NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project

Saskatchewan, Canada

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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 Management System	EMS
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
International Congress on Large Dams	ICOLD
Inverse Distance Cubed	ID3
Land and Resource Management Plan	LRMP

Lerchs-Grossman	LG
Life-of-mine	LOM
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
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
Quality control	QC
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
Stanford University Geostatistical Software Library	GSLIB
Tailings storage facility	TSF
Terrestrial Ecosystem Mapping	TEM
Total dissolved solids	TDS
Total Suspended Solids	TSS
Tunnel boring machine	TBM
Underflow	U/F
Valued Ecosystem Components	VECs
Waste rock facility	WRF

Water balance model	WBM
Work Breakdown Structure	WBS
Workplace Hazardous Materials Information System	WHMIS
X-Ray Fluorescence Spectrometer	XRF

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 Foran Mining 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

1.1 Introduction

Foran Mining Corp. (Foran) retained AGP Mining Consultants Inc. (AGP) to prepare a preliminary feasibility study (PFS) of their wholly-owned McIlvenna Bay Project, zinc-copper volcanogenic massive sulphide (VMS) deposit (the Project), located in northeastern Saskatchewan, Canada. This Technical Report summarizes the results of the PFS, including an initial estimate of Mineral Reserves for McIlvenna Bay.

The Project envisaged by the PFS includes a 3,600 tpd (nominal) underground mine, on-site crushing and mineral processing facilities, a paste plant, filtered tailing storage facility and various supporting project infrastructure such as water management/treatment facilities, offices, a workshop, warehouse, mine dry, and first aid facilities.

The Project considers a 9-year life of mine (LOM) and schedules treatment of the full Mineral Reserve of 11.34 million (M) tonnes (t) (Probable) grading 4.01% Zn, 1.14% Cu, 0.54 g/t Au and 20.97g/t Ag. Production of over 800M lbs zinc and over 250 M lbs copper, in flotation concentrates grading 54.7% Zn (zinc concentrate) and 26.8% copper (copper concentrate) is envisaged.

Pre-production capital costs total C$261M for the project, followed by C$339M of sustaining costs for a total capital cost of C$600M. LOM average operating costs of C$69.48 /t ore are expected, and this translates to an overall operating cost of C$99.34 /t ore if the capitalized sustaining costs are included.

The project economics show a pre-tax net present value (NPV) at a 7.5% discount rate of C$219M and a post-tax NPV of $219M. The pre-tax internal rate of return (IRR) is 23.4% and the post-tax IRR is 19.2%. The Project generates a LOM undiscounted post-tax free cashflow of $365.4M.

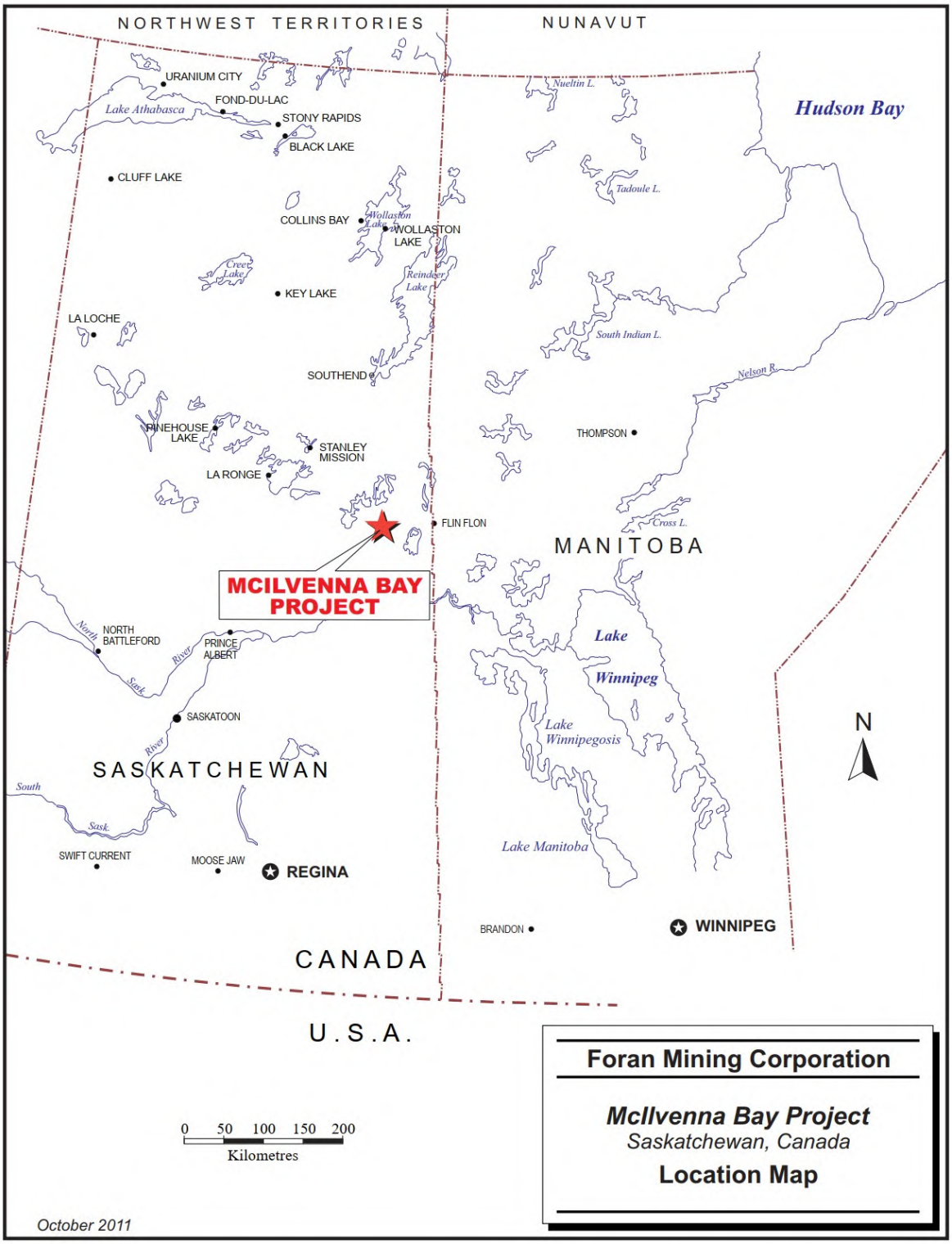
Preparation of the PFS included contributions from several additional consulting firms including, inter alia:

- Micon International Inc. – Mineral Resource Estimate
- Base Metallurgical Laboratories (Base Met Labs) – Metallurgical test work
- BBA (BBA) – Underground Infrastructure designs and costs
- Canada North Environmental Services – Hydrology and Environmental
- Knight Piésold – Tailings Storage Facility design and costs
- Halyard Inc. (Halyard) – Surface Infrastructure and Process Plant design and costs
- Hydro-Resources - Hydrogeology and water quality studies
- RockEng – Underground mine geotechnical designs

1.2 Project Description and Location

The McIlvenna Bay Project lies within Foran's McIlvenna Bay property, which is located approximately 1km south of Hanson Lake, northeastern Saskatchewan (Figure 1-1). The McIlvenna Bay property comprises 38 claims totaling 20,954ha. and is approximately 375km northeast of Saskatoon and 65km west-southwest of Flin Flon, Manitoba.

Figure 1-1: McIlvenna Bay Project Location



1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The McIlvenna Bay project site is accessible via an 18km all-weather, gravel road which connects to Saskatchewan Provincial Highway #106 some 85km southwest of the neighboring towns of Flin Flon, Manitoba and Creighton, Saskatchewan (Figure 1-2). The neighbouring towns represent the largest commercial/residential center in the area. Flin Flon has a long history in mining and provides key infrastructure, such as a heavy rail link that connects the area to the North American railway system. Electrical power is relatively abundant in the region and would be supplied by SaskPower via overhead lines from the hydroelectric station at Island Falls, Saskatchewan.

Figure 1-2: McIlvenna Bay Project Access

In addition to the various highways that connect the towns of Flin Flon and Creighton to other parts of Manitoba and Saskatchewan, Flin Flon is serviced by scheduled daily commercial flights from Winnipeg, Manitoba.

In 2011, Foran built an exploration and development camp on the property. The camp includes a 35-bed trailer camp with office, core shack, shop, and core storage facility. An all-season gravel road has been built through the property to support Foran's exploration programs as well as a historical fracking sand quarry (now reclaimed). An existing 1.2 MVA SaskPower distribution line runs to the property

from Pelican Narrows and although currently deactivated, is still intact and can be brought back into service to support construction activities. A new dedicated distribution line would need to be constructed for the mine and mill as part of the project development.

The climate in the Hanson Lake area is continental, with cold winters and moderate to warm summers. The area is classified as having a sub-humid high boreal eco-climate. The mean temperatures for January and July are -21°C and 18°C, respectively. Temperature ranges from -40°C in the winter to 30°C in the summer can be expected. Annual precipitation averages about 350mm of rain and 1,450mm of snow. There are on average 119 frost-free days per year. Lake ice thaws in April and returns in November.

The property is located within the Boreal Shield Ecozone and is covered with shield-type boreal forest. Topography is flat lying with occasional sharp dolomite cliffs and ridges up to 20m high. Soil thickness on the limestone ridges is minimal, with occasional rock exposure, and the vegetation is dominated by larger conifer and poplar trees. Below the cliffs are poorly drained muskeg swamps with scattered tamarack and black spruce. Throughout the surrounding area, there are numerous lakes and ponds of various sizes.

1.4 History

In 1957, the Parrex Mining Syndicate tested an electromagnetic (EM) conductor delineated under a small bay on the western side of Hanson Lake and intersected impressive zinc-lead massive sulphide mineralization which led to the development of the Hanson Lake (Western Nuclear) Mine. The mine was shut down in 1969.

From 1978 to 1988, Cameco tested selected Aerodat EM anomalies with ground follow-up exploration programs that culminated in the discovery of three new showings, the Miskat Zone (Cu), the Grid B occurrence (Zn), and the Zinc Zone (Zn).

In 1985, the Granges-Troymin joint venture discovered the Balsam Zone, a volcanogenic massive sulphide (VMS) deposit located under the Paleozoic cover, approximately 8km southeast of Hanson Lake. This prompted Cameco to conduct a Mark VI helicopter INPUT survey over the area south of Hanson Lake, which ultimately delineated a 1,200m long INPUT anomaly, striking east-southeast, 1km south of McIlvenna Bay. In 1988, a further geophysical survey defined the anomaly and six holes were subsequently drilled into what is now the McIlvenna Bay deposit. From 1989 to 1991, an additional 61 drill holes were completed by Cameco.

Cameco suspended exploration activities at the McIlvenna Bay property after a corporate decision was made to cease exploration for base metals. The property remained idle until optioned by Foran in 1998.

1.5 Geological Setting and Mineralization

1.5.1 Regional Geology

The McIlvenna Bay Project is located on the western edge of the Paleoproterozoic Flin Flon Greenstone Belt (FFGB) which extends from north central Manitoba into north-eastern Saskatchewan. The FFGB forms part of the Reindeer Zone, a subdivision of the Trans-Hudson Orogen, a continental-scale

tectonic event which occurred approximately between 1.84 Ga and 1.80 Ga (Syme et al., 1999) as a result of the collision between the Superior and Hearne Archean Cratons.

As currently viewed, the FFGB contains eight geographically separate juvenile island arc volcanic assemblages (blocks), each being 20km to 50km across. From east to west, they are known as the Snow Lake, Four Mile Island, Sheridon, Flin Flon, Birch Lake, West Amisk, Hanson Lake, and Northern Lights assemblages (Zwanzig et al., 1997 and Maxeiner et al., 1999). These assemblages are separated by major structural features and/or areas of differing tectonostratigraphic origin. It is unclear whether the eight juvenile arc sequences represent different island arcs, or segments of a larger continuous arc (Syme et al., 1999). Within the belt, each tectonostratigraphic block has been broken into several sub-blocks, usually bounded by local to regional fault systems. Correlation of stratigraphy between sub-blocks is difficult to impossible to determine.

The exposed portion of the FFGB is approximately 250km in an east-west direction by 75km north-south. Although it has an apparent easterly trend, this is an artefact of the belt's tectonic contact with gneissic metasedimentary, metavolcanic, and plutonic rocks to the north (Kisseynew Domain) and the east-trending trace of Phanerozoic platformal cover rocks to the south. In reality, the FFGB extends hundreds of kilometres to the south-southwest beneath a thin cover of essentially flat-lying, Phanerozoic sedimentary rocks.

By Early Ordovician time, the area of northern Saskatchewan and Manitoba had been effectively peneplaned and a regolith was developed on exposed rocks. Inundation by the Ordovician ocean initiated the deposition of the Phanerozoic cover sequence which, in the McIlvenna Bay area, is now represented by the basal Winnipeg Formation sandstone overlain by the Red River Formation dolomite.

In the general Flin Flon area, the predominant direction for the Late Wisconsinan ice-flow indicators is south-southwest indicating that the ice was flowing from a Keewatin dispersal centre. The resulting tills are thin and generally reflect local bedrock lithologies (McMartin et al., 1999).

1.5.2 Local and Property Geology

The Hanson Lake Block, the host terrain of McIlvenna Bay, is bound to the east by the Sturgeon-Weir Shear Zone and to the west by the Tabbernor Fault Zone. The block extends an unknown distance to the south beneath a nearly flat lying cover of Ordovician sandstones of the Winnipeg Formation, and dolomites of the Red River Formation. To the north, the block is bound by the Kisseynew Domain, a gneissic metasedimentary belt and the Attitti Complex. The east end of the block hosts the Hanson Lake Pluton, a large compositionally variable granodiorite to pyroxenite intrusion.

At least two distinct folding events, both having northerly trending fold axes, have influenced the stratigraphy in the Hanson Lake Area. The Hanson Block structural fabric is dominated by a north to northwest-southeast trending, upright regional transposition foliation. A protracted D_2 structural event resulted in tight to isoclinal, southwest plunging F_2 folds and local southwest verging mylonite zones. D_3 deformation resulted in tight north trending folds followed by a brittle D_4 event characterized by north-south trending faults.

Peak regional metamorphism in the areas west and north of Hanson Lake reached upper amphibolite facies as observed by the partial melting of the granodiorite-tonalite assemblage in the Jackpine and Tulabi Lake areas. At McIlvenna Bay, the Proterozoic sequence exhibits a greenschist metamorphic

facies as the deposit alteration assemblages are dominated by sericite and chlorite. The greenschist facies is probably a retrograde event after a previous amphibolite grade since relict cordierite, anthophyllite, garnet and andalusite are commonly observed in the VMS alteration package.

Lacking any outcrop in the area of the deposit, the property geology has been interpreted from the drill core record with help from geophysical surveys.

The stratigraphy of the deposit area, divided into six formations, has been defined over a 2km strike length by a total of 239 drill holes. The lowest formation intersected by drilling both structurally and stratigraphically is the McIlvenna Bay Formation, the host of McIlvenna Bay. The McIlvenna Bay Formation is overlain to the north by the Cap Tuffite Formation. The McIlvenna Bay Formation and the Cap Tuffite Formation may be genetically related but have been separated as they are temporally distinct, as demonstrated by the positioning of the McIlvenna Bay deposit between these two units, an obvious exhalative horizon (and hence a period of clastic and volcanosedimentary quiescence). Overlying the Cap Tuffite Formation is the Koziol Iron Formation, a long and distinctive marker formation traceable for several kilometres along strike by mapping and geophysics. Topping the Koziol Iron Formation is the Rusk Formation, a thick package of mafic volcanics. The Rusk Formation in turn is overlain by the thin HW-A Formation, an exhalative massive sulphide horizon which grades laterally into iron formation. Capping the HW-A Formation is a thick unsorted bimodal package of mafic and felsic volcanics, and mafic intrusions and minor iron formations tentatively called the Upper Sequence which may be thickened due to folding and faulting. The stratigraphic package has been cut by several different intrusions, the largest of which is the Davies Gabbro, represented by a number of sill-like plugs found within the Cap Tuffite Formation. The Proterozoic basement geology is unconformably overlain by the relatively flat lying to shallowly south-dipping Ordovician dolomites and sandstones of the Red River and Winnipeg Formations which have an average total thickness between 20m and 30m.

The McIlvenna Bay Formation, the host formation of the sulphide deposit, is known only to the extent it has been drilled below the footwall of the deposit. The formation is at least 200m thick (true thickness) and comprises the massive and semi-massive sulphides and copper-rich stringer zones that make up the McIlvenna Bay deposit, and a succession of variably altered felsic volcanics, volcaniclastics, and/or volcanic-derived sediments of rhyolitic composition.

1.5.3 Mineralization

McIlvenna Bay is a VMS which consists of structurally modified, stratiform, volcanogenic, polymetallic massive sulphide mineralization and associated stringer style mineralization. The massive to semi-massive sulphides contain copper and/or zinc, with lower concentrations of silver, gold and lead while the stringer style mineralization generally contains elevated copper and gold. The deposit has undergone moderate to strong deformation and upper greenschist to possibly lower amphibolite facies metamorphism. The sulphide lenses are now attenuated down the plunge to the northwest.

The McIlvenna Bay deposit includes five separate zones and two styles of mineralization that are mineralogically and texturally distinct and typical of VMS deposits, including:

- massive to semi-massive sulphide mineralization in the Main Lens and Lens 3
- stockwork-style sulphide mineralization in the Copper Stockwork Zone (CSZ) that directly underlies the Main Lens
- two other small lenses of stockwork-style mineralization:

- the Stringer Zone, which is located between the Main Lens and Lens 3
- the Copper Stockwork Footwall Zone ("CSFWZ"), which occurs as a separate lens underneath the CSZ for approximately 140m of strike length which could represent a fault offset and repetition of the Main Lens and CSZ

1.6 Exploration

On acquisition of the property in 1998, Foran embarked on a diamond drilling program to test new targets as well as in-fill the existing drill pattern on the McIlvenna Bay Deposit. Phase I of this program commenced in December 1998 and carried out through the winter of 1998-1999. A total of 55 holes were drilled during this program, totalling 27,958m. In 1999, Foran initiated environmental baseline studies and commenced engineering work for construction of a road to access the property.

Drilling continued during the winter of 1999-2000 but, was temporarily halted pending financing. Three holes totalling 2,938m were completed in 2000, and an access road was constructed. The mineralization had been delineated to a maximum vertical depth of 1,230m up to this period.

As of May 31, 2000, Foran had drilled an additional 59 holes totalling 33,350m, with 57 holes directly testing the deposit. The first 44 holes were drilled with the objective of upgrading the quality of the resource, down to a depth of 580m, from the Inferred resource category to the indicated resource category. The last 15 holes were drilled below the plunge line and down plunge of the deposit with this drilling successful in extending the deposit an additional 300m vertically below the plunge of the previous resource base.

After 2000, exploration work on the property ceased, and the option agreement with the Hanson Lake Joint Venture was allowed to lapse. Foran acquired a new option agreement in 2005 and resumed work.

In early 2007, Foran completed an airborne deep-penetrating time-domain electromagnetic (VTEM) survey over portions of the Bigstone, Balsam, and McIlvenna Bay properties. The program comprised 404.6 line-km on 150m line spacing over the McIlvenna Bay/Balsam properties and 321 line-km over the Bigstone property.

In the winter of 2007-2008, Foran conducted a diamond drill hole program based on recommendations from the Technical Report on the McIlvenna Bay Project prepared by RPA dated November 27, 2006. Seven diamond drill holes were completed for a total of 6,455m. Drill holes were between 691.5m and 1298.4m in length on sections 9400E through 9700E, with the objective of the program being to tighten drill hole spacing and upgrade the Mineral Resources down plunge on L2MS. A number of drill holes failed to intersect the deposit at depth. Subsequently, Foran determined that the holes which missed their targets were drilled at orientations that made it impossible to intersect the deposit at the targeted depths.

Exploration work underwent a hiatus until 2011 when the company was re-financed, and a new management team was brought in to run the company. That winter, Foran conducted a diamond drilling program consisting of 10 holes totalling 5,056m. This program targeted a portion of the CSZ and was designed to in-fill and prove up the continuity over a portion of the zone in the central part of the deposit. The program was conducted during the late winter and spring and at the same time some of the drill core from the earlier 2007-2008 programs was also relogged and sampled.

The winter 2011 drilling was successful and the Copper Stockwork zone (CSZ) was re-interpreted, using a nominal 0.5% Cu cut-off grade and a minimum apparent thickness of 3m. The other zones were largely unchanged, with the exception of Lens 4, which was incorporated into the FW.

Drilling resumed in August 2011 and ran through to November 2011, with a total of 8,158m completed in 18 holes. The purpose of the drill program was to in-fill the deposit to further increase the confidence in the prior resource estimates, collect sample material for metallurgical test work, and to test the up-dip extension of the CSZ. A re-survey program was completed for all of the drill hole collars that could still be identified on the property. In addition, downhole gyroscopic surveys were carried out in 39 of the historic holes along with the 2011 drill holes.

Foran also completed a helicopter-borne geophysical survey in 2011 that comprised 1,587.4 line-km of time domain electromagnetic (VTEMplus) and horizontal magnetic gradiometer (mag) over those areas of the McIlvenna Bay property not covered in 2007.

In 2012, Foran completed 3,825m of diamond drilling in 15 holes. The drilling was completed during a winter program, which allowed access to areas covered by muskeg that were not accessible during the previous summer. The drilling was directed at near-surface projections of the deposit in order to upgrade the classification and extend the known mineralization. Drilling was dominantly completed utilizing HQ-sized core to provide additional material for future metallurgical test work. Geotechnical and hydrogeological studies were also conducted during the program.

In 2013, three additional drill holes were completed at McIlvenna Bay as part of a more regionally focused winter exploration program targeting other prospective areas on the property. The holes drilled at McIlvenna Bay targeted the updip extension of the CSZ in the central part of the deposit which were accessible from the frozen muskeg.

No further exploration/drilling was conducted on the McIlvenna Bay deposit until the winter of 2018. In December, 2017 Foran signed a Technical Services Agreement with Glencore Canada Corporation, under which Glencore will contribute its professional and technical services, assistance, guidance and advice in connection with the objective of completing a Feasibility Study on McIlvenna Bay Project, in exchange for a exclusive off-take contact to purchase or toll process all of the concentrates and/or other mineral products produced from the Project at prevailing market rates. With this agreement in place, Foran embarked on a large infill and expansion drill program designed to convert as much of the deposit resources as possible into indicated categories which could then potentially be converted into reserves for the upcoming Feasibility study.

In 2018, Foran conducted 26,827m of drilling in 60 drill holes targeting the deposit. The program was completed in two phases, with 14,986.5m in 32 drill holes (including several wedged holes) completed during the phase I winter program and 11,840.5m in 28 holes (including wedges) completed during the phase II summer program. The focus of the winter program was to upgrade both the near surface and deep portions of the deposit which are covered by muskeg and not accessible during summer months. The summer program focused on the middle part of the deposit which was accessible from high ground. Both programs were completed using oriented coring techniques to provide a better understanding of the geological structures in the deposit area. A number of wedge holes were also drilled during the programs in order to provide additional material for metallurgical sampling. In addition to converting resources to the indicated category, other program components included geotechnical, hydrogeological and metallurgical testwork.

Geotechnical components of program included 3,733m of detailed geotechnical core logging on resource drillholes drilled at orientations amenable to both structural and resource studies. In addition to the resource holes, three short geotechnical holes (151.3m) were drilled to characterize the proposed portal location and four short vertical holes (104m) were drilled for piezometer installations to help quantify near surface groundwater flow in the immediate deposit area.

As a part of phase II summer drilling, a downhole resurveying program was also undertaken. A number of holes were identified that did not have a full gyro surveys completed during the 2011 downhole resurvey program due to blockages in drill holes at surface or at depth. Those holes that displayed suspicious or non-existent historic downhole surveys beyond blockages were re-opened with a drill on the pad and re-surveyed with a True North Gyro.

To develop a larger library of ore density measurements across the deposit, Foran employees collected 1,932 bulk density measurements from both 2018 drill holes, and historic core (from 2011, 2012 and 2007), that was not significantly weathered.

As a follow up to both programs, BHEM surveys were completed on a number of holes to look for additional lenses below the level of current drilling. The program was successful in its mandate and culminated with the 2019 resource estimates which is the subject of this report.

1.7 Mineral Processing and Metallurgical Testing

The metallurgical analysis and predictions completed for the PFS were derived mainly from results of the most recent metallurgical testwork program (Base Met Labs, 2019) but also incorporate results and findings from previous mineralogical and metallurgical assessments dating back to 2012. Results of flotation testing from the three metallurgical programs completed to date were generally consistent within each mineralization type over a range of copper and zinc head grades.

The 2019 metallurgical program included flowsheet development work on 3 master composite samples, namely:

- Zone 2 Massive Sulphides (Z2)
- Upper West Massive Sulphides (UWZ)
- Copper Stockwork Zone (CSZ)

Composite head grades are summarized in Table 1-1. The UWZ composite represents the zone of more copper-rich massive sulphides, whilst the Z2 composite is relatively low in copper. Good zinc grades prevail in the massive sulphide zone. The CSZ composite is typical copper stringer type mineralization with significantly lower zinc and iron sulphide concentrations, but higher levels of quartz and mica.

Sample material used to create the 2019 program master composites was a mixture of core pieces and coarse reject material from recent drilling programs. The mass, quantity and distribution of samples used to form the composites is acceptable for a prefeasibility study, and the scope of sampling for the program is sufficient to ensure appropriate representation of the deposit in terms of grade, mineralization style and spatial diversity.

The massive sulphides make up 68% of the PFS reserve tonnage.

Table 1-1: Master Composite Head Assays

Composite	Assays							
	Cu, %	Pb, %	Zn, %	Fe, %	Ag, g/t	Au, g/t	S, %	C, %
Z2 Composite	0.33	0.44	6.65	20.1	16	0.20	25.2	1.93
UWZ Composite	1.93	0.18	4.25	22.3	22	0.97	24.8	1.27
CSZ Composite	1.24	0.04	0.29	6.37	8	0.64	4.48	0.09

In addition to the master composites, testing of several variability composites and ore type blends was undertaken to assist in the definition of operating envelopes for the selected flowsheet under a range of feed conditions.

It was noted throughout testing that a grind of 80% -75µm gave superior metallurgical response and therefore was selected for inclusion in the process design criteria.

A number of key flowsheet developments and incremental improvements to previous metallurgical predictions were made during the 2019 testwork program. Most notably:

- Tests on a range of ore type blends revealed no significant detrimental effects to flowsheet performance. The results confirmed that the CSW mineralization would not have to be processed separately from the massive sulphides. This is contrary to previous studies and means that underground mixing of ore types can be tolerated by the process plant – making the mining operation simpler and more cost-effective.
- A new reagent scheme was also developed during 2019, which used sulphur dioxide (or sodium metabisulphite) as an alternative to the zinc sulphate/sodium cyanide depressant recipe that had been used exclusively in prior work. The selectivity of copper against pyrite, sphalerite and particularly galena was improved significantly when using either chemical in the rougher and cleaner circuits.

At the time of commencing the metallurgical study, processing of McIlvenna Bay mineralization was anticipated to be carried out either via a nearby 3rd party concentrator (existing HudBay operation in Flin Flon), or a new purpose-built mineral processing plant immediately adjacent to the underground mine site. The metallurgical program scope therefore considered the Flin Flon concentrator flowsheet, but also looked at other optimizations that could easily be incorporated into a new mill.

Flotation work culminated in a series of locked cycle tests on the master composites, using a sequential selective approach similar to the Flin Flon concentrator flowsheet. One difference noted however, was that for McIlvenna Bay, regrind circuits were required for both copper and zinc rougher concentrates to improve overall metal recovery and final concentrate grade quality, particularly for the massive sulphide composites.

Locked cycle products were used to carry out physical and chemical characterization work. The chemical anlysis of concentrates did highlight the presence of several deleterious elements in both copper and zinc concentrates, in concentrations that might attact penalties at certain smelters.

Metallurgical Performance Predictions

The locked cycle test results, together with open circuit cleaner test results from the three metallurgical studies, were used to develop performance models related to head grade for copper, zinc, silver and gold. The selected data was curve-fit to derive mathematical functions suitable for use in block models and financial models, with grade or recovery generally specified as a function of feed grade or the recovery of a related metal. Separate models for CSZ and MS ore types were developed.

The metallurgical models were applied to mine production schedules as part of the financial modelling and the resultant life of mine (LOM) average recoveries are shown in Table 22-1 below:

Table 1-2: LOM Average Metallurgical Recoveries

Parameter	Units	Copper	Zinc	Gold	Silver
Massive Sulphide Recovery	%	80.9	81.8	68.8	53.7
Copper Stockwork Recovery	%	96.2	10.0	97.5	78.5
Average Blend Recovery	%	88.2	80.0	79.1	58.0

Over the LOM, the average copper grade in copper concentrate is forecast to be 26.8% and the average zinc grade in zinc concentrate is forecast to be 54.7%.

