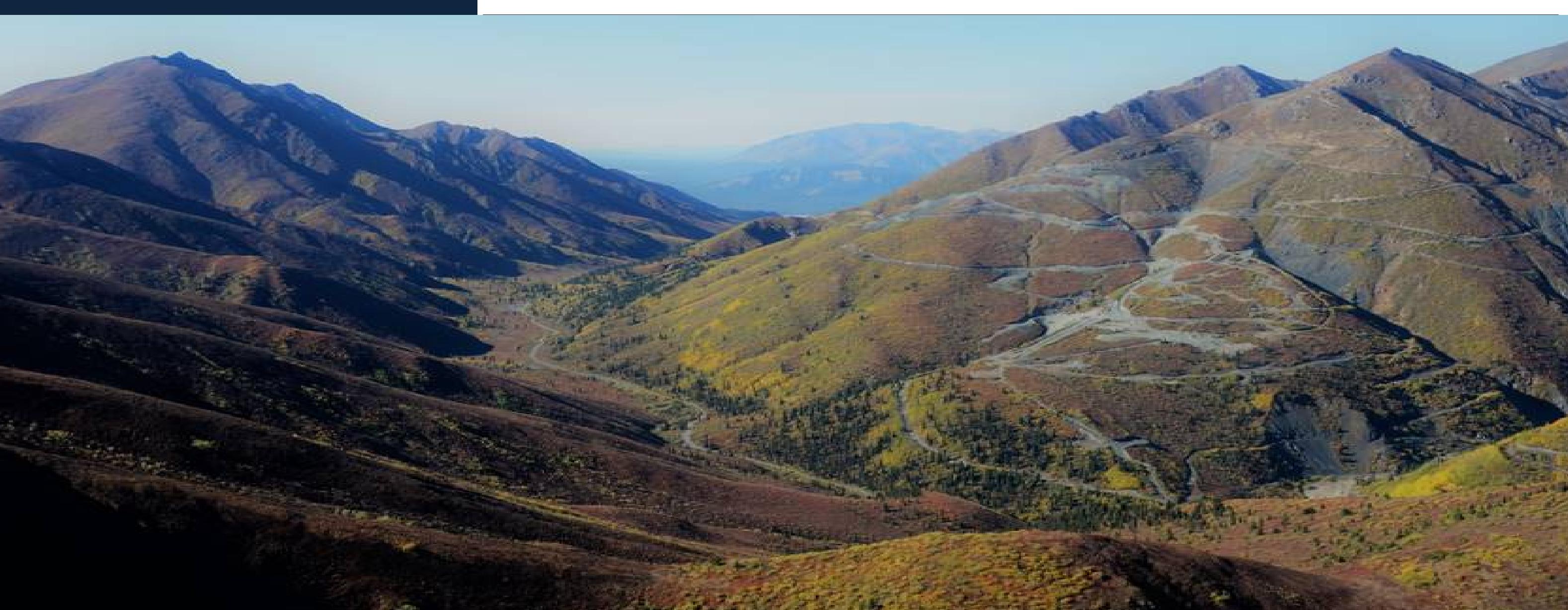
Nickel Creek Platinum Corp.

Ni-Cu-PGM Project
2018 NI 43-101 Resource Update
Yukon, Canada

Effective Date: September 25, 2018



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IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Nickel Creek Platinum Corp. (Nickel Creek or the Company) by AGP Mining Consultants Inc. (AGP) and Independent Mining Consultants (IMC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AGP and IMC's services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Nickel Creek subject to terms and conditions of their contract with AGP and IMC. Their contract permits Nickel Creek to each file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101 Standards of Disclosure for Mineral Projects. Except for the purposes legislated under applicable Canadian provincial, territorial, and federal securities law, any other use of this report by any third party is at that party's sole risk.

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Units of Measure

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm3
Cubic feet per minute	cfm
Cubic feet per second	ft3/s
Cubic foot	ft3
Cubic inch	in3
Cubic metre	m3
Cubic yard	yd3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	C$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t

Greater than	>
Hectare (10,000 m2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	“
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/min
Megabytes per second	Mb/sec
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm

Million	M
Million bank cubic metres	Mbm3
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	sec
Specific gravity	SG
Square centimetre	cm2
Square foot	ft2
Square inch	in2
Square kilometre	km2
Square metre	m2
Thousand tonnes	kt
Three Dimensional	3D
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm3
Total	T
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt

Abbreviations and Acronyms

Absolute Relative Difference	ABRD
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Alpine Tundra	AT
Atomic Absorption Spectrophotometer	AAS
Atomic Absorption	AA
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
British Columbia	BC
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEA Act
Canadian Environmental Assessment Agency	CEA Agency
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Canadian National Railway	CNR
Carbon-in-leach	CIL
Caterpillar's® Fleet Production and Cost Analysis software	FPC
Closed-circuit Television	CCTV
Coefficient of Variation	CV
Copper equivalent	CuEq
Counter-current decantation	CCD
Cyanide Soluble	CN
Digital Elevation Model	DEM
Direct leach	DL
Distributed Control System	DCS
Drilling and Blasting	D&B
Environmental Assessment	EA
Environmental Management System	EMS
Environmental Monitor	EM
Flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
General and administration	G&A
Gold equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma	ICP
Inspectorate America Corp.	Inspectorate
Interior Cedar – Hemlock	ICH

Internal rate of return	IRR
Inverse Distance Squared	ID^2
Inverse Distance Cubed	ID^3
Land and Resource Management Plan	LRMP
Lerchs-Grossman	LG
Life-of-mine	LOM
Light Detection and Ranging	LiDAR
Load-haul-dump	LHD
Locked cycle tests	LCTs
Loss on Ignition	LOI
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
Metres East	mE
Metres North	mN
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
Mini Pilot Plant	MPP
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Invoice Value	NIV
Net Present Value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS
Ordinary Kriging	OK
Organic Carbon	org
Potassium Amyl Xanthate	PAX
Predictive Ecosystem Mapping	PEM
Preliminary Assessment	PA
Preliminary Economic Assessment	PEA
Qualified Persons	QPs
Quality Assurance	QA
Reverse Cirrculation Drilling	RC
Rhenium	Re
Rock Mass Rating	RMR '76
Rock Quality Designation	RQD
SAG Mill/Ball Mill/Pebble Crushing	SABC
Semi-autogenous Grinding	SAG
Standards Council of Canada	SCC
Static Pressure Test	STP

Forward Looking Statements

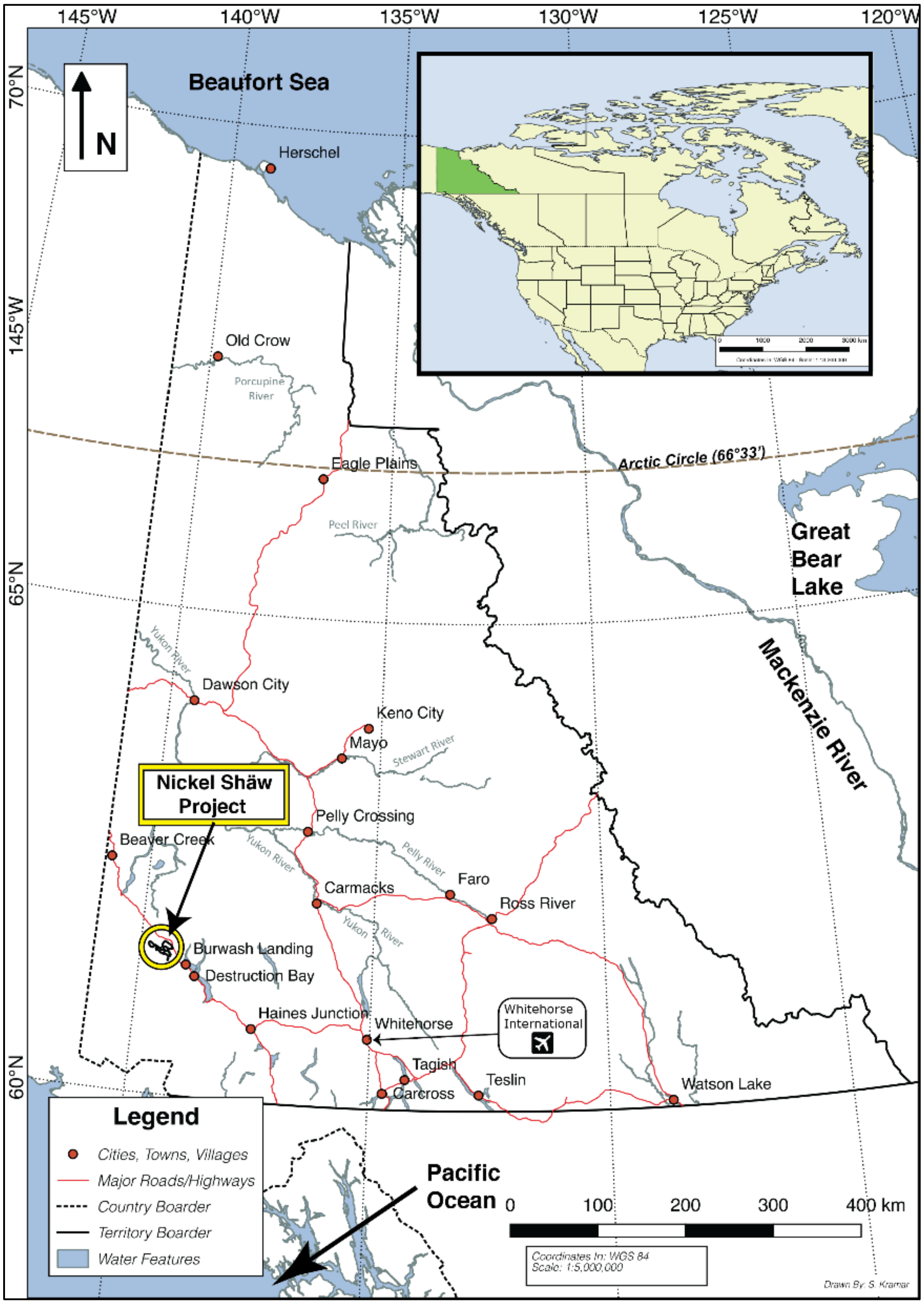
This Technical Report, including the economics analysis, contains forward-looking statements within the meaning of the United States Private Securities Litigation Reform Act of 1995 and forward-looking information within the meaning of applicable Canadian securities laws. While these forward-looking statements are based on expectations about future events as at the effective date of this Report, the statements are not a guarantee of Nickel Creek Platinum Corp. future performance and are subject to risks, uncertainties, assumptions and other factors, which could cause actual results to differ materially from future results expressed or implied by such forward-looking statements. Such risks, uncertainties, factors and assumptions include, amongst others but not limited to metal prices, mineral resources, smelter terms, labour rates, consumable costs and equipment pricing. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements.

1 SUMMARY

This document reports the mineral resources at the Nickel Shäw Ni-Cu-PGM Project (the Project) in Yukon Territory, Canada. The Project is 100% owned by Nickel Creek Platinum Corp. (Nickel Creek or the Company). The Company assembled a team of contractors and corresponding Qualified Persons (QPs) to assemble this statement of mineral resources. The QPs and their affiliations are summarized in Section 2.0.

The Project is located approximately 317 km northwest of Whitehorse in southwestern Yukon, at an approximate latitude of 61°28’N, and longitude of 139°32’W (Figure 1-1). It is accessible by a 14-km road from the paved all-weather Alaska Highway to the northeast. The nearest villages are Destruction Bay and Burwash Landing, both located on the Alaska Highway. The Project lies within the Kluane First Nation “core area” as defined under the Umbrella Final Agreement between the Government of Canada, Government of Yukon, and Council for Yukon Indians (now Council for Yukon First Nations).

Figure 1-1: Project Location Map



Source: Nickel Creek, 2018

The Project contains potentially economic values of nickel, copper, platinum, palladium, cobalt, and gold. It is located within the Insular Superterrane that is comprised of island arc and ocean floor volcanic rocks overlain by thick assemblages of oceanic sedimentary rocks that are Pennsylvanian to Permian in age. Those units were intruded by ultramafic units of the Quill Creek Complex. The mineralization occurs within the Quill Creek Complex of variably serpentenized ultramafic-gabbroic units.

The ultramafic ore hosts are broadly segregated into peridotite, clinopyroxenite, and gabbro. The mineralization of the Wellgreen Deposit strikes nearly east-west for roughly 2 km, with a width of 200 to 400 m in the north-south direction and varies in depth up to 650 m. The ultramafic intrusion is in contact with barren metasedimentary rocks and volcaniclastic units to the north. The highest-grade mineralization lies at that contact within the ultramafic units. The most abundant economic minerals are pentlandite-pyrrhotite and chalcopyrite (see

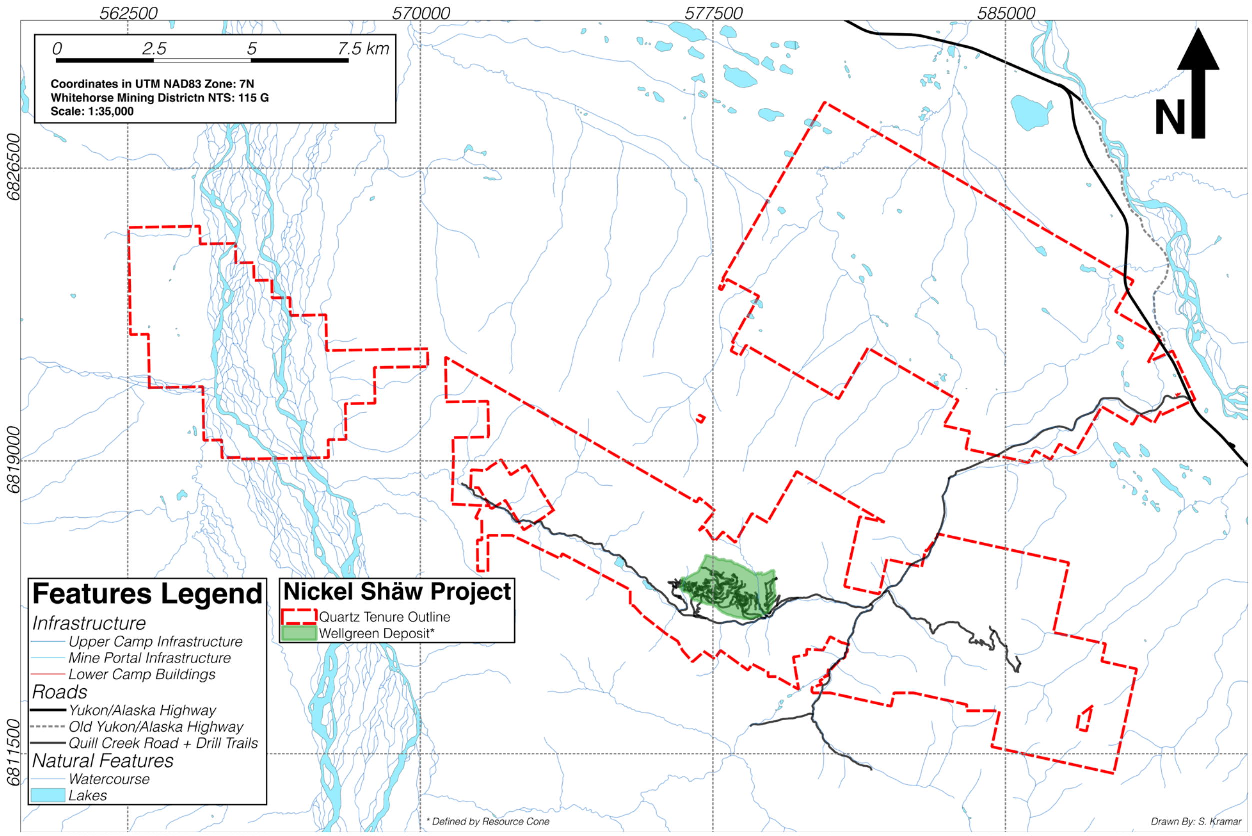


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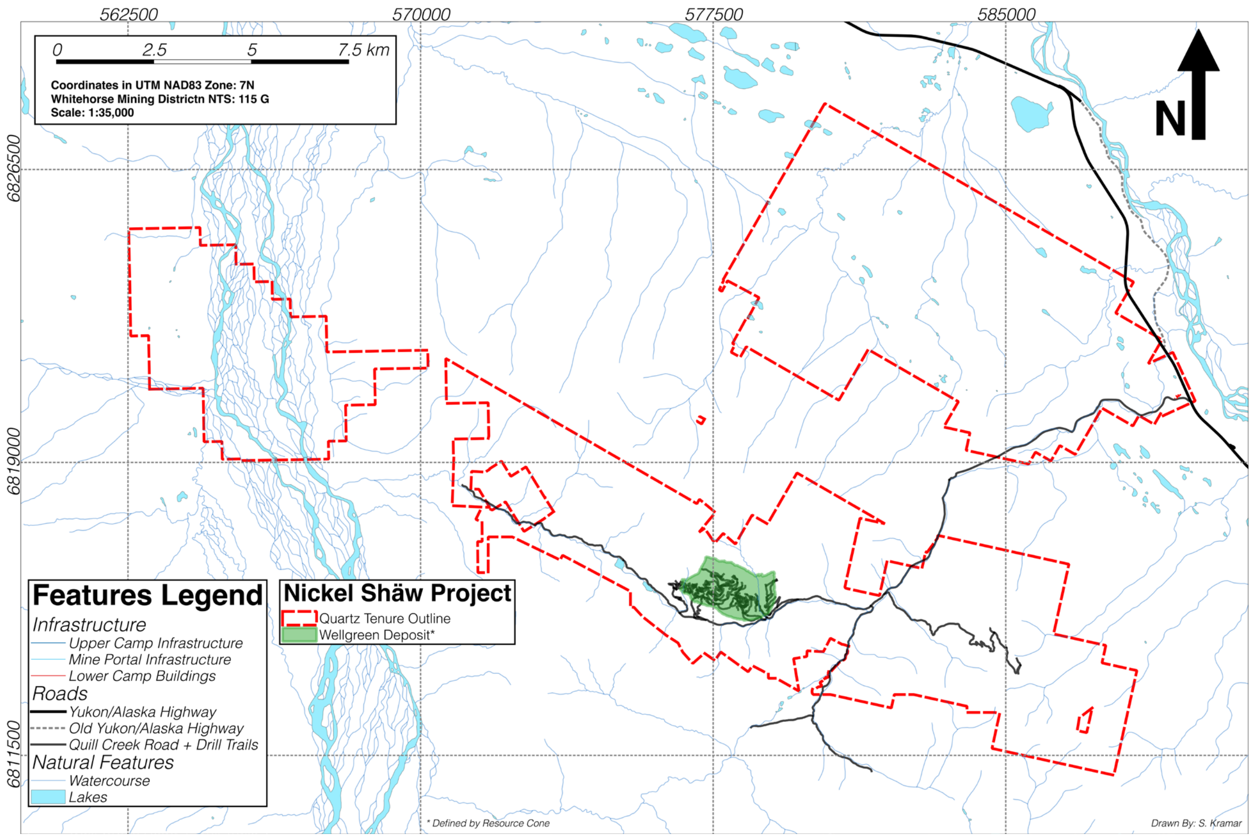
Figure 1-2: Wellgreen Deposit Area



Source: Nickel Creek, 2018

Nickel Creek previously published mineral resources on June 26, 2017. Since that report, the Company drilled an additional 15 diamond drill holes during 2017 comprising a combination of infill and infill/metallurgical holes. Several of these holes did not reach the planned target depths due to poor ground conditions. These holes were determined to not make a material change and were not incorporated into the new model.

A considerable amount of technical work has been performed on the Project over the past 24 months. Due to the changes in the resource estimate, improved understanding of the geologic model, advanced metallurgical testwork, a review of the plant and tailings facilities locations, and other factors that have changed since the publication of the PEA filed on SEDAR by the Company on March 19, 2015 (the 2015 PEA), the Company advised in a news release dated June 26, 2017, that the 2015 PEA has become outdated and should not be relied upon.

John Marek of Independent Mining Consultants Inc. (IMC) is the QP for the mineral resource supported by this technical report and has studied the reliability of the current and historic drilling at the Project. As a result, he formed the opinion that the historic assay information prior to 1987 should not be used for the estimation of mineral resources.

The resulting data base includes drilling completed during 1987 through 2016. The 1987 to 1988 drilling has been re-sampled and assayed using the same techniques currently in use at Nickel Creek. The mineral resource is based on a total of 386 holes, containing roughly 23,730 assays for the economic minerals from 62,799 m of drilling.

The mineral resource for the Project was developed using a computer-based block model of the deposit. The block model was assembled based on the drill hole data base and interpreted geology by Nickel Creek Chief Geologist James Berry after review and verification of that information by the QP (John Marek). Mineral resources were estimated using the block model and the Lerchs-Grossman open pit software to establish the component of the deposit with reasonable prospects of economic extraction. John Marek of IMC acted as the QP for the development of the block model and the estimation of mineral resources.

The final statement of mineral resources reflects material that is inside of a computer-generated pit. The purpose of using Lerchs-Grossman is to provide some assurance the mineral resource has "reasonable prospects of economic extraction" as required by CIM best practices. The economic assumptions used for that pit are broadly summarized in the footnotes below the table.

The block model was assembled using blocks that are 10 x 10 x 10 m. Grade domain boundaries were evaluated and respected where appropriate during the estimation process. The Inverse Distance squared (ID^2) method was applied for block grade estimation within the respective grade domains.

A mineral resource is a concentration or occurrence of diamonds, natural solid inorganic material or natural solid fossilized organic material - including base and precious metals, coal, and industrial minerals in or on the earth's crust in such form and quantity, and of such a grade or quality, that it has reasonable prospects for economic extraction. The location, quantity, grade, geologic characteristics, and continuity of a mineral resource are known, estimated, or interpreted from specific geological evidence and knowledge.

The phrase 'reasonable prospects for economic extraction' implies a judgment by the QP in respect to the technical and economic factors likely to influence the prospects of economic extraction. A mineral resource is an inventory of mineralization that, under realistically assumed and justifiable technical and economic conditions, might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

The current process concept envisions a large-scale process facility that produces and markets a bulk nickel concentrate and a bulk copper concentrate. To capture the potential economic contributions of multiple metals and process recovery formulas, a Net Smelting Return (NSR) value was estimated for each mineralized block and used for cut-off application. The internal or marginal mill cut-off is equal to the sum of the process, G&A, and tailing management operating costs, because the NSR value considers process recoveries, assumed smelter terms, and concentrate transport costs. The process recoveries vary by head grade and sulphur content.

Table 1-1 summarizes the resulting mineral resources. The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be realized, or that they will convert to mineral reserves. John

Marek of IMC is the QP for this statement of mineral resources. Currently there are no mineral reserves at the Project.

The risks associated with this statement of mineral resources include metal price impacts, changes to process recovery as more testing is complete, and permit risks that are typical of any North American mineral development.

Mineral resource classification was determined based on the number of drill holes, number of composites, and the average distance of composites to the estimated block. Classification was completed, by reference to the definitions within NI 43-101 and the CIM Definition Standards.

Figure 1-3 is an illustration of the mineral resource blocks within the resource pit geometry looking to the northwest.

Table 1-1: Project Mineral Resources September 25th, 2018

										Contained Metal					
		Nickel	Copper	Cobalt	Platinum	Palladium	Gold	Mg	Sulfur	Ni	Cu	Co	Pt	Pd	Au
Class	Ktonnes	%	%	%	gm/t	gm/t	gm/t	%	%	M Lbs	M Lbs	M Lbs	K Ozs	K Ozs	K Ozs
Measured	93,300	0.25	0.17	0.015	0.262	0.244	0.054	15.7	0.85	514	350	31	786	732	162
Indicated	230,100	0.27	0.15	0.015	0.249	0.259	0.043	16.8	0.74	1,370	761	76	1,842	1,916	318
Total M+I	323,400	0.26	0.16	0.015	0.253	0.255	0.046	16.5	0.77	1,884	1,111	107	2,628	2,648	480
Inferred	108,100	0.29	0.15	0.016	0.256	0.279	0.040	16.2	0.72	691	357	38	890	970	139

Notes:
Mineral Resources do not have demonstrated economic viability
The QP for the Mineral Resource is John Marek RM-SME, Professional Engineer Yukon Territory
Average grade calculations on this table are impacted by rounding.
Tonnages are reported in units of 1,000 metric tonnes (Ktonnes)
Contained Base Metal reported in units of 1,000,000 lbs, M Lbs
Contained Precious Metal reported in units of 1,000 troy ounces, K Ozs
Metal Prices and Summarized Costs for Resource Determination in USD:
Nickel: $8.25/lb, Copper: $3.00/lb, Cobalt: $24.00/lb
Platinum: $1,200/troy oz, Palladium: $900/troy oz, Gold: $1,300/troy oz
Net of Smelting (NSR) cut-off grades range from $11.51 to $11.74 U.S. Dollars (Bulk vs Split Con)
Mining Cost vary by bench, separately for ore and waste
Average Mining Costs within the resource pit are $1.48 USD per total tonne moved
The average strip ratio for the resource pit is: 2.97 to 1
Average calculated process recoveries combining the bulk and split concentrates approach:

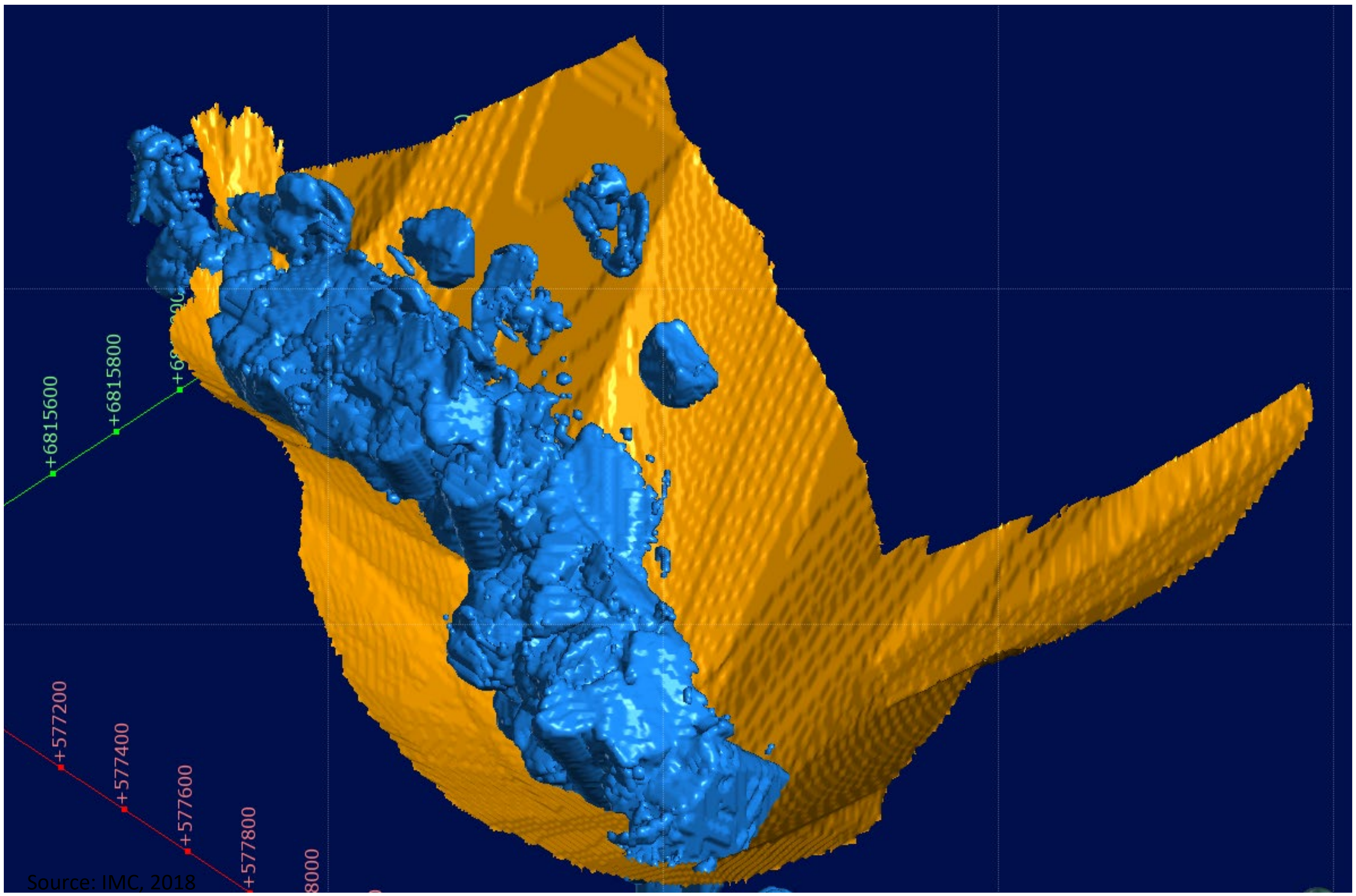
Ni	Cu	Co	Pt	Pd	Au
48.0%	62.2%	60.0%	47.8%	54.0%	47.1%

Average Smelting + Transport Costs, and Losses in Terms of Cost Per Unit in Concentrate:

Ni/lb	Cu/lb	Co/lb	Pt/oz	Pd/oz	Au/oz
$3.26	$1.14	$15.68	$578.89	$434.50	$1,179.00

Slope Angles vary from 33 to 44 Degrees

Figure 1-3: Mineral Resource in Resource Pit Geometry ($11.51 USD NSR/tonne) 315 NW & Down 41 Degrees



Source: IMC, 2018

1.1 Conclusions and Recommendations

The mineral resources at the Project were estimated using conventional mineral resource modeling techniques based on 386 reliable drill holes. The mineral resources are contained within a pit geometry with reasonable prospects of economic extraction.

There is potential to add to the deposit with additional drilling and there are additional exploration targets on the Project controlled by Nickel Creek to the east and west.

2 INTRODUCTION

2.1 General

This report was prepared at the request of the Company to provide an updated mineral resource on the Project.

This report was prepared in accordance with the standards and requirements set out in the Canadian Securities Administrators National Instrument 43-101 'Standards of Disclosure for Mineral Projects'.

2.2 Qualified Persons

This report was prepared under the direct supervision of:

John Marek, P.E. – President of Independent Mining Consultants, Inc. (IMC), is a Registered Member of the Society of Mining Engineers, and a registered professional engineer in Yukon Territory. Mr. Marek visited the Project site from April 25th to 27th, 2017 to review drill core logging and sampling procedures, verify drill hole collar locations, and gain knowledge of the geological setting of the deposit. Mr. Marek is the QP for the estimation of mineral resources. Mr. Marek's responsibility excludes the portion of the report dealing with land title, permits, legal, political, environmental, socio-economic, and tax matters as indicated in Section 3 titled "Reliance on Other Experts."

Gordon Marrs, P. Eng. – Consulting Metallurgist with XPS Expert Process Solutions (XPS), is a registered professional engineer in the province of Ontario with extensive metallurgical experience on projects worldwide. Mr. Marrs did not visit the Project site. Mr. Marrs supervised all the testwork and results contracted by the Nickel Creek metallurgical laboratories. He is responsible for Sections 13, and portions of the Summary, Section 25, and Section 26 that pertain to the metallurgical aspects of the Project.

Andy Holloway, CEng., P. Eng. – Principal Process Engineer with AGP Mining Consultants (AGP) is a registered professional engineer in the province of Ontario with extensive metallurgical experience on projects worldwide. Mr. Holloway did not visit the Project site. Mr. Holloway reviewed all the testwork and results contracted by the Nickel Creek metallurgical laboratories. He is responsible for Section 17, and portions of the Summary, and Section 25 that pertain to the processing aspects of the Project.

Gordon Zurowski, P. Eng. – Principal Mine Engineer with AGP is a registered professional engineer in the provinces of Saskatchewan, Ontario, and Newfoundland with extensive mining experience worldwide. Mr. Zurowski visited the Project on April 29th and 30th, 2017 to review drill core, gain knowledge of the geologic setting of the deposit, and other potential mining and infrastructure considerations. He is responsible for Sections 2, 3, 15, 16, 18, 19, 20, 21, 22, 24 and portions of the Summary and Section 26.

All QP's listed are independent of Nickel Creek or any associated company.

2.3 Site Visits and Responsibilities

IMC and AGP have conducted site visits to the Project as shown in Table 2-1.

Table 2-1: Date of Site Visits and Areas of Responsibility

QP Name	Site Visit Dates	Area of Responsibility
John Marek	25 – 27 April 2017	Sections 1, 4 to 12, 14, 23, and 25.1
Andy Holloway	No Site Visit	Sections 17, and 25.3
Gordon Zurowski	29 – 30 April 2017	Sections 2, 3, 15,16,18,19, 20, 21, 22, 24, and 26.2
Gordon Marrs	No Site Visit	Section 13, 25.2, and 26.1

Mr. Roland Tosney of JRT Geoengineering (JRT) contributed to the geotechnical slope components of the resource shell. Mr. Zurowski accepts responsibility for the geotechnical contribution provided by JRT.

2.4 Effective Dates

The effective date of this technical report is September 25, 2018. It is noted that this technical report is based on drill data and information for the Project that is current to September 7, 2016.

2.5 Previous Technical Reports

Previous NI 43-101 technical reports on the Project are listed below:

- McCracken, T., 2011. Technical Report on the Wellgreen Ni-Cu-Pt-Pd Project, Yukon, Canada. Report to Prophecy Resource Corp. and Pacific Coast Nickel Corp. Wardrop Document No. 1055400400-REP-R0001-04. Effective Date: April 14, 2011.
- McCracken, T., 2011. Technical Report and Resource Estimate on the Wellgreen Platinum-Palladium-Nickel-Copper Project, Yukon, Canada. Report to Prophecy Platinum Corp. Wardrop Document No. 1155400200-REP-R0001-02. Effective Date: July 21, 2011.
- Carter, A., Corpuz, P., Brisson, P., McCracken, T., 2012. Wellgreen Project Preliminary Economic Assessment, Yukon, Canada. Report to Prophecy Platinum Corp. Wardrop Document No. 1193460500-REP-R0001-02. Effective Date: August 1, 2012.
- Simpson, R.G., 2014. 2014 Mineral Resource Estimate on the Wellgreen PGM-Ni-Cu Project, Yukon, Canada. Report to Wellgreen Platinum Ltd. Effective Date: September 8, 2014.
- Makarenko, M., Eggert, J., Simpson, R.G., Levy, M., Darling, G., 2015. Preliminary Economic Assessment Technical Report Wellgreen Project, Yukon, Canada. Report to Wellgreen Platinum Ltd. Effective Date: February 2, 2015.

- Marek, J., Jones, L., Zurowski, G., Mani, H., 2017 Mineral Resource Estimate on the Wellgreen Ni-Cu-PGM Project, Yukon, Canada. Report to Wellgreen Platinum Ltd. Effective Date: June 26, 2017.

These reports are filed on the SEDAR website (www.sedar.com). Background information and a portion of the technical data for this report were obtained from these reports. For the avoidance of doubt, this technical report replaces and supersedes all prior technical reports of the Company.

3 RELIANCE ON OTHER EXPERTS

AGP and IMC have followed standard professional procedures in preparing the content of this resource estimation report. Data used in this report has been verified where possible, and the report is based upon information believed to be accurate at the time of completion.

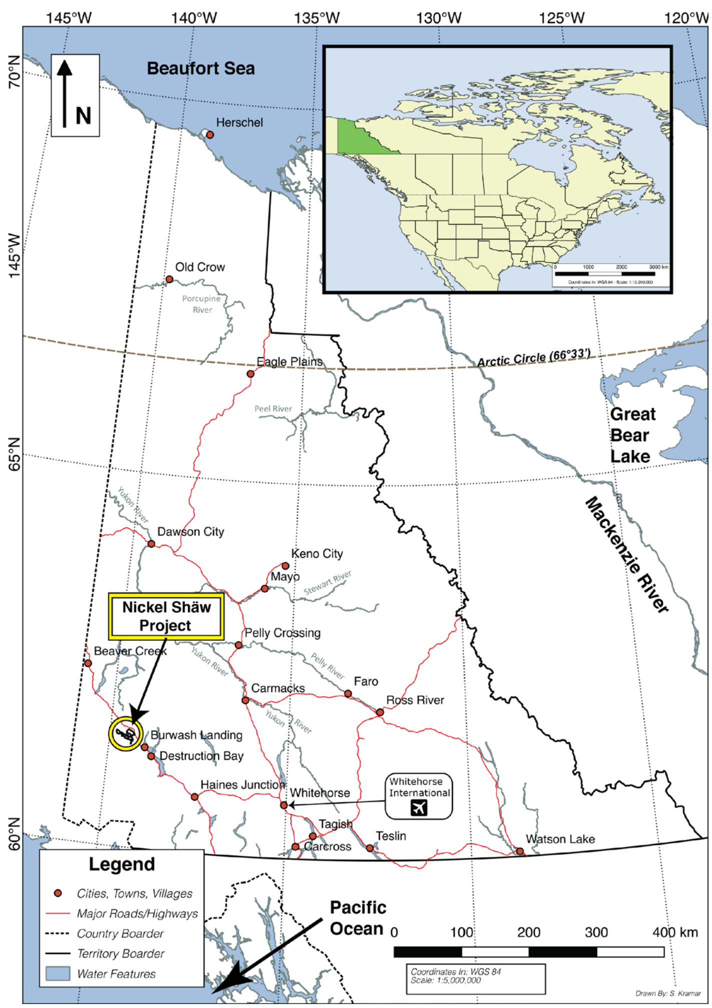
AGP and IMC have not verified the legal status, legal title to any permit, or the legality of any underlying agreements for the subject properties regarding mineral rights, surface rights, permitting, and environmental issues in sections of this technical report. AGP and IMC have relied upon information provided by Mr. James Berry, Chief Geologist for Nickel Creek, which forms the basis for Section 4 of this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located approximately 317 km northwest of Whitehorse in southwestern Yukon, at an approximate latitude of 61°28'N, and longitude of 139°32'W on NTS map sheet 115G/05, 115G/06 and 115G/11 (Figure 4-1). The Project is accessible by an all-weather 14-km road from the paved Alaska Highway to the northeast. The Project lies within the Kluane First Nation core area as defined by the Umbrella Final Agreement (UFA) with the Government of the Yukon, the Council of Yukon First Nations, and the Government of Canada.

Figure 4-1: Nickel Creek Platinum Location Map



Source: Nickel Creek, 2018

4.2 Tenure History

Prospectors W. Green, C. Aird and C. Hankins staked the first recorded mineral claims on the Project property in 1952. Underground mining operations were initiated in 1972 by Hudson Yukon Mining Co. Ltd. (Hudson-Yukon), a subsidiary of Hudson Bay Mining & Smelting Co. Ltd. (HudBay) and ceased in 1973. The Project property has changed ownership several times over the last sixty years as outlined in Section 6. Nickel Creek has had ownership of the Project property since 2011.

4.3 Mineral Tenure

The description below and the list of claims provided in Table 4-1 have been derived from records and information supplied by Nickel Creek and sourced from the Yukon Mining Recorder. A map of the Project claims is shown in Figure 4-2.

The Project is comprised of 711 mineral claims and 91 mining leases in seven (colloquially named) groups totaling 14,650 ha. The claims were staked as early as 1952 and as recent as 2017, with expiry dates that range from December 2018 to February 2036. The claims cover the known Wellgreen Deposit as well as the Quill, Burwash, Arch, and Formula targets. The resource shell is located on 30 Quartz Mining Leases (as defined below), which all have an expiry date of December 5, 2020. The Arch, Quill, Burwash, and additional Project claims/leases are located contiguous to the known deposit, whereas the Formula and Musk claims are separate packages near the contiguous claims. The Project claims are 100% owned, directly or indirectly, by Nickel Creek.

In Yukon, all work undertaken on the surface for hard rock mineral claims and leases is regulated under the Quartz Mining Act (QMA) through the Quartz Mining Land Use Regulation and is managed by the Energy, Mines and Resources branch of the Yukon Territorial Government, and claims management by the Yukon Mining Recorders Office.

A mineral claim is a parcel of land, no larger than 20.9 ha in area granted for the purposes of securing sub-surface mineral rights with the intention of hard rock exploration ultimately leading to hard rock mining. A claim also includes any ditches or water rights used for mining the claim, and all other things belonging to, or used in, the working of the claim for mining purposes. The holder of a mineral claim is entitled to all minerals found in veins or lodes, together with the right to enter on, and use and occupy, the surface of the claim for the efficient and miner-like operation of the mines and minerals contained in the claim. Continued tenure to the mineral rights is dependent upon work performed on the claim or a group of claims. Renewal of a quartz claim requires C$100 of work be done per claim, per year. Where work is not performed, the claimant may make a payment of C$100 in lieu of work.

A Quartz Mining Lease is the most secure form of mineral title in Yukon as the claims are held for a longer period of time (21 years) instead of annually. Mining leases are physically surveyed such that the tenure and boundaries cannot be called into contest. A lease is applied for when a company is contemplating production and would like to advance their claims to a longer, more secure form of tenure in anticipation of long-term land use. This relieves the company of the annual work requirement; there are however, annual rental fees of C$200 per lease. Quartz Mining Leases are issued for 21 years and can be renewed for an additional 21-year term, provided that during the original term of the lease, all conditions of the lease and provisions of the legislation have been adhered to.

Nickel Creek Platinum's interest in the Project property also consists of two surface leases issued by the Government of Canada and administered by the Government of Yukon: Lease 115G05-001 and 115G11-003, as described below and in Table 4-2.

Lease 115G05-001 covers a 69.7 ha parcel of land located near the headwaters of Nickel Creek proximal to the known Wellgreen Deposit (Figure 4-3). Northern Platinum held a lease on this same area from the early 1990s until October 31, 2011. Prior to expiration, the 21-year lease was assigned to Prophecy Platinum Corp. (now Nickel Creek Platinum), who then applied for renewal of the lease. This lease was renewed on June 1, 2013 and expires on May 31, 2034.

Table 4-1: Mineral Claims

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YF44098	ARCH	97	0905144 B.C. Ltd	5.76	2019-02-13
YF44099	ARCH	98	0905144 B.C. Ltd	14.09	2019-02-13
YF44100	ARCH	99	0905144 B.C. Ltd	20.88	2019-02-13
YF44101	ARCH	100	0905144 B.C. Ltd	14.45	2019-02-13
YF44102	ARCH	101	0905144 B.C. Ltd	20.89	2019-02-13
YD87935	ARCH	102	0905144 B.C. Ltd	3.72	2019-02-13
YD87936	ARCH	103	0905144 B.C. Ltd	20.84	2019-02-13
YD87937	ARCH	104	0905144 B.C. Ltd	18.70	2019-02-13
YD87938	ARCH	105	0905144 B.C. Ltd	20.89	2019-02-13
YD87939	ARCH	106	0905144 B.C. Ltd	20.89	2019-02-13
YD87940	ARCH	107	0905144 B.C. Ltd	20.89	2019-02-13
YD87941	ARCH	108	0905144 B.C. Ltd	19.57	2019-02-13
YD87942	ARCH	109	0905144 B.C. Ltd	20.89	2019-02-13
YD87943	ARCH	110	0905144 B.C. Ltd	13.48	2019-02-13
YD87944	ARCH	111	0905144 B.C. Ltd	20.89	2019-02-13
YD87945	ARCH	112	0905144 B.C. Ltd	6.96	2019-02-13
YD87946	ARCH	113	0905144 B.C. Ltd	20.89	2019-02-13
YD87947	ARCH	114	0905144 B.C. Ltd	1.05	2019-02-13
YD87948	ARCH	115	0905144 B.C. Ltd	20.28	2019-02-13
YD87949	ARCH	116	0905144 B.C. Ltd	14.81	2019-02-13
YD87950	ARCH	117	0905144 B.C. Ltd	8.29	2019-02-13
YD87951	ARCH	118	0905144 B.C. Ltd	1.94	2019-02-13
YD87952	ARCH	119	0905144 B.C. Ltd	0.60	2019-02-13

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD87953	ARCH	120	0905144 B.C. Ltd	7.57	2019-02-13
YD87954	ARCH	121	0905144 B.C. Ltd	3.52	2019-02-13
YD87955	ARCH	122	0905144 B.C. Ltd	0.01	2019-02-13
YD87956	ARCH	123	0905144 B.C. Ltd	17.16	2019-02-13
YD87957	ARCH	124	0905144 B.C. Ltd	15.65	2019-02-13
YD87958	ARCH	125	0905144 B.C. Ltd	12.23	2019-02-13
YD87959	ARCH	126	0905144 B.C. Ltd	11.07	2019-02-13
YD87960	ARCH	127	0905144 B.C. Ltd	5.37	2019-02-13
YD87961	ARCH	128	0905144 B.C. Ltd	8.57	2019-02-13
YD87962	ARCH	129	0905144 B.C. Ltd	0.63	2019-02-13
YA94968	BARNY	1	0905144 B.C. Ltd	21.77	2019-02-11
YA94969	BARNY	2	0905144 B.C. Ltd	20.91	2019-02-11
YA94970	BARNY	3	0905144 B.C. Ltd	21.3	2019-02-11
YA94971	BARNY	4	0905144 B.C. Ltd	20.27	2019-02-11
YA94972	BARNY	5	0905144 B.C. Ltd	21.28	2019-02-11
YA94973	BARNY	6	0905144 B.C. Ltd	20.66	2019-02-11
YA96002	BARNY	7	0905144 B.C. Ltd	21.86	2020-02-11
YA96003	BARNY	8	0905144 B.C. Ltd	14.28	2020-02-11
YA96004	BARNY	9	0905144 B.C. Ltd	21.82	2020-02-11
YA96005	BARNY	10	0905144 B.C. Ltd	21.33	2020-02-11
YA96006	BARNY	11	0905144 B.C. Ltd	21.45	2020-02-11
YA96007	BARNY	12	0905144 B.C. Ltd	20.97	2020-02-11
YA96008	BARNY	13	0905144 B.C. Ltd	18.56	2020-02-11
YA96009	BARNY	14	0905144 B.C. Ltd	17.43	2020-02-11
YA96867	BARNY	19	0905144 B.C. Ltd	21.4	2020-02-11
YA96868	BARNY	20	0905144 B.C. Ltd	21.55	2020-02-11
YA96869	BARNY	21	0905144 B.C. Ltd	21.28	2020-02-11
YA96870	BARNY	22	0905144 B.C. Ltd	21.46	2020-02-11
YA96871	BARNY	23	0905144 B.C. Ltd	22.38	2020-02-11
YA96872	BARNY	24	0905144 B.C. Ltd	22.2	2020-02-11
YA96873	BARNY	25	0905144 B.C. Ltd	10.01	2020-02-11
YA96874	BARNY	26	0905144 B.C. Ltd	17.26	2020-02-11
YA96875	BARNY	27	0905144 B.C. Ltd	17.67	2020-02-11
YA96876	BARNY	28	0905144 B.C. Ltd	17.86	2020-02-11
YA96877	BARNY	29	0905144 B.C. Ltd	17.61	2020-02-11
YA96878	BARNY	30	0905144 B.C. Ltd	8.9	2020-02-11

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YA96879	BARNY	31	0905144 B.C. Ltd	13.52	2020-02-11
YA96880	BARNY	32	0905144 B.C. Ltd	20.44	2020-02-11
YA97896	BARNY	33	0905144 B.C. Ltd	5.83	2020-02-11
YA97897	BARNY	34	0905144 B.C. Ltd	12.61	2020-02-11
YA97898	BARNY	35	0905144 B.C. Ltd	17.53	2020-02-11
YA97899	BARNY	36	0905144 B.C. Ltd	15.97	2020-02-11
YA97900	BARNY	37	0905144 B.C. Ltd	17.73	2020-02-11
YA97901	BARNY	38	0905144 B.C. Ltd	11.22	2020-02-11
YA97902	BARNY	39	0905144 B.C. Ltd	11.49	2020-02-11
YA97904	BARNY	41	0905144 B.C. Ltd	19.04	2020-02-11
YA97905	BARNY	42	0905144 B.C. Ltd	14.77	2020-02-11
YA97906	BARNY	43	0905144 B.C. Ltd	13.13	2020-02-11
YA97908	BARNY	45	0905144 B.C. Ltd	14.8	2020-02-11
YA97910	BARNY	47	0905144 B.C. Ltd	15.04	2020-02-11
YA97911	BARNY	48	0905144 B.C. Ltd	9.37	2020-02-11
YA97912	BARNY	49	0905144 B.C. Ltd	12.96	2020-02-11
YB08307	BARNY	50	0905144 B.C. Ltd	5.32	2020-02-11
63029	BETTY	1	0905144 B.C. Ltd	10.38	2020-12-05
63030	BETTY	2	0905144 B.C. Ltd	11.58	2020-12-05
63031	BETTY	3	0905144 B.C. Ltd	11.83	2020-12-05
63032	BETTY	4	0905144 B.C. Ltd	10.93	2020-12-05
63033	BETTY	5	0905144 B.C. Ltd	18.41	2020-12-05
63034	BETTY	6	0905144 B.C. Ltd	17.59	2020-12-05
63035	BETTY	7	0905144 B.C. Ltd	19.5	2020-12-05
63036	BETTY	8	0905144 B.C. Ltd	21.2	2020-12-05
YC26564	BUR	1	0905144 B.C. Ltd	20.9	2032-02-23
YC26565	BUR	2	0905144 B.C. Ltd	20.92	2032-02-23
YC26566	BUR	3	0905144 B.C. Ltd	20.9	2032-02-23
YC26567	BUR	4	0905144 B.C. Ltd	20.9	2032-02-23
YC26568	BUR	5	0905144 B.C. Ltd	20.9	2032-02-23
YC26569	BUR	6	0905144 B.C. Ltd	20.9	2032-02-23
YC26570	BUR	7	0905144 B.C. Ltd	20.89	2032-02-23
YC26571	BUR	8	0905144 B.C. Ltd	20.9	2032-02-23
YC26572	BUR	9	0905144 B.C. Ltd	20.88	2032-02-23
YC26573	BUR	10	0905144 B.C. Ltd	20.9	2032-02-23
YC26574	BUR	11	0905144 B.C. Ltd	20.91	2032-02-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YC26575	BUR	12	0905144 B.C. Ltd	20.9	2032-02-23
YC26576	BUR	13	0905144 B.C. Ltd	20.9	2032-02-23
YC26577	BUR	14	0905144 B.C. Ltd	20.9	2032-02-23
YC26578	BUR	15	0905144 B.C. Ltd	20.86	2032-02-23
YC26579	BUR	16	0905144 B.C. Ltd	20.9	2032-02-23
YC26580	BUR	17	0905144 B.C. Ltd	20.88	2032-02-23
YC26581	BUR	18	0905144 B.C. Ltd	20.88	2032-02-23
YC26582	BUR	19	0905144 B.C. Ltd	20.86	2032-02-23
YC26583	BUR	20	0905144 B.C. Ltd	20.9	2032-02-23
YC26584	BUR	21	0905144 B.C. Ltd	20.86	2032-02-23
YC26585	BUR	22	0905144 B.C. Ltd	20.9	2032-02-23
YC26586	BUR	23	0905144 B.C. Ltd	20.86	2032-02-23
YC26587	BUR	24	0905144 B.C. Ltd	20.9	2032-02-23
YC26588	BUR	25	0905144 B.C. Ltd	20.86	2032-02-23
YC26589	BUR	26	0905144 B.C. Ltd	20.9	2032-02-23
YC26590	BUR	27	0905144 B.C. Ltd	20.9	2032-02-23
YC26591	BUR	28	0905144 B.C. Ltd	20.9	2032-02-23
YC26592	BUR	29	0905144 B.C. Ltd	20.9	2032-02-23
YC26593	BUR	30	0905144 B.C. Ltd	20.9	2032-02-23
YC26594	BUR	31	0905144 B.C. Ltd	20.9	2032-02-23
YC26595	BUR	32	0905144 B.C. Ltd	20.9	2032-02-23
YC26596	BUR	33	0905144 B.C. Ltd	20.9	2032-02-23
YC26597	BUR	34	0905144 B.C. Ltd	20.9	2032-02-23
YC26598	BUR	35	0905144 B.C. Ltd	20.9	2032-02-23
YC26599	BUR	36	0905144 B.C. Ltd	20.84	2032-02-23
YC26600	BUR	37	0905144 B.C. Ltd	20.9	2032-02-23
YC26601	BUR	38	0905144 B.C. Ltd	20.9	2032-02-23
YC26602	BUR	39	0905144 B.C. Ltd	20.9	2032-02-23
YC26603	BUR	40	0905144 B.C. Ltd	20.9	2032-02-23
YC26604	BUR	41	0905144 B.C. Ltd	20.9	2032-02-23
YC26605	BUR	42	0905144 B.C. Ltd	20.9	2032-02-23
YC26606	BUR	43	0905144 B.C. Ltd	20.9	2032-02-23
YC26607	BUR	44	0905144 B.C. Ltd	20.9	2032-02-23
YC26608	BUR	45	0905144 B.C. Ltd	20.93	2032-02-23
YC26609	BUR	46	0905144 B.C. Ltd	20.9	2032-02-23
YC26610	BUR	47	0905144 B.C. Ltd	20.9	2032-02-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YC26611	BUR	48	0905144 B.C. Ltd	20.9	2032-02-23
YC26612	BUR	49	0905144 B.C. Ltd	20.9	2032-02-23
YC26613	BUR	50	0905144 B.C. Ltd	20.9	2032-02-23
YC26614	BUR	51	0905144 B.C. Ltd	20.9	2032-02-23
YC26615	BUR	52	0905144 B.C. Ltd	20.9	2032-02-23
YC26616	BUR	53	0905144 B.C. Ltd	20.9	2032-02-23
YC26617	BUR	54	0905144 B.C. Ltd	20.9	2032-02-23
YC26618	BUR	55	0905144 B.C. Ltd	20.9	2032-02-23
YC26619	BUR	56	0905144 B.C. Ltd	20.9	2032-02-23
YC26620	BUR	57	0905144 B.C. Ltd	20.9	2032-02-23
YC26621	BUR	58	0905144 B.C. Ltd	20.9	2032-02-23
YB36423	BURWASH	1	0905144 B.C. Ltd	20.9	2036-02-23
YB36424	BURWASH	2	0905144 B.C. Ltd	20.9	2036-02-23
YB36425	BURWASH	3	0905144 B.C. Ltd	20.9	2036-02-23
YB36426	BURWASH	4	0905144 B.C. Ltd	20.9	2036-02-23
YB36427	BURWASH	5	0905144 B.C. Ltd	20.9	2036-02-23
YB36428	BURWASH	6	0905144 B.C. Ltd	20.9	2036-02-23
YB36429	BURWASH	7	0905144 B.C. Ltd	20.9	2036-02-23
YB36430	BURWASH	8	0905144 B.C. Ltd	20.9	2036-02-23
YB36431	BURWASH	9	0905144 B.C. Ltd	20.9	2036-02-23
YC18485	BURWASH	10	0905144 B.C. Ltd	17.35	2032-02-23
YC18486	BURWASH	11	0905144 B.C. Ltd	3.55	2032-02-23
YC18487	BURWASH	12	0905144 B.C. Ltd	20.9	2032-02-23
YC18488	BURWASH	13	0905144 B.C. Ltd	20.9	2032-02-23
YC18489	BURWASH	14	0905144 B.C. Ltd	20.9	2032-02-23
YC18490	BURWASH	15	0905144 B.C. Ltd	20.9	2032-02-23
YC18491	BURWASH	16	0905144 B.C. Ltd	20.89	2032-02-23
YC18492	BURWASH	17	0905144 B.C. Ltd	20.9	2032-02-23
YC18493	BURWASH	18	0905144 B.C. Ltd	20.9	2032-02-23
YC18494	BURWASH	19	0905144 B.C. Ltd	20.9	2032-02-23
YC18495	BURWASH	20	0905144 B.C. Ltd	20.9	2032-02-23
YC18496	BURWASH	21	0905144 B.C. Ltd	20.9	2032-02-23
YC18497	BURWASH	22	0905144 B.C. Ltd	20.9	2032-02-23
YC18498	BURWASH	23	0905144 B.C. Ltd	20.92	2032-02-23
YC18499	BURWASH	24	0905144 B.C. Ltd	20.9	2032-02-23
YC18500	BURWASH	25	0905144 B.C. Ltd	20.92	2032-02-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YC18501	BURWASH	26	0905144 B.C. Ltd	20.88	2032-02-23
YC18502	BURWASH	27	0905144 B.C. Ltd	20.9	2032-02-23
YC18503	BURWASH	28	0905144 B.C. Ltd	20.9	2032-02-23
YC18504	BURWASH	29	0905144 B.C. Ltd	20.9	2032-02-23
YC18505	BURWASH	30	0905144 B.C. Ltd	20.9	2032-02-23
YC18506	BURWASH	31	0905144 B.C. Ltd	20.9	2032-02-23
YC18507	BURWASH	32	0905144 B.C. Ltd	20.9	2032-02-23
YC18508	BURWASH	33	0905144 B.C. Ltd	20.9	2032-02-23
60775	DISCOVERY	1	0905144 B.C. Ltd	10.49	2020-12-05
60776	DISCOVERY	2	0905144 B.C. Ltd	10.5	2020-12-05
60777	DISCOVERY	3	0905144 B.C. Ltd	16.08	2020-12-05
60778	DISCOVERY	4	0905144 B.C. Ltd	16.82	2020-12-05
60779	DISCOVERY	5	0905144 B.C. Ltd	13.35	2020-12-05
60780	DISCOVERY	6	0905144 B.C. Ltd	16.69	2020-12-05
60781	DISCOVERY	7	0905144 B.C. Ltd	13.66	2020-12-05
60782	DISCOVERY	8	0905144 B.C. Ltd	11.57	2020-12-05
YE60861	FORMULA	1	1043704 B.C. Ltd	20.78	2019-07-23
YE60862	FORMULA	2	1043704 B.C. Ltd	13.74	2019-07-23
YE60863	FORMULA	3	1043704 B.C. Ltd	20.78	2019-07-23
YE60864	FORMULA	4	1043704 B.C. Ltd	13.74	2019-07-23
YE60865	FORMULA	5	1043704 B.C. Ltd	20.78	2019-07-23
YE60866	FORMULA	6	1043704 B.C. Ltd	20.78	2019-07-23
YE60867	FORMULA	7	1043704 B.C. Ltd	20.78	2019-07-23
YE60868	FORMULA	8	1043704 B.C. Ltd	20.78	2019-07-23
YE60869	FORMULA	9	1043704 B.C. Ltd	20.78	2019-07-23
YE60870	FORMULA	10	1043704 B.C. Ltd	20.78	2019-07-23
YE60871	FORMULA	11	1043704 B.C. Ltd	20.78	2019-07-23
YE60872	FORMULA	12	1043704 B.C. Ltd	20.78	2019-07-23
YE60873	FORMULA	13	1043704 B.C. Ltd	20.78	2019-07-23
YE60874	FORMULA	14	1043704 B.C. Ltd	20.78	2019-07-23
YE60875	FORMULA	15	1043704 B.C. Ltd	20.78	2019-07-23
YE60876	FORMULA	16	1043704 B.C. Ltd	20.78	2019-07-23
YE60877	FORMULA	17	1043704 B.C. Ltd	20.78	2019-07-23
YE60878	FORMULA	18	1043704 B.C. Ltd	20.78	2019-07-23
YE60879	FORMULA	19	1043704 B.C. Ltd	20.78	2019-07-23
YE60880	FORMULA	20	1043704 B.C. Ltd	20.78	2019-07-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YE60881	FORMULA	21	1043704 B.C. Ltd	20.78	2019-07-23
YE60882	FORMULA	22	1043704 B.C. Ltd	20.78	2019-07-23
YE60883	FORMULA	23	1043704 B.C. Ltd	20.78	2019-07-23
YE60884	FORMULA	24	1043704 B.C. Ltd	20.78	2019-07-23
YE60885	FORMULA	25	1043704 B.C. Ltd	20.78	2019-07-23
YE60886	FORMULA	26	1043704 B.C. Ltd	20.78	2019-07-23
YE60887	FORMULA	27	1043704 B.C. Ltd	20.78	2019-07-23
YE60888	FORMULA	28	1043704 B.C. Ltd	20.78	2019-07-23
YE60889	FORMULA	29	1043704 B.C. Ltd	20.78	2019-07-23
YE60890	FORMULA	30	1043704 B.C. Ltd	20.78	2019-07-23
YE60891	FORMULA	31	1043704 B.C. Ltd	20.78	2019-07-23
YE60892	FORMULA	32	1043704 B.C. Ltd	20.78	2019-07-23
YE60893	FORMULA	33	1043704 B.C. Ltd	20.78	2019-07-23
YE60894	FORMULA	34	1043704 B.C. Ltd	20.78	2019-07-23
YE60895	FORMULA	35	1043704 B.C. Ltd	20.78	2019-07-23
YE60896	FORMULA	36	1043704 B.C. Ltd	20.78	2019-07-23
YE60897	FORMULA	37	1043704 B.C. Ltd	20.78	2019-07-23
YE60898	FORMULA	38	1043704 B.C. Ltd	20.78	2019-07-23
YE60899	FORMULA	39	1043704 B.C. Ltd	20.78	2019-07-23
YE60900	FORMULA	40	1043704 B.C. Ltd	20.78	2019-07-23
YE60901	FORMULA	41	1043704 B.C. Ltd	20.78	2019-07-23
YE60902	FORMULA	42	1043704 B.C. Ltd	20.78	2019-07-23
YE60903	FORMULA	43	1043704 B.C. Ltd	20.78	2019-07-23
YE60904	FORMULA	44	1043704 B.C. Ltd	20.78	2019-07-23
YE60905	FORMULA	45	1043704 B.C. Ltd	20.78	2019-07-23
YE60906	FORMULA	46	1043704 B.C. Ltd	20.78	2019-07-23
YE60907	FORMULA	47	1043704 B.C. Ltd	20.78	2019-07-23
YE60908	FORMULA	48	1043704 B.C. Ltd	20.78	2019-07-23
YE60909	FORMULA	49	1043704 B.C. Ltd	20.78	2019-07-23
YE60910	FORMULA	50	1043704 B.C. Ltd	20.78	2019-07-23
YE60911	FORMULA	51	1043704 B.C. Ltd	20.78	2019-07-23
YE60912	FORMULA	52	1043704 B.C. Ltd	20.78	2019-07-23
YE60913	FORMULA	53	1043704 B.C. Ltd	20.78	2019-07-23
YE60914	FORMULA	54	1043704 B.C. Ltd	20.78	2019-07-23
YE60915	FORMULA	55	1043704 B.C. Ltd	20.78	2019-07-23
YE60916	FORMULA	56	1043704 B.C. Ltd	20.78	2019-07-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YE60917	FORMULA	57	1043704 B.C. Ltd	20.78	2019-07-23
YE60918	FORMULA	58	1043704 B.C. Ltd	20.16	2019-07-23
YE60919	FORMULA	59	1043704 B.C. Ltd	20.78	2019-07-23
YE60920	FORMULA	60	1043704 B.C. Ltd	20.19	2019-07-23
YE60921	FORMULA	61	1043704 B.C. Ltd	20.78	2019-07-23
YE60922	FORMULA	62	1043704 B.C. Ltd	20.22	2019-07-23
YE60923	FORMULA	63	1043704 B.C. Ltd	20.78	2019-07-23
YE60924	FORMULA	64	1043704 B.C. Ltd	20.25	2019-07-23
YE60925	FORMULA	65	1043704 B.C. Ltd	20.78	2019-07-23
YE60926	FORMULA	66	1043704 B.C. Ltd	20.28	2019-07-23
YE60927	FORMULA	67	1043704 B.C. Ltd	20.78	2019-07-23
YE60928	FORMULA	68	1043704 B.C. Ltd	20.3	2019-07-23
YE60929	FORMULA	69	1043704 B.C. Ltd	20.78	2019-07-23
YE60930	FORMULA	70	1043704 B.C. Ltd	20.34	2019-07-23
YE60931	FORMULA	71	1043704 B.C. Ltd	20.78	2019-07-23
YE60932	FORMULA	72	1043704 B.C. Ltd	20.36	2019-07-23
YE60933	FORMULA	73	1043704 B.C. Ltd	20.78	2019-07-23
YE60934	FORMULA	74	1043704 B.C. Ltd	20.39	2019-07-23
YE60935	FORMULA	75	1043704 B.C. Ltd	20.78	2019-07-23
YE60936	FORMULA	76	1043704 B.C. Ltd	20.42	2019-07-23
YE60937	FORMULA	77	1043704 B.C. Ltd	20.78	2019-07-23
YE60938	FORMULA	78	1043704 B.C. Ltd	20.44	2019-07-23
YE60939	FORMULA	79	1043704 B.C. Ltd	20.78	2019-07-23
YE60940	FORMULA	80	1043704 B.C. Ltd	20.47	2019-07-23
YE60941	FORMULA	81	1043704 B.C. Ltd	20.78	2019-07-23
YE60942	FORMULA	82	1043704 B.C. Ltd	20.78	2019-07-23
YE60943	FORMULA	83	1043704 B.C. Ltd	20.78	2019-07-23
YE60944	FORMULA	84	1043704 B.C. Ltd	20.78	2019-07-23
YE60945	FORMULA	85	1043704 B.C. Ltd	20.78	2019-07-23
YE60946	FORMULA	86	1043704 B.C. Ltd	20.78	2019-07-23
YE60947	FORMULA	87	1043704 B.C. Ltd	20.78	2019-07-23
YE60948	FORMULA	88	1043704 B.C. Ltd	20.78	2019-07-23
YE60949	FORMULA	89	1043704 B.C. Ltd	20.78	2019-07-23
YE60950	FORMULA	90	1043704 B.C. Ltd	20.78	2019-07-23
YE60951	FORMULA	91	1043704 B.C. Ltd	20.78	2019-07-23
YE60952	FORMULA	92	1043704 B.C. Ltd	20.78	2019-07-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YE60953	FORMULA	93	1043704 B.C. Ltd	20.78	2019-07-23
YE60954	FORMULA	94	1043704 B.C. Ltd	20.78	2019-07-23
YE60955	FORMULA	95	1043704 B.C. Ltd	20.78	2019-07-23
YE60956	FORMULA	96	1043704 B.C. Ltd	20.78	2019-07-23
YE60957	FORMULA	97	1043704 B.C. Ltd	20.78	2019-07-23
YE60958	FORMULA	98	1043704 B.C. Ltd	20.78	2019-07-23
YE60959	FORMULA	99	1043704 B.C. Ltd	20.78	2019-07-23
YE60960	FORMULA	100	1043704 B.C. Ltd	20.78	2019-07-23
YE60961	FORMULA	101	1043704 B.C. Ltd	20.78	2019-07-23
YE60962	FORMULA	102	1043704 B.C. Ltd	20.78	2019-07-23
YE60963	FORMULA	103	1043704 B.C. Ltd	20.78	2019-07-23
YE60964	FORMULA	104	1043704 B.C. Ltd	20.78	2019-07-23
YE60965	FORMULA	105	1043704 B.C. Ltd	20.78	2019-07-23
YE60966	FORMULA	106	1043704 B.C. Ltd	20.78	2019-07-23
YE60967	FORMULA	107	1043704 B.C. Ltd	20.78	2019-07-23
YE60968	FORMULA	108	1043704 B.C. Ltd	20.78	2019-07-23
YE60969	FORMULA	109	1043704 B.C. Ltd	20.78	2019-07-23
YE60970	FORMULA	110	1043704 B.C. Ltd	20.78	2019-07-23
YE60971	FORMULA	111	1043704 B.C. Ltd	20.78	2019-07-23
YE60972	FORMULA	112	1043704 B.C. Ltd	20.78	2019-07-23
YE60973	FORMULA	113	1043704 B.C. Ltd	20.77	2019-07-23
YE60974	FORMULA	114	1043704 B.C. Ltd	20.78	2019-07-23
YE60975	FORMULA	115	1043704 B.C. Ltd	20.78	2019-07-23
YE60976	FORMULA	116	1043704 B.C. Ltd	13.74	2019-07-23
YD80503	GWG	1	1043704 B.C. Ltd	20.89	2019-06-22
YD80504	GWG	2	1043704 B.C. Ltd	20.89	2019-06-22
YD80505	GWG	3	1043704 B.C. Ltd	20.89	2019-06-22
YD80506	GWG	4	1043704 B.C. Ltd	20.89	2019-06-22
YD80507	GWG	5	1043704 B.C. Ltd	20.89	2019-06-22
YD80508	GWG	6	1043704 B.C. Ltd	20.89	2019-06-22
YD80509	GWG	7	1043704 B.C. Ltd	20.89	2019-06-22
YD80510	GWG	8	1043704 B.C. Ltd	20.89	2019-06-22
YD80511	GWG	9	1043704 B.C. Ltd	20.89	2019-06-22
YD80512	GWG	10	1043704 B.C. Ltd	20.89	2019-06-22
YD80513	GWG	11	1043704 B.C. Ltd	20.89	2019-06-22
YD80514	GWG	12	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80515	GWG	13	1043704 B.C. Ltd	20.89	2019-06-22
YD80516	GWG	14	1043704 B.C. Ltd	20.89	2019-06-22
YD80517	GWG	15	1043704 B.C. Ltd	20.89	2019-06-22
YD80518	GWG	16	1043704 B.C. Ltd	20.89	2019-06-22
YD80519	GWG	17	1043704 B.C. Ltd	20.89	2019-06-22
YD80520	GWG	18	1043704 B.C. Ltd	20.89	2019-06-22
YD80521	GWG	19	1043704 B.C. Ltd	20.89	2019-06-22
YD80522	GWG	20	1043704 B.C. Ltd	20.89	2019-06-22
YD80523	GWG	21	1043704 B.C. Ltd	20.89	2019-06-22
YD80524	GWG	22	1043704 B.C. Ltd	20.89	2019-06-22
YD80525	GWG	23	1043704 B.C. Ltd	20.89	2019-06-22
YD80526	GWG	24	1043704 B.C. Ltd	20.89	2019-06-22
YD80527	GWG	25	1043704 B.C. Ltd	20.89	2019-06-22
YD80528	GWG	26	1043704 B.C. Ltd	20.89	2019-06-22
YD80529	GWG	27	1043704 B.C. Ltd	20.89	2019-06-22
YD80530	GWG	28	1043704 B.C. Ltd	20.89	2019-06-22
YD80531	GWG	29	1043704 B.C. Ltd	20.89	2019-06-22
YD80532	GWG	30	1043704 B.C. Ltd	20.89	2019-06-22
YD80533	GWG	31	1043704 B.C. Ltd	20.89	2019-06-22
YD80534	GWG	32	1043704 B.C. Ltd	20.89	2019-06-22
YD80535	GWG	33	1043704 B.C. Ltd	20.89	2019-06-22
YD80536	GWG	34	1043704 B.C. Ltd	20.89	2019-06-22
YD80537	GWG	35	1043704 B.C. Ltd	20.89	2019-06-22
YD80538	GWG	36	1043704 B.C. Ltd	20.89	2019-06-22
YD80539	GWG	37	1043704 B.C. Ltd	20.89	2019-06-22
YD80540	GWG	38	1043704 B.C. Ltd	20.89	2019-06-22
YD80541	GWG	39	1043704 B.C. Ltd	20.89	2019-06-22
YD80542	GWG	40	1043704 B.C. Ltd	20.89	2019-06-22
63001	IRISH	1	0905144 B.C. Ltd	19.66	2020-12-05
63002	IRISH	2	0905144 B.C. Ltd	15.14	2020-12-05
63003	IRISH	3	0905144 B.C. Ltd	11.06	2020-12-05
63006	IRISH	6	0905144 B.C. Ltd	16.41	2020-12-05
64742	JEEP	96	0905144 B.C. Ltd	11.93	2020-12-05
64828	JEEP	234	0905144 B.C. Ltd	4.22	2020-12-05
64830	JEEP	236	0905144 B.C. Ltd	5.61	2020-12-05
64122	JEEP	238	0905144 B.C. Ltd	6.75	2020-12-05

