NI 43-101 Technical Report

PRELIMINARY ECONOMIC ASSESSMENT

BAPTISTE NICKEL PROJECT

British Columbia, Canada

Prepared for:

FPX Nickel Corporation

FPX Nickel Corp.
TSX-V:FPX

By Qualified Persons:

Angelo Grandillo, P. Eng. .... BBA Inc.
Ronald Voordouw, P. Geo. .... Equity Exploration Consultants Ltd.
Ronald G. Simpson, P. Geo...... GeoSim Services Inc.
Gordon Chen, P. Eng. .... Stantec
Sean Ennis, P. Eng.................. Stantec
Jeff Austin, P. Eng. .... IME Inc.

Effective Date: **September 9, 2020**

Signature Date: **September 29, 2020**

BBA EQUITY GeoSim Services Stantec IME

DATE AND SIGNATURE PAGE

This report is effective as of the 9th day of September 2020.

"Original signed and sealed on file"	September 29, 2020
Angelo Grandillo, P. Eng., M. Eng. BBA Inc.	Date
"Original signed and sealed on file"	September 29, 2020
Ronald Voordouw, P. Geo. Equity Exploration Consultants Ltd.	Date
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CERTIFICATE OF QUALIFIED PERSON

Angelo Grandillo, P. Eng., M. Eng.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective as of September 09, 2020.

I, Angelo Grandillo, P. Eng., M. Eng., do hereby certify that:

1. I am an Associate and a Project Manager in the consulting firm BBA Inc., 2020 Robert-Bourassa Blvd., Suite 300, Montréal, Québec, Canada, H3A 2A5.
2. I graduated from McGill University of Montréal with a B. Eng. in Metallurgical Engineering in 1981, and M. Eng. In Metallurgical Engineering in 1988.
3. I am in good standing as a member of the Order of Engineers of Québec (#38342) and Engineers and Geoscientists of British Columbia, (#51074).
4. I have practiced my profession continuously since my graduation in 1981. My relevant experience includes technical, operations and executive management for a major Canadian steel producer and providing consulting and project management services on iron ore and gold projects.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI 43 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I am responsible for the coordination, consolidation and review of this NI 43-101 Technical Report. I have also authored and am responsible for Chapters: 1 to 3, 17, 18 (except section regarding the TSF), 19 to 22, and 24 to 27. Chapters 1, 25, 26 and 27 had contributions from other QPs.
8. I have not personally visited the property.
9. I have had no prior involvement with the Property that is the subject of the Technical Report.
10. I have read National Instrument 43-101, Form 43-101F1 and the Technical Report has been prepared in compliance with this Instrument.
11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report and the parts that I am responsible for, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 29th day of September 2020.

"Original signed and sealed on file"

Angelo Grandillo, P. Eng., M. Eng.

1238 – 200 Granville Street
Vancouver, BC V6C 1S4

CERTIFICATE OF QUALIFIED PERSON

Ronald J Voordouw, Ph.D., P.Geo.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective as of September 09, 2020.

I, Ronald J Voordouw, Ph.D., P.Geo., do hereby certify that:

1. I am a Partner and Director of Geoscience of Equity Exploration Consultants Ltd., a mineral exploration management and consulting company with offices at 1238 – 200 Granville Street, Vancouver, British Columbia, V6C 1S4.
2. I am a graduate of University of Calgary (2000) with a Bachelor of Science degree in Geology and am a graduate of the Memorial University of Newfoundland (2006) with a Doctor of Philosophy degree in Geology.
3. I am a Professional Geologist in good standing with Engineers and Geoscientists of British Columbia (#50515) and the Professional Engineers and Geoscientists of Newfoundland and Labrador (#06962).
4. Since 2006, I have been involved with mineral exploration and research for precious and base metal deposits in Canada, South Africa and Brazil. I completed PhD and post-doctoral research projects focussed on or around Ni-Cu-PGE deposits in the Voisey's Bay and Bushveld camps, respectively, and was lead author on the 2018 NI 43-101 report on the Decar Property
5. I have read the definition of "Qualified Person" (QP) in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and according to NI 43-101 I am a qualified person owing to my education, experience and registration with professional associations.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I have authored and am responsible for chapters 4, 5, 6, 7, 8, 9, 10, 11, 12, and 23, and am partially responsible for chapters 1, 25 and 26.
8. I completed a site inspection on 21-23 September 2017 as part of writing the 2018 NI 43-101 report but have not visited the property since.
9. My only prior involvement with the Property was as a QP on the 2018 NI 43-101 report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed this 29th day of September 2019.

"Original signed and sealed on file"

Ronald J Voordouw, Ph.D., P.Geo.

GeoSim Services Inc
807 Geddes Road
Roberts Creek, BC, Canada V0N 2W6
(604) 803-7470

CERTIFICATE OF QUALIFIED PERSON

Ronald G. Simpson, P.Geo.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective as of September 09, 2020.

I, Ronald G. Simpson, P.Geo., do hereby certify that:

1. I am employed as a Professional Geoscientist with GeoSim Services Inc., 807 Geddes Road, Roberts Creek, British Columbia, Canada, V0N 2W6.
2. I graduated with a Bachelor of Science in Geology from the University of British Columbia, May 1975.
3. I am a Professional Geoscientist (19513) in good standing with the Association of Professional Engineers and Geoscientists of British Columbia
4. I have practiced my profession continuously for 45 years. I have been directly involved in mineral exploration, mine geology and resource estimation with practical experience from feasibility studies.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I am entirely responsible for Section 14 of the Technical Report and as a co-author for Sections 1, 25, 26 and 27.
8. I have not personally visited the property.
9. I have had prior involvement with the Property. I have previously co-authored a Technical Report on the Project titled "2018 Technical (NI 43-101) Report on the Decar Nickel Iron Alloy Project".
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with this Instrument.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report and the parts that I am responsible for, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 29th day of September 2020.

"Original signed and sealed on file"

Ronald G. Simpson, P.Geo.

Stantec

Stantec Consulting Ltd.
1100-111 Dunsmuir Street, Vancouver BC V6B 6A3

CERTIFICATE OF QUALIFIED PERSON

Gordon Chen, P. Eng.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective of September 9, 2020.

I, Gordon Chen, (Kuan Yu Chen), P. Eng., do hereby certify that:

1. I am currently employed as Senior Consultant – Mining Engineer by Stantec Consulting Ltd., 1100-111 Dunsmuir St., Vancouver, British Columbia, Canada, V6B 6A3.
2. I graduated with a Mining Engineer Bachelor of Science degree from University of British Columbia in 2007.
3. I am a member in-good-standing of the Engineers and Geoscientists of British Columbia (Member # 138756).
4. I have practiced my profession continuously since my graduation in 2007. My relevant experience includes technical and operational experience in surface mining projects for coal, gold and base metals.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI 43 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 16 and contributed portions of Section 18, 20, 21, 25 and 26 of the report.
8. I have not personally visited the property.
9. I have had no prior involvement with the Property.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.

Signed this 29th day of September 2020.

"Original signed and sealed on file"

Gordon Chen, P. Eng.

Stantec Consulting Ltd.
1100-111 Dunsmuir Street, Vancouver BC V6B 6A3

CERTIFICATE OF QUALIFIED PERSON

Sean Ennis, P. Eng.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective as of September 09, 2020.

I, Sean Ennis, P. Eng., do hereby certify that:

1. I am currently employed as Vice President, Mining with Stantec Consulting Ltd., 1100-111 Dunsmuir St., Vancouver, British Columbia, Canada, V6B 6A3.
2. I graduated with a Bachelor of Science degree in Mining Engineering from the University of Alberta in 1991 and with a Master's in Engineering Degree in Geo-Environmental Engineering from the University of Alberta in 1997.
3. I am a member in-good-standing of the Engineers and Geoscientists of British Columbia (Member # 24279).
4. I have worked as a Mining Engineer for 29 years. My experience has included working at coal, gold, base metals and oil sands operations. I have been involved in the evaluation of open pit projects, including coal, base, precious metals and industrial minerals projects, from the preliminary economic assessment through to feasibility level and provided operations support to projects. My experience includes the investigation, evaluation and design of mine tailings facilities for a range of commodities and includes tailings management methods for conventional slurry, thickened and filtered tailings deposits.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI 43 101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am independent of the issuer as described in Section 1.5 of NI 43-101.
7. I am responsible for the portions of Sections 18, 20, 21, 25 and 26 of the Report.
8. I have not personally visited the property.
9. I have had no prior involvement with the Property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Report, the omission to disclose which makes the Report misleading.

Signed this 29th day of September 2020.

"Original signed and sealed on file"

Sean Ennis, P. Eng.

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CERTIFICATE OF QUALIFIED PERSON

Jeffrey B. Austin, P. Eng.

This certificate applies to NI 43-101 Technical Report for the "Preliminary Economic Assessment of the Baptiste Nickel Project" prepared for the FPX Nickel Corp. issued on September 29, 2020 (the "Technical Report") and effective as of September 09, 2020.

I, Jeffrey B. Austin, P. Eng., do hereby certify that:

1. I am an independent consultant of International Metallurgical & Environmental Inc., located at 906 Fairway Crescent, Kelowna, B.C., and incorporated in 1995.
2. I graduated with a B.A.Sc. degree from the University of British Columbia in 1984.
3. I am a member, in good standing, of the Association of Professional Engineers and Geoscientists of British Columbia, license number 15708.
4. I have practiced my profession continuously for 33 years and have been involved in the design, evaluation and operation of mineral processing facilities during that time. A majority of my professional practice has been the completion of test work and test work supervision related to feasibility and pre-feasibility studies of projects involving flotation technologies.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Section 13 of the Technical Report.
8. I have not personally visited the property.
9. I have had no prior involvement with the Property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed this 29th day of September 2020.

"Original signed and sealed on file"

Jeffrey B. Austin, P. Eng.

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Abbreviation	**Description**
3-D	Three dimensional
AA	Atomic absorption
ABA	Acid base accounting
ACME	Acme Analytical Laboratories
Actlabs	Activation Laboratories Limited
Aeroquest	Aeroquest International Limited
Ai	Abrasion index
AIA	Archaeological impact assessment
AISC	All-in sustaining cost
ALS	ALS Metallurgy
AOA	Archaeological overview assessment
ARD	Acid rock drainage
ASL	Above seal level
ATV	Acoustic televiewer
Aw	Awaruite
Bandstra	Bandstra Transportation Systems Ltd.
BBA	BBA Inc.
BC	British Columbia (province in Canada)
BLK	Blank
BMP	Best management practice
BWi	Bond work index
CAN, CAD or C$	Canadian dollar
CAPEX	Capital expenditure
Caracle Creek	Caracle Creek International Consulting Incorporated
CCME	Canadian Council of Ministers of the Environment
CDC	Conservation Data Centre
CDE	Canadian Development Expenses
CEAA	Canadian Environmental Assessment Agency
CEE	Canadian Exploration Expenses
Champ Nav	Champ Navigator
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Cliffs	Cliffs Natural Resources Limited
CN, CNR	Canadian National Railway
CO_2	Carbon dioxide
COG	Cut-off grade
COSEWIC	Committee on the Status of Endangered Wildlife in Canada
CRM	Certified reference material

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
CV	Coefficient of variation
CVave	Average coefficient of variation
DCIP	Downhole induced polarization and resistivity
DDH	Diamond drill hole
DFO	Fisheries and Oceans Canada
DGI	DGI Geoscience Incorporated
DGPS	Differential GPS
DRI	Direct reduced iron
DRIPA	Declaration of the Rights of Indigenous Peoples Act (BC)
DTR	Davis Tube Recoverable
DTR Ni	Davis Tube recoverable nickel
DWT	Drop weight test
EA	Environmental assessment
EAC	Environmental Assessment Certificate
EAF	Electric arc furnace
EAO	Environmental Assessment Office
EBS	Environmental Baseline Study
ECCC	Environment and Climate Change Canada
EM	Electromagnetic
EMA	Environmental Management Act
EMP	Environmental management plan
EMPA	Electron microprobe analysis
EMPR	BC Ministry of Energy, Mines and Petroleum Resources
ENV	BC Ministry of Environment and Climate Change
EOH	End of hole
EPD	Environmental Protection Division
EPCM	Engineering, Procurement, Construction Management
EPP	Environmental protection plan
ERT	Emergency response team
ESG	Environment, social, and governance
ESSFmvp	Engelmann Spruce – Subalpine Fir (Moist Very Cold Parkland)
et al.	et alla (and others)
EV	Electric vehicle
EW	Electrowinning
F_{80}	80% passing - Feed size
Fe-Carb	Fe-Carbonate
FeCb_AltUM	Fe-Carbonate Altered Ultramafic

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
FeCb_Listwanite	Listwanite and Fe-Carbonate Altered Ultramafic
FeNi	Ferronickel
FIFO	Fly in fly out
FLNRORD	Ministry of Forests, Lands and Natural Resource Operations and Rural Development
FPX	FPX Nickel Corp.
FS	Feasibility study
FSR	Forestry service road
FW	Foot wall
G&A	General and Administration
GAT	Gravity amenability test
GEMS	Geovia GEMS software
GHG	Greenhouse gas
Gn	Non-Sulphide minerals
GNSS	Global navigation satellite system
GPS	Global positioning system
GSC	Geological Survey of Canada
HADD	Harmful alteration, disruption, and destruction
HPAL	High Pressure Acid Leach
HPGR	High-pressure grinding roll
HQ	HQ- Caliber drill hole
HVAC	Heating, ventilation, and air conditioning
HW	Hanging wall
IAAC	Impact Assessment Agency of Canada
IBA	Impact and Benefits Agreement
ICP	Inductively coupled plasma
ICP-OES	Inductively coupled plasma optic emission spectrometry
ID	Internal Diameter
ID^2	Inverse distance square
IEC	International Electrotechnical Commission
INS	Inertial Navigation System
I/O	Input/output
IP	Induced Polarization
IRR	Internal rate of return
ISO	International Standards Organization
IT	Information technology
K_{80}	80% passing – Particle size
Lidar	Light detection and ranging

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
LIMS	Low intensity magnetic separation
LME	London Metal Exchange
LOM	Life of mine
M	Million
m.a.s.l.	Metres above sea level
Ma	Magnetite
mA	Apparent chargeability
MA	Mines Act (BC)
MDMER	Metal and Diamond Mining Effluent Regulations
MDRC	Mine Development Review Committee
Mg	Magnesium
MIBC	Methyl isobutyl carbinol
ML	Metal leaching
MMU	Mobile manufacturing unit
MOU	Memorandum of Understanding
Mpa	Mega pascals
MTO	Mineral Titles Online
MTOs	Material take-offs
MVA	Mega volt ampere
NAD-83	North American Datum (1983)
NAG	Non-acid generating
NI 43-101	National Instrument 43-101
NN	Nearest-neighbour
No.	Number
NPI	Nickel pig iron
NPP	Navigation Protection Program
NPV	Net present value
NQ	NQ- Caliber drill hole
NSR	Net smelter return
NTS	National topographic system
OF, O/F	Overflow
OK	Ordinary kriging
OPEX	Operational expenditure
OTV	Optical televiewer
P_{80}	80% passing - Product size
PCS	Process Control System
PDC	Process Design Criteria

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
PEA	Preliminary economic assessment
Pe	Pentlandite
PFD	Process Flow Diagram
PFR	Preliminary field reconnaissance
PFS	Pre-Feasibility Study
PGE	Platinum group elements
pH	Potential of hydrogen
portable XRF	portable XRF Niton NLp 502 Analyzer
Prd_Min	Peridotite - Mineralized
Prd_MM	Peridotite – Massive
PSD	Particle size distribution
QA	Quality Assurance
QC	Quality Control
QEMSCAN	Quantitative evaluation of minerals by SEM
QP	Qualified person
Recoverable Ni	Nickel hosted in awaruite and sulphide minerals
RKEF	Rotary kiln – electric furnace
ROM	Run of mine
RQD	Rock quality designation
RTK	Real time kinematic
RWi	Rod work index
SAF	Submerged arc furnace
SAG	Semi-autogenous grinding
SARA	Species at Risk Act
SD	Standard deviation
SDM	Statutory decision maker
SEDAR	System for electronic document analysis and retrieval
SEM	Scanning electron microprobe
SFE	Shake flask extraction
SG	Specific gravity
SI	International System of units
SIPX	Sodium Isopropyl Xanthate
SMC	SAG mill comminution
SMU	Service meter unit
TEM	Terrestrial Ecosystem Mapping
Terra Remote	Terra Remote Sensing Incorporated
TMF	Tailings management facility

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
TMI	Total magnetic intensity
TOH	Top of hole
Total Ni	Nickel hosted in all minerals (i.e. awaruite, sulphide, oxide, silicate)
TSF	Tailings storage facility
TSS	Total suspended solids
USD or US$	United States dollar (examples of use: USD2.5M / US$2.5M)
U/F, UF	Underflow
UTM	Universal Transverse Mercator
VC	Valued components
VLF-EM	Very low frequency EM
VMS	Volcanogenic massive sulphide
VWPs	Vibrating wire piezometers
w/w	Weight per weight
WAAS	wide area augmentation system
Walcott	Peter E. Walcott & Associates Limited
WBS	Work breakdown structure
WQG	Water quality guideline
WRA	Whole rock analysis
wt%	Weight percent
XRD	X-ray diffraction
XRF	X-Ray fluorescence

TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Unit	**Description**
Metric	
deg. or °	angular degree
Cm	centimetre
m^3	cubic metre
m^3/h	cubic metres per hour
Mm^3	Cubic millimetre
d	day (24 hours)
°C	Degrees Celsius
$/t	Dollars per metric tonne
$/lb	Dollars per pound
G	Giga
g	gram
GWh	Gigawatt hour
g/cm^3	Grams per cubic metre
g/L	grams per Litre
g/y	grams per year
ha	hectare
h	hour (60 minutes)
kg	Kilogram
km	kilometres
km/h	Kilometres per hour
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kW	kilowatt
$kWhr/m^3$	kilowatt hour per cubic metre
kWhr/t	kilowatt hour per tonne
L	Litre
MW	Megawatt
m	metre
mA	milliamps
mg	Milligram
ml	Millilitre
µm	Micron or micro-meter
mm	millimetre

TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Unit	**Description**
Metric	
Mt	Million metric tonne
Mt/a	Million metric tonne per annum
min	minute (60 seconds)
Ω m	Ohm metre
ppb	part per billion
ppm	part per million
%	Percent
% solids	percent solids by weight
km^2	square kilometre
m^2	square metre
mm^2	square millimetres
K	Thousand (000)
t	tonne (1,000 kg) (metric ton)
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour
W	Watt
wt%	weight percent
y	year (365 days)

1. SUMMARY

1.1 Introduction

The following Technical Report (the "Report") presents a summary of the Preliminary Economic Assessment (PEA) conducted for the FPX Nickel Corp. (FPX) Baptiste Nickel Project (the "Project"), within the Decar Nickel District (the "District" or the "Property") in the Omineca Mining Division, British Columbia. In February 2020, FPX retained the services of BBA Inc. (BBA) to lead this Study. This Report was prepared at the request of Mr. Martin Turenne, President and CEO of FPX.

This Technical Report titled "Preliminary Economic Assessment of the Baptiste Nickel Project", concerning the development of the Baptiste Deposit, was prepared by Qualified Persons (QP) following the guidelines of the "Canadian Securities Administrators" National Instrument 43-101 (effective June 30, 2011), and in conformity with the 2014 guidelines of the Canadian Mining, Metallurgy and Petroleum (CIM) Standard on Mineral Resources and Reserves.

Since the current study is a PEA, NI 43-101 Guidelines do not permit the disclosure of Mineral Reserves. NI 43-101 Guidelines do allow for Inferred resources to be included in an economic analysis for a PEA, as long as the appropriate cautionary language is used to qualify such an analysis. Mineral resources which are not mineral reserves do not have demonstrated economic viability. It is reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. It should also be noted that a Preliminary Economic Assessment is preliminary in nature and there is no certainty that the Project described in this PEA Report will be realized.

1.1.1 Background and History

An initial Preliminary Economic Assessment on the Baptiste Project was completed in March 2013 and the NI 43-101 Technical Report was filed on Sedar in August 2013. The study was performed under the guidance of Cliffs Natural Resources (Cliffs), the previous majority owner of the Project. In November 2015, First Point Minerals Corp. (now FPX Nickel Corp.) purchased Cliffs' share of the Project and established 100% ownership of the Project.

In February 2018, FPX filed an updated Mineral Resource Estimate Technical Report following the completion of a 2,000-metre drilling program.

In September 2018, a metallurgical testwork program was initiated. The objectives of the program were to improve nickel recovery and product quality (compared to the 2013 PEA) using conventional mineral processing technologies.

1.1.2 Scope of Study

The Baptiste Project scope covered in this PEA is based on the construction of a facility having an annual throughput of 120,000 tpd or 43,800,000 tpa. Initially, during Phase 1 of the Project, covering Year 1 to Year 21 of operations, the run of mine (ROM) material is processed at a primary grind size P_{80} of 300 µm, as dictated by the tailings deposition strategy. In Project Phase 2, starting in Year 22 when in-pit tailings deposition is implemented, the primary grind size is reduced to a P_{80} of 170 µm in order to achieve improved Ni recoveries.

1.2 Tenure, Geological Setting and Exploration

The Property is situated approximately 90 km northwest of Fort St. James, BC (population 1,510) and is situated within the Decar Nickel District which consists of 62 mineral claims covering 24,740 hectares that are 100% owned by FPX. The Property is road accessible from Fort St. James via a network of province-maintained paved roads and forestry-maintained gravel roads. The Canadian National Railway company owns an inactive railway line that passes through the northeastern-most part of the Property.

The Property is underlain by bedrock of the Cache Creek terrane, which includes an obducted Upper Paleozoic and Lower Mesozoic ophiolite sequence referred to as the Trembleur ultramafite unit. Other rocks underlying the Property include metasedimentary and metavolcanic rocks of the Sitlika assemblage and Sowchea succession. Ultramafic rocks of the Trembleur unit are variably serpentinized and are host to multiple occurrences of disseminated awaruite on the Property. Awaruite is a nickel-iron alloy (formula $Ni_{2-3}Fe$) that forms under low oxygen and sulphur fugacity during serpentinization of nickeliferous olivine in peridotite. Awaruite is strongly ferro magnetic and exhibits a high density compared to associated gangue minerals that include serpentine, pyroxene, and magnetite.

The earliest publicly available reports of exploration on and around the Property date from 1974 and were focused on evaluating the potential of the area to contain chromite and gold-hosted listwanite mineralization. Awaruite was first discovered in the area as part of an academic thesis in 1983, followed by sporadic assessment through rock sampling and petrographic work between 1996-2005. In 2006, FPX staked 33 claims to establish the Property, focusing on exploration of awaruite mineralization in what are now the Baptiste Deposit and the Sid, Target B, and Van showings.

From 2006-2009, FPX conducted staking, property-scale airborne geophysics, prospect-scale ground-based induced polarization and resistivity (IP) surveys, rock sampling, geological mapping, petrography and scanning electron microprobe (SEM) analysis. This work identified Baptiste as the primary exploration target, with the Target B, Sid and Van targets also returning positive results.

On 12 November 2009, FPX entered into an option agreement with Cliffs Natural Resources Limited ("Cliffs") pursuant to which Cliffs could earn up to a 75% undivided interest in the Property.

During the option earn in, Cliffs incurred approximately US$22 million of expenditures, on or for the benefit of the Property, that culminated in a Preliminary Economic Assessment ("PEA") on the development of the Baptiste Deposit. Upon completion of the PEA, Cliffs secured a 60% undivided interest in the Property. In August 2014, Cliffs informed FPX that it would divest its 60% undivided interest in the Property and, on 8 September 2015, FPX announced that it had entered into a binding agreement with Cliffs to purchase Cliffs' 60% undivided interest in the Property for US$4.75 million. The repurchase of Cliffs' 60% was financed through a loan agreement that saw the lender receive a 1% NSR royalty. Following approval of the purchase by FPX shareholders on 15 November 2015, FPX re-acquired 100% ownership of the Property subject to a 1% NSR effective 18 November 2015.

Work funded by Cliffs on the Property included drilling a total of 30,223 m in 80 holes. Drilling was completed predominantly on the Baptiste Deposit (27,670 m), with lesser drilling on the Sid Target (847 m) and Target B (305 m). Drilling of 1,401 m of hydrogeological monitoring wells at the Baptiste Deposit was also completed to help inform an environmental baseline study. Drilling results yielded a maiden mineral resource estimate for the Baptiste Deposit published on 25 May 2012, an updated mineral resource estimate for the Baptiste Deposit on 23 January 2013 which later formed the basis of a PEA on 22 March 2013. Other work undertaken by or on behalf of Cliffs included downhole geophysical rock property surveys as well as mineral processing and metallurgical testing.

The sample database supplied for the Baptiste Deposit contains results from 83 surface drillholes completed since 2010, or 96% of all metres drilled on the Property. In comparison to the 2013 resource estimate (Ronacher et al., 2013), the 2020 resource estimate incorporates an additional eight diamond drillholes (totaling 1,917 metres) completed during the summer of 2017, one hole drilled during the 2012 drilling campaign (which was not included in the 2013 resource estimate), and an additional 2,053 samples from core re-sampling of 2010 and 2011 drillholes completed in 2012. The average drillhole spacing in the Baptiste Deposit is 150 metres to a vertical depth of 540 metres.

1.3 Mineral Resource Estimate

An updated resource estimate for the Baptiste Deposit includes all data from the 83 surface drillholes completed since 2010 and 2,053 samples from a re-sampling program of 2010/2011 drill core that was carried out in 2012. The estimate is geologically constrained within four mineralized domains and is reasonably comparable among different estimation methods (i.e. ordinary kriging, inverse distance squared weighting, nearest neighbour).

The 2018 resource model comprises a large, delta shaped volume that measures approximately 3.0 km in length and 150 to 1,080 m in width and extends to a depth of 540 m below the surface. The Baptiste Deposit remains open at depth over the entire system and is covered by an average of 12 metres of overburden.

Table 1-1 presents the mineral resource estimate for the Baptiste Deposit at a range of cut-off grades with the base case, at a cut-off grade of 0.06% DTR Ni, in bold face.

Table 1-1: Indicated and inferred resources for the Baptiste Deposit

INDICATED				INFERRED			
% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)	% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)
0.02	2,076,969	0.119	2,471,593	0.02	750,633	0.098	735,620
0.04	2,055,578	0.120	2,466,694	0.04	659,900	0.107	706,093
0.06	**1,995,873**	**0.122**	**2,434,965**	**0.06**	**592,890**	**0.114**	**675,895**
0.08	1,871,412	0.126	2,357,979	0.08	499,993	0.122	609,991
0.10	1,617,364	0.131	2,118,747	0.10	399,801	0.130	519,741

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. using ordinary kriging with an effective date of September 9, 2020.
2. Indicated mineral resources are drilled on approximate 200 x 200 metre drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximate 300 x 300 metre drill spacing.
3. An optimized pit shell was generated using the following assumptions: US$6.35 per pound nickel Price; a 45° pit slope; assumed mining recovery of 97% DTR Ni and process recovery of 85% DTR Ni, an exchange rate of $1.00 CAN = $0.76 US; and mining costs of US$2.75 per tonne, processing costs of US$4.00 per tonne. A US$1.00 per tonne minimum profit was also imposed to exclude material close to the break-even cut-off.
4. A base case cut-off grade of 0.06% DTR Ni represents an in-situ metal value of approximately US$7.00 per tonne which is believed to provide a reasonable margin over operating and sustaining costs for open-pit mining and processing.
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.4 Mining Methods

A conceptual mine plan was developed in this PEA, based on the mineral resource estimate and its underlying geological block model. The mine plan is based on a three-phase open pit mine development. The first phase of the proposed mine plan uses an external tailings storage facility (TSF) for disposing of tailings generated while mining the Phase 1 pit which will be mined for the first 21 years. After the Phase 1 pit is mined out, the tailings produced from Phase 2 and Phase 3 of the mine plan will be placed in the mined-out Phase 1 pit. A pit rim dam will be constructed in Year 25 to accommodate the additional tailings that will be stored in the Phase 1 and Phase 2 pits. The Phase 2 and Phase 3 pits will be mined from Year 22 to Year 35.

Mining will be conducted using conventional truck and shovel methods. Large-scale open pit mining will provide the mineral processing plant feed at a rate of 120,000 t/d, or 43.8 Mt/a which was based on processing capacity inputs provided. Annual mine production of mill feed and waste will peak at 80.1 Mt/a with a life-of-mine (LOM) stripping ratio of 0.40:1 including preproduction (0.32 during the first 10 years of operation, and 0.22 over the first 16 years of operations). Ultimate pit quantities with corresponding Davis Tube Recoverable (DTR) Ni grades are shown in Table 1-2.

Table 1-2: Ultimate Design Pit Quantities

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	1,326	0.124%
Inferred	177	0.102%
Total for processing	**1,503**	**0.121%**
Waste Rock	540	
Overburden	55	
Total Waste	**596**	
Total material mined	**2,098**	
Note**:** Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

1.5 Metallurgy and Mineral Processing

The metallurgical testwork performed for this PEA was focused on the following:

- Magnetic separation tests at a range of primary grind sizes (P_{80} from 57 µm to 360 µm);
- Magnetic cleaning tests to 25 µm final regrind size;
- Flotation testwork on the magnetic cleaner concentrate under various conditions and reagent additions;
- Mineralogical assessment of the head sample and some products generated in the testwork.

The testwork was performed on a composite sample representing the first seven years of the mine life. A limited amount of grindability data was generated on this sample and grinding circuit conceptual design was largely based on assumptions derived from similar minerals.

The testwork results indicated that at a primary grind of 300 µm, it is possible to produce a 63.4% Ni concentrate with a DTR NI recovery of 84.7%. In Year 22, when in-pit tailings deposition is implemented, a finer primary grind of 170 µm can be achieved through the addition of a third ball mill resulting in a Ni recovery of 89.5%. A conceptual mineral processing flowsheet was developed as the basis for this PEA Study. The process flowsheet can be summarized as follows:

- ROM material crushing takes place in a single primary gyratory crusher located in the vicinity of the open pit;
- Secondary crushing takes place in two cone crushers operating in closed circuit with classification screens;
- Crushed material is stored in a stockpile;
- The crushed material from the stockpile is reclaimed and conveyed to two separate and independent processing lines starting with a high-pressure grinding roll (HPGR) circuit operating in closed circuit with classification screens;

- The screened product from the HPGR circuit is subsequently conveyed to a primary grinding ball mill circuit, in closed circuit with hydrocyclones where the product is ground to a P_{80} of 300 µm;
- The O/F from the hydrocyclones is directed to a rougher / scavenger low intensity magnetic separation (LIMS) circuit and the products are treated as follows:
  - The LIMS magnetic concentrate is directed to an open-circuit vertical stirred mill regrind circuit to generate a product at P80 of 25 µm and,
  - The LIMS non-magnetic tailings are pumped to hydrocyclone clusters where coarse sand tailings (hydrocyclone U/F) are produced and ultimately pumped to the TSF for sand cell dike construction whereas the fine tailings fraction (hydrocyclone O/F) are directed to the tailings thickener;
- The reground product from the vertical stirred mills circuit is then processed through a cleaner magnetic separation LIMS circuit whereby:
  - The LIMS fine magnetic concentrate from the two processing lines is directed to the flotation circuit and,
  - The LIMS fine non-magnetic tailings are directed to the tailings thickener;
- The awaruite flotation circuit produces:
  - A high-grade Ni concentrate that is dewatered using dewatering LIMS, filtered to a filter cake and briquetted into the final saleable product;
  - A magnetite rich tailings stream having the potential to be subsequently valorized as a saleable product that is dewatered using dewatering LIMS and, for this PEA, assumed to be disposed of as tailings;
  - The water from the dewatering LIMS is recirculated to the head of the flotation circuit.
- The U/F from the tailings thickener, consisting of fine tailings, is pumped to the TSF for disposal.

1.6 Project Infrastructure

Project infrastructure is classified as off-site and on-site. The three major elements of off-site infrastructure are:

- The site access road, having a total length of 121 km, consisting of an existing paved road segment and an existing forestry service road (FSR) requiring a new 110-m span bridge, a new 4.5 km FSR segment and upgrades to a 20-m span bridge and to about 11.5 km of existing FSR;
- A rail terminal to be constructed in the vicinity of the existing CN main rail line in Fort St. James used exclusively for transloading containerized FeNi briquettes onto railcars for transportation to the Prince Rupert port terminal and for receiving, storing and transloading rail tankers containing sulphuric acid used in the flotation process at the mine site;
- A 120 MW, 230 kV power transmission line with an approximate length of 98 km.

A conceptual site plot plan was developed for this PEA showing the location of the major on-site infrastructure. The ultimate footprints for the open pit, the waste piles, the external TSF and the pit rim dam were first incorporated into the site plan and the rest of the major site infrastructure was then located. The main site infrastructure is summarized as follows:

- The ultimate life-of-mine open pit footprint for the Baptiste Deposit, as developed by Stantec;
- The waste dumps designed to store waste materials generated during the mine life;
- The external TSF, serving for Years 1 to 21 of the mine life, as conceptually designed by Stantec;
- The mine services area pad, which includes the mine garage, truck wash, warehouse and mine employee facilities as well as the emergency response team facility;
- The explosives plant pad is in an isolated area to the east of the open pit;
- The primary crusher building and pad, is in a designated area in the vicinity of the initial open pit ramp exit;
- The secondary crushing and screening, crushed material stockpile and HPGR and screening buildings, all connected by a conveyor system, are located within a cleared corridor (incorporating an access road, power line and other services) connecting these facilities to the concentrator area;
- The main process plant area pad includes the concentrator building housing all mineral processing equipment and FeNi briquette storage and the thickeners as well as the main 230 kV electrical substation;
- The fresh water pumphouse is located at Trembleur Lake, about 7 km to the south of the concentrator;
- Mine road network dedicated to heavy traffic and controlled for other vehicles;
- Site road network for general infrastructure access and access to the FSR;
- The camp for employees and administration building pad is located to the southeast of the concentrator, on the FSR road accessing the Property.

1.7 Market Study

Metallurgical testwork performed for this PEA Study has shown that the Baptiste Deposit can produce a clean, high-grade, ferronickel (FeNi) concentrate through a conventional mineral processing flowsheet. The concentrate, agglomerated in briquette form, constitutes the final saleable product generated by the Project for consumption by stainless steel producers. Preliminary tests have also shown that the Baptiste FeNi concentrate can potentially be used to produce nickel sulphate for the electric vehicle battery value chain. The projected product specification for the Baptiste briquettes is presented in Table 1-3.

Table 1-3: Projected product specification for the Baptiste FeNi briquetted concentrate

Projected Product Specification	
Ni	60% - 65%
Fe (total)	30% - 32%
Awaruite (Ni_3Fe metallic alloy)	77% - 83%
Metallic Fe in Awaruite	19% - 21%
Magnetite (Fe_3O_4)	13% - 18%
Co	1% typical
Cu	0.7% typical
P	0.02% typical
S	0.6% typical
MgO	1% typical
SiO_2	1.5% typical
Cr_2O_3	0.4% typical

The selling price to be obtained from the sale of the Baptiste FeNi briquette to stainless steel melt shops will generally be a function of two variables: 1) the LME nickel price; and 2) a discount or premium to the LME nickel price, based on the market positioning of the Baptiste FeNi briquette in relation to competing sources of nickel feedstock to stainless producers, being primarily stainless steel scrap, nickel pig iron ("NPI"), standard FeNi and Class 1 Ni. The selling price determined by the analysis of these two components is the price used for the Economic Analysis performed for this PEA Study.

FPX provided long term projected Ni price data published by several reputable analysts. The most current update of this data is dated August 2020. A long-term LME base nickel price assumption of $17,070 per tonne ($7.75 per pound) is assumed in this PEA Study which is consistent with the average long-term nickel price of forecasts given by six base metals analysts.

In order to assess the potential payability for the Baptiste FeNi product, stated as a % of the LME base price, the following sources of information were considered:

- The results of the FPX's preliminary product market testing undertaken with stainless steel and ferronickel producers;
- Preliminary market feedback based on informal discussions with nickel consumers and traders, including an independent consultant to FPX and representatives of large international trading houses specializing in nickel products;
- Benchmarking with typical specifications for standard FeNi and nickel pig iron ("NPI") products from various producers;
- The author's technical knowledge of the steelmaking process based on his over 20 years of experience in a steel smelter;
- Historic premium / discount data for standard FeNi.

The analysis, in consideration of the aforementioned information sources, concluded that a discount of 2% applied to the base LME price provides a reasonable assumption for determining the selling price to be used for the Economic Analysis for this PEA Study. The assumed selling price is $16,743/t of contained Ni.

1.8 Environmental, Permitting and Community Relations

1.8.1 Provincial Regulatory Framework

In British Columbia, reviewable mining projects must attain an Environmental Assessment Certificate (EAC) prior to obtaining the required construction and operating permits. As a principal planning tool, reviewable projects are subject to the revitalized BC Environmental Assessment Act of 2018, currently in force, using a phased approach with imposed regulatory timelines. The EA is generally structured as follows:

- Identification and assessment of potential environmental, social, economic, cultural, and health impacts;
- Development of an acceptable scope and methodology for conducting the effects assessment of a selection of valued components (VCs);
- Characterization of residual effects potential for VCs after avoidance, mitigation measures, standard best management practices (BMPs) and monitoring programs are implemented;
- Prediction of the likelihood of significant residual effects occurring;
- Development of acceptable compensation measures to offset residual effects and maintain compliance with provincial and federal regulatory requirements as well as to effectively accommodate adversely affected Indigenous groups;
- Participation in Crown consultation proceedings to provide opportunities for Indigenous, federal, provincial, and local governments, stakeholders, special interest groups, and members of the public to learn about the Project, identify potential issues, provide input to potential avoidance/mitigation measures, and accommodate any infringement of Indigenous title and rights;
- Incorporate economic, social, cultural, health, and environmental factors into proponent and government decision making processes.

Key phases of the EA process in BC include:

- Early engagement phase (minimum of 90 days) occurring prior to submitting the detailed Project Description (s. 13 and s. 15 of the Act) and designed to attain consensus among participating Indigenous groups. It includes alternative dispute resolution options and leads to a Summary of Engagement and the Detailed Project Description.

- Remaining phases required to obtain an EA certificate include:
  - EA Readiness phase and decision (s. 16(2), s. 17 or s. 18; 60 days minimum, but timeline is variable);
  - Process Planning phase (120 days);
  - Application Development and Review phase (minimum of 180 days) and submission of final application;
  - Effects Assessment and Recommendation phases (150 days maximum); and
  - Decision phase (30 days maximum).

The project will be bound by the conditions of the EAC. Post-certificate activities include mitigation effectiveness reports and may include audits, certificate amendments, extensions, and transfers.

1.8.2 Federal Regulatory Framework

The EA process also takes place under the Impact Assessment Act, 2019 for federally designated projects. Projects that are not designated federally may still require a screening in coordination with the provincial EA process.

Government agencies responsible for coordinating the EA processes include the BC Environmental Assessment Office (EAO) and the Impact Assessment Agency of Canada (IAAC).

The project will likely require federal authorization by Fisheries and Oceans Canada (DFO) under paragraph 35(2)(b) of the Fisheries Act to commit harmful alteration, disruption, and destruction (HADD) of fish habitat or paragraph 34.4(2)(b) for any death of fish. An authorization will be issued by the Minister under the new paragraphs in the amended Act (2019). Habitat and/or productivity offsetting requirements must be acceptable to DFO and seek to accommodate participating Indigenous groups.

Additional EBS work, including the Terrestrial Ecosystems Mapping and determination of the extent of fish habitat, wetlands, and presence listed species and ecosystems will provide information for federal regulators to consider in the EA process.

1.8.3 Permitting

If a project receives an EAC, the proponent must obtain the required authorizations for activities such as water use, timber cutting, access roads, stream crossings, and other mine-related permits. In general, provincial and federal EA processes are finalized before permits can be issued.

1.8.4 Closure and Reclamation

BC Ministry of Energy, Mines and Petroleum Resources (EMPR) will provide the regulatory framework for FPX's obligations for decommissioning, closure, reclamation and rehabilitation for the Project. Acceptable practices will result from effective Crown consultation, ongoing engagement with indigenous groups, and effective planning throughout the EA process.

The Mines Act permit requires a closure plan with the appropriate reclamation security paid. Annual reclamation reports are filed with EMPR as a permit condition under the Health, Safety and Reclamation Code for Mines in BC. The security is collected upon initial permit issuance and adjusted through operational lifespan of the mine to accurately cover the cost of the liability. It must be acceptable to the Mines Inspector determining the appropriate bond amount over time.

A Regional Reclamation Bond Calculator provides the Regional Inspector with a means of assessing reclamation liability that avoids undue financial risk and liability to the public. The assessed security represents the cost of mine reclamation to the Province while promoting transparency to Indigenous groups and the public. The security is returned once the mine site has been reclaimed to a satisfactory level and no longer requires monitoring or maintenance. It ensures that mine sites do not leave an ongoing legacy or require public funds for clean-up activities.

FPX's mine closure and reclamation plan will aim to reclaim and rehabilitate the Project footprint to ensure that, upon termination of mining, land, watercourses and cultural heritage resources will be returned to a safe and environmentally sound condition and to an acceptable end land use that considers previous and potential uses.

1.8.5 Community Relations

Local perspectives and opinions are critical to FPX's decision-making process throughout all aspects of the Project and are an integral part of ongoing consultation and engagement with local communities and First Nations. Enduring relationships must be built with Indigenous groups and community stakeholders. It is based on trust: transparency, accountability, mutual understanding and respect for rights and title, continuous active engagement, and a long-term commitment to shared value.

The Project lies exclusively in the traditional territory of the Tl'azt'en Nation, which has been the focus of Indigenous engagement and consultation activities to date. FPX signed an exploration Memorandum of Understanding (MOU) for the Project on May 22, 2012. On March 12, 2019, Binche Whut'en was constituted as a newly recognized First Nation by the Canadian government, officially separating from the Tl'azt'en Nation. Of the four Keyoh families who are signatories to the MOU, two are associated with Tl'azt'en Nation, and two are associated with the newly-formed Binche Whut'en Nation. FPX is engaged in discussions with Tl'azt'en Nation, Binche Whut'en and the four constituent Keyoh families to amend the MOU to reflect the new administrative structure entailed by the separation of Binche Whut'en from Tl'azt'en Nation.

The MOU formalizes protocols for continuing the cooperative working relationship established between the Tl'azt'en Nation and the Binche Whut'en Nation, including constituent Keyoh families, regarding exploration activities for the Project. The MOU confirms the Tl'azt'en Nation's and Binche Whut'en Nation's support for the exploration activities and acknowledges, as well as describes, how project activities will be managed with respect to:

- Cultural and environmental interests of the Tl'azt'en and Binche Whut'en communities;
- Ongoing engagement and internal community consultation activities;
- Socio-economic benefits to the Tl'azt'en Nation and the Binche Whut'en Nation communities through community contribution funds and business opportunities.

The MOU also establishes processes for the future negotiation of a comprehensive Impact and Benefits Agreement (IBA) for when the Project proceeds to mine development. This IBA emphasizes mutual respect and positive long-term relationship between the parties during all phases of the Project.

1.9 Capital and Operating Costs

Capital, sustaining capital and operating cost estimates were developed for this PEA Study. Costs relating to mineral processing, site infrastructure, G&A and product handling and transport were developed by BBA. Costs relating to mining, the TSF and surface water management were developed by Stantec. All costs are presented in USD. The exchange rate used is $1.00 CAN = $0.76 US. The base date for the cost estimate is Q3-2020. Table 1-4 presents a summary of the estimated capital cost for initial pre-production, in-pit tailings deposition and sustaining capital.

Table 1-4: Summary of capital cost estimate (US$)

Category	Pre-Production M$	In-Pit Deposition M$	Sustaining M$	Total LOM M$
Direct Costs				
Mobile Equipment	$155.1	$0.0	$353.5	$508.6
TSF	$137.9	$14.5	$534.3	$686.6
Mine and TSF Site Preparation	$95.5	$0.0	$90.4	$185.9
Mineral Processing	$610.0	$88.4	$18.2	$716.6
Off-Site Infrastructure	$64.4	$0.0	$0.0	$64.4
On-Site Infrastructure	$66.4	$0.0	$6.8	$73.2
Total Direct Costs	**$1 129.3**	**$102.9**	**$1 003.2**	**$2 235.4**
Indirect Costs	$291.8	$0.0	$8.2	$300.1
Contingency	$253.7	$0.0	$0.0	$253.7
TOTAL PROJECT CAPITAL COST	**$1 674.8**	**$102.9**	**$1 011.5**	**$2 789.2**

The total initial capital cost, including direct costs, indirect costs and contingency was estimated at **$1,674.8M**. This represents the pre-production capital expenditure required to support start-up of operations in Year 1. The capital cost related to the in-pit tailings deposition implementation was estimated at $102.9M. This is the capital expenditure specifically required to allow for finer primary grinding (resulting in improved Ni recovery) and for pumping tailings to the mined-out pits for in-pit deposition, starting in Year 22 of the mine life. This cost also includes the cost for constructing the pit rim dike for containing tailings to the end of the mine life. Sustaining capital costs (which excludes the capital cost related to the implementation of finer primary grinding and in-pit deposition) were estimated at $1,011.5M. These costs include items such as mine equipment fleet additions and replacements, facilities additions and improvements and costs relating to TSF sand cell construction and surface water management which are incurred over the LOM starting at Year 1 of operation. It should be noted that closure and reclamation costs are excluded from the stated capital costs but are included as a separate item in the Economic Analysis.

Table 1-5 presents a summary of the estimated average operating costs for the initial Phase 1 (Years 1 to 21), Phase 2 (Years 22 to 35) and for the Life-of-mine (LOM), expressed in USD/t of dry material processed (milled). Averages include ramp-up years (Year 1 and Year 22).

Table 1-5: Total estimated phase and average LOM operating cost (US$/t milled)

Estimated Average LOM OPEX	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
Mining	$2.28	$2.66	$2.43
Mineral Processing	$2.71	$2.91	$2.79
Briquette Transport	$0.19	$0.18	$0.19
Rail Terminal and Access Road	$0.05	$0.05	$0.05
General Site Services	$0.62	$0.62	$0.62
General and Administration	$0.25	$0.25	$0.25
TOTAL Opex	**$6.09**	**$6.66**	**$6.32**

It should be noted that royalties and working capital are not included in the operating cost estimate presented but are treated separately in the Economic Analysis.

Table 1-6 presents additional metrics of costs incurred annually over the operating years of the mine and include the following:

- ‘C1’ cost defined as follows: “The costs of mining, milling and concentrating, onsite administration and general expenses, property and production royalties not related to revenues or profits, metal product treatment charges, and freight and marketing costs less the net value of by-product credits, if any. These are expressed on the basis of ‘per unit Ni content’ of the sold product.”
- ‘AISC’ or ‘all-in sustaining costs’ defined as follows: “These costs comprise the sum of C1 costs, sustaining capital, royalties and closure expenses. These are expressed on the basis of ‘per unit Ni content’ of the sold product.”

Table 1-6: C1 costs and AISC costs (US$)

	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
C1 costs ($/lb Ni)	$2.61	$2.94	$2.74
C1 cost ($/metric tonne Ni)	$5,753	$6,488	$6,038
AISC cost ($/lb Ni)	$3.13	$3.11	$3.12
AISC cost ($/metric tonne Ni)	$6,897	$6,867	$6 885

1.10 Economic Analysis

Table 1-7 presents the results of the Economic Analysis for the Project. Taxation calculations were provided by FPX. Table 1-8 presents the results of the post-tax economic analysis.

Table 1-7: Economic analysis results (pre-tax) (US$)

IRR = 22.5% Payback = 3.5 years Discount Rate	NPV (M$)
0%	$13,656 M
5%	$5,003 M
8%	**$2,927 M**
10%	$2,069 M

Table 1-8: Economic analysis results (post-tax) (US$)

IRR = 18.3% Payback = 4.0 years Discount Rate	NPV (M$)
0%	$8,725 M
5%	$3,091 M
8%	**$1,721 M**
10%	$1,149 M

1.11 Other Relevant Information

1.11.1 Opportunity for FeNi in EV Battery Application

As part of its product development program, FPX mandated Sherritt Technologies to perform scoping-level, batch pressure leach tests on a sample of Baptiste FeNi flotation product. Two batch pressure leach tests were conducted in an autoclave (pressure chamber) with conditions designed to approximate proposed commercial production conditions.

The quality of the nickel chemical solution generated from the batch tests was excellent, with the low acid and iron content indicating relatively low downstream requirements for neutralization and iron removal. Recovery of nickel was rapid in the batch tests, with over 98% extraction achieved in 60 minutes toward ultimate extractions of 98.8% and 99.5% in 180 minutes. The iron (Fe) in the concentrate feedstock was almost entirely precipitated, resulting in low iron content in the pregnant leach solution.

It is expected that the nickel-cobalt solution produced from the Baptiste concentrate will be an ideal feedstock for the production of nickel sulphate and cobalt sulphate. Downstream processing of the Baptiste nickel-cobalt solution would conceptually entail neutralization (to remove acid and other impurities) and solvent extraction to produce nickel sulphate and cobalt sulphate as two separate products. The low levels of impurities (such as acid and iron) in the Baptiste nickel-cobalt solution suggest that downstream refinement into sulphate products is achievable within conventional operating parameters. Additional test work is required to further evaluate the optimization of any downstream hydrometallurgical processing requirements.

This positive result provides FPX with an opportunity to pursue an alternative marketing route for part of its Baptiste FeNi production. This would allow FPX to become a player in the EV battery value chain. As the Project advances, this opportunity would need to be supported with more testwork and a validation of process economics.

1.11.2 Opportunity for Sale of Iron Concentrate

The process flowsheet developed in this PEA Study generates a flotation tailing with a high Fe content (in the form of magnetite), which can potentially be marketable as a magnetite iron ore concentrate and generate additional financial benefit to the project. The annual production of this co-product would be in the order of 2.0 Mt. A high-level assessment to evaluate the economic viability of this product was performed based on various scenarios for adding value to the product to improve its marketability.

The 'as-is' flotation co-product has a total Fe content of about 61%. The product contains a relatively high level of MgO and SiO_2 as well as a high level of Cr. Testwork will need to be performed to evaluate mineral processing requirements to reduce MgO and SiO_2 levels. The major constraint with this Fe concentrate is its high Cr content. Initial indications are that Cr may not be amenable to removal by mineral processing. If this is shown to be the case, this concentrate may be used in the carbon steel value chain but only in limited quantities and possibly with payability discounts. This concentrate, however, should be usable in the stainless steel value chain, which is a much smaller market than that of carbon steel. It should be noted that the concentrate cannot be used directly by steel smelters and will need a reduction step to make an intermediate iron product (blast furnace or direct reduction). As a next step, a more detailed logistics and marketability analysis to further develop this opportunity would be required.

1.11.3 Project Execution Plan

Following this PEA Study for the Baptiste Project, it is expected that FPX will proceed with a pre-feasibility study of the Baptiste Deposit to further develop the concepts presented in this PEA Study and to further de-risk the project. This should logically then be followed by a feasibility study. Concurrently, FPX will also need to proceed with environmental assessment, studies, community relations development and other permitting activities. This will also likely be complemented by additional infill drilling, metallurgical sampling and testwork and geotechnical surveys. A duration of about 2.5 years should be planned for these activities, to completion of the feasibility study, after which FPX can proceed directly to detailed engineering procurement and construction. After award of permits, about three years will be required to complete engineering, procurement and construction and for commercial production to begin.

1.12 Conclusions and Recommendations

The Project, as presented in this PEA Report, is conceptual in nature and needs to be further developed at a PFS level.

1.12.1 Project Risks

A high-level risk register was initiated during this PEA for FPX to use as an internal planning and risk management tool to be carried through and developed in more detail during the next study stages. The following list describes the key risks that were identified which can have a significant impact on the results of this PEA Study. Recommendations for mitigating these risks are proposed in Chapter 26 of this Report.

- Designs of the external TSF, pit rim dike, waste dumps, open pit mine, surface water management infrastructure and process plant areas were not supported by any foundation geotechnical data. For this PEA, typical dike slopes for engineered fill were assumed, with a nominal sub-excavation, but poor foundation conditions could significantly increase cost or cause the relocation of the structure. Conversely, good foundation conditions could lead to reduced excavation and fill requirements and therefore reduced costs.
- There is currently no geotechnical data for the pit area. For the PEA, an overall pit wall angle of 45 degrees is assumed for the ultimate pits which may need to be adjusted based on the results of the geotechnical data. The change in pit wall angle will have an impact to the overall costs, footprint area, and strip ratio.
- Borrow material for the construction of the external TSF starter dike and the pit rim dike were assumed to be sourced from within the open pit mine. This study assumed that 80% of the material excavated from the pit (overburden and waste rock) is be suitable for TSF starter dike construction. Should more material (or less material) be suitable, development costs will be impacted.
- Coarse sand generated from cycloning the tailings produced by the primary magnetic separation circuit (300 µm primary grind size), was assumed to be of sufficient quantity and quality to be used in the construction of the external TSF sand cells. Should this material not be suitable, the TSF retention capacity and/or the primary grind size could be impacted resulting in higher costs and lower Ni recovery if a coarser grind is required.

- The process design criteria for grinding and primary magnetic concentration at 300 µm was based on limited testwork data. Ore hardness is a critical parameter for estimating grinding power and circuit design which can have a significant impact on throughput, capital and operating costs.
- The conceptual project execution schedule presented in Chapter 24 of this Report was based on an optimized environmental permitting process based on minimum statutory timelines. Should there be delays in compiling data or permit related activities take longer than planned, there is a risk that environmental permitting will be on the critical path of the schedule.

1.12.2 Recommendations

Based on the results of this PEA Study, BBA recommends that a Pre-Feasibility Study (PFS) be undertaken on the Baptiste Project in order to advance the Project to its next phase. The proposed PFS would serve as a stage-gate for FPX to determine if the Project should be subsequently advanced further. The PFS is intended to confirm the results of this PEA, to a higher level of resource definition and cost estimation accuracy, supported by more developed engineering and design and so de-risking the Project to the next study stage. The following recommendations are made for work to be undertaken as part of the PFS Study in order reduce Project uncertainty and to mitigate risks, as well as to evaluate opportunities to improve the Project. Work related to environmental permitting is also recommended in parallel to the PFS in order to maintain the targeted overall project implementation schedule.

Geology and Drilling

Database maintenance: Review of the Baptiste Project drilling database for the purposes of this report recognized that the QA/QC and re-assay results for drill core have not been incorporated into the Project's database and exist as stand-alone spreadsheets. The recommended improvements to the database include rebuilding the Project's drillhole database from original assay certificates and incorporating existing QA/QC data into the Project's drillhole database.

QA/QC: Assays of CRMs used on the Project averaged 0.5 to 1 standard deviation higher than their certified mean, suggesting that calculated DTR Ni grades for the Baptiste Deposit could have inaccuracies of up to 4%. A re-assay program is recommended to evaluate positive bias seen in CRM assays in addition to determining the data adequacy of DTR Fe and Co for future use in resource estimates.

Drilling: Additional drilling on the Property is recommended for the Baptiste Deposit to improve the certainty of near surface Inferred Resources greater than 0.13% DTR Ni. A total of 2,725 m of drilling on ten holes is recommended.

Metallurgical Testwork

Grindability: Specific testwork on representative composite samples related to grindability (crushing, HPGR, ball milling and regrinding) should be performed.

Concentration: Magnetic concentration and flotation testwork should be undertaken on representative composite samples in order to confirm grade/recovery relationship. This should include testwork to confirm DTR Ni recovery at various primary grinds with varying magnetic intensity. Magnetic concentration testwork should be performed with a complete DTR Ni and Fe balance. Also, an analysis of primary magnetic tailings PSD and characteristics of sand tailings and fine tailings (compaction, hydraulic conductivity, strength parameters, ML/ARD potential) should be undertaken in order to confirm TSF sand dike design parameters.

Leaching: More detailed leaching tests should be performed to evaluate metallurgical performance and costs related to production of Ni sulphate.

Additional testwork: Thickening, rheology and filtration testwork should be undertaken.

Variability assessment: To assess the deposit's heterogeneity, variability sampling should be completed. Approximately 25 samples are recommended to assess potential variability within the Baptiste Deposit with respect to crushing, grinding and recovery.

Product marketability sample: The metallurgical testwork should be undertaken in consideration that a 5 to 10 kg representative final FeNi product sample needs to be generated. This will allow FPX to undertake more meaningful and credible discussions with potential users.

Geotechnical

Site surveys: A test pit / geotechnical drilling program is recommended to be carried out in the area of the external TSF, waste dumps, pit, water management dams and area surrounding the pit to collect geotechnical data in order to mitigate risks associated with lack of data on foundation conditions.

Pit slope stability: There is currently no geotechnical and hydrogeology data for the pit area. The projected pit walls are very high (700 m) and therefore small changes in angle could have a significant impact on the mine plan. Core logging and characterization is required to allow for development of rock mass strength models and development of pit wall designs including bench configurations. The hydrogeology data will help determine the amount of ground water the mine will need to manage during operation as well as potential requirements for dewatering related to wall stability. Geotechnical and hydrological information on the pit walls will be required for the PFS.

Other: Geotechnical data from the pit and tailings footprint areas will be required to more accurately determine borrow material suitability for construction of the TSF starter dam and other site infrastructure. Also, additional surveys for site infrastructure such as process plant location and roads are recommended.

Environmental, Permitting and Community

Baseline studies: Further environmental baseline studies should be completed, as recommended in Chapter 20 of this Report. This activity is a critical element of the EA and permitting process and project implementation schedule.

EA planning: A gap analysis (current status vs final requirements) should be performed and a comprehensive action plan should be developed to undertake the permitting process in parallel to the PFS.

Community Engagement: It is recommended that FPX pursue engagement activities with Indigenous groups early in the PFS as a parallel activity.

Product Marketing

FeNi briquette: FPX should undertake a more formal product marketability analysis and directly approach potential users to validate assumptions made in this PEA regarding payability and potential impact of deleterious elements.

Fe concentrate: FPX should undertake exploratory discussions with potential users of the Fe concentrate in order to evaluate product value. This can help guide metallurgical testwork aimed at upgrading the product to enhance value. A more detailed logistics / cost analysis for getting the product to market should be included.

Studies and Other PFS Elements

Trade-off studies: Early in the PFS risks and opportunities identified in the PEA should be reviewed and required trade-off studies should be performed to carry more optimal solutions into the PFS.

Third-party discussions: Early in the PFS, it is recommended that FPX begin exploratory discussions with any potential third party related to off-site infrastructure.

1.12.3 Budget for Next Project Stage

The estimated budget for undertaking the work required to complete the PFS is summarized in Table 1-9. FPX should obtain firm pricing for these activities based on a specific scope of work in order to better estimate its next project phase financing requirements. The PFS, and related activities, are expected to take about 12 to 18 months to complete. The budget estimate excludes costs related to maintaining claims as well as other corporate costs.

Table 1-9: Estimated required budget for next study phase

Activity	Description	Cost (K$ US)
Geology and Drilling	Improve the Project's database, update near-surface inferred resources	$ 880
Metallurgical Testwork	Drilling and sample collection, grindability/magnetic/flotation recovery, leaching, variability assessment, thickening, FeNi product sample	$ 1,400
Geotechnical	External TSF, waste dumps, pit walls, site	$ 980
Environmental and Community	Baseline study, community engagement activities	$ 2,300
Studies	Pre-Feasibility Study	$ 1,500
Owner	PFS owner's team costs and expenses	$ 1,140
TOTAL Opex		**$ 8,200**

2. INTRODUCTION

2.1 Scope of Study

The following Technical Report (the "Report") presents a Preliminary Economic Assessment (PEA) that summarizes the results of a study conducted for the FPX Nickel Corp. (FPX), Baptiste Nickel Project (the Project), within the Decar Nickel District (the "District" or the "Property") in the Omineca Mining Division, British Columbia. In February 2020, FPX retained the services of BBA Inc. (BBA) to lead this Study. This Report was prepared at the request of Mr. Martin Turenne, President and CEO of FPX, a Canadian publicly traded company listed on the TSX Venture Exchange under the symbol 'FPX'. FPX is incorporated under the Business Corporations Act (Alberta) and its registered office is located at:

Suite 620 – 1155 West Pender Street
Vancouver, BC
Canada, V6E 2P4
Tel: (604) 681-8600

This Technical Report titled "Preliminary Economic Assessment of the Baptiste Nickel Project", concerning the development of the Baptiste Deposit, was prepared by Qualified Persons (QP) following the guidelines of the "Canadian Securities Administrators" National Instrument 43-101 (effective June 30, 2011), and in conformity with the 2014 guidelines of the Canadian Mining, Metallurgy and Petroleum (CIM) Standard on Mineral Resources and Reserves.

Since the current study is a PEA, NI 43-101 Guidelines do not permit the disclosure of Mineral Reserves. NI 43-101 Guidelines do allow for Inferred resources to be included in an economic analysis for a PEA, as long as the appropriate cautionary language is used to qualify such an analysis. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The PEA is preliminary in nature and 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. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. It should also be noted that a Preliminary Economic Assessment is preliminary in nature and there is no certainty that the Project described in this PEA Report will be realized.

This Report is considered effective as of September 9, 2020.

2.2 Background and Project History

An initial Preliminary Economic Assessment on the Decar Property was completed in March 2013 and the NI 43-101 Technical Report was filed on Sedar in August 2013. The Study was performed under the guidance of Cliffs Natural Resources (Cliffs), the previous majority owner of the Project. In November 2015, First Point Minerals Corp. (now FPX Nickel Corp.) purchased Cliffs' share of the Project and established 100% ownership of the Project.

In February 2018, FPX filed an updated Mineral Resource Estimate Technical Report following the completion of a 2,000-metre drilling program.

In September 2018, a metallurgical testwork program was initiated. The objectives of the program were to improve nickel recovery and product quality (compared to the 2013 PEA) using conventional mineral processing technologies.

2.3 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions.

- Ronald Voordouw, P. Geo. — Equity Exploration Consultants Ltd.
- Ronald G. Simpson, P.Geo. — GeoSim Services Inc.
- Gordon Chen, P. Eng. — Stantec
- Sean Ennis, P. Eng. — Stantec
- Angelo Grandillo, P. Eng. — BBA Inc.
- Jeff Austin, P. Eng. — IME Inc.

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included in this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables and portions of Chapters 1 (Summary), 25 (Interpretation and Conclusions), and 26 (Recommendations). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

Mr. Trevor Rabb, P.Geo., is the former Vice-President Exploration of FPX and is currently a Partner, Resource Geologist with Equity. Mr. Rabb provided expertise on geological modelling of the Baptiste Deposit and on the historical work done by FPX on the Decar Property. He is not considered a Qualified Person for the purpose of this Report.

Table 2-1: Qualified Persons and areas of report responsibility

Chapter	Description	Qualified Person	Company	Comments and exceptions
1.	Executive Summary	A. Grandillo	BBA	Contribution from all QPs
2.	Introduction	A. Grandillo	BBA	
3.	Reliance on other Experts	A. Grandillo	BBA	
4.	Project Property Description and Location	R. Voordouw	Equity	
5.	Accessibility, Climate, Local Resource, Infrastructure and Physiography	R. Voordouw	Equity	
6.	History	R. Voordouw	Equity	
7.	Geological Setting and Mineralization	R. Voordouw	Equity	
8.	Deposit Types	R. Voordouw	Equity	
9.	Exploration	R. Voordouw	Equity	
10.	Drilling	R. Voordouw	Equity	
11.	Sample Preparation, Analyses and Security	R. Voordouw	Equity	
12.	Data Verification	R. Voordouw	Equity	
13.	Mineral Processing and Metallurgical Testing	J. Austin	IME	
14.	Mineral Resource Estimate	R.G. Simpson	GeoSim	
15.	Mineral Reserve Estimate	G. Chen	Stantec	Not Applicable
16.	Mining Methods	G. Chen	Stantec	
17.	Recovery Methods	A. Grandillo	BBA	
18.	Project Infrastructure	A. Grandillo S. Ennis	BBA Stantec	Stantec section on TSF
19.	Market Studies and Contracts	A. Grandillo	BBA	
20.	Environmental Studies, Permitting, and Social or Community Impact	A. Grandillo	BBA	
21.	Capital and Operating Costs	A. Grandillo	BBA	Stantec mining and TSF
22.	Economic Analysis	A. Grandillo	BBA	
23.	Adjacent Properties	R. Voordouw	Equity	
24.	Other Relevant Data and Information	A. Grandillo	BBA	
25.	Interpretation and Conclusions	A. Grandillo	BBA	Contribution from all QPs
26.	Recommendations	A. Grandillo	BBA	Contribution from all QPs
27.	References	A. Grandillo	BBA	Contribution from all QPs

2.4 Sources of Information

FPX has provided BBA with historic information and past and current study reports which BBA has, in part, relied upon to undertake this Study.

For this PEA, BBA has performed the economic analysis on a pre-tax basis and has relied on FPX and specialists retained by FPX to provide annual tax payment estimates for performing the post-tax economic analysis, as outlined in Chapter 22 of this Report.

Chapter 27 of this Report contains a list of references that have been used and referred to in this PEA Report. Past technical reports on the Project can be accessed from SEDAR's electronic database https://www.sedar.com/.

2.5 Site Visits

Ronald G. Simpson, Gordon Chen, Sean Ennis, Jeff Austin and Angelo Grandillo, have not visited the Project site.

Ronald Voordouw visited the Project site between 21-23 September, 2018.

2.6 Terms of Reference

Unless otherwise stated:

- All units in this Report are in the metric system;
- Unless otherwise stated, all costs are expressed in US Dollars ($ or USD or US$);
- The exchange rate used in this Study is $1.00 CAN = $0.76 US.

3. RELIANCE ON OTHER EXPERTS

For Section 4.0, the authors have relied on FPX, without independent investigation, for information with respect to underlying joint venture and royalty agreements that FPX could have with former option partners and/or shareholders, or the underlying interests in any of these agreements. Also, for Section 4.0, the authors have relied entirely on information from the Minerals Titles Branch of the Ministry of Energy, Mines and Petroleum Resources (Government of British Columbia) regarding property status and legal title for the Project. The authors have not relied upon a report, opinion or statement of another expert concerning legal, political, environmental or tax matters relevant to the technical report.

Mr. Trevor Rabb, P.Geo., is the former Vice-President Exploration of FPX and is currently a Partner and Resource Geologist with Equity. Mr. Rabb provided expertise on geological modelling of the Baptiste Deposit and on the historical work done by FPX on the Decar Property.

4. PROPERTY DESCRIPTION AND LOCATION

The Decar Property comprises 62 contiguous mineral claims that cover 24,740 hectares (247 km2) in the Omineca Mining Division of central BC, Canada (Figure 4-1, Figure 4-2). The approximate centre of the Property is at 54˚54'30.5" north latitude and 125˚21'31" west longitude (NAD-83 UTM Zone 10N: 6,087,000 m N 350,000 m E) on NTS map-sheets 93K083, 084, 085, 093, 094 and 095.

Mineral Titles Online (MTO) is a mineral claim registry maintained by the Government of BC. MTO claim boundaries are defined by latitude and longitude so that they form a seamless grid without overlap. "Legacy" mineral claims were staked on the ground prior to the introduction of the MTO system and take precedence over MTO claims.

MTO claim data for the Decar Property is summarized in Appendix A and shown in Figure 4-2. Five claims along the southern boundary of the Property overlap with Rubyrock Lake Provincial Park and 10 claims along the northeastern margin overlap with Mineral Reserve Site 326751. No exploration can occur in those portions of claims that are overlapped by the Provincial Park whereas exploration within the mineral reserve is either restricted or not permitted. Together, these overlaps cover 823.9 ha and reduce the size of the Decar Property by 3.3% to 23,917 ha. All known Fe-Ni alloy deposits and targets on the Property are located at least 2.5 km from overlaps with the Mineral Reserve and 5 km from the Provincial Park boundary.

Claims are registered with the Government of BC under FPX Nickel Corp. (Client ID 139385). Additional information on the Registration of Documents, Claim Registration, Exploration and Development Work/Expiry Date Change, and Transfer of Ownership events are provided on the MTO website.

The claims confer title only to minerals as defined by the Mineral Tenure Act (British Columbia). Surface rights over non-overlapping MTO claims are held by the Crown, as administered by the Government of BC. The ownership of other rights (placer, timber, water, grazing, trapping, outfitting, etc.) affecting the Property was not investigated by the authors.

British Columbia law requires assessment expenditures to maintain tenure ownership past the current expiry dates. As of July 1, 2012, annual assessment requirements were set at C$5/hectare (ha) for years 1 and 2, C$10/ha for years 3 and 4, C$15/hectare for years 5 and 6, and C$20/hectare for all subsequent years. Exploration expenditure can be distributed over contiguous claims for up to 10 years into the future. As of June 2020, 59 of the 62 Decar claims are in good standing to late 2028. One claim expires 26 September 2023 and two others are protected from expiry in July 2020 through an Order of the Chief Gold Commissioner that states all non-terminated mineral claims with an expiry date prior to 31 December 2021 are amended from their current expiry date to 31 December 2021.

Another seven claims overlap with legacy claims and 14 claims overlap with District Lots. The mineral rights within the overlapping legacy claims are held by Ursula Mowat. These overlapping claims cover 270.0 ha, or 1.1%, of the Decar Property. District Lots are surveyed land parcels that convey title (ownership) of the surface rights to the purchaser when the District Lots are sold by the Crown. Exploration on those claims overlapping District Lots is permitted but requires notification of the surface rights holder (if any). The Van Target lies within 1 km of District Lots 2036 and 2047 but the Baptiste Deposit, Sid Target and B Target are located >4.5 km from District Lot boundaries.

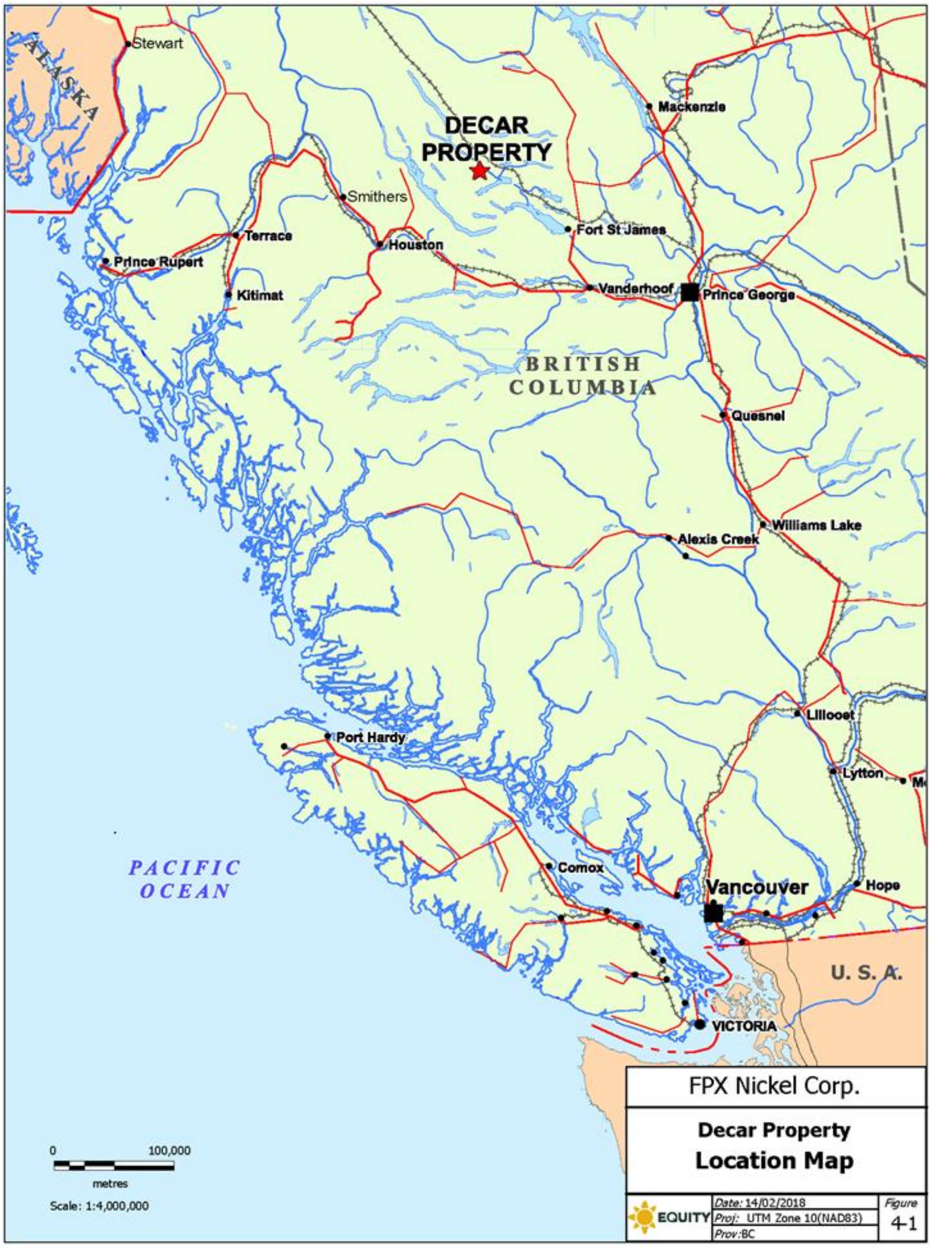


Figure 4-1: Location map for the Decar Property. Source: Equity Exploration (2018)

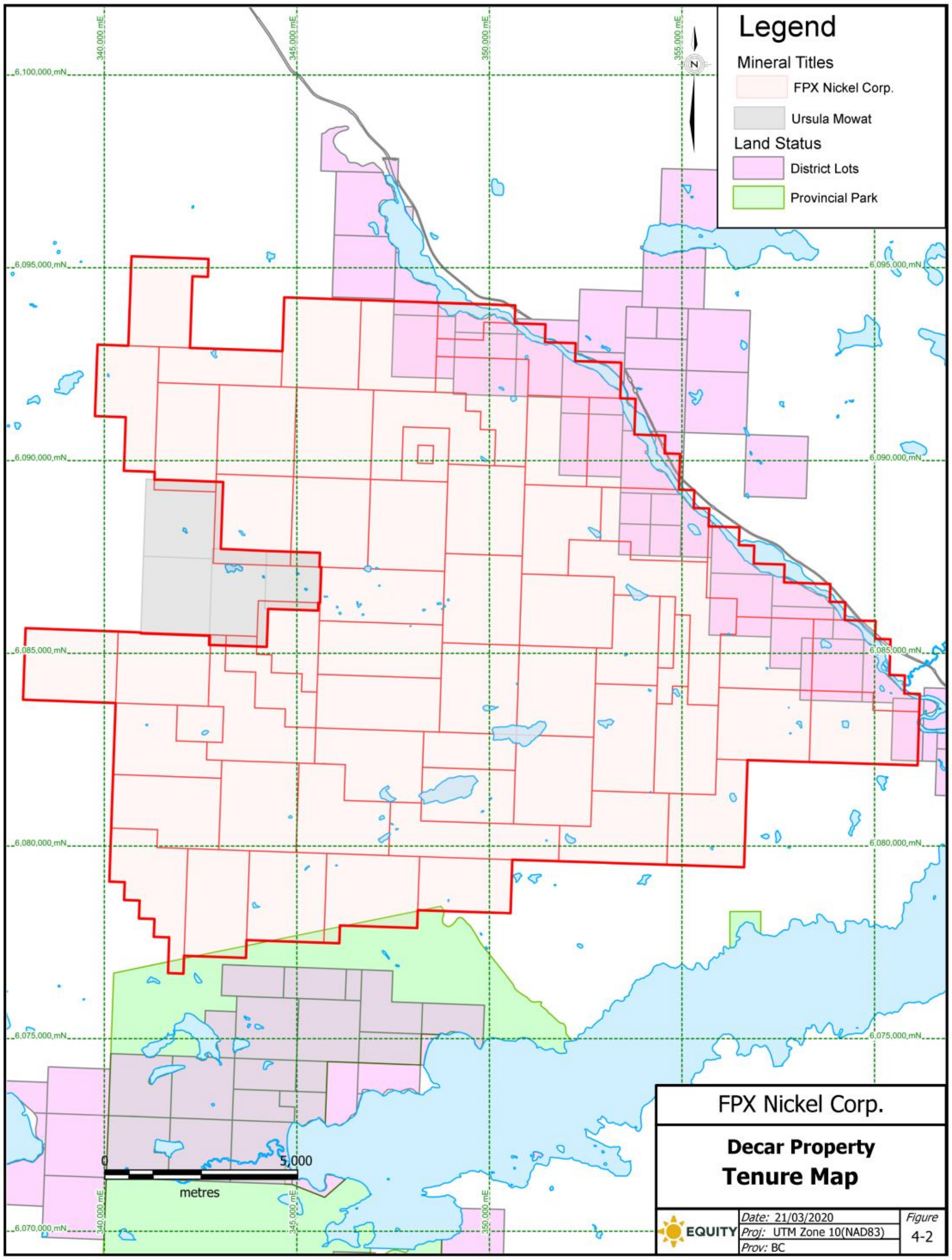


Figure 4-2: Decar Property tenure map. Source: Equity Exploration (2020)

All claims are registered in the name of, and are 100% owned by, FPX, who purchased Cliffs' 60% interest in the project in November 2015 (FPX Nickel, 2015). Cliffs optioned the Decar Property in November 2009, and earned a 60% interest by incurring approximately US$22 million in exploration and related expenditures (FPX Nickel, 2014) which included completion of a PEA (McLaughlin et al., 2013). FPX acquired Cliffs' 60% ownership interest with a cash payment of US$4.75 million, provided by a Lender who earned a 1% NSR royalty over the Baptiste Project (FPX Nickel, 2015). This loan is secured against the Baptiste Project and all related assets. To the authors' knowledge, the Decar Property has no other royalties, back-in rights or other agreements and encumbrances.

No material environmental liabilities were noted by author Voordouw during a visit to the Decar Property in September 2017. Between May 15 and June 15 each year, exploration on the Property is restricted to protect the mountain caribou calving season (BC Ministry of Environment Order U-7-003).

Exploration programs that necessitate mechanical disturbance (e.g. drilling or trenching) and some ground-based geophysical methods require permits issued by the Ministry of Energy and Mines of the Government of BC. FPX is currently applying for two exploration permits that will allow for 5 years of work on the Baptiste Deposit and Van Target.

The Decar Property lies within the traditional territory of the Tl'azt'en Nation. A memorandum of understanding (MOU) was signed between the Tl'azt'en Nation, FPX and Cliffs in June 2012 (FPX Nickel, 2012) that formalized protocols for the working relationship between the parties and confirms the Tl'azt'en Nation's support for exploration activities. The MOU contains specific provisions which, following the sale by Cliffs of its 60% interest in the Property to FPX in 2015, have both enabled and required FPX to assume all of the project proponent's rights and obligations under the MOU. The material impact of future agreements between the FPX, the Government of BC and local stakeholders is not evaluated in this technical report.

To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

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

The Decar Property is road-accessible and reasonably proximal to resources and infrastructure required to develop the project. There are several current- and past-producing mines in the area, including Mount Milligan, located 100 km east-northeast of the Decar Property, and Endako located 95 km to the south. Mount Milligan is a conventional truck-shovel, copper-gold, open pit mining operation with an on-site mill capable of processing 60,000 tonnes per day. The current Life of Mine plan for Mount Milligan forecasts production through to 2028 (Fitzgerald et al., 2020). Endako is an open pit molybdenum operation with a concentrator that has been on care and maintenance as of July 2015 (Centerra Gold, 2018).

5.1 Accessibility

The Decar Property is road-accessible from the town of Fort St. James, BC, through a 170-km long network of paved and gravel forestry roads (Figure 5-1). The most direct route heads 2.3 km north of Fort St. James along Stuart Lake Highway/BC-27N to the Tachie Road, then follows the Tachie Road for 39 km to the Leo Creek Forestry Road (FR). The Leo Creek FR is then followed northwest for 38.5 km, followed by 48 km on the Leo-Kazcheck Forestry Road (300 Road), 2 km on the to Leo-Sakenichie FR (900 Road) to cross the Middle River and then 38 km on the Leo-Middle Forestry Road (700 Road) to reach a 1-km long road leading to the reclaimed Decar Camp site.