1.8 Mineral Resource Estimate

The Mineral Resource Estimate (MRE) used within this PFS remains unchanged from the MRE reported by Foran on July 11, 2019 and is summarized in Table 1-3 below. The Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-3: 2019 Mineral Resources for the McIlvenna Bay Deposit, reported at NSR of US$60/t

Classification Category	Mineralized Domain (Zone)	Tonnage (Mt)	Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)
Indicated	Main Lens – Massive Sulphide	9.25	0.90	6.43	0.40	0.52	25.97
	Lens 3	1.99	0.85	3.29	0.14	0.27	14.71
	Stringer Zone	0.70	1.38	0.62	0.04	0.35	13.34
	Copper Stockwork Zone	10.30	1.43	0.28	0.02	0.40	9.30
	Copper Stockwork Footwall Zone	0.71	1.60	1.04	0.04	0.54	11.47
	Total	**22.95**	**1.17**	**3.05**	**0.19**	**0.44**	**16.68**
Inferred	Main Lens – Massive Sulphide	2.97	1.29	4.79	0.29	0.47	23.58
	Copper Stockwork Zone	8.18	1.42	0.76	0.03	0.47	11.63
	Total	**11.15**	**1.38**	**1.83**	**0.10**	**0.47**	**14.81**

The Mineral Resources presented here were reviewed and audited by Micon's QPs using the CIM Definitions and Standards on Mineral Resources and Reserves as of May 10, 2014. Mineral Resources unlike Mineral Reserves do not have demonstrated economic viability. At the present time, neither Micon nor the authors of this report believe that the Mineral Resource estimate is materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

1.9 Mineral Reserve Estimate

The McIlvenna Bay detailed mine design and production schedules used to convert the Mineral Resources to Mineral Reserves are based on the geological block model and Mineral Resource estimate of July 2019 and described in Section 14.0. Mineral Reserves were estimated based on the LOM plan that was prepared In Deswik mine software.

Appropriate mining recovery and dilution factors have been applied for a sub-level transverse and longitudinal retreat mine design and are summarized below.

Table 1-4 outlines the Probable Mineral Reserve Estimate, effective February 17, 2020.

Table 1-4: McIlvenna Bay Reserve Table, reported at NSR of US$100

	Probable Tonnes	Grade			
		Zn (%)	Cu (%)	Au (g/t)	Ag (g/t)
Massive Sulphide	7,773,176	5.71	0.88	0.51	25.24
Copper Stockwork Zone	3,566,067	0.31	1.70	0.60	11.65
Total	**11,339,243**	**4.01**	**1.14**	**0.54**	**20.97**

Notes:

1. Mineral Reserves have an effective date of February 17, 2020.
2. The Qualified Person for the estimate is Denis Flood, P.Eng.
3. The Mineral Reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves
4. The Mineral Reserves are supported by a detailed mine plan, based on a preliminary NSR cut-off value calculation. Inputs to the value calculation include:
    a. NSR Cut off value of US$100
    b. Metal prices of Zn US$1.25/lb, Cu US$3.30/lb, Au US$1310/oz and Ag US$16.20/oz
    c. Average total operating cost of $100/t, consisting of $62.5/t for mining, $31.0/t for processing and $6.5/t for G&A
    d. Metallurgical Recoveries of 81.1% Zn; 88.8% Cu:69.7% Au; and 56.8% Ag
    e. Smelter terms of US$90/t for Cu and US$215/t for Zn
    f. Concentrate transportation costs of US$188/t for Cu and US$97/t for Zn
5. The Mineral Reserve Estimate incorporates a mining recovery of 95% and dilution of 10% globally.
6. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

Detailed information on mining methods, metallurgy, processing and other relevant factors are discussed within this technical report, and together demonstrate that engineering work has been conducted to an appropriate level of detail and that the project as described is economically viable.

1.10 Mining Methods

The McIlvenna Bay deposit will be extracted using conventional longhole mining methods, namely sublevel transverse stoping and Avoca stoping. The orebody geometry and rock characteristics indicate that these are appropriate for safe and efficient production. Ore will be drilled using a top hammer drill, then blasted and mucked by conventional diesel load haul dump(LHD).

Ore will be hauled to surface by battery electric vehicle (BEV) haul trucks early in the mine life and will be hauled to an underground crusher in later years, allowing more cost-effective transfer of crushed ore to surface by vertical conveyor.

Transverse stopes will be backfilled with pastefill, using filtered tailings from the processing facility and Avoca stopes will be backfilled with waste rock generated by underground development. Conventional trackless mining equipment will be used to execute lateral development required to access the orebody. Ore will be produced at a nominal rate of 3,600 tonnes per day (tpd) with a mine life of 9 years, including an initial ramp up period of 2 years.

The mining operation labour force will consist of mine management and technical support staff, development and production crews, maintenance crews and miscellaneous support staff. The labour schedules calculated for the mining operation describe a complement of up to 201 personnel.

The fleet of mobile mining equipment will include LHD's, haul trucks (BEV's), jumbos, bolters, top hammer drills and a shotcrete unit. A number of key mobile equipment items will be purchased through a lease to own program.

The mine ventilation system is designed around Canada Centre for Mineral and Energy Technology (Canmet) requirements for the described mobile fleet and takes the use of BEV's into account for a total airflow requirement of 200m^3/s. The design includes two permanent exhaust fan systems installed over two perimeter located exhaust raises, and the decline will serve as the primary intake route. Intake air will be heated by a propane-fired heating system located at the portal.

Table 1-5 below outlines the annual plant feed schedule by ore zone. Both zones are to be mined and processed together but are illustrated separately below as the metallurgical performance is dictated by the balance of tonnes/grades from the two zones.

Table 1-5: Plant Feed Schedule

Mining Year		TOTAL	1	2	3	4	5	6	7	8	9	10	11
Tonnes Mined (Dilution and Mining Recovery Applied)													
Massive Sulphide	**kt**	**7,773**	**89.4**	**545.5**	**798.8**	**909.9**	**897.7**	**817.2**	**824.2**	**937.8**	**1135.6**	**816.3**	**0.7**
Silver Grade	g/t	25.24	18.38	44.31	33.18	27.48	22.07	22.51	21.51	27.05	18.07	20.86	18.61
Gold Grade	g/t	0.51	0.18	0.49	0.65	0.58	0.55	0.46	0.44	0.48	0.57	0.40	0.90
Copper Grade	%	0.88	0.13	0.63	0.86	0.87	1.06	0.78	0.76	0.85	1.23	0.73	1.68
Zinc Grade	%	5.71	8.68	7.87	7.11	6.13	5.55	5.45	5.33	5.31	4.34	5.31	3.02
Lead Grade	%	0.40	0.11	0.62	0.54	0.40	0.42	0.42	0.39	0.47	0.22	0.30	0.11
Copper Stockwork	**kt**	**3,566**	**0.5**	**120.6**	**468.3**	**521.5**	**474.8**	**590.5**	**601.2**	**481.8**	**272.7**	**34.4**	**0.0**
Silver Grade	g/t	11.65	17.14	12.17	12.82	11.46	11.68	12.32	10.61	10.01	12.74	17.55	15.41
Gold Grade	g/t	0.60	0.43	0.68	0.70	0.66	0.74	0.64	0.46	0.48	0.51	0.42	0.76
Copper Grade	%	1.70	1.27	1.41	1.65	1.65	1.79	1.95	1.67	1.56	1.57	1.50	1.75
Zinc Grade	%	0.31	0.18	0.31	0.29	0.21	0.25	0.31	0.34	0.33	0.56	0.64	0.97
Lead Grade	%	0.02	0.02	0.02	0.02	0.02	0.03	0.02	0.02	0.04	0.02	0.04	0.03
Total Ore Mined		11,339	89.9	666.1	1267.1	1431.4	1372.5	1407.7	1425.4	1419.5	1408.3	850.6	0.7

1.11 Recovery Methods

Concentration and recovery of copper and zinc minerals will be carried out within an on-site process plant. The plant has been designed with an initial capacity of 3,400 tpd that is stepped up to a nominal 3,700 dmtpd capacity in year 3 via the addition of standby equipment. Short term surges in mine production would be handled using the ROM surface stockpiles

In the early years of mine production, ore will be hauled to surface in 50 tonne trucks and dumped into a surface crushing facility. As mine development continues below the 0m AMSL level (year 3), an underground crushing station will be constructed to feed -6" ore onto a vertical conveying system. The surface crushing station would continue to operate while ore is being truck-hauled to surface, and thereafter would remain available as a standby/emergency crushing facility. The vertical conveyor would be tied into the surface crushing facility using a transfer conveyor.

Irrespective of station location, ROM ore will be crushed to a nominal 100% -150mm, (80% -80mm) size. Conveyors will transfer the coarse crushed ore to one of the two surface stockpiles in preparation for secondary crushing. The secondary crushing circuit will consist of a cone crusher in closed circuit with a screen and is required to reduce the ore size in preparation for ball milling.

The grinding circuit consists of a two-stage ball mill combination which is well suited to handling the variable hardness expected from mixtures of the high-silica copper stockwork material and the softer massive sulphide rock. The grinding circuit is designed to reduce the particle size of flotation feed slurry to a nominal 80% -75µm as indicated by recent metallurgical testwork. Cyclone overflow slurry from the secondary mill would be directed to the flotation section for sequential copper/lead and zinc concentrate recovery. The copper and zinc circuits would be similar in nature, with each circuit producing rougher concentrates prior to regrinding and multi-stage cleaning. Two saleable flotation concentrates – copper and zinc – will be produced separately. The copper and zinc rougher/scavenger concentrates would both be reground, with copper concentrates requiring a P_{80} of approximately 20 to 25µm and zinc concentrates requiring a P_{80} of approximately 25 to 30µm.

Final concentrates from the copper and zinc cleaner circuits would be pumped to the copper and zinc concentrate thickeners to recover water and produce a 50-60% solids underflow slurry suitable for pressure filtration. A vertical pressure filter would be used to further dewater both copper and zinc thickened concentrates in batches to provide two stockpiles of product filter cake suitable for bulk transportation to toll smelters by road/rail.

Zinc flotation tailing slurry would pass through a small desulphurization flotation circuit. Cells would be similar in size/design to the zinc flotation cells, with reagents added to non-selectively recover residual sulphide minerals (mainly pyrite) to a sulphide concentrate. The concentrate will be dewatered separately and directed to the paste backfill circuit, for incorporation into the backfill mixture and safe storage underground.

Low sulphide tailing slurry would be pumped to a high compression tailings thickener for dewatering, storage and transfer across to the paste plant. Water would be recovered from the tailing slurry and returned to the process water tank for re-use. Thickened underflow slurry (approximately 65 to 70% solids) would then be pumped to storage tanks ahead of the paste plant. The paste plant would treat 100% of flotation tailings and would make a filtered cake suitable for stacking on a nearby tailings

storage facility (TSF), or for mixing with water and Portland cement to make a paste backfill slurry for use underground.

The process plant would include various water reticulation and air services, in addition to HVAC and dust extraction modules. Most plant areas would be contained within one large building. The tailing thickener would be located outdoors but would be partially clad to ensure proper operation during winter months. A reagent storage area would be located alongside the main building. Reagents would be stored and transferred to the reagent day tank and dosing area within the main building. Reagents include frothers, collectors and promotors, sulphide and talc depressants and flocculants.

1.12 Project Infrastructure

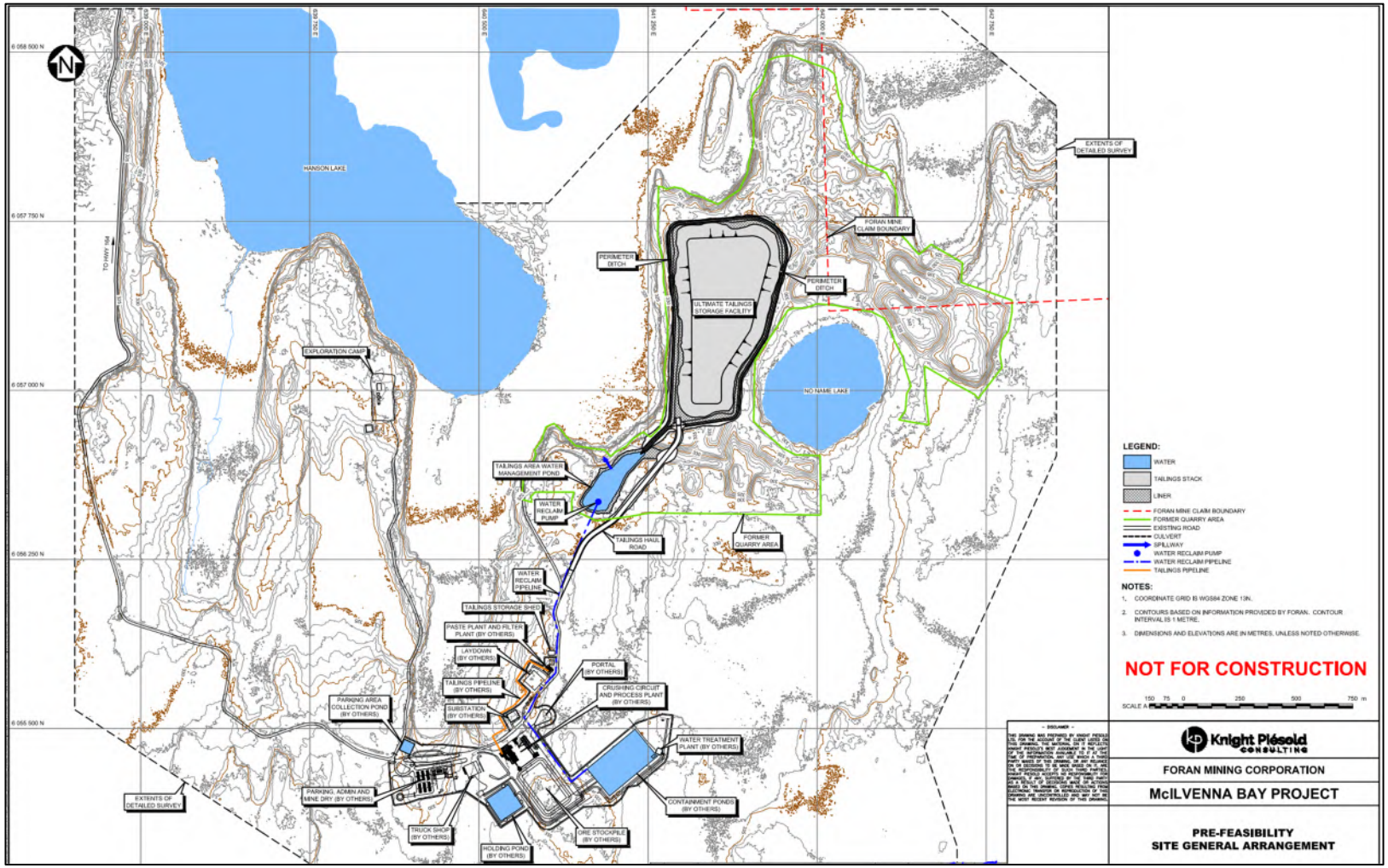
The Tailings Storage Facility (TSF) is located approximately 1 km north of the portal, at the site of the old Preferred Sands Mine (now closed down and rehabilitated). Tailings from the flotation plant will be desulphurized and dewatered to form a filter cake, which will be trucked to the TSF. Approximately half the mine production tonnage would be stored as tailings, with the remaining mass stored underground as paste fill.

Figure 1-3 with facilities to support mining operations including:

- access roads, haul roads, service roads, parking and laydown areas
- truck shop and warehouse
- administrative office and Dry complex
- lunchroom, first aid, and mine rescue trailer
- explosives storage
- propane and diesel storage
- power supply and distribution infrastructure
- waste rock dump and ore stockpiles
- water management systems (civil works , pumps, pipelines and treatment plants)
- waste storage areas
- plant tailings facilities, including haul road, TSF and water return equipment

The Tailings Storage Facility (TSF) is located approximately 1km north of the portal, at the site of the old Preferred Sands mine (now closed down and rehabilitated). Tailings from the flotation plant will be desulphurized and dewatered to form a filter cake, which will be trucked to the TSF. Approximately half the mine production tonnage would be stored as tailings, with the remaining mass stored underground as paste fill.

Figure 1-3: McIlvenna Bay Site Layout (Knight Piésold, 2020)



1.13 Market Studies and Contracts

Copper and zinc concentrates will be produced for sale into established base metal markets. The copper concentrate is anticipated to contain 26.8% Cu and the zinc concentrate is expected to contain 54.7% zinc.

Minor (deleterious) elements are recovered to these concentrates in concentrations that may attract penalties at certain smelters. Elevated mercury concentrations were noted in the zinc concentrate but will be diluted to lower levels in the copper concentrate as a result of dilution with the much cleaner CSZ material. Iron content in zinc concentrate at 9-10% may incur penalties. Iron penalties can be controlled in practice by maintaining zinc concentrate grades around 54% Zn. Selenium was elevated in copper concentrates for all ore types, and fluorine was noted to be a possible complication in previous studies. Magnesium and silica were elevated in the massive sulphide copper concentrates (likely as talc), but this will be lowered significantly with blending of copper concentrate from CSZ. Further metallurgical studies focused on concentrate quality are recommended.

Foran entered into a Technical Services Agreement (TSA) with Glencore Canada Corporation (Glencore) in December 2017, and this TSA included provision for off-take agreements for the McIlvenna Bay metal concentrates. Pricing used in the PFS financial model was based on standard commercial terms available at the time.

1.14 Environmental Studies, Permitting and Social/Community Impact

The Project lies in the Boreal Plain Ecozone on the boundary of Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion, and the Flin Flon Plain landscape area of the Churchill River Upland Ecoregion. The boundary between these two ecoregions passes through McIlvenna Bay on Hanson Lake, such that the northern part of the baseline study area lies in the Churchill River upland, and the southern part lies in the Mid-Boreal Lowland. Extensive mining and exploration activities associated with other metal and silica sand mining projects have occurred in the Project area; therefore, the area does not represent undisturbed baseline conditions. Comprehensive environmental baseline studies for McIlvenna Bay were completed by Canada North Environmental Services (CanNorth) in 2012. The baseline program was designed to prepare the Project for future licensing and regulatory requirements, and included collection of a full suite of environmental data including:

- climate and meteorology
- noise
- surface water hydrology
- water and sediment quality
- plankton, benthic invertebrate, and fish communities
- fish habitat
- fish spawning
- fish chemistry
- ecosite classification
- vegetation communities

- species at risk
- wildlife communities
- heritage resources

Follow-up hydrological studies were completed between 2013 and 2014 and in 2018 and 2019 to extend the hydrological data set and to characterize the hydrologic regime of the local area.

The Project lies within the area traditionally occupied by the Peter Ballantyne Cree Nation (PBCN) and is located approximately 40km southeast of the settlement of Deschambault Lake and approximately 50km west of the community of Denare Beach. Approximately 1,500 PBCN members reside in these communities. The Project is also located within the Métis Nation of Saskatchewan Eastern Region 1. Foran has been meeting with members of the communities of Deschambault Lake and Denare Beach to update them about the Project since 2012. Foran also initiated a Traditional Land Use/Knowledge Inventory Study which was completed by ASKI Resource Management and Environmental Services (a corporation of the PBCN) in 2012.

More recently, Foran has entered into discussion with the PBCN with the objective of negotiating a Memorandum of Understanding that outlines the terms and details of an understanding focused on areas of community engagement, environmental stewardship, training and employment opportunities, and business development.

1.15 Capital and Operating Costs

1.15.1 Capital Costs

Capital costs were prepared using information from a variety of sources, including derivation from first principles, equipment quotes, and factoring from other costs within the PFS. Capital costs are split into pre-production costs and sustaining costs and estimated to an accuracy of +/- 25%. All costs are expressed in Canadian dollars unless stated otherwise and based on Q1 2020 pricing. Capital cost estimates reflect the project scope and designs described in this report and are presented in summary format in Table 1-6 below. Pre-production capital costs are estimated to be $261.3 million and sustaining capital costs are another $338.6 million for a total LOM capital cost of $600 million.

Table 1-6: Capital Cost Summary

Area	Capital Cost, $ Million		
	Initial	Sustaining	Total
Mine	72.7	273.9	346.6
Process Plant	100.6	7.2	107.8
Infrastructure	50.8	-	50.8
G&A	0.7	-	0.7
Tailings	5.9	11.8	17.6
Closure	-	6.4	6.4
Contingency	30.7	39.4	70
Total CAPEX	**261.3**	**338.6**	**600.0**

* All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines

1.15.2 Operating Costs

Operating costs (OPEX) are based on the current mine design, mine production schedule and quantities. The LOM operating costs are presented in summary format in Table 1-7 below and compared with those in the earlier years of the project. Costs are presented in 2019 Canadian dollars.

Table 1-7: OPEX Cost Summary

Area	Operating Cost, $ per tonne ore	
	Year 1-5	Life of Mine
Mine	43.08	41.19
Process Plant	19.86	19.55
Infrastructure	2.92	2.82
General and Admin	4.37	4.13
Tailings	1.78	1.78
Total OPEX	**72.01**	**69.48**

The $338.6 million of capitalized sustaining cost is equivalent to $29.86 per tonne ore and can be added to the OPEX above to obtain a total operating cost (including sustaining costs) of $99.34 per tonne ore. This is in line with other operations in the region.

1.16 Economic Analysis

Economic analysis of the McIlvenna Bay project was modelled using a discounted annual cash flow (DCF), with revenues and expenditures estimated for each annual period of the project. The cashflows were discounted using an annual rate of 7.5% and a CAD:USD exchange rate of 1.30 was used where necessary.

Key results of the financial analysis are given in Table 1-8 below.

Table 1-8 : Financial Metric Summary

Pre-Tax NPV (7.5%) Pre-Tax IRR	$218.6 M 23.4%
Post-Tax NPV (7.5%) Post-Tax IRR	$147.1 M 19.2%
Undiscounted After-Tax Free Cash Flow (LOM, before pre-production capital deductions)	$626 M
Undiscounted After-Tax Free Cash Flow (LOM, Net of pre-production capital)	$365.4 M
Payback Period from start of processing (undiscounted, after-tax cash flow)	3.8 years
Pre-Production Capital Expenditures (rounded)	$261.3 M
LOM Sustaining Capital Expenditures (including closure)	$338.6 M
LOM Cash Cost (net of by-products) per lb. Zinc LOM Cash Cost (net of by-products) per lb. Copper	US$ 0.41 US$ 0.44

Long term commodity price assumptions used for the study are summarized in Table 1-9 below. The base case price forecast uses 3-year trailing average price data, with a base date of 20 January 2020.

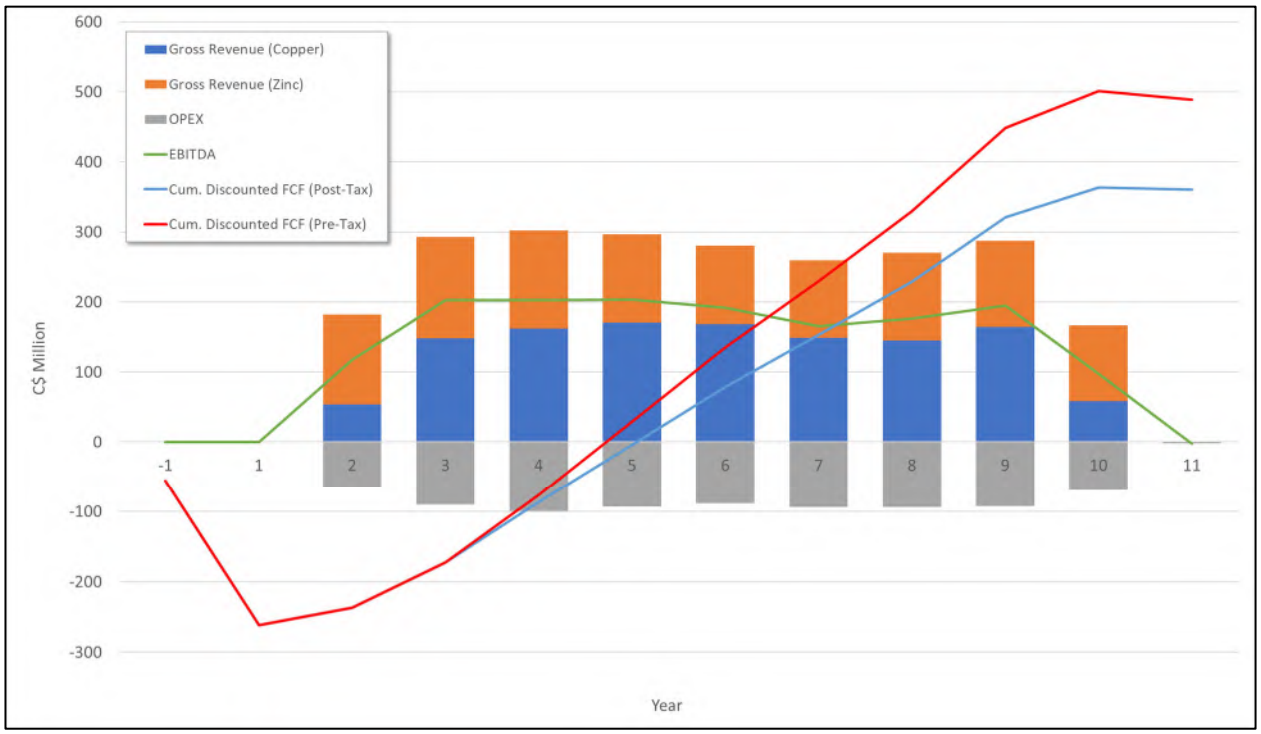
Table 1-9: Long Term (Base Case) Commodity Prices

Metal Price	Unit	Value
Copper Price	US$/lb	2.82
Zinc Price	US$/lb	1.26
Lead Price	US$/lb	0.95
Gold Price	US$/oz (troy)	1,312
Silver Price	US$/oz (troy)	16.30

The cash flow model is based in Canadian dollars. The cash flow model assumes no price inflation in metal prices or in costs.

Net Present Value (NPV) of the project cash flow pre-tax is $218.6 million at a discount rate of 7.5%, and $147.1 million post- tax. Figure 1-4 below shows the annual cash flow compared to project cumulative discounted value.

Figure 1-4 Annual Project Financial Analysis



The undiscounted cash flow is $365.4 million after tax. The pre-tax IRR is 23.4%. Sensitivities to the base case were run for changes to commodity prices (revenues) and costs. Salient production and financial metrics for each set of forecasts are presented below in Table 1-10.

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Table 1-10: Pre and Post Tax NPV $M Sensitivity Table

	80%	90%	100%	110%	120%
PRE TAX NPV, $M					
Metal Price	-69	75	219	362	506
CAPEX	310	264	219	173	128
OPEX	315	267	219	170	122
USD:CAD FX Rate	-22	98	219	339	459
POST TAX NPV, $M					
Metal Price	-69	41	147	253	358
CAPEX	238	193	147	102	56
OPEX	218	183	147	112	76
USD:CAD FX Rate	-32	58	147	236	324

Pre-tax NPV and IRR sensitivities are also shown graphically in Figure 1-5 and Figure 1-6 below.

Figure 1-5 Pre Tax NPV Sensitivity to Key Variables

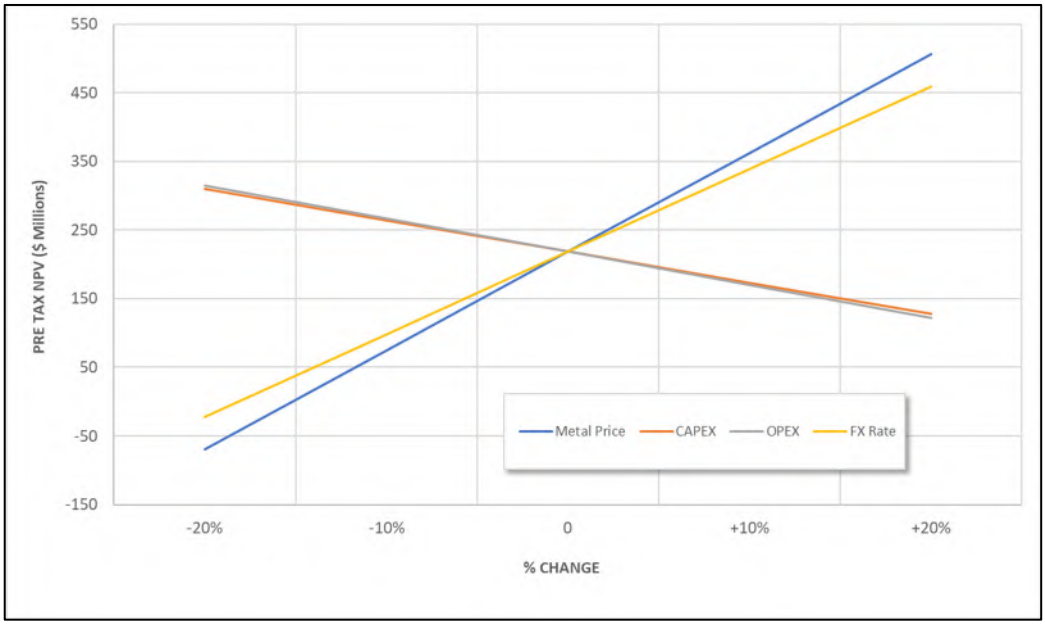
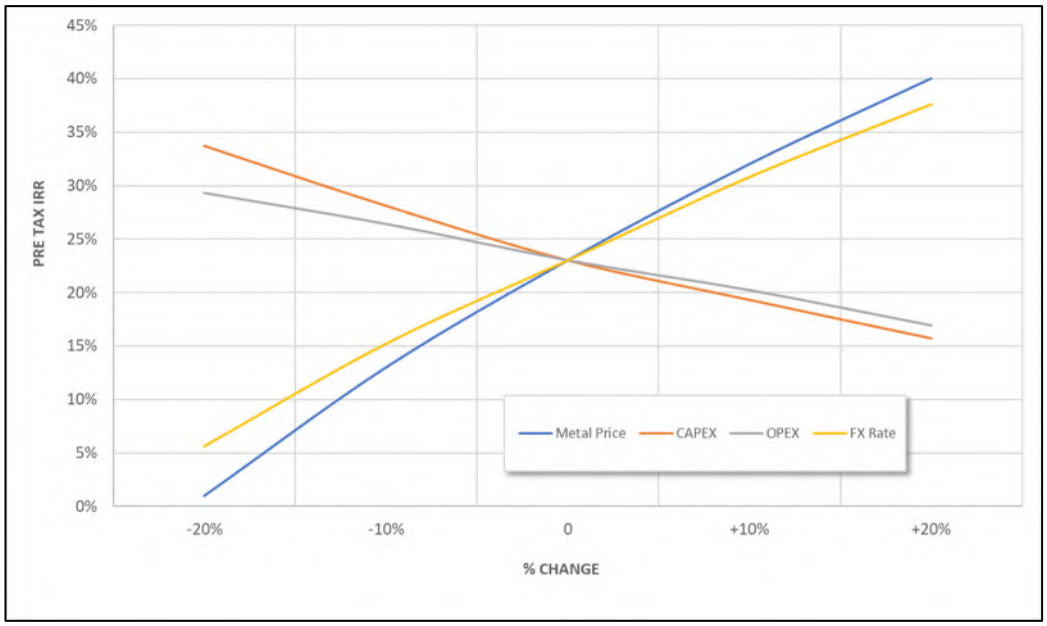


Figure 1-6 Pre Tax IRR Sensitivity to Key Variables



1.17 Interpretations and Conclusions

The PFS concludes that the McIlvenna Bay Project is economically viable based on the current project configuration with the applied economic and financial modelling parameters. Foran is satisfied that the work that is summarized within this Technical Report has been completed to established standards for prefeasibility studies, and that is meets the requirements of National Instrument 43-101 Standards for Disclosure for Mineral Projects.