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
64832	JEEP	240	0905144 B.C. Ltd	6.21	2020-12-05
64834	JEEP	242	0905144 B.C. Ltd	8	2020-12-05
64836	JEEP	244	0905144 B.C. Ltd	12.24	2020-12-05
66569	JEEP	265	0905144 B.C. Ltd	9.98	2020-12-05
66571	JEEP	267	0905144 B.C. Ltd	19.7	2020-12-05
66572	JEEP	268	0905144 B.C. Ltd	18.46	2020-12-05
YD127061	KAT	1	0905144 B.C. Ltd	17.6	2019-12-05
YD127062	KAT	2	0905144 B.C. Ltd	20.9	2019-12-05
YD127063	KAT	3	0905144 B.C. Ltd	18.08	2019-12-05
YD127064	KAT	4	0905144 B.C. Ltd	14.39	2019-12-05
YD127065	KAT	5	0905144 B.C. Ltd	16.65	2019-12-05
YD127066	KAT	6	0905144 B.C. Ltd	10.11	2019-12-05
YD127067	KAT	7	0905144 B.C. Ltd	16.45	2019-12-05
YD127068	KAT	8	0905144 B.C. Ltd	6.6	2019-12-05
YD127069	KAT	9	0905144 B.C. Ltd	16.1	2019-12-05
YD127070	KAT	10	0905144 B.C. Ltd	3.06	2019-12-05
YD127071	KAT	11	0905144 B.C. Ltd	5.63	2019-12-05
YD127072	KAT	12	0905144 B.C. Ltd	19.87	2019-12-05
YD127073	KAT	13	0905144 B.C. Ltd	2.73	2019-12-05
YD127074	KAT	14	0905144 B.C. Ltd	20.57	2019-12-05
YD127075	KAT	15	0905144 B.C. Ltd	5.94	2019-12-05
YD127076	KAT	16	0905144 B.C. Ltd	20.9	2019-12-05
YD127077	KAT	17	0905144 B.C. Ltd	6.52	2019-12-05
YD127078	KAT	18	0905144 B.C. Ltd	20.9	2019-12-05
YD127079	KAT	19	0905144 B.C. Ltd	11.07	2019-12-05
YD127080	KAT	20	0905144 B.C. Ltd	20.9	2019-12-05
YD127081	KAT	21	0905144 B.C. Ltd	15.54	2019-12-05
YD127082	KAT	22	0905144 B.C. Ltd	20.9	2019-12-05
YD127083	KAT	23	0905144 B.C. Ltd	10.86	2019-12-05
YD127084	KAT	24	0905144 B.C. Ltd	20.9	2019-12-05
YD127085	KAT	25	0905144 B.C. Ltd	13.9	2019-12-05
YD127086	KAT	26	0905144 B.C. Ltd	20.9	2019-12-05
YD127087	KAT	27	0905144 B.C. Ltd	7.65	2019-12-05
YD127088	KAT	28	0905144 B.C. Ltd	15.69	2019-12-05
YD127089	KAT	29	0905144 B.C. Ltd	7.86	2019-12-05
YD127090	KAT	30	0905144 B.C. Ltd	2.44	2019-12-05

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD127091	KAT	31	0905144 B.C. Ltd	2.1	2019-12-05
YD127092	KAT	32	0905144 B.C. Ltd	0.92	2019-12-05
YD127093	KAT	33	0905144 B.C. Ltd	1.14	2019-12-05
YD127094	KAT	34	0905144 B.C. Ltd	2.84	2019-12-05
YD127095	KAT	35	0905144 B.C. Ltd	5.49	2018-12-05
YD127096	KAT	36	0905144 B.C. Ltd	3.26	2018-12-05
YD127097	KAT	37	0905144 B.C. Ltd	16.92	2018-12-05
YD127098	KAT	38	0905144 B.C. Ltd	20.02	2018-12-05
YD127099	KAT	39	0905144 B.C. Ltd	16.97	2018-12-05
YD127100	KAT	40	0905144 B.C. Ltd	20.02	2018-12-05
YD127101	KAT	41	0905144 B.C. Ltd	16.02	2018-12-05
YD127102	KAT	42	0905144 B.C. Ltd	20.02	2018-12-05
YE70953	KAT	43	0905144 B.C. Ltd	14.24	2018-12-05
YE70954	KAT	44	0905144 B.C. Ltd	20.02	2018-12-05
YE70955	KAT	45	0905144 B.C. Ltd	10.36	2018-12-05
YE70956	KAT	46	0905144 B.C. Ltd	20.02	2018-12-05
YE70957	KAT	47	0905144 B.C. Ltd	17.69	2018-12-05
YE70958	KAT	48	0905144 B.C. Ltd	13.71	2018-12-05
YE70959	KAT	49	0905144 B.C. Ltd	20.9	2018-12-05
YE70960	KAT	50	0905144 B.C. Ltd	19.89	2018-12-05
YE70961	KAT	51	0905144 B.C. Ltd	20.9	2018-12-05
YE70962	KAT	52	0905144 B.C. Ltd	13.92	2018-12-05
YE70963	KAT	53	0905144 B.C. Ltd	20.9	2018-12-05
YE70964	KAT	54	0905144 B.C. Ltd	12.49	2018-12-05
YE70965	KAT	55	0905144 B.C. Ltd	20.9	2018-12-05
YE70966	KAT	56	0905144 B.C. Ltd	20.9	2018-12-05
YE70967	KAT	57	0905144 B.C. Ltd	20.9	2018-12-05
YE70968	KAT	58	0905144 B.C. Ltd	20.9	2018-12-05
YE70969	KAT	59	0905144 B.C. Ltd	20.9	2018-12-05
YE70970	KAT	60	0905144 B.C. Ltd	20.9	2018-12-05
YE70971	KAT	61	0905144 B.C. Ltd	20.9	2018-12-05
YE70972	KAT	62	0905144 B.C. Ltd	20.9	2018-12-05
YE70973	KAT	63	0905144 B.C. Ltd	20.9	2018-12-05
YE70974	KAT	64	0905144 B.C. Ltd	20.9	2018-12-05
YE70975	KAT	65	0905144 B.C. Ltd	20.9	2018-12-05
YE70976	KAT	66	0905144 B.C. Ltd	20.9	2018-12-05

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YE70977	KAT	67	0905144 B.C. Ltd	20.9	2018-12-05
YE70978	KAT	68	0905144 B.C. Ltd	20.9	2018-12-05
YE70979	KAT	69	0905144 B.C. Ltd	16.97	2018-12-05
YE70980	KAT	70	0905144 B.C. Ltd	19.65	2018-12-05
YE70981	KAT	71	0905144 B.C. Ltd	8.54	2018-12-05
YE70982	KAT	72	0905144 B.C. Ltd	19.65	2018-12-05
YE70983	KAT	73	0905144 B.C. Ltd	14.09	2018-12-05
YE70984	KAT	74	0905144 B.C. Ltd	18.21	2018-12-05
YE70985	KAT	75	0905144 B.C. Ltd	2.86	2018-12-05
YE70986	KAT	76	0905144 B.C. Ltd	7.56	2018-12-05
YE70987	KAT	77	0905144 B.C. Ltd	4.35	2018-12-05
YE70988	KAT	78	0905144 B.C. Ltd	8	2018-12-05
YE70989	KAT	79	0905144 B.C. Ltd	9.84	2018-12-05
YE70990	KAT	80	0905144 B.C. Ltd	8.44	2018-12-05
YE70991	KAT	81	0905144 B.C. Ltd	10.92	2018-12-05
YE70992	KAT	82	0905144 B.C. Ltd	5.71	2018-12-05
YE70993	KAT	83	0905144 B.C. Ltd	11.7	2019-12-05
YE70994	KAT	84	0905144 B.C. Ltd	19.6	2019-12-05
YE70995	KAT	85	0905144 B.C. Ltd	8.78	2019-12-05
YE70996	KAT	86	0905144 B.C. Ltd	19.49	2019-12-05
63021	MAC	1	0905144 B.C. Ltd	12.62	2020-12-05
63022	MAC	2	0905144 B.C. Ltd	12.47	2020-12-05
63023	MAC	3	0905144 B.C. Ltd	14.2	2020-12-05
63024	MAC	4	0905144 B.C. Ltd	11.19	2020-12-05
63025	MAC	5	0905144 B.C. Ltd	9.82	2020-12-05
63026	MAC	6	0905144 B.C. Ltd	8.44	2020-12-05
63027	MAC	7	0905144 B.C. Ltd	7.64	2020-12-05
63028	MAC	8	0905144 B.C. Ltd	13.84	2020-12-05
YA94966	MUS	5	0905144 B.C. Ltd	20.87	2020-02-11
YA94967	MUS	6	0905144 B.C. Ltd	20.74	2020-02-11
YA96015	MUS	12	0905144 B.C. Ltd	20.99	2020-02-11
YA96017	MUS	14	0905144 B.C. Ltd	20.37	2020-02-11
YA96019	MUS	16	0905144 B.C. Ltd	16.12	2020-02-11
YD80544	MWSK	42	1043704 B.C. Ltd	20.89	2019-06-22
YD80545	MWSK	43	1043704 B.C. Ltd	20.89	2019-06-22
YD80546	MWSK	44	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80547	MWSK	45	1043704 B.C. Ltd	20.89	2019-06-22
YD80548	MWSK	46	1043704 B.C. Ltd	20.89	2019-06-22
YD80549	MWSK	47	1043704 B.C. Ltd	20.89	2019-06-22
YD80550	MWSK	48	1043704 B.C. Ltd	20.89	2019-06-22
YD80551	MWSK	49	1043704 B.C. Ltd	20.89	2019-06-22
YD80552	MWSK	50	1043704 B.C. Ltd	20.89	2019-06-22
YD80553	MWSK	51	1043704 B.C. Ltd	20.89	2019-06-22
YD80554	MWSK	52	1043704 B.C. Ltd	20.89	2019-06-22
YD80555	MWSK	53	1043704 B.C. Ltd	20.89	2019-06-22
YD80556	MWSK	54	1043704 B.C. Ltd	20.89	2019-06-22
YD80557	MWSK	55	1043704 B.C. Ltd	20.89	2019-06-22
YD80558	MWSK	56	1043704 B.C. Ltd	20.89	2019-06-22
YD80559	MWSK	57	1043704 B.C. Ltd	20.89	2019-06-22
YD80560	MWSK	58	1043704 B.C. Ltd	20.89	2019-06-22
YD80561	MWSK	59	1043704 B.C. Ltd	20.89	2019-06-22
YD80562	MWSK	60	1043704 B.C. Ltd	20.89	2019-06-22
YD80563	MWSK	61	1043704 B.C. Ltd	20.89	2019-06-22
YD80564	MWSK	62	1043704 B.C. Ltd	20.89	2019-06-22
YD80565	MWSK	63	1043704 B.C. Ltd	20.89	2019-06-22
YD80566	MWSK	64	1043704 B.C. Ltd	20.89	2019-06-22
YD80567	MWSK	65	1043704 B.C. Ltd	20.89	2019-06-22
YD80568	MWSK	66	1043704 B.C. Ltd	20.89	2019-06-22
YD80569	MWSK	67	1043704 B.C. Ltd	20.89	2019-06-22
YD80570	MWSK	68	1043704 B.C. Ltd	20.89	2019-06-22
YD80571	MWSK	69	1043704 B.C. Ltd	20.89	2019-06-22
YD80572	MWSK	70	1043704 B.C. Ltd	20.89	2019-06-22
YD80573	MWSK	71	1043704 B.C. Ltd	20.89	2019-06-22
YD80574	MWSK	72	1043704 B.C. Ltd	20.89	2019-06-22
YD80575	MWSK	73	1043704 B.C. Ltd	20.89	2019-06-22
YD80576	MWSK	74	1043704 B.C. Ltd	20.89	2019-06-22
YD80577	MWSK	75	1043704 B.C. Ltd	20.89	2019-06-22
YD80578	MWSK	76	1043704 B.C. Ltd	20.89	2019-06-22
YD80579	MWSK	77	1043704 B.C. Ltd	20.89	2019-06-22
YD80580	MWSK	78	1043704 B.C. Ltd	20.89	2019-06-22
YD80581	MWSK	79	1043704 B.C. Ltd	20.89	2019-06-22
YD80582	MWSK	80	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80583	MWSK	81	1043704 B.C. Ltd	20.89	2019-06-22
YD80584	MWSK	82	1043704 B.C. Ltd	20.89	2019-06-22
YD80585	MWSK	83	1043704 B.C. Ltd	20.89	2019-06-22
YD80586	MWSK	84	1043704 B.C. Ltd	20.89	2019-06-22
YD80587	MWSK	85	1043704 B.C. Ltd	20.89	2019-06-22
YD80588	MWSK	86	1043704 B.C. Ltd	20.89	2019-06-22
YD80589	MWSK	87	1043704 B.C. Ltd	19.45	2019-06-22
YD80590	MWSK	88	1043704 B.C. Ltd	20.89	2019-06-22
YD80591	MWSK	89	1043704 B.C. Ltd	20.89	2019-06-22
YD80592	MWSK	90	1043704 B.C. Ltd	20.89	2019-06-22
YD80593	MWSK	91	1043704 B.C. Ltd	20.89	2019-06-22
YD80594	MWSK	92	1043704 B.C. Ltd	20.89	2019-06-22
YD80595	MWSK	93	1043704 B.C. Ltd	20.89	2019-06-22
YD80596	MWSK	94	1043704 B.C. Ltd	20.89	2019-06-22
YD80597	MWSK	95	1043704 B.C. Ltd	20.89	2019-06-22
YE32227	MWSK	96	1043704 B.C. Ltd	20.89	2019-06-22
YE32228	MWSK	97	1043704 B.C. Ltd	20.89	2019-06-22
YD80600	MWSK	98	1043704 B.C. Ltd	20.89	2019-06-22
YD80601	MWSK	99	1043704 B.C. Ltd	20.89	2019-06-22
YD80602	MWSK	100	1043704 B.C. Ltd	20.89	2019-06-22
YD80603	MWSK	101	1043704 B.C. Ltd	20.89	2019-06-22
YD80604	MWSK	102	1043704 B.C. Ltd	20.89	2019-06-22
YD80605	MWSK	103	1043704 B.C. Ltd	20.89	2019-06-22
YD80606	MWSK	104	1043704 B.C. Ltd	20.89	2019-06-22
YD80607	MWSK	105	1043704 B.C. Ltd	20.89	2019-06-22
YD80608	MWSK	106	1043704 B.C. Ltd	20.89	2019-06-22
YD80609	MWSK	107	1043704 B.C. Ltd	20.89	2019-06-22
YD80610	MWSK	108	1043704 B.C. Ltd	20.89	2019-06-22
YD80611	MWSK	109	1043704 B.C. Ltd	20.89	2019-06-22
YD80612	MWSK	110	1043704 B.C. Ltd	20.89	2019-06-22
YD80613	MWSK	111	1043704 B.C. Ltd	20.89	2019-06-22
YD80614	MWSK	112	1043704 B.C. Ltd	20.89	2019-06-22
YD80615	MWSK	113	1043704 B.C. Ltd	20.89	2019-06-22
YD80616	MWSK	114	1043704 B.C. Ltd	20.89	2019-06-22
YD80617	MWSK	115	1043704 B.C. Ltd	20.89	2019-06-22
YD80618	MWSK	116	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80619	MWSK	117	1043704 B.C. Ltd	20.89	2019-06-22
YD80620	MWSK	118	1043704 B.C. Ltd	20.89	2019-06-22
YD80621	MWSK	119	1043704 B.C. Ltd	20.89	2019-06-22
YD80622	MWSK	120	1043704 B.C. Ltd	20.89	2019-06-22
YD80623	MWSK	121	1043704 B.C. Ltd	20.89	2019-06-22
YD80624	MWSK	122	1043704 B.C. Ltd	20.89	2019-06-22
YD80625	MWSK	123	1043704 B.C. Ltd	20.89	2019-06-22
YD80626	MWSK	124	1043704 B.C. Ltd	20.89	2019-06-22
YD80627	MWSK	125	1043704 B.C. Ltd	20.89	2019-06-22
YD80628	MWSK	126	1043704 B.C. Ltd	20.89	2019-06-22
YD80629	MWSK	127	1043704 B.C. Ltd	20.89	2019-06-22
YD80630	MWSK	128	1043704 B.C. Ltd	20.89	2019-06-22
YD80631	MWSK	129	1043704 B.C. Ltd	20.89	2019-06-22
YD80632	MWSK	130	1043704 B.C. Ltd	20.89	2019-06-22
YD80633	MWSK	131	1043704 B.C. Ltd	20.89	2019-06-22
YD80634	MWSK	132	1043704 B.C. Ltd	20.62	2019-06-22
YD80635	MWSK	133	1043704 B.C. Ltd	20.89	2019-06-22
YD80636	MWSK	134	1043704 B.C. Ltd	20.89	2019-06-22
YD80637	MWSK	135	1043704 B.C. Ltd	20.89	2019-06-22
YD80638	MWSK	136	1043704 B.C. Ltd	20.89	2019-06-22
YD80639	MWSK	137	1043704 B.C. Ltd	20.89	2019-06-22
YD80640	MWSK	138	1043704 B.C. Ltd	20.89	2019-06-22
YD80641	MWSK	139	1043704 B.C. Ltd	20.89	2019-06-22
YD80642	MWSK	140	1043704 B.C. Ltd	20.89	2019-06-22
YD80643	MWSK	141	1043704 B.C. Ltd	20.89	2019-06-22
YD80644	MWSK	142	1043704 B.C. Ltd	20.89	2019-06-22
YD80645	MWSK	143	1043704 B.C. Ltd	20.89	2019-06-22
YD80646	MWSK	144	1043704 B.C. Ltd	20.89	2019-06-22
YD80647	MWSK	145	1043704 B.C. Ltd	20.89	2019-06-22
YD80648	MWSK	146	1043704 B.C. Ltd	20.89	2019-06-22
YD80649	MWSK	147	1043704 B.C. Ltd	20.89	2019-06-22
YE32229	MWSK	148	1043704 B.C. Ltd	20.89	2019-06-22
YE32230	MWSK	149	1043704 B.C. Ltd	20.89	2019-06-22
YD80652	MWSK	150	1043704 B.C. Ltd	20.89	2019-06-22
YD80653	MWSK	151	1043704 B.C. Ltd	20.89	2019-06-22
YD80654	MWSK	152	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80655	MWSK	153	1043704 B.C. Ltd	20.89	2019-06-22
YD80656	MWSK	154	1043704 B.C. Ltd	20.89	2019-06-22
YD80657	MWSK	155	1043704 B.C. Ltd	20.89	2019-06-22
YD80658	MWSK	156	1043704 B.C. Ltd	20.89	2019-06-22
YD80659	MWSK	157	1043704 B.C. Ltd	20.89	2019-06-22
YD80660	MWSK	158	1043704 B.C. Ltd	20.89	2019-06-22
YD80661	MWSK	159	1043704 B.C. Ltd	20.89	2019-06-22
YD80662	MWSK	160	1043704 B.C. Ltd	20.89	2019-06-22
YD80663	MWSK	161	1043704 B.C. Ltd	20.89	2019-06-22
YD80664	MWSK	162	1043704 B.C. Ltd	20.89	2019-06-22
YD80665	MWSK	163	1043704 B.C. Ltd	20.89	2019-06-22
YD80666	MWSK	164	1043704 B.C. Ltd	20.89	2019-06-22
YD80667	MWSK	165	1043704 B.C. Ltd	20.89	2019-06-22
YD80668	MWSK	166	1043704 B.C. Ltd	20.89	2019-06-22
YD80669	MWSK	167	1043704 B.C. Ltd	20.89	2019-06-22
YD80670	MWSK	168	1043704 B.C. Ltd	20.89	2019-06-22
YD80671	MWSK	169	1043704 B.C. Ltd	20.89	2019-06-22
YD80672	MWSK	170	1043704 B.C. Ltd	20.89	2019-06-22
YD80673	MWSK	171	1043704 B.C. Ltd	20.89	2019-06-22
YD80674	MWSK	172	1043704 B.C. Ltd	20.89	2019-06-22
YD80675	MWSK	173	1043704 B.C. Ltd	20.89	2019-06-22
YD80676	MWSK	174	1043704 B.C. Ltd	20.89	2019-06-22
YD80677	MWSK	175	1043704 B.C. Ltd	20.89	2019-06-22
YD80678	MWSK	176	1043704 B.C. Ltd	16.02	2019-06-22
YD80679	MWSK	177	1043704 B.C. Ltd	20.89	2019-06-22
YD80680	MWSK	178	1043704 B.C. Ltd	20.89	2019-06-22
YD80681	MWSK	179	1043704 B.C. Ltd	20.89	2019-06-22
YD80682	MWSK	180	1043704 B.C. Ltd	20.89	2019-06-22
YD80683	MWSK	181	1043704 B.C. Ltd	20.89	2019-06-22
YD80684	MWSK	182	1043704 B.C. Ltd	20.89	2019-06-22
YD80685	MWSK	183	1043704 B.C. Ltd	20.89	2019-06-22
YD80686	MWSK	184	1043704 B.C. Ltd	20.89	2019-06-22
YD80687	MWSK	185	1043704 B.C. Ltd	20.89	2019-06-22
YD80688	MWSK	186	1043704 B.C. Ltd	20.89	2019-06-22
YD80689	MWSK	187	1043704 B.C. Ltd	20.89	2019-06-22
YD80690	MWSK	188	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80691	MWSK	189	1043704 B.C. Ltd	20.89	2019-06-22
YD80692	MWSK	190	1043704 B.C. Ltd	20.89	2019-06-22
YD80693	MWSK	191	1043704 B.C. Ltd	20.89	2019-06-22
YD80694	MWSK	192	1043704 B.C. Ltd	20.89	2019-06-22
YD80695	MWSK	193	1043704 B.C. Ltd	20.89	2019-06-22
YD80696	MWSK	194	1043704 B.C. Ltd	20.89	2019-06-22
YD80697	MWSK	195	1043704 B.C. Ltd	20.89	2019-06-22
YD80698	MWSK	196	1043704 B.C. Ltd	20.89	2019-06-22
YD80699	MWSK	197	1043704 B.C. Ltd	20.89	2019-06-22
YD80700	MWSK	198	1043704 B.C. Ltd	20.89	2019-06-22
YD80701	MWSK	199	1043704 B.C. Ltd	20.89	2019-06-22
YD80702	MWSK	200	1043704 B.C. Ltd	20.89	2019-06-22
YD80703	MWSK	201	1043704 B.C. Ltd	20.89	2019-06-22
YD80704	MWSK	202	1043704 B.C. Ltd	20.89	2019-06-22
YD80705	MWSK	203	1043704 B.C. Ltd	20.89	2019-06-22
YD80706	MWSK	204	1043704 B.C. Ltd	20.89	2019-06-22
YD80707	MWSK	205	1043704 B.C. Ltd	20.89	2019-06-22
YD80708	MWSK	206	1043704 B.C. Ltd	20.89	2019-06-22
YD80709	MWSK	207	1043704 B.C. Ltd	20.89	2019-06-22
YD80710	MWSK	208	1043704 B.C. Ltd	20.89	2019-06-22
YD80711	MWSK	209	1043704 B.C. Ltd	20.89	2019-06-22
YD80712	MWSK	210	1043704 B.C. Ltd	20.89	2019-06-22
YD80713	MWSK	211	1043704 B.C. Ltd	20.89	2019-06-22
YD80714	MWSK	212	1043704 B.C. Ltd	20.89	2019-06-22
YD80715	MWSK	213	1043704 B.C. Ltd	20.89	2019-06-22
YD80716	MWSK	214	1043704 B.C. Ltd	20.89	2019-06-22
YD80717	MWSK	215	1043704 B.C. Ltd	20.89	2019-06-22
YD80718	MWSK	216	1043704 B.C. Ltd	20.89	2019-06-22
YD80719	MWSK	217	1043704 B.C. Ltd	20.89	2019-06-22
YD80720	MWSK	218	1043704 B.C. Ltd	20.89	2019-06-22
YD80721	MWSK	219	1043704 B.C. Ltd	20.89	2019-06-22
YD80722	MWSK	220	1043704 B.C. Ltd	20.89	2019-06-22
YD80723	MWSK	221	1043704 B.C. Ltd	20.89	2019-06-22
YD80724	MWSK	222	1043704 B.C. Ltd	20.89	2019-06-22
YD80725	MWSK	223	1043704 B.C. Ltd	20.89	2019-06-22
YD80726	MWSK	224	1043704 B.C. Ltd	20.89	2019-06-22

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD80727	MWSK	225	1043704 B.C. Ltd	20.89	2019-06-22
YD80728	MWSK	226	1043704 B.C. Ltd	20.89	2019-06-22
YD80729	MWSK	227	1043704 B.C. Ltd	20.89	2019-06-22
YD80730	MWSK	228	1043704 B.C. Ltd	20.89	2019-06-22
YD80731	MWSK	229	1043704 B.C. Ltd	20.89	2019-06-22
YD80732	MWSK	230	1043704 B.C. Ltd	20.89	2019-06-22
YD80733	MWSK	231	1043704 B.C. Ltd	20.89	2019-06-22
YD80734	MWSK	232	1043704 B.C. Ltd	14.92	2019-06-22
YD80735	MWSK	233	1043704 B.C. Ltd	14.92	2019-06-22
YD80736	MWSK	234	1043704 B.C. Ltd	14.92	2019-06-22
YD80737	MWSK	235	1043704 B.C. Ltd	14.92	2019-06-22
YD80738	MWSK	236	1043704 B.C. Ltd	14.92	2019-06-22
YD80739	MWSK	237	1043704 B.C. Ltd	14.92	2019-06-22
YD80740	MWSK	238	1043704 B.C. Ltd	20.89	2019-06-22
YD80741	MWSK	239	1043704 B.C. Ltd	20.89	2019-06-22
YD80742	MWSK	240	1043704 B.C. Ltd	20.89	2019-06-22
YD80743	MWSK	241	1043704 B.C. Ltd	20.55	2019-06-22
YD80744	MWSK	242	1043704 B.C. Ltd	20.89	2019-06-22
YD80745	MWSK	243	1043704 B.C. Ltd	20.89	2019-06-22
YD80746	MWSK	244	1043704 B.C. Ltd	20.89	2019-06-22
YD80747	MWSK	245	1043704 B.C. Ltd	20.89	2019-06-22
YD80748	MWSK	246	1043704 B.C. Ltd	20.89	2019-06-22
YD80749	MWSK	247	1043704 B.C. Ltd	20.89	2019-06-22
YD80750	MWSK	248	1043704 B.C. Ltd	20.89	2019-06-22
YD80751	MWSK	249	1043704 B.C. Ltd	20.89	2019-06-22
YD80752	MWSK	250	1043704 B.C. Ltd	20.89	2019-06-22
YD80753	MWSK	251	1043704 B.C. Ltd	20.89	2019-06-22
YD80754	MWSK	252	1043704 B.C. Ltd	20.89	2019-06-22
YD80755	MWSK	253	1043704 B.C. Ltd	20.89	2019-06-22
YD80756	MWSK	254	1043704 B.C. Ltd	20.89	2019-06-22
YD80757	MWSK	255	1043704 B.C. Ltd	20.89	2019-06-22
YD80758	MWSK	256	1043704 B.C. Ltd	20.89	2019-06-22
YD80759	MWSK	257	1043704 B.C. Ltd	20.89	2019-06-22
YD80760	MWSK	258	1043704 B.C. Ltd	9.33	2019-06-22
YD80761	MWSK	259	1043704 B.C. Ltd	5.92	2019-06-22
YD80762	MWSK	260	1043704 B.C. Ltd	16.94	2019-06-22

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YD80763	OX	261	1043704 B.C. Ltd	15.49	2019-06-22
YD80764	OX	262	1043704 B.C. Ltd	16.19	2019-06-22
YD80765	OX	263	1043704 B.C. Ltd	9.29	2019-06-22
YD80766	OX	264	1043704 B.C. Ltd	3.28	2019-06-22
YD80767	OX	265	1043704 B.C. Ltd	1.09	2019-06-22
YD80768	OX	266	1043704 B.C. Ltd	3.11	2019-06-22
YD80769	OX	267	1043704 B.C. Ltd	9.34	2019-06-22
YD80770	OX	268	1043704 B.C. Ltd	11.76	2019-06-22
YD80771	OX	269	1043704 B.C. Ltd	5.13	2019-06-22
YD80772	OX	270	1043704 B.C. Ltd	0.94	2019-06-22
YF35387	QC	1	0905144 B.C. Ltd	20.89	2019-01-09
YF35388	QC	2	0905144 B.C. Ltd	20.89	2019-01-09
YF35497	QC	3	0905144 B.C. Ltd	20.89	2019-01-09
YF35498	QC	4	0905144 B.C. Ltd	20.89	2019-01-09
YF35499	QC	5	0905144 B.C. Ltd	20.89	2019-01-09
YF35500	QC	6	0905144 B.C. Ltd	20.89	2019-01-09
60767	QUILL	1	0905144 B.C. Ltd	16.78	2020-12-05
60768	QUILL	2	0905144 B.C. Ltd	17.13	2020-12-05
60769	QUILL	3	0905144 B.C. Ltd	20.89	2020-12-05
60770	QUILL	4	0905144 B.C. Ltd	20.55	2020-12-05
60771	QUILL	5	0905144 B.C. Ltd	20.78	2020-12-05
60772	QUILL	6	0905144 B.C. Ltd	20.78	2020-12-05
60773	QUILL	7	0905144 B.C. Ltd	14.01	2020-12-05
60774	QUILL	8	0905144 B.C. Ltd	16.52	2020-12-05
70829	QUILL		0905144 B.C. Ltd	11.14	2020-12-05
60791	RAM	1	0905144 B.C. Ltd	15.76	2020-12-05
60792	RAM	2	0905144 B.C. Ltd	20.88	2020-12-05
60793	RAM	3	0905144 B.C. Ltd	20.07	2020-12-05
60794	RAM	4	0905144 B.C. Ltd	19.86	2020-12-05
60795	RAM	5	0905144 B.C. Ltd	7.89	2020-12-05
60796	RAM	6	0905144 B.C. Ltd	22.07	2020-12-05
60797	RAM	7	0905144 B.C. Ltd	16.18	2020-12-05
60798	RAM	8	0905144 B.C. Ltd	13.55	2020-12-05
63037	RED	1	0905144 B.C. Ltd	15.34	2020-12-05
63038	RED	2	0905144 B.C. Ltd	13.53	2020-12-05
63039	RED	3	0905144 B.C. Ltd	16.09	2020-12-05

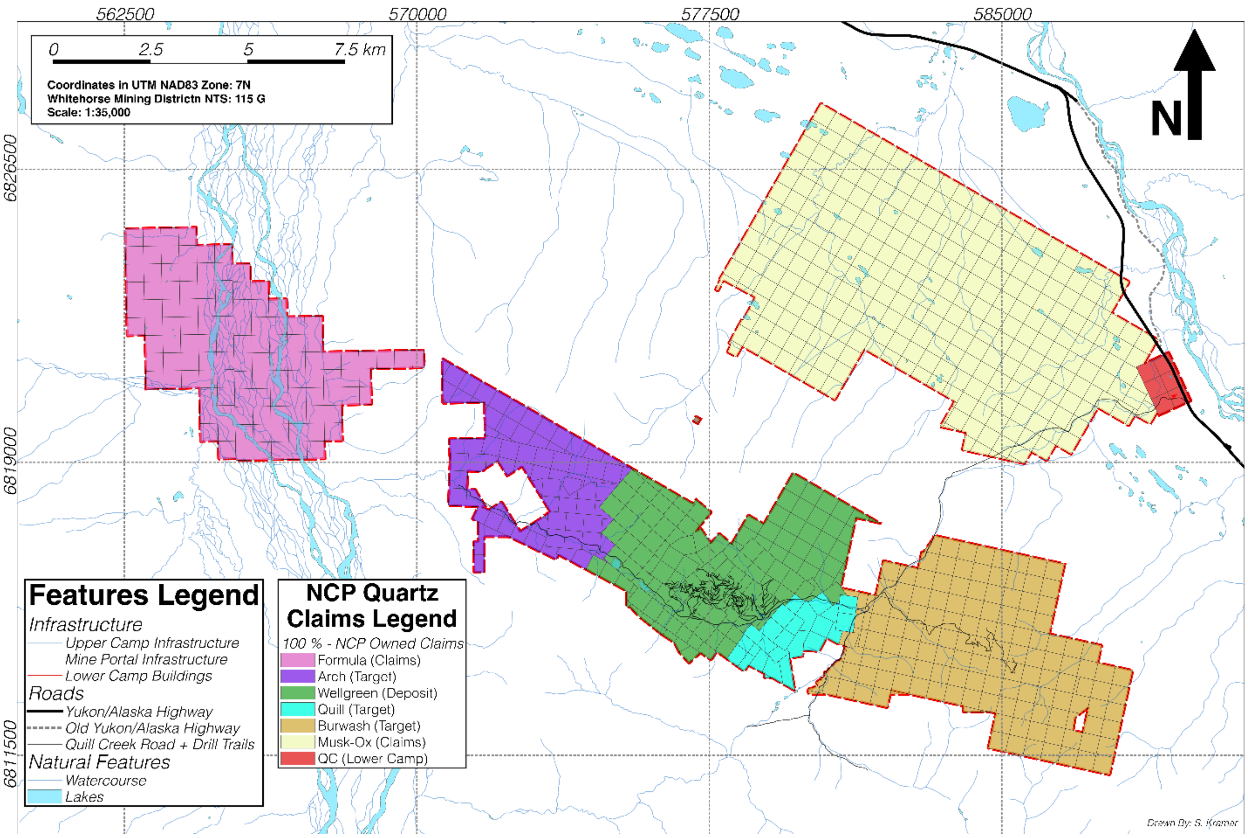
Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
63040	RED	4	0905144 B.C. Ltd	20.69	2020-12-05
63041	RED	5	0905144 B.C. Ltd	20.87	2020-12-05
63042	RED	6	0905144 B.C. Ltd	15.65	2020-12-05
63043	RED	7	0905144 B.C. Ltd	15.46	2020-12-05
63044	RED	8	0905144 B.C. Ltd	19.1	2020-12-05
71432	ROSS	1	0905144 B.C. Ltd	16.47	2020-12-05
71433	ROSS	2	0905144 B.C. Ltd	19.75	2020-12-05
71434	ROSS	3	0905144 B.C. Ltd	13.18	2020-12-05
71435	ROSS	4	0905144 B.C. Ltd	11.97	2020-12-05
64076	ROSS	15	0905144 B.C. Ltd	20.74	2020-12-05
64077	ROSS	16	0905144 B.C. Ltd	20.74	2020-12-05
64066	ROSS	25	0905144 B.C. Ltd	15.94	2020-12-05
64086	ROSS	85	0905144 B.C. Ltd	20.88	2020-12-05
64087	ROSS	86	0905144 B.C. Ltd	21.11	2020-12-05
64084	ROSS	94	0905144 B.C. Ltd	22.04	2020-12-05
64085	ROSS	95	0905144 B.C. Ltd	23.86	2020-12-05
64587	ROSS	96	0905144 B.C. Ltd	23.98	2020-12-05
YC40144	RUB	1	0905144 B.C. Ltd	20.9	2029-02-23
YC40145	RUB	2	0905144 B.C. Ltd	20.9	2029-02-23
YC40146	RUB	3	0905144 B.C. Ltd	20.9	2029-02-23
YC40147	RUB	4	0905144 B.C. Ltd	20.9	2029-02-23
YC40148	RUB	5	0905144 B.C. Ltd	20.9	2029-02-23
YC40149	RUB	6	0905144 B.C. Ltd	20.9	2029-02-23
YC40150	RUB	7	0905144 B.C. Ltd	20.9	2029-02-23
YC40151	RUB	8	0905144 B.C. Ltd	20.9	2029-02-23
YC40152	RUB	9	0905144 B.C. Ltd	20.9	2029-02-23
YC40153	RUB	10	0905144 B.C. Ltd	20.9	2029-02-23
YC40154	RUB	11	0905144 B.C. Ltd	20.9	2029-02-23
YC40155	RUB	12	0905144 B.C. Ltd	20.9	2029-02-23
YC40156	RUB	13	0905144 B.C. Ltd	20.9	2029-02-23
YC40157	RUB	14	0905144 B.C. Ltd	20.9	2029-02-23
YC40158	RUB	15	0905144 B.C. Ltd	20.9	2029-02-23
YC40159	RUB	16	0905144 B.C. Ltd	20.9	2029-02-23
YC40160	RUB	17	0905144 B.C. Ltd	20.9	2029-02-23
YC40161	RUB	18	0905144 B.C. Ltd	20.9	2029-02-23
YC40162	RUB	19	0905144 B.C. Ltd	20.9	2029-02-23

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YC40163	RUB	20	0905144 B.C. Ltd	20.9	2029-02-23
YC40164	RUB	21	0905144 B.C. Ltd	20.77	2029-02-23
YC40165	RUB	22	0905144 B.C. Ltd	20.9	2029-02-23
YC40166	RUB	23	0905144 B.C. Ltd	14.03	2029-02-23
YC40167	RUB	24	0905144 B.C. Ltd	20.9	2029-02-23
YC40168	RUB	25	0905144 B.C. Ltd	20.9	2029-02-23
YC40169	RUB	26	0905144 B.C. Ltd	20.9	2029-02-23
YC40170	RUB	27	0905144 B.C. Ltd	20.9	2029-02-23
YC40171	RUB	28	0905144 B.C. Ltd	20.9	2029-02-23
YC40172	RUB	29	0905144 B.C. Ltd	20.9	2029-02-23
63013	SAM	1	0905144 B.C. Ltd	6.04	2020-12-05
63014	SAM	2	0905144 B.C. Ltd	9.72	2020-12-05
63015	SAM	3	0905144 B.C. Ltd	15.78	2020-12-05
63016	SAM	4	0905144 B.C. Ltd	10.64	2020-12-05
63017	SAM	5	0905144 B.C. Ltd	12.55	2020-12-05
63018	SAM	6	0905144 B.C. Ltd	16.92	2020-12-05
63019	SAM	7	0905144 B.C. Ltd	14.27	2020-12-05
63020	SAM	8	0905144 B.C. Ltd	10.32	2020-12-05
60783	WAGONER	1	0905144 B.C. Ltd	18.46	2020-12-05
60784	WAGONER	2	0905144 B.C. Ltd	18.46	2020-12-05
60785	WAGONER	3	0905144 B.C. Ltd	13.58	2020-12-05
60786	WAGONER	4	0905144 B.C. Ltd	14.37	2020-12-05
60787	WAGONER	5	0905144 B.C. Ltd	16	2020-12-05
60788	WAGONER	6	0905144 B.C. Ltd	16	2020-12-05
60789	WAGONER	7	0905144 B.C. Ltd	13.88	2020-12-05
60790	WAGONER	8	0905144 B.C. Ltd	15.14	2020-12-05
YD87963	WG	1	0905144 B.C. Ltd	1.99	2019-02-13
YD87964	WG	2	0905144 B.C. Ltd	3.51	2019-02-13
YD87965	WG	3	0905144 B.C. Ltd	3.92	2019-02-13
YD87966	WG	4	0905144 B.C. Ltd	20.89	2019-02-13
YD87967	WG	5	0905144 B.C. Ltd	1.58	2019-02-13
YD87968	WG	6	0905144 B.C. Ltd	1.10	2019-02-13
YD87969	WG	7	0905144 B.C. Ltd	2.34	2019-02-13
YD87970	WG	8	0905144 B.C. Ltd	0.00	2019-02-13
YD87971	WG	9	0905144 B.C. Ltd	2.55	2019-02-13
YD87972	WG	10	0905144 B.C. Ltd	4.86	2019-02-13

Grant Number	Claim Name	Claim Number	Owner	Area (ha)	Expiry Date
YD87973	WG	11	0905144 B.C. Ltd	0.71	2019-02-13
YD87974	WG	12	0905144 B.C. Ltd	20.07	2019-02-13
YD87975	WG	13	0905144 B.C. Ltd	16.83	2019-02-13
YD87976	WG	14	0905144 B.C. Ltd	13.43	2019-02-13
YD87977	WG	15	0905144 B.C. Ltd	10.03	2019-02-13
YD87978	WG	16	0905144 B.C. Ltd	6.64	2019-02-13
YD87979	WG	17	0905144 B.C. Ltd	3.24	2019-02-13
YE91102	LYNX	1	0905144 B.C. Ltd	20.90	2019-08-28
YE91103	LYNX	2	0905144 B.C. Ltd	20.90	2019-08-28
YE91104	LYNX	3	0905144 B.C. Ltd	20.90	2019-08-28
YE91105	LYNX	4	0905144 B.C. Ltd	20.90	2019-08-28
YE91106	LYNX	5	0905144 B.C. Ltd	3.16	2019-08-28
YE91107	LYNX	6	0905144 B.C. Ltd	16.23	2019-08-28
YE91108	LYNX	7	0905144 B.C. Ltd	20.90	2019-08-28
YE91109	LYNX	8	0905144 B.C. Ltd	20.90	2019-08-28
YE91110	LYNX	9	0905144 B.C. Ltd	20.90	2019-08-28
YE91111	LYNX	10	0905144 B.C. Ltd	20.90	2019-08-28
YE91112	LYNX	11	0905144 B.C. Ltd	20.90	2019-08-28
YE91113	LYNX	12	0905144 B.C. Ltd	4.88	2019-08-28
YE91114	LYNX	13	0905144 B.C. Ltd	10.68	2019-08-28
YE91115	LYNX	14	0905144 B.C. Ltd	20.90	2019-08-28
YE91116	LYNX	15	0905144 B.C. Ltd	20.15	2019-08-28
YE91117	LYNX	16	0905144 B.C. Ltd	3.42	2019-08-28

Lease 115G11-003 covers a 21.7 ha parcel of land located adjacent to kilometre 1727 on the Alaska Highway (Figure 4-3). This 10-year lease was granted on November 1, 2012 and expires on October 31, 2022. Northern Platinum held a similar but larger (62.7 ha) lease parcel from November 1, 2001 until October 31, 2011 that included the historic Hudson-Yukon mill site used in the 1970s. Pursuant to the requirements of the previous surface lease, which included the Mill Site, Northern Platinum finalized a Reclamation Plan for the Mill Site, which was approved by the Government of Yukon in early 2010. Upon approval, the footprint of the mill area was separated from 115G11-003, and the current footprint of the lease is presented in Figure 4-3. Final accepted closure of the Reclamation Plan remains ongoing and is still being evaluated and is to be cost shared between HudBay, the Government of Yukon, and Nickel Creek.

Figure 4-2: Mineral Tenure



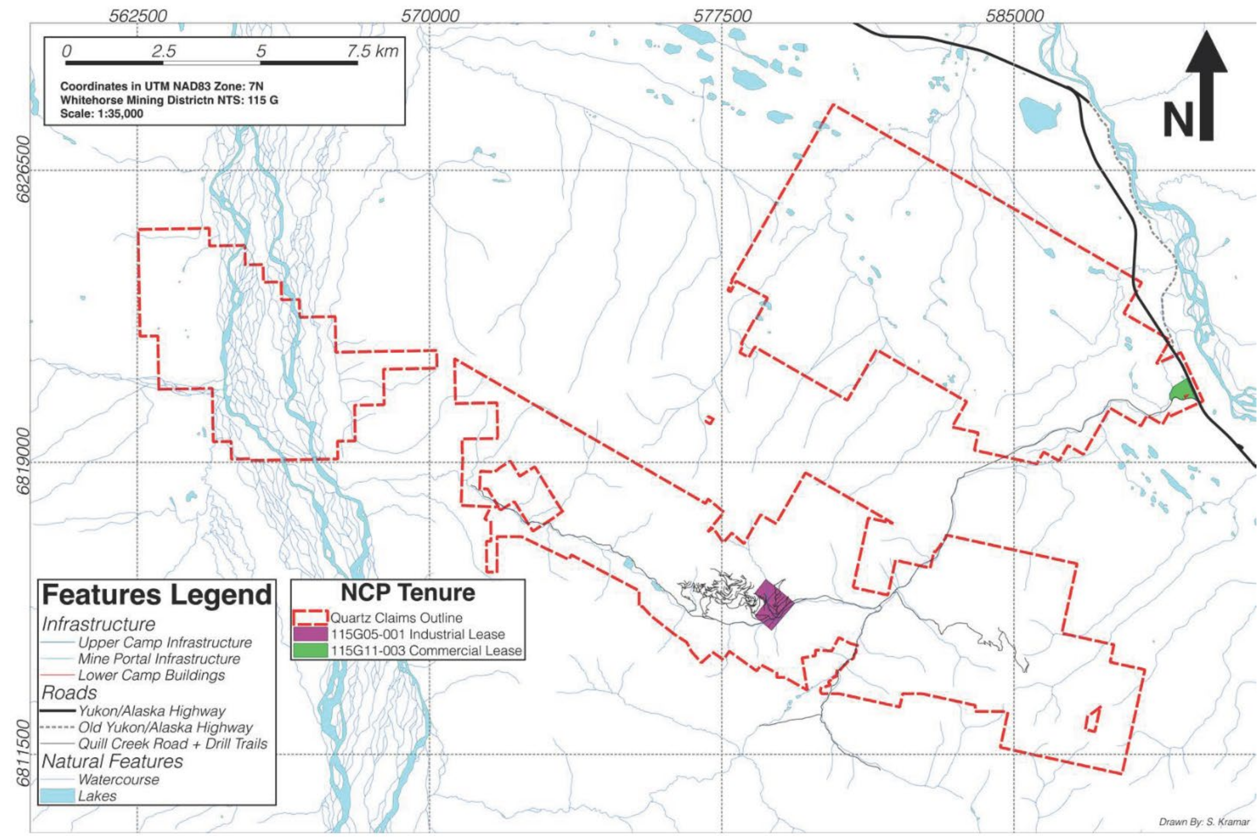
Source: Nickel Creek, 2018

Table 4-2: Surface Leases

Land Disposition #	Pid	Application	Disposition	Tenure Purpose	Area (ha)	Disposition Date	Expiry Date
2753634	100015069	2363L	115G05-001	Industrial	69.7	24/08/1971	31/05/2034
2753541	100023288		115G11-003	Commercial	21.7	20/01/1971	31/10/2022

Source: Nickel Creek, 2018

Figure 4-3: Surface Leases



Source: Nickel Creek, 2018

4.4 Property Ownership and History

Nickel Creek has owned a consolidated 100% interest in the Project property since June 2011. Details of how Nickel Creek acquired its 100% ownership of the Project property are summarized below.

An underlying agreement dated April 27, 1999 relating to Northern Platinum's interest in the Arch Joint Venture (Arch Agreement) was entered into between Kaieteur Resource Corporation (Kaieteur) (formerly International All-North Resources Ltd. (All-North)), Northern Platinum, and J. Patrick Sheridan. Under the Arch Agreement, Northern Platinum agreed to purchase from Kaieteur all of its All-North interest in the Project property, and its interest in the Arch Joint Venture on an "as is" basis for a sum of CDN $62,500 to be paid in cash and shares. The Arch

Agreement acknowledged that Northern Platinum had already earned a 20% interest in the Project property and, under the Arch Agreement, Northern Platinum acquired the remaining 80% interest. Kaieteur warranted it was the beneficial owner of All-North's interest in the Project property interest but did not provide the same warranties for the Arch Joint Venture because certain historical documentation for underlying agreements was incomplete; hence the "as is" stipulation. On September 22, 2010, Northern Platinum (who at that time owned a 100% interest in the Project property, subject to a 50% back-in right held by Belleterre Quebec Mines Ltd.) was acquired by Prophecy Resource Corp. As a result, Prophecy Resource Corp. became the owner of a 100% interest in the Project property (subject to the 50% back-in right held by Belleterre Quebec). Subsequently on September 24, 2010, Prophecy Resource Corp. acquired the 50% back-in right held by Belleterre Quebec, resulting in Prophecy Resource Corp. acquiring a 100% interest in the Project, free of any back-in rights.

In June 2011, Prophecy Resource Corp. spun out all of its North American platinum and nickel assets, including its entire 100% interest in the Project, to 0905144 B.C. Ltd., a wholly-owned subsidiary of Pacific Coast Nickel Corp. (Wellgreen Platinum and Nickel Creek Platinum's predecessor company). As a result of the spin-out transaction, Pacific Coast Nickel Corp. acquired 100% ownership of the Project.

Immediately upon completion of this spin-out transaction, Pacific Coast Nickel Corp. changed its name to Prophecy Platinum Corp., and in December 2013, Prophecy Platinum Corp. changed its name to Wellgreen Platinum Ltd. In January 2018, Wellgreen Platinum Ltd. changed its name to Nickel Creek Platinum Corp.

4.5 Royalties

On November 4, 2015, the Company entered into a transaction whereby it sold to Resource Capital Fund VI L.P., Australind Limited, and Vernon Taylor III, collectively, an aggregate 1% NSR royalty on future production from the Project.

4.6 Permits

In Yukon, the Quartz Mining Land Use Regulation and the Placer Mining Land Use Regulation consist of a classification system based on varying levels of specific activities. These threshold levels categorize exploration activities into four classes of operation. Classes 1 through 4 represent activities with increasing potential to cause adverse environmental impacts.

Nickel Creek currently holds one Class 4 approval through Energy, Mines and Resources (EMR) Land Use Division (see Figure 4-4).

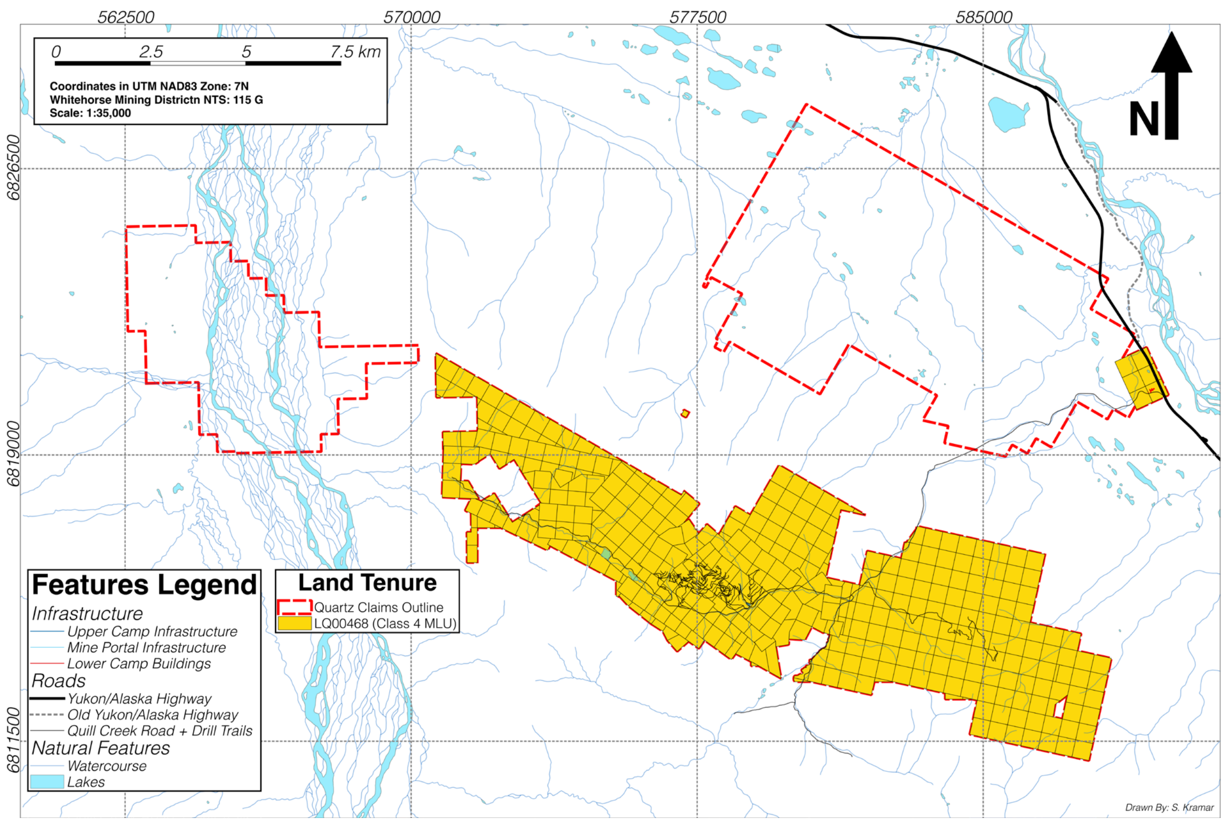
Class 4 Permit LQ00468 covers the claims on which the current mineral resource has been delineated, as well as the historic upper camp of the Project located on surface Lease 115G05-001. This permit expires February 8, 2028.

Class 4 Programs require:

- submission of a detailed operating plan to the Mining Lands Office
- assessment through Yukon Environmental and Socio-economic Assessment Board (YESAB)
- that the operating plan be approved before any other exploration activities can proceed

The operating plan may entail multi-year exploration programs to allow greater flexibility for the operator. Class 4 Program terms and conditions are presented in Table 4-3

Figure 4-4: Class 4 Operating Permit



Source: Nickel Creek Platinum, 2018

Table 4-3: Class 4 Operating Permit Terms

Element	Terms and Conditions
Establishing new access roads per program	On claims for 1 km
Off-Road use of vehicles in summer	Yes
Corridor width	
Lines	1 m wide x 20 km
Establishment of trails per program	4 m wide x 20 km
# of clearings per claim, including existing clearings	Up to 30 clearings per claim
Surface area of each clearing	Approximately 400 m^2.
Total volume of trenching	Up to 2000 m^3
# of person days per camp	18,250
# of persons in a camp at any one time	50 persons
Fuel Storage in a stationary container	Diesel: 198,362 L
	Gasoline: 12,200 L
Upgrading of access roads per	On claims for approximately 27 kms
Use of vehicles on existing roads or trails	January 1st to December 31st

Source: Nickel Creek, Yukon Government - Energy, Mines and Resources, 2018 (for entirety of operating conditions, please see public document LQ00468 NCP Class 4 approval)

4.7 Environmental Liabilities

Nickel Creek has cleaned up surface debris at the old mill site and removed contaminated soils, pursuant to the Reclamation Plan referred to in Section 4.3 and in accordance with the terms of the old surface lease. These activities were initiated in 2009 and completed in 2013 under the direction of Access Consulting Group of Whitehorse. The majority of the contaminated soils on the existing Lease 115G11-003 have now been removed and disposed of in Tervita's Northern Rockies Landfill in Fort Nelson, B.C.

Some additional reclamation activities remain outstanding associated with the historic HudBay Mill Site and 1970s tailings impoundments which are not on Nickel Creek controlled lands. The Company has entered into a preliminary cooperative working arrangement with Yukon Government and HudBay to assess the reclamation work that will need to be conducted. The financial effect and timing of the reclamation work is indeterminable at this time. Once the assessment is completed and a contractual agreement is entered into, a portion of the financial cost for reclamation may be incurred by the Company.

4.8 First Nations

Surface rights legislation for Yukon First Nations is provided under the Umbrella Final Agreement between the Government of Canada, Government of Yukon, and The Council for Yukon First

Nations. This legislation provides a mechanism to resolve disputes over access rights (Mining Yukon 2011 and Minister of Public Works and Government Services Canada 2003).

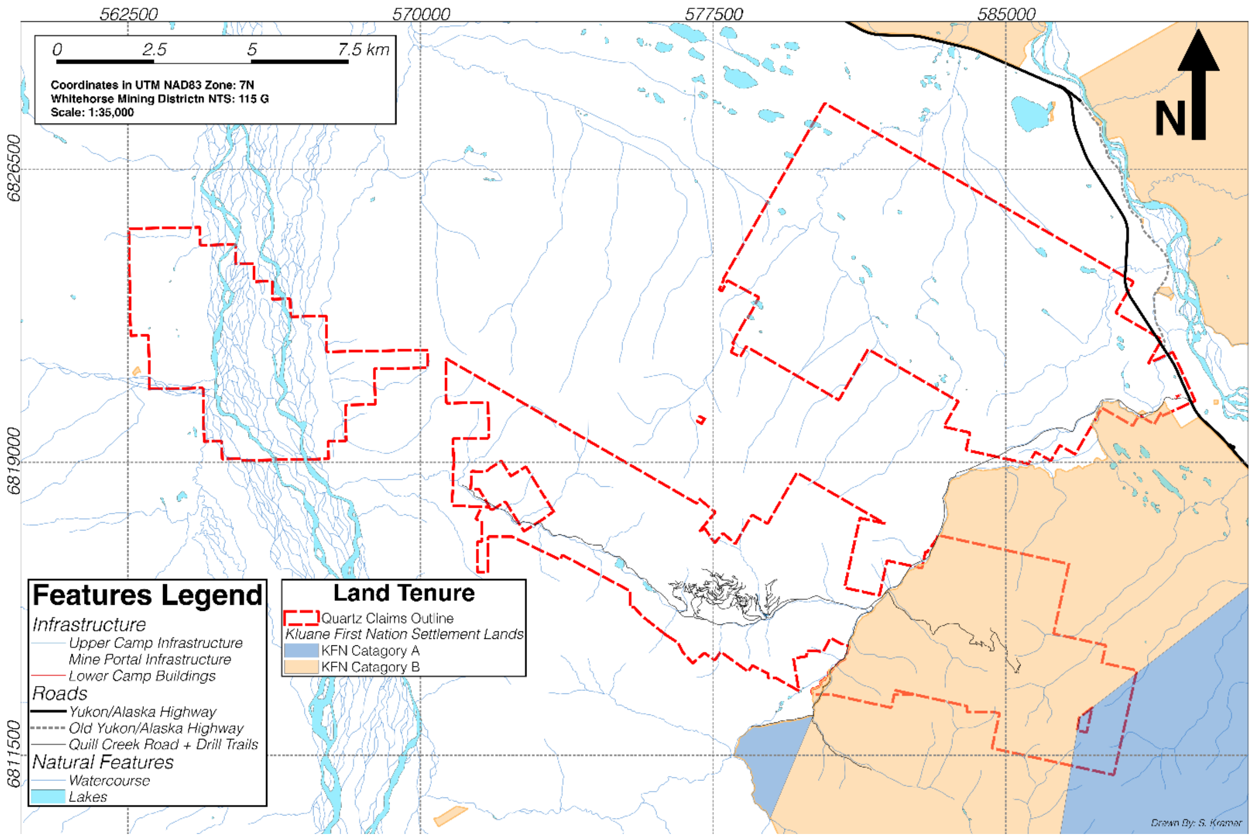
The Kluane First Nation has a settled land claim, which provides them with access, rights and obligations to land and resources, and the right to govern their own affairs. The Kluane First Nation signed final and self-government agreements with the Yukon and Canadian governments on October 18, 2003. The effective date of these agreements was February 2, 2004 (Yukon ECO 2011a).

The Project is located in the "core area" of the Kluane First Nation as defined by the Umbrella Final Agreement. The Project partially overlaps on Category B land (R-49 B) and Category A (R-01A) land owned by the Kluane First Nation (Figure 4-5) (Minister of Public Works and Government Services Canada 2003). As of the signing of the Kluane First Nation Final Agreement, the Kluane First Nation holds both the surface rights and the subsurface/mineral rights on Category A land, while on Category B land the Kluane First Nation owns the surface rights to this land, but not that which is below the surface. However, land belonging to persons holding a right, title, interest, license, and permit on the land prior to the time the area was claimed as Settlement Land are not subject to this legislation (Minister of Public Works and Government Services Canada 2003). The Burwash claims, which are on Category B land, were held prior to the settlement agreement.

The White River First Nation finalized negotiations toward final and self-government agreements with the Canadian and Yukon governments in 2002, when a Memorandum of Understanding (MOU) was signed, signifying the completion of the negotiation process. However, the White River First Nation decided not to ratify the negotiated agreements and there have been no negotiations since. As such, the White River First Nation does not have a settled land claim. Under the terms of the Umbrella Final Agreement, the White River First Nation was allocated Category A and Category B land in their "core area", which have been "interim protected" from third-party interests, pending the settlement or abandonment of a land claim agreement (Yukon ECO 2011b). The "core area" for White River First Nation lies well to the west and north of the Project and is separated from the Kluane First Nation "core area" by an area of overlapping traditional use. On December 18, 2014, the White River First Nations and Government of the Yukon Territory jointly announced the two parties have initiated preliminary negotiations with the goal of reaching a reconciliation agreement. The intent of the reconciliation agreement discussions is to provide the parties with a process to constructively resolve issues relating to land use and other matters.

Nickel Creek signed an Exploration Co-operation Agreement (ECA) with the Kluane First Nation effective August 1, 2012, pursuant to which regular ECA meetings are held between Nickel Creek and the Kluane First Nation. The ECA also provides that Nickel Creek will continue to engage the White River First Nation with respect to discussions related to community presentations as well as training and employment opportunities.

Figure 4-5: Kluane First Nation Land Status



Source: Nickel Creek, 2018

4.9 Significant Risk Factors

Other than as set out in this Section 4, to the extent known, there are no other environmental liabilities to which the Project is subject and no other significant factors that may affect access, title, or the right or ability to perform work on the Project.

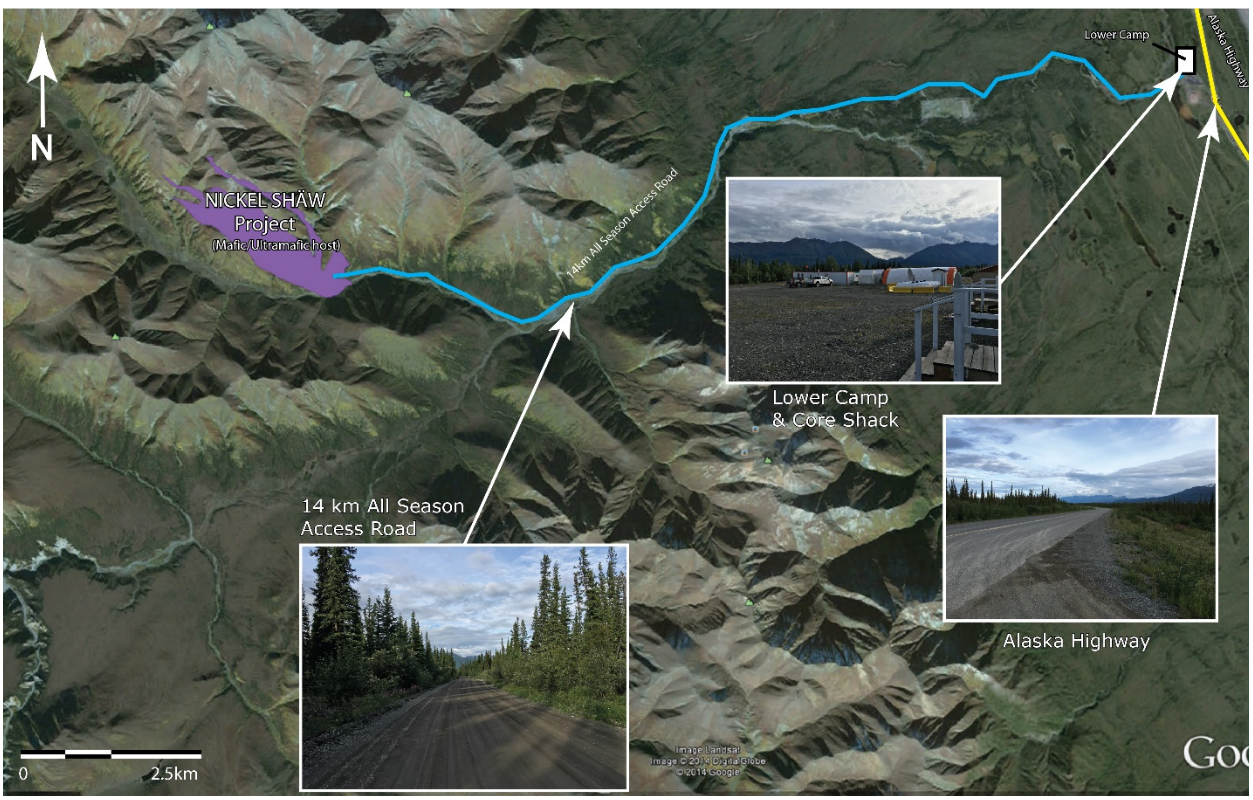
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The Project is located approximately 317 km northwest of Whitehorse, Yukon and can be reached via the paved Alaska Highway which is maintained by the Government of Yukon (approximately kilometre 1727). From the highway to the Project travel is by all-weather, gravel road that runs southwest beside Quill Creek for a distance of 14 km (Figure 5-1).

An all-weather airstrip is located approximately 28 km southeast of the Project at Burwash Landing. It is maintained by NAV CANADA and presently sees limited winter maintenance.

All-season, deep-sea ports are located in Haines, Alaska, 390 km to the southeast, and Skagway, Alaska, which is currently utilized by Capstone Mining Corp. and G&T Resources Corp. for the transport of mining concentrate material on bulk container ships to smelters. Both ports are year-round ice-free ports and are accessible by high-quality paved highways.

Figure 5-1: Project Access and Location



Source: Nickel Creek, 2018

5.1 Climate

The regional climate is semi-arid, sub-arctic with relatively warm, dry summers and winters characterized by relatively dry, cold interior conditions but tempered by west coast climate influences. Weather records have been historically recorded at the Burwash Landing weather station (806.8 masl). The area lies in the rain shadow of the Saint Elias Mountains, with average annual total precipitation for the Burwash Landing station of 27.97 cm (11 inches) of which 19.2 cm (7.6 inches) typically falls as rain in summer and the remainder as snow in winter.

Exploration drilling has historically been done sporadically throughout the year, but potential future operations would be configured for year around operations.

Tetra Tech (EBA) from Whitehorse installed a meteorological station near the historic Upper Camp, approximately 600 m southeast of the historic mine portal on October 27, 2012. It consists of a standard 10 m tower with instrumentation to measure wind speed and direction, air temperature, relative humidity, barometric pressure, incident solar radiation, and water-equivalent precipitation. An evaporation pan was installed in June 2013 at the same location to enable evaporation rates to be recorded over the summer months. Data is collected and stored on a regular basis by EBA.

Data collection recorded over the past five years returned the following details:

- maximum air temperature was 27.5°C on June 10, 2017
- minimum air temperature was -37.4°C on January 28, 2013
- greatest monthly precipitation was 27.5 cm in December 2013
- least monthly precipitation was 0.0 cm in November 2014

5.2 Local Resources Infrastructure

The villages of Burwash Landing and Destruction Bay are located approximately 27 km and 43 km, respectively, southeast from the Project. In addition to the airstrip at Burwash Landing, these villages have lodging, food, and fuel with the potential for future subdivision development to provide housing for mining personnel.

5.2.1 Power

Generators installed for the exploration programs currently supply power to the Project. Haines Junction is the current limit of the high capacity grid and hydroelectric system of Yukon Energy Corporation (YEC), which is approximately 150 km from the Project along the Alaska Highway. Currently it is believed there are 20 megawatts of surplus capacity on the YEC grid.

5.2.2 Water

Water supply adequate for drilling operations can be pumped from local creeks. Potable and non-potable water has been sourced from a shallow well at Lower Camp. In 2015, a new well was drilled at Lower Camp to provide water to the lodging facilities during exploration. It is assumed sufficient water supplies from pit dewatering and surface run-off will be available for the mill processing needs of the Project.

5.2.3 Mining Personnel

Yukon has no net government debt, no territorial sales tax, and a highly competitive taxation regime; all of which encourage investment in the mining sector. Skilled labour and equipment are available in the city of Whitehorse (population approximates 30,000) and the community of Haines Junction (area population of approximately 900). Limited services are also available in the two closest communities, Burwash Landing and Destruction Bay.

5.3 Physiography

The Project is located in the Kluane Ranges, which are a continuous chain of foothills situated along the eastern flank of the Saint Elias Mountains. The topography across the Project is typical of the interior Yukon with slopes of 250 to 300 m, and the highest peaks exceed an elevation of 1,800 m.

The main mineralized zone on the Project lies between an elevation of 1,250 m and 1,700 m on a moderate to steep south-facing slope. Water drainage on the Project is mainly east and then north into the Quill Creek drainage.

Vegetation consists of typical alpine vegetation on the hillsides, along with a mixture of pine, spruce, and poplar trees located in the lower elevations and creek beds.

6 HISTORY

6.1 Prior Ownership and Ownership Changes

W. Green, C. Aird, & C Hankins were the prospectors who discovered the surface showing near Arid Creek in 1952. The prospectors optioned the Project property to Hudson Bay Exploration and Development and subsequently it was optioned to Yukon Mining Corporation Limited (YMC), a subsidiary of Hudson Bay Mining and Smelting Co. Ltd (HudBay) that same year. Furthermore, the Project property was then transferred again, to another subsidiary of HudBay called Hudson-Yukon in 1955. In 1969, Hudson-Yukon completed a detailed feasibility study for a mining and milling operation at the Project property.

The Project was optioned to the Kluane Joint Venture between All North Resources Ltd. (All-North) and Chevron Minerals Ltd. (Chevron Minerals) in 1986 which acquired a 50% interest in the Project. In 1987, Galactic Resources Ltd. purchased Hudson-Yukon's interest and NSRs royalty on the Project property and merged with All-North. In 1989, All-North purchased Chevron Minerals interest to acquire 100% interest in the Project. Other joint ventures were formed on the Arch property, which lies west of the Wellgreen Deposit.

In 1994, Northern Platinum acquired an 80% interest in the Project from All-North, with the remaining 20% purchased in 1999. Coronation Minerals optioned the Project in 2005 but dropped the option in 2009. The Project was then returned to Northern Platinum.

Prophecy Resource Corp. purchased Northern Platinum near the end of 2010. The Project property and other nickel assets were spun out to its subsidiary Pacific Coast Nickel Corp., which then changed its name to Prophecy Platinum Corp. in 2011. Prophecy Platinum Corp. changed its name to Wellgreen Platinum Ltd. in 2013. In January 2018, Wellgreen Platinum Ltd. changed its name to Nickel Creek Platinum Corp.

6.2 Previous Exploration and Development

During the tenure of Hudson-Yukon, a total of 25,017 m of drilling was completed in 60 surface holes and 481 underground drill holes. Additionally, HudBay undertook 4,267 m of underground development including internal shafts. Ground geophysics and a soil geochemical survey were also conducted.

Between 1987 and 1988 during the Kluane JV, 16,648 m of drilling was completed in 83 surfaces and 34 underground holes with some rehabilitation of the underground workings and slashing of new drill stations. Additional exploration included geological mapping and sampling, Very Low Frequency Electromagnetics (VLF), magnetic surveys, and surface trenching.

From 1996 to 2005, Northern Platinum drilled 4,471 m of surface diamond (10 holes) and reverse circulation (57) holes.

Coronation drilled 7,248 m in 24 surfaces and 3 underground holes from 2006 to 2008. This program resulted in the discovery of the deep mineralization in the East Zone. An aeromagnetic survey of 854 linear km was also carried out.

In 2009 and 2010, Northern Platinum drilled 4,190 m in 16 core holes, prior to its acquisition by Prophecy Resources Corp. Prophecy Resources Corp. drilled one more 117 m hole.

In 2011, Prophecy Platinum Corp. (now Nickel Creek) drilled 1,925 m in 6 core holes. This drill program resulted in an updated resource and PEA.

In 2012, Prophecy Platinum Corp. (now Nickel Creek) drilled 10,983 m in 51 core holes.

In 2013, Prophecy Platinum Corp. (now Nickel Creek) drilled 104 m in one diamond hole, 831 m in one diamond-tail hole and 1,858 m in 25 reverse circulation (RC) holes, totalling 2,793 m of new drilling, along with assaying another 8,462 m of core from approximately 21,784 m of re-logged historical drill core from 108 holes.

In 2014, Wellgreen (now Nickel Creek) drilled 773 m in one diamond hole, 2,024 m in 4 diamond-tail holes and 120 m in one RC hole, totalling 2,917 m of new drilling. The resource was updated during the year to include the 2013 drilling.

In 2015, Wellgreen (now Nickel Creek) drilled 5,668 m in 21 diamond holes and 3,336 m in 27 RC holes, totalling 8,904 m of new drilling.

In 2016, Wellgreen (now Nickel Creek) drilled 1,364 m in seven diamond holes and 1,139 m in 11 RC holes, totalling 2,503 m of new drilling.

In 2017, Wellgreen (now Nickel Creek) drilled 2,720 m in 15 diamond holes for infill and metallurgical samples.

Additional information regarding a brief description of the exploration programs, to the extent known, is discussed in Section 9.

6.3 Historic Mineral Resource and Reserve Estimates

The QP (John Marek) has not completed sufficient work to classify any historical estimates as current mineral resources or mineral reserves. Therefore, Nickel Creek is not treating the historical estimates as mineral resources or mineral reserves. Any previous statements of mineral resources have been superseded by the current resource presented in this document.

6.4 Historic Production

Hudson-Yukon commenced commercial production in 1972. Mined mineralized material was trucked down from the mine to the mill site near the current lower camp beside the Alaska Highway at approximately kilometer 1727. Production ceased in 1973 due to falling metal prices and discontinuous massive sulphide horizons. A total of 171,652 tonnes grading 2.23 % Ni, 1.39 % Cu, 1.3 g/t Pt, 0.92 g/t Pd, 0.17 g/t Au, 0.40 g/t Rh, 0.42 g/t Ru, 0.25 g/t Ir, 0.20 g/t Os, and 0.20

g/t Re were milled to produce 33,853 tonnes of concentrate, which was shipped to Sumitomo in Japan.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located within the Insular Superterrane, which is dominantly composed of two older terranes (Wrangellia and Alexander) that were amalgamated at approximately 320 million years (Ma) (Figure 7-1). These terranes are comprised of island arc and ocean floor volcanic rocks overlain by thick assemblages of oceanic sedimentary rocks that range in age from 220 to 400 Ma. Wrangellia exhibits a package of platform-type limestones that are several kilometres thick conformably overlying a 230 Ma old package of volcanic rocks (the Nikolai Group) that are present on the Project.