5.2 Local Resources and Infrastructure

The closest municipality to the Property is Fort St. James, which has a population of 1,510 (Statistics Canada, 2017) and offers a range of services and supplies that include labour, gas stations, freight, rental heavy equipment, groceries and hardware. The nearby city of Prince George (population 74,003), which lies 152 km by paved roads to the southeast of Fort St. James, provides a broader range of services and supplies, along with daily commercial flights to Vancouver. The town of Smithers (population 5,351) lies 120 km due west of the Property and can be accessed by Forestry Roads and a seasonal barge that crosses Babine Lake. Smithers also offers a full range of services and supplies for mineral exploration, in addition to daily commercial flights to Vancouver.

CN Rail operates a railway through Smithers and Terrace to the port of Prince Rupert (Figure 5-1). An out-of-service rail line, also owned by CN Rail, follows the east bank of Middle River and runs through the northeastern margin of the Property.

The nearest power corridor reaches to within 3 km of the Decar Property and serves the settlement of Middle River. BC Hydro's Glenannan Substation connects to a 500 kV trunk line and is located 90 km south-southeast of the Decar Property.

McLaughlin et al (2013) suggested that concentrate could be shipped to a proposed transload facility located adjacent to the CN rail line near the Middle River, a trip distance of approximately 15 km. Concentrate would then be transferred to railroad cars for shipment to the seaport of Prince Rupert, which is located 325 km west-southwest of the Property.

Surface rights over the Decar Property are mostly owned by the Crown and administered by the Government of BC and would be available for any eventual mining operation. Parts of several claims at the east end of the Property overlap with District Lots (see Section 4.0) where exploration activity by FPX would require notification of the surface rights holder (if any). No exploration can be conducted on those parts of the Decar Property that overlap with Rubyrock Lake Provincial Park and, possibly, Mineral Reserve Site 326751.

The Property has abundant water and water rights could be obtained for milling. Potential tailings storage areas, potential waste disposal areas and potential processing plant sites are described in Chapter 16 of this technical report.

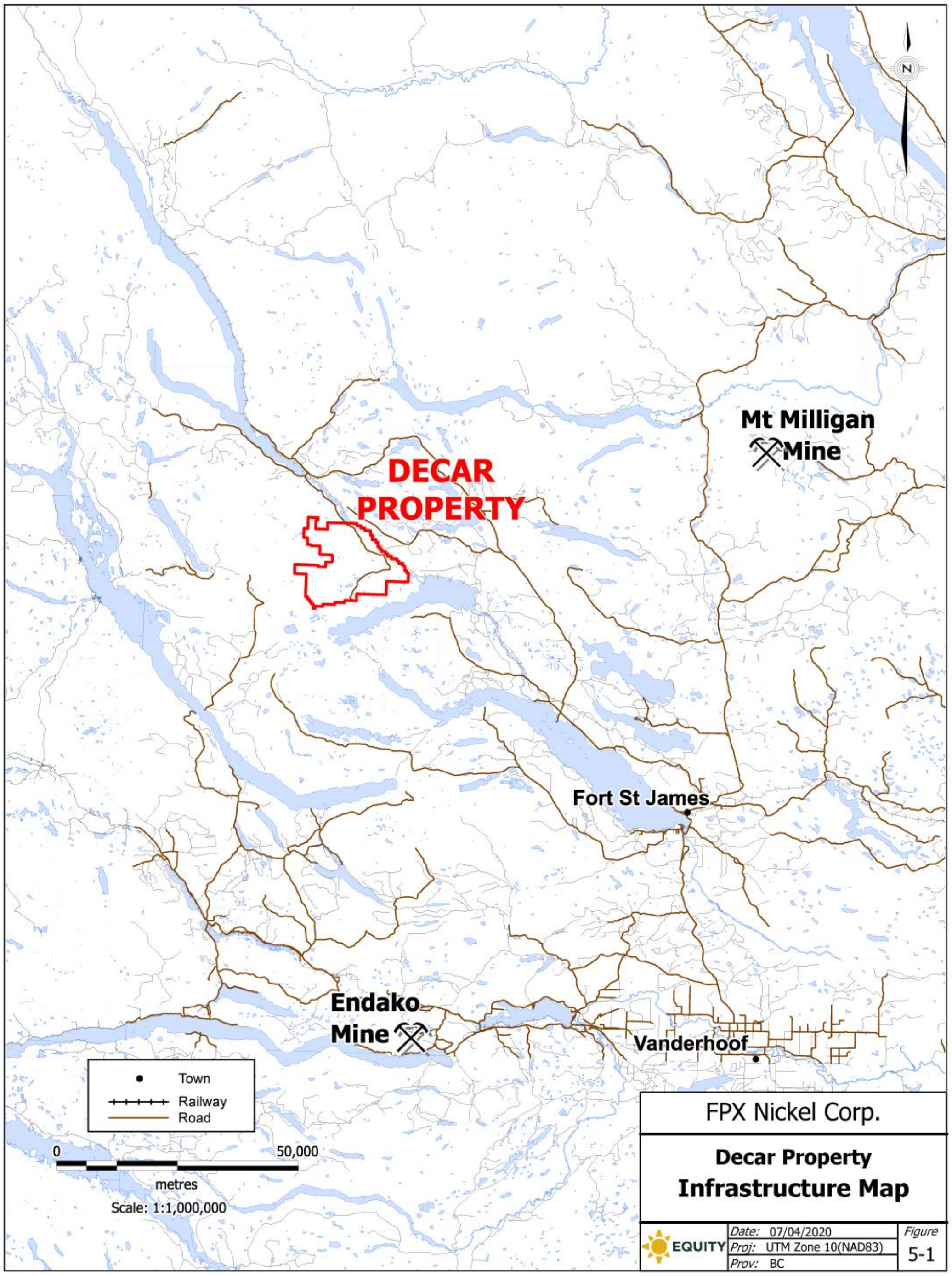


Figure 5-1: Decar Property access and infrastructure map. Source: Equity Exploration (2020)

5.3 Physiography and Climate

The Decar Property lies in the sub-boreal spruce ecological zone, which characterizes the gently rolling terrain of British Columbia's interior plateau. Elevation across much of the Property ranges from 900-1400 m above seal level ("ASL"), dropping to 700 m ASL near the Middle River and reaching up to 1983 m ASL at the top of Mount Sidney Williams. The Property boundaries approach the Middle River in the northeast and Trembleur Lake in the south.

The sub-boreal spruce ecological zone comprises a rolling landscape with dense coniferous forests, with variation related to elevation as well as dry and wet micro-climates. Property-scale baseline environmental studies were done from 2011 to 2014 but were not reviewed as part of preparing this technical report.

The climate in the Decar area is "northern temperate" or "sub-boreal spruce zone" and is characterized by cold snowy winters and warm summers. Average Environment Canada climate data for Fort St. James in the period 1981-2010 (Environment Canada, 2018) indicate daily average temperatures ranging from -9.5°C in January to 15.4°C in July. The highest average monthly accumulations are 50.6 millimetres of rain in June and 43.3 cm of snow in January. Average snow depth peaks in February at 54 cm.

The physiography and climate at the Decar Property are amenable to year-round mineral exploration activities that include diamond drilling and geophysical surveys. Geological mapping and geochemical sampling can be conducted from June to October when there is no snow cover.

6. HISTORY

Exploration activity on, or immediately adjacent to, the Decar Property is summarized in Table 6-1 and began with the discovery of several chromite pods by the Geological Survey of Canada ("GSC") in 1942 (Armstrong, 1949). This work initiated a phase of chromite-focused exploration between 1975-1982, which identified sub-economic occurrences. Starting in 1987, the exploration focus shifted to listwanite-hosted gold that had also been reported by early GSC work (Armstrong, 1949) and prospector anecdotes (Mowat, 1988a), with the bulk of drilling activity occurring between 1990-1994. The strongly magnetic and high-density Ni-Fe alloy "awaruite" was first described in the area by Whittaker (1983). Sporadic awaruite-focused petrographic and metallurgical work was completed between 1996 and 2006. From 2007 onwards, concerted efforts were made to quantify recoverable Ni present as awaruite, define awaruite deposits and demonstrate the economic viability of their extraction.

Chromite pods are a source of chromium (Cr) and, potentially, platinum group elements (PGE) and were first described in the Decar area by the GSC (Armstrong, 1949). A GSC mapping program in 1942 discovered nine chromite deposits hosted by dunite and peridotite (Armstrong, 1949) within what was later mapped as the Trembleur ultramafite unit. The largest of these bodies was described as 5 x 25 feet in size with >50% chromite (Armstrong, 1949). In 1975, Douglas Stelling prospected for chromite on the Pauline Group of claims, near what is now the center of the Decar Property. This work found mostly disseminated chromite grading 0.2-0.4% Cr, with one select sample of massive chromite-bearing dunite returning 9.8% Cr (Stelling, 1975). Subsequent work in the same area by Mountaineer Mines re-discovered the 5 x 25 feet zone identified by Armstrong (1949) and returned assays of 17.8% and 38.9% Cr, but otherwise failed to locate larger chromite bodies (Guinet, 1980). Three years later, Northgane Minerals Ltd subcontracted Western Geophysical Aero Data Ltd to fly a 310 line-km very low frequency electromagnetic (VLF-EM) and magnetometer survey with the aim of defining the extent of ultramafic rocks and identifying trends that could be favourable for chromite mineralization (Pezzot and Vincent, 1982). This survey identified highly magnetic areas of possible serpentinized peridotite and dunite, showing that at least three chromite showings are associated with very high magnetic response (Pezzot and Vincent, 1982). However, no further chromite-focused exploration was completed.

The discovery of "listwanite" bodies with elevated gold-pathfinder elements, like arsenic and antimony, brought renewed exploration interest to the Decar area. Listwanite is partially silicified, carbonate-altered ultramafic rock that is a favourable host for gold, with examples including the Cassiar deposits in northern BC and Motherlode district in California, USA. Geologist Ursula Mowat established a property that overlaps with what is now the central and western half of the Decar Property, and, in 1987, began exploration for gold-hosted listwanite through an option agreement with Lacana Mining Corporation (Mowat, 1988a). The next year, surface sampling and trenching of seven listwanite zones returned Au-enriched channel samples (Mowat, 1988b). Coincident prospecting, in what is now the southern part of the Decar Property, returned ultramafic rock samples with 80-90 ppb Au (Forbes, 1988).

Diamond drilling of listwanite prospects in 1990-1994 (1,541 m in 22 drillholes) were financed through option agreements between Ursula Mowat and Viceroy Resource Corporation (Mowat, 1990, 1991, 1994), Minnova Incorporated (Mowat, 1991) and Teryl Resources Corporation (Mowat, 1994). The first of these drilled 305 m in seven holes, with six of these returning intercepts of 1-6 g/t Au over intervals ranging from 0.4 to 9 m (Mowat, 1990). This would turn out to be the most successful program. The next year, another five holes (for 511.4 m) were drilled on the Stibnite and Upper listwanite zones but returned mostly disappointing results (Mowat, 1991). In 1994, another 10 holes (for 724.7 m) tested EM conductors, soil anomalies, listwanite zones and/or IP features (Mowat, 1994) with drilling generally failing to intersect gold mineralization or explain geophysical anomalies.

Following a brief lull, exploration work resumed in 1998 to focus on gold in the previously unexplored West Peak area as well as on strongly talc-altered ultramafic rocks near Baptiste Creek (Mowat, 1998). Results were used to suggest that glacial detritus could be masking porphyry and/or gold systems on the Property. The following year, a three-phase work program was used to define, and then follow-up on, weak Au and PGE anomalies on the Mid claim (Mowat, 1999). Mappers identified additional listwanite and talc-rich alteration zones with elevated arsenic and Ni, as well as glassy volcanic rocks with weakly anomalous PGE. The following year, new outcrops of ultramafic were discovered in the West Peak area, although sampling returned no elevated base or precious metals (Mowat, 2000). From 2002-2004, small surface programs outlined new listwanite occurrences with coincident Au-arsenic soil anomalies (Mowat, 2002), new outcrops of listwanite and serpentinized peridotite in clear cuts (Mowat, 2004), and Au-enrichment in soil (Mowat, 2005). A small-scale soil and rock sampling program, completed in 2006, returned negligible results (Mowat, 2007).

In 2007, AMARC conducted a significant silt sampling campaign on claims that lie mostly west of the current Decar Property, but also overlap with the southwestern-most of these claims. This work returned anomalous values for molybdenum, copper and zinc (Ditson et al., 2008).

Awaruite in the Decar area was first recognized by Whittaker (1983) while completing a PhD thesis on several belts of ultramafic rocks in central BC. In 1996, geochemical and petrographic work on ultramafic rocks sampled on claims lying near the centre of the present day Decar Property showed that nickel is hosted in awaruite and other low-sulphur nickel minerals, including heazlewoodite, bravoite and pentlandite (Mowat, 1997a, b). In 1997, First Point Minerals Corp. (now named “FPX Nickel Corp.”) optioned the Klone property from Mowat specifically to explore for awaruite as a possible bulk tonnage open pit deposit. It was recognized that since awaruite, magnetite and chromite were all magnetic, ore grade material could be produced by magnetic separation. This work was followed up with a sampling and metallurgical program that showed awaruite may be economically extractable through a simple grind and magnetic separation process. However, with nickel below US$5/lb First Point dropped its option (Mowat, 1997c). The 1999 program included metallurgical testwork on different grind sizes, with results showing that a 150-mesh grind produces higher Ni values than 100-mesh (Mowat, 1999).

Besides rudimentary prospecting, petrographic and metallurgical work, little further work was completed on Ni-Fe alloy mineralization until 2007, when First Point Minerals Corp. (now named “FPX Nickel Corp.”) staked 15 claims that form what is now the core of the Decar Property (Voormeij and Bradshaw, 2008). The Decar Property then grew to 33 claims by 2009 (Britten, 2010), 37 claims by 2010 (Britten and Rabb, 2011), 59 claims by 2012 (Ronacher et al., 2012b) and then to its current 62-claim size by 2018. Work by FPX is further described in sections 9, 10 and 11. FPX optioned the Decar Property to Cliffs in November 2009 and then re-acquired 100% ownership of the Property in 2015.

Table 6-1: Overview of historical work done on and around the Decar Property

Company	Year	Work Type	Commodity	Production	Reference
GSC	1942	Prospecting	Cr	Regional mapping	Armstrong (1949)
D. Stelling	1975	Chip sampling	Cr	38 rocks	Stelling (1975)
Mountaineer Mines	1979	Prospecting	Cr	4 rocks	Guinet (1980)
Northgane Minerals	1982	Airborne VLF-EM and mag	Cr	310 line-km	Pezzot and Vincent, 1982)
Lacana Mining Corp	1987	Surface sampling	Au, PGE, Cr	304 rocks, 95 silt, 180 soil, 9 pan	Mowat (1987)
	1988	Surface sampling	Au, PGE, Cr	276 rock, 58 silt, 2593 soil, 52 trench chip	Mowat (1988)
J.R. Forbes	1988	Surface sampling	Au	30 rock, 20 silt, 3 pan	Forbes (1988)
Viceroy Resource	1990	Drilling, surface sampling	Au	305 m in 7 DDH; 8 rock, 6 silt, 2 soil	Mowat (1990)
Minnova	1991	Drilling	Au	511 m in 5 DDH	Mowat (1991)
Teryl Resources	1994	Drilling, surface sampling	Au	725 m in 10 DDH; 58 soil	Mowat (1995)
U. Mowat	1996	Geochemistry, petrography	Ni	7 rock	Mowat (1997)
	1997	Geochemistry, metallurgy	Ni	262 rock, 32 silt, 1 pan	Mowat (1997)
	1998	Mapping, sampling	Au, PGE	62 rock, 1 silt, 8 soil	Mowat (1998)
	1999	Mapping, sampling	Au, PGE	63 rock, 3 silt	Mowat (1999)
	2000	Mapping, sampling	Au, PGE	40 rock	Mowat (2001)
	2002	Sampling	Au, PGE	46 rock	Mowat (2002)
	2003	Sampling	Au, PGE	13 rock	Mowat (2004)
	2004	Sampling	Au, PGE	8 rock, 75 soil	Mowat (2005)
	2006	Sampling	Au, PGE	291 silt	Mowat (2007)
AMARC Resources	2007	Silt sampling	Mo, Cu, Zinc	60 rock	Ditson et al. (2008)

Au = gold, Cr = chromium, Mo = molybdenum, PGE = platinum group elements

7. GEOLOGICAL SETTING AND MINERALIZATION

Mineralization on the Decar Property is hosted in strongly altered ("serpentinized") ultramafic rocks of the Cache Creek terrane. The following sections establish the regional- to property-scale context of these ultramafic rocks and associated awaruite mineralization.

7.1 Regional Geology and Mineralization

The Decar Property lies in the Paleozoic to Mesozoic Intermontane Belt (Figure 7-1), which is formed by accreted outboard components ("terranes") of oceanic affinity that include Stikine, Quesnel and Cache Creek. The Stikine and Quesnel terranes (or Stikinia, Quesnellia) are both volcanic arcs whereas the Cache Creek terrane consists of ophiolite, marine sedimentary rocks and locally developed seamount-like successions that likely formed in an oceanic and/or back-arc setting (Monger et al., 1991). The Decar Property is entirely underlain by rocks of the Cache Creek terrane.

The Cache Creek terrane is exposed in three main areas; north-central BC to southern Yukon, central BC near Fort St. James, and in southern BC where it serves as the type locality (Monger et al., 1991). Each of these areas is underlain by lithologies that include Carboniferous to Jurassic radiolarian chert and argillite, shallow-water carbonate, ophiolite (basalt, gabbro, ultramafic rocks) and calc-alkaline volcanic with related volcano-sedimentary rocks. The serpentinized ultramafic rocks of ophiolitic-affinity are the host of Ni-Fe alloy (awaruite) mineralization on the Decar Property.

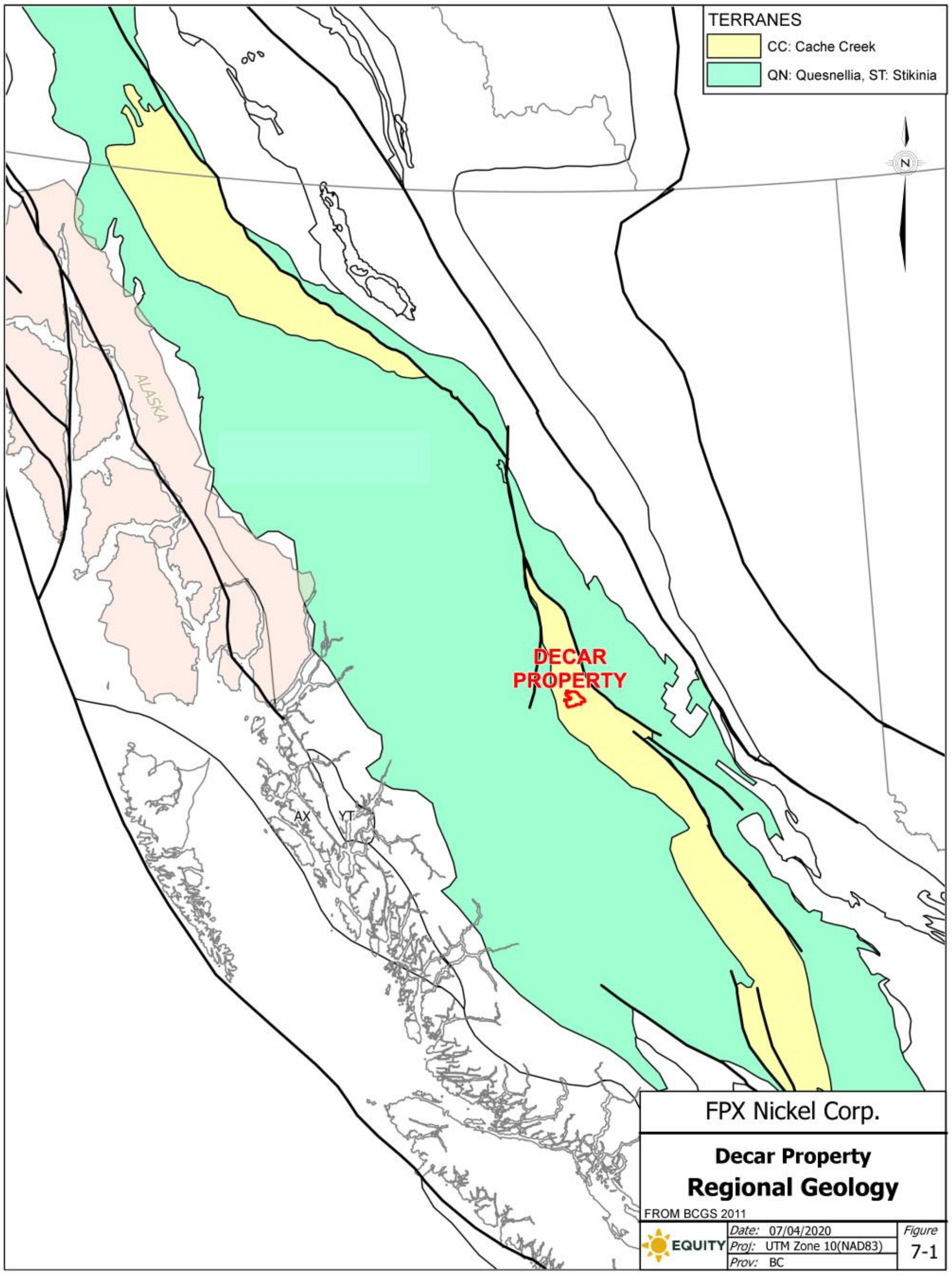


Figure 7-1: Regional geological setting of the Decar Property showing the Cache Creek (CC), Stikine (ST) and Quesnel (QN) terranes. Source: Equity Exploration (2020)

All three Cache Creek areas are bound by regional-scale faults and show significant internal stratigraphic and structural disruption (Monger et al., 1991). Bounding structures include the Thibert-Kutcho-Pinchi faults that separate Cache Creek from Quesnel terrane to the east, and the King Salmon and Takla faults forming its western boundary with the Stikine terrane. An estimated 115 km of dextral movement had occurred on the Thibert-Kutcho-Pinchi fault system by the Upper Cretaceous and was followed by another 170 km of movement on the north-south trending Thudaka-Finlay-Ingenika-Takla fault system into the Eocene (Gabrielse, 1985). This protracted history of accretion and lateral transport resulted in significant faulting internal to the Cache Creek terrane, and likely channeled the fluids that caused serpentinization of ultramafic rocks as well as awaruite mineralization (Britten, 2016).

Mineral deposits in the Cache Creek terrane include several types within the ultramafic ophiolitic rocks, as well as Noranda- and Kuroko-type copper-lead-zinc volcanogenic massive sulphide (VMS), molybdenum-copper porphyry, vein-hosted gold and surficial placer gold. Deposit types formed in ultramafic rocks include the awaruite prospects described in this report (see also Britten, 2016), Alaskan-type Ni-copper-PGE, podiform chromium, jade/nephrite, listwanite-hosted gold, talc-magnesite, cryptocrystalline magnesite veins, magmatic Ni-Cu sulphide and asbestos.

The bulk of the historical and current mineral production from Cache Creek rocks is from relatively small-scale jade/nephrite, placer gold and industrial mineral operations. The nearby Mount Milligan and Endako mines are underlain by the Quesnel and Stikine terranes respectively.

7.2 Local Geology

The Cache Creek terrane that underlies the Decar area is subdivided into (from southwest to northeast) the Rubyrock igneous complex, Sitlika assemblage, Trembleur ultramafite, Sowchea succession and Copley limestone (Figure 7-2). These subdivisions are structurally intercalated by faults internal to the Cache Creek terrane. The western margin of the Cache Creek terrane is bound by the Takla fault, which separates it from Stikine Terrane, whereas the eastern boundary with the Quesnel terrane is marked by the Pinchi fault.

The Rubyrock igneous complex consists of upper Paleozoic to Triassic gabbro, basalt, diabase and micrograbbro (Struik et al., 2001) that could be analogous to the upper part of a typical ophiolite succession (see Boudier and Nicolas, 1985). These rocks occur mostly immediately southwest of the Property with intercalations occurring within the Trembleur ultramafite further to the northeast (Struik et al., 2001). Quartz-carbonate veining and strong chlorite-epidote ± pyrite alteration occur locally.

The Sitlika assemblage underlies the southwestern portion of the Property and consists of slate, phyllite, siltstone, sandstone and conglomerate, with minor abundances of limestone and chert (Struik et al., 2001). Some sedimentary units host felsic volcanic and plutonic clasts, and bedding orientations are typically subvertical. A fault separates Sitlika rocks from the Trembleur ultramafite to the northeast. Sitlika sedimentary rocks are interpreted as a distal to proximal turbidite succession.

The Trembleur ultramafite unit forms the core the Decar Property with many additional occurrences to the south and north (Figure 7-2). Rock types consist mostly of peridotite with lesser amounts of dunite, pyroxenite and gabbro (Britten, 2016; Struik et al., 2001). Most of these rocks range from partially to fully serpentinized (Britten, 2016) with other alteration minerals including Mg-Fe carbonate, talc and/or silica. The Trembleur unit is interpreted as the mantle and lower crustal portion of a typical ophiolite succession and hosts the Ni-Fe alloy mineralization that is the focus of this technical report.

The Sowchea succession is subdivided into units of fine clastic and undivided sedimentary rocks (Logan et al., 2010). The fine clastic unit includes phyllite, slate, siltstone, siliceous argillite, quartzite, conglomerate and chert, with lesser amounts of recrystallized limestone. The undivided unit contains similar sedimentary rocks along with chlorite schist and metabasalt, and is cut by dikes and sills of greenstone, diabase and diorite. The Sowchea succession lies mostly northeast of the Trembleur unit although smaller fragments also occur to the southwest, presumably as fault-bounded panels. In the previous technical reports (McLaughlin et al., 2013; Ronacher et al., 2013) these rocks were referred to as the North Arm succession after Schiarizza and MacIntyre (1999). The Sowchea succession is likely equivalent to the pillow basalt and deep-sea sedimentary rocks that comprise the top of the typical ophiolite column.

Units of Copley limestone are mostly enveloped by Sowchea rocks and occur near the eastern boundary of the Cache Creek terrane. Rock types include mostly Permian (and possibly undifferentiated Triassic) micritic to clastic limestone, massive recrystallized limestone, lesser bedded limestone and minor amounts of marble, in addition to minor abundances of greenstone, chert and argillite (Logan et al., 2010). These limestone units are interpreted as the top parts of paleo-seamounts.

Contacts between the Rubyrock, Sitlika, Trembleur, Sowchea and Copley units are typically structural, comprising thrust and/or transform faults most likely initiated during obduction of the Cache Creek terrane and/or the long-lived transform faulting that followed obduction. The Takla and Pinchi faults, which bound the Cache Creek terrane to the west and east respectively, are both northwest trending and are cut by northeast trending dextral strike-slip faults like the Trembleur Lake and Tildesly Creek faults (Britten and Rabb, 2011). The Pinchi Fault lies 18 km northeast of the Property whereas the Takla Fault lies 10 km to the southwest.

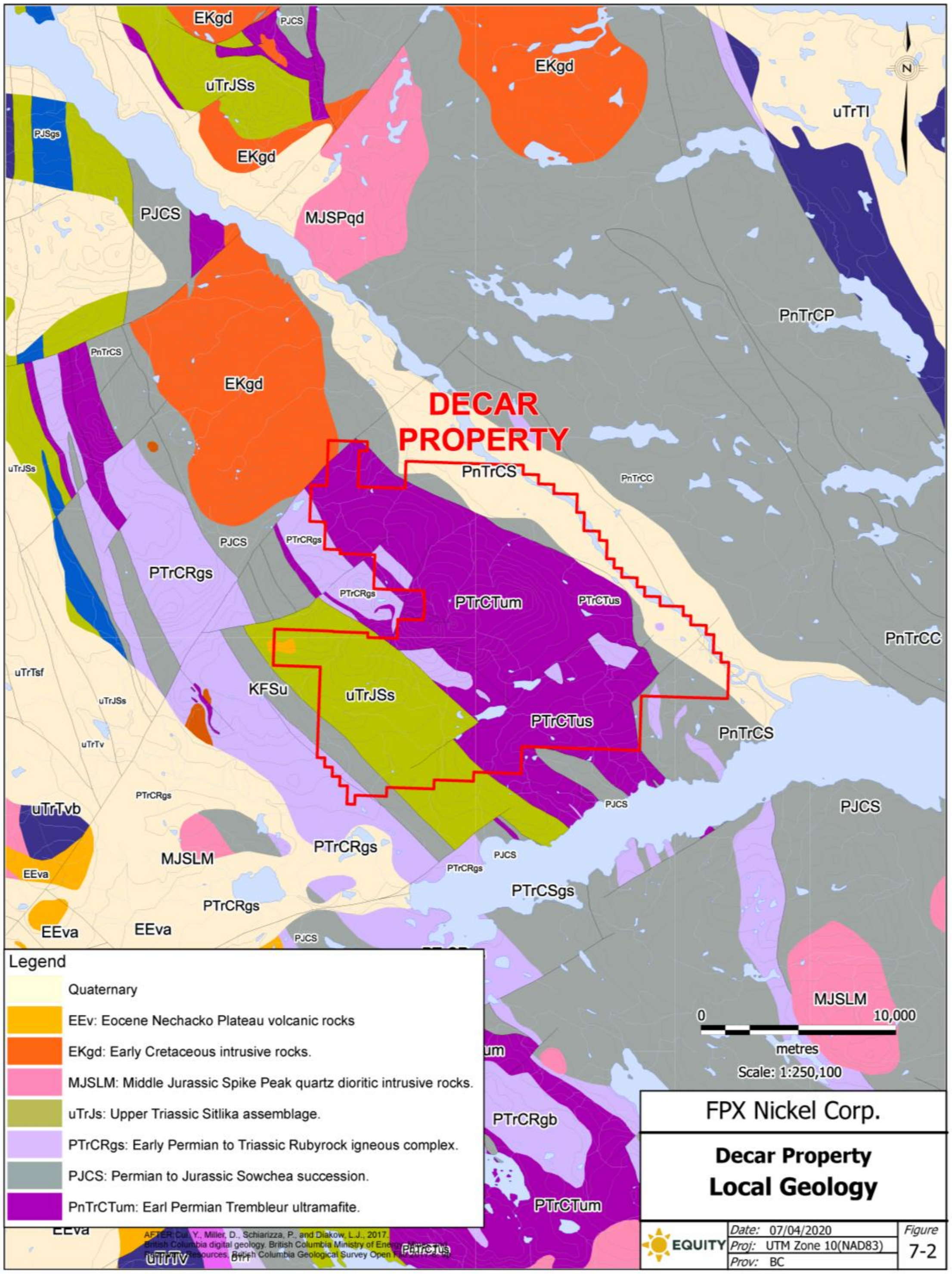


Figure 7-2: Decar Property local geology map. Source: Equity Exploration (2020)

7.3 Property Geology

7.3.1 Lithology

The core of the Decar Property is underlain by the Trembleur ultramafic unit, with Sitlika assemblage rocks underlying the southwest part of the Property and Sowchea succession underlying the northeast (Figure 7-3). Geological mapping and drilling has mostly focused on the Trembleur rocks, although several holes were collared into Sitlika rocks to drill northeast into the ultramafic. The main rock types within each of these three units, as they were mapped and relogged in FPX and Equity's exploration work, are described below.

Mapping and core logging has characterized eight types of ultramafic related to the Trembleur unit. The most abundant of these is peridotite, which comprises 66.7% of all logged metres, and massive peridotite comprising 10.5% of the total metres. In this scheme, peridotite is more serpentinized than massive peridotite, although even the least altered rocks contain at least 60% serpentine. Pre-alteration mineralogy is estimated at 65-80% olivine, 20-30% orthopyroxene, <5% clinopyroxene and <0.5% chromite (Britten, 2016). The abundance of secondary serpentine is manifested in rock colour, with the least altered peridotite appearing medium grey whereas those with >95% serpentine are dark green to black-brown.

Other peridotite subtypes include cataclastic, mylonitized and hornfelsed, which collectively comprise 2.1% of all drill core logged on the Property. Mylonitized and cataclastic peridotite are essentially synonymous, with both overprinted by a prominent network of hairline and brecciated fractures. It is plausible that these rocks mark the structures that channeled the hydrothermal fluids related to awaruite mineralization. Hornfelsed peridotite occurs next to gabbro and altered dikes, forming dark, very fine-grained and locally non-magnetic selvages that range from 0.1-14.4 m in core width (average 3.7 m).

Dunite forms layers, lenses and pods within the more abundant peridotite units. It is typically fine-grained, dark to medium grey and massive, and consists of >95% serpentinized olivine (Britten, 2016). Most dunite occurs along the western margin of the Baptiste Deposit and on the Mount Sidney Williams arête. Fragments of peridotite occur in pods of dunite and both rock types host awaruite, although grades within the dunite are mostly below 0.06% DTR nickel cut-off grade. Of the 32,032 m drilled on the Baptiste Project, 3.4% was logged as dunite.

Some units were logged or mapped as altered ultramafic rock, including 2.3% of all drilled metres as FeCb_AltUM and another 2.3% as FeCb_Listwanite. Intervals logged as FeCb_AltUM are marked by moderately pervasive Mg-Fe carbonate ± talc alteration and are associated with fault zones and intrusive bodies. The FeCb_Listwanite code records listwanite units characterized by near-total replacement of ultramafic protoliths by Mg-Fe carbonate and silica. The FeCb_AltUM and FeCb_Listwanite units therefore form a continuum of Mg-Fe carbonate alteration and both types are antithetic to awaruite mineralization. Listwanite units were explored as part of early gold-focused programs (e.g. Mowat, 1990, 1991, 1994).

The Trembleur and adjacent Sitlika units are cut by gabbro and altered dikes. Gabbro dikes are massive and fine- to medium-grained and are typically northeast to east striking or stock-like in form (Ronacher et al., 2012a). Core widths total 1.8% of all drill metres with an average thickness of 3.0 m, and in several cases occur immediately adjacent to altered dikes and/or carbonate-altered ultramafic. Altered dikes have a wide range of inferred protoliths and contain variable abundances of garnet, serpentine, carbonate, chlorite, hematite, silica, albite and/or fuchsite. Garnet- and pyroxene-rich dikes are referred to as “rodingite”. Core widths total 1.9% of all drilled metres and average 2.0 m in width. Altered dikes likewise appear to show a preferential association with carbonate-altered ultramafic rocks, gabbro dikes and fault zones.

Argillite (2.1% of all drilled metres) and volcanic (0.9%) lithologies comprise part of the Sitlika assemblage lying immediately southwest of the Trembleur ultramafic rocks. The argillite unit, which is referred to as mudstone and phyllite in the 2012 report (Ronacher et al., 2012a), comprises carbonaceous argillite, black phyllite, mudstone and slate with lesser limestone and chert. This unit is tectonically interleaved with Trembleur ultramafic and cut by both gabbroic and altered dikes. Volcanic rocks occur southwest of argillite and consist of green to medium greyish green chloritic phyllite that are locally interleaved with graphitic argillite.

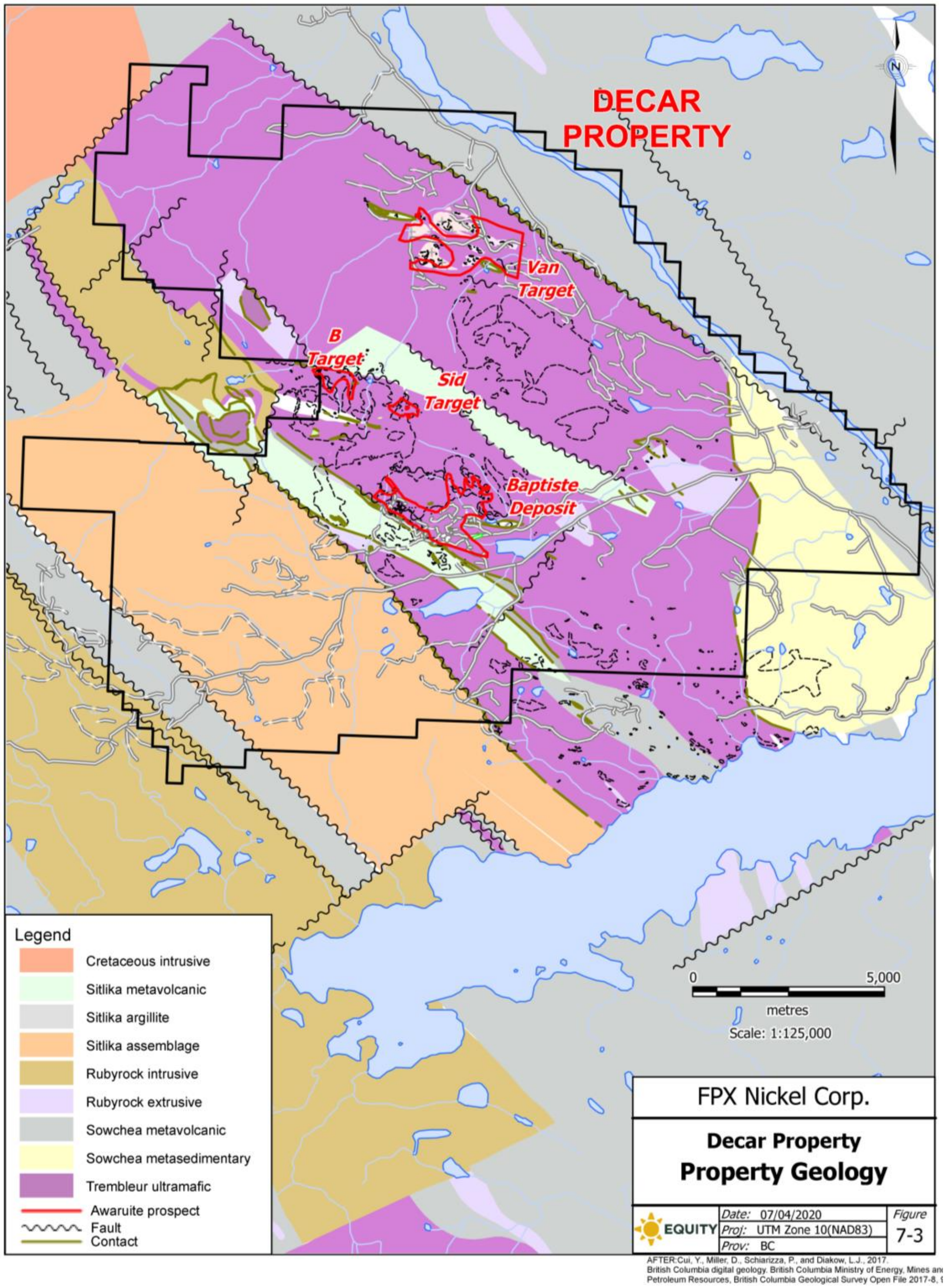


Figure 7-3: Decar Property, property geology map at 1:25,000. Source: Equity Exploration (2020)

Approximately 1.8% of all drill core is logged as Fault Zone, which includes both healed faults and fault gouge. Healed faults show cataclastic and/or mylonitic textures and are possibly related to cataclastic and mylonitized peridotite. These faults developed under brittle to ductile conditions that resulted in localized development of penetrative deformation fabrics and serpentinized fractures. Younger faults are marked by strongly broken core and clayey gouge. Overall, geotechnical logging suggests slightly lower-than-average recovery and rock quality within intervals logged as Fault Zone.

Diamond drilling indicates that true overburden depths average 12.0 m with a maximum depth of 42.2 m. Only one of the 88 holes drilled by FPX on the Decar Property was collared on bedrock. Surficial mapping by the GSC (Plouffe, 2000) indicates that most of the Property is covered by till blanket and veneer, with bedrock outcrops, slope colluvium and talus occurring at higher elevations, and glaciolacustrine deposits occurring near Trembleur Lake and the Middle River.

7.3.2 Structure

Ultramafic rocks exhibit a gradation in deformation features that are likely coeval with serpentinization. The least deformed rocks on the Property are generally massive with rare, suspect, cumulate layers (Britten and Rabb, 2011). Petrographic study shows that the more deformed types appear to have undergone multiple breakage and brecciation events prior to and during serpentinization, resulting in a pseudobreccia texture that is cross-cut by serpentine-filled veins (Britten, 2016; Britten and Rabb, 2011).

Strong penetrative foliation is locally developed in ultramafic rocks as well as the adjacent metavolcanic and metasedimentary units. The eastern and western extents of the Baptiste Deposit, for example, are bound by 50-100 m wide zones of subvertical, northwest-trending and strongly developed penetrative deformation fabric that trends parallel to the long axis of the Deposit (Britten, 2016). Deformation fabric intensity decreases gradually outwards into more massive and undeformed ultramafic rock. The Sid and Van targets are elongated in the same orientation and show a similar spatial association with bounding structures defined by penetrative deformation fabric. Unmineralized peridotite occurring between mineralized zones is typically blocky and massive and appears not to have been deformed prior to serpentinization (Britten, 2016).

Post-alteration fault and shear zones are marked by slickensides, gouge, strong penetrative foliation, fault breccia and/or shear fabric. There are several significant northwest-trending, subvertical, fault zones on the Property, including the fault that separates the Trembleur and Sitlika rocks and another fault that bounds the western side of the Baptise Deposit. Previous mapping by the GSC (MacIntyre and Schiarizza, 1999) also defined a northeast trending fault that appears to have caused 1500 m of dextral offset of the Trembleur unit in the southern part of the Decar Property. Several additional structures occur in the same orientation.

7.3.3 Alteration

Serpentinization and Mg-Fe carbonate alteration are two predominant types of alteration within the Trembleur ultramafic body. Serpentinization is the most widespread, having affected all ultramafic rocks to some degree and with significant areas comprising >90% serpentine. On the Decar Property, serpentinization is defined by the replacement of olivine and orthopyroxene with antigorite and lizardite, which are both more abundant than chrysotile (Britten, 2016). Magnetite and awaruite form as part of the serpentinization process. Serpentinized rocks are cut by rare, discontinuous, crack-seal carbonate micro-veinlets (Britten and Rabb, 2011).

Mg-Fe carbonate alteration forms carbonate-dominant and carbonate-silica (i.e. "listwanite") assemblages. The weak (or incipient) variety of this alteration is logged as FeCb_AltUM, which is non- to weakly magnetite destructive and characterized by selective replacement texture. More pervasive, moderate to strong, alteration is logged as FeCb_Listwanite, which is typically texturally and magnetite destructive. This alteration is spatially associated with fault and/or unit contact zones, as well as small feldspar porphyry intrusions in the southeast part of the Property. The most significant of these is a rusty-weathering, elliptical, 1000 x 1800 m, zone of Mg-Fe carbonate alteration formed around a lens-shaped, east-west trending, feldspar porphyry intrusion. This intrusion shows pervasive alteration to sericite, chlorite, Mg-Fe carbonate, and pyrite, and is cut by north to northeast trending, moderately east dipping, later stage en echelon quartz veins.

Near-total carbonate alteration of ultramafic rocks results in the precipitation of silica (i.e. quartz) and the formation of listwanite. Several listwanite bodies are known to occur within the Decar Property, and several of these host pyrite, rare chalcopyrite and trace amounts of gold (Britten and Rabb, 2011).

7.4 Property Mineralization

Historical exploration on the Property has focused on three deposit types: (1) Ni-Fe alloy (awaruite) in serpentinized ultramafic rock, (2) listwanite-hosted gold, and (3) chromite pods in ultramafic. Each of these is briefly described below even though, at this time, only the Ni-Fe alloy deposits are potentially economic.

Ni-Fe alloy deposits are an atypical deposit type formed by the serpentinization of magmatic olivine that leads to the liberation of nickel and iron (Britten, 2016) and subsequent formation of the alloy. Awaruite occurs throughout the entire extent of the peridotite on the Decar Property, but four zones of more abundant mineralization and larger grain size have been delineated; Baptiste, Sid, B and Van (Figure 7-3). The Baptiste Deposit is the most advanced of these four target areas (see Section 14.0). The other three targets are defined through surface mapping, sampling and/or diamond drilling, with three holes drilled at Sid Target and one drilled at B Target. High-grade awaruite appears to trend NW-SE, parallel to the orientation of lithological contacts and major fault structures.

The Baptiste Deposit is currently defined for approximately 3000 m in an east-west direction and for 600-1500 m from north to south. Most holes end in mineralization at true vertical depths of 250 m below the surface, with 14 of the 2012 holes ending in mineralization at vertical depths between 436-544 m. The Deposit is fault-bound in the southwest and grades into massive peridotite with lower-grade mineralization to the north and northwest. Drilling at the Sid and B targets is sufficient only to demonstrate that Baptiste-like mineralization occurs in these targets as well, but insufficient to determine the length, width, depth and continuity of the mineralization. No drilling has been completed at the Van Target.

Exploration for listwanite-hosted gold deposits peaked in 1990-1994, with 1541 m of diamond drilling over 22 holes to test the most prospective zones (Mowat, 1990, 1991, 1994). By 1994, drilling had identified at least 17 listwanite zones (1994) on or around the current Decar Property (Mowat, 1988a). Pathfinder elements for listwanite-hosted gold occurrences include iron, arsenic, lead, copper, zinc, nickel, cobalt and antimony. Possible origins of the gold-listwanite association include (1) carbon dioxide and hydrogen sulphide immiscibility with attendant gold deposition, (2) reduction of the mineralizing fluid by gold deposition, (3) sulphide precipitation promoted by Fe-rich lithologies, and (4) precipitation of silica, pyrite, arsenide and gold when an acidic gold-bearing solution enters reduced and alkaline carbonatized rocks (Kerrich, 1989; Dussel, 1985).

Chromite pods in the Trembleur ultramafite were discovered by the GSC in 1942 (Armstrong, 1949) and drove some of the first hard-rock exploration efforts in the Decar area (Guinet, 1980; Pezzot and White, 1984; Stelling, 1975). These occurrences are examples of podiform chromite that formed in the ultramafic part of an ophiolite complex. Mapping by the GSC identified nine chromite pods in the Trembleur ultramafic rocks within or nearby the Decar Property, with follow-up work suggesting these pods are generally too small to be economic.

8. DEPOSIT TYPES

Disseminated awaruite (Ni_2Fe to Ni_3Fe) forms an unusual nickel deposit type, with occurrences on the Decar Property comprising the most advanced projects of this type in the world (Britten, 2016). Terrestrial awaruite was first described in heavy black sand from the South Island of New Zealand (Ulrich, 1980), and has since been found as a minor component in altered ultramafic rocks all over the world. It forms during serpentinization of peridotite whereby nickeliferous olivine is altered to serpentine minerals and awaruite + magnetite under conditions of low oxygen fugacity (Frost, 1985). A general unbalanced reaction that illustrates this mineralogical and metal exchange is as follows (from Britten, 2016):

$$\underbrace{4(MgFeNi)_2SiO_4}_{\text{olivine}} + \underbrace{2H_2O}_{\text{water}} \leftrightharpoons \underbrace{2Mg_3Si_2O_5(OH)_4}_{\text{serpentine}} + \underbrace{Fe_3O_4}_{\text{magnetite}} + \underbrace{Ni_3Fe}_{\text{awaruite}} + H_2(g)$$

The alteration of olivine-rich ultramafic rocks to 60-80% serpentine results in a density decrease from 3.3 to 3.4 g/cm^3 for olivine-rich rocks to 2.7 g/cm^3 for serpentinite, and a volume increase of 18% to 55% related to a gain of 10-14 wt% H_2O (Britten, 2016).

A recent overview of the awaruite deposits hosted in Cache Creek terrane (Britten, 2016) suggests that a key part of the ore forming process was a prolonged period of post-accretionary transpression, which resulted in significant strike-slip displacement and, more importantly, ingress of relatively clean and possibly oxygenated meteoric water. Deformation generated high porosity zones up to several hundreds of metres in width that are now marked by foliation, crackle breccia and microfracture textures. Subsequent processes then necessary to produce awaruite included the hydration of olivine to serpentine minerals, ingress of water with low sulfur and CO_2 activity, oxidation of iron to produce magnetite, the maintenance of low oxygen fugacity and, eventually, addition of H_2 through reduction of Fe and Ni. Hydration at temperatures of <100 to 200°C is likely capable of producing fine-grained awaruite (<20 µm) in association with low-temperature serpentine minerals (e.g. lizardite, chrysotile) and brucite (Britten, 2016). Higher temperatures (200 to >400°C) are probably necessary to form the larger grains like those on the Decar Property that are associated with antigorite. The highest temperature (>450°C) conditions produce the highest amount of magnetically recovered awaruite, in association with the metamorphism of serpentine and magnetite to olivine and diopside (Britten, 2016).

Besides the Decar Property, other awaruite occurrences occur in the northern outcropping areas of Cache Creek terrane (see Figure 7-1) and in the Dumont deposit of Québec, Canada. Prospects in the northern Cache Creek terrane include Orca, Wale, Letain and Mich, and are similar to those at Decar (see Britten, 2016). At the Dumont deposit, awaruite occurs as pervasively disseminated grains, between <50 to 400 µm in size, hosted in serpentinite and spatially associated with magnetite and chromite (Staples et al., 2011). Although sulfides are widespread in the Dumont deposit, there are zones where only the Ni-Fe alloy is present. Minor abundances of awaruite also occur together with nickel and copper sulfide in the Duluth complex of Minnesota, USA, and appears to be of magmatic, rather than secondary, origin.

Awaruite is highly ferromagnetic and dense (ρ = 8.2 g/cm^3) and is therefore amenable to concentration by mechanical processes (i.e. magnetic, gravity separation). In addition, the ultramafic tailings from awaruite concentrate production could potentially be used for CO_2 sequestration (e.g. Vanderzee et al., 2018), offering a significant environmental advantage over Ni-sulphide sources.

Because metallurgical properties play such a vital role in the economics of awaruite projects the grades are presented as Davis Tube Recoverable (DTR) nickel. The Davis Tube consists of an inclined water-filled tube placed between electromagnets (Svoboda, 2004) and is used to split finely-ground powder into magnetic and non-magnetic fractions. DTR nickel is calculated as follows:

$$DTR\ Ni = wt\%NiO * 0.7858 * \frac{weight\ magnetic\ fraction}{(weight\ magnetic\ fraction + weight\ nonmagnetic\ fraction)}$$

Data required to calculate DTR Ni is provided by the analytical lab, which besides reporting weight percent nickel oxide (wt%NiO) and Ni (%) also report the weights of the magnetic and non-magnetic fractions split with the Davis Tube.

9. EXPLORATION

In 1997, First Point Minerals Corp. (now named "FPX Nickel Corp.") optioned the Klone property from Ms. Ursula Mowat specifically to explore for awaruite as a possible bulk tonnage open pit deposit and did a number of metallurgical tests to determine how amenable the awaruite was to recovery (Mowat, 1997c). However, the price of nickel was below US$5/lb, and the option was dropped. When the price of nickel increased to well over $10/lb in 2006, FPX staked about half of what is currently the Decar Property and subsequently embarked on a four-year campaign that included additional staking, geological mapping and sampling, geophysical surveys, and metallurgical testwork before optioning the project to Cliffs in 2009. Cliffs then funded exploration on the Property from 2010 to 2013, publishing a maiden resource for the Baptiste Deposit in 2012 (Ronacher et al., 2012a) followed by an updated resource (Ronacher et al., 2013) and PEA in 2013 (McLaughlin et al., 2013). Since 2013, FPX has completed additional mapping and surface sampling work, in addition to expansion drilling on the Baptiste Deposit in 2017.

This section describes geological mapping, surface sampling and geophysical surveys that have been carried out by FPX and Cliffs on the Decar Property since 2006. Descriptions include procedures and parameters used to execute the work, sampling methodology and QA/QC, as well a summary of significant results and interpretation. Diamond drilling is summarized in Section 10.0.

9.1 Geological Mapping and Surface Sampling

Geological mapping, rock sampling and petrographic work done on the Decar Property is summarized in Table 9-1, with the sample totals shown derived from the FPX database. Mapping and rock sampling effectively define the Baptiste, Sid, B and Van targets on the Property. Sampling locations are shown in Figure 9-1.

In 2007, FPX conducted a prospecting and rock sampling program over a 4 x 6 km area of the Decar Property that included the Van, B and Sid targets (Voormeij and Bradshaw, 2008). Sample location coordinates were recorded using a handheld Garmin GPS 60, which typically has a precision of 5-10 m. Samples were mostly serpentinized peridotite (N = 49) of which 42 were analyzed by Acme Analytical Laboratories (ACME) of Vancouver, BC. Total nickel in these samples averaged 0.21% Ni with a range of 0.12% to 0.28%.

The following year, FPX completed geological mapping, rock sampling and stream sediment sampling (Britten, 2009). Four surface bulk samples weighing 20 to 120 kilograms were also collected for metallurgical work. All sample locations were marked with a handheld Garmin GPS 60. Rock samples were collected from outcrop, analyzed with a portable XRF Niton NLp 502 Analyzer (portable XRF) and then slabbed with a diamond saw to help make more accurate visual estimates of awaruite content and grain size. Results showed that awaruite occurs widely across the Property and forms zones of consistently larger grain size (100-400 µm) to define the Baptiste Deposit and Sid Target (Britten, 2009). SEM analysis of 105 awaruite grains from 13 samples indicates an average of 77% Ni and range from 68 to 85% Ni (Le Couteur, 2008).

Seventeen of the 37 sediment samples were sieved to -80 mesh and then analyzed with the portable XRF, whereas a heavy magnetic fraction was separated from the remaining 20 samples by panning over a strong magnet (Britten, 2009). On average, the heavy magnetic fractions were found to contain 2-4 times more Ni than the 80 mesh sediments.

Geological mapping, rock sampling and stream sediment sampling was expanded in 2009 (Britten, 2010) using methods similar to the 2008 program. 130 rock samples and 50 stream sediment samples were collected. Rock sampling focused mostly on the Van and Baptiste areas increase the surface expression of the Baptiste Deposit to a length of 1,750 m and width of 800-1,300 m (Britten, 2010). Three patches of coarse-grained awaruite, separated by overburden, were mapped at the Van Target and could be linked into a single zone up to 700 m in length (Britten, 2010).

Stream sediment samples were dried, sieved to -60 mesh, separated with a pencil magnet and then analyzed with a portable XRF. These magnetic fractions returned ~1,200-4,800 ppm Ni, with those collected over the Baptiste Deposit consistently in the 3,000-4,800 ppm range and those collected over the Van Target between 2,000-3,000 ppm.

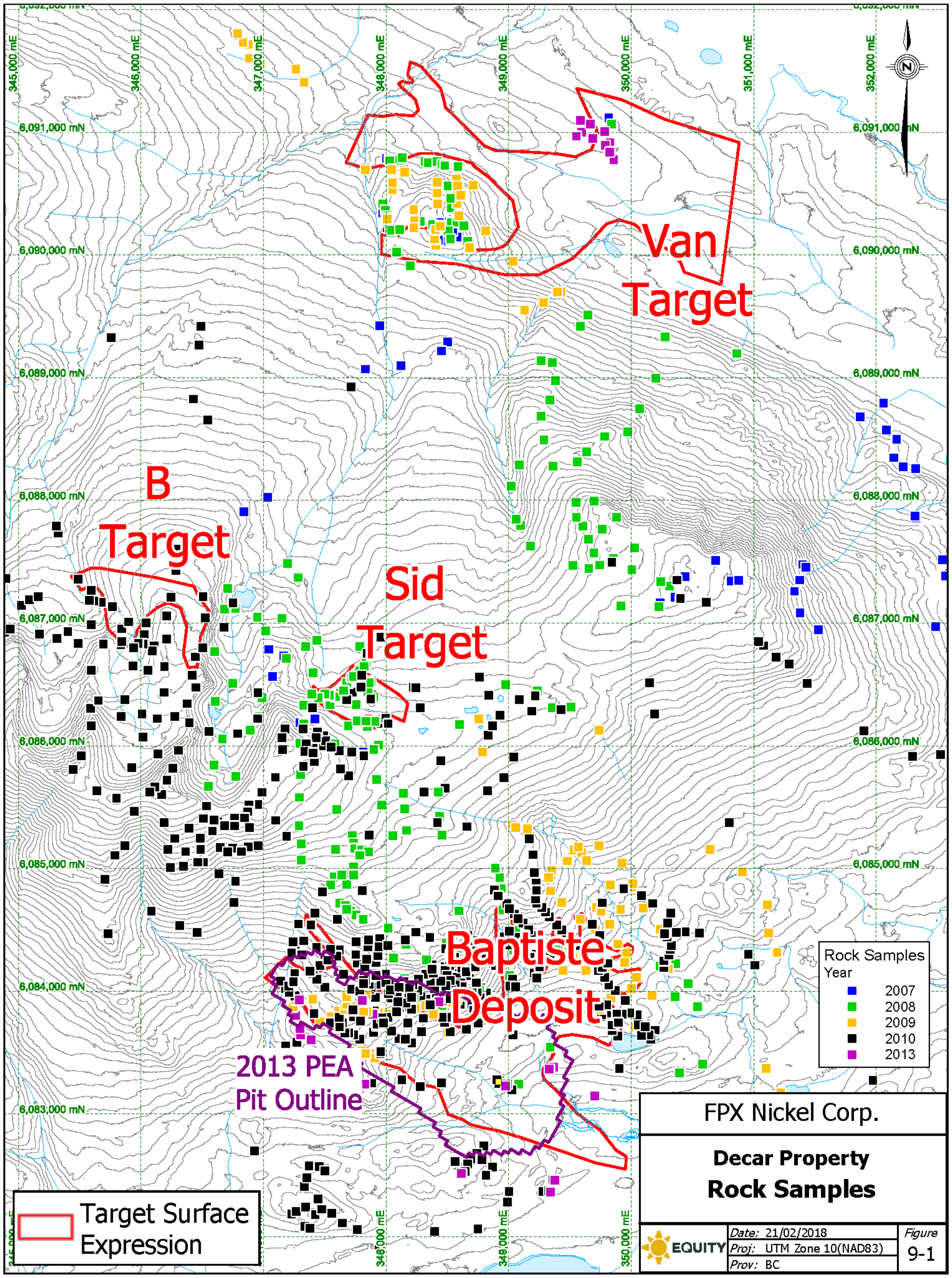


Figure 9-1: Decar Property rock sample map
Source: Equity Exploration (2018)

Table 9-1: Overview of FPX surface samples from the Decar Property

Year	Rock (No.)	Stream Sediment (No.)	Petrography (No.)	Geochemical Assay	Reference
2007	60			ACME	Voormeij and Britten, 2008
2008	226*	37	13	Portable XRF, some ACME	Britten, 2009; FPX database
2009	130	50	6	Portable XRF, some ACME	Britten, 2010
2010	561	24		Portable XRF, some ACME	FPX database
2013	35			Portable XRF, some Actlabs	FPX database
2014	138			Portable XRF, some Actlabs	FPX database
Total	**1,150**	**111**	**19**		

The most extensive rock sampling campaign was completed in 2010, with 561 samples collected from the Baptiste, Sid and B target areas, as well as from elevated ground in the southeastern part of the claim block. Sample descriptions record rock type, serpentinization, magnetic susceptibility and awaruite grain size. Subsequent rock sampling campaigns focused on the Baptiste (2013), Van (2014) and Sid (2014) areas. This work resulted in the successful delineation of the Baptiste Deposit as a zone of coarse awaruite grain size and also demonstrated significant potential at the Van Target, where larger awaruite grains occur over approximately 2.9 km^2 and assays have returned DTR Ni grades similar to Baptiste (FPX Nickel, 2018).

9.2 Geophysical, Televiewer and Lidar Surveys

The formation of magnetite together with awaruite during serpentinization suggests that geophysical surveys (e.g. magnetic susceptibility) could delineate awaruite-rich zones. Early testwork on hand samples by Walcott (2011) showed that awaruite-rich zones may also be chargeable with low resistivity. As a result, several property- and prospect-scale geophysical surveys have been carried out over the Decar Property and within individual drill holes, with the aim of establishing correlation between geophysical properties and mineralization (Table 9-2). Televiewer and Lidar surveys, which are used to collect relatively high resolution geotechnical and topographic data respectively, are also summarized in this section.

Table 9-2: Overview of FPX geophysical, televiewer and Lidar surveys

Survey Type	Contractor	Survey Name	Year	Production	Reference
Airborne magnetic	Aeroquest International Ltd	Decar	2010	1,638.9 line-km	Britten and Rabb, 2011
Ground IP and magnetics	P.E. Walcott & Associates	Baptiste grid	2010	20.1 line-km	Britten and Rabb, 2011
		Sidney grid	2010	9.0 line-km	Britten and Rabb, 2011
Downhole IP	Caracle Creek	Vertical profile (VP)	2011	17 drillholes	Palich and Qian, 2012
		X-hole tomography	2011	8 drillhole pairs	Palich and Qian, 2012
Rock Properties	DGI Geoscience	Poly-electric	2011/12	47 drillholes	Ronacher et al., 2013
		Natural gamma	2011/12	46 drillholes	Ronacher et al., 2013
		Magnetic IC	2011/12	42 drillholes	Ronacher et al., 2013
		Focused density	2011/12	21 drillholes	Ronacher et al., 2013
		Neutron	2011/12	21 drillholes	Ronacher et al., 2013
Televiewer	DGI Geoscience	Optical	2011/12	31 drillholes	Ronacher et al., 2013
		Acoustic	2011/12	36 drillholes	Ronacher et al., 2013
Lidar	Terra Remote Sensing	Decar	2012	389 km^2	Ronacher, 2013

9.2.1 Airborne Gradient Magnetic Survey

FPX contracted Aeroquest International Limited of Mississauga, Ontario, (Aeroquest) to survey a portion of the Decar Property using their airborne gradient magnetometer. The following description of this program is adapted from Aeroquest's logistical report (Aeroquest, 2010).

The 2010 survey was conducted using Aeroquest's Bluebird Heli-TAG tri-axial magnetic gradiometer, aided by a GPS navigation system, radar altimeter, laser altimeter, orientation sensor, digital video acquisition system and base station magnetometer (Aeroquest, 2010). Total survey length is 1,667.8 line-km of which 1,638.9 km fell within the pre-defined project area coordinates, providing coverage over a 220 km^2 area at 150 m line-spacing. Lines were flown at an azimuth of 038°-218°. Nominal tower-bird clearance was between 30-50 m but was periodically higher due to terrain and the capability of the aircraft. The survey speed of 100 km/hr and sampling rate of 10 Hertz produced a reading about every 1.5 to 3.0 m along the flight lines.

Aeroquest delivered six 1:20,000 maps that showed total magnetic intensity (TMI), gradient enhanced TMI with line contours, measured vertical gradient, measured transverse gradient, measured longitudinal gradient and a digital terrain model. All maps were projected in NAD83 UTM Zone 10. No further interpretation was provided by Aeroquest.

Britten and Rabb (2011) used the airborne data to show that much of the Decar Property has a northwest-southeast trending pattern of magnetic highs and lows (Figure 9-2), interpreted as regional-scale strike-slip faults that juxtapose different stratigraphic levels of the Cache Creek terrane. They also suggested that the Baptiste, Sid and Van targets all occur within, or close to the borders of, magnetic highs that enclose irregular-shaped subdued magnetic highs.

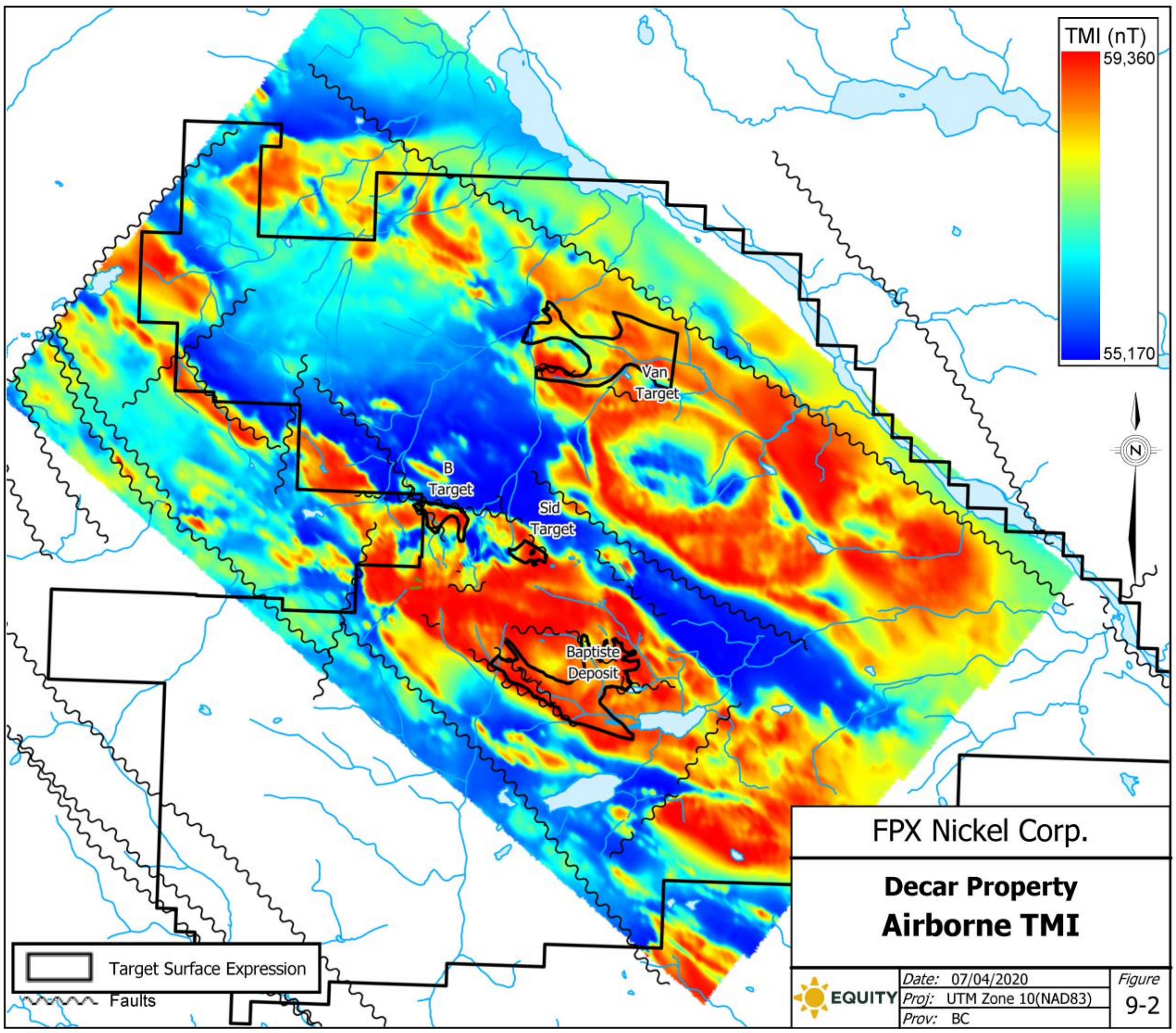


Figure 9-2: Total magnetic intensity (TMI) from the 2010 airborne survey
Source: drafted by Equity Exploration (2020) using imagery from Aeroquest (2010)

9.2.2 Ground-based Induced Polarization (IP) and Resistivity

In 2010, FPX contracted Peter E. Walcott & Associates Limited of Coquitlam, BC, (Walcott) to conduct a ground-based induced polarization and resistivity (IP) survey on the Decar Property and the section below is mostly summarized from their logistical report (Walcott, 2011).

The 2010 IP survey was designed to delineate regions of increased chargeability after laboratory tests showed awaruite-bearing serpentinized peridotite has elevated chargeability (Walcott, 2011). A total of 29.1 line-km was surveyed over two grids, with 20.1 km on Baptiste and 9.0 km on Sid (Figure 9-3). Survey data was collected with a pulse type system using a pole-dipole array. The transmitter was powered with a 7.5 kW 400 Hz three phase generator, providing up to 7.5 kW of direct current to the ground. The transmitter cycling rate was set at 2 seconds each for "current on" and "current off", with the pulses reversing continuously in polarity. Apparent chargeability (mA) is presented as a direct readout of millivolt per volt (mV/V), using a 200-millisecond delay and a 1,000 millisecond sample window by the receiver. Locations of stations were acquired using a wide area augmentation system (WAAS) equipped Garmin GPSMAP 60Cx unit.

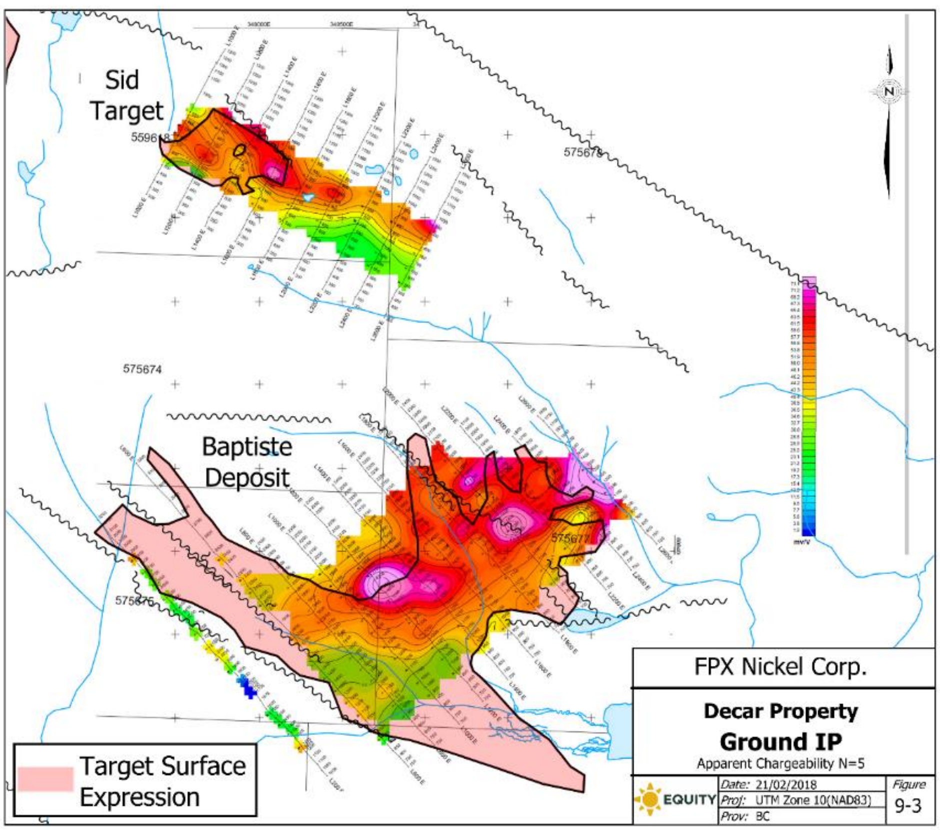


Figure 9-3: Plan map of contoured chargeability at N = 5 separations for the Baptiste and Sidney grids Outlines of the Baptiste Deposit and Sidney target, as defined by surface rock sampling are shown as thick black lines. Source: Equity Exploration (2018) using imagery from Walcott (2011)

IP data was presented as individual pseudo-section plots of apparent chargeability and resistivity at 1:10,000 scale. In addition, the third and fifth separation readings were contoured on a plan map at 1:10,000 (Figure 9-3). Ground magnetic data was also contoured at the same scale.

Strong chargeability and resistivity responses were found to correlate with major lithological breaks and/or fault contacts, with no obvious correlation between IP response and awaruite enrichment. A very strong chargeability response was inferred to reflect 3% modal magnetite masking 0.1-0.3% nickel-iron alloy, although this is not uniform (Britten and Rabb, 2011).

9.2.3 Downhole Induced Polarization (IP) and Resistivity

Downhole induced polarization and resistivity (DCIP) surveying was completed in September and October 2011 and comprised vertical resistivity and chargeability profiling of 17 boreholes in addition to cross-hole tomographic imagining of eight drillhole pairs (Palich and Qian, 2012). These surveys were completed by Caracle Creek International Consulting Incorporated of Sudbury, Ontario, (Caracle Creek) with the objectives of (1) correlating DCIP and 2010 ground IP surveys, (2) mapping DCIP anomalies between boreholes, and (3) correlating DCIP features to lithology and awaruite concentrations (Palich and Qian, 2012).

Data was collected with Caracle Creek's proprietary EarthProbe DCIP system. Vertical resistivity and chargeability profiling was achieved by placing a standard current and potential electrode down a single borehole, with measurements taken in time-domain mode using an 8,192-millisecond current injection square waveform (Palich and Qian, 2012). Based on an electrode separation ("a-spacing") of 4 m and 24 electrodes on each cable, the theoretical formation penetration is about 25 m off-hole. The EarthProbe survey down 11BAP019 was run with a longer cable (300 m) that allowed for theoretical penetration of 70 m. Cross-hole tomographic surveys were conducted by injecting electrical current between two electrodes across two drillholes, and then measuring the potential difference at the two electrodes immediately below the current injection electrodes (Palich and Qian, 2012). Results provide sections of resistivity and chargeability distribution between the two drillholes and therefore can assist in mapping conductor continuity.

IP data was presented as resistivity and chargeability strip logs and pseudosections plotted together with lithology, DTR Ni, Cr, Fe_2O_3 and magnetic susceptibility (Figure 9-4). Results indicate awaruite-bearing peridotite shows a general trend of low resistivity (<120 Ω·m) and high chargeability (>25 mV/V) (Palich and Qian, 2012), which is consistent with the findings of surface IP. More resistive zones, with or without chargeability, can be associated with unmineralized altered dykes and granitoid rocks.

Cross-hole tomographic data also shows that the more awaruite-enriched peridotite correlates with low resistivity and high chargeability (Palich and Qian, 2012). Low chargeability zones, with or without high resistivity, correlate with lower DTR Ni and typically do not exhibit connectivity between boreholes.

In conclusion, both surface and downhole IP indicate a weak positive correlation between DTR Ni and chargeability coupled with weak negative correlation between resistivity and DTR Ni (Palich and Qian, 2012). However, Palich and Qian (2012) suggest that this correlation could also reflect magnetite abundance, grain size and/or multi-element concentration as opposed to just awaruite enrichment, and so suggest caution is needed when using IP data for awaruite targeting.

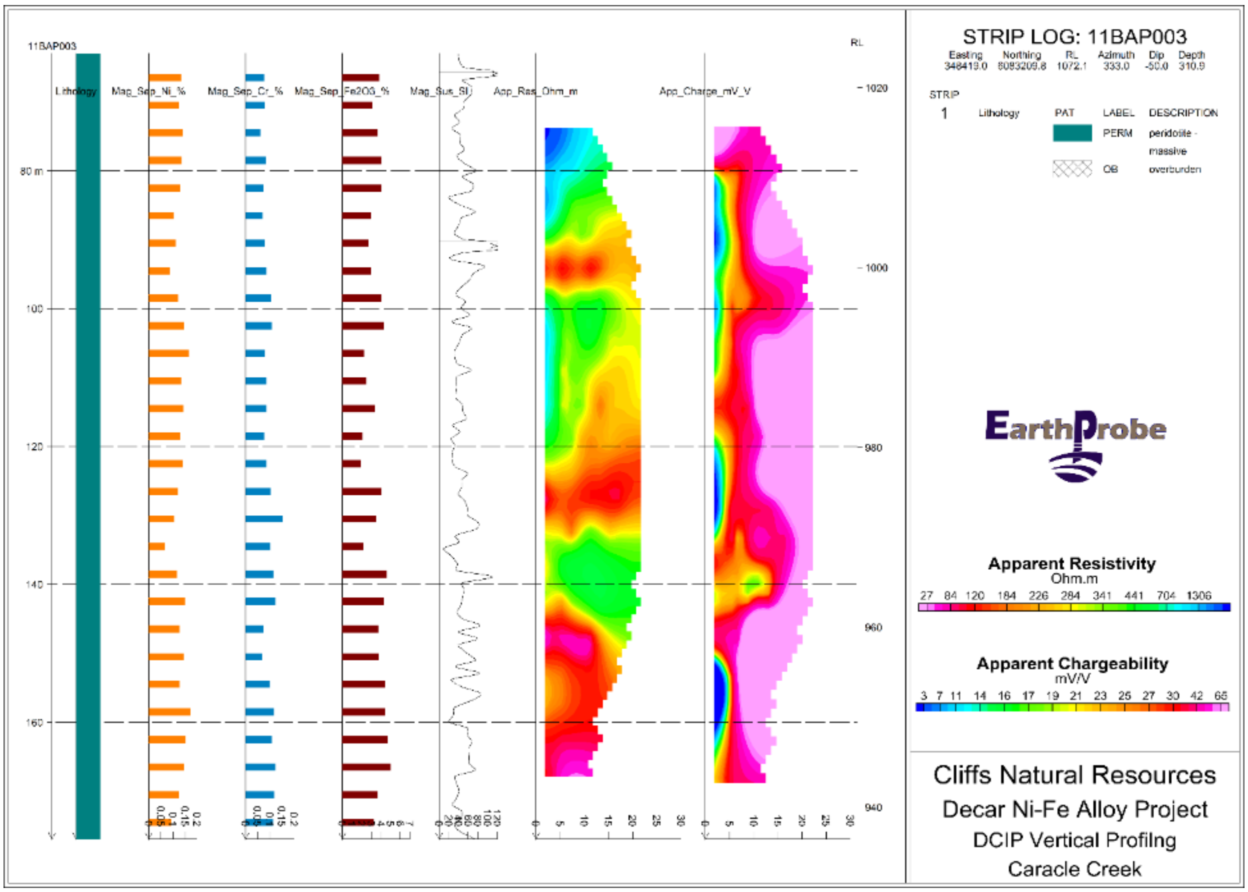


Figure 9-4: Example of a downhole IP strip log generated for drillhole 11BAP003 showing (from left to right) lithology, %Ni in magnetic separate, %Cr in magnetic separate, %Fe_2O_3 in magnetic separate, magnetic susceptibility, resistivity and chargeability. Source: Caracle Creek (2011)

9.2.4 Downhole Physical Rock Property Surveys

From 2010 to 2012, Cliffs sub-contracted DGI Geoscience Incorporated of Toronto, Ontario, (DGI) to complete downhole physical rock property surveys on 2010, 2011 and 2012 drillholes. These surveys were conducted after the drill rig had moved off the collar location and, consequently, a winch was used to move the various probes into and out of each drillhole. The 2010 program was a failure as all the holes had collapsed even though they had been lined with PVC, thereby preventing entry of the probes. Out of the 67 holes drilled in 2011 and 2012, twenty-one were sufficiently open so that they could be surveyed from the top (TOH) to end of hole (EOH), 31 were partially surveyed, and 15 were either obstructed at TOH or not surveyed.

Physical property surveys are used to characterize rock types, define property-specific domains and constrain geophysical modelling. The geophysical probes used for the physical property surveys acquire in-situ data at 10 cm intervals while being lowered and/or raised in the drillhole, with the probes in constant communication with the logging computer at surface. A complete list of holes surveyed as part of this program is given in the previous technical report (Ronacher et al., 2013). DGI delivered the physical property data as databases and strip logs (Figure 9-5).

Poly-electric data was collected in 47 drillholes with a 2PEA-1000 PolyElectric probe, which measures normal resistivity, fluid resistivity and fluid temperature (DGI, 2012b) along a vector perpendicular to the drill string. Normal resistivity can mark lithological changes whereas fluid resistivity is needed to correct for the influence of drilling mud and borehole fluid. Changes in fluid temperature can mark zones of water movement.

The natural gamma (γ) probe measures variations in the presence of natural radioactivity emitted by uranium, thorium and potassium, and consequently records changes in lithology. In addition, the natural gamma probe acquires spontaneous potential and single point resistance data, which provides additional data on lithology, borehole salinity and/or formational clay content (DGI, 2012b). Measurements were made on 46 drillholes using a 2PGA-1000 Natural Gamma probe.

Magnetic susceptibility and inductive conductivity was measured on 42 drillholes with a MagIC probe and helps delineate lithology by characterizing changes in the abundance of magnetic minerals (DGI, 2012b).