The PFS includes the initial statement of Mineral Reserves for McIlvenna Bay. The statement of reserves consists only of Probable Mineral Reserves and are those Resources in the Indicated category that have been calculated to be economically extractable after incorporation of the relevant Modifying Factors. The Reserve Estimate is effective as of February 17, 2020 and is a sub-set of the Mineral Resource Estimate (effective date of May 7, 2019) described in Section 14.

Modifying factors applied to the Probable Mineral Reserve statement include prefeasibility level calculations and assumptions related to mining, processing and metallurgy, infrastructure, economics, marketing, legal, environmental, social and governmental factors. All key calculations and assumptions used in the determination of Mineral Reserves are discussed in this Technical Report.

1.18 Recommendations

A number of future studies and site investigations have been identified to bring the Project up to bankable feasibility level, and it is recommended that Foran continue to advance the Project towards production by mitigating the identified risks and following up on the identified opportunities. A feasibility study is recommended as a next step, and this is estimated to cost $3.7 million.

Key opportunities for the development at the McIlvenna Bay Project are as follows:

- Further optimization of the mine plan and stope sequence.
- Optimization of cut off NSR to increase resource conversion and extend mine life.
- Reduction of capital cost estimates, through more detailed studies, collection and analysis of additional site data including geotechnical, geochemistry, hydrological, and hydrology data (i.e.: foundation designs, geotechnical designs, water management/treatment designs, etc.).
- Investigation of further mine automation to increase productivity and reduced operational expenditures. Current analysis utilizes assumptions that the mine is partial automated with benchmarked production and cost information.
- Increase confidence in Inferred resources enhancing total reserve base and extending expected mine life.
- Evaluate opportunities to share infrastructure with other potential projects in the region.

Risks that could be present in the development of the project are summarized below:

- The project is sensitive to metal prices and CAD:USD FX rates. In times of high market volatility, accurate long-term prediction of cost and revenue is challenging.
- Increased costs for skilled labour, power, fuel reagents, trucking, etc. that require an increase in the cut-off grade and decrease the level of Mineral Reserves.
- Delay of critical path schedule items such as:
  - design, permitting, and construction of the tailings facility
  - design, permitting, and construction of the power line
  - portal development due to unforeseen geotechnical complications
- Changes to the design of the TSF as a result of limited geotechnical and hydrological site data at the selected site.

2. INTRODUCTION

2.1 Terms of Reference

At the request of Mr. Patrick Soares, President and CEO of Foran, AGP has completed a PFS for the potential development of an underground mining operation at McIlvenna Bay. Foran is a Vancouver based company, trading on the TSX Venture Exchange under the symbol FOM.V. The company owns 100% of the mineral claims and mining leases associated with McIlvenna Bay.

The purpose of this report is to summarize the findings of the PFS on the Project that was completed on the Effective Date given in 2.3 below.

The user of this report is encouraged to ensure that this is the most recent Technical Report for the Property, as it becomes invalid as soon as a new Technical Report is issued.

The quality of information contained within this report is considered to be consistent with the level of effort involved in the services provided, and is based on:

- data supplied by outside sources
- information available at the time of preparation, and
- qualifications and assumptions discussed within this report

The PFS covered the following key activities:

- review and acceptance of the current resource model for the McIlvenna Bay deposit
- conduct economic trade off work to compare various production scenarios
- prepare an underground mine design and production schedule using supporting hydrogeology and geotechnical information from 3rd party consultants
- calculate a Probable Mineral Reserve estimate, using suitable modifying factors
- design a metallurgical flowsheet and specifications for a new onsite mineral processing facility
- prepare specifications for all supporting project infrastructure including access roads, power supply, water treatment, ventilation, buildings, tailings storage facilities, etc.
- summarize the environmental baseline work and permitting requirements
- compile capital expenditure and operating expenditure estimates for the Project
- build an economic model of the project, complete with discounted cashflows and sensitivity analyses
- make recommendations for further work required for development of the project

This report discloses technical information, including the first disclosure of Mineral Reserves for the Project. The presentation of this information requires Qualified Persons (QPs) to derive sub-totals, totals and weighted averages that inherently involve a degree of rounding and, consequently, introduce a margin of error. Where these occur, the authors and AGP do not consider them to be material.

The conclusions and recommendations in this report reflect the QPs best independent judgment in light of the information available to them at the time of writing. AGP and the QPs reserve the right, but

will not be obliged, to revise this report and conclusions if additional information becomes known to them subsequent to the date of this report. Use of this report acknowledges acceptance of the foregoing conditions.

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines (2014). The resource statement in this PFS includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. The Mineral Reserve statement in this PFS is the first disclosure of Mineral Reserves for the Project and does not include any Inferred Mineral Resources.

2.2 Site Visits, Scope of Personal Inspection and Qualified Persons

The authors of this Technical Report are specialists in the fields of Mineral Resource/Reserve Estimation, mining, geology, metallurgical process engineering, civil, mechanical and electrical engineering, cost estimating (capital and operating), and mineral economics. The individual consultants responsible for the PFS and for this technical report are all considered to be Qualified Persons by virtue of their education, experience and professional associations.

None of the QP's responsible for the preparation of this report has any beneficial interest in Foran. The Consultants are not insiders, associates, or affiliates of Foran. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Foran and AGP. The consultants are being paid a fee for their services in accordance with normal professional consulting practice. Table 2-1 lists the individual QP's responsible for the preparation of this report, their titles, areas of responsibility and the dates of any site visits. QP Certificates are given in Section 28 of the report.

Table 2-1: Details of QPs and Site Visits

Qualified Person	Title	Area of Responsibility	Site Visit Date(s)
William J. Lewis, B. Sc., P. Geo.	Senior Geologist, Micon International Ltd.	1.2 to 1.6, 1.8, 4.0 - 12.0, 14.0, 25.1 to 25.4, 26.2	August 2018
Andrew Holloway, P.Eng.	Principal Process Engineer, AGP Mining Consultants Inc.	1.1, 1.7, 1.11, 1.15, 1.17, 1.18, 2.0, 3.0, 13.0, 17.0, 21.1.1, 21.1.6 – 21.1.9, 21.2.2, 24.0, 25.7, 25.11, 25.13, 25.14, 26.1, 26.4, 27, 21.2.4, 21.2.6, 23	September 2018 January 2019
Stephen Cole, P. Eng.	Independent Consultant	1.13, 1.16, 19.0, 22.0, 25.9, 25.12, 26.6, 26.8	None
Denis Flood, P. Eng.	Senior Mining Engineer, AGP Mining Consultants Inc.	1.9, 1.10, 15.0, 16.0, 21.1.2, 21.2.1, 25.5, 25.6, 26.3	April 2020
Manoj Patel, P. Eng.	Project Manager, Halyard Inc.	1.12, 18.0 (except 18.8), 21.1.3, 21.1.4, 21.2.3, 25.8, 26.5	January 2019
Jocelyn Howery, M.Sc., Pag	Hydrology Division Manager, Canada North Environmental Services Ltd.	1.14, 20.0, 25.10, 26.7	May 2013, September 2013, and October 2014
Alex McIntyre, P. Eng.	Senior Engineer, Knight Piésold Ltd.	18.8, 21.1.5, 21.2.5	None

2.3 Effective Dates

- The effective date of the Resource Estimate is 7th May 2019.
- The effective date of the Reserve Estimate is 17th February 2020
- The effective date of the Prefeasibility Study is 12th March 2020.

2.4 Information Sources and References

Key sources of information for this report include:

- The previous technical reports listed in Section 2.6 below
- Documents referenced in Section 27.0 of the report
- Information provided by the Foran Management team, as described in Section 3.0 of the report.

2.5 Units of Measure

Currency units used in this study and report are Canadian Dollars except where noted. Any reference to “dollars” or “$” within this report means Canadian Dollars. The metric system has been used

throughout the study and this report. Any reference to "tons" or "tonnes" means dry metric tonnes unless otherwise noted. One metric tonne equals 1,000 kg or 2,204.6 lb.

2.6 Previous Technical Reports

The last Technical Report to outline project economics was the Preliminary Economic Estimate (PEA), reported by JDS Energy and Mining in 2015. The PEA revised report title is "Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan Canada" and the report has an effective date of November 12, 2014.

The Mineral Resource Estimate was subsequently updated in 2019, as reported in the Micon International Ltd. Technical Report, entitled "Technical Report for the 2019 Resource Estimate on the McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of May 07, 2019

3. RELIANCE ON OTHER EXPERTS

In the preparation of TSF designs and cost estimates, Alex McIntyre, P.Eng. relied on information supplied by the owner of CPI Construction Inc., including reports, drawings and other technical details. CPI Construction were intimately involved in the closure and rehabilitation of the Preferred Sands Quarry, including the proposed site of the TSF.

Andrew Holloway, P.Eng., Stephen Cole, P.Eng. and William Lewis B.Sc., P.Geo., have relied upon Patrick Soares (CEO of Foran) for information related to royalties as disclosed with this report and as included within the financial analysis (December 2019).

In preparing the project financial analysis, Stephen Cole, P.Eng. has relied upon Tim Thiessen (CFO of Foran) for information related to taxes levied against the project, including the details of applicable Mineral Property Tax Pools (December 2019).

Mineral Tenure has not been confirmed by independent legal review, and the QP's have relied on records and information provided by the Foran Management Team in this regard (January 2020).

4. PROPERTY DESCRIPTION AND LOCATION

The following section has been extracted from the previous 2019 Micon International Limited (Micon) Technical Report for the McIlvenna Bay Project and updated or edited where necessary.

4.1 General Description and Location

McIlvenna Bay Project (deposit) occurs within Foran's McIlvenna Bay property located approximately 1km south of Hanson Lake, Saskatchewan. The property is also approximately 375 km northeast of Saskatoon and 65 km west-southwest of Flin Flon, Manitoba (Figure 4-1). McIlvenna Bay is located within Canadian National Topographic System (NTS) sheet 63L10 and the plan projection of the deposit is centred on UTM coordinates 640,600 E and 6,056,200 N (NAD 83, Zone 13). The corresponding geographic coordinates are 102°50' W and 54°38" N. The McIlvenna Bay deposit is located well within the property boundaries.

4.2 Ownership, Land Tenure and Property Agreements

4.2.1 Ownership and Land Tenure

Foran owns 100% of the McIlvenna Bay property. The entire McIlvenna Bay property comprises 38 claims totalling 20,954 ha (Figure 4-2). The tabulation of the relevant claim information is summarized in Table 4-1. The claims are listed in the name of Foran and are kept in good standing at the discretion of Foran. Foran has engaged an independent firm to track and maintain the claims in good standing. The information contained within this report was provided by Foran and/or their designates.

4.2.2 Property Agreements

On January 25, 2005, Foran announced that it had entered into a definitive agreement with Cameco Corporation (Cameco) and Billiton Metals Canada Inc. (BHP Billiton), collectively the Hanson Lake Joint Venture, which allowed Foran to acquire a 100% interest in the McIlvenna Bay property (including the McIlvenna Bay copper-zinc deposit). Foran would acquire 100% of the McIlvenna Bay property by:

- paying $1,500,000 to the Hanson Lake Joint Venture
- paying a further $2,000,000 to the Hanson Lake Joint Venture before May 31, 2006
- providing the Hanson Lake Joint Venture with a 1% Net Smelter Return (NSR), with a buy-out provision in favour of Foran for the purchase of the whole NSR for $1,000,000 at any time

Figure 4-1: McIlvenna Bay Project Location Map

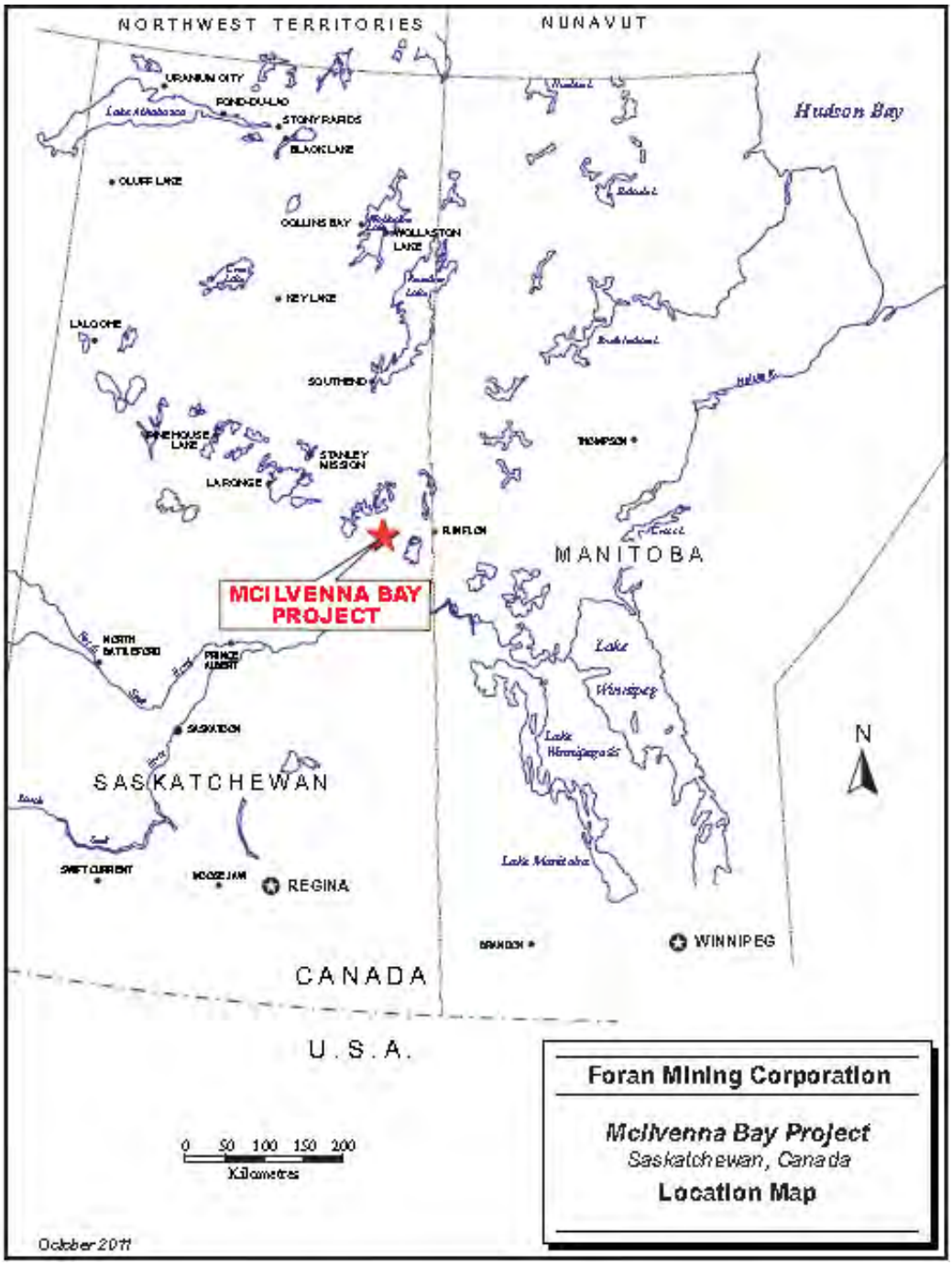


Figure extracted from the 2015 JDS Technical Report, figure originally Foran, 2011.

Figure 4-2: McIlvenna Bay Project Property Map

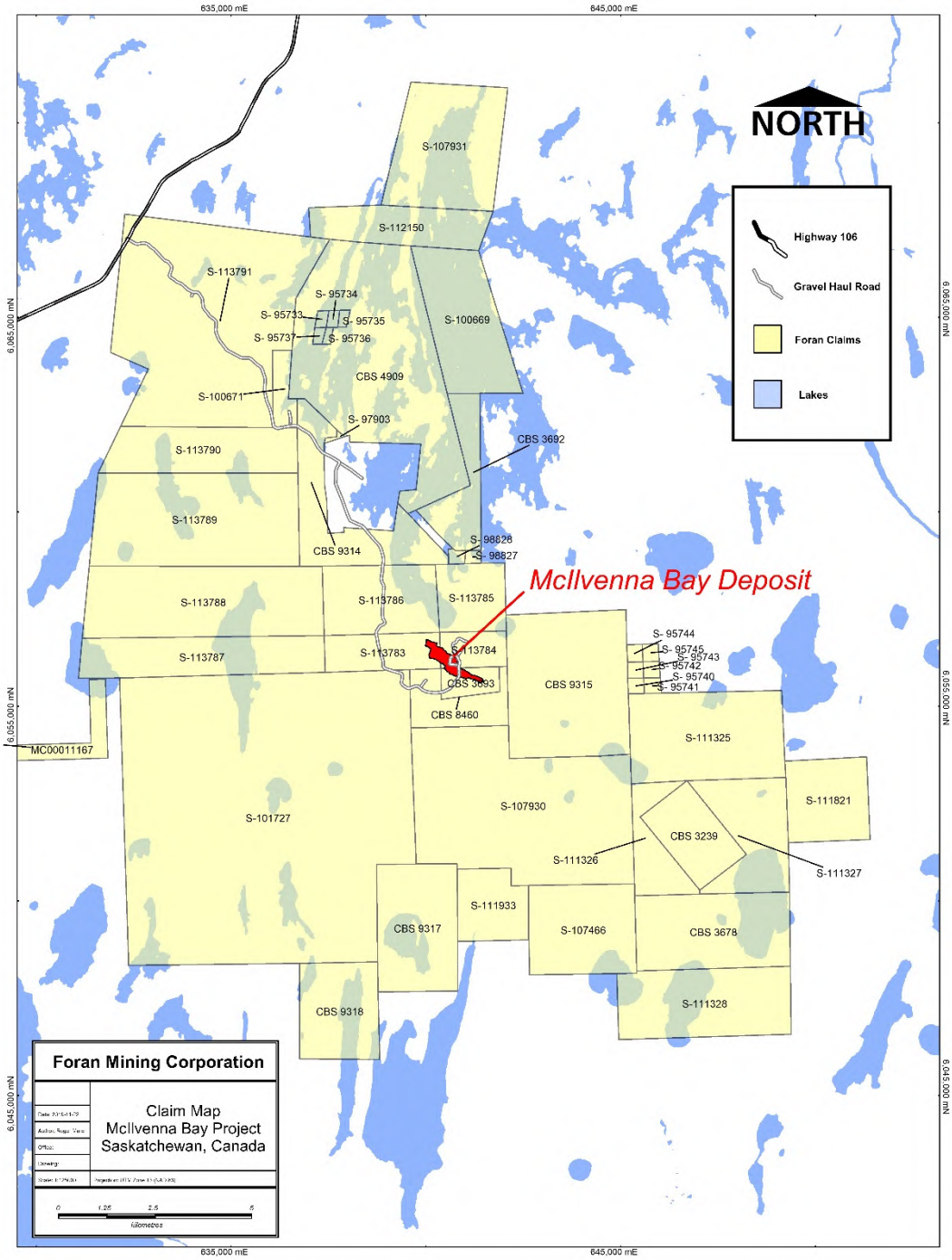


Figure supplied by Foran, June 2019.

Table 4-1: Claim Status for the McIlvenna Bay Property

Property	Disposition No	Owners	Claim Staking Date	Claim Expiry Date	Hectares
McIlvenna Bay	S-113791	Foran Mining Corporation	2011/03/21	2028/06/18	2,255.55
McIlvenna Bay	S-113790	Foran Mining Corporation	2011/03/21	2028/06/18	571.124
McIlvenna Bay	S-113789	Foran Mining Corporation	2011/03/21	2028/06/18	1,261.65
McIlvenna Bay	S-113788	Foran Mining Corporation	2011/03/21	2028/06/18	1,107.29
McIlvenna Bay	S-113787	Foran Mining Corporation	2011/03/21	2028/06/18	624.66
McIlvenna Bay	S-113786	Foran Mining Corporation	1976/12/01	2030/02/28	518.836
McIlvenna Bay	S-113785	Foran Mining Corporation	1976/12/01	2030/02/28	305.373
McIlvenna Bay	S-113784	Foran Mining Corporation	1976/12/01	2030/02/28	157.614
McIlvenna Bay	S-113783	Foran Mining Corporation	1976/12/01	2030/02/28	278.443
McIlvenna Bay	S-101727	Foran Mining Corporation	1991/01/08	2028/04/06	5,283.66
McIlvenna Bay	CBS 8460	Foran Mining Corporation	1988/03/14	2028/06/11	270.35
McIlvenna Bay	S-95733	Foran Mining Corporation	1978/05/01	2028/07/29	12.63
McIlvenna Bay	S-95734	Foran Mining Corporation	1978/05/01	2028/07/29	12.57
McIlvenna Bay	S-95735	Foran Mining Corporation	1978/05/01	2028/07/29	10.58
McIlvenna Bay	S-95736	Foran Mining Corporation	1978/05/01	2028/07/29	8.71
McIlvenna Bay	S-95737	Foran Mining Corporation	1978/05/01	2028/07/29	11.42
McIlvenna Bay	S-97903	Foran Mining Corporation	1990/06/12	2027/09/09	5.51
McIlvenna Bay	CBS 3693	Foran Mining Corporation	1988/02/22	2031/05/22	107.65
McIlvenna Bay	S-111933	Foran Mining Corporation	2011/03/21	2028/06/18	318.68
McIlvenna Bay	S-100671	Foran Mining Corporation	1989/10/19	2029/01/16	102.71
McIlvenna Bay	S-112150	Foran Mining Corporation	2011/03/21	2028/06/18	434.06
McIlvenna Bay	S-107931	Foran Mining Corporation	2006/06/12	2026/09/09	859.02
McIlvenna Bay	S-95741	Foran Mining Corporation	1978/05/01	2028/07/29	17.15
McIlvenna Bay	S-95742	Foran Mining Corporation	1978/05/01	2028/07/29	17.88
McIlvenna Bay	S-95745	Foran Mining Corporation	1978/05/01	2028/07/29	18.79
McIlvenna Bay	CBS 3692	Foran Mining Corporation	1989/06/20	2025/09/17	315.53
McIlvenna Bay	S-100669	Foran Mining Corporation	1989/04/24	2028/07/22	683.88
McIlvenna Bay	CBS 4909	Foran Mining Corporation	1977/04/14	2027/07/12	1,845.78
McIlvenna Bay	CBS 9314	Foran Mining Corporation	1976/12/01	2027/02/28	587.28
McIlvenna Bay	CBS 9315	Foran Mining Corporation	1976/12/01	2027/02/28	1,147.90
McIlvenna Bay	CBS 9317	Foran Mining Corporation	1976/12/01	2027/02/28	675.21
McIlvenna Bay	CBS 9318	Foran Mining Corporation	1976/12/01	2027/02/28	504.09
McIlvenna Bay	S-95740	Foran Mining Corporation	1978/05/01	2028/07/29	16.29
McIlvenna Bay	S-95743	Foran Mining Corporation	1978/05/01	2028/07/29	16.54
McIlvenna Bay	S-95744	Foran Mining Corporation	1978/05/01	2028/07/29	17.84
McIlvenna Bay	S-98827	Foran Mining Corporation	1986/04/07	2029/07/05	13.63
McIlvenna Bay	S-98828	Foran Mining Corporation	1986/04/07	2029/07/05	15.21
McIlvenna Bay	MC00011167	Foran Mining Corporation	2018/05/28	2020/08/20	543.25
				Total:	**20,954.34**

Claim data supplied by Foran, 2019.

Notes: 1 Foran owns 100% of the claims

Foran agreed to assign its interest in the Property Option Agreement between Foran, Cameco, and BHP Billiton to Copper Reef Mines Ltd., newly named Copper Reef Mining Corporation (Copper Reef), a private company organized under the laws of Manitoba. Copper Reef had funded the initial $1.5 million payment and agreed to issue to Foran 5,500,000 common shares of Copper Reef. Subject to regulatory approval, Foran also agreed to subscribe for 2,500,000 units of Copper Reef at a price of $0.20 per unit, which gave Foran a 48.41% equity interest in Copper Reef. Copper Reef is a public company organized under the laws of the Province of Manitoba that trades on the Canadian Stock Exchange.

In a subsequent event, Foran and Copper Reef were in dispute regarding the assignment agreement concerning the Property Option Agreement for McIlvenna Bay. This matter was resolved on May 24, 2006, and under that settlement, Foran made a payment of $2,000,000 for McIlvenna Bay. Foran's $1,500,000 payment to the Hanson Lake Joint Venture on behalf of Copper Reef (Foran contributed $500,000 to Copper Reef for that payment on January 25, 2005) stayed in the Project. Foran gave Copper Reef a 25% interest in the claims, retained 75% for itself, and entered into a joint venture agreement with Copper Reef in which Foran was the operator. Foran retained approximately 25% of shares of Copper Reef and could maintain that percentage through participation in future Copper Reef fund raising. The original 1% NSR in favour of the original Hanson Lake Joint Venture remained the responsibility of the current Foran-Copper Reef joint venture.

On November 3, 2010, Foran announced the closure of an agreement for acquisition of Copper Reef's 25% interest in the McIlvenna Bay property. The deal included transfer to Foran of 3,000,000 Copper Reef shares, and the nearby North Hanson property. In exchange, Copper Reef received 4,000,000 Foran shares (to hold 8% on a non-diluted basis), $1,000,000 cash, a Net Tonnage Royalty of $is0.75/t on future ore produced from the property, and five Manitoba properties selected by Copper Reef from Foran's portfolio.

4.3 Mining Rights in Saskatchewan

Overall regulation of tenure over Mineral Resources in Saskatchewan is conducted under the Crown Minerals Act. The disposition of mineral tenures in Saskatchewan is administered by the Mineral, Lands, and Policy Division of the Ministry of the Economy. Claims on open Crown land, not otherwise reserved from staking, can be applied for via an online facility called the Mineral Administration Registry Saskatchewan (MARS). Mineral tenures comprise claims, permits, and leases. Dispositions acquired before the implementation of MARS are termed "legacy" dispositions, and these are allowed to be held as is until they have been cancelled, surrendered, or otherwise terminated.

Mineral Permits are conveyed for a two-year non-renewable term and may range from 10,000 ha to 50,000 ha in size. The boundary of the area claimed must be configured such that the length is no more than six times the width. They require the posting of a $30,000 performance bond and require expenditures of at least $5.25 per ha over the two-year term of the permit. The bond is refunded when the holder of the permit has complied with the expenditure requirements. All or part of a permit may be converted to a Mineral Claim.

Mineral Claims are smaller but may be maintained for a longer time period than a Mineral Permit. Claims may range from 16 ha to 6,000 ha in size, again, with dimensions such that the length must not exceed six times the width. The term of the tenure is one year, which is renewable upon exploration expenditures according to the following schedule:

- year two to year ten: $15/ha
- thereafter: $25/ha

Both Permits and Claims grant the exclusive right to explore Crown lands, but not the right to remove minerals from the tenure, except for the following activities:

- assaying and testing
- metallurgical, mineralogical, or other scientific studies
- bulk sampling may be conducted, although any minerals recovered in the program remain the property of the Crown

4.4 Permitting, Environmental and Surface Rights

4.4.1 Permitting and Surface Rights

Foran acquired one Industrial Lease for the current exploration camp (#303228), established in 2011, and a second lease for the old campsite located near the deposit (#303458), along with one Miscellaneous Use Permit (MUP #603298) for the camp wastewater lagoon from the Ministry of Environment. These leases/permits are in addition to the pre-existing MUP #602369 for maintenance of the last 8.6 km of private road from the gate at the old Hanson Lake Mine site (public road) to McIlvenna Bay.