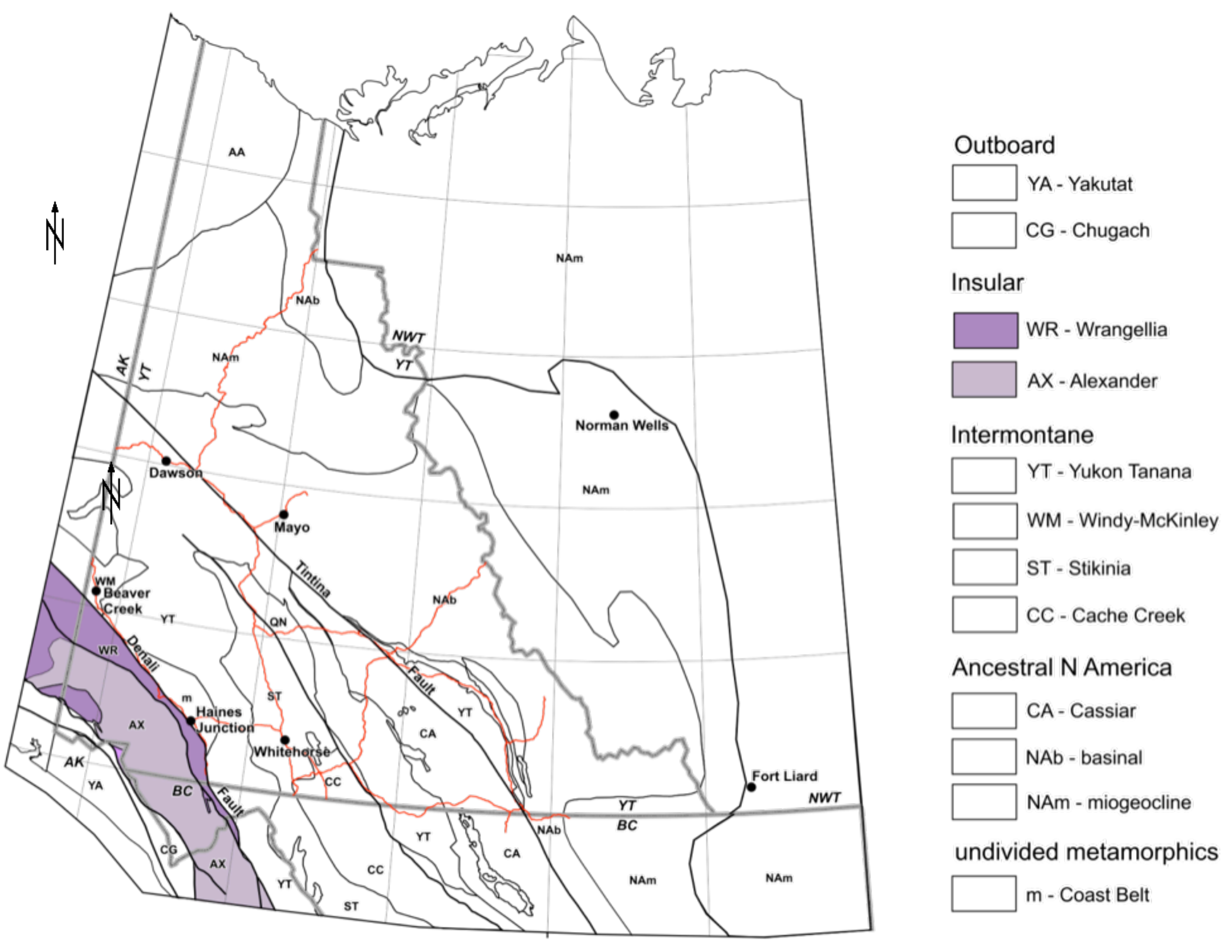
The Project is part of the Kluane Ultramafic Belt, situated in the southwest portion of the Wrangellia Terrane that spans from Vancouver Island, north through British Columbia (BC), into Alaska (Figure 7-2). The Northern Wrangellia terrane is fault bound by the dextral strike-slip Denali Fault to the northeast (Yukon-Tanana Terrane) and the Duke River Thrust Fault to the southwest (Alexander Terrane, Cobbett and others, 2010). In the southwest Yukon, Wrangellia comprises Paleozoic through to mid-Mesozoic volcanic and sedimentary rocks that are overlain by Triassic subaerial flood basalts and complementary intrusive rocks and is designated a Large Igneous Province (LIP). The ultramafic intrusives of the Wrangellia Terrane represent one of the largest tracts of nickel-copper-PGM mineralization in North America, second in size to the Proterozoic Circum-Superior Belt in Northern Quebec that rims the Archean Superior province (Hulbert, 1997).

The oldest stratified rocks that represent the base of the Yukon Wrangellia Terrane belong to the Skolai Group (Smith and MacKevett, 1970; Read and Monger, 1976). This group consists of the Pennsylvanian to Permian Station Creek and the Hasen Creek Formations. The Station Creek Formation, named for the type of section in eastern Alaska, includes Early Mississippian (354 Ma) mafic volcanic rocks overlain by volcanic breccia, tuffs, and volcanogenic sandstone. The Station Creek Formation is considered to represent back-arc oceanic crust that was overlain by arc volcanic detritus. Conformably overlying the Station Creek Formation is the Hasen Creek Formation, a sequence of conglomerate, sandstone and siltstone turbidites, and limestone. The Hasen Creek Formation is Permian in age and is likely the result of sedimentation occurring during the subsidence of the Mississippian-Pennsylvanian Arc.

The Skolai Group is unconformably overlain by the Middle and Late Triassic Nikolai Group generally consisting of basalt flows with minor intercalated limestone. The basalt is the hallmark of Wrangellia and is found throughout the terrane from Alaska to Vancouver Island (Karmutsen Formation). The Nikolai volcanic rocks are up to 3000 m thick and mainly subaerial, vesicular to amygdaloidal flows. Rare pillows occur near the base of the formation, and these volcanic rocks are overlain and occasionally intercalated with carbonate horizons of the Chitistone Limestone. The limestones are likely atoll reefs, formed as the volcanic plateau subsided. Deeper marine sedimentary rocks of the McCarthy Formation overlie the carbonate rocks.

Accompanying the eruption of the Nikolai volcanic rocks are the voluminous mafic-ultramafic intrusions. These mafic and ultramafic intrusions are common throughout the area and are generally located near the contact between the Station Creek and Hasen Creek Formations. These include gabbro, pyroxenite, peridotite and dunite of the Kluane mafic-ultramafic complex. The intrusions commonly exhibit magmatic sulphide associated nickel-copper-PGM and gold mineralization. These sills, which represent individual members of the Kluane Ultramafic Belt, along with the 232 + 1 Ma Maple Creek Gabbro (Mortensen and Hulbert, 1991) are interpreted as feeders for the Nikolai Formation flood basalts (Israel and van Zeyl 2005). The Maple Creek Gabbro occurs as a series of dikes and plugs that are observed to crosscut the sills of the Kluane Ultramafic Belt and in one case are exposed as feeders to the Nikolai Group basalt (Hulbert, 1997).

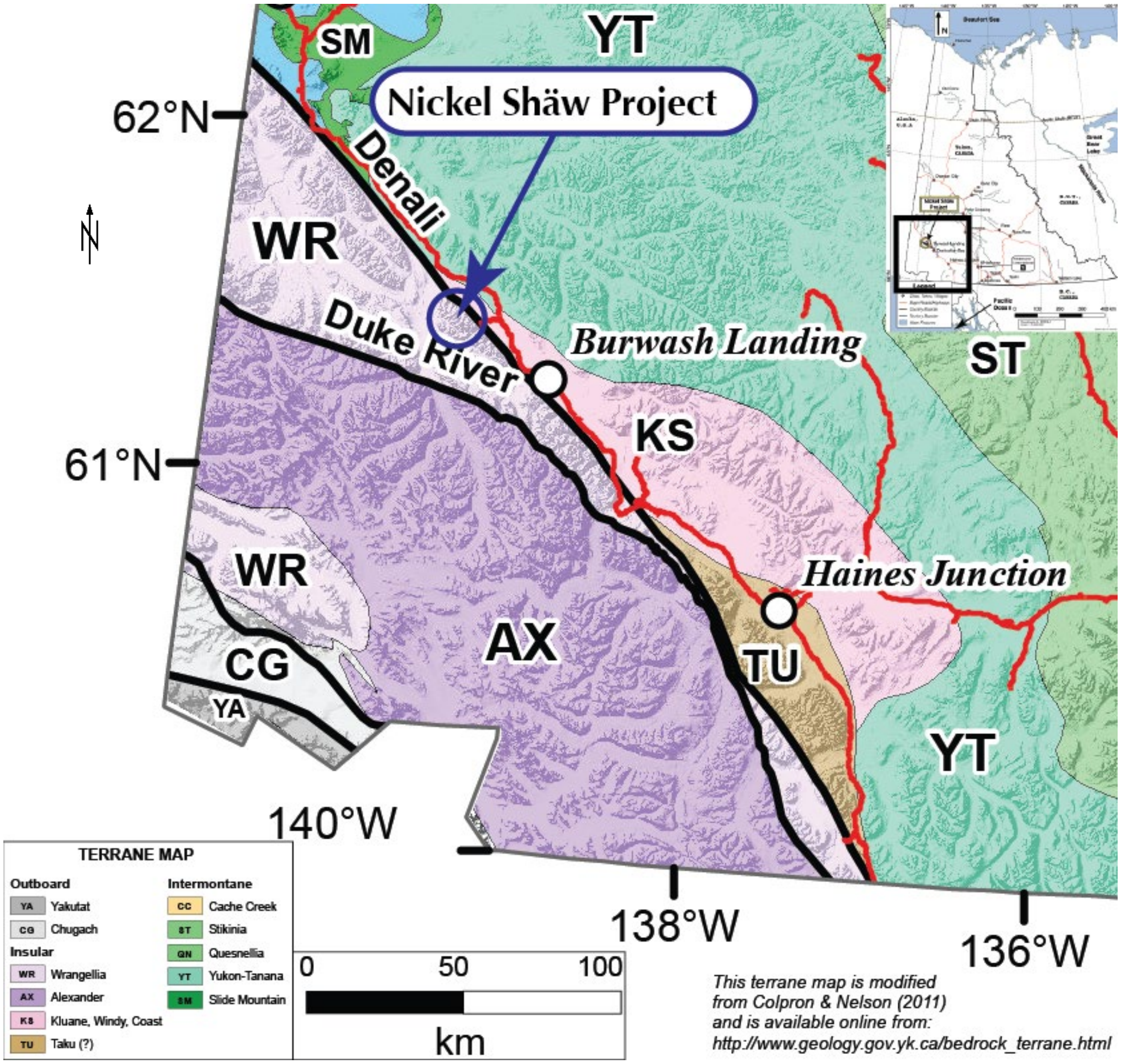
Figure 7-1: Regional Geological Setting



Source: Modified from Yukon Geological Survey, 2016

500 km

Figure 7-2: Regional Geologic Setting

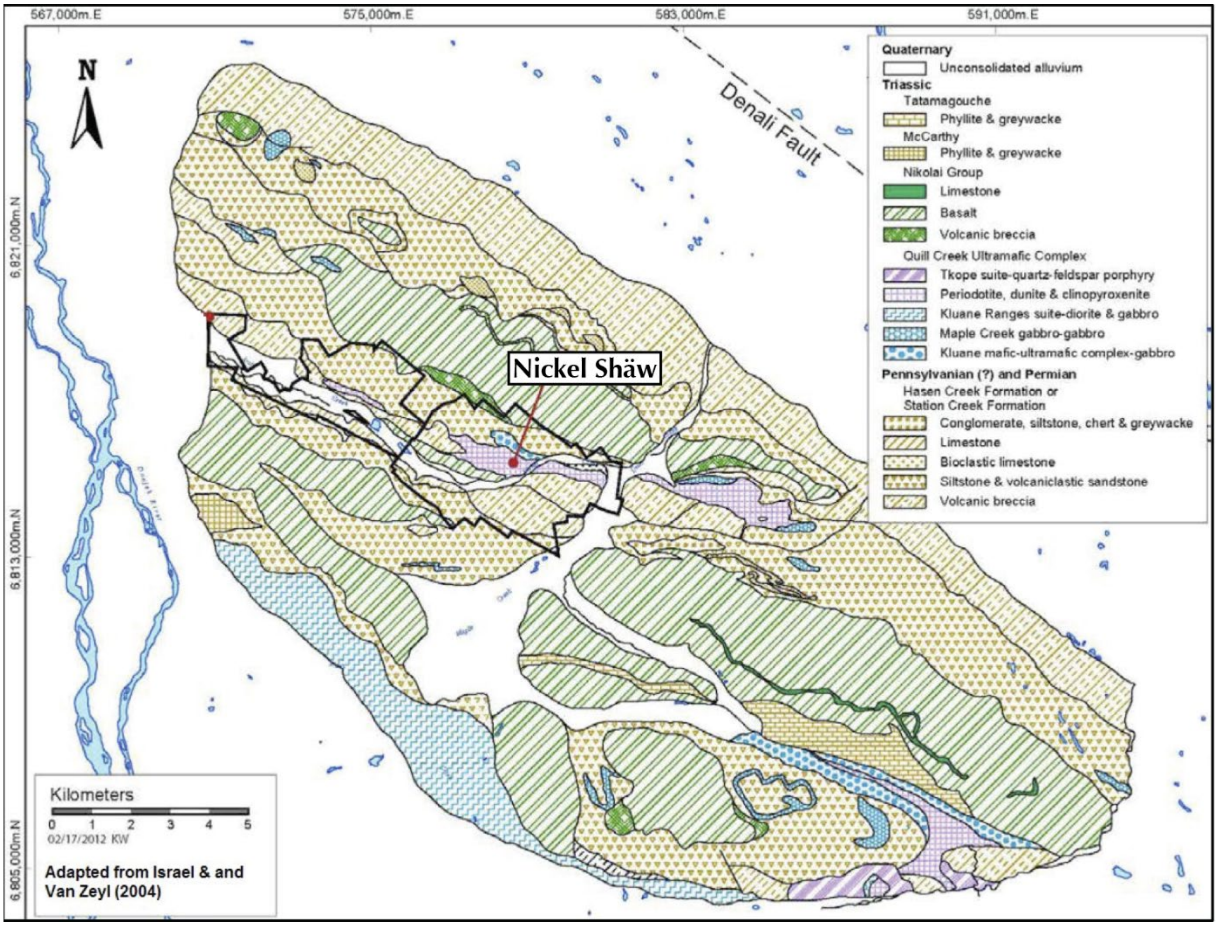


Source: Colpron & Nelson, 2016

7.2 Local Geology

Israel and van Zeyl (2004) provide the most recent, complete regional geological mapping for the Project as illustrated in Figure 7-3. Hulbert (1997) also provides a description and discussion of detailed geology and interpretation covering the Wellgreen Deposit area from maps completed by Archer, Cathro and Associates, who have compiled and reinterpreted exploration results for the Kluane JV programs carried out on behalf of All-North. However, the descriptions and classifications of the geological framework for the Project from these sources are not consistent.

Figure 7-3: Geology of the Quill Creek Area



Source: Israel & van Zeyl, 2004

The oldest rocks of the Skolai Assemblage are represented by the Pennsylvanian Station Creek Formation. The Station Creek Formation underlies significant portions of the Project and is interpreted to be roughly a few hundred m's thick. The formation is composed of basaltic to andesitic, light to medium green volcanic flows, breccias and tuffs that grade into tuffaceous sandstones, moving up section. Pyroclastic breccias and limestone are locally present in this formation but are discontinuous.

The Station Creek Formation is conformably overlain by the Permian Hasen Creek Formation and is defined as beginning where pyroclastic deposition of Station Creek is no longer apparent. The Hasen Creek Formation can be divided into two end-members (upper and lower). The lower end-member is dominantly composed of grey-black phyllite, quartzite, greywacke, cherty argillite, and siltstone. The upper member is dominated by shaley to massive limestone. Discontinuous beds of red-brown conglomerates, massive greywacke, and sandstone interbedded in the limestone horizons are also present. These rocks are folded into a series of parallel, sometimes overturned, synclines and anticlines.

The Hasen Creek Formation rocks are unconformably overlain by locally amygdaloidal flood basalt, volcanic breccias, and limestones of the Middle to Late Triassic Nikolai Group. This sequence of basalt flow contains minor interbedded limestone, and the sequence is capped by a carbonate unit. The flows are generally thin, vesicular to amygdaloidal, and locally haematitic suggesting either a shallow water or subaerial depositional environment. The Nikolai Group rocks are also folded into a series of southeast-northwest trending anticlines and synclines.

In the Wellgreen Deposit area, Nikolai Group mafic volcanics occur in the area immediately south of the Quill Creek Complex. The volcanics have been interpreted to be in fault contact with the upper part of the Quill Creek Complex and Station Creek Formation rocks (Israel and van Zeyl 2005).

There is an abundant series of relatively small intrusions into Paleozoic metasedimentary rocks and the Quill Creek Complex. They are mapped as andesitic to gabbroic dikes and plugs that are part of the Maple Creek Gabbro and are likely correlated with the Nikolai Formation. Hulbert (1997) describes these same rocks as felsic dikes, which may have been gabbro dikes that experienced post-emplacement alteration. Many of these small intrusions are associated with the northeast-southwest oriented faults that cut the stratigraphic sequence and the Quill Creek Complex, while others are parallel to the structural grain of the Station Creek and Hasen Creek Formations.

The middle to late Triassic Kluane mafic-ultramafic suit is volumetrically important in the Kluane Range. These mafic-ultramafic intrusions occurred preferentially between the Station Creek Formation and the Hasen Creek formation and appear to be sill-like in nature. This complex consists of strongly serpentinized dunite, peridotite, clinopyroxenite, and a marginal gabbro unit along the contact of the footwall rocks. This discontinuous gabbro unit occurs at the base of the sill. It is in the pyroxene rich and gabbro phases that the higher-grade, disseminated, net-textured, and massive sulphide mineralization occurs.

The Early Cretaceous intermediate and felsic intrusives belonging to the Kluane Ranges Suite represent the youngest rocks on the Project. These felsic dikes commonly cross-cut the mafic-ultramafic units and have been observed parallel to bedding in the Hasen Creek Formation.

Longitudinal faults and/or shears are common in the ultramafic rocks and some of these faults occur along lithological contacts. Hulbert (1997) describes two western faults as west-dipping reverse faults. Two faults present in the western portion of the Wellgreen Deposit intrusion offset the mafic-ultramafic rocks and dip steeply to the southeast.

7.3 Property Geology

The Wellgreen Deposit occurs within, and along, the lower margin of an Upper Triassic (Kluane) ultramafic-mafic body, within the Quill Creek Complex. This assemblage of mafic-ultramafic rocks is 20 km long and closely intrudes along the contact between the Station Creek and Hasen Creek formations. The main mass of the Quill Creek Complex, the Wellgreen Deposit, and Quill intrusions, is 4.7 km long and up to 1 km wide. A smaller mass of similar intrusive, located along strike to the northwest, is known as the Arch intrusive. The Burwash intrusion is located to the southeast and is likely a continuation of the Quill intrusion.

The Wellgreen Deposit portion of the Quill Creek Complex consists of a main intrusion and an associated group of upright to locally overturned, steeply south dipping sills. Based on drill information, the northernmost sill called the North Arm, and the main Wellgreen Deposit sill appear to be contiguous at depth in the eastern end of the deposit.

The Quill Creek Complex layered intrusion gradationally transitions from peridotite to clinopyroxenite to gabbro with a corresponding increasing sulphide and mineralization content through this sequence toward contact with the Paleozoic sedimentary country rocks (Figure 7-4). The intrusions are serpentinized and locally deformed. Locally, the sills have a lower gabbroic margin adjacent to a chilled contact with Paleozoic rocks. Recent observations indicate that many of these marginal gabbros may actually be endo-skarn units that appear to be the direct result of digestion and hybridization of limestone present in the Hasen Creek country rocks by the Wellgreen Deposit parent magma(s). Mafic-rich exo-skarns also occur in the floor rocks adjacent to the marginal gabbro, particularly where the metasedimentary host rock includes limestone or calcareous rocks. The intrusives are zoned upwards/southward away from the lower gabbroic zone through zones of clinopyroxenite and peridotite. This zonation may be directly related to the degree of interaction with the reactive wall-rocks and appears to reflect the relative sulphide content of the rocks with the highest sulphide content at the lower margins grading up to the least sulphide content in the upper parts of the tabular intrusion, mostly as peridotite.

Table 7-1 lists the regionally mapped units and how they relate to the lithologies used in the geologic model for the resource.

Table 7-1: Nickel Shäw Lithologies

Model Code	Lithologic Description	Mineralization Status	Regionally Mapped Lithology
7	Clinopyroxenite	Ore Host	Quill Creek Clinopyroxenite
20	Mineralized Gabbro	Ore Host	Quill Creek Marginal Gabbro
24	Peridotite	Ore Host	Quill Creek Peridotite
29	Massive Sulfide	Ore Host	Quill Creek Massive-Sulphide
26	Metasedimentary Rocks	Sometimes Ore Host at Contact	Hasen Creek Formation
5	Basalt	Barren	Nikolia Basalt
21	Maple Creek Gabbro	Barren	MC Gabbro
32	Volcaniclastic	Barren	Station Creek Formation

7.4 Mineralization

Mineralization on the Project occurs dominantly within the Quill Creek Complex except for a small portion at the contact within the metasedimentary host rocks. This serpentinized, ultramafic-gabbroic body intrudes the Pennsylvanian-Permian sedimentary and volcanic rocks of the Station Creek and Hasen Creek formations. The main zone of mineralization has a strike length of 1.7 km and thickness ranges from 20 m on the western end to almost 300 m at the eastern end. Drilling intercepts have indicated the mineralization ranges in depth from several metres at the west of the deposit to over 500 m at the eastern side. Discontinuous massive and semi-massive sulphide zones are significantly thinner (centimeters to a few metres), are located near the footwall contact and transition into disseminated sulfide zones above. Historic exploration and development programs defined two main zones of gabbro-hosted massive and disseminated sulphide mineralization known as the East Zone and West Zone. These zones have since been determined to be contiguous and have been further broken up and now are known as the Far East, East, West, and Far West Zones with the connecting Central Zone. Figure 7-4 presents the current zonation of the deposit. The North Arm Zone is interpreted to be a splay off of the Far East Zone. Geologic controls on mineralization are discussed in Section 14.

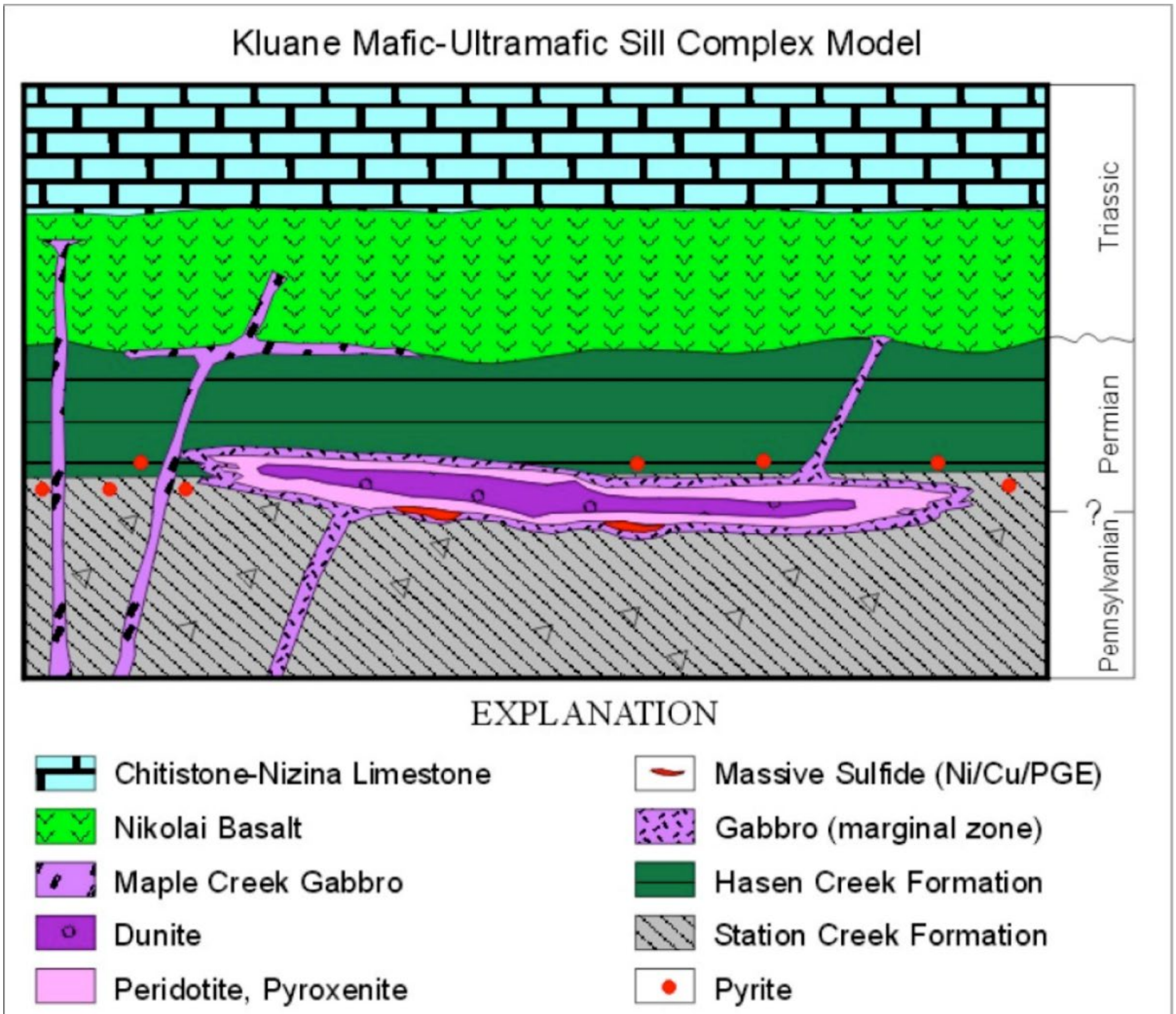
7.4.1 Far East Zone

The Far East Zone represents the easternmost part of the Wellgreen Deposit intrusion. The zone lies between 578250E and Arid Creek, at approximately 578750E (Figure 7-5). In both the current East and Far East Zones, historic exploration efforts focused on defining massive sulfide horizons and lenses near the contact between the Project Intrusion and Hasen Creek metasedimentary rocks and as such this contact is very well defined. This sedimentary contact was historically interpreted to be the steeply dipping southern footwall to mineralization based on the data available at the time, but more recent work in the East Zone showed the sedimentary contact was a wedge of metasedimentary rocks in a much larger ultramafic body. This change in understanding the nature of the sedimentary contact was demonstrated in the Far East Zone by drill holes 154, 160, and 165.

The typical steeply-dipping lithological sequence of peridotite, clinopyroxenite, and gabbro with massive sulphide is very well defined in the Far East Zone. The core of the Far East Zone shows a sulphide-rich, clinopyroxenite, and gabbro/skarn horizon with a second clinopyroxenite and gabbro enriched zone at the lower contact with the metasediments.

In the easternmost portion of the Far East Zone, all lithologies exhibit a similar sub-horizontal dip to the symmetrical sequence further west: peridotite transitioning to clinopyroxenite, and gabbro with skarn units and massive sulphide immediately prior to the basal contact with Station Creek volcaniclastics and Hasen Creek metasedimentary rocks. This lower sequence is interpreted to be contiguous with the basal sequence observed 350 m farther to the west. In addition, the foot-wedge pinches out to the east such that in the upper portion of the intrusion, the various contact-proximal lithologies are absent.

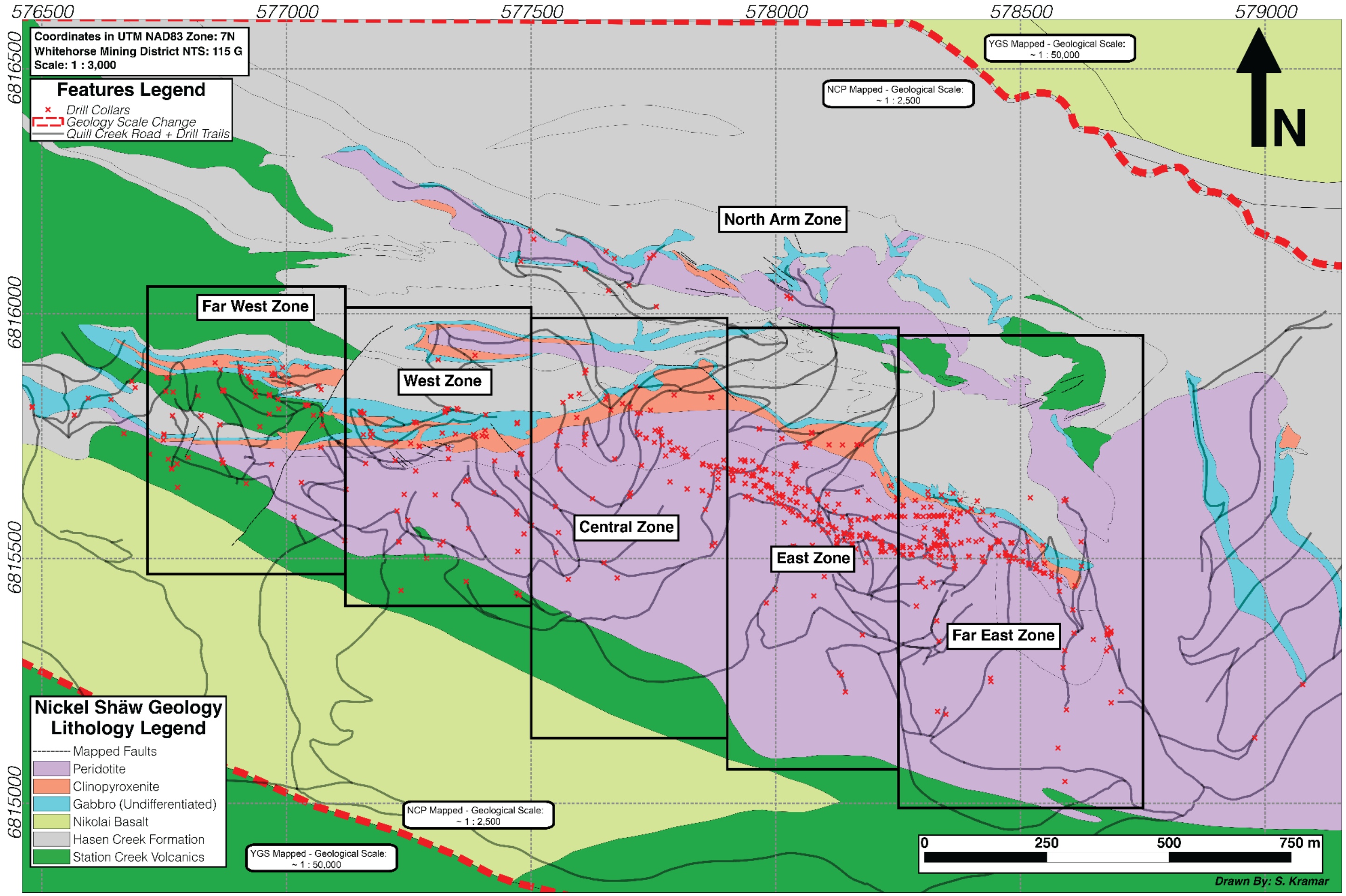
Figure 7-4: Kluane Mafic-Ultramafic Sill Complex Model



Schematic section of typical Kluane mafic-ultramafic intrusive complex model (pre-deformation) showing the outer marginal gabbroic envelope, and the zonal development of progressively more ultramafic rocks (i.e. Pyroxenite, peridotite and dunite, respectively) towards the core of the intrusion. Note the massive sulphide concentrations at the base of the complex and the intrusions preferential emplacement at or near the pyritic Station Creek-Hasen Creek Formations contact zone. (L. Hulbert 1995, GSC OF3057)

Source: L. Hulbert, 1995

Figure 7-5: Property Geology



Source: Nickel Creek, 2018

7.4.2 East Zone

The East Zone lies between 577900E and 578250E and was historically explored for massive sulphide at the footwall contact. As mentioned above, this zone was the first in which the change in the footwall contacts' orientation was observed in drill core. The peridotite-clinopyroxenite-gabbro sequence is observed to wrap around the base of the wedge in the East Zone.

The historic East Zone (current East and Far East Zones combined) was mined by Hudson-Yukon in 1972 and 1973, and approximately 171,652 t of mineralized material was extracted.

7.4.3 Central Zone

The Central Zone lies between 577500E and 577900E. The eastern portion of the zone is similar to the East Zone whereby well-mineralized peridotite gradationally transitions to clinopyroxenite and gabbro, and units are observed near the contact with dominantly Hasen Creek metasedimentary rocks. The western portion of the Central Zone exhibits a sub-horizontal, symmetrical, mineralized unit similar to that intersected at depth in the Far East Zone. Additional drilling will be required to test whether the higher-grade, sub-horizontal, mineralization intersected in the Central Zone connects with that in the East and Far East zones. This represents a high priority exploration target, and currently is the least drilled zone on the Project.

7.4.4 West Zone

The West Zone lies between 577120E and 577500E. Similar to the western portion of the Central Zone, well-mineralized peridotite overlies a comparatively thick package of clinopyroxenite and gabbro with significant semi-massive and massive sulphide zones.

7.4.5 Far West Zone

The Far West Zone lies between 576720E and 577120E, and the northern part of the zone is interpreted to be a branching sill from the main Project intrusion. This sill is generally zoned outwards, well mineralized in the centre, grading from peridotite to clinopyroxenite and gabbro towards the contact with the metasedimentary country rocks. Grades in the Far West Zone are significantly elevated starting at surface with high sulphide content.

7.4.6 North Arm Zone

The North Arm Zone is located in the east-central portion of a narrow 1,200 m long sill, positioned approximately 150 m below the main Project intrusion. It was discovered by Hudson-Yukon in the 1950s and explored in 1987 with three drill holes by All-North. All of these drill holes intersected mineralization. The geology of this zone is similar to both the East and West Zones. Mineralization consists of massive sulphide lenses, disseminated sulphide in gabbro and clinopyroxenite, and fracture fillings in footwall Hasen Creek metasedimentary rocks. The North Arm Zone was tested in 1988 and 2005 by limited drilling and was determined to have a sub-vertical dip. The information collected to-date suggests the North Arm Zone is relatively narrow in comparison with the main Project body at surface, but it does represent a prospective area of nickel-copper mineralization that warrants further work and may be contiguous with the main Project intrusion at depth towards the eastern end of the deposit.

7.5 Prospects / Exploration Targets

7.5.1 Quill Target

The Quill ultramafic body is interpreted to be contiguous (based on surface mapping and geophysical response) with both the Wellgreen Deposit to the west and Burwash to the east and is the least explored ultramafic occurrence in the Project's land package. Surface mapping in 2017 along with mapping by Archer Cathro in 1988 shows a continuous sill of peridotite. Geophysical response (magnetics, VLF & EM) suggest the same type response in magnitude for both the ultramafic body, and contact with the footwall host. Massive or semi-massive sulfide occurrences at the surface have not been observed which may explain the limited exploration. The Quill target however, has exploration potential and limited drilling had intersected ultramafic material which is encouraging (Figure 7-6).

7.5.2 Burwash Target

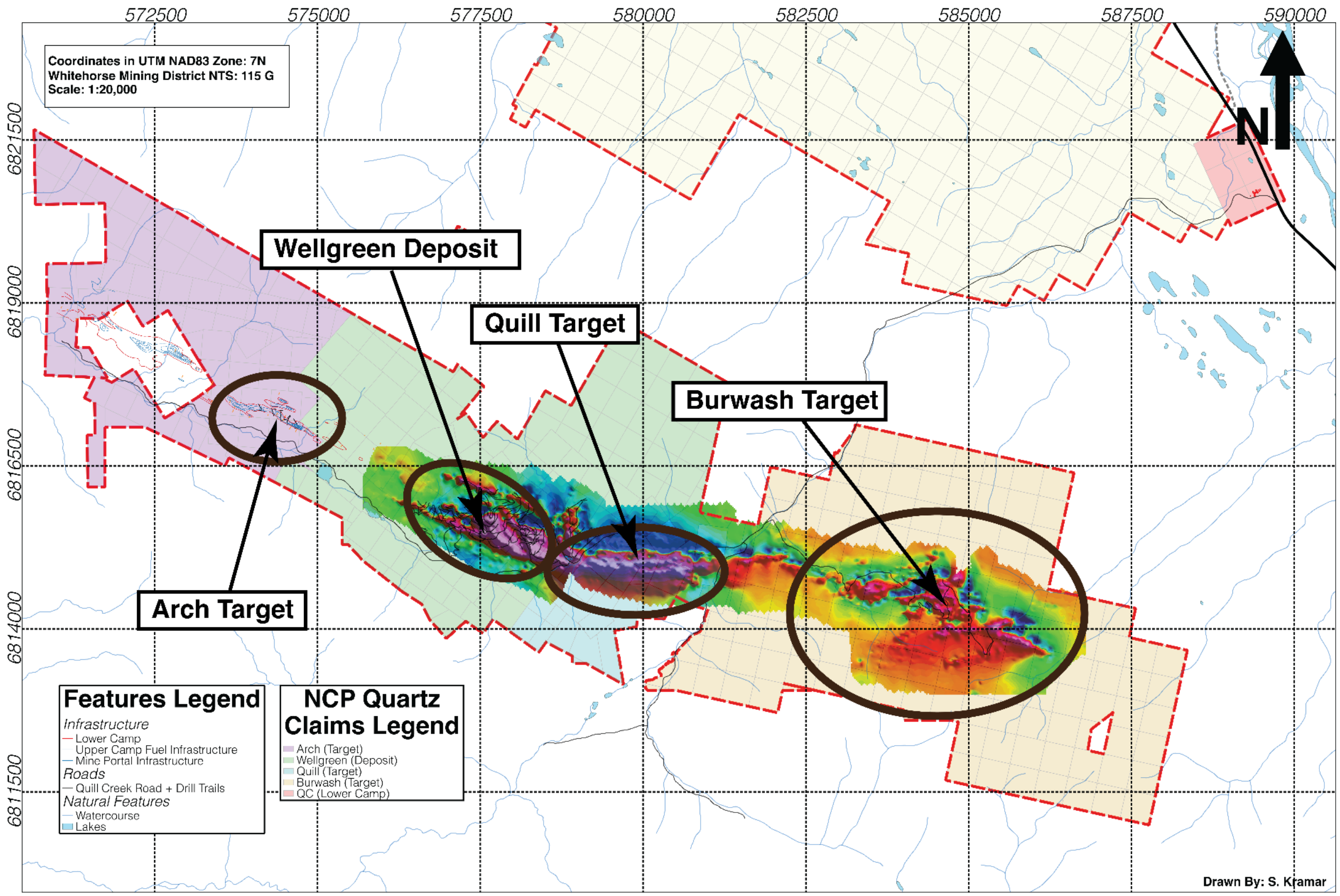
The Burwash Target area (Figure 7-6) covers part of the Project's ultramafic complex and is located east of both the Wellgreen Deposit and Quill Target areas. The northern and southeastern portions of the Burwash area have had little exploration carried out and/or reported. The northern part of Burwash only has limited soil sampling. A Mag/VLF ground geophysical survey and a small soil-sampling program were conducted in the central part of the Burwash Target since the Project was acquired by the Company, with geophysical anomalies similar to that of the Quill Target and the Wellgreen Deposit.

Historic drilling efforts at Burwash have been limited to a number of shallow holes, which appear to have targeted showings and areas of magnetic anomaly and/or soil geochemical highs. These historic campaigns were testing primarily for high-grade massive sulfide zones. Based on subsequent work by Nickel Creek, it is now recognized that mineralization may occur throughout the ultramafic bodies as disseminated mineralization, in addition to contact related mineralization; however, it is to be noted that the Burwash target is interpreted to be a 'stacked sill' system, and ultramafic material may occur in thin stacked sills rather than a large sill system like the Wellgreen Deposit. With that in mind, the Burwash target still has exploration potential (Figure 7-6).

7.5.3 Arch Target

The Arch Property was discovered as a western extension of the Wellgreen Deposit (Figure 7-6). Over the last 40 years, mapping, geochemical soil sampling, geophysics, and trenching were performed on the Arch Target. Historical drilling produced some encouraging results having intersected ultramafic material and massive sulfides in four drill holes. Exploration efforts have been minimal and sporadic, and the Company believes that Arch remains a good exploration target.

Figure 7-6: Project Regional Targets



Source: Arch, Quill & Burwash – defined by Mag/VLF; Arch – defined by Mag Contours, 2012

7.6 Minerals

Table 7-2, Table 7-3, and Table 7-4 after Cabri et al. (1993) list the opaque minerals and PGM-bearing minerals found in the deposit. The elevated presence of rhodium, iridium, osmium, rhenium, and ruthenium within the mineral suite provide an opportunity for additional potential economic contributions from these metals. Rhodium is present on the Project in highly anomalous concentrations as compared to the concentrations found in Noril'sk ores in Russia and other significant ultramafic systems globally (Hulbert 1997).

Table 7-2: Opaque Minerals Observed in the Wellgreen Deposit

Major Minerals*	
Pyrrhotite	$Fe_{1-x}S$
Pentlandite	$(Fe, Ni)_9S_8$
Chalcopyrite	$CuFeS_2$
Magnetite	Fe_3O_4
Ilmenite	$FeTiO_3$
Less Common to Rare Minerals *	
Violarite	$FeNi2S_4$
Sphalerite	(Zn,Fe)S
Chromite	$FeCr_2O_4$
Cobaltite**	CoAsS/NiAsS
Arsenopyrite	FeAsS
Ullmannite	NiSbS
Siegenite Argentopentlandite	$(Ni, Ag)(Fe, Ni)_8S_8$
Gold/Electrum	(Au/Ag)
Melonite	$NiTe_2$
Bismuth Tellurides	Bi-Te (?)
Galena	PbS
Altaite	PbTe
Kickline	NiAs
Covellite	CuS
Breuithauptite	NiSb
Barite	$BaSO_4$
Titanite Hessite	$CaTiSiO_2Ag_2Te$
Matildite	$AgBiS_2$
Undefined	Cu-Fe-Ba-S**

Source: Cabri et al., 1993
Notes: *Ideal Formula
Unidentified mineral of the cobalt-gersdorffite series

Table 7-3: Primary PGM-Bearing Minerals

Mineral	Formula
Sperrylite	$PtAs_2$
Sudburyite	PdSb
Testibiopalladite	PdSbTe
Merenskyite	$PdTe_2$
Moncheite	$PtTe_2$
Michernerite	PdBiTe
Stibiojaiadinite	Pd_5Sb_2
Mertielte II	Pd_8Sb_3
Geversite	$PtSb_2$
Hollingworthite	RhAsS
Froodite	$PdBi_2$
Unidentified	$(Pd,Ni)_2(Te,Sb)_3$
Unidentified	$(Pd,Ni)_3(Te,Sb)_4$
Unidentified	Pd(Bi,Te)
Unidentified	$Pd_3Ni(Sb,Te,Bi)_5$
Laurite	RuS_2
Kotuiskite	$PdTe_2$
Pt-Fe alloy(s)	Pt_3Fe or PtFe(?)
Unidentified	Re>Ir>Os>Ru alloy
Unidentified	Pd-Hg
Iridium	Ir
Unidentified	Re sulphide (?)

Source: Cabri et al., 1993

Table 7-4: Additional PGM-Bearing Minerals

Mineral	Formula	Metal Content
Melonite	$(Ni,Pd,Pt)Te_2$	Up to 15.1%Pd; up to 9.37% Pt
Unidentified	$(Ni,Pd)_2(Te,Sb)_3$	Up to 22.8% Pd
Unidentified	$(Ni,Pd)_3(Te,Sb)_4$	Up to 15.9% Pd
Breuithauptite	(Ni,Pd)Sb	Up to 18.9% Pd
Hextestibio-panickelite	$(Ni,Pd)_2SbTe$	Up to 15.9% Pd
Ullmannite	(Ni,Pd)SbS	Up to 0.09% Pd
Cobaltite	(Co,Rh)AsS	Up to 2.7% Rh, in zones
Pentlandite	(Pt,Rh,Ru)*	Up to 34 Pd, 12 Rh, 13 Ru (ppm)
Chalcopyrite	(Ru,Rh,Pd)*	Up to 10 Ru, 10 Rh, 9 Pd (ppm)
Pyrrhotite	(Pd)*	Up to 5.6 Pd (ppm)

Source: Cabri et al., 1993

Note: *Trace levels as determined by proton microprobe

8 DEPOSIT TYPES

The Wellgreen Deposit is hosted in the Quill Creek Complex, one of a number of mafic-ultramafic sills that are enriched in nickel-copper-PGM mineralization that outcrop within the Kluane Ultramafic Belt of the Wrangellia Terrane in southwestern Yukon. The sills which form the Kluane mafic-ultramafic complex are thought to be part of a sub-volcanic system that feed the Nikolai Formation flood basalts and have been compared to the Noril'sk Deposit in Russia.

Similar deposits also occur elsewhere in Canada (Franklin sills; Bedard et al., 2011; Cape Smith Belt; Giovenazzo et al., 1989), in China (Yangluiping Instrusions; Xie-Yan Song et al. 2003, Jinchuan; Tonnelier, 2010), and southern Africa (Uitkomst intrusion; Maier et al., 2013, floor of eastern Bushveld Complex; Maier et al., 2001).

Many sill-hosted Ni-Cu-PGM deposits are generally considered to be part of a large, interconnected magmatic system that feed voluminous flood basalts and result from the impingement of a mantle plume upon the base of the crust. At Noril'sk, the main sulfide bodies formed from segregated sulfide at the base of magmatic conduits through which multiple pulses of magma travelled, and this mechanism is also believed to have occurred at the Project. Settling of immiscible sulfides, sulfur scavenging from adjacent country rock and silica contamination resulting in sulfur saturation, are thought to be important mechanisms for metal enrichment at the Project. Based on this genetic model, exploration programs have targeted the contact between the Hasen Creek Formation and the mafic-ultramafic host rocks. The Quill Creek complex intruded a Pennsylvanian-Permian island arc, whereas many of the other deposits are Precambrian and all intruded into cratons. Greene *et al*. (2010) offers compelling evidence that the mafic-ultramafic intrusions and flood basalts of Wrangellia were formed in an oceanic plateau, which itself was formed by a mantle plume (Richards, 1991) and the terrane was subsequently accreted to the margin of North America in the Jurassic. These circumstances make the Project unique among other sill-hosted Ni-Cu-PGM deposits.

9 EXPLORATION

Historic exploration carried out by previous operators is summarized in Section 6. Exploration relevant to the mineral resource update is presented below.

9.1 Exploration Potential

The Project extends over an 18-km mineralized trend with multiple exploration targets. Identified major zones of Kluane ultramafic from mapping, soil sampling and geophysics include (from east to west) Formula Target, Arch Target, Wellgreen Deposit, Quill Target, and Burwash Target. Figure 9-1 presents physical locations of targets described above (Formula is further west, off the map).

9.2 Grids and Surveys

In 2013, the Company conducted a collar monument and surveying program. This effort was undertaken to modernize the Project's drill database by changing the coordinate system for all data from local mine grid to Universal Transverse Mercator (UTM), North American Datum 1983 (NAD83), Zone 7N. Many drill holes on the Project were never surveyed or designated with monuments, and those that were surveyed used the mine grid coordinate system. A differential global position system (DGPS) was used to survey 58 holes. Most collar positions were changed by a few metres however, some collars were more than 30 m away from their supposed locations.

For road and trail surveys, the Trimble GPS unit was carried on the operator's back while they were driving an all-terrain vehicle (ATV). The instrument took a measurement every few seconds. For drill collar surveys, the Trimble was activated directly over the collar and its position was measured every few seconds for one minute. The average of the measurements was then corrected using the base station located in Juneau, Alaska for post processing purposes.

9.3 Geological Mapping

In 2013, a three-day mapping program was undertaken on the eastern portion of the Project, east of Arid Creek and northeast of the upper camp. Parts of this area were exposed by undocumented bulldozer trenching. This mapping effort led to a better understanding of the contacts between the Project intrusion, the Maple Creek Gabbro, and the Hasen Creek sedimentary rocks. In 2015 and 2017 a mapping program was completed over the Wellgreen Deposit, conducted by Dr. Craig Bow in order to better understand the site geology. The results of this mapping have been utilized in the current site geologic model and to identify potential exploration targets. In 2018, geologic mapping was completed to extend Dr. Bow's work beyond the deposit.

9.4 Geochemical Sampling

In 2012, a soil sampling survey was undertaken over the Wellgreen Deposit, Quill, Burwash and Arch targets.

Soil samples were taken on a 25-m nominal spacing, along virtual gridlines across the Project. Soil augers and mattocks were the primary tools used to access the 'B' and/or 'C' soil horizon profiles. Samples were placed in Kraft sample bags and shipped to the ALS Global preparation facility in Whitehorse, YT. Sample pulps were then sent to the ALS Global lab in Vancouver, BC for geochemical assay. The soil sampling procedure and analytic method are appropriate for exploration geochemistry and potential target refinement however, were not used for determination of mineral resources since they may not be representative of the mineralization.

9.5 Geophysics

In 2007, Geotech Ltd. conducted an air-borne versatile time domain electromagnetic (VTEM) and magnetic (MAG) survey over the Project, notably the deposit and several new (at the time) areas of interest, hoping to gain information below overburden.

A helicopter-borne magnetic (MAG) and electromagnetic (DIGHEM) survey was flown in 2008 by Fugro over the Project area, hoping to detect zones of conductive mineralization and provide information that could be used during the course of surficial geological mapping. The magnetic response was effective at mapping the host mafic-ultramafic units and the survey returned several anomalous features that could be considered moderate to high priority exploration targets.

In 2009, ground-based geophysics were utilized including a horizontal loop electromagnetic (HLEM) survey and bore hole-Induced Polarization/DC Resistivity (IP) survey conducted by Aurora Geosciences Ltd. as a test, to determine the effectiveness of these geophysical methods. The HLEM survey was ineffective at producing the required depths of penetration along the proposed target lines however, the down-hole IP survey did succeed in showing anomalies over already known mineralization.

In 2012, three ground based geophysical surveys were utilized: a combination total field magnetics (MAG) and Very Low Frequency (VLF) to provide higher resolution data (compared to an air based geophysical method) over the deposit, Quill, Burwash, and parts of Arch, and an Extremely Low Frequency (ELF) survey as a test of the methods effectiveness. Aurora Geosciences Ltd. conducted all the geophysics totalling 148.93 line-kilometres with 62.74 line-kilometres specifically over the deposit/Quill area. The ELF survey was but a few short test lines conducted near the west end of the deposit. Figure 9-1 below presents the area of coverage (gridded) from the MAG/VLF surveys. Magnetic highs correlate well with mapped areas of mafic-ultramafic lithologies and this information has been used to improve the geologic model and to identify exploration targets.

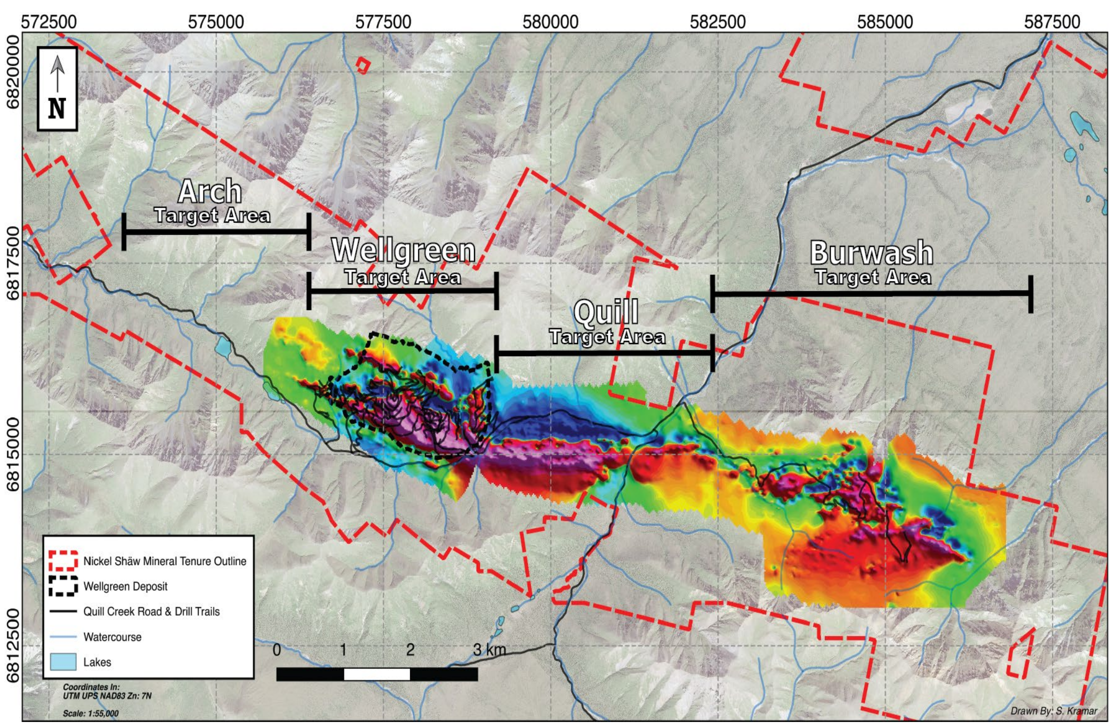
In 2015, Discovery International Geophysics Inc. carried out a borehole electromagnetic (BHEM) and surface Transient Electromagnetic (TEM) survey at the Project. Four boreholes were surveyed and approximately 2 km was surveyed with the TEM sensor in an effort to target higher grade mineralization based on down hole response.

Another Borehole Electromagnetic (BHEM) survey was conducted by Aurora Geosciences at the Project in 2016 to assist in locating massive sulphide ore in the near proximity of select diamond drill holes. Five boreholes were tested, and those holes were selected for proximity to areas of interest.

In 2018, an IP/DC resistivity survey (IP) was done at the Project by Aurora Geosciences Ltd. The survey was designed to test geophysical response on areas with geological control over the known deposit and apply findings to areas of interest for exploration. The survey consisted of nine lines and totaled 11.6 km. The IP survey appears to determine the location of mafic-ultramafic rocks below the surface and has identified target areas that may warrant additional work. At the back of the IP survey, one partial IP line, and subsequent virtual lines adjacent was selected for another test of the ELF geophysical system. Three lines were surveyed for a total of three line-km, also by Aurora Geosciences Ltd. To further bolster the data set collected by the IP survey, drill core samples were sent to Aurora Geoscience's Whitehorse laboratory, to test geophysical (and physical) properties and eventually compare 'micro' results (i.e.: drill core) with 'macro' results (i.e.: IP stations at the deposit).

These geophysical programs, where appropriate, have been used to develop the geological model and identify potential exploration targets.

Figure 9-1: Magnetic-VLF Survey Extent



Source: Nickel Creek, 2018

9.6 Petrology, Mineralogy, and Research Studies

There have been several petrological, mineralogical, and geological studies done at the Project. A Ph.D. thesis was done by S. W. Campbell in 1981, a M.Sc. thesis by S. Miller in 1991, and a B.Sc. thesis by M. Fayak in 1989. Earlier petrologic, mineralogic, and isotopic studies are provided in Hulbert's 1997 Geological Survey of Canada Bulletin (506) of the Ni-Cu-PGE deposits in the terrane. Israel and van Zeyl (2005) completed a preliminary regional geology report on the area. Vancouver Petrographics (VanPetro) completed petrography on the Project samples in 2010, and subsequent petrographic studies were again undertaken by the Company in 2015 using VanPetro as the scientific investigator.

9.7 Geotechnical and Hydrological Studies

SRK Consulting conducted a site visit and logged one core hole in 2015 in order to make recommendations for future geotechnical work. In 2017, nine holes that were drilled for infill/metallurgical purposes were logged in detail for geotechnics and were selectively sampled for laboratory strength testing under the direction of AGP. AGP conducted a site visit in 2017 and provided a Preliminary Mining Geotechnical Assessment for pit slope design. This work was followed up in 2018 by surface geotechnical mapping by AGP's geotechnical engineer and NCP geologists. Further geotechnical work will be required to determine pit slopes and pit phase sequencing.

Since 2013, 27 monitor wells have been drilled within and adjacent to the deposit in order to conduct baseline water quality studies. This work will be used to characterize the background groundwater conditions around the site. Additional groundwater studies will be required to further advance the Project.

9.8 Metallurgical Studies

Metallurgical testwork is discussed in detail in Section 13 of this document.

9.9 Priority Exploration Targets

Exploration potential exists throughout the Kluane Ultramafic Belt, as these rocks are known to be elevated in Ni, Cu, and PGM's. Current understanding indicates higher grade mineralization occurs where magmatic melts have interacted with quartz and carbonate bearing country rock. Potential areas where such interaction is thought to occur are targets for exploration that include the Arch, Quill, and Burwash areas.

The Arch Target area is located 2 km west and on strike with the Wellgreen Deposit. The target was discovered in 1952 and has been explored by surface sampling, geologic mapping, geophysics, trenching and minor drilling. Three holes have been drilled on the prospect and occurred in 1955, 1988, and 2001. The drilling done in 2001 encountered massive sulfides that contained 1 m of 4.18% Cu and 1.98% Ni.

The Quill Target is located east and along strike with the Wellgreen Deposit and may even be contiguous. A magnetic high (warm colours) trend from the Wellgreen Deposit to Quill, as shown in Figure 9-1 suggests the mafic-ultramafic sill is continuous between these two areas, and the inferred ultramafic target (Quill) is defined by magnetic high. Subsequent surface mapping done in 2017 further supports this conclusion. The magnetic low trend (cool colours) north of the Quill mass, suggests a counterpart rock assemblage to the magnetic low north of the Wellgreen Deposit. If this is the case, then similar metasomatic processes may have acted on the Quill target as the Wellgreen Deposit, making Quill a priority exploration target for follow-up.

The Quill Target area has a strike length of 2 km and surface soil samples across the area are anomalous in Ni and Cu. Peridotite and favorable country rock have been mapped within the target area. Three holes were drilled in the target area during the 1950's when HudBay was exploring for higher-grade massive, and/or semi-massive sulfides. The lack of additional drilling indicates they did not encounter the material deemed "higher-grade" in these holes, and subsequently ceased drill activities. However, these holes did encounter ultramafic material (dominantly peridotite), but with lack of massive or semi-massive sulfides, the determination was not to have the core sampled by assay.

The Burwash Target is east of Quill and is along strike with the Wellgreen Deposit. The target area begins about 4 km east of the Wellgreen Deposit and has a strike length of almost 3 km. The magnetic high observed at the Wellgreen Deposit and Quill, broadens and becomes subtler at Burwash. The Ni and Cu geochemical soil sampling results at Burwash are similar in magnitude to the Wellgreen Deposit, indicating a favourable area of exploration interest.

Drilling at Burwash and geologic mapping indicate the ultramafics are a series of bifurcating thin sills and are not as wide as the main sill at the Wellgreen Deposit, and sills are "layer cake" separated by country rock, and is suggested to be a smaller, "stacked sill" system. Drilling was conducted at Burwash in two drill campaigns, one in 2005 and one in 2008. As an example of the exploration potential, one hole drilled in 2008 (BR-08-05) encountered 67.8 m of 0.363 ppm Pt+Pd+Au, 0.22% Ni and 0.07% Cu.

10 DRILLING

Several companies have completed drilling over an extended period of time on the Project property, and Table 10-1 summarizes the drilling history. Section 12 establishes that drilling completed before 1987 was not reliable and consequently not used in the estimation of the mineral resource. Drilling completed during 2017 was also not utilized in the update of mineral resources. The drill count at the bottom of the table illustrates the 1987 through 2016 drilling that was used for the determination of mineral resources. The QP, John Marek of IMC, holds the opinion that the 1987 through 2016 drilling can be used for the determination of mineral resources.

The 2017 drill program generally did not reach the planned target depths in many of the drill holes due to some factors. John Marek completed several tests reported in Section 14 that show the 2017 drilling would not make a material change to the mineral resource and was consequently not incorporated into the model.

The following information regarding historic drilling has been summarized from the Technical Report “Mineral Resource Estimate of the Wellgreen Ni-Cu-PGM Project” June 26, 2017.

10.1 Historic Drilling

Considerable surface and underground drilling were completed in the 1950s by Hudson-Yukon, an operating subsidiary of HudBay. Additional drilling was completed under the auspices of the Kluane JV (All-North, Chevron and Galactic Resources) in the 1980s by Archer, Cathro & Associates Ltd. Drill logs and assay summaries and certificates for many of these historic drill holes are available and have been compiled into a database along with more recent drill data. This historic work has not been thoroughly documented, however, much of the data has been located and digitized. Drilling prior to 1987 has not been used for the resource estimation other than to guide the construction of the geologic model. The pre-1987 drilling was removed because 1) long intervals of the holes were not assayed, 2) the criteria to assay or not assay does not appear consistent, and 3) high-grade intervals seem to be highly biased relative to drilling after 1987.

10.2 Northern Platinum Drilling

Northern Platinum conducted numerous drill campaigns on the Project property between 1996 and 2010. The drilling conducted by Northern Platinum in 2009 and 2010 was designed to extend and expand the potential resource of the Wellgreen Deposit by targeting mineralization up-dip of the East Zone and east along strike. Drilling was completed by E. Caron Diamond Drilling Ltd. of Whitehorse. All holes drilled in 2009 and 2010 were HQ diameter (63.5 mm core size), and all drilling was run in five-foot intervals (1.52 m). Ten holes were drilled in the East Zone in 2009, totalling 2051.75 m. In 2010, prior to its acquisition by Prophecy Resources Corp., Northern Platinum drilled six holes in the East Zone. After the acquisition, one more hole was drilled, bringing the 2010 total to 2,254.77 m.

Table 10-1: Project Drill Hole Summary

Surface Drilling						Underground Drilling		Total	
Year	Company	Diamond		RC or Partial RC		Diamond Drilling		Reported Drilling	
		Holes	Metres	Holes	Metres	Holes	Metres	Holes	Metres
1952	Yukon Mining	18	1,982					18	1,982
1953	Yukon Mining	27	2,500			27	693	54	3,192
1954	Yukon Mining	2	193			159	3,940	161	4,132
1955	Hudson Yukon Mining					154	9,019	154	9,019
1956	Hudson Yukon Mining					38	1,904	38	1,904
1969	Hudson Yukon Mining	13	1,314					13	1,314
1971	Hudson Yukon Mining					81	2,522	81	2,522
1972	Hudson Yukon Mining					23	990	23	990
1987	All North / Galactic Resources	46	5,027					46	5,027
1988	All North / Chevron	37	6,050			34	5,571	71	11,621
1996	Northern Platinum			57	3,874			57	3,874
2001	Northern Platinum	6	530					6	530
2005	Northern Platinum	4	67					4	67
2006	Coronation Minerals	11	2,017					11	2,017
2007	Coronation Minerals					3	577	3	577
2008	Coronation Minerals	13	4,655					13	4,655
2009	Northern Platinum	10	2,052					10	2,052
2010	Northern Platinum	7	2,255					7	2,255
2011	Wellgreen Platinum	6	1,925					6	1,925
2012	Wellgreen Platinum	22	5,566			29	5,417	51	10,983
2013	Wellgreen Platinum	1	104	26	2,689			27	2,793
Drilling Added Since 2015 Resource Statement									
2014	Wellgreen Platinum	1	773	7	2,144			8	2,917
2015	Wellgreen Platinum	21	5,668	27	3,336			48	9,005
2016	Wellgreen Platinum	7	1,364	11	1,139			18	2,503
2017	Nickel Creek Platinum	15	2,720					15	2,720
Total Project Drilling to Date		267	46,761	128	13,182	548	30,633	943	90,575
The Mineral Resource is Based on									
Drilling from 1987 through 2016		192	38,053	128	13,182	66	11,565	386	62,799

10.3 1996 Drill Program

In 1996, Northern Platinum conducted a reverse circulation (RC) program that focused on the historic East and West Zones. Drilling was completed by Northern Platinum staff using an Ingersoll Rand ECM-350 3.5" diameter RC drill. A total of 57 holes totalling 3,873.7 m were drilled, and drilling was run on five-foot intervals (1.52 m).

10.4 2001 Drill Program

The 2001 program targeted mineralization along the historic footwall contact and drill-tested the Middle Arm, a splay off the main property intrusion in the West Zone. Drilling was conducted by E. Caron Diamond Drilling Ltd. of Whitehorse. A total of six drill holes were completed on the Project property and one hole on the adjacent Arch property, for a total of 591.92 m. All 2001 drilling was HQ diameter, sampled on 5 ft intervals (1.52 m).

10.5 2005 Drill Program

A small program was conducted in 2005. This program focused on the North Arm and the drilling was completed by Northern Platinum staff using an Ingersoll Rand (ECM)-350 3.5" diameter RC drill. A total of four holes were completed totalling 67.05 m. Sampling was on 5 ft intervals (1.52 m).

10.6 2006-2008 Coronation Minerals Drill Program

The holes drilled on the Project property by Coronation Minerals in 2006 were to validate the historical drilling done by the Kluane JV in 1987 and 1988. The program was designed by Watts, Griffis, and McQuat, Ltd. (WGM) with a total of 24 holes proposed. Coronation Minerals engaged E. Caron Diamond Drilling Ltd. of Whitehorse, Yukon as the drilling contractor. All the surface drilling was HQ, and holes were reduced to NQ (47.6 mm core diameter), as the depth increased and ground conditions became unfavourable. The underground drilling was all BTW (42 mm) core size. The drilling began in late July 2006, and a total of 11 holes were completed for 2,016.87 m. Ten of the holes drilled in 2006 were drilled to "twin" historical holes drilled by the Kluane JV. In 2007, three underground holes were completed totalling 576.99 m. Two of the holes were designed to "twin" historical holes. In 2008, 13 additional surface diamond drill holes were drilled by Coronation Minerals.

10.7 2011 Wellgreen Drill Program

The drilling conducted by Wellgreen in 2011 was designed initially to delineate the resource potential of the deposit by targeting the area between the East and West Zones to prove the zones are not separate, but rather one continuous zone. The focus of the program evolved to test the hanging wall disseminated sulphides located in the ultramafic unit. Drilling was completed by

E. Caron Diamond Drilling Ltd. of Whitehorse. A total of nine drill holes were completed during the 2011 drill program from June to October. However, three collar locations were never recorded and are considered lost. All holes were drilled HQ, and all drilling was run in 5 ft intervals (1.52 m). Including the lost holes, a total of 2269.17 m was drilled in 2011.

10.8 2012 Wellgreen Drill Program

The surface drilling conducted by Wellgreen in 2012 was designed to infill the potential resource of the Wellgreen Deposit in the East and West Zones. The underground program focused on upgrading the resource category of the high-grade hanging-wall gabbro in the East Zone. Surface drilling was completed by Foraco International SA of Toronto, ON, while DMAC Drilling of Aldergrove, BC completed underground drilling. A total of 22 drill holes from surface and an additional 29 drill holes from underground were completed during the 2012 drill program from February to November, totalling 10,983.11 m. All the holes were HQ diameter.

10.9 2013 Wellgreen Drill Program

The drilling conducted by Wellgreen in 2013 was designed to extend, expand, and upgrade the resource of the Wellgreen Deposit. The program initially focused on defining and expanding the Far East Zone and a second program drilled in-fill holes in the resource with dual purpose geologic definition and groundwater monitoring wells in the Project and areas of potential future mine infrastructure. The first drill program was completed by Boart Longyear of South Jordan, Utah, USA. A total of nine drill holes were completed during the 2013 drill program from July to October, totalling 2,027 m. Eight of the nine holes were drilled with 5.5” RC, one of which was continued in HQ and later downsized to NQ, and one other hole was drilled HQ. All drilling was run in 3 m intervals. Midnight Sun Drilling of Whitehorse completed the second program. A total of 18 vertical holes were completed during the program from October to November, totalling 765.93 m. All of those holes were drilled with 4.5” RC and were run in 5 ft intervals (1.52 m).

10.10 2014 Wellgreen Drill Program

During October and November of 2014, the Company completed 2,916.49 m of drilling in eight holes. Most holes were started with an RC rig and finished with HQ core.

10.11 2015 and 2016 Wellgreen Drill Program

The Company completed a mix of core and RC drilling during 2015 and 2016. Starting in 2015, HQ drill core was sawn in half, and half of the core was sawn again to generate ¼ core samples that were used for assay. This procedure was intended to ensure that samples for metallurgical testing were available (½ core) but still retain ¼ of the core for verification.

10.12 2017 Wellgreen Drill Program

Fifteen diamond core holes were drilled during the last half of 2017 at the Project property. Four of the holes were vertical, and the remaining 11 were angle holes collared on the south side of the main mineralization, and oriented to plunge downward to the north. The plan for these holes was to infill zones within the deposit.

The challenges of a late fall/early winter drill campaign and collar locations that were not ideal due to surface permit constraints limited the effectiveness of the 2017 program in achieving the stated goals.

IMC tested the impacts of the late 2017 drilling on the mid-2017 resource model and found that the new holes made less than a 1% impact on the contained metal of nickel and copper within the mineral resource. As a result, the resource model was not updated with the 2017 drill holes. The tests on the 2017 drilling impact are reported in Section 14.

10.13 Re-Sampling of Historic Drill Core

The Company sampled and assayed previously non-sampled core intervals and re-assayed all available sampled intervals from the 1987-1988 programs in 2013. A total of 3,087 samples were analyzed from 108 holes (8,462 m). The existing half core was sawn so that ¼ core samples were sent for assay and the Company's preparation and assay protocols were applied.

Missing intervals within the 2006 through 2007 programs were also resampled with ¼ core and assayed using the Company's preparation and assay protocols.

10.14 Collar Survey Procedures

Before the 2013 field season, drill collars were spotted with a compass and chain off the local mine grid, with the last completed collars surveyed with a hand-held GPS, compass and chain or a total station GPS, or not at all. In 2013 all collars were spotted using a hand-held GPS and later surveyed with a differential GPS.

All early drilling in mine grid has been converted to the UTM Zone 7N coordinates.

10.15 Downhole Survey Procedures

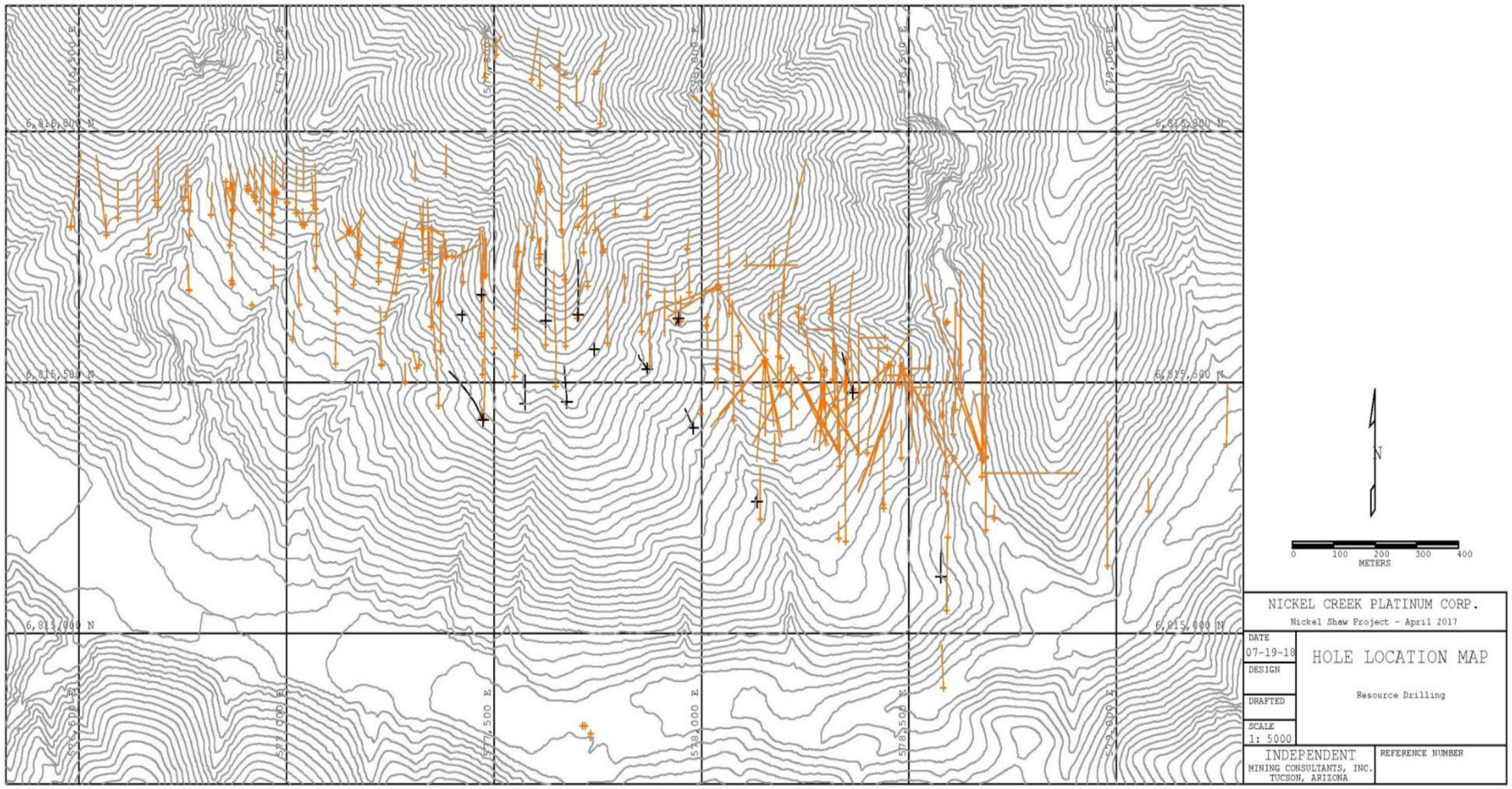
Down-hole surveys were performed differently in different years depending on the operator at the time. HudBay, Archer-Cathro, and Northern Platinum (from 1996-2005) used acid dip tests to determine hole deviation, either at regular intervals or, in the case of Northern Platinum, at the end of each hole. Coronation Minerals used acid dip tests in 2006 and 2007 and used a Reflex Single Shot magnetic tool in 2008. Northern Platinum (from 2009-2010) and Prophecy Resources Corp. (2011) reported the use of a ReflexIt© tool, and survey readings were collected approximately 9 m off the bottom of the hole and at about 152 m intervals up the hole; however, no azimuth data was recorded.