Rock density was measured in 21 drillholes using a 2GDA Focused Density probe. This probe uses a cesium-137 source to bombard wallrock with intermediate gamma ray energies that are then backscattered and received by the detectors to measure in situ density.

The neutron porosity probe uses an alpha emitting radioactive source, americium-241, mixed with beryllium to obtain relative neutron counts that are mostly related to hydrogen ion concentration (DGI, 2012b). Changes in these relative neutron counts could correlate with changes in lithology and/or porosity. Surveys were done on 21 holes using a 2LLP Neutron Probe. Twenty-one holes were also analyzed with a sonic probe.

Cluster analysis of physical property data was used to define seven groups, two of which (PP1, PP2) show a strong correlation with high values of DTR nickel (DGI, 2012a). The PP1 group is marked only by low resistivity whereas PP2 is defined by low resistivity, natural gamma and chargeability, as well as high conductivity and magnetic susceptibility. Intermediate DTR Ni grades are associated with low natural gamma and high resistivity (PP5), or low resistivity, chargeability and natural gamma coupled to high conductivity (PP7).

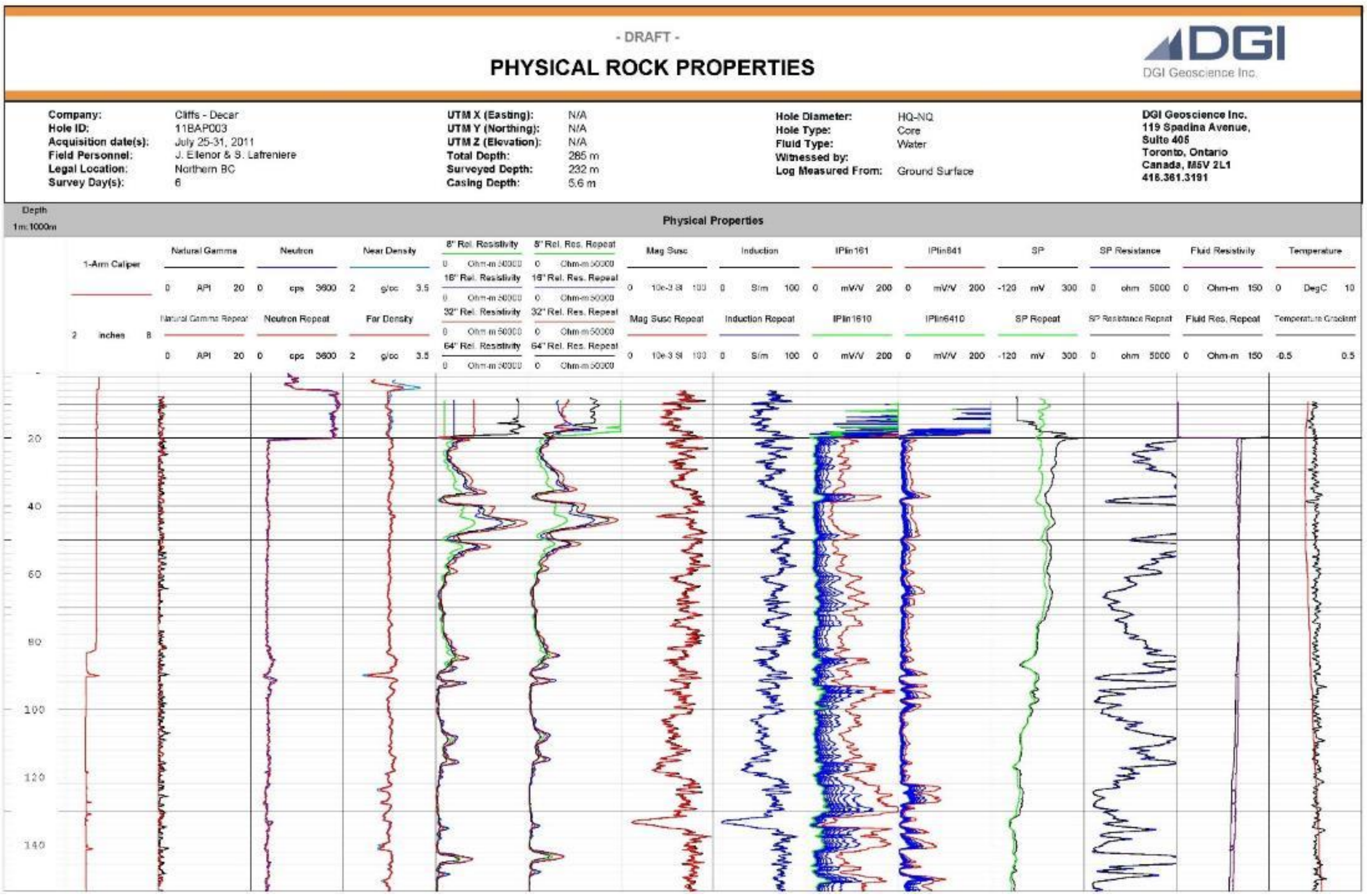


Figure 9-5: Example of physical rock property strip logs provided by DGI and showing (from left to right), borehole diameter, natural gamma, neutron, near density, relative resistivity, relative resistivity repeat, magnetic susceptibility, induction, IP in 161, IP in 641, SP, SP resistance, fluid resistivity and temperature. Source: DGI (2011)

9.2.5 Televiewer Surveys

Acoustic and optical televiewer surveys were completed by DGI together with the downhole rock property measurements (see Section 9.2.4). The acoustic televiewer (ATV) probe produces acoustic images of the borehole wall that can be compared with other geological logs to measure the in situ (or true) orientation of structural features like bedding planes, faults and fractures (DGI, 2012b). ATV surveys were completed on 17 of the 2011 holes and 19 holes from 2012, for a total of 36 surveys.

The optical televiewer (OTV) probe acquires a high-resolution digital image of the borehole wall under in situ stress, pressure and temperature conditions (DGI, 2012b), with the aim of identifying and characterizing structural features such as bedding, vein intersections, fractures and faults. Positional data is measured by an on-board magnetometer and inclinometer sensors, which allow the OTV data to be corrected to true azimuth and dip. OTV surveys were completed on 17 of the 2011 holes and 14 of those from 2012, for a total of 31.

FPX has summarized ATV and OTV data within spreadsheets that record the depth, true dip direction, true dip angle, width (in millimetres) and code for each structural feature. These features include major open, partially open and minor joint/fractures, bedding/banding/foliation, cleavage, water level, vein, fold, lithology contact, faults and shear zones. No publicly filed or internal reports are available that summarize the televiewer work, although the data is well-documented so that it can be effectively used for future work.

9.2.6 Lidar

A light detection and ranging (Lidar) survey was completed over most of the Property in 2012, by Terra Remote Sensing Incorporated of Sidney, British Columbia (Terra Remote). Data processing was completed in fall 2012 and the final report (Terra Remote Sensing Inc., 2013) is publicly filed as an appendix in the 2012 assessment report (Ronacher, 2013). The survey covered a 388.7 km^2 area at a line-spacing of 700 m. Terra Remote also took a digital 1:10,000 orthophoto of the Property.

The Lidar survey was flown with a Piper Navajo fixed-wing aircraft based in Burns Lake, BC, approximately 80 km southwest of the Property. The aircraft was equipped with a combined GPS/Inertial Navigation System (GPS/INS) to follow the pre-determined flight lines and maintain a nominal height of 1,150 m above ground level. An average speed of 234 km/hr was maintained along with a pulse repetition frequency of 100 kHz and swath speed of 34 times/second, thereby achieving a survey density of 1-2 points/m^2 (Terra Remote Sensing Inc., 2013).

Quality assurance and quality control (QA/QC) of Lidar data was monitored with four control points located within the project area. The difference between known and measured elevations for these control points ranged from +8 cm to -30 cm, and averaged -13 cm (Terra Remote Sensing Inc., 2013). Relative and absolute accuracy is subsequently estimated at ±15 cm and ±20 cm respectively. Lidar points were converted to 1 m contours to provide a more accurate topographic map of the Property.

10. DIAMOND DRILLING

Diamond drilling for Ni-Fe alloy deposits on the Decar Property was initiated in 2010, with subsequent campaigns during 2011, 2012 and 2017. Drilling focused on exploration (2010, 2011), resource definition (2011,2012) and resource expansion (2017), as well as infrastructure planning (2012) to support PEA-level studies (McLaughlin et al., 2013). In total these campaigns completed 88 holes for 32,140 m (Table 10-1), most of which was focused on the Baptiste Deposit. Other drill-tested targets include B and Sid. Depth, collar location, azimuth and dip data for the Baptise drilling is shown on Figure 10-1.

The following sections describe the four FPX / Cliffs drilling campaigns completed between 2010 and 2017. Similar procedures were followed in all programs. Most holes were drilled at dips of -50° to -60° through vertically-oriented mineralization, so that the horizontal and vertical extents of the mineralization is equal to 50-65% and 75-85%, respectively, of their downhole length. Fifty-four of the 76 holes (70%) drilled to delineate the Baptiste Deposit terminate in Awaruite mineralization (Figure 10-2), with several others mineralized to within 10 m from the end of hole.

Composites presented in Table 10-2 to Table 10-5 were calculated for intervals with contiguous assays exceeding 0.06% DTR Ni and containing less than 15 m of consecutive samples with grade below 0.06% DTR Ni. When an interval of >15 m of <0.06% DTR Ni was encountered, the composite was split.

Table 10-1: Summary of drilling done by FPX on the Decar Property

Year	Target	DDH (No.)	DDH (Total m)	Avg. m/DDH
2010	Baptiste	7	1,710.8	244.4
	Sid	3	847.3	282.4
2011	Baptiste	35	10,863.6	310.4
	B Target	1	304.5	304.5
2012	Baptiste	27	15,095.8	559.1
	Hydrogeological	7	1,401.0	200.1
2017	Baptiste	8	1,917.5	239.7
Total		88	32,140.5	

10.1 2010 Drilling Program

The 2010 drilling program on the Decar Property comprised 10 holes for 2558.1 m, with seven drilled on the Baptiste Deposit and three drilled on the Sid Target. Drilling took place during July and August 2010 and was completed by Radius Drilling of Prince George, BC who supplied one skid mounted rig that was later broken down to its components to complete the fly drilling program on the Sidney target.

Holes were spotted using a handheld Garmin GPS 60CSx device with a nominal precision of 5-10 m. After completion of the hole, FPX surveyed the collar locations with the same handheld GPS unit.

During drilling, boreholes were surveyed by the drill crew using a Reflex single shot instrument. Results show differences of 1.5° to 12.2° between planned azimuths and the upper-most single shot test, suggesting the magnetic nature of the rock may have offset drill alignment, single shot surveys or both. Subsequent downhole testing indicates that for every 100 m of drilling, holes steepen 0.6° and rotate 1.3° clockwise. This sort of consistency is atypical for tests affected by magnetic rocks, suggesting the 2010 single shot data may be more reliable than would be expected for surveys in magnetic rocks.

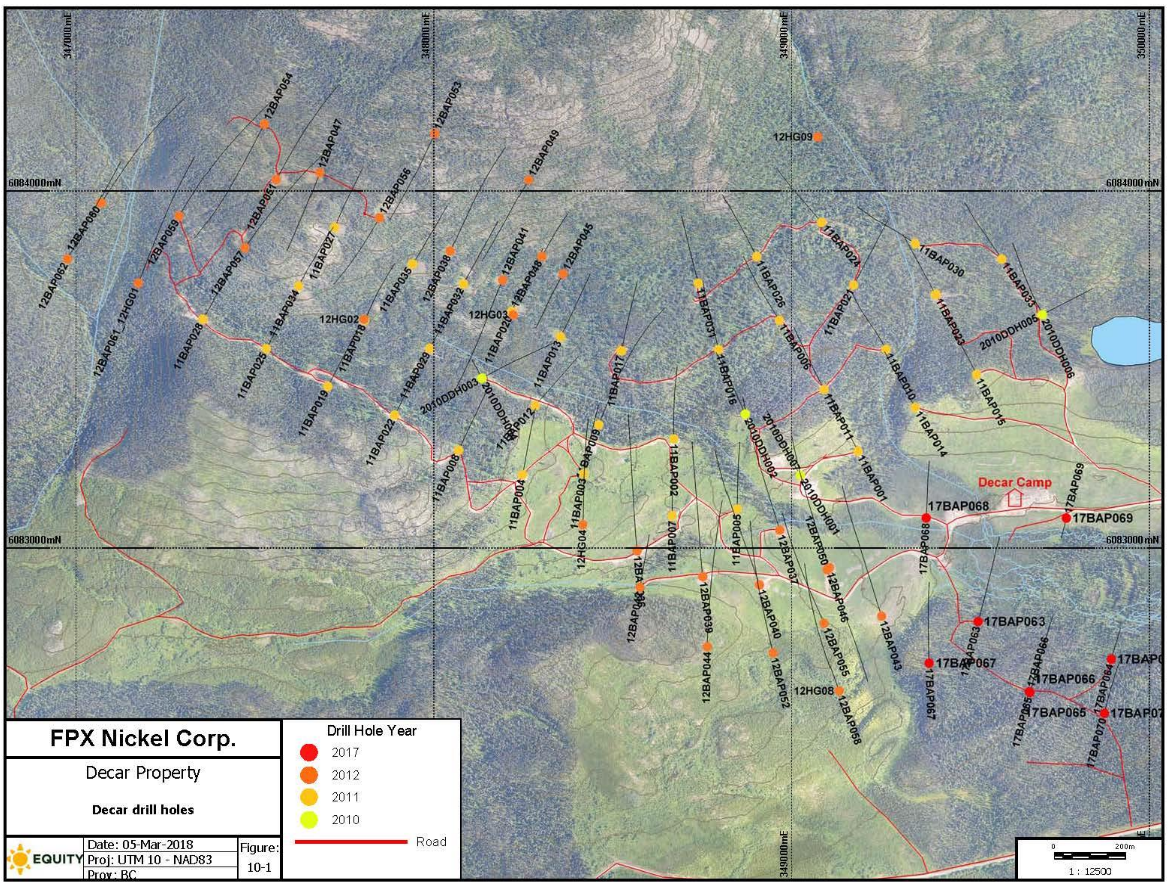


Figure 10-1: Dill collar map for the Baptiste Deposit. Source: Equity Exploration (2018)

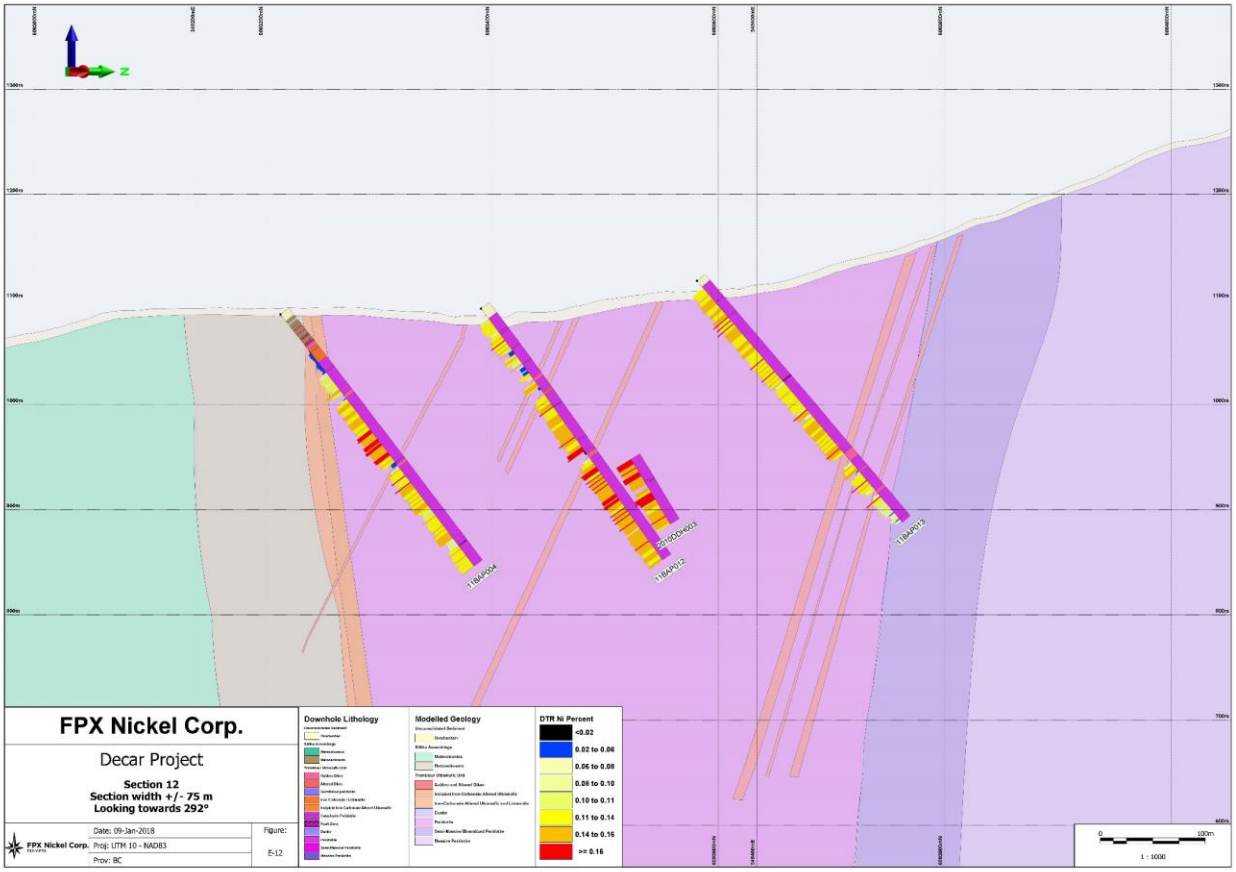


Figure 10-2: Section 12 through the Baptiste Deposit showing four drillholes that all end in mineralization
Source: FPX (2018)

The diameter of all 2010 core is NQ (47.6 mm), with the exception of 2010DDH001 which was drilled using HQ from surface to 17.68 m. Drill core was placed in wooden core trays at the drill site, that were then labelled with the hole ID and box number and transported to the Decar Camp for logging. Each core box was labelled with an aluminum tag indicating the hole number and core interval stored in each box. All 2010 core is currently stored in a fenced compound in Fort St. James, where it is ordered in core storage racks.

Recovery averaged 90.9% for the seven holes drilled at Baptiste and was slightly higher (91.8%) for the three Sid holes. Rock quality designation (RQD), which is measured by summing the length of all core fragments >10 cm long within each 3 metre run, is 50.6% for the Baptiste holes and 51.5% for Sid, with individual holes ranging from 43-63%. This RQD straddles the boundary between weathered (25-50%) and moderately weathered rock (50-75%). Magnetic susceptibility was measured at 1 m intervals and is very high, averaging 27-46 SI units for all 10 holes.

Drill core was logged with a focus on lithology, awaruite content and magnetic susceptibility. By logged metres, the lithology of the Baptiste holes comprised 72% serpentinized peridotite along with 14% other peridotite (i.e. massive, hornfelsed, cataclastic, mylonitized), 8% dikes and faults, and 7% overburden. Six of the seven holes hit long intervals of awaruite mineralization, with the exception being 2010DDH002. Just under 65% of the metres drilled at Sid returned serpentinized peridotite, with the balance intersecting unmineralized massive peridotite (21% of metres), listwanite (9%), dikes and faults. Long intervals of large awaruite grains were logged in two of these three holes.

Drill core sampling, assay methods and quality assurance/quality control (QA/QC) data is summarized in Section 11.0 of this report. Assay composites for the 2010 drilling are shown in Table 10-2, with the best intervals at Baptiste returned from holes 2010DDH001, 003 and 006. Hole 2010DDH001 is the central-most of these three holes, with 003 located 740 m west and 006 located 860 m to the northeast. Together with surface data, these three holes suggest contiguous awaruite mineralization over 1.5 km of strike length and to a true vertical depth of 250 m below ground surface. Hole 2010DDH007 also intersected high-grade DTR Ni but the hole was abandoned in a fault zone at 70.1 m depth.

Table 10-2: Composites >20 DTR Ni%*m from 2010 drilling on Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni	Grade * L	Comments
2010DDH001	3.1	322.2	319.1	0.123	39.337	Starts and ends in mineralization
including	3.1	172.5	169.5	0.132	22.291	
2010DDH003	47.2	340.5	293.2	0.138	40.471	Top to bottom
2010DDH006	13.5	337.0	323.5	0.112	36.172	Top to 3.5 m from bottom

10.2 2011 Drilling Program

The 2011 drilling program comprised 36 holes for 11,168.1 m, with 35 of these holes drilled on the Baptiste Deposit and one drilled at the B Target. Drilling took place during July to October 2011 and was completed by three contractors; Apex Diamond Drilling Limited of Smithers, BC, Element Drilling Limited of Winnipeg, Manitoba, and Midpoint Drilling Limited of Langley, BC. These three contractors supplied four drill rigs, two of which were helicopter-portable, one skid drill rig and one heli-portable rig that was converted to a skid rig.

The 2011 drilling program was managed by Caracle Creek. Drill collar locations were measured with a differential GPS (DGPS) system except for the single hole drilled at the B Target, which was surveyed with handheld GPS. DGPS uses a network of fixed ground-based and GPS satellite systems to achieve nominal accuracy of 10-15 cm. Handheld GPS uses only satellites and has nominal accuracy of 5-10 m.

The drill plan was designed to test the Baptiste Deposit on 13 sections at 200 m spacing, with each section containing one to five drillholes. Section orientation ranges from 028° at the western end of the Deposit area to 332.5° at the eastern end. Each hole from the 2011 program was drilled off its own pad.

The Baptiste holes were surveyed downhole with a Reflex Gyro (N = 22) or EZ-shot (N = 13) survey tool. There are no downhole surveys for the B Target hole. Gyro survey data suggests that for every 100 m of drilling, holes steepen by 0.1° and rotate 0.4° clockwise. EZ-shot data shows slightly higher deviation rates (-0.5°/100 m for dip; +1.6°/100 m for azimuth) which again suggests the single-shot tools performed adequately given the magnetic nature of the bedrock.

The upper-most 100-150 m of each hole was drilled as HQ diameter core (63.5 mm) followed by reduction to NQ (47.6 mm) for the remainder of the hole.

Drill core was placed in wooden core trays at the drill site, labelled with the hole ID and box number and then transported to the Decar Camp for core logging. Each core box was labelled with an aluminum tag recording the hole number, box number and core interval stored in each box. All the 2011 core is currently stored in a fenced compound in Fort St. James.

Recovery averaged 91.9% for the 35 holes drilled at Baptiste and 87.5% for the one hole drilled at B Target. RQD averaged 49.8% for the Baptiste holes with a range of 32-74%, corresponding to rock mass quality of weathered to moderately weathered rock. Only 16 of the 35 Baptiste holes have magnetic susceptibility data, with 14 of these averaging a very high 50-100 SI units and two averaging just 5 SI units. The two holes averaging 5 SI units (11BAP025, 028) intersected significant non-magnetic carbonate-altered ultramafic, dike and/or argillite.

Drill core was logged for lithology, structure and awaruite content. Lithologies intersected in the 2011 drilling are similar to the 2010 logging, with 72% comprising serpentinized peridotite, 15% comprising other types of peridotite and/or dunite, and the remaining 13% including dikes, argillite, chert, volcanic, overburden and fault zone. The hole drilled at B Target also was comprised of mostly peridotite.

Drill core sampling, assay methods and QA/QC data is reviewed in Section 11.0. Assay composites are summarized in Table 10-3. The best intervals at Baptiste were intersected in holes 11BAP003, 005, 007, 029 and 030, all of which returned 0.13-0.14% DTR Ni over 255-305 m. Another 17 holes returned composites averaging 0.11-0.13% DTR Ni over 190-295 m of core. Ten of the remaining 14 holes have intervals of >0.11% DTR Ni that are intercalated with >15 m thick intervals of unmineralized material, or mineralized intervals that start lower down the hole and continue to the end-of-hole (EOH). The four holes that returned the lowest-grade material were drilled on what is now the south margin of the Baptiste Deposit.

A composite calculated for the B Target drillhole returned 271 m of 0.134% DTR Ni, which is based on 1-metre samples taken every 4 metres.

Table 10-3: Composites >20 DTR Ni%*m from 2011 drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni	Grade * L	Comments
11B001	30.0	301.0	271.0	0.134	36.4	Starts and ends in mineralization; samples 1 every 4 m
11BAP001	86.14	275.2	189.1	0.129	24.3	Ends in mineralization
11BAP003	5.6	310.9	305.3	0.126	38.6	Starts and ends in mineralization
11BAP004	71.0	304.5	233.5	0.126	29.4	Ends in mineralization
11BAP005	45.0	304.5	259.5	0.143	37.1	Starts and ends in mineralization
11BAP007	48.0	304.5	256.5	0.150	38.5	Ends in mineralization
11BAP009	6.0	229.0	223.0	0.131	29.2	Starts in mineralization
and	249.0	400.0	151.0	0.134	20.3	
11BAP012	104.5	301.4	196.9	0.141	27.8	Ends in mineralization
11BAP013	9.0	216.0	207.0	0.135	27.9	Starts in mineralization
11BAP014	41.0	265.0	224.0	0.121	27.2	
11BAP017	17.0	304.9	287.9	0.118	34.1	Starts and ends in mineralization
11BAP018	6.3	301.0	294.8	0.117	34.5	Starts and ends in mineralization
11BAP020	7.0	288.0	281.0	0.122	34.3	Starts and ends in mineralization
11BAP021	21.0	302.0	281.0	0.122	34.2	Starts and ends in mineralization
11BAP022	109.6	301.5	191.9	0.116	22.3	Ends in mineralization
11BAP023	33.0	301.0	268.0	0.127	34.1	Starts and ends in mineralization
11BAP027	118.0	310.5	192.5	0.114	21.9	Ends in mineralization
11BAP029	11.0	298.4	287.4	0.127	36.6	Starts and ends in mineralization
11BAP030	13.0	301.5	288.5	0.127	36.7	Starts and ends in mineralization
11BAP032	6.0	258.0	252.0	0.116	29.2	Starts in mineralization
11BAP034	6.0	298.4	292.4	0.108	31.6	Starts and ends in mineralization
11BAP035	6.0	298.4	292.4	0.112	32.6	

10.3 2012 Drilling Program

The 2012 drilling program on the Decar Property comprised 34 holes for 16,496.8 m, with 27 of these drilled on the Baptiste Deposit (12BAP series) and seven drilled for hydrogeological purposes (12HG series). Drilling occurred from 25 June to 3 October 2012 and was done with three drill rigs contracted from Apex Diamond Drilling Limited of Smithers, BC, including two helicopter-portable rigs and one skid rig.

The 2012 drilling program was again managed by Caracle Creek and followed more-or-less the same procedures as the 2011 program, with the main change being continuous sampling of core as opposed to using regularly spaced intervals. Infill sampling of 2010 and 2011 drill core was also done to provide continuous sampling for those holes as well, although only results up to 11BAP011 (and none of the 2010 infill samples) were assayed by the cut-off date for inclusion in the 2013 resource update (Ronacher, 2013). The drill plan was designed to provide infill drilling within the western part of the Baptiste Deposit, and expand it to the west and southeast. Most holes were drilled off their own pad with the exception of 12BAP046/050 and 12BAP058/12HG08. Drill collar locations were measured with a DGPS.

The 27 holes drilled on the Baptiste Deposit have lengths between 300-603 m, with 25 of these ranging from 477-603 m. No holes were lost or abandoned due to bad drilling conditions. Hole azimuth was dependent on the collar location, with azimuths ranging from 028° (or 208°) on the west side of the Deposit to 340° (or 160°) on the east side. Twenty holes were drilled at an inclination of -50°, four were drilled at -60° and three at -65° to -70°. Six of the seven hydrogeological holes were drilled vertically (i.e. -90°) and ranged from 75 to 501 m in length.

Downhole surveys on 26 of the 27 Baptiste holes were done with the Reflex Gyro, with one hole collapsing before it could be surveyed (Ronacher et al., 2013). Only one of the seven hydrogeological holes was surveyed since six of them were drilled vertically. This survey was also done with the Gyro. All Gyro surveys were done from the top to the bottom of the hole at 10 m measurement intervals. Results indicate that for every 100 m of drilling the holes steepen by -0.2°/m and show 1.3° of clockwise rotation.

Recovery averaged 93.4% for the 27 holes drilled at Baptiste and 92.1% for the seven hydrogeological holes. Rock quality designation (RQD) averages 48.1% at Baptiste with a range of 30-75% (weathered to moderately weathered), which is almost identical to the 2011 data. Hydrogeological holes show RQD of 40.6% with a range of 14-69%.

Only 26 of 34 holes have magnetic susceptibility data, with 22 of these averaging 34-110 SI units and the remaining four averaging 0-5 SI units. Holes with low average susceptibility have higher proportions of non-magnetic lithology like Fe-carbonate altered peridotite, listwanite, mafic volcanic, argillite and dikes.

Drill core was logged for lithology, structure and awaruite content. Lithology comprised significantly less peridotite (63% of all core) than in previous years, with the intersection of more massive (i.e. unmineralized) peridotite suggesting that, in places, the edge of the Baptiste Deposit was likely reached. Other kinds of peridotite and dunite averaged just 9% whereas the balance of 12% is formed by argillite, dike, volcanic, overburden and fault zone.

Drill core sampling, assay methods and quality assurance/quality control data are reviewed in Section 11.0 and assay composites are summarized in Table 10-4. The best intervals at Baptiste were intersected in holes 11BAP036, 039 and 040, all of which returned 0.15% DTR Ni over 550-

570 m. Another 15 holes returned composites averaging 0.11-0.16% DTR Ni over 210-600 m of core, with higher grades (0.15-0.16% DTR Ni) occurring over shorter intervals (230-470 m). The eight holes with weak to negligible mineralization were drilled along the northern and southern margins of the Deposit.

Table 10-4: Composites >20 DTR Ni%*m from 2012 drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni	Grade * L	Comments
12BAP036	31.2	600.2	569.0	0.154	87.8	Starts and ends in mineralization
12BAP037	64.0	216.0	152.0	0.147	22.3	
and	298.0	600.0	302.0	0.146	44.1	Ends in mineralization
12BAP039	38.2	594.1	555.9	0.152	84.5	Ends in mineralization
12BAP040	33.0	588.0	555.0	0.152	84.3	Ends in mineralization
12BAP041	10.0	568.0	558.0	0.136	75.9	Starts in mineralization
12BAP043	33.3	426.0	392.7	0.155	60.9	
12BAP044	240.0	477.4	237.4	0.154	36.5	
12BAP045	6.0	487.0	481.0	0.139	66.9	Starts and ends in mineralization
12BAP046	28.5	292.0	263.5	0.142	37.3	Starts in mineralization
and	308.0	494.4	186.4	0.146	27.3	
12BAP047	6.0	600.0	594.0	0.128	75.8	Starts and ends in mineralization
12BAP050	34.5	249.0	214.5	0.140	30.1	Starts in mineralization
12BAP051	182.8	386.0	203.2	0.116	23.7	
12BAP052	271.0	600.2	329.2	0.154	50.6	Ends in mineralization
12BAP053	334.0	600.0	266.0	0.105	27.9	Ends in mineralization
12BAP054	2.7	600.0	597.4	0.127	75.9	Starts and ends in mineralization
12BAP055	106.0	569.7	463.7	0.156	72.4	Ends in mineralization
12BAP056	5.7	600.0	594.3	0.134	79.7	Starts and ends in mineralization
12BAP057	2.4	554.0	551.7	0.110	60.6	Starts and ends in mineralization
12BAP059	3.8	451.0	447.3	0.136	61.0	Starts in mineralization
12BAP060	156.0	404.0	248.1	0.150	37.1	
12BAP061	332.0	532.0	200.0	0.120	24.0	
12HG02	16.0	300.0	284.0	0.131	37.1	Ends in mineralization
12HG03	5.3	300.0	294.7	0.134	39.6	Starts and ends in mineralization
12HG04	176.0	380.0	204.0	0.130	26.6	

10.4 2017 Drilling Program

The 2017 drilling program on the Decar Property comprised eight holes for 1,917.5 m, all of which were drilled on the southeastern extension of the Baptiste Deposit. Drilling occurred from 19 August to 17 September 2017 and was done with one drill rig contracted from Apex Diamond Drilling Limited of Smithers, BC.

The 2017 drilling program was managed by Equity and followed similar procedures set out in the 2011 and 2012 programs. The objective of the drilling program was to expand near-surface, high-grade, mineralization at the southeastern end of the Baptiste Deposit. Seven of the eight holes were drilled off their own pad, with holes 17BAP065 and 066 drilled from the same pad but at azimuths of 014° and 194° respectively. Drill collar locations were initially spotted with a handheld GPS and then surveyed by HGH Land Surveying from Smithers, BC with a global navigation satellite system (GNSS) base station and real time kinematic (RTK) rover. This survey also verified the location of some 2011 and 2012 drillholes, as well as the Lidar base station location points.

Six of the holes were drilled from south to north (356° to 014°), with five of these drilled to 250-390 m depth. The aim of these holes was to expand the Baptiste Deposit to the southeast. The sixth hole was stopped at 141 m depth because of the strong diking present from the top to bottom of the hole. The two other holes were drilled from north to south (azimuth 194°-195°) to depths of 90-96 m, with the aim of closing off the southwestern margin of the Deposit. All holes were drilled at a dip of -50°.

Downhole surveys were done with the Champ Navigator (Champ Nav), a multifunctional solid-state gyro system with azimuth accuracy of ±0.75° and ±0.15° for dip. All Champ Nav surveys were done from the top to the bottom of the hole at 5-10 m measurement intervals. Hole deviation rates are similar to previous campaigns, averaging 0.4° m of dip steepening and 1.1° of clockwise rotation for every 100 m of drilling.

Recovery and RQD measurements were also similar to the 2012 campaign, averaging 93.8% and 46.1% respectively. Magnetic strength was measured with a KT-10 magnetic susceptibility meter, with all eight holes averaging 40-120 SI units that indicates strong magnetic susceptibility. Those holes with the highest average readings (>80 SI units) intersected long stretches of serpentinized peridotite with higher proportions of magnetite and awaruite whereas holes with lower averages contained more carbonate-altered and hornfelsed peridotite, as well as dikes.

Drill core was logged for lithology, structure and awaruite content. Lithology comprised significantly more overburden (13% of all logged core) and cataclastic peridotite (16%) than previous campaigns, and consequently less mineralized peridotite (52%). The balance is formed by hornfelsed peridotite (6%), dikes (5%), Fe-carbonate altered peridotite and listwanite (4%), dunite (2%) and fault zone (2%).

Drill core sampling, assay methods and QA/QC data are reviewed in Section 11.0 and assay composites are summarized in Table 10-5. Two holes (17BAP065 and 17BAP067) returned intercepts averaging 0.12-0.15% DTR Ni over 290-325 m. Three other holes returned higher grades over shorter intervals (i.e. 0.156% DTR Ni over 117.2 m in 17BAP063) or grades of 0.10-0.13% DTR Ni over 140-200 m of core length. Holes 17BAP064, 066 and 069 returned negligible results and so define the southeastern margin of the Baptiste Deposit.

Table 10-5: Composites >20 DTR Ni%*m from 2017 drilling on the Decar Property

Drillhole ID	From (m)	To (m)	Length (m)	DTR Ni	Grade * L	Comments
17BAP065	29.0	351.0	322.0	0.126	40.6	Starts and ends in mineralization
17BAP067	55.1	348.5	293.4	0.145	42.6	Starts and ends in mineralization
17BAP070	44.0	243.0	199.0	0.100	20.0	

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

Drill core sampling for the Baptiste Project was completed during six campaigns, with four of these occurring concurrent with drilling programs (2010, 2011, 2012, 2017) and two comprising infill sampling programs of 2010 and 2011 drill core in 2012. During the 2010 and 2011 drilling programs, samples were collected at regularly spaced intervals so that only approximately 13-24% of mineralized rock was initially sampled. Another 50% of 2010 core and 71% of 2011 core was sampled later as part of the 2012 infill sampling program. Samples taken concurrent with the 2012 and 2017 drilling programs typically run contiguously from the top to the bottom of the hole. Collectively, these six campaigns have sampled and analysed 94.8% of the core drilled by FPX on the Decar Property. The 5.2% of unsampled core consists of non-ultramafic rock types (e.g. dikes, volcanic, argillite) and the three holes drilled on the Sid Target in 2010.

The following sections describe how drill core was secured and handled, prepared for analysis, analysed and how the quality of these data were monitored using QA/QC protocols. All analyses used in the resource estimate were completed by Activation Laboratories Limited of Kamloops, BC, and, in 2010, Ancaster, Ontario (Actlabs). Actlabs is an independent commercial laboratory that has ISO 17025 accreditation. Core samples analysed by Acme Labs in 2010 were re-assayed by Actlabs to generate a consistent assay database.

11.1 Core Handling and Security

Core handling and security methods used for the 2010 program are not recorded but are probably similar to procedures used on the 2011 and 2012 programs, which are described by Ronacher et al (2013). Initial sampling of 2010 core was conducted as 1 m samples every fifth metre along the drill core, so that only approximately 13% of the core was sampled concurrent with drilling (Table 11-1).

Table 11-1: Overview of Decar drill core sampling campaigns

Campaign	Assay Samples			External* QA/QC Samples						
	N	*Ave L (m)*	*% of m*	*CRM***	*Blanks*	*Core Dup*	*Crush Dup*	*Pulp Dup*	*Total*	*%QA/QC*
2010 drilling	308	1.0	13.1%	18	8	17	nc	nc	43	12.3%
2011 drilling	2605	1.0	24.3%	222	72	144	83	216	737	22.1%
2010 resampling	330	3.7	49.8%	33	10	17	nc	nc	60	15.4%
2011 resampling	2594	2.9	70.5%	218	73	51	nc	nc	342	11.6%
2012 drilling	4153	3.7	96.5%	352	123	243	191	198	1107	21.0%
2017 drilling	460	3.4	93.9%	45	16	13	8	nc	82	15.1%
Total	10450	2.7	94.8%	888	302	485	282	414	2371	18.5%

*External = inserted by FPX, Caracle Creek or Equity; internal lab QA/QC was not compiled
**2-3 standards inserted for one CRM sample ID for the 2010 and 2011 drilling campaigns
nc = not compiled

Core sampling concurrent with the 2011 drill program collected 1 m samples every fourth metre along the drill core, irrespective of rock type (Ronacher et al., 2013). The sampling protocol was originally designed to preserve core for potential follow up metallurgical testing. The selected drill core intervals were cut with a core saw, with one half placed into a plastic bag with a pre-numbered sample tag ("core sample") and the other half placed back in the core box for reference ("reference core"). Certified reference material (CRM) and blank samples were alternately inserted as every 20th sample. One core duplicate was also inserted in every 20 samples and comprised a quarter-core sample obtained by splitting the reference core, with the core duplicate then placed in its own plastic bag with a pre-numbered sample tag. All plastic bags containing core samples were sealed into rice bags. Sample transport from camp to Smithers, BC, was handled by CJL Enterprises Ltd of Smithers, BC, and Bandstra Transportation Systems Ltd. (Bandstra) then shipped these samples to the Actlabs facility in Ancaster, Ontario, for analysis.

The 2012 core and infill sampling programs employed similar operating procedures to the 2011 program, with a few exceptions that are described here. Core sampling was done in contiguous 4 m intervals, as opposed to regularly spaced intervals, so that nearly 97% of the 2012 core had been sampled by completion of the drilling program. Infill sampling of 2010 and 2011 core was also completed to fill in the 3-4 m gaps between the regularly spaced 1 m samples that were taken concurrent with those programs. Sample transport from the Decar camp to Smithers, BC, was handled by Rugged Edge Holdings Ltd of Smithers, BC, and Bandstra then transported these samples to the Actlabs facility in Kamloops, BC, for analysis.

The 2017 drilling program employed similar core handling and security procedures to those used in 2012. Sampling was again done in contiguous 4 m intervals, and the core-cutting, shipment, QA/QC insertion rates were all similar to previous programs. Core samples were sealed in plastic bags, then aggregated into groups of 5-10 that were sealed into rice bags with a uniquely numbered security tag. Samples were transported to Prince George, BC, by Equity and then to Actlabs in Kamloops, BC, by Bandstra.

Drill core from the 2010, 2011, 2012 and 2017 programs has been transported to Fort St. James, BC, where it is stored in a fenced compound owned by Russell Transfer Ltd. Core boxes are stored in metal racks and are generally well-preserved as of the last site visit conducted by the QP in October 2017.

The core cutting, bagging and transport procedures for all programs are industry standard. The authors are unaware of any security concerns related to drill core from the 2010, 2011 and 2012 programs, and none were mentioned in the preceding technical reports (McLaughlin et al., 2013; Ronacher et al., 2012a, 2013), which were written by the managers of the 2011 and 2012 exploration programs. No security concerns were reported for the 2017 program. Core sampling is comprehensive and consistent, with assays for 94.8% of all bedrock drilled and all analyses conducted at Actlabs. The QA/QC sample insertion rate of 18.5% is within the recommended best practise range of 15-20% (Abzalov, 2008; Mendez, 2011) and does not include check assays or duplicates analyzed by Actlabs. Adding these in would likely increase the insertion rate to >20%.

11.2 Analytical Techniques

This section describes the analytical methods used for assay of drill core and QA/QC samples, followed by summaries of the CRM, blank and core duplicate performance in Section 11.3. A detailed description of the 2011 and 2012 QA/QC monitoring is found in the 2013 technical reports by Ronacher et al (2013) and McLaughlin et al. (2013).

All assay results for drill core samples are compiled into a single database, although without results for QA/QC and re-assayed samples. For the purposes of this NI 43-101 report, QA/QC data was compiled from several Excel spreadsheets and Access exports provided to Equity by FPX. The omission of re-assays from the master database was discovered through a random spot check of 100 XRF and 100 ICP analyses as part of the data validation procedure for this report. In Section 18.0, we recommend that both the QA/QC and re-assay data is integrated into the core assay database by rebuilding it directly from the original assay certificates.

Geochemical assay of samples collected concurrent to 2010 drilling were completed by Acme Analytical Laboratories Ltd of Vancouver, BC (Acme), an ISO 9001 accredited laboratory that was purchased by Bureau Veritas Mineral Laboratories in 2012. All subsequent assays were done by Actlabs, at their Ancaster, Ontario facility in 2010 and Kamloops, BC facility from 2011 to 2017. In addition, coarse rejects from the 2010 samples analyzed by Acme were re-analyzed by Actlabs to generate consistency among all analyses. Both Actlabs facilities are ISO/IEC 17025:2005 accredited, meaning they meet the general competency requirements to carry out tests and/or calibrations using standard, non-standard and laboratory-derived methods (ISO, 2005).

All mineralized core samples were assayed for “total nickel” and “recoverable nickel” (Table 11-2), with recoverable nickel analyses measuring only the nickel hosted in awaruite and sulphide minerals (e.g. heazlewoodite, bravoite, pentlandite) whereas the total nickel analyses measured both recoverable and refractory nickel, the latter hosted in silicate phases like olivine and, to a lesser extent, serpentine. Sample preparation procedures comprise crushing of the entire core sample followed by pulverizing a 250 g sub-sample to 95% passing 75 µm (200-mesh). Pulp for total Ni analysis is then fused with lithium metaborate/tetraborate flux and analysed by inductively coupled plasma optical emission spectrometry (ICP-OES). Recoverable nickel is determined by first running the pulp through a Davis Tube magnetic separator (Davis Tube), which splits the pulp into a magnetic and non-magnetic fraction. The magnetic fraction is then fused with lithium metaborate/tetraborate flux and analysed by X-ray fluorescence (XRF). Davis Tube Recoverable nickel (DTR Ni) is calculated using the equation shown in Section 8.

In 2010, Acme determined total nickel with a 4-acid digestion and ICP-OES finish whereas recoverable nickel was assayed with a non-standardized technique that used a proprietary selective extraction for nickel-in-alloy and an ICP finish (8FPX method). The selective extraction targeted metallic nickel from non-silicate phases, specifically awaruite (T. Rabb and P. Bradshaw. personal communication, 09 January 2018). The proprietary selective extraction method was used only concurrent with the 2010 drill program, after which FPX switched to Actlabs and re-assayed the 2010 coarse rejects with XRF on the Davis tube magnetic fractions.

Table 11-2: Overview of analytical methods for drill core used by FPX

Campaign	Lab	Preparation		Total Nickel		Magnetic Separation		Recoverable Nickel	
		Code	*Method*	*Code*	*Method*	*Code*	*Method*	*Code*	*Method*
2010 drilling	Acme	R200 -250	250 g passing 75 µm	1E	4-acid digest, ICP	n/a	n/a	8FPX	"metallic Ni by FPX method"
2010 re-assay 2011 drilling 2010 infill 2011 infill 2012 drilling 2017 drilling	Actlabs	RX-1SD	237-250 g passing 75 µm	4B	$LiBO_2$-LiB_4O_7 fusion, ICP-OES	8-DTMS	Davis Tube magnetic separation	4C	$LiBO_2$-LiB_4O_7 fusion, XRF

11.3 Quality Assurance and Quality Control (QA/QC)

QA/QC samples monitor analytical accuracy and precision with CRM and duplicate sample analysis respectively, as well as potential cross-contamination during sample preparation (with blank sample analysis).

CRMs were used to monitor accuracy of both total and recoverable nickel assays, as well as Davis Tube magnetic separation. Total Ni assays were predominantly monitored with OREAS 13b, which is certified for a 4-acid digestion whereas the Decar samples were prepared with lithium-borate fusion. This mismatch in digestion methods does not follow industry best practise of matching the digestion method of CRM and core samples and reduces the relevance of CRM performance. Calculation of “Z-scores” (number of standard deviations an element is from the mean) shows that only a single CRM exceeds the widely used threshold of ± 3 for quality control failure, suggesting that assays are generally accurate. On the other hand, 85% of the OREAS 13b analyses returned a Z-score >0, with an average of +0.9 (Table 11-3). These results suggest there is a positive bias in the analytical data. Assays of OREAS 72b, 74b, 75b, which were certified with the same borate fusion used by Actlabs, are similarly biased, with 74% of Z-scores >0 (Figure 11-1).

Recoverable nickel assays were completed on magnetic separates generated by the Davis Tube, and were monitored with higher-grade CRMs that have certified means between 0.4 to 5.4% Ni (Table 11-3). Between 2010 and 2012 the most frequently used CRM was OREAS 73a, where 81 analyses returned an average Z-score of +0.1 which suggests accurate and unbiased analyses. OREAS 72b, 74b and 75b, on the other hand, returned average Z-scores between +0.6 to +0.9 (Table 11-3) that again is suggestive of a positive analytical bias (Figure 11-2). All of these CRMs were certified with the same lithium borate fusion and XRF finish used by Actlabs.

In 2017, recoverable nickel assays were monitored with CRMs CDN-ME-9 and CDN-ME-10, both of which were certified through methods that are different from those used by Actlabs (Table 11-3). Again, such a mismatch does not follow industry best practise. A total of 24 analyses were done on these two CRMs, returning no QA/QC failures but average Z-scores that, again, suggest a positive bias in the analytical data.

Starting in 2011, the accuracy of Davis Tube magnetic separation was monitored with an internal (or non-certified) DTR replicate developed by Cliffs from 50 kg of pulverized, awaruite-mineralized, ultramafic rock taken from the Baptiste Deposit (Ronacher et al., 2013). The DTR replicate was used to monitor the Davis Tube magnetic separation and XRF analysis of the magnetic separate. 257 analyses of the DTR replicate reside within the FPX database, with percent DTR Ni values returning an average coefficient of variation (CVave) of 11%, with 95% of the data returning CVave of 7%. In comparison, the CRMs used on the Baptiste Project have CVave of <5% for nickel. Based on the mean and standard deviations determined from the assay data, overall failure rates (i.e. Z-score >±3) for the DTR replicate were very high at 21%. The DTR replicate therefore performed significantly worse than a CRM.

A new set of internally derived DTR Ni replicates was used for the 2017 program and was made from outcrop samples of awaruite-mineralized peridotite collected from the Decar Property in 2007 (Voormeij and Bradshaw, 2008), 2008 (Britten, 2009) and 2013. The 21 replicates assayed along with the 2017 core samples show CVave of <5% for percent DTR Ni, indicating CRM-like levels of precision.

Table 11-3: Overview of CRM details and their performance for FPX on the Decar Property

CRM or replicate	Monitors?	Details of Certification			Performance in FPX work				
		Method	*Mean %Ni*	*1SD %Ni*	*Method*	*N**	*Mean Ni*	*Z-score Ave*	*Z-score >±3*
Certified CRM									
OREAS13b	Total Ni	4-A, ICP	0.2247	0.0155	Fusion, ICP	275	0.2383	0.9	1
OREAS 72b		Fusion, ICP	0.7050	0.0253	Fusion, ICP	58	0.7250	0.9	1
OREAS 74b		Fusion, ICP	3.4286	0.1186	Fusion, ICP	57	3.4396	0.5	0
OREAS 75b		Fusion, ICP	5.3621	0.1804	Fusion, ICP	52	5.4498	0.7	2
Total or average						**442**		**0.8**	**4**
OREAS 72b	Recoverable Ni	Fusion, XRF	0.7086	0.0178	Fusion, XRF	66	0.7250	0.9	0
OREAS73a		Fusion	1.44	0.06	Fusion, XRF	81	1.44	0.1	0
OREAS 74b		Fusion, XRF	3.3933	0.0932	Fusion, XRF	65	3.4454	0.6	0
OREAS 75b		Fusion, XRF	5.3825	0.1020	Fusion, XRF	57	5.4400	0.6	0
CDN-ME-9		4-A, ICP	0.912	0.031	Fusion, XRF	11	0.956	1.4	0
CDN-ME-10		4-A, ICP	0.428	0.012	Fusion, XRF	13	0.438	0.8	0
Total or average						**293**		**0.6**	**0**
Non-certified replicates									
DTR	DTR Ni	DTR, fusion, XRF	0.1266	0.0058	DTR, fusion, XRF	257	0.1293	0.5	54
07PXB028		DTR, fusion, XRF	0.0574	0.0031	DTR, fusion, XRF	6	0.0586	0.4	0
08RMB214		DTR, fusion, XRF	0.0327	0.0022	DTR, fusion, XRF	9	0.0330	0.1	0
13TAR001		DTR, fusion, XRF	0.0739	0.0064	DTR, fusion, XRF	6	0.0730	-0.1	0
Total or average						**278**		**0.4**	**54**

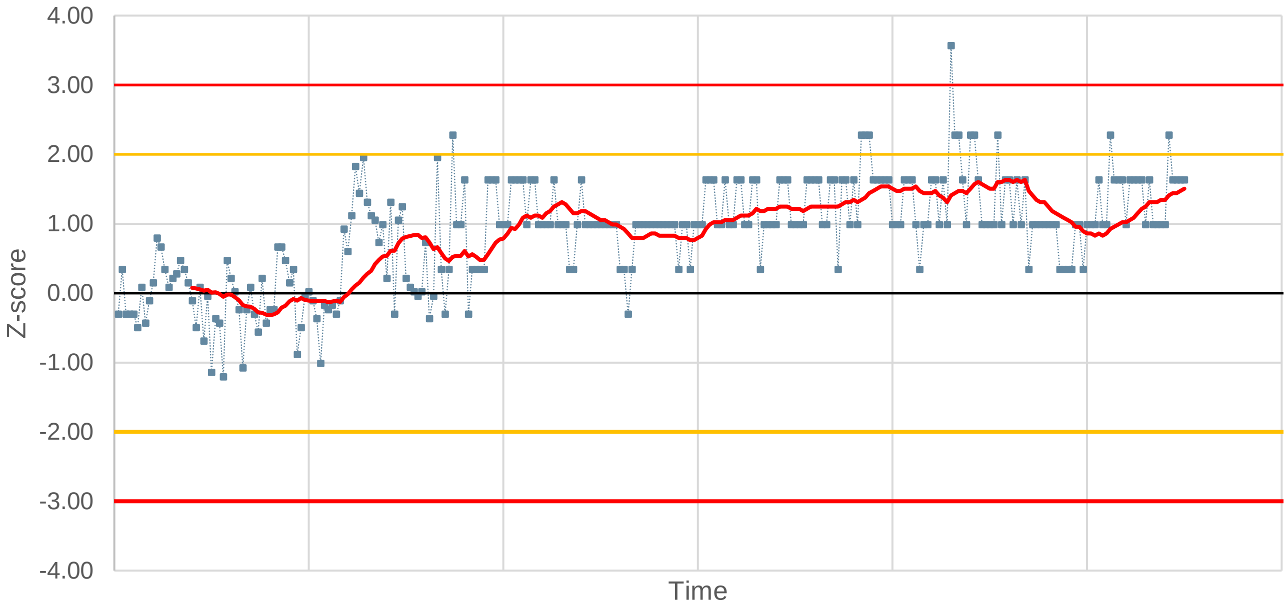


Figure 11-1: Shewhart control chart showing Z-scores against time for OREAS 13b which tracks total nickel analyses by ICP. Positive and negative bias is indicated by Z-scores >0 and <0 respectively. Red and orange lines show Z-scores of ±3 and ±2 respectively. Source: Equity Exploration (2020)

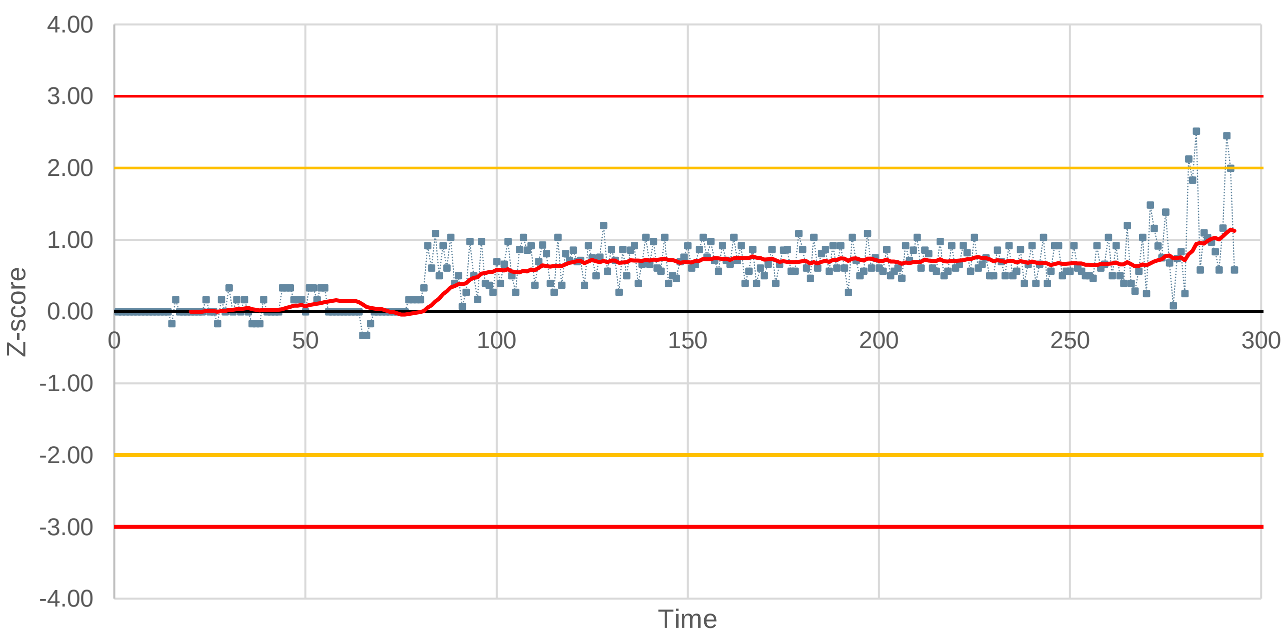


Figure 11-2: Shewhart control chart of Z-scores for CRM used to track XRF analyses of magnetic separate Positive and negative bias is indicated by Z-scores >0 and <0 respectively. Red and orange lines show Z-scores of ±3 and ±2 respectively. Source: Equity Exploration (2020)

Cross-contamination at the crushing and/or pulverization stages was monitored with samples of blank material. In 2011 and 2012 this material is referred to as “quartz” and “quartz sand” (here assumed to be the same material) whereas blank used for the 2017 program comprised unmineralized bedrock from the Sitlika assemblage. Quartz sand would only assess contamination in the pulverization stage since the sand would not undergo crushing during sample preparation. Out of the 274 blank assays in the database, 97% of the XRF and 95% of ICP-OES analyses were at or below detection. Two of the XRF assays exceeded the threshold of 10x detection limit for contamination and none did for ICP-OES. These results suggest that cross-contamination at the preparation stage is insignificant.

The Sitlika bedrock blank used in 2017 was used to monitor potential cross-contamination within the Davis Tube. Prior to sampling, outcrops were tested with a KT-10 magnetic susceptibility meter to ensure that they were non-magnetic. This blank material was then crushed and pulverized in the lab and ran through the Davis Tube magnetic separator, yielding magnetic separates weighing between 0.004-0.145 g from a 30 g parent sample. In comparison, 30 g of mineralized peridotite typically yields close to 2 g of magnetic separate, or approximately 10-500 times that within the 2017 blank material.

Duplicate samples are used to quantify the reproducibility of assays in core, coarse crush and pulp material, typically showing increased precision from core through to pulp. Core duplicates consist of quarter core collected on site, crusher duplicates are split off the parent between the coarse crushing and pulverization stages at the lab, and pulp duplicates are split after pulverization. Precision of duplicate %Ni analyses can be quantified by calculating the average coefficient of variation (CVave), which can then be compared to published values (Abzalov, 2008) to provide some context (Table 11-4). For borate fusion and ICP-OES analysis of core, crush and pulp duplicates, CVave is within the best practise ranges provided by Abzalov (2008) for Cu-Mo-Au porphyry (Table 11-4), which are comparable to Decar in that they are also bulk tonnage deposits. Likewise, all three types of duplicate XRF analyses and magnetic separations also fall within the acceptable to best practise ranges proposed by Abzalov (2008).

Table 11-4: Average coefficient of variation for Decar duplicates by different analytical methods

Analysis	Duplicate	N @100%	CVave @ 100%	N @95%	CVave @ 95%	CVave Practice
Ni by borate fusion, ICP	Core	426	8.7%	405	5.6%	Best
	Crusher	79	7.6%	75	6.1%	Best
	Pulp	406	5.2%	386	3.7%	Best
Ni by borate fusion, XRF	Core	390	13.9%	371	10.6%	Acceptable
	Crusher	229	6.6%	218	4.3%	Best
	Pulp	189	3.6%	180	2.9%	Best
Magnetic separation (>0.1 g)	Core	393	14.7%	373	11.5%	Acceptable?
	Crusher	237	14.6%	225	9.4%	Acceptable?
	Pulp	181	7.0%	172	3.1%	Best?

11.4 Data Adequacy

It is author Voordouw's opinion that the methods used to split, sample, secure and transport drill core are industry standard, and that the QA/QC procedures are in line with industry best practise. QA/QC sample insertion rates exceed industry best practice.

The analytical methods and QA/QC results are also considered adequate for the purposes of this report; however, two recommendations derived from author Voordouw's review of the analytical data are as follows:

- Integrate QA/QC and re-assay data into the project geochemistry database by rebuilding the database directly from the original assay certificates.
- Determine the reason for the positive bias in nickel analyses, possibly through a reanalysis of CRMs with methods used to certify them and those methods used to analyse the Decar samples.

12. DATA VERIFICATION

Data verification by author Voordouw was done from 19 September to 21 September 2017. No drilling or other exploration work has been done on the Decar Property since this 2017 site visit and so no new inspection was done for the purposes of this report.

Besides the site visit, other data verification work done by Voordouw in 2017 included (1) 100 spot-checks of analytical data in the database against original laboratory certificates, (2) re-calculation of composite intervals presented in sections 10.1, 10.2, 10.3 and 10.4 of this report, and (3) independent compilation and interpretation of QA/QC data. In addition, the 2017 exploration program on the Decar Property was managed by author Voordouw's employer (Equity), which is independent of FPX as defined in NI 43-101. The 2011 and 2012 exploration programs were also managed by independent consultants.

The first day of the 2017 site visit occurred two days after the completion of the 2017 drilling program, although core processing was still on-going for drillhole 17BAP070. Author Voordouw liaised with the Equity project geologist who had been on site since 6 September 2017, and had directly overseen the drilling, logging and sampling of holes 17BAP067, 068, 069 and 070. Several poly-ethylene sample bags with core and QA/QC samples were examined (Figure 12-1a). Samples comprised serpentinized peridotite with visible awaruite that subsequently assayed 0.099% DTR Ni over 90 m. All reference core for 17BAP068 was examined and found to host relatively abundant visible awaruite from 26 m to 174 m core depth (Figure 12-1b). This interval assayed 0.127% DTR Ni over 148 m.

The Baptiste Deposit was traversed on 20 September 2017 to confirm the location of 2017 and historical drill collars (Figure 12-1c), as well as outcrops of Trembleur ultramafite mapped by FPX. Collar locations were measured with a handheld Garmin GPSMAP62. Offsets between 15 collar locations measured by author Voordouw and their surveyed locations in the database range from 0.4 - 4.1 m (average 2.3 m), which is within the ±5-10 m precision typical for handheld GPS. Elevations are within 6 m of those surveyed with DGPS. The collar azimuth and dip measured on the drill casings were also comparable to those recorded in the database.

Several outcrops examined at the northwestern end of the Baptiste Deposit were found to consist of peridotite and dunite (Figure 12-1d). Other outcrops found along the road include argillite and mafic volcanic of the Sitlika assemblage, the distribution of which is consistent with property-scale mapping.

The historical core yard in Fort St. James was visited on 21 September 2017 (Figure 12-1e), with the aim of cataloguing the inventory and reviewing some key intervals of drill core. The inventory was verified at approximately 30,200 m of core, which is consistent with the sum of metres drilled in 2010, 2011 and 2012. Historical core is clearly labeled and well-preserved. Historical core intervals that were reviewed include 18-64 m from 11BAP008, 22-39 m in 11BAP027 and 45-125 m in 12BAP043. The interval in 11BAP008 includes the Sitlika argillite and mafic volcanic that bound the Baptiste Deposit to the southwest, 11BAP027 includes an internal of (low-grade) mineralized dunite, and 12BAP043 comprises one of the higher-grade DTR nickel intervals intersected at Baptiste (Figure 12-1f).

Based on the above steps taken by Voordouw, it is his opinion that the Baptiste Project data is adequate for the purposes of resource estimation as presented in this report.

Figure 12-1: Photos from Voordouw's 2017 site visit that show (a) poly-ethylene bags with samples from 17BAP070, (b) awaruite mineralization in serpentinized peridotite, 17BAP067 at approximately 125 m depth, (c) reclaimed drill pad and casing for 17BAP068, with camp in the background, (d) outcrop of peridotite at northwest end of Baptiste Deposit, (e) part of the historical core storage yard in Fort St. James, and (f) serpentinized peridotite with high-grade awaruite mineralization at 85 m depth. Source: photographs by Ron Voordouw (2017).

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A significant volume of metallurgical testwork has been completed toward the development of a successful process design for the Baptiste Project of FPX Nickel Corporation (FPX). This work has spanned approximately 10 years and a number of different metallurgical test facilities using representative sample materials have been obtained from the Baptiste Project. The objective of this testwork has been to provide a means to recover nickel into a saleable concentrate and also to look at the possibility of producing a by-product magnetite concentrate.

Table 13-1 outlines the various testwork programs, sample materials and dates of testing for the metallurgical development outlined in this section. It should be noted that the nature of nickel mineralization within the Baptiste Deposit is somewhat unique and industry experience in the treatment of this type of mineralization is rather limited. FPX has needed to develop this process from essentially first principles using industry proven, conventional mineral processing methods.

Table 13-1: Summary of various testwork programs for the Baptiste Project

Date	Test facility	Test objectives	Sample size (kg)
Jun-Sep. 2010	Knelson (KRTC)	Gravity concentration of nickel values	20
Feb 2012	SGS - Lakefield	Evaluation of magnetic and gravity recovery of nickel	600
Mar 2012	ALS Metallurgy	Magnetic and gravity pilot plant testing	16,000
2018-2019	ALS Metallurgy	Magnetic and flotation testing	500
Oct 2019	Sherritt Technologies	Evaluation of pressure leaching	0.13

SGS and the KRTC were engaged by Cliffs Natural Resources (the former joint venture partner of FPX, then known as First Point Minerals Corp.) to perform initial mineralogical evaluations and nickel upgrading testwork. The objective of this work was to determine the major mineralogical characteristics of the samples and to provide preliminary separation testwork. Magnetic and gravity processes were tested extensively in attempts to upgrade nickel values to marketable concentrate grades. This was subsequently followed by a pilot plant programme aimed primarily at producing a sizeable concentrate sample to be used for product marketability development with potential users.

In early 2018, a review of previous testwork for the Baptiste Project was completed and a new track of testwork was initiated at ALS Metallurgy of Kamloops, BC. Testwork conducted at ALS in 2018 was focused on magnetic separation followed by the recovery of a separate nickel concentrate using flotation. A new composite sample was used for the 2018 ALS testwork, representing early stage (starter pit) mine production.

In October 2019, Sherritt Technologies was mandated by FPX to evaluate the behaviour of awaruite/nickel concentrate from the Baptiste Project to pressure leaching. This is a first step in evaluating the prospects of producing a nickel sulphate product for potential use in batteries for electric vehicles.

Metallurgical results have improved over the 10 years of testwork and the most recent ALS testwork results are considered to be substantially better than earlier testwork results. Additional support work has been completed in the form of mineralogical analysis, gravity recovery testwork and assaying to support the various metallurgical programs.

Gravity testwork conducted by Knelson Research is not discussed in this Chapter as the proposed process flowsheet does not incorporate gravity separation. Results from the ALS pilot testing are also not discussed in this Chapter as this work was directed mainly for producing a large product sample and not for process development. Current testwork results were however compared with some of the results from the pilot program.

13.2 Summary of Metallurgical Testwork Results

Various testwork programs completed for the Baptiste Project have used distinct process options in attempts to obtain suitable metallurgical objectives in terms of nickel recovery and concentrate grades. Within the various testwork programs, mineralogical analyses of the feed samples have consistently shown little variation of the nickel and iron mineralogy for widely dispersed composite samples. This is consistent with the overall geological description of the deposits, with respect to the awaruite occurrence. Key mineralogical parameters are shown in Table 13-2 for a number of composite samples from various testwork programs.

Table 13-2: Comparison of key mineralogical parameters

Report date	Test facility	Total Ni (%)	Ni in Aw[1] (%)	DTR[2] Ni (%)	Magnetite[1] (%)
February 2012	SGS Lakefield	0.25	74.5	63.0	5.0
March 2013	ALS Met.	0.21	79.0	60.0	4.5
2018 - 2019	ALS Met	0.23	78.0	69.0[3]	4.5

[1]From QEMSCAN, [2]DTR refers to a "Davis Tube Recovery", which is a magnetic recovery test conducted at approximately 75 µm. It is a common test used in the iron ore mining industry, [3]Estimated from UBC analysis

The recovery of nickel from the Baptiste Project is complicated by a number of unique key geological influences that are not common within base metal deposits. These unique geological features include:

1. The nickel mineral awaruite is formed by reducing nickel originally contained in the host rock by metamorphic processes, resulting in the mineral awaruite being formed and disseminated throughout the host rock. This process has resulted in a wide-range of awaruite grain sizes within the rock mass. It has also made the deposit very consistent in terms of contained nickel values and grain textures, which may be of some benefit in operations. Awaruite at Baptiste has the form Ni_3Fe and contains approximately 76% Ni by weight.
2. Not all the contained nickel in the deposit has been converted to awaruite, and as such, some nickel is contained in the form of sulphide and silicate minerals. These phases of nickel are not recovered in processes used to recover awaruite. This fact complicates the ability to easily account for nickel in metallurgical results. This dual mineralogical nature of nickel could be compared to copper sulphide and copper oxide minerals existing within a deposit and complicating the copper recovery calculation. In the iron ore industry, for deposits containing magnetite, magnetically recoverable iron values (using low intensity magnetic separation), are quantified using a Davis Tube Recovery (DTR) test. Attempts have been made to use the Davis Tube as a means of quantifying the recoverable nickel content (awaruite) of the various Decar materials. Awaruite is strongly ferromagnetic and recovered concurrently with magnetically susceptible iron minerals such as magnetite. Davis Tube results do not correlate exactly with the observed awaruite content and the Davis Tube results are likely impacted by liberation issues in a manner similar to recovery processes.
3. The high-grade nature of awaruite (76% contained metallic Ni and corresponding 24% metallic iron) is significant, as concentrates of very high Ni grade can potentially be produced by concentration processes for this mineral. Nickel concentrates from the Baptiste Project could be unique in the world of nickel mining and processing.

Preliminary metallurgical testwork at SGS was conducted in 2011 and 2012 using composite sample material from the deposit as well as a number of individual samples. Liberation analysis of these materials showed poor liberation of nickel minerals above 75 µm in terms of primary grind particle size distributions. Irrespective of this mineralogical conclusion, testwork progressed using coarse primary grind particle size distributions of approximately 600 µm and regrind particle size distributions of approximately 100 µm, which did not provide suitable liberation of the awaruite. Test results obtained by SGS were based on magnetic separation for preliminary upgrading of nickel and iron values followed by gravity processes to generate final nickel concentrates. Overall results from this testwork predicted nickel recoveries in the range of 47% of total Ni to a concentrate of approximately 4% Ni.

Mineralogical analysis of all the sample materials used in the various testwork programs show the Decar material to be very consistent in terms of mineral composition and grain texture.

Metallurgical testwork was subsequently conducted at ALS Metallurgy in Kamloops, BC in late 2012 to follow the testwork results of SGS and to generate larger samples of nickel concentrate for use in marketing studies. Metallurgical parameters generated at SGS in 2011 were used in a pilot plant test of approximately 16 tonnes of material, and a total nickel recovery of approximately 19% was obtained to a concentrate of 12.5% Ni. This recovery is in-line with liberation analysis of this material at the grind size used, which showed 23% of awaruite being liberated. It is expected that only liberated awaruite would be recovered in a gravity circuit, as middling particles of awaruite and silicates would mimic the density of host rock and magnetite particles and be rejected in a gravity process devoted to nickel recovery.

Metallurgical testwork was subsequently re-visited in 2018 at ALS using a new 600 kg composite sample of Baptiste material. This testwork employed significantly finer primary grinds in the range of 100 to 300 µm for the magnetic concentration of nickel and iron values and flotation techniques (at a regrind size of 25 µm) for the separation and recovery of nickel from the magnetic concentrates. The recovery of nickel in a traditional magnetic separation circuit followed by flotation returned nickel recovery data in the range of 58% to 60% of overall Ni to a concentrate in the range of 55% Ni. Liberation of awaruite is still considered sub-optimal in this process testwork as only 77% of the total awaruite is estimated to be recovered.

Testwork in the ALS program followed the flowsheet outlined in Figure 13-8, presented later in this chapter. Magnetic concentrates, after being regrind to 25 µm, comprised almost entirely of magnetite and awaruite, have been consistently produced and typically contained 2.5% to 3% Ni. This magnetic concentrate was used for subsequent flotation testing for the production of a high-grade nickel concentrate. Approximately 90% to 93% of the nickel contained in the magnetic concentrate (predominantly in the form of awaruite) was shown to be recovered to a high-grade flotation concentrate with grades in excess of 55% Ni. Overall total nickel recovery to a high-grade flotation concentrate is expected to be in the range of 58% to 60% of the nickel contained in the run-of-mine material. More details on the nickel concentrate are presented later in this chapter.

The testwork results obtained by ALS can be related to the mineral resources estimated by FPX Nickel for use in calculating project economics. When using comparable particle size distributions, the Davis Tube Recoverable Nickel test (DTR) is very similar to the result obtained from low-intensity magnetic separation (LIMS), which is the basis of most of the magnetic separation testwork conducted in this testwork program. Significant testwork was conducted at SGS in 2012 conducting Davis Tube Recoverable nickel tests and various grind sizes. These results correlate well with the more recent LIMS testwork by ALS.

The flotation tailings are composed almost entirely of magnetite and is a potential saleable iron concentrate. The iron concentrate produced in the testwork at ALS contains approximately 60% to 62% Fe. More details on the iron concentrate are presented later in this chapter.

Additional testwork is recommended for the project to optimize nickel recovery. This is discussed later in this chapter.

13.3 Mineralogical Characterization of Decar Materials

This program was conducted by SGS in Lakefield, Ontario, using material collected during the 2010 drilling campaign (SGS 2011). One master composite and five variability composites, identified as indicated below, were created using this material.

- 2010 Master Composite
- 2010 Baptiste 3
- 2010 Sidney 10
- 2010 Grain Size A
- 2010 Grain Size B
- 2010 Grain Size C

Analysis of the 2010 SGS Master Composite, containing 0.28% Aw, using x-ray diffraction (XRD) determined that serpentine is the major component of the mineralized rock, followed by minor amounts of magnetite, brucite, olivine, and trace amounts of awaruite and pentlandite. The Quantitative Evaluation of Minerals by Scanning electron microscopy (QEMSCAN®) data confirmed and quantified these results. This rock characterization tends to be common to all materials tested in the various metallurgical testwork programs. Results of the SGS Qemscan analysis is contained in Table 13-3. Qemscan analysis conducted by ALS in 2018 were in line with those from the 2012 SGS work.

Table 13-3: Summary of Qemscan modal results – SGS master composite

Mineral species	Modal %
Serpentine	85.3
Magnetite	5.0
Clinopyroxene	3.1
Olivine	2.9
Brucite	1.3
Awaruite	0.30
Pentlandite	0.05
Hazelwoodite	0.04

Electron microprobe analysis was used to determine the deportment of nickel in the various species present in the SGS Master Composite sample. Awaruite was found to be the major nickel containing mineral in the sample. The distribution of nickel among the other nickel containing species is presented in Table 13-4.

Table 13-4: Nickel deportment in various minerals – SGS master composite

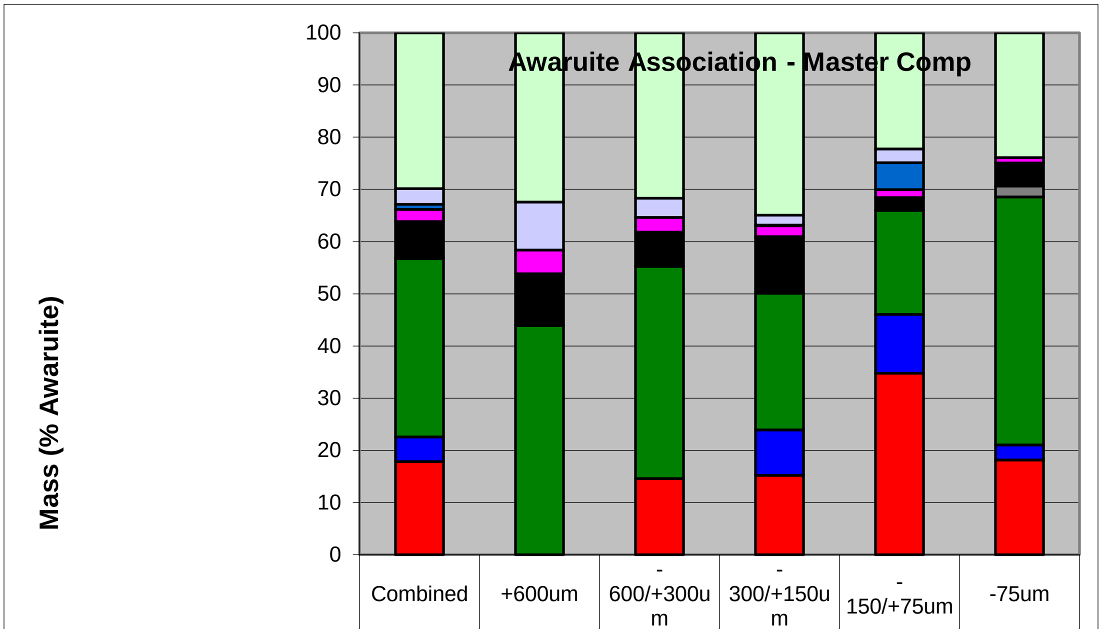
Nickel containing minerals	Percentage of overall contained nickel
Awaruite	74.5
Hazelwoodite	10.0
Pentlandite	4.6
Serpentine	6.3
Magnetite	3.4
Olivine	0.9
Clinopyroxene	0.3

Qemscan was also used to determine the mineral fraction particle sizes, liberation and association of various nickel bearing minerals. Roughly 23% of the awaruite within the master composite is present as liberated grains. Of the remainder, approximately 34% exists locked with serpentine, approximately 30% is contained in ternary or more complex associations (including magnetite), approximately 7% is in a ternary association of awaruite-serpentine-magnetite, and finally, less than 1% of the awaruite is present in binary associations with either magnetite or nickel sulphides. This degree of mineral liberation is insufficient to support a high recovery of nickel using gravity or flotation processes. This degree of liberation may allow for meaningful recovery of nickel using magnetic separation to produce low-grade nickel concentrates, which will likely contain a majority of the nickel as unliberated particles of awaruite associated with silicates or magnetite.

Figure 13-1 presents a graphical summary of the liberation data for the SGS Master Composite sample and it shows increasing liberation characteristics as finer fractions of the sample are evaluated. The results of the very fine fraction appear anomalous as this sample shows a reduction in awaruite liberation, contrary to the trending data. This anomaly is not well understood at this time.

Mineralogical analysis for the five variability samples showed similar liberation issues with respect to awaruite and liberation of awaruite was generally poor for all samples observed. A summary of the liberation of awaruite is shown in Table 13-5 for the variability samples in the 2010 SGS program.

The mineralogical analysis completed by SGS did confirm the consistent nature of the awaruite mineralization across a widely dispersed range of samples. It should also be noted that later mineralogical analysis of materials by ALS generally confirmed the results of the SGS mineralogical analysis with additional sample materials.



	Combined	+600um	-600/+300um	-300/+150um	-150/+75um	-75um
Other	0.00	0.00	0.00	0.00	0.00	0.00
Complex Aw Bearing	29.8	32.5	31.7	34.9	22.3	23.9
Aw-Serp-Bruc-Mag Quaternary	2.99	9.15	3.66	1.87	2.60	0.00
Aw-Pent-Heaz Ternary	1.02	0.00	0.00	0.13	5.17	0.00
Aw-Serp-Brucite Ternary	2.32	4.53	2.85	2.09	1.51	1.04
Aw-Serp-Magnetite Ternary	6.86	9.98	6.52	10.8	2.48	4.42
Aw-Magnetite Binary	0.26	0.00	0.00	0.04	0.00	2.08
Aw-Serpentine Binary	34.1	43.9	40.7	26.2	19.9	47.5
Sub-Liberated Aw	4.70	0.00	0.00	8.71	11.3	2.86
Liberated Aw	17.9	0.00	14.6	15.2	34.8	18.2

Figure 13-1: Awaruite Liberation and Association Analysis – SGS master composite sample

Table 13-5: Summary of Liberation and Associations for awaruite within SGS variability samples

Mineral	Grain size A (%)	Grain size B (%)	Grain size C (%)	Sydney 10 (%)	Baptiste 3 (%)
Liberated Awaruite	36.3	17.0	22.6	10.4	40.2
Awaruite + Magnetite	33.1	60.6	35.7	54.9	25.5
Awaruite – Silicates	30.6	22.4	41.8	34.8	34.4
Total	100.0	100.0	100.0	100.0	100.0

Mineralogical analysis of the 2018 composite sample used at ALS indicated that approximately 78% of the Ni contained in the sample occurred within the mineral awaruite, the balance of the nickel found in minor sulphide minerals and as soluble nickel in silicates. Generally, the Qemscan analysis of the 2018 composite sample mirrored the Qemscan analysis of the 2010 SGS master composite. A summary of the Qemscan data for the 2018 ALS Metallurgy sample is shown in Table 13-6. Of note is the comparison of overall liberation of the awaruite mineral at 22% which corresponds to the observed liberation of the 2012 SGS sample of 23%. A large majority of un-liberated awaruite is observed attached to silicate minerals (ALS, 2020).