There is an old silica sand quarrying operation near McIlvenna Bay which ceased operations in 2014. The site has subsequently been re-claimed and Foran has purchased five quarry dispositions that overlap the McIlvenna Bay deposit. Some additional quarry staking took place west and northwest of McIlvenna Bay in January and February 2012. On December 8, 2012, the Saskatchewan Ministry of Energy and Resources placed a Crown Reserve (CR #965) over McIlvenna Bay that restricts additional quarry staking in the deposit area and subsequently the quarry disposition regulations were amended by the Saskatchewan Government to remove areas of existing mineral tenure from availability for the granting of new dispositions.

The company reports that with the purchase of the over-lapping quarry dispositions, the establishment of the Crown Reserve and changes to the quarry disposition staking regulations by the Saskatchewan Government, the potential land-use conflict between the development of the McIlvenna Bay deposit and quarrying operations has been effectively addressed. The overlapping quarry dispositions were purchased from Preferred Sands on December 22nd, 2014. The QP is not aware of any other constraints on access rights to the property.

Surface rights for the McIlvenna Bay property are retained by the Saskatchewan government and are subject to potential further Industrial Licences and permits should Foran need to expand it footprint on the property.

4.4.2 Social, Community and Land Claims

McIlvenna Bay is located near Hanson Lake in east-central Saskatchewan, approximately 375 km northeast of Saskatoon, Saskatchewan. The closest large communities include Creighton, Saskatchewan and Flin Flon, Manitoba, which are located approximately 65 km west-southwest of the Project. Creighton and Flin Flon have a combined population of approximately 7,100 residents, with

5,600 living in Flin Flon and the remainder in Creighton (Statistics Canada 2012a, 2012b). The economy of the area is primarily based on copper and zinc mining, while tourism and forestry are also of some importance.

HudBay Minerals Inc. (HudBay) operates several mines in the Flin Flon/Snow Lake areas as well as a mill and zinc processing plant in Flin Flon. The 777 Mine located in Flin Flon is nearing the end of its operating life. HudBay recently announced that the mine will reach the end of its reserve life in Q2 2022 (HudBay news release May 6, 2019). Currently, it is unclear what the future plans are for the company or the operations in the area. This is potentially an unfortunate occurrence for the community but should Foran proceed with developing the Project, once a Feasibility Study is completed, it will mean that potentially there will be a trained workforce available for the Project.

McIlvenna Bay lies within the area traditionally occupied by the Peter Ballantyne Cree Nation (PBCN), which is made up of approximately 9,000 members living on more than 36 reserves and/or settlements. The PBCN's traditional territory encompasses roughly 52,000 km^2, from the Saskatchewan/Manitoba border west to the west end of Trade Lake, north to Reindeer Lake, and south to Sturgeon Landing. The Project is located approximately 55 km southeast of the settlement of Deschambault Lake and approximately 100 km west of the community of Denare Beach. Approximately 1,500 PBCN members reside in these communities.

The isolated nature of these communities creates special circumstances for PBCN members working to strengthen their local economies and personal economic well-being. Although rich in natural resources, this sparsely populated region is challenged by infrastructure, education levels, and average income when compared to the rest of the province.

Foran has conducted consultation sessions for the Project in the communities of Deschambault Lake and Denare Beach. Foran also initiated a Traditional Land Use/Knowledge Inventory Study which was completed by ASKI Resource Management and Environmental Services in 2012 (ASKI, 2012). During the study, members of the PBCN communities surveyed clearly articulated their continuing reliance on large game, fish, and waterfowl as well as innumerable plant species, to provide for the physical, social, and spiritual needs of the boreal forest inhabitants.

While most acknowledged that the mining sector does provide the potential for employment and to create spin-off opportunities such as service business in catering, janitorial, trucking, security, grocery and retail supplies, such development must be tempered against the continued reliance of PBCN members on the waters, lands and forests relied on for sustenance, livelihood and spiritual support. As the Project proceeds, Foran will continue to engage the traditional users of the Project area in order to receive input on potential ways and means to minimize, to the extent possible, negative impacts on the traditional use of the lands in the vicinity of McIlvenna Bay site.

4.4.3 Environmental

The Project area lies in the Boreal Plain Ecozone on the boundary of two Ecoregions: the Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion, and the Flin Flon Plain landscape area of the Churchill River Upland Ecoregion. The boundary between these two ecoregions passes through McIlvenna Bay on Hanson Lake, such that the northern part of the study area lies in the Churchill River upland, and the southern part lies in the Mid-Boreal Lowland.

The Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion is characterized by a gently undulating to nearly level landscape, featuring deciduous and coniferous forests with numerous wetlands. Vegetation is generally influenced by landscape and soil types. Peatlands, which comprise approximately one third of the ecoregion, typically consist of tamarack and black spruce interspersed with wet meadows. The Flin Flon Plain landscape area of the Churchill River Upland Ecoregion lies in eastern Saskatchewan's southernmost stretch of Precambrian Shield. Bedrock predominates in this area, with thin deposits of sandy glacial till or glaciolacustrine silt and clay. Vegetation of the Flin Flon Plain landscape is characterised by mixed wood forests. Black spruce is the most common tree species and is largely found in poorly drained peaty areas along with tamarack; however, black spruce is not as abundant as it is in other landscape areas of the boreal shield.

Extensive mining and exploration activities associated with other metal and silica sand mining projects have occurred in the Project area; therefore, the area does not represent undisturbed baseline conditions. Exploration of McIlvenna Bay began in 1988, when it was discovered by Cameco and Esso Minerals Canada (Esso). Cameco suspended exploration in 1991. The Project was optioned by Foran in 1998. Several drill programs were completed between 1998 and 2000, and again between 2011 and 2013. Drilling programs have also been conducted by Foran from 2014 to present during both the winter and summer months.

The site of the past-producing Hanson Lake Mine, operated by Western Nuclear Mines Ltd., (Western Nuclear) lies approximately 5 km north of McIlvenna Bay on the western shore of Bertrum Bay in Hanson Lake. The mine operated between 1966 and 1969 and mined a high-grade copper/zinc/lead VMS deposit. A natural basin north of the mine site was dammed for tailings containment, and runoff from the tailings area originally reported to Bertrum Bay; however, surface flows from the former site currently enter both Bertrum Bay and Mine Bay.

A number of remediation efforts have been completed for the Saskatchewan Ministry of Environment (MOE) regarding this abandoned mine.

A silica sand mine operated by Preferred Sands was located in the immediate vicinity of the Project, approximately 3.6 km from McIlvenna Bay. Production from the site ceased in 2014 and the development area has been re-claimed. This mine was formerly operated by Winn Bay Sand Limited Partnership. Another silica sands project in the area operated by Strong Pine Energy Services (formerly Hanson Lake Sands Company Ltd.) is in the exploration phase.

Aquatic Resources

The aquatic study area (ASA) includes a number of lakes and streams, all of which ultimately flow into Hanson Lake, which drains into the Sturgeon-Weir River. The Sturgeon-Weir River then flows through several large lakes (Amisk Lake, Namew Lake, and Cumberland Lake) to join the Saskatchewan River near Cumberland House. The Saskatchewan River forms part of the Nelson River system, which ultimately discharges into Hudson Bay.

At least 15 species of fish are known to be present in McIlvenna Bay ASA, including lake whitefish, northern pike, walleye, white sucker, and yellow perch; however, none of these species are considered to be of conservation concern. Unnamed Pond is the only waterbody in the Project ASA which does not contain fish. Aquatic habitat mapping indicated a variety of habitat types are present in McIlvenna Bay ASA, with suitable habitat for fish spawning, rearing, feeding, and overwintering provided by most

waterbodies. Evidence of spawning (i.e., eggs) by northern pike and yellow perch was abundant throughout most of the ASA, and the Bad Carrot River was found to be an important spawning migration route/area for white sucker, walleye, northern pike, and yellow perch.

Terrestrial Resources

A number of vegetation species considered rare in the province of Saskatchewan were identified in the Project local study area (LSA) and regional study area (RSA), with conservation rankings ranging from S1 to S3S4 (rare to uncommon). It is noted that the provincial Activity Restriction Guidelines for Sensitive Species apply to vegetation species with conservation rankings between S1 and S3, thus, mitigation for these species may be required (MOE 2014).

Additionally, 63 of the plant species observed within the Project LSA and RSA have documented traditional uses by the Cree and/or Dene people of northern Saskatchewan (Marles 1984; Marles et al 2008, Moerman 2010), although it should be noted that many of these plants are common and widely distributed in the Mid-boreal Lowland and/or Churchill River Upland ecoregions.

A total of 15 species of provincial and federal conservation priority were observed during wildlife field surveys and incidentally in the Project LSA and RSA. Seven of these species are listed federally as species at risk, including common nighthawk (threatened), olive-sided flycatcher (threatened), rusty blackbird (special concern), barn swallow (special concern), horned grebe (special concern), northern leopard frog (special concern), and boreal woodland caribou (threatened). Other observed species that are not federally listed but are considered sensitive in Saskatchewan include bald eagle, Franklin's gull, osprey, American white pelican, double-crested cormorant, common tern, and Canadian toad. McIlvenna Bay LSA and RSA are considered to provide a moderate to high amount of suitable habitat for the species listed above based on field data and supervised satellite image habitat classification.

Heritage Resources

One previously unrecorded heritage resource, GdMq-1, was discovered during the HRIA conducted in the Project LSA during the baseline program. GdMq-1 was found to be of significance due to the discovery of a quartz biface, which is a stone cutting tool or knife that has been flaked on both sides and may have been hafted to a handle (Kooyman, 2000). Additionally, upon further investigation of GdMq-1, three deeply incised dolomite rock crevices were observed in a shelter bay that were large enough to conceal a person, suggesting that this area may have been used as a hunting blind or temporary shelter during the winter.

Environmental Permitting

McIlvenna Bay will most likely require a number of approvals, permits, and authorizations during all stages of the Project following release from the potential provincial and federal EA processes in accordance with various standards outlined in legislation, regulations, and guidelines. Foran will also be required to comply with any other terms and conditions issued by regulatory agencies associated with release from the EA process. Permits and authorizations may also be required from other jurisdictions, such as municipalities, if any are affected.

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

The following section has been extracted from the previous 2019 Micon Technical Report for the McIlvenna Bay Project and updated or edited where necessary.

5.1 Accessibility

McIlvenna Bay is located 1 km south of Hanson Lake, Saskatchewan, and approximately 95 km by road west of Flin Flon, Manitoba. The deposit is located 5 km southeast of the Western Nuclear (or Hanson Lake) Mine, a former producer located on the western shore of Hanson Lake. The McIlvenna site is accessible via an 18 km long all-weather gravel road which connects to Saskatchewan Provincial Highway #106.

The regional mining towns of Flin Flon, Manitoba/Creighton, Saskatchewan (population 7,100), represents the largest commercial/residential centre in the area. Flin Flon provides a railhead that connects the area to the North American railway system. Electrical power would be available from SaskPower at Creighton and/or Island Falls, Saskatchewan.

In addition to the various highways that connect the towns Flin Flon, Manitoba/Creighton, Saskatchewan to various other parts of Manitoba and Saskatchewan, Flin Flon has daily commercial flights to and from Winnipeg, Manitoba.

5.2 Climate

The climate in the Hanson Lake area is continental, with cold winters and moderate to warm summers. The area is classified as having a sub-humid high boreal eco-climate. The mean temperatures for January and July are -21°C and 18°C, respectively. Temperature ranges from -40°C in the winter to 30°C in the summer can be expected. Annual precipitation averages about 350 mm of rain and 1,450 mm of snow. There are on average 119 frost-free days per year. Lake ice thaws in April and returns in November.

In general, exploration can be conducted on a year-round basis except for the fall freeze up and spring break-up periods. Due to the nature of the swampy and muskeg ground conditions the majority of the drilling on the property is confined to winter conditions when the ground is frozen, and access is available.

5.3 Physiography

The property is located within the Boreal Shield Ecozone and is covered with shield-type boreal forest. Topography is flat lying with occasional sharp dolomite cliffs and ridges up to 20 m high. Soil thickness on the limestone ridges is minimal, with occasional rock exposure, and the vegetation is dominated by larger conifer and poplar trees. Below the cliffs are poorly drained muskeg swamps with scattered tamarack and black spruce. Throughout the surrounding area, there are numerous lakes and ponds of various sizes.

McIlvenna Bay of Hanson Lake is at an elevation of approximately 318 m. The base station on the survey grid over the deposit is at an elevation of 325.13 m.

5.4 Local Resources

The Flin Flon-Creighton area has a mining history dating back to the 1920s. Road and rail access is good. General labour, experienced mining professionals and a variety of contractors are available in the area. Local communities are generally supportive of mining.

5.5 Infrastructure

In 2011, Foran permitted and built a new exploration and development camp on the property. This new camp includes a 35-bed trailer camp with office, core shack, shop, and core storage facility.

A gravel road has been built through the property south from Highway 106 to support Foran's exploration programs as well as an adjacent quarrying operation (subsequently re-claimed).

5.6 Sufficiency of Surface Rights

Foran's mineral concessions encompass sufficient area for the construction of all necessary surface works, including tailings storage facilities, temporary stockpiles, processing plant, waste disposal, etc. The local region, including the communities of Flin Flon and Creighton, have enough capacity to house mining personnel.

Power to the project can be provided by SaskPower via a combination of new and existing overhead hydropower lines from existing hydropower infrastructure at Island Falls, SK.

Water for the proposed mining and processing operation could be drawn from water wells and/or one of the local lakes. In addition, water seepage into the underground workings can provide additional water as the workings extend to greater depths.

6. HISTORY

The following section has been extracted from the previous 2019 Micon Technical Report for the McIlvenna Bay Project and updated or edited where necessary.

6.1 General Exploration History Prior to 1998

In 1957, the Parrex Mining Syndicate (Parrex) tested an electromagnetic (EM) conductor delineated under a small bay on the western side of Hanson Lake and intersected impressive zinc-lead massive sulphide mineralization which led to the development of the Hanson Lake (Western Nuclear) mine. The mine operated between 1967 and 1969 and produced 162,200 tons of material averaging 9.99 % Zn, 5.83% Pb, 0.51% Cu, and 4.0 oz/t Ag prior to being shut down. An undisclosed tonnage of unmined resource exists below the workings of the mine. Figure 6-1 is a historical view of the Hanson Lake mine.

Figure 6-1: Historical View of the Hanson Lake Mine

In 1976, the Saskatchewan Mineral Development Corporation (SMDC), the provincial government exploration vehicle that eventually became Cameco, acquired a large exploration lease centered on Hanson Lake. The permit area covered much of the exposed portion of the Hanson Lake Block and extended several kilometres south of the present McIlvenna Bay Property. In 1977, SMDC flew an Aerodat helicopter-borne EM survey across much of the permit area with lines-oriented east-west.

From 1978 to 1988, Cameco tested selected Aerodat EM anomalies with ground follow-up exploration programs consisting of grid establishment, geological mapping (in the exposed portions of the belt),

and ground geophysical surveys which included Horizontal Loop EM (HLEM), Time-Domain EM (TEM), and Surface Pulse EM surveys. Diamond drilling led to the discovery of three new showings, the Miskat Zone (Cu), the Grid B occurrence (Zn), and the Zinc Zone (Zn).

In 1985, the Granges-Troymin joint venture discovered the Balsam Zone, a volcanogenic massive sulphide (VMS) deposit located under the Paleozoic cover, approximately 8 km southeast of Hanson Lake. This prompted Cameco to re-evaluate their existing airborne EM data between the new discovery and Hanson Lake and resulted in a decision to conduct a Mark VI helicopter INPUT survey over the area south of Hanson Lake, with flight lines oriented northeast-southwest. The survey delineated a 1,200 m long INPUT anomaly, striking east-southeast, 1 km south of McIlvenna Bay.

In January 1988, a ground magnetometer and HLEM survey defined the anomaly and six holes were subsequently drilled into what is now McIlvenna Bay. From 1989 to 1991, an additional 61 drill holes were completed. Fifty-six of the holes were drilled to test the deposit, of which only five failed to intersect economically significant mineralization.

Cameco suspended exploration activities at the McIlvenna Bay property after a corporate decision was made not to explore for base metals. Cameco stopped work on the property in 1991 and the property remained idle until optioned in 1998 by Foran.

6.2 Historical Resource and Reserve Estimations

Prior to the McIlvenna Bay Project being originally optioned by Foran in 1998 there were no Mineral Resource or reserve estimations conducted on the property.

Prior to this Technical Report, Foran has issued NI 43-101 Technical Reports containing Mineral Resource Estimates for the McIlvenna Bay Project.

Neither nor the QPs for this report have reviewed any of the previous Mineral Resource Estimates or assessed them for compliance with current CIM Mineral Resource Standards and Definitions as published on May 10, 2014. Foran is not relying on the previous estimates which have been superseded by the current estimate contained in Section 14 of this Technical Report. Therefore, the previous estimates will not be discussed further in this Technical Report.

6.3 Production from the McIlvenna Bay Project

There has been no mineral production on the McIlvenna Bay Project as it relates to the base and precious metal mineralization which Foran has been exploring and drilling.

Silica (fracking) sand quarrying operations have been carried out near McIlvenna Bay and there are quarry dispositions that overlap Foran mineral claims. The quarry dispositions that were overlapping part of the McIlvenna Bay deposit were acquired by Foran from the owner when the sand quarry ceased operation in 2104.

At the current time, the quarrying operations have been shutdown and the site re-claimed.

7. GEOLOGICAL SETTING AND MINERALIZATION

The following section has been extracted from the previous 2019 Micon Technical Report for the McIlvenna Bay Project and updated or edited where necessary.

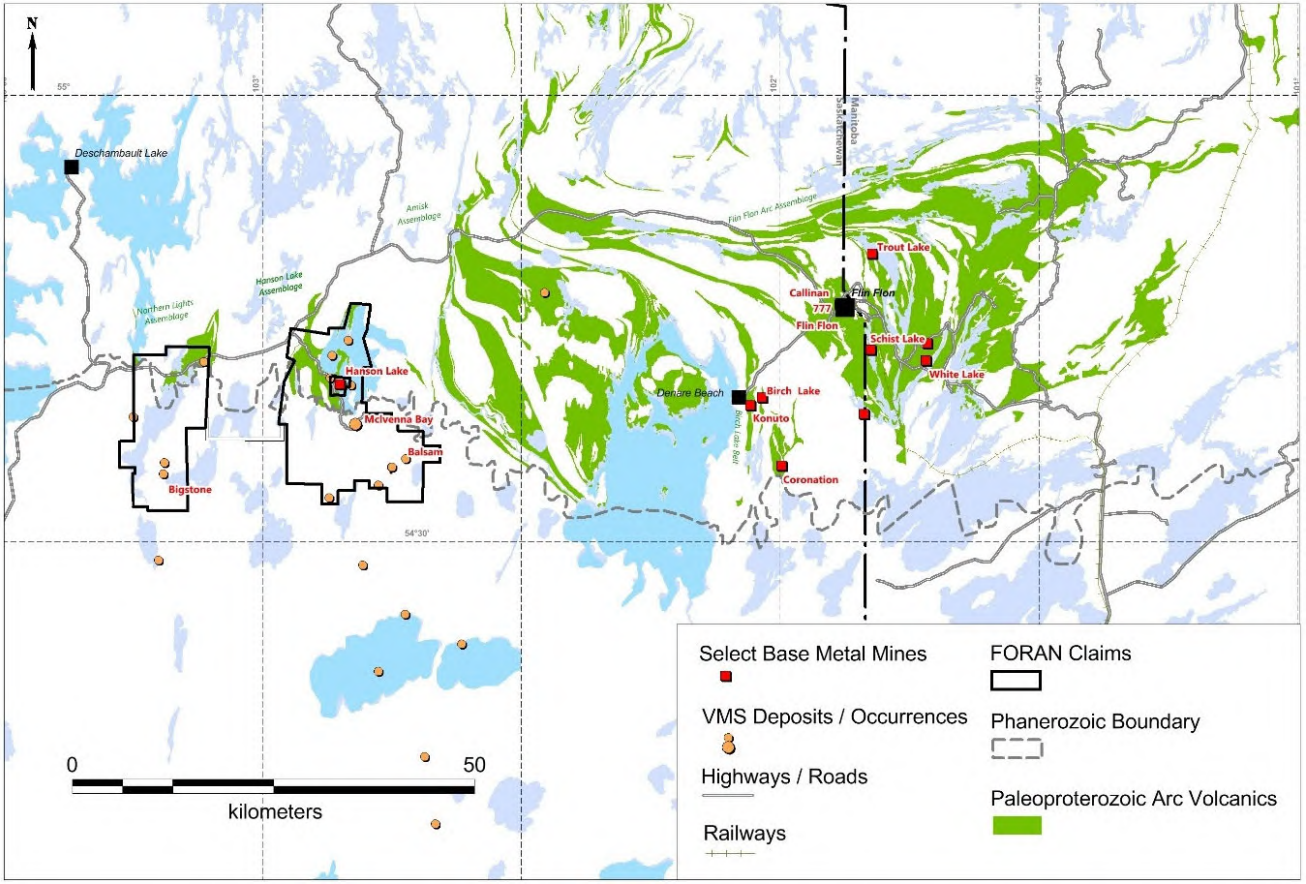
7.1 Regional Geology

The McIlvenna Bay Project is located within a sequence of Paleoproterozoic volcano-plutonic and related sedimentary rocks, termed the Hanson Lake assemblage (HLA), near the western edge of the Flin Flon Greenstone Belt (FFGB) which extends from north central Manitoba into northeastern Saskatchewan (Maxeiner et al., 1999). The FFGB extends 250 kilometres from the Snow Lake mining district in central Manitoba west across the Saskatchewan border (Figure 7.1) and is host to numerous economically significant VHMS deposits. The FFGB is one of the most prolific Cu-Zn-Au-Ag mining belts in the world.

The FFGB is composed of structurally juxtaposed panels of Paleoproterozoic volcanic and related sedimentary and plutonic assemblages that were emplaced in a variety of tectonic environments. The major 1.92-1.88 Ga components include locally significant juvenile arc and juvenile ocean-floor rocks, and minor ocean plateau/ocean island basalt. The juvenile arc assemblage comprises tholeiitic, calc-alkaline, and lesser shoshonitic and boninitic rocks similar in major and trace element geochemistry to modern intra-oceanic arcs. Ocean-floor basalt sequences are exclusively tholeiitic and are geochemically similar to modern N- and E-type Mid-Ocean Ridge Belts (MORBs) erupted in back-arc basins. Evolved arc assemblages and Archean crustal slices are present within the FFGB as minor components.

Collectively, these tectonostratigraphic assemblages were juxtaposed in an accretionary complex ca. 1.88-1.87 Ga, presumably as a result of arc-arc collisions. The collage was basement to 1.87-1.83 Ga, post-accretion arc magmatism, expressed as voluminous calc-alkaline plutons and rarely preserved calc-alkaline to alkaline volcanic rocks. Unroofing of the accretionary collage and deposition of continental alluvial-fluvial sedimentary rocks (Missi Group) and marine turbidites (Burntwood Group) occurred ca. 1.85-1.84 Ga, coeval with the waning stages of post-accretion arc magmatism. The sedimentary suites were imbricated with volcanic assemblages in the eastern FFGB during 1.85-1.82 Ga juxtaposition of the supracrustal rocks along pre-peak metamorphic structures.

Figure 7.1: Regional Geology Map



As currently viewed, the FFGB contains eight geographically separate juvenile island arc volcanic assemblages, each being 20 km to 50 km across. From east to west, they are known as the Snow Lake, Four Mile Island, Sheridon, Flin Flon, Birch Lake, West Amisk, Hanson Lake, and Northern Lights assemblages (Zwanzig et al., 1997 and Maxeiner et al., 1999). These assemblages are separated by major structural features and/or areas of differing tectonostratigraphic origin. It is unclear whether the eight juvenile arc sequences represent different island arcs, or segments of a larger continuous arc (Syme et al., 1999). Within the belt, each tectonostratigraphic block has been broken into several sub-blocks, usually bounded by local to regional fault systems. Correlation of stratigraphy between sub-blocks is difficult to impossible to determine.

The exposed portion of the FFGB is approximately 250 km in an east-west direction by 75 km north-south. Although it has an apparent easterly trend, this is an artefact of the belt's tectonic contact with gneissic metasedimentary, metavolcanic, and plutonic rocks to the north (Kisseynew Domain) and the east-trending trace of Phanerozoic platformal cover rocks to the south. In reality, the FFGB extends hundreds of kilometres to the south-southwest beneath a thin cover of essentially flat-lying, Phanerozoic sedimentary rocks.

By Early Ordovician time, the area of northern Saskatchewan and Manitoba had been effectively peneplaned and a regolith was developed on exposed rocks. Inundation by the Ordovician ocean initiated the deposition of the Phanerozoic cover sequence which, in the McIlvenna Bay area, is now represented by the basal Winnipeg Formation sandstone overlain by the Red River Formation dolomite.

In the general Flin Flon area, the predominant direction for the Late Wisconsinan ice-flow indicators is south-southwest indicating that the ice was flowing from a Keewatin dispersal centre. The resulting tills are thin and generally reflect local bedrock lithologies (McMartin et al., 1999).

7.2 Local Geology

The Hanson Lake Assemblage, the host terrain of McIlvenna Bay, is bound to the east by the Sturgeon-Weir Shear Zone and to the west by the Tabbernor Fault Zone. The block extends an unknown distance to the south beneath a nearly flat lying cover of Ordovician sandstones of the Winnipeg Formation, and dolomites of the Red River Formation. To the north, the block is bound by the Kisseynew Domain, a gneissic metasedimentary belt and the Attitti Complex. The east end of the block hosts the Hanson Lake Pluton, a large compositionally variable granodiorite to pyroxenite intrusion.

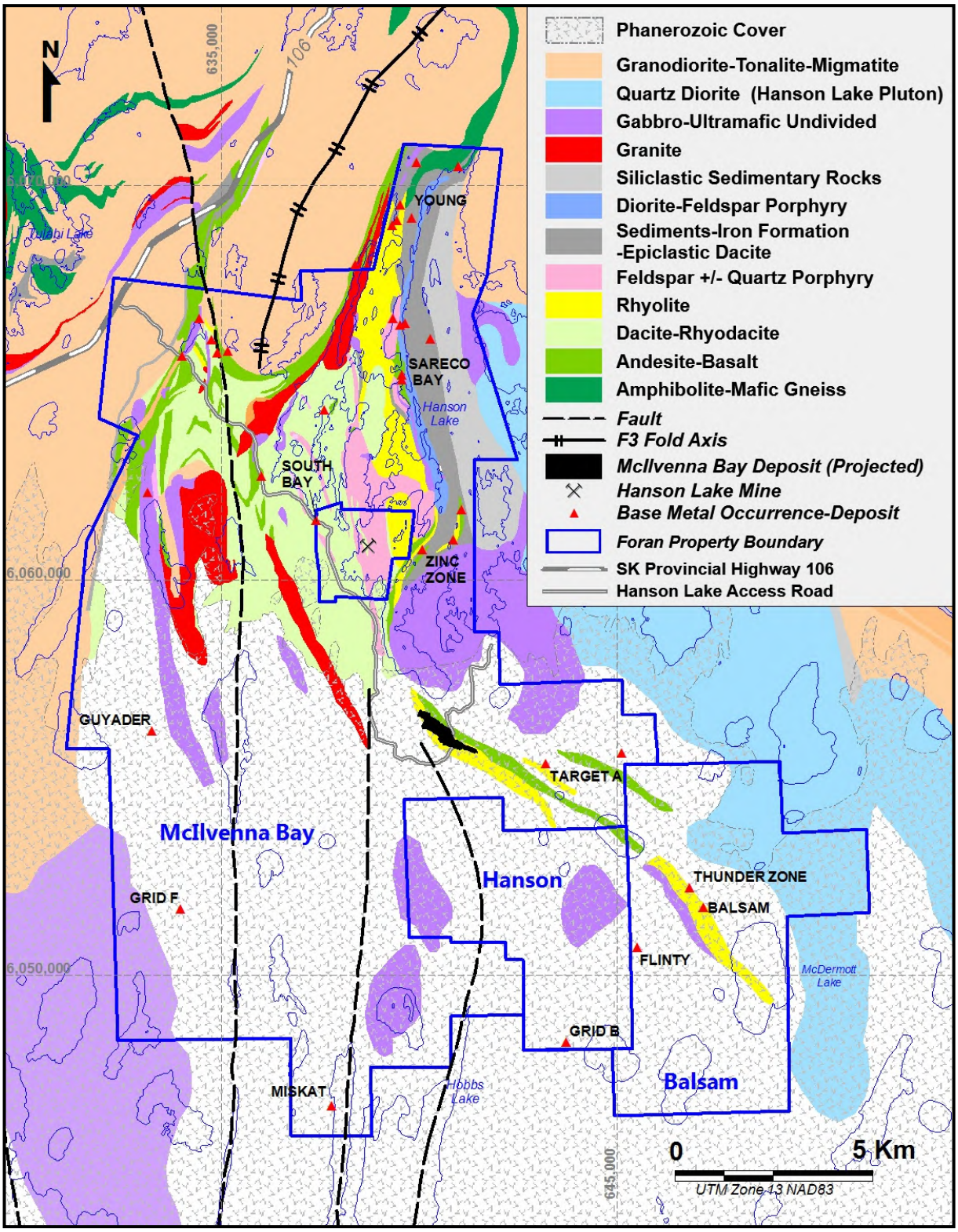
In the Hanson Lake area, north of the Paleozoic margin, the exposed Proterozoic rocks of the Hanson Lake Assemblage (HLA) are dominated by juvenile island arc, felsic to intermediate metavolcanic rocks, with subordinate amounts of mafic volcanics, minor intermediate volcanics, and greywackes. The westernmost part of the HLA is dominated by a succession of chemically distinct metavolcanic dacite, rhyodacite and subordinate mafic and intermediate volcanics, with decreasing mafic constituents and increasing rhyolite to the east (Morelli, R.M and Maxeiner, R.O, 2013). High Zr/TiO2 rhyolite, E-MORB basalt and associated iron formation in the west-central Hanson Lake area and at the McIlvenna Bay and Balsam deposits suggest a rift-related tectonic setting (Morelli, R.M. 2012). The property geology map is provided in Figure 7.2. This figure shows the mapped geology for the exposed northern part of

the property, while the geology for the areas under cover is Inferred based on geophysics and the results of limited exploration drilling in the area.

