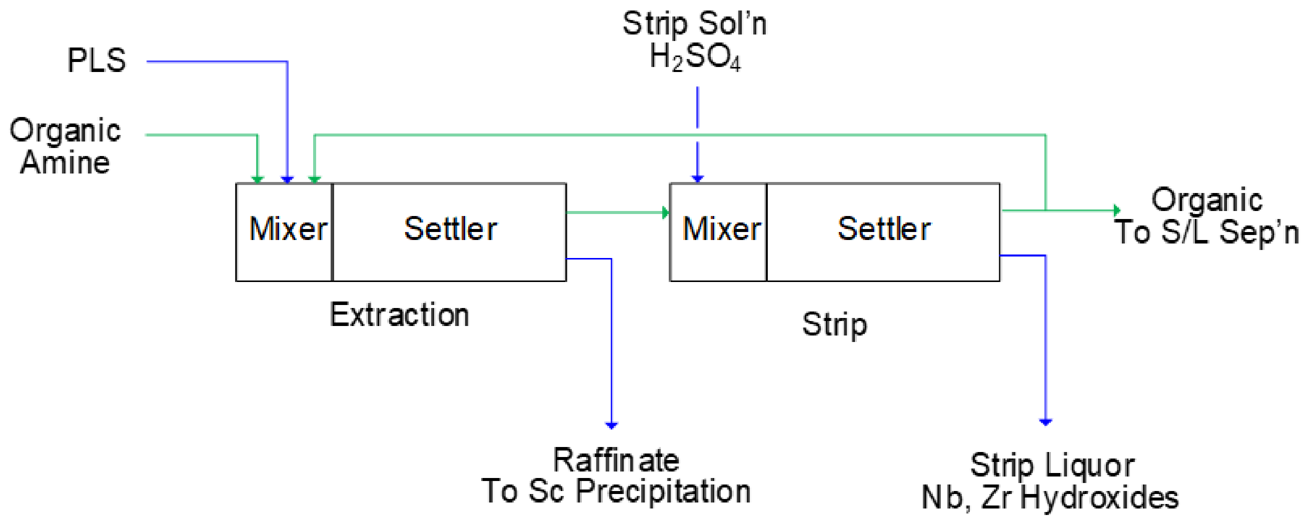
Solvent Extraction SX-II

The second solvent extraction process SX-II will operate under the same principles as the first SX-I, the key difference being that the compounds extracted by the organic phase are impurity metals, while the raffinate aqueous phase will contain Scandium. This process will not require a scrubbing stage. Figure 17-4 summarizes the solvent extract SX-II process. The unit operation will consist of two stages:

— <u>SX-II Extraction:</u> The extraction circuit will load the organic phase with impurity metals such as Nb and Zr to produce a Sc rich raffinate aqueous phase. The raffinate will be sent for further treatment to precipitate Sc. The organic reagent selected to extract impurity metals from the leach solution is an Amine extractant in Shellsol D90.

— <u>SX-II Stripping</u>: Sulfuric acid (H_2SO_4) solution will be used for stripping the Nb and Zr from the amine extractant organic. The resulting strip liquor containing niobium and zirconium will be directed to residue neutralisation. After stripping, the organic will be recycled to the extraction stage.

Figure 17-4: Solvent Extraction Process SX-II



Sc Precipitation

Scandium precipitation is performed in an agitated covered precipitation vessel where Oxalic acid ($C_2H_2O_4$) and dilution water are added to the raffinate. The resulting precipitate will be in the form of Scandium Oxalate ($Sc_2(C_2O_4)_3$). Sulfuric acid will be a by-product of this reaction and will be used to regenerate Hydrochloric acid. Precipitation involves the following reaction:

$$Sc_2(SO_4)_3 + 3C_2H_2O_4 \rightarrow Sc_2(C_2O_4)_3(s) + 3H_2SO_4$$

The precipitation residence time will be 1 hour. The estimated precipitate product solids rate will be 23 kg/h.

Sc Filtration

The Scandium Oxalate precipitate will be filtered, while the Sulfuric acid filtrate will be directed to the HCl regeneration process. This can be done by vacuum belt filter or filter press or even centrifuge.

Sc Calcination

Scandium Oxalate filter cake will be conveyed to a calcination kiln where its temperature slowly rise up to 1000 °C. Initially, the free water will evaporate from the surface of the $Sc_2(C_2O_4)_3$ particles, followed by a thermal decomposition yielding Scandium Oxide (Sc_2O_3) according to the following reaction:

$$Sc_2(C_2O_4)_3(s) \rightarrow Sc_2O_3(s) + 3CO_2(g) + 3CO(g)$$

No additives are required for the calcination The calcination residence time will be 1 hour. The estimated calcined product rate will be 11 kg/h. The product temperature will be reduced to about 80 °C in an indirect contact cooler with air as the cooling medium. The cooled product will be conveyed to a storage silo while the heated air will partially serve as secondary air in the kiln. A 12-hour retention time in the storage silo will offer a degree of buffering between the production of Sc_2O_3 and the downstream Al-2% Master Alloy production.

17.2.6 REE Recovery Systems

The aqueous raffinate, as well as the spent scrub solution from SX-I will undergo selective precipitation of Fe at pH 3, followed by precipitation of REE's at pH 6.

Oxidation and Precipitation

Limestone will be used for HCl neutralisation, as well as Fe removal according to the following reactions:

$$2HCl + CaCO_3 \rightarrow CaCl_2 + H_2O + CO_2$$

$$2FeCl_3 + 3CaCO_3 + 3H_2O \rightarrow 2Fe(OH)_3 + 3CaCl_2 + 3CO_2$$

The removal of impurity iron by precipitation will occur at a pH of 3 and a temperature of 60 °C.

Filtration

The impurity metal hydroxides will be filtered out and directed to residue neutralisation, while the remaining filtrate solution containing CaCl2 will be directed to a REE precipitation vessel.

REE Precipitation

Lime will be used to precipitate the REEs according to the following reaction:

$$2ReCl_3 + 3Ca(OH)_2 \rightarrow Re_2(OH)_3 + 3CaCl_2$$

The removal of REEs by precipitation will occur at a pH of 6 and a temperature of 60 °C.

REE Filtration

The REEs will be filtered out as final by-product, while the remaining filtrate solution containing $CaCl_2$ will be directed to HCl regeneration.

17.2.7 HCl Regeneration

HCl Recovery from $CaCl_2$

Sulfuric acid and acid (H_2SO_4) from the Sc precipitation step and Calcium chloride ($CaCl_2$) from the REE precipitation step will be used to regenerate HCl according to the following reaction:

$$CaCl_2(aq) + H_2SO_4(aq) + 2H_2O \text{ ---> } 2HCl(aq) + CaSO_4.2H_2O(s)$$

No laboratory testwork on the HCl regeneration has been done at this stage of the project development, however, a 95% $CaCl_2$ transformation rate has been assumed. The reaction will be performed in a covered vessel with an estimated one hour residence time. The resulting gypsum by-product will be filtered and directed to the residue neutralisation process, while the HCl filtrate solution will be processed through an evaporator to remove enough water so that its strength can be increased to approximately 20%. At this point it will be suitable for re-use in the high pressure caustic leach filter cake leaching step.

17.2.8 Residue Management

The calcium silicate from caustic regeneration, acid leach residue filter cake, the SX-II strip solution containing Nb and Zr impurities, together with the acid recovery filter cake will be neutralized with calcium oxide (CaO) in a covered vessel. The resulting vessel discharge will be filtered. The filter cake material will be dry-stacked to tailings disposal, while the filtrate will be pumped to a disposal area for further water treatment.

17.2.9 Al-2%Sc Master Alloy

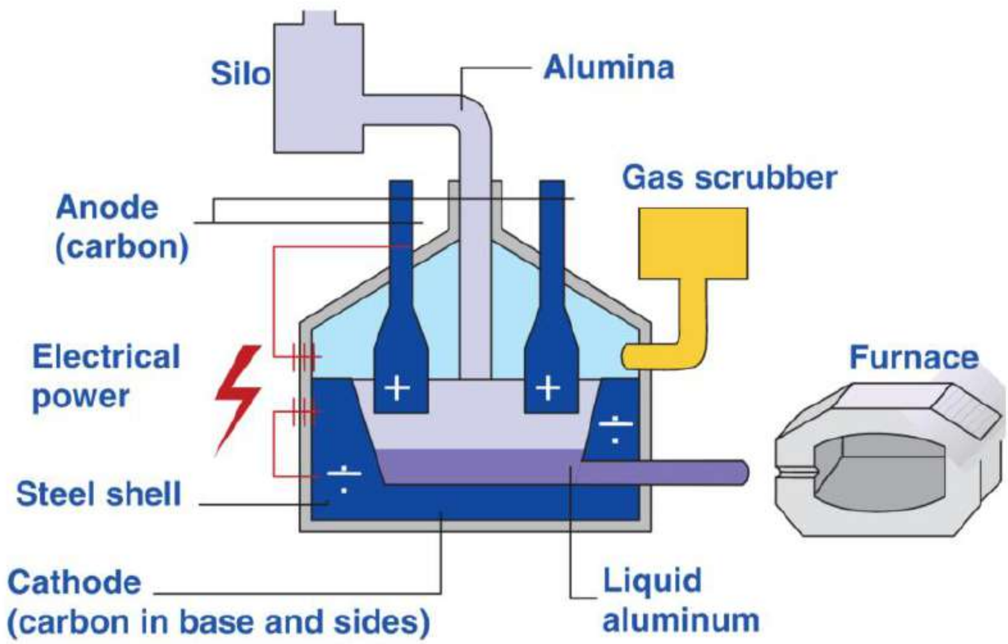
The Al-2%Sc master alloy will be produced by direct electrolysis of alumina (Al_2O_3) and scandia in a process similar to the Hall-Héroult electrolytic smelting process. The co-deposition of aluminum and scandium occurs according to the following reactions:

$$2Al_2O_3 + 3C \rightarrow 4Al_{(l)} + 3CO_{2\,(g)} \quad \text{and} \quad 2Sc_2O_3 + 3C \rightarrow 4Sc_{(l)} + 3CO_{2\,(g)}$$

Both direct Sc electrolysis into an aluminum cathode and co-deposition of Al and Sc have been investigated at a bench scale by researchers. The co-deposition has the advantage of minimizing the formation of intermetallics with high scandium content. The development of Al-2%Sc master alloy flowsheet is yet to be developed for this project and will be completed at the next stage of project development. However, co-deposition of Al and Sc in an electrolytic bath of mainly molten cryolite (Na_3AlF_6) at a temperature of approximately 960 °C has been assumed.

Prebake cells will be used for this project and the carbon anodes will be purchased as a raw material. The cells will be the centre worked (CWPB) prebaked cells with automatic multiple point feeding of alumina and scandia. Three cells will be considered for an annual alloy production of about 2,500 tonnes. In addition to the prebake cells, auxiliary infrastructures are required to operate the complete Al-2%Sc master alloy production plant. These include the alumina receiving, storage and feed preparation system, the gas scrubber system, the electrolytic cells, the holding furnace, and the alloy ingot caster. Figure 17-5 illustrates the basic industrial aluminum production process. Kvande, H. and Drablos, PA. 2014. The Aluminum Smelting Process and Innovative Alternative Technologies.

Figure 17-5: Al-2%Sc Alloy Production Process



17.2.10 Utilities and Chemicals

Power

The hydrometallurgical plant will have an estimated 6850 kW of installed power. More than 50% of the power requirement will be for the Al-2% master alloy electrolysis process and about 30% used for the caustic pressure leaching process. Electrical power will be supplied from the grid.

Steam

A steam production and distribution system will be required to produce saturated steam for process heating in the caustic leaching process, water evaporation and other miscellaneous usages. The pressure and temperature ratings of the feedwater, condensate and saturated steam pipework will be according to ANSI B16.5 Class 150.

Water

Process and boiler make-up water will be required. The plant water pipes and fittings will be carbon steel, with pressure and temperature ratings according to ANSI B16.5 Class 150. The chilled and hot water piping will be insulated for heat conservation. The river water supply will be used for the cooling tower make-up water.

Compressed Air

Compressed air will be required for general plant air and instrumentation air. The compressed air system will include water cooled compressors, vertical air receivers (wet and dry), one instrument air dryer, and the related auxiliary equipment (oil separator and filters). The compressed air will be supplied at a nominal pressure of 100 psig for both instrument air and plant air distribution networks. The instrument air will be purified according to ISO 8573.1 Class 1-1-1 quality requirements (dew point of -70 ºC), whereas the plant air will be purified according to ISO 8573.1 Class 1-6-1.

Chemicals

The chemical products required for the plant operation can be classified in three main categories:

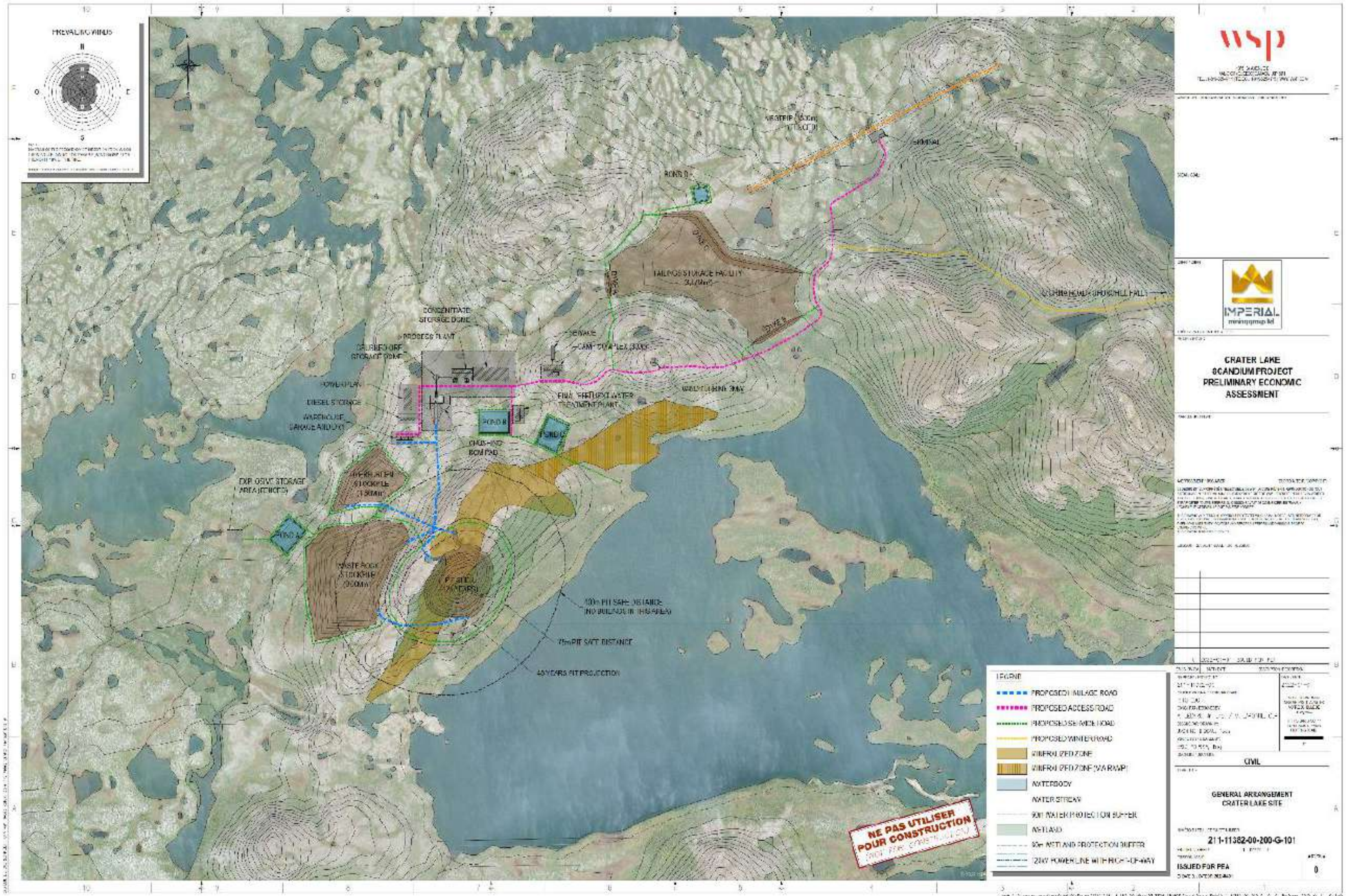
- Inorganic liquids shipped by tanker truck and consumed in large quantities during normal operation. This category includes:
- Sodium hydroxide (NaOH) for caustic pressure leaching and as strip solution for SX-I
- Hydrochloric acid (HCl) for primary leaching of the metals of interest and re-leaching of $Sc(OH)_3$ precipitate,
- Sulfuric acid (H_2SO_4) for scrub solution in the SX-I process, as well as strip solution for the SX-II process and for the regeneration of HCl.
- Hydrogen peroxide (H_2O_2) in combination with H_2SO_4 for scrub solution in the SX-I process.
- Organic solvents shipped in tote tanks. These chemicals are consumed in small quantities during normal operation and include:
- Di (2-Ethylhexyl) Phosphoric Acid (D2EHPA) used as the primary organic reagent for extraction in the SX-I process.
- Shellsol D90 used as an organic diluent for the SX-I process.
- Amine as the organic reagent in the extraction step of SX-II process.
- Miscellaneous solid chemicals shipped in bulk and used for various purposes in the plant, including:
- Quick Lime (CaO) for the caustic leach process to regenerate NaOH thru conversion of sodium silicate. It is also used in the precipitation of REE's and as a neutralizing agent for the final residue streams.
- Limestone ($CaCO_3$) for HCl neutralisation and removal of Fe and Al in the REE recovery process.
- Iron Filings (Fe) for the reduction of iron from ferric to ferrous to keep it in solution.
- Oxalic Acid ($C_2H_2O_4$) for the precipitation of Sc after solvent extraction.
- Alumina (Al_2O_3) for the Al-2%Sc electrolysis process
- Aluminum Fluoride (AlF_3) for the Al-2%Sc electrolysis process
- Cryolite (Na_3AlF_6) as the electrolyte in the Al-2%Sc electrolysis process
- Carbon Anode (C) for the Al-2%Sc electrolysis process

18 PROJECT INFRASTRUCTURE

The project includes three different sites: Crater Lake Project Site, Emeril Rail loading station, and Sept-Îles Hydrometallurgical Plant.

A map of the infrastructure of Crater Lake is included as Figure 18-1.

Figure 18-1: Crater Lake Site Infrastructure Map



The Project envisions the construction of the following key infrastructure items:

Crater Lake Project Site

- Access, haulage, and service roads
- Open pit mine
- Waste rock and overburden stockpiles
- Camp complex and services
- Multiservice building (truck shop, warehouse, dry and mine offices)
- Surface water management ponds, ditches, and pumping stations
- Tailings storage facility (TSF)
- Final effluent water treatment plant (WTP)
- Mineralized material cruising station and storage dome
- Process plant
- Concentrate storage area
- Diesel generators power plant and storage
- Air strip with terminal building
- Wind turbine
- Explosive and magazine storage

Emeril Rail Loading Station

- Access road
- Concentrate storage area
- Rail loading facility
- Railway connection

Sept-Îles Site

- Access, haulage, and service roads
- Hydrometallurgical plant
- Surface water management ponds, ditches, and pumping stations
- Filtered tailings storage facility
- Final effluent water treatment plant
- Gatehouse
- Warehouse and reagent storage area
- Railway connection and concentrate unloading area
- Concentrate storage area
- Electrical substation

18.1 Site Roads

18.1.1 Crater Lake Site

The main access road to the site will connect to the road to Emeril to reach the airstrip, the camp complex, the process plant, the multiservice building, and the final effluent water treatment plant. This service road is an 8m wide gravel road with ditches on both sides.

The haulage road will be 21m wide and will connect the open pit, the waste stockpile, the overburden stockpile, the truck shop, and the crushing station. Contact water will be collected by ditches and directed to collection ponds.

The service roads are single-track 5m wide roads for inspection and maintenance around the stockpiles, ponds, and open pit.

18.1.2 Sept-Îles Site

The main access road to the site will connect to the existing Lac Daigle Street and will be capped with gravel. The single-track service road will allow TSF and pond inspection and maintenance, while the haulage road will be used to transport the filtrated tailings on the TSF. All roads will be constructed with ditches on each side to collect water.

18.1.3 Logistics - Route to Emeril Rail Station

— Figure 18-2 shows the roads to the Emeril Rail Station.

— Presented in blue is the winter road. It will be recreated every year. It will be used from the beginning of December to the end of March by bulk concentrate trucks hauling the prior year's production output and by tractor trailers moving in most of the supply requirements for the ensuing year's production. Its route was determined by IPG and crosses land and water. It is 152km in length from thee Crater Lake site to the end of the existing Orma Lake Road. Drilling and blasting of rock cuts, profiling with fill, ditching, culverts, and bridges are not included. An average travel speed of 40 km/hr is anticipated. More than 43 haul trucks and an estimated 18 supply trucks will travel the road daily. Convoying of the trucks has been assumed with four trucks per convoy leading to a determination of the need for 16 passing lane locations along the route resulting in the convoys meeting each other every 30 minutes. The empty trucks will pull over to let the loaded trucks pass without slowing down.

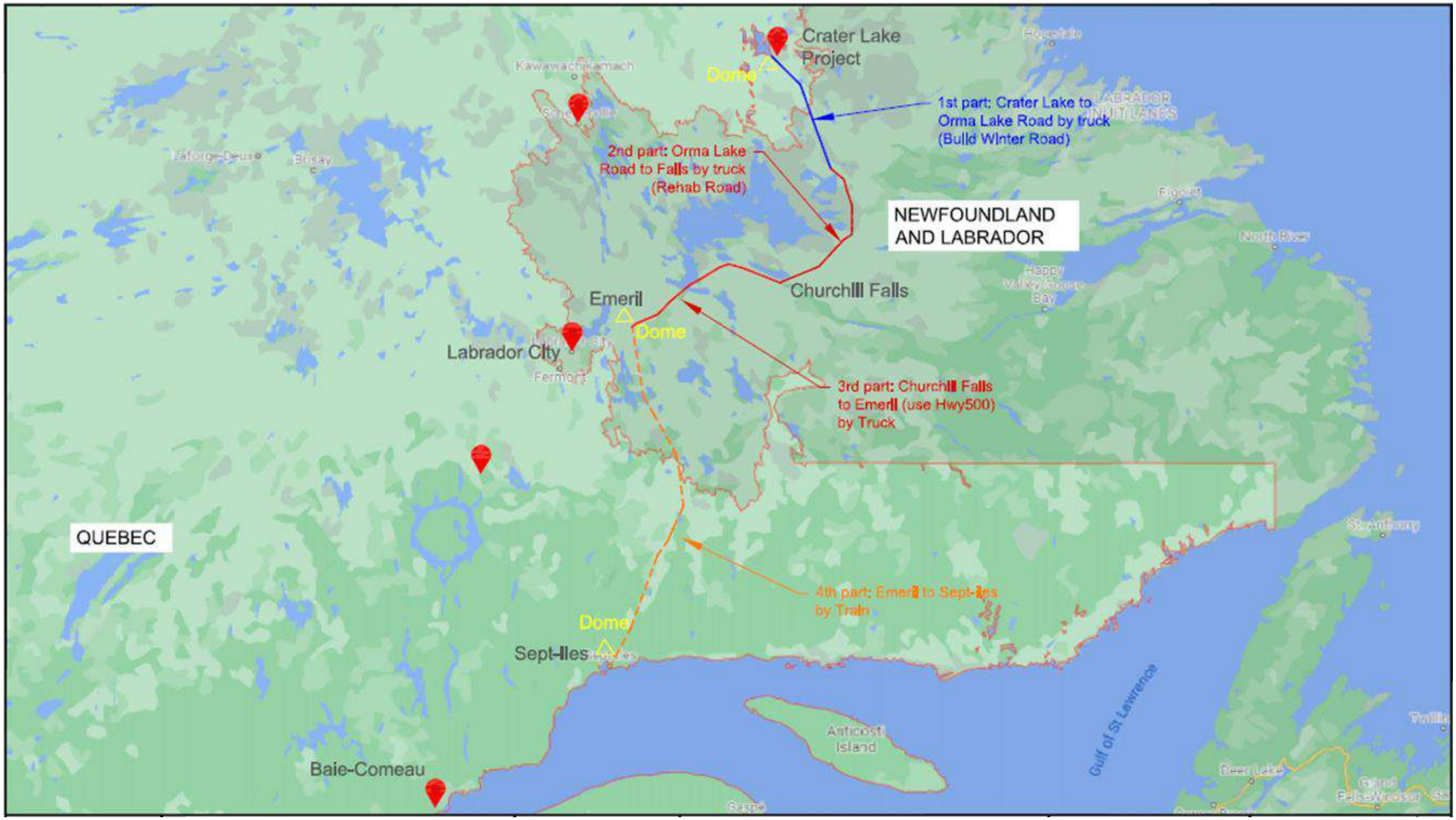
— Presented in red is the existing Orma Lake Road. It will be refurbished as a one-time event during the pre-production period for use of the haul trucks as a double lane gravel road that would have load restrictions were it used during the spring thaw. It is 150km long from the north side of Orma Lake to Churchill Falls and the highway 500 intersection. Existing single lane bridges will be retained and upgrade to an all-season access road will not be required. An average travel speed of 43 km/hr is anticipated due to it being a double lane road with minimal time loss during passing. More than 43 haul trucks and an estimated 18 supply trucks will travel the road daily. Convoying of the trucks has been assumed with four trucks per convoy.

— From the Churchill Falls intersection, the haul trucks will travel 199km to the west along the existing and provincially maintained year-round highway 500 to the Emeril railroad station at an average speed of 80km per hour. The round trip for a haul truck is calculated to be 19.7 hours, leading then to an assumed 2 operators per truck requirement.

— From the Churchill Falls intersection, the supply trucks will travel 289km to the east along the existing and provincially maintained highway 500 to the port of Happy Valley, NL at an average speed of 80km per hour. Existing port facilities are expected to store the supplies brought to the Happy Valley Port by Ocean Freighters during the summer season for highway haulage during the winter months. Other than Crater Lake site warehousing, no added storage facilities for supply storage are allowed for along the overall 638km length route. A round trip for a supply truck is calculated to be 22.6 hours. The supply trucks will thereby also need two operators per truck to avoid the doubling up on truck traffic.

— With the lag between production and shipment, concentrate storage domes are included at both the Crater Lake and Emeril locations. Each storage dome location (Emeril and Crater Lake) will be capable of holding a full year's supply of concentrate. The full year sized storge dome at Emeril allows the rail shipments to as required by the hydrometallurgical facility at Sept-Îles. As such, no storage dome is needed at the Sept-Îles facility.

Figure 18-2: Access Road from Crater Lake to Emeril



18.2 Power Supply

18.2.1 Crater Lake Site

The Crater Lake site will not be connected to the power grid. Therefore, a diesel power plant with five 2.7MW generator set will be installed. The configuration will be N + 2 at a voltage of 4160 Volts. Main switchgear will also be installed in the power plant building. Expected load in operation is 6.2 MW.

Site distribution overhead lines will be at a voltage of 25kV. A 12 pair fibre optic will be installed along this overhead line, connecting all buildings and infrastructure.

The diesel storage farm is sized to accommodate all year storage, so four 10-m diameter tanks are planned for a total storage capacity of 24 million liters.

A 3MW windmill is also planned to be installed in year five of the mine operation to reduce electricity operational cost and significantly decrease carbon footprint. Such bi-energy systems are now commonly installed for mining projects.

18.2.2 Sept-Îles Site

The Sept-Îles site will be connected to the Hydro-Quebec grid. Expected load in operation is 7.3 MW. Next phase will allow to confirm connection point with Hydro-Quebec. The main substation will be installed on site and distribute 25kV feeders to process plant and other infrastructure.

18.2.3 Emeril Site

The Emeril Rail Loading Station is assumed to get power from existing infrastructure.

18.3 Site infrastructure

18.3.1 Crater Lake Site

The Crater Lake Site Camp complex will be a single-story pre-engineered, and prefabricated modular building including a 200 rooms dormitory, a reception area, a kitchen, a dining room, a lounge, an exercise room, and an infirmary. The modules will be installed on a gravel pad on hydraulic jacks and connected with closed corridor.

The multiservice building will include a truck shop (3 maintenance bays, 1 wash bay), a warehouse, the mine rescue office, the locker room / dry area, the mine offices, and conference / training rooms.

To allow site access all year round, a 1,830m long (6,000 ft) airstrip on gravel surface will be constructed to allow take-off and landing of Boeing 737-200 or equivalent planes. The 30 m wide airstrip will be equipped with all required instrumentation and equipment to meet applicable aviation regulations and standards. The airstrip area will be fenced for safety and a terminal building will erected nearby.

The potable water is expected to be pumped from a well, further studies will be required to confirm this assumption. The potable water treatment and distribution system will be shipped in a prefabricated building installed near the camp complex. The sewage treatment system will also need more studies for soil characterization but should also be modular type. A fire protection pumping station with adequate water storage will be installed with hydrants to protect personnel, equipment, and buildings.

Each water catchment pond will have a barge-type pumping station and HDPE piping to redirect water to the final effluent water treatment plant. The open pit will have a mobile diesel pumping station. Each will be controlled either locally or remotely in automatic mode with a level sensor.

The explosive and magazine storage will meet the regulations for the minimal distances. The building will be supplied by vendor.

Site Internet access will be via satellite connection. A Wi-Fi network will be installed to allow communication inside the buildings and an optic fibre network will link all buildings and infrastructure together for IP phone, corporate network, automation network, cameras, and fire alarm systems

18.3.2 Sept-Îles Site

As shown in Figure 18-3, a modular gatehouse will be installed to monitor and control site access. The site area will be fenced and equipped with camera system for security issues.

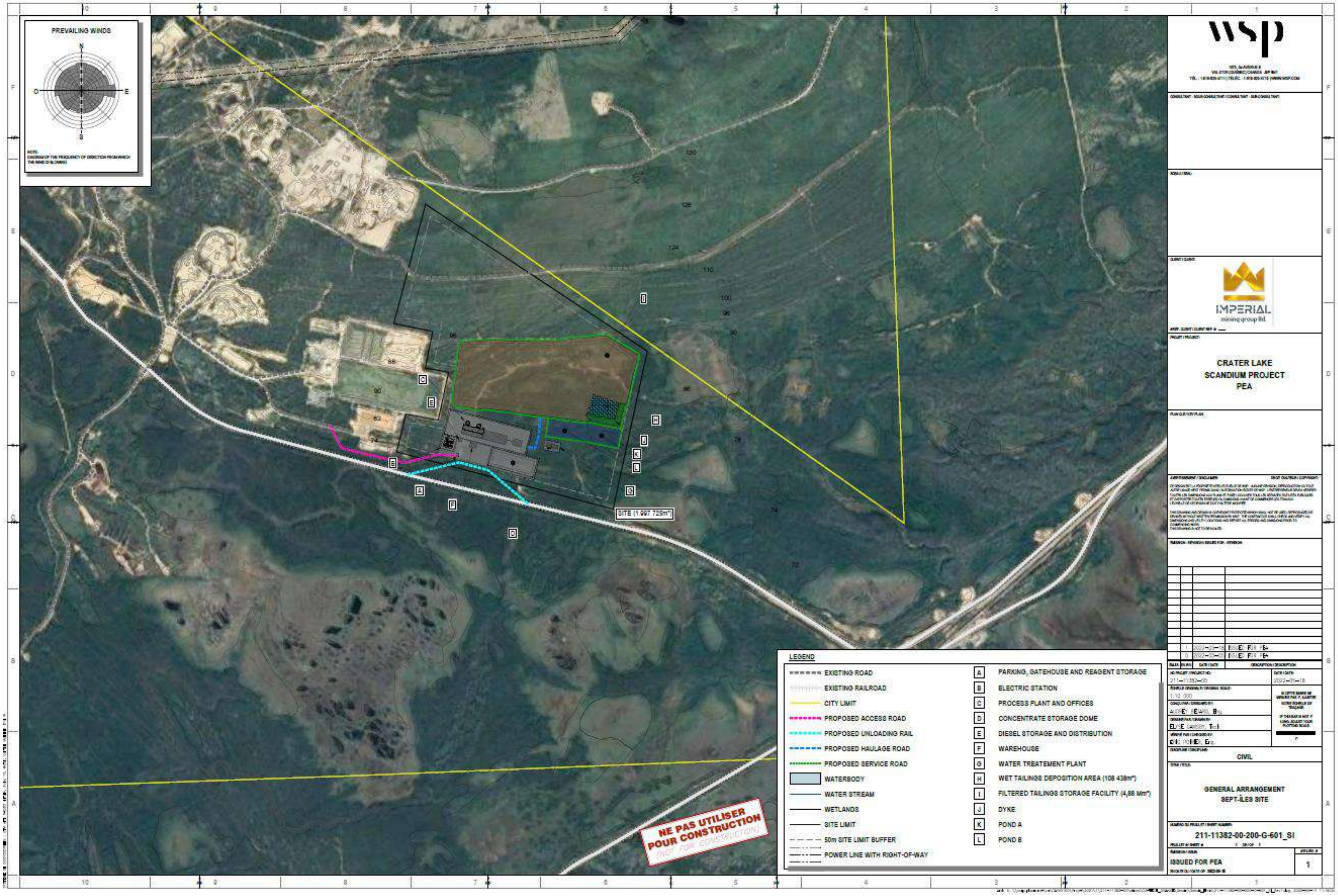
For the concentrate loadout area, a railway will be connected to the existing rail running south of the property, as per owner's standards. The bulk concentrate covered wagons will be unloaded from the railcars by an excavator and stored prior to feeding the hydrometallurgical plant.