Table 13-6: Summary of Liberation and Associations for awaruite (2018 ALS metallurgy composite sample)

Size range	**>150 µm**				**<150>75 µm**				**<75>38 µm**			
Mineral Status	Pe	Aw	Ma	Gn	Pe	Aw	Ma	Gn	Pe	Aw	Ma	Gn
Liberated	0.0	0.6	0.2	17.1	0.0	9.5	1.3	23.0	3.3	5.1	3.7	14.4
Binary – Pe		0.0	0.0	0.0		1.4	0.0	0.0		1.7	0.0	0.0
Binary – Aw	0.0		0.0	0.1	0.7		0.0	0.0	6.7		0.0	0.0
Binary – Ma	0.0	0.0		2.5	0.0	0.0		2.2	0.8	0.6		0.8
Binary – Gn	9.8	19.2	20.7		4.9	27.3	31.3		11.7	13.5	16.6	
Multiphase	2.0	1.2	0.2	0.0	0.0	6.8	0.1	0.0	5.0	2.8	0.0	0.0
Total	**11.9**	**21.0**	**21.1**	**19.6**	**5.6**	**45.0**	**32.8**	**25.3**	**27.6**	**23.6**	**20.4**	**15.2**

Mineral status	**<38µm**				**Mineral liberation-2 dimensions**			
	Pe	Aw	Ma	Gn	Pe	Aw	Ma	Gn
Liberated	34.7	6.8	14.1	39.1	38.0	22.0	19.3	93.5
Binary – Pe		0.6	0.1	0.0		3.7	0.1	0.1
Binary – Aw	6.7		0.0	0.0	14.1		0.0	0.1
Binary – Ma	0.0	0.1		0.7	0.8	0.7		6.2
Binary – Gn	13.5	2.6	11.6		40.0	62.6	80.3	
Multiphase	0.0	0.3	0.0	0.0	7.1	11.1	0.3	0.0
Total	**54.9**	**10.5**	**25.8**	**39.8**	**100**	**100**	**100**	**100**

Notes:

1. Pe-Pentlandite, Cobalt Pentlandite and Millerite/Polydimite, Aw-Awaruite, Ma-Magnetite, Chromium Magnetite, Chromite, Iron/Steel, Hematite and Goethite/Limonite, Gn-Non-Sulphide minerals including Chalcopyrite, Pyrite and Sphalerite.
2. 0.0 indicates these minerals were not observed during the counting procedure.
3. The 150, 75 and 38 µm sizing fractions correspond to the Tyler 100, 200 and 400 mesh sieves
4. The Total line is the distribution of mineral in the size fraction. Original data is from the size by assay and distribution tables.

The overall conclusion of the mineralogical analysis of the various metallurgical samples in the differing testwork programs is that liberation issues will significantly affect the recovery of nickel when using gravity or flotation processes which are significantly affected by grain densities or mineral surface exposure. Regrinding of concentrates to particle sizes in the range of 25 µm or finer (P_{80}) is likely required to achieve concentrates that are predominantly awaruite and to benefit recovery and concentrate grade.

13.4 Bench-Scale Investigations (2012 Test Program)

This bench scale test program was conducted by SGS in Lakefield, Ontario, using material collected during the 2010 drilling campaign. The 2010 Baptiste, 2010 Master Composite, 2010 Grain Size A, 2010 Grain Size B, and 2010 Grain Size C were combined in an appropriate ratio to generate 50 kg of feed for this test program. These were the same samples used in the SGS Mineralogical Characterization program.

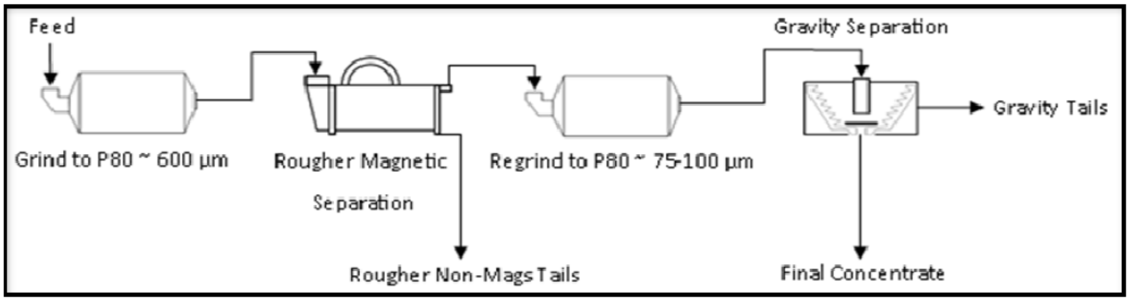
This program was conducted to determine the optimum grind sizes for rougher and/or cleaner magnetic concentration and cleaner or re-cleaner gravity concentration. The magnetic separation tests were carried out using Davis tubes. Gravity testwork was conducted using small samples of magnetic concentrate on a super-panner or wave table, which are bench-scale gravity test equipment.

The total nickel recovery in a rougher magnetic separation was evaluated at a variety of grind sizes ranging from a P_{80} of 150 µm to 600 µm. While the conclusion of SGS was that nickel recovery was not affected by primary grind sizes, liberation data and regrinding testwork did show significant impacts to nickel recovery during upgrading testwork. An objective of producing a 4% Ni concentrate was set for the SGS metallurgical testwork. Key results of the SGS testwork is summarized in Table 13-7 and reflect predicted recovery to a 4% Ni concentrate (SGS, 2012).

Table 13-7: Key metallurgical results from SGS 2012 testwork program

Mag/Gravity test #	Regrind size (µm)	Predicted Rec. to 4% Ni Concentrate
SP-8	114	37.7
SP-7	90	36.3
SP-1	70	47.3
SP-2	46	46.3
SP-3	23	45.1

A flowsheet for the proposed nickel recovery process derived from the SGS 2012 metallurgical testwork is shown in Figure 13-2.



Source: SGS (February 2012)

Figure 13-2: Simplified PFS for proposed Ni recovery process in 2012 testwork program

13.5 ALS 2018 – 2019 Testwork

13.5.1 Sample Selection and Preparation

A detailed review of the testwork conducted by SGS guided the metallurgical work undertaken by ALS Metallurgy in 2018 (ALS, 2020). Metallurgical testwork was initiated at ALS Metallurgy in 2018 to further evaluate the use of magnetic separation for the recovery of nickel and iron, as well to investigate the option of using flotation to separate nickel and iron minerals from a magnetic concentrate. A composite sample was generated from drill core rejects from the 2012 and 2017 drilling program, this sample comprised nearly 500 kg in total mass. Available material used to formulate this composite sample is shown in Table 13-8 and illustrated in Figure 13-3. The composite sample generated is representative of the Project's Baptiste starter pit representing approximately the first seven years of operation. Mining for the subsequent years is planned only from the Baptiste Deposit.

This work used the preliminary results of the work completed at SGS and was focused on using finer primary grinds in magnetic separation and employing regrinding to assist in upgrading magnetic concentrates. The key objective of this preliminary work was to produce a magnetic concentrate that was composed nearly entirely of magnetic minerals, magnetite and awaruite. This high-grade concentrate was subsequently used in flotation testing of the separation of nickel and iron minerals.

ALS performed a duplicate head cut of the starter pit sample received. The average head assay showed a total Ni content of 0.22% and a total Fe content of 5.8%. Sulphur and MgO were also analyzed at 0.06% and 24.0% respectively.

Table 13-8: Summary of drillhole intercepts used in 2018 composite sample generation

Hole ID	Start (m)	End (m)	Length (m)	Approx. Wt. (kg)	DTR Ni (%)	Composite sample		
						Wt (kg)	DTR Ni (%)	Tot Ni (%)
12BAP043	45.6	160.0	115.4	325	0.159	104	0.160	0.195
12BAP043	160.0	193.0	32.0	63	0.172			
12BAP050	34.5	114.0	79.5	224	0.148	106	0.145	0.195
12BAP063	86.0	150.0	64.0	180	0.156	90	0.152	0.207
12BAP067	55.1	150.5	95.4	374	0.167	168	0.169	0.215
12BAP067	159.3	230.1	70.8	139	0.140			
12BAP067	232.5	261.0	28.5	61	0.152			
Totals			**486**	**1,366**	**0.156**	**468**	**0.158**	**0.205**
ALS 2018								**0.220***

*ALS 2018 average of two assays

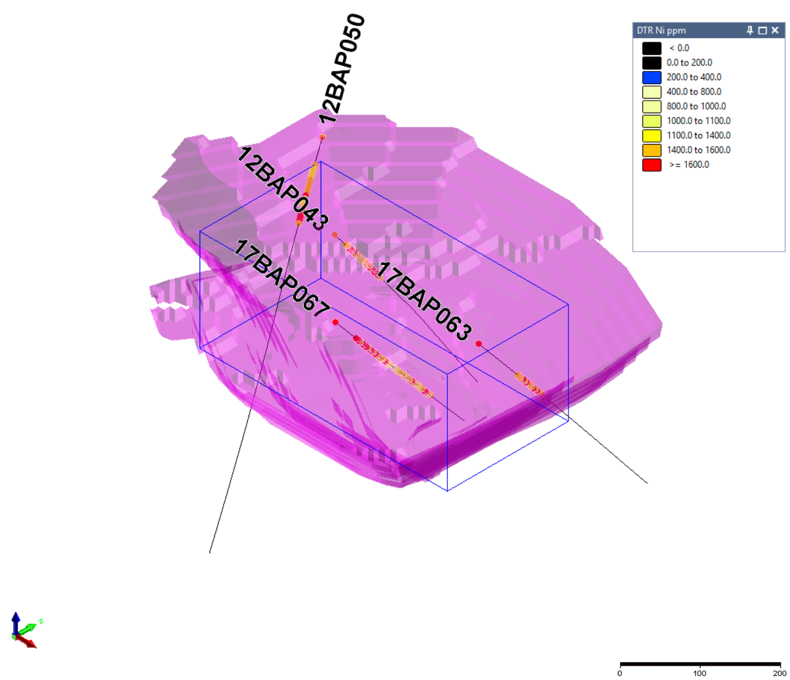


Figure 13-3: Composite sample drillhole location

13.5.2 Magnetic Separation Testwork

A series of magnetic separation tests using LIMS were conducted using the composite sample to evaluate the impact of primary grinding on nickel recovery as concentrate quality. Magnetic separation test results indicate quite consistently that nickel recovery in the magnetic rougher stage is improved as the primary grind is made finer as is shown in Figure 13-4 and Figure 13-5.

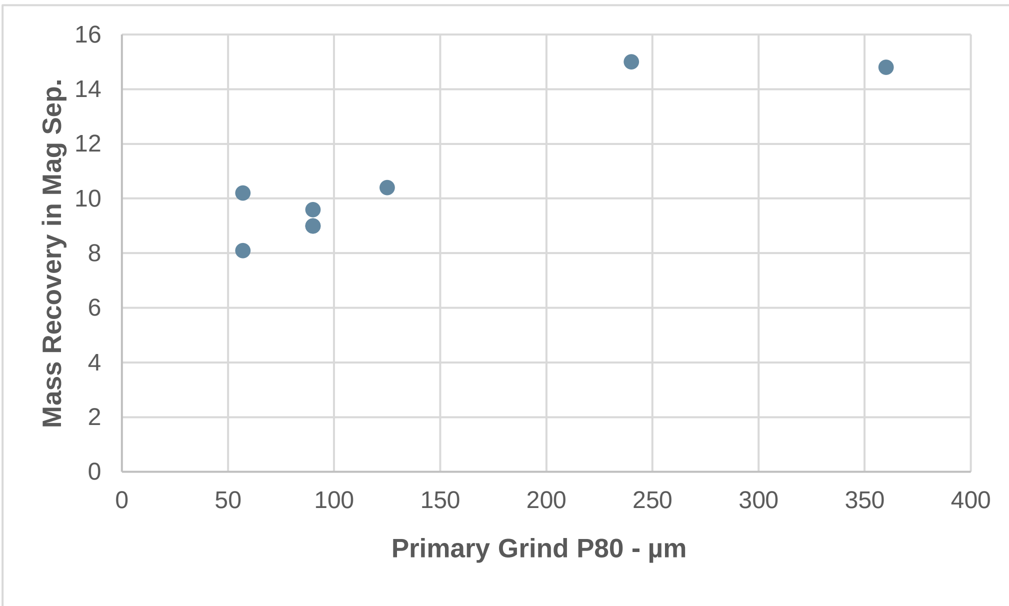


Figure 13-4: Rougher mass of magnetic concentrate as a function of primary grind

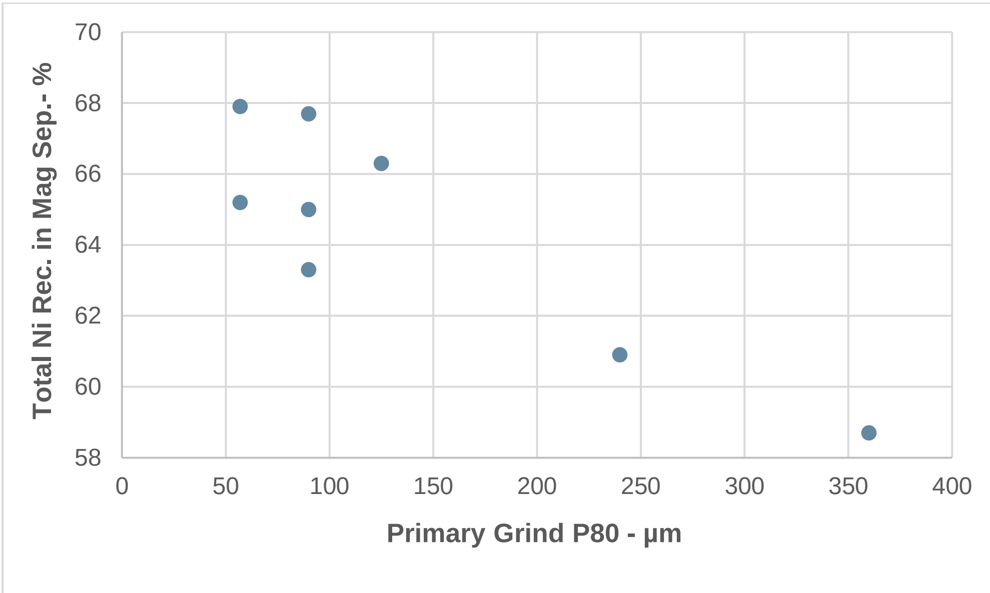


Figure 13-5: Rougher total nickel recovery as a function of primary grind

It is possible to estimate the recovery of awaruite within individual magnetic separation tests and these results are shown in Figure 13-6. This relationship is slightly steeper than the overall nickel recovery relationship as non-awaruite nickel is removed from the coarse grind test results at a more pronounced rate. Awaruite recovery is maximized at about 84% using finer primary grinds.

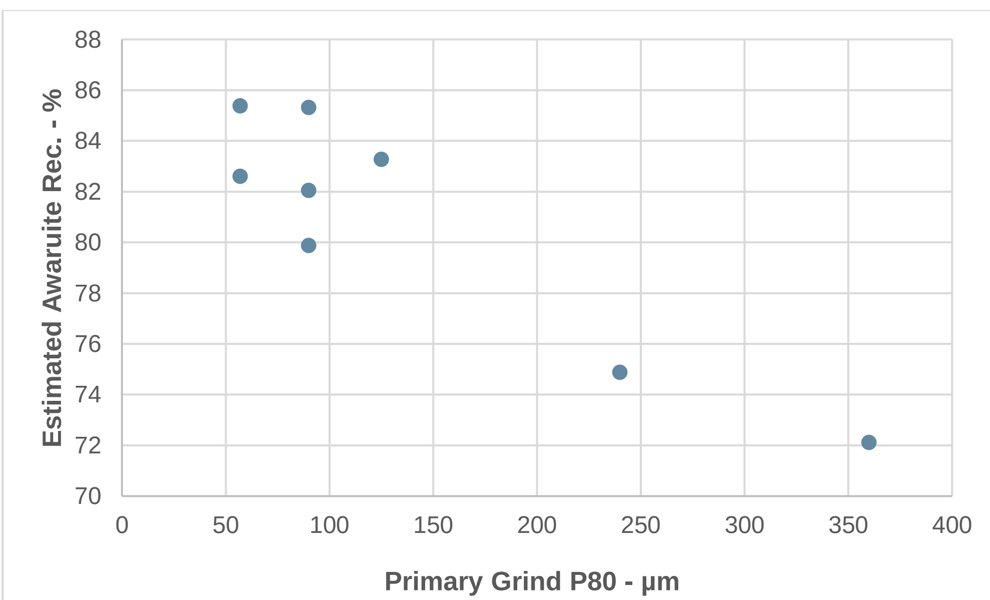


Figure 13-6: Calculated awaruite recovery as a function of primary grind

As approximately 24% of the head overall nickel is non-magnetic, the relationship for magnetic nickel is more pronounced than what is described by the overall nickel recovery relationship. A summary of the primary grind and nickel recovery data is shown in Table 13-9. Also included in this table is the result of DTR nickel.

Table 13-9:Rougher magnetic separation results for various primary grind size distributions

Test No.	Primary grind P_{80} (µm)	Concentrate mass (%)	Overall Ni Rec. (%)
DTR	90	10.3	66.7
5690-1	125	10.4	66.3
5690-2	90	9.0	65.0
2690-3	57	10.2	67.9
5690-6	240	15.0	60.9
5690-7	57	8.1	65.2
5690-15	90	9.0	63.3
5690-16	90	9.6	67.7
5690-19	360	14.8	58.7

A pronounced relationship exists in the data set, with the mass recovery in magnetic separation being strongly influenced by the primary grind size. Overall nickel recovery can be masked by the inclusion of a large volume of nickel in a non-awaruite form and caution needs to be exercised in evaluating overall nickel recovery data to ensure that proper conclusions are being made. The Davis Tube Recoverable nickel results corresponds well with the LIMS testwork. It is also expected that full-scale operation of a grinding plant, with hydrocyclones for use in classification, will result in significantly better liberation of dense minerals owing to the manner in which mineral density impacts classification sizes.

Regrinding and magnetic upgrading of the rougher concentrates resulted in the production of a magnetic concentrate that became flotation feed. The magnetite content of the fresh feed samples is typically 4.5% to 5.0% weight and this represents the typical iron concentrate production in the nickel-iron separation process. A typical magnetic separation test result is shown in Table 13-10.

Table 13-10: Typical magnetic separation test result after regrinding (primary grind 90 µm)

KM5690-28 Starter pit composite 1 Overall metallurgical balance – Magnetic cleaning test								
Product	Weight		Assay %			Distribution %		
	%	g	Ni	Fe	MgO	Ni	Fe	MgO
Magnetic Concentrate	4.8	2842	2.63	65.0	4.70	61.7	52.1	0.6
Magnetic 2nd Clnr Tail	0.3	171.2	0.64	24.3	15.8	0.9	1.2	0.1
Magnetic 1st Clnr Tail	5.0	2951	0.32	4.6	41.1	7.8	3.8	5.2
Magnetic Rougher Tail	89.8	52700	0.07	2.9	41.7	29.6	42.9	94.1
Feed	100.0	58664	0.21	6.0	39.8	100	100	100

13.5.3 Flotation Testwork

As part of the second phase of this 2018 testwork program, ALS conducted a significant volume of flotation testwork to evaluate the possibility of separating awaruite from magnetite using a magnetic concentrate produced using a low-grade magnetic separator, as described earlier. The magnetic concentrate is predominantly a two-mineral concentrate of magnetite and awaruite, as indicated in Table 13-10.

Flotation testwork has included laboratory scale flotation testwork to evaluate nickel flotation from a magnetic concentrate. In summary, awaruite has been shown to be readily recovered with a simple flotation process and concentrates in excess of 60% Ni can be produced. Nickel recovery from the flotation process are estimated to be approximately 90% to 93% of the Ni contained in the magnetic concentrate.

The flotation process uses a xanthate-based reagent scheme at low pH condition to make the awaruite mineral recoverable. A series of open circuit flotation tests were completed, as well as a single locked cycle test. Approximately 40 open circuit flotation tests were conducted in the testwork program at ALS investigating flotation conditions required to recover and upgrade nickel. Flotation conditions, once optimized, are shown in Table 13-11. Additional testwork is recommended to better understand the requirement to reduce and maintain low pH levels as well as to better understand the need to use copper sulphate as an activator for awaruite flotation.

Table 13-11: Flotation conditions typically used in awaruite flotation

Stage	Reagents added g/tonne			Time (minutes)				pH
	SIPX	$CuSO_4$	H_2SO_4	W31	Grind	Cond.	Float	
Natural								
Condition	-	-	11,000	-		5		3.5
Condition	-	600	14,000	-		5		5.0
Condition	100		-	-		1		5.1
Nickel circuit								
Rougher 1	-		980	200		1	2	4.5
Rougher 2	40	100	1,960	200		1	2	4.5
Rougher 3	40	100	580	200		1	2	4.5
Rougher 4	40	100	980	200		1	6	4.5
Cleaner 1	20	-	784	400		1	5	3.5
Cleaner 2	-	-	590	400		1	3	3.5

Shown in Table 13-12 is a metallurgical balance for test 5690-43, which is a test completed in the later stages of the testwork program with near optimal flotation conditions. The feed contained 2.76% total Ni and 91% total Ni recovery was achieved to a concentrate grading 53.2% Ni and 90.1% Ni recovery to a concentrate grade of 63.4% total Ni.

Table 13-12: Metallurgical balance for test 5690-43

KM5690-43 Test 43 Magnetic Cleaner Concentrate Overall metallurgical balance								
Product	Weight		Assay - %			Distribution - %		
	%	g	Ni	Fe	MgO	Ni	Fe	MgO
Nickel Concentrate	3.9	21	63.4	30.2	0.80	90.1	2.0	0.9
Nickel 2nd Clnr Tail	0.8	4.2	3.20	54.4	6.68	0.9	0.7	1.5
Nickel 1st Clnr Tail	3.8	20	1.18	54.2	5.54	1.6	3.4	5.7
Nickel Rougher Tail	91.5	480	0.22	61.0	3.66	7.3	93.8	91.9
Feed	100.0	524	2.76	59.5	3.64	100	100	100

Cumulative metallurgical balance								
Cumulative	Cum. weight		Assay - %			Distribution - %		
Product	%	g	Ni	Fe	MgO	Ni	Fe	MgO
Product 1	3.9	21	63.4	30.2	0.80	90.1	2.0	0.9
Product 1 to 2	4.7	25	53.2	34.3	1.8	91.0	2.7	2.3
Product 1 to 3	8.5	45	30.1	43.1	3.5	92.7	6.2	8.1
Product 4	91.5	480	0.22	61.0	3.7	7.3	93.8	91.9
Feed	100.0	524	2.76	59.5	3.6	100	100	100

Locked cycle testing of the process was also completed. The results of this testwork are shown in Table 13-13. This test shows the flotation process is capable of being stable and consistently produced a very high-grade concentrate. This test reported a nickel recovery of 81% of the available Ni from the magnetic concentrate feed to a very high-grade concentrate of 66% Ni. This high nickel concentrate is likely too high and contributes to the lower than expected Ni recovery. The internal balance of this locked cycle test also reports a rougher flotation concentrate of 54% Ni grade at a recovery of 91% as shown in Table 13-13.

Table 13-13: Locked-cycle test metallurgical balance for test KM5690-25

KM5690-25 Test 20 Magnetic Concentrate Overall cycle test mass and metallurgical balance								
Product	Weight		Assay - % or g/tonne			Distribution - %		
	%	g	Ni	Fe	MgO	Ni	Fe	MgO
Nickel Con I	0.6	15.5	67.4	25.2	0.99	16.4	0.3	0.1
Nickel Con II	0.6	14.8	68.4	23.7	0.85	15.9	0.2	0.1
Nickel Con III	0.7	19.0	64.0	24.7	1.53	19.1	0.3	0.2
Nickel Con IV	0.5	13.9	69.0	21.9	0.75	15.1	0.2	0.1
Nickel Con V	0.6	15.9	64.2	25.7	1.24	16.0	0.3	0.2
Nickel 2nd Clnr Tail	0.1	2.0	33.0	35.0	4.61	1.0	0.0	0.1
Nickel 1st Clnr Tail	0.1	3.3	15.2	42.8	8.29	0.8	0.1	0.2
Nickel Ro Tail I	20.2	530.6	0.36	54.4	4.92	3.0	19.8	20.9
Nickel Ro Tail II	19.8	518.3	0.34	56.0	4.91	2.8	20.0	20.4
Nickel Ro Tail III	19.2	503.2	0.33	56.2	4.79	2.6	19.4	19.3
Nickel Ro Tail IV	19.4	507.8	0.36	57.4	4.74	2.9	20.0	19.3
Nickel Ro Tail V	18.2	476.5	0.60	58.8	4.97	4.5	19.3	19.0
FEED	100	2621	2.43	55.5	4.76	100.0	100.0	100.0

KM5690-25 Test 20 Magnetic Concentrate Mass balance flowsheet and metallurgical balance data								
Flotation Stream		Weight	Assay (% or g/tonne)			Distribution (%)		
No.	Product	%	Ni	Fe	MgO	Ni	Fe	MgO
1	Mag Concentrate (feed)	100.0	2.41	57.1	4.74	100.0	100.0	100.0
2	Nickel Ro Feed	101.1	2.62	56.9	4.76	109.8	100.8	101.6
3	Nickel Ro Tail	97.1	0.48	58.1	4.85	19.1	98.8	99.4
4	Nickel Ro Con	4.0	54.5	28.2	2.59	90.7	2.0	2.2
6	Nickel 1st Clnr Tail	0.7	15.2	42.8	8.29	4.2	0.5	1.2
7	Nickel 1st Clnr Con	3.3	62.4	25.3	1.45	86.4	1.5	1.0
8	Nickel 2nd Clnr Tail	0.4	33.0	35.0	4.61	5.6	0.2	0.4
9	Nickel 2nd Clnr Con	2.9	66.4	23.9	1.0	80.9	1.2	0.6
10	Final Tail	97.1	0.52	57.9	4.87	19.1	98.8	99.4

Analysis of the results indicated that the test conditions were not optimal in terms of pH and regrind size, and that further improvements in grade and recovery are possible, as suggested from the body of the open circuit testwork.

One of the key observations in the completion of the locked cycle test is the loss of recovery in attempting to produce exceedingly high nickel concentrate grades. The liberation issues that are responsible for losses of awaruite in the rougher magnetic circuit have not been completely resolved at a 25 µm regrind size. It is likely that concentrate grades in the range of 50% Ni will have considerable advantage to the overall nickel recovery. Further testwork in this area of the process is recommended and selection of a flotation concentrate grade target is required at some point in time. The nickel concentrate grades produced in flotation for the Baptiste Project are considered well above industry average and the Baptiste concentrates should be readily saleable to nickel consumers in the range from 10% Ni and above.

Analysis of the nickel losses in the flotation process show that most of the losses are liberated and potentially available for recovery, the reason for this fraction being un-recovered is not known, nor has adequate testwork been completed to address this issue. It is expected that additional flotation testwork will be able to recover a significant portion of this liberated awaruite. Table 13-14 shows the Qemscan analysis of a flotation tailings sample from flotation test 5690-30, which had a flotation tailings grade of 0.33% Ni. It is expected that this tailings grade can be reduced by 50% with additional testwork, into the range of 0.16% Ni with adjustments to flotation conditions and regrind optimization. It is also worth noting that nearly 50% of the gangue in the tailings is liberated and the balance of the contained gangue is in simple binary particles with magnetite, indicating that additional gangue rejection could be possible with optimized cleaner magnetic upgrading.

Overall projections for flotation recovery of nickel are expected to be in the range of 94% when optimized to a flotation concentrate in excess of 50% Ni. Current testwork results have achieved approximately 92% recovery to a high-grade nickel concentrate and the estimation of 94% Ni recovery in flotation is warranted, based on the trajectory of the testwork results and the observed nature of the nickel losses in flotation.

Table 13-14: Qemscan results on flotation rougher tails

Summary of Percent Liberation by Size and Class Test 33 Nickel Rougher Tailing KM5690												
Size range	**>38 µm**				**<38>20 µm**				**<20 µm**			
Mineral status	**Pe**	**Aw**	**Ma**	**Gn**	**Pe**	**Aw**	**Ma**	**Gn**	**Pe**	**Aw**	**Ma**	**Gn**
Liberated	0.0	15.3	0.0	0.1	0.1	0.3	0.1	0.0	66.2	60.3	86.9	46.6
Binary – Pe		0.2	0.0	0.0		0.0	0.0	0.0		0.0	0.0	0.0
Binary – Aw	6.0		0.0	0.0	1.6		0.0	0.0	0.0		0.1	0.0
Binary – Ma	0.1	0.5		0.1	1.4	0.2		0.2	8.6	14.4		52.6
Binary – Gn	0.3	0.1	0.0		0.1	0.0	0.0		11.5	1.0	12.7	
Multiphase	0.2	0.5	0.0	0.0	1.0	0.2	0.0	0.0	2.9	6.8	0.1	0.2
Total	**6.6**	**16.6**	**0.0**	**0.3**	**4.2**	**0.8**	**0.1**	**0.2**	**89.3**	**82.5**	**99.9**	**99.5**

Mineral status	**Mineral liberation-2 dimensions**			
	Pe	**Aw**	**Ma**	**Gn**
Liberated	66.3	76.0	87.0	46.8
Binary - Pe		0.2	0.0	0.0
Binary - Aw	7.6		0.1	0.1
Binary - Ma	10.2	15.1		52.9
Binary - Gn	11.9	1.1	12.8	
Multiphase	4.0	7.5	0.1	0.3
Total	**100**	**100**	**100**	**100**

Notes:

1. Pe-Pentlandite, Cobalt Pentlandite and Millerite/Polydimite, Aw-Awaruite, Ma-Magnetite, Chromium Magnetite, Chromite, Iron/Steel, Hematite and Goethite/Limonite, Gn-Non-Sulphide minerals including Chalcopyrite, Pyrite and Sphalerite.
2. 0.0 Indicates these minerals were not observed during the counting procedure.
3. The 38 and 20 µm sizing fractions correspond to the Tyler 400 and 635 mesh sieves.
4. The Total line is the distribution of mineral in the size fraction. Original data is from the size assay and distribution tables.

13.5.4 Testwork Result Consolidation

Predictions of the recovery of nickel for use in Project economic analysis in this PEA, can be estimated from the various phases of testwork completed at ALS in 2018/2019. The magnetic recovery process is shown to be dependent on the primary grind size employed in the magnetic separation process, and subsequent regrinding and flotation recovery of nickel has been adequately demonstrated. As was discussed in Section 13.5.3, a concentrate with a Ni grade between 53% and 64% was achieved with a virtually constant flotation Ni recovery in the order of 90% to 91%. The lower grade Ni concentrate is likely as a result of a small incremental increase in Ni recovery carrying with it a significant amount of magnetite as well as a relatively small quantity of additional MgO/SiO_2 gangue oxides (hence unliberated Ni).

For the purpose of this PEA Study, the metallurgical balances have been performed using the higher-grade Ni concentrate. For product marketability, a concentrate specification has been determined based on the results of analysis of concentrate from test 43 and is presented in Chapter 19 of this Report.

Recovery data can be reported either on a total nickel basis or to reflect a recovery based on the DTR content of the contained nickel in the process feed. All metallurgical testwork has been completed, based on total nickel analysis which is common and based on standard nickel assaying procedures. FPX Nickel reports nickel content in the deposit based on Davis Tube tests and the reporting of recovery data in terms of Davis Tube Recoverable data is possible. For the purpose of this PEA, the conversion of recovery data is simply the ratio of Davis Tube recoverable nickel at the primary grind size used in the standard Davis Tube test. Within the ALS Metallurgy testwork, both Davis Tube testing and LIMS testing was done at 90 µm and it was observed that 66.7% of the total Ni reported in a Davis tube test.

Reporting of nickel recovery data from the proposed overall process is summarized in Table 13-15 and reflected both the reporting of nickel on an overall nickel or DTR basis. This table shows both the expected recovery of nickel in the magnetic separation process (feed to flotation) and the expected recovery of nickel to a final flotation concentrate.

Table 13-15: Recovery data for nickel reported based on total Ni or calculated DTR Ni

Test No.	P_{80} µm	Overall Ni basis		DTR basis	
		Rec. to Mag. %	Rec. to Flot. %	Rec. to Mag. %	Rec. to Flot. %
DTR	90	66.7		100	
5690-1	125	66.3	62.3	99.4	93.4
5690-2	90	65.0	61.1	97.5	91.7
2690-3	57	67.9	63.8	101.8	95.6
5690-6	240	60.9	57.2	91.3	85.8
5690-7	57	65.2	61.3	97.7	91.8
5690-15	90	63.3	59.5	94.9	89.2
5690-16	90	67.7	63.6	101.5	95.4
5690-19	360	58.7	55.2	88.0	82.7

Graphical results of the data shown in Table 13-15 is presented in Figure 13-7 along with linear approximations of the relationship between recovery and primary grind size. It is recommended that Ni recovery be determined by the linear approximations shown on the graph for obtaining estimates of the Ni recovery at various primary grind sizes. It should be noted that the DTR Ni data was generated by calculation and not by direct measurement or assaying. The overall Ni recovery is assumed to apply for final Ni flotation concentrate grades between 55% and 65%.

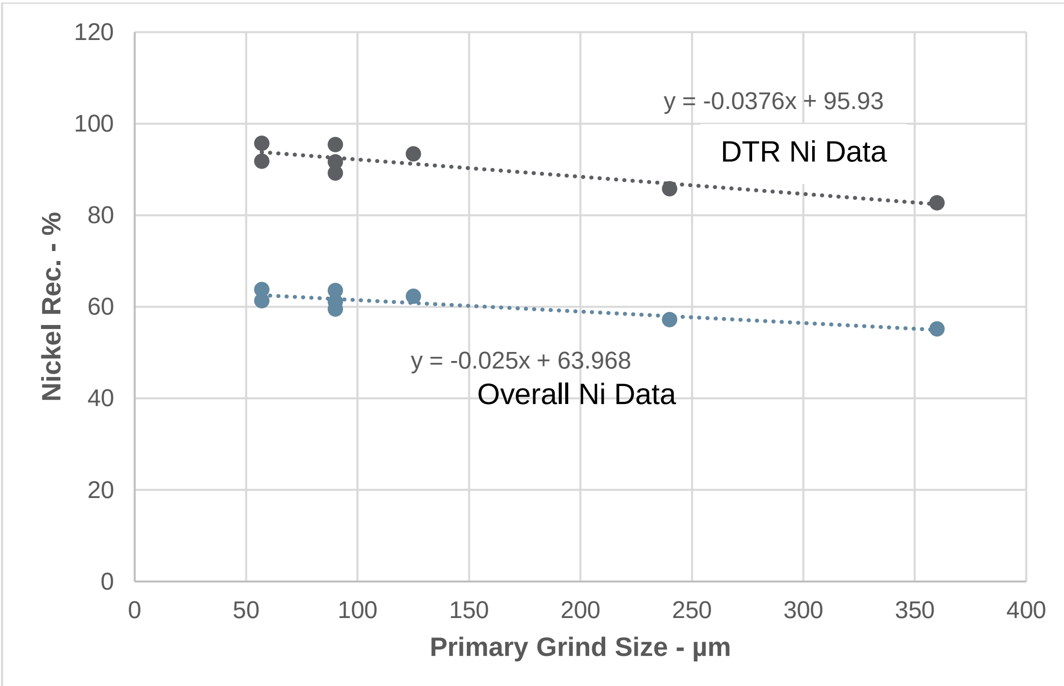


Figure 13-7: Expected Ni recovery versus primary grind size

13.5.5 ALS Testwork Conclusions

A metallurgical test program was conducted on a composite of crushed assay rejects from the Baptiste Deposit. The objective of the test program was to develop a robust process flow sheet to recover the Ni-Fe alloy mineral awaruite (Aw). Froth flotation in acidic pulp conditions was investigated with the objective of producing a highly metallized, high-grade Ni concentrate as the saleable product for the Project.

The assembled Starter Pit Composite sample contained approximately 0.22% total nickel and 5.8% iron. Mineralogical analyses indicated that approximately 78% of the Ni was present in Aw, while the remaining Ni was present in pentlandite and millerite/polydimite. The sample contained approximately 4% magnetite. The host rock was primarily serpentine minerals. Chrysotile, an asbestos mineral, measured about 1.0%.

The mineralogical assessment of the feed was conducted on a sample ground in a rod mill to approximately 150 µm K_{80}. Approximately 22% of the Aw liberated at this primary grind size, and most grains were locked in binary with gangue. The distribution of Aw was somewhat biased to the coarser size fractions, suggesting that the mineral is disseminated within the host rock and not easily liberated during grinding. About 28% of the Aw was present in grains with less than 5 percent surface exposure, as measured in 2 dimensions; these grains would likely be difficult to recover by magnetic means.

The Aw grain size distribution was estimated to have 80% and 50% passing sizes of 96 and 66 µm respectively, based on an effective diameter of the observed cross-sectional areas. This somewhat coarse grain sizing, along with the malleable metallic nature of the mineral, appeared to have the following effects on the mineral processing performance:

- After regrinding the magnetic cleaner concentrate to approximately 25 µm K_{80}, about 62% of the awaruite was present in the >38 µm fraction, a significant portion of which was still coarser than 100 µm.
- In order to mitigate assay variances on the magnetic cleaner concentrates, a screened metallic type method was required to more accurately measure nickel content in the coarser fractions.
- A review of froth flotation rougher tails grades indicated that variances in performance under similar conditions were related to the distribution of coarse nickel remaining in the tails.

Rougher magnetic recovery indicated that about 65% of the total Ni in the feed could be recovered to magnetic concentrate containing about 10% of the feed mass, when processed at a primary grind sizing of about 90 µm K_{80}. Nickel recovery decreased by about 2.5% for each 100 µm increase in the primary grind sizing K_{80} value to the coarsest primary grind sizing tested of 360 µm K_{80}.

The magnetic rougher concentrate was consistently upgraded to a concentrate that contained about 2.5% Ni and 5% of the feed mass. Magnetic cleaning was conducted after regrinding the magnetic rougher concentrate to about 25 µm K_{80}, and Ni losses after the magnetic cleaning steps averaged about 7%. The percentage of the nickel losses to the magnetic cleaner tails as fine Aw and as nickel sulphide minerals was not determined.

Froth flotation tests performed with the magnetic concentrate following conditioning with sulphuric acid and using copper sulphate as an activator for awaruite indicated that 80% to 90% of the Ni could be recovered to a high-grade containing between 51% and 72% Ni. It appears that conditioning at pH levels of 4 or lower provided better metallurgical performance than less acidic conditions. Use of a larger rougher cell and lower slurry solids concentration in initial tests resulted in lower rougher mass recoveries with similar rougher nickel recoveries, and higher nickel concentrate grades, but similar results were recorded in later tests by reducing each stage of conditioning by about one half albeit at a higher sulphuric acid dosage. The nickel flotation performance appeared to be affected by coarse nickel present in the flotation feed, making it difficult to assess optimal pulp chemistry conditions. In flotation tests with higher nickel losses, between 20% and 50% of the nickel lost to the rougher tails was present in the >38µm fraction. In a locked-cycle test with baseline conditions, about 81% of the Ni in the magnetic concentrate was recovered to the concentrate containing about 66% Ni and 1% MgO.

13.6 Grindability Testwork

13.6.1 ALS Testwork (2012 and 2018)

Limited grindability testwork has been done on the materials from the Baptiste Project. Shown in Table 13-16 is the results of testwork completed at ALS Metallurgy in 2012, as well as a single Bond Ball Mill work index determination in 2018 on material from Baptiste only. It should be noted that FPX advised that the 2012 grindability results should not be relied upon as the sample was obtained from a surface bulk sample which was not deemed representative of the deposit. The bulk sample was processed for the sole purpose of generating product which was used for the purpose of introducing it to potential end users for them to provide a preliminary assessment. Significant testwork is recommended for additional samples of material from the Baptiste Project, as grinding costs, both power and grinding media, will be major component of operational costs. For this PEA, crushing and HPGR design parameters were estimated in collaboration with an equipment vendor by benchmarking with other similar material. The 2018 BWi was used for design of the primary ball mill circuit as well as for the regrind circuit.

Table 13-16: ALS grindability testwork results

Sample	BWI kWhr/tonne	RMWI kWhr/tonne	SMC DWi kWhr/m^3	Axb
ALS Met. 2012 Comp.	23.1 (css 300 µm)	18.7 (css 1,180 µm)	4.7	56.5
ALS Met. 2018 Comp.	18.7 (css 150 µm)			

13.7 Sherritt Technologies Pressure Leaching Testwork

In November 2019, Sherritt Technologies performed leaching tests on awaruite flotation concentrate from Baptiste generated by the ALS Metallurgy testwork. This leaching testwork consisted of conducting two pressure leach tests to evaluate the potential to leach the awaruite mineral under chemical conditions common to current autoclave operations. Test results are summarized in Table 13-17 and indicate that very high nickel extractions are expected in the range of 98-99.5% of the contained nickel. These results are consistent with leaching mineralogy and chemistry expectations.

The pressure leach tests were conducted at 150°C with the pressure controlled to 750 kPa (g) with oxygen addition. The conditions and results of the batch tests are also presented in Table 13-17. Sulphuric acid additional are expected to be relatively high, as the nickel concentrate does not carry any sulphur or sulphate component and all sulphate contained in a potential nickel sulphate product is required to be sourced from sulphuric acid additions.

Table 13-17: Pressure leaching results and test conditions

Pressure leach test	Unit	1	2
Charge			
Ni Concentrate	g	40	40
Synthetic feed solution	L	1.8	1.8
Concentrated H_2SO_4	G	50	50
Retention time	Min	180	180
Synthetic feed solution analyses			
H_2SO_4	g/L	14.9	6.91
Ni	g/L	55.9	54.6
Fe	g/L	5.79	2.82
Discharge (pregnant leach) solution analyses			
H_2SO_4	g/L	21.5	12.1
Ni	g/L	69.4	70.1
Fe	g/L	2.45	0.77
Ni Extraction	%	99.5	98.8

The batch pressure leach tests confirmed the nickel concentrate from the Baptiste Project readily soluble in pressure leaching, using industry standard leaching conditions. Greater than 99% nickel extraction is expected with a 60-minute retention time in leaching. The quality of the pregnant leach solution generated from the tests was also excellent, with relatively low requirements for neutralization and iron removal.

It should be noted that for this PEA, the production of nickel sulphate is not considered in neither the process flowsheet nor in the economic evaluation of the Baptiste Project. It is however presented as an opportunity to pursue in Chapter 24 of this Report.

13.8 Proposed PEA Flowsheet and Process Design Parameters

Figure 13-8 presents the proposed conceptual process flowsheet for the current PEA Study. It covers only the beneficiation part of the process. The primary grind size targeted is a P_{80} of 300 µm. Even though the testwork shows that Ni recovery can be significantly improved by providing a finer primary grind (100 µm to 150 µm), the proposed primary grind size is driven by the tailings disposal strategy that has been adopted for this PEA Study by FPX and its tailings consultant, Stantec. This will be discussed further in Chapter 18 of this Report. The process design will be discussed in detail in Chapter 17 of this Report.

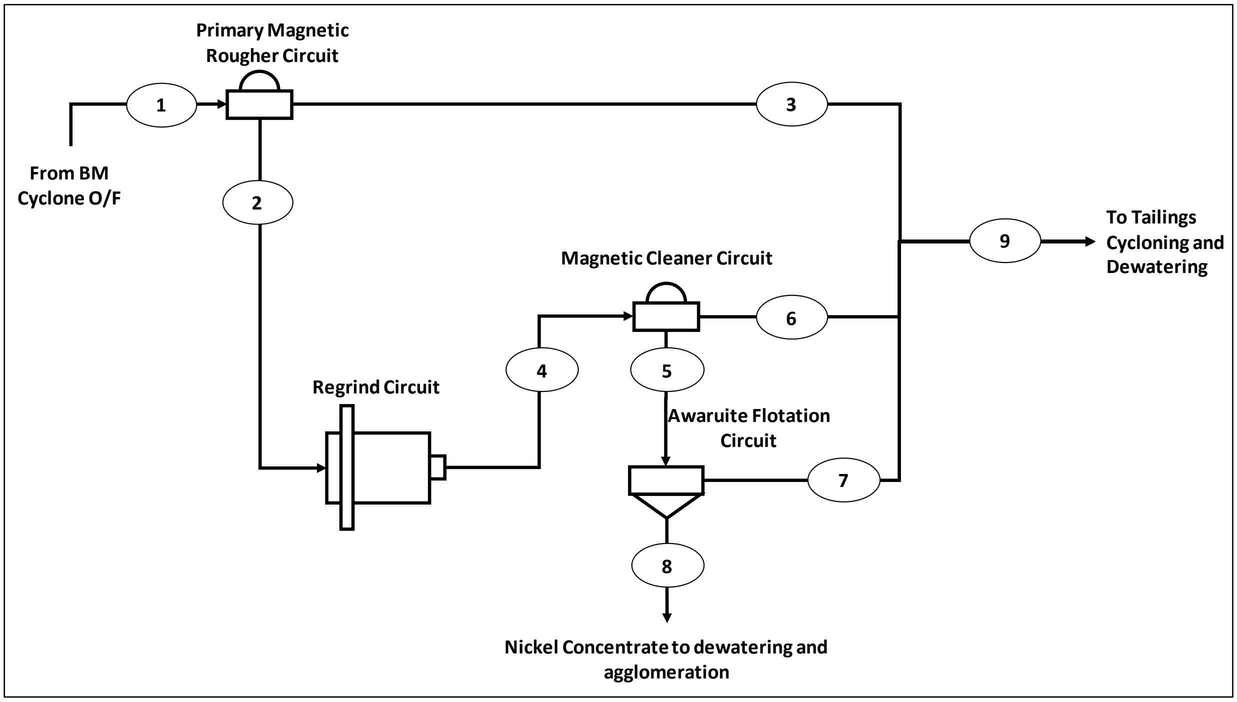


Figure 13-8: Proposed process flowsheet for the PEA study

Table 13-18 present the conceptual mass balance for process streams presented in Figure 13-8, derived from the testwork results presented earlier in this chapter. This data is subsequently used for the development of the process design discussed in detail in Chapter 17 of this Report. The overall DTR Ni recovery applied to the annual mined material head grade is 84.7%, based on a primary grind size of P_{80} = 300 µm. The final Ni concentrate weight recovery assumes a Ni grade of 63.4%.

Table 13-18: Conceptual mass balance from 2018 ALS testwork used for process design

Concentration Circuit Design Parameters		Wt Rec (%)
1	Feed to Primary Magnetic Circuit (Fresh Feed)	100%
2	Feed to Regrind Circuit	15.0%
3	Primary Magnetic Tailings	85.0%
4	Magnetic Cleaner Circuit Feed	15.0%
5	Magnetic Concentrate (Feed to Flotation)	4.8%
6	Magnetic Cleaner Circuit Tailings	10.2%
7	Flotation Tailings (Fe Concentrate)	4.6%
8	Final Flotation Ni Concentrate	0.16%
9	Final Combined Total Tailings	99.8%

13.9 Final Concentrate Specifications

13.9.1 Nickel Concentrate

The testwork indicated that a high-grade Ni concentrate can be produced from the Decar mineralization with the proposed magnetic/flotation process. The concentrate, rich in awaruite, is highly metallized but also contains residual magnetite and other oxides. Some trace elements such as Cu, S and P are also present and need to be taken into account when approaching stainless steel smelters for product marketability and payability analysis for this PEA. Table 13-19 presents an estimated chemical analysis for the Ni concentrate based on the testwork results.

The 2018 ALS testwork indicated that, with the composite sample tested, a Ni concentrate between 55% and 65% Ni can be produced as a flotation concentrate. As the Ni content decreases, the magnetite content increases. The MgO and SiO_2 also increases by a relatively small amount. The Baptiste Ni concentrate is unique when compared to standard ferronickel products available to steelmakers. The Baptiste concentrate will have the advantage of having a significantly higher Ni grade compared to standard ferronickel but is also less metallized and in general contains more impurities. This is because the Baptiste concentrate is the product of a mineral processing operation compared to standard ferronickel which is a product of a pyrometallurgical process.

Table 13-19: Estimated final Ni concentrate analysis (data from Test 43)

	Ni Concentrate	
	65% Ni Con	**55% Ni Con**
Ni	63.4	53.2
Fe (Total)	30.2	34.3
Aw (Ni_3Fe)	83.5	70.1
Fe_3O_4 (Calculated)	13.9	24.1
Co	1.0	1.0
Cu	0.6	0.9
S	0.6	0.7
MgO	0.91	1.8
SiO_2	1.3	2.6

13.9.2 Flotation Tailings

The proposed process flowsheet produces a tailings steam from the flotation step. This tailings stream is rich in iron as magnetite. For the purpose of this PEA, this tailings stream is assumed to be discarded with the other process tailings however there may a real opportunity to valorize this stream as an iron concentrate and generate complimentary revenue for the Project. This is discussed in more detail in Chapter 19 of this Report. Table 13-20 presents the chemical analysis of the flotation tailings, rich in iron in the form of magnetite which can potentially be sold and generate a complementary revenue stream. Further upgrading of this co-product could be required to increase the iron content and decrease the MgO and SiO_2 content to make the product more appealing to end users. Only the key components affecting applicability and payability are presented.

Table 13-20: Estimated final Fe concentrate analysis

Element	Method	Units	PSD P_{80} = 25 µm
			Fe Concentrate
Fe(T)	MA – AA	%	61.0
Fe_3O_4	Calculated	%	84.3
Cr_2O_3	WRA	%	2.81
Cr	Calculated	%	1.92
Fe + Cr	Calculated	%	62.9
MgO	WRA	%	3.75
SiO_2	WRA	%	3.87

13.10 Recommendations for Future Testwork

1. The 2018 ALS Metallurgy testwork program indicated that additional flotation testwork is required to better understand the optimal flotation conditions as well as the regrind requirements of the flotation process. Improvements in flotation results are expected.
2. Additional evaluative work is required to optimize the quality of an iron concentrate, including additional mineralogical work to observe the nature of the contaminants and guide additional testwork.
3. A significant program of grinding testwork is recommended using varied samples from the Baptiste Deposit.
4. Future testwork would involve a significant pilot scale magnetic separation testwork, as well as further bench scale flotation testwork to improve the understanding of the flotation conditions of awaruite. Additional flotation testwork is needed to better understand the regrind requirements of the flotation process and to optimize the quality of the iron concentrate.
5. Testwork involving coarse primary grind size (as dictated by tailings deposition strategy) vs magnetic intensity impact on Ni recovery at rougher magnetic separation.
6. The S content in the Ni concentrate is higher than industry standard. In case it becomes an issue, it may be warranted to undertake some baseline flotation testwork on the final concentrate to see if it is possible to lower the S through sulphide flotation.

14. MINERAL RESOURCE ESTIMATES

This section describes the methods used to produce an updated resource estimate for the Baptiste Deposit on the Decar Property. No estimates of mineral resources or mineral reserves have been made for the Van, Sid and B targets.

14.1 Key Assumptions and Basis of Estimate

The sample database supplied for the Baptiste Deposit contains results from 83 surface drillholes completed since 2010 (Table 14-1) or 96% of all metres drilled as shown in Table 10-1. In comparison to the 2013 resource estimate (Ronacher et al., 2013), the 2020 resource estimate incorporates an additional eight diamond drillholes (totaling 1,917 metres) completed during the summer of 2017, one hole drilled during the 2012 drilling campaign (which was not included in the 2013 resource estimate), and an additional 2,053 samples from core re-sampling of 2010 and 2011 drillholes completed in 2012. The average drillhole spacing in the Baptiste Deposit is 150 metres.

Table 14-1: Summary of drillhole and sample totals used in 2020 resource estimate

Year	Holes	Metres	Samples	Assayed (m)	Comments
2010	7	1,710.8	638	1,533.5	Samples include 2012 re-sampling program
2011	35	10,863.6	5199	10,176.5	Samples include 2012 re-sampling program
2012	32	16,346.8	4153	15,410.9	
2017	8	1,917.5	460	1,565.1	
Total	**83**	**30,838.7**	**10450**	**28,685.9**	

The 2020 resource model comprises a large, delta shaped volume that measures approximately 3.0 km in length and 150 to 1,080 m in width and extends to a depth of 540 m below the surface. The Baptiste Deposit remains open at depth over the entire system and is covered by an average of 12 metres of overburden.

Davis Tube magnetically-recovered (DTR) nickel is the nickel content recovered by magnetic separation using a Davis Tube, followed by fusion with lithium borate and an XRF finish to determine the nickel content of the magnetic fraction; in effect a mini-scale metallurgical test. The Davis Tube method is the industry standard for the quantitative analysis of magnetic minerals (e.g. SGS, 2009).

14.2 Geological Model

The updated geological interpretation was initially compiled in Micromine software using 26 section lines drawn parallel to the drillhole fences (Figure 14-1) and therefore ranging from an orientation of 030° at the western end of the Deposit to 330° in the northeast and 015° in the southeast. Reconciliation of the sectional interpretations was carried out on level plans at 200 m vertical intervals. String files of lithological contacts generated in Micromine were imported into Leapfrog software to generate solid models.

The updated geological model of the Baptiste Deposit was drafted by Equity and consists of four mineralized and six unmineralized (or barren) domains (Figure 14-2). The mineralized domains are:

- **Peridotite - Mineralized:** main host of nickel mineralization, logged as "peridotite" (see Section 7.3.1), includes variably brecciated cataclastic and/or mylonitized peridotite;
- **Dunite:** grade is variable but typically lower than Peridotite – Mineralized, logged as "dunite", forms layers in Peridotite - Mineralized domain (Figure 14-2), marked by relatively fine grain size, locally brecciated.
- **Peridotite - Massive:** weakly mineralized, logged as "massive peridotite" or possibly "peridotite", forms transition from Peridotite - Mineralized to - Low Grade domains (Figure 14-2), more vein-controlled serpentinization as opposed to pervasive serpentinite alteration in Peridotite - Mineralized domain;
- **Fe-Carbonate Altered Ultramafic:** weakly mineralized, logged as "FeCb_AltUM", weak to moderate ("incipient") Fe-carbonate altered peridotite, forms gradation from Peridotite - Mineralized to barren Listwanite & Fe-Cb altered domains, occurs mostly along southern margin of Baptiste Deposit and around the Listwanite ellipse bounding most of the eastern margin;

The six domains modelled as unmineralized (i.e. very low grade to barren) are:

- **Peridotite - Low Grade:** very low grade to barren, logged as "massive peridotite" (see Section 7.3.1), separated from Peridotite - Mineralized by Peridotite - Massive, more vein-constrained serpentinization relative to Peridotite - Mineralized domain;
- **Listwanite and Fe-Carbonate Altered Ultramafic:** very low grade to barren, logged as "FeCb_Listwanite" and possibly the most altered examples of "FeCb_AltUM", pervasively altered peridotite consisting mostly of Fe-carbonate ± silica, separated from Peridotite - Mineralized domain by incipient Fe-Carbonate Altered Ultramafic domain;
- **Altered dikes:** barren, logged as "altered dikes", interpretation based on 2012 model (Ronacher et al., 2012a) extended into 2017 drilling area, sharp contacts with host rocks;
- **Metavolcanic:** barren, logged as volcanic, part of Sitlika succession bounding the southern margin of Baptiste Deposit, fine-grained, mafic to intermediate composition;

- **Metasediments:** barren, logged as argillite, part of Sitlika succession bounding the southern margin of Baptiste Deposit, fine-grained, black;
- **Sitlika Metasediments:** barren, not intersected in any drillholes, distribution based on outcrop mapping, part of Sitlika succession, heterolithic siltstone to sandstone, variably disrupted with bedding-concordant sulphide.

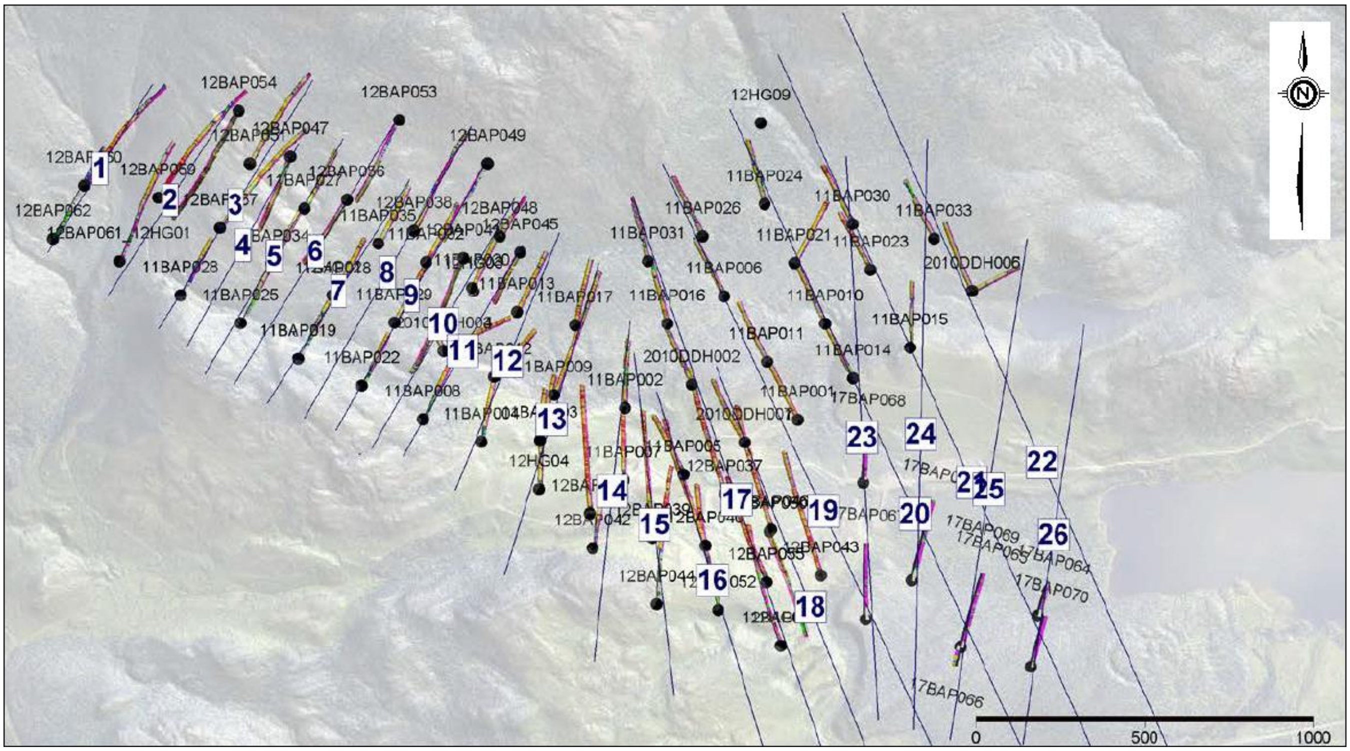


Figure 14-1: Plan map showing drill collar locations and section lines 1-26 on the Baptiste Deposit

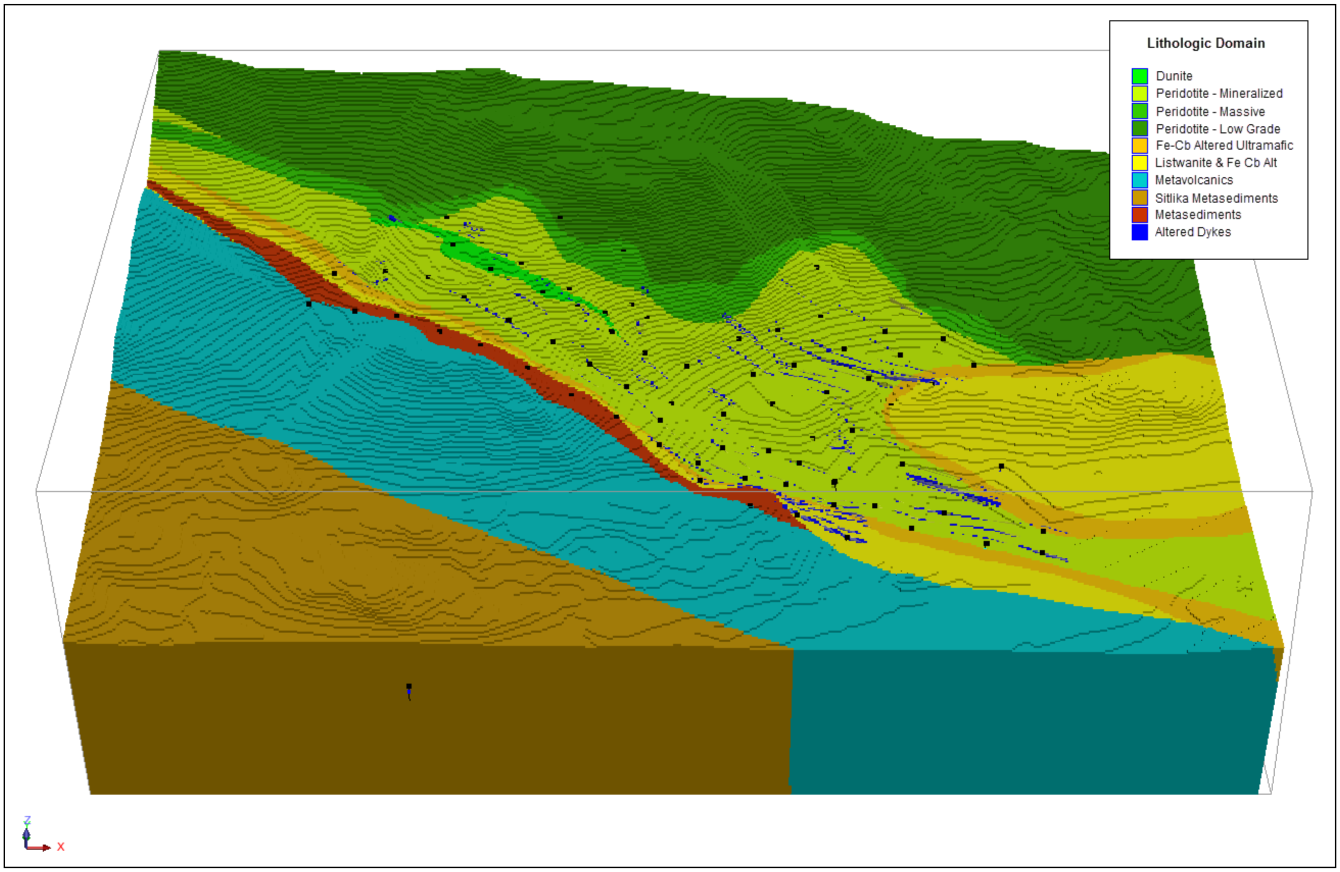


Figure 14-2: Geological model of the Baptiste Deposit showing the 10 lithological domains Highest awaruite grades are hosted in the light green Peridotite - Mineralized domain

14.3 Exploratory Data Analysis

Nominal sample lengths varied from 0.12 to 9.73 m for the various drill programs, with 36% of the samples exactly 4 m in length and only 1.7% of samples exceeding 4 m in length. Four metres was the targeted sample length for the 2012 and 2017 drilling campaigns, and also matches the 1 m original plus 3 m infill sampling done on the 2011 core. Data was therefore composited at 4 m intervals prior to statistical analysis

Statistical analysis of the grade distribution within the mineralized domains shows that the peridotite and dunite are the dominant hosts for the DTR Ni mineralization (Table 14-2, Figure 14-3, Figure 14-4).

Table 14-2: Descriptive statistics of 4 m composites from mineralized domains

Statistic	Peridotite - Mineralized	Dunite	Peridotite - Massive	Fe-Carbonate Altered Ultramafic
Sample count	5226	305	453	404
Minimum % DTR Ni	0.000	0.004	0.000	0.000
Maximum % DTR Ni	0.254	0.190	0.139	0.132
Mean % DTR Ni	0.128	0.108	0.049	0.045
Median % DTR Ni	0.133	0.110	0.041	0.042
Standard deviation	0.034	0.030	0.029	0.024
Variance	0.001	0.001	0.001	0.001
Coefficient of variation	0.269	0.277	0.597	0.524
Kurtosis	1.486	-0.020	0.676	0.602
Skewness	-0.981	-0.154	1.050	0.569

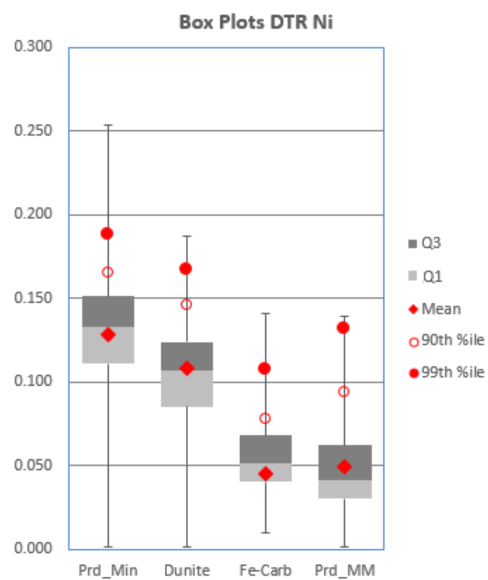


Figure 14-3: Box plots showing the % DTR nickel grade distribution within mineralized domains Prd_Min = Peridotite - Mineralized, Fe-Carb = Fe-Carbonate Altered Ultramafic, Prd_MM = Peridotite – Massive

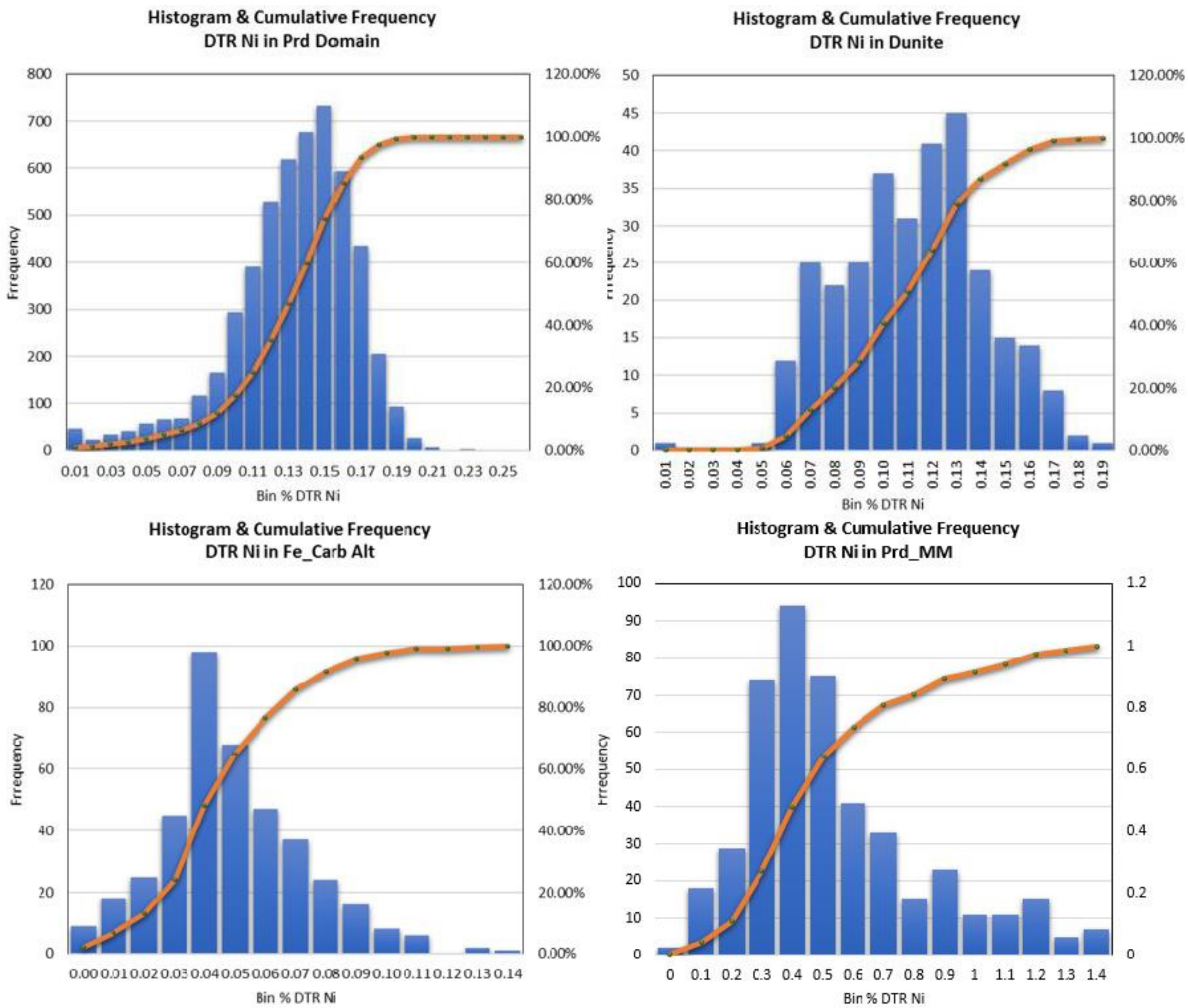


Figure 14-4: Frequency distribution of % DTR Ni grade within mineralized domainsDomain abbreviations are summarized in the caption for Figure 14-3

14.4 Density Assignment

A total of 978 specific gravity measurements were used to assign bulk density values to the lithologic domains (Table 14-3). The median value of the readings for each rock type were used except in the case of the metavolcanics where the single result of 2.23 was deemed unreliable.

The median density for the four ultramafic domains ranges from 2.7 to 2.8 g/ cm3, which is closer to the typical densities for serpentine (2.2-2.9 g/cm3) than it is to olivine (3.2-4.5 g/ cm3). Median domain densities are therefore consistent with the pervasive serpentinization of all ultramafic rocks of the Baptiste Deposit.

Table 14-3: Density assignments for each lithological domain

Lithological domain	Lithological code	Specific gravity measurements (No.)	Median density (g/cm3)
Dunite	1	55	2.66
Peridotite - Mineralized	2	638	2.67
Peridotite - Massive	3	69	2.79
Peridotite - Low Grade	4	33	2.79
Fe-Carbonate Altered Ultramafic	5	57	2.68
Listwanite and Fe-Carbonate Altered Ultramafic	6	65	2.69
Metavolcanic	7	1	2.71*
Sitlika Metasediment	8	1	2.71
Metasediment	9	6	2.71
Altered Dikes	10	53	2.92
Total		978	

*Actual median density is 2.23 g/cm3 but is considered unrepresentative

14.5 Grade Capping and Outlier Restrictions

Grade distribution in the composited data was examined to determine if grade capping or special treatment of high outliers was warranted. Cumulative probability plots were examined for outlier populations and decile analyses was performed for percent DTR Ni within the mineralized domains. Generally, the cutting or restriction of high grades is warranted if the last decile (upper 10% of samples) contains >40% of the metal or more than 2.3 times the metal of the previous decile, or if the last centile (upper 1%) contains >10% of the metal or >1.75 times the metal of the next highest centile.

For the mineralized domains, the last decile contained only 15% of the metal and the last centile contained 1.7% (Figure 14-5). It is therefore concluded that no capping or restriction of higher-grade composites is warranted.

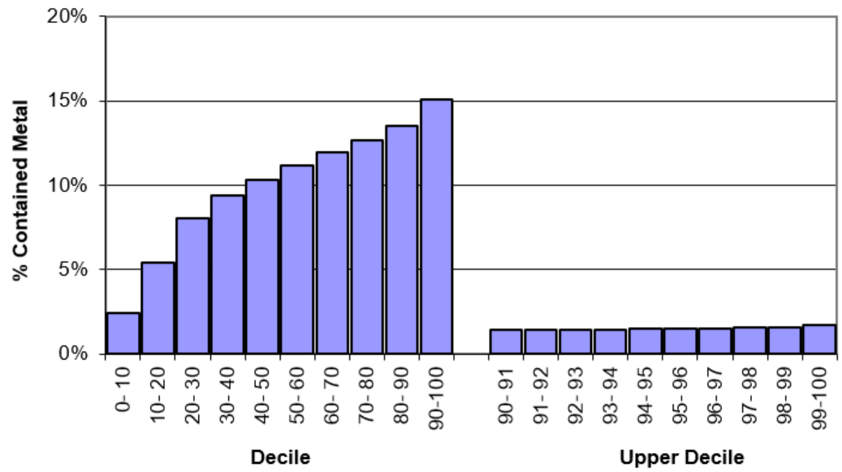


Figure 14-5: Decile analysis for mineralized domains

14.6 Variography

Kriging parameters, search parameters and anisotropy were determined with semi-variograms for percent DTR Ni, using composites falling within the mineralized domains. Nested spherical models showed a maximum range of 540 m (Figure 14-6). The resulting search ellipsoid has a major axis trending at an azimuth of 114° with the semi-major axis plunging steeply to the north-northeast.

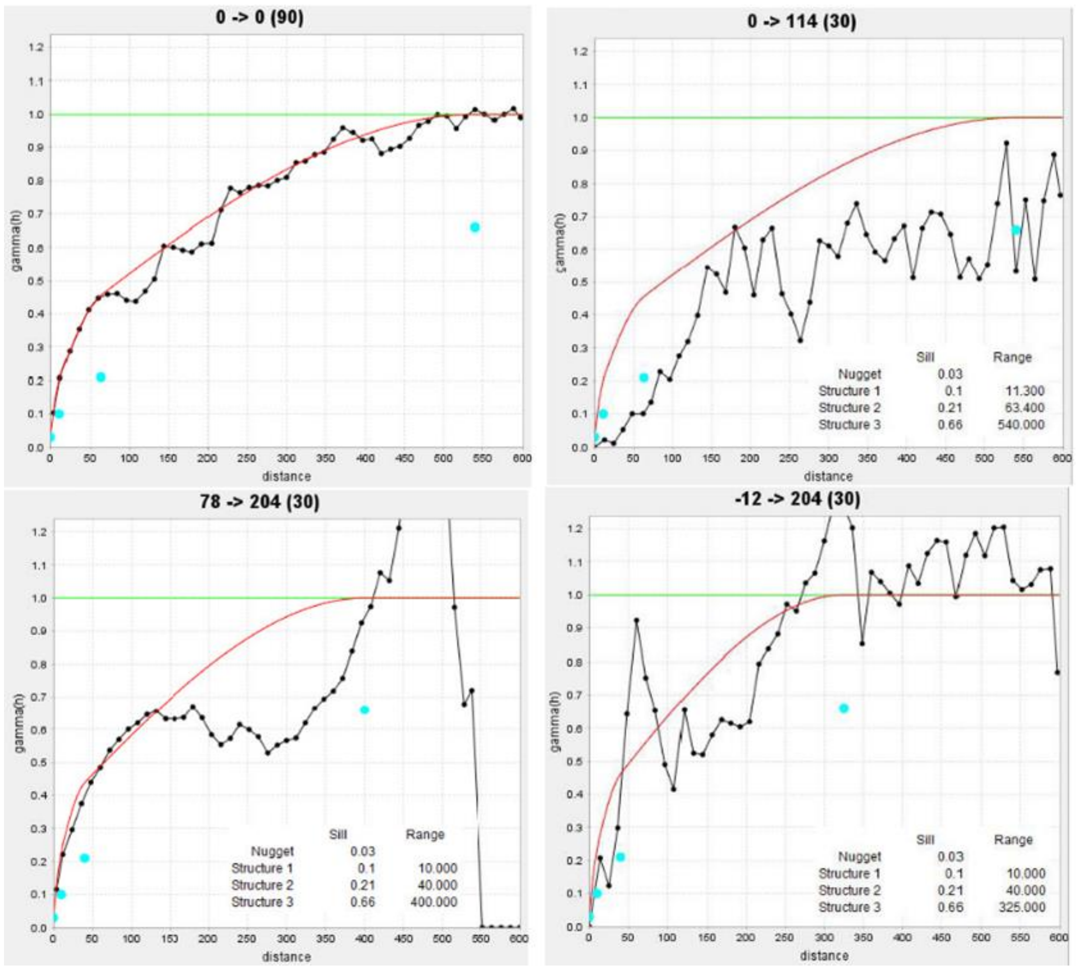


Figure 14-6: Variogram models for different plunges and azimuths of search windows

14.7 Estimation and Interpolation Methods

A block model was created in Geovia-Surpac Vision software using a block size with dimensions of 10 m x 10 m x 10 m. Model extents are presented in Table 14-4.

The model blocks were first coded by the partial percent and below topography and the overburden surface. Lithologic domain codes and density values were then assigned.

Table 14-4: Block model extents

	Easting (NAD83, Zone 10)	Northing (NAD83, Zone 10)	Elevation (m above sea level)
Minimum	346000	6081500	200
Maximum	351000	6085500	2000
Extent (m)	5000	4000	1800
Block size (m)	10	10	10
Blocks (N.o.)	500	400	180

DTR Ni grades within the corresponding lithologic domains were estimated in three passes using ordinary kriging (OK). For comparison purposes an estimate was also carried out using the inverse distance squared weighting method (ID2). A single pass nearest-neighbour (NN) estimate was also performed for use in model validation. Search parameters are outlined in Table 14-5. The anisotropy conforms to the search ellipsoids derived from the variogram models.

Soft boundaries were used between the Peridotite - Mineralized and Dunite domains. Semi-soft boundaries (± 50 m) were used between the Peridotite - Mineralized, - Massive and - Low Grade domains as these contacts are gradational. A narrower semi-soft boundary (± 25m) was used between the Fe-Carbonate Altered Ultramafic and adjacent ultramafic domains, as those contacts are gradational in all directions.

Mineralized domains of the Baptiste Deposit are cut by 34 steeply dipping, non-mineralized dikes, comprising approximately 3% of the rock mass in the classified resource blocks. These dikes are all >5 m thick and were identified as rock units that could be selectively mined as waste; the volume represented by these dikes was subtracted from the resource blocks based on the partial percent of the block within the dyke. Dikes <5 m thick were treated as rock units that are internally dilutive and account for approximately 1% of the rock mass in the classified resource blocks.

Block model grade distribution is illustrated in Figure 14-7 to Figure 14-10.