At least two distinct folding events, both having northerly trending fold axes, have influenced the stratigraphy in the Hanson Lake Area. The Hanson Block structural fabric is dominated by a north to northwest-southeast trending, upright regional transposition foliation. A protracted D_2 structural event resulted in tight to isoclinal, southwest plunging F_2 folds and local southwest verging mylonite zones. D_3 deformation resulted in tight north trending folds followed by a brittle D_4 event characterized by north-south trending faults.

Peak regional metamorphism in the areas west and north of Hanson Lake reached upper amphibolite facies as observed by the partial melting of the granodiorite-tonalite assemblage in the Jackpine and Tulabi Lake areas. At McIlvenna Bay, the Proterozoic sequence exhibits a greenschist metamorphic facies as the deposit alteration assemblages are dominated by sericite and chlorite. The greenschist facies is probably a retrograde event after a previous amphibolite grade since relict cordierite, anthophyllite, garnet and andalusite are commonly observed in the VMS alteration package (Lemaitre, 2000).

Figure 7.2: Project Geology Map (by Foran)



7.3 Deposit Geology

The Proterozoic rocks that host the McIlvenna Bay Deposit are unconformably overlain by an extensive Phanerozoic cover sequence. Due to the lack of outcrop in the area, the geology of the McIlvenna Bay deposit is interpreted from drill core. Stratigraphy in the deposit area strikes between 275° and 295° and generally dips to the north at 65°-70°, becoming near vertical locally. The mineralized zones have the same orientation as the stratigraphy and plunge to the northwest at approximately 45°. Rocks in the host stratigraphy are massive to strongly foliated, the intensity of which depends on the competency of each individual unit and the degree of alteration.

The stratigraphy of the deposit area, defined over almost two kilometres of strike length by 239 drill holes, has been divided into six formations. The formational units are described in detail below. A stratigraphic column displaying relationship between the units is provided in Figure 7.3 and a geological type section through the deposit is provided in Figure 7.4.

Figure 7.3: Stratigraphic Column for the McIlvenna Bay Deposit Area (from Foran, 2011)

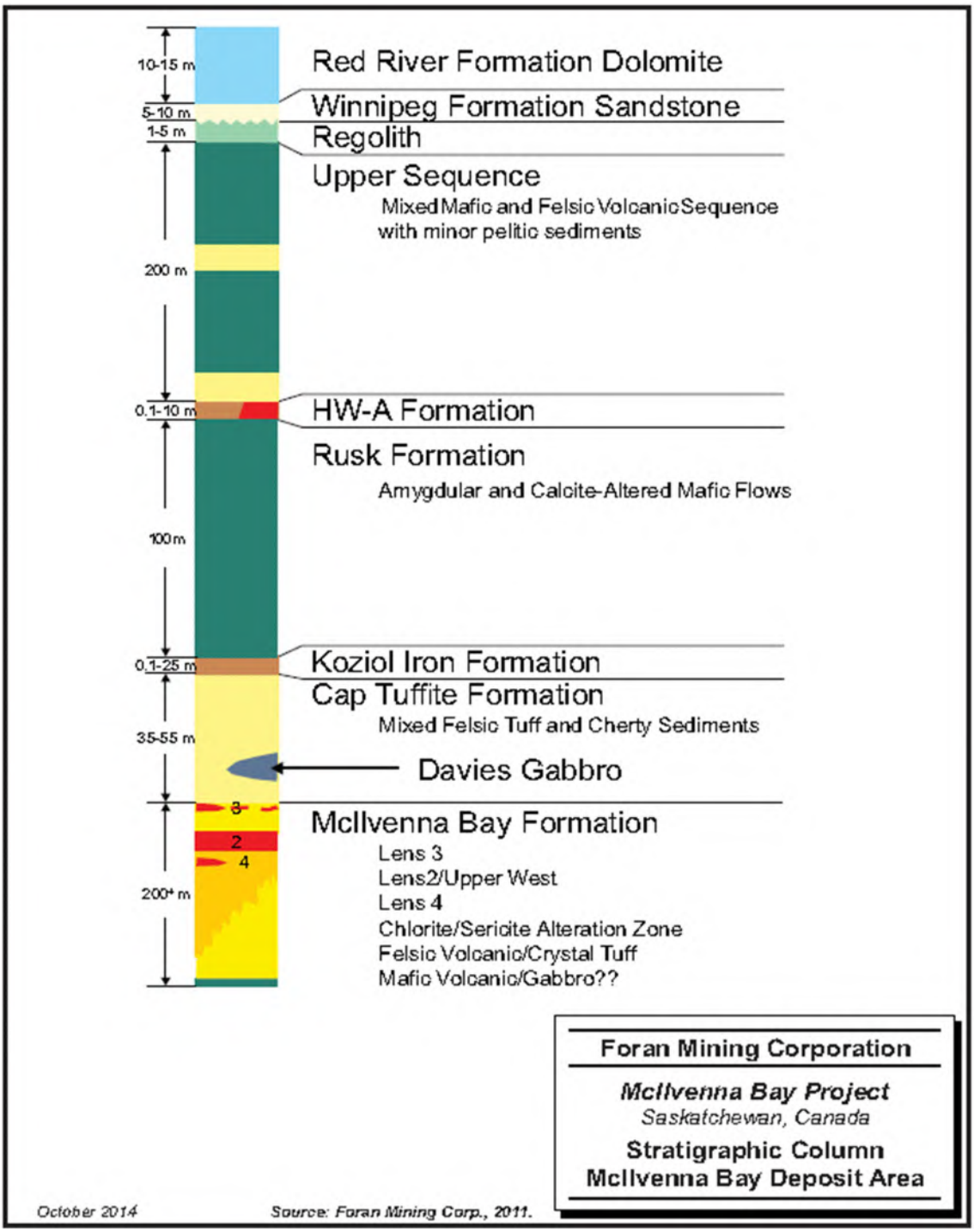
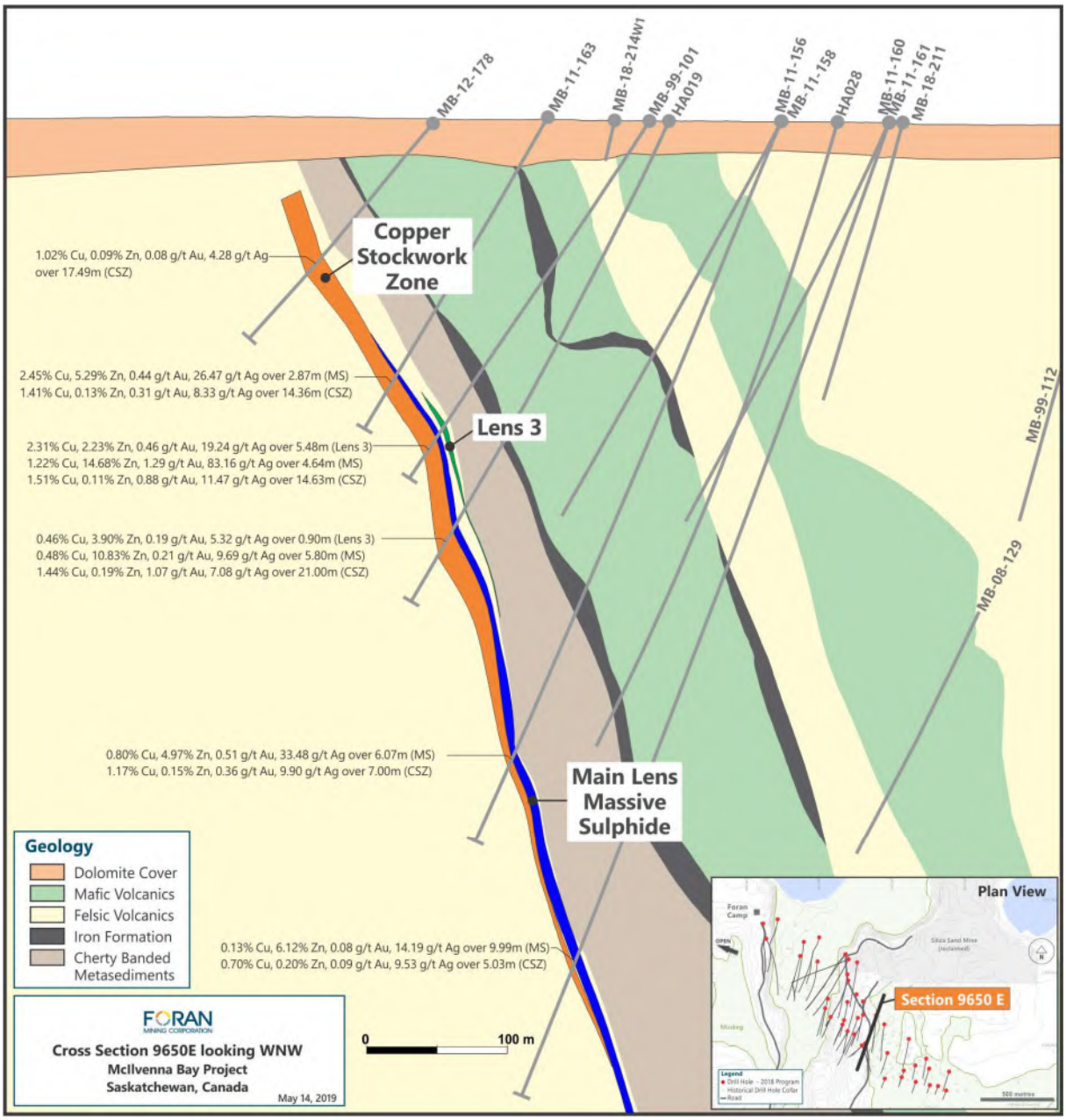


Figure 7.4: Cross-Section 9650E Looking WNW (by Foran, 2019)



The lowest formation intersected by drilling both structurally and stratigraphically is the McIlvenna Bay Formation, the host formation of McIlvenna Bay sulphide deposit. The McIlvenna Bay Formation is known only to the extent it has been drilled below the footwall of the deposit. The formation is at least 200m thick (true thickness) and includes a succession of variably altered felsic volcanics, volcaniclastics, and/or volcanic-derived sediments of rhyolitic composition which are capped by massive and semi-massive sulphides and copper-rich stringer zones that make up the McIlvenna Bay deposit.

The McIlvenna Bay Formation is overlain to the north by the Cap Tuffite Formation. The McIlvenna Bay Formation and the Cap Tuffite Formation may be genetically related but have been separated as they are temporally distinct, as demonstrated by the positioning of the McIlvenna Bay deposit between these two units, an obvious exhalative horizon (and hence a period of clastic and volcanosedimentary quiescence). The Cap Tuffite Formation generally consists of a sequence of intercalated felsic volcanic and cherty metasediments which have been intruded by sills and dykes of the Davies Gabbro (described below). The unit ranges from 35m to 55m thick, is finely banded to finely laminated, and ranges from white to cream to grey-green in colour. Sections of the formation range from very finely laminated, bleached chert to 1 to 10cm thick banded, fine grained, aphanitic rhyolitic tuff. Discrete contacts between the units are nebulous. Instead, wide transitions are observed from one end member to the other. It is believed that the formation represents a sequence of re-deposited, water-lain, distal volcaniclastics and chert.

Stratigraphically overlying the Cap Tuffite Formation is the Koziol Iron Formation, a long continuous exhalative horizon traceable in drill core and by geophysics over several kilometres and, as such, an excellent stratigraphic marker horizon. The unit is a true oxide-facies iron formation that ranges from 0.1m to 25m true thickness and is composed of 1 to 5cm thick bands of fine-grained chert, interbedded with 1 mm to 50 mm massive magnetite bands and 1 cm to 1m thick massive magnetite ± grunerite ± garnet ± chlorite bands. Occasional pyrite and/or pyrrhotite are also observed in selected bands. Near the base of the iron formation is a ±1m thick graphitic shear/fault zone which is oriented sub-parallel to the stratigraphy and/or the S_1 transposition foliation.

Topping the Koziol Iron Formation is the Rusk Formation, a thick package of massive, calcite altered mafic volcanic rocks that are approximately 100m thick. The mafic rocks are likely massive flows, although the thickness of individual flow units cannot be determined from drill core. No distinct flow tops or pillow structures have been observed. The Rusk Formation in turn is overlain by the thin HW-A Formation, an exhalative horizon which ranges from 1cm to 5m thick and shows a transition from oxide-facies iron formation to massive pyrite ± sphalerite.

Overlying the HW-A Formation is the +600m thick Upper Sequence, a bimodal package of volcanic units that have historically been difficult to correlate from hole to hole. With the deep resource drilling completed on the deposit in 2018, additional drill intersections have been obtained through the upper hangingwall sequence which has allowed for additional sub-divisions of this unit to be defined. The Upper Sequence has now been subdivided into three mafic-dominated units interbedded with two felsic-dominated units. Approximately 60-70% of the Upper Sequence is composed of generally fine-grained mafic volcanic rocks (some of which could be mafic intrusions) interbedded with 30-40% generally fine grained to aphanitic, grey, felsic volcanics. The Upper Sequence also contains minor greywacke interbeds (up to about 5% overall) and a number of narrow oxide-facies iron formation horizons that appear to have limited strike length. In the near surface northeast sector of the deposit, there is also a large sill-like granitoid body that intrudes the upper mafic stratigraphy. This unit has been traced along strike for over 300m in drilling, extending downdip for about 400m from the paleosurface with a thickness of up to 200 m.

The stratigraphic package has been cut by several different intrusions, the most significant of which is the Davies Gabbro, represented by a series of sill-like plugs found within the Cap Tuffite Formation. The unit ranges from fine-grained to very coarse grained; the grain size appears to be directly related

to the unit thickness. Chilled margins have been observed on the thicker dykes. It appears that the gabbro intruded along the bedding planes of the wet, cherty banded sediments of the Cap Tuffite.

The Proterozoic basement geology is unconformably overlain by the relatively flat lying to shallowly south-dipping Ordovician dolomites and sandstones of the Red River and Winnipeg Formations which have an average total thickness between 20m and 30m.

7.4 Structure

The McIlvenna Bay stratigraphy appears to have been subjected to at least two main phases of deformation. The first phase of deformation is believed to have been an isoclinal folding event which may have been related to the regional F_2 event (Lemaitre, 2000). This isoclinal folding was responsible for the development the dominant foliation (S_1) in the deposit area, oriented at approximately 280°/65°, and resulted in the transposition of the original bedding into the plane of the S_1 fabric so that the stratigraphy is now oriented sub-parallel to this foliation. The foliation is well developed in the least competent stratigraphic units, particularly the footwall altered rocks.

Isoclinal folding of the iron formation has been observed locally in several drill holes with a plunge that is estimated to be approximately 45° to the west or west-north-west, which is roughly parallel to the plunge of the deposit (Lemaitre, 2000). This may suggest that the plunge of the deposit and the orientation of higher grade/thicker shoots in the deposit may be related to re-orientation during this deformational event.

A strong crenulation (F_3?) of the foliation is locally developed in the stratigraphy, but it is most common in portions of the footwall alteration zone. The plunge of the crenulation is much flatter, usually less than 25°, and trends either north-west or north-east. This trend and plunge of the crenulation appears to be parallel to the fold axis of gentle to open folds observed in banded felsic volcano-sedimentary units both above and below the deposit and may be responsible for the broad warping of the stratigraphy observed in the magnetic maps between the Hanson Lake and the south end of McIlvenna Bay (Lemaitre, 2000).

There is some evidence of faulting documented in drill core in the deposit area. However, it is difficult to determine the orientation, scale, or continuity of any faults at the present time. Often faulting, when present, appears to be oriented sub-parallel to the stratigraphy and may represent discontinuities that helped to facilitate the transposition during deformation.

7.5 Mineralization

McIlvenna Bay is a Volcanogenic Massive Sulphide Deposit (VMS) which consists of structurally modified, stratiform, volcanogenic, polymetallic massive sulphide mineralization and associated stringer style mineralization. The massive to semi-massive sulphides contain copper and/or zinc, with lower concentrations of silver, gold, and lead while the stringer style mineralization generally contains elevated copper and gold. The deposit has undergone moderate to strong deformation and upper greenschist to possibly lower amphibolite facies metamorphism. The sulphide lenses are now attenuated down the plunge to the northwest.

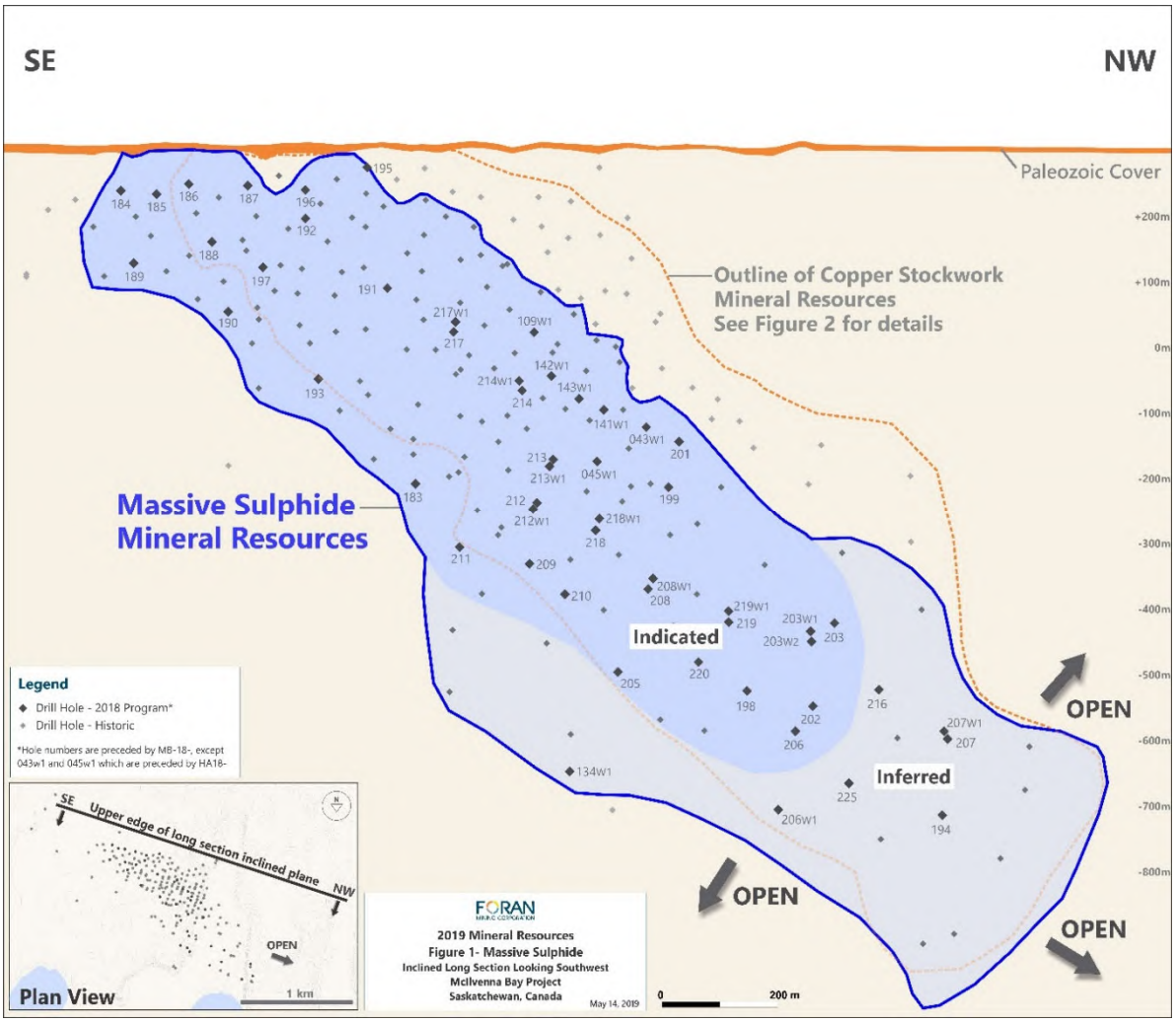
The McIlvenna Bay deposit includes five separate zones and two styles of mineralization that are mineralogically and texturally distinct and typical of VMS deposits, including:

- massive to semi-massive sulphide mineralization in the Main Lens and Lens 3
- stockwork-style sulphide mineralization in CSZ that directly underlies the Main Lens
- two other small lenses of stockwork-style mineralization:
  - The Stringer Zone which is located between the Main Lens and Lens 3.
  - The Copper Stockwork Footwall Zone ('CSFWZ') which occurs as a separate lens underneath the CSZ for approximately 140m of strike length which could represent a fault offset and repetition of the Main Lens and CSZ.

The Main Lens at McIlvenna Bay is a large massive to semi-massive sulphide horizon containing a metal zonation consisting of Cu-Au-rich material near the upper plunge line of the Deposit which transitions down dip into a more Zn-Ag-dominant massive sulphide. In the previous 2013 Resource estimate, the Main Lens was sub-divided into the copper-rich Upper West Zone (UWZ) and the more zinc-rich Zone 2 based on these differences in mineralogy, however for the 2019 Resource Estimate, the Main Lens massive sulphide is reported as a single zone. This change is a result of statistical analysis of the assay grades within the lens, which suggests that there is a gradational transition between the two zones and that a hard boundary is not appropriate; coupled with the fact that they will likely be mined together without any distinction between the zones in the Feasibility Study.

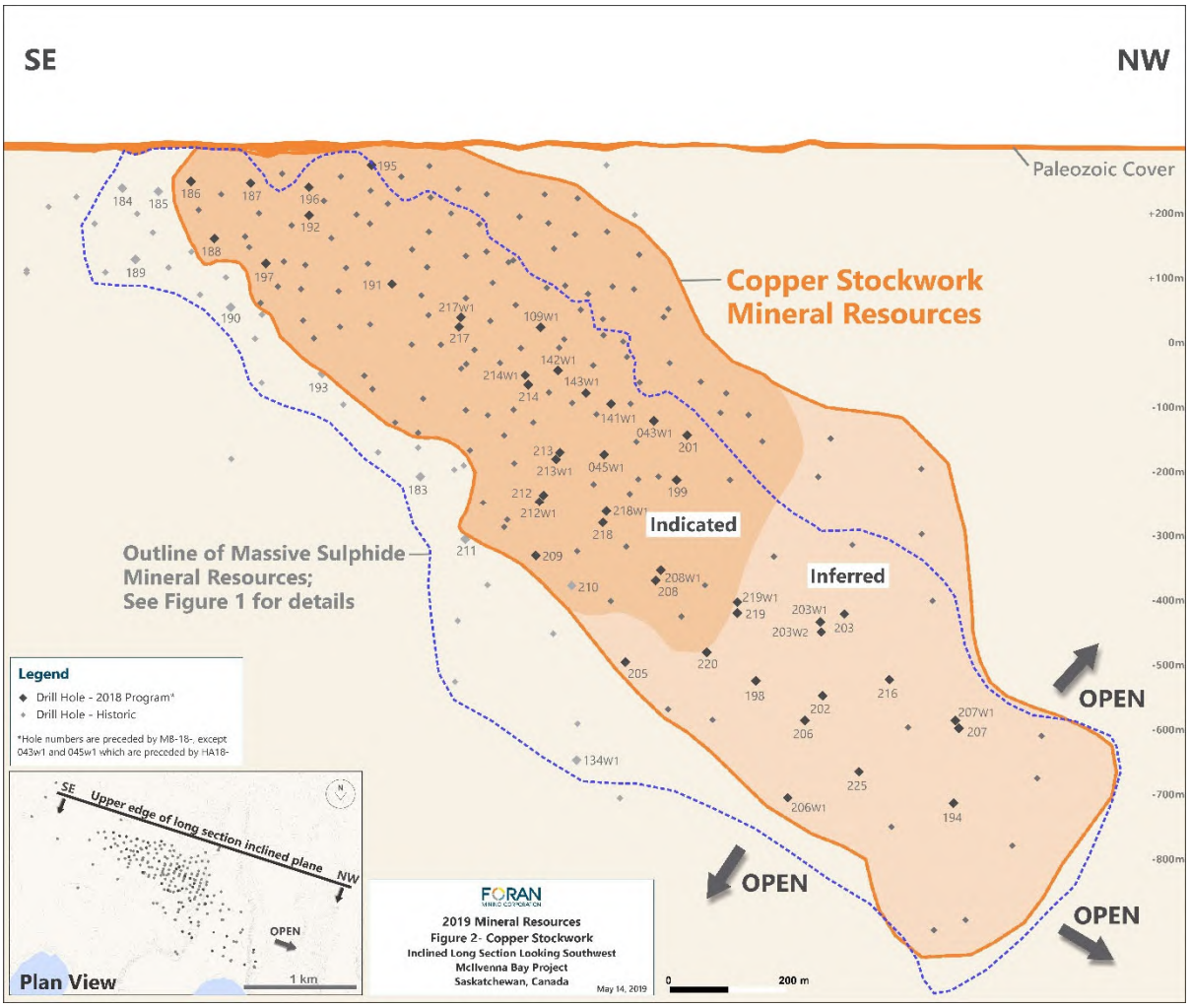
The Main Lens massive sulphide is a continuous mineralized horizon which varies from 0.1 to 36.0 m in thickness and averages 5.5 m overall (Figure 7.5).

Figure 7.5: Longitudinal Section showing the Extent of the Main Lens Massive Sulphide (Foran, 2019)



The CSZ is a zone of stockwork style copper-rich mineralization that directly underlies and is in contact with the massive sulphide. The zone is wedge-shaped, running parallel to the plunge line Main Lens massive sulphide. Based on the limit of current drilling, the zone extends up-dip beyond the upper edge of the massive sulphide for approximately 100-200 m and terminates downdip where it pinches out against the massive sulphide approximately 100-200 m before the Main Lens ends. This unit is interpreted to represent the feeder zone to the massive sulphide system that was transposed into its current geometry during deformation. The CSZ varies from 0.3 to 37.2 m in thickness with an average thickness of 12.1 m. The Main Lens massive sulphide and the underlying CSZ are generally in contact with one another throughout the Deposit, giving the bulk of the Deposit an average thickness of 17.6 m overall. The deposit plunges at approximately 45 degrees from surface for a down plunge length of approximately 2000 m (Figure 7.6).

Figure 7.6: Longitudinal Section showing the Extent of Copper Stockwork Zone (Foran, 2019)



Lens 3 is a massive sulphide lens that sits approximately 10 to 30 m in the hangingwall above the Main Lens and demonstrates the presence of stacked massive sulphide lenses in the Deposit (Figure 7.4). This lens has been traced intermittently along a strike length of 1,440m and plunges parallel to the underlying Main Lens and CSZ. The lens ranges in thickness from 0.1 to 12.5 m and averages 2.8 m.

The Stringer Zone is a narrow intermittent lens of stringer-style sulphide that occurs sporadically between the massive sulphides of the Main Lens and Lens 3 through the Deposit.

The CSFWZ is a separate lens that underlies the CSZ and has been intersected in nine drill holes over approximately 140 m of strike length in the up-dip, central part of the Deposit. The lens varies in thickness from 0.3 to 17 m with an average thickness of 4.4 m. The CSFWZ dominantly consists of stockwork style copper-rich mineralization similar to the CSZ, although in several holes, narrow massive sulphide was also intersected at the top of the interval. It is possible that the CSFWZ represents a fault offset and repetition of the Main Lens and CSZ, but further drilling is required to prove the relationship of this lens to the rest of the Deposit.

Massive to locally semi-massive sulphides are typical of the Main Lens and Lens 3 horizons in the deposit. The massive sulphide mineralization tends to be composed of 70% to 80% medium-sized and sub-rounded pyrite grains resembling 'buckshot' in a fine-grained sphalerite-rich matrix. Sphalerite is hosted as fine-grained and sometimes feathery minerals located in the interstices of the pyrite grains ranging from 5% to 25% of the total unit. The sphalerite is generally dark to medium brown in colour. Faint banding of the massive sulphides is occasionally apparent. Up to 10% fine-grained grey quartz, and occasionally fine calcite, is also observed in the interstices. Subangular to sub-rounded inclusions or fragments of massive black chlorite ranging from 2 to 50 mm in diameter comprise 10% of the unit. Patchy but commonly rounded chert fragments ranging from 1 to 3 cm in diameter can constitute up to 20% of the unit locally. Such chert, when present, is often surrounded by one to three-centimetre-thick zones of enhanced, pale brown sphalerite.