In 2012, Wellgreen completed down-hole surveys using the Reflex Maxibor II© tool. Survey readings were collected every 3 m up the hole. Some measurements or surveys were subject to tool malfunction and deemed unreliable. In 2013, Wellgreen completed down-hole surveys using the Icefield Tools Gyro Shot® tool. Survey readings were collected approximately 9 m off the bottom of the hole and at every 18 m up the hole. Geotechnical/groundwater holes drilled in the Wellgreen deposit were spotted with a hand-held GPS and were surveyed with differential GPS (DGPS). Down-hole surveys were not conducted due to the shallow lengths and vertical dips of the holes.

10.16 Drill Holes for Mineral Resource Estimation.

Figure 10-1 is a map of the drill hole collar locations that have been drilled since the beginning of 1987. The latest 2017 holes are also shown on the map coloured black. As discussed earlier, the 2017 holes have not been included in the resource model and are not material. All of the holes that are plotted as orange on Figure 10-1 represent the holes that were selected for input to the estimation of mineral resources. As noted in earlier, the drill holes before 1987 were not used in this estimate of mineral resources.

Figure 10-1: Drill Hole Location Map, 2017 Holes in Black. 1987 – 2016 Holes Used in the Resource in Blue



Source: IMC, 2018

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Sample collection and preparation protocols have changed over the 65 years of drilling that have occurred at the Project. This section will focus on the procedures that have been followed by the Company on their drilling and sampling as well as the resampling of the 1987 to 1988 historic holes.

The available knowledge of the historic drilling is briefly discussed below, and the outcome of data verification is provided indicating if the information package was utilized in this statement of mineral resources. Information on the historical programs has been summarized from the Technical Report "Mineral Resource Estimate of the Wellgreen Ni-Cu-PGM Project" June 26, 2017.

IMC and John Marek (QP) have formed the opinion that the sampling protocols and procedures that were applied to the data utilized in the mineral resource estimate are appropriate for the determination of mineral resources. Additional discussion of specific programs and procedures are detailed in this section and Section 12.0.

As noted in Section 10, the data from the 2017 drill program has not been incorporated into the mineral resources. The sample preparation and analysis methods that were applied in 2017 are identical to those used in the years 2013 through 2016 as reported in this section.

11.1 Programs Before Wellgreen Platinum

11.1.1 Historic Drill Programs 1952-1988

IMC has not verified sampling details for historic programs. No documented Quality Assurance/Quality Control (QA/QC) programs are available for review. Based on assay results, it appears that Hudson-Yukon only sampled intervals considered to be well mineralized. Long drill intervals were not sampled, and the determination of when to and when not to assay is inconsistent. Hudson-Yukon assayed the core at their internal lab in Flin Flon, Manitoba.

Archer Cathro & Associates Ltd. supervised drill programs in 1987-1988 on behalf of All North Resources, Ltd. Assessment reports filed during these years do not document sampling or analytical details; however, it appears only "mineralized" intervals were sampled.

Archer-Cathro assayed the core at Bondar-Clegg & Company Ltd. in North Vancouver.

File information suggests Archer-Cathro core was analyzed for Pt and Pd by fire assay, and Cu and Ni by atomic absorption (AAS). In addition, some samples were analyzed for the other PMEs and as such underwent neutron activation.

Wellgreen sampled and assayed previously non-sampled core intervals and re-assayed all available sampled intervals from the 1987-1988 programs in 2013. A total of 3,087 samples were analyzed from 108 holes (8,462 m). Most of these samples were ¼ core.

The resampled intervals from 1987 and 1988 were used in the estimation of mineral resources. Otherwise, the pre-1987 data was rejected by IMC and not used in the estimation of mineral resources. Details are provided in Section 12.

11.1.2 Coronation Minerals Programs 2006-2008

The drill core was logged and sampled by the Company's geologist and assistants under the direct supervision of Mr. Rory Calhoun, P.Geo., at the designated facilities of the Coronation Minerals base camp on site. The geologists recorded lithology, mineralization, structures, sample numbers etc., and assistants would record the geotechnical data [rock quality designation (RQD)] and core recovery.

Sample length was variable based on lithology and mineralization observed by the geologist, and the core was marked accordingly. Most sampled intervals were 1.52 m or 5 ft in length. The assistant transported the core into the saw shack and cut it in half using a core saw. After cutting, the core was returned to the core tray, and the geologist would sample it. Half of the split core would be placed in a plastic sample bag with the sample tag. The sample number was also written on the outside of each bag for easy identification. No sample tags were left in the core trays. All of the data from logging the core was recorded in handwritten logs and then transferred to Microsoft Excel™ spreadsheets, for later import into a geological software package.

IMC has verified this data can be used by comparing it to nearby recent Nickel Creek drilling. Details are provided in Section 12.

11.1.3 Northern Platinum Programs 1996-2005 and 2009-2010

There is no available documentation on sampling details for the older Northern Platinum programs; however, based on handwritten assays on the paper drill logs, samples were taken every 5 ft (1.52 m) and were assayed for Cu, Ni, and Co and sometimes for Pt, Pd, and Au.

Northern Platinum sampled core based on lithology and observed mineralization, and where no contacts were present used a nominal 5 ft (1.52 m) sample interval.

Most samples, including field-inserted Standards and Blanks, were sent to Loring Laboratories in Calgary, AB for assaying. In 2009, samples were also analyzed at ALS Global in North Vancouver, BC. Loring Laboratories has ISO 9001:2000 certification and ALS Global has ISO/IEC 17025:2005 and ISO 9001:2000 certification. A 30-element package, including copper, nickel, and cobalt reported in parts per million was analyzed by aqua regia "partial digestion" followed by ICP analyses. Gold, platinum, palladium, and rhodium were analyzed by four acid digestion followed by a 30 g fire assay with an atomic absorption (AA) finish.

IMC has verified the Northern Platinum data from 1996 through 2010 can be used for the economic minerals by comparing it to nearby recent Nickel Creek drilling. Details are provided in Section 12.

11.2 Wellgreen – Nickel Creek Sampling Protocols

11.2.1 Wellgreen – Nickel Creek Programs 2011-2017

The sampling methodology adopted by both Wellgreen and Nickel Creek was as follows:

The drill contractor delivers the drill core to the core shack, and the core boxes are sorted and placed in groups of three. The group of boxes is photographed and run markers and other marker blocks are checked for accuracy. The geologist or technician collects RQD and recovery data, and the geologist logs the core. Prior to 2013 all recovery, RQD, and geology data were hand-written onto paper forms that were then entered into spreadsheets. From 2013 onwards, all of this data is captured digitally in an Access database.

Ideally, only one geologist is logging each individual hole for consistency. Most of the samples vary in length from 0.5 to 3.5 m with 96% of the intervals falling in this range.

In 2013, the sample interval was written on a lab-provided tag that was then stapled onto the box. The tag displays the sample number and interval. Previously, the sample was marked on the box with the footage and sample number in permanent marker. Processed boxes of the core are taken to the core cutting facility for cutting by a technician. The saw uses fresh water for cooling that is not recycled. The core is cut, and the technician places the samples in clean plastic bags with a sample tag. The sample number is written on the outside of the sample bag, and the bag is then sealed using a heavy-duty zip tie rendering it impermeable to outside contamination.

Starting in 2012 through 2017, the core was sawn twice:

- entire core was sawn in half
- one of the core halves was sawn again to generate two, quarter samples

The half core is maintained for possible future metallurgical sampling, while one quarter is left in the box and the other quarter is sent to the lab for assay.

All samples collected in 2011 and 2012, including field-inserted Certified Reference Materials and Blanks, were sent to ALS Global in Vancouver, BC, for assaying. In 2013 all samples were sent to Bureau Veritas (formerly ACME Laboratories) in Vancouver, BC, for analysis. Both labs have ISO/IEC 17025:2005 and ISO 9001:2000 certification and are independent of both Wellgreen and Nickel Creek.

The samples were assayed for copper, nickel, cobalt, gold, platinum, and palladium. The following is a brief description of the sample preparation:

- samples are sorted into numerical order and then dried
- once dried, the material is crushed using a jaw crusher
- the sample is then split to get a 250 g sample for pulverizing
- the total 250 g of split sample is pulverized to 85% passing 75 microns (µm)

- gold, platinum, and palladium are assayed by fire assay fusion of 30 g with an ICP-ES finish; the resulting values reported in parts per million
- copper, nickel, and cobalt are assayed by four-acid "near total" digestion ICP-ES

During 2015 through 2017 the primary assay lab was Bureau Veritas Labs in Vancouver (using Bureau Veritas preparation facility in Whitehorse), and the check lab is AGAT Laboratories. AGAT is ISO 9001:2015 certified. Bags of ¼ core were shipped to ACME in lots of 50 to 100 samples. For every batch of samples, Bureau Veritas sends a second pulp to AGAT as a check assay as directed by the Company's geology staff.

The QA/QC procedure for sample shipment, based on the sequential sample number with samples ending with the following values, is as follows:

- 000 Inserted Certified reference material (CRM) CDN-ME-1309
- 002 Lab Check to AGAT of Sample 001 (Pulp)
- 020 CRM CDN-ME-1310
- 025 Field duplicate (other ¼ of the core, field duplicates not sampled in 2017)
- 030 Coarse Blank
- 040 CRMCDN-ME-09
- 050 Pulp Blank
- 060 CRM CDN-ME-09
- 075 Field duplicate (other 1/4 of the core, field duplicates not samples in 2017)
- 080 CRM CDN-ME-1310
- 090 Coarse Blank

The same QA/QC protocols are used when drilling Reverse Circulation (RC). RC samples are collected at the rig with a rotary splitter. Water was added to all RC sampling to facilitate sample collection.

The sample shipments are transferred from the site to the Bureau Veritas preparation facility in Whitehorse by a third-party transport service. The lab confirms the transmittal list from the mine upon arrival in Whitehorse.

11.2.2 Density Measurement

Both Wellgreen and Nickel Creek completed relative specific gravity measurements with one sample from each core box before sawing the core. Core boxes hold approximately 5 m of core. Samples are solid pieces of core between 10 and 20 cm long.

The sample is weighed directly from the core box for the "in air" weight and suspended in water for the wet weight. Both weights are recorded in the database, and the relative specific gravity is calculated from those values. Samples are air-dried (not oven dried) before testing. There is no provision for sealing the sample in wax or vacuum bags to prevent water from entering the samples.

12 DATA VERIFICATION

The database verification reported here was completed during mid-2017 and addresses the drilling available through the end of 2016. As reported in Sections 10 and 11, there were 15 additional holes drilled in late 2017. The impact of those holes on the 2017 resource model was measured and reported in Section 14. That impact is sufficiently small that it is not material to this statement of mineral resources.

The analysis in this section was originally reported in the Technical Report "Mineral Resource Estimate of the Wellgreen Ni-Cu-PGM Project" June 26, 2017.

The database verification for the Project utilized the following approach:

- The drilling completed by Wellgreen from 2011 through 2016 was confirmed using their QA/QC procedures with check assays, blank insertions, duplicates, and certified reference material.
- Alternative sample methods utilized by Wellgreen were checked against one another including Reverse Circulation to Diamond Drilling and ¼ versus ½ core sampling.
- Once the reliability of the Wellgreen drilling was established, it was used as the basis to compare with the other historic data sets on a nearest neighbour basis.

John Marek of Independent Mining Consultants, Inc. (IMC) acted as the QP for the data verification and determination of mineral resources. As a result of the data verification work that is summarized in this section, Mr. Marek and IMC find the selected database is reliable for the determination of mineral resources. The selected data is the drilling completed between 1987 and 2016 inclusive of the re-assay of core during 2013 that was originally drilled during 1987 and 1988.

12.1 Data Verification

The following checks were performed on the Project data drilled between 2011 to present:

- collar survey check and confirmation of the drill holes
- spot check of certificates of assay versus the electronic database
- statistical analysis of the inserted standards
- statistical analysis of the inserted blank
- statistical analysis of the field duplicates
- statistical analysis of the check assays samples

12.1.1 Drill Hole Collar Survey Checks

Drill hole collars at the Project have not been routinely monumented after completion of each hole. Occasional collars and monuments do exist for verification. Drill pads are prevalent all over the mountain and pads do exist where drill collars are plotted on topographic maps.

During the site visit, the QP and the Company team members hiked to five drill holes that could be observed during an afternoon of walking. Their collar coordinates were spot checked by GPS or by recording the collar ID and back calculating the location against the GPS estimate. Table 12-1 summarizes the field check of the few holes that could be accessed at the time. The error on Table 12-1 is the difference between the handheld GPS and the database coordinates that were based on high precision differential GPS.

Table 12-1: Spot Check of Drill Hole Coordinates

Collar Search	Lat deg-min-sec(dec)	Long deg-min-sec(dec)	UTME (7N) NAD83	UTMN (7N) NAD83	Elevation (m)	Hole ID	Error metres	Comment(s)
1	61°27'53.47”	139°31'48.22”	578,324	6,815,450	1,445	WS09-170	3	PVC w/ concrete rod/monument
2	61°27'55.04”	139°31'46.31”	578,361	6,815,497	1,452	WS52-009	3.5	Possible collar location (from map) no marker found in snow
3	61°27'55.2”	139°31'49.60”	578,312	6,815,502	1,466	WS09-169	6.1	4x4 post with label. PVC pipe with concrete pad
4	61°27'50.28”	139°31'44.19”	578,395	6,815,353	1,373	WS15-271	8	HQ rod or casing @ ground surface
5	61°27'44.76”	139°31'48.23”	578,340	6,815,181	1,364	WS13-215	8	RC casing with HWT insert

12.1.2 QA/QC Verification – Certificate Check

IMC requested the original certificates of assay for 24 drill holes contained in the database. The selection of holes was established by IMC to cover the entire life of the Project drill program from 1988 through the end of 2016 and the spatial distribution of the deposit.

Assay certificates for the historic work before Wellgreen involvement are not available electronically, and the paper files are incomplete. All of the Wellgreen drilled holes in the request list were available and are summarized below in Table 12-2.

Table 12-2: Drill Holes Available for Certificate Check

Year Drilled	Drill Hole Certificates Received			
2012	WS12-211			
2013	WS13-215	WS13-222		
2014	WS14-231			
2015	WS15-255	WS15-257	WS15-263	WS15-271
2016	WS16-283			

The nine holes received contained 1,253 assay intervals or about 5% of the database used for resource estimation. Within those 1,253 intervals, IMC did not find any situation where the Wellgreen database did not match the certificate of assay.

12.1.3 Statistical Analysis of Certified Reference Material

Company geologists inserted certified reference materials with each laboratory submission of samples. Thirteen CRMshave been or are currently in use to monitor laboratory performance.

Six of these are site-specific Certified Reference Materials (CRM) collected from the Project and prepared by CANMET Mining and Mineral Sciences Laboratory in Ottawa as part of the Canadian Certified Reference Material Project (CCRMP). Two of the CRMs were purchased from Ore Research and Exploration Pty. Ltd. (OREAS from Australia), two were purchased from African Mineral Standards (AMIS from South Africa), and three were purchased from CDN Resource Laboratories Limited (CDN from Canada). The certified values of the CRMs are summarized in Table 12-3.

Table 12-3: Certified Values of Certified Reference Materials used by Wellgreen

CRM Name	Nickel %	Copper %	Cobalt %	Platinum gm/t	Palladium gm/t	Gold gm/t
AMIS 0253	0.035	0.014	0.002	4.030	2.340	0.060
AMIS 0326	0.224	0.142	0.006	1.040	1.250	0.170
CDN-ME-09	0.912	0.654	0.017	0.664	1.286	0.154
CDN-ME-1309	0.194	0.519	0.014	0.707	0.363	0.113
CDN-ME-1310	0.379	0.276	0.019	0.433	0.563	0.063
OREAS 13P	0.226	0.250	0.009	0.047	0.070	0.047
OREAS 14P	2.090	0.997	0.075	0.099	0.150	0.051
WGB-1	0.008	0.011	0.003	0.006	0.014	0.003
WMG-1	0.270	0.590	0.020	0.731	0.382	0.110
WMG-1A	0.248	0.712	0.019	0.899	0.484	0.062
WMS-1A	3.020	1.396	0.145	1.910	1.450	0.300
WPR-1	0.290	0.164	0.018	0.285	0.235	0.042
WPR-1a	0.439	0.021	0.021	0.452	0.614	0.050

Provisional Values not Certified

These CRMs reflect a range of values for the six elements that span the grade ranges at the Project. Where certified values were not present, the provisional values were used.

Since the lab does the sample preparation, and the CRMs are pulps, the lab knows which samples are CRMs. However, they do not know which CRM or pulp blank has been inserted in the sample stream.

The CRM results sent to IMC have dates from 2007 through 2016. This period corresponds to the drilling completed by Coronation, Northern Platinum, and Wellgreen. No CRMs were inserted in previous drill programs.

This dataset contained 1,010 CRM (not including blanks). This amounts to roughly one CRM insertion for every 20 assay values during the 2006 to 2016 time frame.

Figure 12-1 and Figure 12-2 are summary plots of the CRM sample values on the X-axis versus the laboratory reported result on the Y-axis. The graph indicates there are numerous sample swaps for all elements being studied. It is likely the wrong CRM was either recorded or inserted in the sample submission.

The CRM WMS-1A shows numerous sample swaps in Ni, Pt, Pd, Au, and Cu, as well as a wide scatter in the high-grade nickel result. The three high values that are shown above the line for the 3.02% Nickel value should have been considered for re-assay.

The OREAS 14P CRM does not have certified values for cobalt, only recommended values which is noticeable in the cobalt graph as a low bias for the sample in the middle of the plot.

The graphs do not indicate any substantial bias in the results for the certified values. Except for the WMS-1A CRMs, the other 12 CRMs perform well in all grade ranges. Gold results indicate some scatter, which prompted the tabulations below.

Table 12-4 summarizes the number of CRMs that are outside of a 10% error band when compared against the CRM value. The detection limit has been added to the 10% error to account for the variation in the very low-grade range.

Table 12-4: CRMs out of 10% Tolerance from CRM Value

Wellgreen CRMs Assay Statistics, 2006 - 2016						
	Ni %	Cu %	Co %	Pt ppm	Pd ppm	Au ppm
Number of CRM Assays	1010	1008	1010	1006	1006	1006
Number Greater than 10% Error	36	25	17	45	38	39
Percentage Outside 10%	3.56%	2.48%	1.68%	4.47%	3.78%	3.88%

The error rates are similar for all metals which may be more of an indication of sample swapping than assay issues.

Figure 12-1: CRMs Results, Ni, Pt, Pd

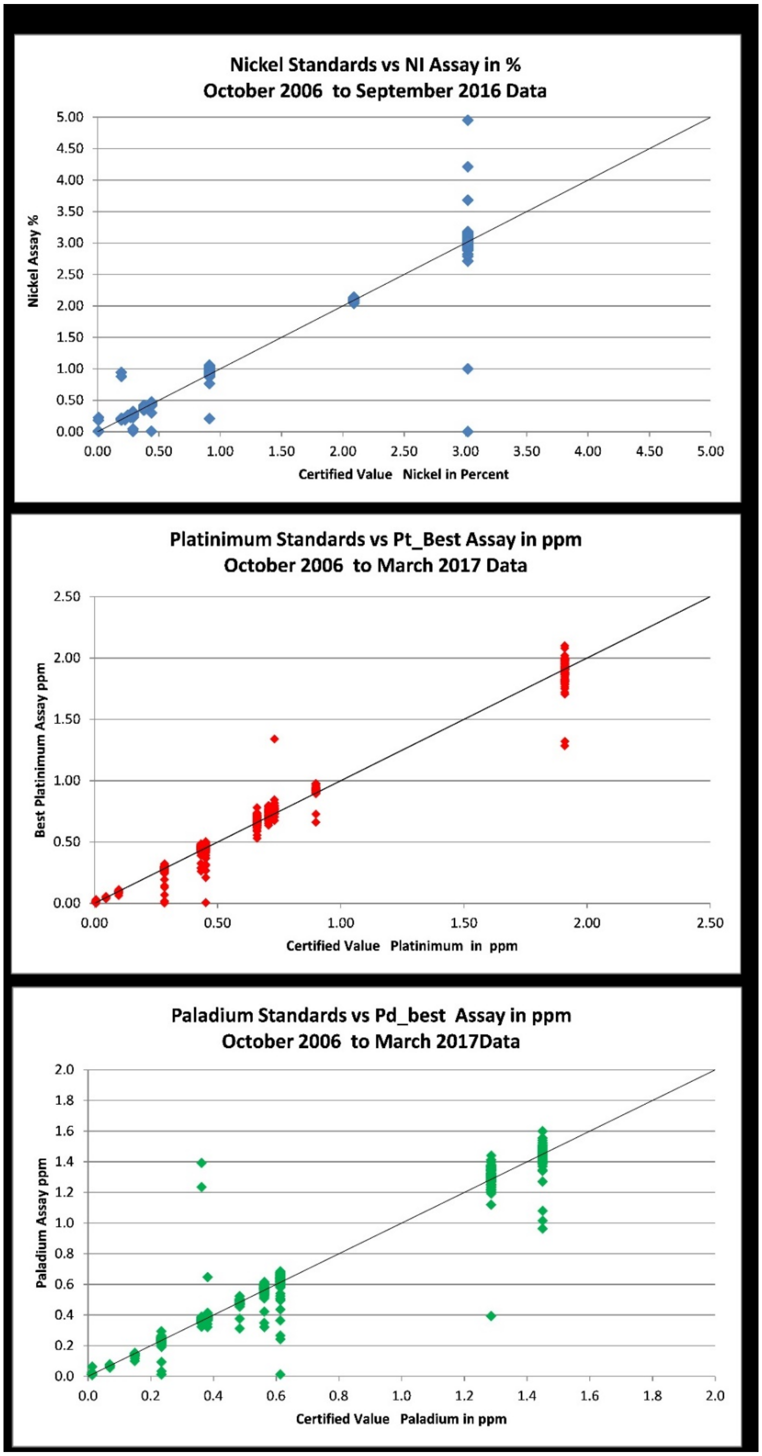
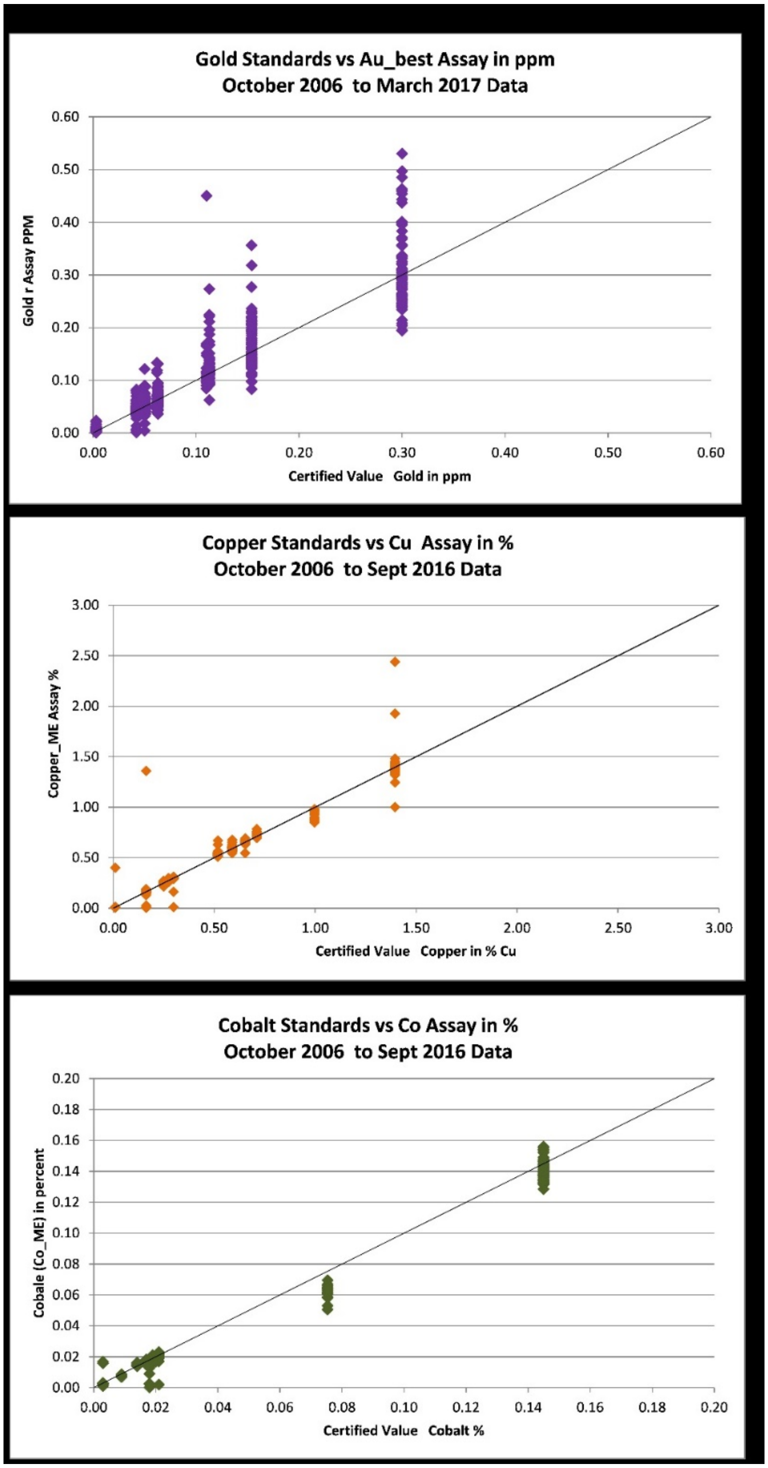


Figure 12-2: CRMs Results, Au, Cu, Co



12.1.4 Statistical Analysis of Blanks

Blank samples were used to check for contamination during sample preparation. The coarse blank material was obtained from two sources: granodiorite from a nearby road quarry, and garden marble from hardware stores in Whitehorse, Yukon. Pulp blanks are sourced from commercial labs.

2015 sample submission protocols state there are two coarse blanks and one pulp blank inserted into every standard submission of samples. The blank data sent to IMC have dates from 2007 through 2016. This period corresponds to the drilling completed by Coronation, Northern Platinum and Wellgreen. No blanks were inserted in previous drill programs.

This dataset contained 909 blanks (not including CRMs). This amounts to roughly one blank insertion for every 20 assay values during 2006 to the 2016-time frame.

Figure 12-3 and Figure 12-4 show the blanks by date assayed for Ni, Pt, Pd, Au, Cu, and Co. IMC established levels for reporting high valued blanks based on detection limit, and low practical values for resource modelling. Table 12-5 summarizes the number of blanks that were higher than expected. In summary, the out of tolerance blanks are few in the data set.

Table 12-5: Blanks above IMC Threshold Value

Wellgreen Blanks Assay Statistics, 2006 to 2016						
	Ni %	**Cu %**	**Co %**	**Pt ppm**	**Pd ppm**	**Au ppm**
Number of Blank Assays	907	907	907	909	909	909
Threshold Grade Level for test	0.02 %	0.02 %	0.002 %	0.05 ppm	0.05 ppm	0.05 ppm
Number Above Blanks Grade Level	7	9	21	10	10	7
Percentage Above Threshold	0.77%	0.99%	2.32%	1.10%	1.10%	0.77%

Figure 12-3: Blank Results, Au, Cu, Co, 2006 to 2016

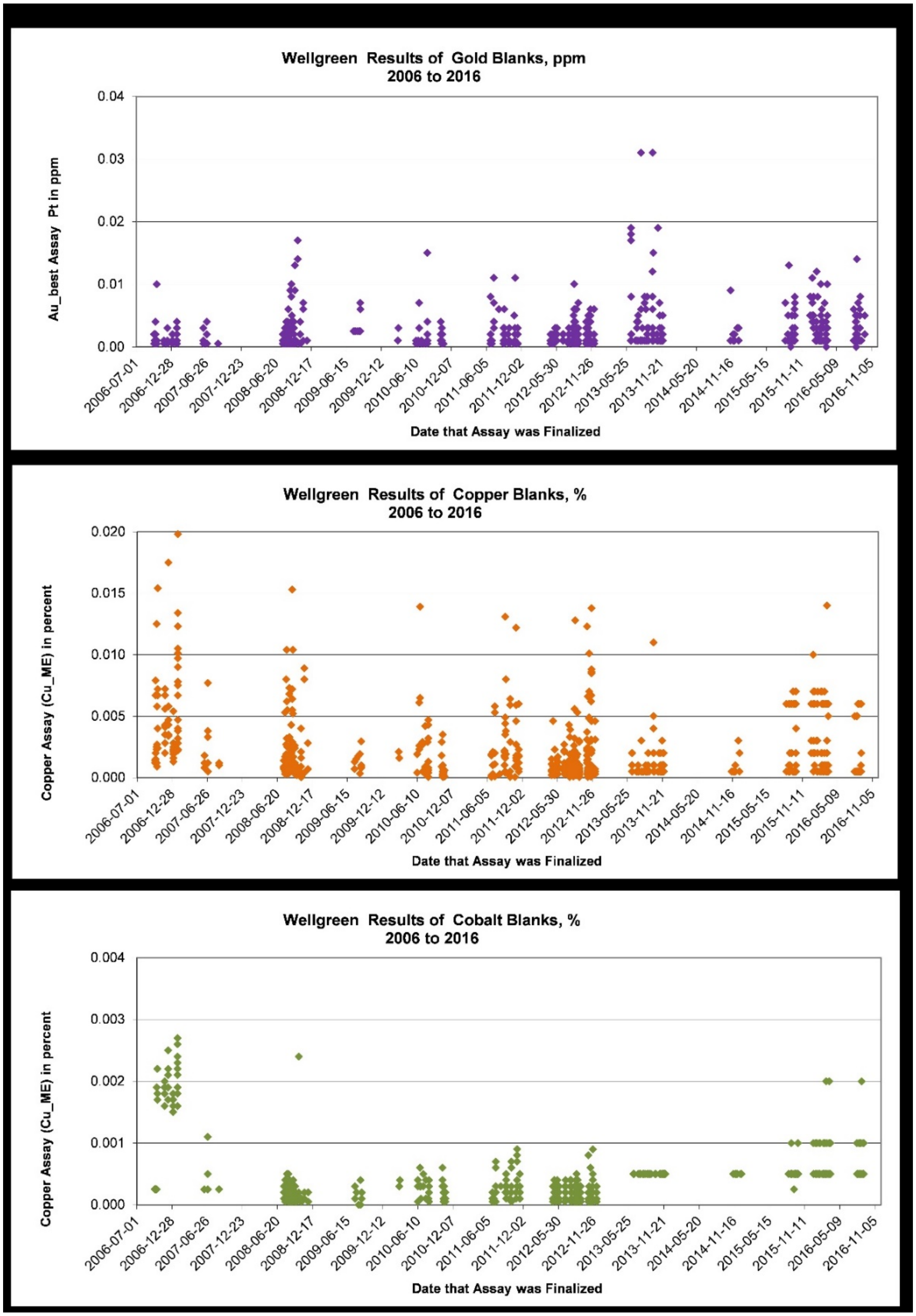
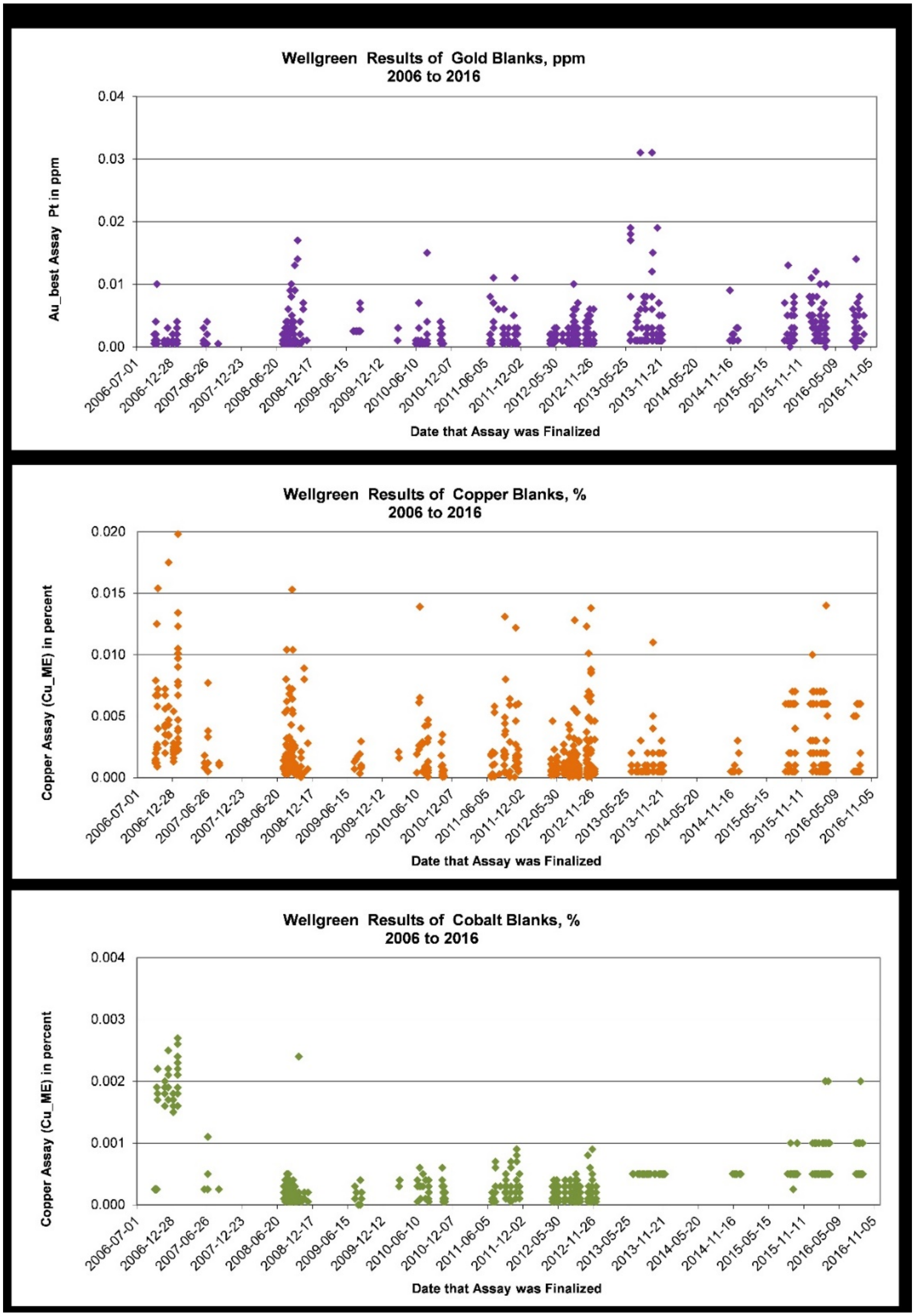


Figure 12-4: Blank Results, Au, Cu, Co, 2006 to 2016



12.1.5 Statistical Analysis of Lab Duplicates

Several types of duplicate samples have been utilized from 2006 through 2016. There is a total of 831 duplicates from 206 drill holes completed during that period.

- 2006 - 2011 ¼ core samples sent as "field duplicates."
- 2012 - 2014 Crusher duplicates from the lab .
- 2015 - 2016 Reverse Circulation duplicates on the RC holes.
- 2016 - 2016 ¼ core samples sent as "field duplicates."

The duplicates are intended to confirm the repeatability of the sample preparation procedures and assay procedures combined. They are not intended to measure bias of sampling or assay.

IMC combined all of the duplicate types for statistical analysis. Figure 12-5 and Figure 12-6 illustrate all of the duplicate results, showing the grade difference between the original and duplicate versus the original sample value.

A 20% error envelope added to the assay threshold is shown on each graph.

Table 12-6 is a count of the number of duplicates that are outside of the 20% error bounds. The detection limit has been added to the 20% error bound to provide more realistic results at grades near detection.

Table 12-6: Duplicate Count Outside of 10% Error

Wellgreen Duplicate Assay Statistics, 2006 to 2016 (All Duplicate Types)						
	Ni%	**Cu%**	**Co%**	**Pt ppm**	**Pd ppm**	**Au ppm**
Number of Duplicate Assays	829	832	829	831	830	832
Number More than 20% Different	46	76	13	67	55	94
Percentage, More than 20% Different	5.55%	9.13%	1.57%	8.06%	6.63%	11.30%
Mean of First Assay	0.240	0.171	0.014	0.252	0.225	0.052
Mean of Second Assay	0.240	0.173	0.014	0.253	0.225	0.054

All of the results are typical for ¼ core and RC samples. The results indicate the level of uncertainty with the ¼ samples. For example, for copper, the results are not repeatable within 20% about 9% of the time.

Figure 12-5: Duplicate Results for Ni, Pt, Pd, All Duplicate Types

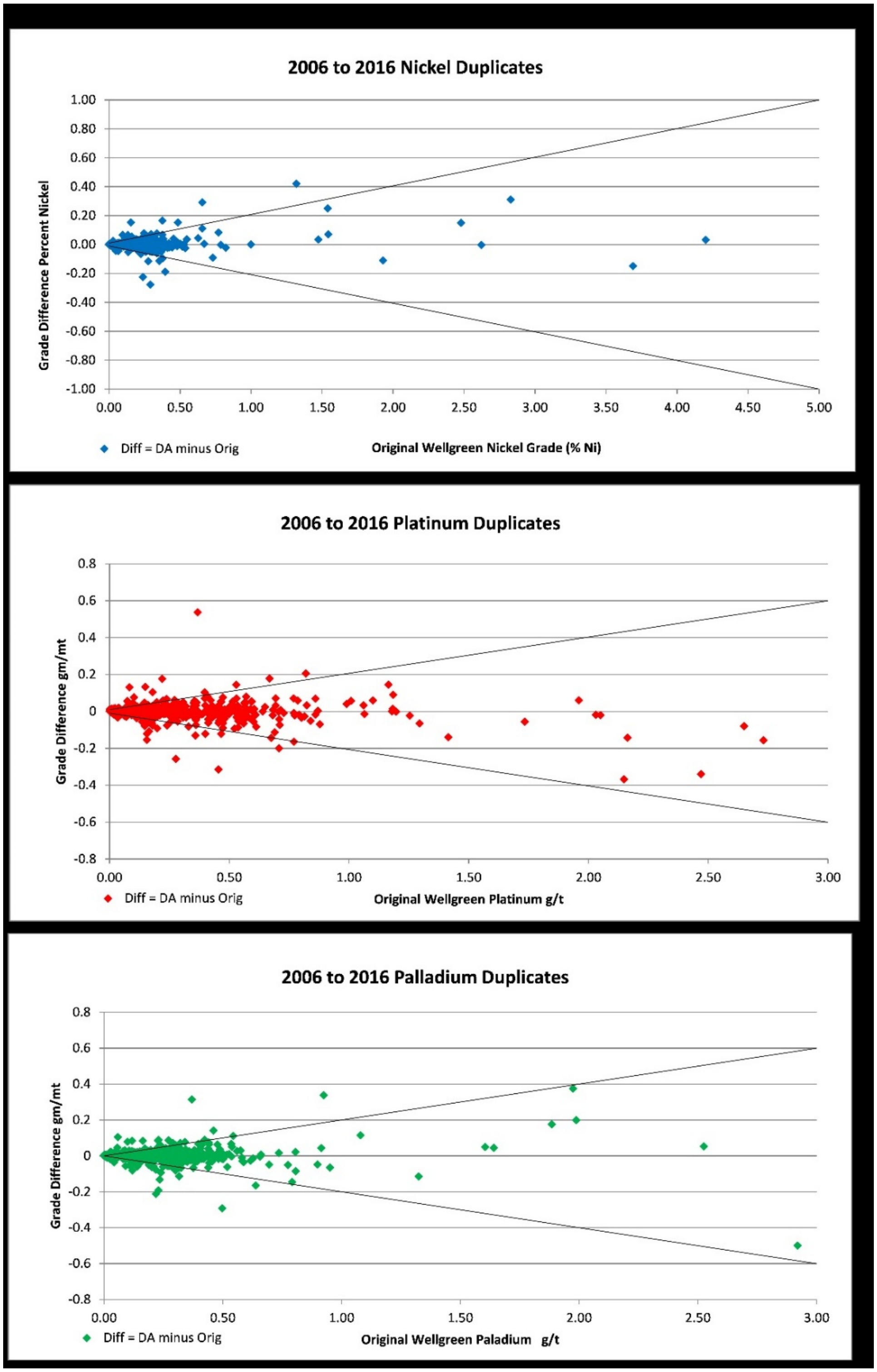
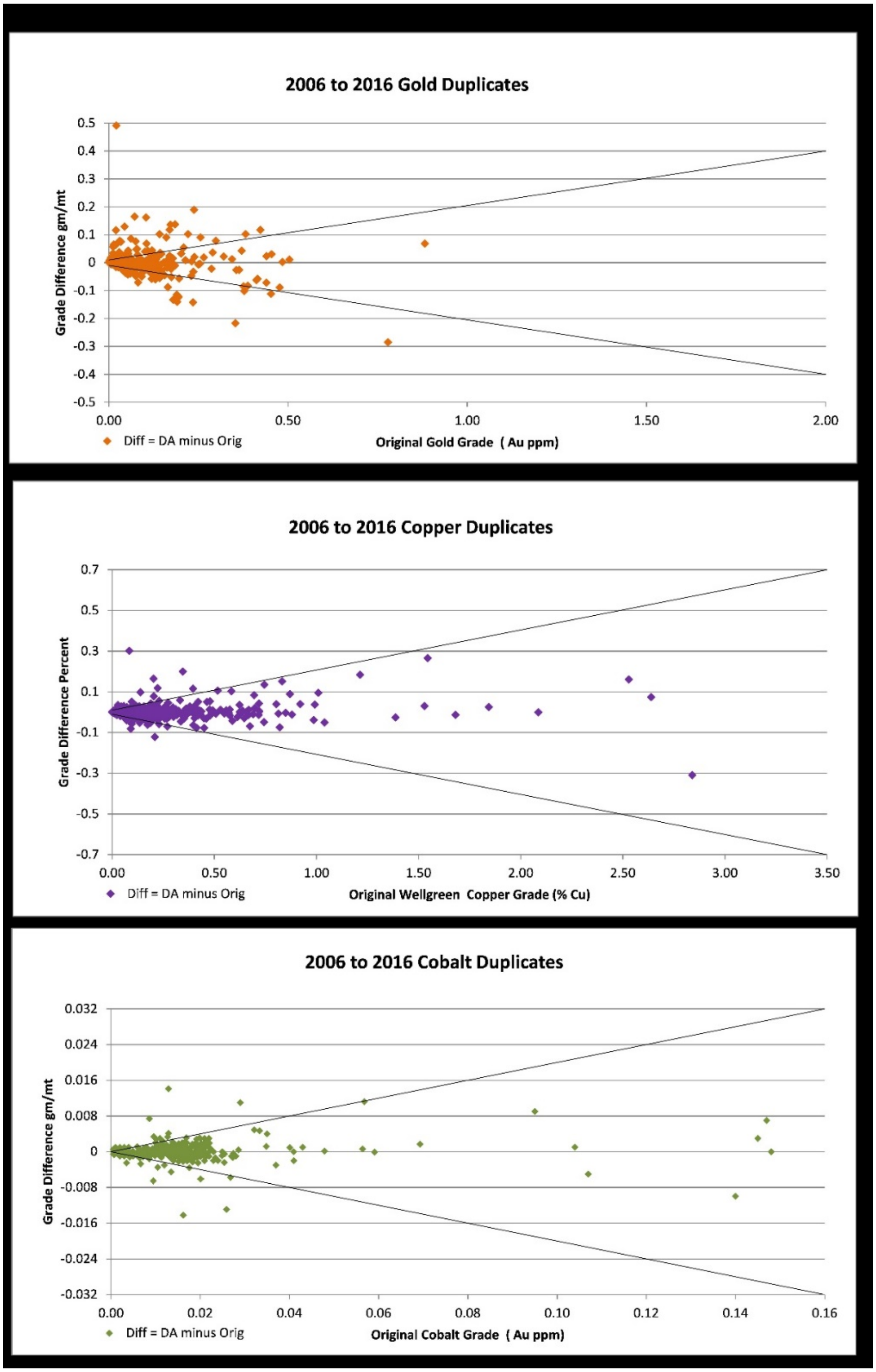


Figure 12-6: Duplicate Results for Au, Cu, Co, All Duplicate Types



12.1.6 Check Assays

In 2014, Wellgreen personnel began requesting the ACME lab send a split of selected assay pulps to the AGAT Laboratory in Whitehorse. This is reportedly every 22nd sample.

These check assays are intended to be a measure on the precision of the ACME pulverizing, pulp splitting, and assay procedure in combination.

Figure 12-7 and Figure 12-8 illustrate XY plots of the lab checks. A scan of the graphs indicates AGAT is reporting low relative to ACME on the same pulps, particularly for the ICP analysis of Ni, Co, and to a lesser degree, Cu. The standards results discussed earlier did not indicate a bias for these metals. The assay of the standards did have substantial scatter due to sample swaps in the data set.

Both AGATS and ACME procedures are four acid digestion with ICP-OES finish for base metals. A review of the standards sent to the AGAT lab should be implemented to confirm there are no issues with the check assay lab.

Table 12-7 is a comparison of the reported means of the original ACME check assays and the AGAT check assays. The table also provides the results of a "students-t" hypothesis test to give an indication of the impact of the bias.

Table 12-7: Check Assay Summary, 2014-2015

Metal	# of Check	ACME Mean	AGAT Mean	Hypothesis Test	
				Students-T	Paired-T
Ni%	119	0.233	0.214	Pass	Fail
Pt gm/t	237	0.260	0.254	Pass	Pass
Pd gm/t	237	0.238	0.234	Pass	Pass
Au gm/t	237	0.050	0.052	Pass	Pass
Cu%	119	0.131	0.135	Pass	Pass
Co%	119	0.014	0.012	Pass	Fail

The Student's-T statistic indicate all the observed bias is sufficiently small and there is a 95% chance the two data sets could have come from the same population. The reason for the bias in Nickel should be further investigated, but there is not sufficient evidence to reject one data set or the other.

Figure 12-7: Check Assays, 2014-2015

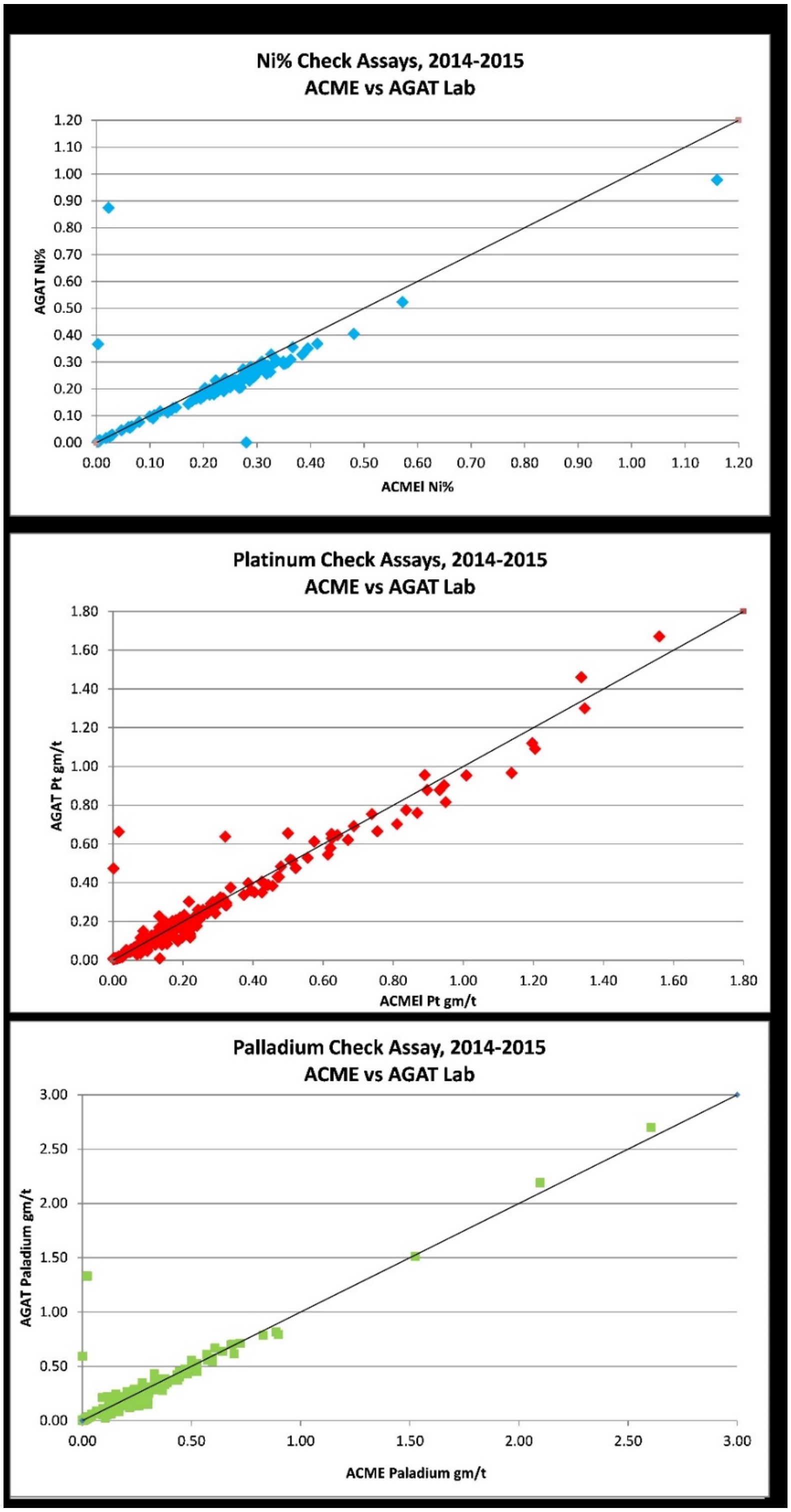
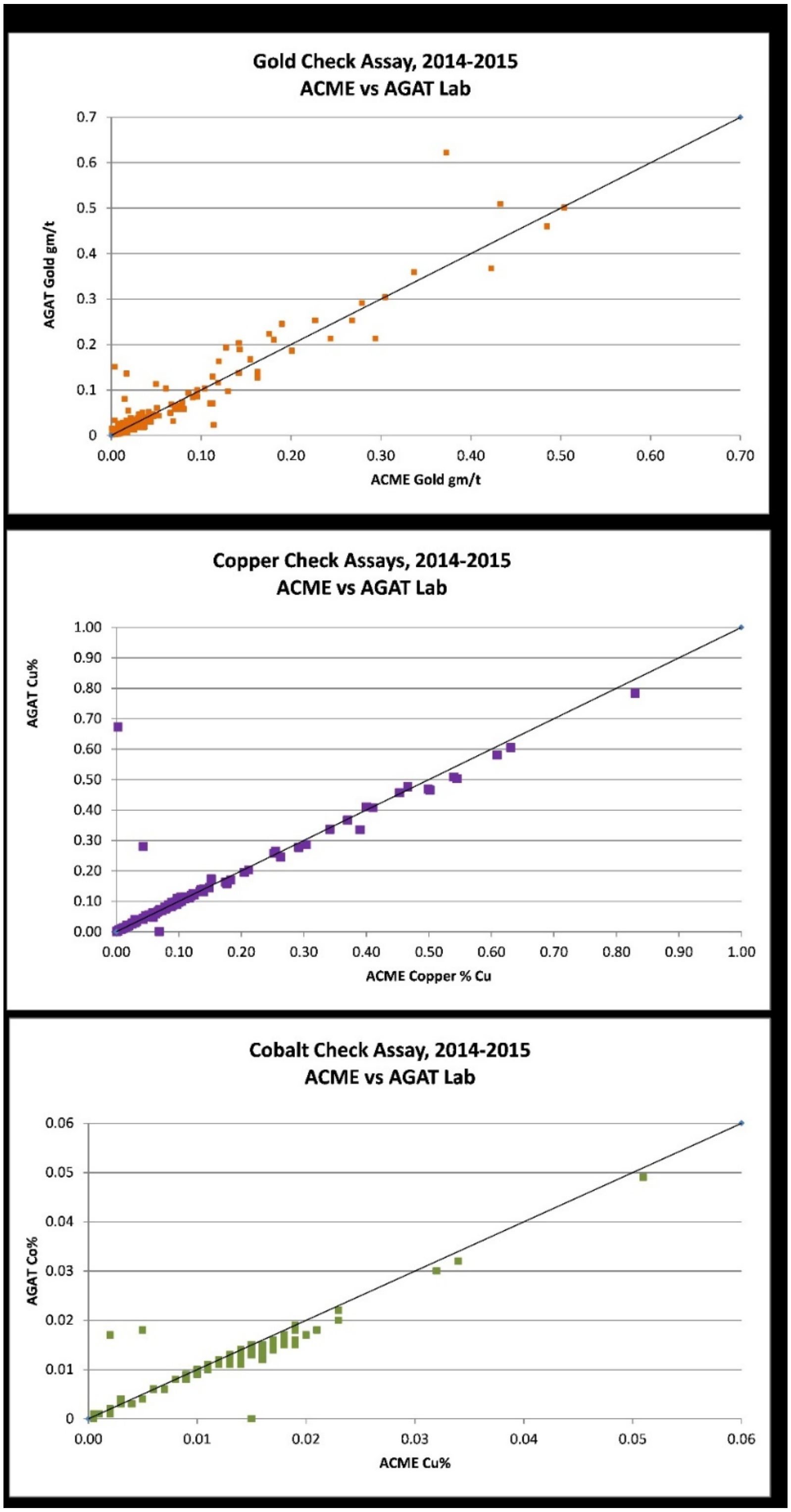


Figure 12-8: Check Assays, 2014-2015



12.2 Verification of Sampling Procedures

12.2.1 Reverse Circulation versus Diamond Drilling

There have been two periods of Reverse Circulation Drilling (RC) at the Project. The most recent was controlled by Wellgreen and amounted to 71 holes from 2013 through 2016. During 1996, Northern Platinum completed 57 holes totalling 3,874 m of drilling.

IMC completed a nearest neighbour (NN) comparison between RC and diamond drilling holes (DDH) based on 10 m composites of the drill database. The composite procedure will be described in Section 14.

The 10 m down hole composites of the RC holes were located, and a search completed to find nearby DDH composites. The paired sets of RC and DDH data were compared statistically at several separation distances. The results are summarized in Table 12-8.

Table 12-8: Nearest Neighbour Comparison RC to DDH Drilling 1987 thru 2016

Metal	Separation Distance	Number Pairs	DDH Mean	RC Mean	Students-T Test	
					T-Statistic	Test Result
Ni %	0 - 10m	33	0.339	0.284	1.339	Pass
	0 - 20m	95	0.256	0.240	0.681	Pass
Pt gm/t	0 - 10m	33	0.451	0.321	1.357	Pass
	0 - 20m	95	0.377	0.308	1.361	Pass
Pd gm/t	0 - 10m	33	0.337	0.253	1.762	Near Fail
	0 - 20m	95	0.258	0.239	0.686	Pass
Au gm/t	0 - 10m	33	0.097	0.094	0.093	Pass
	0 - 20m	95	0.008	0.094	0.433	Pass
Cu%	0 - 10m	33	0.301	0.240	0.756	Pass
	0 - 20m	95	0.274	0.238	0.998	Pass
Co%	0 - 10m	33	0.020	0.018	0.996	Pass
	0 - 20m	95	0.017	0.016	0.898	Pass

The table does indicate a bias where RC drilling averages lower grade than nearby DDH drilling. However, the differences are not sufficiently significant to reject the RC data set.

If the 1987 RC data is analyzed separately from the Wellgreen 2013-2016 data, the 1987 RC information low bias is more apparent. The Wellgreen 2013-2016 RC data compares well to nearby DDH information with minor low bias.

The 1987 RC data statistical analysis indicates it is on the borderline for possible rejection. However, its inclusion is conservative in that the values are lower grade than surrounding DDH. As additional drilling is completed in the future, the inclusion of historical RC data should be reevaluated.

12.2.2 Quarter Core versus Half Core

Starting in 2012, Wellgreen began to sample a quarter of their HQ drill core as opposed to the common practice of assaying half of the core. The core is sawn in half and then half is sawn again. The purpose is to assay a quarter of the core leaving half for metallurgical testing and still have a quarter of the core in the tray for confirmation or check assays. The argument being that a quarter of HQ core is approximately 90% of the volume of half of the NQ core. The same practice was applied to historic core that was drilled in 1987 and 1988 to develop fresh samples to assay that old core reliably.

IMC completed a NN comparison between quarter core and half core based on 10 m composites of the drill database. The composite procedure will be described in Section 14.

The 10 m down hole composites of the quarter core samples were located and a search completed to find nearby half core composites. The paired sets of quarter core and half core data were compared statistically at several separation distances. The results are summarized in Table 12-9.

Table 12-9: Nearest Neighbour Comparison ¼ Core DDH to ½ Core DDH, 1987 thru 2016

Metal	Separation Distance	Number Pairs	¼ Core Mean	½ Core Mean	Students-T Test	
					T-Statistic	Test Result
Ni %	0 - 10m	40	0.288	0.263	0.764	Pass
	0 - 20m	101	0.243	0.237	0.280	Pass
Pt gm/t	0 - 10m	40	0.347	0.272	0.895	Pass
	0 - 20m	101	0.279	0.240	0.950	Pass
Pd gm/t	0 - 10m	40	0.271	0.229	0.995	Pass
	0 - 20m	101	0.220	0.210	0.418	Pass
Au gm/t	0 - 10m	40	0.080	0.075	0.205	Pass
	0 - 20m	101	0.066	0.067	0.129	Pass
Cu%	0 - 10m	40	0.230	0.201	0.470	Pass
	0 - 20m	101	0.201	0.183	0.564	Pass
Co%	0 - 10m	40	0.017	0.016	0.700	Pass
	0 - 20m	101	0.015	0.015	0.544	Pass

The table does indicate the potential for quarter core to be slightly high biased relative to the half core. The hypothesis tests indicate that both the quarter core and the half core can be comfortably merged for determination of mineral resources.

The slight high bias of the quarter core is likely caused by the fact that it has a higher variance than half core. The proportional effect that is common in metal distributions could explain the occurrence.

12.2.3 Comparison of Historic Drill Programs by Company

The drill data collected from 1987 onward has been used for estimation of mineral resources by IMC. That information has been gathered by All North, Coronation Minerals, and Northern Platinum prior to Wellgreen drilling and sampling.

The All North drilling during 1987 and 1988 has been resampled by Wellgreen as well as the same sample practices applied as the 2011 and newer drilling by Wellgreen.

Each of the historic programs was compared with NN methods using the 20 m maximum separation distance and the same 10 m composites as the previous test. Table 12-10 summarizes the results. The test is completed within the primary ore host rock types combined clinopyroxenite, mineralized gabbro, and peridotite.

Table 12-10: Nearest Neighbour Compare, Previous Company Drilling to Wellgreen Drilling 2011-2016

Metal	Company Tested	Separation Distance	Number Pairs	Wellgreen Mean	Company Mean	Students-T Test	
						T-Statistic	Test Result
Ni%	Coronation 2006-2008	0 - 20m	53	0.289	0.260	0.870	Pass
	Northern Plat 96-05, 09-10		42	0.230	0.195	1.287	Pass
	All North 1987 -1988		96	0.242	0.264	0.919	Pass
Pt gm/t	Coronation 2006-2008	0 - 20m	53	0.456	0.439	0.250	Pass
	Northern Plat 96-05, 09-10		42	0.372	0.265	1.310	Pass
	All North 1987 -1988		96	0.296	0.400	2.214	Fail
Pd gm/t	Coronation 2006-2008	0 - 20m	53	0.282	0.280	0.044	Pass
	Northern Plat 96-05, 09-10		42	0.233	0.233	0.019	Pass
	All North 1987 -1988		96	0.217	0.270	2.026	Fail
Au gm/t	Coronation 2006-2008	0 - 20m	53	0.116	0.122	0.298	Pass
	Northern Plat 96-05, 09-10		42	0.095	0.089	0.202	Pass
	All North 1987 -1988		96	0.083	0.105	1.436	Pass
Cu%	Coronation 2006-2008	0 - 20m	53	0.347	0.574	0.711	Pass
	Northern Plat 96-05, 09-10		42	0.245	0.188	1.020	Pass
	All North 1987 -1988		96	0.255	0.298	1.081	Pass
Co%	Coronation 2006-2008	0 - 20m	53	0.020	0.017	1.319	Pass
	Northern Plat 96-05, 09-10		42	0.015	0.014	1.007	Pass
	All North 1987 -1988		96	0.016	0.018	1.317	Pass
Mg%	Coronation 2006-2008	0 - 20m	21	11.64	11.68	0.003	Pass
	Northern Plat 96-05, 09-10		9	19.48	18.93	1.007	Pass
	All North 1987 -1988		94	13.51	13.98	0.617	Pass

The All North program of 1987 and 1988 with the re-assayed intervals illustrate a high bias when compared to the Wellgreen 2011-2016 drilling. Both the platinum and palladium results for the All North data are high biased. The other metals are within tolerance. A partial reason for the observed bias in the All North drilling and sampling could be the quarter sampling that was observed in the previous sub-section.

Observations by Nickel Creek geologist James Berry indicated a number of drill holes with low magnesium (Mg) within rock types that should have higher magnesium due to the observed olivine content. These ten holes were WS09-167 through WS09-176, drilling in 2009 by Northern Platinum. IMC tested the observation by completing a NN comparison of Mg assays in 2009 versus the other approved years drilling, 1987 through 2016. The results indicate the 2009 Mg assays were roughly half the value of nearby holes assayed on different years. It appears many of the 2009 Mg assays were capped at 10% when a significant portion of the host units can average above 20% Mg.

The result of the Mg work was that the Mg values of the 2009 drill holes were coded as "no assay" before additional statistical analysis and block grade assignment. Mg is not an economic metal but was added during 2018 as it has an impact on process response.

12.3 Removal of Pre-1987 Drilling

Hudson-Yukon primarily completed the historic drilling from 1952 to 1972. The location and assay selections from the holes indicate they were likely intended to be stope definition drilling for the underground mine.

The pre-1987 drilling amounted to 542 holes with 25,050 m of drilling and 5,100 nickel assays. The holes were often short underground holes. Assays were selectively high-grade. In the long drill holes, there were often long intervals with no assay data. The indication is the average grades targeted by the Project mineral resource were not of sufficient grade to suggest assay for underground stope design in the 1950's through 1970's. Also, the decision to assay or not assay is inconsistent when viewed in the current context.

One option was to use the pre-1987 (old) data and set the un-assayed intervals to zero as they were likely low-grade. IMC devised a check to determine if that was a sound policy.

All of the pre-1987 drilling that was coded as no-assay was broken into 3 m composite intervals and paired with the 1987 to 2016 drill hole data on a NN basis. Table 12-11 indicates the results of the test.

Table 12-11: Pre-1987 No Assay versus Nearest Neighbour Data 1987 – 2016

Distance to New Data in Metres		Number of New Samples	Mean Ni Grade % of New Samples
From	to		
0	5	34	0.234
5	10	96	0.211
10	15	128	0.220
15	25	373	0.231
25	50	900	0.213
50	75	890	0.228
75	100	883	0.199
100	125	504	0.229

Table 12-11 indicates that as spacings as close as 5 m and as far as 120 m, the average grade of surrounding drill hole data is the average nickel grade of the deposit. If one were to assign zero to the un-assayed intervals, a low bias would be superimposed on the block model.

The option of continuing to treat those intervals as "no-assay" in the estimation process was considered; however, the available grades in the old data appear to be substantially high biased.

Another NN comparison was completed where the assayed values in the pre-1987 work was compared to nearby assays from the 1987 to 2016 drilling and sampling. This comparison selected the high-grade component of the new drilling for comparison.

This analysis used 10 m composites as did many of the previous tests. The pre-1987 data contained many very short intervals, and the composite process was used to reduce the variability of the small samples. Table 12-12 summarizes the results.

Table 12-12: Pre-1987 Assays versus Nearest Neighbour Data 1987-2016

Maximum Separation Meters	Number of Samples Pairs	Old Drilling Mean Ni%	New Drilling Mean Ni%	T Test	Paired T Test
5	11	1.158	0.628	Fail	Fail
10	23	1.021	0.616	Fail	Fail
15	37	1.120	0.654	Fail	Fail
20	65	0.955	0.646	Fail	Fail
25	78	0.977	0.648	Fail	Fail

The test area is the nickel population above 0.35% nickel. The pre-1987 (old) drilling is substantially high biased when compared to the surrounding drilling from 1987 to 2016. All of the hypothesis tests fail. Of interest is the stability in the grade of the selected high-grades from 1987 through 2016, indicating a relatively robust sampling. The old drilling, however, is substantially higher grade and high biased. As a result of these tests, all of the pre-1987 drilling was removed and not used in the estimation of mineral resources for the Project.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical test work has been carried out on samples from the Wellgreen Deposit since 1987.

13.1 Historical Metallurgical Test Work

13.1.1 Lakefield Research 1988

Two test programs were conducted at Lakefield Research (now SGS) in 1988. The first test program evaluated two composites (Comp 1 and Comp 2) containing high sulphide content. Flowsheet development was conducted and Locked Cycle Tests (LCT) were completed for each. The second program evaluated Composite A, which was produced from samples from the first program, as well as an additional Composite B, sample identified as active and a low-grade composite. A Bond ball work hardness was performed on Composite A and was determined to be Wi=17.8 kWhr/t, which is considered moderately hard. The process identified in the final report was a primary grind of 96% +/- 3% minus 200 mesh (d80 ~45 µm), soda ash/sodium silicate for gangue control, xanthate sulphide collector and pine oil frother. After open circuit roughing, the bulk Cu-Ni concentrate was conditioned at high speed before a stage of open circuit cleaning. The Cleaner 1 concentrate was reground to d80 of 25 µm before final upgrading in additional two stages of counter current cleaners. An LCT was completed on Composite A. In the absence of LCT results for Composite B, the low-grade composite recovery at Cleaner 1 is reported as an estimate of composite entitlement. Table 13-1 summarizes the results achieved.

Table 13-1: Lakefield Test Program Results 1988

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
Lakefield	1988	Feb-88 Comp 1	0.65	0.87	6.59	85.30	94.90
Lakefield	1988	Feb-88 Comp 2	0.61	0.90	6.85	76.80	94.10
Lakefield	1988	Nov-88 Comp A	0.57	0.94	5.51	82.10	95.50
Lakefield	1988	Nov-88 Comp A	0.58	0.96	5.46	81.10	96.10
Lakefield	1988	Nov-88 Comp B	0.41	0.57	3.88	79.30	95.40
Lakefield	1988	Low-grade Comp	0.23	0.08	0.25	20.00	33.80

Source: 1988 Report 1 P.40, 1988 Report 2 - P.15, 20, 30

13.1.2 G&T Metallurgical Services 2011

A limited ore characterization and metallurgical program was carried out by G&T in 2011 on a composite sample identified as peridotite Composite 1. The ore characterizations identified the mineralization of half the sample as serpentine followed by amphibole, chlorite, goethite and sulphides. Although Ni and Cu assays are more in line with resource estimates, the S grade of 1.8% was still elevated in comparison to the 2018 resource grade. At a primary grind d80 of 93 µm the liberation of chalcopyrite and pentlandite was low at 35%. The program concluded a grind between d80 of 65 and 93 µm was required. Xanthate (PAX) and MIBC were used as collector and frother. The addition of hexametaphosphate to the cleaners was found to be the most significant for gangue control, however MgO at 11% in concentrate was identified as a potential risk for smelter penalties. No LCT's were performed on this composite but a number of cleaner tests were performed on the optimized rougher conditions. To indicate the entitlement of this composite, the recovery to Cleaner 1 was averaged from the three open circuit tests. The results are presented in Table 13-2.

Table 13-2: G&T Test Program Results 2011

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
G&T	2011	2012 P. Composite 1	0.26	0.29	1.80	65.73	65.10

Source: 2011 Report - P.8, 10, 12

13.1.3 SGS Vancouver 2012

A more extensive metallurgical testing program was conducted at SGS Vancouver in 2012. Several ore type samples were provided and SGS Vancouver produced a master composite for flowsheet development and a high Ni composite for additional testing. In this composite the Ni, Cu, and S assays were elevated above the 2018 resource grades. The master composite was characterized for both mineralogy and grindability. QEMSCAN analysis of the MC composite indicated model mineralogy of 46% serpentine/chlorite, 13% clinopyroxenite, 13% amphibole, 8% orthopyroxenite, and 8% sulphides. At a primary grind of d80 of 80 µm, high liberation (pure, free or liberated) values of 72% for chalcopyrite and 84% for pentlandite were measured. Bond ball grinding tests were completed indicating that the ore as very hard with a Bond Wi at 19.7 kWhr/t to a closing size of 100 µm. An abrasion index (Ai) of 0.088 was measured for the composite, which is low. Two flowsheet configurations were tested; a split float producing a Cu and Ni concentrate, and a bulk float. In both flowsheets the primary grind was at a d80 of 90 µm and xanthate (SIPX) and MIBC were used as collector and frother in the roughers. In the cleaners a 50:50 mixture of CMC and guar gum were used to control MgO recovery and in the split concentrate copper sulphate addition was required for the Ni cleaners. QEMSCAN analysis performed on the LCT flotation products indicated higher Ni recovery would require higher pyrrhotite recovery and that would reduce the grade of concentrate. All the composites were tested to the LCT results, although some were evaluated for the sequential production of Cu and Ni concentrates. For tests where separate Cu and Ni concentrates were produced, the combined recovery to bulk concentrate was calculated. The recovery of all the LCT's is presented in Table 13-3.

Table 13-3: SGS Vancouver Program Results 2012

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
SGS Vancouver	2012	Master Comp 2012	0.48	0.34	2.95	62.80	86.20
SGS Vancouver	2012	Master Comp 2012	0.42	0.33	2.53	63.40	84.90
SGS Vancouver	2012	Master Comp 2012	0.45	0.37	2.92	67.60	87.80
SGS Vancouver	2012	Master Comp 2012	0.44	0.35	2.85	65.70	85.90
SGS Vancouver	2012	High Ni Comp 2012	0.83	0.55	6.73	72.90	88.00

Source: 2012 Report - P.46, 39, 47, 50

13.1.4 SGS Lakefield 2014

A test program was conducted by SGS Lakefield, which issued a report in January 2015. This program was a more extensive program where ten individual variability samples were individually characterized for mineralogy and hardness. The samples were ground to a primary grind at a d80 = 100 µm and sized for QEMSCAN mineralogy analysis. The samples highlighted the variations in serpentine content (80% to 5%) and were classified as six high serpentine samples (peridotite) and four low serpentine samples (gabbro). The high serpentine peridotite samples tended to have lower Ni grade, with a greater proportion of the Ni in non-sulphide minerals, and lower pentlandite liberation. The liberation for pentlandite was between 9% and 50% for the peridotite samples and averaged 70% for the gabbro samples. The hardness of each sample was also measured and ranged from 14.4 kWh/t to 21.3 kWh/t, which is moderate to very hard.

A composite sample identified as LUC was produced from the variability samples for metallurgical testing. A series of open circuit tests identified a sequential flotation of Cu and Ni concentrates of a primary grind d80 of 90 µm was required to produce the best metallurgy. QEMSCAN analysis was performed on the flotation products identifying most of the Cu losses as being attributed to locking, as were the Ni losses to the Mag circuit rougher tails. An LCT was performed using the final conditions and produced separate Cu and Ni concentrates. The combined recovery to the two concentrates was calculated and presented in Table 13-4.

Table 13-4: SGS Lakefield Program Results 2013

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
SGS Lakefield	2013	LUC 2013	0.39	0.41	2.34	68.00	88.50

Source: 2014 Report - P.46

13.1.5 XPS 2014

A test program was conducted by XPS in 2014. The test program included QEMSCAN mineralogy on a composite of peridotite from the previous SGS program labelled 203 Lower. The mineralogy identified 15% of the Ni occurs in silicates and another 15% occurs as ultrafine pentlandite locked in gangue. The flotation Ni recovery entitlement of the sample, based on mineralogy, was determined to be 69%. A test program determined that rougher circuit with a primary grind of d80 = 50 µm, CMC circuit, produced a suitable rougher concentrate that when reground to a d80 of 25 µm and activated with copper sulphate, produced good cleaning results. The test program did not include an LCT. Open circuit testing produced a final concentrate of Cu+Ni of 14.2% and a recovery of 58.2% Ni and 62.7% Cu. The results are presented in Table 13-5.