Municipal services for potable water and sewage are expected to be available to the property limits and connected to the Process plant building. A fire protection pumping station with adequate water storage near the process plant will be installed with hydrants to protect personnel, equipment, and buildings.

Catchment pond will have a barge-type pumping station with HDPE piping to redirect water to the final effluent water treatment plant. The stations will be controlled either locally or remotely in automatic mode with a level sensor.

Site Internet access will be accessible from a local provider and a Wi-Fi network will be installed to allow communication inside the buildings. Cellular phones are functional in this area.

Figure 18-3: Sept-Iles Site Infrastructure Layout

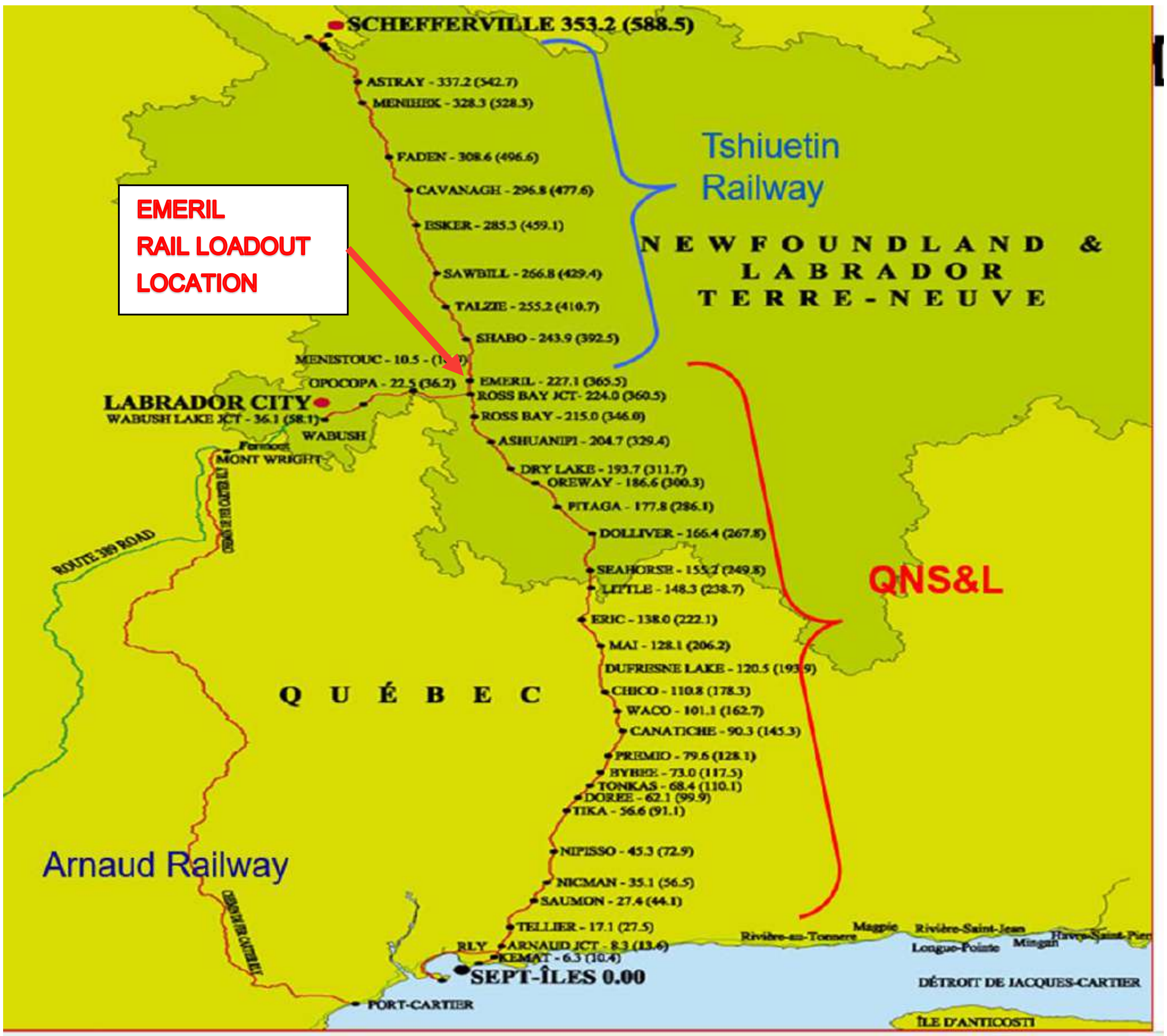


18.3.3 Logistics - Emeril Rail Loadout

WSP assumed for the PEA that QNS&L will be able to ship the bulk concentrate using company owned wagons from Emeril, NL to Sept-Îles, QC. The loading and unloading reflects equipment and labour provided by Imperial Mining Group and includes the handling of the covers and the sealing of the rail cars (wagons) with no involvement of QNS&L. An added siding has been allowed for at the existing Emeril Rail Station for parking and loading of the empty wagons. The acquisition of land for the siding and dome storage must be arranged by IPG as well as confirmation through negotiations with QNS&L of the suitability of this location.

Figure 18-4 provides the details pertaining to the location of Emeril at the junction of the QNS&L line and the Tshiuetin Railway to the north.

Figure 18-4: Emeril Railway Siding Location Map



18.4 Tailings Storage

The following Items describe the Tailings Management Facilities (TMF) on the Crater Lake site and on the Sept-Îles site. The proposed designs are based on the available information and will need to be reviewed at the feasibility stage.

18.4.1 Crater Lake Site Tailings Storage Facility

Design and General Considerations

The TMF at the Crater Lake site will allow for the storage of the total forecasted volume of tailings produced over the LOM in a single cell. The LOM is expected to be 25 years with a tailings production rate of 208,636 tpy (138,908 m^3/year). For the purpose of this PEA, it is assumed that only slurry (unthickened) tailings will be produced from the Crater Lake mill; and therefore, a conventional TMF using raised dykes to contain slurry tailings and process water is proposed.

The proposed TMF is located northeast of the process plant site and is surrounded by three dykes connecting to the natural topography, generating a 5.21 Mt (3.47 Mm^3) capacity cell. Figure 18-1 shows the general arrangement of the TMF and related water management infrastructure. The location was chosen with consideration for the favourable topography which reduces the amount of construction materials required for dyke construction and the proximity of the process plant from which the tailings will be pumped through pipelines to the impoundment, as well as to avoid known wetlands and water bodies. The proposed location of the TSF allows potential increase in the LOM and the capacity of the tailings impoundment, while keeping the same footprint.

The cell will be developed in three stages using the downstream raise method. A starter dyke (Dyke C) will be built on the northern side of the impoundment to store the slurry tailings produced from year 1 to 5. Dyke C will then be raised at year 5, and at year 15 to accommodate the tailings produced until the end of LOM (15 to 25). Dykes A and B will be built at year 15 to close the impoundment.

For the purpose of volume calculations, a slurry density of 1.50 to/m^3 was used. The tailings are expected to have a 50% (weight/weight) water ratio and particle size distribution is expected to be of 80% passing the 100-micron sieve.

The proposed design for the TMF dykes is based on the following geometry:

— Upstream and downstream slopes: 2.5H:1V
— Maximum height: 12 m
— Crest width: 10 m

Preliminary geochemistry hypotheses indicate that the tailings issued from the Crater Lake process plant may be metal leaching for magnesium and magnesium oxide. However, no geochemical analyses have been conducted on the tailings produced at the Crater Lake. Consequently, the proposed TMF is designed to be impervious until further geochemical studies and subsequent seepage modelling have been completed. The upstream slope of the dykes is lined using impervious liners (Bentomat or similar product), a protective geotextile, and a granular material protection layer of 900 mm thickness. The mass backfill of the dyke uses till or other sorted granular materials such as waste rock. The TMF impoundment foundation will also be lined, and it is expected that tailings will be deposited directly on the liners. The proposed construction sequence also includes topsoil stripping, surface preparation, dyke foundation treatment, and drainage trenches.

Geotechnical Considerations

No proper geotechnical data is available for the Crater Lake site and therefore the proposed TMF design assumes, on the basis of the surface deposit maps made available by Données Québec (Ministère de la Faune, des Forêts et des Parcs [MFFP], 2022), that overburden is primarily composed of glacial deposits such as moraines and till. No notable stability issues are foreseen; however, geotechnical investigations will be required to confirm this hypothesis and the proposed design.

For this PEA, it is assumed that dyke foundation treatment will be required as the rock foundation quality in the targeted area is unknown.

18.4.2 Sept-Îles Tailings Storage Facility

Design and General Considerations

Tailings produced at the Sept-Îles site will be filtered to 12% moisture content (88% solids). Consequently, the filtered tailings will be hauled from the process plant to the drystack TMF, spread and compacted.

For the purpose of volume calculations, the filtered tailings are assumed to have a packed density of 2.0 to/m^3 and a bulk density of 1.3 to/m^3, according to preliminary metallurgical assumptions. No sieve analyses have been done on the tailings. A volume contingency of 10% was included to account for the variability of the density obtained after tailings placement and compaction on the drystack.

It is expected that a total of 8.13 Mt (4.47 Mm^3) of filtered tailings will be produced over the LOM. The drystack TMF is subdivided in two construction phases. This allows for sequencing of the construction phases, as well as progressive reclamation. Figure 18-3 shows the general arrangement of the TMF and related water management infrastructures. Consideration was given to the topography, presence of water bodies, and wetlands in the design of the TMF. The design is such that tailings produced from year 1 to year 13 will be stored on site on the eastern side of the TMF. The TMF can then be extended to the West to store the tailings produced from years 13 to the end of the LOM (year 25). The drystack TMF has a total capacity of 4.47 Mm^3.

The proposed design for the drystack TMF is based on the following geometry:

- Maximum height: 15 m
- Bench slope: 4H:1V
- Berm width: 10 m
- Bench height: 7.5 m

A wet tailings disposition area is also included in the design to store unfiltered tailings on site in the event of a breakdown or maintenance of the process plant. An impoundment will be created from a dyke joining the drystack TMF which will allow for the gradual deposition of wet tailings on the downstream side of the impoundment. The wet deposition area is designed to store 0.16 Mt (0.11 Mm^3) which is equivalent to six months of tailings production from the Sept-Îles process plant. A slurry density of 1.5 to/m^3 was used for the purpose of this calculation.

Preliminary geochemistry hypotheses indicate that the tailings issued from the Sept-Îles process plant may be metal leaching for magnesium and magnesium oxide. However, no geochemical analyses have been conducted on the tailings produced at the Sept-Îles process plant. Additionally, the site is located in relative proximity of urban areas, and therefore the design is subject to social acceptability. Consequently, the drystack TMF, and the wet deposition area are designed to be impervious until further geochemical analyses and subsequent seepage rate modelling analyses are completed.

The wet deposition area dyke follows the same design as the dykes proposed for the Crater Lake TMF, as described in the previous Item.

The proposed drystack TMF design is lined using impervious liners (Bentomat or similar product), a protective geotextile, and a granular material protection layer of 300 mm thickness. The design also includes a drainage mat for slope stability. It is expected that the filtered tailings will be placed directly on the liner.

The proposed construction sequence for both the drystack TMF and the wet deposition area also includes topsoil stripping, and surface preparation.

Geotechnical Considerations

No proper geotechnical data was consulted for the purpose of the Sept-Îles site drystack TMF and wet deposition area designs and therefore the proposed design assumes that no notable stability issued is foreseen. A proper review of geotechnical data, and geotechnical investigations will be required to confirm the proposed design.

For this PEA, it is assumed that dyke foundation treatment for the wet deposition area will be required as the rock foundation quality in the targeted area is unknown.

18.5 Stockpiles

18.5.1 Crater Lake Site Waste Rock and Overburden Stockpiles

Design and General Considerations

Waste rock will be produced only on the Crater Lake site. Volumes to be stored have been calculated based on the mining plan described in Item 16. It is expected that 18.79 Mt (9.00 Mm^3) of waste rock will be stored on the waste rock storage facility (WRSF) at the end of the LOM. A density of 2.71 to/m^3 and a swell factor of 1.3 were used to calculate the required storage capacity of the WRSF. It is expected that a certain tonnage of waste rock will be used for construction purposes; however, these volumes were not deducted in the design of the waste stockpile. The location of the proposed WRSF is shown in Figure 18-1.

The proposed design for the WRSF is based on the following geometry:

- Maximum height: 35 m
- Bench slope: 2.5H:1V
- Berm width: 10 m
- Bench heigh: 10 m

The proposed WRSF is located west of the open pit, and south of the mill site, as shown in Figure 18-1. The location of the WRSF accounts for the possible shell expansion of the Crater Lake open pit. The proposed location also aims to reduce the hauling distances from the pit and complies with a Design for Closure approach with a low-profile stockpile to reduce rework during site closure. Site constraints, such as the presence of water bodies, and rugged topography as well as location of the process plant have been considered.

It is expected that the waste rock extracted from the Crater Lake pit is not metal leaching and not potentially acid generating. Therefore, the required construction works for the preparation of the stockpile foundation include topsoil stripping only. Waste rock will be placed directly on the natural ground.

It is assumed that a portion of the overburden produced at the beginning of the LOM will be used for construction purposes, particularly for the access road, pads, dyke construction, and remaining volumes will be used reclamation purposes, and therefore, it is considered that a temporary overburden stockpile may be required on site. The proposed overburden stockpile has a capacity of 1.80 Mm^3 and is located southwest of the process plant. Soil stripping was considered for the overburden stockpile site preparation.

Geotechnical Considerations

The proposed WRSF design is based on limited geotechnical information. It is expected that the bedrock is near the surface and is overlain by a variable thickness of till, on the basis of the surface deposit maps made available by Données Québec (MFFP, 2022). Site geotechnical conditions will be confirmed with field investigations in subsequent engineering studies.

18.5.2 Sept-Îles Site Waste Rock and Overburden Stockpiles

As mentioned previously, no waste rock is generated from the activities on the Sept-Îles site; consequently, no WRSF is required. As for overburden, it is assumed that the excavated volumes will be negligible and will be mostly used as construction materials for surface infrastructure such as roads, ponds, and ditches as well as reclamation purposes. A temporary overburden stockpile may be required; however, no design was provided for this PEA.

18.5.3 Organic Topsoil Stockpiles

For the purpose of this PEA, it is assumed that topsoil (organic material) excavation on the Crater Lake site, located in northern Québec, will be minimal, and therefore no stockpile was designed. For the Sept-Îles site, no permanent stockpile was designed for the storage of topsoil. However, for both sites, it is assumed that topsoil will be stockpiled and reused for progressive reclamation, where possible.

18.6 Water Management

18.6.1 Contact Water Management

18.6.1.1 Crater Lake Site

Water management infrastructure on the site was designed to collect and prevent any discharge of contact water to the environment. The water management on the Crater Lake site includes catchment ponds for the TSF embankments and the WRSF runoff water, a collection pond and a sedimentation pond. The ponds were designed to withhold a 100-year rainfall event while considering a complete discharge of the pond in three days. Climate change effects in the rainfall and snow accumulation has been considered. The recirculation of process water at a rate of 12 m^3/h was also considered.

A system of peripheral ditches is proposed to collect runoff waters which is then transferred to the collection pond and into the Water Treatment Plant (WTP) via system of submersible pumps and pipes. The location of the proposed ponds is shown in Figure 18-1. Small capacity pumping ponds are also proposed on the downstream embankments of dykes A and B to redirect runoffs to the TMF pond, located north of dyke C.

The proposed pond and ditch design uses a combination of geosynthetic materials (geotextiles and geomembranes) and granular protection material to ensure impermeability of the infrastructure.

18.6.1.2 Sept-Îles Site

Water Management infrastructures on the Sept-Îles site use a similar design basis as on the Crater Lake site. A system of peripheral ditches, catchment ponds and pumps ensure that all contact water is collected and treated in the WTP. The proposed water management infrastructure is also lined to ensure that no contact water is discharged into the natural environment.

18.6.2 Water Treatment Plant (Final Effluent)

18.6.2.1 Crater Lake Site

The WTP on the Crater Lake site was designed according to the catchment area, pond volumes and process water inflow. The maximum estimated WTP flow rate is 150 m^3/h. The WTP is located southeast of the plant site. The WTP size used for cost calculation was determined on the basis of the available information (such as process water reuse rate) and preliminary water management concept and design (such as pond and waste infrastructure, and pond size), and will need to be re-evaluated in subsequent engineering phases.

For the purpose of this PEA, it is assumed that possible contaminants include suspended solids, . The design of the WTP is conceptual and will need to be confirmed in subsequent engineering studies. The final effluent location will need to be determined in further studies.

18.6.2.2 Sept-Îles Site

The WTP on the Sept-Îles site was designed according to the runoff areas, pond volumes and process water inflow where applicable.. The maximum estimated WTP flow rate is 185 m^3/h. The WTP location is shown in Figure 18-3.

For the purpose of this PEA, it is assumed that possible contaminants include suspended solids, and dissolved minerals. The design of the WTP will need to be confirmed in subsequent engineering studies. The final effluent locations will need to be determined in further studies.

18.7 Material Handling Logistics

18.7.1 Introduction

The purpose of this Item is to compare the various concentrate transport scenarios from the Crater Lake mine to the Hydrometallurgical facility.

The processing plant at Crater Lake is expected to produce 218,540 tonnes of concentrate per year, which will be sent to the hydrometallurgy plant in Sept-Îles to produce a final Al-2%Sc Master Alloy. The concentrate will be shipped, no moisture.

There is no access to the property via land or water for the material transportation at this current time.

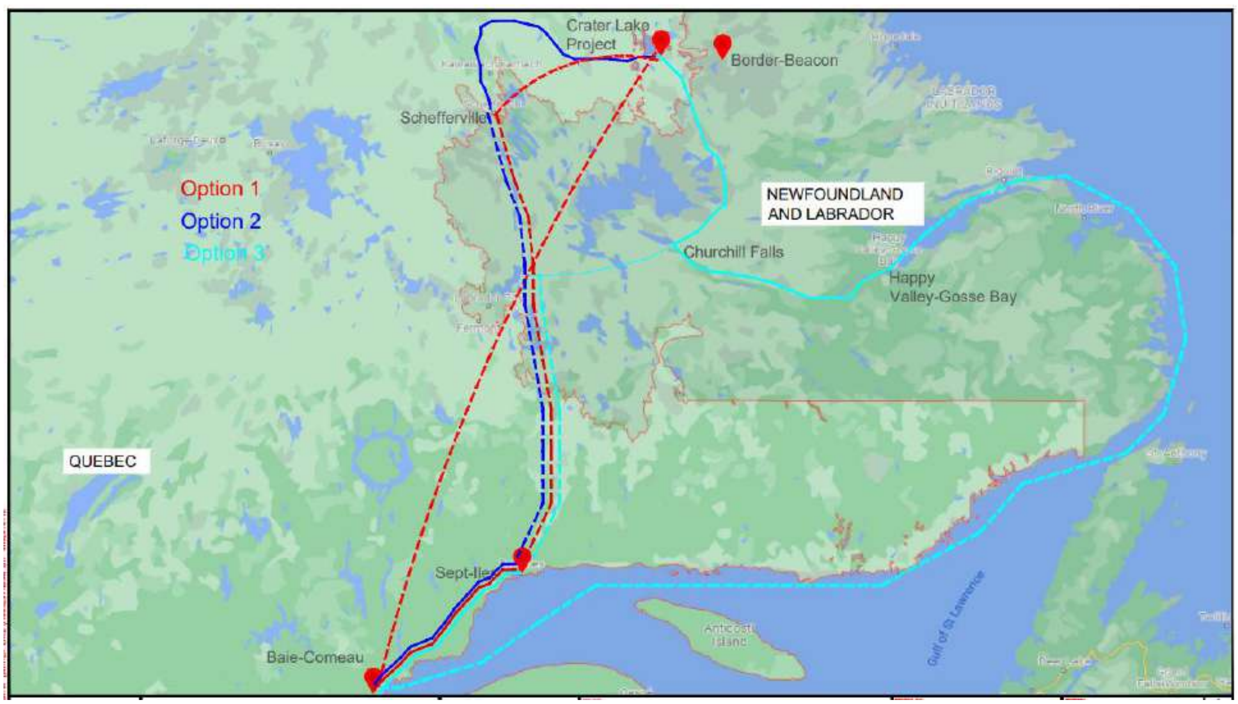
WSP conducted three studies for multiple options applied, which is to determine the different impacts on CAPEX OPEX, total cost NPV, IRR, cycle time, and other factors.

The first study was completed in December 2021, the second in February 2022, and the third in March 2022.

18.7.2 Air versus Rail versus Maritime to Baie-Comeau

This early study looked at three transportation options (airline, railway, and maritime transport) for getting concentrate to Baie-Comeau, as the initial location for the Hydrometallurgical Plant was assumed to be in Baie-Comeau. The alternative possibilities that exist for each option are presented in Figure 18-5.

Figure 18-5: Air Versus Rail Versus Maritime to Baie-Comeau



As shown in Table 18-1, Option 1- Airline option: Uses air transport from mine site airstrip to Baie-Comeau or to Schefferville.

— Scenario 1: Only air transport.
— Scenario 2: Air transport to Schefferville then railway to Sept-Îles, and finally truck to Baie-Comeau.

Option 2 - Railway option: assumes building a winter road that connects the mine site to Schefferville, then use the rail infrastructure to Sept-Îles and haul the material by truck to Baie-Comeau.

— Scenario 1: Winter road to Schefferville, then use train to Sept-Îles and haul the material by truck to Baie-Comeau
— Scenario 2: Hybrid between winter road (wetlands) and permanent road to Schefferville, then use the train to Sept-Îles and haul the material by truck to Baie-Comeau.

Option 3 - Maritime transport: assumes building a new road from the mine to Orma Lake Road, rehabilitation of the existing Orma Lake road to reach Churchill Falls, hauling the concentrate material by truck to Happy Valley Goose Bay and use the maritime transport from to Baie-Comeau. Another potential possibility is to haul the concentrate material by truck from Churchill Falls to Emeril, then use the rail infrastructure to Sept-Îles and haul material by truck to Baie-Comeau.

— Scenario 1: Build a permanent road from Crater Lake to end of Orma Lake road, rehabilitate the existing Orma Lake road to reach Churchill Falls, haul the concentrate material by truck to Happy Valley Goose Bay and use the maritime transport to Baie-Comeau.
— Scenario 2: Build a winter road from Crater Lake to end of Orma Lake road, rehabilitate the existing Orma Lake road to reach Churchill Falls, haul the concentrate material by truck to Happy Valley Goose Bay and use the maritime transport to Baie-Comeau.
— Scenario 3: Build a permanent road from Crater Lake to end of Orma Lake road, rehabilitate the existing Orma Lake road to reach Churchill Falls, move to the concentrate material to Emeril by using Hwy 500, then use the rail infrastructure to Sept-Îles and haul the material by truck to Baie-Comeau.

Four aspects were considered in this high-level study to include operating, costing, environment, and permits.

Table 18-1: Material Handling Logistical Options

	Option 1-Airline		Option 2-Railway		Option 3-Maritime time		
	Scenario 1	Scenario 2	Scenario 1	Scenario 2	Scenario 1	Scenario 2	Scenario 3
NPV M$ CAD	-1521	-359	4431	4294	4546	**4676**	4438
CAPEX $ CAD/t	176	184	44	909	1012	152	160
OPEX $ CAD/t	4098	3347	270	270	98	101	254

The results were ranked, and the ranking resulted in scenario 2 of option 3 maritime transport being the less expensive option and this option was recommended.

18.7.3 Maritime to Baie-Comeau versus Rail to Sept-Îles

A second study was done based on the findings of the first. The CAPEX and OPEX of the railway to Sept-Îles and maritime to Baie-Comeau options were compared in this study.

— Option 1: Build a winter road or a permanent road from Crater Lake to the end of Orma Lake road, rehabilitate the existing Orma Lake road to reach Churchill Falls, move the concentrate material to Emeril by using Hwy 500, then use the rail infrastructure to Sept-Îles .

— Option 2: Build a winter road or permanent road from Crater Lake to the end of Orma Lake road, rehabilitate the existing Orma Lake road to reach Churchill Falls, haul the concentrate by truck to Happy Valley Goose Bay and use the maritime transport to Baie-Comeau.

Figure 18-6 and Figure 18-7, respectively, depict option 1 railway and option 2 maritime.

Figure 18-6: Option 1 Railway

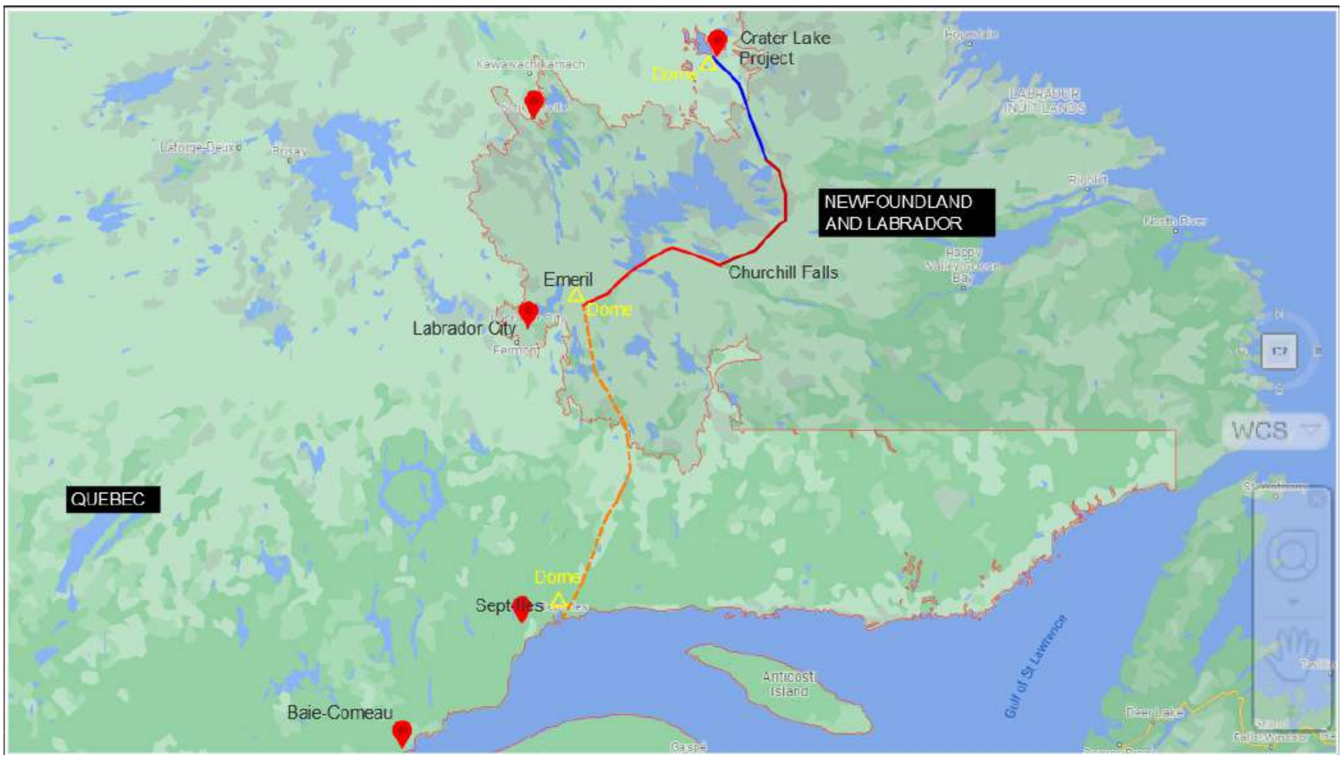
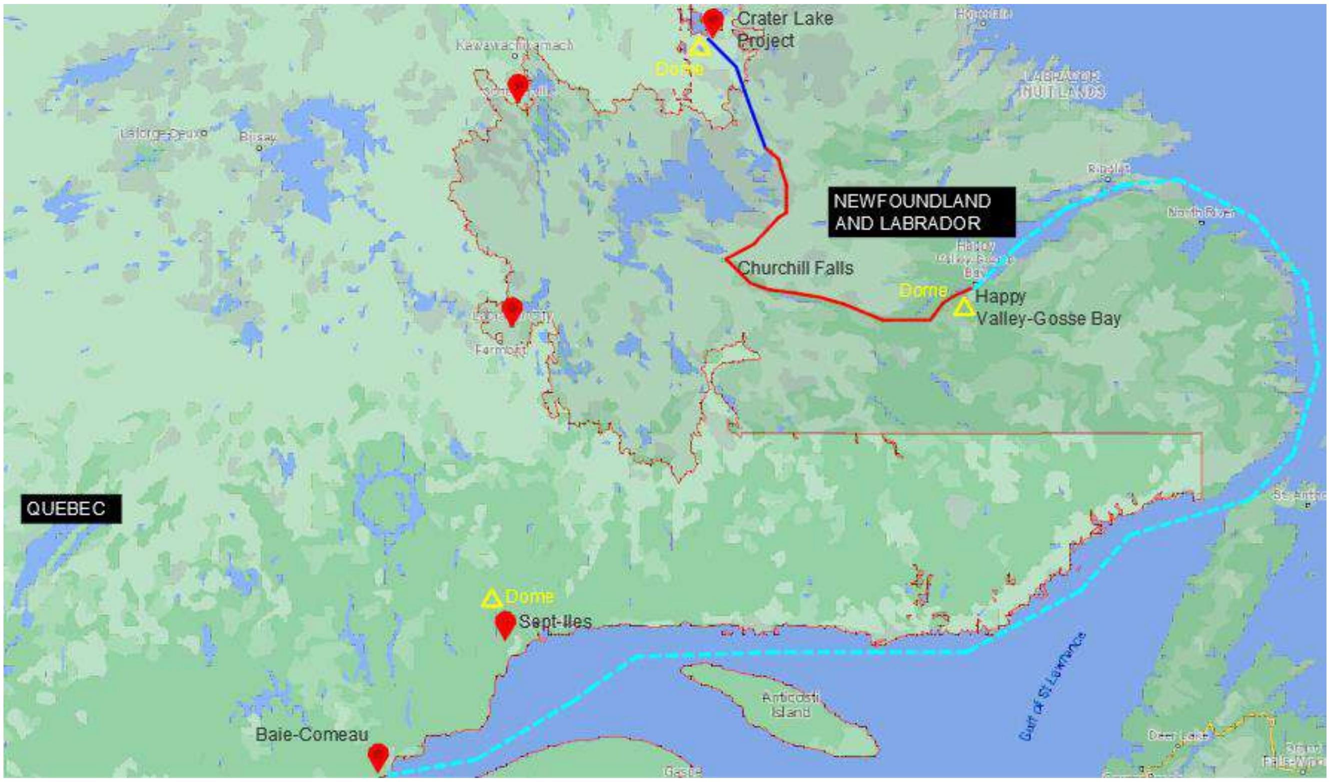


Figure 18-7: Option 2 Maritime



Because the CAPEX of a permanent road is substantially more than that of a winter road, IPG requested that the study be continued with the winter road.

A CAPEX and OPEX comparison is shown in Table 18-2.

Table 18-2: CAPEX/OPEX Comparison

Item	CAPEX	OPEX	Unit
Railway	187	317	$ CAD/t
Boat (using containers)	984	298	$ CAD/t
Boat (using bulk)	246	207	$ CAD/t

The result showed that the lowest total cost is the railway option, it was also found that the harbour only operates six months per year. A decision was made to re-investigate the railway option to look for further opportunities to reduce costs.

18.7.4 Rail to Sept-Îles

WSP continued to review multiple railway options to transport the concentrate from the mine site to Sept-Iles, as Sept-Iles may be a better option as a location for the Hydrometallurgical Plant. The information was ranked without any weightings and the risk was indicated by number of the trucks travailing the route at any given time.

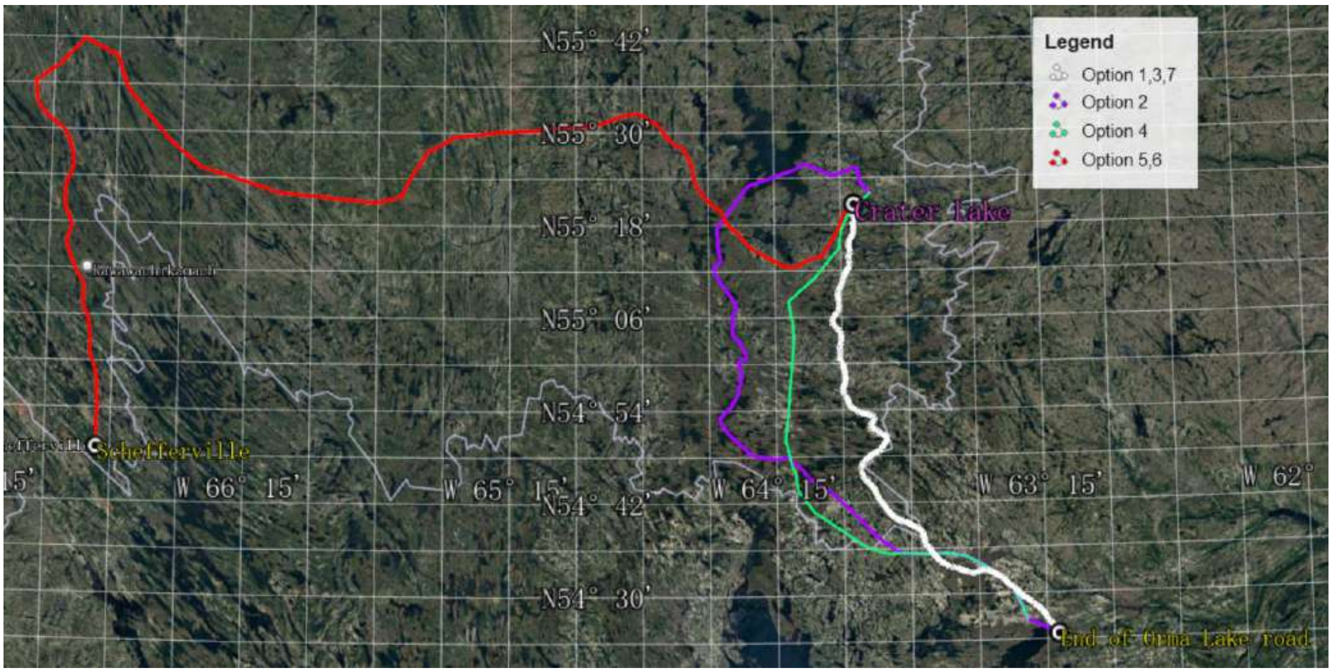
The common part for all the options is from the rail (Emeril or Schefferville) to Sept-Îles. To reach the rail station from the mine, WSP provides 7 roads to railroad options to be considered.