The semi-massive sulphides range from 20% to 60% sulphides which are found as veinlets, veins, and pods within strongly chlorite-altered rock. The sulphide portion tends to be either sphalerite or chalcopyrite dominant, with less than 20% fine-grained pyrite. Sphalerite-dominant portions are generally comprised of reddish or pale brown to blonde sphalerite indicative of zinc-rich and iron-poor sphalerite. Individual veins or pods have been documented to contain up to 56% zinc. Less common are the chalcopyrite-dominant intervals which are composed of 80% chalcopyrite over narrow widths. Veining and replacement textures are common in the semi-massive sulphides.

The CSZ mineralization is confined to the area below the Main Lens massive sulphide, but locally similar stringer styles of mineralization have also been observed between the Main Lens and Lens 3. In these instances, stringer-style mineralization can occur directly above the Main Lens massive sulphide, directly below Lens 3 or in the intervening stratigraphy between the two lenses, where it has been broken out a the "Stringer Zone" in the 2019 resource estimate. The nature of the stockwork zone mineralization varies according to the host rock alteration, but dominantly this style of mineralization is associated with moderate to strong chlorite alteration. Chlorite alteration-hosted copper stockwork mineralization comprises chalcopyrite and pyrrhotite, with occasional pyrite, and is found in veinlets and pods cutting the chlorite. Sericite-quartz altered copper stockwork zones tend to be less prevalent and comprise exclusively chalcopyrite which lines fine, hairline fractures within the strongly silicified host, and as 5 to 10 cm long semi-massive pods containing angular to rounded host rock fragments. These pods and fractures appear to be late brittle features and may suggest that the chalcopyrite was remobilized into fractured rock possibly during deformational events.

The sulphide mineralogy and the size of the alteration footprint suggest the presence of a proximal vent environment along the entire top plunge line of McIlvenna Bay which is represented by the copper-rich portion of the massive sulphide. The location of the Lens 3, and possibly the CSFWZ zones, which respectively overlie and underlie the Main Lens is interpreted by Foran geologists to indicate the occurrences of smaller hydrothermal pulses along different stratigraphic timelines.

In the 2015 report it was noted that "the UW-MS, L2MS, and CSZ all remain open down plunge and, likely, both the zones and the plumbing system underlying them will continue at depth". This point has been demonstrated by subsequent Foran exploration program

8. DEPOSIT TYPES

The following section has been extracted from the previous 2019 Micon Technical Report for the McIlvenna Bay Project and updated or edited where necessary.

The McIlvenna Bay Project hosts a VMS deposit, of a type commonly found in Canada in Precambrian through Mesozoic volcano-sedimentary greenstone belts occupying extensional arc environments such as a rifts or calderas. They are typified by synvolcanic accumulations of sulphide minerals in geological environments characterized by submarine volcanic rocks. The associated volcanic rocks are commonly relatively primitive (tholeiitic to transitional), bimodal and submarine in origin (Galley et al., 2006). The spatial relationship of VMS deposits to synvolcanic faults, rhyolite domes or paleotopographic depressions, caldera rims or subvolcanic intrusions suggests that the deposits were closely related to particular and coincident hydrologic, topographic, and geothermal features on the ocean floor (Lydon, 1990).

VMS deposits are exhalative deposits, formed through the focused discharge of hot, metal-rich hydrothermal fluids. These deposits commonly occur in clusters which form a VMS camp. In many cases, it can be demonstrated that the sub-seafloor fluid convection system was apparently driven by large, 15 to 25 km long, mafic to composite, high level subvolcanic intrusions. The distribution of synvolcanic faults relative to the underlying intrusion determines the size and areal morphology of the camp alteration system and ultimately the size and distribution of the VMS deposit cluster. These fault systems, which act as conduits for volcanic feeder systems and hydrothermal fluids, may remain active through several cycles of volcanic and hydrothermal activity. This can result in several periods of VMS formation at different stratigraphic levels (Galley et al., 2005).

The idealized, undeformed and unmetamorphosed Archean VMS deposit, as exemplified by the Matagami deposits, typically consists of a concordant lens of massive sulphides, composed of 60% or more sulphide minerals (pyrite-pyrrhotite-sphalerite-chalcopyrite with associated magnetite), that is stratigraphically underlain by a discordant stockwork or stringer zone of vein-type sulphide mineralization (pyrite-pyrrhotite-chalcopyrite and magnetite) contained in a pipe of hydrothermally altered rock (Sangster and Scott, 1976). The upper contact of the massive sulphide lens with hanging wall rocks is usually extremely sharp, while the lower contact is gradational into the stringer zone. A single deposit or mine may consist of several individual massive sulphide lenses and their underlying stockwork zones.

It is thought that the stockwork zone represents the near-surface channel ways of a submarine hydrothermal system and the massive sulphide lens represents the accumulation of sulphides precipitated from the hydrothermal solutions, on the sea floor, above and around the discharge vent (Lydon, 1990). VMS deposits are commonly divided into Cu-Zn, Zn-Cu, and Zn-Pb-Cu groups according to their contained ratios of these three metals (Galley et al., 2005).

Most Canadian VMS deposits are characterized by discordant stockwork vein systems or pipes that, unless transposed by structure, commonly underlie the massive sulphide lenses, but may also be present in the immediate hanging wall strata. These pipes, comprised of inner chloritized cores surrounded by an outer zone of sericitization, occur at the centre of more extensive, discordant alteration zones.

The alteration zones and pipe systems often host stringer chalcopyrite-pyrite/pyrrhotite ± Au and may extend vertically below a deposit for several hundred metres or may continue above the deposit for tens to hundreds of metres as a discordant alteration zone (Ansil and Noranda deposits). In some cases, the proximal alteration zone and attendant stockwork/pipe vein mineralization connects a series of stacked massive sulphide lenses (Amulet, Noranda, LaRonde, and Bousquet deposits), representing synchronous and/or sequential phases of mineralization formation during successive breaks in volcanic activity (Galley et al., 2005).

The McIlvenna Bay deposit consists of structurally modified, stratiform, volcanogenic, polymetallic massive sulphide mineralization and associated stringer zone mineralization. The structural deformation and related transposition of the stratigraphy in the deposit area appears to be responsible for the current geometry of the CSZ. This zone of stringer-style mineralization occurs as a compact, continuous zone directly underlying the massive sulphide. The sulphides contain copper and zinc, with low lead and silver and gold values.

The McIlvenna Bay deposit has undergone strong deformation and upper greenschist to amphibolite facies metamorphism. The massive sulphide lenses are now attenuated down the plunge to the northwest. Typical aspect ratios of length down-plunge to width exceed 10:1. The extent of remobilization of sulphides within the deposit is uncertain.

9. EXPLORATION

A portion of the following section has been extracted from the previous 2015 JDS Technical Report and updated to reflect the exploration since the 2015 Technical Report was written. Previous Mineral Resources discussed in this section have all been superseded by the current estimate discussed in Section 14.0 of this report and are only noted for their part in outlining the Foran's sequential exploration history of the McIlvenna Bay Project.

9.1 Foran Exploration 1998 to 2012

9.1.1 Exploration on the McIlvenna Bay Deposit or in the Immediate Area

On acquisition of the property in 1998, Foran embarked on a diamond drilling program to test new targets as well as in-fill the existing drill pattern on the McIlvenna Bay Deposit. Phase I of this program commenced in December 1998 and carried out through the winter of 1998-1999. A total of 55 holes were drilled during this program, totalling 27,958m. Geosight Consulting Canada (Geosight) was retained to prepare a resource estimate using the drill holes completed by previous operators. In 1999, Foran initiated environmental baseline studies and commenced engineering work for construction of a road to access the property.

Drilling continued during the winter of 1999-2000 but, was temporarily halted pending financing. Three holes totalling 2,938m were completed in 2000, and an access road was constructed. M'Ore Exploration Services Ltd (M'Ore) prepared a resource estimate which was released on June 14, 2000. This block model estimate was based on a total of 63,344m of diamond drilling from 124 holes, of which 33,350m of drilling was completed by Foran between December 1998 and May 2000. The mineralization had been delineated to a maximum vertical depth of 1,230 m up to this period.

As of May 31, 2000, Foran had drilled an additional 59 holes totalling 33,350m, with 57 holes directly testing the deposit. The first 44 holes were drilled with the objective of upgrading the quality of the resource, down to a depth of 580m, from the Inferred resource category to the Indicated resource category. The last 15 holes were drilled below the plunge line and down plunge of the deposit with this drilling successful in extending the deposit an additional 300m vertically below the plunge of the previous resource base.

After 2000, exploration work on the property ceased, and the option agreement with the Hanson Lake Joint Venture was allowed to lapse. Foran acquired a new option agreement in 2005 and resumed work. Scott Wilson RPA (a predecessor to RPA Inc.) was retained in 2006 to audit the Mineral Resource Estimate and prepare a NI 43-101 Technical Report (Cook and Moore, 2006). The Mineral Resources dropped significantly owing to an increase in the cut-off grade used, which resulted in removal of much of the Copper Stringer Zone (CSZ) as it was then termed.

In early 2007, Foran completed an airborne deep-penetrating time-domain electromagnetic (VTEM) survey over portions of the Bigstone, Balsam, and McIlvenna Bay properties. The program comprised 404.6 line-km on 150m line spacing over the McIlvenna Bay/Balsam properties and 321 line-km over the Bigstone property.

In the winter of 2007-2008, Foran conducted a diamond drill hole program based on recommendations from the Technical Report on the McIlvenna Bay Project prepared by RPA dated November 27, 2006 (Cook and Moore, 2006). Seven diamond drill holes were completed for a total of 6,455 m. Drill holes were between 691.5m and 1298.4m in length on sections 9400E through 9700E, with the objective of the drilling being to tighten drill hole spacing and upgrade the Mineral Resources down plunge and dip on the L2MS. A number of drill holes failed to intersect the deposit at depth. Subsequently, Foran determined that the holes which missed their targets were drilled at orientations that made it impossible to intersect the deposit at the targeted depths.

Exploration work underwent a hiatus until 2011 when the company was re-financed, and a new management team was brought in to run the company. That winter, Foran conducted a diamond drilling program consisting of 10 holes totalling 5,056m. This program targeted a portion of the CSZ and was designed to in-fill and prove up the continuity of the zone over a portion of the central part of the deposit. Also at that time, some of the drill core from the earlier 2007 to 2008 program was also relogged and sampled.

The winter 2011 drilling was successful in demonstrating good continuity within the CSZ and RPA was retained to update the Mineral Resource Estimate (Rennie, 2011). The zone was re-interpreted, using a nominal 0.5% Cu cut-off grade and a minimum apparent thickness of 3m. The other zones were largely unchanged, with the exception of Lens 4, which was incorporated into the FW. The re-inclusion of the CSZ into the deposit resulted in a large increase in the total 2011 Mineral Resources when compared to the prior 2006 estimate.

Drilling resumed in August 2011 and ran through to November 2011, with a total of 8,158m completed in 18 holes. The purpose of the drill program was to in-fill the deposit to further increase the confidence in the resource, collect sample material for metallurgical test work, and to test the up-dip extension of the CSZ. Detailed geotechnical logging was also conducted, and a suite of samples were collected to initiate geochemical characterization studies of the mineralized zones. Metallurgical sampling was conducted from core collected in a series of HQ-size diamond drill holes. A re-survey program was completed for all of the drill hole collars that could still be identified on the property. In addition, downhole gyroscopic surveys were carried out in 39 of the historic holes along with the 2011 drill holes.

Foran also completed a helicopter-borne geophysical survey in 2011 that comprised 1,587.4 line-km of time domain electromagnetic (VTEMplus) and horizontal magnetic gradiometer (mag) over those areas of the McIlvenna Bay property not covered in 2007 (Figure 9-1).

In 2012, Foran completed 3,825m of diamond drilling in 15 holes. The drilling was completed during a winter program, which allowed access to areas covered by muskeg that were not accessible during the previous summer. The drilling was directed at near-surface projections of the deposit in order to upgrade the classification and extend the known mineralization. Drilling was dominantly completed utilizing HQ-sized core to provide additional material for future metallurgical test work. Geotechnical and hydrogeological studies were also conducted during the program.

Figure 9-1: Geophysical Surveys 2007 to 2014

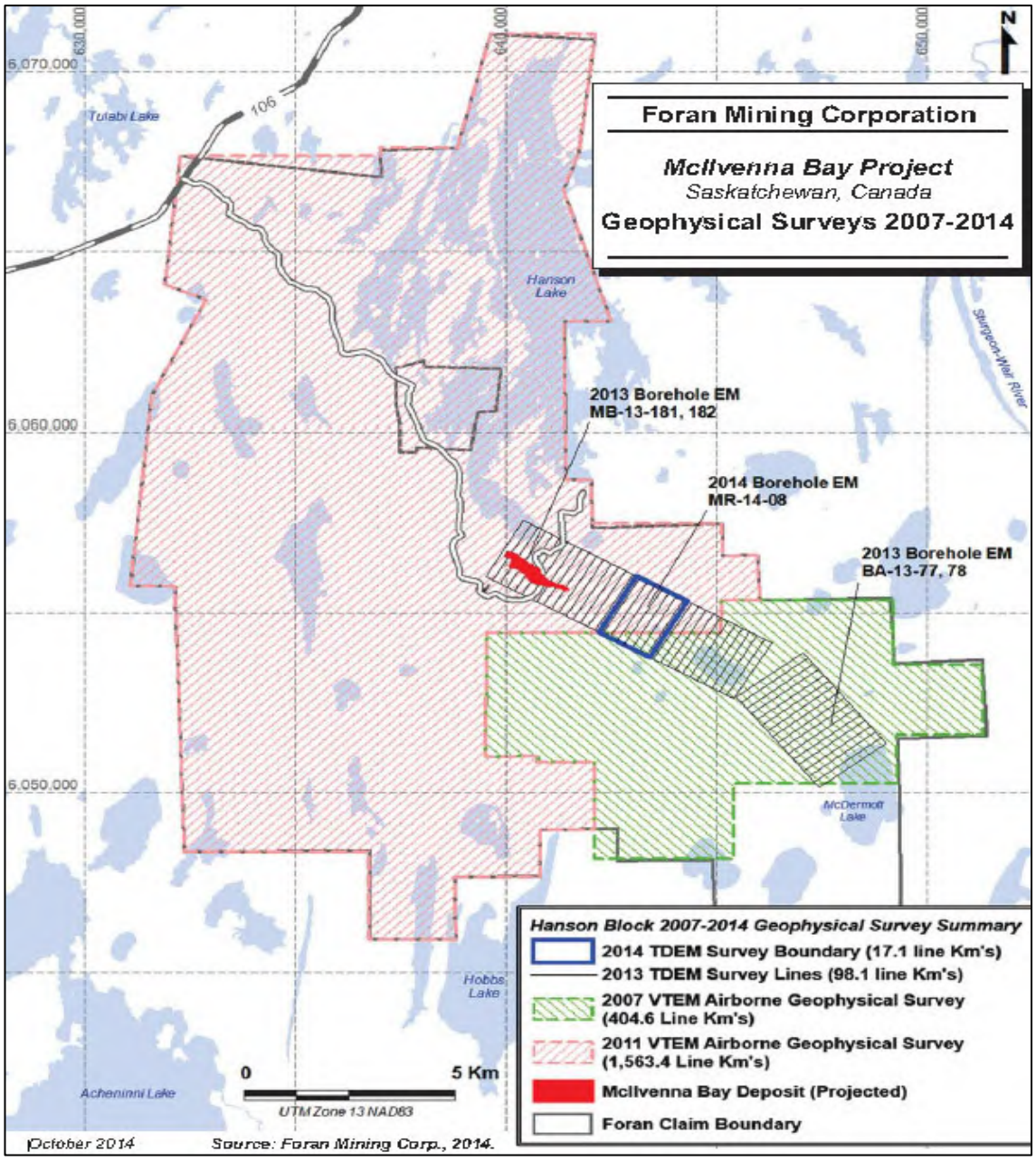


Figure taken from the 2015 JDS Technical Report.

Metallurgical test work on the samples collected from the 2011 drilling was completed in June 2012. The work was carried out by G&T Metallurgical Services Ltd., of Kamloops, BC. Three composite samples, consisting of 516 kg of drill core, were created for each of three different mineralogical domains: the CSZ, L2MS, and UW-MS. The samples were then used in batch and locked cycle flotation testing, as well as determination of Bond Work Indices.

In late 2012, RPA was engaged to prepare an updated Mineral Resource Estimate for the Project, using drill results completed up to that time. The estimate update was completed in March 2013 (Rennie, 2013) and resulted in an increase of 15% in the Indicated tonnage and 18% in the Inferred tonnage. As this increase was not deemed to be material, a new NI 43-101 Technical Report was not triggered. However, the 2013 estimate was used as the basis of the PEA completed by JDS and disclosed in the PEA Technical Report dated January 2015. As with all other Mineral Resources estimated mentioned in this section it has now been superseded by the current Mineral Resource Estimate discussed in Section 14 of this report.

Coincident with the update of the Mineral Resource Estimate, Foran drilled four diamond drill holes totalling 2,243m on the deposit in the winter of 2013. These holes were not incorporated into the previous estimate and a review by RPA concluded that the impact of these holes on the Mineral Resource Estimate used in the 2015 PEA would be negligible. However, these drill holes along with all of the subsequent drilling have been included in the current estimate discussed in Section 14.0.

9.1.2 Exploration Conducted Outside the Immediate Area of the McIlvenna Bay Deposit 2013 to 2014

In addition to the work done on McIlvenna Bay deposit, Foran has conducted exploration activities on the surrounding property area to look for additional deposits. Exploration work carried out in 2013 included 98.1 line-km of ground-based time-domain electromagnetic surveying (TDEM) which covered the McIlvenna Bay deposit and the trend of the geology to the southeast into the Balsam area. The survey grid covered portions of the McIlvenna Bay property, the southeast corner of the Hanson Block claims and a portion of the Balsam property (Figure 9-1). Borehole electromagnetic surveys (BHEM) were carried out in two holes in the Thunder Zone/Balsam areas as well as two others at McIlvenna Bay deposit.

Foran has also drilled a number of holes on regional targets within the property boundary but outside of the immediate McIlvenna Bay area. Figure 9-2 shows the location of these targets and summarizes the amount of drilling done. In 2012 and 2013, Foran drilled six holes, totalling 2,163m on five separate regional targets in the southern portion of the property.

In 2013, nine holes, totalling 3,211m were drilled in the Balsam/Thunder Zone area, located 5 to 7 km southeast of the McIlvenna Bay deposit. Initial drilling during this program targeted areas of known mineralization in the Balsam area to infill them in an attempt to expand the mineralized zones and better understand the stratigraphy of the immediate area. The program was successful in intersecting new mineralization and appeared to indicate that there are several mineralized zones at different stratigraphic levels at Balsam, but that the zones tend to poddy in nature. Near the end of the program a new electromagnetic (EM) conductor was identified as part of the concurrent ground geophysical program. One of the last drill holes of the program tested this anomaly and was successful in intersecting a new zone of mineralization, termed the Thunder Zone along the same geological trend that hosts the McIlvenna Bay deposit. Massive sulphide mineralization was intersected in BA-13-77 which included a 3.66 m intercept grading 4.08% Cu, 0.43 g/t Au, and 27.0 g/t Ag at the Thunder Zone, which appeared to be open for expansion along strike to the northwest.

In 2014, a short geophysical program comprised 17.1 line-km of detailed TDEM was completed along strike to the southeast of the McIlvenna Bay deposit and northwest of the new Thunder zone discovery, to confirm the location and characteristics of a new large deep-seated EM conductor (Target A) also

generated from the 2013 ground geophysical survey. The EM response at Target A had similar characteristics to those observed from the McIlvenna Bay deposit and the late time response of the anomaly suggested a sulphide conductor. Following the detailed geophysics, Foran drilled 1,864m in two holes on Target A, located just east of the McIlvenna Bay deposit (Figure 9-2). The first drill hole was terminated early due to excessive flattening, but the second hole was completed to a depth of 1,683 m, but no significant sulphide mineralization was intersected that would explain the anomaly. The drilling was followed by a BHEM survey, which suggested that the conductor was still present below the hole and the geological logging indicated that the stratigraphy was cut by a dyke at that location of the conductor, so that the source of the conductor was not tested by the drill hole.

Lithogeochemical sampling has been carried out on drill core from McIlvenna Bay, as well as at Thunder Zone/Balsam areas, and in surface exposures in a broad area surrounding Hanson Lake (Figure 9-3). The work was focused on building a chemo-stratigraphy for the rocks of the area. The surface sampling around Hanson Lake was conducted jointly with the Saskatchewan government as part of a company-sponsored master's thesis study. A total of 1,406 samples were collected as part of this program. Final synthesis of the results of this work was included in a M.Sc. Thesis report.

Figure 9-2: Regional Drilling Summary 2011 to 2014

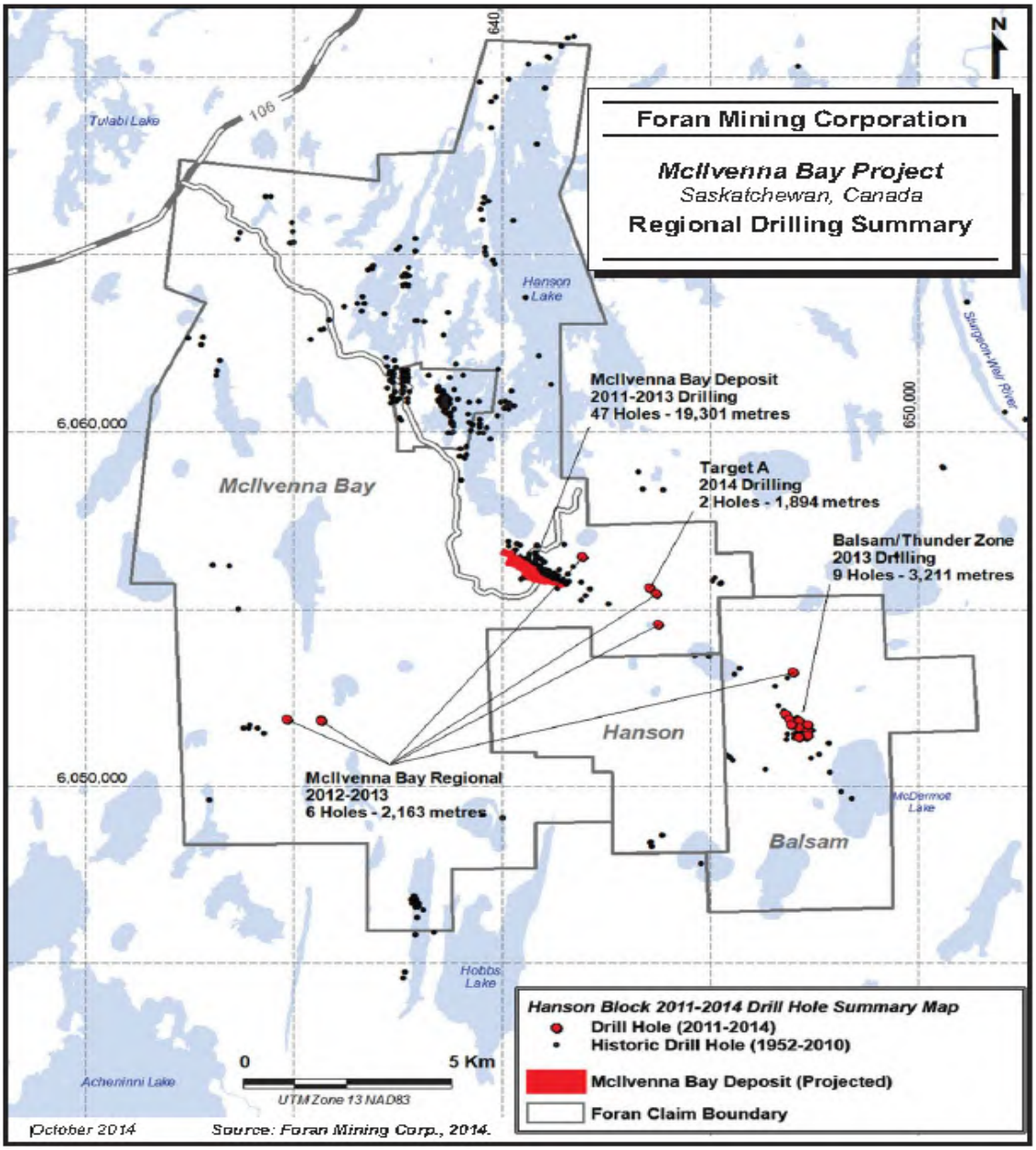


Figure taken from the 2015 JDS Technical Report.

Figure 9-3: Lithogeochemical Sampling Surveys 2012 to 2014

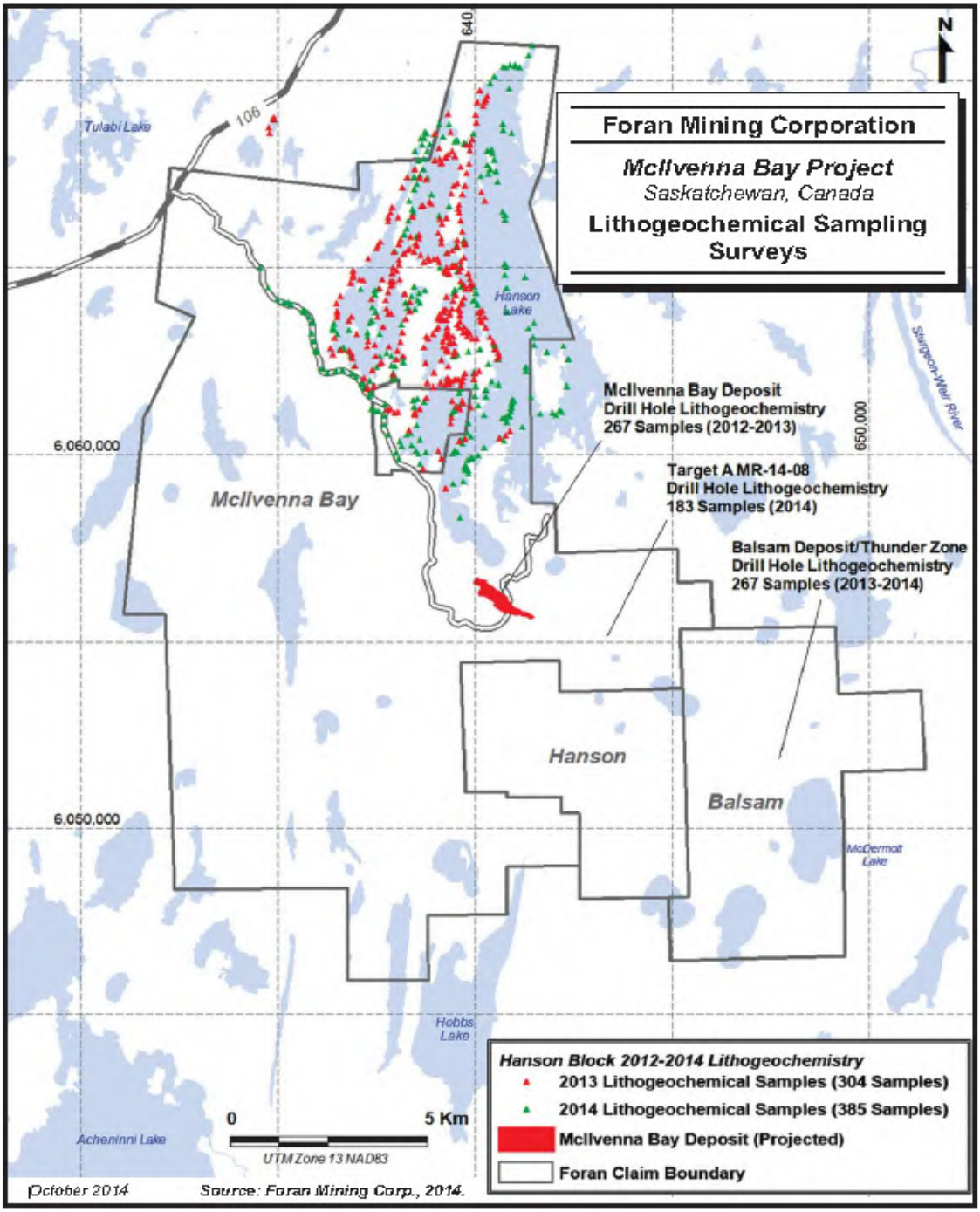


Figure taken from the 2015 JDS Technical Report.

9.2 Foran Exploration 2015 to Present

9.2.1 Exploration on the McIlvenna Bay Deposit or in the Immediate Area

No further exploration/drilling was conducted on the McIlvenna Bay deposit until the winter of 2018. In December, 2017 Foran signed a Technical Services Agreement with Glencore Canada Corporation, under which Glencore will contribute its professional and technical services, assistance, guidance and advice in connection with the objective of completing a Feasibility Study on McIlvenna Bay, in exchange for an exclusive off-take contract to purchase or toll process all of the concentrates and/or other mineral products produced from the Project at prevailing market rates. With this agreement in place, Foran embarked on a large infill and expansion drill program designed to convert as much of the deposit resource as possible into Indicated categories which could potentially be converted into reserves for the upcoming Feasibility study.