Table 14-5: Grade model search parameters

Pass	Search Distances			Composites Used		Maximum per hole
	Major axis	Semi-major axis	Minor axis	Minimum	Maximum	
1	150	118	90	12	36	11
2	300	236	181	12	48	11
3	500	394	301	12	48	

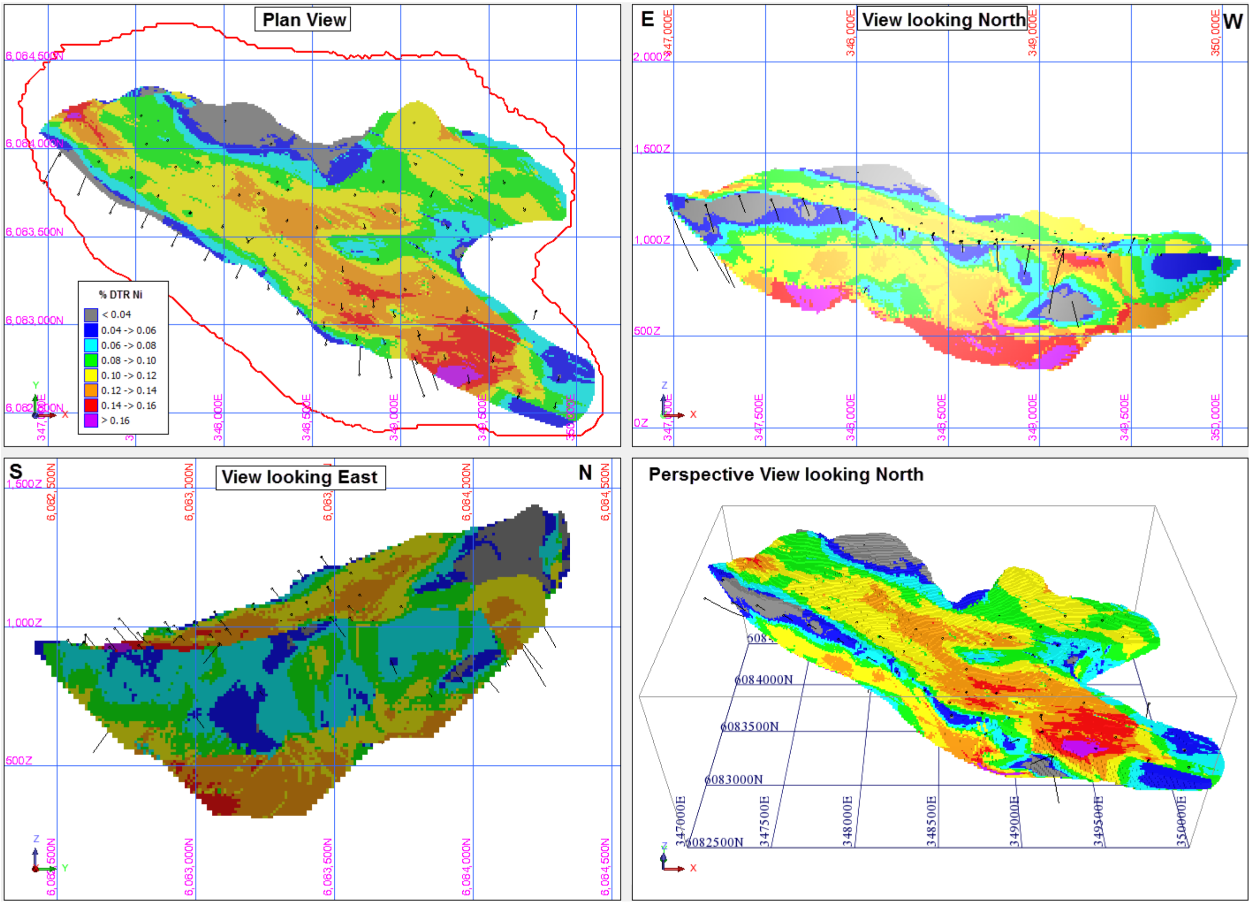


Figure 14-7: Block grade distribution for the Baptiste Depositin (clockwise from top left) plan view, east-west section looking north, tilted view looking north and north-south section looking west

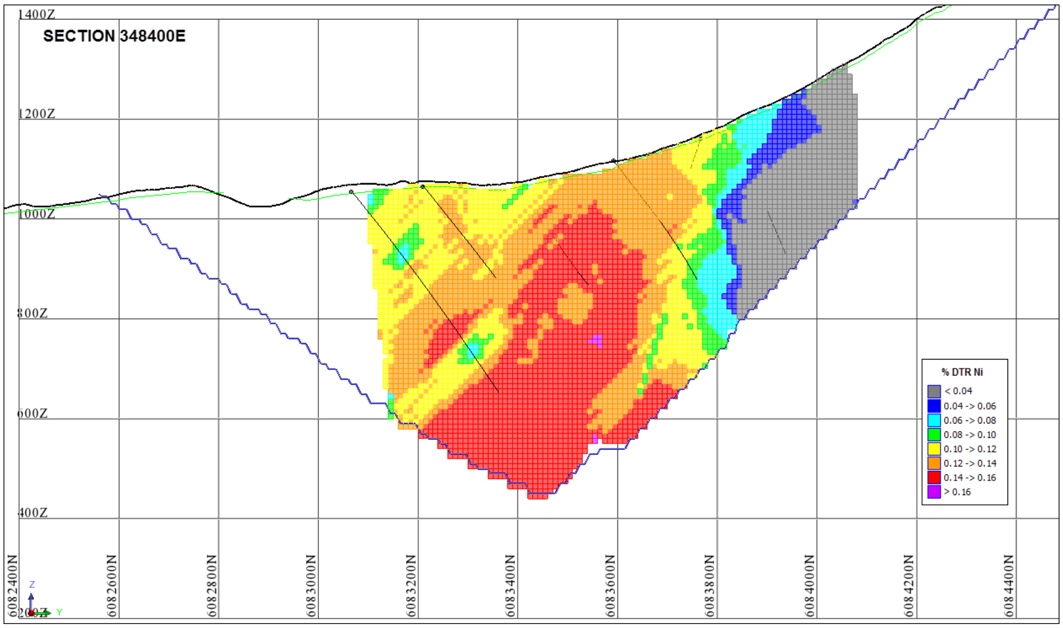


Figure 14-8: Block grade distribution for the Baptiste Deposit on cross section 348400E

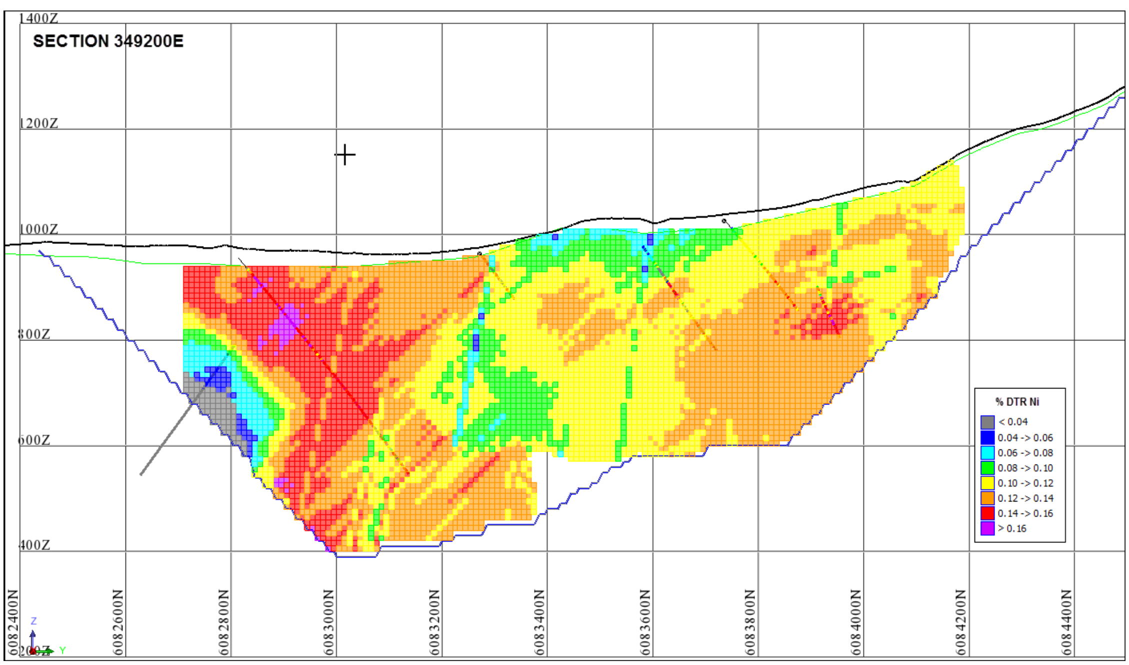


Figure 14-9: Block grade distribution for the Baptiste Deposit on cross section 349200E

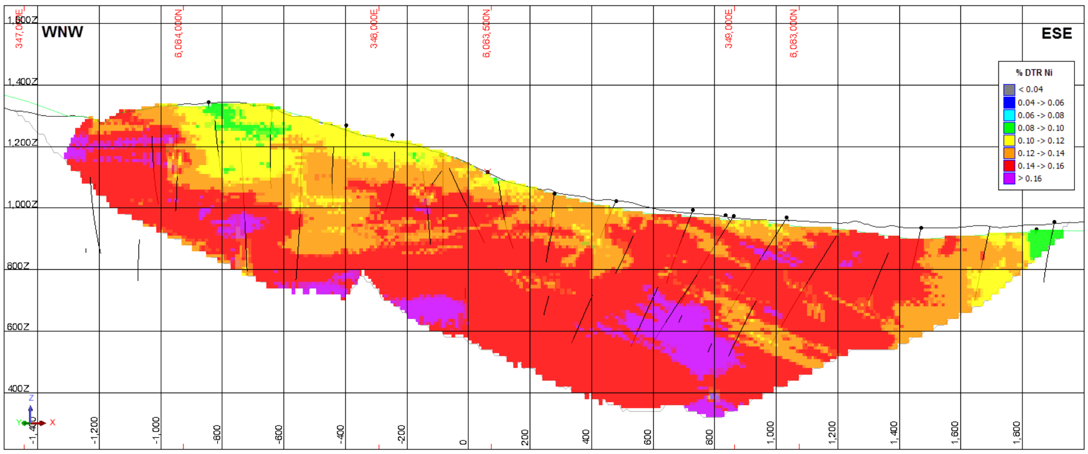


Figure 14-10: Block grade distribution for the Baptiste Deposit on longitudinal section

14.8 Block Model Validation

Block model validation included visual inspection, global bias check and a check for local bias. Each of these is summarized below.

Visual inspection comprised a visual comparison of blocks and composite grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

A global bias check was done by comparing the mean percent DTR Ni grades obtained for indicated and inferred resources through the different estimation methods (Table 14-6). These results show a reasonably close relationship to each other and with composite values.

The local bias check was done with swath plots that were generated to compare OK, ID2 and NN estimates on panels through the Deposit. Results show a reasonable comparison between the methods, as indicated by the bar charts on Figure 14-11 to Figure 14-13, particularly in the main portions of the Deposit.

Table 14-6 Global mean grade comparison

Population	% DTR Ni in indicated blocks	% DTR Ni in inferred blocks
Kriged Blocks	0.120	0.096
ID2 Blocks	0.121	0.096
NN Blocks	0.122	0.099

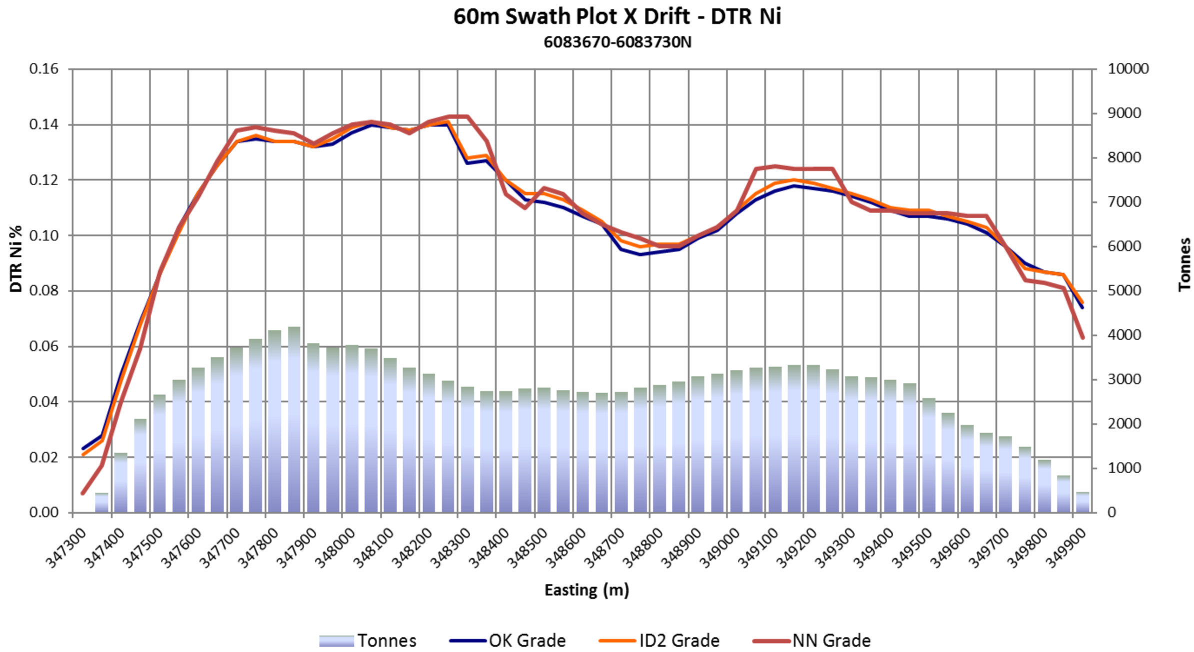


Figure 14-11: East-west trending swath plot at 6083670-3730N

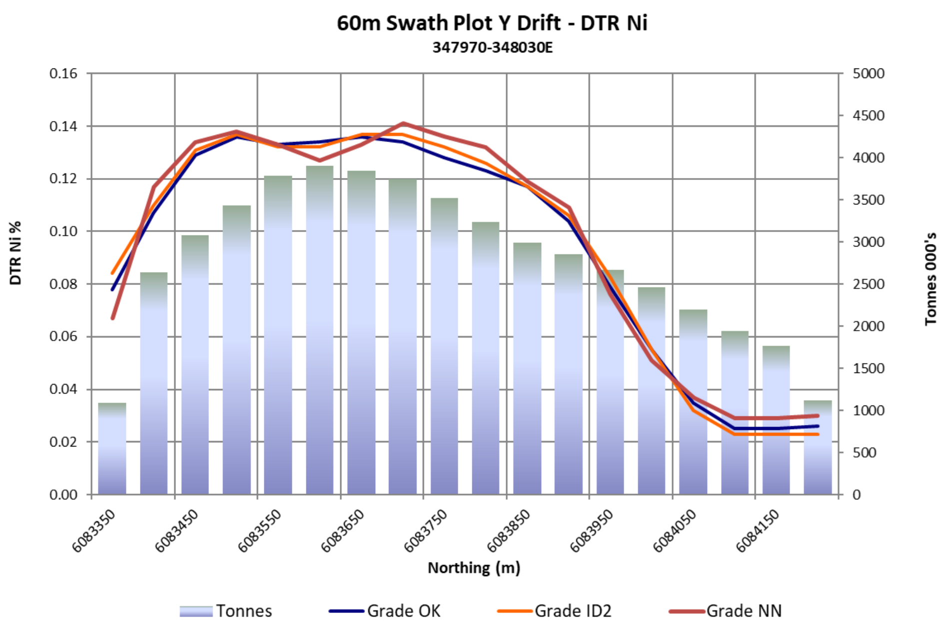


Figure 14-12: South to north oriented swath plot at 347970-8030E

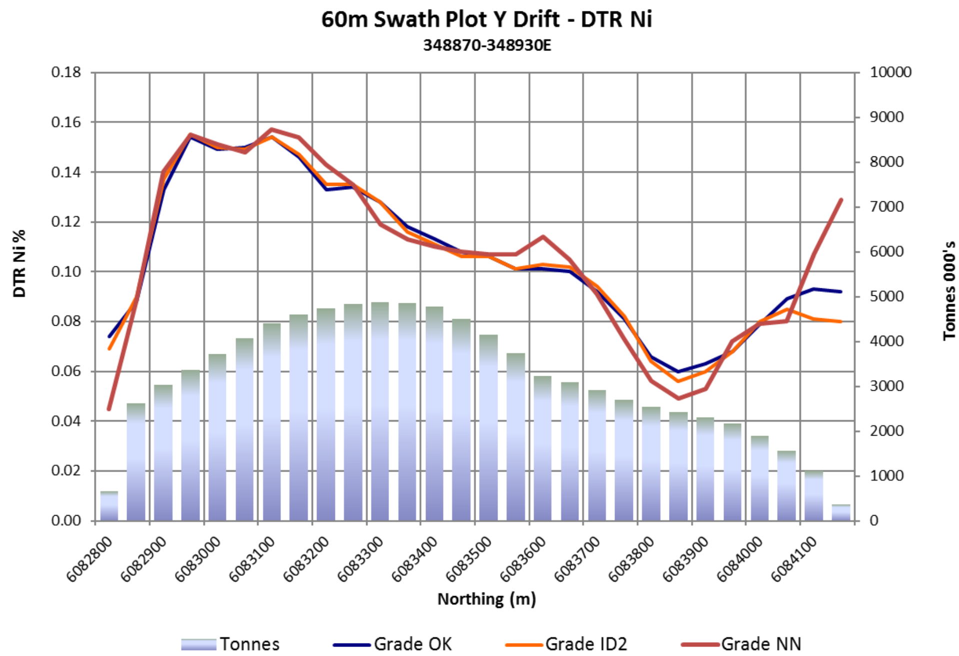


Figure 14-13: South to north oriented swath plot at 348870-8930E

14.9 Classification of Mineral Resources

Resource classifications used in this study conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). In order to be classified as an indicated mineral resource a block had to meet the following conditions:

- Restricted to the one of the four mineralized lithologic domains (i.e. all ultramafic domains except for Peridotite - Low Grade and Listwanite & Fe-Carbonate Alteration);
- Within a 200 m drill spacing

Blocks not classified as indicated mineral resources were assigned to the inferred mineral resource category if they fell within a 300 m drill spacing (Figure 14-14).

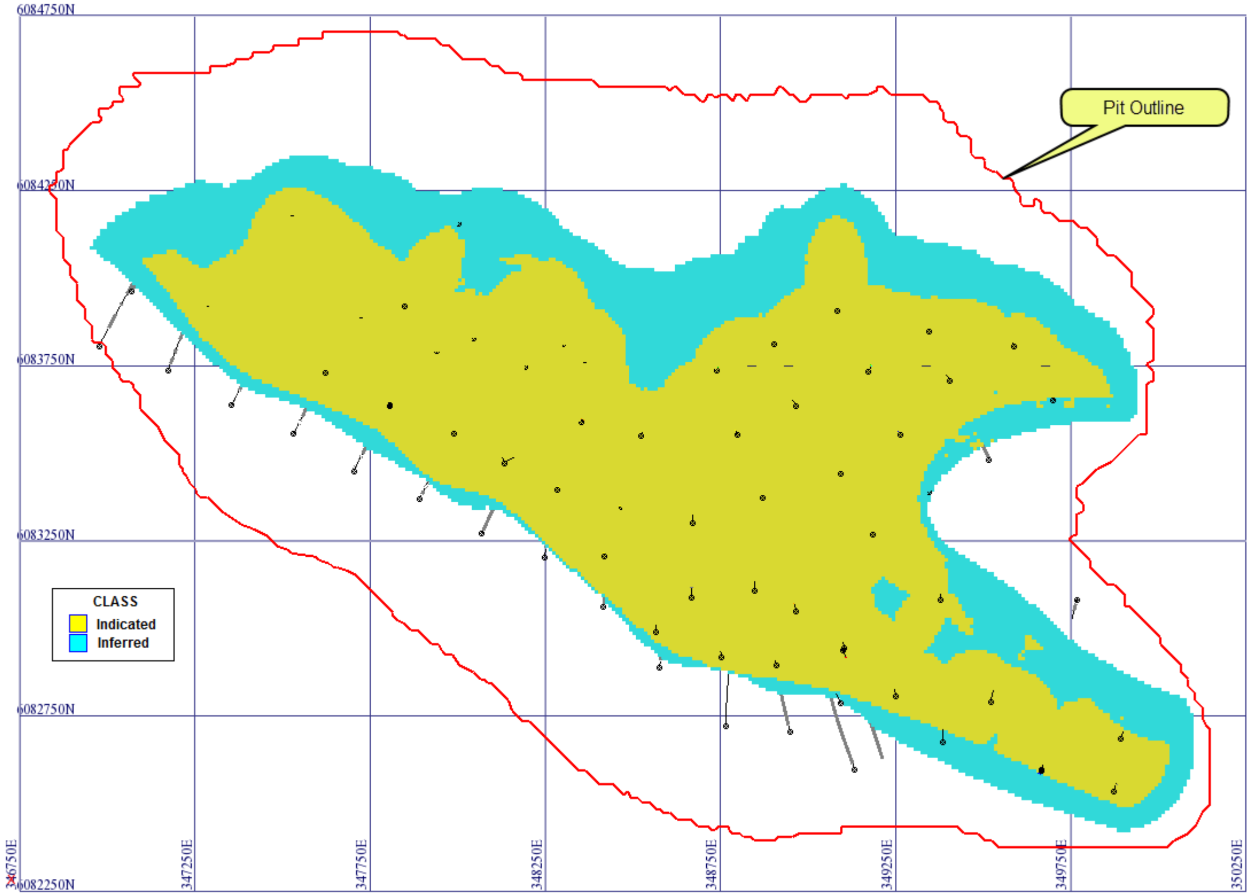


Figure 14-14: Plan view of resource classification for the Baptiste Deposit

14.10 Reasonable Prospects of Economic Extraction

Mineral resources were constrained by an optimized pit shell based on an exchange rate of $1.00 CAN = $0.76 US and a nickel price of US$6.35 per pound with 98.3% payable. Mining costs were assumed to be US$2.75 per tonne and processing plus G&A costs US$4.00 per tonne. A mining recovery of 97% and process recovery of 85% were used in the optimization. A US$1.00 per tonne minimum profit was also imposed to exclude material close to the break-even cut-off. Pit slope was 45˚.

A base case cut-off grade of 0.06% DTR Nickel represents an in-situ metal value of approximately US$7.00 per tonne, which is believed to provide a reasonable margin over operating and sustaining costs for open-pit mining and processing.

14.11 Mineral Resource Statement

Table 14-7 presents the mineral resource estimate for the Baptiste Deposit at a range of cut-off grades with the base case, at a cut-off grade of 0.06% DTR Ni, in bold face. Table 14-8 provides the summary of the 2020 Baptiste Deposit pit-constrained mineral resource estimate.

Table 14-7: Indicated and inferred resources for the Baptiste Deposit

INDICATED				INFERRED			
% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)	% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)
0.02	2,076,969	0.119	2,471,593	0.02	750,633	0.098	735,620
0.04	2,055,578	0.120	2,466,694	0.04	659,900	0.107	706,093
0.06	**1,995,873**	**0.122**	**2,434,965**	**0.06**	**592,890**	**0.114**	**675,895**
0.08	1,871,412	0.126	2,357,979	0.08	499,993	0.122	609,991
0.10	1,617,364	0.131	2,118,747	0.10	399,801	0.130	519,741

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. using ordinary kriging with an effective date of September 9, 2020.
2. Indicated mineral resources are drilled on approximate 200 x 200 metre drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximate 300 x 300 metre drill spacing.
3. An optimized pit shell was generated using the following assumptions: US$6.35 per pound nickel Price; a 45° pit slope; assumed mining recovery of 97% DTR Ni and process recovery of 85% DTR Ni, an exchange rate of $1.00 CAN = $0.76 US; and mining costs of US$2.75 per tonne, processing costs of US$4.00 per tonne. A US$1.00 per tonne minimum profit was also imposed to exclude material close to the break-even cut-off.
4. A base case cut-off grade of 0.06% DTR Ni represents an in-situ metal value of approximately US$7.00 per tonne which is believed to provide a reasonable margin over operating and sustaining costs for open-pit mining and processing.
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Table 14-8: 2020 Baptiste Deposit pit-constrained mineral resource estimate*

Category	Tonnes	Davis Tube recoverable (DTR) nickel content		
		(% Ni)	(Tonnes Ni)	(Pounds Ni)
Indicated	1,995,873,000	0.122	2,434,965	5,368,179,843
Inferred	592,890,000	0.114	675,895	1,490,093,663

*Notes:

The effective date of the 2020 mineral resource estimate is September 9, 2020. See Table 14-7 for additional notes concerning preparation of the mineral resource estimate.

Mineral resources which are not mineral reserves do not have demonstrated economic viability.

It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

14.12 Factors that may Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the mineral resource estimate include:

- Commodity price assumptions;
- Pit slope angles;
- Metal recovery assumptions; and
- Mining and Process cost assumptions.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in British Columbia in terms of environmental, permitting, taxation, socio economic, marketing and political factors. GeoSim is not aware of any legal or title issues that would materially affect the mineral resource estimate.

15. MINERAL RESERVE ESTIMATES

Not applicable to a PEA Study.

16. MINING METHODS

16.1 Introduction

The current mine plan, developed at a conceptual level for this PEA, was based on the mineral resource estimate presented in Chapter 14 of this Report and its underlying geological block model. The mine plan is based on a three-phase open pit mine development. The first phase of the proposed mine plan uses an external tailings storage facility (TSF) for disposing of tailings generated while mining the Phase 1 pit which will be mined for the first 21 years. After the Phase 1 pit is mined out, the tailings produced from Phase 2 and Phase 3 of the mine plan will be placed in the mined-out Phase 1 pit. A pit rim dam will be constructed in Year 25 to accommodate the additional tailings that will be stored in the Phase 1 and Phase 2 pits. The Phase 2 and Phase 3 pits will be mined from Year 22 to Year 35.

Mining will be conducted using conventional truck and shovel methods. Large-scale open pit mining will provide the mineral processing plant feed at a rate of 120,000 t/d, or 43.8 Mt/a which was based on processing capacity inputs provided. Annual mine production of mill feed and waste will peak at 80.1 Mt/a with a life-of-mine (LOM) stripping ratio of 0.40:1 including preproduction (0.32 during the first 10 years of operation, and 0.22 over the first 16 years of operations). Ultimate pit quantities with corresponding Davis Tube Recoverable (DTR) Ni grades are shown in Table 16-1 below. Figure 16-1 shows the site layout of the Project.

Table 16-1: Ultimate design pit quantities

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	1,326	0.124%
Inferred	177	0.102%
Total for Processing	**1,503**	**0.121%**
Waste Rock	540	
Overburden	55	
Total Waste	**596**	
Total Material Mined	**2,098**	
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

The following sections provide the parameters, methodologies and results of the mine planning tasks completed for the Baptiste Project.

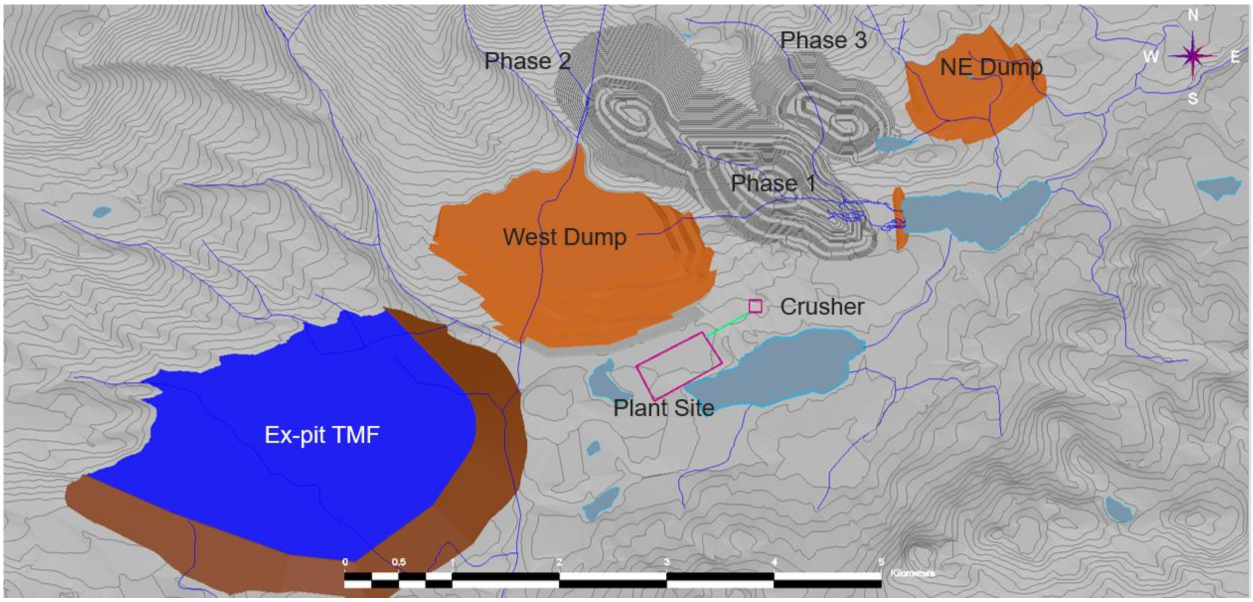


Figure 16-1: General site layout

16.2 Geotechnical

The Baptiste pit is designed for 10 m mining bench heights based on consideration of the loading equipment capabilities (mining height and reach). This may be modified during future detailed planning and equipment selection. In the absence of any geotechnical data for determining pit slope angle, for this PEA, Stantec assumed the same pit slope design parameters as in the 2013 PEA (McLaughlin et al, 2013). These parameters, shown in Table 16-2, have been accepted by Stantec as reasonable for this level of study.

Table 16-2: Pit design geometry

Design Feature	Phase 1 Pit	Phase 2 and Phase 3 Pit	Units
Overall Pit Slope	45	45	Degrees
Inter-ramp Angle	56	45	Degrees
Bench Face Angle	80	80	Degrees
Bench Height*	20 m (10 m x 2 mining benches)	20 m (10 m x 2 mining benches)	metres
Catch Bench Width	10		metres

16.3 Dilution and Mineralization Losses

The open pit is generally contained within the mineralized zone and has limited areas with the potential for dilution. Those areas with potential for dilution and loss include a small portion of the upper south west wall and some larger (>5 m) internal dykes. These waste contacts are very limited in extent and would be expected to contribute dilution tonnages of much less than 1% of the mineral mined. Therefore, a dilution factor has been excluded for this study. Smaller, non-separable, dykes of less than 5 m in width have been included in the tonnes and grades reported. It is recommended that further investigation into potential dilution factors are completed in the future.

During preproduction, A small mining loss of 44,000t at a DTR Ni grade of 0.096% has been assumed to occur as part of preproduction mining. This lost material represents initial base bedding for stockpiles and mill start-up test material. While some of the nickel present in this consumed material may be recovered, it is assumed as lost for the purposes of this evaluation.

16.4 Hydrogeology

Water inflows to the Baptiste open pit will include both groundwater and surface water runoff. The contributions from groundwater will progressively increase as the pit extends below the groundwater table. The contributions from surface water will be via direct precipitation into the pit and runoff from the contributing catchments around the pit excavation. The inflows from direct precipitation will increase with expanding pit area in conjunction with groundwater inflows as the pit increases in depth. No site-specific data for groundwater was available as of the writing of this report.

16.5 Pit Design Basis

The following sections summarize the design basis and assumptions used in developing the pit design.

16.5.1 Specific Gravity

In-situ dry density values for mineralized material and waste rock are provided in Chapter 14 of this Report. The in-situ dry density averages for rock ranged between 2.66 t/m³ and 2.92 t/m³. The in-situ dry density of 2.20 t/m³ was used for overburden. A range of swell factors to determine placed density were used depending on the destination or placement of the material.

16.5.2 Cut-off Grades

A cut-off grade (COG) of 0.06% DTR Ni was used. Using a process cost of $4.00/t milled with $1.00/t milled minimum profit and assuming 85% DTR Ni recovery, cut-off grades were calculated over a range of nickel prices, as shown in Table 16-3. The cut-off grade corresponding to a nickel price of $10,000/t at 98% playability, was selected to provide a reasonable profit margin per tonne of material processed (i.e. limit the amount of marginal material processed).

Table 16-3: Calculation of cut-off grade

Nickel Price ($/t)	Ni Price at 98% payable ($/lb)	COG
8,000	3.57	0.08%
9,000	4.01	0.07%
10,000	**4.46**	**0.06%**
11,000	4.90	0.05%
12,000	5.35	0.05%
14,000	6.24	0.04%
15,000	6.69	0.04%
16,000	7.13	0.04%
17,000	7.58	0.04%

16.5.3 Pit Optimization Method & Parameters

The pit optimization used to develop the mine schedule was run on the geological resource model presented in Chapter 14 of this Report. Both indicated and inferred resource categories are included in the reporting of the pit quantities. The pit optimization was done in two stages. In the first stage, a series of pit shells were generated using nickel prices from $8,000 to $17,000 /t. The results were used to define the Phase 1 pit where all the tailings produced will be stored in the external TSF.

The second stage of pit optimization was completed to define the Phase 2 and Phase 3 mining areas based on the strategy that all tailings from Phases 2 and 3 would be stored in the mined-out Phase 1 pit void.

16.5.3.1 Initial Pit Optimization Stage

The mining cost inputs were based on Stantec's internal cost database. The processing cost inputs and recovery % were provided by BBA.

Table 16-4 summarizes the inputs used in the initial pit optimization stage. Figure 16-2 shows the pit optimization results when varying nickel price from $8,000 to $17,000 /t.

Table 16-4: Pit optimization inputs for economic pit limits

Parameter	Cost ($)	Unit	Source
Mining			
Total Mining Cost	2.51	$/t milled	Stantec
Sustaining Capital	0.24	$/t milled	Stantec
Total	2.75	$/t milled	Stantec
Processing			
Operating Cost (To TSF)	2.89	$/t milled	BBA
Transportation	0.24	$/t milled	BBA
G&A	0.86	$/t milled	BBA
Total Processing Cost	4.00	$/t milled	
Minimum Profit	1.00	$/t milled	
Overall Plant Recovery	85	%	Test work
Overall Pit Wall Angle	45	degrees	2013 PEA (Tetra Tech)
Cut-off Grade (% Ni)	0.06	%	

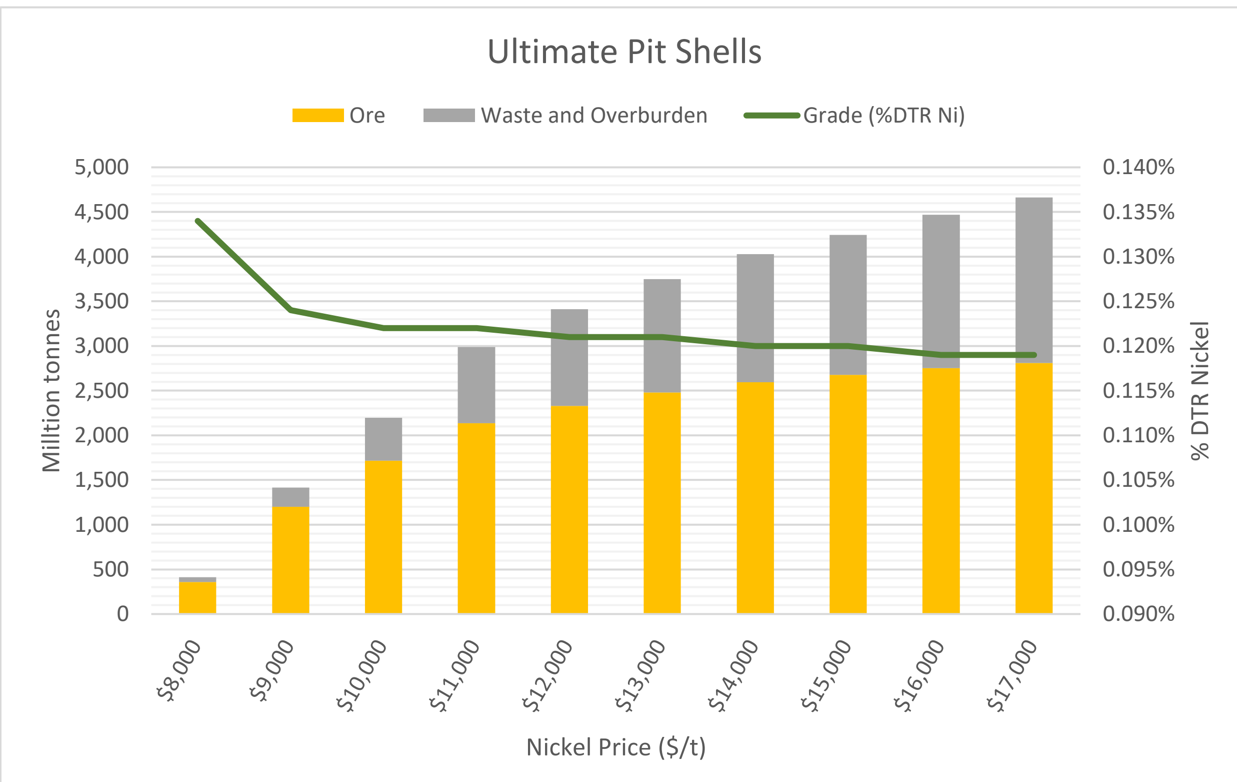


Figure 16-2: Pit optimization results

16.5.3.2 Pit Shell Selection for Phase 1

The $9,000 pit was selected as the basis for the Phase 1 pit following an evaluation to balance the first phase of mining with the capacity of a range of external tailings impoundments. Selection of the initial mining pit was constrained by the estimated capacity of the external tailings facility which in turn was constrained by the topography and major water bodies/rivers. The Phase 1 pit design was completed on the $9,000 pit shell prior to the second stage of pit optimization in order to define the capacity for in-pit tailings storage. Again, the Phase 2 and Phase 3 pit design limits were constrained by the capacity for storage of tailings within the mined-out Phase 1 pit.

16.5.3.3 Pit Optimization for Phase 2 and Phase 3 Area

Parameters used in the Phase 2 and Phase 3 pit areas are slightly different than the parameters used in the initial pit optimization stage because the tailings produced in these areas will be backfilled into the Phase 1 pit. Placing tailings in-pit eliminates the need for cyclone recovery of the coarser tailings sand to support on-going dam expansion. Therefore, the primary grind size at the concentrator can be adapted to maximize metal recovery without the constraint of a maximum fines content. The following assumptions were used when developing the pit optimization parameters for the Phase 2 and Phase 3 areas:

- Lower mining operating cost – eliminating on-going tailings dam construction cost.
- Higher plant recovery – finer grind for the process plant because the plant does not need to produce suitable cyclone sand material for the tailings dam construction.
- Higher plant operating cost – additional operating costs required for the finer grind process.

The pit optimization inputs for the Phase 2 and Phase 3 areas are summarized in Table 16-5.

Table 16-5: Mining and processing costs used in modelling

Parameter	Cost	Unit	Source
Mining			
Total Mining Cost	1.99	$/t milled	Stantec
Sustaining Capital	0.24	$/t milled	Stantec
Total	2.23	$/t milled	
Processing			
Operating Cost (To TSF)	2.90	$/t milled	BBA
Incremental Cost (In-Pit)	0.31	$/t milled	BBA
Transportation	0.24	$/t milled	BBA
G&A	0.86	$/t milled	BBA
Total Processing Cost	4.31	$/t milled	
Minimum Profit	1.00	$/t milled	
Plant Recovery	90	%	
Pit Slope	45	Degrees	
Cut-Off grade	0.06	%	

16.5.4 In-Pit Tailings Considerations for Pit Optimization

When developing the pit optimization for the Phase 2 and Phase 3 pit areas, additional boundary constraints were being applied in order to satisfy the in-pit tailings storage requirement. The optimization for the Phase 2 pit was limited to only mine from the northwest portion of the deposit and to not mine below the 940 m elevation from the Phase 1 pit. This elevation was chosen so that the Phase 1 pit could be backfilled with tailings produced from Phase 2 production without the need for a pit rim dam.

The second optimization was performed to define the extents of the Phase 3 pit. This optimization was constrained to not mine anything below the 980 m elevation from the ultimate Phase 1 and 2 pits. This elevation was the estimated maximum backfill level for tailings produced from Phase 3 production.

Future engineering studies should examine the lowest bench elevation constraint for Phases 2 and 3 to determine if the selected elevations are correct when tailings consolidation, ponded water and opportunities for additional recovery of mineralized material are considered.

16.5.5 Pit Shell Selection

A series of economic pit shells were created by varying the price of nickel in $1,000 /t increments using Minesight 3D©. Indicated and inferred resources were both used for the optimization. Pit shells created at a price of $14,000 /t were selected for both the Phase 2 and 3 ultimate pits. Table 16-6, Table 16-7, Table 16-8 and Table 16-9, and Figure 16-3Figure 16-4 show the variation in mineral, waste, and nickel grade by nickel price. These economic pit shells were selected as the basis for the pit design. Access roads, and catch benches were incorporated into the final pit design which is summarized in Section 16.5.7.

A Mill COG of 0.06% DTR nickel was used for the following resource calculations.

Table 16-6: Phase 2 pit shells by nickel price

Nickel Price ($/t)	Nickel Price ($/lb. at 98.3% payable)	Mineral (Mt)	Waste and Overburden (Mt)	Grade (%DTR Ni)	SR (t:t)
7,000	3.12	13.8	0.4	0.133%	0.029
8,000	3.57	61.8	4.4	0.122%	0.071
9,000	4.01	265.1	88.7	0.118%	0.335
10,000	4.46	345.2	177.4	0.120%	0.514
11,000	4.90	394.4	251.8	0.120%	0.638
12,000	5.35	420.9	301.7	0.120%	0.717
13,000	5.80	444.8	351.9	0.119%	0.791
14,000	6.24	456.4	381.0	0.119%	0.835
15,000	6.69	474.1	424.0	0.119%	0.894
16,000	7.13	490.7	467.3	0.118%	0.952

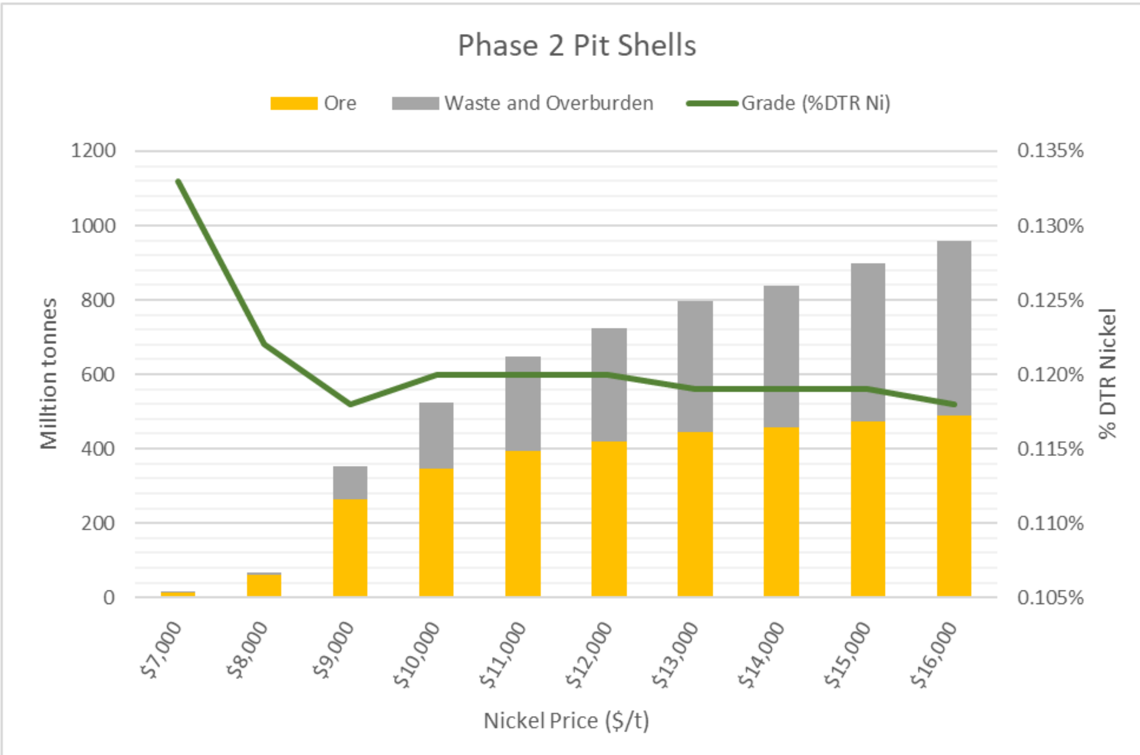


Figure 16-3: Phase 2 pit-by-pit graph

Table 16-7: Material quantities from selected Phase 2 pit shell

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	374,364,000	0.123%
Inferred	82,052,000	0.104%
Total Mineral	**456,416,000**	**0.119%**
Waste Rock	376,771,000	
Overburden	4,258,000	
Total Waste	**318,028,000**	
Total Material Mined	**839,445,000**	
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

Table 16-8: Phase 3 pit shells by nickel price

Nickel Price ($/t)	Nickel Price ($/lb. at 98.3% payable)	Mineral (Mt)	Waste and Overburden (Mt)	Grade (%DTR Ni)	SR
8,000	3.57	4.0	0.08	0.114%	0.020
9,000	4.01	97.9	9.5	0.110%	0.097
10,000	4.46	144.2	18.5	0.107%	0.128
11,000	4.90	175.2	31.7	0.105%	0.181
12,000	5.35	206.1	54.3	0.103%	0.263
13,000	5.80	224.4	71.4	0.102%	0.318
14,000	6.24	246.3	99.1	0.101%	0.402
15,000	6.69	261.1	120.5	0.100%	0.462

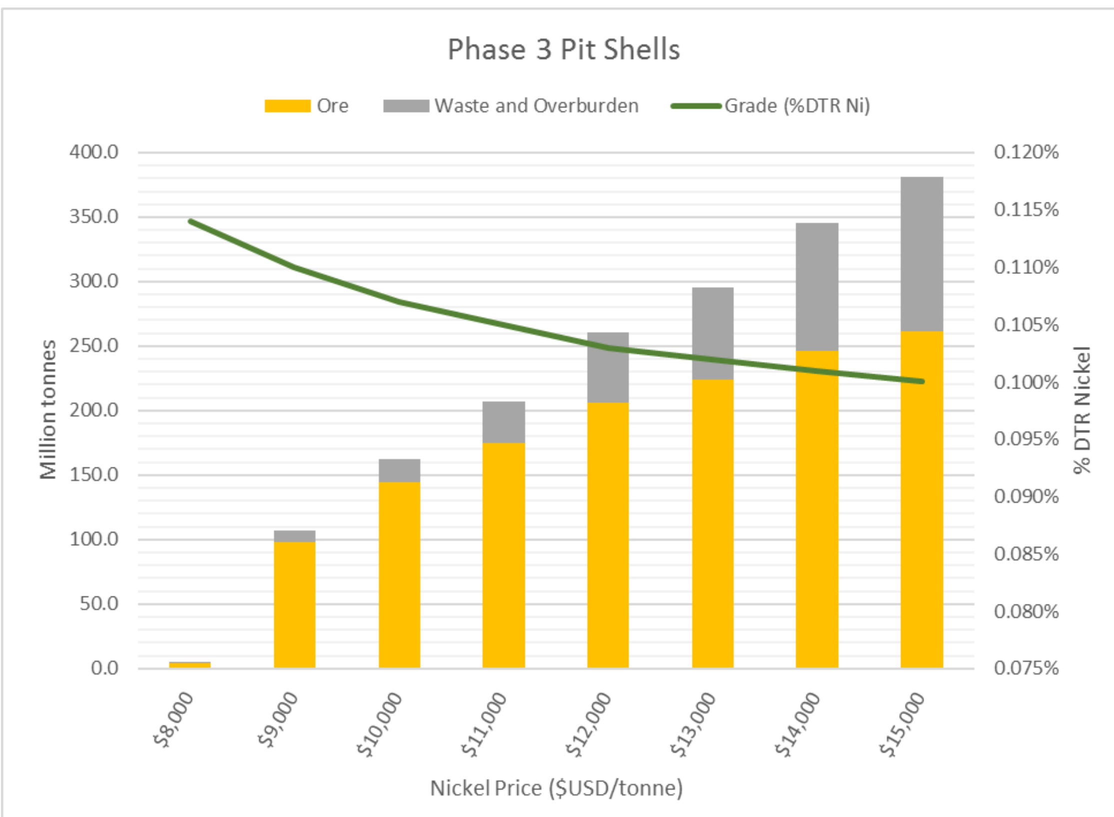


Figure 16-4: Phase 3 pit-by-pit graph

Table 16-9: Material quantities from selected Phase 3 pit shell

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	176,425,000	0.104%
Inferred	69,858,000	0.094%
Total Mineral	**246,282,000**	**0.101%**
Waste Rock	94,841,000	
Overburden	4,215,000	
Total Waste	**99,057,000**	
Total Material Mined	**345,339,000**	
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

16.5.6 Roads

The haul road widths are based on a 220 tonne (Komatsu 830E AC) size haul truck with a width of 7.3 m. Road widths are up to 40 m depending on ditching and berm requirements, and road grades are limited to 10%. British Columbia mine safety regulations require runaway lanes spaced as required for all roads over 5% grade (Ministry of Energy and Mines, 2017). No runaway lanes were explicitly included in the designed pit but opportunities to locate these ramps should be developed as part of future pit designs.

A conceptual haul road alignment leading from the pit to the tailings dam starter dyke was also developed.

Haul cycles were calculated for each key design period and haul simulations were carried out based on the designated 220 tonne haul trucks. Rolling resistance was estimated to be 4%, except within 100 m of a loading source or dumping destination, in which case 6% was used. Truck speeds are summarized in Table 16-10.

Table 16-10: Truck speeds

Speed Parameter	Speed (Km/h)
Maximum Speed	50
Maximum Speed in Switchbacks	15
Maximum Speed for Downhill hauls at a grade steeper than 5% (unloaded/loaded)	30

Truck hours needed for scheduled production were modelled using forty-nine (49) haul cycle routes. Truck hours were modelled annually for the first 5 years, and at 5-year increments after that.

During the pre-production period, a large number of roads are developed to support the pit, dump and TSF areas. These include:

- 12 km of heavy hauler roads (220-tonne trucks).
- 6 km of construction roads (41-tonne trucks).
- 5 km of access roads (pickup-truck).

When the mine expands into the Phase 2 area, which is shown in Figure 16-5, an additional 4 km of heavy hauler roads and construction roads were deemed necessary to provide access and maintain production.

16.5.7 Final Pit Design

The selected pit shells were used to design the ultimate pit shown in Figure 16-5. This three-phase ultimate pit was used as the basis for the mine schedule.

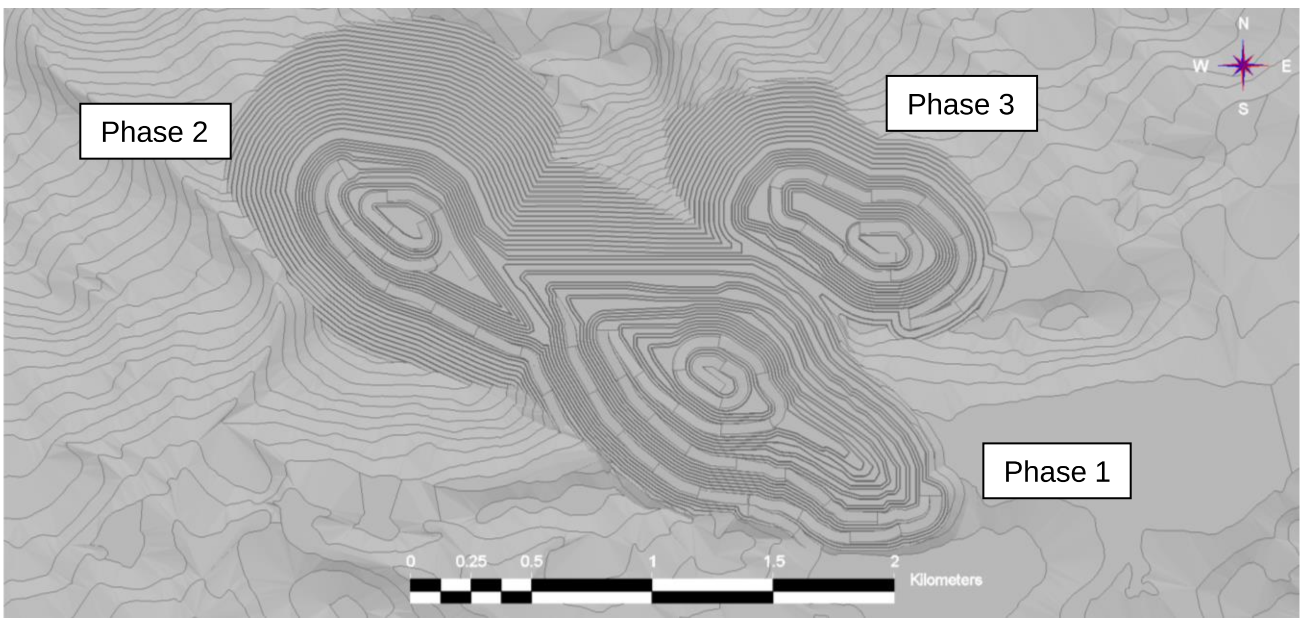


Figure 16-5: Plan of ultimate pit design and surrounding area

The ultimate design pit has a life of mine of 35 years, with three years of pre-production for road and infrastructure development, waste stripping, plant commissioning, and starter dyke construction. The overall pit quantities are summarized in Table 16-11. Table 16-12 shows the pit design quantities by phases and material classifications.

Table 16-11: Overall pit quantities

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	1,326	0.124%
Inferred	177	0.102%
Total Mineral	**1,503**	**0.121%**
Waste Rock	540	
Overburden	55	
Total Waste	**596**	
Total Mined	**2,098**	
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

Table 16-12: Pit quantities by mining phases

Material Classification	Tonnage (Mt)	Grade (%DTR Ni)
Phase 1		
Indicated	803	0.128%
Inferred	42	0.114%
Total Phase 1	845	0.127%
Phase 2 & 3		
Indicated	523	0.117%
Inferred	135	0.099%
Total Phase 2 & 3	658	0.113%
Total LOM	1,503	0.121%

16.6 Waste Dump Design

The proposed mine plan produces approximately 596 Mt of waste materials (waste and overburden), and majority of these materials will be placed in the west dump and the remaining amount will be placed in the northeast dump. Figure 16-1 shows the location of the two external waste dumps. The west dump has a design height of 200 m at an overall angle of 2.5H:1V. The northeast dump has a design height of 100 m at an overall angle of 2.5H:1V. The waste dumps will be constructed in controlled lift heights from the bottom-up to mitigate against flowslides. The waste material will be end-dumped from the haul trucks, and dozers (D10 or equivalent) will be used to push the material towards the dump crest.

16.7 Mine Schedule

The proposed mine schedule is summarized in the following sections. The annual production schedule quantities are shown in Table 16-13 and graphically in Figure 16-6. Table 16-14 shows the schedule quantities by mining phases and by material classifications. Multiple considerations such as ramp-up schedule, TSF construction requirements, pit backfill constraints, and metal production were considered in developing the production schedule.

Table 16-13: Production schedule

Year	-3	-2	-1	1	2	3	4	5	6	7	8
Total Mill Feed	-	-	44,000	32,850,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000
Average Mill Grade	0.000%	0.000%	0.096%	0.141%	0.134%	0.129%	0.137%	0.126%	0.109%	0.108%	0.113%
Total Overburden	-	17,779,000	26,187,000	-	-	-	-	-	-	-	-
Total Waste Rock	-	7,768,000	8,145,000	8,555,000	2,472,000	2,974,000	2,980,000	7,380,000	13,874,000	8,200,000	10,916,000
Total Production	-	25,547,000	34,376,000	41,405,000	46,272,000	46,774,000	46,780,000	51,180,000	57,674,000	52,000,000	54,716,000
Year	9	10	11	12	13	14	15	16	17	18	19
Total Mill Feed	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000
Average Mill Grade	0.119%	0.124%	0.130%	0.140%	0.125%	0.121%	0.121%	0.122%	0.121%	0.127%	0.129%
Total Overburden	-	2,793,000	423,000	2,000	6,000	-	-	-	71,000	105,000	9,000
Total Waste Rock	10,426,000	6,283,000	5,131,000	2,564,000	3,507,000	3,229,000	2,530,000	1,270,000	29,939,000	36,200,000	33,905,000
Total Production	54,226,000	52,876,000	49,354,000	46,366,000	47,312,000	47,029,000	46,330,000	45,070,000	73,810,000	80,105,000	77,714,000
Year	20	21	22	23	24	25	26	27	28	29	30
Total Mill Feed	43,800,000	43,800,000	32,850,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000	43,800,000
Average Mill Grade	0.123%	0.131%	0.105%	0.115%	0.115%	0.118%	0.125%	0.132%	0.119%	0.119%	0.125%
Total Overburden	135,000	3,628,000	-	-	1,000	-	97,000	358,000	58,000	1,652,000	955,000
Total Waste Rock	31,217,000	29,372,000	43,950,000	32,999,000	26,200,000	26,200,000	26,200,000	26,200,000	26,200,000	14,046,000	15,230,000
Total Production	75,152,000	76,800,000	76,800,000	76,799,000	70,001,000	70,000,000	70,097,000	70,358,000	70,058,000	59,498,000	59,984,000
Year	31	32	33	34	35	Total					
Total Mill Feed	43,800,000	43,800,000	43,800,000	43,800,000	35,654,000	1,502,998,000					
Average Mill Grade	0.122%	0.097%	0.095%	0.104%	0.113%	0.121%					
Total Overburden	865,000	214,000	-	-	-	54,456,000					
Total Waste Rock	11,200,000	13,618,000	5,627,000	2,932,000	769,000	539,398,000					
Total Production	55,865,000	57,632,000	49,427,000	46,732,000	36,423,000	2,096,852,000					

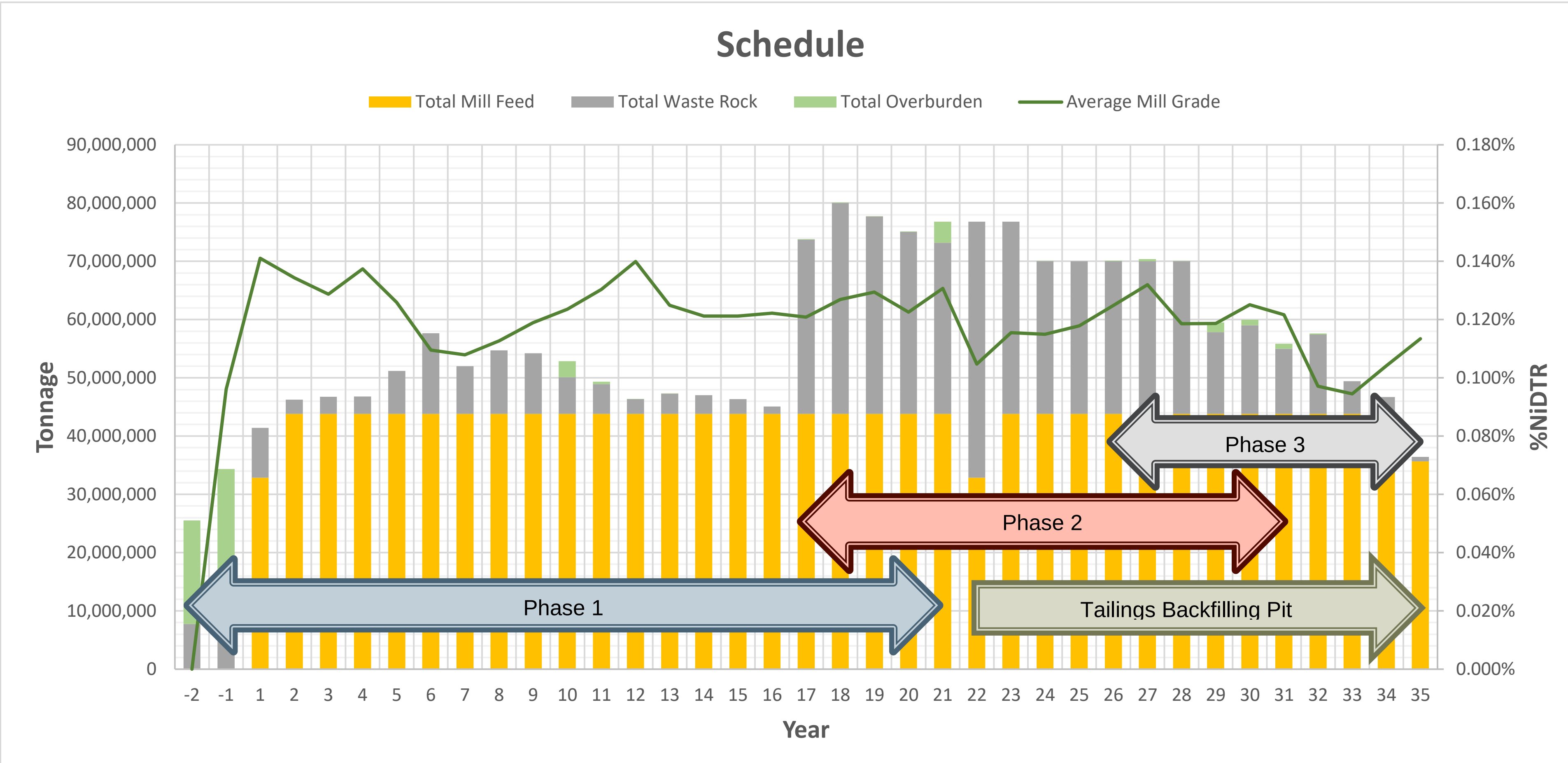


Figure 16-6: Production graph

Table 16-14: Design pit quantities by mining Phases and by Years

Material Classification	Years				
Phase 1	**1-7**	**8-14**	**15-21**	**22-28**	**29-35**
Indicated Tonnage (Mt)	274	289	240	-	-
Indicated Grade (% DTR Ni)	0.126%	0.126%	0.132%	-	-
Inferred Tonnage (Mt)	22	17	3	-	-
Inferred Grade (% DTR Ni)	0.122%	0.103%	0.114%	-	-
Total Phase 1	296	307	243	-	-
Phase 2 & 3					
Indicated Tonnage (Mt)	-	-	50	245	227
Indicated Grade (% DTR Ni)	0.000%	0.000%	0.100%	0.123%	0.115%
Inferred Tonnage (Mt)	-	-	13	51	71
Inferred Grade (% DTR Ni)	0.000%	0.000%	0.098%	0.101%	0.098%
Total Phase 2 & 3	-	-	64	296	298
Total LOM (Mt)	296	307	307	296	298
Note: Does not include pre-production years -2 and -1.					

16.7.1 Pit Phases

A total of ten production sub-phases, and one pre-production phase (Phase 1A) were developed and incorporated into the mine production schedule, as shown in Figure 16-7. These sub-phases are essentially pushbacks within the large pit areas (Phases 1, 2 and 3). Phase 1A was designed to target waste rock for the TSF starter dyke construction during pre-production and the remaining phases were designed to maximize metal grade early in the mine life.

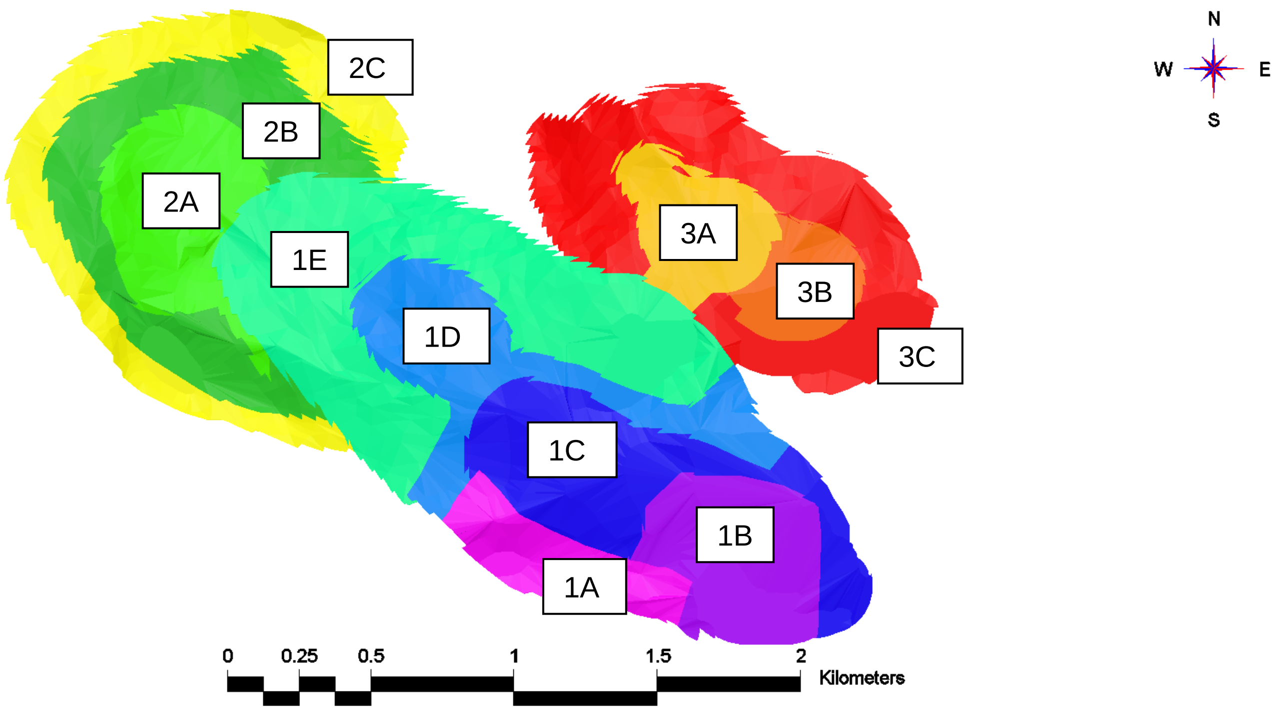


Figure 16-7: Pit phasing

16.7.2 Pit Development

The development and phasing of the Baptiste Deposit is described in the following sections.

16.7.2.1 Year -3 (Pre-production)

Excavation and backfill of the starter dam cut-off trench will be completed using the mine utility fleet or a contractor fleet if contractors are brought in to support pre-production activities.

16.7.2.2 Year -2 (Pre-production)

The primary mining fleet will be commissioned at the start of this period. Most overburden within the Phase 1 pit limit will be removed, with suitable dam construction material sent to the TSF starter dyke. Waste rock from Phase 1A is to be partially mined for starter dam construction. Initial pit development starts at the southwestern extent of the pit and will be mined to the final pit wall. This area will be mined in a stepped fashion from south to north with a maximum depth of 920 m. Unsuitable overburden and waste rock will be placed in the dump to the west of the pit. Initial site roads will also be constructed with a portion of the mined waste rock.

16.7.2.3 Year -1 (Pre-production)

The remainder of overburden will be stripped and hauled to the TSF to finish the construction of the starter dyke. A small amount of mineralized material will be mined from Phase 1A.

16.7.2.4 Year 1

Production will begin with a small pit located at the south end of the ultimate pit and will reach an elevation of 860 m (see Figure 16-8). Access will be gained from the top of the ramp exposed during preproduction. Mineralized material will be hauled west to the crusher, while waste rock will be hauled along the same road to the crusher but will continues west to be placed in the West Dump.

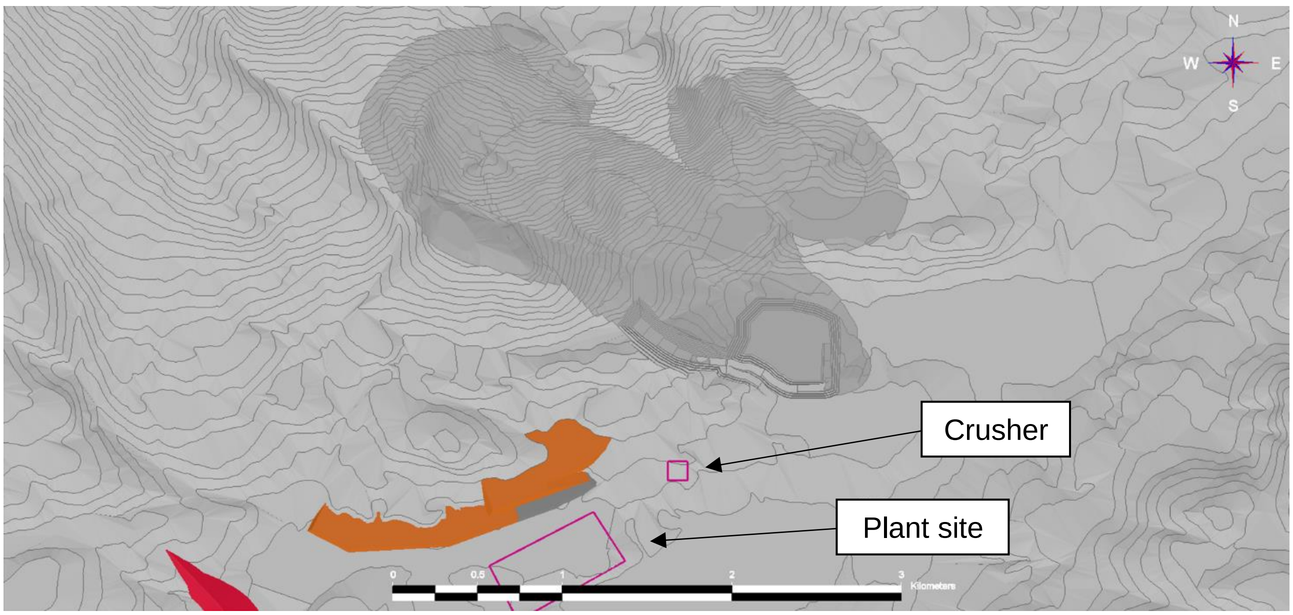


Figure 16-8: End of Year 1 pit

16.7.2.5 Year 2

Phase 1A of the pit will be completed in this year as shown in Figure 16-9. Waste and mineralized material from this phase will be hauled out along the main pit access haul road. Mining of Phase 1D also begins in this year to target a second higher-grade zone. Mining in this phase progresses from an elevation of 1,230 m down to 1,160 m, where mineralized material and waste are both removed along contour towards the West Dump. Mineralized material will be hauled across and down the West Dump to the crusher.

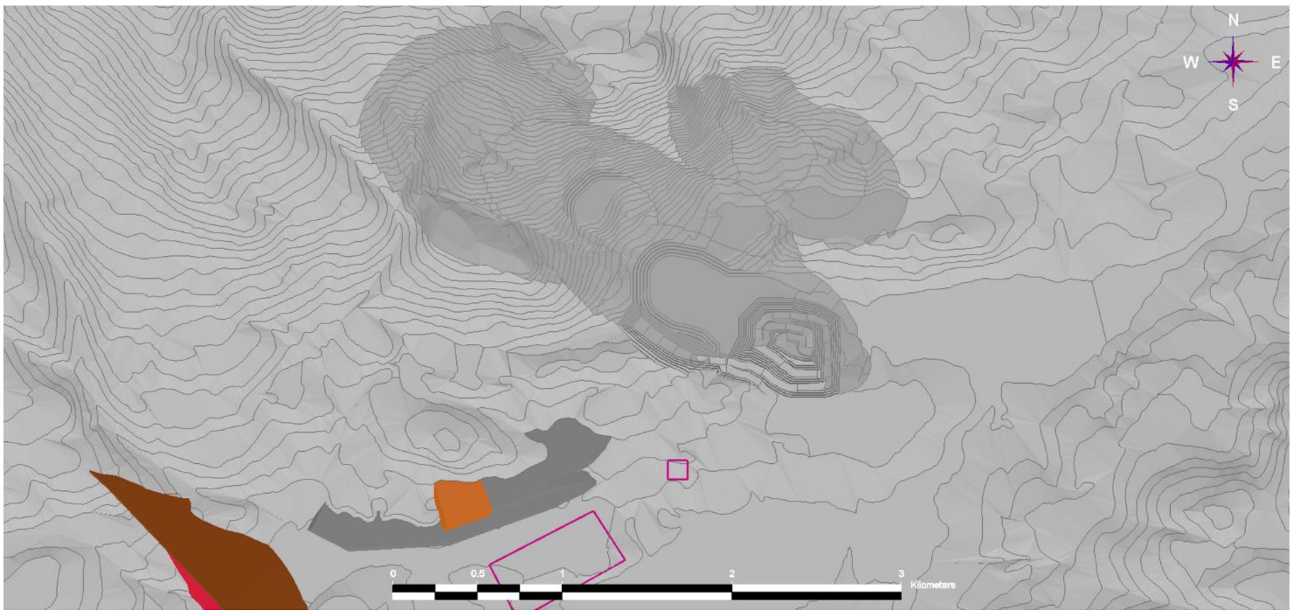


Figure 16-9: End of Year 2 pit

16.7.2.6 Year 3

Mining in Year 3 will progress in Sub-phases 1C and 1D as shown in Figure 16-10. Sub-phase 1C will be mined down to 910 m via the main haul road, and Sub-phase 1D mined down to 1,050 m, with waste and mineralized material being hauled downhill to the west dump and crusher respectively.

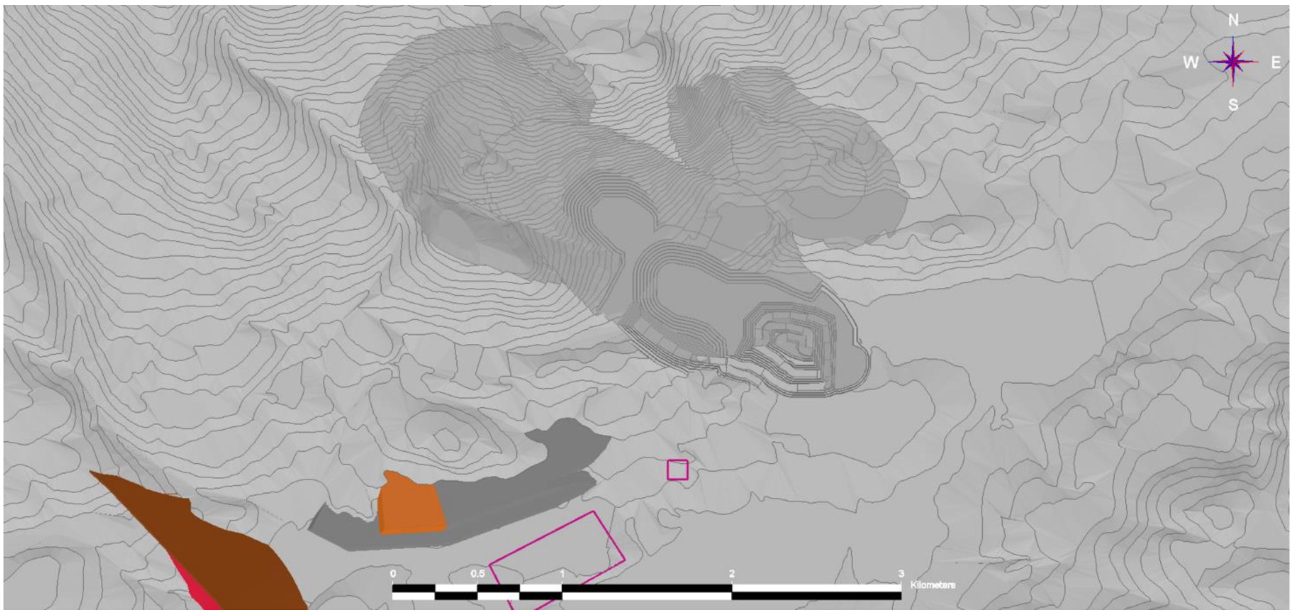


Figure 16-10: End of Year 3 pit

16.7.2.7 Year 4

Mining continues in Sub-phases 1C and 1D through Year 4. Sub-phase 1C will be mined down to 830 m and Sub-phase 1D mined down to 1,040 m. Waste rock will be placed in the west dump while avoiding interfering with the drainage immediately west of the facility (see Figure 16-11).

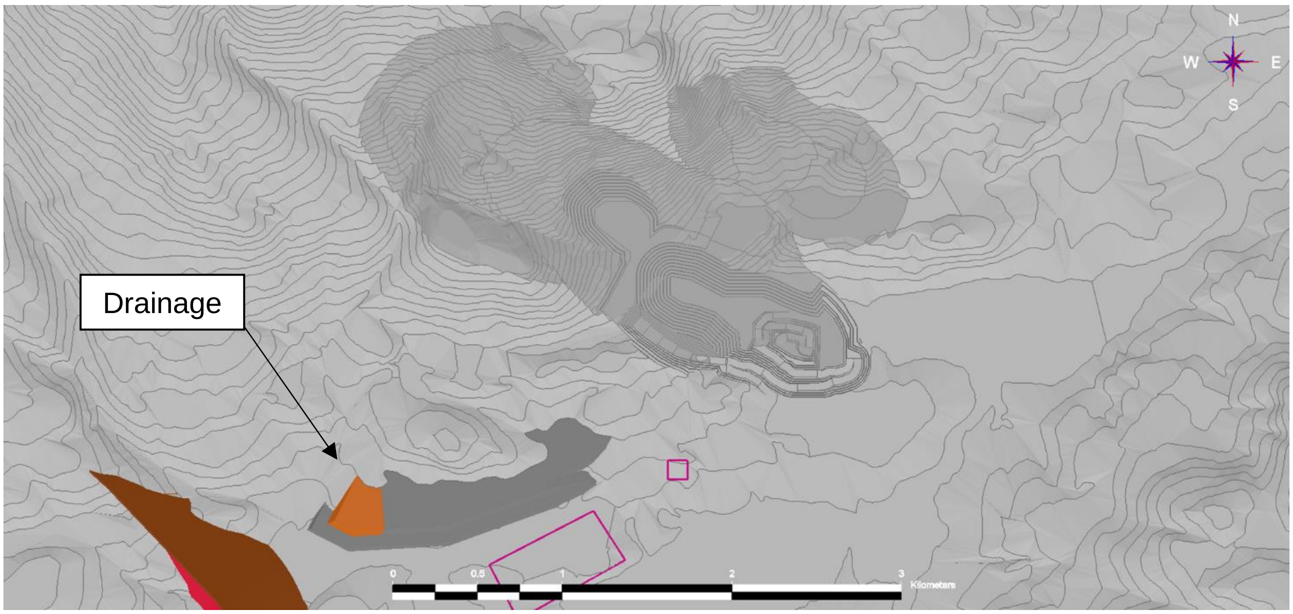


Figure 16-11: End of Year 4 pit

16.7.2.8 Year 5

Mining in Sub-phase 1E begins in Year 5 from 1,390 m to 1,270 m. Sub-phase 1D continues down to 1,020 m, and Sub-phase 1C will be completely mined down to 710 m. Material from Phase 1E will be hauled downhill, the majority of mineralized material and waste from 1D will be hauled downhill and along contours, and all of the material mined in 1C will be hauled uphill along the main haul road as shown in Figure 16-12.

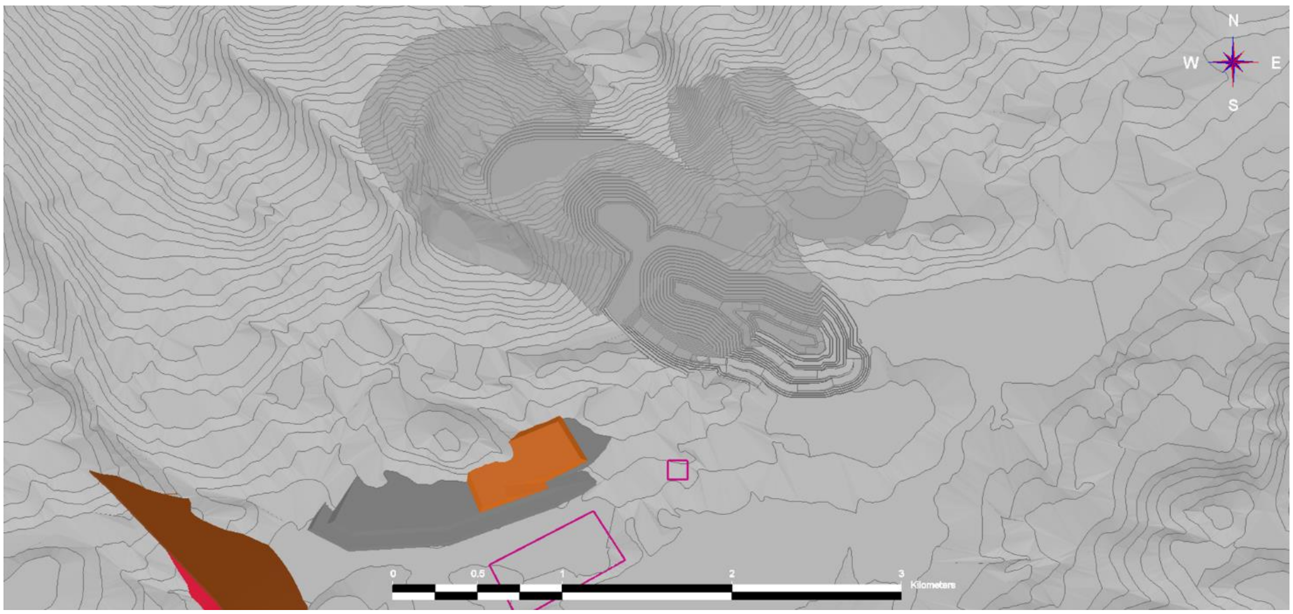


Figure 16-12: End of Year 5 pit

16.7.2.9 Years 5 to 10

Mining in years 5 to 10 lowers Sub-phase 1D from 1020 m down to 850 m and Sub-phase 1E from 1,270 m to 1,010 m as shown in Figure 16-13. Mineralized material and waste from 1D will be hauled up along the main haul road to the crusher and the eastern half of the waste dump respectively. Most of the mineralized material from 1E will be hauled downhill, with some in the later years hauled uphill along a highwall ramp. The waste from 1E will also be placed in the eastern half of the waste rock dump to shorten truck cycle times.