— Option1-Winter Road over land road to Emeril, NL (146km)

- Option2-Winter Road over water to Emeril, NL (199km)
- Option3- Permanent Road over land to Emeril, NL (146km)
- Option4- Permanent Road over land to Emeril, NL (152km, different route than Option 3)
- Option5- Permanent Road over land to Schefferville, QC (325km)
- Option6-All season Road over land to Schefferville, QC (325km)
- Option7-All season Road over land Emeril, QC (146km)

Figure 18-8 shows the 7 options for materials handling logistics.

Figure 18-8: Seven Options for Material Handling Logistics



The challenge of this study is to analyze the costs, the construction periods, the utilization of the total period per year, and the speed limit of the different types of the roads:

- Winter road, a seasonal road, only usable during the winter, they must be re-built every year, the speed limit is between 40-60km/hr.
- All season road is operable all year round, due to the unfinished surface or sufficient width, the expected speed is about 50km/hr.
- Permanent road is an engineered roadbed with engineered curvatures and inclines, with expected speed of about 80km/hr.

Table 18-3 summarizes the results of the cost evaluations.

Table 18-3: Cost Evaluation Results

Option	Descriptions	Trucks req'd	Total Capital	Total Operating	Total Cost	Cost/t Concentrate Capital	Cost/t Concentrate Operating	Cost/t Concentrate Total	NPVx10^6 (M$)	IRR
1	Winter road over land to Emeril	42	$90,535,480	$1,362,742,196	$1,453,277,676	$6.68	$251.13	$267.81	$43,820	499%
2	Winter road over water to Emeril	44	$74,875,407	$1,426,734,740	$1,501,610,147	$13.80	$262.92	$276.72	$43,712	600%
3	Permanent road over land to Emeril	10	$765,575,531	$692,099,916	$1,457,675,446	$141.08	$127.54	$268.62	$39,574	63%
4	Permanent road over land to Emeril		not costed, longer distance than option 3 road, make more expensive							
5	Permanent road over land to Schefferville	6	$727,268,220	$541,152,451	$1,268,420,671	$134.02	$99.72	$233.75	$40,517	67%
6	All season road over land to Schefferville	10	$472,732,768	$692,790,719	$1,165,523,487	$87.12	$127.67	$214.78	$42,930	101%
7	All season road over land to Emeril	15	$451,091,232	$1,233,072,676	$1,684,163,908	$83.13	$227.23	$310.36	$40,994	101%

Table 18-4 shows the ranking results.

Table 18-4: Ranking Seven Options Results

Option	Descriptions	Rankings: Lowest Total Cost	Lowest Operating Cost	Lowest Capital Cost	Numbers of Trucks	NPV	IRR	Totals
1	Winter road over land to Emeril	3	4	2	4	1	2	16
2	Winter road over water to Emeril	4	5	1	4	2	1	17
3	Permanent road over land to Emeril	3	2	5	2	7	6	25
4	Permanent road over land to Emeril							
5	Permanent road over land to Schefferville	2	1	4	1	6	5	19
6	All season road over land to Schefferville	1	2	3	2	3	4	15
7	All season road over land to Emeril	6	3	3	3	5	4	24

Given the above ranking, the following conclusions were made:

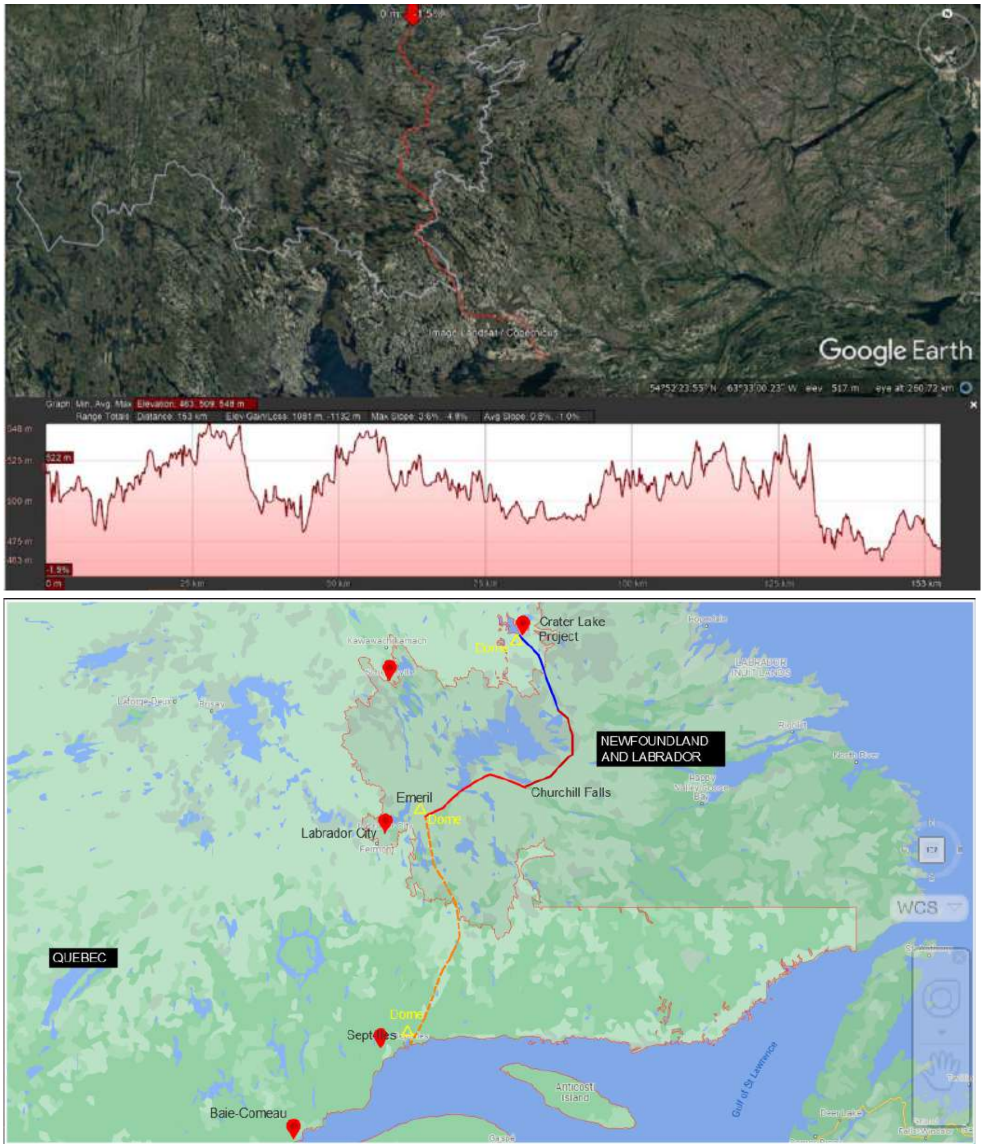
— The rankings showed the all-season road to Schefferville had the best result.

— The winter road over land to Emeril had the second-best result.

18.7.5 Route Selection to Sept-Îles

At the conclusion of the Sept-Îles study, client provided the following route to Sept-Îles, as shown in Figure 18-9.

Figure 18-9: Route to Sept-Îles



The client used 50k scale Topographical maps and Satellite images (ESRI Database) to establish the route for the winter road. Waterways and their widths (Table 18-5) were provided for this route selection. The total length of the route is 152 km.

Table 18-5: Waterways and Widths

Width	Value
More than 30 m	3
20 to 30 m	3
10 to 20 m	5
5 to 10 m	7
3 to 5 m	5

The route provided by the client to reach Sept-Îles involves:

- Winter road overland 152km to Orma lake road
- Refurbish the existing Orma lake road to Churchill Falls
- Travel by Hwy 500 to Emeril
- Establish a railcar load out area at Emeril
- Travel by rail to Sept-Îles
- Establish a railcar unloading area at the hydrometallurgical facility at Sept-Îles

This route is the basis of the PEA.

19 MARKET STUDIES AND CONTRACTS

19.1 Scandium Market

All information pertaining to current and future scandium and master alloy markets has been compiled from publicly available information, publications issued by expert consultants, as well as an internal outlook for the supply of scandium and master alloy products. This Item contains forward-looking information related to master alloy, scandium and rare earth demand and prices for the Project. The factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: Prevailing economic conditions, master alloy, scandium, and rare earth demand and prices are as forecast over the Study period.

19.1.1 Introduction

A market analysis was prepared in February 2022 for Imperial Mining Group by Ernst & Young (EY), an independent consultant. EY specializes in market assessments.

The following is an introductory statement reproduced from the report, "Ernst & Young LLP ("EY") was engaged by Imperial Mining Group Ltd. ("IPG") to conduct a scandium market analysis and assess the potential market size of its use. In preparing this document ("Report"), EY relied upon unaudited data and information from IPG and publicly available data and information (collectively, the "Supporting Information"). EY reserves the right to revise any analyses, observations or comments referred to in this Report, if additional Supporting Information becomes available to us subsequent to the release of this Report. EY has assumed the Supporting Information to be accurate, complete, and appropriate for the purposes of the Report. EY did not audit or independently verify the accuracy or completeness of the Supporting Information. Accordingly, EY expresses no opinion or other forms of assurance in respect of the Supporting Information and does not accept any responsibility for errors or omissions, or any loss or damage as a result of any persons relying on this Report for any purpose other than that for which it has been prepared."

19.1.2 Scandium Utilization History

Both the USGS commodity report and the EY analysis make similar statements in worldwide resources pertaining to scandium, "Despite scandium's relatively high crustal abundance, it is widely dispersed and rarely concentrated in economically exploitable deposits. Owing to its dispersed nature and resulting low concentration, scandium is currently produced exclusively as a by-product during the processing of various ores and has also been recovered from mill tailings."

As stated in the USGS commodity summaries for 2021, "No scandium was recovered from mining operations in the United States. As a result of its low concentration, scandium is produced exclusively as a by-product during processing of various ores or recovered from previously processed tailings or residues. Historically, scandium was produced as by-product material in China (iron ore, rare earths, titanium, and zirconium), Kazakhstan (uranium), the Philippines (nickel), Russia (apatite and uranium), and Ukraine (uranium). Foreign mine production data for 2020 and 2021 were not available. The global supply and consumption of scandium oxide was estimated to be about 15 to 25 tons per year."

19.1.3 Scandium Market Analysis

The EY market analysis then discusses foreseeable trends.

"Certain scandium properties make it an ideal alloying element in aluminium. Studies have shown that the addition of scandium considerably influences the structure and properties of aluminum and its alloys, making the end products lighter, stronger and more resistant to heat and corrosion.

Because of the low market availability and high cost, scandium has received relatively little attention in the manufacturing community."

EY's market analysis found:

"The potential global scandium market size was estimated for the aerospace and defence (A&D), automotive, solid oxide fuel cells (SOFCs), and offshore wind turbines markets. Under most likely scenario:

- Global demand is expected to reach 463 tonnes of scandium oxide in 2030.
- Global demand is expected to continue rising with the introduction of new technologies and manufacturing techniques and reach 1,970 tonnes in 2040.
- The automotive industry is anticipated to account for approximately two-thirds of the scandium demand starting in 2030.
- The Aerospace and Defense sector may account for up to 15% of the demand – scandium adoption is anticipated to grow in defence, space, and commercial aviation markets.
- The offshore wind turbine market may be in the early stages of development, but scandium adoption may increase in the next decade and account for over 15% of the total market in 2040-2045."

Of note, the production plan for Crater Lake shows a maximum metals production potential, should all products be made into Scandium Oxide, of 88.5 tonnes per year, which would be well short of the market potential described by EY in their analysis.

Aerospace and Defense

For A&D, EY's market analysis states:

"The main groups of materials used in aerospace and defense manufacturing are aluminum alloys, titanium alloys, steels, and composites. Scandium may enhance functional properties of aluminum alloys used in the sector include contributing to increased material strength, weight/density ratio, corrosion resistance, heat capacity, and fatigue life.

Light-weighting properties of scandium-aluminum alloys are expected to drive the demand in the aerospace and defense markets. The defense and space sectors may be among the early adopters of scandium aluminum alloys, and commercial aviation sector is expected to have longer lead times due to regulatory standards. In longer term, new emerging scandium applications may include new technologies such as exoskeletons and electric vertical take-off and landing aircraft (EVTOL).

In the most likely scenario, the demand for scandium is anticipated to total ~144 tonnes by 2035 and ~411 tonnes by 2045. The commercial aviation segment is expected to make up close to 50% of the total market.

The aerospace and defense markets are expected to have low price sensitivity as scandium additions can offer cost savings and improved performance. It is estimated that the maximum scandium oxide price that the aerospace and defense manufacturers will be willing to pay is over US$ 3,000.

In commercial aviation, scandium applications may result in up to US$ 2 million operating costs savings over a 10- year operating life of an aircraft.

In space, scandium adoption in light satellite structures may offer launch cost savings of up to US$ 110,000.

Scandium application in launch vehicles may result in potential net new revenue of up to US$ 400,000 from the ability to launch 500kg additional payload."

Automotive

For Automotive, EY's market analysis states:

"Offsetting the substantial mass coming from battery systems is key to a successful electrification strategy in the automotive industry and has stimulated demand for light-weighting solutions. Scandium has shown potential to enhance functional properties of aluminum alloys, including improving tensile strength, elongation, and weld integrity, making it suitable for various uses in the automotive industry.

Scandium additions considerably improve the functional properties of aluminum alloys. This is expected to enable the use of aluminum in many EV components, including (1) structural components; (2) battery cases; (3) A/C heat exchangers & evaporators; (4) heat exchangers for motor housing; (5) battery thermal management systems; and (6) magnet wiring in EV motors, for the benefits of cost savings and light weighting.

In the most likely scenario, the demand for scandium oxide is anticipated to total ~2,138 tonnes by 2045. Battery casing and body & chassis are expected to make up nearly three- quarters of the total market.

Downstream price sensitivity is estimated in Table 19-1 and is based on the cost savings and performance improvements that OEMs could potentially achieve by switching to Sc-Al alloys."

Table 19-1: Ernst & Young Market Analysis, Price Prediction Scandium Oxide in Automotive Industry

Application	Market Segment	Sc loading*	Maximum Sc price/kg
Body and chassis	Premium EV	0.3%	US$ 2,900
	Economy EV	0.3%	US$ 1,670
Battery cases	Commercial EV	0.3%	US$ 1,150
	Premium EV	0.3%	US$ 2,610
	Economy EV	0.3%	US$ 1,150
Magnet wire	Premium EV	0.15%	Over US$ 3,000
	Economy EV	0.15%	Over US$ 3,000
Heat exchangers for motor housing	Passenger EV	0.2%	Over US$ 3,000
	Commercial EV	0.2%	Over US$ 3,000
Battery thermal management systems	Commercial EV	0.2%	Over US$ 3,000
	Premium EV	0.2%	Over US$ 3,000
	Economy EV	0.2%	Over US$ 3,000
R744 MAC systems	Passenger vehicles	0.2%	Over US$ 3,000
	Commercial vehicles	0.2%	Over US$ 3,000

Solid Oxide Fuel Cell

For Solid Oxide Fuel Cells (SOFC), EY's market analysis states:

"Scandium oxide provides better ionic electrical conductivity in SOFC compared to yttrium – the stabilizing agent currently being used. The uses of scandium also lower the temperature required for fuel cells to be operational, which in turn raise the power density of the fuel cell units and considerably reduces the thermal stresses in the cell components and thereby improving fuel cell's lifespan.

Research indicates the use of scandium oxide in fuel cell electrolytes considerably improves SOFC's operating economies. Key drivers, including growing demand for efficient and reliable auxiliary power supplies in data centres and policies improvement of energy system efficiencies and reduction of carbon expected to fuel future demand for SOFCs. In the most likely scenario, the demand for scandium oxide is anticipated to total ~96 tonnes by 2045.

Maximum Sc price per kilogram over US$ 3,000. SOFC manufacturers' scandium oxide price sensitivity is evaluated based on the anticipated cost savings relative to grid power and the value of carbon emission reduction over a SOFC's operating life. Specifically, operating a 1MW SOFC unit with an assumed operating life of ten years may save the owner upwards of \$400,000. In addition, research has shown SOFCs emit significantly lower GHG emissions relative to traditional power sources. For example, on a per MW basis, SOFC produces only 7% of CO_2 relative to that of a traditional gas combined cycle power plant."

Offshore Wind Turbines

For Offshore Wind Turbines, EY's market analysis states:

"Scandium alloy's high strength and corrosion resistance makes it an ideal material for use in the offshore environment. Leading edge protection made of scandium alloy is expected to be more durable and easier to repair using 3D printing technology. The use of scandium alloy may significantly reduce the weight of nacelles, which is expected to lower installation cost.

Scandium alloy's superior strength and corrosion resistance make it an ideal material for offshore wind turbines whose operating environment is often harsh. Scandium's ability to improve the durability of turbine blades and lower the total weight of the nacelle housing may extend the turbine's operating life and lower cost related to installation and the foundation. These benefits are expected to be the primary driver of scandium in the market.

In the most likely scenario, the demand for scandium oxide is anticipated to total ~655 tonnes by 2045. Nacelle housing is expected to make up nearly two-thirds of the market.

Maximum Sc price per kilogram is over US\$3,000. Offshore wind turbine manufacturers' scandium oxide price sensitivity is evaluated based on the anticipated cost savings from lower repair frequency, additional revenues from higher operating efficiency and lower initial capital cost.

Lower repair frequency translates into savings in costs related to the deployment of maintenance ships. More durable leading edges mean the offshore wind turbines can operate at the optimal efficiency for a longer period of time (compared to unprotected leading edges, which will degrade over time, lowering efficiency).

EY's estimates suggest that the more durable leading-edge protection provided by scandium alloys may translate into upwards of \$1.1 M in additional revenues for a 15-MW offshore turbine in a 15-year operating life. In addition, lighter nacelle housing is expected to reduce the cost related to initial installation and the construction of substructure and foundation, which may translate into savings of \$0.5 M per installation.

19.2 Rare Earths

19.2.1 Marketing Plan

Imperial Mining Group plans on marketing a select grouping of rare earth compounds. Of the many rare earths in the mineral mix being mined, the metals recovery plan for the Crater Lake Mill is to ship a mineral concentrate containing scandium and all the rare earth elements to the hydrometallurgical facility for processing to 99.5% purity scandium oxide and rare earth mixed hydroxide. Of all the rare earth elements contained in the mixed hydroxide, only those used for manufacturing of magnets are accrued values for the project:

- Dysprosium Oxide (Dy_2O_3)
- Lanthanum Oxide (La_2O_3)
- Neodymium Oxide (Nd_2O_3)
- Praseodymium Oxide (Pr_2O_3)
- Terbium Oxide (Tb_4O_7)

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19.3 Market Analysis / Metal Pricing Usage

Imperial Mining Group (IPG) provided the prices shown in Table 19-2 for assessment of the economic viability of the Crater Lake Project.

Table 19-2: Imperial Mining Group, Selling Price of Products Produced

Saleable Product	Units	Unit Price Provided
Scandium Oxide (Sc_2O_3)	US$/kg	1,500.00
Master Alloy (Al –Sc 2%)	US$/kg	204.00
Dysprosium Oxide (Dy_2O_3)	US$/kg	128.40
Lanthanum Oxide (La_2O_3)	US$/kg	1.50
Neodymium Oxide (Nd_2O_3)	US$/kg	49.20
Praseodymium Oxide (Pr_2O_3)	US$/kg	49.20
Terbium Oxide (Tb_4O_7)	US$/kg	584.40

WSP compared the pricing structure provided by IPG to those observed by EY in their market analysis and with the USGS spot prices over the last 5 years. The USGS spot prices are shown in Table 19-3 and the average spot price for the last 5 years has been included for the comparison to the IPG provided prices.

Table 19-3: Scandium and Rare Earths Year End Spot Prices, Preceding Five Years

Salient Statistics—United States:	2017	2018	2019	2020	2021e	Spot Price Average
Scandium Compounds, dollars per gram:						
Oxide, 99.99% purity, 5-kilogram lot size	4.6	4.6	3.9	3.8	2.2	3.82
Scandium Metals:						
Scandium-aluminum alloy, 1-kilogram lot size, dollars per kilogram	350	360	300	340	350	340.00
Other Rare Earths:						
Dysprosium Oxide (Dy_2O_3) 99.5% minimum, dollars per kilogram	187	179	239	261	400	253.20
Lanthanum Oxide (La_2O_3) 99.5% minimum, dollars per kilogram	2	2	2	2	2	2.00
Neodymium Oxide (Nd_2O_3) 99.5% minimum, dollars per kilogram	50	50	45	49	49	48.60
Praseodymium Oxide (Pr_2O_3), dollars per kilogram*		63.49	54.02	45.76	58.40	55.42
Terbium Oxide (Tb_4O_7) 99.5% minimum, dollars per kilogram	501	455	507	670	1300	686.60

Note: *Statistica.com for Praseodymium Oxide
Source: USGS Mineral Commodity Summaries, except

Based on the above comparison, WSP deems that the market analysis and prices provided by Imperial Mining Group are generally conservative and thereby acceptable to form the basis of an economic analysis for establishing whether a viable mining operation for Scandium Oxide and a mix of Rare Earth products can be established in Northern Quebec for the purposes of providing feed to a newly constructed Hydrometallurgical facility in Southern Quebec. Sensitivity analysis presented as part of the economic analysis shows project viability is retained when an even more conservative pricing structure is realized.

The market analysis and pricing information as presented does contain forward-looking information related to market demand and prices for Scandium and Rare Earths. The factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts, or projections in the forward-looking information include any significant differences from one or more of the following:

- Prevailing economic conditions
- Demand for Scandium and Rare Earth Products
- Prices as forecast over the Study period

19.4 Contracts

There are no contracts currently in place. Looking forward, IPG has approached industries identified by the EY market analysis with the intent of entering into discussions for the purposes of setting up contracts and other offtake agreements.

20 ENVIRONMENTAL STUDIES PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This Item summarizes the existing environmental and social conditions within the Project area based on data available at this stage of the Project. It also provides the environmental requirements for ore, waste rock and tailings disposal, site monitoring and water management. The regulatory context applicable to the project, including the environmental impact assessment (EIA) process and permitting requirements, is then overviewed, as well as the social and community requirements. Finally, it outlines the mine closure requirements and costs.

20.1 Environmental Studies

The environmental information provided in this chapter were collected from public databases, considering that no ecological studies have been conducted on the mining and the plant sites so far.

20.1.1 Baseline Conditions

The following Items summarize the project's current biophysical environmental conditions.

20.1.2 Hydrological Conditions

Mining Site

The Crater Lake mining site is located within the George River Primary Watershed, in the Ungava Bay Watershed Region. Based on the information available from the project site footprint, almost a quarter of the study site is covered by water bodies.

No data is currently available concerning water quality.

Plant Site

The Sept-Îles site is located within the Sept-Îles watershed, in the estuarine and marine portion of the Saint-Lawrence River Watershed. Based on the information available from the project site footprint, the site has low aquatic surface area, which are essentially associated with intermittent water courses.

No data is currently available concerning water quality.

20.1.3 Soil Quality Assessment

Mining Site

According to the information obtained from the northern ecoforestry mapping (MFFP, 2020a), the mining site is mainly composed of surface deposits of glacial origin, equivalent to approximately 85% of the soil surface. This type of deposit includes undifferentiated till followed by Rogen's moraines distributed over the entire site as well as Drumlins and drumlinoides in the southeast.

Next comes fluvioglacial deposits (about 8%) with the presence of eskers in the center and south of the site, bedrock (5%) in a few places to the north and an organic deposit (1%) at the western end.

No data is currently available concerning soil quality.

Plant Site

According to the information obtained from the Southern Quebec ecoforestry mapping (MFFP, 2020b), the Sept-Îles plant infrastructure footprint is composed at approximately 40% of marine deposit consisting of generally well sorted sand that could contain certain amount of gravel. A marine coastal deposit composed of well sorted and layered sand, gravel and pebbles (30%) and a thin to thick organic layer (30%) is also present within the infrastructure footprint.

No data is currently available concerning soil quality.

20.1.4 Vegetation and Wetlands

Mining site

The data of the Northern Eco-Forestry Inventory Program (MFFP, 2013) and the MELCC potential wetlands (MELCC, 2020a) were used to describe the vegetation and wetlands of the mining site.

Of the total mining property, around 70% is occupied by terrestrial environments while around 5% is made up of wetlands. Water bodies represent around 25% of the site. The terrestrial vegetation is represented by softwood stands dominated by black spruce, with the presence of lichens, deciduous shrubs, mosses and tamarack trees. It covers a very low density of the territory (often less than 25% cover according to the MERN ecodendrometric inventory standard) and is distributed over a little less than half of the site. Wetlands are dominated by open bogs and shrub swamps, followed by a few undifferentiated wetlands. At this stage of the project, the ecological value of the natural environments has not been evaluated.

Plant site

The data of the Southern Quebec ecoforestry mapping (MFFP, 2020b) and the MELCC potential wetlands (MELCC, 2020a) were used to describe the vegetation and wetlands of the plant infrastructure area.

Arround 40% of the Sept-Îles infrastructure is occupied by terrestrial environments while around 60% is made up of wetlands. The terrestrial vegetation is represented by softwood stands dominated by black spruce, with the presence of moss, heath, or sphagnum. A fir stand is also present in the norhtestern portion of the infrastructure footprint, which represent less then 5% of the area. Wetlands are dominated by swamps and open bogs followed by a few undifferentiated wetlands. At this stage of the project, the ecological value of the natural environments has not been evaluated.

Wetland Off-setting Plan

The preliminary assessment using geomatics data estimates that more than 250 ha of wetlands could be impacted by both the mining and the plant sites. Under the Quebec Regulation respecting compensation for adverse effects on wetlands and bodies of water, Imperial Mining Group Ltd. will have to compensate for the loss of wetlands. In accordance with the second paragraph of Section 46.0.5 of the Quebec Environmental Quality Act (EQA), the replacement of all or part of the payment of the financial contribution can be replaced by work carried out to restore or create wetlands or bodies of water in the case of the work for mining mineral substances within the meaning of Section 1 of the Quebec Mining Act. A compensation program will have to be submitted by IPG to the MELCC for approval.

20.1.5 Fish and Fish Habitat

Based on available data, portions of the project infrastructure are potentially located in fish habitat. Those potential fish habitat existence will need to be confirmed on the field if the project proceeds.

In case where infrastructure are confirmed to be located in fish habitat, the project could result in the harmful alteration, disruption or destruction of fish habitat as defined by the Federal Fisheries Act (e.g., installation of water crossings or intake and discharge structures). An application for a Section 35 (2) (b) Fisheries Act authorization will need to be provided to Fisheries and Oceans Canada (DFO). If required, the company will have to work with DFO on the details of the compensation requirements.

Moreover, in case where natural water body frequented by fish are used for tailings or waste rocks storage, this will require an amendment to the Metal and Diamond Mining Effluent Regulations (MDMER), which will consist in adding the affected water bodies to Schedule 2 of this Regulation. An authorization under Section 36 of the Fisheries Act will also be required.

IPG will have to produce an assessment of alternatives for mine waste disposal in accordance with federal guidelines and produce a fish habitat offsetting plan for authority approval as part of the EIA.

20.1.6 Species at Risk

Mining Site

There are no protected areas directly on the Crater Lake site. According to the data on wildlife and plant species in a precarious situation processed by the "Centre du patrimoine naturelle du Québec" (CDNPQ) (MELCC, 2022), there are no fauna or flora species at risk within an 8 km radius of the Crater Lake mining site.

It should be noted, however, that a species of interest likely to frequent the study area of the Crater Lake project is the Migratory caribou of the George River population. This caribou population represents one of the species of particular interest, particularly for the Indigenous community, and also because it is emblematic of the maintenance of regional biodiversity in northern Quebec.

At its highest population level in 1993, the George River Migratory caribou numbered some 800,000 individuals. It fell to an estimated 5,500 individuals in 2018, a decline of around 99%. Although the situation seems to have improved in recent years, the status of this population remains at a very critical level and the situation is being closely monitored by provincial and federal governments who are promoting conservation measures with regard to this caribou population.

Plant Site

There are no protected areas directly on the Sept-Îles plant site. However, according to the data on wildlife and plant species in a precarious situation processed by the CDNPQ (MELCC, 2022), there are two flora species (michigan moonwort and pale moonwort) and two fauna species (rock vole and short-eared owl) liable to be designated as vulnerable within an 8 km radius of the plant site. There is also one fauna species (bank swallow) that is a candidate to be designated as being in a precarious situation in the western portion of the plant site infrastructure.

20.2 Ore, Waste Rock, Tailings and Water Management Requirements

The following Items describe the environmental requirements for mining materials storage management facilities based on available information. The "Directive 019 sur l'industrie minière" Guideline (DIR-019) of the Quebec Government (MDDEP, 2012) is the main permitting guideline for mineralized material, waste rock, tailings, and water environmental management requirements with among others as Mining Association of Canada (MAC) (Guide to the Management of Tailings Facilities), Canadian Dam Association (CDA) and International Council in Mining and Metals (ICMM).

20.2.1 Geochemical Assessment

A study aiming to define the geo-environmental properties of the ore body, tailings, and waste rock to be generated at the Crater Lake mining site may be required to establish the environmental risk of those materials, including the status of acid drainage generation potential and metal leaching. The results must then be used to classify the materials according to the "Guide de caractérisation des résidus miniers et du minerai" of the Quebec Government (MELCC, 2020) (Provincial Guide). Results must also be used to classify tailings and waste rock under DIR-019 of the Quebec Government (MDDEP, 2012) and adapt specific designs, or potential re-use, according to the geochemistry of each material.

20.2.2 Waste Rock Management

Based on DIR-019, in case where ore or waste rock from Crater Lake site is acid-generating, leachable or radioactive, material stockpiling must be designed to prevent the transport of contaminants to groundwater. Under these circumstances, a 3D groundwater modeling is requested to address the percolation rate of the geological setting or foundation of each facility and adapt the design accordingly. A percolation water collection network, including drainage ditches around the stockpiles must also be installed in order to convey the contact water collected to the appropriate treatment facilities.

Based on the results of the chemical characterization of Crater Lake project ore and waste rock, management approach of this material will have to comply with DIR-019 provisions.

For the purpose of this PEA, it is assumed that waste rock from the Crater Lake site is non-acid generating and non metal leaching. Consequently, it is assumed that protection measures such as impervious liners are not required. The waste rock will be deposited on the natural ground. However, peripheral ditches are planned to collect runoff from the WRSF waters and direct them to a catchment pond. No waste rock is produced on the Sept-Îles site.

20.2.3 Tailings Management

As per DIR-019, the management of cyanide, acid-generating, leachable, contaminated by organic or radioactive compounds tailings generated by an ore or an ore concentrate treatment process requires protection measures meeting DIR-019 specifications.

The management method for these tailings must be designed to respect a maximum daily percolation flow of 3.3 l/m^2 for the bottom of the tailings management facility. It must also be demonstrated, through a modeling study, that the natural foundation or a designed sealing measures will prevent any significant deterioration in the quality of groundwater, failing which a change of tailings management facility design may be reviewed.

The modeling study must highlight the fact that the hydrogeological conditions in place, the physicochemical nature of the substrate on or in which the tailings will be stored and the design of the tailings management method, including the management of the water on the mining site, allow compliance with the groundwater protection DIR- 019 objectives.

Based on the results of the chemical characterization of Crater Lake project tailings, management approach of this material will have to comply with DIR-019 provisions.

For the purpose of this PEA, it is assumed that tailings produced on both the mining site and the plant site will be leachable for magnesium and magnesium oxide and process water will most likely contain reactive contaminants. The proposed TMFs designs for both site use liners and water management systems to collect all contact water from the impoundment, or drystack TMF.

20.2.4 Water Management

This Item provides a general description of the surface water management plan for the mine site. A description of the water management preliminary structures at the mine site is provided in Chapter 18.