In 2018, Foran conducted 26,827m of drilling in 60 drill holes targeting the deposit. The program was completed in two phases, with 14,986.5m in 32 drill holes (including several wedged holes) completed during the phase I winter program and 11,840.5m in 28 holes (including wedges) completed during the phase II summer program. The focus of the winter program was to upgrade both the near surface and deep portions of the deposit which are covered by muskeg and not accessible during summer months. While the summer program focused on the middle part of the deposit which was accessible from high ground. Both programs were completed using oriented coring techniques to provide a better understanding of the geological structures in the deposit area. A number of wedge holes were also drilled during the programs in order to provide additional material for metallurgical sampling. In addition to converting resources to the Indicated category, other program components included geotechnical, hydrogeological, and metallurgical testwork.

Geotechnical components of program included 3,733m of detailed geotechnical logging on resource drillholes drilled at orientations amenable to both structural and resource studies. In addition to the resource holes, three short geotechnical holes (151.3m) were drilled to characterize the proposed portal location and four short vertical holes (104m) were drilled for piezometer installations to help quantify near surface groundwater flow in the immediate deposit area.

Material for metallurgical testwork was collected from all phase I and II drillholes, with either a quarter or half of each piece of sampled material submitted for testing. Metallurgical work is being carried out by Base Metallurgical Laboratories Ltd., of Kamloops, BC. A total of 1440.96 kg of drill core was provided from 2018 drilling, supplemented with 712.4 kg of coarse rejects from assayed material from the 2018 program. Another 38.34 kg of core material from 2011 drilling was collected for HLS testing. Test work consisting of grinding and floatation circuit tests, including some DMS studies, have been completed on these samples. The results of this test work have been incorporated into the Pre-feasibility Study that is the subject of this report.

As a part of phase II summer drilling, a downhole resurveying program was also undertaken. A number of holes were identified that did not have a full gyro surveys completed during the 2011 downhole resurvey program due to blockages in drill holes at surface or at depth. Those holes that displayed suspicious or non-existent historic downhole surveys beyond blockages were re-opened with a drill on the pad and re-surveyed with a True North Gyro.

To develop a larger library of ore density measurements across the deposit, Foran employees collected 1932 bulk density measurements from both 2018 drill holes, and historic core (from 2011, 2012 and 2007), that was not significantly weathered. Bulk density measurements were matched to sampled intervals, with individual pieces labelled to ensure correct wet and dry weights. Samples were measured using a larger scale than the regular specific gravity measurements. The precision of the scale used was within 1 g (0.5 g for skilled operators), therefore the larger sample sizes (often between 2 and 4 kg) minimized the error introduced by the 1.0 g precision. These bulk density samples are much more representative of the actual density of the mineralized material in the ground compared measurements taken from isolated random small samples of core.

As a follow up to both programs BHEM surveys were completed on a number of holes to look for additional lenses below the level of current drilling. The program was successful in its mandate and culminated with the 2019 resource estimates which is the subject of this report.

9.2.2 Exploration Conducted Outside the Immediate Area of the McIlvenna Bay Deposit

Since 2015 Foran has completed several drill programs in the McIlvenna Bay area, targeting geophysical anomalies generated from the 2013 ground TDEM survey discussed above.

In 2015, Foran completed five drill holes encompassing 1,914m at the Thunder zone to follow up on the new discovery from 2013 discussed above. The program was successful in intersecting massive and stringer sulphide mineralization in four of the five holes drilled which defines a mineralized zone over approximately 300m of strike length that remains open for expansion. The best result from the program came from the last hole, BA-15-83 which intersected two zones, an upper zone containing 2.04% Cu, 3.47% Zn, 0.37 g/t Au and 11.6 g/t Ag over 3.46 m followed down hole by a second zone containing 0.62% Cu, 3.41% Zn, 0.36 g/t Au and 27.24 g/t Ag over 8.39m (including an interval of 3.70m grading 7.16% Zn).

During the winter of 2017, Foran returned to follow up on the Target A EM conductor first drilled in 2014. One hole was drilled during the 2017 program to attempt to intersect the conductor down dip of the 2014 drill hole. The hole (MR-17-09) was drilled to a depth of 1,323m (short of the target depth) before an early spring thaw forced the shutdown of the drilling. The rods were left hanging in the hole when the drill rig was demobilized to facilitate the completion of the hole during the winter 2018 program.

During the 2018 winter program at McIlvenna Bay, MR-17-09 was extended to completion, reaching a final depth of 1,542m. The hole hit a zone of exhalative material/iron formation from 1,386 to 1,396m with strong silicification, garnet growth and chlorite alteration along with net textured to locally semi-massive pyrrhotite (2% to 40% overall) and trace pyrite and/or chalcopyrite. It was determined with a subsequent BHEM survey that this zone likely represented the source conductor, but based on the response, the hole had only intersected the upper edge of the modelled plate and the strongest part of the conductor still lay below the hole.

In 2019, Foran returned to Target A to continue the drilling and attempt to get a better test of the conductor by targeting the centre of the modelled conductor plate. Initially the plan was to wedge a short hole off of MR-17-09, but due to technical difficulties, multiple attempts to wedge the hole failed and it was decided to collar a new hole targeting the centre of the conductor plate. This hole MR-19-10 was drilled to a depth of 1,749m and intersected a package dominantly consisting of altered and

silicified exhalate/iron formation from 1,547 to 1,572m, which was similar to the zone from MR-17-09. The zone contained variable amounts of pyrrhotite with trace pyrite and chalcopyrite as above, but the interval was also cut by a number of graphitic shears/faults which may have contributed to the EM response. There were no appreciable base metals associated with the zone at this location but given the large size of the modelled conductor plate, further follow-up exploration is warranted.

9.3 Qualified Person's Comments

The exploration programs conducted by Foran to date on the Project have continued to outline the extent of the mineralization at the McIlvenna Bay deposit. Further work will be needed to determine the full extent of the mineralization both in the down plunge direction and at depth. Due to the nature of the mineralization, should Foran take the steps necessary to put the Project into production, the extent of the mineralization either down plunge or at depth would be more economically defined by underground drilling.

Further exploration programs will be necessary to identify the extent and tenure of the mineralization in a number of secondary zones which have been identified either historically or more recently by Foran. Further exploration will also be able to determine if these secondary zones of mineralization are economically viable to be used as secondary sources for material for the purposes of exploitation.

10. DRILLING

The first portion of this Section 10.1 has been extracted from the previous 2015 JDS Technical Report. The second portion of this Section discussing the drilling since 2014 reflects the work conducted by Foran since the January 2015, Technical Report was published.

10.1 Drilling to 2014 (RPA Discussion)

Diamond drilling has spanned a fairly broad period, starting with Cameco in 1988. Cameco (and partners) drilled 68 holes, of which 56 targeted the McIlvenna Bay deposit. All other drilling in and around the Project area has been completed by Foran. A summary of drilling within McIlvenna Bay deposit is provided in Table 10-1.

Table 10-1: McIlvenna Bay Deposit Diamond Drilling Summary to August 2014

Company	Year	Number of Holes	Metres Drilled (m)
SMDC (with partners Esso, Tri-gold)	1988	26	7,702.00
Cameco (SMDC) (with partner Trimin)	1989	30	14,550.53
Cameco (with partner Billiton)	1990	13	7,693.70
Foran	1998	3	997
Foran	1999	62	28,992.70
Foran	2000	3	2,938.30
Foran	2007	3	3,214.20
Foran	2008	4	3,310.70
Foran	2011 Phase I	10	5,056.00
Foran	2011 Phase II	18	8,158.00
Foran	2012	15	3,825.00
Foran	2013	4	2,243.00
	TOTAL	**191**	**88,681.13**

Table taken from the 2015 JDS Technical Report.

RPA noted that the totals provided by Foran for the Cameco-era drilling do not match that contained in the database. The database contains 68 of these holes totalling 30,905.6m of drilling versus 69 holes and 29,946.2m of drilling as listed in Table 10-1. The apparent discrepancies were due to holes that were lost and re-collared, and other holes that were drilled by Cameco and subsequently lengthened by Foran. Some holes that were collared and then abandoned appear in the database, and some do not, so it is was not really possible to reconcile the drilled totals. The metres from the lengthened holes are contained within the database as though they were drilled by Cameco, but they should have been recorded as drilled by Foran. For some of the abandoned and lengthened holes, the records are not complete. Consequently, it is not possible to fully reconcile what is in the database, which is supported by logs, and what is reported. In some instances, Foran has re-logged older drill core to update the records.

The incidents of apparent discrepancies have been investigated by Foran personnel and documented as follows:

- Hole 22, collared by SMDC/Esso in 1988, was deepened by Foran in 1999
- log for Hole 7 is missing
- Holes 35 and 40, collared by Cameco/Trimin in 1989, were lost and re-collared as 35A and 40A, respectively; original drilled intervals not recorded
- log for Hole 42 is missing
- Hole 43, also collared by Cameco/Trimin in 1989, was deepened by Foran in 1999
- Holes 58, 66, and 67, collared by Cameco/Billiton in 1990, subsequently deepened by Foran
- Holes 62 and 63 also appear to have been deepened, but it is not clear by whom
- no logs were available for holes 62 or 58D
- Holes 68, 120, and 121 were collared by Foran, lost, and re-drilled; now recorded as 68A, 120A, and 121A, respectively
- Hole 122W1 was drilled as a wedge
- Hole 123 was not drilled in the deposit area, and therefore not included in McIlvenna Bay database
- Holes 126, 130, and 131 were planned but not drilled, and so records with these hole numbers do not exist

In RPA's opinion, these apparent discrepancies have been adequately explained and do not present a significant concern for the drill hole database particularly as the only data used for resource estimation is recorded in logs and verifiable or has been re-acquired through logging of early core.

Cameco and Foran employed similar drilling procedures on McIlvenna Bay. The top of the holes from surface down through the Paleozoic cover sequence was drilled with HQ equipment. The drill string was reduced to NQ for drilling below the Proterozoic regolith. All but a handful of the Cameco holes, and all of the Foran holes still have their HQ rod string in the hole allowing one to locate the holes on surface and to re-enter them if necessary.

Downhole surveying of Cameco holes HA-60 through HA-65 was completed using acid tests only. Holes HA-01 through HA-17, and HA-66 and HA-67 were completed using Tropari and acid test measurements. All other Cameco holes were surveyed using the Techdel International Light-Log system.

Initially, downhole surveying on the Foran holes was done using a combination of Tropari measurements and acid tests. Due to the presence of magnetic rocks in the stratigraphy, especially the iron formations, Tropari azimuths were sometimes inaccurate and were occasionally ignored in order to get reasonably accurate hole locations. Tropari measurements were taken at approximately 75m intervals, and acid tests were taken every 50 m.

The use of Tropari measurements was considered acceptable for the shorter holes as the influence of the one or two iron formation horizons intersected in such holes could be eliminated by careful analysis of the Tropari data, logging of the core, and magnetic susceptibility measurements of the core from area around the survey location. However, the Tropari instrument was found to be totally inadequate as a surveying tool for the deep, step-out holes 67, 111, 120A, 122, 122W1, 124, and 125. Foran concluded that the locations of the intersections of these holes had an estimated error of ±50m in the east-west direction and ±25m in the vertical direction (Lemaitre, 2000).

Starting with the winter program of 2011, the holes were surveyed initially with a Reflex EZ Shot instrument by the drillers during the drilling process as a means of tracking the trend of the drill hole during drilling. The EZ Shot tool provides an accurate dip, but also uses a magnetic compass to determine the azimuth. At the completion of the program, holes MB-11-136 to -145, inclusive, were re-surveyed using a Gyro tool from Reflex Instruments, which is not affected by magnetics. There were significant differences found between the results for the two instruments. Based on this result, the gyro tool was deemed to provide the most accurate survey result and this tool was used for all subsequent downhole surveys used in the database. For all future drill programs, a similar protocol was followed, with an EZ Shot tool employed by the drillers for routine tracking of the hole at 50 m intervals during drilling and a final gyro survey completed at the end of the hole to provide an accurate hole trace for the database.

In 2011, a program of re-surveying was also conducted to re-located as many of the older drill collars as possible to validate the historic database. Where the casing could be found and the holes were still open, a downhole survey was redone using the Gyro instrument. This resulted in revisions to the locations and paths of some holes, which impacted the geological interpretations and grade interpolations. In RPA's opinion, this was a prudent and worthwhile exercise, as there were some significant changes made to the projected path of some holes.

A drill hole location map showing the drill holes up to August 2014, is provided in

Figure 10-1. In RPA's opinion, the drilling and surveying conducted on the property has been done to industry standards and there are no apparent issues that would have a significant deleterious impact on the estimation of Mineral Resources.

Figure 10-1: Drill Collar Locations to August 2014

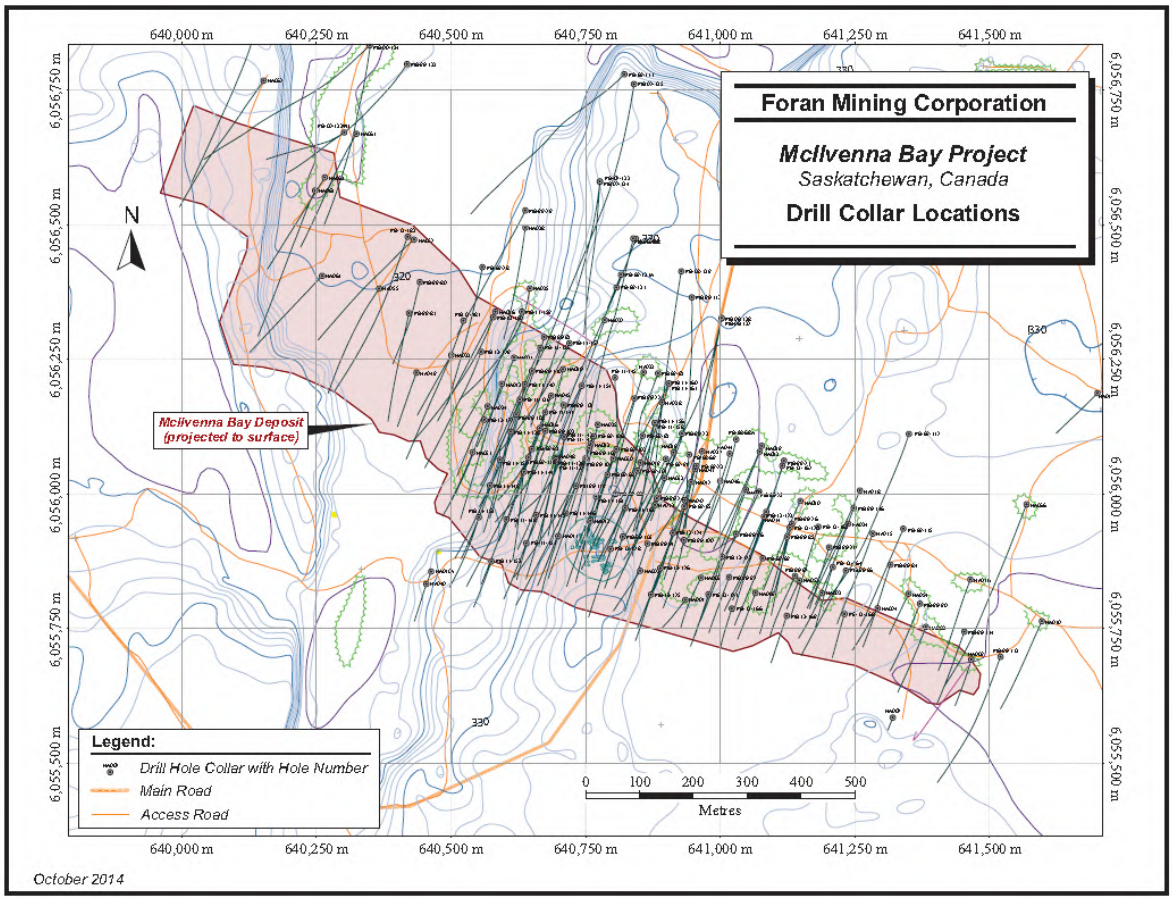


Figure taken from the 2015 JDS Technical Report.

10.2 Foran Diamond Drilling 2014 to Present

No further drilling was conducted on the McIlvenna Bay deposit until the winter of 2018, when Foran embarked on a large infill and expansion drill program at the deposit designed to convert as much of the deposit resource as possible into Indicated categories which could potentially be converted into reserves for the further studies.

The 2018 program consisted of 26,827m of drilling in 60 drill holes which was completed in two phases, with 14,986.5m in 32 drill holes (including several wedged holes) completed during the phase I winter program and 11,840.5m in 28 holes (including wedges) completed during the phase II summer program.

Table 10-2 and Table 10-3 provide detailed information on the drill holes from the 2018 program and plan map showing the collar locations and hole traces is provided in Figure 10-2.

Table 10-2: Summary of the 2018 Phase I Diamond Drilling Program, McIlvenna Bay Deposit

Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation	Azimuth from Total Station	Dip	Length (m)
MB-18-183	640961.55	6056152.38	330.46	189.25	-73.82	701.00
MB-18-184	641386.19	6055696.85	332.07	199.52	-55.52	113.00
MB-18-185	641330.43	6055727.59	331.93	198.80	-56.72	110.50
MB-18-186	641273.78	6055733.70	331.92	198.46	-56.02	100.50
MB-18-187	641177.32	6055756.53	331.91	197.89	-55.79	137.50
MB-18-188	641269.95	6055847.54	331.97	197.00	-57.21	215.00
MB-18-189	641406.73	6055826.48	332.30	198.94	-57.81	248.00
MB-18-190	641249.16	6055915.98	331.97	189.86	-66.86	320.00
MB-18-191	640983.05	6055954.84	330.94	193.44	-62.89	317.00
MB-18-192	641095.20	6055834.00	331.88	197.64	-56.61	176.00
MB-18-193	641130.52	6056051.29	332.40	190.16	-65.72	447.00
MB-18-194	640229.57	6056875.69	319.32	183.27	-79.81	1160.00
MB-18-195	640964.92	6055778.35	331.40	199.61	-55.19	119.00
MB-18-196	641080.70	6055787.40	331.73	198.17	-56.34	122.00
MB-18-197	641174.58	6055869.36	332.00	191.63	-68.25	251.00
MB-18-198	640503.65	6056584.79	319.71	184.74	-76.98	918.00
MB-18-199	640574.40	6056328.22	320.40	193.51	-71.15	655.00
MB-18-200	640409.75	6056719.88	319.27	183.02	-75.68	496.00
MB-18-201	640555.88	6056262.74	320.76	196.51	-72.11	565.00
MB-18-202	640409.792	6056719.76	319.26	186.02	-71.84	1007.00
MB-18-203	640384.69	6056622.34	319.40	192.57	-71.79	861.00
MB-18-203-W1	640384.69	6056622.34	319.40	192.57	-71.79	201.00
MB-18-203-W2	640384.69	6056622.34	319.40	192.57	-71.79	169.00
MB-18-204	640515.00	6056752.00	319.39	184.00	-75.00	27.00
MB-18-205	640708.67	6056497.43	327.53	187.13	-73.89	932.00
MB-18-206	640515.03	6056750.93	319.38	186.48	-75.14	1032.00
MB-18-206-W1	640515.03	6056750.93	319.38	186.48	-75.14	579.00
MB-18-207	640130.36	6056846.12	329.76	171.41	-72.81	1068.00
MB-18-207-W1	640130.36	6056846.12	329.76	171.41	-72.81	198.00
MB-18-208	640713.94	6056454.73	329.54	200.63	-68.72	841.00
MB-18-209	640815.50	6056310.08	330.45	181.57	-71.29	429.00
MB-18-210	640772.07	6056358.76	329.83	186.86	-72.15	471.00
				Total Metres		**14,986.50**

Table supplied by Foran in June 2019.

Table 10-3: Summary of the 2018 Phase II Diamond Drilling Program, McIlvenna Bay Deposit

Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation	Azimuth from True North Gyro	Dip	Length Drilled (m)
HA067	640152.50	6056767.00	329.60	234.59	-78.46	201
HA18-043w1	640594.60	6056204.00	326.61	197.47	-74.99	172.5
HA18-045w1	640686.20	6056182.00	328.64	197.96	-75.91	153.5
MB-18-109w1	640757.00	6056085.00	332.07	199.07	-62.47	111.5
MB-18-134w1	640776.40	6056580.00	327.85	185.25	-78.04	122
MB-18-141w1	640675.10	6056151.00	328.73	190.22	-72.32	265.5
MB-18-142w1	640710.20	6056106.00	331.76	185.89	-69.17	100.5
MB-18-143w1	640710.20	6056106.00	331.73	184.83	-72.02	90.5
MB-18-208w1	640713.90	6056455.00	329.54	202.15	-68.97	240
MB-18-209	640815.50	6056310.00	330.45	182.68	-71.46	357
MB-18-210	640772.10	6056359.00	329.83	183.60	-72.24	348
MB-18-211	640905.90	6056215.00	331.78	182.09	-74.18	755
MB-18-212	640816.60	6056219.00	332.30	199.27	-74.67	696
MB-18-212w1	640816.60	6056219.00	332.30	199.27	-74.67	136
MB-18-213	640780.60	6056198.00	332.16	199.35	-70.87	648
MB-18-213w1	640780.60	6056198.00	332.16	199.35	-70.87	422.5
MB-18-214	640831.70	6056022.00	333.38	231.26	-75.10	555
MB-18-214w1	640831.70	6056022.00	333.38	231.26	-75.10	147
MB-18-215	640694.80	6056635.00	323.75	221.10	-62.32	606
MB-18-216	640150.40	6056744.00	329.10	153.29	-71.01	1,050
MB-18-217	640807.00	6056008.00	335.31	155.90	-71.93	528
MB-18-217w1	640807.00	6056008.00	335.31	155.90	-71.93	145.5
MB-18-218	640708.70	6056306.00	327.48	187.76	-74.18	708
MB-18-218w1	640708.70	6056306.00	327.48	187.76	-74.18	115
MB-18-219	640693.90	6056633.00	323.72	213.83	-63.52	942
MB-18-219w1	640693.90	6056633.00	323.72	213.83	-63.52	130.5
MB-18-220	640716.50	6056601.00	325.66	205.55	-68.09	1,002
MB-18-225	640715.90	6056601.00	325.58	245.91	-72.29	1,092
					Total Metres	**11,840.5**

Table supplied by Foran in June 2019.

Drill hole collars were located in the field by a surveyor/geologist with a survey transit or Differential GPS and two foresite pickets were placed in front of the drill to allow the drill to be aligned at the proper azimuth. The drill holes were started with HQ sized core and drilled until they passed through the dolomite cap rock and sand layer and/or through the regolith. Once into solid bedrock the rod string was reduced to NQ size and the holes was drilled to depth, leaving the HQ rod string as casing. Once the drill hole was reduced to NQ the surveyor completed a 'heads and tails' survey of the rod string to obtain an accurate azimuth and a final collar location for the hole.

Figure 10-2: Drill Collar Locations to September 2018

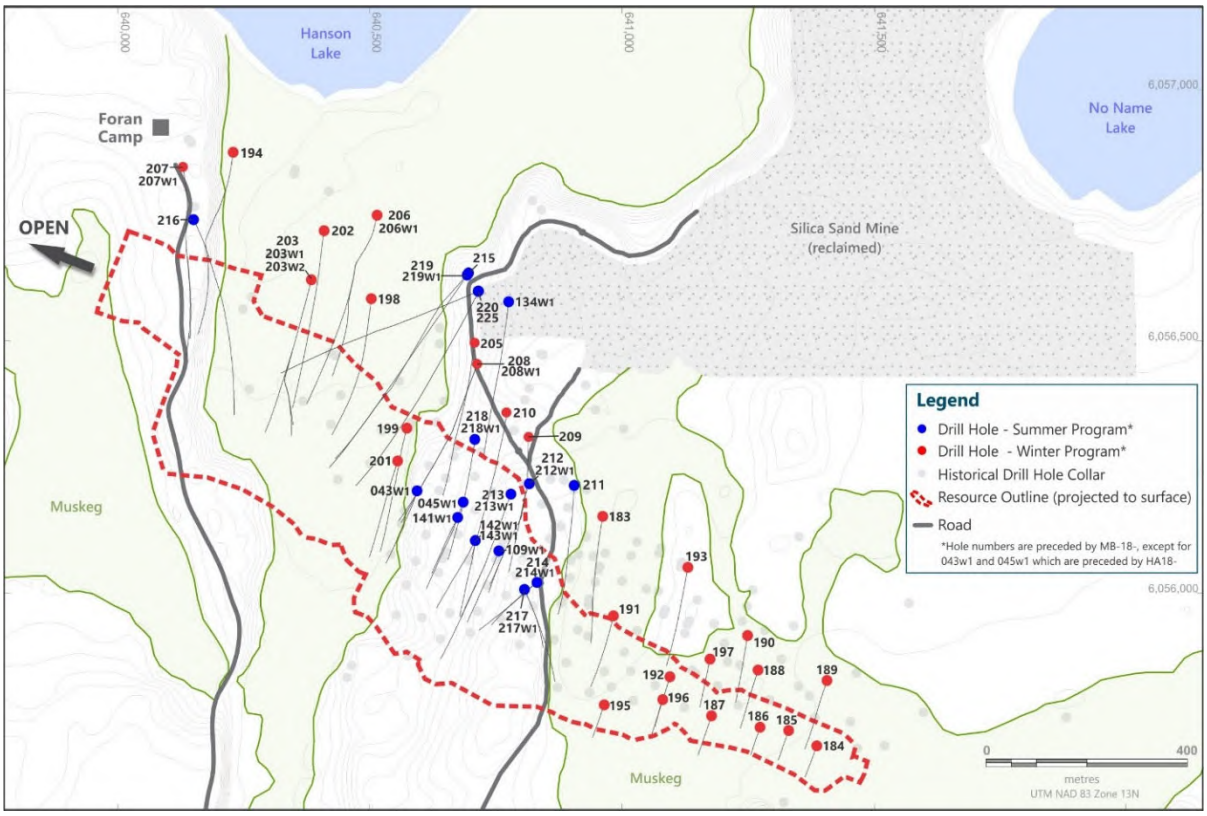


Figure supplied by Foran in June 2019.

During drilling, downhole survey readings were routinely collected by the drill crew at 50m intervals as the holes progressed, utilizing an EZshot survey tool to track the progress of the hole. The EZshot tool provides an accurate reading for the dip of the hole, however, the tool uses a magnetic compass to determine the azimuth. Due to the occurrence of some magnetic units in the stratigraphy at McIlvenna Bay, the azimuth data for some readings from the EZshot tool may be dubious, but they provide a back up of survey data for the hole in the event that it is lost and not available for surveying at the end of drilling. Due to the magnetic parts of the stratigraphy, all drill holes had a separate downhole survey conducted once drilling was complete, as described below, to ensure that accurate survey data was available for each hole.

At the completion of each drill hole a downhole survey was completed using a MEMS Gyro Tool, provided by Reflex Instruments, which provides an accurate trace of the drill hole at depth. The gyro tool is a downhole survey instrument that is magnetically independent making use of gyros rather than a magnetic compass to determine orientations and therefore the results are not affected by magnetic rock units such as iron formations, etc. in the stratigraphy. Surveys were generally conducted from the bottom of the hole up, with measurements collected at five or ten metre intervals throughout the hole. The survey data collected by the gyro tool is taken as the most accurate source and these results are used in the drill hole database. Part way through the winter 2018 drill program, North Seeking Gyro tools were obtained from Reflex Instruments, and Stockholm Precision tools. North seeking gyro

instruments are unaffected by magnetic terrain similar to MEMS instruments but have the added benefit of not requiring a collar survey to calculate the holes azimuth; instead, the tool calculates the station azimuth independently.

Once the core was received at the McIlvenna Bay core shack, geological and geotechnical core logging was completed. Geospark Consulting Inc. (Geospark) core logging software, under license from Geospark, was used to collect all the pertinent geological data from the drill core along with a detailed description of the rock units and sample information. All drill core was logged by Foran employees at the McIlvenna Bay core shack.

For the 2018 drill program, all drill holes were completed using the ACT III digital core orientation system from Reflex Instruments to provide oriented drill core. The system allows the bottom of each run to be marked by the driller's helper before the core is retrieved from the core tube and placed in the core boxes. Prior to logging, the core was aligned on a section of angle iron relative to that mark and a 'bottom' reference line was marked on the core. This provides a reference line which can be used to take structural measurements of fabrics in the rock which are aligned as they would have been in the ground prior to drilling. This process provides valuable information on the true orientation of structures in the ground a will greatly assist in the interpretation of the geology of the deposit.

10.3 Qualified Person's Comments

The QP has reviewed and discussed the drilling programs with Foran personnel both during the site visit and at other times throughout its audit and review of the Mineral Resource estimate. The QP believes that the programs have followed the best practices guidelines as outlined by the CIM for exploration.

In the opinion of the QP for this section, Foran has achieved its objective of outlining the mineralization in the McIlvenna Bay deposit with its diamond drilling programs. The drilling programs on the McIlvenna Bay deposit was sufficiently extensive to be used as the basis of a Mineral Resource Estimate at the McIlvenna Bay Project.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

Section 11 below has been extracted from the previous 2015 JDS Technical Report. Section 11.2 and beyond reflect the exploration since the 2015 Technical Report was written.

11.1 Sample Preparation Analysis and Security (1988 to 2013)

This section describes, to the best of RPA's knowledge, the historical procedures employed initially by Cameco and later by Foran.