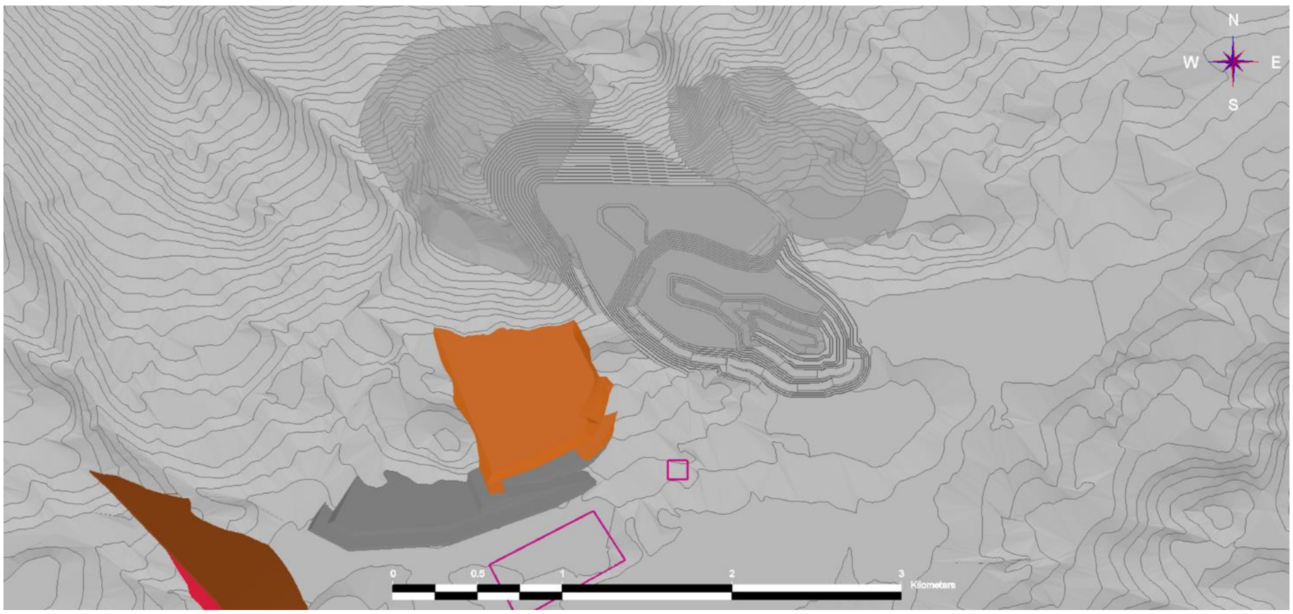


Figure 16-13: End of Year 10 pit

16.7.2.10 Years 10 to 15

All the mineralized material and waste from the Phase 1 pit will be hauled uphill along highwall ramps during years 10 to 15. Sub-phase 1D will be mined down from 850 m to 670 m and 1E will be lowered from 1,010 to 880 m. This waste will be placed in the western half of the West dump as shown in Figure 16-14.

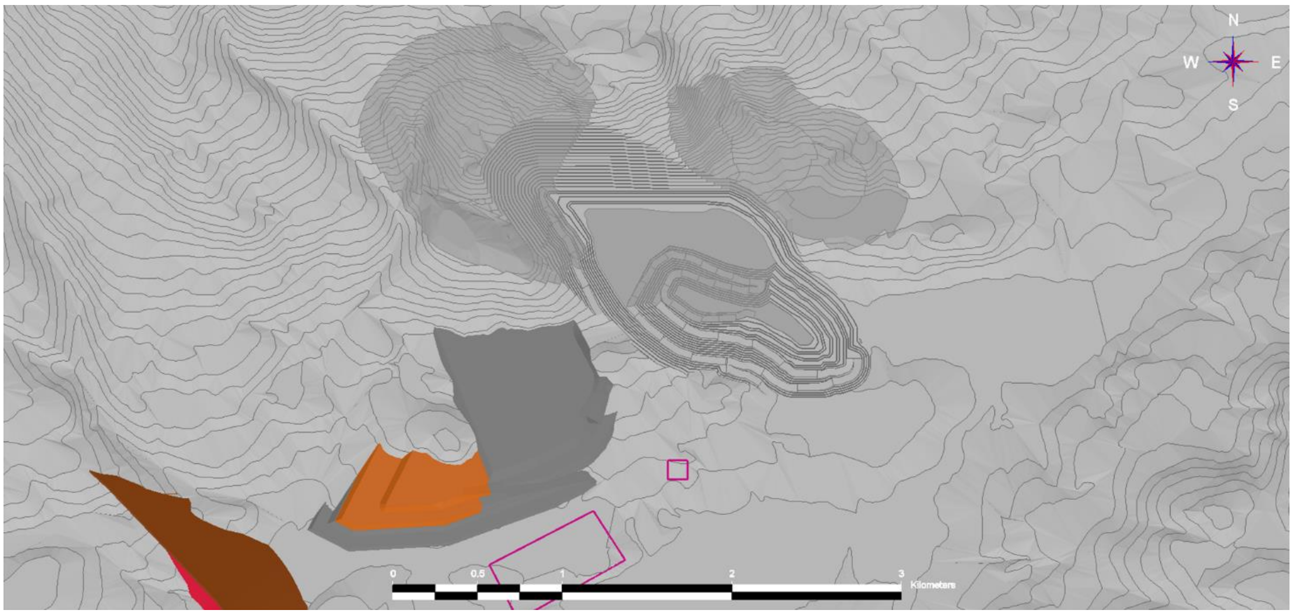


Figure 16-14: End of Year 15 pit

16.7.2.11 Years 15 to 20

Mining in years 15 to 20 completely mines out mineralized material in Sub-phase 1D and begins the Phase 2 pit as shown in Figure 16-15. Sub-phase 1E will be mined from 670 m to 630 m during this period. Sub-phase 2A will be mined from 1,410 m to 1260 m, while 2B and 2C are both mined together to the 1,280 m bench.

Phase 2 will initially be accessed via a pioneer road for articulated haul trucks that will work to establish a 40 m wide haul road for primary haul trucks. Ten (10) additional 220 tonne haul trucks will be required to maintain feed to meet the plant capacity during the early years of Phase 2 mining due to its increased strip ratio. Waste rock from both pits will be placed to the west of the waste rock dump to expand the facility to its full footprint area.

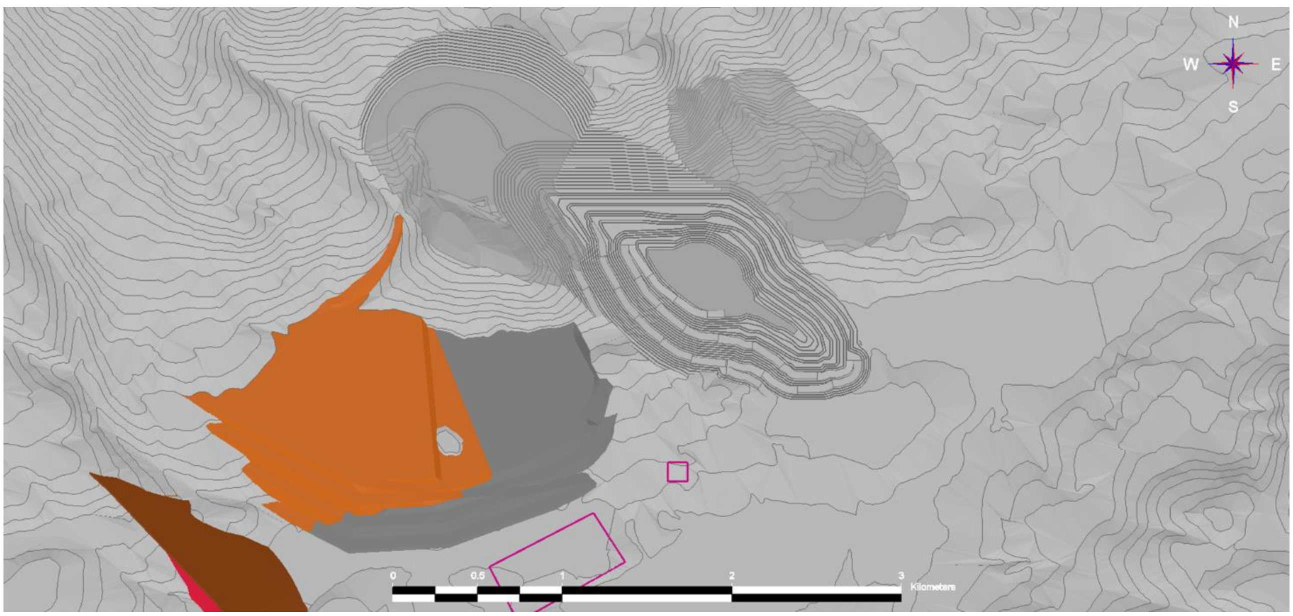


Figure 16-15: End of Year 20 pit

16.7.2.12 Years 20 to 25

The remainder of the Phase 1 pit will be completely mined out in Year 21 (see Figure 16-16). This pit will be ready to be backfilled with tailings upon completion of Phase 1 and final tailings placement in the ex-pit TSF. The low strip ratio Sub-phase 2A pit will be completely mined out. Sub-phases 2B and 2C are mined down from 1,280 m to 1,070 m and 1,170 m, respectively.

The ultimate pit rim dam will be built to 957 m (17 m high) at the end of this period so that it can be used to retain future tailings production from Phase 3. The dam serves a dual purpose of retaining tailings and facilitating access to the future Phase 3 pit and providing access to Phase 3.

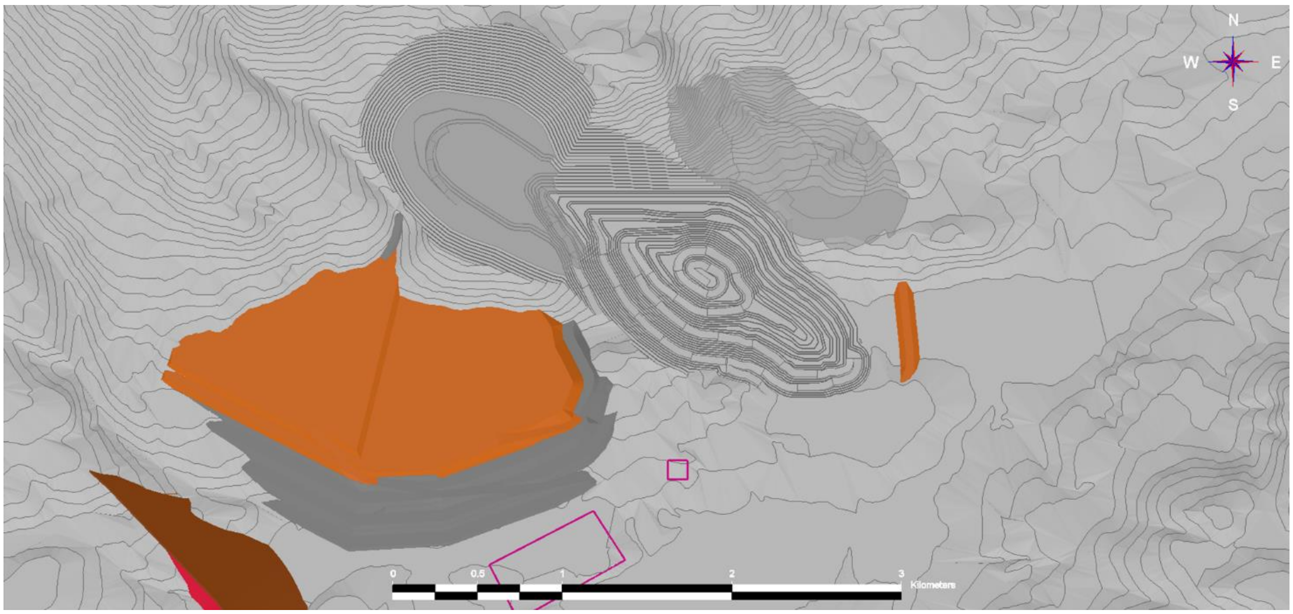


Figure 16-16: End of Year 25 pit

16.7.2.13 Years 25 to 30

Mining in years 25 to 30 will continue to advance the Phase 2 pit and begin production of the Phase 3 pit. Sub-phase 2B will be completed and Sub-phase 2C mined down to 870 m. A pit wall between Phase 1 and 2 will be maintained at the 940 m elevation to act as dam for backfilling Phase 1 with tailings (backfilling shown in teal below in Figure 16-17). This will make up the final significant waste placement on the West dump, which will raise the dumping platform to 1,175 m.

Sub-phases 3A and 3B will be mined together down to 1,010 m, while Sub-phase 3C will be mined to 1,040 m. The majority of the mineralized material haul is downhill to the crusher via the pit rim dam. Phase 3 waste rock will be placed in the northeast dump adjacent to the pit.

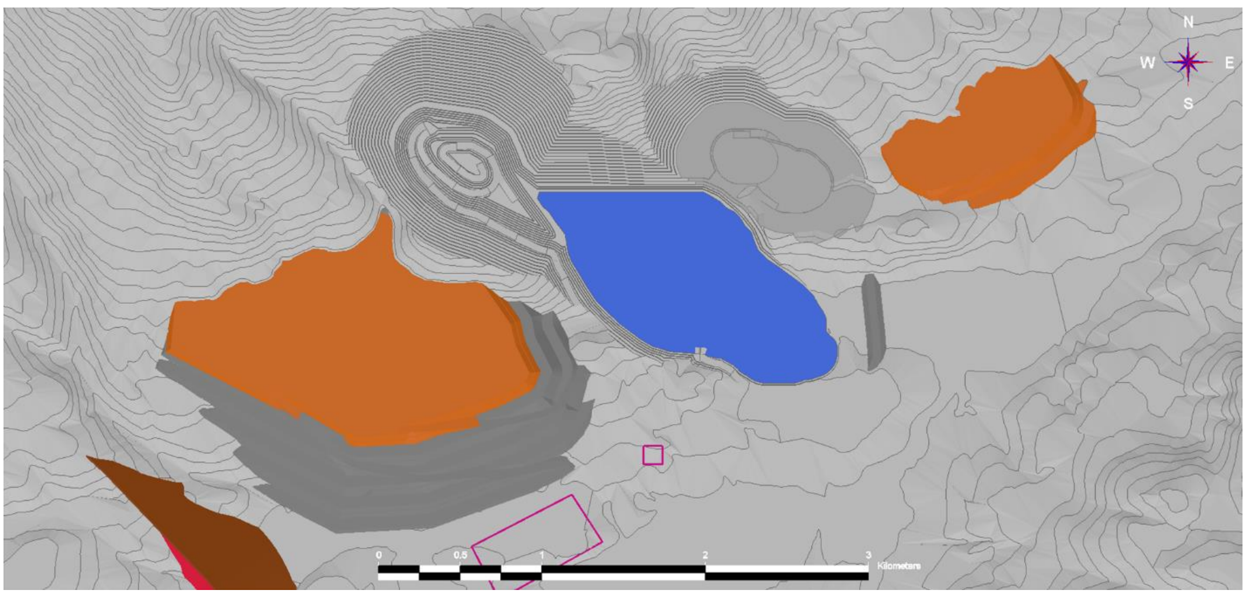


Figure 16-17: End of Year 30 pit

16.7.2.14 Years 30 to 35

The Phase 2 pit will be completed in Year 31 (see Figure 16-18) so that tailings produced from Phase 3 production can be stored in Phase 2. Completion of the Phase 3 pit in Year 35 represents the end of mining and completion of the northeast dump.

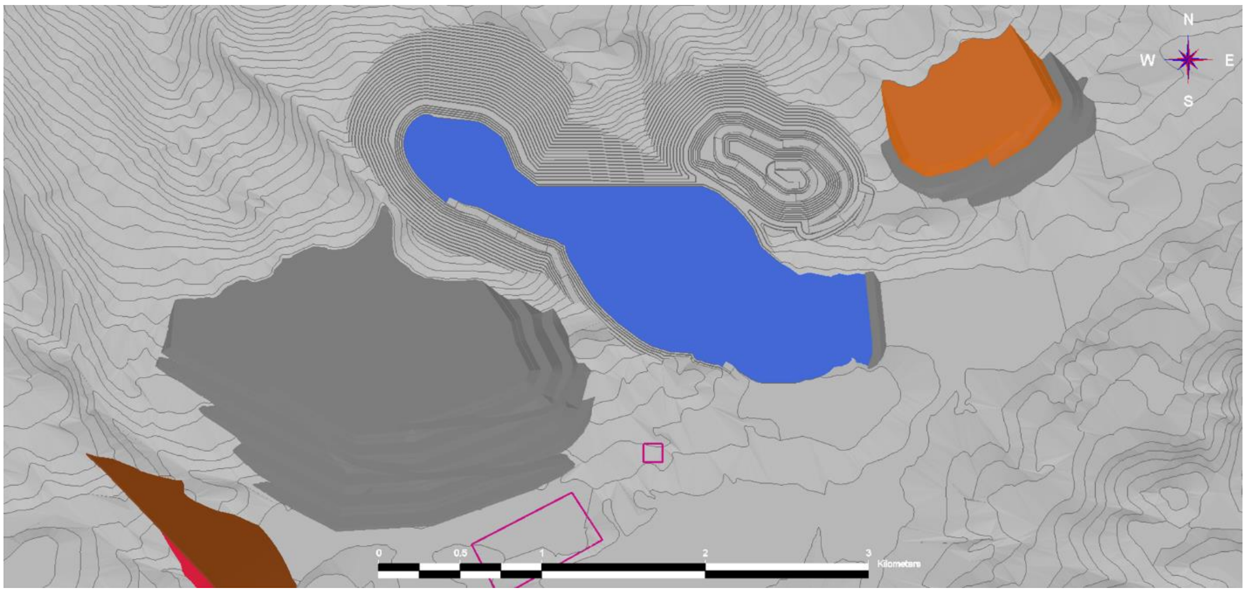


Figure 16-18: End of Year 35 pit

16.8 Production Rates and Recovery

The process facility is designed to process 120,000 tpd, 365 days per year, for a total annual throughput of 43.8 Mt. The mine will operate on two 12-hour shifts per day.

During the ramp-up period, a plant production capacity of 75% of the full production rate (32.85 Mtpa) was assumed for the schedule in order to allow for a gradual ramp-up to full production. This lower plant production rate also allows for commissioning and development of the tailings handling system.

Mining recovery is assumed at 97%.

Table 16-13 summarizes annual mining/plant feed schedule. The schedule is based on indicated and inferred mineral resource tonnages. Waste rock and overburden have a zero grade and are therefore automatically assigned to waste.

16.9 Mining Equipment

The Baptiste Project is envisioned as a conventional truck and shovel operation. Loading will primarily be carried out by hydraulic shovels loading trucks. Equipment is sized as follows:

- 29 m^3 Hydraulic Shovel (EX5600) (5 required).
- 240-ton Rigid Frame Haul Truck (Komatsu 830E) (Maximum fleet size of 29).

Trucks of 220-tonne (240-ton) capacity have been selected as a productive match for the loading equipment, permitting five-pass loading by the shovels. Figure 16-19 shows the graph summarizing the truck requirements per year. The reference to a specific haul truck manufacturer is not an endorsement of the specific machine by Stantec; alternate trucks of similar capacity and performance can be used.

To meet the targeted production level while maximizing grade, the mine plan relies on sufficient truck capacity to maintain the productivity of the primary loading units. The changing configuration of the mine over the Project life means that haul distances for both mineralized material and waste will vary significantly, and consequently the truck fleet requirements will also vary. For the first 16 years of the mine life, the mine is developing in the Phase 1 area which is closer to the plant and to the west waste dump. Stantec estimated 19 haul trucks are required during these periods. From years 17 to 20, where the mine is developing the high-strip area, Stantec estimated 29 trucks will be required during these years. From Year 21 to Year 34, the number of trucks decreases to 25.

Drill and blast operations will be carried out continuously as part of the normal mining operation. The drills will be required to drill to a depth to allow for blasting benches up to 10 m high (estimated 11-11.5 m depth with sub-grade). Five drills will be required at peak mining production.

The exact size and configuration of the support equipment fleet will be finalized at a later stage, but is expected to include units of the following type:

- Track-type dozers (450 kW, 250 kW).
- Wheel-type dozers (370 kW).
- Motor graders (16 ft blade, 14 ft blade).
- Hydraulic backhoes (350 kW, 300 kW, 150 kW).
- Articulated haul truck (45 ton).
- Water trucks for road dust control.
- Various maintenance vehicles: fuel/lube trucks, tow truck + flat deck trailer, mobile cranes, tire handler, FELs, buses, compactors, etc.

Some of the support equipment will initially be used for construction of the TSF starter dyke before moving to a general support role as the mine goes into production. Table 16-15 summarizes the primary and support equipment requirements over the LOM (including pre-production).

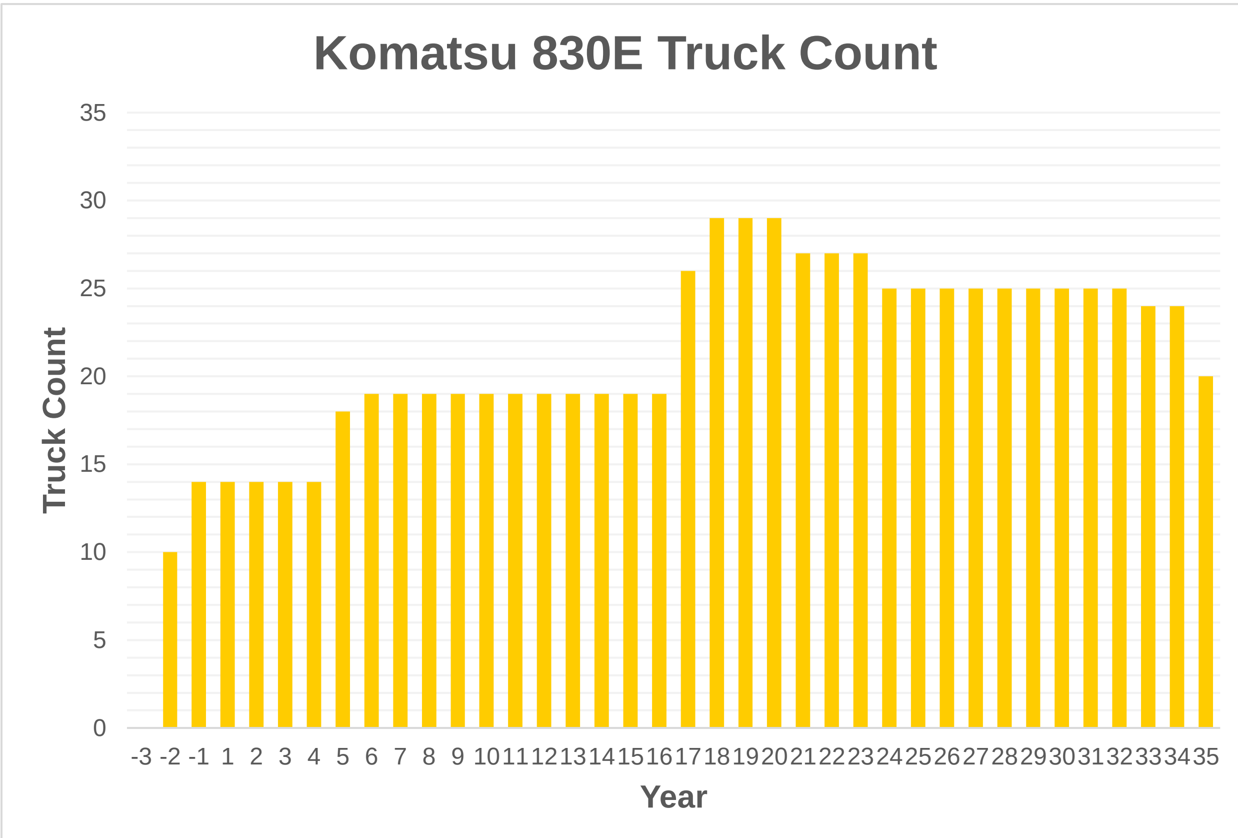


Figure 16-19: Fleet size by year

Table 16-15: Major mining equipment fleet size

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Shovel/Excavators																				
EX5600 (29m3)	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	5
350kW (CAT374)	2	2	2	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
300kW (CAT336)	2	2	2	3	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
150kW (CAT318)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2
Loaders																				
CAT980 (6m3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Trucks																				
830E (240ton)	0	10	14	14	14	14	14	18	19	19	19	19	19	19	19	19	19	19	19	26
CAT745 (45ton)	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Drills																				
PV271	0	1	1	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	7
Utility	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dozers																				
D10 (450kW)	0	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	9
D8 (250kW)	3	3	3	9	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14
370kW Wheeled	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2
Graders																				
16M (16ft blade)	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3
14M (14ft blade)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Year	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	Total
Shovel/Excavators																				
EX5600 (29m3)	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4	4	4	4	0	5
350kW (CAT374)	0	0	0	0	0	0	0	1	0	0	0	0	0	0	0	0	0	0	0	2
300kW (CAT336)	5	5	4	4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	5
150kW (CAT318)	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	0	2
Loaders																				
CAT980 (6m3)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Trucks																				
830E (240ton)	29	29	29	27	27	27	25	25	25	25	25	25	25	25	25	24	24	20	0	29
CAT745 (45ton)	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	4	4	0	6
Drills																				
PV271	7	7	6	7	7	7	7	7	7	7	7	6	6	6	6	5	5	4	0	7
Utility	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	1
Dozers																				
D10 (450kW)	9	9	9	9	8	7	7	7	7	7	7	5	5	5	5	4	4	3	0	9
D8 (250kW)	14	14	12	11	2	2	2	2	0	0	0	0	0	0	0	0	0	0	0	14
370kW Wheeled	2	2	2	2	2	2	2	2	2	2	2	1	1	1	2	1	1	1	0	2
Graders																				
16M (16ft blade)	4	4	4	4	3	4	3	3	3	3	3	3	3	3	3	3	3	3	0	4
14M (14ft blade)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.10 Blasting and Explosives

Blast patterns and loading parameters are based on 10 m bench heights with sub-grade drilling. The selected blast hole diameters are 200 mm in mineralized zones and 250 mm in waste rock. An average powder factor of 0.3 kg/tonne blasted has been used.

It is anticipated that a full-service vendor will provide the main explosives plant and storage, mobile manufacturing unit (MMU), loading personnel, and support vehicles. This is a common arrangement in the western Canadian mining industry.

17. RECOVERY METHODS

The testwork performed for the Baptiste Project using samples from the Baptiste Deposit is described in Chapter 13 of this Report. The testwork showed that a final flotation concentrate grade of 63.4% Ni can be achieved via milling, magnetic recovery and flotation. The final grind size (P_{80} of 25 µm) is required to achieve proper awaruite liberation.

From the testwork performed, the following recovery equation was developed to estimate DTR (Davis Tube Recoverable) Ni recovery as a function of the primary grind size:

- DTR Ni Rec % = (-0.0376 X Primary grind size µm) + 95.93

This recovery factor is applied to the head %DTR Ni in the ROM material mined and processed. The final flotation concentrate weight recovery is calculated based on the targeted final %Ni grade, assumed to be 63.4% Ni for this PEA.

The current PEA is based on a project with a nominal mineral processing capacity of 120,000 tpd, equating to 43.8 Mtpa. The initial primary grind size has been set to 300 µm. This grind size is dictated by the tailings deposition strategy and the TSF design, as described in Chapter 18 of this Report. The 300 µm primary grind size is deemed to be the finest grind size that will generate sufficient sand tailings to accommodate the initial tailings deposition plan, which covers Years 1 to 21 of the project life, referred to as Phase 1 of the Project.

Starting in Year-22, process tailings will be disposed of using an in-pit deposition strategy. As such, the primary grind size is no longer constrained. BBA performed a high-level analysis to determine a more optimal primary grind size that would provide improved DTR Ni recovery for an estimated increase in capital and operating costs. The results of this analysis indicated that a 170 µm primary grind size should be targeted post Year-21 of the project life, referred to as Phase 2 of the Project.

The current PEA is based on primary and secondary crushing with screening, stockpiling, HPGR crushing with screening and ball milling to the primary grind size. This is followed by a primary magnetic separation step and its concentrate is subject to regrind to the final grind size of 25 µm and further magnetic separation. The fine magnetic concentrate is then subjected to a flotation step and the final concentrate is dewatered and briquetted into the final saleable product.

17.1 Process Design Basis

Design of the concentrator and ancillary facilities is based on a plant typically operating 24 h/d, 7 d/w, 365 d/y. The primary and secondary crushing plant utilization has been assumed to be 70% and the plant, downstream of the crushed product stockpile utilization, has been assumed at 92% to account for planned and un-planned downtime. Most equipment incorporates a design factor of 15% above nominal to account for operational variability. The first year of operation is assumed to be a ramp-up year with mill throughput at 75% of nominal capacity. Year-22 is also assumed at

75% of nominal throughput to account for construction and tie-ins to incorporate the added grinding equipment to achieve the targeted finer primary grind. Table 17-1 presents the general design basis for both Phase 1 and Phase 2:

Table 17-1: Process design basis

Parameter	Unit**	Phase 1*	Phase 2*
Total feed processing rate	Mt/a	43.8	
Primary grind size	µm	300	170
DTR Ni recovery	%	84.7	89.5
Mine head grade DTR Ni	%	0.125	0.115
Final concentrate Ni grade	%	63.4	
Briquetted concentrate weight recovery	%	0.167	0.162
Total contained Ni produced	tpa	46 373	45 081
Total briquetted concentrate produced	tpa	73 143	71 106
Crushing			
Crushing circuit utilization	%	70	
Nominal crushing rate	tph	7 143	
Design crushing rate	tph	8 214	
Concentrator (incl. HPGR)			
Concentrator utilization	%	92	
Nominal concentrator fresh feed rate	tph	5 435	
Design concentrator fresh feed rate	tph	6 250	
Nominal concentrate production rate	tph	9.1	8.8

* Phase 1 from Yr-1 to Yr-21. Phase 2 from Yr-22 to Yr-35. Excludes ramp-up years Yr-1 & Yr-22.
** All tonnages are in dry metric tonnes.

17.2 General Process Design

As part of its conceptual process design work for this PEA Study, BBA developed the following, some of which were provided to FPX as internal documents:

- A Process Flow Diagram (PFD) outlining the major processing areas and equipment;
- A Process Design Criteria (PDC);
- A Mass and Water Balance;
- A major equipment list outlining equipment size and number as well as motor power;
- High-level process plant General Arrangement and 3-D model.

These were developed based on testwork results, experience on other similar projects and vendor recommendations. Figure 17-1 and Figure 17-2 present, respectively, the simplified PFD with mass balance for the major streams, and the general site water balance for Nominal operating conditions.

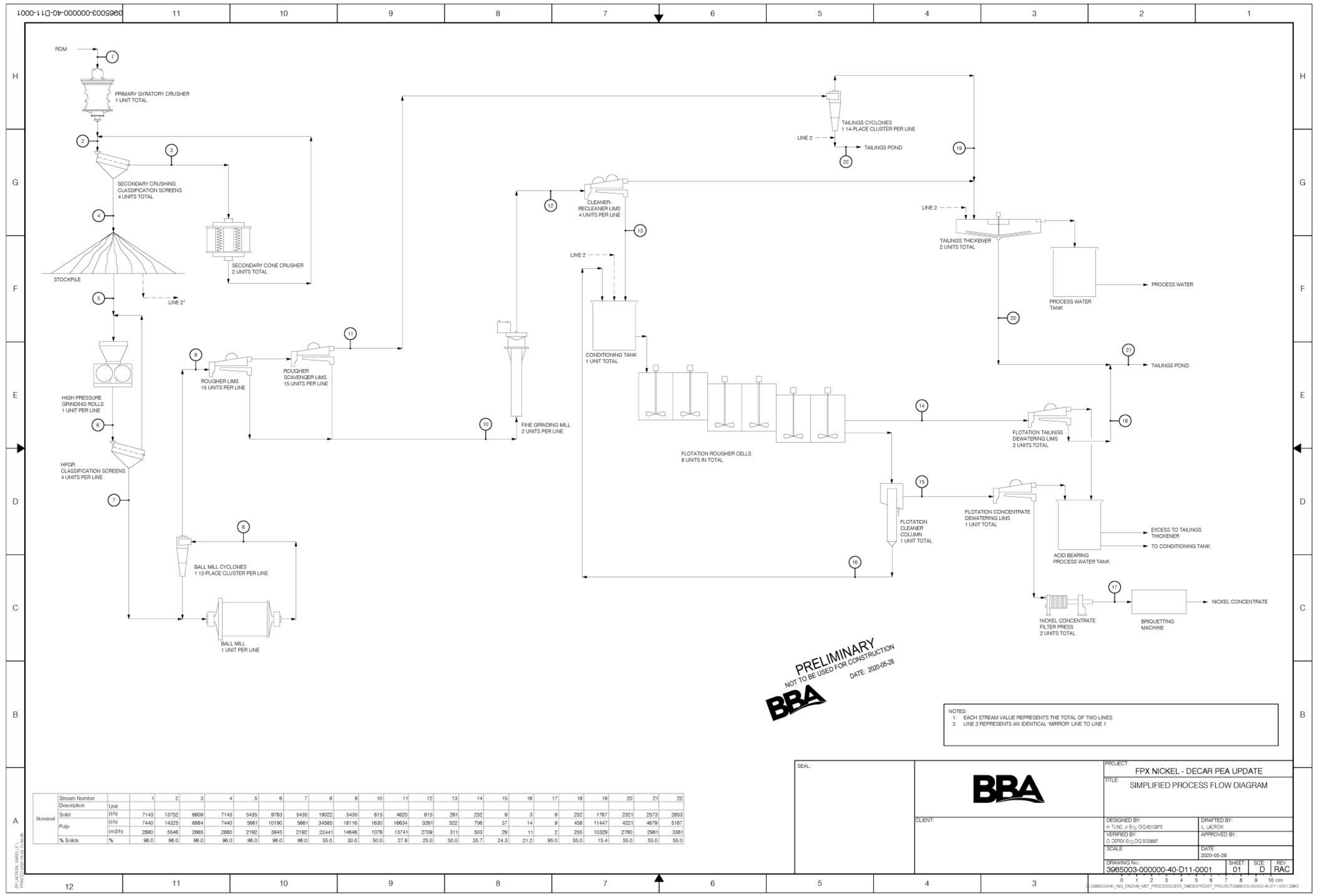


Figure 17-1: Simplified process flow diagram

	Stream Number		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
	Description	Unit																						
Nominal	Solid	(t/h)	7143	13752	6609	7143	5435	9783	5435	19022	5435	815	4620	815	261	252	9	3	9	252	1767	2321	2573	2853
	Pulp	(t/h)	7440	14325	6884	7440	5661	10190	5661	34585	18116	1630	16634	3261	522	706	37	14	9	458	11447	4221	4679	5187
		(m3/h)	2880	5546	2665	2880	2192	3945	2192	22441	14646	1078	13741	2709	311	503	29	11	2	255	10329	2760	2961	3381
	% Solids	%	96.0	96.0	96.0	96.0	96.0	96.0	96.0	55.0	30.0	50.0	27.8	25.0	50.0	35.7	24.3	21.2	95.0	55.0	15.4	55.0	55.0	55.0

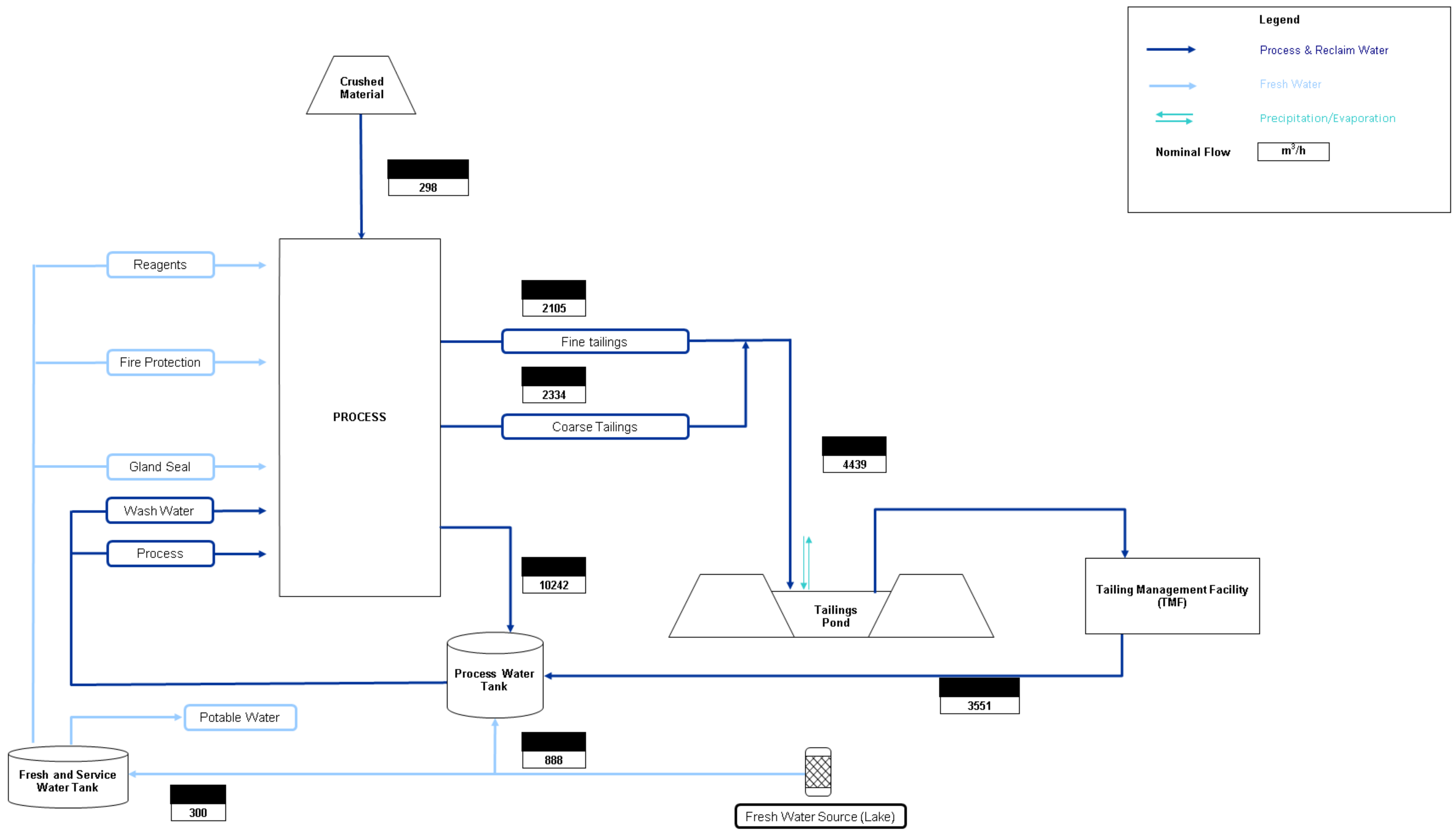


Figure 17-2: General site water balance for nominal, steady-state conditions

17.3 Conceptual Plant Design and Configuration

The general location of the mineral processing facilities is described and indicated on the site plot plan presented in Chapter 18 of this Report. The primary crushing station is placed in an area in the vicinity of the open pit so as to minimize mine truck hauling distances. Run-of-mine material for processing is crushed and stockpiled. Downstream of the crushed material stockpile, the major process equipment number and size allows for a two-line configuration, each capable of operating independently. As such, this proposed configuration offers the flexibility of operating the plant at full capacity, or at half capacity when a unit of major equipment is down for maintenance on one of the two lines. As will be described herewith in more detail, each processing line will have one primary grinding circuit, one primary magnetic separation circuit, one fine grinding circuit with secondary magnetic separation and, finally, a common flotation circuit. The tailings produced by each line are cycloned to separate the coarse fraction from the fine fraction. The fine fraction is further dewatered with thickeners. The coarse tailings and the fine tailings are pumped separately to the TSF, as required by the tailings deposition plan.

17.4 Crushing, Screening, Conveying and Stockpiling

17.4.1 Primary Crushing

Material from the mine will be delivered by haul trucks to a single, 6.2 m x 10.3 m (60” X 110”), gyratory crusher equipped with a 1500 kW motor. Crushing and conveying utilization have been assumed at 70%. Material is crushed to a P_{80} (80% passing) of 226 mm and is conveyed to the secondary crusher screening system.

17.4.2 Secondary Crushing and Screening

Secondary crushing is performed using two, 1,865 kW, cone crushers in closed circuit with screens. Screening is performed using four double-deck, 4.25 m X 8.5 m (14’ x 28’), classification screens. Two parallel conveyors transport the product of the primary crusher and the oversize of two classification screens to one secondary crusher.

The screen undersize is conveyed to the stockpile by two parallel conveyors. The P_{80} of the circuit (screen undersize) is approximately 38 mm. Metal detectors and removal devices will be used in the crushing circuit to sense and extract tramp metallic objects and decrease the risk of substantial damage to downstream equipment. With this configuration, the crushing circuit can still operate at, at least, half capacity should one of the secondary crushers be down for maintenance.

17.4.3 Stockpile

The secondary crusher screen undersize conveyors transport the crushed material to an uncovered stockpile that has a live capacity of 12 hours, and a total capacity of 48 hours, of concentrator operation. Crushed material from the stockpile is reclaimed by a series of apron feeders onto two parallel conveyors running in two tunnels underneath the stockpile. Each conveyor feeds one HPGR line.

17.5 High Pressure Grinding Rolls (HPGR)

Each of the two HPGR units has a tyre size of 3.0 m in diameter by 2.0 m in width with an installed power of 7,500 kW. Each HPGR discharges into a bin and the material is conveyed to eight double deck dry vibrating classification screens (four per HPGR line). Each screening unit has an effective screening area of 4.27 m by 8.53 m (14' width x 28' length). The screen oversize from each HPGR line is conveyed back to the HPGR units for further size reduction. The screen undersize, with a P_{80} of approximately 2.0 mm, is conveyed to two parallel Ball Mill grinding lines. Appropriate surge capacity will be provided ahead of the HPGR units. Dust will be controlled along the dry circuit by dust collectors, exhaust fans and water sprays.

17.6 Primary Grinding Circuit

The HPGR product is conveyed by two parallel conveyors each feeding one, 8.5 m dia by 12.8 m long, Ball Mill. Each of the two Ball Mills is of gearless drive design (wrap-around) with an installed power of 22 MW. The ball mills are operated in closed circuit with hydrocyclones to obtain the target P_{80} grind size of 300 µm. Each Ball Mill discharges into its dedicated pump box equipped with pumps, each feeding one 1.2 m diameter, 12-place, hydrocyclone cluster. The hydrocyclone U/F is recirculated back to the ball mill while the O/F is fed to the primary magnetic separation circuit. This configuration applies to Phase 1 of the Project. The modifications for Phase 2 will be discussed later in this Chapter.

17.7 Primary Magnetic Separation

The overflow of each of the two hydrocyclone clusters is collected in its own dedicated pump box, where the %-solids are adjusted with dilution water before the slurry is pumped to distributors feeding the Low Intensity Magnetic Separators (LIMS) circuit. Each Ball Mill hydrocyclone cluster feeds a LIMS circuit comprised of fifteen rougher LIMS, 1.2m dia. X 3.75m width. The tailings from the rougher LIMS flow by gravity to an equal number of rougher-scavenger LIMS to ensure minimal DTR nickel losses in the primary magnetic circuit. The concentrate from both the rougher and rougher/scavenger LIMS flows to the fine grinding feed pump boxes.

The rougher-scavenger LIMS tailings are collected in pump boxes and are subsequently directed to hydrocyclones for dewatering and size classification.

17.8 Fine Grinding Circuit

The fine grinding circuit consists of four, 5,500 KW, vertical stirred mills in open circuit with a resulting P_{80} of 25 µm. Each processing line has two mills. The fine grinding circuit is configured as an open circuit. The magnetic concentrate from the rougher and rougher / scavenger LIMS circuit is pumped to the bottom-fed, fine grinding mills after being diluted to the targeted solids density. The discharge from the fine grinding mills is then directed to the secondary magnetic separation circuit.

17.9 Secondary Magnetic Separation

The ground product from the fine grinding mills is fed to a total of eight cleaner-recleaner double-drum, 1.2m dia. X 3.75m width, LIMS (four per processing line) after being diluted to the required solids density. The magnetic concentrate is directed to the awaruite flotation circuit. Tailings from the LIMS units are collected and pumped to the thickeners.

17.10 Awaruite Flotation Circuit

The finisher LIMS concentrate streams from Line 1 and Line 2 are combined in the awaruite flotation conditioning tank at the head of the flotation circuit. The conditioning process in the tank involves the addition of an activator and a pH modifier. High-shear agitation is provided to this conditioning tank in order to maximize the reagent adsorption by particles. The retention time in the conditioning tank is approximately 5 minutes.

The conditioned slurry is pumped to awaruite flotation rougher cells, where reagents, such as the activator, pH modifier, collector, and frother are added at various points within the flotation circuit. The roughing stage in the flotation circuit is comprised of six 70 m^3 flotation cells which allow approximately 30 minutes of residence time. The tailings stream from the roughing stage is pumped to the dewatering system.

The rougher flotation concentrate is sent to one awaruite flotation cleaner column for further upgrading. The cleaner column diameter and height are 2.4 m and 7 m, respectively (providing a volume of 32 m^3). The flotation column tailings are recirculated back to the rougher conditioning tank and the concentrate is pumped to concentrate dewatering.

The air supply to the flotation cells and column is provided by dedicated blowers.

17.11 Concentrate Dewatering, Agglomeration and Handling

The first step of concentrate dewatering consists of a single, 1.2m dia. X 0.61m width, dewatering LIMS. This product is then directed to two filter presses working in parallel, each having the capacity for filtering all of the concentrate produced. The targeted moisture content for the filter cake is 5%. The operation of every filter press is a batch process and involves feeding, pressing, washing, air drying and cake discharge.

The filter cake discharged from the filters is conveyed to the briquetting machine to convert the fine concentrate to its final briquetted form. For this PEA, it is assumed that briquetting is performed without the use of binders, but this needs to be confirmed with testwork. The briquetting operation includes a screening step to ensure that fines are minimized in the final product and are recycled back to the head of briquetting. The final briquetted product is conveyed onto a pad where it is subsequently loaded into containers using a front-end loader. The containers are trucked to the Fort St. James rail terminal for ultimate transport to market. The details of concentrate handling and logistics are described in Chapter 18 of this Report.

17.12 Coarse Tailings Management

For each processing line, coarse, non-magnetic tailings stream generated by the rougher/scavenger LIMS is pumped to one 800 mm diameter, 14-place, tailings hydrocyclone cluster with a cut-size of 75 µm. The overflow from the cyclone is directed to the tailings thickener. The underflow is directed to a coarse tailings pump box for pumping to the TSF. These coarse tailings serve for dike construction. This is described in more detail in Chapter 18 of this Report.

17.13 Fine Tailings Management

The fine tailings are generated from three process streams: the coarse tailings hydrocyclone overflow, the secondary magnetic separation dewatering LIMS and the flotation tailings. The hydrocyclone overflow from the two processing lines is discharged to two 55 m diameter high-rate tailings thickeners (one thickener per line). The non-magnetic fraction from the secondary magnetic separation stage is also directed to the thickeners. Flocculant is added to the thickeners to promote the settling of fine solids and to preserve overflow clarity.

The underflow from the thickeners is pumped to the fine tailings pump box, while the overflow (clarified water) is directed by gravity to the process water storage tank.

The flotation tailings are dewatered via two, 1.2m dia. X 3.75m width, dewatering LIMS. The stream is dewatered in order to recover acidified process water which can then be recycled to the conditioning tank at the head of the flotation circuit in order to reduce the sulphuric acid consumption. The dewatered flotation tailings are combined with the tailings thickener underflow in the fine tailings pump box and is pumped to the tailings pond. This is described in more detail in Chapter 18 of this Report.

17.14 Project Phase 2

The project is initially implemented based on a primary grind size of 300 µm, driven by the need to generate coarse sand tailings for dike construction for tailings impoundment. When the initial TSF runs out of storage capacity at the end of Year-21, the mine plan allows for tailings to be disposed of in an exhausted portion of the open pit mine. For this type of deposition, coarse tailings are no longer required, and a finer primary grind can be implemented in order to benefit from improved DTR Ni recoveries. For this PEA Study, it is assumed that, starting in Year-22, a third 22 MW Ball Mill will be added in a new building extension, in order to achieve a primary grind of 170 µm. The testwork provides the estimated DTR Ni recovery for a 170 µm primary grind size.

For this PEA, no other process development work or design, other than a power calculation and estimated Ni production has been performed. The methodology for estimating capital and operating costs for Phase 2 is outlined in Chapter 21 of this Report.

17.15 General Plant Utilities and Services

The plant utilities and services have been specifically quantified based on the mineral processing plant requirements. The capital costs have been estimated accordingly. The general utilities and services for the mineral processing plant are described herein.

17.15.1 Water Management System

The water management strategy is discussed in detail in Chapter 18 of this Report. The strategy is based on process water requirements and maximizes water recycling to assure that no net effluent is discharged to the environment.

17.15.2 Reagents Storage and Handling

The reagents storage and mixing facilities are located within the concentrator building. The following reagents are required in the proposed flowsheet:

- Collector (SIPX);
- Frother (W31);
- Activator (copper sulphate);
- pH modifier (sulfuric acid); and
- Flocculant.

The flocculant is used to aid in the sedimentation of the fine tailings in the thickeners and the remaining reagents are used in the awaruite flotation circuit. The reagents storage and handling area will be equipped with dedicated sump pumps which deliver the product to collection locations in the event of spillage. Moreover, a separator wall will be installed to isolate this area from the main mill building, to protect personnel from hazards. In addition, the facility will have all the necessary safety equipment, including a fire-detection system, ventilation system, safety showers and eye-wash stations. Fresh water will be used for reagent preparation.

17.15.3 Compressed Air

Dry compressed air is required in the concentrate filtration area and flotation area, as well as for general use throughout the plant. Compressed air for use in the plant is supplied via a compressed-air distribution network. Two different qualities of air will be supplied to different consumers, plant air and instrument air. The air supply system includes the appropriate number of dryers and filters in order to supply the required quality of air. Compressed air, for the specific operation of the filter presses, is provided by dedicated compressors.

17.15.4 Electric Power

Power requirements, transmission and distribution are discussed in Chapter 18 of this Report.

17.15.5 Plant Instrumentation and Process Control

A Process Control System (PCS) will be implemented, along with the automated instrumentation to allow for the automatic control of the plant. A centralized control room will allow the operator to monitor and control all aspects of the process. Remote monitoring and/or control will also be possible with the appropriate network design. A cyber security strategy will also be implemented.

18. PROJECT INFRASTRUCTURE

This Chapter describes the major infrastructure required to support the Baptiste Project, as developed in this PEA. Infrastructure requirements are both on-site and off-site.

18.1 Off-Site Infrastructure and Plot Plan

Off-site infrastructure incorporates the following major elements:

- Access road;
- Rail terminal;
- High-voltage power line.

18.1.1 Access Road

The Decar Property project area is connected to the proposed rail terminal (described later in this section) and to the Town of Fort St. James by an existing road network consisting mainly of forestry service roads (FSR). In this PEA, BBA performed an analysis of the existing roadwork in order to determine its current condition and status so a capital cost related to upgrade and new road construction can be estimated. An assessment of road maintenance requirements was also performed so an operating cost can be estimated. This is documented in a BBA memo (BBA, April 27, 2020). Figure 18-1 presents the proposed access to the Property with the various road segments.

The existing road network is about 166 km in length. This distance can be shortened to about 121 km if a bridge is constructed to cross Middle River at a more southerly point than the existing crossing. For this PEA, it is assumed that the shorter route will be adopted. The capital and operating cost estimate related to the access road, presented in Chapter 21 of this Report, are based on the following:

- Of the 121 km road length connecting to the Property, 105 km are existing and require no upgrades by FPX;
- FPX would need to construct a new 4.5 km FSR segment with a 110 m bridge over Middle River, upgrade an existing 20 m bridge span and upgrade an existing 11.5 km FSR segment;
- Most of the road length is maintained by others and only 40 km require road maintenance and snow plowing by FPX.

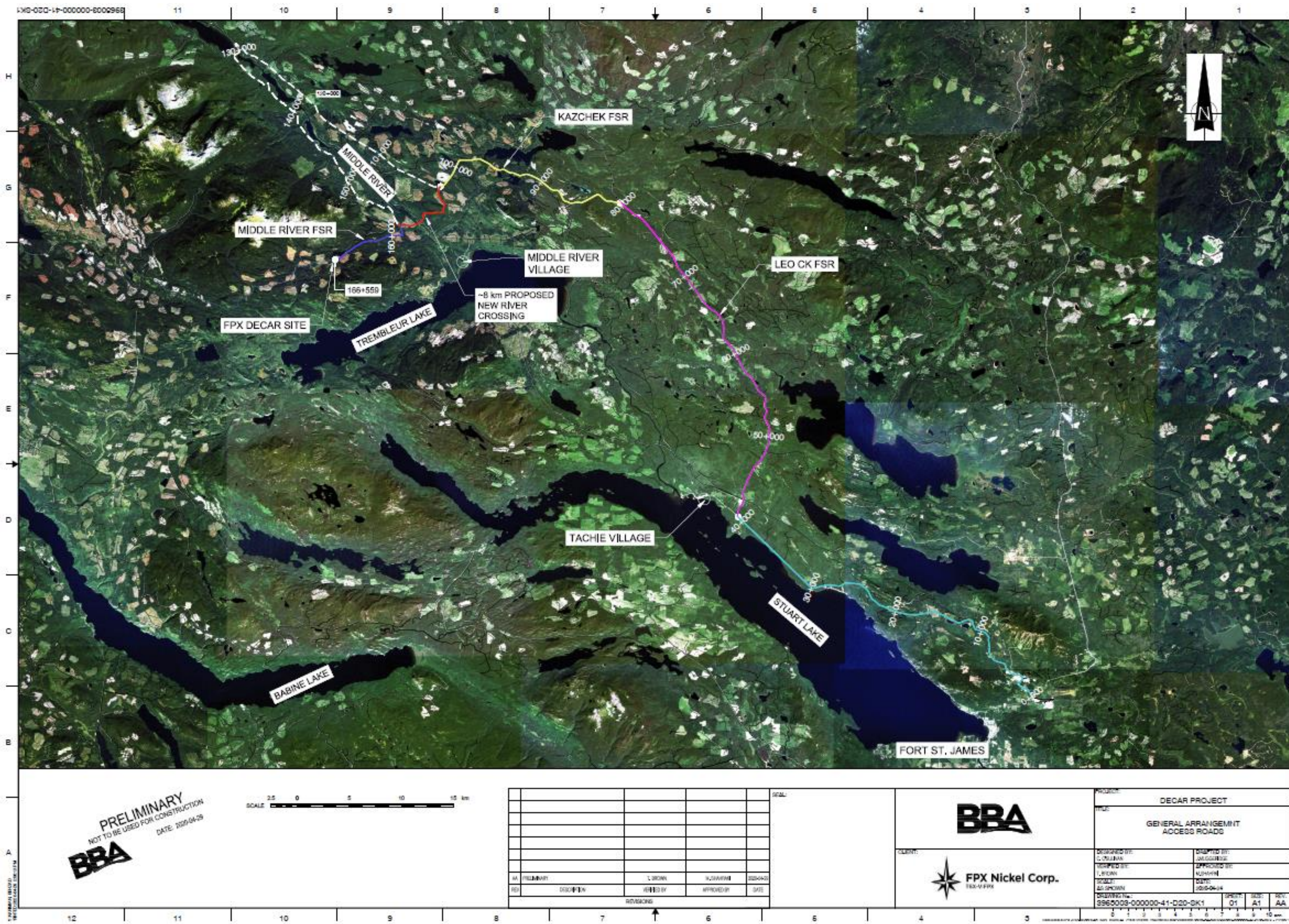


Figure 18-1: Access road from Fort St. James to the Decar Property

18.1.2 Fort St. James Rail Terminal

FPX will be required to construct a rail terminal at Fort St. James that connects to the CN main line. This infrastructure will serve primarily as a transfer point for shipping FeNi product to the Prince Rupert port terminal and, subsequently, to a destination port in Asia. The facility will also serve to receive and store rail tanker cars holding sulphuric acid that is subsequently trucked to the mine site. For this PEA, it is assumed that the rail terminal will be somewhere in the vicinity of the existing CN main rail line. As an exact location has not been defined, it is recommended that in the next study stage, FPX perform a site location study to evaluate options for possible sites.

The FeNi briquettes are loaded into containers at the mine site and trucked by FPX to the rail terminal. Rail transportation, container handling at the port terminal and maritime transport to the destination port will be provided by third party service providers, including fees related to use of containers. These services are included in operating costs as are costs for trucking from the mine site and operating and maintaining the Fort St. James rail terminal.

For this PEA, it is assumed that all other materials, fuel, reagents and consumables required at the mine site will be trucked directly and not handled at the rail terminal.

The rail terminal will consist of the following infrastructure:

- A cleared and fenced area for conducting materials handling activities related to containers containing FeNi briquettes;
- A 500-metre rail spur for handling railcars for loading and unloading FeNi containers;
- A 300-metre rail spur for storing and unloading tanker railcars containing sulphuric acid;
- A 10,000-litre diesel fuel tank for haul trucks;
- An office and facility trailer for personnel;
- Water, power and communications services.

The Project will, on average, produce about 72,000 tonnes of ferronickel concentrate briquettes annually which averages about 200 tonnes per day. For this PEA it is assumed that the product will be loaded into standard intermodal 20 ft containers, holding 24 tonnes of product. The product will be hauled from the mine to the rail terminal with tractor / trailers configured as B-trains. As such each load will comprise of two trailers, each with one container, for a total of 48 tonnes per load. This implies that four loads per day (eight containers) leave the mine. A front-end loader will be used to load briquettes into the containers which will be handled by 30-tonne capacity lift trucks; one at the mine site and one at the rail terminal.

The containers are unloaded at the rail terminal and laid in the stockyard or directly into the railcars. Empty containers are hauled back to the mine site. Railcar loading configuration will be based on type of railcars available. The rail spurs provided will have the capacity to handle more than one week of containerized briquette production.

It is estimated that about 39,000 tonnes per year (107 tonnes per day) of sulfuric acid will be required at the mine for mineral processing. Standard tanker railcars have a capacity of 90 tonnes. For this PEA it is assumed that the rail terminal will hold two weeks of storage, or 17 railcars. Sulphuric acid will be transferred to tanker trucks for daily delivery to the mine site based on requirements. The tanker storage area will have appropriate emergency spill containment.

18.1.3 High Voltage Power Line

Based on the analysis performed in this PEA, a power transmission line with a capacity of 120 MW will be required for the life of the operation. Power requirements are driven mainly by crushing and grinding operations, as defined by the process flowsheet presented in Chapter 17 of this report.

In this PEA, power connection options were evaluated and documented in a technical memorandum (BBA, April 23, 2020). At the required load, the lowest transmission voltage that can be utilized for the Project is a single, 230 kV circuit. For this PEA, the capital cost estimate has been based on the shortest technically plausible tie-in point located at a distance of approximately 98 km from the Property. The powerline is assumed to be constructed using a wood pole H-frame structure.

18.2 General Decar Site Plot Plan

The location of major site infrastructure was conceptually defined based on site topographical features and in consideration of the location of the Baptiste open pit, the designated TSF and the waste pile locations. It was assumed that the areas selected for site infrastructure do not have any economic underlying mineralization. A general plot plan is presented in Figure 18-2. The design is based on the following:

- The ultimate life-of-mine open pit footprint for the Baptiste Deposit, as developed by Stantec and described in Chapter 16 of this Report;
- The waste dumps, as described in Chapter 16 of this Report, have been designed to store waste materials generated during the mine life;
- The initial TSF, serving for Years 1 to 21 of the mine life, as designed by Stantec and described later in this Chapter;
- The mine services area pad, which includes the mine garage, truck wash, warehouse and mine employee facilities;
- The explosives plant pad is in an isolated area to the east of the open pit;
- The primary crusher building and pad, is in a designated area in the vicinity of the initial open pit ramp exit;

- The secondary crushing and screening, crushed material stockpile and HPGR and screening buildings, all connected by a conveyor system, are located within a cleared corridor (incorporating an access road, power line and other services) connecting these facilities to the concentrator area;
- The main process plant area pad includes the concentrator building housing all mineral processing equipment and FeNi briquette storage and the thickeners as well as the main 230 kV electrical substation;
- The fresh water pumphouse is located at Trembleur Lake, about 7 km to the south of the concentrator;
- Mine road network dedicated to heavy traffic and controlled for other vehicles;
- Site road network for general infrastructure access and access to the FSR;
- The camp for employees and administration building pad is located to the south-east of the concentrator, on the FSR road accessing the Property;
- Excluding the open pit, waste dumps and TSF, the estimated area for infrastructure pads is 210,000 m^2.

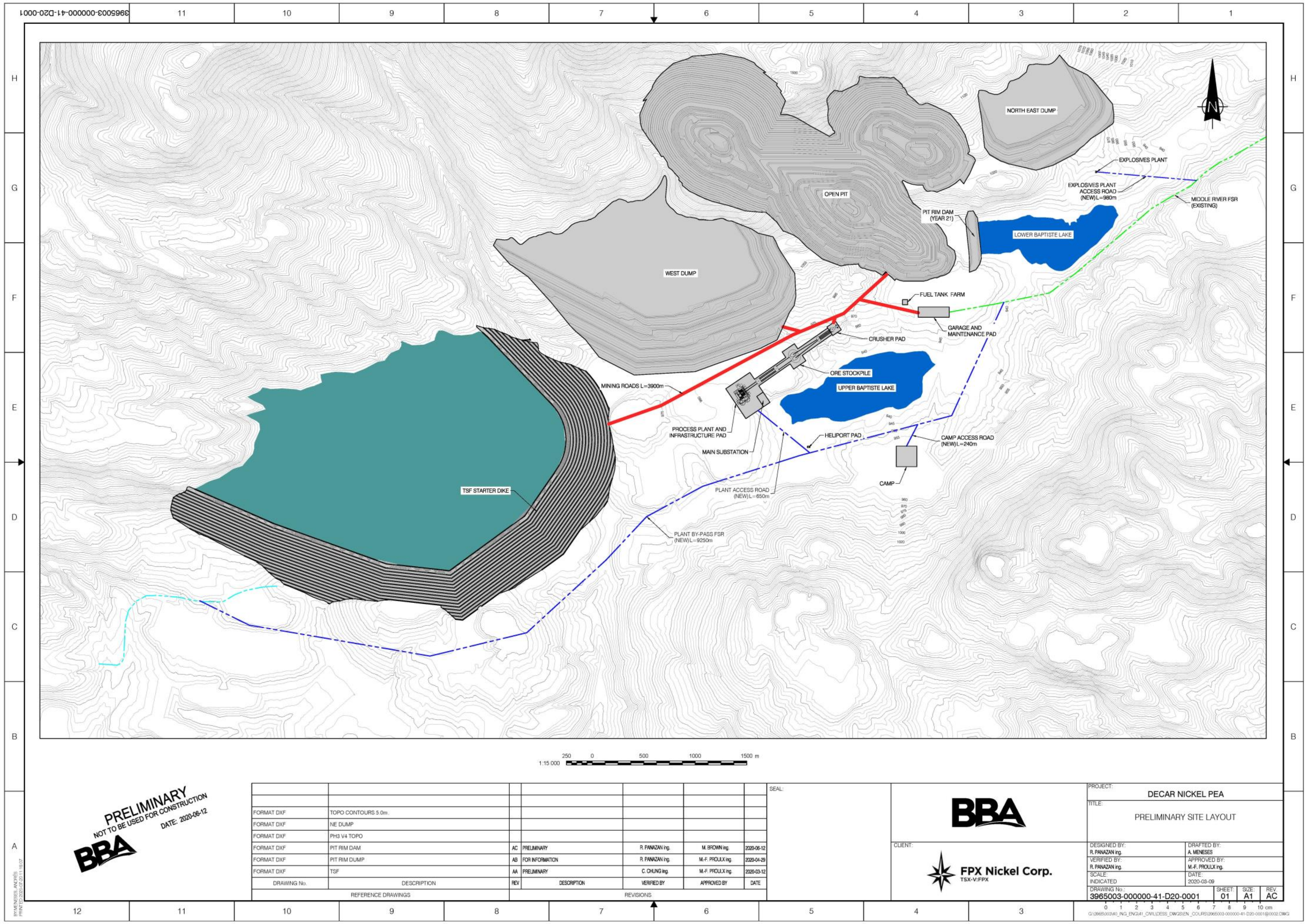


Figure 18-2: General site plan

18.3 Mine Maintenance Facilities and Other Mine Infrastructure

The mine equipment maintenance facilities are located within a dedicated pad. The main features of the facilities include the following:

- Initially, four bays for mining and auxiliary equipment maintenance will be provided (with two bays added in Year 6 and two more bays added in Year 17);
- A vehicle wash bay including an oil/water separation system allowing for recycling of water;
- A welding bay;
- Warehouse;
- Office space for 30 people with conference room;
- Lunchroom for mine personnel, and other employee facilities;
- Parking areas for service equipment, light vehicles and separate area for mine heavy equipment;
- Enclosed parking and maintenance areas for mine rescue and emergency response vehicles;
- Explosives pad (150 m^2) provided in an appropriate location.

On-site fuel storage is based on about 10-day storage capacity. A tank farm is provided for fuel storage with an appropriate fueling station. The fuel storage reservoirs will be contained within a bermed area and designed to meet applicable regulations. Diesel equipment operating inside the open pit will be serviced by tanker truck. The following storage capacity is assumed:

- Diesel storage capacity for 500,000 litres is provided, including a "Robo-Fuel" fueling device;
- Gasoline for small tools and equipment, all-terrain vehicles and snowmobiles will be stored in a 50,000-litre reservoir servicing the whole site, including a "Gas-Boy" for standard vehicles.

18.4 Mineral Processing Infrastructure

Mineral processing infrastructure includes pads and buildings housing the crushing and screening operations, HPGR and screening, grinding and concentrating, tailings dewatering and pumping and FeNi briquette handling. These are all located within a corridor pictured in Figure 18-3. The concentrator building houses all major processing equipment including the ball mills, primary magnetic separation LIMS, regrinding vertical stirred mills, final magnetic concentration LIMS and flotation, as described in the process flowsheet in Chapter 17 of this Report. It is assumed that the concentrator building will also house ancillary facilities such as reagents area, E-rooms, maintenance shop, offices and concentrator employee facilities and the laboratory.

In Year 22, a building extension for the additional ball mill will be added adjacent to the initial concentrator building. Figure 18-4 provides a conceptual, general plan view of major equipment layout.

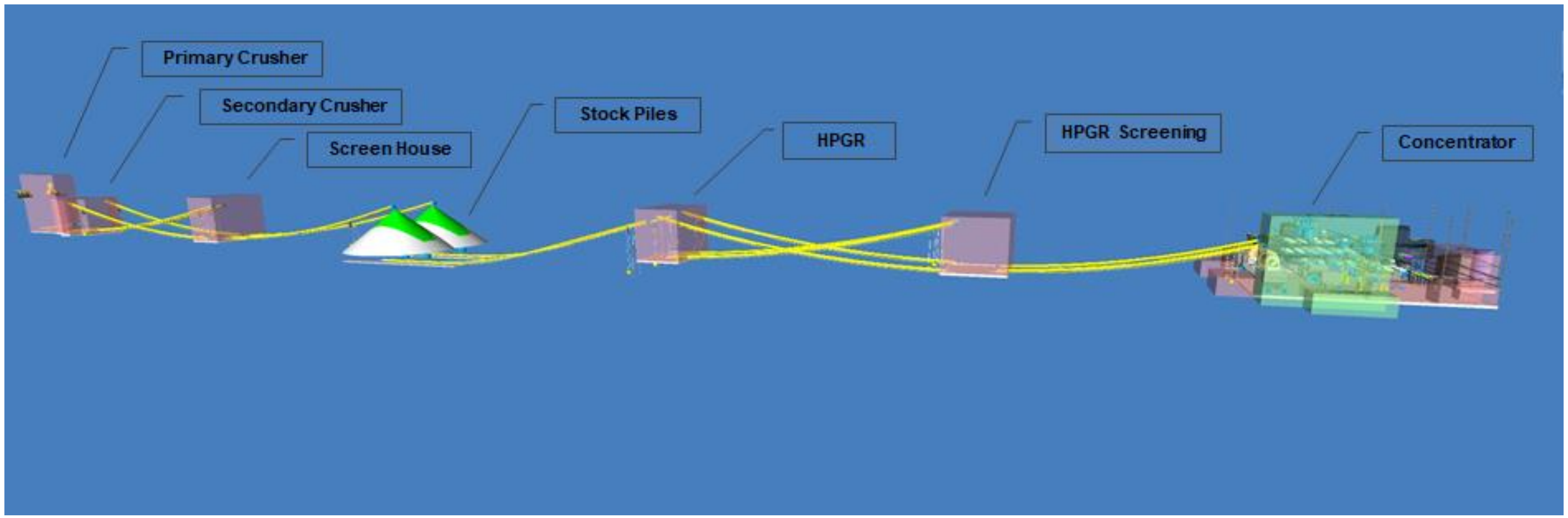


Figure 18-3: Conceptual layout of mineral processing areas

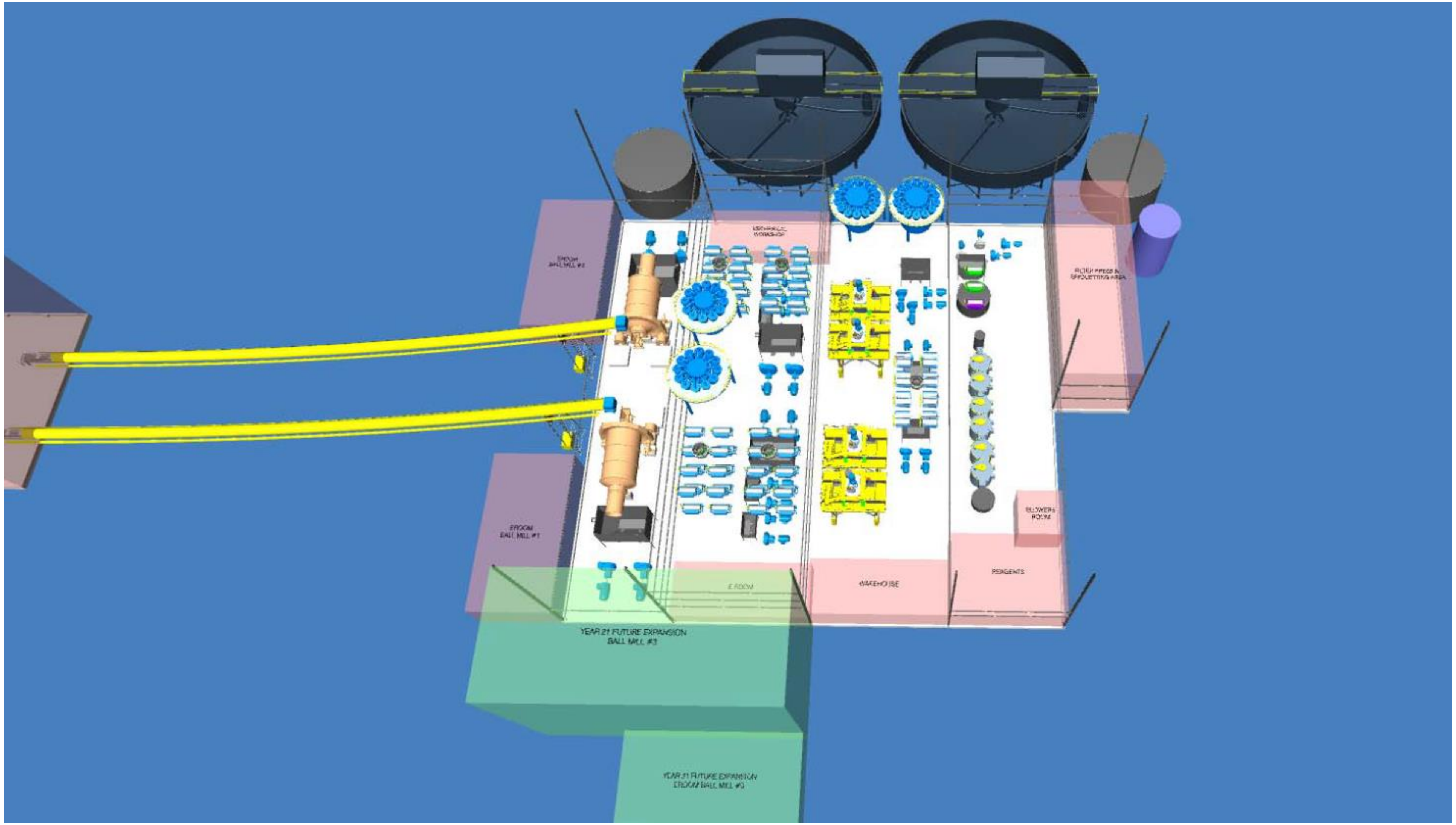


Figure 18-4: Conceptual plan view of concentrator building general arrangement

18.5 Tailings Pumping

Coarse and fine tailings are pumped separately from the concentrator to the TSF. The pipelines for each stream extend over a 7 km distance from the concentrator to the furthest deposition point in the TSF. The initial installation requires a two-stage pumping system for each of the tailings streams. The pumps are located in the concentrator building. In Year 3, a booster station is required in order to pump both tailings streams to the required dike crest elevation. This will be housed in a separate building. In Year 9, additional pumping capacity will be required in the booster station pumphouse to provide sufficient pumping power to reach the ultimate pumping elevation that will be reached in Year 21.

Starting in Year 22, the coarse and fine tailings streams will be pumped to the exhausted open pit for in-pit tailings deposition. This will require a new pipeline for each tailings stream, 4 km in length. The initial pumping system in the concentrator will be sufficient to pump the tailings to the end of the mine life in Year 35 and the booster station for pumping to the TSF will no longer be required.

18.6 Tailings Storage Facility

The tailings disposal strategy for the Baptiste Project is based on two distinct methodologies:

- For Phase 1 of the Project spanning from Year 1 to Year 21, tailings are disposed of within an external tailings storage facility (TSF);
- For Phase 2 of the Project, starting in Year 22 and ending in Year 35, tailings are disposed of within the exhausted open pit based on an in-pit disposal strategy.

These described herewith.

18.6.1 External TSF

The proposed external TSF will be constructed using the centerline construction method with a downstream slope of 3H:1V. The overall height of the tailings dam is approximately 170 m. The external TSF will be constructed primarily with cycloned sand produced in a mineral processing plant. Approximately 162.5 Mm^3 of cycloned tailings sand is required for the tailings dam construction. The TSF was designed to retain 440 Mm^3 of tailings. This is sufficient capacity for the first 21 years of production based on the mine schedule described in Chapter 16 of this Report.

18.6.1.1 Starter Dike Construction

During pre-production, a starter dyke with an initial 18-month storage capacity will be constructed using overburden and waste rock from the open pit area. It is assumed that 80% of the overburden and 90% of waste rock is suitable for the starter dyke construction. Testing of these materials to determine their suitability as construction fill should be carried out as part of future site investigations.

Depending on the availability/volumes of low permeability overburden, the overburden would be used to construct an upstream seepage barrier. An internal drain system to control seepage would be constructed as part of the starter dam. A liner system is not currently contemplated but may be required if a suitable volume of low permeability fill is not available.

The starter dyke is divided into east and west cells. The two separate cells allow the plant to be operational earlier when compared to the traditional single cell design. This is because tailings can be placed in one cell while the dyke of the second cell is being completed. A total of 14 Mt of waste rock and 37 Mt of overburden are required to construct the starter dyke in Year 2 and Year 1. The starter dyke configuration had a 2H:1V upstream slope and a 3H:1V downstream slope.

There is the potential to decrease starter dyke borrow if cyclone sand construction can be accelerated to assist in early development of the eastern tailings cell. In addition, sourcing of local borrow within the TSF pond footprint may be a cost-effective option or provide a source of specialty fills (ex. sand/gravel).

18.6.1.2 Cycloned Sand Cell Construction

The plan assumes the use of a starter dam constructed from overburden and pit-run waste material followed by a series of dam raises using the compacted sand-size tailings stream. The tailings dam raises would follow as centerline construction methodology with hydrocyclones being used to size and partially dewater the tailings stream into a coarse (sand) and fine stream, as described in Chapter 17 of this Report. The design assumes that the sand-sized tailings (size gradation 2 mm – 75 µm) will drain sufficiently to allow compacted lifts to be constructed with typical construction methods using mechanical equipment to place and compact the sand.

The fine tailings stream will be a slurry which is impounded within the main tailings dam. The tailings beach will be managed such that the pond is maintained against the back of the impoundment against natural ground. In addition to the tailings process water, incident precipitation and run-off will accumulate in the tailings impoundment. It is expected that the reclaim water intake would be located near the abutment of the main dam and projected out into the pond area to allow for a minimum depth of water for pumping. A floating barge or decant tower system could be utilized however each option would require periodic relocation as the overall tailings dam height increases.

For the current level of design evaluation, the fines content (-75 µm) of the sand stream planned for dam raise construction should be less than 15%. Stantec completed simple hydrocyclone simulations for key tailings streams to develop an initial understanding of what proportion of tailings stream might be useable to tailings dam cell construction. This simulation assumed the whole tailings steam was cycloned in a single stage. Multistage processing, or different treatment of multiple tailings streams from the plant processes, were not considered but could lead to material improvement. Stantec recommended the tailings stream parameters shown in

Table 18-1 be considered when developing the process flowsheet in order to allow for enough quantities of well-graded sand size material for TSF shell construction. Figure 18-5 shows a schematic cross-section of the tailings dam construction.

Table 18-1: Sand-size tailings stream recommended parameters

Design Parameter	Value (Units)	Comments
Maximum fines content	15%	Percentage of particles <75 µm
Size gradation for sand-size stream	+ 75 µm to 2 mm	Based on hypothetical simulations as no testwork has been completed to date
Coefficient of uniformity (Cu)	≥6	$Cu = D_{60} / D_{10}$ for sand-size stream
Coefficient of curvature (Cz)	1 ≤ Cz ≤ 3	$Cz = (D_{30}^2) / (D_{60} * D_{10})$
Sand content of overall tailings stream	>50%	Variable depending on distribution of particle sizes

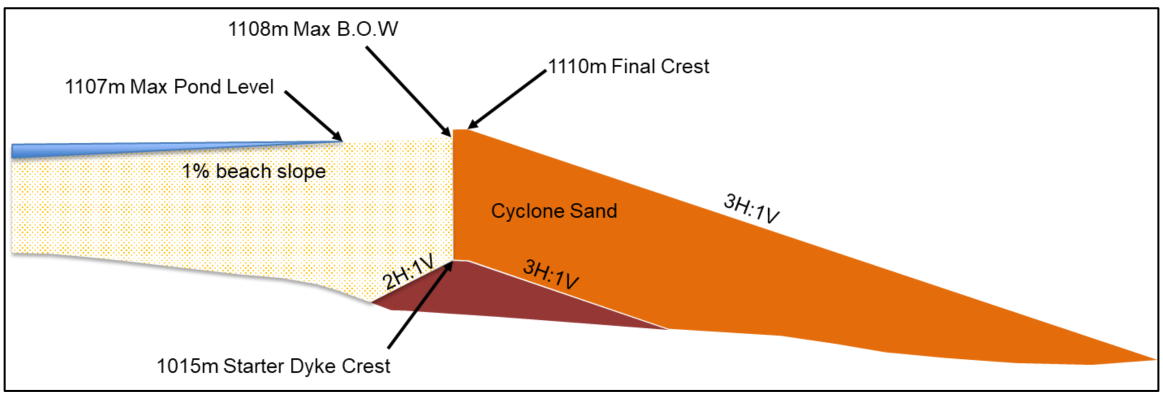


Figure 18-5: Schematic cross-section of tailings dam including starter dike

18.6.1.3 Geotechnical Assumptions

There is currently no foundation information available in the external TSF area so the proposed 3H:1V downstream slope may require revision (steeper or shallower) once foundation information becomes available. A terrain stability assessment was completed in 2018 (Polar Geoscience Ltd. Feb. 2018) based on air photo data in order to identify any critical instability features. The study identified till as the most common surficial material present within the tailings study area. The thickest and most extensive morainal till deposits are present in the south end of the study area that underlies most of the tailings facility footprint. The terrain stability assessment noted that glaciolacustrine sediments were not mapped within the study area, although there is a possibility that these soils were deposited and may be buried beneath till and glaciofluvial sediments. Foundation investigations would be required to determine if this is the case.