According to Section 4 (1) of the MDMER, for the purposes of paragraph 36(4)(b) of the Fisheries Act, the owner or operator of a mine is authorized to deposit, or to permit the deposit of, an effluent containing any deleterious substance that is prescribed in section 3 in any water or place referred to in subsection 36(3) of the Act if:

a) the concentration of the deleterious substance in the effluent does not exceed the maximum authorized concentrations that are set out in columns 2, 3, and 4 of:
   - (i) Table 1 of Schedule 4, in the case of a mine in respect of which these Regulations apply for the first time on or after June 1, 2021, or in the case of a recognized closed mine that returns to commercial operation on or after June 1, 2021, or
   - (ii) Table 2 of Schedule 4, in any other case
b) the pH of the effluent is equal to or greater than 6.0, but is not greater than 9.5
c) the effluent is not acutely lethal

The targeted deleterious substance of Schedule 4 of the MDMER are the following: Arsenic, Copper, Cyanide, Lead, Nickel, Zinc, Suspended Solids, Radium 226 and, un-ionized ammonia.

In order to comply with MDMER, water runoff and seepage from mining and plant sites (contact water) will be collected, treated, and characterized to ensure the concentration of the deleterious substance in the effluent does not exceed the maximum authorized concentrations of the MDMER prior to discharging it to the environment.

Mining site and plant sites water management will include:

- Diverting non-contact water from undisturbed areas through diversion channels, to the practicable extent.
- Collecting runoff and seepage from mine facilities in collection ponds for reuse in the mine process, with excess conveyed to the water treatment plants and monitoring control before discharge to the environment.

A treatment plant will be constructed at the mining and the plant sites, with separate treatment units to account for specific treatment requirements for the different types of mine water. The water treatment plant is described in chapter 18. Surface water Geochemical transportation modeling will need to be conducted in order to identify the contaminants expected in each of the site effluents in order to properly design the water treatment plant.

Water treatment will allow to ensure that the mining effluent discharge meets DIR-019 and MDMER quality criteria. Additional environmental discharge objectives defined by MELCC may also be added to the discharge quality objectives. Those objectives will be defined by the MELCC during the permitting process.

20.2.5 Site Monitoring

The objective of the environmental monitoring program is to detect and document any changes in the environment in relation to the baseline, to verify the impact assessment and to evaluate the effectiveness of the mitigation or compensation measures proposed in the project impact assessment report. IPG will implement an environmental monitoring program, which will include the main following components:

- Effluents Quality Monitoring (DIR-019 and MDMER).
- Groundwater Quality and Piezometric Level (DIR-019).
- Water Quality Monitoring Studies (MDMER).
- Biological Monitoring Studies (MDMER).
- Mitigation Measures Monitoring (air quality, noise, vibration, runoff, etc.).

Additional monitoring component could be required as a condition of the issuance of permit or authorization from the government.

20.3 Regulatory Context

The regulatory context described in the following Items is based on regulations and acts in force at the time of the preparation of this report.

20.3.1 Environmental Impact Assessment Process

Quebec Authorities

Mining Site

The Crater Lake mining site is located in the Moinier region of the Quebec Environmental Protection Regime Application. The legislation associated with this region is described in the Regulation respecting the environmental impact assessment and review applicable to a part of the northeastern Québec region (chapter Q-2, r. 24). According to this regulation, as the Moinier Region is mentioned in Schedule A of the Quebec Environmental Quality Act (EQA), the project is subject to the environmental impact assessment and review procedure provided for in subdivision 4 of Division II of Chapter IV of Title I of the EQA for this region and must be the object of an authorization issued by the Quebec Government pursuant to Sections 31.1 and 31.5 of the EQA.

Based on the Moinier EIA procedure, Sections 4 and 5 and the first paragraph of Section 7 of the Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec (chapter Q-2, r. 25) apply to the Crater Lake mining site. Based on this same procedure, the Crater Lake mine site EIA has to be adapted by including the necessary modifications applying to projects undertaken in the de Moinier region and, for the purposes of this Regulation (chapter Q-2, r. 25), the Native people governed by the Regulation must include the Naskapis in particular.

Division III, Section 6 and Division V of the Regulation respecting the environmental impact assessment and review of certain projects (chapter Q-2, r. 23.1) also apply to the Crater Lake mine site, with the necessary modifications, to projects undertaken in the Moinier region and referred to this Regulation (chapter Q-2, r. 23.1).

Besides the usual consultation of the EIA, additional consultations methods are provided in the Regulation (chapter Q-2, r. 23.1), and the Minister must, immediately after making public the EIA pursuant to the first paragraph of Section 31.3 of the EQA with respect to a project undertaken in the Moinier region, transmit a copy thereof, as well as a copy of the related documents, to the Naskapi village referred to in subsection 13 of Section 131 of the EQA. The latter must transmit its comments and ask the Minister to hold a public hearing, where applicable, within 45 days following the date on which it received the file unless the Minister grants an additional delay owing to the nature or the importance of the project, in accordance with Section 31.8 of the EQA.

Upon failure to submit its comments within the delay prescribed pursuant to the first paragraph, the Naskapi village is deemed to have no objection to the carrying out of the project. In such a case or after receiving the comments of the Naskapi village, the file follows its course within the context of the environmental impact assessment and review procedure stipulated in Sections 31.3 to 31.8 of the EQA and in accordance with the Regulation respecting the environmental impact assessment and review of certain projects.

To initiate the EIA and review process, the Crater Lake project will have to be at a feasibility stage. It must be noted that the scope of the feasibility study may include mine site, plant site and roads in order to have a complete portrait of the project to submit to the government.

Plant Site

The Sept-Îles plant site is located in the Southern Quebec Environmental Protection Regime Application. Based on the available information, the plant site is qualified under Schedule 1 of the Regulation relating to the assessment and review of the environmental impacts of certain projects (chapter Q-2, r. 23.1) under the following provisions:

- Schedule 1 - Section 23. ORE PROCESSING

  The following projects are subject to the procedure:

— (1) the construction of a processing plant for:

— (a) rare earth ore

The plant site is therefore subject to the provincial EIA and review procedure provided in sub-section 4 of Section II of chapter IV of title I of the EQA and must be subject to an authorization from the Quebec government prior to proceed with construction.

Access Road

The planned access road will cross two distinct territories of the Quebec Environmental Protection Regime Application: 1) Moinier and 2) North of 55th parallel of the James Bay and Northern Québec Agreement (JBNQA).

Based on the available information, the access road is qualified under Schedule A of the EQA as a designated project subject to an environmental impact assessment and review procedure under the following provisions:

— Paragraph p) any access road to a locality or road infrastructure for a new project.

The same environmental impact assessment and review procedure as the one for the Crater Lake mining site is applicable for the portion of the road located within the Moinier region.

Portions of the access road located North of 55th parallel of the JBNQA would be subject to the specific environmental impact assessment and review procedure for this region (EQA, Title II, Chapter II).

The Kativik Environmental Quality Commission (KEQC), composed of Québec and Inuit representatives, will be responsible for assessing and reviewing portion of the access road project located in this region.

EIA Procedure Application

It must be noted that at the time this report was written, no project has gone through the Moinier EIA procedure. The first project to go through this procedure will be a first experience for the provincial authorities and the First Nations involved in the process (no project was conducted so far in this region).

Compare to the Moinier EIA procedure, the southern EIA procedure and the North of 55th parallel of the JBNQA are well known processes that the provincial government is used to deal with. However, it is impossible to identify how the government would consider the EIA procedure application for the mining site, the plant site and the road at this point, as they are located in three different application regimes. Project statement describing those three component may have to be submitted to the provincial government in order to allow it to determine the applications terms for the entire project.

Newfoundland and Labrador Authorities

The planned access road will extend into Newfoundland and Labrador province. Based on the available information, the access road is qualified under Section 35 of the Environmental Assessment Regulations (2003) under the Environmental Protection Act (O.C. 2003-220), Newfoudland and Labrador Regulation 54/03:

> "Section 35. (1) An undertaking that will be engaged in construction projects other than buildings that involve the:
> (b) construction of roads or the relocation or realignment of existing roads where a portion of the road will be more than 500 metres from an existing right of way"

Therefore, the preliminary analysis of the project indicates that the access road is subject to an environmental assessment under the Newfoundland and Labrador legislation.

The environmental assessment process will have to be initiated with a formal registration of the project, submitted in a prescribed format, to the Newfoundland and Labrador Department of Environment and Climate Change. The Registration will have to outline the proposed project and describe how it will affect the bio-physical and socio-economic environment. IPG will demonstrate in the Registration how the best practicable technology and methods will be used to minimize harmful effects. At the conclusion of the review period, the Minister will advise IPG whether the project will require an Environmental Preview Report, an Environmental Impact Statement, or if the project his released or rejected.

Federal Authorities

A modification of the Impact Assessment Act ("IAA") came into force on August 28, 2019, along with a new set of regulations. The IAA repeals the Canadian Environmental Assessment Act of 2012 (CEAA, 2012), but continues the approach taken under CEAA 2012 to designate projects by type and thresholds prescribed by regulation. The provisions in the schedule to the Physical Activities Regulations associated to the mining operations of the Crater Lake project, in whole or in part are the following:

> "Section 18(c) The construction, operation, decommissioning and abandonment of a new rare earth element mine with an ore production capacity of 2 500 t/day or more;"

With its average extraction rate of 2,350 t/day and a process plant with a capacity of 2,350 t/d, the preliminary analysis of the project indicates that the Crater Lake mining operations is not subject to the IAA.

However, the Crater Lake project involves the construction of an 1,830 m airstrip on the mining property. This infrastructure is qualified under the Physical Activities Regulations as a designated project under the following provisions:

Section 46 The construction, operation, decommissioning, and abandonment of one of the following:

— A new aerodrome with a runway length of 1 000 m or more

— A new aerodrome that is capable of serving aircraft of Aircraft Group Number IIIA or higher

— A new runway at an existing aerodrome with a length of 1 000 m, or more

The preliminary analysis of the project indicates that the Crater Lake airstrip is subject to the IAA. IPG will submit a project description to the Impact Assessment Agency of Canada so that it can rule on whether or not the project is subject to a federal EIA and determine if and how the decision would apply to the whole Crater Lake project or only a portion of it (e.g., the airstrip).

20.3.2 Permitting Requirements

Following release from the provincial decree and federal authorization (EIA approval if applicable), the project will require several approvals, permits and authorizations to initiate the construction phase, operate and close the project. In addition, Imperial Mining Group Ltd. will be required to comply with any other terms and conditions associated with the decree and authorization issued by the provincial and federal authorities. A preliminary and non-exhaustive list of permits, approvals, and authorizations that may be required for the Carter Lake project is presented in Table 20-1. Permits and authorizations may also be required from other jurisdictions, such as municipalities, if any are affected.

Table 20-1: Preliminary and Non-Exhaustive List of Permitting Requirements

Activities	Type of request	Authority
Quebec		
Rehabilitation and restoration plan	Approval	Quebec Ministry of Energy and Natural Resources
Mine waste management facilities and processing plant location	Approval	Quebec Ministry of Energy and Natural Resources
Infrastructure implantation on public land	Lease	Quebec Ministry of Energy and Natural Resources
Mining operations	Lease	Quebec Ministry of Energy and Natural Resources
Construction and operation of an industrial establishment	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Withdrawal of water, including associated infrastructure	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Establishment of drinking water, sanitary waters and mine waste waters management and treatment facilities	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Installation and operation of any other apparatus or equipment designed to treat water, in particular in order to prevent, abate or stop the release of contaminants into the environment	Authorization	Quebec Ministry of Environment and Fight Against Climate Change

Activities	Type of request	Authority
Installation and operation of an apparatus or equipment designed to prevent, abate or stop the release of contaminants into the atmosphere	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Construction on land formerly used as a waste elimination site and any work to change the use of such land	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Industrial depollution attestation	Attestation	Quebec Ministry of Environment and Fight Against Climate Change
Work, structures or other interventions carried out in wetlands and water bodies and waterstreams	Authorization	Quebec Ministry of Environment and Fight Against Climate Change
Quarry development	Authorization	Quebec ministry of Energy and Natural Resources
Carry out an activity likely to modify a wildlife habitat	Authorization	Quebec Ministry of Fauna, Forest and Parks
Forest intervention licence	Licence	Quebec Ministry of Fauna, Forest and Parks
Harvest wood on public land where a mining right is exercised	Permits	Quebec Ministry of Fauna, Forest and Parks
Build or improve a multi-use road	Permits	Quebec Ministry of Fauna, Forest and Parks
High-risk petroleum equipment	Permit	Régie du Bâtiment du Québec
Explosive’s possession, magazine and transportation	Permit	Sureté du Québec
Newfoundland and Labrador		
Release from Environmental Assessment Process		NFL Department of Environment and Climate Change
Permit to Occupy Crown Land	Permit	NFL Department of Fisheries, Forestry and Agriculture
Land Lease	Lease	NFL Department of Fisheries, Forestry and Agriculture
Species at risk permit	Permit	NFL Department of Environment and Climate Change
Nuisance Wildlife Permit	Permit	NFL Department of Fisheries, Forestry and Agriculture
Commercial Cutting Permit	Permit	NFL Department of Fisheries, Forestry and Agriculture
Permit to Alter a Body of Water	Permit	NFL Department of Environment and Climate Change
Federal		
Construct, place, alter, rebuild, remove or decommission an infrastructure in, on, over, under, through or across any navigable water	Notice, Approval and/or Exemption ,	Transport Canada
Harmful alteration, disruption or destruction of fish habitat	Authorization ,	Federal Department of Fisheries and Ocean
Explosives manufacturing plant and magazine	Licence	Federal Ministry of Natural Resources
Use of nuclear substances and radiation devices	Licence	Canadian Nuclear Safety Commission
Storage of mining waste in fish habitats & MDMER Schedule 2 Modification	Authorization	Environment and Climate Change Canada

Activities	Type of request	Authority
Notice and Environmental Emergency Response Plan	-	Environment and Climate Change Canada

20.4 Social or Community Considerations

20.4.1 Consultation Activities

No information and consultation activities were initiated with First Nations, local communities, or stakeholders so far.

20.4.2 Social Components and Related Requirements

Land Planning and Land Use

Mining site

The Project is located in the administrative region of Nord-du-Québec (10) in the RMC Kativik Regional Administration. The limits of the land are located on the « unorganized territory of the municipality of Rivière-Koksoak.

No permanent residences are located on the project site. The nearest settlement is the town of Schefferville, located in the neighbouring Regional County Municipality of Caniapiscau, in the North Shore administrative region. Schefferville is located approximately 188 km (lineal distance) from the western limits of the Crater Lake site.

Archaeological potential study aiming at acquiring sufficient information about archaeological heritage that may be present within the Crater Lake mining property may be useful to avoid potential issues with regard to archaeological aspects of the project prior to its construction.

Plant Site

See Figure 18-1 depicts the plant site.

Proximity of Local and First Nations

Mining Site

The closest First Nations to the project site (included in the province of Quebec) are the following:

- the Naskapi community of Kawawachikamach (187 km).
- the Matimekush-Lac John Innu community (approximately 193 km). This community is spread out over 2 territories, the first one in Matimekush and the second one at Lac John: 3.5 km from Matimekush and the center of Schefferville.

On the Labrador side, the closest First Nation is the Mushuau Innu First Nation (182 km), bordering the Labrador Sea.

No land in a reserve is located within the proposed layout. The project area is, however, located on land that is subject to a comprehensive land claims agreement or a self-government agreement, within the territory of the James Bay and Northern Quebec Agreement (JBNQA). In addition, the Project covers territory that is within the 1978 Northeastern Quebec Agreement (NEQA). Several comprehensive land claim agreements also overlap on the portion of the territory contemplated by the company. They are as follows:

- The Labrador Inuit Association (1976), a Quebec claim.
- Labrador Innu (1977).
- NunatuKavut Community Council (1991).
- The Naskapi Band of Quebec (1995), Labrador Claim.

The project is therefore located on a common territory of interest for the Inuit, Innu, and the Naskapi.

Social Related Requirements

Engagement Activities

The Provincial and Federal governments recommend that project initiators engage in good faith, as soon as possible, in a process of information and consultation with locals and indigenous communities, with an approach based on respect, transparency and collaboration.

The MELCC published the "Guide sur la démarche d'information et de consultation réalisée auprès des communautés autochtones par l'initiateur d'un projet assujetti à la procédure d'évaluation et d'examen des impacts sur l'environnement" (MELCC, 2020) for the implementation of an information and consultation process with indigenous communities for projects subject to the EIA and review procedure. The "Ministère de l'Énergie et des Ressources naturelles" (MERN) also published a Native Community Consultation Policy specific to the mining sector (MERN, 2019).

No engagement activities were initiated with First Nations, local communities, or stakeholders along with the Crater Lake project.

20.5 Mine Closure Requirements

Under the Mining Act, a person who performs prescribed exploration or mining work must submit a closure plan for the land affected by their operations, subject to approval by the MERN and is conditional upon receipt of a favourable decision from the MELCC. This approval is required for the release of the mining lease and the mining operations to begin (including the construction phase).

The main objective of a mining closure plan is to return the site to an acceptable condition.

Protection, reclamation and closure measures that will be presented to the government will aim to return the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety.
- Limiting the production and spread of contaminants that could damage the receiving environment and, in the long term, aiming to eliminate all forms of maintenance and monitoring.
- Returning the site to a condition in which it is visually acceptable (reclamation).
- Returning the infrastructure areas (excluding the tailings impoundment and waste rock piles) to a state that is compatible with future use (rehabilitation).

According to the Regulation respecting Mineral Substances other than Petroleum, Natural Gas and Brine of que Mining Act, mining companies must now provide a financial guarantee. This financial guarantee ensures that funds will be available to carry out the work provided for in the closure plan in the event of default by the proponent. It covers the entire cost of land rehabilitation and reclamation work for the entire mine site as provided for in the closure plan.

In November 2017, the MERN published the Guidelines for the preparing mine closure plans in Québec (MERN, 2019). A detailed breakdown of the dismantling cost for all infrastructure built on-site must now be provided and the engineering and supervision fees (indirect costs) have been fixed to a minimum of 30% of the direct cost (at conceptual stage), which include the post-restoration monitoring. A mandatory contingency of minimum 15% must be added to the estimated cost. The proponent who engages in mining operations must pay the financial guarantee according to the following terms:

- The guarantee must be paid in three payments:
- The first payment must be made within 90 days of receiving the plan's approval.
- Each subsequent payment must be made on the anniversary of the plan's approval.

— The first payment represents 50% of the total amount of the guarantee, and the second and third payments represent 25% each.

Total guarantee for the Crater Lake Project is estimated at 145,0 M$, including the direct and indirect costs, and a 15% contingency. Closure costs are estimated to a total of 74.4 M$ for Crater Lake Site and 65.1 M$ for the Sept-Îles Site. This cost includes site rehabilitation and restoration as well as the post-restoration monitoring for 5 to 10 years of both sites. The guarantee must remain in effect until the certificate of release provided for in Section 232.10 of the Mining Act has been issued.

All buildings, and surface infrastructure will be dismantled, including electrical and support infrastructure such as both process plants, as well as water management facilities (ponds, WTP, and ditches), unless it is shown that they are necessary to achieve and maintain a satisfactory condition, or to support the area's socio-economic development.

All areas affected by mining operations (for example, building sites, TMFs, WRSF, and road surfaces) will be covered or vegetated to control erosion and to return the site to a natural appearance integrated in the surrounding landscape. The cost of reclaiming the Crater Lake site TMF, as well as the Sept-Îles site TMF are included.

Before the revegetation of the affected areas, a characterization study certified by an expert authorized under Section 31.65 of the EQA must be submitted to the regional branch of the MELCC. If the study reveals the presence of contaminants in a concentration exceeding the regulatory limit values, a land rehabilitation plan must be submitted for approval.

The Tailings and Waste rock areas will be reclaimed to ensure geotechnical stability and to prevent any discharge of contaminant into the environment. The reclamation concept for the Crater Lake mining and plant site's TMFs consists in the implementation of an engineered cover. Since no notable issues with waste rock geochemistry is foreseen, the WRSF (mining site only) will be covered with granular material to prevent wind and hydraulic erosion of the pile.

A post-closure monitoring and maintenance program will be carried out to ensure the physical stability of all infrastructure and the effectiveness of any remedial measures applied at the site. The post-closure monitoring and maintenance program will include:

— A physical stability monitoring and maintenance program.
— An environmental monitoring program.
— An agronomical monitoring program.

Closure work will begin within three years of the cessation of operations. A certificate of release may be issued when:

— The MERN is satisfied that the closure work has been completed in accordance with the closure plan approved by the MERN, and no sum of money is due to the MERN with respect to the performance of the work.
— The MERN is satisfied that the condition of the land affected by the mining operations no longer poses a risk for the environment or for human health and safety.
— The MERN receives a favourable notice from the MELCC.

21 CAPITAL AND OPERATING COSTS

The Capital and Operating Costs Item of the report is based on design criteria and engineering performed by the various QPs. Each QP contributed the cost information that is pertinent to their work.

The overall capital and operating costs for the Project are presented in Table 21-1.

All capital works and the associated Capital Costs are at the sites associated with the Project proper. No Capital cost for upgrading infrastructure at locations other than the Project proper is included in this feasibility study.

Sources for the Capital costs include vendor quotations, historical data, similar projects, CostMine information, and empirical factors. Hourly rate costs for installation of equipment and for rental of construction equipment were based on similar projects.

Table 21-1: Crater Lake Mine / Mill and Sept-Îles Hydrometallurgy Facility Capital and Operating Cost Listing

CAPITAL COSTS	MCA$	($/t milled)
Direct Costs		
Mine Equipment, Crater Lake	$13.7	$1.29
Mill Plant Construction, Crater Lake	$63.7	$6.02
Hydrometallurgical Facility Construction, Sept-Iles	$160.1	$15.13
Electrical Distribution, At Both Crater Lake and Sept-Îles Sites	$14.1	$1.33
Infrastructure, At Both Crater Lake and Sept-Îles Sites	$113.5	$10.73
Tailings Storage Facilities including Water Management, At Both Crater Lake and Sept-Îles Sites	$69.3	$6.55
Initial Winter Road plus Orma Lake Road Rehabilitation	$46.6	$4.40
Between Sites Concentrate Handling Infrastructure	$27.5	$2.60
Crater Lake Camp (200 Person Capacity)	$26.5	$2.50
Pre-Production Mining Licenses	$0.2	$0.02
Pre-Production Mine and Mill Expenses to Capital	$67.9	$6.42
Subtotal Direct Costs	***$602.9***	***$57.00***
Owner's Costs	$6.8	$0.64
Indirect Costs (19% of Directs) *	$102.0	$9.64
Contingency (25% of Directs and Indirects) *	$159.2	$15.05
Total Capital Cost (All In, Taxes Extra)	**$870.1**	**$82.33**

OPERATING COSTS	MCA$	($/t milled)
Mine Crater Lake	$248.5	$23.49
Mill Crater Lake	$430.9	$40.73
Power Plant Crater Lake	$36.0	$3.40
Surface Mobile Equipment Crater Lake	$27.5	$2.60
Water Management Crater Lake	$192.4	$18.19
Lodging Crater Lake	$140.1	$13.25
Transportation + Domes Between Sites	$1,387.6	$131.18
Hydromet Facility Sc_2O_3 at Sept-Iles	$58.5	$5.53
Hydromet Facility Al + 2% Sc Master Alloy at Sept-Iles	$1,139.8	$107.75
Surface Mobile Equipment at Sept-Iles	$24.8	$2.35
Water Management at Sept-Iles	$108.6	$10.27
less Mine and Mill Pre-Production Expenses to Capital	($67.9)	($6.42)
Total Operating Cost	**$3,726.8**	**$352.32**
Selling General & Administrative Costs	$83.6	$7.90
Royalties (one time buy out)	$2.0	$0.20
Sustaining Capital Costs + Restoration	$207.7	$19.63
Total Operating Cost (All in)	**$4,020.1**	**$380.05**

Note* - capital indirects and contingencies are exclusive of pre-production expenses, mining licenses and owner's costs.

21.1 Capital Expenditures

21.1.1 Responsibility Matrix

Responsibility for the cost estimates has been divided amongst the study contributors as follows:

- General site infrastructures at both Crater Lake and Sept-Îles locations, including site roads, earthworks, and buildings; power distribution; diesel power plant or connection to Hydro-Québec grid; surface water management infrastructure; final effluent water treatment; process plant buildings and ancillary installations.
- Scandium Oxide process plant at Crater Lake, including crushing section, scandium oxide and REE recovery section, including concentrate thickening and filtration, magnetics, kiln, final tailings thickening, filtration, and dry tailings.
- Hydrometallurgical plant at Sept-Îles including a Master Alloy (AL-Sc2%) production section separate from the Scandium Oxide and REE Oxide production sections.
- All pre-production mining related activities at Crater Lake, such as overburden removal and the drilling, blasting, loading, and hauling of the rock material, as well as the purchase of the mining equipment.
- Logistics between the Crater Lake and Sept-Îles sites including all access roads, railroad sidings, concentrate storages, transfer stations for concentrate loading and unloading.

21.1.2 Basis of Estimate

The purpose of the Basis of Estimate is to describe the methodology used in the development of the Capital Expenditures (CAPEX) estimate. The CAPEX estimate has been structured based on the Work Breakdown Structure (WBS). The CAPEX estimate has been designed to provide the details required to establish a cashflow for financial evaluation. The Base Date of the CAPEX estimate is Q1 2022.

The accuracy of the estimate is ±35%, based on a global engineering completion of approximately 10% (Class 4 according to AACE 47r-11 recommended practice).

The CAPEX estimate is assembled in Canadian dollars (CAN$) and all sales taxes are excluded from the estimate. No escalation factor was applied to equipment and material quotes received. For financial modelling purposes, estimates in local currencies have been time-phased separately for inclusion in the financial model, in order to be able to perform cost and exchange rate sensitivities on the complete financial model.

21.1.3 Work Breakdown Structure

The Capex estimate and documentation has been structured on the Work Breakdown Structure (WBS) and the cost coding structure defined for the Project. Table 21-2 shows the Work Breakdown Structure used for the Project.

Table 21-2: Work Breakdown Structure – Level 1

Area	Description
000	General Site
100	Off Site Infrastructure
200	On-site Surface Facilities
300	Mineral Process Plant & Mining Equipment
400	Environment
500	Municipal Infrastructure
800	Tailings and Water Management
700	PCM, contingency, other indirect costs
900	Pre-production (Mining – Pre-strip, Mill Startup)

21.1.4 Infrastructure Supply

Infrastructure supply costs are based on prior projects with escalation factors applied.

21.1.5 Budgetary Supply Quotations

Quotations were based on prior projects and memorandums between WSP and prospective suppliers.

The pricing recommendation for the CAPEX estimate was identified and selected in collaboration with Imperial Mining Group.

21.1.6 Labour Hours

Labour hours were estimated for all construction tasks. Hours were estimated using experience from similar projects or handbooks. Direct field supervision hours are included in the labour hours of each item and trades (foremen).

A Productivity Factor adjustment of 1.2 was integrated to all disciplines labour hours to account for local site conditions such as Project location / size, labour availability, working schedule, workforce skills and availability, distance from camp to site, weather, working conditions, contract strategy, staff breaks, daily / weekly coordination and health and safety meetings.

21.1.7 Direct Labour Rates

WSP considering 70 hours per week, 14 days in / 14 days out schedule, supervisor/foreman, overtime, benefits, tools, individual protective equipment, transportation to site premium, insurances, contractor's administrative fees and profit. A 20% indirect supervision factor was considered for the contractor's high-level project management team (construction supervisor, administrative clerk, procurement / logistics, HSSE agent, etc.) and also the required construction equipment supply / rental for each discipline (mobile and lifting equipment, expensive specialized tools).

Room and board, mobilization / demobilization, field site temporary facilities, temporary construction infrastructures (scaffolding, platform, etc.), consumables (fuel, lubricant, etc.) and winter conditions are not included. These costs are all included in the construction indirect costs.

21.1.8 Material Take-Offs and Unit Costs

Material take-offs (MTOs) are based on neat quantities, with applied factors for waste and details not shown in actual documentation. However, no design growth factor was applied on these quantities.

21.1.9 Earthworks

Earthwork quantities are generated from grading designs using Autodesk Civil 3D software.

Unit costs were established based on past project productivity and references. Cost of operated machinery was based on the rates of the *Taux de location de machinerie lourde avec opérateur* published by the *Centre de services partagés du Québec* for the year 2021. This reference recommends using a 10% overhead on equipment rental cost for works north of the 49th parallel when compared to rates used in southern Québec. Those rates include the cost of rental, operation (oil and gas), and the operator man-hour. Aggregate preparation costs are based on similar projects.

Roads and buried services (potable water, fire water, freshwater and sewage) piping lengths were based on plot plan developed for the project.

The key assumptions are as follows:

- A layer of organic matter of 250 mm (average) was considered.
- Cut and fill activities include excavation of second-class material, haulage within 500 m from its point of origin, and reuse in the backfill of roads or pads below the infrastructure line.
- Excavation activities include excavation of second-class material and haulage within 1,000 m from its point of origin.
- If open pit waste rock is used for the backfill, no cost was considered, since doing a mass backfill with muck is comparable to the disposal in a waste dump by the mining contractor.
- No pavement is required on both sites.

21.1.10 Concrete

Concrete is included with the building based on similar projects.

21.1.11 Structure and Architecture

Structural steel is included with the building based on similar projects.

21.1.12 HVAC

HVAC is included with the building based on similar projects.

21.1.13 Electrical and Instrumentation

Major Electrical Distribution and Substation requirements were based on similar projects. Man-hours for the installation of the equipment, services, grounding, cable trays, and cables were based on an estimation book.

Electrical distribution within buildings is included with the purchase of the building.

21.1.14 Factors Applied to Direct Costs

The following direct cost factors were considered and applied:

- Design growth – Not factored on direct costs, design growth is considered a component of contingency in indirect costs.
- Construction waste – To be considered at next stage of design.
- Productivity factor – As described in the hourly rates Item above.

— Seasonal influence – There was no cost added in direct costs for seasonal influence.

21.1.15 Estimate Exclusions

The following costs are not included in the CAPEX estimate.

Schedule delays and/or associated costs, such as those caused by:

- Unexpected site conditions
- Unidentified ground conditions
- Labour disputes
- Force majeure
- Permit applications
- Foreign currency changes from Project exchange rates
- Economy factors/pressure on labour productivity (less skilled workforce)
- Weather related issues.
- Sales Taxes and Duties.

21.2 Direct Cost Estimates

CAPEX summaries and details are presented in Table 21-3.

Table 21-3: Summary of Direct Capital Costs

CAPITAL COSTS	MCA$	($/t milled)
Direct Costs		
Mine Equipment, Crater Lake	$13.7	$1.29
Mill Plant Construction, Crater Lake	$63.7	$6.02
Hydrometallurgical Facility Construction, Sept-Iles	$160.1	$15.13
Electrical Distribution, At Both Crater Lake and Sept-Îles Sites	$14.1	$1.33
Infrastructure, At Both Crater Lake and Sept-Îles Sites	$113.5	$10.73
Tailings Storage Facilities including Water Management, At Both Crater Lake and Sept-Îles Sites	$69.3	$6.55
Initial Winter Road plus Orma Lake Road Rehabilitation	$46.6	$4.40
Between Sites Concentrate Handling Infrastructure	$27.5	$2.60
Crater Lake Camp (200 Person Capacity)	$26.5	$2.50
Total Direct Costs	***$534.8***	***$50.56***

21.3 Indirect Capital Costs

21.3.1 Basis of Estimate for Indirect Capital Costs

Summary of Indirect Capital Costs

The indirect Capital cost covers for administration and overhead, project development, and EPCM and other indirects.

Indirect Capital costs are summarized in Table 21-4.

Table 21-4: Summary of Indirect Capital Costs

Item	Initial Capital (M CAN$)
Owner's Costs	6.8
Engineering	28.9
Construction – Health and Safety	2.7
Construction Management, Contractor Indirect, temporary camp	48.1
Commissioning and Startup	10.7
Freight	6.1
Spares	5.5
Total Indirect Capital Estimate	108.8

Scope and Basis of Estimation of Indirect Costs

Engineering represents the detailed engineering during and ahead of the construction period. Engineering for studies preceding the work or for establishing the budgets for the work are excluded. Construction Health and Safety is a component of the PCM for a project as is Construction Management, Contractor Indirect and in this case the temporary camp requirements. Temporary camp includes such services as air and ground transportation, electricity, LNG, camp services, water management, site security, road maintenance, general liability and construction insurances, and purchase of service equipment. Upon completion of construction separate expenditures are shown for commissioning and start-up. With no evidence of freight and spares being included in the direct costs of procuring the equipment, they are being shown as indirects for this project. Cost estimation is based on requirements and proposed budget unit costs.

Project development excludes studies related to the environment, water treatment, permitting and power distribution.

EPCM is made up of the engineering, construction health and safety and Construction Management and Commissioning and Startup components of the above table 22.6. It includes detailed engineering, procurement, construction management, commissioning, and start-up. The Crater Lake EPCM oversight team is included separately as part of the Owner's Costs.

Excluding freight and spares the $90.4 million for EPCM cost is based on factors totalling 16.9% on all construction estimates except for pre-production activities, which will be managed by the pre-production mining and milling teams.

Spare parts and freight are based on factors on materials and equipment. Spare parts and freight for the mining equipment are excluded from the equipment purchase costs.

21.3.2 Design Allowances

A provision of $127.4M is included in the initial capital for design growth allowance realizing, as noted in Item 21.1.8, that no design growth factor was applied as part of the direct cost estimations.