Table 13-5: XPS Program Results 2014

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
XPS 2014	2014	203 Lower Variability	0.33	0.18	1.24	58.20	62.70

Source: 2014 Report - P.13

13.1.6 XPS 2017 – Phase 1

A comprehensive program was completed in 2017 and early 2018 by XPS. The objective of this work was to maximize Ni recovery to a bulk concentrate and evaluate variability across the resource. Composites were produced from the drill core covering proposed mining years 1-16 (Yr. 1-16), years 1-5 (Yr. 1-5), year 2 (Yr. 2) and years 6-10 (Yr. 6-10) from both peridotite and clinopyroxenite domains. Also, from drill core, 26 variability samples spanning the range of Ni heads, along geologically recognizable domains of peridotite and clinopyroxenite, were identified. A precious metal deportment study identified that up to 1/3rd of the Pt and Pd was locked with magnetite confirming the requirement for magnetic separation circuit.

The Bond ball work hardness was confirmed to be very high with all composite samples having a Wi measuring between 19.0 and 21.5 kWhr/t which is very high and consistent with previous studies. A flowsheet was developed which includes a magnetic circuit after roughing to enhance PGM recovery. MgO flotation was identified as a challenge to concentrate grade with the peridotite composite. To control MgO, in order to obtain concentrate grade, the flotation process required lower feed density and froth depressants. LCT's of the optimized process were performed on the two prevalent geometallurgical composites and the results are summarized in Table 13-6 below.

Table 13-6: XPS Program Results 2017

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu
XPS 2017	2017	PERD Comp Yr. 1-16	0.29	0.15	0.81	58.80	73.40
XPS 2017	2017	CLPX Comp Yr. 1-16	0.25	0.37	1.46	70.50	93.20
XPS 2017	2017	Peridotite Comp - Ph 2a	0.30	0.17	0.87	50.40	60.90

Source: 2017 Report - P.19, 20

Variability testing on the 26 samples identified a relationship between Ni recovery and the S/Mg ratio of the feed. For resource estimation, a projected recovery versus head grade was identified for each of the peridotite and clinopyroxenite.

13.2 Current Metallurgical Test Work – Phase 2

Based on the XPS Phase 1 program recommendations, additional metallurgical samples were obtained by drilling to provide material for a Phase 2 Study. The Phase 2 Study was designed to optimize the bulk flotation process, to evaluate Cu-Ni separation, and to characterize tailings using products obtained from a Mini Pilot Plant (MPP) campaign. As the drill core sample for piloting was not available at the beginning of the Phase 2 program, process optimization was performed on a blend of composites remaining from the previous XPS program.

Control of recovery of MgO to concentrate has been identified as a challenge for this resource. The process optimization phase, which is discussed later, found a low pH acid circuit provided more effective depression of the MgO than did the CMC circuit. The lower pH also resulted in an increase in total S recovery to final concentrate. Grinding size, magnetic separation, and cleaning were also optimized in Phase 2 testing.

With this fundamental change in process, it was considered necessary to repeat the variability testing of peridotite samples from the previous XPS 2017 test work. During this test program, the inherent variability of the resource, based primarily on S levels, became increasingly significant and the Ni and Cu recovery relationships were refined from previous studies.

13.2.1 Sample Variability and Impact on Metallurgy

Several samples were selected to be re-tested using the low pH acid circuit. The samples consisted of one clinopyroxenite intersection and twelve peridotite intersections based on their location in the most recent mine plan. Table 1 lists the samples tested. Sample description indicates the original lithological ore type followed by Ni feed grade. Designation of either A or B indicates there was originally two intersections tested in Phase 1 for that specific Ni feed grade from special different locations. A summary of the samples is presented in Table 13-7.

Table 13-7: Variability Intersections Tested for Phase 2

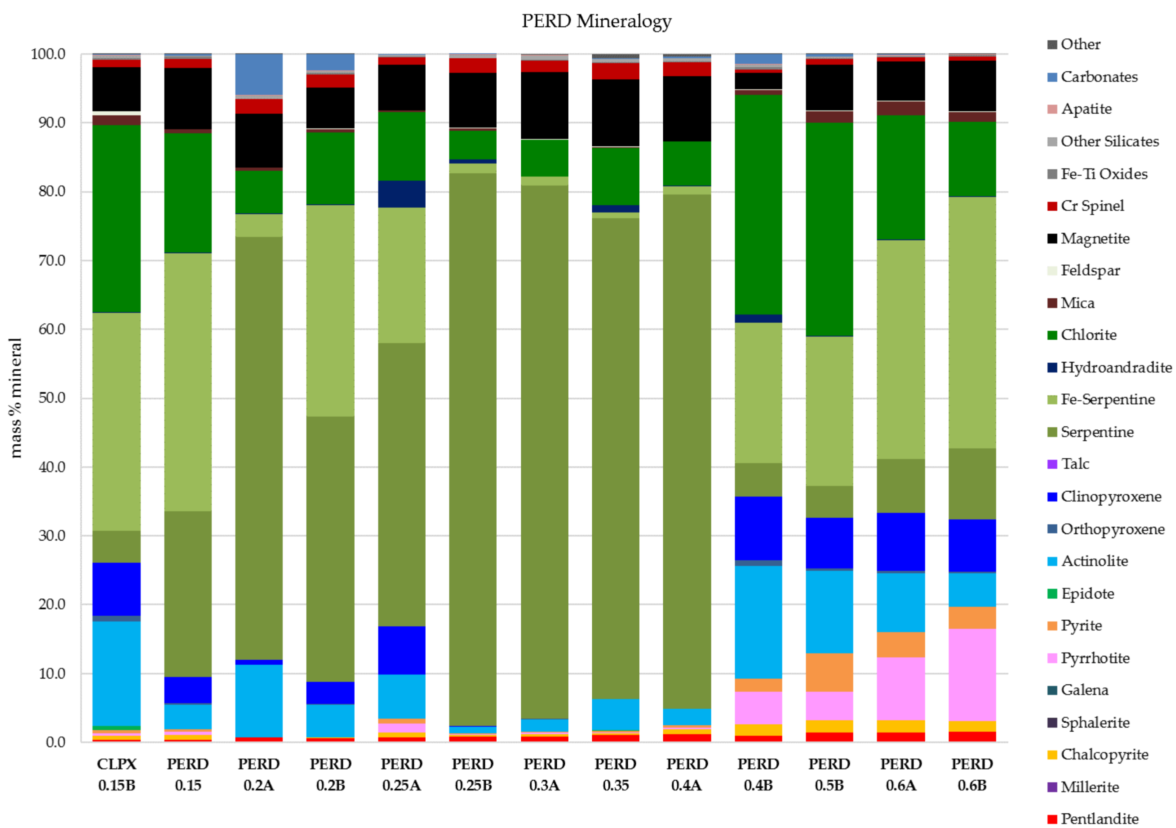
Sample Description	Met ID	Head Assays (%)						
		Ni	Cu	*Cu+Ni*	S	Fe	Co	MgO
CLPX 0.15B	WM16-112	0.14	0.19	*0.32*	0.77	11.40	0.02	29.02
PERD 0.15	WM16-336	0.14	0.18	*0.32*	0.61	11.55	0.02	34.49
PERD 0.2A	WM16-326	0.21	0.02	*0.23*	0.10	8.50	0.01	34.99
PERD 0.2B	WM16-317	0.20	0.03	*0.23*	0.17	9.38	0.01	35.74
PERD 0.25A	WM16-323	0.24	0.21	*0.45*	1.43	12.10	0.02	32.92
PERD 0.25B	WM16-330	0.27	0.06	*0.33*	0.45	8.83	0.02	38.89
PERD 0.3A	WM16-319	0.28	0.12	*0.40*	0.50	9.99	0.02	38.97
PERD 0.35	WM16-331	0.32	0.10	*0.42*	0.59	9.70	0.02	37.81
PERD 0.4A	WM16-334	0.37	0.17	*0.54*	0.85	10.15	0.02	37.89
PERD 0.4B	WM16-284	0.38	0.50	*0.88*	3.74	15.05	0.03	24.87
PERD 0.5B	WM16-329	0.49	0.55	*1.04*	4.97	16.95	0.03	26.62
PERD 0.6A	WM16-328	0.55	0.53	*1.08*	6.12	18.30	0.04	25.62
PERD 0.6B	WM16-327	0.60	0.48	*1.08*	7.96	21.75	0.05	24.63

Source: XPS 2018 Variability Report - P.2

Mineralogy, Head Grade and Entitlement Sensitivities

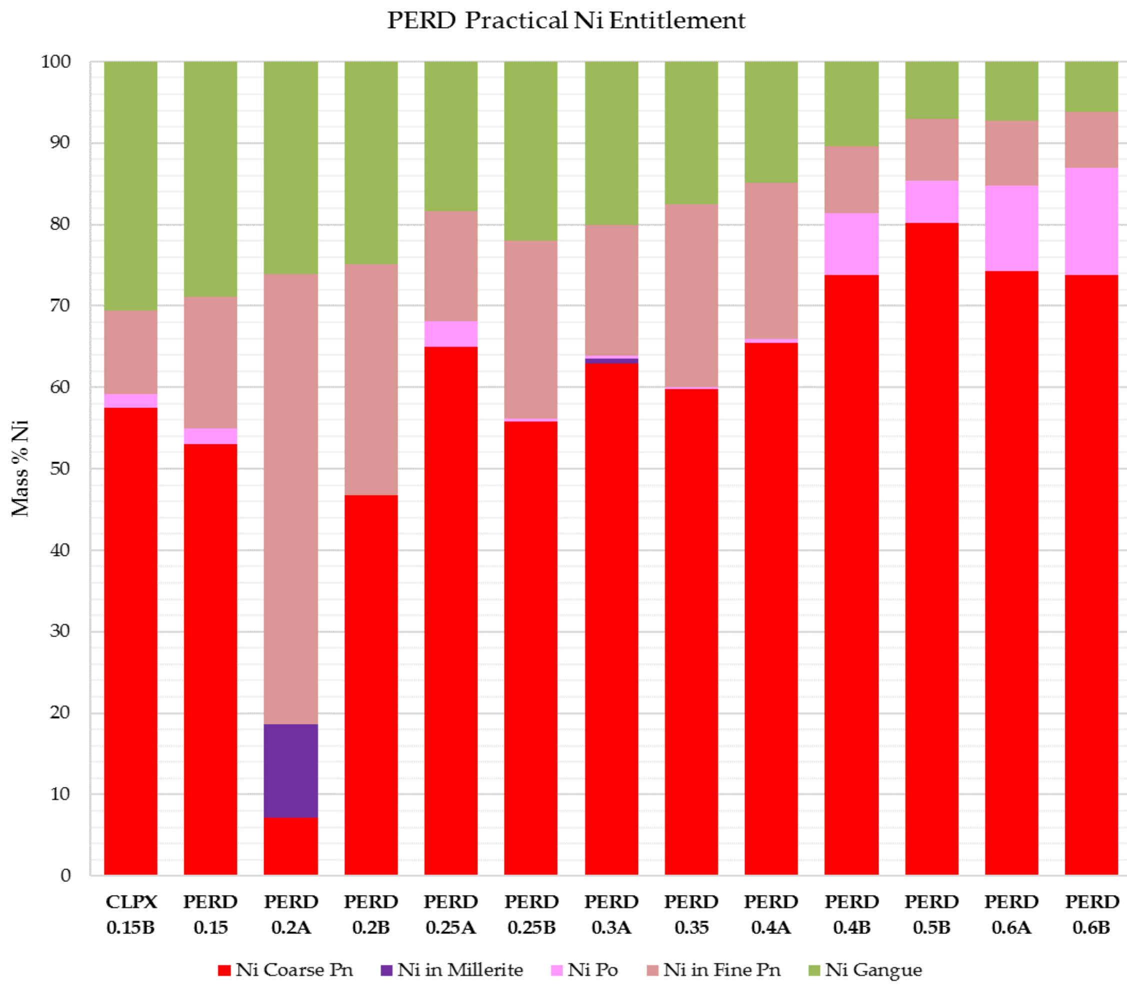
A summary of the mineralogy for the selected intersections is shown in Figure 13-1. Ni deportment was also measured to indicate the percentage of total Ni in Sulphides. Of the Ni in sulphides, an amount occurs as <5µ pentlandite inclusions in serpentine which are not recoverable. Figure 13-2 shows the results of these measurements indicating the remaining Ni in sulphides (shown in red) as the mineralogically measured practical Ni entitlement. Practical Ni entitlement is the maximum amount of Ni in the sample which is recoverable by flotation.

Figure 13-1: Variability Samples Mineralogy



Source: XPS 2018 Variability Report - P.4

Figure 13-2: Ni Entitlement (deportment) (left) and Practical Ni Entitlement (right) for the Variability Intersections Reassessed for Phase 2; Data Presented as Mass Percent of Ni in Each Sample



Source: XPS 2018 Variability Report - P.4

The following mineralogical conclusions were made on intersections tested:

- Mineralogy identified total sulphide and pentlandite content increasing with Ni head grade.
- Magnetite content did not increase with head grade and was relatively constant across the variability samples.
- Trend of increasing pentlandite, chalcopyrite, and pyrrhotite grain size with increased sulphide content and Ni feed grade. (Not observed in CLPX samples). This correlates with a decrease in "unrecoverable" Ni in serpentine.
- Trend of increasing pentlandite, chalcopyrite, and pyrrhotite liberation with increased sulphide content and Ni feed grade. (Not observed in CLPX samples).
- Constant Ni in silicate composition was observed, independent of head grade.

- A base Ni grade of between 0.04% and 0.06% Ni was determined to be in gangue with the average at 0.06%.
- Pyrrhotite did show some levels of increasing Ni in solid solution with increasing pyrrhotite content.

Flotation Results Analysis

The aim of the Phase 2 variability program was to determine Ni recovery sensitivities as a function of Ni head grade with the optimized low pH acid flowsheet (Figure 13-3). The high pH, soda ash/CMC relationship of head grade to rougher Ni recovery had previously been generated using XPS Phase 1 test conditions (Figure 13-4) and it was expected a similar relationship would be seen between the circuits.

Figure 13-3: Variable Testing Flowsheet Phase 2

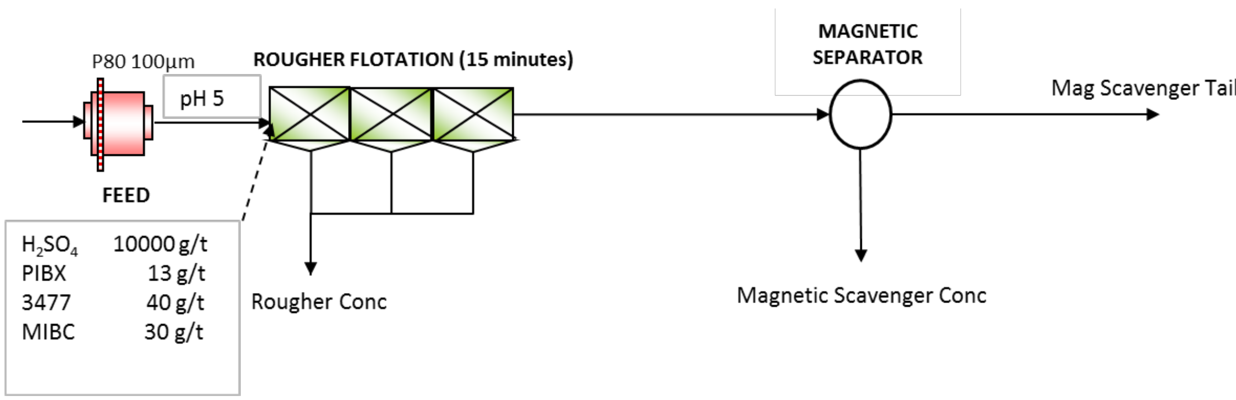
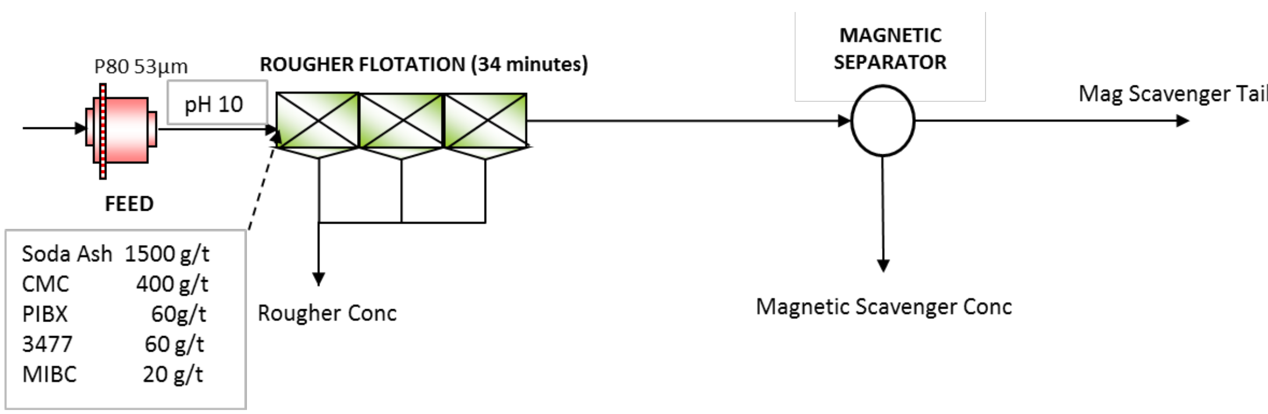


Figure 13-4: Variable Testing Flowsheet Phase 1



A summary of the head grade, rougher concentrate recovery, magnetic scavenger concentrate recovery, and magnetic scavenger tails grade is shown in Table 13-8.

Table 13-8: Concentrate Recovery and Tailings Grade for Variability Tests

Sample Description	Met ID	Head Grade			Rougher Conc				Magnetic Scavenger Conc				Mag Scav Tail	
		Ni	Cu	S	wt%	R%Ni	R%Cu	R%MgO	wt%	R%Ni	R%Cu	R%MgO	Ni	Cu
CLPX 0.15B	WM16-112	0.14	0.19	0.77	8.6	63.4	85.0	8.4	15.9	7.3	3.2	16.1	0.06	0.04
PERD 0.15	WM16-336	0.14	0.18	0.61	4.3	55.2	61.2	3.3	30.2	22.0	18.2	27.6	0.05	0.07
PERD 0.2A	WM16-326	0.21	0.02	0.10	25.0	38.5	54.7	27.4	16.0	17.0	12.7	14.6	0.18	0.02
PERD 0.2B	WM16-317	0.20	0.03	0.17	7.2	25.3	30.7	6.8	29.0	28.5	12.5	24.4	0.17	0.05
PERD 0.25A	WM16-323	0.24	0.21	1.43	12.1	79.3	74.3	8.7	18.9	7.2	7.5	17.1	0.05	0.06
PERD 0.25B	WM16-330	0.27	0.06	0.45	26.9	55.6	38.1	29.1	18.9	24.6	19.7	16.7	0.11	0.05
PERD 0.3A	WM16-319	0.28	0.12	0.50	8.1	37.4	17.3	8.5	27.8	33.5	27.7	23.4	0.14	0.12
PERD 0.35	WM16-331	0.32	0.10	0.59	13.4	50.1	38.2	13.6	19.2	28.5	19.9	13.8	0.11	0.06
PERD 0.4A	WM16-334	0.37	0.17	0.85	17.8	68.4	56.2	17.0	15.0	16.0	12.3	10.6	0.11	0.10
PERD 0.4B	WM16-284	0.38	0.50	3.74	13.9	84.8	88.3	5.5	10.7	3.1	3.1	9.5	0.07	0.06
PERD 0.5B	WM16-329	0.49	0.55	4.97	17.3	89.1	89.3	5.7	7.7	1.7	1.7	7.9	0.07	0.07
PERD 0.6A	WM16-328	0.55	0.53	6.12	19.9	89.7	86.1	6.2	17.8	3.3	4.2	18.9	0.07	0.09
PERD 0.6B	WM16-327	0.60	0.48	7.96	26.0	94.9	89.8	7.7	19.9	1.9	3.4	24.7	0.04	0.07

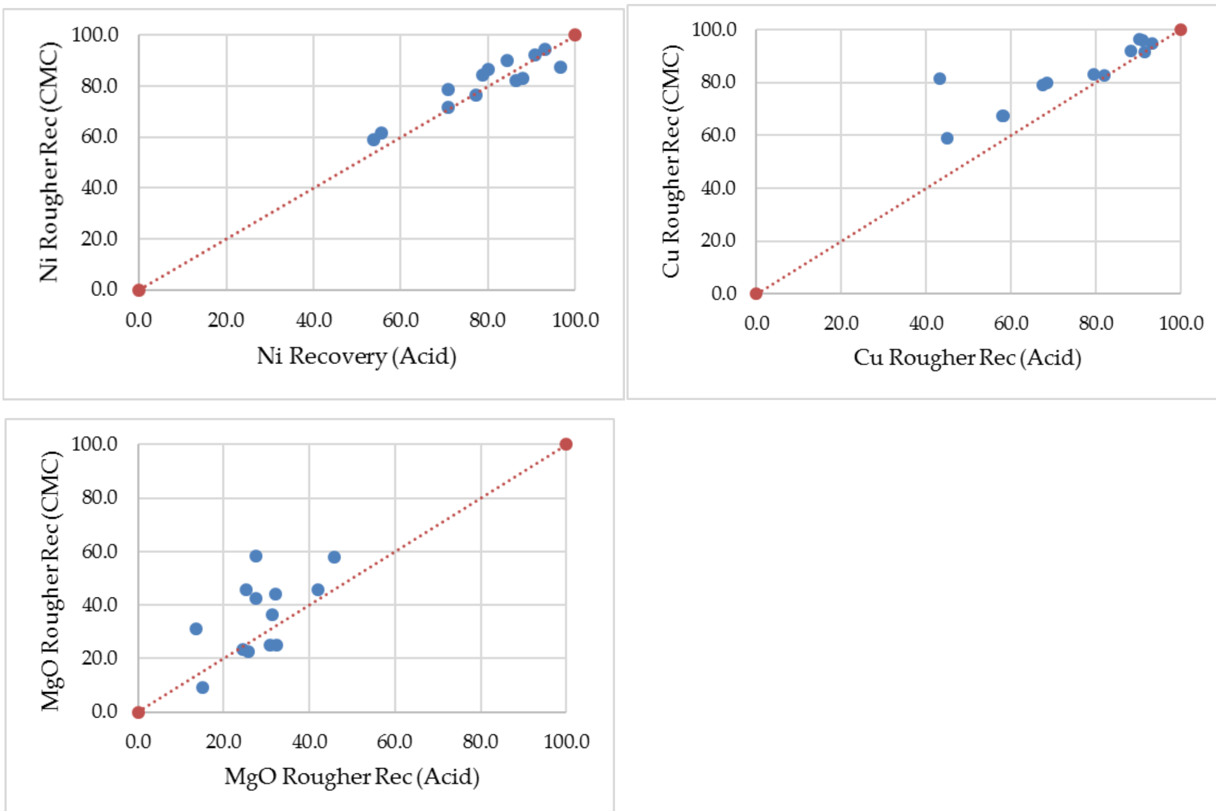
Source: XPS 2018 Variability Report - P.7

The Ni feed grade relationship to rougher recovery for Cu and Ni contained significant scatter. Even with the change in flowsheet, the mass pull to the rougher concentrate was highly variable as was observed in Phase 1. This variability was hypothesised to be related to changes in ratios of the type of serpentine-antigorite, lizardite, and clinochrysotile. The difference in flotation properties drives the MgO recovery to rougher concentrate and results in differences in mass pull. This scatter was somewhat reduced with the low pH circuit and/or the reduced retention time in the optimized (Phase 2) flowsheet rather than CMC and soda ash added in the Phase 1 testing.

The optimized rougher results from the current phase was made with the same samples from Phase 1 and have been compared to review differences in results. Note that primary grind (53µm vs. 100µm), collector dose, MgO depression strategy (low pH acid vs. CMC) and retention time are different between the flowsheets.

A comparison of the total rougher recovery (rougher float + magnetic scavenger) is provided in Figure 13-5 and shows comparable results between the two flowsheets for Ni recovery, and lower recovery for both Cu and MgO.

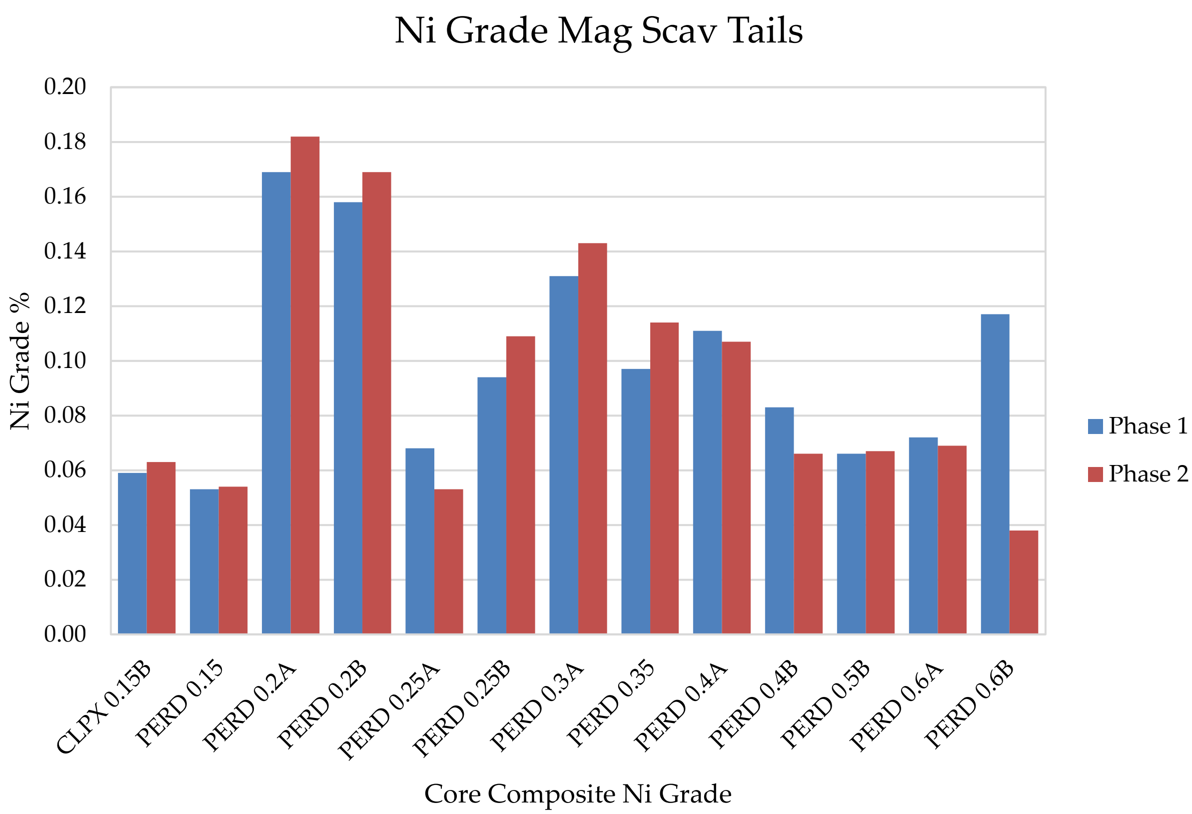
Figure 13-5: Comparison for Ni (top left) and Cu (top right) and MgO (bottom) Total Rougher Recoveries (float + mag scavenger) from CMC Process and Acid Process



Source: XPS 2018 Variability Report - P.7, 8

The Ni grade of the magnetics scavenger tails (Mag-Scav-Tails) was also compared from the two flowsheets and the results are presented in Figure 13-6 . These show Ni tailing grades are within 0.01% Ni of both flowsheets following flotation and magnetic scavenging with the exception of the highest Ni variability sample and in that sample the tail was lower for the new circuit.

Figure 13-6: Comparison of Mag Scav Tail Ni Grade for Phase 1 and Phase 2 Flow Sheets



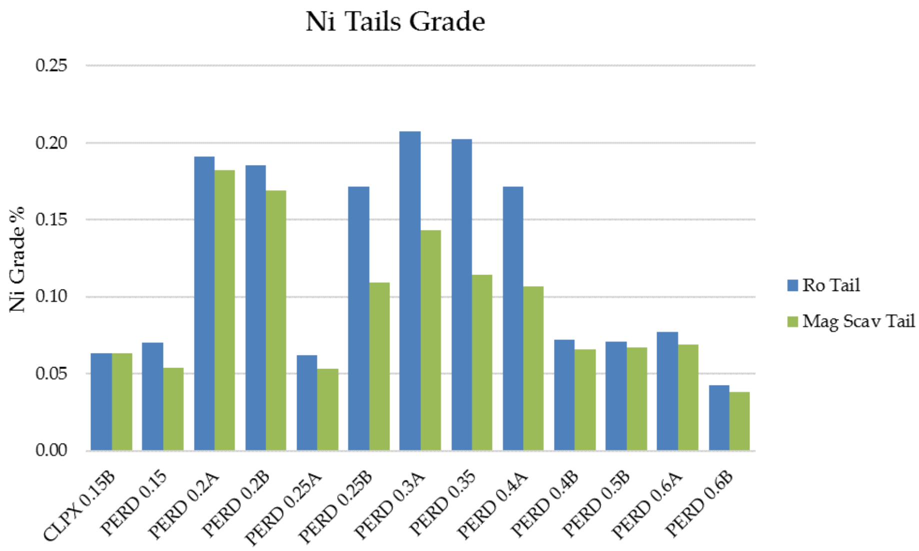
Source: XPS 2018 Variability Report P.8

The rougher tails and Mag-Scav-Tails are plotted for the variability samples in Figure 13-7. This data shows the rougher tails and Mag-Scav-Tails generally trend together except for a few samples. Visually, three populations of "type" of samples emerge from the dataset.

- high rougher and high Mag-Scav-Tails
- high rougher and low Mag-Scav-Tails
- low rougher and low Mag-Scav-Tails

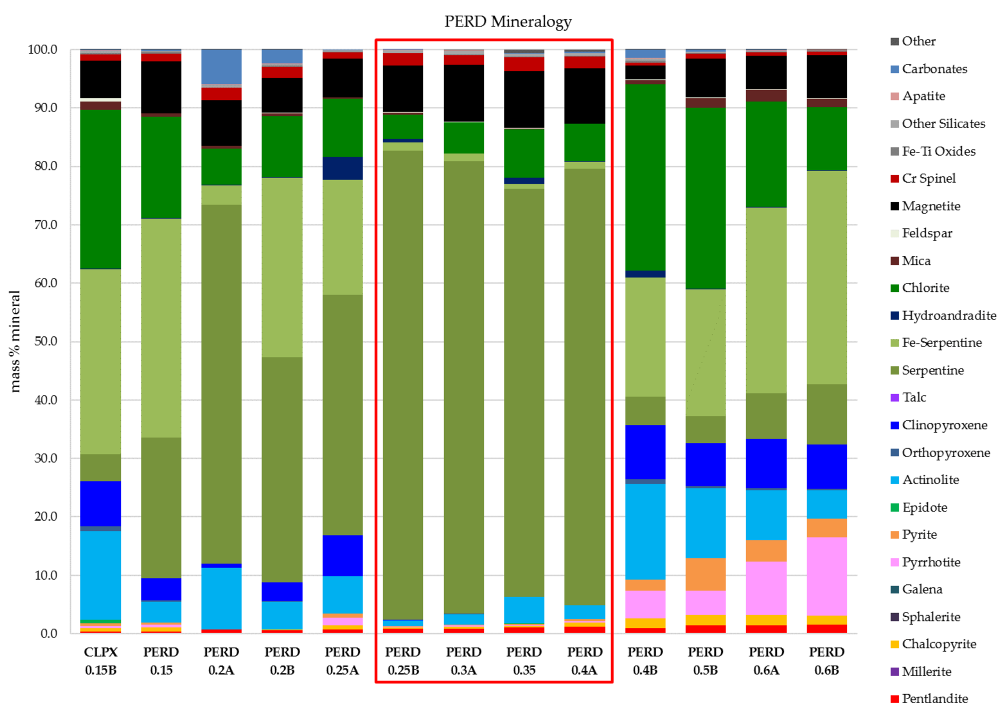
The four samples demonstrating high rougher tails grade and low Mag-Scav-Tails grade were investigated further and show these samples with elevated MgO (high serpentine) as shown in Figure 13-8.

Figure 13-7: Bulk Mineralogy of Variability Sample Highlighting High Serpentine Mineralogy for Samples Showing High Rougher Tails Grade and Low Mag Scav Tails Grade



Source: XPS 2018 Variability Report

Figure 13-8: Bulk Mineralogy of Variability Sample Highlighting High Serpentine Mineralogy for Samples Showing High Rougher7: Rougher Ni Tails Grade and Low Mag Scav Tails Grade



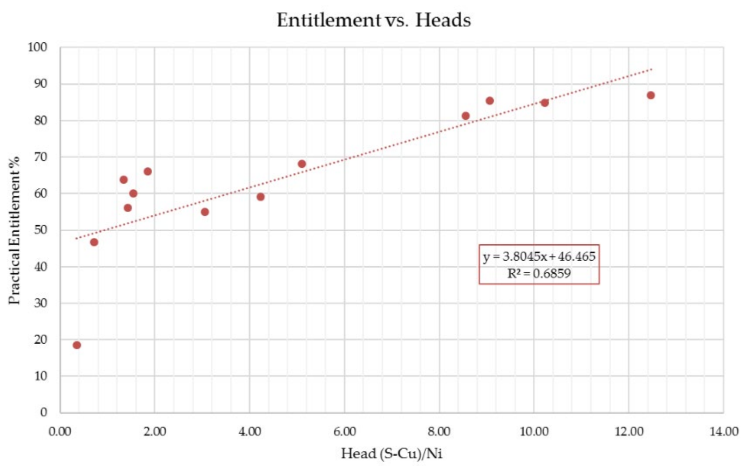
Source: XPS 2018 Variability Report - P.10

Recovery Models

The Phase 1 recovery models were updated using the information obtained in Phase 2 to better predict recovery from the variability samples. A ratio of the Mag-Scav-Tail Ni grade over the Ni feed grade as a predictor of total rougher recovery (rougher + Mag-Scav-Conc) was found to correlate well with achieved recoveries. This ratio was also found to correlate well with the practical entitlement of the variability samples determined by the mineralogy.

A number of assay ratios using S in the heads were modelled against the calculated practical Ni entitlement and the ratio of head (S-Cu)/Ni assays was identified as the most reliable correlation (Figure 13-9).

Figure 13-9: Feed (S-Cu)/Ni Ratio Correlation with Feed Practical Ni Entitlement

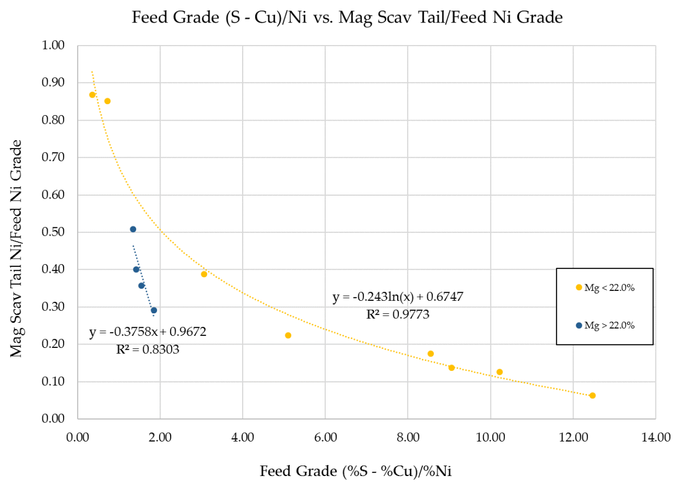


Source: XPS 2018 Variability Report - P.12

This feed ratio also best correlated with the Mag-Scav-Tail Ni grade to head Ni ratio for most of the variability samples. The four samples identified in Figure 13-10 as behaving differently had elevated MgO content and did not fit the curve. These samples, defined as having >22% Mg or 36.5% MgO, show an independent relationship for the (S-Cu)/Ni feed grade ratio to tails/head ratio (Figure 13-10).

The highest MgO samples show higher recoveries than would be predicted by the primary relationship calculated from the other samples. The equation provided below is based on only four samples and may lack a statistical robustness to generate a true relationship. However, only 4.3% of the total resource (indicated and inferred) meets this criteria.

Figure 13-10: Relationship for Feed (S-Cu)/Ni Feed Ratio vs. Mag Scav Tails to Head Ni Ratio



Source: XPS 2018 Variability Report - P.13

Equation 1: (The formula for samples <22% Mg)

Mag-Scav-Tail/Head Ni ratio = -0.243ln(x) + 0.6747

Where; x is feed ratio of (S-Cu)/Ni up to a maximum ratio of 12

Equation 2: (The formula for samples >22% Mg)

Mag-Scav-Tail/Head Ni ratio = -0.3758x + 0.9672

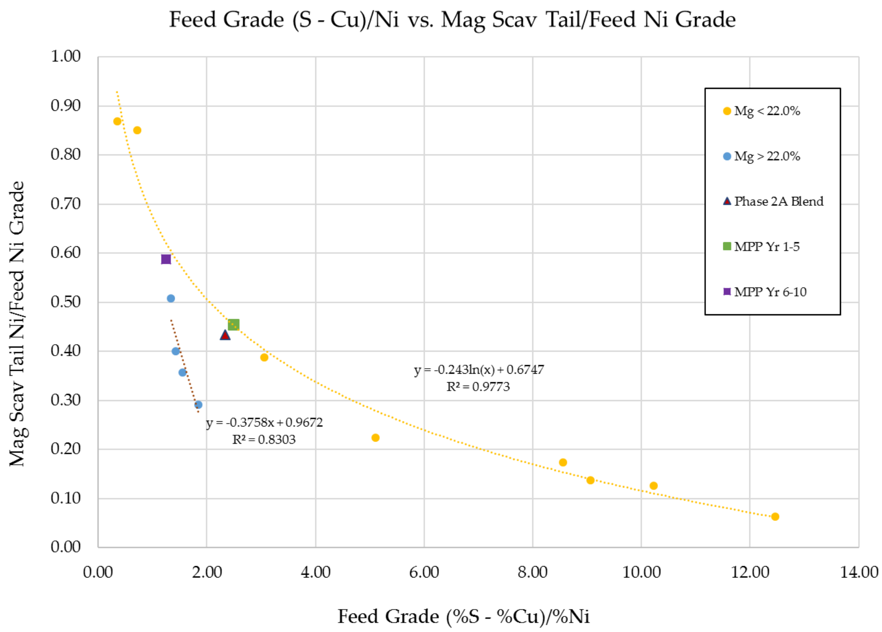
Where; x is feed ratio of (S-Cu)/Ni

The relationship in

Source: XPS 2018 Variability Report

Figure 13-8 is plotted again in Figure 13-11 with the addition of the MPP results from MPP Yr. 1-5 and Yr. 6-10 composites and the LCT results from Phase 2A Blend. Both MPP composites and the Phase 2A Mag-Scav-Tails fit the curve.

Figure 13-11: Relationship for Feed (S-Cu)/Ni Feed Ratio vs. Mag-Scav-Tails to Head Ni Ratio for the Variability Samples and Including the MPP Composites and Phase 2A Blend



Source: (XPS 2018 Variability Report P.14)

The model (S-Cu)/Ni feed ratio is used to predict the Mag-Scav-Tails Ni/Head Ni ratio. This ratio can then be used to estimate final recovery. The ratio can be converted to a Ni recovery by using the formula in Equation 3.

Equation 3: Ni Recovery

Ni Recovery =(1-y) *100

Where; y is the Mag-Scav-Tail Ni/Head Ni predicted by Equation 1 or Equation 2

Cu recovery model was also estimated. The Cu tails grade centres around 0.06% for most of the variability samples. For samples where Cu head grade is 0.06% or lower, the measured head is assigned as the Mag-Scav-Tail grade meaning no Cu recovery to final concentrate.

Equation 4: Cu recovery

Cu Recovery = (Cu Hd - 0.6)/Cu Hd

Where; Cu Hd is Cu head assay

The final metal recovery to bulk concentrate, prior to Cu-Ni separation, used in the resource calculations were calculated based on the above equations.

Application of Recovery Estimates to Current and Historic Metallurgical Studies

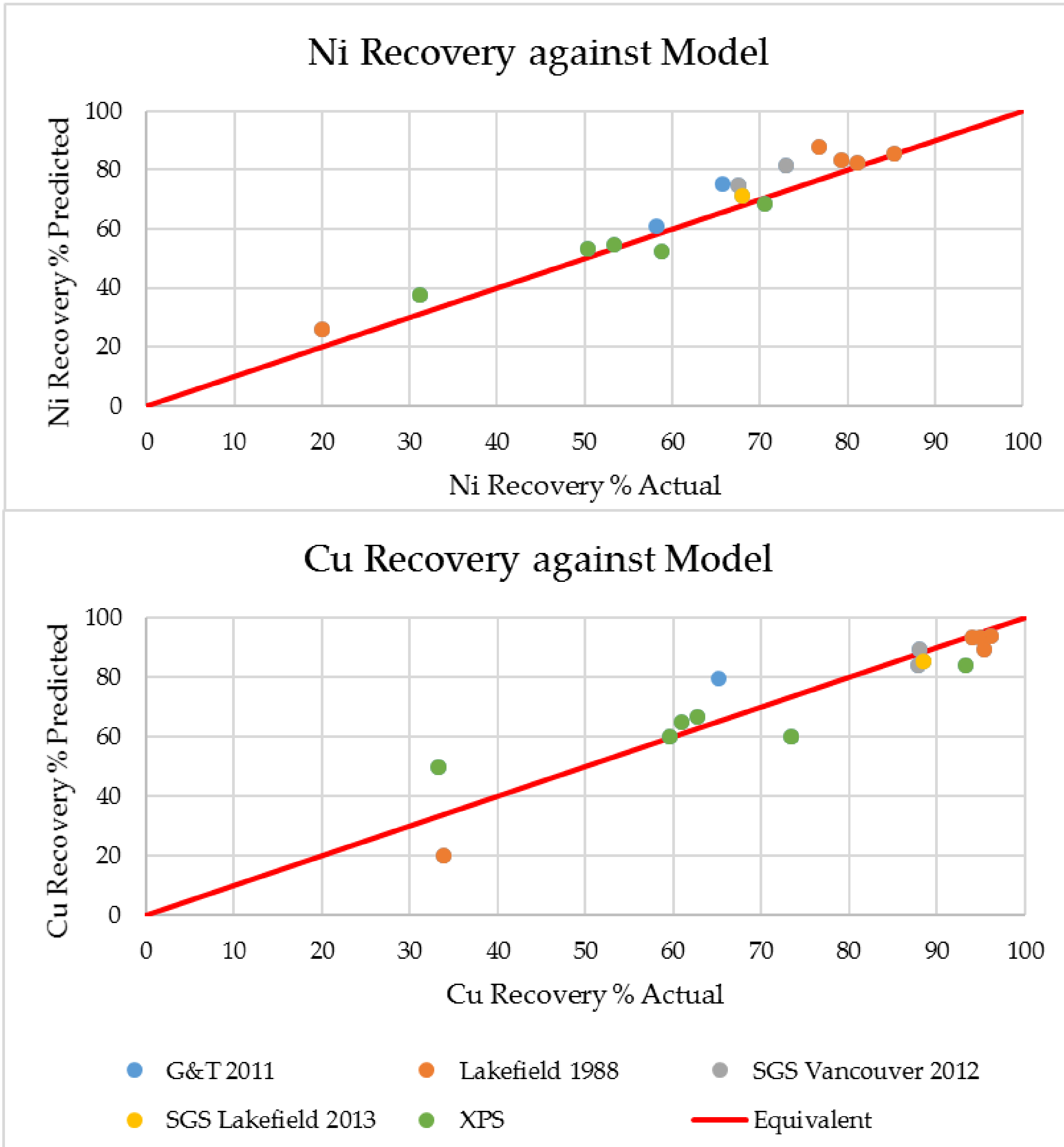
The Ni and Cu resource recovery models developed by XPS were based on a data set of 13 variability samples selected by geomet unit and Ni grade. To test the robustness of the models a review of previous test work was completed on previous composite samples. There have been 15 composites that have been produced and tested from the resource since 1988. The results of these tests are presented in Table 13-9. Twelve of these composites were evaluated with LCT's whereas the remaining three culminated in open circuit cleaning tests. The results from previous composite samples tested to a final flowsheet configuration were plotted against Ni and Cu recovery calculated by the XPS model in Figure 13-12.

The results indicate that the model is predictive of the results of current and historic studies, which increases confidence in the model. The relationship also suggests that the two processing strategies developed, CMC and low pH, results in similar recoveries.

Table 13-9: Summary of Historic Test Work and XPS Model Calculation (from previously reported data)

Laboratory	Year	Composite Sample	Heads Assay			Test Recovery (%)		XPS Model Recovery (%)	
			Ni (%)	Cu (%)	S (%)	Ni	Cu	Ni	Cu
Lakefield	1988	Feb-88 Comp 1	0.65	0.87	6.59	85.30	94.90	85.38	93.10
Lakefield	1988	Feb-88 Comp 2	0.61	0.90	6.85	76.80	94.10	87.88	93.33
Lakefield	1988	Nov-88 Comp A	0.57	0.94	5.51	82.10	95.50	83.11	93.62
Lakefield	1988	Nov-88 Comp A	0.58	0.96	5.46	81.10	96.10	82.32	93.75
Lakefield	1988	Nov-88 Comp B	0.41	0.57	3.88	79.30	95.40	83.28	89.47
Lakefield	1988	Low-grade Comp	0.23	0.08	0.25	20.00	33.80	25.89	20.00
G&T	2011	2012 P. Composite 1	0.26	0.29	1.80	65.73	65.10	75.28	79.31
SGS Vancouver	2012	Master Comp 2012	0.48	0.34	2.95	62.80	86.20	73.68	82.35
SGS Vancouver	2012	Master Comp 2012	0.42	0.33	2.53	63.40	84.90	72.77	81.82
SGS Vancouver	2012	Master Comp 2012	0.45	0.37	2.92	67.60	87.80	74.68	83.78
SGS Vancouver	2012	Master Comp 2012	0.44	0.35	2.85	65.70	85.90	74.75	82.86
SGS Vancouver	2012	High Ni Comp 2012	0.83	0.55	6.73	72.90	88.00	81.32	89.09
SGS Lakefield	2013	LUC 2013	0.39	0.41	2.34	68.00	88.50	71.39	85.37
XPS 2014	2014	203 Lower Variability	0.33	0.18	1.24	58.20	62.70	60.89	66.67
XPS 2017	2017	PERD Comp Yr. 1-16	0.29	0.15	0.81	58.80	73.40	52.51	60.00
XPS 2017	2017	CLPX Comp Yr. 1-16	0.25	0.37	1.46	70.50	93.20	68.31	83.78
XPS 2017	2017	Peridotite Comp - Ph 2a	0.30	0.17	0.87	50.40	60.90	53.12	64.71
XPS 2018	2018	MPP 1-5 Comp - Ph 2b	0.33	0.15	1.00	54.40	57.10	55.52	60.00
XPS 2018	2018	MPP 1-5 Comp - Ph 2b	0.33	0.15	0.97	53.30	59.60	54.65	60.00
XPS 2018	2018	MPP 6-10 Comp - Ph 2b	0.31	0.12	0.50	31.20	33.20	37.48	50.00

Figure 13-12: Comparison of XPS Ni & Cu Models vs Current & Historic Studies (graph display of above data)



13.3 Process Optimization

13.3.1 Sample for Process Optimization

A test program to optimize the process based on recommendations of XPS Phase 1 test work was conducted. The optimization was completed prior to the completion of the drilling for a new metallurgical sample so a sample was created for optimization testing from the inventory of composites available from the previous stage. The remaining PERD Comp Yr. 1-16 augmented with composites to match as close as possible to the LOM grades (Table 13-10Table 13-10). To rule out sample aging, flotation tests were completed on the PERD Yr. 1-16 to show no detrimental impact of storage on the sample.

Table 13-10: Phase 2a Blend Recipe

		Blend Ratio	Cu	Ni	Fe	MgO	S	Au	Pd	Pt
Composite 001	PERD Yr1-16	28.00%	0.15	0.30	10.13	32.70	0.87	0.04	0.25	0.23
Composite 002	PERD Yr1-5	10.00%	0.02	0.30	10.50	29.90	1.05	0.08	0.29	0.32
Composite 003	PERD Yr6-10	42.00%	0.11	0.29	9.48	33.20	0.61	0.05	0.24	0.17
Composite 004	PERD Yr2	10.00%	0.35	0.30	11.63	28.10	1.80	0.08	0.35	0.43
Composite 009	PERD Yr1-5 Spatial	10.00%	0.21	0.31	10.13	32.10	0.86	0.06	0.31	0.32
Calculated Phase 2 A Blend		100.00%	0.17	0.30	10.05	32.12	0.87	0.06	0.26	0.24
Actual Phase 2 A Blend			**0.17**	**0.29**	**10.15**	**31.09**	**0.89**	**0.05**	**0.26**	**0.25**

Source: XPS 2018 Development - P.3

QEMSCAN quantitative mineralogy was determined on the new composite and compared to the PERD Yr. 1-16 to confirm the sample was consistent with the previous phase.

13.3.2 MgO Control

Control of the MgO flotation in both the rougher and cleaner circuits had been identified in previous studies as being critical to the production of saleable concentrates. The previous phase of testing had identified soda ash, CMC, and low-density solids were required for the control of MgO recovery to rougher concentrate. In this phase, it was necessary to scale up from 1.2 kg per test to 2 kg and then to 5 kg to facilitate production of sufficient bulk concentrate for Cu-Ni separation. The increasing density associated with the scale-up resulted in an increase in the amount of CMC required to control MgO flotation. The amounts of CMC required became uneconomic and alternate MgO control strategies were evaluated.

Acid (H_2SO_4) was added to lower the pH to 5 to reduce the MgO entering the froth. The high acid dosage was successful at depressing MgO flotation in the initial stages of flotation. With added flotation time, similar grades and recoveries were achieved but with lower MgO recovery.

Alternatives to the acid (H_2SO_4) addition were also evaluated in the optimization program however, in conclusion, the high acid H_2SO_4 circuit was included in the final process. With the low pH acid circuit, the MgO control was possible with lower residence times and at higher densities. Rougher retention time was reduced to 25 minutes and the rougher flotation pulp density of 22%, was raised to 35% to be more in line with standard rougher operations. These conditions established were shown to be scalable to larger feed masses.

13.3.3 Rougher Collector Addition

Another factor to be optimized was collector dosage, which in excessive amounts, not only adds to operating costs but also negatively impacts downstream Cu/Ni separation. Collector dosage was reduced to a total of 70g/t at a ratio of 3:1 3477: PIBX (from 120g/t total at a ratio of 1:1 3477: PIBX).

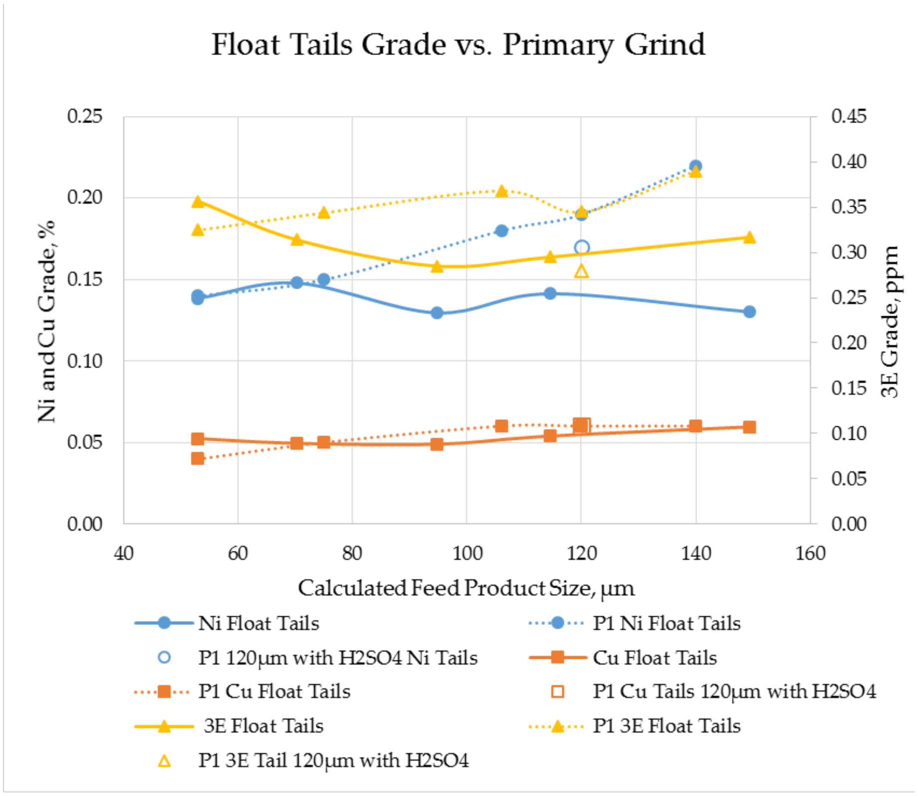
13.3.4 Primary Grind

The primary grind requirement was re-evaluated in this phase. Grind targets of d80 53µm, 75µm, 100µm, 120µm, and 140µm were included in the program. Data showed there was little difference in the sulphide tailings grade between these primary grinds for Ni. PGM tailings grade improved with coarser primary grind. Cu tails increased marginally with grind size.

The flotation tailings analysis by primary grind size is shown in Figure 13-13. The plot of tailings grade-by-grind size compares the earlier Phase 1 sample results using CMC + soda ash (dashed lines) and Phase 2 sample results with H_2SO_4 (solid lines). A repeat test at 120µm on a Phase 1 sample using Phase 2 conditions including H_2SO_4 is also shown.

A grind target of 100µm was chosen for further testing of the ore.

Figure 13-13: Flotation Tailing Grades for Ni, Cu and 3E by Grind Size



Source: XPS 2018 Development - P.12

Magnetic Separation XPS Phase 1 testwork identified that magnetite carries a significant portion of the PGE through locking of precious metal minerals at magnetite grain boundaries. That phase showed between 30-40% of the PGEs are recovered to the magnetic scavenger concentrate after regrinding to a d80 of 10 µm.

For Phase 1, a magnetic concentrate was collected using a laboratory hand magnet. Approximately 20% mass was collected to the concentrate. Mineralogy showed this required regrinding to 10µm to aid in PGE mineral liberation. For this phase of testing, and to better represent industrial practice, an Eriez Electro Drum magnet was used. The operation of the drum magnet was adjusted to get the mass pull to 25%. In Phase 2a a two-pass approach was adopted which was replaced in Phase 2b with a single pass procedure targeting the same mass pull (25%).

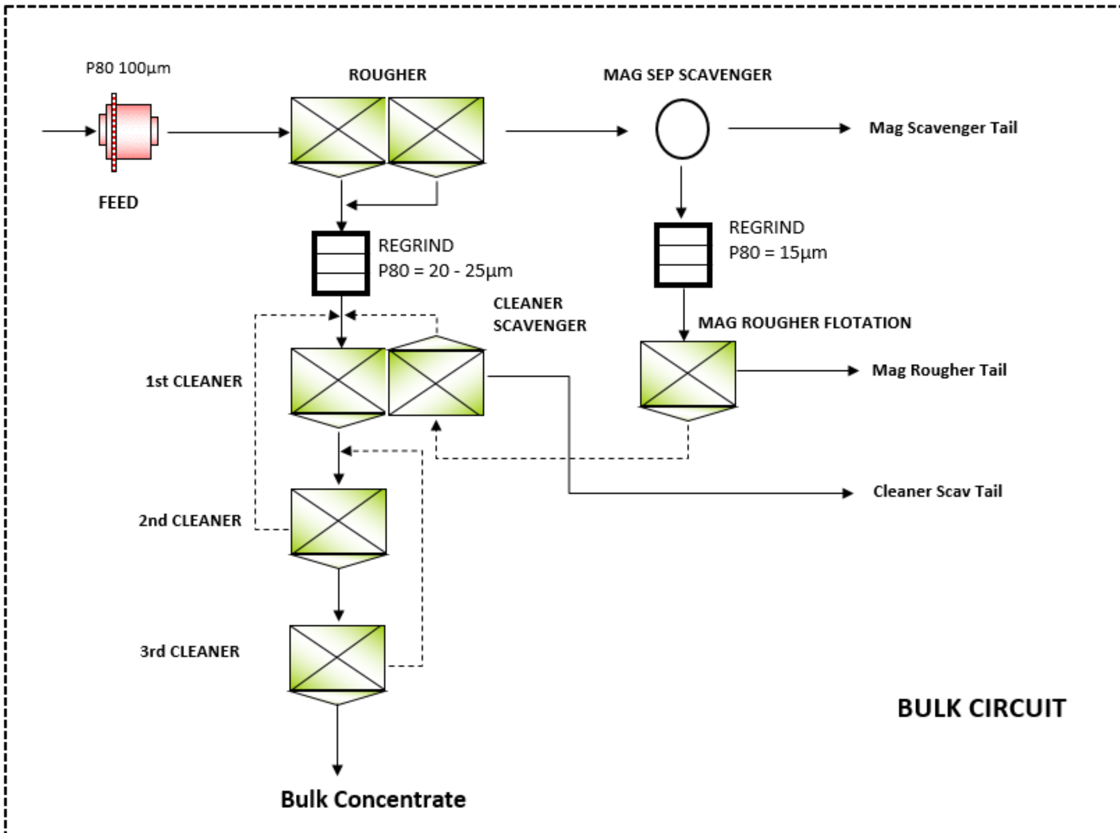
13.3.5 Cleaner Flotation

Cleaner circuit modifications were made to match the cleaner performance of previous test programs. Modifications to the Cleaner 1 conditions included an addition of 10g/t $CuSO_4$ based on cleaner feed and an additional two minutes was added to the retention time. The cleaner scavenger conditions were also modified by increasing collector doses of xanthate and increasing retention to ten minutes which significantly lowered the cleaner scavenger tailings grade.

13.3.6 Final Phase 2 Flowsheet

Based on the optimization phase, the flowsheet for MPP evaluation was finalized and is presented in Figure 13-14.

Figure 13-14: Final Phase 2 Flowsheet



Source: XPS 2018 MPP - P.25

13.4 MPP Operation

13.4.1 MPP Objectives

The objectives of the MPP test program were:

- to generate representative sample of ROM composites for Yr. 1-5 and Yr. 6-10
- to benchmark these composites on the flowsheet recommended from Phase 2 testing

- to complete comminution testing
- mini piloting to confirm overall circuit performance and to generate products for:
- Cu/Ni separation on bulk concentrate
- settling, filtration, process water, and environmental testing on flotation tailings.

13.4.2 MPP Composite Selection

Geology Representation

The MPP composites are representative of production periods and a crosscheck on proportional ratios of rock types within a composite has been completed.

For both composites, the PERD (peridotite) lithology is over represented in the MPP composite compared to the parent population. It was mentioned that the LOM distribution is now defined as 86% PERD and 14% CLPX. The average of the two composites in terms of peridotite distribution (PERD+FPERD) is 88.9%. Clinopyroxenite ore is low for both Yr. 1-5 and Yr. 6-10 MPP composites, however, some of this lithology has been captured. Yr. 1-5 Composite over represents lithology defined as SKAR or massive sulphide.

Grade and Grade Distribution

Grade comparison tables are presented in Table 13-11 and Table 13-12. The tables highlight the ROM target grades, the block model calculated grades, and the average grade of the DDH of the parent population compared against the planned grades of the individual composite recipes as well as the variance between them.

The rows of each table are defined as follows:

- ROM Target Grade – Nickel Creek target grades based on revised block model and mine plan (2017).
- Block Model Population Calculated Parent Grade – Block model grades of the Yr. 1-5 and Yr. 6-10 pits calculated from the same solids that were used for determining the samples for the parent populations and is based on a 0.2% Ni cut-off grade.
- DDH Samples Parent Population – calculated grades (weighted by mass) from all diamond drill holes (DDH) sample intervals extracted from the block model solids that represent the parent populations with 0.2% Ni cut-off grade.
- MPP Composite Calculated Grade – expected grade (weighted by mass) of MPP (MPP) composites using sample assays from split core weighted by measured mass of each sample.
- MPP Composite Actual Grade – Robust measured average grade of between 3-10 sub-samples of MPP composite after crushing, blending, representative sub-sampling.

Note the comparison to the ROM Target grades are most significant in the context of this review.

Table 13-11: Comparison Table of Grades for Yr. 1-5 Target, Parent and MPP Composite Populations

		Ni	Cu	Pt	Pd	Au	# Holes	# Samples	Mass (kg)
		%		ppm					
1	Yr. 1-5 ROM Target Grade	0.26	0.14	0.210	0.240	0.040			
2	Yr. 1-5 Block Model Population Calculated Parent Grade	0.27	0.14	0.213	0.255	0.042			
3	Yr. 1-5 DDH Samples Parent Population Calculated Grade	0.31	0.22	0.317	0.285	0.058	152	2780	
4	Yr. 1-5 MPP Composite Calculated Grade	0.32	0.14	0.218	0.278	0.033	6	213	1744
5	**Yr. 1-5 MPP Composite Actual Grade**	**0.33**	**0.16**	**0.227**	**0.287**	**0.036**			
	Calculated Composite (4) % Variance to Target (1)	**22%**	**-1%**	**4%**	**16%**	**-19%**			
	Calculated Composite (4) % Variance to Block Model Parent (2)	18%	-1%	2%	9%	-22%			
	Calculated Composite (4) % Variance to DDH Parent (3)	4%	-38%	-31%	-2%	-44%			
	Measured Composite (5) % Variance to Target (1)	**27%**	**14%**	**8%**	**20%**	**-10%**			
	Measured Composite (5) % Variance to Block Model Parent (2)	22%	14%	7%	13%	-14%			
	Measured Composite (5) % Variance to DDH Parent (3)	8%	-28%	-28%	1%	-38%			

Source: XPS 2018 Development - P.8

Table 13-12: Comparison Table of Grades for Yr. 6-10 Target, Parent and MPP Composite Populations

		Ni	Cu	Pt	Pd	Au	# Holes	# Samples	Mass (kg)
		%		ppm					
1	Yr. 6-10 ROM Target Grade	0.26	0.11	0.190	0.230	0.030			
2	Yr. 6-10 Block Model Population Calculated Parent Grade	0.28	0.10	0.187	0.241	0.029			
2	Yr. 6-10 DDH Samples Parent Population Calculated Grade	0.29	0.14	0.228	0.269	0.031	86	1907	
4	Yr. 6-10 MPP Composite Calculated Grade	0.30	0.10	0.171	0.244	0.027	9	206	1730
5	**Yr. 6-10 MPP Composite Actual Grade**	**0.31**	**0.12**	**0.178**	**0.266**	**0.022**			
	Calculated Composite (4) % Variance to Target (1)	**16%**	**-6%**	**-10%**	**6%**	**-11%**			
	Calculated Composite (4) % Variance to Block Model Parent (2)	10%	-1%	-8%	1%	-8%			
	Calculated Composite (4) % Variance to DDH Parent (3)	2%	-27%	-25%	-9%	-13%			
	Measured Composite (5) % Variance to Target (1)	**19%**	**9%**	**-6%**	**16%**	**-27%**			
	Measured Composite (5) % Variance to Block Model Parent (2)	13%	15%	-5%	10%	-24%			
	Measured Composite (5) % Variance to DDH Parent (3)	5%	-15%	-22%	-1%	-28%			

Source: XPS 2018 Development - P.8

The following comments and observations are noteworthy:

- A recommendation from the previous sample selection review indicated new drilling (>2015) be added to the parent population dataset. The parent population dataset used here contains all drilling up to 2017, including the samples used for MPP composite selection.
- The Block model parent population grade and the calculated DDH parent population grades are listed as separate numbers. It is observed that the calculated grades of the DDH parent populations are high for both Cu and Pt. These discrepancies are artefacts associated with the estimation process within the block model and how internal dilution is accounted for by compositing the samples. The Cu and Pt grades drop off significantly more from the high- grade zones than the Ni and Pd.
- The calculated MPP composite Ni grades for both the Yr. 1-5 and Yr. 6-10 composites are higher than the target grades. Target grades are 0.26% Ni with Yr. 1-5 calculated as 0.32% and Yr. 6-10 calculated as 0.30%.
- The calculated MPP composite Cu grades are well matched to the targets.
- The calculated precious metal grades for the MPP composite are overall closer to target for the Yr. 6-10 samples. The Pt for the Yr. 1-5 is on target, however, Pd is higher and Au is lower. Given the low-grades of the precious metal variance, within +/-20% is considered acceptable.
- S assays were not directly compared for this dataset, however an assessment by Nickel Creek geologists indicated that S grades were similar to what should be expected based on the sulphur data available in the resource model.
- Overall grade representativity for all elements, except Ni, is acceptable and close to target grades.

The main purpose of attempting to match grade distributions is to help capture inherent geomet variability with the composites (textures, grain sizes, hardness etc.). Grades and grade variability can be a proxy for changes in metallurgical response, so efforts to gather samples which capture grade variability can enhance overall composite representativity.

Grade variability with the two composites captured by the samples selected from the 2017 drilling program have been compared to the parent population grade distributions. Distributions were compared as measured and weighted by length and mass due to the significant difference in interval length distribution observed between the parent populations and the MPP composite populations.

The following general comments and observations are presented:

- Ni distribution for the MPP composite Yr. 1-5 match very well with the parent population.
- Cu and precious metal distributions are generally narrowed compared to the parent populations. These are centred on the mean and it is observed that attempts were

made to proportionally include some low- and higher-grade samples to improve composite representativity.

Spatial Representation

Spatial representativity of the composite is dependent on sample availability. Completing large testwork programs on composites composed of material from highly clustered drill holes or in the centre of the ore body would be considered a potential representativity risk. As such, for the MPP composites, new drilling was completed with a series of 10 holes to generate sufficient sample mass. The location of these drill holes was selected by the Nickel Creek Project team to obtain the best spatial representativity they considered possible.

The following conclusions on representation of the samples tested are noteworthy:

- The number and composition of the composites were selected by the Nickel Creek Project team and were based on updated production profile data (reviewed and revised in 2017); sequential open pit mining Yr. 1-5 and Yr. 6-10.
- There is a recognized grade discrepancy between the target grade and the built MPP composites for both the Yr. 1-5 and Yr. 6-10 periods. For Yr. 1-5 Ni is calculated at 22% higher than target and 16% higher than the target for Yr. 6-10.
- Cu and precious metals show good grade representativity for the calculated MPP composites compared to the target grades.

With respect to grade distribution, Ni is well matched for both composites to the parent population. Cu and precious metals are generally narrowed compared to the parent population, however, attempts were made to pull both lower and higher-grade intervals.

13.4.3 Comminution Testing

Two separate composites of ~90 kg each were generated for comminution testing. Comminution testing was completed at SGS Lakefield and standard sampling protocols were followed. The hardness tests completed were SMC, Bond work index, abrasion testing, and static pressure test (SPT).

The Project samples tested fell at the 40th percentile of hardest measured samples in the JK Drop test database and are classified as moderately soft. The tα was at the 37th percentile and the SCSE was at the 39th percentile. The relative density of the samples was 2.72 and 2.73.

The Bond test generates a Bond work index estimate for use in the sizing of mills. Both Rod and Ball Bond tests were performed. The Rod Bond tests were performed at a closing size of 1180µm, whereas the Bond Ball tests were performed at a closing size of 106µm. The measured Rod bond work index of 15.2 kWh/t is at the 62nd percentile of the hardest of all samples evaluated by SGS and are rated as moderately hard. The measured Ball bond work index of between 19.8 and 21.4 kWh/t is at 92 and 96 percentiles of the hardest of all samples evaluated by SGS and are rated as very hard.

The Bond abrasion test is used to determine wear rates of media in grinding. The results of 0.012 g and 0.005 g which is under the 10th percentile of materials tested, indicated the samples are very mild in terms of abrasion.

The STP test was performed to estimate the energy required for high pressure grinding rolls (HPGR) to achieve a product of 3.35 mm. At 14.4 kWhr/t and 16.5 kWhr/t the samples are classified as medium in terms of HPGR hardness.

The results of the hardness testing are summarized in Table 13-13. Comminution Parameters for MPP Composites Yr. 1-5 and Yr. 6-10

Table 13-13: Summary of MPP Composite Hardness Results

Sample Name	Relative Density	JK Parameters				RWI (kWh/t)	BWI (kWh/t)	AI (g)	HPI (kWh/t)
		A x b[1]	A x b[2]	t_a[3]	t_a[4]				
Composite Yr 1-5	2.72	51.8	49.9	0.55	0.47	15.2	19.8	0.012	14.5
Composite Yr 6-10	2.73	-	49.0	-	0.46	15.2	21.4	0.005	16.5
Average	**2.73**	**-**	**49.4**		**0.47**	**15.2**	**20.6**	**0.009**	**15.5**

Source: SGS Grinding Report - P.ii

13.4.4 MPP Sample

Each of the two composites, Yr. 1-5 and Yr. 6-10, were assembled from 1,670 kg of half core for bench and pilot scale test work. The composites were stage crushed individually to -10 mesh (-1.7 mm) and blended to homogenize the contents of each composite.

260 kg of the homogenized feed from each composite was split into 2 kg test charges for bench scale testing. The remaining material was then split into 6 kg ore charges for the MPP campaign.

Sub-samples for head analyses were taken using a spin riffler. Head assays were completed in triplicate on the two ore blends. A summary of the head analyses is presented in Table 13-14

Table 13-14: Composite Head Assays

		Ni %	Cu %	Co %	Fe %	S %	MgO %	Au g/t	Pd g/t	Pt g/t
Yr. 1 - 5	**Average**	0.058	0.206	0.007	5.02	0.15	33.69	0.003	0.055	0.083
	Rel. Stdv	0.100	0.400	0.000	13.80	0.80	29.90	0.000	0.300	0.300
Yr. 6 - 10	**Average**	0.000	0.000	0.000	0.00	0.00	0.00	0.000	0.000	0.000
	Rel. Stdv	0.000	0.000	0.000	0.00	0.00	0.00	0.000	0.000	0.000

Source: XPS 2018 Development - P.13

Relative deviation values of less than 5% on the base metal elements indicate the head analysis was robust and representative of the blended composite. Relative deviation values are typically higher for the precious metals, particularly Au, due to nuggeting effects and generally lower grades.

Bench and MPP scale results were reconciled against the measure head analyses.

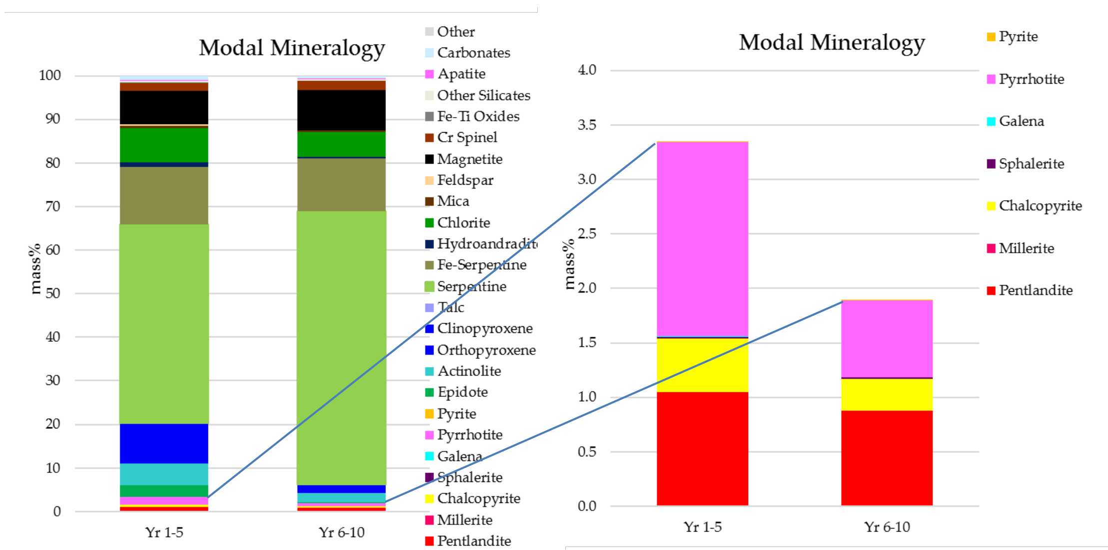
MPP Composite Mineralogy

Figure 13-15 shows the bulk mineralogy from size-by-size measurements for the Yr. 1-5 and Yr. 6-10 samples.

The mineralogy observed for the MPP composites is consistent to what is expected based on the lithology proportions in each. MPP Yr. 1 - 5 shows higher levels of clinopyroxenite, actinolite and epidote minerals consistent with a higher proportion of clinopyroxenite ore type in this sample. The MPP Yr. 6 - 10 sample is predominantly peridotite ore, and this is reflected in the higher proportion of serpentine mineralogy.

XRD was completed on each sample to review the types of serpentine present. No significant changes in peak ratios between the serpentines (antigorite, lizardite and clinochrysotile) were visible in the patterns. There is a significant difference in total sulphide content in Yr. 6 - 10 compared to Yr. 1 – 5.