For this PEA, it was assumed that the primary grind of 300 µm and hydrocycloning will generate enough sand tailings with the right characteristics for sand cell construction. Some testing was performed in 2017 which included grain size, specific gravity, relative density, standard Proctor and direct shear on a selected sample blend. The tailings, as tested, had similar properties to some blends of copper tailings including SG and grain size distribution. Therefore, data available for copper tailings blends related to consolidation and placed density has been used to support the TSF design. Table 18-2 summarizes the test data results.

Table 18-2: Test data results

Test	Property	Value
Standard Proctor	Optimal moisture content	11%
	Maximum dry density	1,913 kg/m^3
Specific gravity	S.G. at 20ºC	2.62
Relative density	Maximum	1,674 kg/m^3
	Minimum	1,291 kg/m^3
Direct shear	Cohesion	0 kPa
	Friction Angle	34 degrees

18.6.1.4 Hydrological Design Parameters

The hydrological estimates are based on regional data, estimated catchment area and site-specific topography. The current TSF storage capacity has an allowance for a 1 in 200-year storm; however, the volume hasn't been tied to an overall water balance and schedule so the storage allowance will need to be verified and reviewed at the PFS stage. The proposed TSF design includes a costing of emergency spillway to handle flood events larger than the estimated 1 in 200-year volume. The tailings water was deemed suitable for discharge under emergency conditions.

The dam was sized to maintain 3 m of freeboard above the operating pond level for all phases of its operational life.

18.6.2 In-Pit Tailings Deposition

Tailings produced in excess of the external TSF's capacity will be backfilled into the mined-out pits. Upon completion of mining of the Phase 1 pit in year 21, the pit will then start being backfilled with tailings produced while processing material mined in the Phase 2 pit, starting in Year 22. The Phase 1 pit has a storage capacity of 220 million m^3 of tailings, which is more than sufficient to retain tailings produced while mining the Phase 2 pit. A highwall will be left in place at elevation 940 m between the mining Phase 1 and Phase 2 pits to act as an in-situ dam. Characterization of the rock mass that will make up this in-situ dam will be a critical item for site investigation to support future permitting.

Tailings produced during Phase 3 of the mine plan will be backfilled into the Phase 2 pit once this pit is mined out in Year 31. The pit rim dam located to the south east of the Phase 1 pit will increase tailings capacity in the Phase 1 and 2 pits to allow all of Phase 3 tailings to be stored within the pit footprint. The pit rim dam is a significant structure and was designed to be 50 m wide along the crest, 17 m high, with 2H:1V side slopes. Currently, there is no geotechnical information for the dam foundation in this area. Approximately 800,000 m^3 of waste rock and overburden (placed volume) is required for rim dam construction. The pit rim dam would be a zoned construction with overburden used to construct a lower permeability upstream blanket. Given the annual mine production, the required rock fill volume is relatively small and sufficient material will be present even if the dam slopes are shallowed. This structure will need to be in place by the end of Year 25 so that the crest can be used to access the Phase 3 pit for pre-production stripping. It is assumed that suitable rock and overburden will be available from the pit or in stockpiles to facilitate construction.

Water would be available from the external TSF while initial in-pit placement occurs as consolidation and incidental precipitation and run-off accumulation would likely maintain the pond at least in the near-term (months-years). The volume of water within the mined-out Phase 1 pond at the start of tailings placement would be dependent on the timing of cessation of mining as well as rates of groundwater inflow. For example, if mining in Phase 1 ceases during the winter, the spring freshet could lead to substantial volumes of water accumulating in the pit while only a small volume of tailings and associated water has actually been deposited.

18.7 Water Management

18.7.1 Recycled Water From the TSF

Based on Stantec's preliminary analysis, it is assumed in this PEA that 80% of the water pumped with the tailings is available for recycling back to the process water tank at the concentrator. As there may be periods during the year where more water may be available (from snow melts and precipitation), the pumping system will be sized to be able to provide 100% of the water required, according to the nominal process water balance presented in Chapter 17 of this Report. Design pumping capacity is estimated at 4,500 m^3/h. The pumphouse will be mounted on a floating barge. The pumping distance is estimated at 4 km.

Starting in Year 22, when in-pit tailings deposition will be implemented, water will be returned from the pit. It is assumed that a new barge-mounted pumphouse will be required as well as a 4 km pipeline for returning water to the process water tank. The system at the external TSF will be maintained operational and provide water as available and as required. This allows additional flexibility for water management.

18.7.2 Fresh and Make-up Water

Fresh water / make-up water will be pumped from Trembleur Lake, which is located approximately 7 km south of the concentrator. A corridor comprising of a maintenance road, the pipeline and power line will connect the pumphouse to the concentrator. Fresh water is normally required for gland seal, cooling, fire protection and potable water, distributed from a central main water tank. Make-up water is required in the process water tank in order to comply with the process water balance mainly when water availability from the TSF is reduced. This may be seasonal, when freezing may reduce available water or for periods of low precipitation.

For this PEA, the lake pumphouse is assumed to have a capacity to pump 40% of the process plant requirement in addition to the required service water. Based on the nominal site water balance presented in Chapter 17 of this Report, design pumping capacity is estimated at 1700 m^3/h. The pumphouse will be constructed on the shore of the lake. A corridor from the concentrator to the pump house will include a service vehicle road, a pipeline and electric power line.

18.7.3 Surface Water Management

Infrastructure pads, roads and corridors will incorporate proper ditching, water collection systems and sedimentation ponds to ensure water quality for streams directed to the local watersheds.

Approximately 5 km of major diversion ditches will be required for the TSF and a further 9 km will be required for a clean water diversion above the pit and the two external dump areas. Approximately 16 km of smaller ditches will also be required to manage surface water run-off across the pit, waste dump and TSF areas. Both major and minor ditches will require an access road to be built alongside the ditch for maintenance and inspections.

It is anticipated that several temporary ponds will need to be constructed to control surface water during the removal of overburden and waste rock in pre-production years. An allowance of three temporary ponds has been made for this time period.

Permanent ponds will be required for the collection of both non-contact and contact water from major and minor ditches. Over the LOM, 6 major ponds, and 6 minor ponds will be required.

As the TSF increases in height, the toe will move further out and eventually encroach upon an existing creek. This will require a major channel diversion which has been assumed to be built-in Year 15. A more detailed tailings plan will be required to determine the actual timing of this diversion.

Most of the permanent water management structures will need to be in place by the end of Year 1, with the majority requiring construction within the first pre-production year. Additional ponds will be constructed in Year 17 and Year 25 to support expansions of the pit and dumps.

18.8 Site-wide General Infrastructure

18.8.1 Mine Roads

Mine road requirements and design are described in Chapter 16 of this Report. These roads provide access to the open pit, waste dumps, the TSF and the mine services facility. The TSF is accessible to light traffic with controlled and escorted access.

18.8.2 Site Roads

Currently, an FSR crosses the Property and allows access to the west of the Property. This road is used by third parties and access must be maintained. Considering that the existing road will no longer be available as the open pit and TSF run through it, it must be relocated and by-pass the Property in order to provide continued access to third party users. This by-pass FSR segment, as previously shown in Figure 18-2, is estimated to be 9.3 km in length. A cost estimate is provided for the construction of this road segment. Other site roads include the following:

- The mine garage and maintenance, is accessed through the existing FSR;
- The explosives plant is accessed from existing FSR through a new 1-km road segment;
- The mineral processing areas, including the concentrator and the corridor running from the concentrator to the crusher, are accessed from the south from the by-pass FSR and through a new 650-m road segment;
- The employee camp and administrative area is accessed from the by-pass FSR through a new 240-m road segment;
- Access to the fresh water pumphouse at Trembleur Lake was not developed and assumed to require a 7-km road and corridor.

18.8.3 Warehouses and Storage

A heated and insulated structural steel building (25 m by 50 m) will be incorporated in the concentrator area. A cold storage warehouse (25 m by 50 m), of foldaway type will also be provided.

18.8.4 Camp Accommodations and Offices

The employee camp is located on the FSR road to the south of the mining complex. The initial camp will have 340 rooms on two floors. Fifty rooms will be added in Year 17 to accommodate the mine plan requirements. The camp will comprise of single-occupancy bedrooms with a shared shower and toilet for two rooms. Common areas include lounges, recreational areas, a fitness area, a kitchen and lunchrooms. Administration offices are integrated in the facility.

18.8.5 Emergency Response

A facility for the emergency response team (ERT) is provided. It consists of a vehicle garage for ambulance, fire truck and rescue truck as well as an adjoining medical center and storage space. This facility can be part of the mine services infrastructure. A helipad is included as part of the on-site infrastructure.

18.8.6 Site Communications

A communication infrastructure will be installed on the site. The following communication systems are included:

- Telephone network;
- Internet access;
- Computer network;
- Automation network (for instrumentation/control);
- Surface radio system;
- Cable television network (camps only).

18.8.7 Sewage and Waste Management

Sanitary sewage is collected in pits at the mine garage and concentrator where it is subsequently syphoned by vacuum truck and transported to a central treatment system located in the vicinity of the camp site.

Solid waste is disposed of with a batch incinerator within an enclosed building.

18.8.8 Explosives Plant

A dedicated, fenced area will be prepared and available to the explosives contractor to store and prepare explosives for the mining operation. Electric power will be provided to the explosives plant battery limit.

18.9 Electric Power

18.9.1 Power Requirements

The estimated power demand and annual power consumption serving to estimate operating costs related to electric power consumption are presented in Table 18-3 for Phase 1 and Phase 2 of the Project. During Phase 1, additional tailings pumping capacity will be required in Yr 3 and Yr 9. This adds about 3.5 MW to the indicated power demand and 28 GWh to the annual electric power consumption. It should be noted that the indicated power requirements do not consider electric space heating for the buildings. A trade-off study should be performed in the next project phase to determine the best option for building heating.

Table 18-3: Estimated electric power requirements

Category	Phase 1 (Yr 1 – Yr 21)		Phase 2 (Yr 22 – Yr 35)	
	Power Demand (MW)	Consumption (GWh)	Power Demand (MW)	Consumption (GWh)
Mineral processing	93	733	103	813
Site infrastructure and services	14	115	15	117
Total	**107**	**848**	**118**	**930**

18.9.2 Main Electrical Substation and Site Distribution

The main electrical 230 kV, 120 MW substation will be located in proximity of the concentrator. Electric power to the complex will be provided by BC Hydro through a new high-voltage power line described earlier in this Chapter. The substation will contain the required stepdown transformers, switchgear and electrical equipment to distribute power to the various areas of the mine site at the appropriate voltage.

18.9.3 Emergency Power

Four emergency diesel gensets are provided; one for the camp, one for the mine garage and two for the concentrator.

19. MARKET STUDIES AND CONTRACTS

19.1 Introduction

Metallurgical testwork, described in Chapter 13 of this Report, has shown that the Baptiste Project can produce a clean, high-grade, ferronickel (FeNi) concentrate through a conventional mineral processing flowsheet. The concentrate will be agglomerated in briquette form, as described in Chapter 17 of this Report. The principal mineral recovered in the concentrate is awaruite, which is a naturally occurring alloy of nickel and iron (Ni_3Fe). For this PEA, the intended application for the FeNi briquette is as a nickel additive for stainless steel producers.

FeNi is a nickel-iron alloy that is almost exclusively used as a raw material in stainless steelmaking. Unlike pure nickel metal products (e.g. cathode), which are graded as Class I by the London Metal Exchange (LME), FeNi is a Class II nickel product which, by definition, means that it contains less than 99.8% Ni. The FeNi briquette, like typical FeNi products, is not LME deliverable. Consequently, FeNi has no terminal market and is sold directly to stainless steel melt shops.

The price to be obtained from the sale of the FeNi briquette to stainless steel melt shops will generally be a function of two variables: 1) the LME nickel price; and 2) a discount or premium to the LME nickel price, based on the market positioning of the FeNi briquette in relation to competing sources of nickel feedstock to stainless producers, being primarily stainless steel scrap, nickel pig iron (NPI), standard FeNi and Class 1 Ni.

This Chapter describes how the selling price for the FeNi briquette was derived as the basis for the Baptiste Project Economic Analysis presented in Chapter 22 of this Report.

19.1.1 Alternative Marketing Routes for Baptiste FeNi Product

The Economic Analysis of this PEA assumes that 100% of the Baptiste FeNi briquette will be sold directly to stainless steel producers over the entire life of the Project, in the form of a high Ni grade briquetted product.

In addition to the stainless steel market, there are two potential alternative markets for the sale of the FeNi concentrate:

1) Standard Ferronickel Producers: There may be an opportunity to sell Baptiste FeNi concentrates to standard FeNi smelters, which could use the concentrate to upgrade the Ni content of their feedstock; the sale of FeNi concentrates to FeNi smelters is not considered in the Economic Analysis of this PEA.
2) Electric Vehicle Battery Producers: FPX has conducted a baseline assessment to explore the amenability for the FeNi concentrate to subsequent acid leaching to produce a high purity nickel-cobalt feedstock for subsequent production of nickel sulphate and cobalt sulphate for the electric vehicle (EV) battery market. While this market alternative is not considered in the Economic Analysis of this PEA, a brief discussion of this opportunity is presented in Chapter 24 'Other Relevant Information' of this Report.

19.1.2 Iron Concentrate By-product

In addition to the primary FeNi product, the Baptiste Project also generates an iron-rich by-product that can potentially be sold as an iron ore concentrate on a standalone basis. For this PEA, the sale of this iron ore by-product is not considered in the Economic Analysis. However, a discussion regarding this opportunity is presented in Chapter 24 'Other Relevant Information', of this Report.

19.2 Characteristics of the Baptiste Nickel Product

The FeNi concentrate produced by the Baptiste Project is in the form of a fine powder in the range of 25 µm in particle size. The Baptiste process flowsheet has incorporated a briquetting step to agglomerate the fine concentrate into 20 mm x 15 mm briquettes. The projected product specification for the briquettes is presented in Table 19-1.

Table 19-1: Projected product specification for the Baptiste FeNi briquetted concentrate

Projected Product Specification	
Ni	60% - 65%
Fe (total)	30% - 32%
Awaruite (Ni_3Fe metallic alloy)	77% - 83%
Metallic Fe in Awaruite	19% - 21%
Magnetite (Fe_3O_4)	13% - 18%
Co	1% typical
Cu	0.7% typical
P	0.02% typical
S	0.6% typical
MgO	1% typical
SiO_2	1.5% typical
Cr_2O_3	0.4% typical

In contrast to the Baptiste FeNi briquette, which is produced via a mineral processing route, standard FeNi and NPI are produced by pyrometallurgical route and are highly metallized and refined products with a carbon content up to 5%. FeNi and NPI are typically produced from nickel laterite ores. Nickel content of such ores is usually in the range of 0.8% Ni to 3% Ni. The predominant production route for ferronickel is the pyrometallurgical RKEF (rotary kiln – electric furnace) process. The RKEF process typically yields a product grading between 15% Ni and 30% Ni. Laterite ores can also be processed to produce NPI. This blast furnace route can produce an NPI grading between 5% Ni and 10% Ni. The submerged arc furnace (SAF) route can produce high-grade NPI grading 10% Ni to 15% Ni. Table 19-2 presents a comparison of the nickel content for the various nickel products discussed.

Table 19-2: Nickel content comparison of various nickel products

Element	Baptiste FeNi	Class 1 Ni	FeNi	High-Grade NPI	Low-Grade NPI
% Ni	60% - 65%	≥99.8%	15% - 40%	10% - 15%	5% - 10%
% Fe (metallic)	19% - 21%	NIL	balance	balance	balance
% Other	14% - 21%*	NIL	1% - 4%	5% - 10%	7% - 13%

*Consists mainly of magnetite Fe_3O_4 and other oxides.

As shown in Table 19-2, the Baptiste FeNi briquette has some unique characteristics compared to other products. Its most distinguishable feature is its high Ni content. Compared to standard FeNi, significantly less material needs to be added to the steel melt to achieve the same nickel addition. Compared to high-grade NPI, four to five times less material is required per Ni unit and compared to low grade NPI, six to twelve times less material is required per Ni unit.

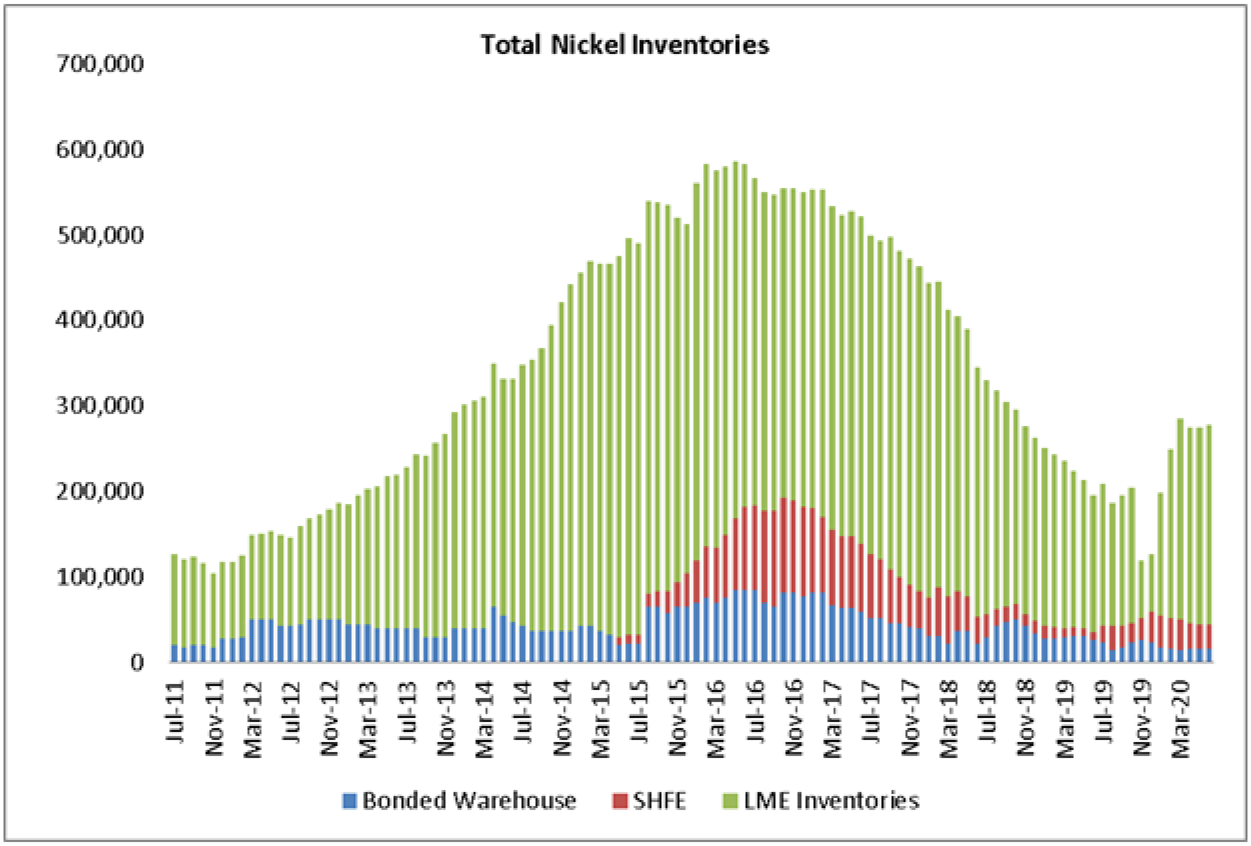
The Baptiste FeNi briquette has relatively high purity, considering that this is a product of a mineral separation process rather than a pyrometallurgical process. After Awaruite (Ni_3Fe), the next most abundant component in the Baptiste FeNi briquette is iron oxide in the form of magnetite; the Baptiste FeNi briquette does not contain any carbon. The specification for the Baptiste FeNi product suggests that it would be directly useable in the stainless steel smelting process without having to go through an intermediary smelting or refining process. The Baptiste FeNi briquette could, thus, be used to substitute or complement other forms of nickel in stainless steel smelters.

19.3 Global Nickel Supply and Demand

According to Scotiabank, global refined nickel production in 2019 was 2.41 Mt. Approximately 60% of current nickel production originates from sulphide ores and 40% from laterites.

The stainless steel industry accounts for approximately 70% of the current global nickel production. A further 24% of current global nickel production is consumed for plating, alloys and castings, with approximately 5% used in the production of batteries.

According to Scotiabank, the global refined nickel market posted a cumulative supply deficit of approximately 240,000 tonnes from 2016 to 2019, leading to a sustained decline in global nickel inventories. This is indicated in Figure 19-1.



Source: Scotiabank

Figure 19-1: Historical data of nickel inventories

According to Wood Mackenzie, global nickel demand is expected to grow to over 3 million tonnes by 2030, with demand growth to be underpinned by expanded consumption of nickel in EV batteries and stainless steel. Wood Mackenzie further estimates that the nickel market will require the addition of over 525,000 tonnes of annualized production capacity from new projects over the period to 2032, in order to meet the projected growth in demand. Wood Mackenzie further estimates that the nickel market will require the addition of over 1.5 million of annualized production capacity from new projects over the period to 2040, in order to meet the projected growth in demand.

Nickel has two classifications: Class 1 nickel has a purity of >99.8% and Class 2 nickel has a purity of <99.8%. Class 1, being of high purity, is generally used in the production of high-quality stainless steel, nickel alloys and batteries. Class 2, which includes FeNi and the lower-grade variant NPI, is almost exclusively used in the production of stainless steel. It is primarily produced in a value chain originating with laterite (oxide) ores.

Stainless steel producers have three sources of nickel for their furnaces: recycled stainless steel scrap, Ni/Fe alloy (standard FeNi and NPI) and Class 1 nickel. The availability of these three primary nickel feedstocks is highly regional and subject to local and global supply-demand dynamics at a given time. As such, the use and pricing of scrap, FeNi, NPI and Class 1 nickel are interrelated and interdependent.

19.4 LME Nickel Base Price

FPX provided long term projected Ni price data published by several reputable analysts. The most current update of this data is dated August 2020. A long-term nickel price assumption of $17,070 per tonne ($7.75 per pound) is assumed in this PEA Study which is consistent with the average long-term nickel price of forecast given by six base metals analysts, as seen in Table 19-3.

Table 19-3: Projected analyst LME benchmark price

	Projected Price	
	$/lb	$/tonne
Analyst 1	$8.00	$17,621
Analyst 2	$8.00	$17,621
Analyst 3	$7.50	$16,520
Analyst 4	$8.00	$17,621
Analyst 5	$7.00	$15,419
Analyst 6	$8.00	$17,621
Average	**$7.75**	**$17,070**

19.5 Product Payability Analysis

In order to assess the potential payability (stated as a % of the LME base price outlined in Section 19.4), BBA has considered the following sources of information:

- The results of the FPX's preliminary product market testing undertaken with stainless steel and ferronickel producers;
- Preliminary market feedback based on informal discussions with nickel consumers and traders, including an independent consultant to FPX and representatives of large international trading houses specializing in nickel products;
- Benchmarking with typical specifications for standard FeNi and NPI products from various producers;
- The author's technical knowledge of the steelmaking process based on his over 20 years of experience in a steel smelter;
- Historic premium / discount data for standard FeNi.

The Baptiste process flowsheet incorporates a briquetting operation to agglomerate the fine concentrate into 20mm X 15mm briquettes. This sizing will ensure that no excess fines are shipped to customers and avoids any potential payability discount related to excessive fine material.

The product specification for the Baptiste FeNi briquettes was presented in Table 19-1. The key secondary elements of concern for nickel-bearing feedstocks like FeNi and NPI are: C, Si, S, P, Cr, Cu and Co. These, and other elements that are specific to the Baptiste FeNi product and their assumed impact on payability for this PEA Study, are elaborated on in Table 19-4.

Table 19-4: Baptiste FeNi secondary elements and impact on payability assumed in this PEA

Element	Content (%)	Technical observations	Payability considerations
Ni	60 – 65	Ni predominantly present as Awaruite (Ni_3Fe)	Higher nickel content may have a value to stainless steelmaking in maintaining the iron balance in the bath
Fe	30 - 32	Metallic Fe associated with Awaruite (Ni_3Fe) Oxide Fe in the form of magnetite (Fe_3O_4)	A premium can be expected for the metallic portion of the iron which may be reduced if carbon additions are required to reduce the magnetite component
Co	1	Not expected to be an issue as it benchmarks at equivalent or below standard FeNi specifications	No discount or premium assumed.
P	0.02	Not expected to be an issue as it benchmarks at equivalent or below standard FeNi specifications	No discount or premium assumed.
Cr_2O_3	0.4	Not expected to be an issue as it benchmarks at equivalent or below standard FeNi specifications	No discount or premium assumed.
Cu	0.7	Content is relatively high compared to typical standard FeNi but the amount of Cu that can be tolerated will depend on scrap mix and specification of steel grade produced	No discount or premium assumed.
MgO	1	Relatively small content, MgO is normally added as dolomitic lime in EAF	No discount or premium assumed.
SiO_2	1.5	Relatively small content; will report to the EAF slag. Si is present in standard FeNi and NPI	No discount or premium assumed.
S	0.6	Content is relatively high compared to standard FeNi and NPI and can impact the steelmaking process	No discount or premium assumed.

The impact of the various secondary elements in the Baptiste FeNi product should be evaluated in context of the addition of FeNi made to a steel batch as well as the elemental specifications of the grade of stainless steel produced. As previously mentioned, the higher nickel grade Baptiste FeNi briquetted product will result in less material addition per nickel unit, thus proportionally reducing the impact of the secondary elements. This is especially pertinent to copper and sulphur. Nickel additions from FeNi will also depend on the nickel content as well as the sulphur and copper in the scrap charge to the EAF. The more the nickel coming from the scrap, the less the FeNi required. As such, as previously mentioned, the amount of scrap utilization is specific to each stainless steel smelter and dependent on regional dynamics which influence stainless steel scrap price and availability.

This preceding analysis was performed to assess the Baptiste FeNi briquette applicability from a stainless steel smelter operations point of view. Based on this analysis, it is reasonable to assume that the Baptiste product will attract payability factors in line with standard FeNi. Table 19-5 presents historical premium/discount data for standard FeNi.

Table 19-5: Historical premium / discount data for standard FeNi

Year	Average LME Price ($/tonne)	Average Realized Price ($/tonne)	Premium / Discount (%)
2014	$16,865	$16,116	-4%
2015	$11,817	$10,979	-7%
2016	$9,612	$9,502	-1%
2017	$10,406	$10,494	+1%
2018	$13,118	$12,963	-1%
2019	$13,933	$13,757	-1%
Average			-2%

Source: Anglo American annual reports

Given the relative novelty of the Baptiste FeNi briquette, the Project may have to prove its capabilities before the product is fully accepted as a viable alternative or complement to current stainless steel producers. It is recommended that FPX develop a strategic marketing plan to promote the product, including the production of additional samples for marketing tests with nickel traders and consumers. For the Economic Analysis of this PEA, no consideration or payability discount was applied to take into account market acceptance of the novel Baptiste FeNi briquette.

For this PEA it is assumed that the Baptiste FeNi is sold into the Asian market. It should be noted that the cost of transport from the mine site to a port in Asia is included in the operating costs of the product; therefore, no adjustment to payability has been made to account for product transportation. Transportation costs are discussed in Chapter 21 of this Report.

19.6 Selling Price used for Economic Evaluation

As mentioned previously, the LME price and payability factor (expressed as a discount or a premium) for FeNi products is dependent on many factors driven mainly by the stainless steel scrap market and the availability of standard FeNi and NPI products on a local and global basis at any given time. Historically, nickel values have fluctuated at both the LME base price level but also in terms of the discount or premium paid for FeNi and NPI products. These fluctuations arise as a result of supply/demand imbalances and geographic variation in feedstock availability. Based on the average projected analyst LME benchmark nickel price and the payability analysis presented earlier in this chapter, Table 19-6 provides the basis for the selling price assumed in the Economic Analysis of this PEA presented in Chapter 22 of this Report.

Table 19-6: Selling price for PEA Economic Evaluation (based on 63.4% Ni contained)

	$/t Contained Ni
LME price based on analyst average	$17,085
Payability (discount)	2%
Selling price used in PEA Economic Analysis	**$16,743**
Payability (% of LME price)	98%

19.7 Contracts

FPX does not have any contracts, agreements or any commercial engagements with regards to the sale of the Baptiste product or to the development of the Project.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Baseline Studies

Preliminary environmental baseline studies (EBSs) were conducted in 2011 and 2012 (KCB 2012a; 2012b). EBSs were continued and expanded in 2012 using Allnorth Consultants Ltd. (Allnorth 2012 1-9). The EBS components initiated in 2011/2012 are summarized in Table 20-1.

Table 20-1: EBS Components

EBS Component	Scope of Work
Climate	▪ A meteorological monitoring station has been installed (August 2012) and is equipped with sensors for wind direction and speed, precipitation (tipping bucket), temperature, relative humidity, snow depth and water equivalent (ultrasonic), and evaporation.
Hydrology and Surface Water Quality	▪ The Project is located in the Stuart-Takla watershed and encompasses three sub-watersheds associated with Baptiste Creek, Sidney Creek and Van Decar Creek. Van Decar Creek and Baptiste Creek flow into Middle River approximately 18 km and 5 km, respectively, northwest of Trembleur Lake; Sidney Creek flows south from the western side of Mt. Sidney Williams, directly into Trembleur Lake. The outflow from Trembleur Lake flows southeast into Stuart Lake, which is the lowest catchment of the Stuart-Takla watershed. Baptiste Creek is the main watercourse draining the Baptiste target. Baptiste Creek receives water from Upper and Lower Baptiste Lakes, two small headwater lakes and their tributaries, one of which flows adjacent to the exploration camp (Camp Creek), and a second, Nickel Creek, which drains Nickel Lake. ▪ Monthly water quality monitoring has been undertaken at 10 to 15 sites along the main watercourses and their relative tributaries since May 2011. ▪ Manual flow monitoring has been undertaken (irregularly) at the surface water monitoring stations. ▪ Three hydrometric stations, consisting of a Solonist pressure transducer and stream stage gauge, have been installed since October 2011. Continuous monitoring occurs every 10 minutes.
Hydrology and Groundwater Quality	▪ Eight hydrogeologic monitoring wells were installed in September 2012 and configured with vibrating wire piezometers (VWPs) or data loggers, as appropriate. ▪ Lugeon packer tests and/or falling head tests were undertaken in the hydrogeologic monitoring wells to determine hydraulic conductivity within the proposed pit area. ▪ Water quality and levels were collected from five non-vertical, open exploration drillholes in October 2011. ▪ Water quality and levels are regularly being collected from the hydrogeologic monitoring wells.

EBS Component	Scope of Work
Acid Rock Drainage (ARD)/Metal Leaching (ML)	▪ 218 samples were analyzed for a combination of static acid base accounting (ABA) parameters, metals, and shake flask extraction (SFE).
Vegetation	▪ Review of existing information from recognized government databases was completed. ▪ White bark pine surveys, including geo-referencing and flagging, were undertaken over the areas impacted by the 2011/2012 drill area. Detailed review of information relating to white bark pine was completed. ▪ Vegetation surveys over the Project footprint were undertaken to develop a database of rare and invasive plant information, plant tissue, and Terrestrial Ecosystem Mapping (TEM) ground trothing data.
Terrestrial Wildlife	▪ Review of existing information from recognized government databases was completed. ▪ Wildlife habitat mapping has been undertaken. ▪ Wildlife transects have been seasonally completed, as appropriate, to determine the presence of mammal, bird, and amphibian species. ▪ A winter caribou survey was undertaken and discussions with regulators regarding caribou monitoring and management commenced.
Aquatic Environment	▪ Fish population data and stream inventory (fish habitat) assessments were undertaken in Nickel Creek, Camp Creek, Baptiste Creek, Sidney Creek, Van Decar Creek, Upper/Lower Baptiste Lake and Nickel Lake. A limited number of opportunistic fish tissue samples have been collected and analyzed. ▪ Spawning assessments were undertaken in Baptiste Creek and Van Decar Creek. ▪ Sediment quality sampling was undertaken. ▪ Periphyton, phytoplankton, zooplankton, and benthic invertebrate sampling were undertaken.
Archaeology	▪ An archaeology overview assessment (AOA) and preliminary field reconnaissance (PFR) survey was conducted for the 2011/2012 drillhole locations.

20.1.1 Results and Identified Environmental Sensitivities

EBS results from preliminary studies determined the following:

- Geologic materials likely to be extracted from the ground during mining exhibit both low-sulphide concentrations and high-acid neutralizing capacity and are expected to be non-acid generating (NAG), except for one lithology (metasediments) of minor occurrence (Palich 2012);
- Metasediments were found to be enriched in boron, bismuth, and sulphur with potential enrichment of chromium, nickel, arsenic, and molybdenum. Bismuth and chromium appear to be present in mineral forms with a low leachability potential;

- All rock types were enriched in boron, although only the mineral form of boron found in the peridotite, overburden, and metasediments appears to be leachable;
- The overburden, peridotite, quartz monzonite, and mafic dykes exhibited some enrichment of chromium, nickel, and magnesium with limited leachability potential;
- Selenium may be enriched in all rock types and may be present in leachable mineral forms in the metasediments, although further assessment using lower multi-element analysis detection limits is required to confirm this preliminary assessment;
- Water quality in Baptiste Creek and its tributaries generally exhibited higher metal concentrations than Sidney and Van Decar Creeks, with more frequent exceedances in provincial and federal WQGs (Allnorth 2013);
- Federal (i.e., CCME) water quality guideline (WQG) exceedances were observed for aluminum at 10 of 15 sites;
- Chromium exceeded both provincial and federal WQG criteria at all established monitoring sites. Exceedances (either provincial or federal or both) were also observed at 9 of 15 sites for iron and 6 of 15 sites for copper and cadmium;
- Several sites that exhibited chromium and iron exceedances are distally located to exploration activities, so elevated concentrations are interpreted to be naturally occurring in the Project area;
- Although surface water quality results overall for the region contained low levels of suspended sediment, observed WQG exceedances for these metals may be linked to elevated total suspended solids (TSS) associated with seasonal changes in turbidity;
- Arsenic, cadmium, copper, lead, nickel, selenium, silver, vanadium and zinc exceedances of provincial and/or federal guidelines often occurred during the onset of freshet (April and May);
- Lugeon packer tests in the pit area showed hydraulic conductivity (K) within the pit area to be relatively consistent in the order of 10-9 m/s;
- Provincial and/or federal groundwater quality guideline exceedances have also been observed in the initial samples collected up to 2013. Total arsenic, cadmium, lead, iron, silver, vanadium, and zinc exceedances were observed in many samples, but do not have corresponding exceedances for dissolved metals. Exceedances may be positively correlated to observed high turbidity in the samples based on corresponding surface water quality observations during freshet;
- Dissolved chromium, aluminum, and selenium WQG exceedances are likely reflective of the natural groundwater conditions and will require periodic evaluation during ongoing surface and groundwater monitoring activities at control and impact sites as part of a comprehensive environmental management plan (EMP);

- Baptiste Creek and Van Decar Creek targets were assessed as having low archaeological potential;
- Other sites within the Project area have had limited examination in terms of archeological and cultural resources. Additional studies related to current and traditional use of local resources, which include biophysical resources important to Indigenous groups, will be assessed in ensuing Project phases.

Fish and wildlife species considered to be subject to environmental sensitivities were identified as known occurrences or are anticipated to be present in the Project area:

- Known occurrence and observed presence of Whitebark Pine, which is on the provincial Blue-List (CDC 2011, 2013) and is federally listed as Endangered (COSEWIC 2010). Scattered Whitebark Pine occurrences have been reported in the ESSFmvp biogeoclimatic ecosystem classification subzone variant. It is unknown whether observations near the Property constitute critical habitat under the federal Species at Risk Act (SARA);
- Known occurrence of the provincially protected Upper Fraser River Population of White Sturgeon, which is on the provincial Red-List (CDC 2018) and is federally listed as Endangered (COSEWIC 2012) under SARA. Local sturgeon presence is associated with the Trembleur Lake survey with precise locations unknown. The sturgeon fishery is considered valuable to local Indigenous groups; however, evidence provided has been anecdotal (CDC 2014);
- Known occurrence of the provincially protected Northern Mountain Population of Woodland Caribou, which is on the provincial Blue-List (CDC 2014) and is federally listed as a Special Concern (COSEWIC 2014). Local Woodland Caribou presence is associated with the Takla Lake survey (Poole et al. 2000; CDC 2013);
- Observed presence of other provincial Blue-List species within the Project area include Wolverine (CDC 2010), Grizzly Bear (CDC 2005), Mountain Goat (CDC 2015), Olive-sided Flycatcher (CDC 2010b), and Rusty Blackbird (CDC 2007);
- Observed presence of Bull Trout (Blue-List; CDC 2011, updated 2018) in nearby creeks;
- Observed presence of provincially secure (i.e., Yellow-List) wildlife species of local and ecological importance including Moose (CDC 1994, updated 2015), Clark's Nutcracker (CDC 2001, updated 2015) and Western Toad (CDC 2010c, updated 2016).

Any Project activities that result in disturbance or habitat destruction for sensitive species have the potential to result in elevated risk to the Project and will require a habitat management strategy and/or specific environmental protection plans (EPP) to mitigate any conservation risks. Based on the information available to date, it is not expected that the environmental sensitivities identified will be limiting to the Project development.

20.1.2 Future Work

Additional environmental baseline work remains to be completed for the Project so EMPs and EPPs can be prepared as part of the environmental assessment (EA) and regulatory permitting processes. This includes, but is not be limited to:

- Continuous local climate data acquisition;
- Air quality and greenhouse gas (GHG) monitoring;
- Hydrology and surface water quality monitoring;
- Hydrogeologic and groundwater quality monitoring;
- Sediment and soil monitoring;
- Kinetic acid rock drainage testing;
- Terrain stability assessments;
- Terrestrial ecosystem mapping (TEM);
- EA-specific fish and wildlife studies and critical habitat mapping;
- Visual and noise studies;
- Socio-economic studies;
- Current and traditional use studies;
- Traditional knowledge and cultural resources assessment;
- Site-wide archaeological overview assessment (AOA) and potentially archaeological impact assessment (AIA) as required by local Indigenous groups;
- Assessment and evaluation of biophysical social, cultural, and health valued component (VC) risks for the Project area, including the transmission corridor and other lateral infrastructure.

20.2 Site Management

20.2.1 Waste Management

Two waste dumps will be constructed, one to the west and one to the east of the open pit. The design of the waste piles is described in more detail in Chapter 16 of this Report. Suitable waste rock and overburden will also be used for the construction of the starter dykes for the initial tailings storage facility as well as for the dyke for the in-pit tailings containment area, as described in Chapter 18 of this Report. Small quantities of waste rock will also be used for the construction of site roads and pads.

All domestic waste generated over the course of the Project will be disposed of using acceptable practices, such as on-site incineration or through off-site disposal.

A central sanitary sewage system will be installed in the vicinity of the employee camp site. Sanitary sewage generated in the mine garage area and concentrator area will be collected and transported by vacuum truck to the central treatment system. Sanitary sewage is proposed to be treated using an above-grade, mechanical treatment system. Treated sanitary wastewater is expected to be discharged to the environment under permitted conditions with the opportunity to recover effluent as makeup water once the process plant comes online. Sludge generated as a by-product of sewage treatment is assumed to be trucked away by a licensed contractor for appropriate disposal.

Hazardous petrochemical wastes that will require collection and disposal include waste oil, filters, hydraulic oil, and glycol. Waste oil will be collected in above grade tanks within a lined containment area and may be used as a fuel source for the on-site incinerator, if deemed an acceptable management practice to regulators. All other petrochemical wastes will be stored on-site at the waste management facility and final disposal will be contracted to a qualified hazardous waste disposal service.

Hazardous liquid wastes such as laboratory chemical wastes and paints will be collected and stored on-site, with final disposal off-site by a licensed contractor. Other liquid wastes such as equipment and floor wash down water from both process and non-process areas will be collected and recycled as makeup water to the process plant, with excess reporting to the TSF for storage and ultimate use as makeup for the process plant. No untreated liquid wastes shall be released to the environment.

20.2.2 Tailings Disposal

The technical design and operational aspects related to the tailings storage facility (TSF) are presented in more detail in Chapter 18 of this Report. Based on the information available to date, the tailings generated are considered to be non-acid generating and of low metal leaching potential.

The proposed location for the initial, external TSF is to the southwest of the open pit and the process plant. Other potential locations were assessed at a high level within a 10 km radius from the open pit. The tailings dam is proposed to be designed using the 'centerline' construction method with a 3 m freeboard. An initial starter dike, 53 m in height, will be constructed using compacted mine waste rock and overburden. Cycloned tailings sand will be used to raise the dam to its final height of approximately 170 m at elevation 1,110 m. The storage capacity of the facility is 440 Mm^3 of tailings. This provides sufficient capacity for the first 21 years of mine operations. The total affected footprint is approximately 884 hectares.

Coarse sand tailings and fine tailings are pumped separately from the process plant. The tailings pipelines will be positioned around the perimeter of the TSF to allow for tailings deposition based on the deposition plan. The coarse tailings are placed and compacted using tracked equipment. An internal drainage system will be incorporated to manage seepage and internal pore pressure. Seepage will report to a collection pond located downstream of the TSF which will be monitored as part of acceptable ongoing monitoring and management requirements under the range of seasonal flows. Total suspended solids from surficial run-off will be settled out prior to discharge to the environment. Water from the tailing slurry will be collected within a confined area of the TSF and recycled back to the process plant.

Once the external TSF has reached its capacity, and starting in Year 22 of operations, tailings will be disposed of within the mined out early phase pits, as they become available. New pipelines for pumping tailings will be installed to deliver the tailings to the designated deposition points. This will provide enough storage capacity to support operations to the end of the current mine plan in Year 35 of operations. A pit-rim perimeter dam will be required and will be constructed in Year 25 of operations to provide sufficient capacity for the in-pit storage plus sufficient freeboard.

20.2.3 Water Management

Water management is based around current industry best management practices (BMP) and regulatory requirements:

- Control surface water in order to prevent potential contamination of clean or non-impacted water resources;
- Control and divert clean or non-impacted water away from mine affected areas including the open pit, waste dumps, haul roads and the TSF;
- Divert and/or control excess runoff and groundwater ingress that may interfere with mine operations;
- Control erosion of the site to limit sediment runoff that may impact receiving waters;
- Carry out progressive reclamation and revegetation to restore natural cover and reduce erosion.

During the operations phase of the Project, impacted water is defined as water that comes into contact with the processing mill, open pit, mineral and waste stockpiles and the other plant site facilities. Impacted water will be collected from the various sources and, as much as possible, be recycled by the process plant. Excess impacted water will be discharged to the environment under permitted conditions in order to meet acceptable provincial and federal regulatory standards.

Surface water runoff from catchments above the open pit, TSF, plant site, and waste rock stockpile areas will be diverted away from the disturbed lands and collected in diversion channels to minimize surplus water volume in the water management circuit. Diverted non-impacted water will be returned to natural drainage channels further downstream of the impact areas and subject to ongoing aquatic ecosystem management under provincial EMPs as well as federal environmental effects monitoring under the Metal and Diamond Mining Effluent Regulations (MDMER).

Permanent ponds will be required for the collection of both impacted and non-impacted water streams.

The process flowsheet is designed to dewater tailings at the plant, using hydrocyclones and thickeners, in order to increase water recycling to the process. As such, tailings slurries will be pumped at densities in the order of 50% solids to the TSF. It is estimated that about 80% of the water pumped to the TSF will be available for recycling back to the process plant. The remaining will be lost through evaporation or will remain with the deposited tailings.

Surface runoff from the plant site and other infrastructure pads will be routed to the sediment control pond prior to discharge to the environment.

Though much of the water required for processing will be provided via recovery from the TSF, fresh water makeup will be required. Freshwater supply will be considered for the Project from Trembleur Lake. It will also serve as the source of potable water, used where recycled wastewater does not meet required water quality standards, and retained as a top-up for firewater.

20.3 Environmental Assessment Process

20.3.1 British Columbia

In BC, reviewable mining projects must attain an EA certificate (EAC) prior to obtaining the required construction and operating permits. As a principal planning tool, reviewable projects are subject to the revitalized BC Environmental Assessment Act, 2018 currently in force using a phased approach with imposed regulatory timelines. The EA is generally structured as follows:

- Identification and assessment of potential environmental, social, economic, cultural, and health impacts;
- Development of an acceptable scope and methodology for conducting the effects assessment of a selection of valued components (VCs);
- Characterization of residual effects potential for VCs after avoidance, mitigation measures, standard BMPs, and monitoring programs are implemented;
- Prediction of the likelihood of significant residual effects occurring;
- Development of acceptable compensation measures to offset residual effects and maintain compliance with provincial and federal regulatory requirements as well as to effectively accommodate adversely affected Indigenous groups;
- Participation in Crown consultation proceedings to provide opportunities for Indigenous, federal, provincial, and local governments, stakeholders, special interest groups, and members of the public to learn about the Project, identify potential issues, provide input to potential avoidance/mitigation measures, and accommodate any infringement of Indigenous title and rights;
- Incorporate economic, social, cultural, health, and environmental factors into proponent and government decision making processes.

Key phases of the EA process in BC include:

- Early engagement phase (minimum of 90 days) occurring prior to submitting the detailed Project Description (s. 13 and s. 15 of the Act) and designed to attain consensus among participating Indigenous groups. It includes alternative dispute resolution options and leads to a Summary of Engagement and the Detailed Project Description.
- Remaining phases required to obtain an EA certificate (EAC) include:
  - EA Readiness phase and decision (s. 16(2), s. 17 or s. 18; 60 days minimum, but timeline is variable);
  - Process Planning phase (120 days);
  - Application Development and Review phase (minimum of 180 days) and submission of final application;
  - Effects Assessment and Recommendation phases (150 days maximum); and
  - Decision phase (30 days maximum).

The Project will be bound by the conditions of the EAC. Post-certificate activities include mitigation effectiveness reports and may include audits, certificate amendments, extensions, and transfers.

20.3.2 Government of Canada

The EA process also takes place under the Impact Assessment Act, 2019 for federally designated projects. Projects that are not designated federally may still require a screening in coordination with the provincial EA process.

Government agencies responsible for coordinating the EA processes include the BC Environmental Assessment Office (EAO) and the Impact Assessment Agency of Canada (IAAC).

The Project will likely require federal authorization by Fisheries and Oceans Canada (DFO) under paragraph 35(2)(b) of the Fisheries Act to commit harmful alteration, disruption, and destruction (HADD) of fish habitat or paragraph 34.4(2)(b) for any death of fish. An authorization will be issued by the Minister under the new paragraphs in the amended Act (2019). Habitat and/or productivity offsetting requirements must be acceptable to DFO and seek to accommodate participating Indigenous groups.

Additional EBS work, including the TEM and determination of the extent of fish habitat, wetlands, and presence listed species and ecosystems will provide information for federal regulators to consider in the EA process.

20.4 Permitting Process

20.4.1 Concurrent Permitting

If a project receives an EAC, the proponent must obtain the required authorizations for activities such as water use, timber cutting, access roads, stream crossings, and other mine-related permits. In general, provincial and federal EA processes are finalized before permits can be issued. In BC, the Concurrent Approval Regulation under the revitalized BC Environmental Assessment Act, 2018 allows for concurrent reviews of provincial permit applications and applies to provincial permits contained within the scope of the EA. Concurrent permitting does not apply to federal permits.

20.4.2 Joint Permits

The Mines Act and Health, Safety and Reclamation Code for Mines in British Columbia protect workers, the public, and the environment to minimize the mining-related health, safety and environmental risks. Whether or not the Project is deemed reviewable, mines must acquire authorization under the BC Mines Act/Environmental Management Act (MA/EMA) joint permitting process. A joint permit must be obtained with an appropriate reclamation security payable before construction and operational activities can commence.

Applications for the joint permit are submitted to:

- BC Ministry of Energy, Mines and Petroleum Resources (EMPR); and
- BC Ministry of Environment and Climate Change Strategy (ENV).

The Chief Inspector of Mines (or a delegated Inspector of Mines) is the statutory decision maker (SDM) for MA permits. Authorizations for mining under EMA are issued by ENV's Environmental Protection Division (EPD) mining authorizations team. An integrated MA/EMA joint permitting process includes detailed technical reviews of environmental, geotechnical, and geoscience information, as well as the development of EMPs and mine reclamation and closure plans. The ministries offer specific guidance on other required authorizations, including additional permits issued by the Ministry of Forests, Lands and Natural Resource Operations and Rural Development (FLNRORD).

20.4.3 Coordinated Authorizations

Coordinated authorizations are implemented through a project-specific regional Mine Development Review Committee (MDRC) to coordinate the multiple authorizations that may be required from the various agencies. Examples of authorizations for which FLNRORD is responsible include:

- Issuing *Land Act* tenure;

- Licensing or approving water use, water storage facilities, or diversion channels under the *Water Sustainability Act* and the BC Dam Safety Regulation;
- Authorizing access roads, utility corridors, and other improvements related to the mine operation on Crown land outside of mine areas under the *Land Act* or *Forest Act;*
- Issuing *Wildlife Act* authorizations;
- Administering the *Heritage Conservation Act* and associated authorizations; and
- Reviewing/authorizing any cutting or spoiling of Crown trees under the *Forest Act*.

Permitting requirements may also come from additional agencies during the mine development process as a result of future legislative and/or policy adjustments.

20.4.4 Referrals

The Province of BC upholds its fiduciary duty to consult with Indigenous groups regarding potential adverse effects of activities on treaty rights or claimed/proven Indigenous rights or title. Proponent applications and/or Project notifications that did not involve prior consultation or engagement as part of the EA process will be referred to potentially-affected Indigenous groups. Indigenous groups are typically given a 30-day period to review and respond with any issues or concerns identified.

20.4.5 Federal Authorizations

The Project may include federal requirements under enabling legislations such as:

- *Fisheries Act:*
  - Authorization under the Policy for Applying Measures to Offset Adverse Effects on Fish and Fish Habitat administered by DFO (2019);
  - Environmental effects monitoring under the Metal and Diamond Mining Effluent regulations (MDMER) administered by Environment and Climate Change Canada (ECCC).
- *Canadian Navigable Waters Act* – approval from the Navigation Protection Program (NPP) administered by Transport Canada;
- SARA – permits issued by DFO and/or the Canadian Wildlife Service (CWS) of ECCC.

20.5 Community

Local perspectives and opinions are critical to FPX's decision-making process throughout all aspects of the Project and are an integral part of ongoing consultation and engagement with local communities and First Nations. Enduring relationships must be built with Indigenous groups and community stakeholders. It is based on trust: transparency, accountability, mutual understanding and respect for rights and title, active engagement, and a long-term commitment to shared value.

FPX has developed an effective public and First Nations engagement process to foster its relationships with local communities, maximize shared value, and minimize adverse effects. The approach is based on principles of sustainable development and corporate responsibility and considers environment, social, and governance (ESG) factors. The management of external relations is defined in separate stages: mid-stage exploration, advanced exploration and, specifically, the early engagement phase of the EA process. Overall objectives of the Decar external relations engagement process are to:

- Engage in consistent and respectful dialogue with statutory decision-makers (SDM), stakeholders, and Indigenous groups on the Project;
- Ensure that Indigenous and stakeholder engagement strive to maximize mutual understanding of interests, values, priorities, and concerns related to the Project;
- Collaborate with Indigenous groups, community stakeholders, and SDMs on Project planning;
- Acknowledge and respect the role of Indigenous governing bodies as decision-makers under the BC Declaration of the Rights of Indigenous Peoples Act (DRIPA, 2019);
- Maximize potential economic, social, cultural, health, and environmental values from the Project to the local and broader public interest.

20.5.1 Local First Nations

The Project lies exclusively in the traditional territory of the Tl’azt’en Nation and Binche Whut’en Nation, which have been the focus of Indigenous engagement and consultation activities to date. FPX signed an exploration Memorandum of Understanding (MOU) for the Project on May 22, 2012. The MOU formalizes protocols for continuing the cooperative working relationship established between the Tl’azt’en Nation including constituent Keyoh families and FPX, regarding exploration activities for the Project. The MOU confirms the Tl’azt’en Nation’s support for the exploration activities and acknowledges, as well as describes, how Project activities will be managed with respect to:

- Cultural and environmental interests of the Tl’azt’en communities;
- Ongoing engagement and internal community consultation activities;
- Socio-economic benefits to the Tl’azt’en Nation communities through community contribution funds and business opportunities.

The MOU also establishes processes for the future negotiation of a comprehensive Impact and Benefits Agreement (IBA) for when the Project proceeds to mine development. This IBA emphasizes mutual respect and positive long-term relationship between the parties during all phases of the Project.

On March 12, 2019, Binche Whut'en was constituted as a newly recognized First Nation by the Canadian federal government, officially separating from Tl'azt'en Nation. Of the four Keyoh families who are signatories to the MOU, two are associated with Tl'azt'en Nation, and two are associated with the newly-formed Binche Whut'en Nation. FPX is engaged in discussions with Tl'azt'en Nation, Binche Whut'en and the four constituent Keyoh families to amend the MOU to reflect the new administrative structure entailed by the separation of Binche Whut'en from Tl'azt'en Nation.

20.6 Mine Closure and Reclamation

EMPR will provide the regulatory framework for FPX's obligations for decommissioning, closure, reclamation and rehabilitation for the Project. Acceptable practices will result from effective Crown consultation, ongoing engagement with indigenous groups, and effective planning throughout the EA process.

The Mines Act permit requires a closure plan with the appropriate reclamation security paid. Annual reclamation reports are filed with EMPR as a permit condition under the Health, Safety and Reclamation Code for Mines in BC. The security is collected upon initial permit issuance and adjusted through operational lifespan of the mine to accurately cover the cost of the liability. it must be acceptable to the Mines Inspector determining the appropriate bond amount over time.

A Regional Reclamation Bond Calculator provides the Regional Inspector with a means of assessing reclamation liability that avoids undue financial risk and liability to the public. The assessed security represents the cost of mine reclamation to the Province while promoting transparency to Indigenous groups and the public. The security is returned once the mine site has been reclaimed to a satisfactory level and no longer requires monitoring or maintenance. It ensures that modern mine sites do not leave an ongoing legacy or require public funds for clean-up activities.

FPX's mine closure and reclamation plan will aim to reclaim and rehabilitate the Project footprint to ensure that, upon termination of mining, land, watercourses and cultural heritage resources will be returned to a safe and environmentally sound condition and to an acceptable end land use that considers previous and potential uses. Components of the mine closure and reclamation plan will address the following:

- End Land Use Objectives: End land use objectives will be developed based on returning the Project area to an acceptable end land use that considers previous and potential users;
- Productivity or Capability Objectives: Productivity or capability objectives, how they will be achieved, and how reclamation success will be measured, will be defined;

- Long-term Stability: Long-term stability, both physical and chemical, must be adequately addressed for all structures and discharges from the mine site;
- Treatment of Structures and Equipment: To the extent possible, all structures and/or equipment will be removed, or alternatively disposed of following site decommissioning. All at-grade concrete will be broken, covered with soil, and vegetated;
- Watercourse Reclamation: Re-establishment of post-mine watercourses will be undertaken as appropriate. Water structures not needed for longer-term water management will be rehabilitated to ensure proper site drainage;
- Road Reclamation: Roads will be re-vegetated where applicable and decommissioned to ensure geotechnical and hydraulic stability;
- Disposal of Toxic Chemicals: Chemicals and reagents will be taken off-site using the same modes of transport, containment, and emergency response as utilized during site operations;
- Operational and Post-Closure Monitoring: A long-term monitoring plan will be developed to address geotechnical, ARD/ML, re-vegetation, sedimentation or other long-term Project risks.

The external TSF, in-pit tailings dam, sedimentation ponds and the waste dumps area will be reclaimed after mining is complete in Year 35. Parts of the external TSF can be reclaimed during and after Year 23, when the tailings storage transitions from the external TSF into in-pit. The waste dumps will be progressively re-sloped and reclaimed during operation when possible. The pit rim dam will be reclaimed after Year 35, along with any other mining structures. The following summarizes various reclamation activities for the various mining areas:

Waste dumps: After active mining, the dumps will be re-sloped down to 2H:1V slope or flatter in accordance with BC mine reclamation requirements and drainage features will be incorporated into the final rock pile landforms. Soil will be placed and revegetated by seeding with native vegetation mixture and planting tree seedlings. For this PEA study, it is assumed there is sufficient amount of topsoil stockpiled during mining that can be used for reclamation purposes.

Pit wall and benches: The pit walls and benches will be left in their post-mining configuration. The exposed, gently sloping pit floors will be covered with soil and revegetated. Water management channels within the post-closure pit will be developed, where required, to minimize erosion.

Sedimentation ponds: The ponds will be regraded, and any impoundments will be breached to prevent the accumulation of runoff water. This will allow surface waters to flow along the natural local drainage systems. Depending on erosion/sediment management requirements, portions of the existing water management system can be left in place for an extended period while vegetation develops, and the post-closure landscape stabilizes.

Tailings Storage Facilities: The external tailings dam will likely remain a dam structure following the cessation of operations because the deposited slurry tailings will require long-term containment. It is expected that minor re-sloping and topsoil placement could occur on the downstream slopes of the dam. Consolidation of the impounded tailings will probably require many years (>10 years) so while capping of the tailings is a goal, it may take several years before the impounded tailings can be trafficked such that a closure cover can be placed. It may be necessary to place the internal soil cover in multiple thin lifts over the tailings beach in order to manage the on-going consolidation of the tailings deposit.

The final tailings dam should be reconfigured to allow for release of extreme storm event run-off via an emergency spillway structure. Diversions of run-off upstream of the impoundment should be developed to limit the volumes of water reporting to the impoundment area during reclamation. Similarly, the in-pit tailings storage should include an overflow spillway unless the containment within the pit has capacity for the maximum precipitation event.

Once a decision has been made to permanently close the site, it is anticipated that the major closure activities that have not been completed progressively during the LOM would be completed within a period of approximately three years, with the exception of the TSF which will require a longer period for tailings consolidation. Closure costs have been estimated based on evaluation of comparable mining projects in Canada and are estimated at $130M. These costs have been included in the financial evaluation of the Project.

20.7 References

The following list of references pertain to the published literature on environmental sensitivities and conservation statuses as currently understood.

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21. CAPITAL AND OPERATING COSTS

The Baptiste Project scope covered in this PEA is based on the construction of a facility having an annual throughput of 120,000 tpd or 43,800,000 tpa. Initially, during Phase 1 of the Project, covering Year 1 to Year 21 of operations, the ROM material is processed at a primary grind size P_{80} of 300 µm, as dictated by the tailings deposition strategy. In Project Phase 2, starting in Year 22 when in-pit tailings deposition is implemented, the primary grind size is reduced to a P_{80} of 170 µm in order to achieve improved Ni recoveries.

The capital cost, sustaining capital cost and operating cost estimates for this PEA were developed based on the methodology described in this chapter. Costs relating to mineral processing, site infrastructure, G&A and product handling and transport were developed by BBA. Costs relating to mining, the tailings storage facility (TSF) and related surface water management were developed by Stantec. All costs in this chapter are presented in USD unless otherwise stated. The exchange rate used is $1.00 CAN = $0.76 US. The base date for the cost estimate is Q3-2020.

Table 21-1 presents a summary of the estimated capital cost for initial pre-production, in-pit tailings deposition and sustaining capital.

Table 21-1: Summary of capital cost estimate

Category	Pre-Production M$	In-Pit Deposition M$	Sustaining M$	Total LOM M$
Direct Costs				
Mobile Equipment	$155.1	$0.0	$353.5	$508.6
TSF	$137.9	$14.5	$534.3	$686.6
Mine and TSF Site Preparation	$95.5	$0.0	$90.4	$185.9
Mineral Processing	$610.0	$88.4	$18.2	$716.6
Off-Site Infrastructure	$64.4	$0.0	$0.0	$64.4
On-Site Infrastructure	$66.4	$0.0	$6.8	$73.2
Total Direct Costs	**$1 129.3**	**$102.9**	**$1 003.2**	**$2 235.4**
Indirect Costs	$291.8	$0.0	$8.2	$300.1
Contingency	$253.7	$0.0	$0.0	$253.7
TOTAL PROJECT CAPITAL COST	**$1 674.8**	**$102.9**	**$1 011.5**	**$2 789.2**

The total initial capital cost, including direct costs, indirect costs and contingency was estimated at **$1,674.8M**. This represents the pre-production capital expenditure required to support start-up of operations in Year 1. The capital cost related to the in-pit tailings deposition implementation was estimated at $102.9M. This is the capital expenditure specifically required to allow for finer primary grinding (resulting in improved Ni recovery) and for pumping tailings to the exhausted mine pits for in-pit deposition, starting in Year 22 of the mine life, as described in Chapter 16 of this Report. This cost also includes the cost for constructing the pit rim dike for containing tailings to the end of the mine life. Sustaining capital costs (which excludes the capital cost related to the implementation of finer primary grinding and in-pit deposition) were estimated at $1,011.5M. These costs include items such as mine equipment fleet additions and replacements, facilities additions and improvements and costs related to TSF sand cell construction and surface water management which are incurred over the LOM starting at Year 1 of operation. It should be noted that closure and reclamation costs are excluded from the stated capital costs but are included in the economic analysis and discussed in Chapter 22 of this Report.

Table 21-2 presents a summary of the estimated average operating costs for the initial Phase 1 (Years 1 to 21), Phase 2 (Years 22 to 35) and for the Life-of-mine (LOM), expressed in USD/t of dry material processed (milled). Averages include ramp-up years (Year 1 and Year 22).

Table 21-2: Total estimated phase and average LOM operating cost ($/t milled)

Estimated Average LOM OPEX	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
Mining	$2.28	$2.66	$2.43
Mineral Processing	$2.71	$2.91	$2.79
Briquette Transport	$0.19	$0.18	$0.19
Rail Terminal and Access Road	$0.05	$0.05	$0.05
General Site Services	$0.62	$0.62	$0.62
General and Administration	$0.25	$0.25	$0.25
TOTAL Opex	**$6.09**	**$6.66**	**$6.32**

It should be noted that royalties and working capital are not included in the operating cost estimate presented but are treated separately in the Economic Analysis and discussed in Chapter 22 of this Report.

Table 21-3 presents additional metrics of costs incurred annually over the operating years of the mine and include the following:

- ‘C1’ cost defined as follows: “The costs of mining, milling and concentrating, onsite administration and general expenses, property and production royalties not related to revenues or profits, metal product treatment charges, and freight and marketing costs less the net value of by-product credits, if any. These are expressed on the basis of ‘per unit Ni content’ of the sold product.”
- ‘AISC’ or ‘all-in sustaining costs’ defined as follows: “These costs comprise the sum of C1 costs, sustaining capital, royalties and closure expenses. These are expressed on the basis of ‘per unit Ni content’ of the sold product.”

Table 21-3: C1 costs and AISC costs

	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
C1 costs ($/lb Ni)	$2.61	$2.94	$2.74
C1 cost ($/metric tonne Ni)	$5,753	$6,488	$6,038
AISC cost ($/lb Ni)	$3.13	$3.11	$3.12
AISC cost ($/metric tonne Ni)	$6,897	$6,867	$6,885

21.1 Basis of Capital Cost Estimate and Assumptions

The capital cost estimate for this PEA was developed by BBA and Stantec to an accuracy of +/-35% and is generally based on an Engineering, Procurement and Construction Management (EPCM) project execution strategy. Some elements were costed on a 'turnkey, lump-sum basis' while others were based on an 'owner-perform' basis where FPX acts as the construction manager.

21.1.1 Direct Cost Estimate for Mobile Equipment

Mobile equipment capital cost expenditures incurred over the pre-production years include the following, as estimated by Stantec:

- Mining equipment purchased by FPX for pre-stripping overburden and waste rock required for the construction of the external starter dike;
- Mobile equipment required for construction of the starter dike and for other site support during construction.

The initial mobile equipment fleet serves to support pit, waste dump and TSF development during the pre-production years as well as mining operations beginning in Year 1. All equipment is assumed to be bought and paid for in the year prior to its use on-site, except for some smaller equipment bought in Year 3 to conduct work required at that time. This smaller equipment has short erection times and will not be required to operate until several months into Year 3. Direct costs for this component include the purchase of all mobile mining equipment including delivery and erection costs, as well as the mobile equipment used at the TSF. This estimate is based on recent vendor quotes and Stantec's internal cost database.

Mobile equipment capital costs also include the following, as estimated by BBA:

- A mobile equipment fleet required for site maintenance and operations support;
- Mobile equipment for product material handling and transportation to the Fort St. James rail terminal.

Costs for service vehicles and plant mobile equipment for site maintenance and support were estimated by factoring a BBA reference project. As such, a similar equipment fleet was assumed but adapted to the Baptiste Project requirements.

Equipment costs for briquette loading into containers, container handling at the mine and at the rail terminal and for hauling the containers to the Fort St. James rail terminal were estimated based on recent vendor prices.

21.1.2 Direct Cost Estimate for Mining and TSF (by Stantec)

In general, Stantec performed its capital cost estimate based on the mine plan and TSF construction requirements, as discussed in Chapter 16 of this Report. Components of direct capital costs include the following:

- Pre-production (mine pre-stripping) to generate construction materials for the TSF starter dike and mine roads;
- TSF starter dike and cut-off trench construction (placement and compaction of construction materials);
- Mine and TSF site preparation consisting of:
  - Surface water management in areas affecting the open pit, waste dump and TSF;
  - Initial mine roads;
  - Site clearing/preparation.

The following describes Stantec's methodology for estimating direct capital costs for the various components of the mine and TSF infrastructure:

- Mine pre-production consists mainly of removing overburden and waste rock and delivering the material to the TSF for construction of the starter dike. The pre-production mine operation will be carried out by FPX employees and equipment, under the direct supervision of FPX. The cost estimate is based on a combination of material take-off (MTO) data, vendor quotes, Stantec's internal database, manufacturers' information, and industry standards and rates.
- TSF starter dike construction consists of placing and compacting the materials from the mine. Other related costs included in this item consist of the construction of a cut-off trench and dike instrumentation. Construction will be performed by FPX employees and equipment, under the direct supervision of FPX. The cost estimate is based on a combination of material take-off (MTO) data, vendor quotes, Stantec's internal database, manufacturers' information, and industry standards and rates.

- Mine and TSF site preparation was estimated as follows:
  - Surface water management covers the construction of both temporary and permanent water management structures at the waste dump, open pit and TSF including:
    - Large and small sediment ponds and related dams/impoundment structures;
    - Small temporary ponds/sumps;
    - Ditches for impacted and non-impacted water streams including adjacent maintenance access;
    - Creek diversions;
    - Tailings pond spillway.

    This work will be carried out by a specialized contractor but will be under the direct supervision of FPX. Estimates were based on Stantec's internal database and quotes for similar work in the area.
  - The construction of initial mine roads during the pre-production period ranging from pickup truck roads to roads suitable for 220-tonne haul trucks. Road costs include:
    - Pioneering;
    - Fill placement;
    - Culverts.

    This work will be carried out by FPX employees and equipment, under the direct supervision of FPX. Estimates were based on Stantec's internal database and experience in designs completed in similar terrain.
  - Site preparation consists of work related to the clearing of the initial footprints of the pit, dump, roads, water management structures and TSF dike. The tailings basin is assumed to be uncleared other than for soil recovery where feasible. This item covers:
    - Soil stripping and stockpiling;
    - Clearing and grubbing (logging is not included);
    - Initial pit dewatering.

    This work will be carried out by contractors, under the direct supervision of FPX. Estimates are based on Stantec's internal database and quotes for similar work.

21.1.3 Direct Cost Estimate for Mineral Processing and Infrastructure (by BBA)

In general, BBA performed its capital cost estimate based on MTOs estimated from the conceptual site plan and 3-D model, major equipment lists and quantities for certain commodities based on reference projects of similar scope. The reference data was reviewed and adjusted for date, scale of size and project context. Additional costing information was obtained from equipment vendors, internal databases and first principles as described in this Chapter. The capital cost estimate is based on conceptual the Project development and construction schedule presented in Chapter 24 of this Report.

The capital cost estimate developed by BBA is based on the construction of a greenfield facility at the Decar Property. For this PEA, the conceptual process flowsheet and plant design are largely based on metallurgical testwork results, reference projects and BBA's experience in operating plants and on recent projects. Major process equipment costs were estimated based on budgetary proposals from reputable vendors.

The following describes BBA's methodology for estimating direct capital costs for the various mineral processing areas:

- Direct costs comprise the sum of labor, materials and equipment:
  - Labor costs were estimated based on estimated standard hours per activity and regional productivity factors and hourly unit crew rates per discipline are considered 'all-in' and include direct and indirect labor rates as well as equipment rates. A 3-in/1-out rotation was assumed with a 70-hour work week, with an allowance for travel time as well for travel costs (airfare and bus) to and from site;
  - Commodity materials such as concrete and structural steel were estimated based on either a material take-off based on the 3-D model or a factored estimate based on experience on other similar projects. Unit prices for the commodities were estimated based on experience on other similar projects;
  - Major process equipment costs were estimated based on vendor budget prices. Major pumps were sized based on pumping capacity and estimated head and costs were estimated based on BBA's data base. Conveyors were sized based on capacity and lengths from the 3-D model and costs were estimated based on BBA's database.
- Direct cost estimate for the mineral processing infrastructure include these four areas:
  - <u>Crushing:</u> this area includes the primary crusher station, secondary crushing area, screen house and connecting conveyors, including the conveyors discharging onto the crushed material stockpile;
  - <u>HPGR:</u> this area includes the crushed material stockpile and reclaim conveyor system, the HPGR building and the screenhouse, including the conveyors that feed the primary grinding ball mills;
  - <u>Concentrator:</u> this area includes primary grinding and magnetic separation, magnetic concentrate regrinding and secondary magnetic concentration, flotation and final concentrate briquetting. This area also includes tailings cycloning, thickening and first stage of fine and coarse tailings pumping;
  - <u>Tailings pumping and process water management:</u> this area includes the coarse and fine tailings pipelines, reclaim water barge-mounted pumphouse and reclaim water pipeline and the fresh water pumphouse at Trembleur Lake and pipeline. These were estimated based on piping lengths from the site plot plan and assumed materials standard costs in BBA's database and pumphouse capacities and costs developed on other similar projects.

- Direct costs for the mineral processing areas include the following elements and were estimated as follows:
  - Major process equipment was selected and sized based on the conceptual process flowsheet. Budgetary proposals were obtained from equipment vendors. Allowances or factored estimates were made for secondary equipment such as pumps, conveyors and platework;
  - Building and equipment foundations and concrete works, structural steel elements and architectural works were evaluated based on the conceptual 3-D model developed in this PEA using a combination of MTOs and factors to similar reference projects.
  - Electrical, automation and piping were factored based on similar projects;
  - A budget estimate for site preparation and earthworks was included to cover the estimated concentrator area pads as well as pads required for other site infrastructure.

The following describes BBA's methodology for estimating direct capital costs for off-site and on-site infrastructure:

- Off-site infrastructure is described in Chapter 18 of this Report and consists of the following three major elements:
  - The high voltage 230 kV transmission powerline was estimated based on an assumed tie-in to the BC Hydro power grid and topographical data. A 'turn-key, lump-sum' price was obtained from a specialized contractor based on its experience and as-built cost data for a similar powerline in the general vicinity of the Decar Property;
  - The access road from Fort St. James to the Decar Property was estimated based on an assessment of the existing road and assumptions on required upgrades and the construction of a new road segment and bridge, as described in Chapter 18 of this Report. Unit costs related to this work were estimated by BBA based on experience;
  - The Fort St. James rail terminal was estimated based on a scope of work that provides the functionality required to support the Project. The estimate was generally based on a reasonable allowance for individual component within the infrastructure.
- On-site infrastructure is described in Chapter 18 of this Report. The capital cost estimates were generally developed as 'turnkey lump-sum' costs based on a contractor design-build strategy. These were based on data available to BBA from as-built costs for similar facilities on other projects, adapted to the Baptiste Project requirements. Direct costs include the following:
  - The emergency vehicle garage and emergency response team (ERT) facility;
  - The fuel storage facility;
  - The mine services building including mine equipment maintenance bays, truck wash, warehouse, offices and employee facilities;
  - The permanent camp and administration office and ancillary infrastructure;

- The incinerator building;
- Cold and heated storage buildings;
- Potable water and sewage treatment systems;
- Emergency generators;
- Site roads;
- 230 kV electrical substation;
- Electric power distribution to the various site buildings from the main substation;
- Fresh water supply to the camp and to the mine garage and truck wash station;
- Communication tower;
- Site fire protection system.

21.1.4 Indirect Capital Costs

Project indirect costs generally include the following components:

- Owner's costs: These include items such as Owner's project management team salaries and expenses, insurance, authorization certificates and permits, compensation for environmental and affected stakeholders, costs related operational readiness such as hiring and training operations and supervisory personnel, etc.;
- Engineering, Procurement, and Construction Management (EPCM) services;
- Costs related to the construction of temporary facilities (office trailers, fuel storage, laydown and storage, power generators, etc.) as well as operations and maintenance of these;
- Temporary construction camp construction and dismantling;
- Construction camp operation;
- Freight to transport equipment to site;
- Commissioning spare parts and first fills;
- Vendor representatives;
- Commissioning.

These indirect cost components are typically factored as a percentage of direct costs, based on project execution methodology and experience on projects of similar size, scope and location remoteness. For this PEA, indirect cost factors were applied as follows:

- No indirect costs were applied to mobile equipment purchased during pre-production and used for construction and required for first year of operation;
- For mine pre-stripping, starter dike construction and site preparation, Stantec compiled a list of indirect costs which include the Owner's team for construction technical support and supervision, spare parts for equipment, software and communication systems, consultants and certain construction site maintenance and upkeep costs. Indirect costs, as a percentage of direct costs (excluding mobile equipment) amount to 26% of direct costs;

- For the process plant and off-site and on-site infrastructure, an effective 31% indirect cost factor was applied, built up from assumed factors for the various indirect cost component. Within this factor, an estimate of construction camp requirements was performed. It was estimated that a 620-room temporary construction camp will be required to support peak construction, including construction personnel at the mine and TSF. Camp operating costs were estimated on a per diem basis based on the estimated total construction hours for the Project, including mine and TSF construction, concentrator and off-site and on-site infrastructure.

On a consolidated basis, excluding mobile equipment, the indirect costs in the capital cost estimate amounts to 30% of total direct costs.

21.1.5 Contingency

Contingency provides an allowance to the capital cost estimate for undeveloped details within the scope of work covered by the estimate. Contingency is not intended to take into account items such as labour disruptions, weather-related impediments, changes in the scope of project from what is defined in this study, nor does contingency take into account price escalation or currency fluctuations. The following contingency factors were applied for to the capital cost estimate for this PEA Study:

- No contingency was applied to the cost of mobile equipment used for mine pre-stripping, TSF construction, site support and product handling;
- A 25% contingency was applied to the direct cost of mine pre-stripping, starter dike construction and instrumentation, site preparation and some indirect cost components;
- A 20% contingency was applied to the sum of direct and indirect costs for the mineral processing and on-site infrastructure executed in an EPCM regime;
- A 10% contingency was applied to the sum of direct and indirect costs to infrastructure components whose direct costs were estimated on the basis of a ‘turnkey, lump-sum’ estimate.

On a consolidated basis, excluding mobile equipment, the contingency in the capital cost estimate amounts to 20.0% of total direct plus indirect costs.

21.1.6 Exclusions

The following items are not included in this Capital Cost Estimate:

- Inflation and escalation. The estimate is in constant Q3-2020 United States Dollars;
- Costs associated with hedging against currency fluctuations;
- All taxes, duties and levies;
- Working capital (included in the Financial Analysis but not in the capital or operating costs);
- Sunk costs;
- Risk mitigation costs;
- Project financing costs including but not limited to interest expense, fees and commissions.

21.2 Estimated Capital Costs

Mobile equipment: The initial mobile equipment fleet acquired during pre-production by FPX has an estimated cost of **$155.1M**. Equipment for fleet additions and replacements over the LOM are included in sustaining capital and its capital cost is estimated at $353.5M.

TSF: The initial direct capital cost incurred during pre-production for the construction of the engineered containment area for the external TSF, including the cost for the pre-stripping of the open pit mine, was estimated at **$137.9M**. Sustaining capital incurred mainly from Year 1 to Year 21 for raising the dike by constructing sand cells using coarse tailings is estimated at $534.3M. In addition to these elements, an estimated cost $14.5M is incurred after Year 25, to construct a small pit perimeter dike in order to allow for in-pit tailings deposition during Phase 2 of the Project.

Mine and TSF site preparation: During the pre-production period, direct cost related to site preparation, consisting mainly of site clearing of the initial open pit footprint and the TSF starter dike areas, mine road construction and temporary and permanent surface water management systems was estimated at **$95.5M**. Related sustaining capital was estimated at $90.4M and is comprised mainly of LOM surface water management and progressive site preparation of the open pit mine as the footprint expands.

Mineral processing: This area includes two-stage crushing and screening, stockpiling and reclaiming, HPGR and screening, grinding, primary magnetic concentrating, regrinding and secondary magnetic concentrating and flotation. In addition, FeNi concentrate briquetting, tailings cycloning, dewatering as well as the pumps and pipelines to the TSF and the reclaim water and fresh water pumping systems are also included. The estimated pre-production direct capital costs for mineral processing is **$610.0M**. Sustaining capital, associated mainly with the additions of tailings booster pumping capacity as tailings to the external TSF are pumped to progressively higher elevations, was estimated at $18.2M. For Phase 2 of the Project, where the primary grind size becomes finer by the addition of a third Ball Mill in a separate building and tailings are pumped to the open pit, the capital cost was estimated at $88.4M.

Off-site infrastructure: This area includes the construction of the 230-kV electric power transmission line, the upgrade and construction of new segments of the site access road and the construction of the Fort St. James rail terminal. The estimated direct capital costs for this area is **$64.4M**.

On-site infrastructure: This area includes a number of sub-areas and systems such as the mine garage and ancillary facilities, the permanent camp, site roads, emergency vehicle building, fuel storage, and storage facilities. The estimated direct capital cost for this area is **$66.4M**. Sustaining capital was estimated at $6.8M and is related to the addition of truck bays to the mine garage and the addition of rooms to the permanent camp, over the LOM.

Indirect costs: The components of indirect costs have been described earlier in this Chapter. Indirect costs have been estimated at **$291.8M**. Indirect costs as part of sustaining capital have been estimated at $8.2M.

Contingency: This has been described previously in this Chapter and has been estimated at **$253.7M**.

21.3 Operating Costs

The average operating cost for the LOM of the Baptiste Project has been estimated at **$6.32 /t** of milled material. This amount represents the cost to produce one tonne of FeNi briquettes and ship this product to a port in Asia.

Operating costs were based on the following unit prices:

- Electric power US $0.053 ($0.07 CAN) / kWh;
- Diesel fuel US $0.654 ($0.86 CAN) / litre, based on average 2019 price;
- Labor costs were based on the collective bargaining agreement of a nearby BC mine with an added 60% factor for burden and benefits.

Operating costs, as defined in this Section, exclude costs which are related to site reclamation, royalties and working capital. These are treated separately and discussed in the Economic Analysis presented in Chapter 22 of this Report.