21.3.3 Contingency

Additional to the design growth funds there is a provision of $31.8M is included in the initial capital for contingency bringing the total for growth and contingency to $159.2M, based on the level of development stage of the Project.

In order to meet the budget established for the Project in this estimate, it is expected that sufficiently developed engineering, adequate project management, realistic construction schedule and appropriate controls will be implemented.

21.3.4 Mine Rehabilitation Bond

The total estimated rehabilitation cost for the both the mine and mill and for standalone hydrometallurgical site over the life of the project is $139.5M.

The rehabilitation fund may be replaced by a bond issued by a reputable insurance company. Imperial Mining Group will obtain a bond to secure its rehabilitation obligations toward the Province of Québec. The cost of the bond has yet to be determined by Imperial Mining Group and accordingly has not been included in the estimate.

21.4 Pre-Production and Ongoing Capital Costs

Pre-production costs directly related to the mining operation were estimated by WSP. Hydrometallurgical facility start-up defines the beginning of the production period. The Cost estimate for the pre-production period includes all costs associated with a year of mining and milling along with the transportation of 25% of the concentrate to the rail line at Emeril. The pre-production cost breakdown is shown in Table 21-5.

Pre-production is $67.9M, and there are the initial capital expenditures of $802.9M for a total of $870.9M. Additionally, there will be $37.7M of upfront working capital that then is credited back to the ongoing capital over the life of the project. The overall funding prior to generation of revenue from sales is then shown in Table 21-5 as $908.6M. Table 21-11 presents both the pre-production and sustaining Capital costs. The Project capital costs were discussed independently in Items 21.2 and 21.3. As well as the capital costs over the life of mine there are restoration costs upon completion of mining, milling, and processing, which then identifies all liabilities to be $1,078.6M. Capital cost estimates are based on previous projects that included quotes for major mining equipment and where such quotes are missing, then based upon CostMine data.

Table 21-5: Pre-Production and Sustaining Capital Costs

Mining Capital Cost Items ($M)	Pre-Production	Ongoing	Total
Mining equipment, Replacements and licences	13.7	24.5	38.2
Processing and TSF	132.9	42.7	175.6
Site Infrastructure including Camp	154.0	1.0	155.0
Concentrate Transportation	74.1	0	74.1
Hydrometallurgical facility	160.1	0	160.1
SUB-TOTAL DIRECTS CONSTRUCTION	534.8	68.2	603.0
Construction Indirects + Contingencies	268.1	Opex	268.1
(A) SUB-TOTAL CONSTRUCTION	**$802.9**	**$68.2**	**$871.1**
OTHER COST LIABILITIES			
Pre-Production Mine and Mill and Logistics	67.9	Opex	67.9
Working Capital	37.7	-37.7	0
Restoration	0	139.5	139.5
(B) TOTAL PRE-PRODUCTION and RESTORATION CAPITAL	**$112.7**	**$101.8**	**$207.4**
INITIAL PROJECT CAPITAL (A + B)	908.6		
ONGOING CAPITAL plus RESTORATION (A + B)		170	
TOTAL CAPITAL plus OTHER LIABILITIES (A + B)	**$908.6**	**$170.0**	**$1,078.6**

21.5 Mine Capital Costs

The cost of pre-production work includes all mining operations during two six-month yearly periods. In the first yearly 6-month period, the owner's mining team will oversee the pre-production work for mining, which includes items such as, but not limited to, overburden removal, drilling, blasting, loading, and hauling of all rock material, and all other auxiliary work. The owner's mining team is included in the cost per tonne for mining and added indirects for EPCM of the mining are not needed.

This initial period includes the mobilization of a mining contractor for the pre-strip activities. In parallel, the EPCM team will oversee the mobilization of a construction contractor for the mill and other surface infrastructure construction activities.

The second yearly pre-production period is again 6 months in duration and involves the owner's mining team having hired their own workers to mine mineralized material for that year's six months of milling. Following the mining and milling, the site waits until the onset of winter to create a winter road for haulage of the concentrate to the rail line.

Overall mining equipment purchases and ongoing replacements total $38.2M of which $13.7M is incurred in the initial project period. This cost includes all the main mining equipment (i.e., trucks, drills, excavators, etc.) and all the support equipment (i.e., pick-up trucks, pumps, cables and sub-station for the electric front shovel, tower lights, etc.).

Table 21-6 presents the purchasing and replacement schedule for all the main mining equipment. The equipment purchases are incurred in the year the equipment is needed. These costs are not depreciated over time and do not consider a salvage value at the end of equipment life. Contingencies assigned to the initial capital considered that a lower amount was needed for these vehicles and the ongoing capital does not consider any contingency amounts for the equipment replacements. During the negotiation process for the purchase of the equipment, it could be advantageous to consider a financing plan to spread out these costs over time.

Table 21-6: Main Mining Equipment Purchasing and Replacement Schedule

Equipment	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	LOM
Hydraulic Excavator (5.8 m³)	1	0	0	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Loader (6.4 m³)	1	0	0	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Rigid Truck 47 tonnes	3	0	0	0	0	0	0	0	1	0	0	3	0	0	0	0	0	0	1	0	0	0	0	0	0	0	**8**
Articulated Truck 41 tonnes	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Surface Drill 127 mm (5 inch)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Dozer (D8 or simillar))	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	**3**
Motor grader (CAT14M or simillar)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	**3**
Small loader (Cat972 or simillar)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Small Excavator (Cat352or simillar)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	**3**
Water Truck (Cat 745 or simillar)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Compactor (Cat CS76B or simillar)	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Tractor	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Service Truck (Mechanical)	2	0	0	0	0	0	0	0	0	0	2	0	0	0	0	0	0	0	0	0	2	0	0	0	0	0	**6**
Personal carrier	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	**3**
Tyre Handler	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1**
50t Crane	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1**
Fuel Lube truck	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	**3**
Flat Bed truck	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**
Lighting Tower	7	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**7**
Pick-up truck	8	0	0	0	8	0	0	0	8	0	0	0	8	0	0	0	8	0	0	0	6	0	0	0	3	0	**49**
Boom truck	1	0	0	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**2**

New Purchase	Replacement	Used Purchase

Given that the production period is defined as the start-up of the hydrometallurgical facility, no capitalized revenue is generated during the pre-production period.

21.6 Mill Project Capital

The Capital costs for the beneficiation plant were estimated by WSP. Capital cost estimate for the beneficiation plant is based on the construction of a milling facility at the Project site. The cost estimation is based on 426,100 tonnes of mineralized material milled per year.

The process facilities include the primary and secondary crushing sections, crushed ore storage dome, the concentrator, and kiln and magnetics for scandium oxide concentrates. The office and laboratory is included. Tailings will be sent by trucks to a Tailings Storage Facility.

21.6.1 Mill Process Equipment

The costs for major process equipment were obtained from qualified suppliers and the remaining equipment costs were estimated from database or in-house estimation.

21.6.2 Mill Electrical and Communications

The electrical and communication costs were obtained from in-house database.

21.6.3 Mill Instrumentation and Control

Instrumentation and control costs were estimated from database and in-house estimation.

21.6.4 Mill Piping

For the beneficiation plant, piping costs were included with the building based on similar projects.

21.6.5 Mill Equipment Installation

Installation costs were estimated from database and in-house estimation.

The working capital required for plant start-up (first three months of operation) is included with the overall site requirements.

21.6.6 Mill Buildings

The costs for the beneficiation plant buildings and ancillary installations were estimated by WSP based on preliminary process flow diagrams.

21.7 Hydrometallurgy Project Capital

The Capital costs for the process plant were estimated by WSP. Capital cost estimate for the Aluminum-2% Scandium plant is based on the construction of a Hydrometallurgy facility at a site in Sept-Iles, Quebec. The cost estimation is based on 218,540 tonnes of scandium oxide concentrate feed per year.

The hydrometallurgical facility includes an alumina receiving & storage system, a pot gas scrubber system, electrolytic cells (350 kA, three), a pot house service crane, power transformer & rectifier, alloy transfer ladle, alloy holding furnace and an alloy casting system. An office area and laboratory including assay lab is also included. Tailings will be sent by truck to a TSF. Tailings will be deposited in a dry stack tailing storage facility.

21.7.1 Hydrometallurgy Process Equipment

The costs for major process equipment were estimated from database or in-house estimation.

21.7.2 Hydrometallurgy Electrical and Communications

The electrical and communication costs were obtained from in-house database.

21.7.3 Hydrometallurgy Instrumentation and Control

Instrumentation costs were obtained from database and in-house estimation.

21.7.4 Hydrometallurgy Piping

For the hydrometallurgy plant, piping costs were included with the building.

21.7.5 Hydrometallurgy Equipment Installation

Installation costs were estimated from database and in-house estimation.

The working capital required for plant start-up (first three months of operation) is included with the overall site requirements.

21.7.6 Hydrometallurgy Buildings

The costs for the hydrometallurgical facility buildings and ancillary installations were estimated by WSP based on process flow diagrams.

21.8 Operating Costs

The Life of Operations Plan (LoOP) operating cost breakdown is shown in Table 21-7.

Table 21-7: Life of Operations Plan (LoOP), Total Operating Costs by Area

PLANT / AREA	Total Operating Costs CA$M	Average LoOP $/t Mill Feed
Mine Crater Lake	$249	$23.49
Mill Crater Lake	$431	$40.73
Power Plant Crater Lake	$ 36	$ 3.40
Surface Mobile Equipment Crater Lake	$ 28	$ 2.60
Water Management Crater Lake	$192	$18.19
Lodging Crater Lake	$140	$13.25
Transportation, Dome and Road, Crater Lake to Churchill Falls	$222	$20.95
Transportation, Truck Between Crater Lake and Emeril	$916	$86.56
Transportation, Rail Between Emeril and Sept-Iles	$ 238	$22.49
Transportation, Rail Car Unloading System Usage at Sept-Iles	$ 12	$ 1.17
Hydrometallurgy Facility Sc2O3 Sept-Iles	$ 59	$ 5.53
Hydrometallurgy Facility Al - Sc2% Sept-Iles	$1,140	$107.75
Surface Mobile Equipment Sept-Iles	$ 25	$ 2.35
Water Management Sept-Iles	$109	$10.27
Less the Pre-Production Operating Costs to Project Capital	-$ 68	-$ 6.42
TOTAL OPERATING COSTS before G&A and Royalties	**$3,727**	**$352.32**
Corporate General and Administrative (G&A, Selling Costs)	$ 90	$8.54
Royalties	$ 2	$ 0.19
Less the Pre-Production G&A (Owner's Costs) to Project Capital	-$ 7	-$ 0.64
TOTAL OPERATING COSTS	**$3,812**	**$360.41**

21.8.1 Mine Operating Costs

The mining costs reflect the LOM plan for an Open Pit operation at Crater Lake that was prepared by WSP based upon a mineral resource block model provided by InnovExplo and have been divided into the following categories:

- Loading;
- Hauling;
- Drilling;
- Blasting;
- Stockpile and road maintenance;
- Mine services;
- Engineering department;
- Geology department;
- Maintenance;
- Overburden removal;
- General and management.

The total Mine Operating costs for the Project are $248.5M, or $7.80/t mined. Table 21-8 presents the total and unit Mine Operating costs for each category for the entire Project.

Table 21-8: Mine Operating Costs by Category

Mine Operating Cost Categories	Unit Cost ($/t mined)	Total Cost ($M)
Loading	0.50	16.0
Hauling	0.76	24.3
Drilling & Blasting	0.60	19.1
Auxiliary Equipment	1.82	57.9
Manpower	4.08	129.9
Aggregates	0.04	1.3
TOTAL	**$7.80**	**$248.5**

The Loading, Hauling, and Drilling categories are comprised mainly of the costs incurred for the operator's salaries and benefits, energy (fuel) and the main consumables (e.g. ground engaging tools, drill bits and rods, tires, etc.). A diesel fuel cost of $CA0.80/l including escalation at 2% pa was used from CostMine 2017.

The Blasting costs are comprised mainly the explosives, and the accessories. It should be noted that it is possible to purchase the explosives site plant from the supplier and reduce the operating costs over the mine life. However, this option requires an initial capital investment.

A detailed list of the manpower requirements is presented in Item 16.10. This list does not include contractor personnel.

The labour costs are all in rates based on a similar project that involve practices such as, but not limited to, a northern allowance, a production bonus and lodging on site. Going forward, some overtime and yearly bonuses would also be considered, and these would vary based on the position.

21.8.2 Mill Operating Costs

Annual and unit process operating costs for the beneficiation process plant that will be located at Crater Lake were determined for an annual mill ore tonnage of 426,100 metric tons of run of mine mineralized material that will produce 218,540 t/y of mineral concentrate. The estimated beneficiation plant operating costs are summarized in Table 21-9 and include manpower, requirements, electrical power cost, consumables and wear parts, maintenance, grinding media and reagents and mobile equipment. The total mill operating costs were estimated to be $17,355,760 per year or $40.73 per ton of processed feed material. The sources of information to develop the operating costs include in-house databases and outside sources.

Table 21-9: Beneficiation Plant Operating Costs

Operating Cost by Category	$/y	$/t	% of Costs
Manpower	2,803,119	6.58	16.2%
Utilities	10,050,510	23.59	57.9%
Consumables and Wear Parts	2,336,680	5.48	13.5%
Maintenance	1,176,376	2.76	6.8%
Mobile Equipment	197,653	0.46	1.1%
Grinding Media and Reagents	791,422	1.86	4.6%
Total Process Operating Cost	**17,355,760**	**40.73**	**100.0%**

Manpower, Beneficiation Plant

The manpower requirements for the beneficiation pant will be 30 people as listed on Table 21-10. The workforce includes the area of mill operation including administration, operations, maintenance, metallurgist, and technicians. The labour rates and benefits were based on the rates for similar jobs. Overheads of 25% of the base salary was considered. The total estimate is $2,803,119 per year, which equates to $6.58 per tonne of ore processed.

Table 21-10: Beneficiation Plant Workforce Costs

Description	Employees Qty	Total cost $/y	Unit cost $/t
Operations	21	1 949 280	4.57
Maintenance	9	853 839	2.00
Total Process Manpower	**30**	**2 803 119**	**6.58**

Utilities, Beneficiation Plant

The electric power costs were calculated using the total estimated load of the milling operation. The total power demand for the plant was estimated at 4,392 kW, which is equal to 13.02 MWh per year. The electrical power cost was estimated to be $0.25/kWh based on other similar remote northern projects. Genset operators and maintenance/overhaul costs not included.

Propane costs were based on estimated consumption for plant HVAC, at an average of $0.50/L.

Diesel costs were based on estimated consumption for the kiln olivine conversion, at an average $1.92/L.

Total utilities operating costs are listed on Table 21-11.

Table 21-11: Beneficiation Plant Utilities Costs

Description	$ / y	$ / t
Electrical Power	3,254,515	7.64
Propane	500,000	1.17
Diesel	6,295,996	14.78
Total Utilities Operating Cost	**10,050,510**	**23.59**

Consumables and Wear Parts, Beneficiation Plant

Consumables include mainly milling equipment wear parts such as crushing equipment wear parts, grinding equipment liners, pumping liners and parts, cyclone liners, filter cloths, etc. The total cost was estimated at $2,336,680 per year or $5.48 per tonne of ore processed and is based on comparable projects.

Maintenance, Beneficiation Plant

Equipment maintenance material costs have been established as a function of the relative equipment capital cost and factored based on benchmarking studies. The estimated maintenance material costs cover mechanical spares and wearing parts.

Total equipment cost is approximately $32M. Spare parts and miscellaneous costs have been estimated at 3.5% of the total equipment cost. Overall, the plant maintenance spares and supplies represent $1.18M per year.

Mobile Equipment, Beneficiation Plant

A mobile equipment list has been estimated, including the fuel consumption, the availability of the equipment, their utilization and the costs for consumables and parts. Table 21-12 summaries the mobile equipment requirements.

Table 21-12: Beneficiation Plant Mobile Equipment Costs

Description	# of units	Fuel Consumption L/y	Fuel Cost $/y	Maintenance Cost $/y	Total Cost $/y
4WD Double Cab Pickup (4 passengers)	1	17,520	18,057	5,959	24,017
Plant Forklift (4t)	1	6,570	6,164	5,087	11,251
Bobcat	1	13,140	12,328	5,087	17,415
Front-end Loader	1	52,560	49,311	8,642	57,953
Stand-by Generator - 800 kVA	1	50,000	46,909	3,303	50,213
Portable Pumps Generator - 200 kVA	1	27,375	25,683	1,982	27,665
Generator - 5 kVA	1	1,600	1,501	1,321	2,822
Portable Compressor	1	3,200	3,002	1,321	4,324
Diesel Welder	1	1,280	1,201	793	1,994
Total Mobile Equipment Operating Cost			**164,156**	**33,497**	**197,653**

Grinding Media and Reagents, Beneficiation Plant

The total operating costs for grinding media (SAG mill and ball mill) and reagents (flocculant preparation) were estimated at $791,422 per year, or $1.86 per tonne of ore processed. The grinding media consumption rates and unit costs were based on similar projects. The flocculant consumption and unit cost were also based on similar projects.

21.8.3 Power Plant, Crater Lake Operating Costs

The Crater Lake Power plant operating costs reflect the cost of maintenance. The basis is a determination of the annual cost based on 7000-hour annual operation over a 25-year period for seven diesel operated Caterpillar C175-16 gensets as depicted by Table 21-13.

Table 21-13: Crater Lake Power Plant Annual Maintenance Cost Determination

Power Plant Maintenance Items	Frequency Over 25 Year Period	Cost Per Maintenance Item	Total Cost Over 25 Year Period	Annual Maintenance Cost	Annual Maintenance Cost Per Tonne Mill Feed
Preventative Maintenance	245	$7,000	$1,715,000	$68,600	$0.16
First Overhaul	49	$240,000	$11,760,000	$470,400	$1.11
Second Overhaul	49	$460,000	$22,540,000	$901,600	$2.13
Total Maintenance			**$36,015,000**	**$1,440,600**	**$3.40**

The diesel fuel for the power plant is costed separately. See Table 21-10 for the mill electrical power cost determination which is representative of the cost of fuel expressed as a cost per kilowatt hour. See Table 21-16 for the cost of labour associated with the power plant operators.

21.8.4 Surface Mobile Equipment, Crater Lake Operating Costs

The Crater Lake Surface Mobile Equipment operating costs are based on an estimate of operating hours for the listed equipment as depicted in Table 21-14. The resultant CA$M1.1 per year equates to $2.60 per tonne mill feed.

Table 21-14: Crater Lake Surface Mobile Equipment Annual Operating Cost Determination

Description	Units	Operating Hours per Year	Maintenance Cost per Operating Hour	Fuel liters per Hour	Diesel Fuel Liters per Year	Fuel Cost per Year at $1.25/l	Maintenance Cost per Year (including tires)	Total Fuel and Maintenance Cost per Year	Sustaining Cost per Year	Total Yearly Operating Cost
Caterpillar 988 XE Loader- Surface loader (6-7 m3) with rock bucket	1	3000	$82.00	40	120,000	$150,000	$246,000	$396,000	$15,180	$411,180
Caterpillar 950 Loader - Surface service loader (3 m3) with forks, snow bucket & snowplow	1	3000	$53.00	14	42,000	$52,500	$159,000	$211,500	$15,180	$226,680
P6000 Lift truck - Fork Lift (6,000 / 8,000 lbs)	2	1000	$ 11.53	0	0	$0	$23,060	$23,060	$10,120	$33,180
Skid loader 242D3 CCE	2	1000	$23.00	10	20,000	$25,000	$46,000	$71,000	$10,120	$81,120
14M grader - Surface (14 ft blade with side wing)	1	2500	$60.00	16	40,000	$50,000	$150,000	$200,000	$12,650	$212,650
Pickup truck, 3/4 tonnes	5	1000	$ 9.60	10	50,000	$62,500	$48,000	$110,500	$25,300	$135,800
Total					272,000	$340,000	$672,060	$1,012,060	$88,550	$1,100,610

21.8.5 Water Management, Camp Maintenance, Crater Lake Operating Costs

The Crater Lake Water Management and labour for Camp Maintenance yearly costs are derived as shown in Table 21-15. The labour cost determinations shown as a cost in

Table 21-32 are depicted in Table 21-33 and Table 21-34. The resultant CA$M7.5 per year until year 14 and CA$M7.7 per year thereafter for water management and camp maintenance equates to an $18.19 per tonne of mill feed average cost over the LoOP.

Table 21-15: Crater Lake Water Management and Camp Maintenance Annual Operating Cost Determination

Item	Quantity	Unit of Measure	Unit Cost / Year	Year -2	Year -1	Years 1 to 14	Years 15 to 25
Crater Lake							
Water Management				Capitalized		Operating Expense	Operating Expense
WTP operation costs	400	m3/h	$ 0.10		$ 350,400	$ 350,400	$ 350,400
Ditch maintenance	According to ditch length on site	l. m.	$ 350.00		$1,625,050	$1,625,050	$1,825,950
Pond Maintenance	15% of CL Pond CAPEX	lot	$ 650,000		$ 650,000	$ 650,000	$ 650,000
Labour							
Waste and Water Management	See Table 21.17	lot	$1,118,000	$1,118,000	$1,118,000	$1,118,000	$1,118,000
Camp and Site Management	See Table 21.18	lot	$3,059,731	$1,529,866	$3,059,731	$3,059,731	$3,059,731
Engineering, Monitoring and reporting	10% of above costs		$ 680,318		$ 680,318	$ 680,318	$ 700,408
Total Crater Lake				**$2,647,866**	**$7,483,499**	**$7,483,499**	**$7,704,489**

Table 21-16 and Table 21-17 represent the surface labour requirements on site at any given time. For a rotation, such as14 days in and 14 days out, the surface labour payroll and thereby the yearly labour cost shown in the tables would normally be doubled. With the Crater Lake site operating six months of the year, the labour cost, as shown in the tables below, then allow for a 14/14 rotation without a doubling of the labour cost.

Table 21-16: Crater Lake Waste and Water Management Labour Cost Determination

Occupation	Function	Number of Operators	Shift	Base Salary	Benefits Ratio	Yearly Each	Yearly Total
Superintendent		1	Day	$124,800	1.25	$156,000	**$156,000**
Engineer – Tailings Management Facility		1	Day	$104,000	1.25	$130,000	**$130,000**
Engineer – Water		1	Day	$104,000	1.25	$130,000	**$130,000**
Supervisor		1	Day	$83,200	1.25	$104,000	**$104,000**
Operator / Repairer	Machine operator	2	Day and Night	$72,800	1.25	$91,000	**$182,000**
Operator / Repairer	Maintenance work	1	Day	$72,800	1.25	$91,000	**$91,000**
Operator / Repairer	Pumps, conduits, Utilities	1	Day	$72,800	1.25	$91,000	**$91,000**
Daily	Inspections	1	Day	$41,600	1.25	$52,000	**$52,000**
Technician	WTP maintenance and operation	2	Day and Night	$72,800	1.25	$91,000	**$182,000**
TOTAL ON SITE		**11**					**$1,118,000**

Table 21-17: Crater Lake Camp and Site Management Labour Cost Determination

Camp and site management	Number of Operators	Shift	Base Salary	Benefits Ratio	Yearly Each	Yearly Total
Camp						
Camp manager	1	Day	$90,000	1.25	$112,500	**$112,500**
Reception desk	2	Day	$70,000	1.25	$87,500	**$175,000**
Housekeeping	3	Day	$63,000	1.25	$78,750	**$236,250**
Site						
Site manager	1	Day	$90,400	1.25	$113,000	**$113,000**
Security guard	2	Day + night	$73,450	1.25	$91,813	**$183,625**
Loader / grader Operator	2	Day	$84,750	1.25	$105,938	**$211,875**
Journey man	1	Day	$70,625	1.25	$88,281	**$88,281**
Warehouse management	3	2 Day + 1 Night	$76,840	1.25	$96,050	**$288,150**
Dry and warehouse housekeeping	1	Day	$79,840	1.25	$96,050	**$96,050**
Power Plant						
Manager	1	Day	$105,000	1.25	$131,250	**$131,250**
Electrical Engineer	1	Day	$95,000	1.25	$118,750	**$118,750**
Operator / technician	6	4 Day + 2 Night	$82,000	1.25	$102,500	**$615,000**
Truck shop						
Maintenance foreman	1	Day	$84,000	1.25	$105,000	**$105,000**
Mechanics	5	3 Day + 2 Night	$78,000	1.25	$97,500	**$487,500**
Welder	1	Day	$78,000	1.25	$97,500	**$97,500**
TOTAL ON SITE	**31**					**$3,059,731**

21.8.6 Lodging, Crater Lake Operating Costs

The Crater Lake lodging and travel to and from site costs are presented as an annualized amount in Table 21-18.

The resultant $5,538,000 per year for lodging and travel costs associated with a fly in, fly out operation equates to $13.25 per tonne of mill feed.

Table 21-18: Crater Lake Lodging (Room and Board) and Travel Cost Determination

Item	Quantity	Unit Cost	Year -2	Year -1	Years 1 to 25
Crater Lake Camp			Capitalized		Operating Expense
Food by employer by day including shipment by plane to site	200 people per day	$85 each for 182 days	Construction Contractor Provides		$3,094,000
Cook	4 cooks for 26 weeks	$3,500 per cook per week			$ 364,000
Transportation (fly in, fly out)	2600 trips per year	$800 per trip			$2,080,000
Total Yearly Camp Cost					**$5,538,000**

21.8.7 Transportation of Concentrate Operating Costs

The concentrate is kiln dried and thereby of a dry powdery consistency with no appreciable moisture content. The following is the basis of estimate for the transportation of the bulk concentrate without having to use costly containers:

- During the six months of temperate weather, all 218,540 tonnes of concentrate will be taken from the kiln discharge into a bulk storage facility (dome) until the onset of Winter.
- The domes will be sized to allow trucks and loaders entry to avoid loading in the open and associated losses.
- Yearly, a contractor will construct a winter road from Crater Lake to Orma Lake Road.
- Upon preparation of the winter road, the 218,540 tonnes of stored concentrate will be loaded by contractor into covered and sealed 35 tonne capacity highway haul trucks and moved over a four-month period to a second IPG year-round storage (dome) at IPG's newly constructed railroad siding in Emeril, Newfoundland.
- The trucks will unload within the dome at Emeril and the bulk concentrate will be stored until the hydrometallurgical facility calls for it to be loaded into rail cars (wagons).
- Daily, the concentrate will be loaded into eight covered and sealed wagons provided by Imperial Mining Group. A portable crane at the Emeril railroad siding will remove and install the sealable covers.
- The wagons, holding about 90 tonnes each, will be transported by the existing QNS&L railroad to Sept-Îles over a 305 days period.
- Simultaneously, another eight empty wagons with sealed covers provided by Imperial Mining Group will be returned by QNS&L to Emeril.
- At Sept-Iles, the sealed covers will be removed from the wagons by means of a portable crane and the wagons will be unloaded directly into the hydrometallurgical facility, avoiding need of a third concentrate storage.
- Once unloaded the covers will be returned onto each rail car by means of the same portable crane. The covers will be inspected at the Sept-Îles location in the event the seal is compromised.

The unit operating cost for concentrate transportation is summarized in Table 21-19.

Table 21-19: Concentrate Transportation Costs

Item	Cost Per Tonne of Concentrate	Cost Per Tonne Mill Feed
Dome and Road Costs	$ 39.89	$20.95
Truck Transportation, Crater Lake to Emeril	$168.73	$86.56
Rail Transportation, Emeril to Sept-Iles	$ 43.85	$22.49
Unloading System at Sept-Iles	$ 2.29	$ 1.17
Total Concentrate Transportation Cost	**$254.76**	**$110.23**

21.8.8 Storage Domes and Road Yearly Costs

A shipping contractor provided an expected unit cost for the equipment usage within a storage. The yearly winter road reconstruction cost is reflective of other projects in the WSP database, as is the yearly maintenance cost per kilometer. Table 21-20 shows how these factors were used to arrive a cost per tonne of concentrate for the winter road and dome storages.

Table 21-20: Road and Dome Component of Concentrate Transportation Costs

Item	Cost Measurement	Value
Dome Usage Cost Crater Lake	Cost per Concentrate Tonne	$2.25/t
	Concentrate Tonnes	218,589 t
	Yearly Crater Lake Dome Usage Cost	**$500,000**
Winter Road Reconstruction Cost	Distance	152 km
	Mobilization/Demobilization	$917,914
	Reconstruction Unit Cost	$37,500/km
	Number of Passing Lanes (Highway to Site)	16 each
	Length of Passing Lanes	0.15 km
	Passing Lanes Reconstruction Unit Cost	$37,500/km
	Yearly Winter Road Reconstruction cost	**$6,709,350**
Winter Road Maintenance Cost	Distance	152 km
	Maintenance Unit Cost	$5,000/km
	Yearly Winter Road Maintenance Cost	**$ 760,000**
Orma Lake Road Maintenance Cost	Maintenance Unit Cost	$5,000/km
	Orma Lake Road Distance	150 km
	Yearly Orma Lake Road Maintenance Cost	**$750,000**
Total Yearly Cost Dome and Road	**Yearly Cost to Move Concentrate**	**$8,719,350**
Unit Cost Dome and Road Component	**Yearly Cost Per Tonne Concentrate**	**$39.89**

21.8.9 Truck Transportation, Crater Lake to Emeril

Measured distances and assumed speeds establish that a round trip using a winter road 152km in length followed by use of Orma Lake Road for another 150km and ensuing travel on Highway 500 for another 199km will take about 20 hours. Two drivers were assigned to each truck to allow one to be sleeping while the other is driving. Cost factors from the mine truck workups were applied to then arrive at an hourly cost for each truck of $300 per operating hour. The total cost for each trip of $5,900 to move 35 tonnes of concentrate then provides the $168.73 per tonne concentrate for the truck transportation component of the overall cost of moving the yearly Hydrometallurgical feed. Table 21-21 provides the details pertaining to the cost determination for the truck transportation component of the concentrate delivery.

Table 21-21: Truck Component of Concentrate Transportation Costs

Item		Truck Capacity	35 tonnes
Mobilization / Demobilization	$ 917,914 per yr	6,245.4 trips at 19.69 hrs. per trip	$7.47/hr.
Truck Labour	2	drivers	$120.00/hr.
Truck Overhauls			$8.57/hr.
Truck Fuel and Lubes			$33.55/hr.
Truck Maintenance			$ 15.90/hr.
Truck Acquisitions	$ 950,000 each	50,000 hrs. life	$19.00/hr.
Truck Wear Materials		Tires	$9.01/hr.
Contractor Indirects	5%	assumed	$10.67/hr.
Contractor's Direct Supervision			$1.74/hr.
Contractor Overheads	10%	assumed	$22.59/hr.
Contractor Markups	10%	assumed	$2.80/hr.
Profits	15%	assumed	$37.70/hr.
Contractor Workers' Room and Board	215$/day	2 people	$21.84/hr.
Total Hourly Unit Cost per Truck			$310.85/hr.
Total Hourly Cost Rounded			$300.00/hr.
Truck loading	5	min	0.08 hrs.
Travel Section 1	33 kms	40 km/hr.	0.8 hrs.
Travel Section 2	28 kms	40 km/hr.	0.7 hrs.
Travel Section 3	39 kms	40 km/hr.	1.0 hrs.
Travel Section 4	25 kms	40 km/hr.	0.6 hrs.
Travel Section 5	27 kms	40 km/hr.	0.7 hrs.
Travel Orma Lake	150 kms	43 km/hr.	3.5 hrs.
Travel Hwy	199 kms	80 km/hr.	2.5 hrs.
Truck unload	3	min	0.05 hrs.
Cycle time	One Way	Total of Above	9.93 hrs.
Cycle time	Round Trip		19.69 hrs.
Cost Per Trip	19.69 hrs.	$300.00 / hr.	$5,905.52
Unit Cost Truck Component	**$5,905.52**	**35 tonnes**	**$168.73/t**

21.8.10 Rail Transportation Emeril to Sept-Îles

A shipping contractor provided an expected unit cost for the equipment usage within a storage. An expectation of the bulk shipping cost by rail was provided by the Sept-Îles port authority. Going forward, Imperial Mining Group will need to reach an agreement with QNS&L on the shipment of bulk concentrate using company owned wagons. The loading and unloading costs reflect equipment and labour provided by Imperial Mining Group and includes the handling of the covers and the sealing of the rail cars (wagons) with no involvement of QNS&L. Table 21-22 provides the details pertaining to the cost determination for the rail transportation component of the concentrate delivery.

Table 21-22: Rail Component of Concentrate Transportation Costs

Item	Cost Measurement	Value
Bulk Tonnes Shipped	Yearly Concentrate Shipped Tonnes	218,589 t
Dome Usage Cost Emeril	Cost per Concentrate Tonne	$2.25
	Yearly Emeril Dome Usage Cost	**$500,000**
Train loading cost	Train Loading Unit Cost per Bulk Tonne	$ 7.00/t
	Yearly Train Loading Cost at Emeril	**$1,530,125**
Rail Transport cost	Railroad Unit Cost per Bulk Tonne	$26.56/t
	Yearly Rail Shipping Cost Between Emeril and Sept-Iles	**$5,805,732**
Train unloading cost	Train Unloading Unit Cost per Bulk Tonne	$ 8.00/t
	Yearly Train Unloading Cost at Sept-Iles	**$1,748,714**
Total Yearly Cost Rail	**Yearly Cost to Move Concentrate**	**$9,584,571**
Unit Cost Rail Component	**Yearly Cost Per Tonne Concentrate**	**$43.85**

21.8.11 Unloading at Sept-Îles

The same storage usage cost was assigned to the unloading system at Sept-Iles. The cost is specific to the unloading system itself and does not allow for the labour and equipment assigned to unloading the rail cars. Table 21-23 provides the details pertaining to the cost determination for the unloading system at Sept-Îles component of the concentrate delivery.