Figure 13-15: Comparison of Mineralogy of the Nickel Creek Calculated Phase 2A and MPP Composites; Full Mineralogy is Shown Left While Only Sulphides are Shown Right

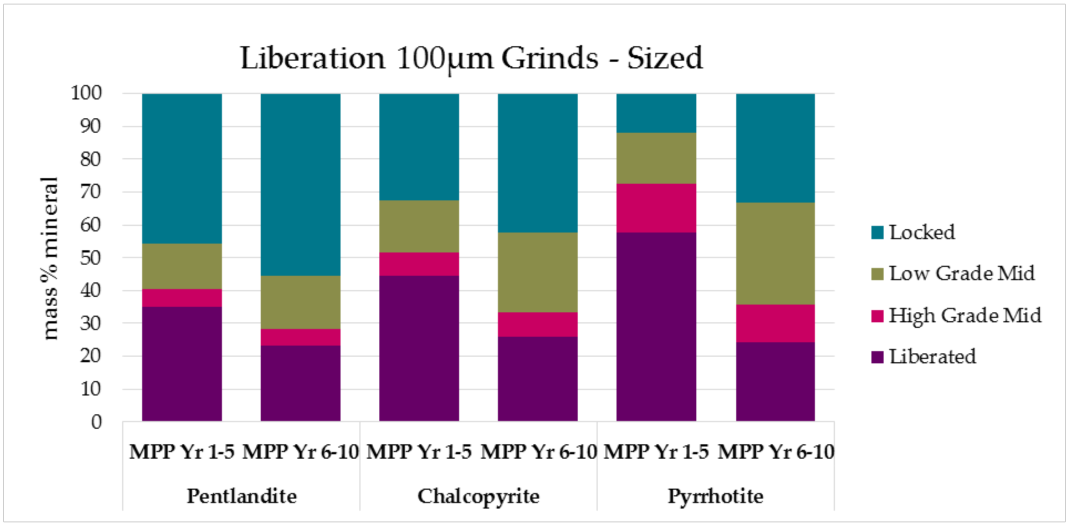


Source: XPS 2018 Development P.14

Using the compositions of minerals determined by EPMA, Ni deportment was determined to be 83% in sulphide for Yr. 1-5 vs. 80% in sulphide for Yr. 6-10. For both samples, 30% of the pentlandite occurs as fine pentlandite/serpentine texture.

Liberation for the sized samples is shown in Figure 13-16. Yr. 6-10 samples had a significantly lower liberation for all sulphides than Yr. 1-5 samples.

Figure 13-16: Sulphide Liberation for MPP Composites Displayed as Mass % of Mineral in Each Sample



Source: XPS 2018 Development - P.16

13.4.5 MPP Results Yr. 1 – 5 Composite

Objectives

The main objectives for the Yr. 1 – 5 composite was to produce a minimum of 25 kg of bulk concentrate for subsequent Cu/Ni separation test work and to demonstrate the performance of the low pH flowsheet. The secondary objective was collection of tailings products for environmental and geotechnical assessment.

Metallurgical Results

The closed-circuit product only survey balance (Table 13-15) showed a bulk concentrate recovery of 53.3% Ni, and 59.6% Cu was achieved at grades of 6.08% Ni and 3.06% Cu. Pt and Pd recoveries of 53.9% Pd and 47.9% Pt was achieved at grades of 4.85 g/t Pd and 3.64 g/t Pt.

Table 13-15: Yr. 1 - 5 Closed Circuit – Products Only Balance

Closed Sample 3	Mass		Assays (%,ppm)									
	(g)	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt
Reference Head			0.33	0.15	0.48	0.018	9.7	0.97	30.0	0.036	0.287	0.227
Balanced Head	200.9	100.0	0.35	0.16	0.51	0.018	10.2	1.06	31.3	0.022	0.280	0.237
Call Factor			**108**	**104**	**106**	**102**	**105**	**109**	**104**	61	98	104
Mag Scav Tail	114.7	57.1	0.15	0.06	0.21	0.007	5.0	0.15	33.7	0.003	0.055	0.083
Mag Ro Tail	48.0	23.9	0.16	0.06	0.22	0.007	15.6	0.20	29.9	0.003	0.190	0.122
Clnr Scav Tail	31.9	15.9	0.27	0.10	0.37	0.013	13.8	0.79	29.9	0.020	0.330	0.295
3rd Clnr Conc	**6.3**	**3.1**	**6.08**	**3.06**	**9.13**	**0.330**	**43.9**	**25.59**	**5.6**	**0.525**	**4.853**	**3.643**
	Mass		**Distribution (%)**									
	(g)	**(%)**	**Ni**	**Cu**	**Cu+Ni**	**Co**	**Fe**	**S**	**MgO**	**Au**	**Pd**	**Pt**
Balanced Head	200.9	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Mag Scav Tail	114.7	57.1	23.7	20.9	22.8	22.2	28.2	8.1	61.5	8.0	11.2	20.0
Mag Ro Tail	48.0	23.9	10.8	9.6	10.4	9.3	36.7	4.5	22.8	3.0	16.2	12.3
Clnr Scav Tail	31.9	15.9	12.2	9.9	11.5	11.5	21.7	11.9	15.2	14.6	18.7	19.8
3rd Clnr Conc	**6.3**	**3.1**	**53.3**	**59.6**	**55.3**	**57.0**	**13.5**	**75.5**	**0.6**	**74.4**	**53.9**	**47.9**

Source: XPS 2018 MPP - P.29

Table 13-16 illustrates the internal balance for the closed-circuit. Rougher selectivity achieved the expected performance. Ni grade of the Mag-Scav-Tailing was higher than the LCT by 0.01%, but the 8% higher mass pull by the magnet pushed the base and precious metals recovery in the MPP over that of the LCT.

The Mag Scavenger concentrate had a density of 37% solids. After regrinding to 17µm, the pulp appeared more viscous than expected. Water was added to the front end of the mag rougher flotation stage to dilute the slurry to approximately 20% solids. Froth was weak after dilution and the frother change to W55 helped stabilized the froth and improved recovery. Since the weak froth was likely dilution induced, an increase in MIBC dosage may be sufficient as an alternative to W55.

The cleaners showed excellent gangue rejection; reducing MgO from 22.7% in the cleaner feed to 5.6% in the final bulk concentrate. Pyrrhotite was the main diluent in the bulk concentrate as the acidic nature of the flowsheet encouraged flotation of all sulphide species.

Sample Generation

All of the bulk concentrate produced during the Yr. 1 – 5 run, except for the portion consumed for assays, was captured and blended for the subsequent Cu/Ni separation testing. The bulk concentrate assays averaged moderately lower grade than the reported balanced assays because it represents concentrate produced during the entire run and not just during the period of the plant survey. Bulk concentrate totalling around 25 kg was split into 50 batches of 500 g. The concentrate was filtered and frozen until used.

Table 13-16: Comparison of Total Bulk Sample produced vs. Survey Result

	Assay (%,gpt)										
	Ni	Cu	Cu+Ni	Co	S	Fe	MgO	Pt	Pd	Au	3E
Bulk Sample for Cu-Ni Sep	5.64	3.37	8.95	0.35	24.5	41.2	6.56	3.06	4.48	0.502	8.04
MPP Survey Sample	6.08	3.06	9.14	0.33	25.6	43.9	5.60	3.64	4.85	0.525	9.02

Source: XPS 2018 MPP - P.39

Tailing samples, as individual tailings streams and combined, were collected as per instruction from Alexco Environmental Group (AEG). Samples were prepared and shipped to SGS Lakefield for environmental and geotechnical assessment. Additional tailings material was shipped to Outotec's testing facility in Sudbury for de-watering and filtration testing.

13.4.6 MPP Results Yr. 6 – 10 Composite

Objectives

The Yr. 6 – 10 composite showed poor flotation selectivity and recovery during the bench scale testing. Uncertainty over the performance of this composite postponed its intended MPP campaign. When the Yr. 1 – 5 campaign met its objectives ahead of schedule, an opportunity arose for an abbreviated (16 hour) run using the Yr. 6 – 10 composite to evaluate and attempt to demonstrate the flotation performance on a continuous basis.

The primary objective of this run was to achieve steady state in the metallurgy for the Yr. 6 – 10 composite. The flotation circuits stabilized very quickly after the removal of the excess frother from the process allowing for collection of a closed-circuit survey. A longer run would have permitted further optimization and the results of the abbreviated run are presented below.

Metallurgical Results

Overall, the bulk concentrate recovered 31.2% Ni and 33.2% Cu at grades of 9.44% Ni and 3.76% Cu. Concentrate grade was higher because of the lower pyrrhotite content in feed. The MgO grade in concentrate was 7.1%. Precious metal recovery to the bulk concentrate was 54.3% Au, 28.5% Pd, and 27.8% Pt at grades of 1.1 g/t Au, 6.8 g/t Pd, and 4.6 g/t Pt. The products only balance from the survey is presented in Table 13-17.

Table 13-17: Yr. 6 - 10 Closed Circuit – Products Only Balance

	Mass		Assays (%,ppm)									
	(g)	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt
Reference Head			0.31	0.12	0.42	0.016	9.4	0.50	29.4	0.022	0.266	0.178
Balanced Head	151.4	100.0	0.32	0.12	0.44	0.016	9.6	0.53	35.0	0.022	0.253	0.173
Call Factor			**104**	**103**	**104**	**95**	**102**	**105**	**119**	100	95	98
Mag Scav Tail	103.0	68.0	0.20	0.07	0.27	0.008	5.7	0.21	37.4	0.006	0.096	0.068
Mag Ro Tail	34.9	23.0	0.19	0.07	0.26	0.009	17.6	0.23	30.7	0.012	0.261	0.159
Clnr Scav Tail	12.0	7.9	0.51	0.16	0.67	0.029	15.9	1.11	31.0	0.044	0.695	0.533
Bulk Clnr Conc	**1.6**	**1.1**	**9.44**	**3.76**	**13.20**	**0.542**	**39.2**	**23.22**	**7.1**	**1.134**	**6.846**	**4.589**
	Mass		**Distribution (%)**									
	(g)	**(%)**	**Ni**	**Cu**	**Cu+Ni**	**Co**	**Fe**	**S**	**MgO**	**Au**	**Pd**	**Pt**
Balanced Head	151.4	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Mag Scav Tail	103.0	68.0	42.2	42.4	42.2	35.2	40.1	26.9	72.6	17.1	25.9	26.6
Mag Ro Tail	34.9	23.0	13.9	13.9	13.9	13.4	42.4	10.0	20.2	12.8	23.8	21.2
Clnr Scav Tail	12.0	7.9	12.7	10.6	12.1	14.8	13.2	16.8	7.0	15.8	21.8	24.4
Bulk Clnr Conc	**1.6**	**1.1**	**31.2**	**33.2**	**31.7**	**36.6**	**4.3**	**46.4**	**0.2**	**54.3**	**28.5**	**27.8**

Source: XPS 2018 MPP - P.33

The rougher concentrate showed grades similar to that of the Yr. 1 – 5 composite albeit at a much lower recovery. Detailed analysis of the internal streams indicated the rougher tail was higher than achieved in a comparable bench scale rougher test and may indicate an opportunity to improve recovery with further optimization.

Grade and recovery of the Mag Scavenger concentrate was comparable to that of Yr. 1 – 5. The mag rougher flotation stage recovery for all pay metals was under 50% compared to over 50% with the Yr. 1–5 composite. The recovered concentrate showed significantly higher grades.

The cleaner scavenger showed a similar response to the mag rougher - high upgrading ratio, but poor recovery. The excellent selectivity shown by both the Mag Scavenger and cleaner scavenger appeared to indicate the possibility of recovery improvement. It was uncertain whether the recovery improvements were limited by reagent dosage or mineral liberation.

The cleaners performed as expected in gangue rejection. Final bulk concentrate grades were higher because of less pyrrhotite content in the feed.

13.5 Cu/Ni Separation Test Work

All Cu/Ni separation test work was completed on the bulk concentrate produced from MPP processing of the Yr. 1-5 composite.

Cu separation from Ni is typically achieved by elevating the bulk concentrate pulp to a 12-pH using lime which depresses the Ni mineral pentlandite allowing the Cu concentrate to be floated off. The process is sometimes aided by the addition of sodium cyanide.

Separation test work performed as part of the optimization program utilized these conditions. It was found at the time that a 20-minute aeration of the pulp prior to Cu flotation, that bulk concentrate regrind and a small NaCN dose was required to achieve the required selectivity.

The MPP Yr. 1-5 sample produced a persistent froth during Cu-Ni separation and required several changes to the separation process. The cause of the persistent froth may be linked to the use of a stronger frother in the MPP or the impact of the magnetic rougher concentrate, which was finely ground to a d80 of 15 µm.

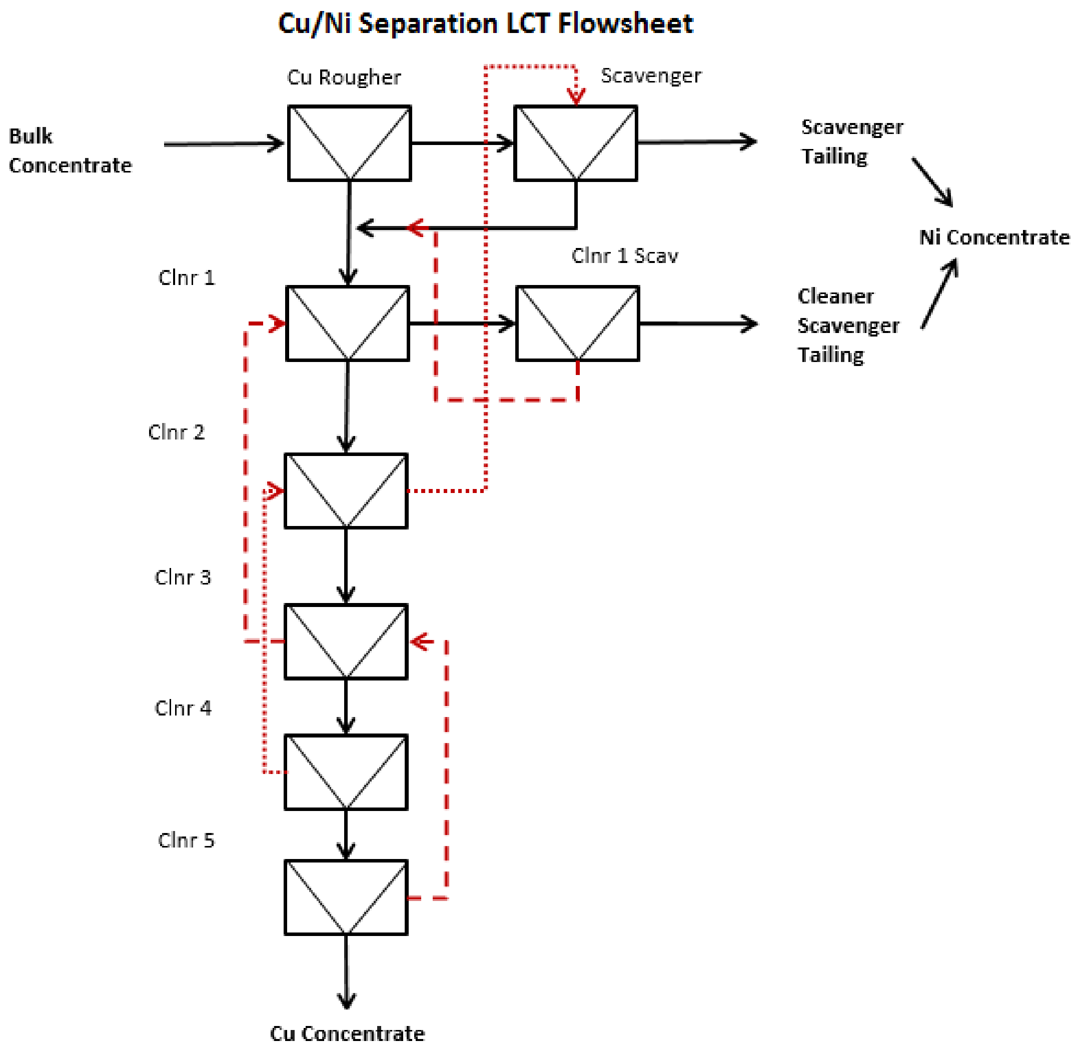
The additional changes incorporated into the separation flowsheet included a conditioning stage with activated carbon to remove excess frother, a more aggressive aeration stage with the addition of sodium metabisulfite (SMBS) and the use of CMC to disperse the fines during the cleaning stages. Once the conditions for Cu-Ni separation were established, an LCT to measure the effect of recirculating loads on the process was performed.

13.5.1 Cu/Ni Separation Locked Cycle Test

The LCT flowsheet is illustrated in Figure 13-17. The pre-test simulations for the LCT determined the number of conventional stages required would not reach steady state in the typical 6 cycles. The decision was made to include only 5 cleaning stages and conduct an extended test of 10 cycles over two days; as such, 5 cycles per day and the internal products at the end of Day 1 were frozen overnight and used the following day. The objective of the LCT was to determine closed-circuit split factors that could be used for the simulation of an industrial Cu-Ni separation process.

Lime was used to increase and maintain a pulp pH of 12 throughout the rougher and cleaner stages. Activated carbon was used in the initial conditioning stage to remove residual xanthate and frother from the bulk flotation circuit. SMBS was added to aid the aeration process and refresh mineral surfaces. Finnfix 150 was used to depress silicate fines and provide better froth drainage. Several smaller additions of Finnfix were applied in the cleaners as the recirculating loads reduced the fluidity of the froth. A small amount of MIBC was added in the later parts of the cleaner circuit as the pulp density decreased and the amount of fresh water added to top up the flotation cell increased.

Figure 13-17: Cu/Ni Separation LCT Flowsheet



Source: XPS 2018 MPP - P.46

Although there were some fluctuations from cycle to cycle, the test achieved an overall balanced state when averaging out the feed in and products out from cycles 4 to 10. All metal units in the product out were within 5% of the feed in when measured over the 7 cycles.

The formal balance for the LCT is presented in Table 13-18. The recalculated head assay from the 7-cycle average compares well against the Bulk Concentrate head assays from the MPP closed circuit balances.

The LCT produced a Cu concentrate at 13.8% Cu and 1.1% Ni grade with 53.2% Cu and 2.5% Ni recovery. Pt and Pd recoveries to Cu concentrate were below 10% at grades of over 1g/t.

31% of Au in bulk concentrate reported to the Cu concentrate at a grade of 1.2 g/t. The balance of the metals reported to Ni concentrate.

Cu concentrate was diluted mainly by pyrrhotite; evident with the grades of 43.7% Fe and 34.4% S. The pyrrhotite in the recirculating loads may be responsible for the visual thickening of the froth as the test progressed and this may have led to higher pyrrhotite recovery to the Cu concentrate. MgO grade in Cu concentrate was 1.6%.

Table 13-18: Cu/Ni Separation LCT Balance

Cycle 4 - 10	Mass		Assays (%,ppm)										
	(g)	(%)	Cu	Ni	Cu+Ni	Co	S	Fe	MgO	Pt	Pd	Au	3E
Bulk Con Calc Head Assay			**3.25**	**5.70**	**8.95**	**0.32**	**24.6**	**41.3**	**6.9**	**3.1**	**4.5**	**0.5**	**8.04**
Cyc 4-10 Calc Head Assay	3401.5	100.0	3.37	5.77	9.14	0.32	24.5	41.1	6.8	3.0	4.4	0.5	7.94
Call Factor			**104**	**101**	**102**	**101**	**100**	**100**	**100**	**98**	**99**	**100**	**99**
Cycle 4 - 10 Scav Tail	1921.3	56.5	1.22	7.67	8.89	0.43	20.9	38.6	8.7	3.2	4.6	0.3	8.13
Cycle 4 - 10 Clnr Scav Tail	1037.9	30.5	2.91	4.23	7.14	0.23	27.0	44.5	5.5	3.3	4.7	0.5	8.54
Cycle 4 - 10 Ni Conc	**2959.2**	**87.0**	**1.81**	**6.46**	**8.28**	**0.36**	**23.0**	**40.7**	**7.6**	**3.3**	**4.6**	**0.4**	**8.27**
Cycle 4 - 10 Cu Conc	**442.3**	**13.0**	**13.8**	**1.09**	**14.9**	**0.05**	**34.4**	**43.7**	**1.6**	**1.4**	**3.2**	**1.2**	**5.72**
Cycle 4 - 10	**Mass**		**Distribution (%)**										
	(g)	**(%)**	**Cu**	**Ni**	**Cu+Ni**	**Co**	**S**	**Fe**	**MgO**	**Pt**	**Pd**	**Au**	**3E**
Product Out Cycle 4 -10	3401.5	100.0	146.8	197.5	178.8	198.1	181.8	186.2	196.9	194.2	190.7	168.6	190.6
Cycle 4 - 10 Scav Tail	1921.3	56.5	20.4	75.2	55.0	76.5	48.2	53.1	72.2	60.5	58.5	36.2	57.8
Cycle 4 - 10 Clnr Scav Tail	1037.9	30.5	26.4	22.4	23.8	21.6	33.6	33.1	24.7	33.7	32.2	32.3	32.8
Cycle 4 - 10 Ni Conc	**2959.2**	**87.0**	**46.8**	**97.5**	**78.8**	**98.1**	**81.8**	**86.2**	**96.9**	**94.2**	**90.7**	**68.6**	**90.6**
Cycle 4 - 10 Cu Conc	**442.3**	**13.0**	**53.2**	**2.5**	**21.2**	**1.9**	**18.2**	**13.8**	**3.1**	**5.8**	**9.3**	**31.4**	**9.4**

Source: XPS 2018 MPP - P.48

13.5.2 Cu/Ni Mineral Based Simulation

Assays of the recirculating streams from the 10th cycle were used to calculate the secondary split factors in order to build a predictive model for the varying bulk concentrate grades. The secondary split factors are presented in Table 13-19. These secondary split factors also allow for the simulation of the Cu concentrate grade and recovery for changing feed grades and alternative configurations of the separation circuit.

Table 13-19: Secondary Split Factors Captured on Cycle 10

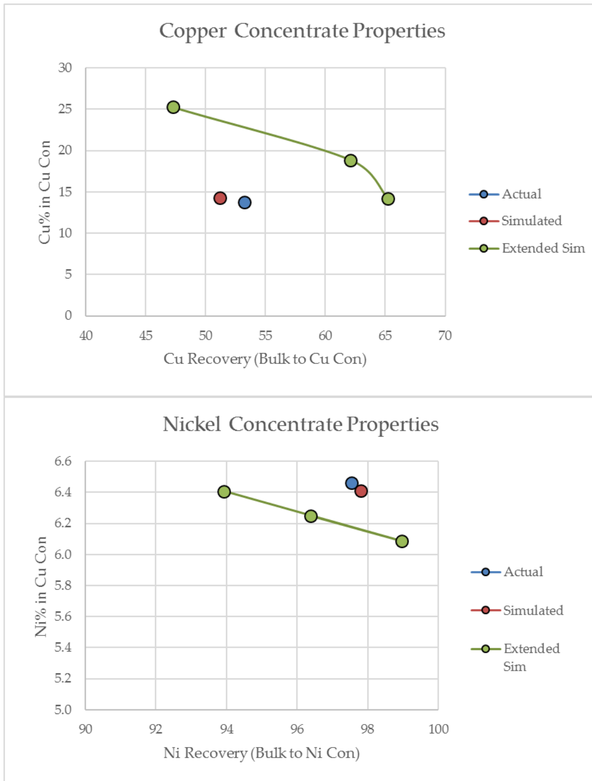
Stage	Mass	Cu	Ni	Co	S	Fe	MgO	Pt	Pd	Au	3E	Cp	Pn	Po	NSG
Rougher	53	83	32	30	62	58	35	50	52	72	53	83	30	66	36
Cleaner 1	45	68	29	27	50	46	31	40	45	58	44	68	27	49	30
C1 Scav	50	43	50	50	50	50	48	51	49	48	50	43	50	51	50
Cleaner 2	63	79	53	51	65	63	54	58	64	73	63	79	52	62	54
Cleaner 3	62	76	53	51	63	61	56	50	52	67	53	76	52	60	54
Cleaner 4	68	83	59	58	69	66	57	57	63	72	63	83	58	64	57
Cleaner 5	74	88	62	61	75	72	63	60	67	75	67	88	60	68	61

Source: XPS 2018 MPP - P.48

A simulation utilizing the mineral split factors was constructed using Excel. As a test of the simulation, the original data from the LCT was entered into the simulation. The result is shown as the red dot in Figure 13-18 whereas the actual LCT result is shown as the blue dot. The offset between the two dots is the result of the merged split factor for rougher and scavenger. As a result, the simulation is expected to produce a conservative estimate.

A simulation was performed of extended cleaners to 10 stage via mechanical cells and added an additional increment of scavenger concentrate to emulate a more aggressive pull in the rougher/scavengers. The simulation and their grade recovery curves are shown in Figure 13-18.

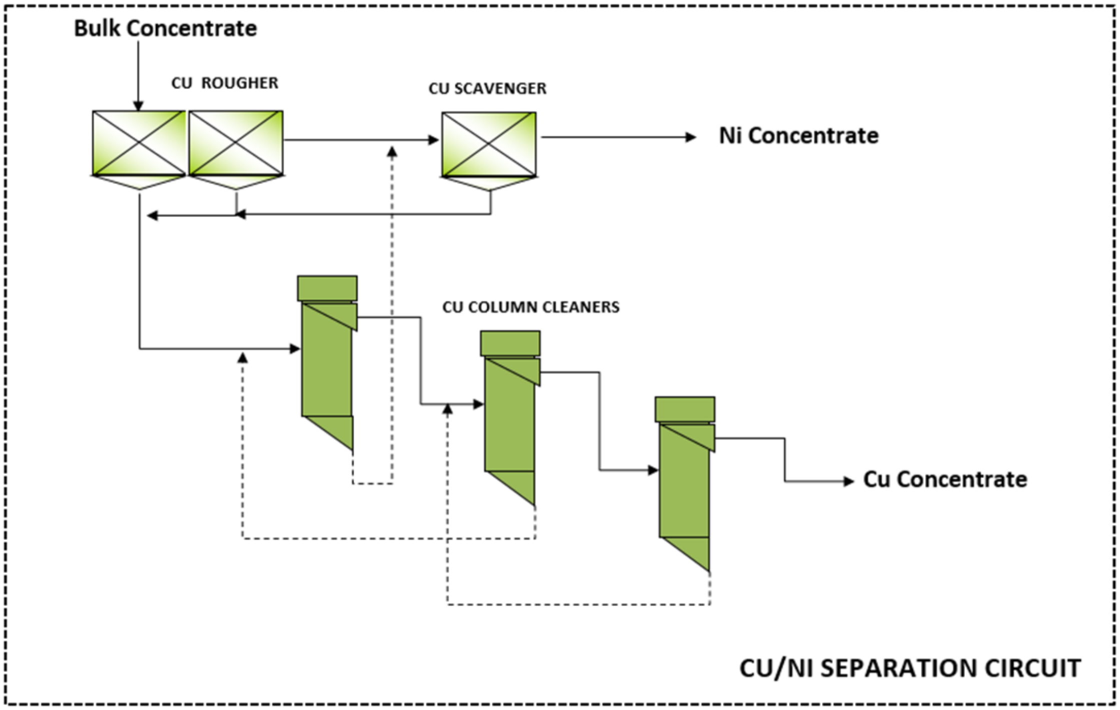
Figure 13-18: Simulated Cu and Ni Concentrate Grade Recovery



Source: XPS 2018 MPP - P.49, 50

Simulated Cu and Ni concentrate grade recovery. Ten cleaner stages via mechanical cells are approximately equivalent to three industrial columns in series. Figure 13-19 illustrates the Cu/Ni separation circuit layout as intended by the extended simulation. By extending the cleaner stages, the expectation is an improvement of Cu recovery to around 65% at the 14% grade, or an improvement of grade to 18% Cu at 62% Cu recovery. The Ni concentrate will contain 98% of the Ni however, the Ni concentrate grades are expected to see a drop as the pyrrhotite units are rejected from the Cu concentrate into Ni concentrate.

Figure 13-19: Cu/Ni Separation Circuit Flowsheet as per Simulation



Source: XPS 2018 MPP - P.49

The overall split of bulk concentrate into Cu concentrate and Ni concentrate is dependent on the ratio of Cu, Ni, and pyrrhotite in the bulk concentrate. Based on this simulation, the bulk concentrate produced from composite Yr. 1-5 would separate into Cu and Ni concentrate according to the results presented in Table 13-20. Based on the MPP close balance of composite Yr. 1-5 an overall balance incorporating Cu-Ni Separation is presented in Table 13-21.

Table 13-20: Simulated Final Concentrates using the MPP Year 1 – 5 Bulk Concentrate

	Mass	Assays (%,ppm)										Mineral Concentration (%)			
	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt	Cp	Pn	Po	NSG
Bulk Concentrate		6.08	3.06		0.330	43.9	25.59	5.6	0.525	4.853	3.643	8.81	16.02	43.38	31.80
Cu Concentrate	10.5	1.10	18.02		0.060	42.9	35.34	0.7	1.578	2.820	0.734	51.92	2.45	41.59	4.04
Ni Concentrate	89.5	6.66	1.29		0.362	44.0	24.45	6.1	0.400	5.093	3.986	3.73	17.61	43.59	35.06
	Mass	Distribution (%)										Mineral Distribution (%)			
	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt	Cp	Pn	Po	NSG
Bulk Concentrate		100.0	100.0		100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Cu Concentrate	10.5	1.9	62.1		1.9	10.3	14.5	1.3	31.7	6.1	2.1	62.1	1.6	10.1	1.3
Ni Concentrate	89.5	98.1	37.9		98.1	89.7	85.5	98.7	68.3	93.9	97.9	37.9	98.4	89.9	98.7

Source: XPS 2018 MPP - P.51

Table 13-21: Overall MPP Balance Yr 1-5 – Ore Feed to Separated Cu and Ni Concentrate

	Mass	Assays (%,ppm)										Mineral Concentration (%)			
	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt	Cp	Pn	Po	NSG
Ore Feed	**100**	**0.35**	**0.16**	**0.51**	**0.018**	**10.2**	**1.06**	**31.3**	**0.022**	**0.280**	**0.237**	**0.46**	**0.95**	**1.53**	**97.06**
Mag Scav Tail	57.1	0.15	0.06	0.21	0.007	5.0	0.15	33.7	0.003	0.055	0.083	0.17	0.40	0.00	99.43
Mag Ro Tail	23.9	0.16	0.06	0.22	0.007	15.6	0.20	29.9	0.003	0.190	0.122	0.18	0.44	0.00	99.38
Clnr Scav Tail	15.9	0.27	0.10	0.37	0.013	13.8	0.79	29.9	0.020	0.330	0.295	0.29	0.73	1.13	97.86
Bulk Concentrate	**3.1**	**6.08**	**3.06**	**9.13**	**0.330**	**43.9**	**25.59**	**5.6**	**0.525**	**4.853**	**3.643**	**8.81**	**16.02**	**43.38**	**31.80**
Cu Concentrate	***0.3***	***1.10***	***18.02***	***19.12***	***0.060***	***42.9***	***35.34***	***0.7***	***1.578***	***2.820***	***0.734***	***51.92***	***2.45***	***41.59***	***4.04***
Ni Concentrate	***2.8***	***6.66***	***1.29***	***7.96***	***0.362***	***44.0***	***24.45***	***6.1***	***0.400***	***5.093***	***3.986***	***3.73***	***17.61***	***43.59***	***35.06***
	Mass	Distribution (%)										Mineral Distribution (%)			
	(%)	Ni	Cu	Cu+Ni	Co	Fe	S	MgO	Au	Pd	Pt	Cp	Pn	Po	NSG
Ore Feed	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
Mag Scav Tail	57.1	23.7	20.9	22.8	22.2	28.2	8.1	61.5	8.0	11.2	20.0	20.9	24.3	0.0	58.5
Mag Ro Tail	23.9	10.8	9.6	10.4	9.3	36.7	4.5	22.8	3.0	16.2	12.3	9.6	11.0	0.0	24.5
Clnr Scav Tail	15.9	12.2	9.9	11.5	11.5	21.7	11.9	15.2	14.6	18.7	19.8	9.9	12.2	11.7	16.0
Bulk Concentrate	**3.1**	**53.3**	**59.6**	**55.3**	**57.0**	**13.5**	**75.5**	**0.6**	**74.4**	**53.9**	**47.9**	**59.6**	**52.5**	**88.3**	**1.0**
Cu Concentrate	***0.3***	***1.0***	***37.0***	***12.2***	***1.1***	***1.4***	***11.0***	***0.0***	***23.6***	***3.3***	***1.0***	***37.0***	***0.8***	***8.9***	***0.0***
Ni Concentrate	***2.8***	***52.3***	***22.6***	***43.1***	***56.0***	***12.1***	***64.5***	***0.5***	***50.8***	***50.6***	***46.9***	***22.6***	***51.7***	***79.4***	***1.0***

Source: XPS 2018 MPP - P.51

13.6 Application of Metallurgical Results to the Resource

In summary, the Ni and Cu recovery models were developed from variability test results and then validated against historic test results. These models calculated the total recovery to bulk concentrate. Co and precious metal recoveries to bulk concentrate provided were based on the stable results of the pilot plant composite Yr. 1-5. The recovery of metals to separate Cu and Ni concentrates from the total bulk recovery was determined by the mineral based simulated Cu-Ni separation model developed by XPS.

The initial application of modelled Cu-Ni separation results to the block model identified the Cu distribution in the resource was not constant. In many of the blocks, low Cu heads resulted in better economics with a bulk concentrate. In the blocks with high Cu heads, production of separate Cu and Ni concentrates provided superior economics.

In reviewing the areas of the resource which contained either the low or the high Cu, it was determined that an opportunity existed to separately process these materials. In processing these materials separately, the conditions could be selected to optimize concentrate grade and hence overall Project economics.

The low Cu material, containing an average Cu/Ni ratio of only 0.37, also contained low (0.52%) S values and was most similar to the Phase 2 composites tested at XPS. These composites responded well with the low pH circuit. The low pH circuit has the benefit of reducing MgO content while increasing pyrrhotite recovery. Based on the two MPP results a weighted average concentrate grade of Cu+Ni of 10.4% was calculated with an MgO content of 5.6%. These values were used to determine bulk concentrate production.

The remaining high Cu material, containing an average Cu/Ni ratio of 1.12, also contained higher (1.32%) S values and was more comparable to samples tested prior to Phase 2. There are numerous tests performed on higher sulphur samples however the closest match was the 203 lower peridotite sample tested by XPS in 2014 which contained 1.24% S. With the higher sulphur values, the CMC circuit which demonstrated superior pyrrhotite rejection albeit at a higher MgO content, was selected as the preferred processing route. To determine the concentrate tonnage produced from the higher Cu material the results from the 2014 study with a grade of Cu+Ni of 14.2% and an MgO content of 9.8% was used.

The bulk concentrate produced from the higher Cu material was entered in the simulated Cu-Ni separation circuit designed by XPS. With the higher Cu grade in the feed, the simulation projected a higher grade of 25.6% Cu although, Cu recovery to Cu concentrate was unchanged at 62.3% of the Cu reporting to bulk concentrate.

A summary of the metal recoveries, for metallurgical calculation purposes, is presented in Table 13-22. For resource calculation purposes, a constant concentrate grade was used.

Table 13-22: Summary Table for Metallurgical Calculations

Metal Recovery	
Ni Recovery based on variability testing	
Where Mg%	<22,
Total Ni Recovery	= (1-(-0.243*LN((HdS%-HdCu%)/HdNi%)+0.6747))*100;
Where Mg%	>22
Total Ni Recovery	= (1-(-0.3758*((HdS%-HdCu%)/HdNi%)+0.9672))*100;
Cu Recovery based on variability testing	
Where CuHd%	>0.06%
Total Cu Recovery	=(HdCu%-0.06)/HdCu%*100;
Co, Au, Pt, Pd Recoveries fixed based on MPP results	
Total Co Recovery Total Au Recovery Total Pt Recovery Total Pd Recovery	= 57% = 74.4% = 47.8% = 54%
Concentrates Production	
Low Cu/S Resource Blocks Produce Bulk Concentrate	
Bulk concentrate grade based on MPP results	
Bulk Cu+Ni Grade	=10.4%
High Cu/S Resource Blocks Produce Cu and Ni Concentrates from Bulk	
Intermediate Bulk concentrate grade based on XPS 2014 Program	
Bulk Cu+Ni Grade	=14.2%
Split Cu and Ni Concentrates based on XPS Simulation and LCT results	
Cu Concentrate split from High Cu/S Bulk Concentrate	
Cu recovery to Cu conc Grade of Cu Conc Pd recovery to Cu conc Au recovery to Cu conc	= 62% of Cu in high Cu/S bulk concentrate = 25.6% Cu, 1.1% Ni = 5.9% = 31.7%
Ni Concentrate split from High Cu/S Bulk Concentrate	
Ni recovery to Ni conc Cu recovery to Cu conc Co recovery to Ni conc Au recovery to Ni conc Pt Recovery to Ni conc Pd Recovery to Ni conc	= based on 1.1% Ni in Cu Conc (96.2% of Ni in bulk conc) = remaining 38% of Cu in bulk concentrate = 98.1% of high Cu/S bulk concentrate = 68.3% of high Cu/S bulk concentrate = 98.1% of high Cu/S bulk concentrate = 94.1% of high Cu/S bulk concentrate

13.7 Future Work

With the improved understanding of the impact of Cu and S values on the metallurgy, combined with the distribution of the higher Cu and S within the resource, separate testing of these two geomet units is recommended. Whereas much of the testing performed over the years was done on the higher sulphur material there was little work done on samples less than 0.8% S. At such time the Company proceeds to a PFS, a further metallurgical testwork program to include additional variability test work on low S samples is recommended to increase confidence in the results.

14 MINERAL RESOURCE ESTIMATES

The mineral resource for the Project was developed using a computer based block model of the deposit. The block model was assembled based on the drill hole data base and interpreted geology by Nickel Creek Platinum's Chief Mine Geologist James Berry after review and verification of that information by IMC. Mineral resources were estimated using the block model and the Lerchs-Grossman open pit software to establish the component of the deposit with reasonable prospects of economic extraction. John Marek, of IMC acted as the qualified person for the development of the block model and the estimation of mineral resources.

The final statement of mineral resources is presented at the end of this section and reflects material that is inside of a computer-generated pit. The Lerchs-Grossman pit algorithm was used to provide some assurance that the mineral resource has "reasonable prospects of economic extraction" as required by CIM best practices. The economic assumptions that were used for that pit are also summarized later in text.

14.1 Model Location

The Project block model was assembled using the project coordinate system of: UTM North American Datum 1983, Zone 7. The model blocks are 10 x 10 x 10 m cubes.

Table 14-1 below summarizes the size and location of the block model.

Table 14-1: Nickel Shäw Model Size and Location

Outside Edges of the Model				
	Coordinates (metres)		Number of Blocks	Block Size
East	576,325	579,505	318	10 m
North	6,814,500	6,816,800	230	10 m
Elevation	600	1,960	136	10 m

Future work should consider alternative block sizes once the process plant production rate is well established.

14.2 Data Base

Section 12, regarding data verification, has indicated the opinion of the qualified person that the historic assay information prior to 1987 should not be used for the estimation of mineral resources. As a result, Table 14-2 summarizes the amount of drilling and raw assay information within the block model volume that was used to estimate this statement of mineral resources.

As reported in Sections 10 and 11, there were 16 additional holes drilled in late 2017. The impact of those holes on the 2017 resource model is measured and reported later in this section after the model description of the model methods. That impact is sufficiently small that it is not material to the statement of mineral resources.

Table 14-2: Assay Information Used to Develop the Block Model

1987 and Newer Drilling Used in the Model	
386	Drill holes
24,341	Sample Intervals
62,799	Meters of Drilling
Number of Assays Used for Modeling	
23,732	Ni
23,732	Cu
23,635	Co
23,730	Pt
23,730	Pd
23,650	Au
20,622	Ag
19,410	Mg (by ICP)
19,266	Sulfur (by ICP)

14.2.1 Bench Height for Compositing

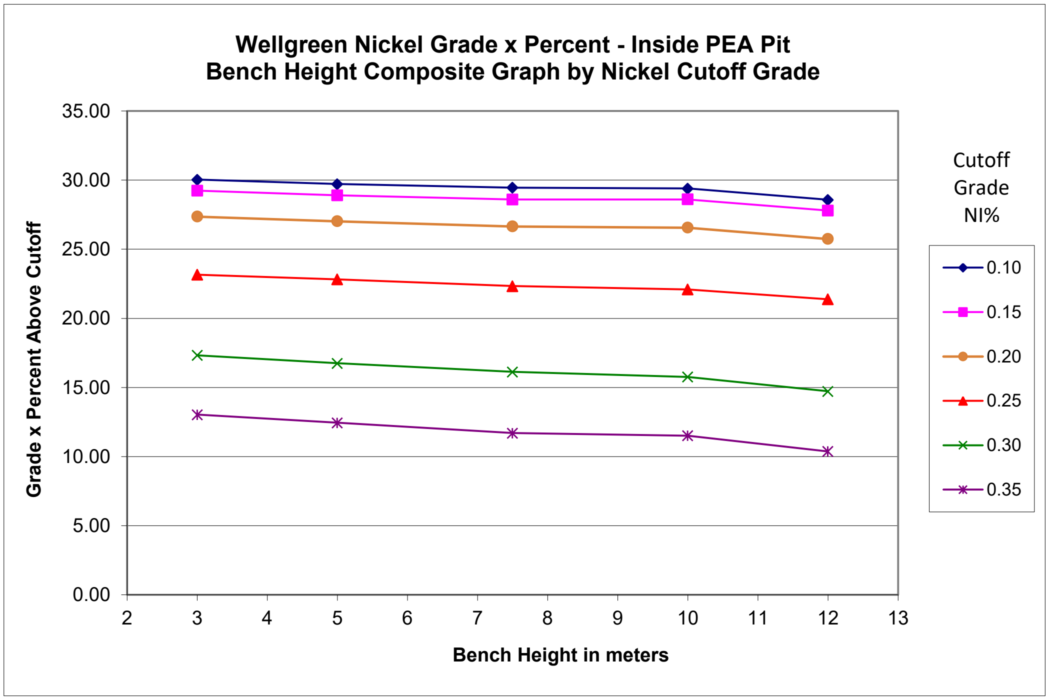
A bench height analysis was completed to measure the potential change in metal production due to alternative mining bench heights. As higher bench heights and larger blocks are utilized, additional dilution would be incorporated into the feed to the process plant.

As a preliminary measure for model assembly, IMC completed the following test:

- drill hole data was composited at alternative sample lengths of: 3,5,7.5, 10, and 12 m
- for each bench height (composite length), the mean grade for nickel and the number of composites above alternative cutoffs were tabulated. Cutoffs were: 0.10, 0.15, 20, 0.25, 0.30, and 0.35% Ni
- the product of mean grade x composite count is used as a relative measure of contained metal
- all the composites were contained within an earlier resource pit to establish a consistent total volume for comparison

Figure 14-1 summarizes the results. There is little change in contained metal between 7.5 m and 10 m bench heights. A bench height (composite length) of 12 m begins to show losses of metal in the range of 3% in the cutoff range of 0.10% to 0.25% nickel. As a result, the 10 m bench height and composite length was selected.

Figure 14-1: Composite Length and Bench Height Analysis for Contained Metal



14.3 Geology and Data Populations

The geologic interpretation was discussed in Sections 7 and 8. The resulting interpretation includes the rock types detailed in Table 14-3 below.

Table 14-3: Interpreted Rock Types

Model Code	Lithologic Description	Mineralization Status
7	Clinopyroxenite	Ore Host
20	Mineralized Gabbro	Ore Host
24	Peridotite	Ore Host
29	Massive Sulfide	Ore Host
26	Metasedimentary Rocks	Generally Barren
5	Basalt	Barren
21	Maple Creek Gabbro	Barren
32	Volcanoclasitic	Barren

As noted in table 14.3, clinopyroxenite, mineralized gabbro, peridotite, and massive sulfide are rock types that can act as hosts to mineralization. Metasedimentary rocks can host sulfide mineralization when in close contact with the intrusive units of clinopyroxenite, mineralized gabbro, and peridotite. The extent of ore into the sedimentary rocks is minor, but it can be locally high grade.

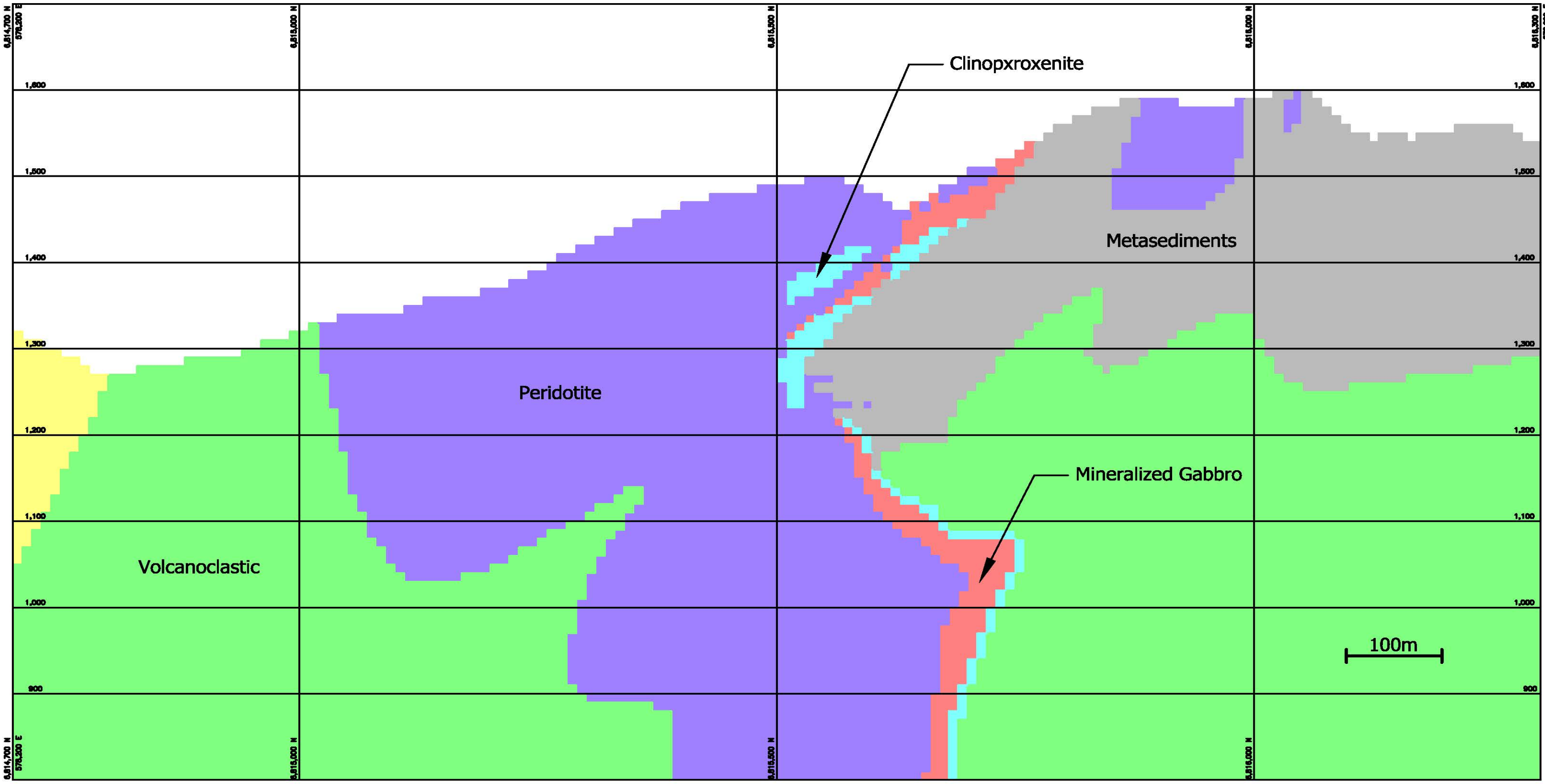
The recoverable metal in the deposit is associated with sulfide mineralization. The sulfides are associated with the contact alteration at the intrusive – sediment contact. The mineralized gabbro, and clinopyroxenite units are generally near to the sediment contact and they are consistently mineralized. Massive sulfide material is also typically close to the contact and is consistently high grade.

Peridotite is well mineralized near the contact and becomes lower grade as one migrates to the south, away from the sediment contact. The southernmost distal peridotite mineralization reflects the nickel contained in olivine/serpentine and is not expected to be recovered.

Peridotite near the contact contains more sulfides and is consequently higher grade and more recoverable.

Figure 14-2 is a north-south cross section through the deposit looking west. The mineral bearing rock types, as well as the barren sediments and volcanoclastics, are shown on the section.

Figure 14-2: North-South Section 578,200 E, Looking West



Source: IMC, 2018)

14.4 Statistical Evaluation

Basic statistics and variography were completed to establish the best method for block grade estimation. All statistics and analysis presented in this section apply to the drilling and re-assay completed from 1987 through 2016. No further reference will address the historic drilling completed before 1987.

Sulfur and magnesium have been estimated because they were used to establish process recovery estimates for the resource. There is no commercial benefit from magnesium or sulfur, so they are not included in the final statement of mineral resources.

14.4.1 Grade Capping

Cumulative frequency plots were completed for each metal, in each of the potential ore host rock types. High grade outliers were determined by studying those plots. A cap level was established for each population so that any assay that was above that value was replaced with the cap value prior to further statistical analysis or used in the block model. Table 14-4 summarizes the cap values applied to the individual assays.

Table 14-4: Cap Values Applied to Assay Intervals

Rock Type	Model	Metal or Element to be Estimated							
	Code	Ni%	Cu%	Co%	Pt gm/t	Pd gm/t	Au gm/t	S%	Mg%
Clinopyroxenite	7	3.00	3.00	0.150	4.00	3.50	1.50	10.0	No Cap
Mineralized Gabbro	20	1.50	3.00	0.150	4.50	3.50	1.50	10.0	No Cap
Peridotite	24	3.50	3.20	0.150	4.50	3.00	1.00	10.0	No Cap
Massive Sulfide	29	3.20	2.70	0.150	2.00	2.50	1.50	9.0	No Cap
Metasedimentary Rocks	26	4.00	2.50	0.150	2.50	2.70	0.80	10.0	No Cap
Volcanoclasitic	32	1.20	1.00	0.060	1.90	1.50	0.40	4.0	No Cap

There were three Mg assays that were in error in the data base with values above 100% Mg. They were set to a code for no assay rather than being capped. Mg assays completed during 2009 were found to be significantly low biased compared to all other years of Mg assay. The 2009 drilling was consequently not used for estimation of Mg.

14.4.2 Compositing

The rational for a 10m target composite length was reported on previous pages. The procedure for compositing respected the rock type boundaries and resulted in composites that vary in length around the 10m target.

The procedure is as follows:

- the length of each of the rock types is established within each drill hole
- that length is divided by 10 m and the resulting number of composites rounded to an integral number
- the integral number of composites defines a new composite length within each rock type intercept within each drill hole
- the down hole composites are calculated at the target length that respects the rock type boundaries

The population statistics for rock types, that are the primary hosts for mineralization, are summarized in Table 14-5.

Table 14-5: Basic Statistics of 10m Composites

Rock Type	Statistic	Statistics of Nominal 10m Down Hole Composites							
		Ni%	Cu%	Co%	Pt gm/t	Pd gm/t	Au gm/t	S%	Mg%
Clinopyroxenite	Number	830	830	823	830	830	823	675	684
	Mean	0.235	0.254	0.016	0.357	0.264	0.090	1.10	13.54
	Std Dev	0.128	0.219	0.001	0.275	0.177	0.092	0.86	4.00
Mineralized Gabbro	Number	585	585	576	585	585	576	462	468
	Mean	0.212	0.286	0.014	0.339	0.235	0.073	1.31	7.70
	Std Dev	0.220	0.286	0.012	0.379	0.253	0.090	1.03	4.13
Peridotite	Number	3,094	3,094	3,073	3,094	3,094	3,077	2,610	2,613
	Mean	0.253	0.124	0.015	0.209	0.224	0.040	0.61	17.98
	Std Dev	0.108	0.139	0.005	0.200	0.133	0.057	0.72	4.77
Massive Sulfide	Number	21	21	21	21	21	21	14	14
	Mean	1.148	1.054	0.056	0.870	0.772	0.121	4.780	4.34
	Std Dev	0.649	0.735	0.029	0.577	0.587	0.088	2.83	5.67

14.4.3 Domain Boundaries

Table 14-5 indicates that the massive sulfide is a different high grade population from the rest of the rock types. To determine the proper treatment of rock type or other boundaries, a statistical evaluation was completed. Sometimes called "boundary analysis", the procedure pairs composites from opposite sides of rock type borders and compares their statistical properties to understand if they are similar or different populations.

Table 14-6 summarizes the results of the boundary analysis. The term "hard" means that block estimation will only use composites where the rock type matches the composite rock type. "Soft" boundaries allow for composites from either side of the rock type boundary to be used for grade estimation.

In all cases, the massive sulfide rock type (Code 29) appears to be a separate population from all others and is treated as a hard boundary.

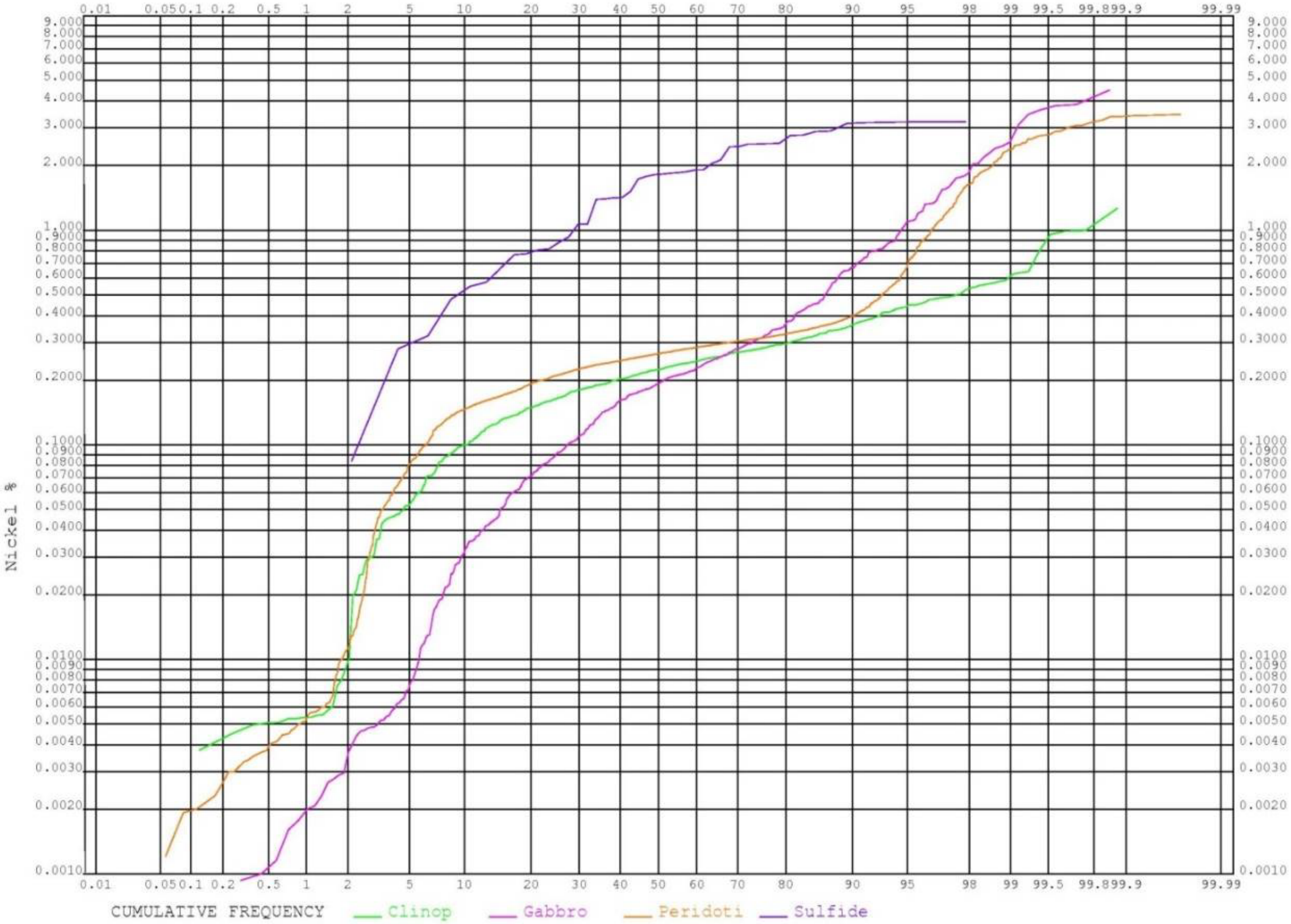
Table 14-6: Summary Results of Boundary Analysis

Rock Type	Paired With	Summary of Boundary Analysis							
		Ni%	Cu%	Co%	Pt gm/t	Pd gm/t	Au gm/t	S%	Mg%
Clino	Gabbro	soft	soft	hard	soft	soft	soft	soft	hard
Clino	Peridotite	soft	soft	soft	soft	soft	hard	soft	hard
Clino	Mass Sulf	hard	hard	hard	hard	hard	hard	hard	hard
Gabbro	Peridotite	soft	hard	soft	soft	soft	hard	hard	hard
Gabbro	Mass Sulf	hard	hard	hard	hard	hard	hard	hard	hard
Peridotite	Mass Sulf	hard	hard	hard	hard	hard	hard	hard	hard

Table 14-6 indicates that for nickel, the only boundary should be massive sulfide and all other rock types should be allowed to be treated as soft boundaries. This prompted more investigation of the grade distribution for nickel.

Figure 14-3 is a cumulative frequency plot showing the distribution of the nickel composites for each of the four mineralized host rock types. The clinopyroxenite, peridotite, and gabbro all show a population change at about 0.35% Ni.

Figure 14-3: Cumulative Frequency Plot for 10 m Nickel Composites



A review of cross sections also indicates that each of the rock types contain a component of +0.35% Ni with limited areal extent. Those high grade zones often appear to connect across the rock type boundaries. This outcome prompted a different approach for nickel estimation compared with the other rock types.

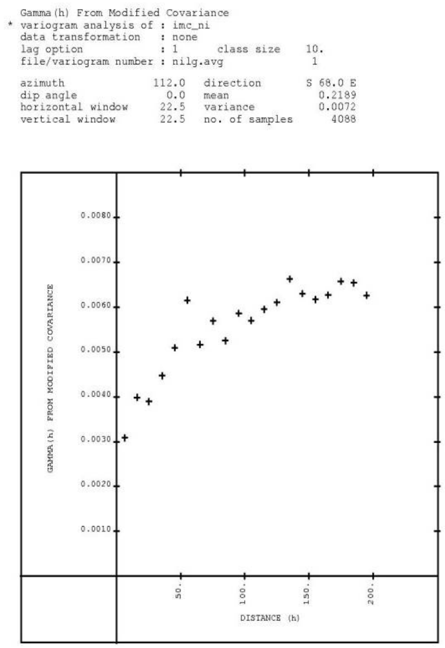
A separate indicator estimation was completed for nickel to establish the volume of the 0.35% population, independent of rock type boundaries. Once the 0.35% volume was defined, the grade inside was estimated with that boundary and treated as a "hard" bound.

14.4.4 Variography

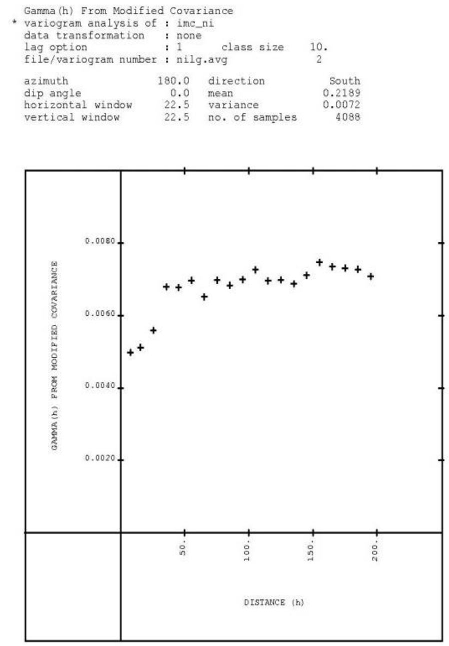
Variograms were developed for each metal in each rock type as a guide to the search radius for block grade estimation. In addition to the variograms on grade, a series of 0.35% Ni indicator variograms were run for nickel to set the parameters for the indicator estimate of that grade range.

Figure 14-4 and Figure 14-5 are examples of the variograms obtained for nickel. The summarized results from the variograms are shown in the tables defining the estimation methods in the next sub-section.

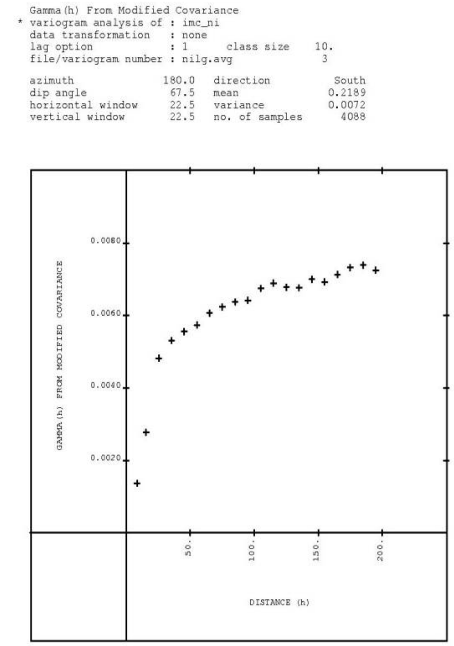
Figure 14-4: Example Indicator Variograms for Nickel Indicator at 0.35% Ni



Strike

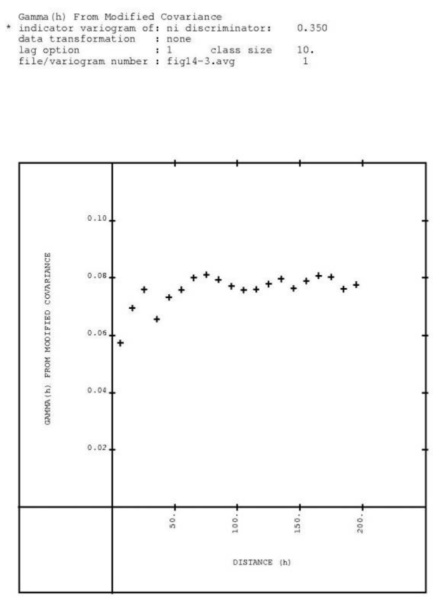


Down Dip

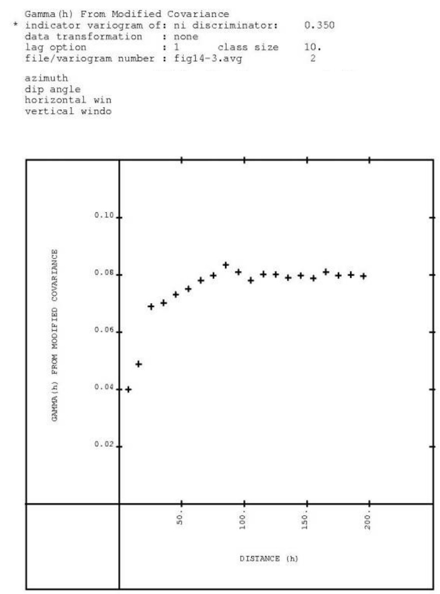


Perpendicular

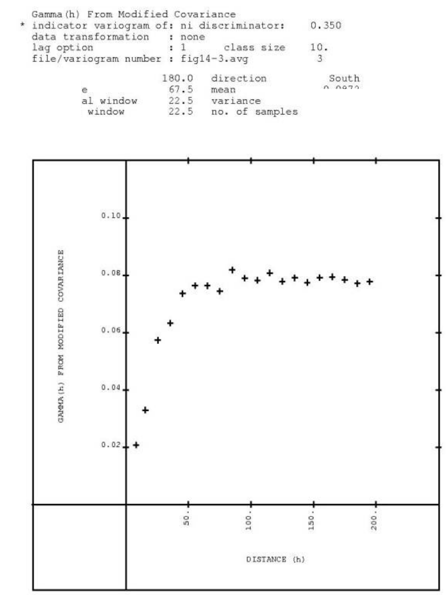
Figure 14-5: Example Grade Variograms for Nickel Less than 0.35% Ni



Strike



Down Dip



Perpendicular

14.5 Block Model Assembly Procedures

The Nickel Shäw block model was assembled with conventional methodology. Block grades were assigned using Inverse Distance squared (1/D2) for all economic metals. Mg was assigned by 1/D3. Nickel used an indicator boundary rather than a rock type boundary. All other metals utilized some combination of rock type boundaries for estimation.

The indicator process for nickel did allow some mineralization to be estimated in the metasedimentary rocks and volcanoclastics. However, the amount of that material was specifically limited to be only one block width into either rock type. The mineralization on the sediment or volcanoclastic contact is of minor tonnage and does not penetrate those units any substantial distance. The mineralized blocks in sediment or volcanoclastics combined, amount to less than 0.33% of the total number of block grades assigned in the model.

Block grades of economic metals were assigned by 1/D2 methods. Tests were completed comparing ordinary kriging with the 1/D2 method which indicated that ordinary kriging would tend to smooth the grade distribution more than 1/D2.

The grades of the economic metals are generally highest near the intrusive – sediment contact. As one moves southward, the metal grades begin to reduce. It is important that the relatively high grade values near the contact not be smeared to the south as that would overestimate mineable tonnage above cut-off. As a result, the estimation methods maintained relatively tight search radii perpendicular to the contact and selected the 1/D2 method so that the block grades would look like the local composite grades.

The economic mineralization is generally conformal to the intrusive-sediment contact. The higher grade values tend to be proximal to the contact. To model that occurrence, a series of sub-domains were established that allow the search orientations to be parallel and perpendicular to the intrusive-sediment contact. The domains were selected on plan and section to reflect the changes in strike and dip of the intrusive sediment contact. There were 19 sub-domains established to reflect the variability in the contact. Those domains were not hard boundaries for grade estimation but reflected a local change in search orientation.

Magnesium does not follow the same trend, because the magnesium grade is indicative of the host rock chemistry rather than the sulfide mineralization. The rock type boundaries were particularly important for Mg, but the multiple search orientations were not applied. Mg will be used to establish the metallurgical response based on a Mg grade boundary. To best define that boundary, the 1/D3 method was utilized to limit the areal smearing of high grade values. The low biased 2009 drilling was removed so the volume of low value Mg would not be overestimated for the process recovery determination.

Table 14-8 illustrates the sediment contact orientation in each domain. Domain boundaries are established by coordinate and elevation limits across the deposit. In some areas, the dip orientation changes three times with increasing depth. Detailed illustration of the domain boundaries would take several level maps and sections to present, but the codes are available in the model.

Table 14-9 summarizes the estimation parameters by metal, indicator, and rock type. References to strike and dip reference the orientations in Table 14-8.

The following rock types were estimated: Clinopyroxenite, Mineralized Gabbro, Peridotite, Massive Sulfide, and the few boundary blocks of sediments and volcanoclastics.

The results of the Mg estimation indicated that there were a number of blocks that had Ni grades but did not have Mg grades. This is due to the fact that three are not as many Mg assays and that the 2009 drilling was removed from estimation. In order to provide information for process evaluation, the average Mg grade of each rock type was assigned to those blocks that were not estimated by the 1/D3 process. The mean values of Mg assigned where estimation did not cover are summarized below on Table 14-7.

Table 14-7: Mg Default values if Not Estimated and Ni is Present

Mg Default Values		
Rock Type Code	**Rock Type**	**Assigned Average Mg%**
7	Clinopyroxenite	13.76
20	Min - Gabbro	8.01
21	Maple Crk Gabbro	4.65
24	Peridotite	18.25
26	Metasedimentary Rocks	3.73
29	Massive Sulfide	2.55
32	Volcanics	4.78

Table 14-8: Search Orientation by Domain for Economic Metals

Search Domain Orientations		
Domain	**Strike Degrees**	**Dip Degrees**
1	90	45 S
2	90	90 S
3	90	90 S
4	90	60 S
5	120	65 SW
6	120	90 SW
7	120	45 SW
8	120	65SW
9	300	60 NE
10	120	60 SW
11	120	45 SW
12	300	60 NE
13	120	75 SW
14	120	45 SW
15	300	45 NE
16	120	75 SW
17	120	50 SW
18	120	75 SW
19	120	50 SW

Table 14-9: Block Grade Estimation Parameters

Orientation Domains	Search Distance Meters Strike	Down Dip	Perpendicular	Estimation Method	High Grade Limit	Search Limit on HG meters
Nickel Indicator at 0.35% Ni Discriminator						
3 and 4	75	120	15	1/D2		
1, 2 and 5 to 19	75	90	15	1/D2		
Nickel Grade inside the Indicator Zone						
3 and 4	75	120	15	1/D2	3.00%	25
1, 2 and 5 to 19	75	90	15	1/D2	3.00%	25
Nickel Grade Outside the Indicator Zone						
3 and 4	75	120	75	1/D2	0.40%	25
1, 2 and 5 to 19	75	90	75	1/D2	0.40%	25
Copper Grade						
3 and 4	75	120	75	1/D2	1.00%	25
1, 2 and 5 to 19	75	90	75	1/D2	1.00%	25
Cobalt Grade						
3 and 4	75	120	75	1/D2	0.05%	25
1, 2 and 5 to 19	75	90	75	1/D2	0.05%	25
Platinum Grade						
3 and 4	75	120	75	1/D2	1.00 gm/t	25
1, 2 and 5 to 19	75	90	75	1/D2	1.00 gm/t	25
Palladium Grade						
3 and 4	75	120	75	1/D2	0.75 gm/t	25
1, 2 and 5 to 19	75	90	75	1/D2	0.75 gm/t	25
Gold Grade						
3 and 4	75	120	75	1/D2	0.30 gm/t	25
1, 2 and 5 to 19	75	90	75	1/D2	0.30 gm/t	25
Sulfur Grade						
3 and 4	75	120	75	1/D2	10.00%	25
1, 2 and 5 to 19	75	90	75	1/D2	10.00%	25
Magnesium Grade, Stk 90, Dip 90						
3 and 4	180	180	120	1/D3		Not Applied
1, 2 and 5 to 19	180	180	120	1/D3		Not Applied

All grade estimates use the following number of composites:
Maximum = 10, Minimum = 1, Maximum per hole = 3

14.5.1 Additional Sulfur Estimation

The procedures for estimation of sulfur presented on Table 14-9 did not estimate sulfur grade everywhere that there was a nickel grade in the model. As noted in Table 14-2, there were 19,266 sulfur assay available compared to 23,732 assays for nickel and copper. The primary reason for the shortfall in sulfur assays was that the 2009 drilling was not assayed for sulfur. The missing sulfur assays from 2009 primarily impacted the east side of the block model in domain 19.

There were 19,581 blocks (7.7% of the total required) that did not receive a sulfur estimate using the parameters on the previous pages. The total of 19,581 was 87% contained in peridotite, 60% contained in the eastern most estimation domain 19, and 92.5% inferred. Efforts to expand the search did not fill in a substantial number of the remaining blocks due to the lack of assaying in 2009 contained in the east side peridotite.

In order to provide an estimate of sulfur for metallurgical recovery estimation in these predominately inferred blocks, a series of correlations were developed based on the nickel grade of the block and the relative position from north to south.

There is a general correlation of sulfur versus nickel. Higher grade nickel is associated with higher grade sulfur. There is also a correlation of sulfur with distance south from the contact between the mafic units and the sedimentary units to the north. As one moves south from the contact, there is a reduction in sulfur grade. Both correlations were calculated for the un-estimated blocks and the lessor of the two correlation estimates was selected as the sulfur grade assignment for those block not estimated by the 1/D2 procedure.

The nickel correlation equations were differed by rock type and are listed below:

Peridotite
- 0.0001 <+ Ni < 0.168, Sulfur = 0.2615
- NI > = 0.168, Sulfur = Ni x 3.834502-0.380817

Clinopyroxenite
- Sulfur = Ni x 4.209504+0.110338

Gabbro
- Sulfur = Ni x 4.690852 + 0.298162

The estimation by location was based on the northing coordinate of the model block Table 14-10 summarizes the coordinate correlation procedure.

Table 14-10: North Coordinate Correlation to Sulfur

Range of North		Sulfur Correlation to North Coord	Factor +	Constant
From	To	in Peridotite Only		
6,814,895	6,815,075	Sulfur=(((North-6814705)/10)+1) x	0.00717	-0.14233
6,815,075	6,815,155		0.00100	0.09200
6,815,155	6,815,265		0.00836	-0.25927
6,815,265	6,815,435		0.01741	-0.75947
6,815,435	6,815,505		0.00057	0.48671
6,815,505	6,815,585		0.01763	-0.89463
6,815,585	6,815,655		-0.00286	0.92829
6,815,655	6,815,765		0.01427	-0.71682

Once both correlation procedures were developed the lower value of the two was used as the best estimate for sulfur that was outside of the 1/D2 estimate. Of the 19,581 blocks that were not estimated by 1/D2, 48% were assigned by the coordinate correlation with 52% assigned by correlation to nickel grade.