21.3.1 Mine Operating Costs

The mine operating costs, as estimated by Stantec, consist of the following components:

- Mining activities include drilling, loading, hauling, mining support, and equipment maintenance. Equipment operating costs, including maintenance and operating hourly employees, were estimated from the total annual operating hours based on the equipment productivities and the cost per service meter unit (SMU) hour to operate the equipment;

- Blasting is assumed to be performed by a contractor and estimated to cost $0.20/t. The estimate was obtained from Stantec's internal database;
- Staff includes mine department salary staff comprising of mine operations, mine maintenance, environmental, engineering, geology and geotechnical (including TSF QA/QC). During full production 69 personnel are included in this category;
- General site support includes equipment and personnel to support mining operations. Road maintenance equipment includes graders, water trucks and sand trucks. Mobile maintenance equipment including mechanics trucks, welding trucks, a mobile crane and tire handler. Other general support equipment including front end loaders, fuel/lube trucks, lowboy, light plants, crew bus and pickup trucks. A 12-person general utility crew (3 per shift) and 8 first aid personnel are also included;
- Supplies and services include:
  - Operating supplies and minor items estimated as a factor of 5% of mining, blasting, TSF, and general site support operating costs;
  - Software/technology, where an allowance of $1.0 million/year was assumed for mine technology projects, the specifics of which to be defined in the future;
  - Miscellaneous contracting/consultants, where an allowance to employ contractors for small jobs and hire consultants to carry out inspections and support operations was provided.

The total estimated operating cost for all mining activities is $1.79/t mined, or $2.43/t milled, averaged over the LOM. These are shown in Table 21-4 and presented graphically in Figure 21-1.

Over the first 16 years, the cost per tonne of mineralized material (or per tonne milled) ranges from $1.76 to $2.27, with an average of $1.99. From Year 17 to Year 23 where the strip ratio increases in the mine phase 2 area, the operating cost ranges from $3.05 to $4.03 with an average of $3.29 over the 7 years. From Year 24 to Year 35, the operating cost ranges from $2.29 to $2.79 with an average of $2.53.

Table 21-4: Average LOM mining operating costs

Mining Opex	$/t milled	$/t mined
Drilling	$0.16	$0.12
Loading	$0.36	$0.26
Hauling	$0.96	$0.71
Mining Support	$0.17	$0.12
Blasting	$0.27	$0.20
Staff	$0.19	$0.14
General Site Support	$0.19	$0.14
Supplies and Services	$0.13	$0.10
TOTAL Mining Opex	**$2.43**	**$1.79**

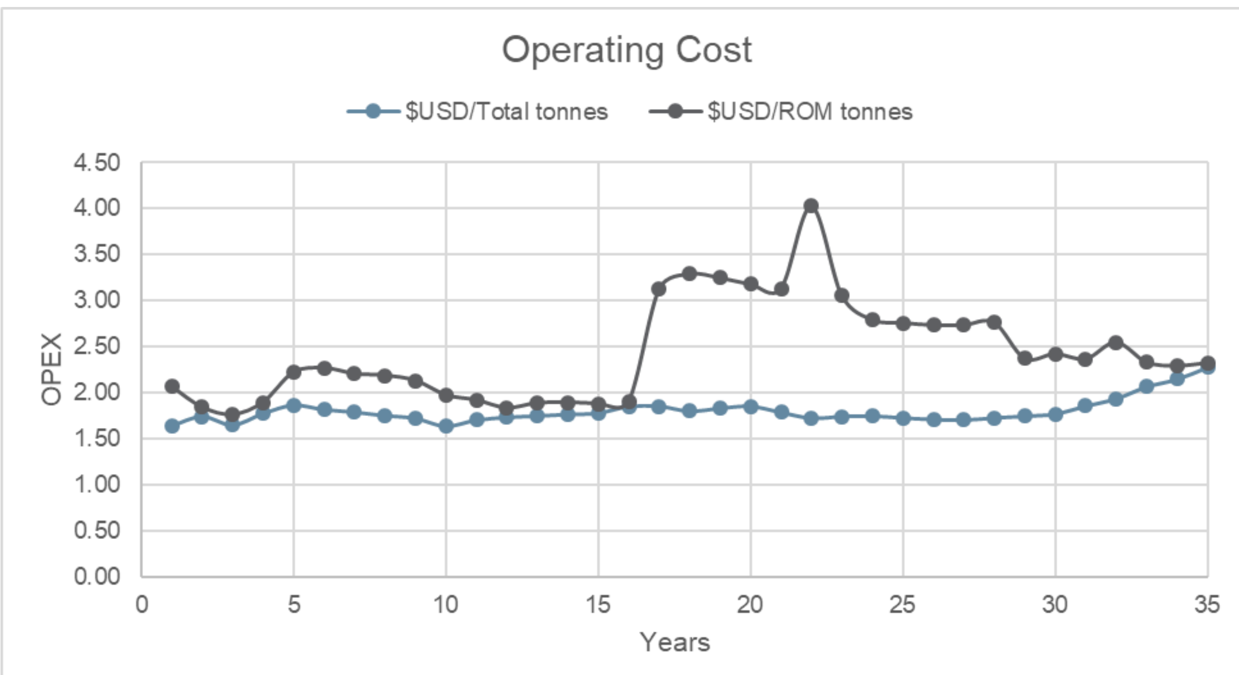

Figure 21-1: LOM annual mining operating costs

For this PEA, operating costs are tracked on a LOM basis but also as Project Phase 1 (Yr 1 to Yr 21) and Phase 2 (Yr 22 to Yr 35) where tailings disposal is transferred from the external TSF to in-pit deposition. Table 21-5 presents average mining costs for the periods covered by Project Phase 1 and Phase 2 on a $/t milled basis.

Table 21-5: Average LOM mining operating costs by Project phase

Mining Opex ($/t milled)	Phase 1	Phase 2	Avg. LOM
	Yr 1-21	Yr 22 - 35	
Drilling	$0.16	$0.17	$0.16
Loading	$0.34	$0.38	$0.36
Hauling	$0.85	$1.12	$0.96
Mining Support	$0.15	$0.19	$0.17
Blasting	$0.26	$0.29	$0.27
Staff	$0.19	$0.20	$0.19
General Site Support	$0.20	$0.17	$0.19
Supplies and Services	$0.13	$0.14	$0.13
TOTAL Mining	**$2.28**	**$2.66**	**$2.43**

21.3.2 Mineral Processing Operating Costs

Operating costs for processing at the Baptiste Project are shown in Table 21-6. The following values include costs associated with crushing, HPGR, grinding, magnetic separation, regrinding, flotation, dewatering, briquetting, conveying, process pumping and tailings pumping. These costs were derived from metallurgical testwork, the conceptual 3-D model, supplier information, BBA's database, or factored from similar operations.

Table 21-6: Mineral processing operating costs

Mineral Processing Opex ($/t milled)	Phase 1 Yr 1 – 21	Phase 2 Yr 22 - 35	Avg. LOM
Labour	$0.33	$0.35	$0.34
Electric power	$0.92	$0.99	$0.95
Process consumables (mantles, tires and liners)	$0.49	$0.48	$0.49
Grinding media	$0.53	$0.65	$0.58
Reagents	$0.29	$0.29	$0.29
Maintenance and supplies	$0.14	$0.14	$0.14
TOTAL Mineral Processing	**$2.71**	**$2.91**	**$2.79**

Labour: In Phase 1, it is estimated that 149 supervisory and hourly personnel will be required to operate and maintain the various areas of the mineral processing plant. For Phase 2, concentrator labour requirements are estimated to increase to 157.

Electric power: Electric power for the Baptiste Project is provided by BC Power. Annual power consumption at the concentrator was discussed in Chapter 18 of this Report.

Process consumables: Process consumables include crusher concaves, mantles, HPGR tires, screen decks, mill liners, filter cloths and lab consumables. Consumption rates were estimated in collaboration with vendors as well as BBA's database and other similar projects as a function of throughput. It should be noted that no testwork was available in this PEA to support the assumed consumption rates. Unit costs were recently generally provided by vendors for major cost items.

Grinding media: Grinding media is consumed in the ball mill and vertical stirred mills. Media consumption was estimated based specific power consumption as well as benchmarking with other operations. No abrasion testwork has been performed for this PEA.

Reagents: Reagents consist mainly of flotation reagents and flocculant for the thickeners. Reagent consumption was estimated based on testwork results and benchmarking with other operations.

Maintenance and supplies: Costs for maintenance parts and supplies (excluding labour) were estimated using a 3% factor of major concentrator equipment costs.

21.3.3 Briquette Transport

Costs related to briquette transportation from the mine to the Fort St. James rail terminal and subsequently to a port in Asia are presented in Table 21-7. A description of the logistics and design parameters for estimating the loading and hauling fleet was provided in Chapter 18 of this Report. Briquette transportation costs from the mine site to Fort St. James include costs for personnel, fuel and equipment maintenance and operation. Briquette transportation cost from rail terminal to Asia was estimated on a cost per container basis by a logistics service provider familiar with this type of material handling in Western Canada. This cost also includes container cost, container handling costs at the port and shipping costs to Asia.

Table 21-7: Briquette transportation costs

Briquette Transport ($/t milled)	Phase 1 Yr 1 - 21	Phase 2 Yr 22 - 35	Avg. LOM
Briquette transport from mine to rail terminal	$0.02	$0.02	$0.02
Briquette transport from rail terminal to Asia port	$0.17	$0.17	$0.17
TOTAL Briquette Transport	**$0.19**	**$0.18**	**$0.19**

21.3.4 Rail Terminal and Access Road

The operating cost related to the access road maintenance and Fort St. James rail terminal are presented in Table 21-8. A labor headcount was estimated for the operation of the rail terminal and a general allowance was included for site services. Also, an allowance was provided for contracted services for maintenance of parts of the FSR roads accessing the Decar Property.

Table 21-8: Rail terminal and access road operating costs

Rail Terminal and Access Road ($/t milled)	Phase 1 Yr 1 - 21	Phase 2 Yr 22 - 35	Avg. LOM
Labour	$0.03	$0.03	$0.03
Access road maintenance (allowance)	$0.02	$0.02	$0.02
Rail terminal (allowance)	$0.01	$0.01	$0.01
TOTAL Rail Terminal and Access Road	**$0.05**	**$0.05**	**$0.05**

21.3.5 General Site Services

Operating costs related to general site services at the Decar Property are shown in Table 21-9.

Table 21-9: Site services operating costs

General Site Services ($/t milled)	Phase 1 Yr 1 - 21	Phase 2 Yr 22 -35	Avg. LOM
Labour	$0.06	$0.06	$0.06
Electric power	$0.14	$0.14	$0.14
Building heating (Allowance)	$0.09	$0.09	$0.09
Site Maintenance (Mobile Equipment)	$0.04	$0.04	$0.04
Fly in fly out (FIFO) air and land transport	$0.07	$0.07	$0.07
Camp Operations	$0.19	$0.19	$0.19
General Site Buildings (Allowance)	$0.02	$0.02	$0.02
TOTAL General Site Services	**$0.62**	**$0.62**	**$0.62**

Labour: In both Phase 1 and Phase 2, it is estimated that 27 supervisory and hourly personnel will be required to operate and maintain the general site infrastructures (excluding areas related to the mine and TSF). Employees include site mobile equipment operators assigned to tasks such as road maintenance, snow removal and operating general site mobile equipment as required.

Electric power: For both phases, power requirements for infrastructure (other than mineral processing related power) have been estimated as an allowance based on other projects. Transmission line power losses are also included in this cost.

Building heating: A general allowance was provided for building heating based on a specific energy consumption of 0.34 GJ/m^3 of building volume. The concentrator building volume was derived from the 3-D model and other buildings were estimated or factored.

Site maintenance mobile equipment: The cost related to the site maintenance mobile equipment was estimated as a factor of the site maintenance equipment fleet capital cost based on a reference project.

FIFO air and land transport: The FIFO cost includes airfare and bus transportation to the site from Prince George. It's assumed that 50% of the employees are flying to Prince George and taking the bus to the mine site. The remaining 50% of the employees are assumed to be from local communities and use land transportation to reach the site.

Camp operations: The cost related to camp operations was estimated based on a per diem allowance per room and it's assumed that all rooms will be occupied. Based on the mine plan, in Yr 17 the camp operations cost will increase due to the increase of required rooms.

General site buildings: This item was estimated as an allowance and it covers the costs of general maintenance in for the various site buildings.

21.3.6 General and Administration

G&A operating costs at the Baptiste Project are shown in Table 21-10.

Table 21-10: G&A operating costs

G&A ($/t milled)	Phase 1 Yr 1 - 21	Phase 2 Yr 22 - 35	Avg. LOM
Labour	$0.13	$0.13	$0.13
Materials, services (allowance)	$0.12	$0.12	$0.12
TOTAL G&A	**$0.25**	**$0.25**	**$0.25**

Labour: In both phases, it is estimated that 43 salaried employees will be required for general management, finance and human resources and support to the operations of the facilities. The labour cost excludes technical services costs as these were included in mining costs. It also excludes corporate office personnel.

Materials and services: This item was estimated as a general allowance and includes general corporate and administrative expenses including maintenance of mining leases, municipal taxes, site insurance and other miscellaneous supplies.

22. ECONOMIC ANALYSIS

The Economic Analysis for the Baptiste Project PEA Study was performed using a discounted cash flow model on both a pre-tax and post-tax basis. The Capital and Operating Cost Estimates, presented in Chapter 21 of this Report, are based on the mining and processing plan developed in this Study to process 120,000 tpd or 48.3 Mtpa over the estimated 35-year life-of-mine (LOM).

The internal rate of return (IRR) on total investment was calculated based on 100% equity financing. The net present value (NPV) was calculated for discounting rates between 0% and 10%, resulting from the net cash flow estimated to be generated by the Project. The Project Base Case NPV was calculated based on a discounting rate of 8%. The payback period on the undiscounted annual cash flow of the Project, is also presented. Furthermore, a sensitivity analysis was performed for the base case on both pre-tax and post-tax basis, to assess the impact of a +/-20% variation of the Project initial pre-production capital cost, annual operating costs (including the cost for shipping FeNi briquette to a port in Asia) and the assumed Ni selling price. The Economic Analysis was performed with the following assumptions and basis:

- Project Execution is based on key Baptiste milestones presented in Chapter 24 of this Report;
- The Economic Analysis was performed based on the 35-year mine plan developed in this Study;
- The price of Ni (based on contained Ni in the 63.4% Ni briquette product sold) assumed in the base case Economic Analysis is US $16,743/t. This price was derived based on the methodology presented in Chapter 19 of this Report;
- Exchange rate of $1.00 CAN = $0.76 US;
- All costs and sales estimates are in constant Q3-2020 dollars;
- The Economic Analysis includes working capital that was estimated based on a 30-day operating cost basis;
- All sunk costs are not considered in this Economic Analysis;
- A 1% NSR (Net Smelter Return) royalty is payable based on annual sales less cost of transportation of the product to market;
- Reclamation costs (total estimated costs of $129.8 M) related to the posting of a security bond were assumed to be incurred in an initial pre-production amount followed by progressive payments every five years, based on the expanding footprint of the Baptiste.

Table 22-1 presents the undiscounted, pre-tax cash flow projection for the Project. Assumptions were made regarding disbursement of initial pre-production capital costs over the construction period. Details regarding mill feed, head grades and other mining parameters are detailed in the mine production schedule found in Chapter 16 of this Report.

Table 22-1: Baptiste Project table of un-discounted, pre-tax, cash flow (M$ US)

Description / Year	PP3	PP2	PP1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Contained Ni Sold (Kt)	-	-	-	39.2	49.8	47.8	51.0	46.7	40.6	40.0	41.8	44.1	45.8	48.4	51.9	46.4	45.0	45.0	45.3	44.8
Ni Selling Price ($/t)	-	-	-	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743
Revenue (M$)				**656.8**	**834.4**	**799.7**	**853.3**	**782.0**	**680.1**	**670.3**	**700.1**	**738.6**	**767.7**	**809.7**	**869.4**	**776.1**	**752.8**	**753.0**	**759.2**	**750.5**
Operating Costs (M$)				205.6	245.9	242.9	249.2	262.9	263.9	261.1	260.6	259.3	252.7	250.6	247.5	249.0	249.0	248.5	249.5	304.6
Royalties(M$)				6.5	8.3	7.9	8.4	7.7	6.7	6.6	6.9	7.3	7.6	8.0	8.6	7.7	7.4	7.5	7.5	7.4
Capital Costs (M$)	488.6	462.1	608.1	149.1	27.9	37.2	46.6	33.1	30.5	31.6	35.2	53.4	48.6	41.8	39.3	36.2	31.0	43.3	108.7	64.8
Reclamation (M$)			45.3	0.0	0.0	0.0	0.0	25.5	0.0	0.0	0.0	0.0	17.1	0.0	0.0	0.0	0.0	3.1	0.0	0.0
Cash Flow (Undiscounted)																				
Total Operating + Royalties (M$)				212.1	254.2	250.8	257.7	270.6	270.7	267.8	267.5	266.6	260.3	258.7	256.1	256.6	256.5	255.9	257.0	312.0
Total Capital + Reclamation (M$)	488.6	462.1	653.4	149.1	27.9	37.2	46.6	58.6	30.5	31.6	35.2	53.4	65.7	41.8	39.3	36.2	31.0	46.4	108.7	64.8
Working Capital (M$)				16.9	3.3	(0.3)	0.5	1.1	0.1	(0.2)	(0.0)	(0.1)	(0.5)	(0.2)	(0.3)	0.1	0.0	(0.0)	0.1	4.5
Annual Cash Flow	(488.6)	(462.1)	(653.4)	278.7	548.9	512.0	548.6	451.6	378.9	371.2	397.4	418.7	442.2	509.4	574.2	483.1	465.2	450.8	393.4	369.1
Cumulative Cash Flow	(488.6)	(950.7)	(1,604.1)	(1,325.4)	(776.4)	(264.4)	284.1	735.8	1,114.6	1,485.8	1,883.2	2,301.9	2,744.1	3,253.4	3,827.6	4,310.7	4,776.0	5,226.8	5,620.2	5,989.3

Description / Year	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	Total
Contained Ni Sold (Kt)	47.1	48.0	45.5	48.5	30.8	45.3	45.1	46.2	48.9	51.7	46.5	46.5	49.0	47.7	38.1	37.1	40.8	36.2	**1,572**
Ni Selling Price (USD/t)	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743	16,743
Revenue (M$)	**788.5**	**804.0**	**761.1**	**811.9**	**515.5**	**757.8**	**754.6**	**773.7**	**819.5**	**866.2**	**778.5**	**778.7**	**821.1**	**798.4**	**637.7**	**620.3**	**683.4**	**606.1**	**26,331**
Operating Costs (M$)	312.3	310.6	307.1	305.6	276.3	308.1	296.6	295.3	294.7	295.2	295.5	278.4	281.0	278.4	284.7	275.1	274.0	223.9	9,495.6
Royalties (M$)	7.8	8.0	7.5	8.0	5.1	7.5	7.5	7.7	8.1	8.6	7.7	7.7	8.1	7.9	6.3	6.1	6.8	6.0	260.5
Capital Costs (M$)	34.7	28.5	50.0	113.1	27.5	22.5	13.1	61.7	2.2	0.7	11.5	0.5	2.2	0.0	2.3	0.3	0.7	0.3	2,789.2
Reclamation (M$)	0.0	0.0	15.8	0.0	0.0	0.0	0.0	5.6	0.0	0.0	0.0	0.0	10.9	0.0	0.0	0.0	0.0	6.5	129.8
Cash Flow (Undiscounted)																			
Total Operating + Royalties (M$)	320.1	318.5	314.7	313.6	281.4	315.6	304.1	303.0	302.9	303.8	303.2	286.1	289.2	286.3	291.0	281.2	280.7	229.9	9,756.1
Total Capital + Reclamation (M$)	34.7	28.5	65.8	113.1	27.5	22.5	13.1	67.3	2.2	0.7	11.5	0.5	13.1	0.0	2.3	0.3	0.7	6.8	2,919.0
Working Capital	0.6	(0.1)	(0.3)	(0.1)	(2.4)	2.6	(0.9)	(0.1)	(0.0)	0.0	0.0	(1.4)	0.2	(0.2)	0.5	(0.8)	(0.1)	(22.5)	0.0
Annual Cash Flow	433.1	457.1	381.0	385.3	209.0	417.1	438.4	403.5	514.4	561.7	463.8	493.5	518.7	512.3	343.8	339.6	402.1	392.0	13,656
Cumulative Cash Flow	6,422.4	6,879.5	7,260.5	7,645.8	7,854.8	8,271.9	8,710.3	9,113.7	9,628.2	10,190	10,654	11,147	11,666	12,178	12,522	12,862	13,264	13,656	

A discount rate is applied to the cash flow to derive the NPV for each discount rate. The payback period is presented for the undiscounted cumulative NPV. The NPV calculation was done at 0%, 5%, 8% and 10%. The Base Case NPV was assumed at a discount rate of 8% following discussions with FPX. Table 22-2 presents the results of the Economic Analysis for the Project, based on the assumptions and cash flow projections presented previously.

Table 22-2: Economic analysis results (pre-tax)

IRR = 22.5% **Payback = 3.5 years**	**NPV (M$)**
Discount Rate	
0%	$13,656 M
5%	$5,003 M
8%	**$2,927 M**
10%	$2,069 M

22.1 Corporate Taxation

The current Canadian tax system applicable to Mineral Resources Income was used to assess the annual tax liabilities for the Project. This consists of federal and provincial corporate taxes, as well as provincial mining taxes.

Corporate taxpayers, resident in Canada, are subject to a federal income tax rate of 15% and taxpayers resident in British Columbia are subject to a further 12%, for a total combined corporate income tax rate of 27%.

Taxable losses generated in a given year may be carried forward 20 years and applied to taxable income carried back 3 years.

Costs associated with exploration and development are allocated to certain resource pools and deductible against taxable income. Canadian Exploration Expenses (CEE) may be carried forward indefinitely and are fully deductible against taxable income. Canadian Development Expenses (CDE) may be carried forward indefinitely and are deductible against taxable income up to a maximum of 30% per year on a declining balance basis.

A depreciation rate of 25% per year is applied to capital expenditures on mining production equipment. In 2018, certain accelerated depreciation provisions were adopted by the Canadian Government. The provisions that apply to this Baptiste include suspending the half-year rule for determining the Capital Cost Allowance. This provision is currently planned to be phased out by 2028. For this Baptiste, the half-year rule for capital expenditures incurred from the inception of this Baptiste through the first year of operations has been suspended. After the second year of operations, the half-year rule is assumed to be reinstated.

Currently the provincial government in British Columbia collects taxes relating to mineral production referred to as BC Mineral Tax. BC Mineral taxes are assessed under a two-part system, made up of Net Current Proceeds Tax and Net Revenue Tax.

Net Current Proceeds Tax applies at a rate of 2% to operating cash flow from production. This tax applies until the producer has recovered applicable capital investments and a reasonable rate of return, at which time the Net Revenue Tax will apply at a rate of 13%. The total tax collected under both Net Revenue Tax and Net Current Proceeds Tax will not exceed 13%.

The development of the Baptiste will be eligible for a new mine allowance under the BC Mineral Tax. The new mine allowance is calculated as 1/3 of eligible capital expenditures from the development of the new mine and is applied in determining the Net Revenue Tax. BC Mineral taxes are deductible against corporate income taxes.

Taxation calculations, based on the aforementioned applicable taxes, were provided by FPX and its tax consultant. Table 22-3 presents the results of the post-tax economic analysis.

Table 22-3: Economic analysis results (post-tax)

IRR = 18.3% **Payback = 4.0 years** **Discount Rate**	**NPV (M$)**
0%	$8,725 M
5%	$3,091 M
8%	**$1.721 M**
10%	$1,149 M

22.2 Sensitivity Analysis

A sensitivity analysis was performed on a pre-tax and post-tax basis, whereby initial pre-production capital cost, annual operating costs and product selling price were individually varied between +/- 20% to determine the impact on Project IRR and NPV at an 8% discount rate. Results are presented graphically in Figure 22-1 and Figure 22-2 for the pre-tax analysis and in Figure 22-3 and Figure 22-4 for the post-tax analysis. Results are also presented in tabular form in Table 22-4. As can be observed, economic performance is most sensitive to the FeNi selling price.

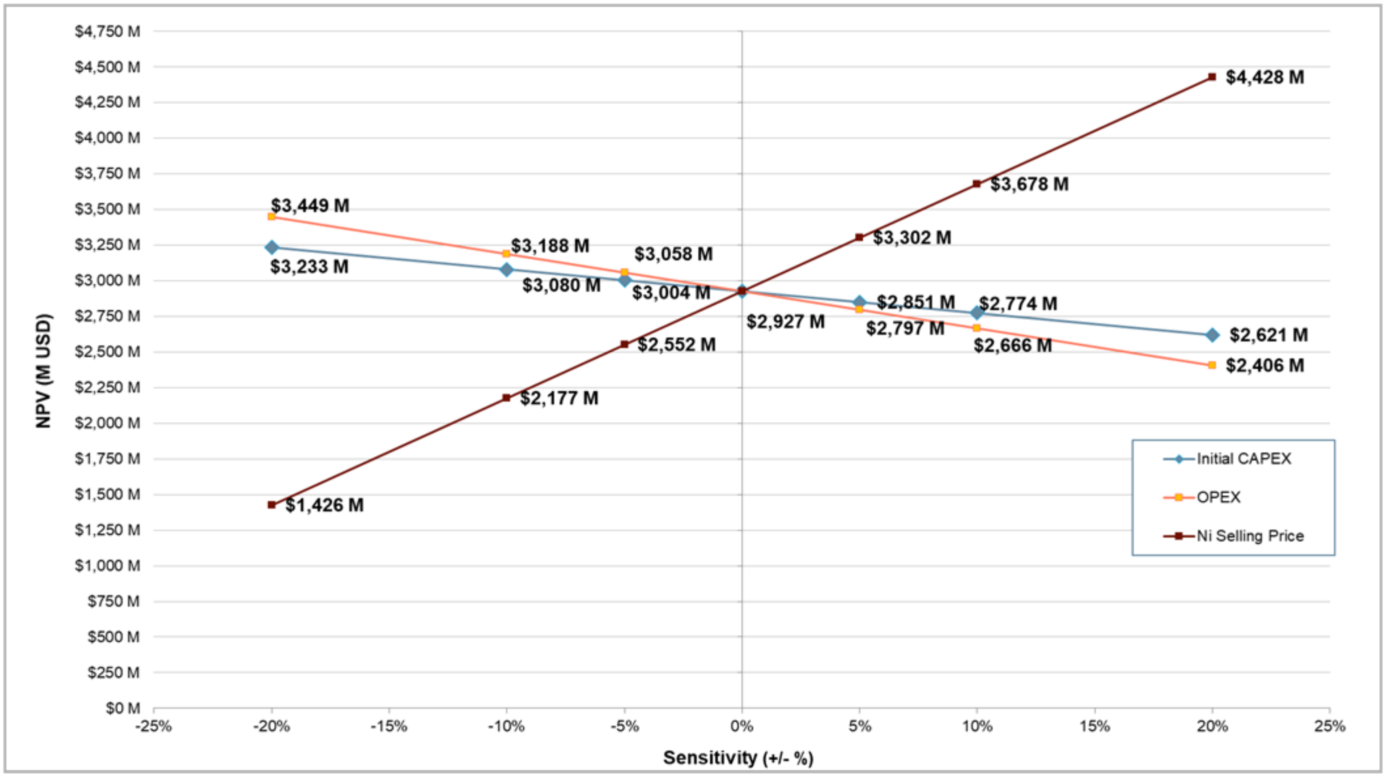

Figure 22-1: Sensitivity analysis graph for NPV (pre-tax)

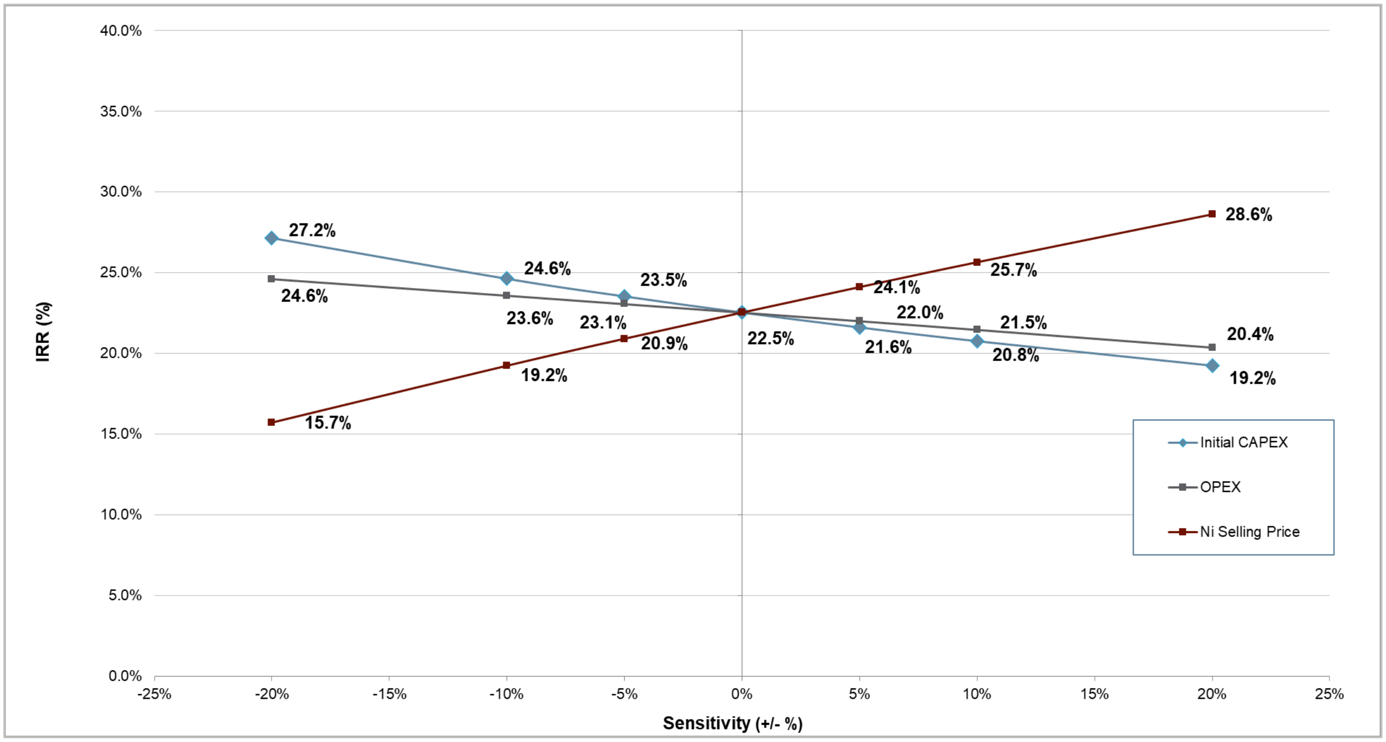

Figure 22-2: Sensitivity analysis graph for IRR (post-tax)

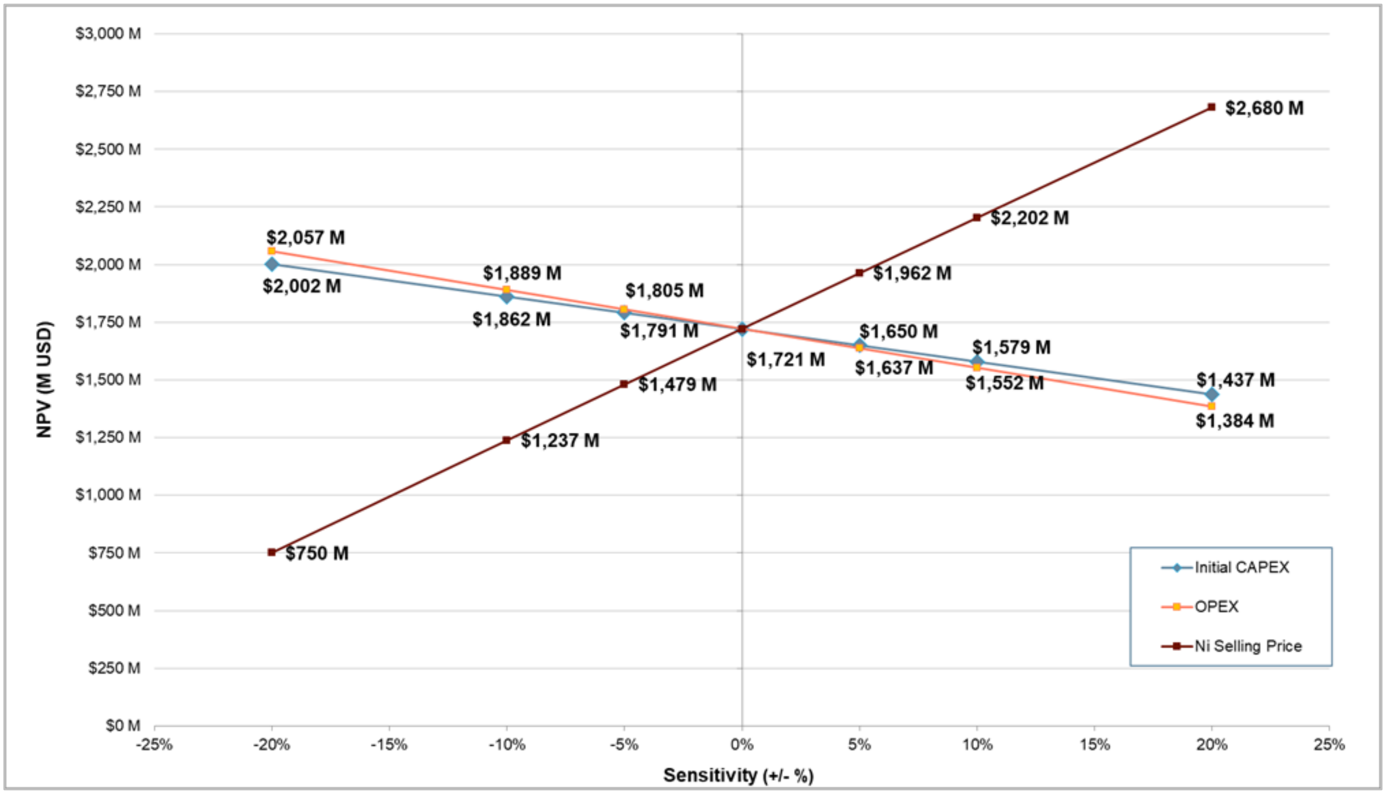

Figure 22-3: Sensitivity analysis graph for NPV (post-tax)

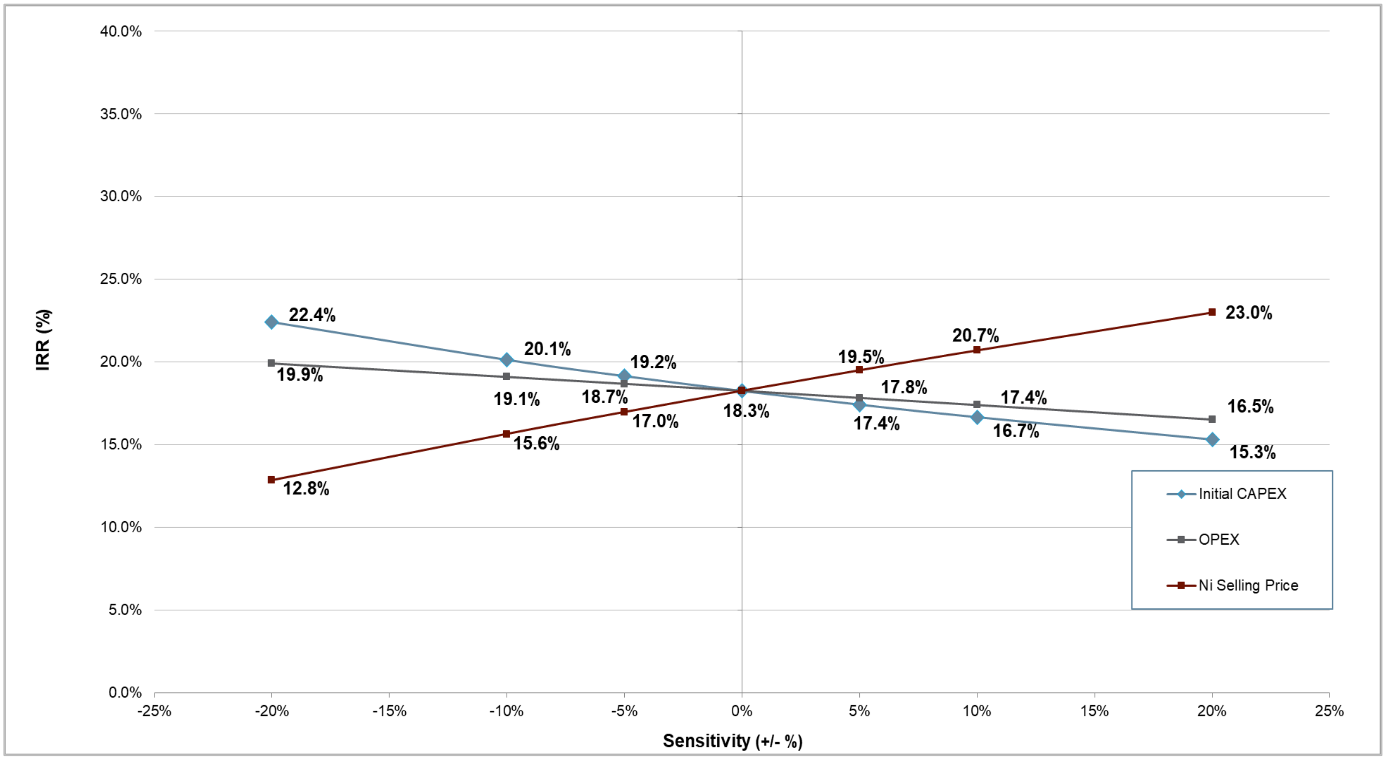

Figure 22-4: Sensitivity analysis graph for IRR (post-tax)

Table 22-4: Summary of sensitivity analysis results

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Operating Costs					
Pre-Tax					
NPV	$3,449 M	$3,188 M	**$2,927 M**	$2,666 M	$2,406 M
IRR	24.6%	23.6%	**22.5%**	21.5%	20.4%
Post-Tax					
NPV	$2,057 M	$1,889 M	**$1,721 M**	$1,552 M	$1,384 M
IRR	19.9%	19.1%	**18.3%**	17.4%	16.5%
Capital Costs					
Pre-Tax					
NPV	$3,233 M	$3,080 M	**$2,927 M**	$2,774 M	$2,621 M
IRR	27.2%	24.6%	**22.5%**	20.8%	19.2%
Post-Tax					
NPV	$2,002 M	$1,862 M	**$1,721 M**	$1,579 M	$1,437 M
IRR	22.4%	20.1%	**18.3%**	16.7%	15.3%
Ni Selling Price					
Pre-Tax					
NPV	$1,426 M	$2,177 M	**$2,927 M**	$3,678 M	$4,428 M
IRR	15.7%	19.2%	**22.5%**	25.7%	28.6%
Post-Tax					
NPV	$750 M	$1,237 M	**$1,721 M**	$2,202 M	$2,680 M
IRR	12.8%	15.6%	**18.3%**	20.7%	23.0%

23. ADJACENT PROPERTIES

There are two early stage properties immediately adjacent to the Decar Property: Mount Sidney Williams and Mac. The information source for the Mount Sidney Williams property is a summary of 2013 fieldwork written by Mowat (2013). Information for the Mac property is from a resource estimate, on behalf of Stratton Resources Inc, disclosed by Giroux and Moore (2012). Only the Mount Sidney Williams property has documented occurrences of awaruite that may be similar to the style of mineralization found on the Decar Property. Both the Mac and Mount Sidney Williams Properties are located directly west adjoining the Decar claim group.

QP Ron Voordouw has not verified the information for either the Mount Sidney Williams or Mac properties. Furthermore, available information for the Mount Sidney Williams and Mac properties is not necessarily indicative of the mineralization at Decar.

23.1 Mount Sidney Williams Property

Mowat (2013) describes fine grained awaruite occurring within ultramafic rocks on the Mount Sidney Williams property, including coarse-grained awaruite on the Klone 6 claim that is locally visible without a hand lens. The dimensions of the conceptual exploration target on the Mount Sidney Williams property are unknown. Work on the property has been limited to mapping and sampling, and diamond drilling of 10 exploratory drill holes.

23.2 Mac Property

The Mac property is an early stage molybdenum-copper porphyry deposit that is not indicative of the mineralization found on the Decar Property. A resource estimate by Giroux and Moore (2012) reported Indicated resources of 70,360 kt at 0.063% Mo and 0.100% Cu containing 97.74 Mlbs molybdenum and 155.14 Mlbs copper. Inferred resources are estimated to be 177,934 kt at 0.042% Mo and 0.050% Cu containing 164.78 Mlbs molybdenum and 196.17 Mlbs copper. Drilling on the property has totalled 22,377m from 104 holes.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Opportunity for Baptiste FeNi in EV Battery Application

As part of its product development program, FPX mandated Sherritt Technologies to perform scoping-level, batch pressure leach tests on a sample of Baptiste FeNi flotation product. The testwork has been previously summarized in Chapter 13 of this Report. (Sherritt, 2019). The results were announced by FPX in a News Release dated January 7, 2020.

Two batch pressure leach tests of Baptiste concentrates were undertaken at Sherritt Technologies in Fort Saskatchewan, Alberta. The pressure leach tests were conducted in an autoclave (pressure chamber) with conditions designed to approximate proposed commercial production conditions.

The quality of the nickel chemical solution generated from the batch tests was excellent, with the low acid and iron content indicating relatively low downstream requirements for neutralization and iron removal. Recovery of nickel was rapid in the batch tests, with over 98% extraction achieved in 60 minutes toward ultimate extractions of 98.8% and 99.5% in 180 minutes. The iron (Fe) in the concentrate feedstock was almost entirely precipitated, resulting in low iron content in the pregnant leach solution.

It is expected that the nickel-cobalt solution produced from the concentrate will be an ideal feedstock for the production of nickel sulphate and cobalt sulphate. Downstream processing of the nickel-cobalt solution would conceptually entail neutralization (to remove acid and other impurities) and solvent extraction to produce nickel sulphate and cobalt sulphate as two separate products. The low levels of impurities (such as acid and iron) in the nickel-cobalt solution suggest that downstream refinement into sulphate products is achievable within conventional operating parameters.

Additional test work is required to further evaluate the optimization of any downstream hydrometallurgical processing requirements.

The operating parameters for the leaching of Baptiste concentrates into nickel-cobalt solution are potentially favourable because they are based on significantly lower sizing, power consumption, pressure and temperature requirements than typical HPAL operations treating laterite ore.

Furthermore, it is expected that utilizing a high-grade Baptiste concentrate feed with a consistent specification for mineral content, moisture and particle size would entail lower potential operational risk than a typical HPAL plant accepting “whole ore” with inconsistent, grades sourced from inherently variable run-of-mine laterite deposits, ranging from 0.7% to 1.7% nickel content.

This positive result provides FPX with an opportunity to pursue an alternative marketing route for part of its FeNi production. This would allow FPX to become a player in the EV battery value chain. As the Project advances, this opportunity would need to be supported with more testwork and a validation of process economics.

24.2 Opportunity for Sale of Iron Concentrate

The process flowsheet developed in this PEA Study generates a flotation tailing with a high Fe content (in the form of magnetite), which can potentially be marketable as a magnetite iron ore concentrate and generate additional financial benefit to the Project. The annual production of this co-product would be in the order of 2.0 Mt. FPX mandated BBA to perform a high-level assessment to evaluate the economic viability of this product. The results of this analysis are presented in an internal technical memorandum (BBA, 2019).

Table 24-1 presents the chemical analysis for the Fe concentrate, indicated by the testwork performed and described in Chapter 13 of this Report, produced directly by the flotation process.

Table 24-1: Chemical analysis for the Baptiste Fe concentrate

Element	Units	PSD P_{80} = 25 µm
		Fe Concentrate
Fe(T)	%	61.0
Fe_3O_4	%	84.3
Cr_2O_3	%	2.81
MgO	%	3.75
SiO_2	%	3.87

The concentrate is high in Fe, in the form of magnetite. Based only on the Fe content, this would be a product that can be potentially used as a feed for ironmaking (as a precursor to steelmaking). The economic viability analysis performed by BBA considered four options:

- Selling the concentrate as is;
- Upgrading the concentrate to increase Fe content and decrease the MgO and SiO_2 to levels for direct reduced iron (DRI) applications and selling as concentrate;
- Pelletizing the upgraded concentrate and sell the product as pellets;
- Produce DRI with the upgraded pellets and sell the product as DRI.

Testwork will need to be performed to evaluate mineral processing requirements to reduce MgO and SiO_2 levels. The major constraint with this Fe concentrate is its high Cr content. Initial indications are that Cr may not be amenable to removal by mineral processing. If this is shown to be the case, this concentrate may be used in the carbon steel value chain but only in limited quantities and possibly with payability discounts. This concentrate, however, should be usable in the stainless steel value chain, which is a much smaller market than that of carbon steel. It should be noted that the concentrate cannot be used directly by steel smelters and will need a reduction step to make an intermediate iron product (blast furnace or direct reduction).

Selling the Fe concentrate in any form will require a more elaborate material handling system, especially at the rail terminal, which will need a capacity to handle about 60 railcars of product per day for rail transportation to a port terminal. Pelletizing and DRI production require significant additional infrastructure and related capital expenses but generate a higher margin product. Given the right market conditions, selling the product as an upgraded concentrate may provide FPX with a good alternative to disposing of this product with the process tailings, which is the base case for this PEA Study. The sale of the magnetite iron ore concentrate would have the added benefit of reducing the tonnage impounded in the tailings facility by in excess of 70 Mt over the mine life. In its report, BBA recommended that in the next Project study phase, FPX perform testwork on the upgrading of the Fe concentrate and perform a more detailed logistics and marketability analysis to further develop this opportunity.

24.3 Project Execution Plan

Following this PEA Study for the Baptiste Project, it is expected that FPX will proceed with a prefeasibility study to further develop the concepts presented in this PEA Study and to further de-risk the Project. This should logically then be followed by a feasibility study. Concurrently, FPX will also need to proceed with environmental assessment, studies, community relations development and other permitting activities. This will also likely be complemented by additional infill drilling, metallurgical sampling and testwork and geotechnical surveys. A duration of about 2.5 years should be planned for these activities, to completion of the feasibility study, after which FPX can proceed directly to detailed engineering procurement and construction.

The key to success for implementation of the Baptiste Project, rests with planning of logistics and construction. Early in detailed engineering and with the support of procurement resources, an early works package, which includes the development of temporary and permanent infrastructure and site preparation, including earthworks for water management to support construction, will be of prime importance. This infrastructure includes the following:

- Access road construction and upgrade as well as other site road works;
- Temporary construction camp and office trailers;
- Fuel storage;
- Equipment laydown areas (on-site and possibly also in Fort St. James);
- Power generators.

Construction of these aforementioned areas should begin over the third year (Yr-3) before start of production. This will be followed by engineering, procurement and construction activities for all Project areas. The overall project execution schedule should take into account delivery lead times for key elements and engineering and procurement priorities should be given to these. This may even require that some critical early infrastructure packages be developed during the FS. The construction plan should also consider the seasonal impacts on equipment and materials delivery to site and taking advantage of the summer months to strategically advance work in order to properly plan and execute work during the winter months.

Figure 24-1 presents a high-level schedule for key activities and milestones, including studies, environmental and engineering and construction, leading to operations start-up.

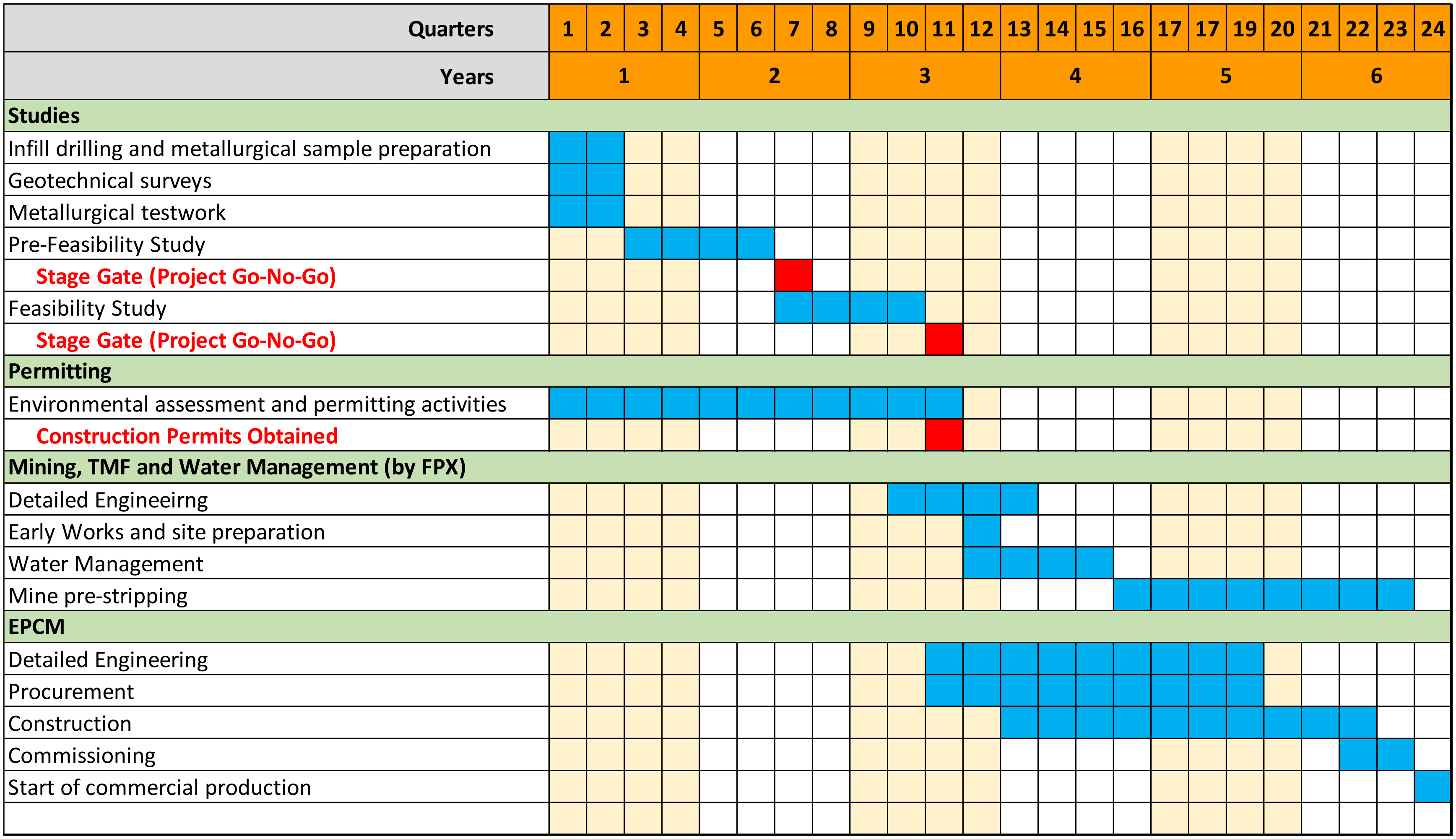

Figure 24-1: High-Level project implementation schedule

25. INTERPRETATION AND CONCLUSIONS

25.1 Overview

The Decar Property, located in central British Columbia, hosts the Baptiste Deposit which is characterized as a disseminated Ni-Fe alloy (awaruite) Deposit. In addition to the Baptiste Deposit, there are three other known target areas on the Property that bear similarities to the Baptiste Deposit, all of which occur within the serpentinized ultramafic rocks of the Cache Creek terrane. The Property consists of 62 claims that cover 24,740 ha and is 100% owned by FPX after re-purchase of the Decar Property from option partner Cliffs in 2015. This purchase was made with a loan granting the Lender a 1% NSR over the Property. FPX has signed a MOU with the Tl'azt'en Nation that formalized protocols for the working relationship between the company and the Tl'azt'en Nation and allows continued support for exploration activities.

The lower elevations of the Property are road accessible from the town of Fort St. James (population 1,510) via a network of province-maintained paved and forestry-maintained gravel roads. CN Rail owns an inactive railway that passes through the northeastern-most claims of the Property. Electrical infrastructure reaches to within 5 km of the Property and a hydro substation capable of powering a mine operation is located 90 km to the south. The Mount Milligan porphyry copper-gold open pit mine is the nearest producing mine to the Decar Property and is located 90 km to the northeast.

Disseminated awaruite is hosted in the Trembleur ultramafic unit (ultramafite). Awaruite is strongly magnetic and has a high density (8.2 g/cm^3) compared to associated gangue minerals of magnetite (5.2 g/cm^3) and serpentine (2.2-2.9 g/cm^3). Awaruite occurrences were first discovered on the Property as part of an academic thesis in 1983, then sporadically assessed through rock sampling and petrographic work from 1996-2005. In 2006, FPX staked 33 claims to establish the Decar Property and explore the areas now known as the Baptiste Deposit, Sid, Target B and Van target areas. An overview of this work (Britten, 2016) has suggested that most awaruite formed during serpentinization of nickeliferous olivine in peridotite within broad fault and/or shear zones.

From 2006-2009, FPX conducted staking, airborne geophysics, ground-based IP surveys, rock sampling, geological mapping, petrography, and SEM analysis. Mapping and outcrop sampling proved to be effective exploration tools and were used to identify Baptiste as the primary exploration target on the Property.

In 2009, FPX entered an option agreement that allowed Cliffs to earn an interest in the Property through meeting several project milestones, including the completion of NI 43-101 compliant PEA, prefeasibility and feasibility studies. In 2010, Cliffs completed seven diamond drill holes for 1,711 m on the Baptiste Deposit and another three holes for 847 m on Sid Target. In 2011 and 2012, 70 holes for 27,665 m of diamond drilling were completed that focused mostly on the Baptiste Deposit (25,959 m) with one hole drilled at Target B (305 m). In addition, seven hydrogeological monitoring wells, for 1,401 m, were drilled in the Baptiste area. These programs provided the necessary data to inform a maiden mineral resource estimate on the Baptiste

Deposit in 2012 (Ronacher et al., 2012a) followed by an updated resource (Ronacher et al., 2013) that formed the basis for a PEA completed in 2013 (McLaughlin et al., 2013). Additional drilling on the Baptiste Deposit, in 2017, triggered an update to the resource estimate as described in Section 14.0 of this technical report.

The geochemical assay method using the Davis tube was developed by Cliffs in 2010, with all assays done by Actlabs. This method involves running a slurry containing 30 g of pulverized sample material through a Davis tube magnetic separator and recovering magnetic and non-magnetic fractions. The magnetic fraction is then analyzed by fusion XRF. DTR Ni is calculated by multiplying the Ni content of the mass percent of the magnetic sample recovered (see Section 8.0). Reviews of QA/QC data suggests that assays are uncontaminated, precise and accurate, but show some positive bias thought to be partially explained by the fusion XRF analytical method.

25.2 Mineral Resource Statement

An updated resource estimate for the Baptiste Deposit includes all data from the 83 surface drillholes completed since 2010 and 2,053 samples from a re-sampling program of 2010/2011 drill core that was carried out in 2012. The estimate is geologically constrained within four mineralized domains and is reasonably comparable among different estimation methods (i.e. ordinary kriging, inverse distance squared weighting, nearest neighbour).

The 2018 resource model comprises a large, delta-shaped volume that measures approximately 3 km in length and 150 to 1,080 m in width and extends to a depth of 540 m below the surface. The Baptiste Deposit remains open at depth over the entire system and is covered by an average of 12 metres of overburden.

Table 25-1 presents the mineral resource estimate for the Baptiste Deposit at a range of cut-off grades with the base case, at a cut-off grade of 0.06% DTR Ni, in bold face.

Table 25-1: Indicated and Inferred resources for the Baptiste Deposit

INDICATED				INFERRED			
% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)	% DTR Ni Cut-off	Tonnes 000's	DTR Ni (%)	Contained Ni (Tonnes)
0.02	2,076,969	0.119	2,471,593	0.02	750,633	0.098	735,620
0.04	2,055,578	0.120	2,466,694	0.04	659,900	0.107	706,093
0.06	**1,995,873**	**0.122**	**2,434,965**	**0.06**	**592,890**	**0.114**	**675,895**
0.08	1,871,412	0.126	2,357,979	0.08	499,993	0.122	609,991
0.10	1,617,364	0.131	2,118,747	0.10	399,801	0.130	519,741

Notes:

1. Mineral resource estimate prepared by GeoSim Services Inc. using ordinary kriging with an effective date of September 9, 2020.
2. Indicated mineral resources are drilled on approximate 200 x 200 metre drill spacing and confined to mineralized lithologic domains. Inferred mineral resources are drilled on approximate 300 x 300 metre drill spacing.
3. An optimized pit shell was generated using the following assumptions: US$6.35 per pound nickel Price; a 45° pit slope; assumed mining recovery of 97% DTR Ni and process recovery of 85% DTR Ni, an exchange rate of $1.00 CAN = $0.76 US; and mining costs of US$2.75 per tonne, processing costs of US$4.00 per tonne. A US$1.00 per tonne minimum profit was also imposed to exclude material close to the break-even cut-off.
4. A base case cut-off grade of 0.06% DTR Ni represents an in-situ metal value of approximately US$7.00 per tonne which is believed to provide a reasonable margin over operating and sustaining costs for open-pit mining and processing.
5. Totals may not sum due to rounding.
6. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

25.3 Metallurgy and Mineral Processing

The metallurgical testwork performed for this PEA indicated that at a primary grind of 300 µm, it is possible to produce a 63.4% Ni concentrate with a DTR Ni recovery of 84.7%. In Year 22, when in-pit tailings deposition is implemented, a finer primary grind of 170 µm can be achieved through the addition of a third ball mill resulting in a Ni recovery of 89.5%.

The mineral processing facilities are proposed to be located in an area southwest of the open pit mine. The primary crushing station is placed in an area in the vicinity of the open pit so as to minimize mine truck hauling distances. Run-of-mine material for processing is crushed (two stage crushing and screening) and stockpiled. Downstream of the crushed material stockpile, the major process areas consisting of HPGR and screening, ball milling and hydrocycloning, primary magnetic separation, magnetic concentrate regrinding, secondary magnetic separation and flotation produce a final FeNi concentrate. The fine concentrate is briquetted and transported to market. The process tailings produced are separated into a coarse sand stream (for sand cell dike construction at the external TSF) and a fine tailings stream for impounding within the TSF. In Year 22, the tailings are pumped for in-pit deposition.

25.4 Mining Methods

The mine plan, developed at a conceptual level for this PEA, was based on the mineral resource estimate, presented in Chapter 14 of this Report, and its underlying geological resource block model. The mine plan is based on a three-phase open pit mine development. The first phase of the proposed mine plan uses an external tailings storage facility (TSF) for disposing of tailings generated while mining the Phase 1 pit, which will be mined for the first 21 years. After the Phase 1 pit is mined out, the tailings produced from Phase 2 and Phase 3 of the mine plan will be placed in the mined-out Phase 1 pit. A pit rim dam will be constructed in Year 25 to allow access from the Phase 3 pit to the plant area and also to accommodate the additional tailings that will be stored in the Phase 1 and Phase 2 pits. The Phase 2 and Phase 3 pits will be mined from Year 22 to Year 35.

Mining will be conducted using conventional truck and shovel methods. Large-scale open pit mining will provide the mineral processing plant feed at a rate of 120,000 t/d, or 43.8 Mt/a which was based on the processing capacity inputs provided. Annual mine production of mill feed and waste will peak at 80.1 Mt/a with a life-of-mine (LOM) stripping ratio of 0.40:1 including preproduction (0.32 during the first 10 years of operation, and 0.22 over the first 16 years of operations). Ultimate pit quantities with corresponding Davis Tube Recoverable (DTR) Ni grades are shown in Table 25-2.

Table 25-2: Ultimate design pit quantities

Material Classification	Tonnage (Mt)	Grade (% DTR Ni)
Indicated	1,326	0.124%
Inferred	177	0.102%
Total for processing	**1,503**	**0.121%**
Waste Rock	540	
Overburden	55	
Total Waste	**596**	
Total material mined	**2,098**	
Note: Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.		

25.5 Capital and Operating Costs

Table 25-3 presents a summary of the estimated capital cost for initial pre-production, in-pit tailings deposition and sustaining capital. Table 25-4 presents a summary of the estimated average operating costs for the initial Phase 1, Phase 2 and LOM, expressed in USD/t of material milled. Table 25-5 presents C1 and AISC costs on a per unit Ni sold basis.

Table 25-3: Summary of capital cost estimate

Category	Pre-Production M$	In-Pit Deposition M$	Sustaining M$	Total LOM M$
Direct Costs				
Mobile Equipment	$155.1	$0.0	$353.5	$508.6
TSF	$137.9	$14.5	$534.3	$686.6
Mine and TSF Site Preparation	$95.5	$0.0	$90.4	$185.9
Mineral Processing	$610.0	$88.4	$18.2	$716.6
Off-Site Infrastructure	$64.4	$0.0	$0.0	$64.4
On-Site Infrastructure	$66.4	$0.0	$6.8	$73.2
Total Direct Costs	**$1,129.3**	**$102.9**	**$1,003.2**	**$2,235.4**
Indirect Costs	$291.8	$0.0	$8.2	$300.1
Contingency	$253.7	$0.0	$0.0	$253.7
TOTAL PROJECT CAPITAL COST	**$1,674.8**	**$102.9**	**$1,011.5**	**$2,789.2**

Table 25-4: Total estimated phase and average LOM operating cost ($/t milled)

Estimated Average LOM OPEX	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
Mining	$2.28	$2.66	$2.43
Mineral Processing	$2.71	$2.91	$2.79
Briquette Transport	$0.19	$0.18	$0.19
Rail Terminal and Access Road	$0.05	$0.05	$0.05
General Site Services	$0.62	$0.62	$0.62
General and Administration	$0.25	$0.25	$0.25
TOTAL Opex	**$6.09**	**$6.66**	**$6.32**

Table 25-5: C1 costs and AISC costs

	Phase 1	Phase 2	Total
	Yr 1 - 21	Yr 22 - 35	LOM
C1 costs ($/lb Ni)	$2.61	$2.94	$2.74
C1 cost ($/metric tonne Ni)	$5,753	$6,488	$6,038
AISC cost ($/lb Ni)	$3.13	$3.11	$3.12
AISC cost ($/metric tonne Ni)	$6,897	$6,867	$6,885

25.6 Economic Analysis

Table 25-6 presents the results of the Economic Analysis for the Project, based on the assumptions and cash flow projections presented previously. Taxation calculations were provided by FPX. Table 25-7 presents the results of the post-tax economic analysis.

Table 25-6: Economic analysis results (pre-tax)

IRR = 22.5% Payback = 3.5 years Discount Rate	NPV (M$)
0%	$13,656 M
5%	$5,003 M
8%	**$2,927 M**
10%	$2,069 M

Table 25-7: Economic analysis results (post-tax)

IRR = 18.3% Payback = 4.0 years Discount Rate	NPV (M$)
0%	$8,725 M
5%	$3,091 M
8%	**$1,721 M**
10%	$1,149 M

25.7 Conclusion

The Project, as presented in this PEA Report, is conceptual in nature and needs to be further developed at a PFS level.

25.7.1 Project Risks and Opportunities

A high-level risk register was initiated during this PEA for FPX to use as an internal planning and risk management tool to be carried through and developed in more detail during the next study stages. The following list describes the key risks that were identified which can have a significant impact on the results of this PEA Study. Recommendations for mitigating these risks are proposed in Chapter 26 of this Report.

- Designs of the external TSF, pit rim dike, waste dumps, open pit mine, surface water management infrastructure and process plant areas were not supported by any foundation geotechnical data. For this PEA, typical dike slopes for engineered fill were assumed, with a nominal sub-excavation, but poor foundation conditions could significantly increase cost or cause the relocation of the structure. Conversely, good foundation conditions could lead to reduced excavation and fill requirements and therefore reduced costs.
- There is currently no geotechnical data for the pit area. For the PEA, an overall pit wall angle of 45 degrees is assumed for the ultimate pits which may need to be adjusted based on the results of the geotechnical data. The change in pit wall angle will have an impact to the overall costs, footprint area, and strip ratio.
- Borrow material for the construction of the external TSF starter dike and the pit rim dike were assumed to be sourced from within the open pit mine. This study assumed that 80% of the material excavated from the pit (overburden and waste rock) is be suitable for TSF starter dike construction. Should more material (or less material) be suitable, development costs will be impacted.
- Coarse sand generated from cycloning the tailings produced by the primary magnetic separation circuit (300 µm primary grind size), was assumed to be of sufficient quantity and quality to be used in the construction of the external TSF sand cells. Should this material not be suitable, the TSF retention capacity and/or the primary grind size could be impacted resulting in higher costs and lower Ni recovery if a coarser grind is required.
- The process design criteria for grinding and primary magnetic concentration at 300 µm was based on limited testwork data. Ore hardness is a critical parameter for estimating grinding power and circuit design which can have a significant impact on throughput, capital and operating costs.
- The conceptual project execution schedule presented in Chapter 24 of this Report was based on an optimized environmental permitting process based on minimum statutory timelines. Should there be delays in compiling data or permit related activities take longer than planned, there is a risk that environmental permitting will be on the critical path of the schedule.

26. RECOMMENDATIONS

Based on the results of this PEA Study, BBA recommends that a Pre-Feasibility Study (PFS) be undertaken on the Baptiste Project in order to advance the Project to its next phase. The proposed PFS would serve as a stage-gate for FPX to determine if the Project should be subsequently advanced further. The PFS is intended to confirm the results of this PEA, to a higher level of resource definition and cost estimation accuracy, supported by more developed engineering and design and so de-risking the Project to the next study stage. The following recommendations are made for work to be undertaken as part of the PFS Study in order reduce Project uncertainty and to mitigate risks previously outlined in Chapter 25 of this Report, as well as to evaluate opportunities to improve the Project. Work related to environmental permitting is also recommended in parallel to the PFS in order to maintain the overall project implementation schedule presented in Chapter 24 of this Report.

Geology and Drilling

Database maintenance: Review of the Baptiste Project drilling database for the purposes of this report recognized that the QA/QC and re-assay results for drill core have not been incorporated into the Project's database and exist as stand-alone spreadsheets. The recommended improvements to the database include rebuilding the Project's drillhole database from original assay certificates and incorporating existing QA/QC data into the Project's drillhole database.

QA/QC: Assays of CRMs used on the Project averaged 0.5 to 1 standard deviation higher than their certified mean, suggesting that calculated DTR Ni grades for the Baptiste Deposit could have inaccuracies of up to 4%. A re-assay program is recommended to evaluate positive bias seen in CRM assays in addition to determining the data adequacy of DTR Fe and Co for future use in resource estimates.

Drilling: Additional drilling on the Decar Property is recommended for the Baptiste Deposit to improve the certainty of near surface Inferred Resources greater than 0.13% DTR Ni. A total of 2,725 m of drilling on ten holes is recommended. Recommended drillhole locations are shown on Figure 26-1.

Metallurgical Testwork

Grindability: Specific testwork on representative composite samples related to grindability (crushing, HPGR, ball milling and regrinding) should be performed.

Concentration: Magnetic concentration and flotation testwork should be undertaken on representative composite samples in order to confirm grade/recovery relationship. This should include testwork to confirm DTR Ni recovery at various primary grinds with varying magnetic intensity. Magnetic concentration testwork should be performed with a complete DTR Ni and Fe balance. Also, an analysis of primary magnetic tailings PSD and characteristics of sand tailings and fine tailings (compaction, hydraulic conductivity, strength parameters, ML/ARD potential) should be undertaken in order to confirm TSF sand dike design parameters.

Leaching: More detailed leaching tests should be performed to evaluate metallurgical performance and costs related to production of Ni sulphate.

Additional testwork: Thickening, rheology and filtration testwork should be undertaken.

Variability assessment: To assess the deposit's heterogeneity, variability sampling should be completed. Approximately 25 samples are recommended to assess potential variability within the Baptiste Deposit with respect to crushing, grinding and recovery.

Product marketability sample: The metallurgical testwork should be undertaken in consideration that a 5 to 10 kg representative final FeNi product sample needs to be generated. This will allow FPX to undertake more meaningful and credible discussions with potential users.

Geotechnical

Site surveys: A test pit / geotechnical drilling program is recommended to be carried out in the area of the external TSF, waste dumps, pit, water management dams and area surrounding the pit to collect geotechnical data in order to mitigate risks associated with lack of data on foundation conditions.

Pit slope stability: There is currently no geotechnical and hydrogeology data for the pit area. The projected pit walls are very high (700 m) and therefore small changes in angle could have a significant impact on the mine plan. Core logging and characterization is required to allow for development of rock mass strength models and development of pit wall designs including bench configurations. The hydrogeology data will help determine the amount of ground water the mine will need to manage during operation as well as potential requirements for dewatering related to wall stability. Geotechnical and hydrological information on the pit walls will be required for the PFS.

Other: Geotechnical data from the pit and tailings footprint areas will be required to more accurately determine borrow material suitability for construction of the TSF starter dam and other site infrastructure. Also, additional surveys for site infrastructure such as process plant location and roads are recommended.

Environmental, Permitting and Community

Baseline studies: Further environmental baseline studies should be completed, as recommended in Chapter 20 of this Report. This activity is a critical element of the EA and permitting process and project implementation schedule.

EA planning: A gap analysis (current status vs final requirements) should be performed and a comprehensive action plan should be developed to undertake the permitting process in parallel to the PFS.

Community Engagement: It is recommended that FPX pursue engagement activities with Indigenous groups early in the PFS as a parallel activity.

Product Marketing

FeNi briquette: FPX should undertake a more formal product marketability analysis and directly approach potential users to validate assumptions made in this PEA regarding payability and potential impact of deleterious elements.

Fe concentrate: FPX should undertake exploratory discussions with potential users of the Fe concentrate in order to evaluate product value. This can help guide metallurgical testwork aimed at upgrading the product to enhance value. A more detailed logistics / cost analysis for getting the product to market should be included.

Studies and Other PFS Elements

Trade-off studies: Early in the PFS risks and opportunities identified in the PEA should be reviewed and required trade-off studies should be performed to carry more optimal solutions into the PFS.

Third-party discussions: Early in the PFS, it is recommended that FPX begin exploratory discussions with any potential third party related to off-site infrastructure.

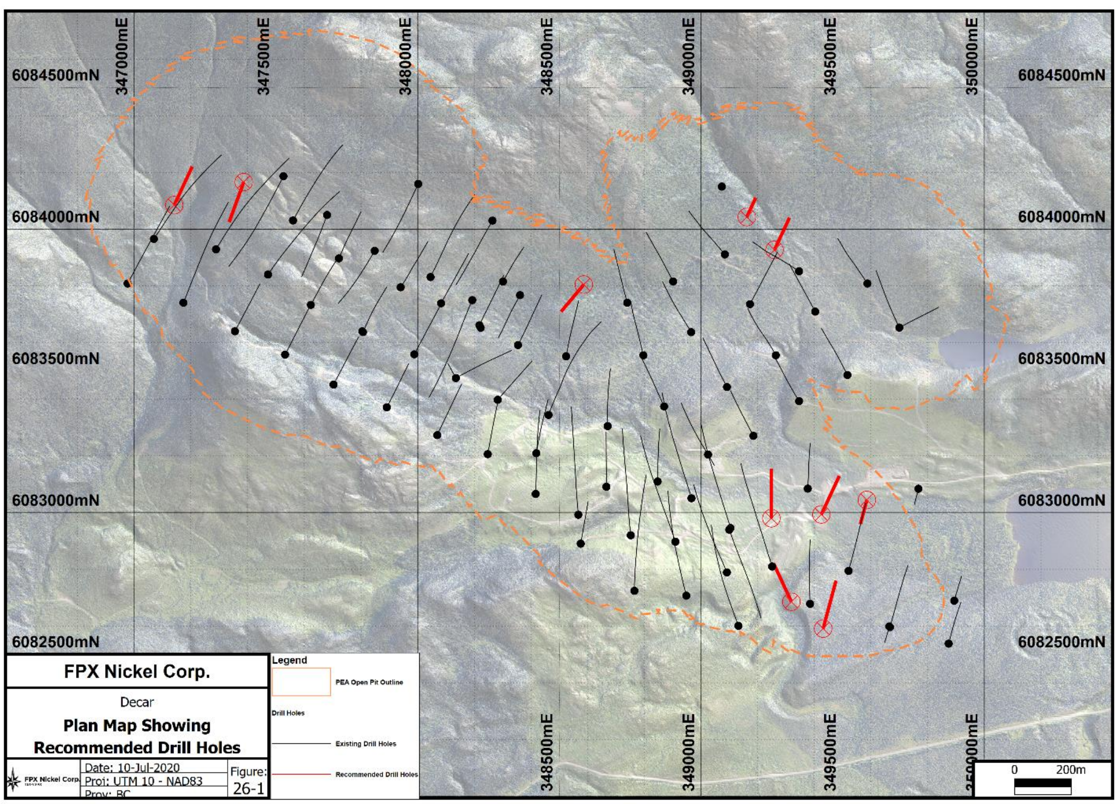

Figure 26-1: Plan map showing recommended drillholes Source: Equity, 2020

26.1 Budget for Next Project Stage

The estimated budget for undertaking the work required to complete the PFS is summarized in Table 26-1. FPX should obtain firm pricing for these activities based on a specific scope of work in order to better estimate its next project phase financing requirements. The PFS, and related activities, are expected to take about 12 – 18 months to complete.

Table 26-1: Estimated required budget for next study phase

Activity	Description	Cost (K$ US)
Geology and Drilling	Improve the Project's database, update near-surface inferred resources	$ 880
Metallurgical Testwork	Drilling and sample collection, grindability/magnetic/flotation recovery, leaching, variability assessment, thickening, FeNi product sample	$ 1,400
Geotechnical	External TSF, waste dumps, pit walls, site	$ 980
Environmental and Community	Baseline study, community engagement activities	$ 2,300
Studies	Pre-Feasibility Study	$ 1,500
Owner	PFS owner's team costs and expenses	$ 1,140
TOTAL Opex		**$ 8,200**

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Appendix A: Tenure Information

Tenure Number	Claim Name	Owner	Issue Date	Good to Date	Area (ha)	Overlap Description
559615	WILL 1	FPX (100%)	31-May-07	14-Nov-28	464.76	Legacy claim
559616	WILL 2	FPX (100%)	31-May-07	14-Nov-28	464.76	Legacy claim
559617	WILL 3	FPX (100%)	31-May-07	14-Nov-28	464.76	
559618	WILL 4	FPX (100%)	31-May-07	14-Nov-28	446.35	
575674	WILL 5	FPX (100%)	08-Feb-08	14-Nov-28	446.49	
575675	WILL 6	FPX (100%)	08-Feb-08	14-Nov-28	446.63	
575677	WILL 7	FPX (100%)	08-Feb-08	14-Nov-28	465.19	
575678	WILL 8	FPX (100%)	08-Feb-08	14-Nov-28	464.95	
575679	WILL 9	FPX (100%)	08-Feb-08	14-Nov-28	464.72	
575680	WILL 10	FPX (100%)	08-Feb-08	14-Nov-28	465.19	
575681	WILL 11	FPX (100%)	08-Feb-08	14-Nov-28	446.38	
575682	WILL 12	FPX (100%)	08-Feb-08	14-Nov-28	297.65	Legacy claim
575683	WILL 13	FPX (100%)	08-Feb-08	14-Nov-28	390.40	
575684	WILL 14	FPX (100%)	08-Feb-08	14-Nov-28	223.37	
575686	WILL 15	FPX (100%)	08-Feb-08	14-Nov-28	316.24	
594247	BAP 1	FPX (100%)	14-Nov-08	14-Nov-28	446.78	
594248	BAP 2	FPX (100%)	14-Nov-08	14-Nov-28	335.14	
594249	BAP 3	FPX (100%)	14-Nov-08	14-Nov-28	465.43	
594250	BAP 4	FPX (100%)	14-Nov-08	14-Nov-28	446.70	
594251	BAP 5	FPX (100%)	14-Nov-08	14-Nov-28	390.88	
594252	KAR 1	FPX (100%)	14-Nov-08	14-Nov-28	464.53	
594254	KAR 2	FPX (100%)	14-Nov-08	14-Nov-28	464.29	
594255	KAR 3	FPX (100%)	14-Nov-08	14-Nov-28	464.29	Mineral reserve, district lot

Tenure Number	Claim Name	Owner	Issue Date	Good to Date	Area (ha)	Overlap Description
594256	KAR 4	FPX (100%)	14-Nov-08	14-Nov-28	427.27	Mineral reserve, district lot
594257	KAR 5	FPX (100%)	14-Nov-08	14-Nov-28	371.63	Mineral reserve, district lot
594258		FPX (100%)	14-Nov-08	14-Nov-28	464.52	
594259	KAR 7	FPX (100%)	14-Nov-08	14-Nov-28	445.97	Legacy claim
594260	KAR 8	FPX (100%)	14-Nov-08	14-Nov-28	297.19	
594262	KAR 9	FPX (100%)	14-Nov-08	14-Nov-28	408.72	
594263	KAR 10	FPX (100%)	14-Nov-08	14-Nov-28	389.92	
602564		FPX (100%)	14-Apr-09	14-Nov-28	18.58	
602566		FPX (100%)	14-Apr-09	14-Nov-28	148.66	
603803	VAN 1	FPX (100%)	03-May-09	14-Nov-28	464.51	District lot
669586	BAP 6	FPX (100%)	16-Nov-09	14-Nov-28	260.51	Legacy claim
669625	BAP 7	FPX (100%)	16-Nov-09	14-Nov-28	446.96	
669645	BAP 8	FPX (100%)	16-Nov-09	14-Nov-28	446.94	
669665	BAP 9	FPX (100%)	16-Nov-09	14-Nov-28	446.91	
839601	MID 1	FPX (100%)	03-Dec-10	03-Dec-28	74.40	
839604	MID 2	FPX (100%)	03-Dec-10	03-Dec-28	446.66	District lot
839607	MID 3	FPX (100%)	03-Dec-10	03-Dec-28	427.88	District lot
839610	MID 4	FPX (100%)	03-Dec-10	03-Dec-28	465.28	Mineral reserve, district lot
839615	MID 5	FPX (100%)	03-Dec-10	03-Dec-28	427.90	Mineral reserve, district lot
839617	MID 6	FPX (100%)	03-Dec-10	03-Dec-28	464.90	Mineral reserve, district lot
839618	MID 7	FPX (100%)	03-Dec-10	03-Dec-28	464.75	Mineral reserve, district lot
839620	MID 8	FPX (100%)	03-Dec-10	03-Dec-28	427.39	Mineral reserve, district lot
839621	MID 9	FPX (100%)	03-Dec-10	03-Dec-28	464.33	Mineral reserve, district lot
839622	MID 10	FPX (100%)	03-Dec-10	03-Dec-28	148.55	Mineral reserve, district lot

Tenure Number	Claim Name	Owner	Issue Date	Good to Date	Area (ha)	Overlap Description
895893	NEY 1	FPX (100%)	02-Sep-11	02-Sep-28	446.56	Legacy claim
895899	NEY 2	FPX (100%)	02-Sep-11	02-Sep-28	465.29	Legacy claim
895902	NEY 3	FPX (100%)	02-Sep-11	02-Sep-28	446.92	
895904	NEY 4	FPX (100%)	02-Sep-11	02-Sep-28	465.52	
895905	NEY 5	FPX (100%)	02-Sep-11	02-Sep-28	390.91	
895907	NEY 6	FPX (100%)	02-Sep-11	02-Sep-28	465.54	
895909	NEY 7	FPX (100%)	02-Sep-11	02-Sep-28	447.11	Provincial Park
895910	NEY 8	FPX (100%)	02-Sep-11	02-Sep-28	447.16	Provincial Park
895911	NEY 9	FPX (100%)	02-Sep-11	02-Sep-28	465.76	Provincial Park
895912	NEY 10	FPX (100%)	02-Sep-11	02-Sep-28	465.74	Provincial Park
895913	NEY 11	FPX (100%)	02-Sep-11	02-Sep-28	335.31	Provincial Park
895914	NEY 12	FPX (100%)	02-Sep-11	02-Sep-28	446.56	
1013225		FPX (100%)	26-Sep-12	26-Sep-23	632.34	District lot
1053094	BT GROUP	FPX (100%)	12-Jul-17	12-Jul-20*	111.63	
1069701	NEY 13	FPX (100%)	16-Jul-19	16-Jul-20*	111.67	
Total	**62**				**24740.26**	
*Protected to 31-Dec-21 by Order of the Chief Gold Commissioner						

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