Table 21-23: Unloading System at Sept-Îles Component of Concentrate Transportation Costs

Item	Cost Measurement	Value
Usage Cost, Unloading System / Silo at Sept-Iles	Yearly Concentrate Shipped Tonnes	218,589 t
	Unloading System / Silo Usage Cost per Bulk Tonne	$2.25/t
	Unloading System Usage Yearly Cost	**$500,000**
Unit Cost Unloading System Component	**Yearly Cost Per Tonne Concentrate**	**$2.29**

21.8.12 Hydrometallurgical Plant Operating Costs

Annual and unit process operating costs for the hydrometallurgical process plant were determined for an annual feed rate of 218,540 metric tons of mineral concentrate with annual production of 1,865 metric tons of mixed REE and 2,562 metric tons of Al-2%Sc Master Alloy.

The scandium oxide or scandia (Sc2O3) extracted in this process will be used to produce the Al-2%Sc Master Alloy, however, for the first 5 years 26% of the Sc2O3 in the feed will go to the production of purified Sc2O3 for sale as Sc2O3 to SOFA while 74% will go to the production of Al-2%Sc Master Alloy, thereafter 100% of the Sc2O3 will be used to produce the Master Alloy.

The estimated operating costs for the Hydrometallurgical plant are summarized in Table 21-24. The total operating costs were estimated to be $49,215,867 per year or $225.20 per ton of processed feed material. The sources of information to develop the operating costs include in-house databases and outside sources particularly for chemical reagents.

Table 21-24: Hydrometallurgical Plant Operating Costs

Operating Cost by Category	$/y	$/t	% of Costs
Manpower	7,120,598	32.58	14.5%
Utilities	2,569,630	11.76	5.2%
Chemicals	35,218,809	161.15	71.6%
Laboratory	512,164	2.34	1.0%
Maintenance	3,529,129	16.15	7.2%
Mobile Equipment	265,536	1.22	0.5%
Total Process Operating Cost	**49,215,867**	**225.20**	**100.0%**

Manpower

The manpower requirements will be 61 people as listed on Table 21-25. The workforce includes the area of mill operation including administration, operations, maint enance, metallurgists, technicians, and assayers. The labour rates and benefits were based on the rates for similar jobs. Overheads of 25% of the base salary, was considered. The total estimate is $7,120,598 per year, which equates to $16.71 per tonne of material processed.

Table 21-25: Hydrometallurgical Plant Workforce Costs

Description	Employees Qty	Total cost $/y	Unit cost $/t
Operations	47	5 166 757	12.13
Maintenance	14	1 953 841	4.59
Total Manpower	**61**	**7 120 598**	**16.71**

Utilities

The electric power costs were calculated using the total estimated load of the process plant operation. For the first two years, the total power demand for the plant was estimated at 3,267 kW, which is equal to 19.75 MWh per year. After year two with the production of Al-2%Sc Master Alloy, the total power demand for the plant was estimated at 6,826 kW, which is equal to 41.26 MWh per year. The electrical power cost was estimated from the Hydro-Québec Rate and averaged at $0.05/kWh.

Propane costs were based on estimated consumption for the boiler and plant HVAC, at an average $0.50/L.

Diesel costs were based on estimated consumption for the calcination kiln, at an average $1.92/L.

Total utilities operating costs are listed on Table 21-26.

Table 21-26: Utilities Costs

Description	$/y	$/t
Electrical Power	2,062,923	4.84
Propane	500,000	1.17
Diesel	6,707	0.02
Total Utilities Operating Cost	**2,569,630**	**6.03**

Chemicals

Chemical reagents quantities were estimated from process design criteria, while reagent costs were obtained from suppliers and internal database. Table 21-27 summarizes the chemical reagents consumption and cost.

Table 21-27: Chemicals Costs

Description	Consumption	Cost	
	t/y	$/y	$/t
Chemical Consumption			
Sodium hydroxide - NaOH	5,225	3 918 750	17.93
Quick lime - CaO	40,925	11 336 225	51.87
Hydrochloric acid - HCl	7,266	3 923 478	17.95
Iron filings - Fe	6,105	1 220 960	5.59
Di (2-Ethylhexyl) Phosphoric Acid - D2EHPA	1.40	16 800	0.08
Shellsol D90	5.50	8 250	0.04
Sulfuric acid - H_2SO_4	27,408	5 481 660	25.08
Hydrogen peroxide - H_2O_2	0.70	371	0.00
Amine	0.012	120	0.00
Oxalic acid - $C_2H_2O_4$	147	308 280	1.41
Limestone - $CaCO_3$	42,668	5 260 915	24.07
Fly Ash	4,850	194 000	0.89
Alumina - Al_2O_3	4,745	2 847 000	13.03
Aluminium Fluoride - AlF_3	51	89 600	0.41
Cryolite - $NaAl_3F_6$	128	102 400	0.47
Carbon Anode - C	850	510 000	2.33
Total Chemicals Operating Costs		**35 218 809**	**161.15**

Maintenance

Equipment maintenance material costs have been established as a function of the relative equipment capital cost. The factors have been determined from benchmarking studies. The estimated maintenance material costs cover mechanical spares and wearing parts.

Total equipment cost is approximately $96M. Spare parts and miscellaneous costs have been estimated at 3.5% of the total equipment cost. Overall, maintenance spares and supplies represent $3.53M per year.

Mobile Equipment

A mobile equipment list has been estimated, including the fuel consumption, the availability of the equipment, their utilization and the costs for consumables and parts. Table 21-28 summaries the mobile equipment requirements.

Table 21-28: Mobile Equipment Costs

Description	# of units	Fuel Consumption L/yr	Fuel Cost $/y	Maintenance Cost $/y	Total Cost $/y
4WD Double Cab Pickup (4 passengers)	4	70,080	72,230	23,838	96,068
8t Twincab 4WD Hiab Truck	1	8,760	8,218	8,642	16,860
3t Tip Truck	1	8,760	8,218	8,642	16,860
Plant Forklift (4t)	2	13,140	12,328	10,175	22,502
Manlift / Scissor Lift	1	7,300	6,849	9,646	16,495
Front-end Loader	1	52,560	49,311	8,642	57,953
Portable Pumps Generator - 200 kVA	1	27,375	25,683	1,982	27,665
Generator - 5 kVA	1	1,600	1,501	1,321	2,822
Portable Compressor	1	3,200	3,002	1,321	4,324
Diesel Welder	1	1,280	1,201	793	1,994
Fusion Butt Welder 90/315	1	1,280	1,201	793	1,994
Total Mobile Equipment Operating Cost			**189,742**	**75,795**	**265,536**

21.8.13 Surface Mobile, Sept-Îles Operating Costs

The Sept-Îles Surface Mobile Equipment operating costs are based on an estimate of operating hours for the listed equipment, as depicted in Table 21-29. The resultant CA$M1.0 per year equates to $2.35 per tonne mill feed.

Table 21-29: Crater Lake Surface Mobile Equipment Annual Operating Cost Determination

Description	Units	Operating Hours per Year	Maintenance Cost per Operating Hour	Fuel liters per Hour	Diesel Fuel Liters per Year	Fuel Cost per Year at $1.25/l	Maintenance Cost per Year (including tires)	Total Fuel and Maintenance Cost per Year	Sustaining Cost per Year	Total Yearly Operating Cost
Caterpillar 988 XE Loader- Surface loader (6-7 m3) with rock bucket	1	3000	$82.00	40	120,000	$150,000	$246,000	$396,000	$15,180	$411,180
Caterpillar 950 Loader - Surface service loader (3 m3) with forks, snow bucket & snowplow	2	3000	$53.00	14	84,000	$105,000	$318,000	$423,000	$30,360	$453,360
P6000 Lift truck - Fork Lift (6,000 / 8,000 lbs)	2	1000	$ 11.53	0	0	$0	$23,060	$23,060	$10,120	$33,180
Skid loader 242D3 CCE	1	1000	$23.00	10	10,000	$12,500	$23,000	$35,500	$5,060	$40,560
14M grader - Surface (14 ft blade with side wing)	0	2500	$60.00	16	0	$0	$0	$0	$0	$0
Pickup truck, 3/4 tonnes	2	1000	$ 9.60	10	20,000	$25,000	$19,200	$44,200	$10,120	$54,320
Total					234,000	$340,000	$629,260	$921,760	$70,840	$992,600

21.8.14 Water Management, Sept-Îles Operating Costs

The Sept-Îles yearly Water Management Costs are derived, as shown in Table 21-30. The labour cost determinations shown as a cost in Table 21-30 are itemized in Table 21-31 and Table 21-32. The resultant CA$M4.0 to CA$M4.6 per year for water management at the Sept-Îles site equates to $10.27 per tonne of mill feed.

Table 21-30: Sept-Îles Water Management Annual Operating Cost Determination

Item	Quantity	Unit of Measure	Unit Cost	Year -2	Year -1	Years 1 to 14	Years 15 to 25
Sept-Iles							
Water Management				Being Constructed		Operating Expense	Operating Expense
WTP operation costs	185	m3/h	$ 0.10			$ 162,060	$ 162,060
Ditch maintenance	Ditch length increases from 2140m to 3745m year 15	l. m.	$ 350.00			$749,000	$1,310,750
Pond Maintenance	15% of SI Pond CAPEX	lot	$127,500			$ 127,500	$ 127,500
Waste Management							
Loader - Tailings loading into hauler	325,315	Wet tonnes	$ 0.23/t			$73,196	$73,196
Hauler - Tailings transport to TSF	2,801	hours	$103.28/hr			$289,236	$289,236
Dozer - Tailings placement on TSF	325,315	Wet tonnes	$ 0.23/t			$73,196	$73,196
Compactor - Tailings compacting	325,315	Wet tonnes	$ 0.23/t			$73,196	$73,196
Labour							
Waste and Water Management	See Table 2131	lot	$1,157,000			$1,157,000	$1,157,000
Site Management	See Table 2132	lot	$933,125			$ 933,125	$ 933,125
Engineering, Monitoring, and Reporting	10% of above costs		$ 363,751			$ 363,751	$ 419,926
Total Sept-Iles						**$4,001,259**	**$4,619,184**

Table 21-31 and Table 21-32 depict the surface labour requirements on site at any given time. Sept-Îles does not need a rotation, such as14 days in and 14 days out, for the surface labour payroll and thereby the yearly labour cost shown in the tables represents annual payroll costs.

Table 21-31: Sept-Îles Waste and Water Management Labour Cost Determination

Occupation	Function	Number of Operators	Shift	Base Salary	Benefits Ratio	Yearly Each	Yearly Total
Superintendent		1	Day	$124,800	1.25	$156,000	**$156,000**
Engineer – Tailings Management Facility		1	Day	$104,000	1.25	$130,000	**$130,000**
Engineer – Water		1	Day	$104,000	1.25	$130,000	**$130,000**
Supervisor		1	Day	$83,200	1.25	$104,000	**$104,000**
Operator / Repairer	Machine Operator - Dozer	2	Day	$72,800	1.25	$91,000	**$182,000**
Operator / Repairer	Machine Operator - Hauler	1	Day	$72,800	1.25	$91,000	**$91,000**
Operator / Repairer	Machine Operator - Loader	1	Day	$72,800	1.25	$91,000	**$91,000**
Operator / Repairer	Maintenance work	1	Day	$72,800	1.25	$91,000	**$91,000**
Operator / Repairer	Pumps, conduits, Utilities	2	Day	$72,800	1.25	$91,000	**$182,000**
Daily	Inspections	1	Day	$41,600	1.25	$52,000	**$52,000**
Technician	WTP maintenance and operation	1	Day and Night	$72,800	1.25	$91,000	**$91,000**
TOTAL ON SITE		**13**					**$1,157,000**

Table 21-32: Sept-Îles Site Management Labour Cost Determination

Camp and Site Management	Number of Operators	Shift	Base Salary	Benefits Ratio	Yearly Each	Yearly Total
Site						
Site manager	1	Day	$80,000	1.25	$100,000	**$100,000**
Security guard	2	Day + night	$65,000	1.25	$81,250	**$162,500**
Loader / grader Operator	2	Day	$75,000	1.25	$93,750	**$187,500**
Journey man	1	Day	$62,500	1.25	$78,125	**$78,125**
Warehouse management	3	2 Day + 1 Night	$68,000	1.25	$85,000	**$255,000**
Offices and warehouse housekeeping	1	Day	$45,000	1.25	$56,250	**$56,250**
Mechanics	1	Day	$75,000	1.25	$93,750	**$93,750**
TOTAL ON SITE	**11**					**$933,125**

21.8.15 Pre-Production Period Operating Costs to Project Capital

The costs associated with the mine and mill prior to the hydrometallurgical facility commencing production are identified as pre-production costs. Any operating costs in that period are credited to the operating costs and debited against the project capital costs. Accordingly, $67,906,652 is removed from the operating costs as a capital expense. This approximates $6.42 per tonne mill feed as a credit against the life of operations plan (Table 21-33).

Table 21-33: Pre-Production Costs to Project Capital

Item	Year -2	Year -1	Total Period
MINE CRATER LAKE	$ 8,670,950	$ 9,764,283	$ 18,435,233
MILL CRATER LAKE		$ 17,355,760	$ 17,355,760
POWER PLANT CRATER LAKE		$ 1,440,600	$ 1,440,600
SURFACE MOBILE EQUIPMENT CRATER LAKE		$ 1,100,610	$ 1,100,610
WATER MANAGEMENT CRATER LAKE		$ 7,483,499	$ 7,483,499
LODGING CRATER LAKE		$ 5,538,000	$ 5,538,000
TRANSPORTATION TO EMERIL		$ 9,220,643	$ 9,220,643
Dome and Road Costs - Crater Lake to Churchill Falls		$ 7,332,306	$ 7,332,306
Total Pre-production Costs to Project Capital	**$ 8,670,950**	**$ 59,235,702**	**$ 67,906,652**

21.8.16 General and Administrative Costs

Imperial Mining Group provided their estimation of the yearly Corporate General and Administrative costs for the PEA.

Table 21-34 depicts the breakdown of those costs. The CA$M3.4 per year for G&A equates to $7.93 per tonne of mill feed. The G&A is reduced in the later years of the LoOP with closure of the mine and mill while the Hydrometallurgical facility continues to operate resulting in the overall LoOP G&A amount being slightly less at $7.90/t.

Table 21-34: General and Administrative Costs, Yearly

Corporate G&A	Yearly Costs	Cost Per Tonne Mill Feed
Plant Management	$17,860	$ 0.04
Finance	$716,760	$ 1.68
Insurance	$865,850	$ 2.03
Legal	$83,740	$0.20
HR (training + entertainment)	$31,810	$0.07
Corporate	$1,374,290	$ 3.23
Environmental	$274,640	$ 0.64
Product shipping expense	$12,240	$ 0.03
Total Corporate G&A	**$3,377,190**	**$7.93**

These General and Administrative costs (Table 21-34) were used to arrive at the Owner's Costs during the pre-production period. They would include management, accounting, and health and safety labour necessary for the detailed engineering and construction period. It also includes such services as air and ground transportation, electricity, LNG, camp services, water management, site security, road maintenance, general liability and construction insurances, and purchase of service equipment.

21.8.17 Royalties Operating Costs

A one time $2 million royalty buyout will be exercised at the commencement of commercial production as an operating expense and over the life of operations plan approximates an overall $0.19 per tonne mill feed cost.

21.8.18 Corporate G&A To Project Capital

The first two years of the Project's General and Administrative costs (Table 21-34) represent the Owner's Costs during the pre-production period before commencement of commercial production. They would include management, accounting, and health and safety labour necessary for the detailed engineering and construction period. It also includes such services as air and ground transportation, electricity, LNG, camp services, water management, site security, road maintenance, general liability and construction insurances, and purchase of service equipment. Accordingly, $6,754,380 is removed from the operating costs as a capital expense. This approximates $0.64 per tonne mill feed as a credit against the life of operations plan.

22 ECONOMIC ANALYSIS

22.1 Introduction

An engineering economic model was prepared for the Project to estimate annual cashflows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the tasks performed by WSP.

The Project includes: at the mine site, an open pit mine, a processing plant for the recovery of a concentrate containing scandium oxide and other rare earths, surface infrastructure to support the mine and mill operations (maintenance and office facilities), water management features, and a tailings storage facility; at the hydrometallurgical site, a facility for the production of a Master Alloy, Al-Sc2% product and of a saleable Scandium Oxide and of a saleable rare earth Mixed REE+Y product with a tailings storage facility and other water management features; and the logistics features between the two sites for the transportation of the concentrate along a winter road to a highway leading to a storage facility located at a railroad siding for year round transportation by rail to the hydrometallurgical site. It is noted that the mine and mill operate 6 months of the year during the temperate season while the hydrometallurgical facility operates year-round. The full year's milled concentrate is stored at the mine site until the end of the year when a winter road can be made available for transportation of the concentrate over a four (4) month winter period to a railroad siding estimated to be 501 kilometers distance from the mine site with a round trip time of about 19 hours and 41 minutes. From the siding the concentrate is moved by rail another 385 km to the hydrometallurgical plant for creation of the saleable products.

The Project's economic analysis inputs and resultant indicators are summarized in Table 22-1. The Project indicates an after-tax cashflow of $4,536 million, after-tax NPV (10%) of $1,121 M and after-tax IRR of 25.5%. The project is most sensitive to Master Alloy (Al-Sc2%) commodity prices and currency exchange rates.

Table 22-1: Summary of Project Economics, Base Case Scenario

Item	Units	Value
Production		
Project life (from start of construction to closure)	years	25
Mine life	years	26
Total Tonnes Potential Mill Feed (PMF)	M t	10.6
Scandium Oxide (Sc2O3) Grade	g/t	268.26
Dysprosium Oxide (Dy2O3) Grade	g/t	62.42
Lanthanum Oxide (La2O3) Grade	g/t	583.01
Neodymium Oxide (Nd2O3) Grade	g/t	566.99
Praseodymium Oxide (Pr2O3) Grade	g/t	152.76
Terbium Oxide (Tb4O7) Grade	g/t	11.13
Sc2O3 Sold	t	110
Tonnes Al – Sc2% Sold	t	57,298
Dy2O3 Sold	t	377
La2O3 Sold	t	3,521
Nd2O3 Sold	t	3,425
Pr2O3 Sold	t	923
Tb4O7 Sold	t	67
Commodity Prices		
Selling Price for Scandium Oxide (Sc2O3)	US$/kg	1,500.00
Selling Price for Master Alloy (Al -Sc2%)	US$/kg	204.00
Selling Price for Dysprosium Oxide (Dy2O3)	US$/kg	128.40
Selling Price for Lanthanum Oxide (La2O3)	US$/kg	1.50
Selling Price for Neodymium Oxide (Nd2O3)	US$/kg	49.20
Selling Price for Praseodymium Oxide (Pr2O3)	US$/kg	49.20
Selling Price for Terbium Oxide (Tb4O7)	US$/kg	584.40
Project Costs		
Mine	$/t PMF	23.49
Mill	$/t PMF	40.73
Hydromet	$/t PMF	113.29
G&A	$/t PMF	43.63
Concentrate Shipment	$/t PMF	131.18
Project Economics		
Gross Revenue	$M	15,200
Total Selling Cost Estimate	$M	86
Total Operating Cost Estimate	$M	3,727
Total Sustaining Capital Cost Estimate	$M	68
Total Capital Cost Estimate	$M	871
Total Other Costs Estimate	$M	139
Taxes	$M	4,050
Pre-Tax Cashflow	$M	10,309
After-Tax Cashflow	$M	6,259
Discount Rate*	%	10.00%
Pre-Tax Net Present Value @ 10%	$M	2,971
Pre-Tax Internal Rate of Return	%	42.9%
Pre-Tax Payback Period	years	2.5
After-Tax Net Present Value @ 10%	$M	1,721
After-Tax Internal Rate of Return	%	32.8%
After-Tax payback period	years	3.0

Note: *Discounting commences with Commercial Production.

22.2 Cautionary Statement

The results of the Economic Analysis are based on forward looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this Item include, but are not limited to, statements with respect to:

- Future prices of Master Alloy (AL-Sc2%), Scandium Oxide (Sc_2O_3), Dysprosium Oxide (Dy_2O_3), Lanthanum Oxide (La_2O_3), Neodymium Oxide (Nd_2O_3), Praseodymium Oxide (Pr_2O_3) and Terbium Oxide (Tb_4O_7).
- Currency exchange rate fluctuations.
- Estimated costs and timing of Capital and Operating expenditures.

This PEA is preliminary in nature. There are no “Measured Resources”. In addition to Indicated Resources, the mine plan presented in this study includes Inferred mineral resources. As such, there is no certainty that this PEA will be realized.

22.3 Principal Assumptions

The cashflow estimate includes only revenue, costs, taxes, and other factors applicable to the Project. Corporate obligations, financing costs, and taxes at the corporate level are excluded.

The model was prepared from mining schedules estimated on an annual basis. The cashflow model was based on the following:

- All costs are reported in Canadian dollars (CAN$) and referenced as '$', unless otherwise stated.
- One hundred percent (100%) equity ownership.
- No cost escalation beyond 2025.
- No provision for effects of inflation.
- Constant year 2022 dollars analysis.
- The economic analysis consists of the technical assumptions outlined in the previous Items, together with the economic assumptions and estimated Capital and Operating costs described in Item 21.
- The economic analysis is based on Imperial Mining Group's preferred scenario of selling three products:
- A Master Alloy, Aluminum and 2% Scandium product throughout all years of Hydrometallurgical facility operation with the first five (5) years consuming 74% of the scandium oxide in the mill concentrate and the ensuing years 100% of the scandium oxide.
- A Scandium Oxide product for the first five (5) years of Hydrometallurgical facility operation, representing 26% of the scandium oxide in the mill concentrate.
- A rare earth REE+Y product throughout all of the years of Hydrometallurgical facility operation
- Annual gross revenue is determined by applying estimated metal prices with payable metal assumption to the annual recovered metal estimated for each operating year.
- As discussed in Item 19, a constant commodity pricing of 1,500 US$/kg. For Scandium Oxide was used in the economic analysis.
- A constant exchange rate assumption of 1 US$: 1.25 CAN$ (1 CAN$: 0.80 US$) was used in the economic analysis.
- Exploration costs whether to prove up the resource within the pit design or to identify additional resources outside of the pit design are deemed outside of the Project.
- Any additional Project study costs to better define the project and / or arrive at a more definitive implementation plan with a control budget and detailed work basis schedule have not been included in the analysis.
- Land acquisition costs have been excluded.
- Independent reclamation costs have been included in the economic analysis at the end of life for each of the mine / mill location and of the hydrometallurgical facility location.
- Annual cost of a bond considering a progressive rehabilitation schedule has been excluded.

22.4 Taxes and Royalties

22.4.1 Taxes

The Project has been evaluated on an after-tax basis. It must be noted that there are many potential complex factors that affect the taxation of a mining project. The taxes, depletion, and depreciation calculations in this PEA economic analysis are simplified and only intended to give a general indication of the potential tax implications; like the rest of the PEA economics, they are only preliminary.

The Project will be subject to the following taxes as they relate to the Project:

- A federal income tax rate of 15%.
- A provincial corporate income tax rate ranging from 11.8% (in 2017) to 11.5% (in 2029 and thereafter).
- A provincial mining tax rate from 16% to 28% depending on the profit margin of the year.

Processing Allowance:

- A company is entitled to deduct a processing allowance in the calculation of its mining profit. Basically, this deduction corresponds to 20% of the original value of an asset used in the ore processing when it involves a hydrometallurgical facility.

Depreciation Allowance:

- A company may claim a depreciation allowance on an asset used in the mining operations at the declining rate of 30%.

22.4.2 Royalties

The Project royalties are described in Item 4. For this PEA, a $2M royalty buyout has been identified as being applicable.

22.5 Economic Results, Base Case

The results are derived from the life-of-mine schedule presented in Item16, from the recovery method discussed in Item 17, and from the Capital and Operating costs presented in Item 21. Table 22-2 summarizes the cost inputs for the economic analysis.

Table 22-2: Summary of Cost Inputs

Cost Item / Description	Pre-Production	Production / Sustaining	Total	$/t mined	$/t product
	M $	M $	M $		
Mining crater lake	18.4	230.1	249	23.49	3,782
Milling crater lake	17.4	413.5	431	40.73	6,556
Power plant use Crater Lake	1.4	34.6	36	3.40	548
Surface mobile equipment use Crater Lake	1.1	26.4	28	2.60	419
Water management Crater Lake	7.5	184.9	192	18.19	2,928
Lodging Crater Lake	5.5	134.6	140	13.25	2,132
Truck transport concentrate to Emeril	9.2	906.4	916	86.56	13,932
Concentrate storage and winter road maintenance	7.3	214.3	222	20.95	3,372
Transport by rail Emeril to Sept Isles	0.0	237.9	238	22.49	3,620
Unload concentrate at Sept Isles	0.0	12.4	12	1.17	189
Hydromet facility Sc2O3 Sept Isles	0.0	58.5	59	5.53	890
Hydromet facility Al - Sc2% Sept Isles	0.0	1,139.8	1140	107.75	17,343
Surface mobile equipment use Sept Isles	0.0	24.8	25	2.35	378
Water management Sept Isles	0.0	108.6	109	10.27	1,652
Pre-production operating costs	(67.9)	0.0	-68	-6.42	-1,033
1 - Total Operating Costs	**0.0**	**3,726.8**	**3727**	**352.32**	**56,707**
(Mining + Processing + GA + Transport)					
Selling general & administrative costs	0.0	83.6	84	7.9	1,272
Royalties	0.0	2.0	2	0.2	30
2 - Subtotal Costs (Operating Costs + Selling Costs + Royalties)	**0.0**	**3,812.4**	**3812**	**360.41**	**58,009**
Crater lake, purchases mining equipment and licenses	0.0	24.5	25	*2.32*	373
Crater lake, processing equipment replacements	0.0	8.2	8	*0.78*	125
Crater lake, tailing storage facility dyke	0.0	23.4	23	*2.21*	356
Crater lake, windmill year 5	0.0	1.0	1	*0.10*	16
Crater lake, added ditches & pond	0.0	2.8	3	0.27	43
Sept Isles, added ditches	0.0	0.8	1	0.07	11
Sept Isles, tailing storage facility dyke	0.0	7.6	8	0.71	115
Crater lake, restoration cost year 26	0.0	74.4	74	7.03	1,132
Sept isles, restoration cost year 27	0.0	65.1	65	6.15	990
3 - Subtotal Sustaining Capital	**0.0**	**207.7**	**208**	**19.63**	**3,160**
3 - Subtotal Costs	**0.0**	**4,020.1**	**4020**	**380.05**	**61,169**
(Operating Costs + Sustaining Capital + Selling Costs)					
Capital Cost Estimate					
Mine - Equipment	13.7	0.0	14	1.29	208
Mill Processing Plant Construction	63.7	0.0	64	6.02	969

Cost Item / Description	**Pre-Production**	**Production / Sustaining**	**Total**	**$/t mined**	**$/t product**
	M $	**M $**	**M $**		
Hydromet Facility	160.1	0.0	160	15.13	2,436
Power & electrical	14.1	0.0	14	1.33	215
Crater Lake and Sept Isles site infrastructure	113.5	0.0	113	10.73	1,727
TSF and water management	69.3	0.0	69	6.55	1,054
Initial winter road + Orma lake road	46.6	0.0	47	4.40	709
Offsite concentrate handling	27.5	0.0	28	2.60	419
Camp (200-person capacity)	26.5	0.0	26	2.50	403
Pre-Production Mining Licences	0.2	0.0	0.2	0.02	3.35
Pre-production (see operating costs above)	67.9	0.0	68	6.42	1,033
Indirect Estimate	108.8	0.0	109	10.28	1,655
Contingency	159.2	0.0	159	15.05	2,422
Total Capital Costs with Contingency	**870.9**	**0.0**	**871**	**82.33**	**13,252**
4 - All-in Costs, Pre-Tax*	**870.9**	**4,020.1**	**4,891**	**462.38**	**74,421**
(Operating Costs + Selling Costs + Royalties + Sustaining Capital + Total Capital; excl. Tax)					
Tax Estimation	0.0	4,050.4	4,050	382.91	61,630
5 - All-in Costs*	**870.9**	**8,070.5**	**8,941**	**845.29**	**136,052**
(All estimated costs, incl. Tax)					
6 – Gross Revenue*	**0**	**15,200.4**	**15,200**	**1436.99**	**231,288**
7 – Surplus*	**-870.9**	**7,130.0**	**6,259**	**591.71**	**95,237**

Note: *Are non-GAAP financial performance measures with no standardized definition.

Figure 22-1 depicts the cashflow.

Figure 22-1:Cashflow Model Results, Base Case

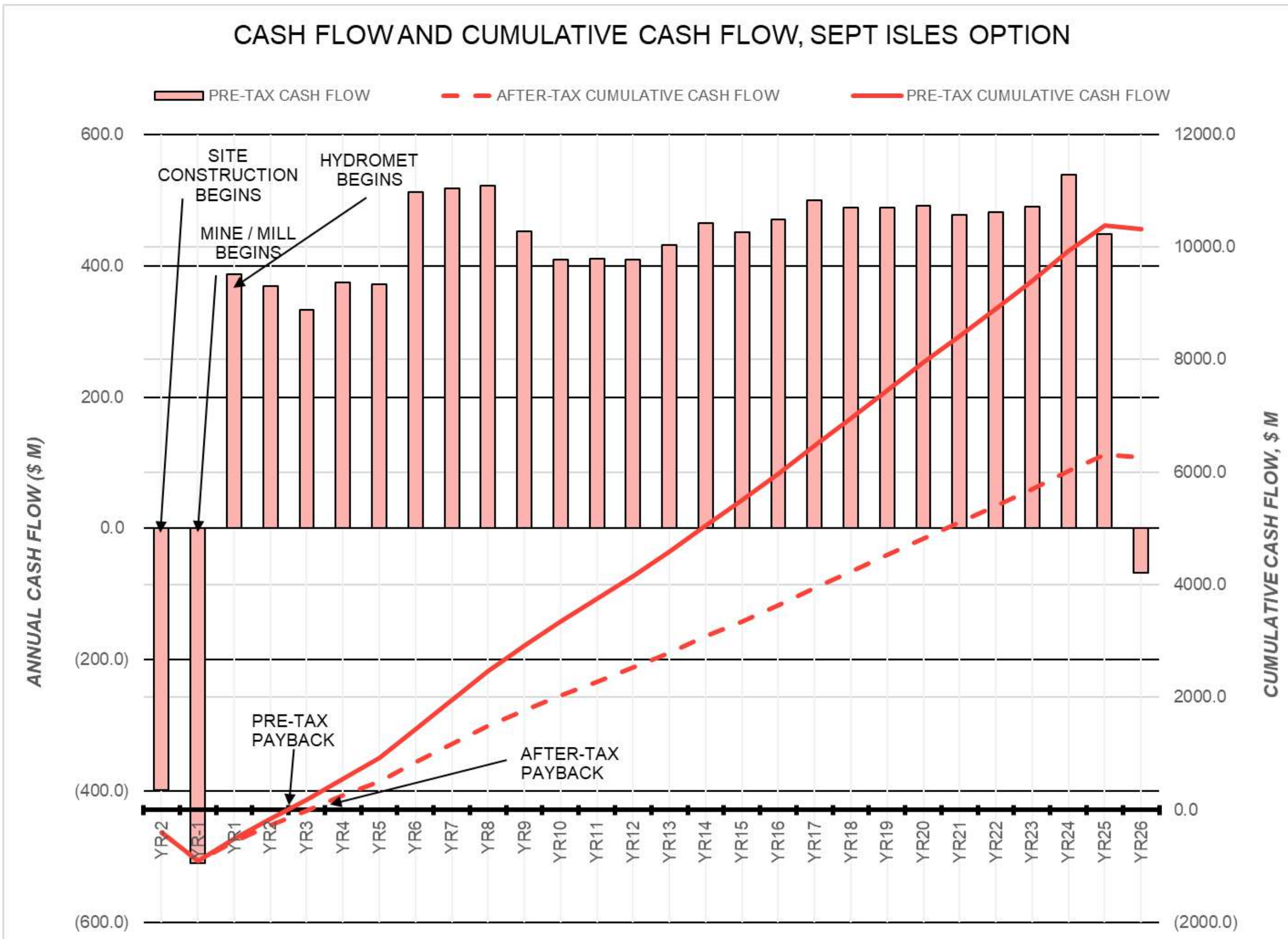


Table 22-3 presents the estimated cashflow model.