14.5.2 Block Grade Check Procedure

IMC utilizes a simple method to compare the block model results against the composite grades used to estimate the block model. The test is applied to each metal separately and is summarized as follows:

The following steps are repeated at multiple cutoff grades to understand the response of the block model relative to composites over a range of cutoff grades:

- a test cutoff grade is selected, and the block model is used to establish the number of blocks above that cutoff and the average grade of those blocks
- the drill hole composites that are contained within that group of blocks are identified, and the average grade of the composites calculated
- the percentage of the composites less than the grade outline value that are contained within the selected cutoff zone is tabulated

The comparison of the composite mean to the block mean should generally result in the average grade of the composites being higher than the average grade of the blocks. This is because the block grades are usually impacted by surrounding lower grade samples. If the block grades are higher than the average of the contained composites, additional investigation is warranted.

The calculation of the percentage of composites that are less than the block outline cutoff is an indication of the amount of smearing of high grade over lower grade that has occurred in the model. For reference, values in the range of 15 to 20% are typical for block models estimated by ordinary kriging.

Table 14-11 summarizes the results of the grade outline to contained composite check for nickel, copper, platinum, and palladium as examples. All metals received the same test procedure.

The results of the test also indicate that reasonable amounts of dilution are included in the model and IMC does not recommend the addition of dilution factors to the block model for determination of mineral resources or any mine planning that may follow.

The test addresses the following combined rock types: Clinopyroxenite, Mineralized Gabbro, Peridotite, and Massive Sulfide.

As noted, the Mg estimates are intended to be used to determine the process plant response. A boundary of 22% Mg is used to define the boundary between process responses. As a further check, the percentage of composites that are contained in the block estimate of plus 22% that are less than 22% is 6.9%. That low value indicates that the 1/D3 methods are respecting the data and providing a reasonable estimate of the material in each process response class.

Table 14-11: Block Model to Composite Check, Selected Metals

Cut-off Grade Tested	Number of Composites	Average Composite Grade	Percentage of Composites Less Than Cutoff	Number of Blocks Above Cut-off	Average Block grade Above Cut-off
Nickel Estimation Test, Grades in Ni%					
0.00	4,530	0.248	0.00%	255,107	0.237
0.05	4,353	0.256	2.85%	250,799	0.242
0.10	4,143	0.266	3.21%	241,018	0.247
0.15	3,858	0.277	4.12%	226,247	0.255
0.20	3,183	0.299	5.94%	181,784	0.274
0.25	2,181	0.332	9.08%	112,309	0.303
0.30	943	0.404	9.86%	32,491	0.378
0.35	417	0.501	15.83%	11,343	0.494
0.40	224	0.617	16.96%	6,463	0.587
0.45	152	0.710	19.08%	4,344	0.667
Copper Estimation Test, Grades in Cu%					
0.00	4,530	0.172	0.00%	259,263	0.128
0.05	3,786	0.200	4.46%	197,008	0.163
0.10	2,558	0.259	10.63%	126,725	0.210
0.15	1,569	0.349	11.98%	70,201	0.282
0.20	1,154	0.416	11.70%	47,000	0.337
0.25	896	0.470	11.72%	34,373	0.379
0.30	691	0.530	11.57%	24,041	0.424
0.35	537	0.589	12.10%	15,986	0.473
0.40	401	0.661	11.97%	10,305	0.529
0.45	313	0.722	11.18%	6,978	0.580
0.50	242	0.791	11.57%	4,301	0.648
Platinum Estimation Test, Grades in Pt gm/t					
0.00	4,530	0.256	0.00%	261,826	0.209
0.10	3,669	0.302	5.39%	198,450	0.256
0.20	2,113	0.416	11.26%	104,620	0.348
0.30	1,192	0.554	10.99%	54,180	0.449
0.40	771	0.668	9.73%	32,543	0.515
0.50	506	0.779	11.86%	14,262	0.605
0.60	306	0.922	11.44%	5,269	0.713
0.70	186	1.079	11.83%	1,737	0.864
0.80	139	1.179	15.83%	832	0.999
0.90	97	1.291	19.59%	524	1.088
1.00	71	1.400	15.49%	299	1.198
Palladium Estimation Test, Grades in Pd gm/t					
0.00	4,530	0.235	0.00%	262,099	0.216
0.10	3,898	0.264	5.54%	262,281	0.240
0.20	2,681	0.319	10.18%	148,492	0.288
0.30	1,017	0.441	11.90%	49,169	0.375
0.40	399	0.582	14.54%	12,051	0.483

0.50	170	0.734	17.65%	3,322	0.606
0.60	93	0.879	17.20%	1,257	0.715
0.70	46	1.063	19.57%	420	0.859
0.80	34	1.129	26.47%	165	0.105
0.90	20	1.313	20.00%	98	1.178
1.00	13	1.473	23.08%	67	1.285

14.5.3 2017 Drilling Impact

The model reported in this section was originally developed during June of 2017. After completion of the model another 16 drill holes were completed in the last half of 2017 that contained 2787.3m of drilling and 897 assay intervals. The 2017 drill holes were for obtaining metallurgical samples. The potential impact of the drilling was tested to determine if the model required update with the 2017 drilling, or if the impact was immaterial and the June 2017 model was reasonable for the definition of mineral resources.

Several tests were completed to understand the impact these holes would have on the block model and mineral resource. The following two tests provide clear indication of the impact of the 2017 drilling:

- The 2017 drill holes were compared to the block grades of the resource model that contained the drill holes. This was the first indication that the new holes were not significantly different than the June 17 model estimate.
- The block grades of Nickel and Copper were quickly updated as a test including the new holes to understand the overall impact on the model and mineral resource.

The 2017 drilling was composited to 10m down hole intervals and the composites were "dipped" in the model. In other words, the model grade of the block that contained the composite was added to the composite data base.

Comparing the grades of the drill hole composites versus the block grades that contain them represents: 1) a validation of the model and, 2) a test of the impact of the new holes on the model if they were to be included. In addition to a simple comparison of the mean, two statistical hypothesis tests were also tabulated to indicate the impact of any differences in the drill hole composite to block grade mean comparison (Table 14-12) .

Table 14-12: Comparison of 2017 Drilling vs June 2017 Model

Metal	10m Composite of 2017 Drilling		Model Blocks Containing Drilling		T-Statistic at 95%	Paired T-Statistic at 95%
	Number	Avg Grd	Number	Avg Grd		
Nickel %	257	0.264	257	0.255	Pass	Pass
Copper %	257	0.078	257	0.081	Pass	Pass
Cobalt %	257	0.015	257	0.014	Pass	Pass
Platinum gm/t	257	0.143	257	0.142	Pass	Pass
Palladium gm/t	257	0.203	257	0.197	Pass	Pass
Gold gm/t	257	0.022	257	0.022	Pass	Pass

The results above indicate that the new drilling is quite similar to the previous model estimate. The expectation from the above work is that a model update with the new drilling would not be materially different than the June 17 model as it exists today.

The model grade estimation methods applied to the June 17 model were applied to the nickel and copper block estimate incorporating the 16 new holes from late 2017. The grade estimation methods for this test replicated the June 17 work and were stored in additional test variables within the same model framework. The difference between the June 17 model estimate and the updated estimate for the two test metals was tabulated as a direct measure of the potential impact of the new drilling.

The resource was not re-tabulated. The comparison was made between the estimated tonnage and grade of nickel and copper within the 2017 resource pit boundary. The results are as follows on Table 14-13.

Table 14-13: Test Model Estimate Including Late 2017 Drilling

Test Metal	26Jun17 Model Estimated Blocks		Add the 2017 Drilling Re-Estimated Blocks		Percent Difference		
	Number	Grade	Number	Grade	Tonnes	Grade	Contained Metal
Nickel	199,348	0.253	199,378	0.253	0.02%	-0.01%	0.01%
Copper	202,494	0.122	207,717	0.120	2.58%	-1.68%	0.85%

Changes on this scale would not be material to the stated mineral resource.

14.5.4 Bulk Density

Bulk density assignment to the model was based on the samples collected at the core shed by staff geologists. Selected samples from each core box are weighed wet and dry to calculate specific gravity. IMC completed the calculation of specific gravity for those samples in the Nickel Shäw data base where the weights had been collected, but the specific gravity field was not populated.

IMC completed several class regression analyses to determine if there was a correlation between fracturing represented by RQD and specific gravity. The average RQD for each rock type was then used to determine the density reduction factor used from the RQD plots. In summary, a 2% reduction of measured specific gravity was applied across the board to all the mineral bearing rock types.

Table 14-14 illustrates the results of the specific gravity tests and the actual density value assigned to the respective rock type in the block model.

Table 14-14: Bulk Density Assignment to the Block Model

Rock Type	Model Code	Number of Samples	Mean of Samples Sp.G	Model Assignment Bulk Density
Clinopyroxenite	7	773	2.919	2.861
Mineralized Gabbro	20	484	3.040	2.979
Peridotite	24	3,251	2.806	2.750
Massive Sulfide	29	13	3.104	3.042
Sedimentary Rocks	26	142	2.747	2.692
Volcanics	32	297	2.814	2.758
Basalt	5			2.770
Maple Creek Gabbro	21			2.800
Unassigned				2.770

Topographic codes are stored in the block model to reflect the amount of each block that exists below topography. Since there has been historic stoping at Nickel Shäw, there is a second model code that reflects the fraction of each block remaining after the underground mining. Those variables have been combined for the determination of mineral resources so that the underground volumes are removed from the calculations of remaining mineral resource.

14.5.5 Classification

The classification categories of Measured, Indicated, and Inferred were based on the number of samples used to estimate the block and the average distance from the block to the data used for block grade estimation.

The classification codes were based on the estimation parameters for nickel. This is appropriate because all economic metals were estimated with nearly the same number of composites.

Measured

- 10 composites were used
- average distance of the searched samples was <= 45 m (½ of range)

Indicated

- at least 4 composites were used (minimum 2 drill holes)
- average distance of the searched samples was <=70m (78% of range)

Inferred

- any remaining block with a nickel grade out to the search distance

14.6 Mineral Resource

Mineral resources for the Wellgreen deposit were developed based on the block model described in this section. A computer-generated pit geometry for the resource was developed by AGP using the Lerchs-Grossman algorithm. John Marek, of IMC, acting as the Qualified Person checked the results using the floating cone algorithm and confirmed the resource pit has reasonable prospects of economic extraction.

A mineral resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material, including base and precious metals, coal, and industrial minerals in or on the earth's crust in such form and quantity, and of such a grade or quality, that it has reasonable prospects for economic extraction. The location, quantity, grade, geologic characteristics, and continuity of a mineral resource are known, estimated, or interpreted from specific geological evidence and knowledge.

The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect to the technical and economic factors likely to influence the prospects of economic extraction. A mineral resource is an inventory of mineralization that, under realistically assumed and justifiable technical and economic conditions, might become economically extractable. These assumptions must be presented explicitly in both public and technical reports. The assumptions are stated below and on the accompanying tables.

The current process approach to Nickel Shäw is a large scale process facility that might process about 45,000 tpd. The process facility is currently planned to have two options of concentrate production:

1. A bulk concentrate of primarily nickel with minor copper plus accessory metals.
2. A split concentrate when copper values in the ground are sufficiently high to warrant two concentrates of:
    a. A nickel concentrate with accessory metals, and
    b. A copper concentrate with accessory metals.

The process recoveries and metal grades of concentrate are based on the results of the metallurgical testing reported in Section 12.0.

The procedure applied to the block model was to calculate the net value after processing for each block of the model for both bulk concentrate and split concentrates. The process option that provided the maximum benefit for processing of that block was applied. As a result, roughly 70% of the reported resource would produce a single bulk concentrate rather than two concentrates of nickel and copper respectively.

Several steps were required to apply the process results and estimated costs for smelting refining and freight. All costs were estimated by AGP with review by IMC on a comparative basis against other active and recent projects.

Table 14-15 summarizes the metal prices, mining costs, process costs, and slope angles that were applied to the open pit algorithm. Costs are presented in both Canadian and U.S. Dollars on the table. Internally, the open pit software utilized Canadian dollars, however, the input parameters and results are reported in U.S. Dollars.

Table 14-16 summarizes the details of process plant recovery depending on the application of split concentrates versus producing a single bulk concentrate. The process recoveries for nickel are based on the presence of Magnesium (Mg) and Sulfur (S). the equations are presented at the bottom of the table. Copper recovery is developed on a constant tail basis applying 0.06% as the constant tail. Split concentrate recoveries for nickel and copper are factored from the bulk concentrate recoveries.

Indicative downstream smelting and refining costs were incorporated for both the split concentrates and bulk concentrate option and are summarized on Table 14-17. Costs and penalties are based on the average concentrate grades reported on Table 14-16.

Cutoff grades are reported in terms of Net Smelting Return (NSR) using the internal or marginal cutoff grade. Process + G&A for bulk concentrate production is $11.51 USD/tonne and $11.74 USD/tonne for split concentrate production. The proper cutoff has been applied to the appropriate blocks in the model.

Table 14-18 summarizes the resulting mineral resources and Table 14-15 through Table 14-17 summarize some of the economic assumptions that were used for this Mineral Resource calculation. The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be realized, or that they will convert to mineral reserves. John Marek of IMC is the QP for this statement of mineral resources. Currently there is no mineral reserve at the Project.

The risks associated with this statement of mineral resources include, metal price impacts, changes to process recovery as more testing is complete, and permit risks that are typical of any North American mineral development.

Figure 14-6 illustrates the mineral envelope on the same section as Figure 14-2. Figure 14-7 presents a 3D illustration of the resource and the resource pit geometry.

Table 14-15: Metal Prices, Site Costs and Slope Angles

	$ CDN	$USD	unit
Metal Prices			
Nickel	$10.58	$8.25	/lb
Copper	$3.85	$3.00	/lb
Cobalt	$30.77	$24.00	/lb
Platinum	$1,538	$1,200	/troy Oz
Palladium	$1,154	$900	/troy Oz
Gold	$1,667	$1,300	/troy Oz
Waste Mining			
Base	1.89	1.4742	
Subtract per bench above 1260	0.006	0.0047	
Add per bench below 1260	0.03	0.0234	
Ore Mining			
Base	1.64	1.2792	
Add per bench above 1260	0.02	0.0156	
Add per bench below 1260	0.02	0.0156	
Process Costs, Split Concentrate			
Process + Tails + Site Supplies	14.76	11.5128	
G&A	0.29	0.2262	
Total Process + G&A	15.05	11.7390	
Process Costs, Bulk Concentrate			
Process + Tails + Site Supplies	14.46	11.2788	
G&A	0.29	0.2262	
Total Process + G&A	14.75	11.5050	
Over Slope Angles for Pit Generation			
Basalt, Volcanoclastic, Maple Creek Gabbro Metasediments + Default	44	Degrees	
Mineralized Gabbro, Clinopyroxenite, Massive Sulfide	35	Degrees	
Peridotite	33	Degrees	
Exchange Rate: $CDN x 0.78 = $ US			

Table 14-16: Process Recoveries for Bulk Concentrate and Split Concentrates

Split Concentrate				Bulk Concentrate	
Nickel Concentrate		Copper Concentrate			
Recovery to Ni Con	**Recovery**	**Recovery to Cu Con**	**Recovery**	**Recovery to Bulk Con**	**Recovery**
Copper	equation below	Copper	equation below	Copper	equation below
Nickel	equation below	Nickel	equation below	Nickel	equation below
Cobalt	55.9%	Cobalt	1.1%	Cobalt	57.0%
Platinum	46.9%	Platinum	0.9%	Platinum	47.9%
Palladium	50.8%	Palladium	3.2%	Palladium	53.9%
Gold	50.8%	Gold	23.6%	Gold	74.4%
Metal Grades in Nickel Concentrate		**Metal Grades in Copper Concentrate**		**Metal Grades in Nickel Concentrate**	
Copper	1.30%	Copper	18.02%	Copper	3.06%
Nickel	6.57%	Nickel	1.10%	Nickel	6.08%
Cobalt	0.36%	Cobalt	0.06%	Cobalt	0.33%
Platinum	3.99 gm/t	Platinum	0.67 gm/t	Platinum	3.06 gm/t
Palladium	5.11 gm/t	Palladium	2.69 gm/t	Palladium	4.85 gm/t
Gold	0.40 gm/t	Gold	1.58 gm/t	Gold	0.50 gm/t
MgO	6.15%	MgO	0.70%	MgO	5.60%
		Sulfur	35.00%		

Recovery Equations for Nickel and Copper:
Define "X" = (%S-%Cu)/%Ni Capped at 12.0%
For Blocks Where Mg > 22%
Ni Recovery to Bulk Con= (1-(-0.3758*X+0.9672)) *100
For Blocks Where Mg <= 22%
Ni Recovery to Bulk Con= (1-(-0.243Ln(X)+0.6747)) *100
Ni Recovery to Ni Con = Ni Recovery to Bulk Con *0.98
Ni Recovery to Cu Con = Ni Recovery to Bulk Con *0.02
Copper Recovery to Bulk Con = ((Cu-0.06)/Cu)) *100, Constant tail at 0.06% Cu
Copper Recovery to Copper Con = Cu Recovery to Bulk Con *0.62
Copper Recovery to Nickel Con = Cu Recovery to Bulk Con *0.38

Table 14-17: Smelting, Refining, and Freight Charges and Multipliers for NSR Determination

Split Concentrate						Bulk Concentrate		
Nickel Concentrate			Copper Concentrate					
TCRC Costs and Penalties per Block Recovered Grade			TCRC Costs and Penalties per Block Recovered Grade			TCRC Costs and Penalties per Block Recovered Grade		
Recovered Copper x 22.0462	3.02	$ / lb	Recovered Copper x 22.0462	0.84	$ / lb	Recovered Copper x 22.0462	1.81	$ / lb
Recovered Nickel x 22.0462	3.23	$ / lb	Recovered Nickel x 22.0462	8.29	$ / lb	Recovered Nickel x 22.0462	3.27	$ / lb
Recovered Cobalt x 22.0462	15.68	$ / lb	Recovered Cobalt x 22.0462	24.12	$ / lb	Recovered Cobalt x 22.0462	15.68	$ / lb
Recovered Platinum / 31.1035	578.89	$ / oz	Recovered Platinum / 31.1035	1206.03	$ / oz	Recovered Platinum / 31.1035	578.89	$ / oz
Recovered Palladium / 31.1035	434.17	$ / oz	Recovered Palladium / 31.1035	449.91	$ / oz	Recovered Palladium / 31.1035	434.17	$ / oz
Recovered Gold / 31.1035	1305.53	$ / oz	Recovered Gold / 31.1035	874.88	$ / oz	Recovered Gold / 31.1035	1306.53	$ / oz
Block Recovered Grade Multipliers to Set NSR			Block Recovered Grade Multipliers to Set NSR			Block Recovered Grade Multipliers to Set NSR		
Recovered Copper x 22.0462	0.00	$ / lb	Recovered Copper x 22.0462	2.16	$ / lb	Recovered Copper x 22.0462	1.19	$ / lb
Recovered Nickel x 22.0462	5.02	$ / lb	Recovered Nickel x 22.0462	0.00	$ / lb	Recovered Nickel x 22.0462	4.98	$ / lb
Recovered Cobalt x 22.0462	8.32	$ / lb	Recovered Cobalt x 22.0462	0.00	$ / lb	Recovered Cobalt x 22.0462	8.32	$ / lb
Recovered Platinum / 31.1035	621.11	$ / oz	Recovered Platinum / 31.1035	0.00	$ / oz	Recovered Platinum / 31.1035	621.11	$ / oz
Recovered Palladium / 31.1035	465.83	$ / oz	Recovered Palladium / 31.1035	450.09	$ / oz	Recovered Palladium / 31.1035	465.83	$ / oz
Recovered Gold / 31.1035	0.00	$ / oz	Recovered Gold / 31.1035	425.12	$ / oz	Recovered Gold / 31.1035	0.00	$ / oz

Table 14-18: Project Mineral Resources on 24 September 2018

										Contained Metal					
Class	Ktonnes	Nickel %	Copper %	Cobalt %	Platinum gm/t	Palladium gm/t	Gold gm/t	Mg %	Sulfur %	Ni M Lbs	Cu M Lbs	Co M Lbs	Pt K Ozs	Pd K Ozs	Au K Ozs
Measured	93,300	0.25	0.17	0.015	0.262	0.244	0.054	15.7	0.85	514	350	31	786	732	162
Indicated	230,100	0.27	0.15	0.015	0.249	0.259	0.043	16.8	0.74	1,370	761	76	1,842	1,916	318
Total M+I	323,400	0.26	0.16	0.015	0.253	0.255	0.046	16.5	0.77	1,884	1,111	107	2,628	2,648	480
Inferred	108,100	0.29	0.15	0.016	0.256	0.279	0.040	16.2	0.72	691	357	38	890	970	139

Notes:

Mineral Resources do not have demonstrated economic viability

The QP for the Mineral Resource is John Marek RM-SME, Professional Engineer Yukon Territory

Average grade calculations on this table are impacted by rounding.

Tonnages are reported in units of 1,000 metric tonnes (Ktonnes)

Contained Base Metal reported in units of 1,000,000 lbs, M Lbs

Contained Precious Metal reported in units of 1,000 troy ounces, K ozs

Metal Prices and Summarized Costs for Resource Determination in USD:

Nickel: $8.25/lb, Copper: $3.00/lb, Cobalt: $24.00/lb

Platinum: $1,200/troy oz, Palladium: $900/troy oz, Gold: $1,300/troy oz

Net of Smelting (NSR) cut-off grades range from $11.51 to $11.74 U.S. Dollars (Bulk vs Split Con)

Mining Cost vary by bench, separately for ore and waste

Average Mining Costs within the resource pit are $1.48 USD per total tonne moved

The average strip ratio for the resource pit is: 2.97 to 1

Average calculated process recoveries combining the bulk and split concentrates approach:

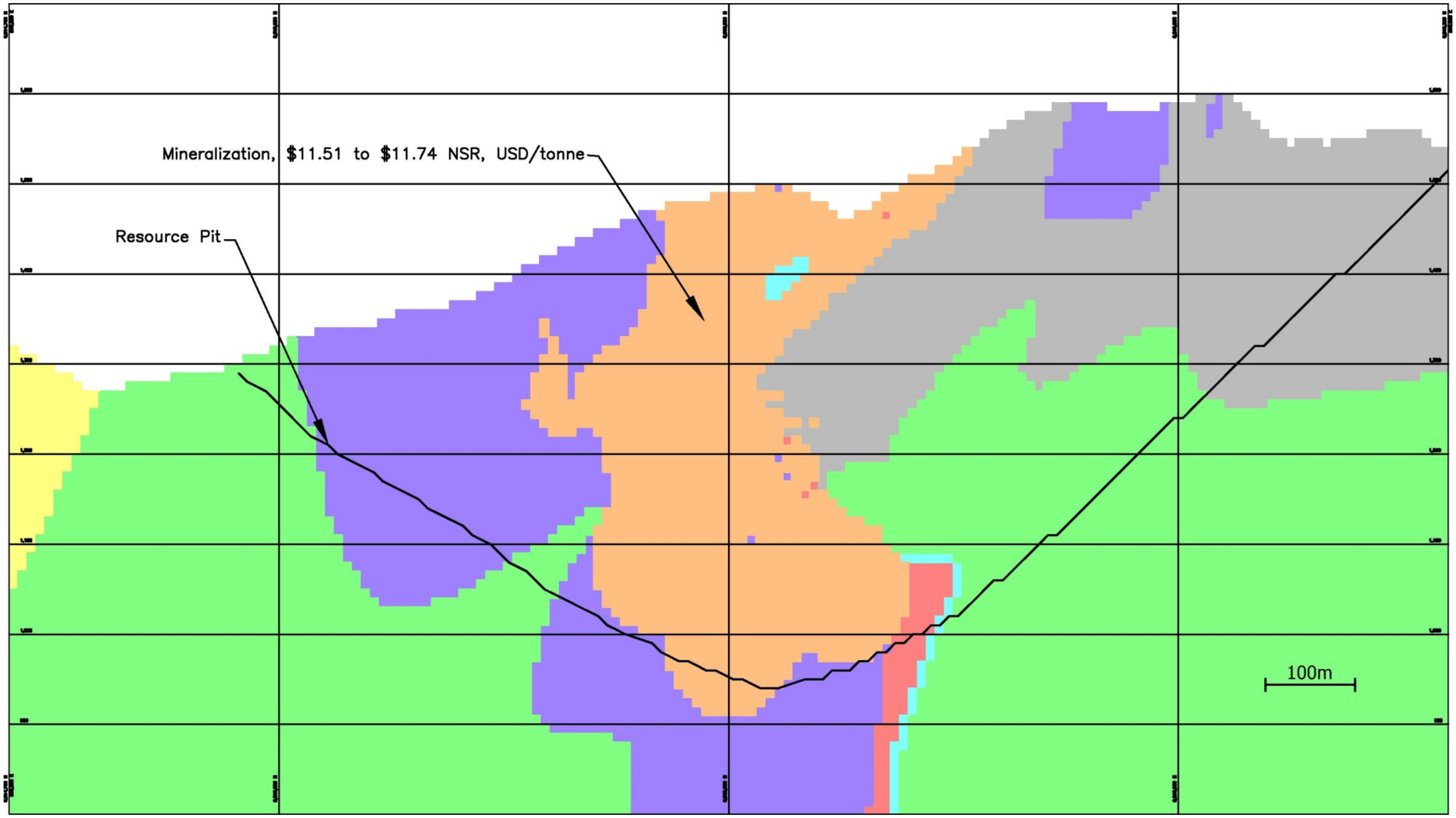
Ni	Cu	Co	Pt	Pd	Au
48.0%	62.2%	60.0%	47.8%	54.0%	47.1%

Average Smelting + Transport Costs, and Losses in Terms of Cost Per Unit in Concentrate :

Ni/lb	Cu/lb	Co/lb	Pt/oz	Pd/oz	Au/oz
$3.26	$1.14	$15.68	$578.89	$434.50	$1,179.00

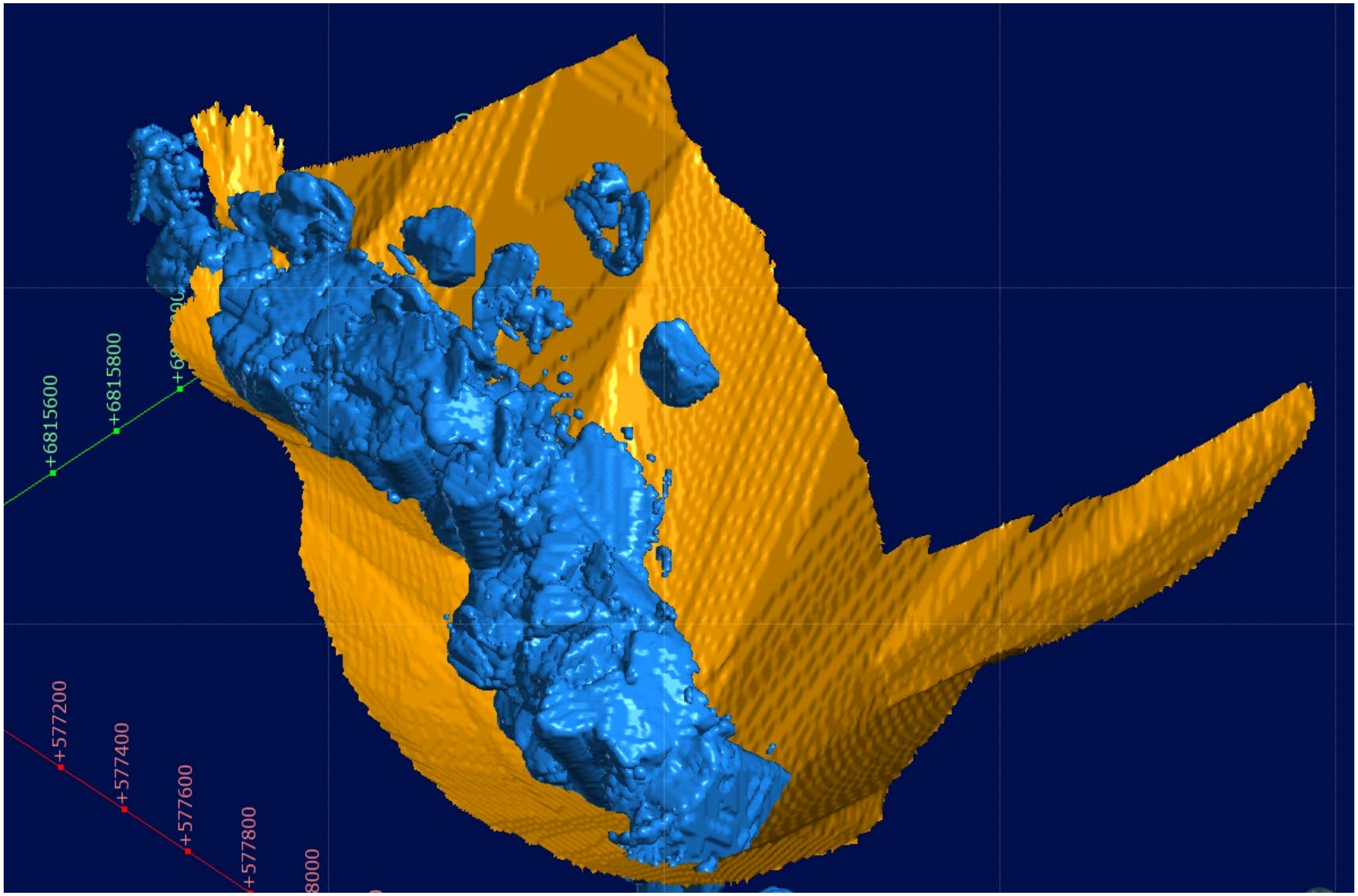
Slope Angles vary from 33 to 44 Degrees

Figure 14-6: North-South Section 578,200 E, looking West, Showing Mineralization Overlay on Geology



Source: IMC, 2018

Figure 14-7: Mineral Resource in the Resource ($11.51 USD NSR/tonne) looking 315 NW & Down 41 Degrees



Source: IMC, 2018

15 MINERAL RESERVE ESTIMATES

This report is an update of the mineral resources present at the Project. No additional work considering economics has been performed to bring the mineral resources to a level of reserves and therefore, no mineral reserves have been estimated.

16 MINING METHODS

16.1 Introduction

Conventional open pit mining was selected as the mining method to examine the reasonable prospects of economic extraction for the Project. This selection was based on the size of the resource, tenor, distribution of the grade, and proximity to topography. AGP believes with current metal prices and knowledge of the mineralization, large scale open-pit mining represents the most viable approach for economic extraction.

16.2 Geotechnical

In support of the current resource update, AGP has completed a compilation, review, and preliminary assessment of available and recently collected geotechnical data and information for the Project. A gap analysis was also completed by AGP to provide guidance and recommendations for the further field studies (geotechnical drilling, outcrop mapping, field index testing), laboratory testing, and geotechnical and hydrogeological design work that is necessary to advance the Project.

Data compiled and reviewed to date includes exploration and geotechnical drilling logs, core photographs, recovery rock quality designation (RQD), rock mass classification data, laboratory test data, geotechnical mapping data, preliminary LG pit shells, geologic models, and relevant background reports.

In 2015, two core holes (WS15-232 and WS15-257) were logged in detail for geotechnical data. In the late summer and fall of 2017, Wellgreen completed a 15-hole 2,720 m drill program with the principal goals of providing additional samples for metallurgical testing and upgrading the Project's mineral resources. AGP assisted NCP with collecting geotechnical data during the program. AGP's scope of work included:

- geotechnical logging training of NCP geologists and field technicians delivered during an early-stage, onsite training course
- provision of site-specific geotechnical reference materials and logging information designed to complement existing/ongoing geo-data collection using GeoSpark database software
- instruction and training on geotechnical sampling for subsequent laboratory testing
- geotechnical mapping of select/representative outcrops
- periodic off-site technical support
- Point Load index strength testing of core samples

Rock mass classification data from the geotechnical holes in Table 16-1 have been compiled and analyzed by AGP to assess the range of conditions likely to be encountered during mining. Although key sectors of the proposed pit remained largely uninvestigated by the 2017 metallurgical drilling program, the geotechnical data and samples collected from the program were none-the-less useful in advancing geotechnical knowledge of the Project.

Table 16-1: 2017 Drill Program – Geotechnical Logging and Sampling

Hole ID	Easting	Northing	Elevation (m)	Max Depth (m)	Geotech	Samples
WS17-287	577741.2	6815565.9	1612.4	61.5		
WS17-288	577741.2	6815565.9	1612.4	360		
WS17-289	577869.5	6815526.1	1551.9	183		
WS17-290	577469.9	6815674.7	1612.9	203		
WS17-291	577625.2	6815622.7	1650.8	248		
WS17-292	577703	6815634.4	1653.3	258.3	Y	Y
WS17-293	577423.6	6815634.7	1576.5	146		
WS17-294	577575.8	6815458	1579.4	82	Y	
WS17-295	577944.9	6815627.4	1540.9	102		
WS17-296	578364.7	6815479.5	1447.6	168	Y	
WS17-297	577474.2	6815424.8	1543.8	291	Y	Y
WS17-298	577675.8	6815460.7	1592.8	201	Y	Y
WS17-299	578576.5	6815113.3	1309.8	237	Y	Y
WS17-300	578133.9	6815262.6	1399.4	75.9	Y	Y
WS17-300B	578133.9	6815262.6	1399.4	82.7	Y	
WS17-301	577980.2	6815409.6	1476.4	88	Y	Y

Geotechnical surface mapping was completed by NCP and AGP during the 2018 summer field season to verify and complement the existing dataset including lithology, rock mass strength, and discontinuity characteristics. The field mapping program collected data from approximately 125 mapping locations covering most of the extents of a potential open pit. No geotechnical drilling, logging, sampling, or laboratory testing were conducted in 2018.

The above data was reviewed and analyzed to develop a 3D approximation of the character and variability of the rock mass conditions in the vicinity of the proposed slopes. From these models, AGP notes a limited amount of drilling data for the bulk of the proposed north slope high-wall (south facing), and an apparent variability of rock quality, even as lithology remains constant over large areas.

The orientation and extent of major structures and jointing are also largely unknown at this time and may have significant impacts on achievable slope angles at later stages of project development. A number of faults and/or fault systems have been mapped and intersected by drill holes and are interpreted to exist within the resource pit extents. The current level of knowledge regarding these faults and their geotechnical conditions is low. Additional work to collect and refine structural information is required as the Project advances.

Hydrogeological conditions are not well known for the site however, it is understood that the lower levels of underground workings, located within the future east-central portion of the proposed pit, are flooded below the portal elevation (approximate elevation 1,280 m), suggesting pit slopes will be at least partially saturated.

Based on the above information, the following slopes (Table 16-2) were applied for the resource constraining pit shell.

Table 16-2: LG Shell Slope Parameters

Slope	Rock type
33˚	Peridotite
35˚	Clinopyroxenite
35˚	Mineralized Gabbro/Massive Sulfides
44˚	Sediments
44˚	Station Creek Volcanics
44˚	Maple Creek Gabbro

16.3 Mining Costs

For the development of the resource constraining pit shell, representative mining costs needed to be developed. Local vendors were contacted for mine equipment costing, tires, explosives, and other consumables. The large-scale nature of the Project meant that 200-ton class trucks and their respective loaders were considered.

The mine costing scenario featured 22 m^3 diesel hydraulic shovels as primary loaders together with the 200-ton haul trucks on 10 m benches. Drilling will be performed by track mounted diesel drills capable of single pass drilling on the planned 10 m bench. A standard suite of support and ancillary equipment was also considered. Truck productivities were estimated based on high-level ore and waste haul profiles at different potential resource pit levels. This was done in an effort to consider the topography present at the location and significant downhill haulage component possible for both ore and waste.

This information was placed in AGP's operating cost model to determine a realistic mining cost representative of the deposit for resource determination. This approach could be considered as somewhat more rigorous than what is normally accomplished for resource pit shells however, it adds to the confidence required for "reasonable prospects of economic extraction" and for Nickel Creek management. The resulting costs are shown in Table 16-3.

Table 16-3: Ore and Waste Mining Costs – Resource Definition

Resource Pit		
Ore		
Base Mining Cost	$CDN/t mined	1.64
Down Increment	$CDN/t/10m bench above	+0.02
Up Increment	$CDN/t/10m bench below	+0.02
Waste		
Base Mining Cost	$CDN/t mined	1.89
Down Increment	$CDN/t/10m bench above	-0.01
Up Increment	$CDN/t/10m bench below	+0.03

Mining costs were determined in Canadian dollars internally as vendor quotes were provided in that manner. The final values were converted to US dollars using an exchange rate of $1.00 CDN = $0.78 US. These costs were provided to IMC to develop the resource pit shell.

17 RECOVERY METHODS

This section describes the parameters used to define a nickel-copper-PGM mineral process plant suitable for the Project located near Burwash, SW Yukon Territory. A nominal throughput of 45,000 metric tonnes per day has been selected for the purposes of this study.

The process flowsheet described herein is based upon the most recent metallurgical work completed by XPS located in Sudbury, Ontario in 2018, and as described in Section 13.0 of this report. The XPS testwork concluded with an MPP campaign, and the MPP flowsheet provides much of the design basis for the process. The process plant mass balance and flotation circuit configuration (arrangement of cells and residence times), grinding targets, energy consumption calculations, and reagent consumptions are derived to a great extent from the MPP conditions and results.

The flowsheet can be described as a conventional mineral processing circuit, consisting of primary crushing (gyratory crusher), SAG and ball mill grinding, froth flotation, low intensity magnetic separation, concentrate dewatering, and tailings thickening. The flotation process includes a circuit designed to separate copper and nickel minerals (Copper-Nickel Separation).

Metallurgical work suggests the process plant would be required to treat high-sulphur content and low-sulphur content mineralization slightly differently within the flotation circuit, and the flowsheet described in this section caters for this requirement.

17.1 Basis for Preliminary Design

The following preliminary documents have been developed for this resource update based on AGP's experience together with the results of recent MPP testwork:

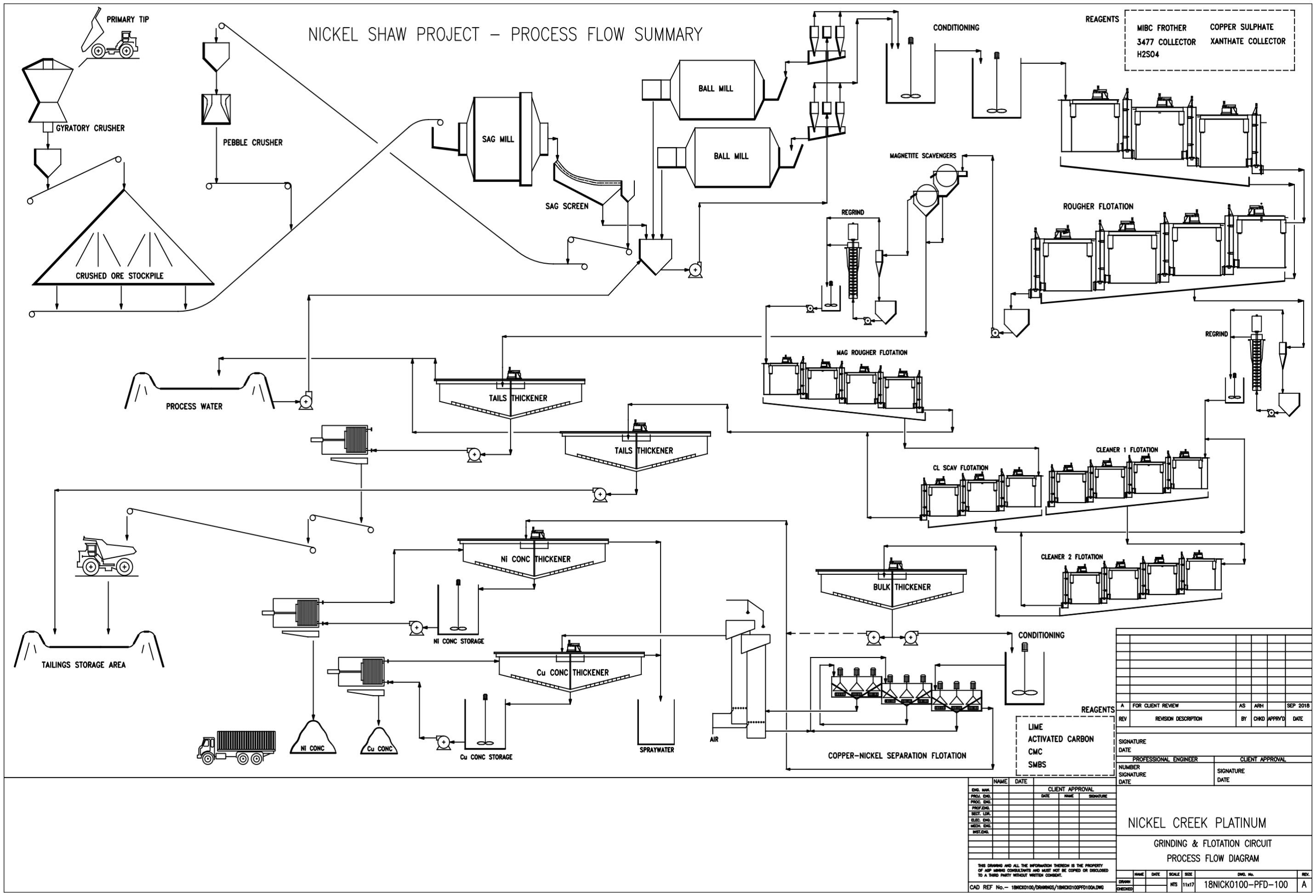
- process flow diagram
- preliminary process design criteria (PDC)
- mass, water and metal balance
- equipment listing

Together, these documents specify a mineral process plant suitable for the desired 45,000 tpd nominal duty. A summary of the key process design criteria is presented in Table 17-1 and a summary schematic of a suitable process flowsheet is presented in Figure 17-1.

Table 17-1: Summary of Process Design Criteria

Parameter	Unit	Design Data
Process Plant Nominal Throughput	dmtpd	45,000
LOM Average Head Grades:		
Nickel	%	0.26
Copper	%	0.15
Sulphur	%	0.76
Platinum	g/t	0.25
Palladium	g/t	0.25
Gold	g/t	0.04
Crushing Circuit Availability	%	75.2
Mill/Flotation Circuit Availability	%	92.6
Particle Sizes:		
Crusher Feed Size, F100	mm	500
Crusher Mill Feed Size, F80	mm	220
SAG Mill Feed Size, F100	mm	350
SAG Mill Feed Size, F80	mm	140
Primary Grind size, P80	µm	110
Rougher Conc Regrind Size, P80	µm	23
Magnetite Regrind Size, P80	µm	17
Bond Ball Wi	kWh/t	20.6
Flotation Nominal Retention times		
Bulk Rougher	min	38
1st Bulk Cleaner	min	30
1st Cleaner Scavenger	min	38
2nd Bulk Cleaner	min	25
3rd Bulk Cleaner	min	14
Magnetite Rougher	min	50
Cu/Ni Rougher/Scavenger	min	55
Cu/Ni 1st Cleaner	min	55
Reagent Dosages		
Frother – MIBC	g/t	120
Collector 1 – PIBX	g/t	40
Collector 2 – 3477 dithiophosphate	g/t	90
Activator – copper sulphate	g/t	50
Modifier – Activated Carbon	g/t	25
pH Adjustment – H2SO4	kg/t	11
pH Adjustment – Lime	g/t	85
Depressant – Finnfix 150 CMC	g/t	35
Sodium Metabisulphite	g/t	50

Figure 17-1: Flowsheet for the Project Processing Plant



Source: Nickel Creek, 2018

17.2 Mass Balance

Table 17-2 provides a summary of the key nominal process flows within the 45,000 tpd process plant.

Table 17-2: Mass/Metal Balance by Area

#	Description	dmtph	% Solids	SG	Flow m^3/h	Cu+Ni % Grade	Cu+Ni % Distribution
Area 100 – Crusher							
1	ROM Delivery to Crusher	2400	97%	2.66	872.7	0.51	100.0
2	Crusher Discharge to Stockpile	2400	97%	2.66	872.7	0.51	100.0
Area 150 – Stockpile							
1	Stockpile Discharge	2025.1	97.0%	2.66	785.9	0.51	100.0
Area 200 – Grinding							
1	SAG Mill Feed	2025.1	97.0%	2.66	785.9	0.51	100.0
2	Pebble Recycle	405.0	95.0%	2.53	168.6	0.36	14.0
3	SAG Mill Discharge	2430.1	70.5%	1.83	1889.5	0.48	114.0
4	Screen Oversize	405.0	95.0%	2.53	168.6	0.36	14.0
5	Screen Undersize	2025.1	54.4%	1.54	2420.9	0.51	100.0
6	Cyclone Feed (Per Ball Mill)	4050.1	56.1%	1.57	4610.6	0.55	215.0
7	Cyclone Underflow	3037.6	70.0%	1.83	2375.2	0.56	165.0
8	Cyclone Overflow	1012.5	35.1%	1.29	2235.4	0.51	50.0
Area 300 – Bulk Flotation							
1	Cyclone O/F A	1012.5	35.1%	1.29	2235.4	0.51	50.0
2	Cyclone O/F B	1012.5	35.1%	1.29	2235.4	0.51	50.0
3	Rougher Feed	2025.1	34.9%	1.29	4492.9	0.51	100.0
4	Rougher Concentrate	208.6	16.7%	1.13	1111.3	2.73	55.2
5	Rougher Tails to Mag Circuit	1816.5	39.0%	1.33	3500.4	0.25	44.8
6	Cleaner 1 Feed	341.0	13.9%	1.10	2220.6	2.26	74.7
7	Cleaner 1 Concentrate	144.9	11.4%	1.09	1170.5	4.58	64.3
8	Cleaner 1 Tails	196.1	15.8%	1.11	1115.2	0.00	0.0
9	Cleaner Scavenger Feed	341.6	16.7%	1.12	1830.0	0.35	11.5
10	Cleaner Scav Concentrate	49.2	11.4%	1.08	399.2	1.22	5.8
11	Cleaner Scav Tails	292.4	17.8%	1.13	1453.0	0.20	5.7
12	Cleaner 2 Feed	148.9	11.1%	1.09	1229.9	4.56	65.7
13	Cleaner 2 Concentrate	65.6	11.4%	1.09	527.3	8.17	51.9
14	Cleaner 2 Tail	83.2	10.6%	1.08	732.1	1.70	13.7
15	Cleaner 3 Feed	65.6	11.4%	1.09	527.3	8.17	51.9

#	Description	dmtph	% Solids	SG	Flow m^3/h	Cu+Ni % Grade	Cu+Ni % Distribution
16	Cleaner 3 Concentrate	61.7	11.4%	1.09	495.7	8.46	50.5
17	Cleaner 3 Tail	3.9	6.3%	1.05	59.4	3.68	1.4
18	Mag Scav Feed	1816.5	39.0%	1.33	3500.4	0.25	44.8
19	Mag Scav Concentrate	544.9	25.3%	1.20	1793.7	0.42	22.0
20	Mag Scav Tail	1271.5	41.0%	1.35	2306.7	0.19	22.8
21	Mag Rougher Feed	544.9	25.3%	1.20	1793.7	0.42	22.0
22	Mag Rougher Concentrate	145.5	16.7%	1.12	780.3	0.82	11.5
23	Mag Rougher Tail	399.4	29.6%	1.25	1078.9	0.27	10.4
24	Bulk Conc. Thickener Underflow	61.7	33.0%	1.33	140.7	8.46	50.5
25	Bulk Conc. Thickener Overflow	0.0	0.0%	1.00	355.0	0.00	0.0
Area 310 – Copper/Nickel Separation Flotation							
1	Bulk Concentrate Underflow	61.7	33.0%	1.33	140.7	8.46	50.5
2	Cu Rougher Concentrate	29.6	24.1%	1.22	100.8	7.90	22.6
3	Cu Rougher Tail	32.1	41.5%	1.45	53.2	8.98	27.9
4	Cu Scavenger Feed	69.4	28.4%	1.27	192.7	7.35	49.4
5	Cu Scavenger Conc	13.9	20.0%	1.17	59.0	7.13	9.6
6	Cu Scav Tail (Nickel Conc)	55.5	30.6%	1.30	139.9	7.40	39.8
7	Cleaner 1 Feed	60.2	22.6%	1.20	222.0	6.97	40.7
8	Cleaner 1 Concentrate	22.9	20.8%	1.18	92.8	8.64	19.1
9	Cleaner 1 Tail	37.3	22.3%	1.20	139.5	5.95	21.5
10	Cleaner 2 Feed	29.8	21.3%	1.19	118.0	7.93	22.9
11	Cleaner 2 Concentrate	13.1	18.3%	1.16	61.8	11.39	14.5
12	Cleaner 2 Tail	16.7	22.4%	1.20	62.1	5.21	8.4
13	Cleaner 3 Feed	13.1	18.3%	1.16	61.8	11.39	14.5
14	Cleaner 3 Concentrate	6.2	14.1%	1.12	39.3	17.94	10.7
15	Cleaner 3 Tail	7.0	22.9%	1.20	25.2	5.59	3.8
Area 360 – Copper/Nickel Concentrate Dewatering							
1	Copper Conc from Flotation	6.2	15.0%	1.13	36.5	17.94	10.7
2	Filtrate from conc filter	0.0	0.0%	1.00	4.4	1.79	0.0
3	Thickener Feed	6.2	13.6%	1.11	40.9	17.94	10.7
4	Thickener O/F	0.0	0.0%	1.00	34.3	0.72	0.0
5	Thickener U/F	6.2	55.0%	1.69	6.6	17.94	10.7
6	Cu Filter Feed	6.2	55.0%	1.69	6.6	17.94	10.7
7	Cu Cake Filtrate	0.0	0.0%	1.00	4.4	1.79	0.0

#	Description	dmtph	% Solids	SG	Flow m^3/h	Cu+Ni % Grade	Cu+Ni % Distribution
8	Cu Concentrate Cake	6.2	90.0%	3.02	2.3	17.94	10.7
9	Nickel Concentrate from Flotation	55.5	30.6%	1.30	139.9	7.40	39.8
10	Filtrate from conc filter	0.0	0.0%	1.00	39.3	0.74	0.0
11	Ni Thickener Feed	55.5	25.1%	1.23	179.2	7.40	39.8
12	Ni Thickener O/F	0.0	0.0%	1.00	119.9	0.30	0.0
13	Ni Thickener U/F	55.5	55.0%	1.70	59.3	7.40	39.8
14	Ni Filter Feed	55.5	55.0%	1.70	59.3	7.40	39.8
15	Ni Cake Filtrate	0.0	0.0%	1.00	39.3	0.74	0.0
16	Ni Concentrate Cake	55.5	90.0%	3.08	20.0	7.40	39.8
17	Spraywater Tank L/C Water	0.0	0.0%	1.00	10.0	0.00	0.0
18	Spraywater Pump Delivery	0.0	0.0%	1.00	164.2	0.00	0.0
Area 400 – Flotation Tail Dewatering							
1	Filtrate Recycle	0.0	0.0%	1.00	927.9	0.03	0.0
2	Tail Thickener Feed	1963.4	28.0%	1.22	5766.5	0.21	39.0
3	Tail Thickener U/F	1963.4	60.0%	1.62	2016.7	0.21	39.0
4	GSW - U/F Pump	0.0	0.0%	1.00	20.0	0.00	0.0
5	Tail Thickener U/F to Filters	1963.4	0.0%	1.00	2036.7	0.21	39.0
6	Tail Thickener O/F	0.0	0.0%	1.00	3749.8	0.00	0.0
8	Tail Filter Feed	1963.4	59.6%	1.62	2036.7	0.21	39.0
9	Tail Filter Filtrate	0.00	0.0%	1.00	897.9	0.03	0.0
10	Level Control Water	0.0	0.0%	1.00	30.0	0.00	0.0
11	Filtrate to Thickener	0.00	0.0%	1.00	927.9	0.03	0.0
12	Tail Filter Cake	1963.4	82.0%	2.10	1138.8	0.21	39.0

17.3 Equipment List

Table 17-3 below provides a summary of the mechanical equipment items selected for the 45,000 tpd mineral processing plant:

Table 17-3: Mechanical Equipment Listing

Equip. ID	#	Name	Description	kW Installed
100-BAA-005	1	Primary Crush Inload Bin	Partially Lined	
100-XLC-010	1	Overhead crane	10t capacity	
100-XBA-015	1	Rock Breaker	For relining and mantle replacement	
100-XXX-020	1	Dust Reduction System		
100-CJA-025	1	Primary Crusher	60 x 89”; 150mm OSS	
100-BAA-030	1	Surge Pocket	civil construction; 600m^3	
100-ZAD-035	1	BAA-030 Discharge Chute	Lined	
100-FDA-040	1	Belt Feeder	Extra Heavy Duty, 8’ x 28’	
100-ZAA-045	1	FDA-040 Discharge Chute	Incl. rockboxes for lining	
100-AAG-055	1	Dust Extraction Unit	Cold weather package; sprays and extraction	
100-EEA-060	1	Tramp Magnet	Cross Belt, Self cleaning	
100-ZAA-065	1	Magnet Scrap Bin		
100-PCD-070	1	Area Spillage Pump	pumps to pit	
100-XXX-075	3	IR Heaters	Local heating	
Power Draw – Area 100				**635**
150-FCV-005	1	Stockpile Feed Conveyor	full cladding on conveyor, none on stockpile	
150-CSP-010	1	Conical Stockpile	55,000 tonne total capacity uncovered	
150-FDA-015	4	Belt Feeders	1.2m x 4.8m integral dribble chute	
150-ZAA-035	4	Stockpile Discharge Chutes	AR450 Lined	
150-ZAA-055	4	FDA-015 Feeder Discharge Chutes	Rockbox, Rail and AR400	
Power Draw – Area 150				**480**
200-PCE-005	1	Ball Bunker Spillage Pump	Submersible	
200-XLC-010	1	Ball Loading Crane	Monorail, 2 tonne cap.	
200-EEC-015	1	Ball Loading Magnet	1200mm diameter	
200-FCV-020	1	SAG Mill Feed Conveyor	Incl. Cladding	
200-MSA-025	1	SAG Mill	38' diameter x 19' EGL	
200-AAA-030	1	SAG Mill Lube System	incl	
200-ZAA-035	1	Discharge Chute	AR400 Lined	
200-ZAD-040	1	Screen Feed Box	MSRL 20mm and AR400 Lined	
200-SVE-045	2	Scalping Screens	12' x 24' Low head, single deck, 25mm cut	
200-ZAC-055	1	Screen Underpan	MSRL, 6mm	
200-FCV-060	3	Pebble Recycle Conveyors	45m, running slow, Extra heavy duty	

Equip. ID	#	Name	Description	kW Installed
200-ZAA-065	3	Conveyor Head Chutes		
200-BAA-080	1	Pebble Crusher Feed Bin	200 tonne capacity (120 m3)	
200-FCA-085	1	Vibrating Feeder		
200-ZAA-090	1	CCB-095 Feed Chute		
200-CCB-095	1	Pebble Crusher	MP800	
200-ZAA-100	1	CCB-095 Discharge Chute		
200-TAA-115	1	Mill Discharge Tank	MSRL, 12mm, 500m^3	
200-PCB-120	3	Mill discharge Pumps	550MMC (4,500 m^3/h @45m TDH)	
200-XLC-135	1	Mill Area Overhead Crane	25t capacity	
200-PCB-140	1	SAG Mill Spillage Pump	Submersible SHW 100-375	
200-MZA-145	1	SAG Mill Liner Handler	7-Axis	
200-MBB-150	2	Ball Mills	26' x 40.5'EGL o/f c/w trommel. Dual pinion	
200-XXX-155	2	Ball Mill Barring gear	supplied with mills	
200-XXX-160	2	Ball Mill Lubrication System	supplied with mills	
200-XXX-165	2	Ball Mill ring gear lubrication	supplied with mills	
200-YAA-170	2	Ball Mill Cyclone Clusters	700CVX10 (8+2), automated valves	
200-XLC-175	1	Ball loading crane	electric winch	
200-PCB-180	1	Mill Area Spillage Pump	Submersible, SHW 100-375	
Power Draw – Area 200				**51,727**
300-TAA-005	2	Conditioner Tanks	5min r/t. MSRL6mm, baffles @120 deg. 185m^3	
300-XSA-010	2	Conditioner Agitators	35 rpm, 2m dia blades, rubber lined	
300-XDB-035	2	Primary Samplers		
300-XDA-040	2	Secondary Samplers		
300-XXX-045	1	Particle Size Analyser	2 Streams - A and B mills	
300-XCB-050	6	Bulk Rougher Cells	500m^3 Tank Cell incl. air flow& L/Control	
300-TAA-080	1	Rougher Conc Tank	60m^3 MSRL; Conical Bottom. 90 sec r/time	
300-PCB-085	2	Rougher Conc Pumps	16/14 G-AH - 3,000 m^3/h (FF=2.0)	
300-ZAD-095	1	Rougher Tail Discharge Launder	MSRL	
300-XDB-100	1	Rougher Tail Sampler (prim)	2200mm cutter width	
300-XDA-105	1	Rougher Tail Sampler (sec)	rotary vezin	
300-TAA-110	1	Rougher Tail Tank	80m^3 MSRL Conical Bottom; 90 sec r/time	
300-PCB-115	2	Rougher Tail Pumps	pumps up to magsep distributors	

Equip. ID	#	Name	Description	kW Installed
300-XLC-125	1	Overhead Crane	5t capacity	
300-PCD-130	1	Rougher Spillage Pump		
300-XCB-135	4	Cleaner 1 Flotation Cells	300m^3 Tank Cell incl. air flow& L/Control	
300-XCB-155	4	Cleaner Scav Flotation Cells	300m^3 Tank Cell incl. air flow& L/Control	
300-TAA-175	1	Cleaner Scav Conc Tank	50m^3, 90 sec r/ time, MSRL; conical bottom	
300-PCB-180	2	Cleaner Scav Conc Pumps	10/8 EAH, 900m^3/h (FF=2.0)	
300-XDB-190	1	Cleaner Scav Tail Sampler (prim)	Cross cut; 1500mm cutting width	
300-XDA-195	1	Cleaner Scav Tail Sampler (sec)	rotary vezin	
300-TAA-200	1	Cleaner Scav Tail Tank	60m^3, 90 sec r/time, MSRL; conical bottom	
300-PCB-205	2	Cleaner Scav Tail Pumps	20/18AH - 3,000 m^3/h (FF=2.0)	
300-TAA-215	1	Cleaner 1 Conc Tank	90 sec r/time, MSRL	
300-PCB-220	2	Cleaner 1 Conc Pumps	16/14AH -2,000 m^3/h	
300-XCB-230	4	Cleaner 2 Flotation Cells	130m^3 Tank Cell; incl. air flow& L/Control	
300-TAA-250	1	Cleaner 2 Conc Tank	90 sec residence time, MSRL 6	
300-PCB-255	2	Cleaner 2 Conc Pumps	12/10AH - 1050 m^3/h Pumps to Cleaner 3 (FF=2.0)	
300-XCB-265	1	Cleaner 3 Flotation Cell	TC e130, incl. air flow& L/Control.	
300-ZAD-270	1	CL3 Conc Sample Launder	MSRL 6mm	
300-XDB-275	1	CL3 Conc Sampler	1200mm cutting width	
300-TAA-280	1	Cleaner 3 Conc Tank	90 sec residence time, MSRL 6mm	
300-PCB-285	2	Cleaner 3 Conc Pumps	12/10AH - 1000 m^3/h (FF=2.0)	
300-ZAA-295	1	Sampling Launder	MSRL, drain, weir etc. 1200mm wide	
300-XDB-300	1	Bulk Concentrate Sampler	1500mm cross cut	
300-ACA-305	1	Bulk Concentrate Thickener	22m diameter, 2.4m wall height	
300-PPA-310	2	ACA-305 Underflow Pumps	58m^3/h nominal 100mm hose	
300-TAA-320	1	Storage Tank	150m^3 Cylindrical Tank; MSRL	
300-XSA-325	1	Agitator	20 rpm, 1m dia twin blades.	
300-PCB-330	2	Transfer Pumps		
300-PCB-340	1	Thickener Area Spillage Pump		
Power Draw – Area 300				**8,108**
320-ZAZ-005	1	Head Box	MSRL; Overflows back to Rougher Tail	
320-ZAZ-010	1	First Stage Distributor	Star Distributor, 4 way (c/w isol. Valves)	

Equip. ID	#	Name	Description	kW Installed
320-ZAZ-015	3	Second Stage Distributors	Star Distributor, 4 way (c/w isol. Valves)	
320-XMS-035	12	Low Intensity Wet MDS – Rougher	1500 Gauss, DD, 1.2m dia x 3m long	
320-XMS-095	8	Low Intensity Wet MDS – Cleaner	1200 Gauss, DD, 1.2m dia x 3m long	
320-TAA-135	1	Mag Sep Concentrate Tank	1800 m^3/h MSRL, 50m^3 live volume	
320-PCB-140	2	Mag Conc Pumps	16/14 GAH, 1,800 m^3/h	
320-ZAA-150	1	Mag Sep Tails Launder	MSRL, 2m wide	
320-XDB-155	1	Mag Sep Tails Sampler (#1)	X-cut. 2200mm sweep.	
320-XDA-160	1	Mag Sep Tails Sampler (#2)	Rotary Vezin, single cutter	
320-TAA-165	1	Mag Sep Tails Tank	2300m^3/h MSRL, 60m^3 live volume	
320-PCB-170	2	Mag Tails Pumps	2300 m^3/h pumps to tails area	
320-PCD-180	1	Mag Sep Spillage Pump		
320-XCB-185	6	Mag Rougher Flotation Cells	300m^3 Tank Cells, incl. air flow& L/Control	
320-ZAA-215	1	Mag Rougher Tails Launder	MSRL, 6mm	
320-XDB-220	1	Mag Rougher Tails Sampler		
320-TAA-225	1	Mag Rougher Concentrate Tank	45m^3. 1,800m^3/h, 1.5min res time, MSRL	
320-PCB-230	2	Mag Rougher Conc Pumps	16/14 GAH, 1,800 m^3/h	
Power Draw – Area 320				**3,678**
400-YAA-005	1	Mag Scav Guard Cyclone	1,410 m^3/h 20um cut point	
400-TAA-010	1	Cyclone Underflow Tank	600 m^3/h 2 min residence time	
400-PCB-015	4	Cyclone Underflow Pump	200 m^3/h pumps to HIG Mills	
400-MBB-035	3	Mag Scavenger Regrind Mill	HIG Mill HIG5000/35000(F)	
400-TAA-050	1	Mill Product Tank	Receives ground slurry and COF	
400-PCB-055	2	Mill Product Pumps	Pumps to Mag Rougher	
400-PCD-065	1	Regrind Area Spillage Pump		
400-XXX-070	1	Ceramic Media Addition System	Auto recovery and addition of media to 3 mills	
400-YAA-075	1	Guard Cyclone	25um cut point	
400-TAA-080	1	Cyclone Underflow Tank	Cylindrical,	
400-PCB-085	2	Cyclone Underflow Pumps	pump through HIG Mill	
400-MBB-095	1	Rougher Conc Regrind Mill	HIG Mill HIG3500/23000(F)	
400-TAA-100	1	Mill Product Tank		
400-PCB-105	2	Mill Product Pump	Pumps to cleaner 1	
400-PCD-115	1	Regrind Spillage Pump		

Equip. ID	#	Name	Description	kW Installed
400-XXX-120	1	Ceramic Media Addition System	Auto recovery and addition of media to HIG3500	
400-XXX-125	1	PSA System	Particle size analyser - 2 streams	
Power Draw – Area 400				**17,705**
450-TAA-005	1	Aeration Tank	25m^3 - Provide 10 min aeration time; MSRL	
450-XSA-010	1	Aeration Tank Agitator	20 rpm	
450-PCB-015	2	Cu Rougher Feed Pumps	6/4 CAH	
450-PCB-020		Cu Rougher Feed Pump S/by	6/4 CAH	
450-XCB-025	3	Copper Rougher Cells	30m^3 Tank Cells, incl. air flow& L/Control.	
450-XCB-040	3	Copper Rougher Scav Cells	30m^3 Tank Cells, incl. air flow& L/Control.	
450-PCF-050	2	Copper Rougher Conc Pumps	VF200	
450-TAA-060	1	Copper Rougher Tails Tank		
450-PCB-065	2	Copper Rougher Tails Pumps	6/4 CAH	
450-XCC-075	3	Cleaner 1 Column Cell	1.8m diameter x 13m h	
450-TAA-090	1	Cleaner 1 Concentrate Tank	400m^3/h incl. FF, 10m3 capacity	
450-PCB-095	2	Cleaner 1 Concentrate Pumps	8/6 AH rubber (FF=2.0)	
450-XCB-105	4	Cleaner 1 Scavenger Cell 1	20m^3 Tank Cells, incl. air flow& L/Control.	
450-TAA-125	1	Cleaner 1 Tails Tank	600m^3/h with FF, 15m^3 capacity	
450-PCB-130	2	Cleaner 1 Tails Pumps	8/6 AH rubber	
450-XCC-140	2	Cleaner 2 Column Cells	1.5m diameter x 13m h	
450-TAA-150	1	Cleaner 2 Concentrate Tank	200m3/h with FF, 8m3 capacity	
450-PCB-155	2	Cleaner 2 Concentrate Pumps	6/4 CAH rubber	
450-TAA-165	1	Cleaner 2 Tail Tank	200m3/h with FF, 8m3 capacity	
450-PCB-170	2	Cleaner 2 Tail Pumps	6/4 CAH rubber	
450-XCC-165	1	Cleaner 3 Column Cell	1.5m diameter x 13m h	
450-TAA-170	1	Cleaner 3 Tail Tank	200m3/h with FF, 8m3 capacity	
450-PCB-175	2	Cleaner 3 Tail Pumps	6/4 CAH rubber	
450-TAA-185	1	Cleaner 3 Concentrate Tank	200m3/h with FF, 8m3 capacity	
450-PCB-190	2	Cleaner 3 Concentrate Pumps	6/4 CAH rubber	
Power Draw – Area 450				**939**

Equip. ID	#	Name	Description	kW Installed
500-ZAB-005	1	Sampling Launder	MSRL6mm, drain, weir etc. 1000mm wide	
500-XDB-010	1	Concentrate Sampler	1200mm cross cut	
500-ACA-015	1	Concentrate Thickener	10m diameter, 2.4m wall height	
500-PPA-020	2	Underflow Pumps	6.6m^3/h; 2 x 80mm hose pumps	
500-TAA-030	2	Storage Tank A	50m^3 Cylindrical Tank	
500-XSA-035	2	TAA-030 Agitator	20 rpm, 1m dia blades. Twin blades	
500-PCB-050	2	Filter Feed Pumps	4/3 AH with mechanical seal	
500-PCB-060	1	Thickener Area Spillage Pump		
500-AAA-065	1	Pressure Filter	Larox PF19M12 1 60	
500-XXX-070	1	Pressure Filter Ancillaries	pumps, compressor, receiver etc.	
500-FCV-075	1	Transfer Conveyor	750mm wide, 30m long, 1m vertical lift	
500-PCB-080	1	Filtrate Tank	unlined	
500-PCB-085	2	Filtrate Pumps	pumps filtrate back to thickener	
500-XXX-100	4	Infrared heaters	Local heating	
Power Draw – Area 500				**293**
550-ZAB-005	1	Sampling Launder	MSRL6mm, drain, weir etc. 1000mm wide	
550-XDB-010	1	Ni Concentrate Sampler	1200mm cross cut	
550-ACA-015	1	Ni Concentrate Thickener	22m diameter, 2.4m wall height	
550-PPA-020	2	Underflow Pumps	58m^3/h 2x100mm hose	
550-TAA-030	2	Storage Tanks	150m^3 Cylindrical Tank	
550-XSA-035	2	Agitators	20 rpm, 1m dia blades. Twin blades	
550-PCB-050	2	Filter Feed Pump A		
550-PCB-060	1	Thickener Area Spillage Pump		
550-AAA-065	1	Pressure Filter	Larox PF132/144M60 1 60	
550-XXX-070	1	Pressure Filter Ancillaries	pumps, compressor, receiver etc.	
550-FCV-075	1	Transfer Conveyor	1.5m wide, 1.2m/s 20 deg. Idlers	
550-PCB-080	1	Filtrate Tank	Unlined, 100m^3	
550-PCB-085	2	Filtrate Pump	pumps filtrate back to thickener	
550-XXX-100	4	Infrared heaters	Local heating	
Power Draw – Area 550				**724**
580-ZAA-005	1	Feed Launder	MSRL	
580-XDB-010	1	Tailings Sampler 1#		
580-XDB-015	1	Tailings Sampler 2#		
580-ZAA-020	1	Splitter Platework		

Equip. ID	#	Name	Description	kW Installed
580-ACA-025	2	Tailings Thickener	45m diameter, autodil, rake lift, 15g/t flocc	
580-PPA-030	4	Underflow Pump		
580-PCB-055	4	Tailings Discharge Pump		
580-PCD-065	1	Tailings Spillage Pump		
580-TAA-070	1	Final Tailings Tank or Filter feed	200m3 capacity	
580-XSA-075	1	TAA-070 Agitator		
580-XXX-100	4	Infrared heaters	Local heating	
Power Draw – Area 580				**779**
600-TAA-005	1	Spray Water Tank		
600-PCB-010	2	Spray Water Pumps		
600-TAA-020	1	Process Water Tank	insulated	
600-PCC-025	2	Process Water Pumps		
600-XXX-035		Process Water Overflow Dam	Earth Dam	
600-PCC-040	2	GHP Pump	water	
600-PEA-050	1	Fire-water Pump	Diesel system with electric jockey pump	
600-PCC-055	2	Clean Water Pump		
600-HAC-065	2	Plant Air Compressor	1 running, 1 standby	
600-HBB-075	4	Flotation Air Blower A	3 running, 1 standby	
Power Draw – Area 600				**2,217**
700-XXX-005	1	PIBX Mixing/Storage/Dosing System	650 tonnes per annum	
700-XXX-010	1	3477 Storage/Dosing System	1460 tonnes per annum	
700-XXX-015	1	MIBC Storage/Dosing System	2000 tonnes per annum	
700-XXX-020	1	CuSO4 Mixing/Storage/Dosing System	810 tonnes per annum	
700-XXX-025	1	CMC Mixing/Storage/Dosing System	570 tonnes per annum	
700-XXX-030	1	SMBS Mixing/Storage/Dosing System	810 tonnes per annum	
700-XXX-035	1	Lime Mixing/Storage/Dosing System	1380 tonnes per annum	

Equip. ID	#	Name	Description	kW Installed
700-XXX-040	1	Act. Carbon/Storage/Dosing System	400 tonnes per annum	
700-XXX-045	1	Sulphuric Acid System	178000 tonnes per annum	
700-XXX-050	1	Flocculant System	630 tonnes per annum	
700-XXX-055	1	Nitric Acid System	4000 tonnes p.a.	
700-XLC-070	1	Reagent Area Crane	5t	
700-XLC-075	1	Reagent Area Crane	15t	
700-PCD-080	8	Spillage Pumps	8 pumps	
Power Draw – Area 700				**985**
1000-XXX-005		Plant Offices & Workshops	Incl. Control room and ablutions	
1000-XXX-010		HVAC Systems		
1000-XXX-015		Courier System		
1000-XXX-020		Stormwater System		
Power Draw – Area 1000				**125**
800—005		Distribution Tanks and Pipes	to feed 30 machines	
800—010		Tailings Filters	25 machines, CX12-204	
800—015		Tailings Filters (Spare)	5 machines, CX12-204	
800—020		Cake Chutes	between filters & conveyors	
800—025		Discharge Conveyors	6 conveyors, 40m each	
800—030		Transfer Conveyor	2 conveyors, 100m each	
800—035		Filtrate Systems	6 sets	
800—040		Acid Systems	6 sets	
--000		Load Out System	2 off, bins, truck drive through etc.	
Power Draw – Area 800				**3215**

The process plant includes equipment with a total installed power rating of 91,300kW. Applied power is calculated to be 73,750 kW, or approximately 34 kWh/t

17.3.1 Summary

The process plant design considers a nominal 45,000 tpd throughput. 150-tonne mine haul trucks would tip into a single gyratory crusher station designed with a planned 80% circuit availability. Should the crusher feed pocket be full on arrival, the haul truck would dump run-of-mine material on the ground for later reclamation.

Surge capacity between the primary crusher station and the mill circuit would be handled by a single 30,000-tonne coarse rock stockpile. Material would then be withdrawn from the stockpile, in a controlled manner, using apron feeders. SAG mill feed control would consist of variable speed feeders with mill feed size distribution measurement. The SAG mill circuit would include a pebble crusher to assist with the comminution of critical size material that could otherwise accumulate within the SAG mill.