Table 22-3: Cashflow Model, Base Case

			PRE-PRODUCTION				PRODUCTION																									
	Total	**Unit**	**YR-4**	**YR-3**	**YR-2**	**YR-1**	**YR1**	**YR2**	**YR3**	**YR4**	**YR5**	**YR6**	**YR7**	**YR8**	**YR9**	**YR10**	**YR11**	**YR12**	**YR13**	**YR14**	**YR15**	**YR16**	**YR17**	**YR18**	**YR19**	**YR20**	**YR21**	**YR22**	**YR23**	**YR24**	**YR25**	**YR26**
MILL FEED	10,578	k t				426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	426	352		
Sc2O3 Grade	268	g/t				283	273	254	276	277	281	284	286	260	240	240	238	251	262	262	263	277	271	272	272	266	267	271	284	302		
Dy2O3 Grade	62	g/t				64	63	60	64	65	67	65	65	62	59	58	60	61	63	62	63	64	64	63	63	61	62	60	60	62		
La2O3 Grade	583	g/t				600	585	583	591	603	616	596	597	575	571	582	580	593	589	578	587	589	591	589	576	564	572	550	550	565		
Nd2O3 Grade	567	g/t				567	568	548	574	583	601	589	589	568	545	544	547	561	570	561	567	579	582	575	567	555	559	549	551	576		
Pr2O3 Grade	153	g/t				150	153	151	154	158	165	158	158	152	147	147	151	152	153	150	152	155	155	153	152	149	151	148	150	154		
Tb4O7 Grade	11	g/t				11	11	11	11	12	12	12	12	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11		
Products Sold																																
Sc2O3 to SOFC	110	t					23	22	20	22	22																					
Al - Sc2%	57,298	t					1,902	1,835	1,707	1,859	1,866	2,555	2,584	2,598	2,362	2,178	2,182	2,163	2,280	2,383	2,381	2,394	2,513	2,460	2,475	2,473	2,417	2,430	2,461	2,577	2,264	-
Dy2O3	377	t					16	15	15	16	16	16	16	16	15	14	14	14	15	15	15	15	16	16	15	15	15	15	15	15	12	-
La2O3	3,521	t					146	142	142	144	147	150	145	145	140	139	142	141	144	143	141	143	143	144	143	140	137	139	134	134	113	-
Nd2O3	3,425	t					138	138	133	140	142	146	143	143	138	133	132	133	137	139	136	138	141	142	140	138	135	136	134	134	116	-
Pr2O3	923	t					36	37	37	38	39	40	39	38	37	36	36	37	37	37	37	37	38	38	37	37	36	37	36	36	31	-
Tb4O7	67	t					3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	-
Gross Revenue	15,200	M$			0	0	543	525	489	532	534	668	675	678	618	570	571	566	597	623	622	626	657	643	647	646	631	635	643	672	590	0
Selling Costs	86	M$			0.0	0.0	5.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	3.4	1.7	0.8
Operating Costs	3,727	M$			0.0	0.0	150.9	151.8	152.5	152.5	151.7	152.4	152.5	152.1	152.0	155.1	153.7	153.7	152.8	151.1	152.3	151.2	151.6	151.6	150.6	150.6	150.5	150.1	150.0	144.8	86.9	1.8
Sustaining Capital	68	M$			0.0	0.0	0.2	0.2	0.4	1.6	7.3	0.6	1.7	0.6	8.7	3.5	3.0	0.6	8.9	4.2	16.8	0.6	1.6	0.6	4.4	0.6	0.6	0.6	0.6	0.5	0.0	0.0
Capital Costs	871	M$			398.8	472.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Other Costs	139	M$			0.0	37.7	0.2	0.2	0.0	-0.2	0.2	0.0	-0.1	0.0	0.8	-0.4	0.0	-0.2	-0.4	0.3	-0.3	0.1	0.0	-0.3	0.0	0.0	-0.1	0.0	-1.3	-14.5	53.1	64.6

			PRE-PRODUCTION				PRODUCTION																									
	Total	Unit	YR-4	YR-3	YR-2	YR-1	YR1	YR2	YR3	YR4	YR5	YR6	YR7	YR8	YR9	YR10	YR11	YR12	YR13	YR14	YR15	YR16	YR17	YR18	YR19	YR20	YR21	YR22	YR23	YR24	YR25	YR26
Cashflows																																
Pre-tax cashflow	10,309	M$			(399)	(510)	387	369	332	374	371	512	517	522	453	409	411	409	432	464	450	471	500	488	488	491	477	481	490	538	448	(67)
Cumulative Pre-Tax Cashflow		M$			(399)	(909)	(522)	(153)	179	554	925	1,436	1,954	2,476	2,929	3,338	3,749	4,158	4,590	5,054	5,504	5,975	6,475	6,963	7,451	7,943	8,420	8,901	9,390	9,928	10,377	10,309
After-tax cashflow	4,536	M$			(399)	(510)	347	296	257	268	254	329	326	325	279	255	255	253	262	282	269	287	303	296	294	297	288	290	296	328	281	(48)
Cumulative After-Tax Cashflow		M$			(399)	(909)	(562)	(266)	(9)	259	513	842	1,168	1,493	1,772	2,027	2,282	2,535	2,797	3,079	3,348	3,635	3,938	4,234	4,527	4,824	5,112	5,403	5,698	6,026	6,307	6,259

NPV calculation was performed to discount all future years of production to the first year of production. The Internal Rate of Return is defined as the rate at which the net present value of the cashflow equals zero. The payback period of the Project is defined as the point when the cumulative cashflow becomes positive.

Table 22-4 summarizes the economic indicators, both pre-tax and after-tax for the estimated cashflow model presented in Table 22-3.

Table 22-4: Economic Indicators, Base Case

Economic Indicators	Units	Pre-Tax	After Tax
Payback Period (From Start of Production)	years	2.5	3.0
Internal Rate of Return, IRR	%	42.9%	32.8%
Net Present Value @ 5%	M CAN$	$5,265	$3,150
Net Present Value @ 7%	M CAN$	$4,145	$2,455
Net Present Value @ 10%	M CAN$	$2,971	$1,721
Net Present Value @ 11%	M CAN$	$2,675	$1,535
Net Present Value @ 12%	M CAN$	$2,413	$1,370
Net Present Value @ 15%	M CAN$	$1,794	$977

22.6 Sensitivity Analysis, Pre-Tax Basis

The pre-tax cashflow was evaluated for sensitivity to commodity prices, operating costs, capital expenditures and US to Canadian dollar exchange rate. All sensitivities were analyzed as mutually exclusive variations.

The Project's pre-tax NPV was most sensitive to the factors impacting revenue, that is, commodity pricing and currency exchange rate. The Project's pre-tax NPV was equally sensitive to operating and capital expenditures. Figure 22-2, Figure 22-3 and Figure 22-4, and Table 22-5 to Table 22-8 summarize the pre-tax sensitivity results.

Figure 22-2: Pre-Tax Sensitivity Analysis on NPV 10%

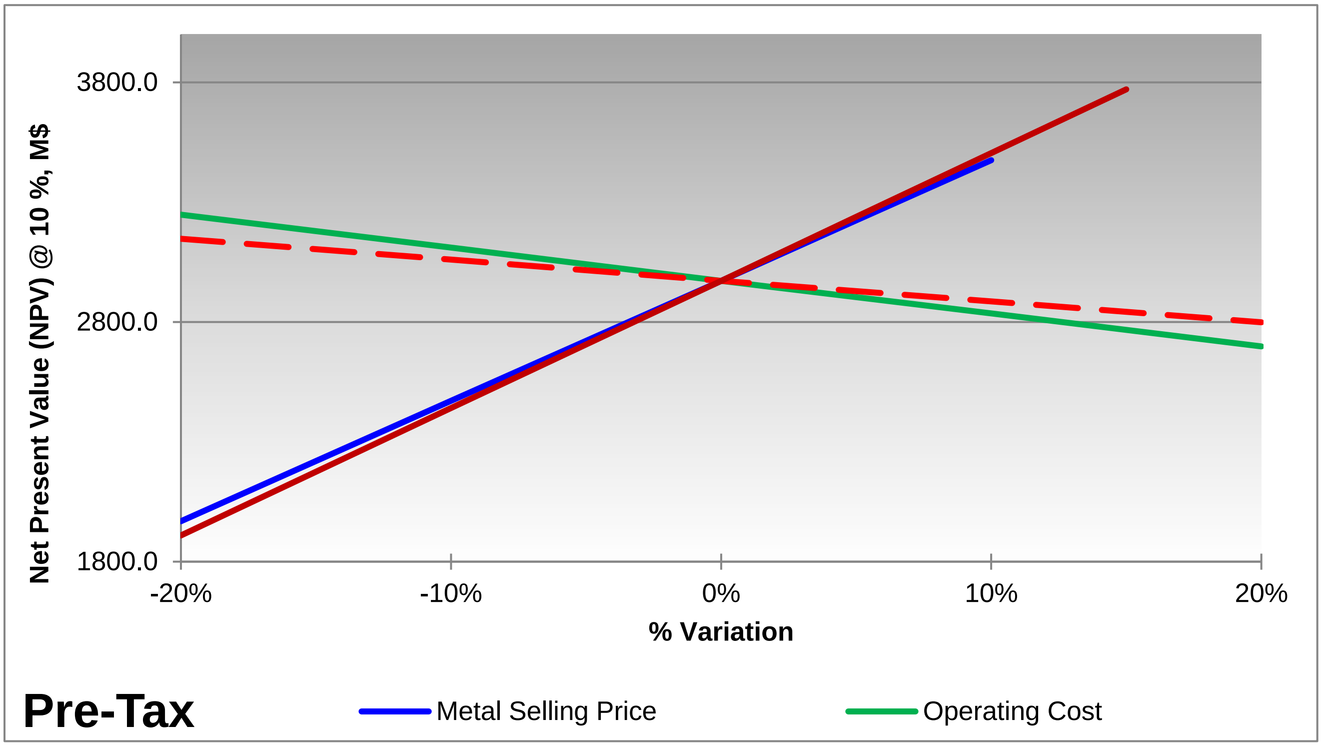


Figure 22-3: Pre-Tax Sensitivity Analysis on IRR

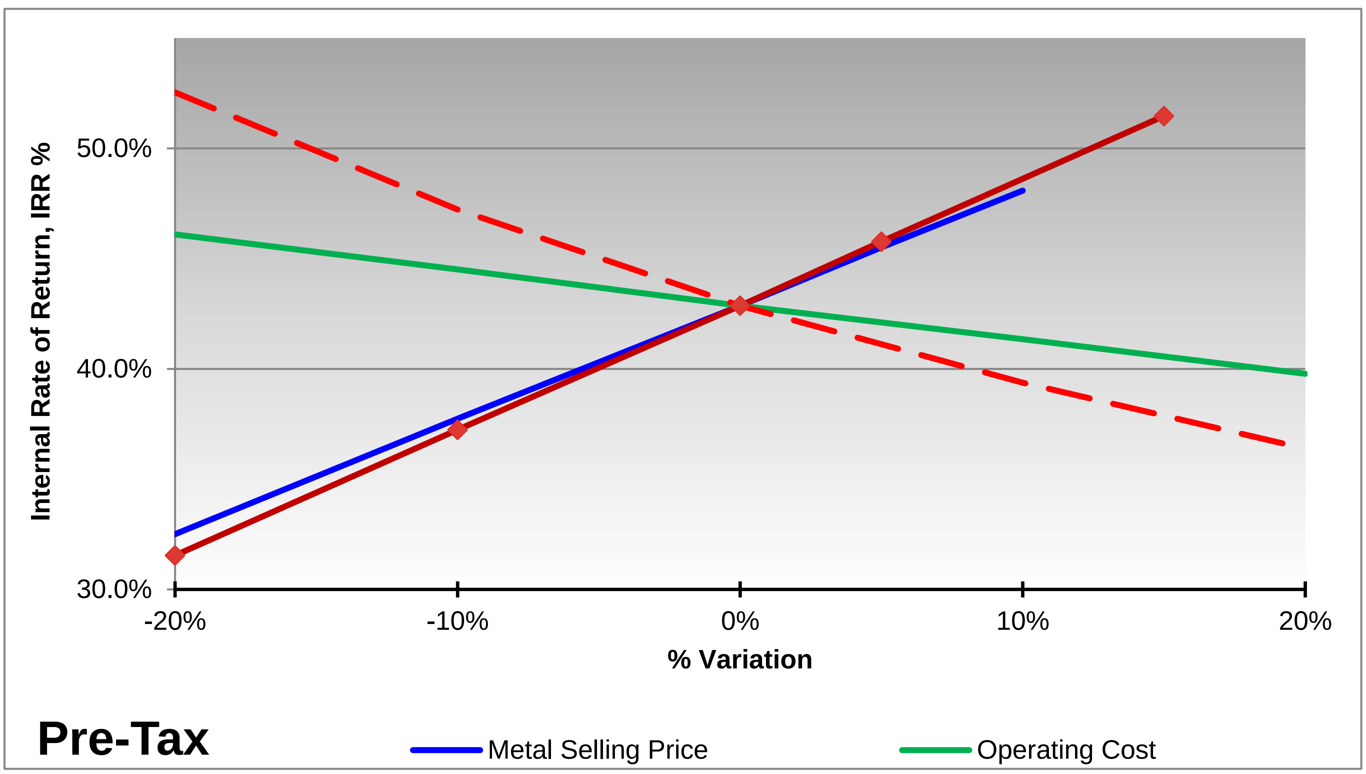


Figure 22-4: Pre-Tax Sensitivity Analysis on Payback Period @ 10%

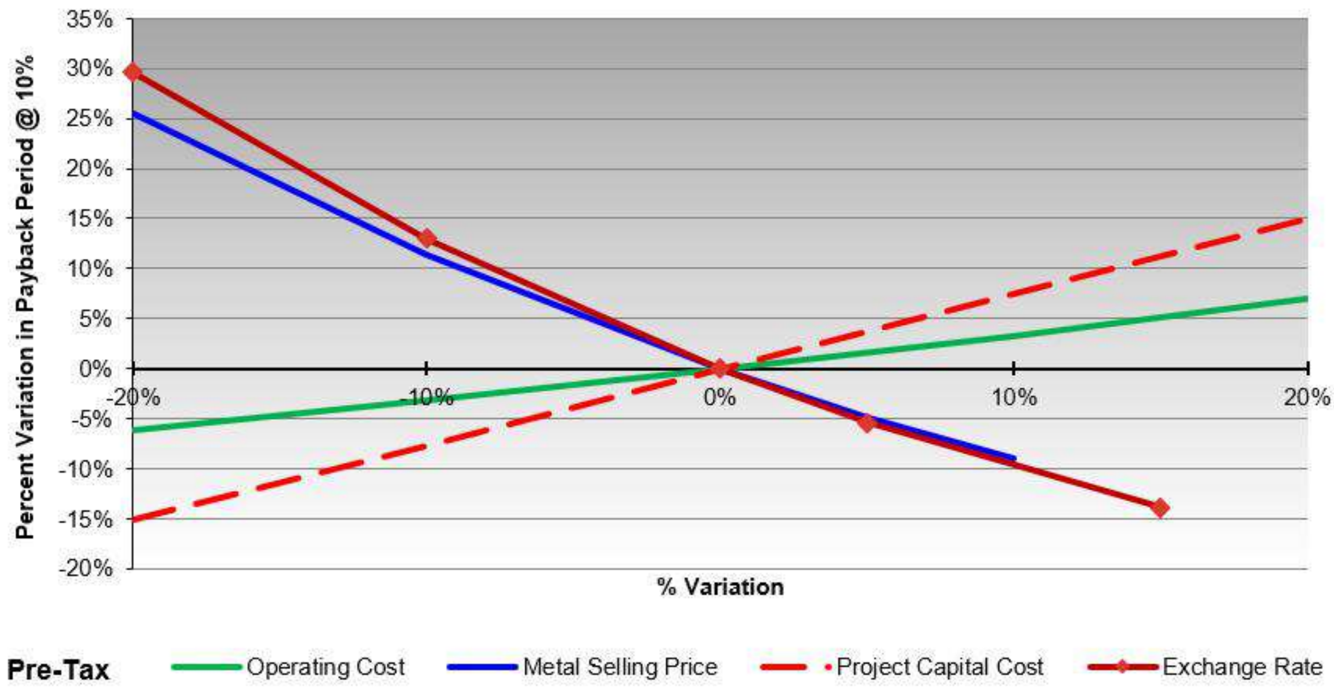


Table 22-5: Pre-Tax Sensitivity on Al-Sc2% Metal Price

Metal Selling Price							
Description		**Unit**	**Net Present Value (M $)**				
% Variation		%	10%	5.0%	0%	-10.0%	-20.0%
Metal Selling Price		CAN$/tonne Al-Sc2	280,500	267,750	255,000	229,500	204,000
PRE-TAX							
Discount Rate	0%	M CAN$	11772.5	11042.0	10309.4	8850.3	7389.2
	5%	M CAN$	6068.0	5667.2	5264.5	4464.9	3663.3
	7%	M CAN$	4802.6	4474.9	4145.3	3491.8	2836.4
	10%	M CAN$	3475.4	3224.3	2971.4	2471.0	1968.8
	11%	M CAN$	3139.8	2908.1	2674.6	2212.9	1749.4
	12%	M CAN$	2844.4	2629.7	2413.3	1985.7	1556.3
	15%	M CAN$	2144.6	1970.3	1794.2	1447.2	1098.5
	51%	M CAN$	0.0	0.0	0.0	0.0	0.0
Internal Rate of Return (IRR)		%	48.1%	45.5%	42.9%	37.7%	32.5%
Payback Period		YEARS	2.1	2.3	2.5	2.9	3.3
YEARS FROM START OF PRODUCTION		% Variation in Payback	-9%	-5%	0%	11%	26%

Table 22-6: Pre-Tax Sensitivity on Operating Cost

Operating Cost							
Description		**Unit**	**Net Present Value (M $)**				
% Variation		%	-20%	-10.0%	**0%**	10.0%	20.0%
Operating Cost		CAN$/tonne ore	281.86	317.09	**352.32**	387.55	422.78
PRE-TAX							
Discount Rate	0%	M CAN$	11056.8	10684.1	10309.4	9938.7	9566.1
	5%	M CAN$	5690.8	5478.6	5264.5	5054.3	4842.1
	7%	M CAN$	4499.0	4323.1	4145.3	3971.3	3795.4
	10%	**M CAN$**	**3248.0**	**3110.6**	**2971.4**	**2835.8**	**2698.4**
	11%	M CAN$	2931.5	2803.9	2674.6	2548.8	2421.2
	12%	M CAN$	2652.8	2533.9	2413.3	2296.2	2177.3
	15%	M CAN$	1992.1	1894.0	1794.2	1697.8	1599.7
	51%	M CAN$	0.0	0.0	0.0	0.0	0.0
Internal Rate of Return (IRR)		%	46.1%	44.5%	**42.9%**	41.4%	39.8%
Payback Period		YEARS	2.2	2.3	**2.5**	2.6	2.7
YEARS FROM START OF PRODUCTION		% Variation in Payback	-6%	-3%	0%	3%	7%

Table 22-7: Pre-Tax Sensitivity on Capital Cost

Project Capital Cost							
Description		**Unit**	**Net Present Value (M $)**				
% Variation		%	-20%	-10.0%	**0%**	10.0%	20.0%
Project Capital Cost		CAN$	697	784	**871**	958	1,045
PRE-TAX							
Discount Rate	0%	M CAN$	10485.6	10398.5	10309.4	10224.3	10137.2
	5%	M CAN$	5440.6	5353.5	5264.5	5179.4	5092.3
	7%	M CAN$	4321.4	4234.3	4145.3	4060.1	3973.0
	10%	**M CAN$**	**3147.4**	**3060.3**	**2971.4**	**2886.1**	**2799.0**
	11%	M CAN$	2850.5	2763.4	2674.6	2589.3	2502.2
	12%	M CAN$	2589.2	2502.1	2413.3	2328.0	2240.9
	15%	M CAN$	1970.1	1883.0	1794.2	1708.8	1621.7
	51%	M CAN$	0.0	0.0	0.0	0.0	0.0
Internal Rate of Return (IRR)		%	52.5%	47.2%	**42.9%**	39.4%	36.4%
Payback Period		YEARS	1.9	2.2	**2.5**	2.7	3.0
YEARS FROM START OF PRODUCTION		% Variation in Payback	-15%	-8%	0%	7%	15%

Table 22-8: Pre-Tax Sensitivity on Exchange Rate

Exchange Rate	Includes Sustaining Capital						
Description		**Unit**	**Net Present Value (M $)**				
% Variation		%	-20%	-10.0%	**0%**	**5%**	**15%**
Exchange Rate		1 US$: CA$	1.00	1.13	**1.25**	1.31	1.44
PRE-TAX							
Discount Rate	0%	M CAN$	7271.3	8791.4	10309.4	11071.4	12591.5
	5%	M CAN$	3584.1	4425.3	5264.5	5687.0	6528.2
	7%	M CAN$	2766.5	3456.9	4145.3	4492.4	5182.7
	10%	**M CAN$**	**1909.4**	**2441.3**	**2971.4**	**3239.1**	**3771.0**
	11%	M CAN$	1692.8	2184.6	2674.6	2922.2	3414.0
	12%	M CAN$	1502.2	1958.6	2413.3	2643.3	3099.7
	15%	M CAN$	1050.8	1423.3	1794.2	1982.2	2354.8
	51%	M CAN$	0.0	0.0	0.0	0.0	0.0
Internal Rate of Return (IRR)		%	31.5%	37.2%	**42.9%**	45.8%	51.5%
Payback Period		YEARS	3.5	2.9	**2.5**	2.3	2.0
YEARS FROM START OF PRODUCTION		% Variation in Payback	30%	13%	0%	-5%	-14%

22.7 Alternative Scenario Analysis

Two scenarios were analyzed as part of the PEA:

1. Concentrate Transportation by Boat from Happy Valley Newfoundland to Baie Comeau, Quebec.
2. Concentrate Transportation by Rail from Emeril, Newfoundland to Sept Isles, Quebec.

Table 22-9 presents a comparison of the high-level alternative scenario analysis.

Table 22-9: Alternative Scenario Analysis

PAYABLE QUANTITIES	Unit	R2 - BAIE COMEAU	R3 - SEPT ISLE	Comments on Variation
Al - Sc2%	t	57,626	57,298	less Al-Sc2 Master Alloy produced with a full 5 years of operation of hydromet in Sept Isles option
Sc_2O_3	kg	98,639	109,811	more Sc_2O_3 produced with a full 5 years of operation of hydromet in Sept Isles option
Dy_2O_3	kg	377,014	377,014	same
La_2O_3	kg	3,521,419	3,521,419	same
Nd_2O_3	kg	3,424,666	3,424,666	same
Pr_2O_3	kg	922,670	922,670	same
Tb_4O_7	kg	67,202	67,202	same
REVENUE GENERATED BY METAL				
Sc_2O_3	M CA$	$184.9	$205.9	more Sc_2O_3 produced with a full 5 years of operation of hydromet in Sept Isles option
Master Alloy	M CA$	$14,694.6	$14,611.0	less Al-Sc2 Master Alloy produced with a full 5 years of operation of hydromet in Sept Isles option
Dy_2O_3	M CA$	$60.5	$60.5	same
La_2O_3	M CA$	$6.6	$6.6	same
Nd_2O_3	M CA$	$210.6	$210.6	same
Pr_2O_3	M CA$	$56.7	$56.7	same
Tb_4O_7	M CA$	$49.1	$49.1	same
Total	M CA$	$15,263.1	$15,200.4	more Sc_2O_3 sold first 5 years means less Al-Sc2 Master Alloy sold with a full 5 years of operation of the hydromet in Sept Isles option versus 4 1/2 years in Baie Comeau option
SELLING COSTS				
SELLING GENERAL & ADMINISTRATIVE	M CA$	$91.2	$90.3	Hydromet operation ends in 2050 versus halfway through 2051
ROYALTIES	M CA$	$2.0	$2.0	Buyout 1st year of revenues.
Capitalized Surface and Selling Costs	M CA$	($6.8)	($6.8)	represents owner's costs during pre-production period
Total	M CA$	$86.4	$85.6	Sept Isles Hydromet operation ends in 2050 versus halfway through 2051 with Baie Comeau option
OPERATING COSTS				
MINE CRATER LAKE	M CA$	$248.5	$248.5	same
MILL CRATER LAKE	M CA$	$430.9	$430.9	same
POWER PLANT CRATER LAKE	M CA$	$36.0	$36.0	same
SURFACE MOBILE EQUIPMENT CRATER LAKE	M CA$	$27.5	$27.5	same
WATER MANAGEMENT CRATER LAKE	M CA$	$192.4	$192.4	same
LODGING CRATER LAKE	M CA$	$140.1	$140.1	same
TRANSPORTATION TO EMERIL	M CA$	$1,020.3	$915.6	shorter distance to Emeril than to Happy Valley reduces trucking OPEX about $20/t concentrate
DOME AND ROAD COSTS - CRATER LAKE TO CHURCHILL FALLS	M CA$	$221.8	$221.6	winter road, Orma Lake road, operating costs of domes remains the same
TRANSPORTATION - RAIL EMERIL TO SEPT ISLES	M CA$	$151.1	$237.9	higher cost to ship by rail versus boat increases cost by about $16/t concentrate
TRANSPORTATION at SEPT ISLES	M CA$	$39.8	$12.4	train offloads directly at Sept Isle saving about $5/t concentrate while boat offloads into a waterfront silo and then trucks move concentrate to the Hydromet site for storage
HYDROMET FACILITY Sc_2O_3 SEPT ISLES	M CA$	$52.7	$58.5	6 added months in 1st 5 years increases amount ot Sc_2O_3 produced and sold

PAYABLE QUANTITIES	Unit	R2 - BAIE COMEAU	R3 - SEPT ISLE	Comments on Variation
HYDROMET FACILITY Al - Sc2% SEPT ISLES	M CA$	$1,146.1	$1,139.8	7 added months in 1st 5 years reduces amount ot Al-Sc2 produced and sold
SURFACE MOBILE EQUIPMENT SEPT ISLES	M CA$	$24.8	$24.8	same
WATER MANAGEMENT SEPT ISLES	M CA$	$189.0	$108.6	less ditches plus production ends 1/2 year sooner in Sept Isle option
Capitalized Operating Costs	M CA$	($69.0)	($67.9)	includes 54,647 tonnes of mill concentrate moved to Emeril dome end of 2025 at $168.73/t trucking cost versus $188.01/t trucking cost to Happy Valley
Total	M CA$	$3,852.0	$3,726.8	shorter distance to Emeril than to Happy Valley reduces logistics (truck/rail/road/storage) OPEX to 254.75/t concentrate versus boat OPEX of 263.11/t concentrate, equals $8.36/t concentrate less costly to ship to Sept Isles
SUSTAINING CAPITAL COSTS				
Crater Lake - Mining Equipment, licenses & equipment	M CA$	$24.5	$24.5	same
Crater Lake - Process concentrator	M CA$	$8.2	$8.2	same
Crater Lake - Tailing Storage Facility (Dyke)	M CA$	$23.4	$23.4	same
Crater Lake - Windmill construction at year 5	M CA$	$1.0	$1.0	same
Crater Lake - Future phase Ditches & Pond	M CA$	$2.8	$2.8	same
Sept Isles - Future phase Ditches	M CA$	$4.9	$0.8	less ditches
Sept Isles - Tailing Storage Facility (Raise Dykes)	M CA$	$6.9	$7.6	larger TSF at Sept Isles
Sept Isles - Equipment Replacements	M CA$	$0.0	$0.0	Surface equipment replacement not foreseen
Total	M CA$	$71.7	$68.2	same
CAPITAL COSTS				
Mine Equipment	M CA$	$13.7	$13.7	same
Mill Plant Construction	M CA$	$63.7	$63.7	same
Hydrometallurgical Facility Construction	M CA$	$160.1	$160.1	same
Power & Electrical Both CL and Met Sites	M CA$	$14.1	$14.1	same
CL and Met Site Infrastucture	M CA$	$113.5	$113.5	same
TSF & Water Management	M CA$	$63.7	$69.3	larger TSF at Sept Isles
Initial Winter Road + Orma Lake Road Rehabilitation	M CA$	$46.6	$46.6	same
Off Site Concentrate Handling Infrastructure	M CA$	$45.8	$27.5	smaller dome at Emeril than Happy Valley plus train unloads directly into storage building without need of a dome at Sept Isle plus no waterfront storage and unloading system at Sept Isles
Camp (200 Person Capacity)	M CA$	$26.5	$26.5	same
Pre-Production Annual Mining Licence Fees	M CA$	$0.2	$0.2	same
Pre-Production (see Capitalized Operating Costs above)	M CA$	$69.0	$67.9	less operating costs get capitalized since includes 54,647 tonnes of mill concentrate moved to Emeril dome end of 2025 at $168.73/t trucking cost versus $188.01/t trucking cost to Happy Valley
Owner's Cost (see Capitalized Surface and Selling Costs above)	M CA$	$6.8	$6.8	same
Engineering fees	M CA$	$29.6	$28.9	percent of direct costs, so reflects savings for no dome and no waterfront unloading with storage silo system at Sept Isle

PAYABLE QUANTITIES	Unit	R2 - BAIE COMEAU	R3 - SEPT ISLE	Comments on Variation
Construction - Health and Safety Training Costs	M CA$	$2.7	$2.7	percent of direct costs, so reflects savings for no dome and no waterfront unloading with storage silo system at Sept Isle
Construction - Contractor Indirect Cost & Management	M CA$	$49.3	$48.1	percent of direct costs, so reflects savings for no dome and no waterfront unloading with storage silo system at Sept Isle
Construction - Commissioning & Start-up Services	M CA$	$11.0	$10.7	percent of direct costs, so reflects savings for no dome and no waterfront unloading with storage silo system at Sept Isle
Construction - Freight / Transportation / Logistics	M CA$	$6.3	$6.2	percent of direct costs, so reflects savings for no dome and no waterfront unloading with storage silo system at Sept Isle
Common Distributables - Spare Parts (Start-up)	M CA$	$5.5	$5.5	same
SubTotal	M CA$	$727.9	$711.7	no dome and no waterfront unloading with storage silo system at Sept Isle
Contingency	M CA$	$163.0	$159.2	Savings with no dome and no waterfront storage at Sept Isle reduces capital and thereby reduces contingency
Total	M CA$	$890.9	$870.9	Sept Isles unloads trains directly into hydromet concentrate building. No dome and no waterfront storage at Sept Isle reduces overall project capital in comparison to Baie Comeau option.
OTHER COSTS				
Working Capital	M CA$	$0.0	$0.0	net zero
Crater Lake - Restoration cost at year 26	M CA$	$74.4	$74.4	same
Sept Isles - Restoration cost at year 27	M CA$	$70.6	$65.1	Less costly closure for Sept Isles versus Baie Comeau
Total	M CA$	$145.0	$139.5	Less costly closure for Sept Isles versus Baie Comeau
PRE-TAX CASHFLOW				
PRE-TAX NPV @ 0%	M CA$	$10,217.1	$10,309.4	This NPV shows no discounting, so better Sept Isle result reflects OPEX / CAPEX savings.
PRE-TAX NPV @ 5%	M CA$	$4,987.7	$5,264.5	Baie Comeau is a higher cost option with a production lag that requires significant amount of working capital which then reduces NPV below that of Sept Isle.
PRE-TAX NPV @ 7%	M CA$	$3,847.5	$4,145.3	
PRE-TAX NPV @ 10%	M CA$	$2,663.1	$2,971.4	
PRE-TAX NPV @ 11%	M CA$	$2,365.9	$2,674.6	
PRE-TAX NPV @ 12%	M CA$	$2,105.3	$2,413.3	
PRE-TAX NPV @ 15%	M CA$	$1,491.6	$1,794.2	
PRE-TAX DCF	M CA$	$2,663.1	$2,971.4	
PRE-TAX IRR		34.1%	42.9%	When compared to Baie Comeau option with working capital increased by production lag, increase in early revenues makes IRR for Sept Isles option better.
PRE-TAX Payback		3.4	2.5	Full year revenue, plus moving 2026 project capital to 2025, makes 2026 cashflow positive, resulting in 0.9-year quicker payback when compare options that include working capital
AFTER TAX CASHFLOW				
AFTER-TAX NPV @ 0%	M CA$	$6,253.3	$6,259.0	This NPV shows no discounting, so better Sept Isle result reflects OPEX / CAPEX savings.
AFTER-TAX NPV @ 5%	M CA$	$2,972.5	$3,150.5	With taxes, Baie Comeau remains a higher cost option with a production lag that requires significant amount of working capital which then reduces NPV below that of Sept Isle.
AFTER-TAX NPV @ 7%	M CA$	$2,254.9	$2,455.2	
AFTER-TAX NPV @ 10%	M CA$	$1,506.7	$1,721.3	
AFTER-TAX NPV @ 11%	M CA$	$1,318.3	$1,534.7	
AFTER-TAX NPV @ 12%	M CA$	$1,152.6	$1,369.9	

PAYABLE QUANTITIES	Unit	R2 - BAIE COMEAU	R3 - SEPT ISLE	Comments on Variation
AFTER-TAX NPV @ 15%	M CA$	$760.7	$976.9	
AFTER TAX DCF	M CA$	$1,506.7	$1,721.3	
AFTER TAX IRR		26.7%	32.8%	When compared to higher cost Baie Comeau option which requires a working capital increase by production lag, IRR for Sept Isles option better.
AFTER TAX Payback		4.0	3.0	Full year revenue, plus moving 2026 project capital to 2025, makes 2026 cashflow positive, resulting in 1.0-year quicker payback when compare options that include working capital.

The shipment of concentrate by rail to Sept Isles provides better economic indicators and accordingly is selected to be the Base Case for the PEA.

Sept Isles is a lesser cost option and provides a full year operation at start-up, versus Baie Comeau waiting until midyear to receive concentrate for start-up.

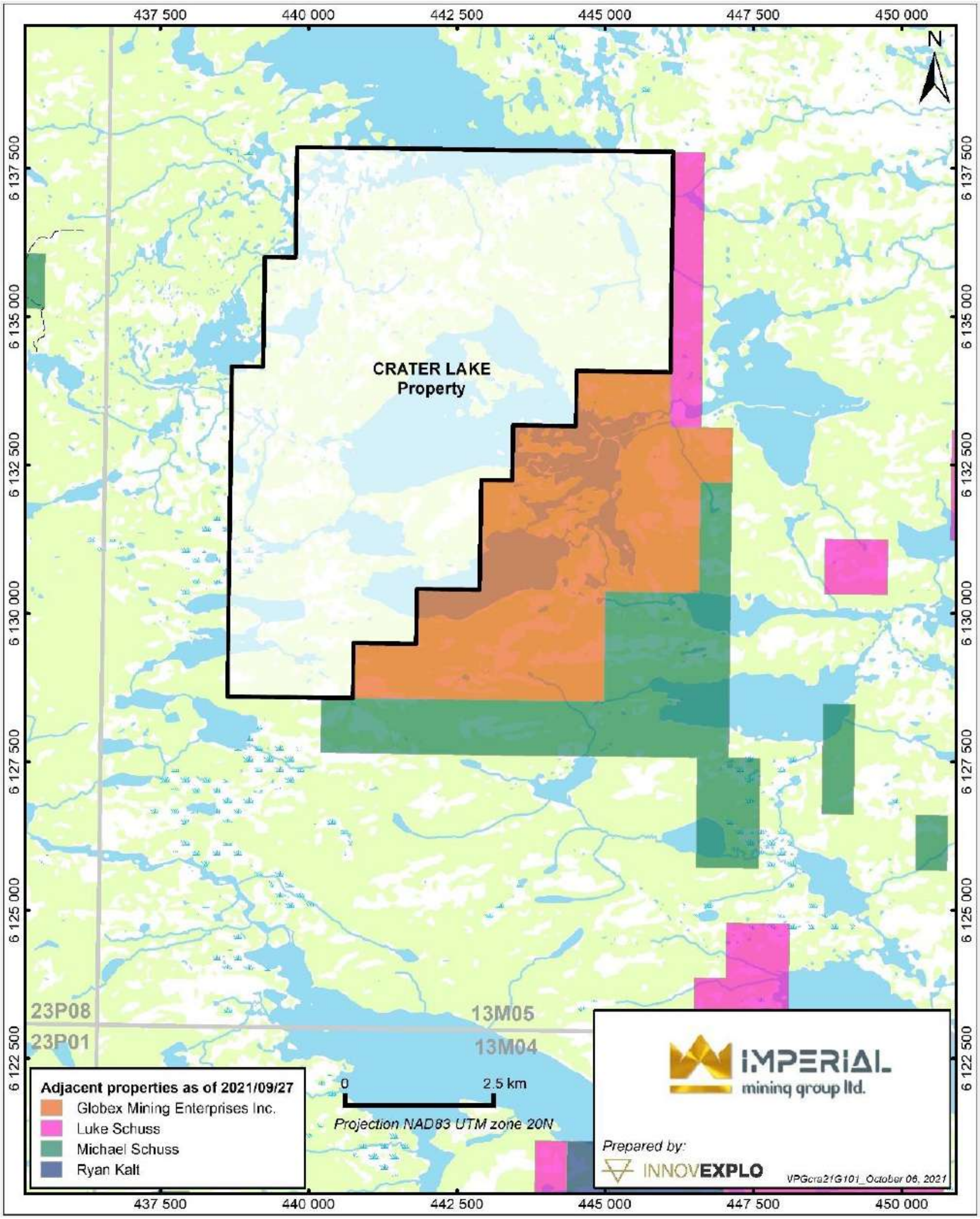
Given the high cost of concentrate transportation, it is recommended that the concentrate transportation alternatives be further assessed at the next level of study.

23 ADJACENT PROPERTIES

As at the effective date of the Technical Report, the online GESTIM claims database shows several properties under different ownership adjacent to the Property (Figure 23-1). This public information has not been verified by the QPs. As at the time of writing, the authors are not aware of any active exploration work in the immediate area of the Property that would be considered relevant to the 2021 MRE.

The Strange Lake deposit, 100% owned by Torngat Metals Ltd, lies approximately 110 km to the north of the Property. The Strange Lake miarolithic pegmatite hosts 4.9 million tonnes of REO. A 2014 PEA detailed a mineral resource estimate for the B Zone, with a cut-off grade of 0.5% TREO, including indicated resources of 278.13 Mt at 0.93% TREO and inferred resources of 214.35 Mt at 0.85% TREO. Torngat Metals Ltd is currently doing piloting and engineering with a PFS to be completed by 2021/2022.

Figure 23-1: Adjacent Properties



24 OTHER RELEVANT DATA AND INFORMATION

There is no Other Relevant Data and Information available at this time.

25 INTERPRETATION AND CONCLUSIONS

Table 25-1 summarizes the interpretations and conclusions discussed in this Item.

Table 25-1: Risks and Opportunities for the Crater Lake Project

RISK	POTENTIAL IMPACT	POSSIBLE RISK MITIGATION
Metallurgical recovery below expectations	Metallurgical tests are preliminary; recovery could be worse than what is currently assumed	Additional metallurgical testwork.
Difficulty in attracting experienced professionals	The ability to attract and retain competent, experienced professionals is a key factor in the success of the Project	The early search for professionals will help identify and attract critical people. It may be necessary to provide accommodation for key people (not included in project costs).
Niche market	It may be difficult to find a buyer	The early search for buyers will ensure commercial opportunities. It will allow the production to be adapted to the needs of the market.
Price of scandium and REE controlled by only a few producers	Price volatility	The early search for buyers will ensure commercial opportunities. It will allow the production to be adapted to the needs of the market.
Geotechnical capacities overestimated for infrastructure	Raised costs and footprint for some infrastructure	Perform a geotechnical study to confirm design parameters
Airstrip location not suitable	Farther airstrip location to comply with regulations	Airstrip location complete study
OPPORTUNITIES	**EXPLANATION**	**POTENTIAL BENEFIT**
Delineation drilling	Lateral and Deep extensions still open	Likely to increase the geological and grade continuities
Exploration potential	The Property contains untested geophysical targets and under-explored targets	Potential to discover a satellite deposit
Metallurgical recovery optimization	Metallurgical tests are preliminary; additional metallurgical testwork could improve the recovery	Recovery could be optimized to be better than what is currently assumed.
Missing Dewatering Data	Increase Mining Cost	Include a Hydrogeology Study in further Studies for this Project
Price of scandium and REE controlled by only a few producers	Price volatility	Potential to benefit from the growth of green energy. The early search for buyers will ensure commercial opportunities. It will allow the production to be adapted to the needs of the market.

25.1 Geology

InnovExplo generated the mineral resource estimate for the scandium-rare earth TGZ target on the Crater Lake Property (the “2021 MRE”).

The QPs from InnovExplo conducted site visits that included but were not limited to a review and validation of the data used for the 2021 MRE, including the geology and mineralization, and a review of the drilling and sampling procedures and processing methods. The QPs also validated the geological information provided by the issuer or obtained from public sources.

The Project is situated in northern Quebec, near the border of Newfoundland and Labrador, approximately 200 km northeast of Schefferville, Quebec. The Property consists of 96 contiguous mineral claims (two claim blocks) that cover the northwestern portion of the Crater Lake syenite batholith and the four principal exploration targets: TGZ, STG, North and Boulder Lake.

Current exploration and drilling on the TGZ target cover an area approximately 500 m along strike and 120 m wide, to a depth of approximately 250 m. The scandium and REE mineralization occurs mainly within the olivine ferro-syenite (OLFESYN), with minor mineralization in the pyroxene ferro-syenite (PXFESYN).

The Hill Top, STG and SCL targets cover an approximate strike length of 700 m based on geophysical surveys. Previous results and current geological observations for the STG target show a similar geological context as TGZ. An exploration program, including channel sampling and two (2) drill holes, was underway on the STG target at the time of writing. No exploration work was being planned for the SCL and Hill Top targets.

For the TGZ target, InnovExplo created a lithological model for the mineralized domains using all available geological and analytical information. To provide accurate resource modelling, the QPs based their wireframe model of mineralized domains on the drill hole database and the interpretation provided by Imperial's geologists.

The following conclusions are based on a detailed review of all pertinent information and results:

— The database supporting the 2021 MRE is complete, valid and up to date.
— The geological and grade continuity of scandium and REE mineralization in the OLFESYN and PXFESYN domains has been demonstrated, supported by a 50-m drilling grid.
— The 2021 MRE is classified as indicated and inferred resources. There are no measured resources.
— The 2021 MRE was prepared for a potential open-pit scenario at an NSR cut-off of 110.80 CA$/t.

The 2021 MRE for the TGZ target at the Crater Lake Project, at a 110.80 CA$/t NSR cut-off grade, comprises:

— Indicated Resource of 7,315,500 t grading 282 g/t Sc_2O_3, 66 g/t Dy_2O_3, 606 g/t La_2O_3, 596 g/t Nd_2O_3, 160 g/t Pr_2O_3, 12 g/t Tb_4O_7 equivalent to a 413 CA$/t NSR
— Inferred Resource of 13,158,400 t grading 264 g/t Sc_2O_3, 62 g/t Dy_2O_3, 569 g/t La_2O_3, 573 g/t Nd_2O_3, 154 g/t Pr_2O_3, 11 g/t Tb_4O_7 equivalent to a 386 CA$/t NSR.
— Table 25-1 identifies important internal risks, potential impacts and possible risk mitigation measures that could affect the economic outcome of the Project. It does not over the external risks that apply to all mining projects (e.g., changes in metal prices, exchange rates, availability of investment capital, change in government regulations, etc.). Significant opportunities that could improve the economics, timing and permitting of the Project are also identified in this table. Further information and evaluation are required before these opportunities can be included in the project economics.

25.2 Mining

The deposit will be mined while prioritizing the mineralized material with higher profit margin. The strip ratio for the Starter pit, the Middle Pit and the Ultimate Pit are respectively 0.59, 1.64 and 2.29 for an overall strip ratio of 2.01. Indicated and inferred in-pit resources are respectively 6.7 Mt and 3.9 Mt.

The life of mine and the mine production schedule is a combination of different mining rates, mill capacity and operations strategy to optimize the profitability and results in a 25-year mine life. The low overburden material to be stripped and the mineralization outcrops are an advantage to lower production costs at the start of the mine.

25.3 Milling

Based on mineral processing and hydrometallurgical testwork scandium can best be recovered using well established metallurgical processes such as magnetic separation to produce a scandium rich mineral concentrate and by treating the concentrate through caustic leaching, hydrochloric acid leaching and solvent extraction to produce a high-grade scandium oxide (Sc_2O_3) and a mixed REE hydroxide product. The scandium oxide or scandia will be processed together with alumina to produce Al-2%Sc master alloy in a conventional direct electrolysis process.

— The beneficiation plant, located at Crater Lake, will operate six months out of the year and will have a throughput capacity of 2380 t/d and will recover 90% of the scandium and 65% of the REE to produce 1220 t/d of concentrate.

— The hydrometallurgical plant, located in Sept-Isles, will operate year-round and, with an expected Sc recovery of 81%, will produce 87.3 t/a of scandia (Sc2O3) and 1,865 t/a of mixed REE hydroxide. The scandia will be processed together with alumina to produce 2,562 t/a of A-2%Sc master alloy.

— Specialized unit operations such as high pressure and temperature caustic leaching, solvent extraction and Al direct electrolysis will need to be evaluated in more detail in the next phase of the project study.

— Chemical reagents used in the hydrometallurgical plant represent over 71% of the total process operating costs. As such, reagents consumption, procurement, transportation, and handling logistics will need to be evaluated in more detail in the next phase of the project study.

25.4 Infrastructure

Please refer to Table 25-1 for infrastructure interpretations and conclusions.

25.5 Geotechnical- Waste Rock and Overburden Stockpiles

Geotechnical investigation for waste and overburden stockpiles are recommended in order to assess soil conditions such as soil stratigraphy, depth to bedrock, groundwater levels and soil properties. Tests pits are also recommended for soil characterization. Borehole and test pit will provide geotechnical and hydrogeological data and required to adapt and optimize the design of infrastructures. Permafrost needs to be address in those investigations.

25.6 Tailings, Waste Rock and Overburden characterization

A comprehensive Geochemical investigation is required at the next step of Engineering in order to confirm the water management, water treatment Plants, tailings management, and waste rock stockpiles design and strategies

Additional bathymetry and lidar in specific areas are also recommended.

Geochemestry comprehensive study is requested, the estimated construction and reclamation costs are highly dependent of site specific Geochemestry (acid generating potential and metal leaching potential) of tailings and waste rock. In the case of a potential acid or significant metal leaching challenges, the TMF and WRD Design, closure and reclamation concept will need to be reviewed, and costs could be significantly modified accordingly.

Additional test for specific tailings density could lead to a change in the overall tailings volume requirement.

25.7 Water Management

The following interpretations and conclusions are noted for water management:

— Surface water geochemical model is required for WTP design and treatment;

— assess liner geotechnical properties and geochemical compatibility with process water;

— Water balance for the overall sites and review of WTP requested capacity;

— Integration of refined climate data analysis is recommended.

25.8 Environment

The next step will be to produce an ESIA, which must meet all regulatory requirements and guidelines. This EIA should address, without be limited to, potential issues or risks highlighted in this PEA: wildlife and plant species having a status, caribou, underground and surface waters, wetlands and fish habitat, and social context. It is recommended to continue discussions throughout the Project and align the different phases of the Project with the Indigenous and regional communities to understand and consider their concerns in the Project.

The exploitation of a rare earth deposit must be the subject of particular attention in terms of geochemical potential issues, water quality and impact on the receiving environment.

It also should be note that the Moinier region has a particular regulatory context that has not been tested so far for permitting purposes with the authorities. Therefore, the Project approach and schedule will need to take this into consideration.

Finally, the 1000m+ Air Strip may trigger federal ESIA, which may have significant impact on EISA work nature and schedule.

25.9 Costs and Economics

Based on the study results, the conclusions are as follows:

1. The overall economic results indicate that the Project will have positive economic returns and generate approximately $6,259 million net after-tax cashflow ($10,309 million pre-tax) over the Project's 25-year life of mine.
2. Total project capital requirements have been estimated at approximately $871 million, with an additional $38 million in working capital, prior to commencement of Master Alloy production, followed by approximately $68 million of sustaining capital over the life of the Project. The project capital is inclusive of contingency.
3. Total operating expenses over the life of the mine are estimated to be approximately $3,727 million or $352.32 per tonne milled or $56,707 per tonne Alloys sold.
4. At the base case metal prices, the Project's post-tax net present value is estimated at approximately $1,721 million at a discount rate of 10%. Discounting commences with Commercial Production of the Hydrometallurgical facility. The post-tax IRR is estimated at 32.8% and post-tax payback has been calculated at 3.0 years from start of production.
5. The project economics are most sensitive to variations in factors impacting revenue such as Al-Sc2% Master Alloy price and currency exchange rates.

26 RECOMMENDATIONS

This NI 43-101 Preliminary Economic Assessment ("PEA") report was prepared by experienced and qualified independent consultants using recognized engineering standards. Based on the results of this study, it is recommended that the Crater Lake Project proceeds to the next phase of the Project development and that additional exploration and engineering work is conducted.

This section provides a summary of the QP's recommendations based on this PEA results and findings for each major area of investigation.

26.1.1 Geology

Based on the results of the 2021 MRE, the following work program is recommended:

- Complete a 4,000 m drilling program on the southern portion of the TGZ target (Sections 0N to 350N) and at depth (sections 450N and 500N). The goal of this program is to investigate the extensions of the TGZ mineralized domains.
- Perform a Lidar topographic survey to cover the entire Property.
- Perform the planned additional metallurgical testwork (solvent extraction flowsheet development and optimization; development of Al-2%Sc master alloy production technology; and processing pilot program).
- Update the mineral resource estimate for the Project using data from the recommended studies and test results.

26.1.2 Mining

In regard to mining, the following recommendations are provided:

- Perform a geotechnical drilling campaign to assess open-pit geotechnical domains and to confirm geotechnical rock properties.
- Update dilution calculation based on updated resources

26.1.3 Testwork

Continue with the bulk treatment of the Sc/REE mineral concentrates, including olivine removal, and hydrometallurgical development program focused on finalizing the Sc SX circuit and optimizing Sc solubilization processes.

Continue with the scheduled Al-2%Sc master alloy development work focused on co-electrowinning of alumina and Sc_2O_3 planned to start late 2022 or early 2023.

Imperial Mining Group in the fall of 2021 has collected 16 t of sample from Crater Lake site to feed its planned pilot programs.

Start the testwork on a 16-t sample from the Crater Lake site, which will feed a mineral processing pilot program, to produce a Sc/REE-rich mineral concentrate to feed the hydrometallurgy pilot program. This program will commence in early to mid 2023.

26.1.4 Infrastructure

Recommendations related to Infrastructure are presented in Table 25-1.

26.1.5 Environment

A baseline study is required to document and address potential issues related to sensitive wildlife and plant species, caribou, esker aquifer potential, wetlands, and fish habitat. It must be noted that this baseline study is also required to support the Environmental and Social impact assessment.

The quality of the watercourses and the positioning of the final effluent must also be studied. Indeed, the exploitation of a rare earth deposit must be the subject of particular attention in terms of geochemical potential issues, water quality and impact on the receiving environment.

26.1.6 Economics

With the project generating a positive post-tax IRR of 32.8% and a post-tax NPV at 10% of 1,721 million, it is recommended Imperial Mining Group implement other post-study recommendations that will allow them to proceed to the next stage of study. A FEL2, Prefeasibility Study (PFS) that further defines the transportation by rail option and resizing of the hydrometallurgical facility resulting from added concentrate deliveries related to a year-round access road opportunity is seen to be the next step.

26.1.7 Infrastructure

Regarding infrastructure the following work is recommended prior to next stage of study:

- Perform a complete geotechnical investigations and studies program along with site surveying to confirm assumptions used for preliminary infrastructure design. These are required to characterize foundation conditions under planned infrastructure to assess soil conditions such as soil stratigraphy, bedrock depth, groundwater levels and soil properties. The investigations should include an extensive laboratory and in situ testing program. Requirements of this geotechnical investigation campaign will be detailed prior to site works to fulfill the necessary gaps for the feasibility study.
- Borrow source identification and assessment for material properties and estimates for borrow source quantities should be undertaken to define available materials for the construction of all surface infrastructure, including roads, pads, stockpiles, ponds, water management, and TMF.
- Geochemistry sampling and characterization of the mineralized material, waste rock and water shall be performed. If waste rock material is identified as non potentially acid generating and non-leachable, this material could be used for road and pad construction, thus reducing borrow source material.
- Perform an airstrip location study to confirm a suitable location to comply with applicable requirements and Canadian Aviation regulations for physical characteristics, dominating winds, obstacle limitation surfaces, visual aids, etc.
- A FEL2 or Prefeasibility Study (PFS) would allow to further define the requirements and capacities for infrastructure.

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- Système d'information géominière du Québec (SIGEOM) :
- World Climate, Border A station:

28 CERTIFICATE OF QUALIFIED PERSON

CERTIFICATE OF QUALIFIED PERSON
Carl Pelletier

I, Carl Pelletier, state that:

(a) I am a professional geoscientist and Co-President Founder at:
Innovexplo Inc.
560, 3e Avenue,
Val-d'Or, QC, J9P 1S4

(b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of: July 26, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. graduated with a Bachelor's degree in Geology (B.Sc.) from Université du Québec à Montréal (Montreal, Quebec) in 1992. I initiated a Master's degree at the same university for which I completed the course program but not the thesis. I am a member of the Ordre des Géologues du Québec (OGQ, No. 384), the Association of Professional Geoscientists of Ontario (PGO, No. 1713), the Association of Professional Engineers and Geoscientists of British Columbia (EGBC, No. 43167), the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG, No. L4160), and the Canadian Institute of Mines (CIM). My relevant experience includes a total of 30 years since my graduation from university. My mining expertise has been acquired at the Silidor, Sleeping Giant, Bousquet II, Sigma-Lamaque and Beaufor mines. My exploration experience has been acquired with Cambior Inc. and McWatters Mining Inc. I have been a consulting geologist for InnovExplo Inc. since February 2004.

(d) The requirement for a site visit is not applicable to me.

(e) I am the co-author of and share responsibility for sections 14, 25 and 26.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the property that is the subject of the Technical Report. I was QP for the NI 43-101 Technical report entitled "NI 43-101 Technical Report and Maiden Mineral Resource Estimate for the Crater Lake Project, Quebec, Canada" (November 4, 2021).

(g) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(h) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Val-d'Or, Quebec, Canada this 28 of July, 2022.

(Original document signed and sealed)

Carl Pelletier
Co-President Founder
Innovexplo Inc.

CERTIFICATE OF QUALIFIED PERSON
ÉRIC POIRIER

I, Éric Poirier, state that:

(a) I am a Project Manager and Electrical Engineer at:
WSP Canada Inc
1075, 3rd Avenue East
Val-d'Or, Québec, Canada J9P 0J7

(b) This certificate applies to the technical report titled Crater Lake Scandium Project, Preliminary Ecomonic Assessment NI-43 101 Report with an effective date of: July 26. 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Université du Québec à Chicoutimi with Electrical Engineering degree (B.Sc., 1996). I am a member in good standing of the Ordre des ingénieurs du Québec (OIQ No. 120063), Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG No. L2229), and Professional Engineers Ontario (PEO No. 100112909). I hold the credential of Project Management Professional (PMP) from the Project Management Institute (PMI No. 6115196). My relevant experience after graduation and over 24 years for the purpose of the Technical Report includes working as multi-disciplinary project manager and discipline lead for surface infrastructure, buildings, water management, electrical distribution, automation, and communications projects.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Section 18.1, 18.2, 18.3.1, 18.3.2, 21.1.4, and portions of Items 1, 2, 3, 25, 26, and 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Val-d'Or, Québec this 28 Day of July 2022.

"Original document signed and stamped by Éric Poirier, P.Eng., PMP

Éric Poirier, P.Eng., PMP (OIQ, No. 120063)
WSP Canada Inc.

CERTIFICATE OF QUALIFIED PERSON
Ewald Pengel, P.Eng.

I, Ewald Pengel, state that:

(a) I am a Principal Metallurgical Process Specialist at:
WSP Canada
1600 René Lévesque Blvd. West
Montréal, Québec, H3H 1P9 Canada

(b) This certificate applies to the technical report titled I Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of: 26 of July 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University at Kingston, Ontario with a B.Sc. in Metallurgical Engineering in 1982 and the University of Pittsburgh, Pittsburgh, PA with a M.Sc. in Mining Engineering in 1985. I am a registered member of Professional Engineers Ontario (90520297). My relevant experience after graduation and over 36 years for the purpose of the Technical Report includes Metallurgical and Mineral Processing Test Work and Operational Experience.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) item 13 and parts of item 1 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Preliminary Economic Assessment of the Crater Lake Project in Québec for which I am responsible has been prepared in compliance with NI 43-101.

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Preliminary Economic Assessment of the Crater Lake Project in Québec for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Montréal, QC this 28 of July, 2022

(Original document signed and sealed)

Ewald Pengel, M.Sc., P. Eng.

CERTIFICATE OF QUALIFIED PERSON
Marina Lund

I, Marina Iund, state that:

(a) I am a senior resource geologist at:
Innovexplo Inc.
725 Boul. Lebourgneuf, Suite 312,
Quebec City, Qc, G2J 0C4

(b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of: July 26, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I graduated with a Bachelor's degree in Geology from Université de Besançon (Besançon, France) in 2008. In addition, I obtained a Master's degree in Resources and Geodynamics from Université d'Orléans (Orléans, France), as well as a DESS degree in Exploration and Management of Non-renewable Resources from Université du Québec à Montréal (Montreal, Quebec) in 2010. I am a member of the Ordre des Géologues du Québec (OGQ No. 1525), the Association of Professional Geoscientists of Ontario (PGO, No. 3123), and the Northwest Territories and Nunavut Association of Professional Engineers and Professional Geoscientists (NAPEG licence No. L4431). I have practiced my profession in mineral exploration, mine geology and resource geology for a total of 13 years since graduating from university. I acquired my expertise with Richmont Mines Inc. and Goldcorp. I have been a project geologist and then a senior resource geologist for InnovExplo Inc. since September 2018.

(d) My most recent personal inspection of each property described in the Technical Report occurred on May 7 to 9, 2021 and was for a duration of 2 days.

(e) I am responsible for Items 4 to 13 and 23 of the Technical Report as well as co-author of and share responsibility for items 1 to 3, 14 and 24 to 26.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the property that is the subject of the Technical Report. I was QP for the NI 43-101 Technical report entitled "NI 43-101 Technical Report and Maiden Mineral Resource Estimate for the Crater Lake Project, Quebec, Canada" (November 4, 2021).

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Quebec City, Quebec, Canada this 28 of July, 2022.

(Original document signed and sealed)

Marina Iund, P.Geo.
Senior Resource Geologist
Innovexplo Inc.

CERTIFICATE OF QUALIFIED PERSON
Mireno Dhe Paganon

I, Mireno Dhe Paganon, state that:

(a) I am a Senior Metallurgist at:
WSP Canada
1600 René-Levesque
Montréal, Québec, H3H 1P9

(b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of July 26th, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of McGill University with a B. Eng. Degree in Metallurgy and Mineral Processing in 1991, and member in good standing with the Ordre des Ingénieurs du Québec (OIQ #118862). My relevant experience after graduation and over twenty-five years for the purpose of the Technical Report includes Applications and Process Engineer for mineral processing equipment suppliers and engineering consulting firms. This includes mineral processing and hydrometallurgy projects involving process design and flowsheet development, equipment recommendations, process support and optimization of unit operations.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item 17 and relevant sections of item 1 and 21 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the 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 Montréal, Québec, 28th of July, 2022.

(Original document signed and sealed)

Mireno Dhe Paganon, ing.

CERTIFICATE OF QUALIFIED PERSON

Paul Daigle, géo., P.Geo.

I, Paul Daigle, géo., P.Geo., state that:

(a) I am an Associate Resource Geologist at:
InnovExplo Inc.
859, Boulevard Jean-Paul Vincent - Suite 201 Longueuil,
QC, Canada, J4G 1R3

(b) (b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of: July 26, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Concordia University with a Bachelor of Science, Geology (1989). I am a member of the:
Ordre des Géologues du Québec (No. 1632).
Professional Geoscientists of Ontario (No. 1592)
Professional Engineers and Geoscientists of Newfoundland and Labrador (No. 06679)
Engineers and Geoscientists of British Columbia (No. 55629)
My relevant experience after graduation and over 32 years for the purpose of the Technical Report includes: the Strange Lake REE deposit in Quebec (technical report and resource estimate), the Two Tom REE deposit in Newfoundland and Labrador (technical report and resource estimate), and the Misery Lake Project (now Crater Lake Project) (technical reports in 2014 and 2017).

(d) My most recent personal inspection of each property described in the Technical Report occurred on 30 September and 1 October 2014 and was for a duration of 2 days.

(d) I am responsible and co-author for Item(s) 14, 25.1 and 26.1 of the Technical Report.

(e) I am independent of the issuer as described in section 1.5 of NI 43-101.

(f) My prior involvement with the property that is the subject of the Technical Report is as follows: I was QP and author for the Crater Lake Project (then Misery Lake Project) technical reports in 2014 and 2017; and QP and co-author in 2021.

(g) I have read the parts of the Technical Report for which I am responsible and has been prepared in compliance with NI 43-101; and

(h) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Toronto, Ontario this 26 of July 2022.

(Original document signed and sealed)

Paul Daigle, géo., P.Geo. (OGQ No. 1632)
Associate Resource Geologist

CERTIFICATE OF QUALIFIED PERSON
SIMON LATULIPPE

I, Simon Latulippe, state that:

(a) I am a senior engineer at:

- Team Lead for geotechnical and Mine waste/environment, with WSP Canada inc. in an office located at 1135, blvd Lebourgneuf, Quebec, Quebec, Canada, G2K OM5.

(b) This certificate applies to the technical report titled Insert Title of Report with an effective date of: insert date of report (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Laval University with a B.Sc. in geological engineering in 1998. I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: (#121692). I am a Member of the Mining Association of Canada (MAC) as part of the Tailings committee. My relevant experience after graduation and over 24 years for the purpose of the Technical Report includes (Selbaie Mine and Gaspé Mine Closure projects, Nunavik Nickel Mine, Royal Nickel Project, Mine Arnaud Project, Arcelor Mittal Mont-Wright, SFPPN Transhipment and Load out facility- water management, Glencore Kidd TMA Closure, Bloom Lake Mine- tailings management, Ministry Orphaned sites New Calumet, Main Mine and Preissac Mine Sites Closure) Environmental engineering, Mine Closure and reclamation, geochemistry, water management and geotechnical engineering for surface infrastructures.

(d) My most recent personal inspection of each property described in the Technical Report occurred on enter date and was for a duration of number days.

(d) the requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) 18.4, 18.5, 18.6 and 20 of the Technical Report, portions of Items 1, 25, 26, and 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Quebec, Quebec, Canada this 28 of July, 2022.

(Original document signed and sealed)

Simon Latulippe, P.Eng. (Ordre des ingénieurs du Québec no. 121692)

CERTIFICATE OF QUALIFIED PERSON

William Richard McBride, P.Eng.

I, William Richard McBride, P.Eng. state that:

(a) I am a Senior Mining Engineer at:
WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario, P3C 4Y1

(b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Quebec with an effective date of: July 26, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University with a Bachelor of Science in Mining Engineering (B.Sc. 1973). I am a member in good standing of the Association of Professional Engineers of Ontario (PEO), License Number 29888013. My relevant experience includes 49 years of experience in mine engineering and operations, including long-range and short-range mine planning and the managing of projects from concept through to start-up. From 1970 to 2008, I worked at Canadian and Central American mines holding positions as a Miner, Certified Open Pit Blaster, Planning Engineer (open pit and underground), Certified Underground Hard Rock Supervisor, Engineering Supervisor, Chief Mine Engineer, Superintendent Technical Services and Senior Projects Manager. From 2009 to 2021, I worked as a consultant Project Manager, a Senior Mining Engineer and a consultant Mining Engineer. Since 2021, I have worked as an Associate Mining Engineer. I have completed scoping studies, prefeasibility studies, feasibility studies, project evaluations, due diligences, technical reviews and economic analyses for nickel sulphide PGE-type deposits, palladium deposits, narrow vein gold deposits, chromite deposits, and lithium deposits. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) 18.3.3, 18.7, 19, 21, 22 and portions of Items 1, 3, 24, 25, 26 and 27 that are based on those Items of the Technical Report.

(f) I am independent of the issuer of the report, as described in section 1.5 of NI 43-101.

(g) I have no prior involvement with the Property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the items of The Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(g) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the items of The Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury / Ontario, 28 July 2022

Original document signed and stamped by Rick McBride, P.Eng.

Rick McBride, P. Eng.
Associate Mining Engineer
WSP Canada Inc.

CERTIFICATE OF QUALIFIED PERSON
Zakaria Ould Moctar

I, Zakaria Ould Moctar state that:

(a) I am a Mining Engineer at:
WSP Canada 1600
René-Levesque
Montréa, Québec, H3H 1P9

(b) This certificate applies to the technical report titled Preliminary Economic Assessment of the Crater Lake Project in Québec with an effective date of: July 26, 2022 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of The National School of Mineral Industry (ENIM) with a Bachelor of Science in Mining Engineering (graduated in 2004). I am a member in good standing of the Ordre des Ingénieurs du Québec (OIQ), License number 5000433 and a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG), License number L5007. My relevant experience after graduation includes 15 years of experience in mining engineering, including drill and blast, geotechnic, short-range and long-range planning, mine economics, the managing of technical and engineering teams as well as the managing of mining projects from inception to completion. From 2004 to 2005, I worked as a junior engineer with SNIM, a national Iron Mining Company in Zouerate, Mauritania. From 2008 to 2010, I worked as a junior mining engineer with Inmet Mining corporation at Troilus gold mine. From 2010 to 2014, I worked as a mine planning engineer with Met-Chem Canada in Montreal, a consulting engineering firm. I also worked temporarily during four months for Arnaud Mine as a project engineer in 2014, a phosphate developing project based in Sept-Ile, Quebec. From 2015 to 2016, I worked as a mining engineer with BBA Inc. in Montreal, a consulting engineering firm. From 2016 to 2017, I worked as an open-pit senior mining engineer at Casa Berardi Mine gold mine, with Hecla-Mining, 90km north of La-Sarre, Québec. From 2017 to 2018, I worked as a senior mining engineer at Essakane gold mine in Burkina Faso, with Iamgold. From 2018 to 2020, I worked as the Chief mine Engineer at Bloom Lake Iron Ore Mine, mine near Fermont, Québec, with Quebec Iron Ore. From 2020 to present (2022), I work as a principal mining engineer with WSP Canada Inc., a consulting engineering firm, based in Montreal.

(d) The requirement for a site visit is not applicable to me.
(e) I am responsible for Item(s) 16 of the Technical Report.
(f) I am independent of the issuer as described in section 1.5 of NI 43-101.
(g) I have not had prior involvement with the property that is the subject of the Technical Report.
(h) I have read NI 43-101 and The Technical Report OR the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report OR the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Ottawa, Ontario this 28 of July, 2022.

(Original document signed and sealed)

Zakaria Ould Moctar
Mining Engineer
WSP Canda Inc.