Secondary milling to a product size of 80% -110 microns would be achieved by a pair of ball mills, operating in parallel. The two ball mills operate in closed circuit with hydro cyclones and discharge into a common mill sump. SAG mill discharge slurry would also gravitate from the SAG mill via a scalping screen into the mill sump.

The contents of the mill sump would be diluted with process water and then pumped to the two parallel cyclone clusters associated with the two ball mills.

Cyclone overflow slurry would gravitate from the cyclones through a sampling station and into a pair of surge/conditioning tanks ahead of the rougher flotation circuit. The rougher flotation plant would consist of six large tank cells in series, with each cell having independent air flow and pairs of cells having separate pulp level control.

Rougher flotation concentrate would be reground in a vertically stirred mill to a P_{80} of 25 µm and then cleaned in a three-stage cleaner circuit (smaller tank cells) and with a cleaner scavenger circuit. Bulk concentrate from cleaner three would be pumped to the bulk concentrate thickener from where the underflow slurry would either be:

- pumped to pressure filtration equipment for dewatering sale as bulk concentrate, or
- pumped to the copper-nickel flotation circuit for separation of copper and nickel minerals to give separate copper and nickel concentrates

Rougher tailing slurry would be pumped to a magnetic separation plant, consisting of two stages of wet separation at approximately 1500 gauss. The magnetic separators would remove magnetite from the slurry and direct this mineral to a regrind mill and rougher flotation circuit for further recovery of valuable minerals. Magnetite rougher flotation concentrate would be pumped to the first sulphide cleaner scavenger whilst magnetite scavenger tailing slurry would be sampled and pumped to the tailing thickener for dewatering and disposal at the TMF.

The copper nickel separation process that would be included is a selective flotation process that occurs at high pH. Lime and other reagents would be added to the bulk concentrate prior to Cu/Ni separation flotation. The pulp must be aerated for approximately 10 minutes with an addition of activated carbon at this point to facilitate adsorption of any excess sulphide collector. In copper nickel separation, the copper minerals are collected, and the nickel minerals are depressed. Column flotation cells would be used for cleaning the Cu/Ni separation rougher concentrate, and column flotation concentrate would be pumped to the copper concentrate dewatering circuit. Cu/Ni separation scavenger tailing pulp would be pumped to the nickel concentrate dewatering circuit.

The plant would be supplied with water (raw/fresh and process/recycled) from various tanks and dams within the plant.

Reagents would be stored, mixed, and distributed from a central reagents area, adjacent to the plant. Frother, collector, promoter, and depressant etc. would pump from the reagents area to day tanks within in the flotation section, from where peristaltic reagent pumps would accurately dose to the process.

17.3.2 Primary Crushing – Area 100

Run-of-Mine (ROM) would be delivered to the primary tip location by haul trucks at an average frequency of roughly 15 trucks per hour. ROM material would be tipped directly into the primary crusher throat at a peak delivery rate of approximately 2,400 dmtph and would be crushed between the mantle and the concave shell. The throat area would be served by a 2,000 ft.lb-class hydraulic rock-breaker to handle oversize rocks.

The selected primary crusher, a 60-90" gyratory unit, can accept a feed size of up to 800 mm and would run with a 165 mm open side setting. Crushed rock would discharge by gravity into an 800-tonne rail-lined surge pocket, which would provide at least 20 minutes of surge capacity between the crusher and the stockpile. A belt feeder would be used to withdraw crushed ROM from the surge pocket onto a short sacrificial conveyor. This conveyor would discharge onto the main overland conveyor, which in turn would transport material down the valley to the crushed ROM stockpile area.

Spillage and run-off in the primary crusher building would be pumped to surface for appropriate handling. The primary crusher could be served either by a 20-ton capacity overhead maintenance crane or a large mobile crane, should that be available on site.

17.3.3 Stockpile – Area 150

An overland conveyor would transport ROM material from the mine to a stockpile in the plant area. The crushed ROM stockpile would provide a live capacity equivalent to roughly 12 hours of plant production. Mill feed would be withdrawn from the stockpile via four lined discharge chutes and four apron feeders (two operating, two standby). Each apron feeder would be variable speed and capable of providing roughly 80% of the nominal mill feed rate. Each apron feeder would discharge via a chute onto the SAG mill feed conveyor.

17.3.4 SAG & Ball Milling – Area 200

From the stockpile discharge apron feeders, mill feed would be withdrawn in controlled quantities onto the SAG mill feed conveyor, which would discharge into the mill feed hopper. Key process variables such as the throughput and the particle size distribution would be monitored using an accurate weightometer and high-speed camera system respectively and controlled using plant control systems.

The SAG mill selected for this plant is 38-foot diameter x 19-foot long, with grate discharge and with a 16,000-kW twin pinion drive system. Fresh ROM rock and water would be added to the SAG mill and slurry and pebbles would exit the mill after passing through the discharge grate or pebble ports onto a vibrating scalping screen. Scalping screen oversize material (consisting mainly of pebbles) would be directed by chute onto the pebble recycle conveyor for crushing and recycling to the mill. Scalping screen undersize slurry (-2 mm) would gravitate into the common mill discharge sump, where it would be mixed with ball mill discharge and further diluted with water.

From the discharge sump, the coarse mill discharge slurry would be pumped to the ball mill cyclones.

SAG mill slurry spillage would be collected in a drive-in sump, and then returned to process by a submersible slurry pump.

The milling area (SAG and ball mill x 2) would be served by a 15t overhead crane. Mill lining replacement would be carried out using a common 5-axis relining machine.

SAG mill grinding media (5" diameter) would be stored in ball bunkers located partway along the mill feed conveyor belt. The bunkers would be served with a small spillage pump and a ball loading crane and magnet. Balls would then be added to mill feed at timed intervals via a ball loading chute.

After SAG milling, the particle size would be further reduced by a pair of conventional, closed-circuit ball mills operating in parallel. Each mill would be 26.5-foot diameter x 40-foot long overflow discharge type. Slurry would be pumped from the common mill discharge sump to two independent clusters of cyclones. Each cyclone cluster would consist of 8 x 28" units and would serve one ball mill. Cyclone underflow of roughly 70% solids would discharge from the hydro cyclone spigot, to the ball mill for more grinding. Each ball mill would be equipped with a 16,000-kW twin pinion drive unit and would discharge ground pulp into the common mill discharge sump.

The overflow from each cyclone cluster would gravitate to a sampling launder and automatic sample cutter before passing into conditioning tanks ahead of flotation. Particle size would be monitored automatically by an online particle size analysis (PSA) system that would provide data viewable in the control room.

Spillage contained in the ball mill area would be pumped to the common mill discharge sump for re-treatment.

Ball mill grinding media would be delivered to the Project site in bulk and would be stored in the ball mill ball bunkers. The ball bunkers would be serviced by a crawl and electric hoist arrangement, allowing balls to be lifted into a kibble using the ball loading magnet, and tipped into the mill feed spout via a rubber lined ball loading chute.

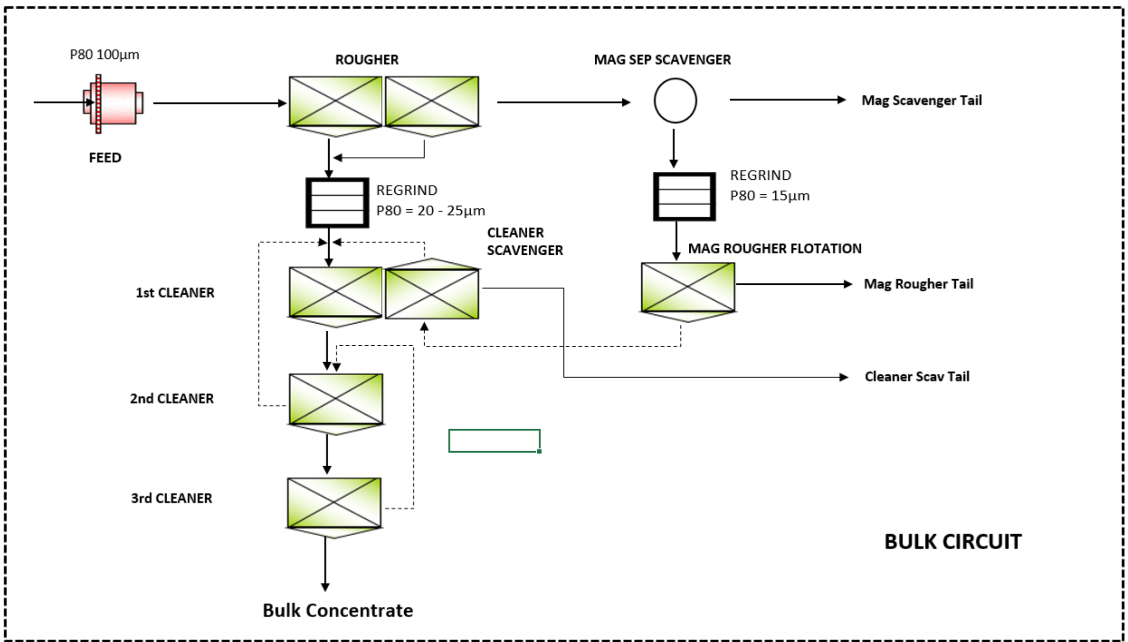
17.3.5 Bulk Flotation – Area 300

Bulk Rougher Flotation

The bulk flotation flowsheet is illustrated in Figure 17-2.

Cyclone overflow pulp would be conditioned in a pair of conditioning tanks – each sized to give six minutes conditioning time ahead of flotation. Each rougher/scavenger bank (four banks total) consists of six 500 m^3 cells operating in series. Flotation air would be supplied by flotation blowers via a low-pressure manifold, and air flow to each cell would be controlled by modulating valves and flow meters. Pulp level within cells would be maintained by modulating dart valves and ultrasonic pulp level instruments.

Figure 17-2: Bulk Concentrate Flowsheet



Source: XPS 2018 MPP - P.25

Bulk rougher concentrate slurry would be collected via HDPE launders and directed into the rougher concentrate pump box. Concentrate slurry would be pumped to the concentrate regrind circuit (Area 400) prior to cleaner flotation.

Spillage in the rougher and regrind sections would be collected in a common sump and pumped back into the first rougher cell using a vertical spillage pump.

Bulk Cleaner Flotation

Reground rougher concentrate would be pumped to the first cleaner circuit, which would consist of 4 x 300 m^3 cleaner tank cells and 4 x 300 m^3 cleaner scavenger tank cells. First Cleaner concentrate would be collected and pumped to the head of cleaner 2, while first cleaner tails would gravitate together with magnetite rougher concentrate into the cleaner scavenger cells. Cleaner scavenger concentrate would be pumped back to the head of the first cleaner and cleaner scavenger tailing slurry would be pumped to one of the tailing thickeners for dewatering and disposal.

Second cleaner feed would consist of first cleaner concentrate plus third cleaner tailing (i.e.: a traditional counter-current cleaner arrangement). Cleaner 2, a collection of 4 x 130 m^3 tank cells, would collect concentrate slurry in a HDPE launder, and would direct it to the second cleaner concentrate pump box. From here it would be pumped up to the head of the third cleaner line for further upgrading. The third cleaner would be a single 130 m^3 tank cell.

Third cleaner concentrate pulp would be collected in a HDPE launder, and would be directed to the final bulk concentrate pump box from where it would be pumped to the bulk concentrate thickener for dewatering. Third cleaner tailing slurry passes by gravity into the second cleaner feed box and second cleaner tailing slurry passes by gravity into the first cleaner feed box in a traditional counter-current manner. The flow of process streams is depicted in the flowsheet summary above.

Low pressure air for bulk flotation would be supplied by several LP compressors, and a low-pressure manifold/distribution system while pulp level in all cells, would be maintained by modulating dart valves and ultrasonic level instruments. Pulp levels and air flowrates would be viewed and controlled from the control room.

Any cleaner area slurry spillage from flotation cells would be collected in bermed areas beneath and directed to the cleaner area spillage pump, which would pump back to the first Cleaner feed box.

Bulk Concentrate Dewatering

Bulk Cu/Ni concentrate slurry would be pumped from bulk cleaner #3 concentrate tank to the bulk concentrate thickener-sampling launder and sampler before entering the concentrate thickener tank for dewatering. This 22 m diameter thickener would be equipped with a rake lift, bed level detection, and bed mass monitoring. Thickener overflow would gravitate to a common spray water tank for recycling into the flotation circuit, while the thickener underflow would be withdrawn from the cone by a centrifugal underflow pump and pumped forward – either to storage tanks ahead of the copper nickel separation circuit, or to the nickel concentrate pressure filter for dewatering and sale as a bulk concentrate.

The bulk concentrate thickener would be located outdoors, with the following heat loss prevention included:

- underflow pump area sheeted and heated
- feed pipe and drive area sheeted and heated
- thickener surface fitted with floating hexagonal cover system

The alternate configuration will be for the bulk flotation concentrate to be directed to copper/nickel separation for sale of separate concentrates.

17.3.6 Magnetite Scavenger Circuit - Area 320

The rougher flotation tailing slurry would be sampled as it discharges the final rougher flotation cell into the rougher tailing tank. From here it would be pumped to a distribution chamber atop the magnetic separation plant. This plant consists of two stages of low intensity wet magnetic separation, with tailing pulp from the second stage being recycled to the rougher tailing tank. Twelve scavenger and six cleaner drum separators have been allowed in the design, with scavenger machines utilizing 50% higher gauss than the cleaner machines.

Magnetic separator concentrate would be washed with water down to the magnetite concentrate regrind circuit (Area 400).

Tailing slurry from the scavenger separators would be combined into the magnetite scavenger tailing tank from where it would be pumped to the tailing system for dewatering and storage/disposal.

Once reground, magnetite concentrate would be pumped to the magnetite rougher flotation circuit, consisting of six 300 m^3 tank cells designed to give 50 minutes nominal retention time. Magnetite rougher concentrate slurry would be collected in HDPE launders and directed to the magnetite rougher concentrate pump box. From there, the magnetite concentrate would be pumped to the sulphide cleaner scavenger cells for mixing and further upgrading with sulphide concentrates.

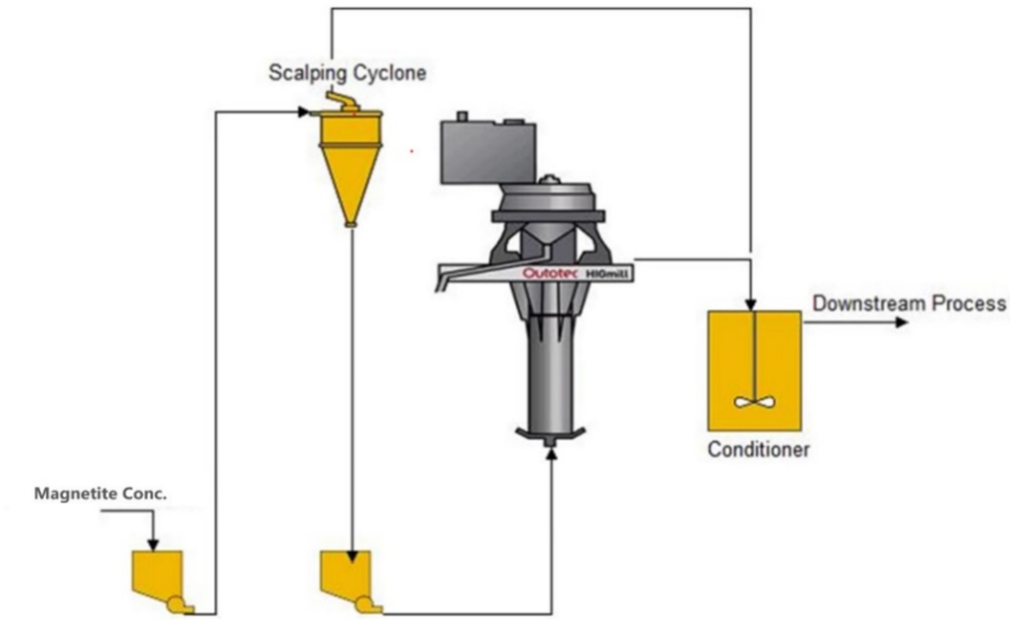
17.3.7 Concentrate Regrinding Circuit – Area 400

The concentrate regrinding area would consist of two independent regrind streams, namely the bulk rougher concentrate regrind circuit and the magnetite concentrate regrind circuit

Magnetite Concentrate Regrind

Magnetic separator circuit concentrate would gravitate to the magnetite regrind circuit, which would grind the concentrate to approximately 80% passing 15 microns. The regrind circuit would consist of a circuit feed tank, a cluster of scalping cyclones, three 5MW vertically stirred mills in parallel, and an agitated product tank (Figure 17-3). The 3 HIG Mills would be served by a single ceramic media addition system.

Figure 17-3: Typical HIG Regrind Circuit



Rougher Concentrate Regrind

Bulk rougher concentrate would gravitate to the bulk rougher regrind circuit, which would grind the concentrate to approximately 80% passing 25 microns. The regrind circuit would consist of a circuit feed tank, a cluster of scalping cyclones, a single 3.5MW vertically stirred mill, and an agitated product tank and would be very similar in configuration to the Magnetite Concentrate Regrind circuit (Figure 17-3). The HIG Mill would be served by a single ceramic media addition system.

Reground concentrate would be pumped to the bulk cleaner flotation circuit (Area 300) for further upgrading.

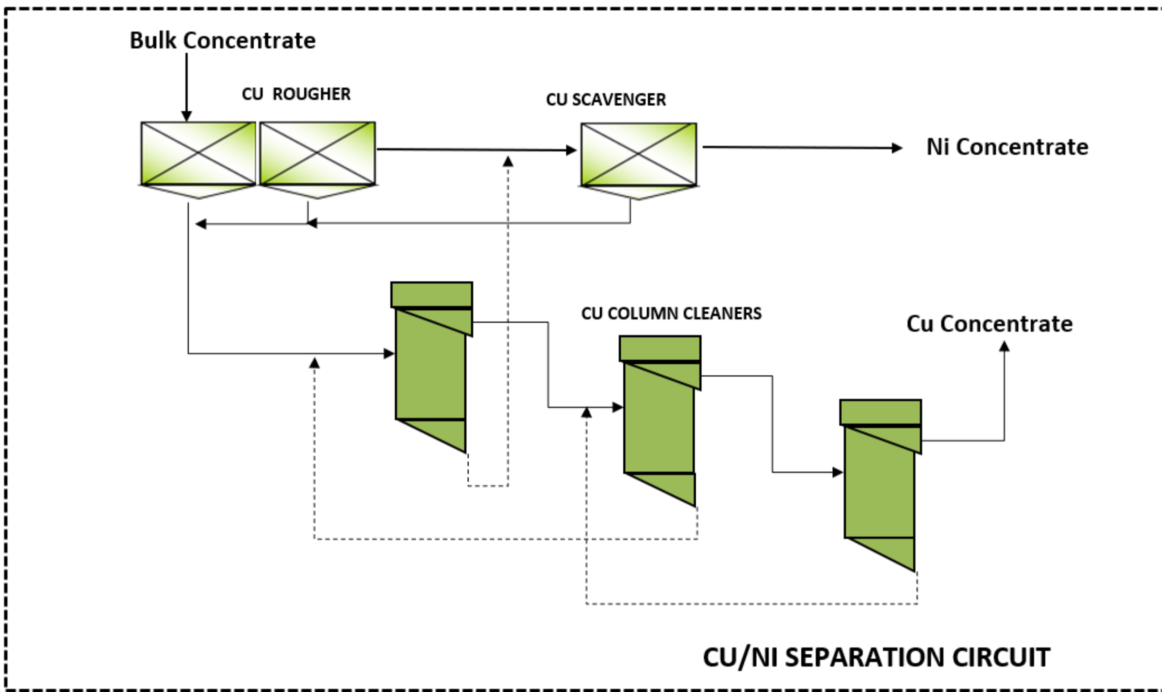
17.3.8 Copper Nickel Separation Flotation – Area 450

The Copper Nickel separation process is illustrated in Figure 17-4 and uses a change to the pulp chemistry to depress nickel and iron sulphide minerals, while allowing copper sulphide minerals to float. In this way, the Cu/Ni separation 'concentrate' becomes enriched in copper, and the Cu/Ni separation 'tailing' stream is depleted in copper, thereby becoming richer in nickel.

The copper nickel separation would be fed from the bulk concentrate thickener underflow surge tank. The rougher/scavenger flotation circuit consists of an agitated and aerated conditioning tank, followed by six 30 m^3 tank cells in series. Rougher and scavenger concentrate would be collected in launders and pumped to the first of three stages of counter-current column type cleaner flotation cells. Cleaner tail slurry from each column would be recycled back to the previous stage using froth pumps, and the concentrate from the third cleaner column would be pumped forward as copper concentrate to the copper dewatering circuit (Area 500). The first column cleaner tailing slurry would be pumped to the copper scavenger feed box.

Tailing slurry from the copper scavenger cells would be low in copper and therefore suitable as the nickel concentrate. Thus, copper scavenger tailing slurry would be pumped to the nickel dewatering circuit (Area 550) for further treatment.

Figure 17-4: Copper/Nickel Separation Flowsheet



Source: XPS 2018 MPP - P.49

17.3.9 Copper Concentrate Dewatering – Area 500

Final copper concentrate slurry would be pumped from the copper concentrate tank (Cu/Ni separation circuit) to the copper concentrate thickener sampling launder and sampler before entering the thickener tank for settling and dewatering. This 10 m-diameter thickener would be equipped with a rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to a common spray water tank for recycling as flotation sprays, while the thickener underflow would be withdrawn from the cone by a centrifugal underflow pump and pumped to the mechanically agitated storage tank ahead of the copper pressure filter.

Copper concentrate slurry would be pumped from the storage tank to the pressure filter for final dewatering (reduction from 45% moisture down to approximately 10% moisture). Filtrate from the pressure filter would be recycled back to the copper concentrate thickener for recycling and additional feed dilution. Auxiliary systems, such as membrane squeeze water and core-blow air equipment, are located on a lower floor and are controlled entirely from the pressure filter control system.

Copper filter cake would be discharged automatically from the press via two cake discharge chutes onto the cake transfer belt, which transfers cake to the concentrate storage shed. A front-end loader serves the cake stockpile and loads cake into side-tipping trucks that transport the concentrate to a toll smelter. Trucks are weighed and auger-sampled at the weighbridge prior to dispatch.

Copper concentrate dewatering area spillage would be recovered from the area floor via vertical spindle pumps and pumped back to the copper concentrate thickener feed launder.

The copper concentrate thickener would be located outdoors, with the following heat loss prevention included:

- underflow pump area sheeted and heated
- feed pipe and drive area sheeted and heated
- thickener surface fitted with floating hexagonal thermal cover system

17.3.10 Nickel Concentrate Dewatering – Area 550

Final nickel concentrate slurry would be pumped from the nickel concentrate tank (Cu/Ni separation circuit) to the nickel concentrate thickener-sampling launder and sampler before entering the thickener tank for settling and dewatering. These 22 m-diameter thickeners would be equipped with a rake lift, bed level detection, and bed mass monitoring. Thickener overflow gravitates to a common spray water tank for recycling as flotation sprays, while the thickener underflow would be withdrawn from the cone by a centrifugal underflow pump and pumped to the mechanically agitated storage tank ahead of the nickel concentrate pressure filter.

Nickel concentrate slurry would be pumped from the storage tank to the pressure filter for final dewatering (reduction from 45% moisture down to approximately 10% moisture). Filtrate from the pressure filter would be recycled back to the nickel concentrate thickener for recycling and additional feed dilution. Auxiliary systems, such as membrane squeeze water and core-blow air equipment, are located on a lower floor and are controlled entirely from the pressure filter control system.

Nickel filter cake would be discharged automatically from the press via two cake discharge chutes onto the cake transfer belt, which transfers cake to the concentrate storage shed. A front-end loader serves the cake stockpile and loads cake into side-tipping trucks that transport the concentrate to a toll smelter. Trucks are weighed and auger-sampled at the weighbridge prior to dispatch.

Nickel concentrate dewatering area spillage is recovered from the area floor via vertical pump and pumped back to the nickel concentrate thickener feed launder.

The nickel concentrate thickener would be located outdoors, with the following heat loss prevention included:

- underflow pump area sheeted and heated
- feed pipe and drive area sheeted and heated
- thickener surface fitted with floating hexagonal thermal cover system

17.3.11 Tailings Dewatering – Area 580

The tailings dewatering circuit would consist of two 45 m-diameter thickeners and a number of disc filters. The magnetic scavenger tailing pulp would report to one thickener while the magnetite rougher tailing pulp and the sulphide cleaner scavenger tailing pulp would be pumped to the second tailing thickener. This split thickening strategy would allow subsequent filtration of the much coarser magnetic scavenger tailing pulp and thickened slurry disposal of the much finer reground slurries. The filtered cake would be transported by truck to the tailing storage area and co-mingled with waste rock, while the thickened slurry would be pumped to the same storage facility for containment within the engineered dam walls.

Overflow from each of the two thickener tanks would gravitate to the process water tank for re-use, while thickener underflow would either be pumped to the final tailings tank (second thickener) or to the tailing filters (first thickener).

The two tailing thickeners would be located outdoors, with the following heat loss prevention included:

- underflow pump area sheeted and heated
- feed pipe and drive area sheeted and heated
- thickener surface fitted with floating hexagonal thermal cover system

Underflow slurry from the first thickener would be pumped to the tailing filtration plant – a collection of disc filters and associated auxiliaries. The filters would produce a cake of approximately 18% moisture, and this would be discharged by conveyor onto a stockpile for loading into 150t trucks.

17.3.12 Services – Area 600

Process water would be stored in two insulated 1,200 m^3 tanks and would be distributed to the plant by the process water pumps. Plant hosing/flushing water would be provided by the hose-down water supply pumps.

One process water tank would also be used to feed the diesel-powered fire water pump from a separate (lower) offtake, thus guaranteeing availability.

Raw water would be piped into the plant from nearby wells and would be stored in the insulated clean water tank. From the tank, clean water would be pumped around the plant for use as reagent mixing water, slurry pump gland seal water, and if required, for mill lubrication system cooling.

Plant and instrument air would be provided by two compressors. Air quality would be maintained by a filtration system. Instrument air would be dried using a refrigeration drier. Separate air receivers would be provided for compressed and instrument air lines to allow for surges in demand.

Low-pressure air would be supplied to the flotation plant by a bank of four blowers. The blowers are fixed speed, with manifold pressure controlled using modulating valves on exhaust ports.

Reagents – Area 600

Frother – MIBC

Liquid methyl isobutyl carbinol (MIBC) would be delivered to site in bulk tankers at a rate of 160 tonnes per month. As delivered (full strength), MIBC would be transferred into holding tanks at the reagent storage facility, from where it would be pumped on demand to a head tank within the flotation plant.

From the head tank, MIBC would be dosed to several points by dedicated peristaltic pumps. Each reagent dosage pump would be equipped with a flowmeter on the delivery line to allow precise control of reagent dosage.

Collector 1 – Potassium isobutyl Xanthate (PIBX)

PIBX would be delivered to site in 50-kg bags a rate of 54 tonnes per month. Bags would be fed into automatic bag breakers and discharged into mixing tanks, where the PIBX pellets are mixed with water to a 12% solution strength. Mixed PIBX would be transferred to the Collector 1 storage tank and from there to a head tank above the flotation circuit.

From the head tank, PIBX would be dosed to several points by dedicated peristaltic pumps. Each reagent dosage pump would be equipped with a flowmeter on the delivery line to allow precise control of reagent dosage.

Suitable ventilation will be provided around mixing and storage facilities to prevent the buildup of HS_2 gas – a flammable decomposition product of xanthate mixtures.

Collector 2 - 3477

3477 dithiophosphate collector would be delivered to site in bulk tankers at a rate of 122 tonnes per month. As delivered (full strength), 3477 would be transferred into holding tanks at the reagent storage facility, from where it would be pumped on demand to a head tank within the flotation plant.

From the head tank, 3477 would be dosed to several points by dedicated peristaltic pumps. Each reagent dosage pump would be equipped with a flowmeter on the delivery line to allow precise control of reagent dosage.

Activator – Copper Sulphate

Copper sulphate would be delivered to site in 50-kg bags at a rate of 68 tonnes per month. Bags would be fed into automatic bag breakers and discharged into mixing tanks, where the copper sulphate solids are mixed with water to a 15% solution strength. Mixed activator would be transferred to the activator storage tank and from there to the activator head tank above the flotation circuit.

From the head tank, copper sulphate would be dosed to several points by dedicated peristaltic pumps. Each reagent dosage pump would be equipped with a flowmeter on the delivery line to allow precise control of reagent dosage.

Depressant – Finnfix 150 (CMC)

Finnfix 150 is a carboxymethyl cellulose powder that would be delivered to site in 1000-kg bags at a rate of 47 tonnes per month. Bags would be hoisted over automatic feed systems that discharged metered amounts of powder into mixing tanks, where the CMC solids are mixed with water to a 2% solution strength. Mixed CMC is transferred to the CMC storage tank and from there to the CMC head tank above the flotation circuit.

From the head tank, CMC solution is dosed to various points by dedicated peristaltic pumps. Each reagent dosage pump is equipped with a flowmeter on the delivery line to allow precise control of reagent dosage.

Non-slip floors would be provided in and around the CMC mixing and storage facilities as the main risk with this substance is slippery floors.

Reagent spillage would be pumped to the tailings tank for disposal in the tailings dam. The reagent areas would be served with safety showers.

Flocculant – Magnafloc 10

Flocculant powder would be delivered to site in 1000-kg bags at a rate of 50 tonnes per month and stored in the reagent storage area. Bags would be lifted by the reagent area crane and added to the flocculant powder hopper. Powder would be withdrawn by the flocculant screw feeder and blown through a venturi to a wetting head located on top of the mechanically agitated mixing tank.

From the mixing tank, mixed flocculant can be fed forward to the storage tanks or recycled back into the mixing tank to aid mixing. Once mixed, the flocculant should be left for several hours to hydrate. A storage tank would provide sufficient volume for storage of flocculant while the mixed batch hydrates in the mixing tank.

From the storage tank, flocculant would be pumped directly to the tailings and concentrate thickeners.

Activated Carbon (ACC)

ACC would be delivered to site in 500 kg bags at a rate of 34 tonnes per month. ACC would be transferred to the plant by forklift and added to the process using water to transport it to the dosage point in the Cu/Ni separation plant.

Sodium Metabisulphite (SMBS)

SMBS would be delivered to site in 1000 kg bags at a rate of 68 tonnes per month. It would be transferred to the plant by forklift and added to the process using water to transport it to the dosage point in the Cu/Ni separation plant.

pH Control (Lime)

Lime would be delivered to site in 1000-kg bags at a rate of 115 tonnes per month. Bags would be fed into automatic bag breakers and discharged into mixing tanks, where the lime would be mixed with water to a 12% solution strength. Mixed lime would be pumped around the plant in a ring main system and drawn off at dosage points for addition to the process.

pH Control (Sulphuric Acid)

Sulphuric Acid would be delivered to site in bulk tankers at a rate of 500 tonnes per day. As delivered (96% strength), acid would be transferred into large holding tanks within deep berms at the reagent storage facility. All normal precautions and procedures for handling sulphuric acid would be executed. From the storage tanks, acid would be pumped around the plant via a ring main system. The ring main would be protected from leaks and damage by double walled piping. The sulphuric acid would be metered out of the ring main at the required rate, at dosage points as required by the process.

18 PROJECT INFRASTRUCTURE

The Project is expected to be a large-scale open pit operation due to the nature and tenor of the grade present. This would include the open pit, waste dumps, access roads, process facility, power generation, concentrate storage, camp facilities, and tailings storage.

Location and types of various infrastructure have only been considered to help in determining potential operating costs for use in the resource constraining pit shell.

Waste dumps were expected to be located in close proximity to the open pit and the nearby valleys would be utilized for storage. Assumed hauls were used to generate the open pit cost for use in the resource pit shell determination.

The process facility is assumed to be located near the Alaska Highway for easy access. Material from the mine would be conveyed to the plant for processing. The cost of the conveying is included in the operating cost estimate for the resource pit generation.

The process facility will require significant power for the considered large scale operation and power of this quantity is not readily available near the Project site. LNG power generation is currently in use in Whitehorse to supply power to the local grid. For the process cost, LNG power generation was considered and included the transportation of the LNG from a facility in British Columbia.

It is anticipated that camp facilities will be required for the work force due to the Project's location. While local road infrastructure is excellent, the population is sparse within reasonable commuting distances. This has been considered and the G&A cost used in the resource pit shell generation reflects this reality.

Supplies for mine operation, processing, and the camp need to be transported from larger centres and allowances for freight costs have been considered as well.

The cost of trucking the concentrate to port facilities in Alaska has also been included in the calculation of the net smelter return.

Tailings facilities may be located near the mine in an adjacent valley or further downstream in the flatter areas near the Alaska Highway. A cost allowance for the operating cost of this facility is also part of the cost package used in the resource pit shell development.

The items discussed above are all expected to be detailed in later engineering studies but are not required for this statement of mineral resources. They were only considered to ensure proper costs were applied to generate the resource constraining pit shell.

19 MARKET STUDIES AND CONTRACTS

According to International Nickel Study Group (INSG) the global nickel market demand slightly outpaced supply in 2016 and registered a deficit of 49.7 mt compared with a surplus of 91.4 mt in 2015. The nickel market has been largely oversupplied for the last five years contributing to a significant build of inventory on the London Metal Exchange (LME). The market surplus was largely a result of unprecedented growth in Nickel Pig Iron (NPI) production in China, which was enabled by the Indonesian laterite ore exports. In January 2014, and in accordance with their mining law, Indonesia implemented the ore bans, however, large inventories of laterite stockpiles plus increase in export of laterite ore from the Philippines supported NPI production in China even after the ore bans, at just below peak levels. In January 2017, Indonesia reversed that ban introducing more uncertainty in the nickel outlook while the supply chain was still dealing with the consequences of the over-supply since 2009 which was by-in-large a response to the price run up in the 2006-2007 era to the US$24-25/lb range.

Multi-year deficits will be required to balance the market with improved demand growth rates in stainless steel being a key driver. Between 1980 to 2015, demand for nickel grew at approximately 5% CAGAR with China accounting for over 50% of the demand by 2015. Most analysts believe there will be a return to the long-term growth rates for stainless steel of about 5% but differ on demand growth rates by region for different periods. Prior to the Indonesian ore ban reversal, there was a general expectation that rate of supply growth will not keep up with demand and a draw-down in stocks will ensue and be supportive to the price. Since the ore ban reversal, there is a question as to whether or not the market can sustain the multi-year deficits. Currently, the nickel price in the first half of 2018 has averaged US$6.31/lb. In the long term, higher prices will be required to incentivize the development of nickel projects which will be needed to avoid another price run-up.

19.1 Commodity Price Projection

A review of metal pricing forecasts from many different analysts and bank forecasts in addition to the trailing averages was conducted to determine average long-term prices to be used by Nickel Creek. These are presented in Table 19-1.

Table 19-1: Long Term Price Projection

	Units	Study Price Used	Analyst Minimum	Analyst Maximum	Spot Price July 2, 2018	3 Year Average	5 Year Average	10 Year Average
Nickel	US$/lb	**8.25**	7.50	10.00	6.61	4.83	5.68	7.03
Copper	US$/lb	**3.00**	2.68	3.30	2.99	2.57	2.76	3.02
Platinum	US$/oz	**1,200**	934	1,400	809	961	1,112	1,307
Palladium	US$/oz	**900**	800	1,125	942	764	769	668
Cobalt	US$/lb	**24.00**	24.00	48.67	34.02	20.80	17.91	-
Gold	US$/oz	**1,300**	1,200	1,513	1,240	1,240	1,248	1,287

19.2 Nickel Concentrate Market

Nickel Creek intends to produce marketable Ni and Cu bearing bulk sulfide concentrates. The concentrates will be transported by the existing roads, rail, and port facilities to the smelter(s). A nickel concentrate market has developed in the last 25 years with growing need by smelters to replace nickel sulfide concentrate units as older mines are depleted. Unlike markets such as copper, zinc, and lead, the nickel concentrate market does not have global benchmarks and commercial terms are negotiated individually with the off-take terms between the buyer and seller held confidentially. No contracts have been negotiated for the Project concentrates.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIO-ECONOMIC IMPACT

20.1 Environmental Assessment & Permitting

20.1.1 Overview

The Project occurs within the northeastern portion of the St. Elias Mountain Ecoregion. The physiography is characterized by mountain ranges and alpine plateaus bisected by broad valleys with wide, braided rivers. The climate consists of short, dry summers and long, cold winters. Precipitation is moderated by a rain shadow effect caused by the St. Elias Mountains. Discontinuous permafrost is widespread across the region. Vegetation zones include the Boreal High below 1,080 masl, the Subapline Shrub between 1,080 to 1,230 masl, and the Alpine Tundra zone above 1,400 masl. Dominant vegetation associations include white and black spruce forests, subalpine willow-scrub birch shrublands, and dwarf shrub-graminoid alpine meadows.

The Project is entirely within the Kluane Wildlife Sanctuary, which limits licensed hunting to two Dall's sheep per year, while respecting First Nations subsistence harvesting rights. The Asi Keyi Natural Environment Park is located approximately 3 km east of the Project area and Kluane National Park and Reserve is located 5 km south of the Project area.

Environmental management for the area is primarily focused on, but not limited to, water quality and proximity to sensitive wildlife areas. The Yukon Environmental and Socio-Economic Assessment Act requires baseline environmental studies to be completed and presented to the Yukon Environmental and Socio-Economic Assessment Board (YESAB–Executive Panel) prior to the granting of permitting and licensing for quartz-mining operations and/or activities. Select environmental baseline studies were initiated and have been ongoing since 2012.

The Project occurs in the Traditional Territory of Kluane First Nation. Nickel Creek is committed to implementing and planning environmental activities with the encouraged participation of First Nations through constructive dialogue to understand any unique environmental or Traditional Socio-Economic values that are of concern. Nickel Creek engages Kluane First Nations to assist with environmental work and traditional knowledge to enhance the environmental program.

20.1.2 Project Proposal

Before projects proceed to the mine licensing phase, they are first assessed through an Environmental Assessment (EA). The Yukon Environmental and Socio-economic Assessment Board (YESAB) administer EA's in Yukon. The Project will be subject to an EA under the YESAA conducted by the YESAB Executive Committee. As part of an EA, a project proposal document will be submitted to YESAB that contains (but may not be limited to) i) a description of the project, ii) potential environmental considerations, iii) potential socio-economic considerations, iv) mitigation strategies for any adverse effects of the project, and v) an environmental strategy for management including a closure and reclamation plan. As part of a project proposal, documented records of all correspondence and consultation with any potentially impacted First Nations and qualified specialists (e.g.: biologists, archeologists) are required.

20.1.3 Licensing, Permitting & Regulatory Authority

Two of the most important permits for quartz mining activity to proceed in Yukon are the Quartz Mining Licence (QML), and the Yukon Water Licence (YWL), both requiring an EA through YESAB and the YESAA.

In Yukon, all hard rock (quartz) mining claims are administered through the Quartz Mining Act (QMA). The QMA enables the Government of Yukon to issue licenses and regulate mining developments. The Government of Yukon's Department of Energy, Mines, and Resources administer the QML following review of the EA. Although permits and licenses cannot be issued in advance of completing the assessment, regulatory processes can be initiated simultaneously while the assessment is underway.

The Yukon Water Board is responsible for licensing the use of water and the discharge of wastes into waters within the Yukon Territory under the Yukon Waters Act. The Project will require a Type A water license.

The Project will also be subject to territorial legislation and will require a number of permits and approvals outside the EA, QML, and YWL. In addition, the Project will also be subject to federal legislation depending upon specific project features and details. However, with comprehensive environmental baseline data, Nickel Creek may be able to apply for other legislative permits concurrently with the EA.

20.1.4 Environmental Assessment

The Project requires an Executive Committee screening as it is a quartz mining project that is a proposed metal mine, other than a gold mine, with an ore production capacity of at least 1,500 tonnes per day. Projects assessed by the Executive Committee of YESAB generally require between one and three years (not more than 923 days, including time required for a government decision) to receive a Decision Document that will inform the QML and YWL.

Detailed information requirements for this process are outlined in the Information Requirements for Executive Committee Project Proposal Submissions under the YESAA, which is available through the YESAB office.

Once assessments are complete, recommendations are forwarded to a decision body or bodies.

20.1.5 Existing Permits

Currently, Nickel Creek does not hold any of the permits required to operate a mine. However, Nickel Creek is permitted for advanced year-round exploration activities and carries out its site and exploration activities under the following permits/licences:

- Class 4 Quartz Mining Land Use Approval, LQ00468 (Exploration Activities)
- Commercial Lease, 115G11-003 (Lower Camp, Camp Facilities)
- Industrial Lease, 115G05-001 (Historic Upper Camp, Fuel Infrastructure)
- Sewage Disposal Permit, #4948 (Lower Camp, Septic Field)
- Storage Tank System Permit, 2017-022 (65,000 L Diesel Fuel Tank)
- Storage Tank System Permit, 2017-031 (4,300 L & 4,460 L Diesel Fuel Tanks)
- Building Permit(s), various (for ATCO trailer camp housing facilities)

In addition to seasonally granted permits for the following, when and if necessary:

- Water License (Supplying exploration drilling, kitchen facilities, core cutting)
- Food Premises Permit (Permission to Operate a Food Premise)
- Land Use Permit, 2017-F776 (Permission to maintain off-claim sections of the Quill Creek access road)
- Open Air Burn Permit(s) (for disposal of scrap wood and paper refuse)

20.2 Baseline Environmental Studies & Environmental Operating Plans

20.2.1 Overview

Baseline studies are required to meet two general requirements within the EA process. The first requirement is to document reference conditions for selected environmental factors referred to as 'valued components' (VC) within the Project area. This typically includes physical VC, such as water quality or trace mineral content in vegetation. The second requirement for baseline studies is to collect information that is used to conduct an effects assessment for VC as part of the EA.

Environmental operating plans are mitigation and/or monitoring plans that are designed to mitigate and or monitor the potential effects of the Project on environmental VC.

Contemporary environmental studies were first undertaken by NCP at the Project in 2012, when an initial baseline environmental program was outlined. From 2012 to 2016 intermittent data was collected on select environmental VC (meteorological, snow, soil, ground and surface water, vegetation, fish and aquatics, and wildlife including sheep) until the fall of 2016.

Although these studies provide useful reference information, concerns about the data being outdated, changes to survey standards, and poor spatial overlap with the current Project preclude using that information directly to support an EA.

In September 2016 Nickel Creek upgraded its environmental program to include a comprehensive 2-year environmental baseline study with the assistance of AEG to undertake the physical environmental studies (e.g.: water quality, data, fisheries) and Environmental Dynamics Inc. (EDI) to conduct the terrestrial studies (e.g.: ungulate monitoring studies) and to assist in training of an Environmental Monitor (EM). For certain activities during 2016 & 2017 Hatfield Consultants and local independent consultants were utilized for a variety of different fieldwork. However, from

2018 onwards AEG and EDI have been retained exclusively as Nickel Creek's environmental consultants as qualified environmental scientists and qualified biologists for the development and implementation of environmental baseline studies.

20.2.2 Environmental Studies

Several environmental studies have occurred in the Project area in the past. Limited environmental studies, focussed on caribou and sheep, were conducted for the Project in 1987 and 1988. Baseline studies were also conducted for the adjacent Alaska Highway Gas Pipeline from 1987 to 1990. Environment Yukon has been conducting intermittent surveys for sheep and moose in the area since 1987 and a graduate student conducted additional sheep surveys from 1988 to 1990 in relation to developing management strategies for the Kluane Wildlife Sanctuary.

Prior to initiating baseline studies, Nickel Creek developed a candidate list of environmental VC that the Project had the potential to affect. A desktop assessment was then conducted to collect existing information for each VC and that information was evaluated in terms of its ability to support the requirements for an EA. For VCs where the existing information was determined to be inadequate, one or more specific variables were selected for measurement via environmental studies.

Environmental baseline surveys are required as part of baseline information, and data is required for submission to the YESAB Executive Committee under the YESAA. This is part of the process for the eventual granting of permits associated with increased development and/or land use, and eventual mining. However, data will also be used to obtain other mining activity related permitting (e.g.: water licenses, transport licenses, etc.) in compliance with Territorial Legislation and Federal Legislation.

Components of the comprehensive environmental baseline study and data acquisition methods are summarized below.

Surface Water Quality

Surface water quality in the Project area is assessed by taking water samples at 18 sites conducted monthly. An additional sample regime was undertaken twice per year (seasonal low and high flow conditions) in 2017 and 2018 where 5 samples were collected in a 30-day period sampling event. All water samples are sent to Maxxam Analytics, a local laboratory in Whitehorse, or expedited (testing dependant) to Burnaby, B.C. QA/QC on all data is conducted by AEG.

Groundwater Quality

Groundwater quality in the Project area is assessed by taking water samples monthly at 15 sites, hosting 31 wells on a quarterly basis. Samples are sent to Maxxam Analytics or expedited (testing dependant) to Burnaby, B.C. QAQC on all data is conducted by AEG.

Hydrology

The main objective of the hydrologic monitoring program has been to characterize the timing and magnitude of stream flow at various locations within the Project area. This is undertaken by visiting watercourses at 15 sites monthly for the whole calendar year. Data is focused on non-analytical (i.e.: elemental) parameters of said watercourse, examining characteristics such as flow rate, stream velocity, and depth. QAQC on all data is undertaken by AEG

Vegetation

Vegetation studies include three components: Ecosystem and Landscape Classification (ELC) mapping, rare plants, and traditional use plants. ELC mapping is a standardized process for mapping ecosystem units (communities) across large landscapes based on patterns of vegetation type, vegetation structure, topography, terrain and soil moisture and nutrients. In addition to ELC mapping being a standalone product required by YESAB to assess vegetation communities in an EA, ELC mapping is a critical product for wildlife habitat interpretation, which is used for i) effectively designing baseline wildlife studies and ii) developing wildlife habitat models which are the primary tool used to assess potential Project effects on wildlife in an EA. Due to its requirement as an input for wildlife studies, ELC mapping was prioritized as one of the first terrestrial baseline study components, and that work was completed in 2017. A total of 2,366 map polygons were delineated across the study area, representing 39 unique ecosystem unit types.

Rare plants (i.e.: species of conservation concern) and traditional use plants (i.e.: plants used by local First Nations for subsistence and medicinal use) are additional vegetation VCs that must be considered in the EA.

Wildlife

Fourteen wildlife species or species groups have been identified as candidate wildlife VC for the Project based on their status as either a species of conservation concern or as a prominent game/subsistence species. Of these 14 potential VCs, Nickel Creek anticipates conducting some level of baseline studies for nine of them. Of the remaining five, woodland caribou and moose have adequate existing baseline data available for them from Environment Yukon, mountain goat data is being obtained opportunistically during the thinhorn sheep studies, wolverine can be assessed without local data, and amphibians are proposed not to be carried forward as a VC. The survey methodology and timing of baseline surveys for the nine remaining VCs varies by species. For all wildlife studies Nickel Creek has extended opportunities for consultation and participation to territorial and federal regulators and Kluane First Nations.

To date, the bulk of survey efforts has been allocated to thinhorn sheep, which are considered the most prominent game and subsistence species in the area. The objectives of thinhorn sheep studies include determining seasonal habitat use requirements and seasonal movement patterns, and monitoring population abundance and distribution. Surveys have utilized a variety of ground and aerial-based survey methods and remote camera monitoring across key sensitive seasonal periods: lambing, rutting, and late winter.

Since studies began in 2015 the minimum annual counts have suggested a relatively stable population. The ratios of adult males to nursery sheep and lambs to nursery sheep have been indicative of healthy reproduction rates. Habitat use surveys have been analyzed, in conjunction with the ELC mapping, to develop seasonal habitat models across the Project area.

In 2018, breeding bird studies were also conducted. Those included three types of survey methods for upland songbirds (n= sites), riverine birds (n= sites) and pond and marsh birds (n= sites). Data from the 2018 bird studies is currently being analyzed and will be available in early 2019. Planning for rare moth and butterfly studies was also initiated in 2018. Surveys for the remaining wildlife VCs will be conducted prior to EA submission.

In addition to EA-related baseline studies, Nickel Creek has been conducting two operational monitoring and mitigation programs associated with its ongoing exploration program. One of these programs is a ground-based, thinhorn sheep-monitoring program. The objective of this program is to monitor the occurrence of sheep within and adjacent to the active exploration areas to determine potential interactions between the Project and sheep and, if negative interactions occur, develop management measures to mitigate effects. That program consists of weekly surveys from fixed observation stations within, and adjacent to, the exploration area to map sheep occurrences and potential interactions with Project activities.

The second operational monitoring program is for a golden eagle breeding area north of the main exploration area. No special management measures for the eagles were applied for the 2018 season.

During times of field activity, in addition to the above, Nickel Creek works collaboratively with Kluane First Nations and EDI to train and assist a wildlife monitor that carries out activity/disturbance related duties to ensure the Company has the lowest impact on the natural environment around the Project area.

Physiography

Additional studies to understand the unique physiography of the Project area include a comprehensive light detection and ranging (LiDAR) and orthophoto survey conducted by Eagle Mapping in 2016. These airborne remote sensing techniques provide high quality data about the natural environment, and potential habitat in the Project area.

Snow Pack Surveys

Snow pack surveys are conducted during the winter months (January - April) to characterize the accumulation of snow in the Project area and investigate the changing density of the snowpack throughout the winter months.

Meteorological and Climate Studies (Automated Weather Station)

Weather and climate are examined through a data-logging automated weather station located adjacent to the historic Upper Camp. The station continuously collects data regarding the current weather and has been operational since 2013. Data is periodically downloaded, and QA/QC is undertaken by AEG.

Trace Metals

In 2017 a trace metals study was conducted to document current levels of trace metal concentrations in soil and vegetation across the Project area. This information is required as baseline information for the Project application and may be used as benchmark data for future monitoring. This data can be used in conjunction with the "non-degradative" philosophy to investigate if exploration/mining activities are the potential cause for any elevated concentrations of metals in the Project area.

Heritage & Archeological Studies

To protect the archaeological potential of heritage use (trails, gathering places, etc.) and artifacts, in 2017 Nickel Creek undertook two archaeological studies: i) Heritage Resource Impact Assessment and ii) Heritage Resource Overview Assessment. Both studies covered the Project area and were conducted by ECOFOR. As a result of these studies, all workers and visitors that come to the Project are given a specific point of orientation for procedure to follow when finding a potential archaeological site(s).

Aquatic Resources & Fisheries Study

In order to understand aquatic biology (e.g.: fish, benthic macroinvertebrates, periphyton) locations on the Kluane River and its tributaries (Nickel, Quill, Swede Johnson, Glacier, and Arch Creeks) that represent the watershed catchments, in and around the Project are monitored for baseline purposes and to understand any potential effects that may be caused in the natural environment from the Project. Study methods may include, but are not limited to, minnow trapping, net fishing, electrofishing, and physical property notes of the watercourse.

20.2.3 Environmental Management & Operating Plans

Nickel Creek has developed a number of management plans and work policies that aim to preserve and reduce potential impact to the Project area. These documents are also part of the YESAB/YESAA process as the Project moves toward development, and eventual mine permitting. These management plans (or policies) include, but are not limited to:

- Spill Response and Documentation (2018 – updated yearly)
- Emergency Response and Health and Safety Guidelines (2018 – updated yearly)
- Fish and Wildlife Adaptive Management Plan (2018 – to be updated yearly)
- Sheep Observation/Monitoring Plan (2018 – adaptive, evolves with scope of activities and biologist's discretion)
- Environmental Monitoring/Observation (2018 – adaptive, evolves with scope of activities)
- Waste Management Plan (2018 – updated yearly and with scope of activities)
- Access Management Plan (2018 – updated yearly)
- Chance Find Procedure for Heritage Resources (2018 – updated as needed)
- Petroleum Fuel and Hazardous Substances Policy (2018 – as per Class IV LMU Permit)

- Water Management & Sediment Control (2018 – as per Class IV MLU Permit)
- Reclamation and/or Decommissioning (2018 – as per Class IV MLU Permit)

20.2.4 Final Site Reclamation & Closure

A site reclamation and closure plan will be required as part of the design and Project proposal submission for quartz mining permits. The expectation would be that all facilities would be removed from the site and that surface disturbance would be modified to minimize the impact upon wildlife and other land users. As part of the Project design, the area of disturbance will be minimized and, as much as possible, there will be progressive reclamation activities concurrent with operations. The site reclamation plan will be developed with input from Kluane First Nation that, at a minimum, meets the requirements outlined in the Yukon Government reclamation policy.

Financial assurance must be posted to secure the rehabilitation work, and the determination of the outstanding mine reclamation and closure liability associated with the Project technical features and structures. The Government of Yukon determines the amount and form of financial assurance to be provided by the Company. The Yukon Government will also ensure financial assurance is maintained at all times. Financial assurance will be comprised of an initial payment, prior to commencement of development, and a periodic adjustment to ensure the full amount of financial assurance is held for outstanding reclamation and closure liability throughout the development, operation, and closure of the mine site. Concurrent reclamation may reduce the amount of financial assurance required to be provided and maintained by the Company.

The Company will file an annual report stating what concurrent reclamation has been accomplished during the course of mining and the results of environmental monitoring programs. The Company will monitor the Project to determine the effectiveness of the mitigation measures as progressive reclamation and closure work is completed.

20.3 Socio-Economic Considerations

20.3.1 First Nations & Project Location

The Project and all current infrastructure are located approximately 30 km north west of Burwash Landing, on leased (commercially and/or industrially) Crown Land and within the Traditional Territory of Kluane First Nation and asserted territory of White River First Nation. The current Class IV MLU permit covers 401 active (quartz claims and mineral leases) tenures and the claim block has partial overlap with Kluane First Nation Settlement Land Parcels R-49B (Category B Settlement Land) and R-1A (Category A Settlement Land). However, the Project including all active land tenures, is entirely in Kluane First Nation Traditional Territory and asserted White River First Nation territory. Kluane First Nation is a self-governing nation with a settled land claim agreement.

20.3.2 Communities

The Project is located in western Yukon, within the Whitehorse Mining District. The primary communities potentially affected by the Project and related infrastructure are Burwash Landing, Destruction Bay, Haines Junction, and Beaver Creek, that are approximately, 30, 50, 100 and 150 km from the Project, respectively.

20.3.3 Community Engagement & Consultation

Community engagement and consultation are core values of Nickel Creek, so that the Project advances responsibly towards mine development and ultimately, mine production. As part of that responsibility, dialogue between Nickel Creek and the local community members must include discussions regarding environmental concerns, health and safety, socio-economic benefits, and other potential effects advancing the Project may have.

Nickel Creek initiated engagement with Kluane First Nation and White River First Nation regarding the Project beginning in 2010. As the relationship between Nickel Creek and Kluane First Nation developed, an Exploration Co-operation Agreement (ECA) was signed on August 1, 2012. This document reflects the Company's commitment to recognize concerns of the community, to ensure that socio-economic benefits from the Project that include employment, training, skill progression, etc. is flowing from the Company to the local community, and to recognize the mine will be on Kluane First Nation Traditional Territory. As part of this commitment, Kluane First Nation and Nickel Creek have regular ECA meetings to discuss changing or new community concerns. The Kluane First Nation is regularly consulted on company policy and management plans for their input.

21 CAPITAL AND OPERATING COSTS

These items are not applicable at this stage of the Project in this Statement of Mineral Resources.

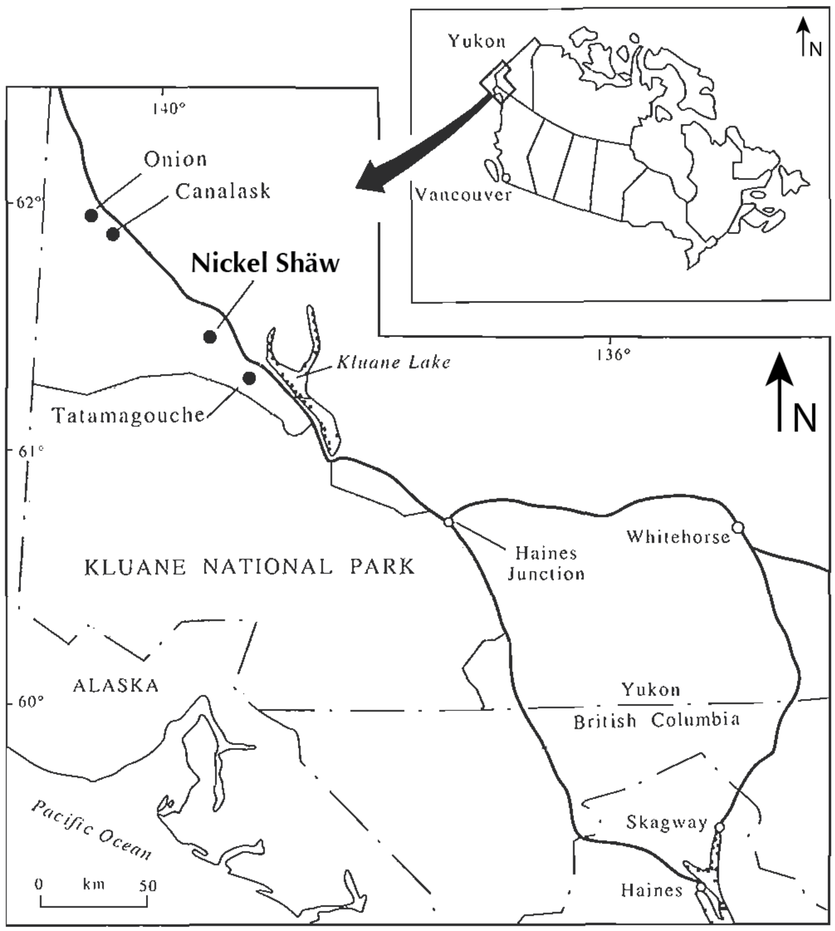
22 ECONOMIC ANALYSIS

These items were not determined nor applicable at the current stage of the Project for this Statement of Mineral Resources.

23 ADJACENT PROPERTIES

Several mafic-ultramafic bodies similar to the Project, extend in a northwest to southeast trend in the southwestern part of the Yukon. The location of these deposits relative to the Project are shown in Figure 23-1. Information for this section has been summarized from Hulbert, 1997, and none of the QP's involved in this report have verified the data presented below or have they visited the properties, and the information provided in this section is not indicative of the mineralization at the Project.

Figure 23-1: Location Map of Ni, Cu and PGM Deposits Discussed in this Section



Source: Hubert, 1997

23.1 Onion

The Onion property is located approximately 80 km northwest of the Project and is hosted within the White River Intrusive Complex. The White River Intrusive Complex is the second largest mafic-ultramafic body in the Kluane belt and is 16 km long. The Onion property was discovered in 1952 and was staked for Prospectors Airways Ltd. Work on the property, up until the time of Hulbert's report (1997), included prospecting, mapping, hand trenching, and geophysics. The property consists of a sill like body of peridotite that intrudes volcanic breccias of the Pennsylvanian Station Creek Formation. The sill is approximately 3 km long, ranges in thickness from 100 to 150 m, and dips at approximately 50 degrees to the southwest. A zone of quartz carbonate alteration rims the sill and reaches thicknesses of 50 m. Ni-Cu-PGE mineralization has been noted at four locations at the lower contact of the sill with the Station Creek volcanics. The Onion's southwest showing contains malachite and minor limonite that assays up to 19.2% Ni, 0.02% Cu, 100 ppb Pd, 50 ppb Pt and 4100 ppb Au. The Discovery showing contains semi-massive to massive pyrrhotite-pendlandite-chalcopyrtie bands that are up to 10 cm thick. Samples from the Discovery showing assay as high as 4.5% Ni, 0.9% Cu, 1700 ppb Pd, 2000 ppb Pt and 56 ppb Au (Hulbert, 1997).

23.2 Canalask

The Canalask property is located approximately 70 km northwest of the Project in the White River Intrusive complex. The property was discovered in 1952 and exploration on the property up to 1997, included 7,317 m of drilling, 518 m of underground drifting, surface sampling, mapping, and geophysics. The length and width of the deposit has been largely determined from the drilling and trenching due to poor surface exposure. The exploration area covers a strike length of 2.7 km and averages 430 m in width. Geophysics and drilling indicate the intrusion has not been significantly folded or faulted and dips approximately 45 degrees to the southwest. Massive sulfide mineralization is found in the footwall within the Station Creek Formation. A 20 m thick zone of mixed mineralized gabbro and sediments is located next to the massive sulfide mineralization and extends eastward for 1 km. The mineralization within this zone is 5 to 8 m thick and contains 5 to 8% disseminated sulphides with occasional net-textured sulphides. This zone assays as high as 0.78% Ni and 0.27% Cu. The massive sulfide mineralization is discordant to the country rock–intrusion contact and is hosted within bedded andesitic tuffs and volcanic breccias. Much of the early work on the deposit has been lost, but available maps show Ni grades as high as 6%. Assays from the gabbro-sedimentary mineralization are also limited but 0.92% Ni and 0.22% Cu have been reported (Hulbert, 1997).

23.3 Tatamagouche

The Tatamagouche Creek Intrusive Complex is the largest mafic-ultramafic body in the Kluane mafic-ultramafic belt. The intrusion is located 40 km southeast of the Project and is 14 km west of Burwash Landing. The property was first staked in 1952 and 998 m of drilling, mapping, surface sampling, and geophysical surveys had been completed on the Project area at the time of

Hulbert's report. The mafic-ultramafic rocks are poorly exposed and the complex has seen limited exploration relative to other deposits in the belt. The intrusion is 17 km in length and is 1.4 km wide in the central portion and thickens to 3 km wide near the northwest and southeast ends. The northern end of the complex intrudes on the Station Creek and Hasen Creek Formations. The southern end of the complex is cut by a latter granitic intrusion. Chilled margins within the ultramafic body and xenoliths of ultramafic rocks indicate this is an intrusive contact. A grab sample of the marginal gabbro containing 50% sulfides assayed at 3.6% Ni and 0.7% Cu. Diamond drilling was conducted near the sample location but the results are unavailable. Two diamond drill holes targeting a geophysical anomaly west of the showing yielded disappointing results. Grab samples near Tatamagouche Creek yielded assays as high as 1.1% Ni and 2% Cu from semi-massive sulfides. Gabbro samples from this site yielded results as high as 0.32% Ni and 0.85% Cu (Hulbert, 1997).

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information for this report.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Resource

The Project is a large tonnage deposit of Ni-PGM-Cu mineralization that has potential for large scale bulk mining. There is sufficient drilling, interpretation, process concept testing, and analysis to estimate a mineral resource.

The mineral resource for the Project was developed using a computer based block model of the deposit using conventional resource modeling techniques based on 386 reliable drill holes. The geologic interpretation was completed by Nickel Creek Chief Geologist James Berry after review and verification of that information by the QP (John Marek). Mineral resources were estimated using the block model and the Lerchs-Grossman open pit software to establish the component of the deposit with reasonable prospects of economic extraction.

The final statement of mineral resources reflects material that is inside of a computer-generated pit. The purpose of using Lerchs-Grossman is to provide some assurance that the mineral resource has “reasonable prospects of economic extraction” as required by CIM best practices.

Table 14-4 summarizes the resulting mineral resources and Table 14-1 and Table 14-3 summarize some of the economic assumptions that were used for this Mineral Resource calculation. The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be realized, or that they will convert to mineral reserves. John Marek of IMC is the QP for this statement of mineral resources. Currently there is no mineral reserve at the Project.

The risks associated with this statement of mineral resources include, metal price impacts, changes to process recovery as more testing is complete, and permit risks that are typical of any North American mineral development.

The current process concept envisions a large-scale process facility that produces and markets a bulk nickel concentrate and a bulk copper concentrate. To capture the potential economic contributions of multiple metals and process recovery formulas, an NSR value was estimated for each mineralized block and used for cut-off application. The internal or marginal mill cut-off is equal to the sum of the process, G&A, and tailings management operating costs because the NSR value considers process recoveries, assumed smelter terms, and concentrate transport costs. The process recoveries vary with total Ni, Cu and sulphur content which were applied to the block model on a block by block basis.

There is potential to add to the Project resource tonnage with additional drilling, and there are additional exploration targets on the property controlled by Nickel Creek to the east and west.

25.2 Metallurgy

Composite samples from the Project have been submitted for several metallurgical testwork programs since 1988. The most recent program was completed in July 2018 at XPS.

The study confirmed that the Wellgreen Deposit samples are amenable to concentration using a conventional SAG mill/ball mill grinding circuit, followed by flotation and magnetic separation. The study resulted in an improved understanding of the impact of resource variations in Cu and S content on the metallurgy, and the following conclusions were made based on the testwork completed to date:

- Variability testing identified a strong correlation between total Ni recovery and sulphur content in the resource. This correlation was compared to, and is consistent with, historic testing.
- Grindability testing of composited samples revealed a relatively hard Bond Ball Work Index ranging from 19.8 kWh/t to 21.4 kWh/t, and JK Drop-weight Axb and ta parameters of 49.9 and 0.49, respectively.
- The composite samples were found to be amenable to conventional concentration using froth flotation followed by magnetic scavenging of the rougher tailings at a primary grind size P80 of 100 µm.
- The rougher concentrate was reground to P80 of 25 µm, and was upgraded to final concentrate in three stages of cleaning. Magnetic scavenger concentrate was reground to a P80 of 10 µm to liberate fine pentlandite and PGM's, prior to rougher flotation and three-stage cleaning.
- MPP testing of the Yr. 1-5 composite produced a bulk final concentrate with a combined Cu+Ni grade of 9.1%, at copper and nickel recoveries of 59.6% and 53.3%, respectively.
- MPP testing of the Yr. 6-10 composite produced a bulk final concentrate with a combined Cu+Ni grade of 13.2%, at copper and nickel recoveries of 33.2% and 31.2%, respectively.
- Cu-Ni separation was performed on the bulk concentrate produced from Yr. 1-5 composite in an LCT. The separation of 53% of the contained Cu to a Cu concentrate of 14% Cu demonstrated the ability to separate, and produced mineral based split factors which were used in the simulation of an industrial circuit.

25.3 Processing

The process plant described in Section 17 is a 45,000 tpd mineral processing facility that includes very conventional processing equipment and flows. The plant design is based on metallurgical test work that includes MPP data (described in Section 13) and is therefore an accurate estimate of processing methods. The plant is designed to produce copper, nickel, and bulk concentrates depending on the plant feed composition.

26 RECOMMENDATIONS

26.1 Metallurgy

To complement the results achieved to date, and to advance the metallurgy to the Pre-Feasibility level, additional variability testwork on low S samples is recommended to increase confidence in the results.

The budget for this phase of metallurgical study is estimated to cost CDN$100,000.

26.2 Drilling

In order to advance the Project to the pre-feasibility stage, it is recommended additional drilling be conducted to better define wall slopes for use in a detailed mine design. As well, drilling will be required to define waste storage area stability as well as plant and tailings foundations. This program is estimated to cost CDN$3 million utilizing a combination of drilling, surface trenches and pits, and other site investigations.

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28 QUALIFIED PERSONS CERTIFICATES

28.1 John M. Marek, P.E.

I, John M. Marek P.E. do hereby certify that:

1) I am currently employed as the President and a Senior Mining Engineer by:

   Independent Mining Consultants, Inc.
   3560 E. Gas Road
   Tucson, Arizona, USA 85714

2) I graduated with the following degrees from the Colorado School of Mines
   - Bachelors of Science, Mineral Engineering – Physics 1974
   - Masters of Science, Mining Engineering 1976

3) I am a Registered Professional Mining Engineer in the State of Arizona USA Registration #12772

   I am a Registered Professional Engineer in the State of Colorado USA Registration #16191

   I am a Professional Engineer, Yukon Territory, Canada

   I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers, Registration #2021600

4) I have worked as a, geoscientist, and reserve estimation specialist for more than 41 years. I have managed drill programs, overseen sampling programs, and interpreted geologic occurrences in both precious metals and base metals for numerous projects over that time frame. My advanced training at the university included geostatistics and I have built upon that initial training as a resource modeler and reserve estimation specialist in base and precious metals for my entire career. I have acted as the Qualified Person on these topics for numerous Technical Reports.

5) I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI43-101.

6) I am responsible for Sections 1, 4 through 12, 14, 23, and 25.1 of the Technical Report titled "2018 NI 43-101 Resource Update on the Nickel Creek Ni-Cu-PGM Project, Yukon Canada" with an effective date of 25 September 2018 and a report date of 9 November 2018.

7) I visited the Nickel Creek Project between April 25-27, 2017 during which times I reviewed the drill core, core handling procedures, sample preparation, core logging and site conditions.

8) Independent Mining Consultants, Inc., and this author worked on the a previous mineral resource and Technical Report for the project that was published 26 June 2017.
9) As of the date hereof, to the best of my knowledge, information, and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) 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 Technical Report, the omission to disclose which makes the Technical Report misleading.
11) I am independent of the issuer applying the tests in Section 1.5 of NI 43-101.
12) I have read national Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.
13) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 9 November 2018

“Signed and Sealed”

John M. Marek, P.E.

28.2 Andy Holloway, CEng., P.Eng.

I, Andy Holloway, CEng., P.Eng. of Peterborough, Ontario Canada as a Qualified Person of this technical report titled "2018 NI 43-101 Resource Update on the Nickel Creek Ni-Cu-PGM Project, Yukon Canada" (the "technical report") prepared for Nickel Creek Platinum Corp. with an effective date of September 25, 2018 and a report date of November 9, 2018, do hereby certify that:

- I am employed as a Principal Process Engineer with AGP Mining Consultants Inc. located at #246-132K Commerce Park Drive, Barrie Ontario L4N 0Z7.
- I graduated from the University of Newcastle upon Tyne, England, B.Eng. (Hons), 1989.
- I am a member in good standing of Professional Engineers of Ontario, membership #100082475.
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience with respect to metallurgy and mining project management includes 28 years' experience in the mining sector covering mineral processing, process plant operation, design engineering, and operations and project management. I have been involved in numerous projects around the world in both base metal, industrial mineral and precious metal deposits.
- I am responsible for Section 17 and Section 25.3 as they pertain to the metallurgical aspects of the technical report.
- I have read the definition of "qualified person" set out in National Instrument 43–101 (NI 43-101) and certify, that by reason of my education, affiliation with a professional associated (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I have not visited the Nickel Creek Yukon Project.
- I am independent of Nickel Creek Platinum Corp. as described by Section 1.5 of the instrument.
- I have had no previous involvement with Nickel Creek Yukon Project.
- I have read NI 43-101, and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated at Peterborough, Ontario Canada this 9th day of November 2018.

"Signed and Sealed"

Andy Holloway, CEng., P.Eng

28.3 Gordon Zurowski, P.Eng.

I, Gordon Zurowski, P.Eng., of Stouffville, Ontario, Canada as a QP of this technical report titled "2018 NI 43-101 Resource Update on the Nickel Creek Ni-Cu-PGM Project, Yukon Canada" (the "technical report") prepared for Nickel Creek Platinum Corp. with an effective date of September 25, 2018 and a report date of November 9, 2018, do hereby certify that:

- I am employed as a Principal Mine Engineer with AGP Mining Consultants Inc. located at #246-132K Commerce Park Drive, Barrie Ontario L4N 0Z7.
- I graduated from the University of Saskatchewan with a B.Sc. Geological Engineering, 1989.
- I am a member in good standing with the Professional Engineers of Ontario (PEO) in Canada, membership #100077750.
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes the design and evaluation of open pit mines for over 25 years.
- I am responsible for Sections 2, 3, 15, 16, 18, 19, 20, 21, 22, 24 and 26.2 of the technical report.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify, that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- I visited the property that is the subject of the technical report on April 29th and 30th, 2017 and inspected the Nickel Creek project site and examined the typical core.
- I am independent of Nickel Creek Platinum Corp. as described by Section 1.5 of the instrument.
- I have had no previous involvement with the Nickel Creek Yukon project.
- I have read NI43-101, and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of the technical report, to the best of my knowledge, information, and belief, the sections of the technical report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated at Stouffville, Ontario Canada this 9th day of November 2018.

"Signed and sealed"

Gordon Zurowski, P.Eng

28.4 Gordon Marrs, P.Eng.

I, Gordon Marrs, P. Eng. am employed as a Consulting Metallurgist with XPS, Expert Process Solutions located at 6 Edison Road, Falconbridge ON P0M 1S0.

This certificate applies to the technical report titled "2018 NI 43-101 Resource Update on the Nickel Creek Ni-Cu-PGM Project, Yukon Canada" (the "Technical Report") with an effective date of September 25, 2018, and report date of November 9, 2018, and I do hereby certify that:

- I graduated from the University of Toronto in 1980 with a BASc in Geological Engineering.
- I have practiced my profession continuously for 38 years since graduation. As a result of my technical and commercial experience, I am a Qualified Person as defined in NI 43–101.
- I have not visited the Nickel Creek project.
- I am responsible for Section 13, Section 25.2 and Section 26.1 of the Technical Report.
- I am independent of Nickel Creek Platinum Corp. as described by Section 1.5 of the instrument.
- I have had no previous involvement with the Nickel Creek Platinum Corp.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated at Falconbridge, Ontario Canada this 9th day of November 2018.

"Signed and sealed"

Gordon Marrs, P.Eng.