11.1.1 Cameco (1988 to 1991)

Little information is available for security measures employed, QA/QC procedures, and who actually prepared the samples. The samples of sawn core were initially sent to TSL Laboratories Inc., (TSL) in Saskatoon. Each sample was crushed to a minimum of 60% passing -10 mesh and was split, with the rejects being stored at TSL's laboratory. A split portion, approximately 250 g, was pulverized to 90% passing -150 mesh. The split halves were assayed by standard Atomic Absorption (AA) techniques for zinc, copper, silver, and lead and by fire assay-atomic absorption (FA-AA) for gold. When the initial assay samples exceeded 1% Zn, 1% Cu, or 1 g/t Au, the sample was re-analysed. Samples from HA-01 to HA-06 were assayed at TSL. The remainder of the samples from HA-07 through HA-67 were assayed at Eco-Tech Laboratories (Eco-Tech) in Creighton, Saskatchewan (Eco-Tech). A total of 152 check assays were performed at TSL, Bondar-Clegg & Company Ltd. (Bondar-Clegg) in Ottawa, and TerraMin Research Labs Ltd. (TerraMin) (Calgary). Cameco was pleased with the Eco-Tech results and believed that TSL returned somewhat lower values for zinc and, to a lesser extent, copper during check assays (MRDI, 1998).

11.1.2 Foran (1998 to 2000)

The bulk of the assaying from the Foran drilling programs was done at TSL. Once sawn, individual samples were packaged in individual plastic sample bags, which were sealed with packing tape, boxed, and taken directly by a Foran representative from the field to Creighton, Saskatchewan. The boxes were shipped via bus to Saskatoon where a representative from TSL collected the boxes and brought them to the lab.

At TSL, each sample was crushed to a minimum of 60% passing -10 mesh and then split, with the rejects being stored at TSL. A split portion, approximately 250 g, was pulverized to 90% passing -150 mesh. All samples were analysed for copper, zinc, lead, gold, and silver, while samples from holes MB-99-78 through 125 were also analysed for iron and sulphur. All samples were also analysed by a 31-element ICAP scan that was completed at the TSL laboratory. Copper, lead, zinc, and silver analyses were done by Atomic Absorption Spectrophotometry, while the gold was determined by standard FA procedures.

One in ten samples assayed by TSL was shipped to the Saskatchewan Research Council's Geoanalytical Services Laboratory (SRC) in Saskatoon for check assay. In the case of a discrepancy between the original and check assay results, the sample was rechecked by XRAL Laboratories Ltd. (XRAL) in Toronto to determine the most accurate result. In their signed assay reports, TSL included the analytical results of all internal repeat samples (duplicates) and TSL in-house or Certified Reference Material standard

samples inserted into the assaying sequence. Foran's experience was that for most elements, TSL assayed very slightly lower (<10% difference) than the corresponding assay done at the SRC. Generally, zinc, lead and silver assays were less than 10% lower at TSL than at SRC, copper assays were less than 5% lower, and gold results were comparable (Lemaitre, 2000).

During the time periods noted, it is not known what the certifications were for the various laboratories mentioned.

The QA/QC procedures used by Foran were not as rigorous as one might expect in a current program. Nonetheless, RPA believes that the work was done in accordance with the best practices of the time and that the results should be reliable.

Specific Gravity Determinations

From hole MB-99-87 to MB-99-125, Foran had specific gravity determinations of each sample done by TSL using the weight in water – weight in air method on the intact core sample. Holes MB-99-78 to MB-99-86 did not have any specific gravity determinations but did have iron and sulphur analytical data. Holes prior to MB-99-78 do not have any specific gravity determinations or any sulphur analytical data.

11.1.3 Foran (2007 to 2008)

All core was split using a diamond saw. Sampling was done on a range of intervals up to a maximum of 1.24m often with breaks at lithological and mineralogical contacts. Assay tags were stapled into the boxes.

Samples were analysed at TSL for gold, silver, copper, lead, and zinc by AA with a four-acid digestion. Samples were analysed for gold, silver, copper, lead, and zinc in all holes except MB-07-135. Over limit gold and silver were rerun using fire assay of a 30 g aliquot with a gravimetric finish. All samples were crushed to 70% -10 mesh, riffle split to a 250 g sub sample, which was then pulverized to 95% -150 mesh.

Samples were in the custody of Foran personnel or their designates until delivered to the lab. The site is fairly remote and, while not fenced, was continually supervised and relatively immune to incursions from unauthorized personnel.

There is no record in the database of any independent assay QA/QC protocols applied for these programs. In RPA's opinion, this is a significant deviation from industry best practices which impacts on the overall perceived reliability of the assay database. It is noted that assay QA/QC protocols have since been adopted by Foran, and this is viewed as a positive step. It is also noted that in 2011, Foran checked the sampling, re-logged the core, and did some re-sampling of the 2007-2008 holes. There was good agreement with the sample and logging records, and therefore, there is no reason to suspect that the assay work done in 2007-2008 is sub-standard.

11.1.4 Foran (2011 to 2013)

The initial winter 2011 program was managed under contract to Equity Exploration Consultants Ltd. Subsequent to that, all exploration work was managed by Foran personnel.

Up until the latter part of the 2011 program, holes were logged in a dedicated facility established in an old office building located near the area of drilling. At the time of the last RPA site visit, Foran was in

the process of moving to a new building constructed specially for core handling. This facility has been fully configured and is presently in use.

Core was logged for lithology, mineralization, and alteration. Geotechnical measurements included recovery, Rock Quality Designation (RQD), and magnetic susceptibility. All core was photographed prior to sampling. The sampling was done using a diamond saw. The maximum sample length was standardized to one metre with breaks at lithological and mineralogical contacts. Routine bulk density measurements were taken from intact core specimens.

RPA inspected several sampled intervals and considers the sampling to have been done properly, in a manner appropriate for the deposit type and mineralization style. In RPA's opinion, the orientation and distribution of the samples are such that they will be representative of the deposit.

Drill core from early programs were either stored in racks or cross-stacked boxes on site. Foran has collected the cross-stacked core, re-boxed it, and placed it in racks. The older Cameco core, although in racks, is exposed to the elements and has suffered some degradation as a result. Foran personnel have reportedly begun re-boxing and storing this core as well.

Assay QA/QC protocols were introduced, in the winter of 2011, which comprised inclusion of a blank, standard, and duplicate into the sample stream at a nominal rate of one for every 20 samples. Duplicates comprised both quarter-cores (field duplicates), as well as splits from pulps (preparation duplicates) which were inserted on a rotating basis. The duplicates were taken at a rate of one in 20 samples; however, they alternated between field and preparation duplicates. Following the winter 2011 program, the protocol was revised slightly so that the lab duplicates were completed by taking a second pulp from the sample reject material rather than a second split from the pulp. Material for the blanks consisted of locally obtained barren carbonate rock. The standard material comprised eight different commercially prepared reference standards, listed below in Table 11.1. .

The samples were analysed at TSL for Cu, Zn, Pb, and Ag by AA following four-acid digestion, as described above. Samples were analysed for Au using fire assay with AA finish and over-limits for Au were re-assayed by fire assay with gravimetric finish. All samples were also routinely analysed separately by a 30 element ICP package following Aqua Regia digestion for trace metal concentrations. A 30 g aliquot was used for the FA-AA analyses, and a 58.32 g aliquot was used for FA-gravimetric assays. As with the 2007-2008 programs, all samples were crushed to 70% -10 mesh, riffle split to a 205 g subsample, which was then pulverized to 95% -150 mesh.

Table 11-1: Reference Standards – 2011 to 2013 Program

Standard	Au (ppb)		Ag (ppm)		Cu (%)		Pb (%)		Zn (%)	
	Mean	SD	Mean	SD	Mean	SD	Mean	SD	Mean	SD
GBM909-11			25.5	1.7	0.5344	0.0195	0.2074	0.0103	1.9486	0.0591
GBM909-12			51.7	3	1.083	0.0339	0.4191	0.0141	4.0073	0.1348
GBM909-13			127.3	6.8	3.2093	0.1295	0.8513	0.0327	6.8362	0.2363
G310-4	430	30								
CDN-ME-11	1,380	100	79.3	6	2.44	0.11	0.86	0.1	0.96	0.06
CDN-ME-17			38.2	3.1	1.36	0.1	0.676	0.054	7.34	0.37
GLG307-1	2.86	1.7								
CDN-GS-P7B	710	70	13.4	1.6						
CDN-FCM-7	896	84	64.7	4.1	0.526	0.026	0.629	0.042	3.85	0.19
CDN-ME-18	512	70	58.2	5.1	1.931	0.086	0.098	0.012	4.6	0.22

Source: RPA, Rennie, 2011

Notes: Standard deviations (SD) are provided by the manufacturer and are derived from umpire assays of the standards. They provide a basis for derivation of error limits. In this table SD refers to +/-2 SD, which is the error limits provided by the manufacturer for the standard based on the results of round-robin testing.

The above table and notes were extracted from the 2015 JDS Technical Report and modified as required.

The QA/QC results were gathered and collated to check for failures. Duplicates were plotted on diagrams comparing the absolute relative difference between duplicate pairs with the mean of the pair. Reasonable agreement was obtained for both the field and prep duplicates. Blanks and standards were plotted in chronological order and compared with the nominated values and acceptable error limits. For blanks, all values returned were extremely low and there were no failures. A number of standards failures were reportedly obtained during the 2011 winter program which resulted in re-assay of partial batches (batch of 20 samples in the sample stream surrounding failure).

In three cases, the failure was determined to have resulted from improper labelling of the standards packets. In all other cases the batches of samples passed on re-assay and those results were used in the database. There were two standard failures during the summer 2011 program. In both cases, batches of 20 samples surrounding the failure were re-assayed. The batches passed on re-run and the results of these re-runs were used in the database. Three standards and one blank failure were obtained in 2012, which resulted in the re-run of the affected batches. The results passed for all samples on re-run and the revised data was incorporated into the database. One batch from the 2013 winter program was re-run owing to a standard failure.

RPA has reviewed the assay QA/QC results for the 2011, 2012, and 2013 programs and concluded that there were no concerns evident.

Equity personnel re-logged five of the seven 2007-2008 drill holes in 2011 and updated the geology, geotechnical data and verified the sample intervals. The core was reported to be completely intact and sample intervals were easily checked with no discrepancies noted. Samples were focused on the mineral zones with one or two shoulder samples from the adjacent rocks. All analytical certificates were available from TSL and corresponded to the sample numbers in the core boxes.

Foran has continued with re-logging of portions of holes in order to help resolve complications in the geological interpretations.

Specific Gravity Determinations

At the time of the resource update, Foran had collected 1,085 density measurements from core specimens. RPA plotted scatter diagrams of the measured density against the sample metal grades and found a reasonably robust linear relationship between density and zinc grade. A regression formula was derived in order to estimate block density from the interpolated zinc grades. This formula is as follows:

$$\boldsymbol{SG = (0.075\, x\, Zn) + 2.8124}$$

The density for each block was calculated from the interpolated zinc grade.

Foran has since made many more density measurements, and at present, there are 2,501 determinations in the database. RPA recommends that the regression formula be updated with this more recent data.

In RPA's opinion, Foran's present logging, sampling, and assaying protocols are consistent with good industry practice. The QA/QC program as designed and implemented by Foran is adequate and the assay results within the database are suitable for use in a Mineral Resource Estimate.

11.2 Micon QP Comments on Sample Preparation Analysis and Security (1988 to 2013)

Where possible Micon's QP was able to review the work conducted by RPA that was commented on in the 2015 JDS Technical Report and agrees with RPA's opinions regarding the work and that it was suitable for conducting Mineral Resource Estimates.

11.2.1 Notes Regarding Assay Laboratories

TSL quality control system conforms to the requirements of ISO/IEC Standard 17025 guidelines and in April 2004, it received its certificate stating accreditation for specific tests from the Standards Councils of Canada, Laboratory Number 538. TSL participates in the proficiency testing program sponsored by the Canadian Certified Reference Materials Project. TSL has qualified for Certificates of Laboratory Proficiency since the program's inception in 1997, and this program is a requirement of its ISO/IEC 17025 accreditation. TSL is independent of both Micon and Foran.

Bondar-Clegg & Company Ltd. was an independent commercial assay laboratory company which was taken over by ALS Chemex Labs Ltd. in December 2001.

Eco-Tech Laboratories in Creighton, Saskatchewan was an independent commercial assay laboratory company which appears to have been struck off the public company registry in Saskatchewan as noted in Part 1 of the December 27, 2002 Sask Gazette. It is recorded as struck off the register pursuant to Section 29.0.

Section 290(1) enumerates the circumstances under which a corporation may have its name struck from the register of corporations. The most common circumstances for striking the name of a corporation from the register are where: the Branch Director does not receive a return, notice or

other document or prescribed fee required by the Act; the corporation gives notice to the Branch Director that it has ceased to carry on business in Saskatchewan; the corporation is not entitled to carry on business under the act of incorporation of the jurisdiction in which it was incorporated; the corporation is issued a Certificate of Discontinuance pursuant to Section 182; the corporation is dissolved; or the corporation is amalgamated with one or more other corporations. Before striking a corporation off the register, the Branch Director will send notice to the corporation advising the corporation of the default under Section 290(1) and stating that unless the default is remedied within 30 days after the date of the notice, the name of the corporation will be struck off the register. If the corporation does not cure the default within the time mentioned in the notice, the Branch Director may strike the name off the register and publish notice thereof in the Saskatchewan Gazette.

No information was obtained regarding the TerraMin Research Labs Ltd. of Calgary, Alberta. and it appears this laboratory is no longer operating.

XRAL Laboratories Ltd. (XRAL) as purchased by the SGS Group in 1988. XRAL was an anacronym that stood for X-Ray Assay Laboratories Ltd.

All of the above laboratories are or were independent laboratories which charged a fee to process a sample. These laboratories are or were independent of the Foran or the other companies which conducted work on the McIlvenna Bay Project. At the time of operation, Micon's QP believes that the laboratories applied the best practices in undertaking their assaying techniques and obtained any certifications necessary to operate as independent laboratories serving the mineral industry.

11.3 Sample Preparation Analysis and Security (2018 to Present)

For the 2018 programs drilling was completed using NQ size diamond drill core for all holes. During the logging process mineralized intersections were marked for sampling by the geologist and given a unique sample number. The samples were sawn in half with a diamond saw blade and the sample interval and sample number was marked on a metal tag that was stapled into the core box at the start of the sample interval as a permanent record. Half NQ core was placed in plastic bags with the sample tag, sealed and submitted for assay, while the second half was returned to the core box for storage on site. The sealed plastic sample bags were placed in labelled rice sacks for hand delivery to TSL by Foran employees. Samples generally averaged one metre in length in homogeneous material, with a maximum of 1.5m or a minimum of 0.20m taken in select circumstances, if required, to conform with geological contacts and/or mineralized zones. Under no circumstances were samples taken across geological boundaries.

QA/QC measures employed by Foran included the insertion of one certified standard, one blank (barren dolomite) and one lab duplicate within every sequence of 20 samples, similar to previous programs completed since 2011. Part-way through the winter 2018 program, however, it was decided to beef up the amount of QA/QC material inserted in the sample stream and to increase the number of duplicate analysis completed by the assay lab. This resulted in a revised protocol which consisted of the use of seven standards of varying grades (high, medium, low), two blanks and two field duplicates inserted in the sample stream for every 100 samples taken prior to shipment of the samples to the laboratory. A list of the certified standards used for the program are provided in Table 11-2.

At the laboratory, a second split was taken from the initial pulp for every tenth sample processed, to represent a pulp duplicate, and a second pulp is created from the original reject for every 11th sample as a prep duplicate. These samples are analysed in order with the original sample stream. All QA/QC reference material was checked for compliance prior to compiling the assay data and any batches with failures of QA/QC material were re-run by the laboratory.

The 2018 samples were analysed at TSL for Cu, Zn, Pb, and Ag by AA following four-acid digestion. Samples were analysed for Au using fire assay with AA finish and over-limits for Au (>1 g/t) were re-assayed by fire assay with gravimetric finish. All samples were also routinely analysed separately by a 30 element ICP package following Aqua Regia digestion for trace metal concentrations. A 30 g aliquot was used for the FA-AA analyses, and a 58.32 g aliquot was used for FA-gravimetric assays. As with the 2007-2013 programs, all samples were crushed to 70% -10 mesh, riffle split to a 205 g subsample, which was then pulverized to 95% -150 mesh.

Table 11-2: Reference Standards – 2018 Program

Standard	Au (ppb)		Ag (ppm)		Cu (%)		Pb (%)		Zn (%)	
	Mean	SD	Mean	SD	Mean	SD	Mean	SD	Mean	SD
CDN-ME-11	1,380	100	79.3	6	2.44	0.110	0.86	0.1	0.96	0.06
CDN-ME-17	452*	58	38.2	3.1	1.36	0.100	0.676	0.054	7.34	0.37
CDN-FCM-7	896	84	64.7	4.1	0.526	0.026	0.629	0.042	3.85	0.19
CDN-ME-18	512	70	58.2	5.1	1.931	0.086	0.098	0.012	4.60	0.22
CDN-ME-1410	542	48	69	3.8	3.80	0.170	0.248	0.012	3.682	0.084
CDN-ME-14	100*	20	42.3	4.2	1.22	0.078	0.495	0.030	3.10	0.28
CDN-ME-1705	3,660	210	78.3	6.4	1.35	0.050	0.058	0.004	0.71	0.04
CDN-ME-1406	678	54	57.1	3.7	0.32	0.012	0.485	0.026	2.27	0.08
OREAS 622	1,850	132	102	6.6	0.486	0.016	2.210	0.134	10.24	0.36
CDN-ME-1707	2,020	214	27.9	2.9	2.72	0.11	0.097	0.006	0.539	0.016

Notes: Standard deviations (SD) are provided by the manufacturer and are derived from umpire assays of the standards. They provide a basis for derivation of error limits. In this table SD refers to +/-2 SD, which is the error limits provided by the manufacturer for the standard based on the results of round-robin testing.

A total of 1,562 samples (including all QA/QC materials) were analysed during the 2018 Phase I program and there were seven standard failures reported from the assaying. The first failure occurred while the historic QA/QC protocols were still in effect, so a batch of 20 samples surrounding the failed standard was re-run. All other instances occurred once the new QAQC protocols had been established and in these cases a group of seven samples was re-run (three samples either side of the failure in the sample stream). In all cases the standard material passed on re-run and the revised assay results for these samples were incorporated into the database.

A total of 1,550 samples (including all QA/QC materials) were analysed during the 2018 Phase II program and there were ten standards and three blanks that failed QAQC protocols during the program. These failed samples and their surrounding groups of samples (generally three samples either side) were re-assayed by the laboratory. In all cases the batches of samples passed on re-run and the revised assay results for these samples were incorporated into the database.

11.3.1 Specific Gravity Determinations

A number of additional specific gravity measurements were completed on intact core during the 2018 program, both through the continued routine measurement of individual core pieces for the different rock units during the logging process, as well as, the collection of 'bulk' specific gravity measurements for complete samples. Specific gravity data was collected on intact core using the weight in water – weight in air method. For the 'bulk density' measurements an apparatus was set up for the weight scale in the core shack utilizing a large basket, which allowed entire sample intervals to be weighed at once and therefore provide a much more representative value.

The database for the deposit consists of 4,435 specific gravity measurements from individual core samples taken from all lithologies in the deposit area measured either on site or at the assay lab. The database also includes 1,932 bulk specific gravity measurements collected for complete sample intervals from the mineralized zones of 61 drill holes spread spatially through the deposit. As discussed above, the bulk density measurements were taken on complete sample intervals and are much more representative of the density of the mineralized material in the ground than small randomly selected core pieces.

11.4 QP Comments on Sample Preparation Analysis and Security (2018 to Present)

The QP was able to review the work conducted by Foran on its 2018 to 2019 drilling programs and is of the opinion that the QA/QC programs have been conducted in line with CIM best practices. The QP believes that the work is suitable for use in conducting a Mineral Resource Estimate on the McIlvenna Bay Project.

12. DATA VERIFICATION

12.1 Site Visit

Mr. William Lewis, P.Geo., a senior geologist with Micon International visited the site in August 2018, during which the McIlvenna Bay property was inspected, and various aspects of the Project were discussed. The exploration programs for the Project were discussed in detail and the onsite exploration QA/QC procedures were reviewed and discussed during a review of the core logging and sampling procedures at the core logging facility.

Mr. Lewis conducted the site visit with the assistance of Roger March, P.Geo., Vice President of Exploration for Foran.

Figure 12-1 shows the core storage area at Foran's McIlvenna Bay camp during the site visit. This storage area holds both the historical core as well as the core from Foran's previous drilling programs.

Figure 12-2 shows the buildings used to log core and prepare samples at Foran's McIlvenna Bay camp during the site visit, and Figure 12-3 shows one of the drills set-up and drilling during the Micon site visit in August, 2018.

Figure 12-1: Core Storage Area at Foran's McIlvenna Bay Camp

Figure 12-2: Buildings Related to Logging and Sample Preparation at Foran's McIlvenna Bay Camp

Figure 12-3: Drill Set-up and Drilling During the 2018 Micon Site Visit

After the site visit, Micon's QP, Mr. Lewis, selected 13 random reject core samples from Foran's McIlvenna Bay drilling samples located at TSL in Saskatoon. Micon requested that TSL re-assay the selected samples and send the results to Micon's Toronto office. The TSL sample preparation procedures and standard assaying procedures are summarized in Table 12-1.

Table 12-1: TSL Sample Preparation and Standard Assaying Procedures

Procedure	Sample Type	No. of Samples	Size Fraction		Sample Preparation
Preparation	Reject	13	Reject approx. 70% - 10 mesh (1.70 mm)		Riffle Split, Pulverize
			Pulp approx. 95% - 150 mesh (106 µm)		
Assay	**Element Name**	**Unit**	**Extraction Technique**	**Lower Detection Limit**	**Upper Detection Limit**
	Au	ppb	Fire Assay/AA	5	3,000
	Au	g/t	Fire Assay/Gravimetric	0.03	100 %
	Ag	g/t	HNO3-HF-NCIO4-HCl/AA	1	1,000
	Cu	%	HNO3-HF-NCIO4-HCl/AA	0.01	80
	Pb	%	HNO3-HF-NCIO4-HCl/AA	0.01	80
	Zn	%	HNO3-HF-NCIO4-HCl/AA	0.01	80
Samples for Au Fire Assay/AA (ppb) are weighed at 30 grams. Samples for Au Fire Assay/Gravimetric (g/t) are weighed at 1 AT (29.16 g). Samples for Ag (g/t), Base Metals (%) are weighed at 0.5 g.					

Table 12-2 summarizes the 13 random reject core samples and descriptions chosen by Micon for re-assaying. All samples were taken from one drill hole, but the samples represent the different mineralized zones encountered by the drill hole and represent various grade ranges.

Table 12-2: Random Reject Core Samples Re-Assayed at Micon's Request

Drill Hole	Mineralized Zone	Sample Number	From (m)	To (m)	Interval (m)
HA-18-045w1	Upper Sx Zone	780581	514.70	515.70	1.00
		780583	516.30	516.80	0.50
		780584	516.80	517.80	1.00
	UWZ	780588	519.34	519.55	0.21
		780593	521.38	521.96	0.58
		780597	523.53	524.30	0.77
	CSZ	780600	526.25	526.72	0.47
		780604	528.50	528.82	0.32
		780607	530.22	530.90	0.68
		780608	530.90	531.27	0.37
		780609	531.27	532.30	1.03
		780614	535.10	536.10	1.00
		780618	538.10	539.10	1.00

Table 12-3 summarizes the results of Micon's re-assaying of the 13 samples chosen from Foran's samples originally submitted for assaying by TSL. Three samples were also chosen for specific gravity testwork.

Table 12-3: TSL Results for the Thirteen Random Samples Chosen by Micon for Re-assaying

Sample Number	Au (ppb)	Au1 (ppb)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Specific Gravity
780581	110			10.8	0.85	0.02	0.7	2.68
780583	140			20.5	2.29	0.02	2.23	
780584	130			16.9	0.62	0.12	1.08	
780588	620			45.7	0.77	0.88	10.3	
780593	420			16.4	1.87	0.04	4.33	
780597	>1,000	>1,000	1.37	34.2	3.25	0.27	2.57	3.08
780600	>1,000		7.27	44.8	5.36	0.05	0.38	
780604	10			0.4	<0.01	<0.01	<0.01	
780607	180			7.4	1.49	<0.01	0.06	
780608	880			35.9	9.05	0.02	0.63	
780609	320			6.5	1.5	<0.01	0.15	2.68
780614	150			3.6	0.66	<0.01	0.04	
780618	35			2.2	0.65	<0.01	0.03	
GS-1P5P	1,450							
GS-7E			7.34					
ME-8				61	0.1	1.94	2	
ME-1411				44.1	1.54	0.26	0.47	

Micon also requested that TSL perform a Multi-Element ICP analysis of the samples using Aqua Regia digestion of the samples.

The ICP-AES, Aqua Regia Leach digestion (HCl-HNO_3) liberates most of the metals noted in Table 12-4 except those marked with an asterisk where the digestion will not be complete.

Table 12-4: Lower Detection Limits for Aqua Regia Leach Digestion

Element Name	Lower Detection Limit	Element Name*	Lower Detection Limit
Ag	0.3 ppm	Mo	1 ppm
Al*	0.01 %	Na*	0.01 %
As	2 ppm	Ni	1 ppm
Ba*	1 ppm	P*	0.001 %
Be*	1 ppm	Pb	3 ppm
Bi	3 ppm	S	0.05 %
Ca*	0.01 %	Sb	3 ppm
Cd	0.5 ppm	Sn*	5 ppm
Co	1 ppm	Sr*	1 ppm
Cr*	1 ppm	Ti*	0.01 %
Cu	1 ppm	V*	1 ppm
Fe*	0.01 %	W*	2 ppm
K*	0.01 %	Y	1 ppm
Mg*	0.01 %	Zn	1 ppm
Mn*	2 ppm	Zr*	1 ppm

Note: * The elements marked with an asterisk indicate that the digestion will not be complete.

Table 12-5 summarizes the assays for the elements using the Multi-Element ICP analysis of the samples using Aqua Regia digestion.

Copies of the TSL assay certificates sent to Micon for the samples are included as Appendix 2.

12.2 Database Review

Micon received the updated database on January 7, 2019, the data was organized in multiple Excel files. Micon proceeded to compile and review the data, no errors were found, however, drill hole MB-99-108 was ignored because of the suspicious collar and down the hole survey location. During the construction of the wireframes, a few records were changed in the mineralized zones table to improve the 3D interpretation of the envelopes.

Micon had previously undertaken an extensive review of Foran's database as part of an independent internal review of its McIlvenna Bay Project. Micon was therefore familiar with the database prior to undertaking the independent review and audit of the current Mineral Resource Estimate.

12.3 Qualified Person's Comments

The QP responsible for reviewing both the exploration work and the mineral resource estimate have reviewed the material and database provided by Foran and found that the data were adequate for the use in undertaking a mineral resource estimate on the McIlvenna Bay Project. The data provided by Foran is suitable to be used as the basis of Mineral Resource and Reserve Estimates for incorporated into this prefeasibility study of the McIlvenna Bay Project.

Table 12-5: Summary of Assay Values for the Multi-Element ICP Analysis, Aquia Regia Leach Digestion Method

Element Units	Ag Ppm	Al %	As ppm	B ppm	Ba ppm	Bi ppm	Ca %	Cd ppm	Co ppm	Cr ppm	Cu ppm	Fe %	Ga ppm	Hg ppm	K %	La ppm	Mg %
780581	10.1	4.00	10	28	16	24	0.57	22.9	27	35	8,172	10.53	25	<1	0.08	32	3.16
780583	20.1	2.96	26	<20	63	6	0.30	80.3	48	62	>10,000	11.18	15	2	0.22	25	2.43
780584	14.8	3.30	30	<20	139	79	0.84	31.0	25	59	5,829	8.57	16	<1	0.42	31	3.00
780588	42.0	0.84	202	24	10	64	5.98	302.6	51	34	6,831	18.64	34	27	0.02	14	4.24
780593	15.4	0.90	119	27	3	7	7.97	146.2	24	15	>10,000	16.53	21	7	0.01	16	6.42
780597	35.2	0.96	272	26	3	31	9.78	98.0	24	17	>10,000	16.97	25	6	<0.01	20	6.33
780600	39.2	1.34	81	<20	30	72	0.05	18.2	87	59	>10,000	14.76	13	2	0.21	12	1.06
780604	<0.3	1.17	18	47	118	<3	0.33	<0.5	6	67	51	1.98	6	<1	0.70	6	0.81
780607	6.1	2.15	27	35	49	28	0.07	3.4	27	61	>10,000	5.10	9	<1	0.40	14	1.79
780608	33.0	1.40	140	27	22	42	0.03	33.5	77	77	>10,000	14.80	15	1	0.14	9	1.14
780609	5.9	2.14	73	39	40	32	0.06	5.4	53	61	>10,000	8.38	9	<1	0.23	16	1.66
780614	2.2	2.74	29	<20	28	5	0.11	1.7	8	78	6,210	4.84	14	<1	0.19	19	2.23
780618	2.0	2.15	5	23	9	4	0.04	1.3	13	70	6,216	3.79	10	<1	0.09	18	1.76

Element Units	Mn ppm	Mo ppm	Na %	Ni ppm	P %	Pb ppm	S %	Sb ppm	Sc ppm	Sr ppm	Th ppm	Ti %	Tl ppm	V ppm	W ppm	Zn ppm
780581	727	2	<0.01	2	0.003	266	3.77	<3	<5	6	3	0.015	<5	<1	<2	6,393
780583	511	1	0.01	4	0.021	217	6.81	<3	<5	14	<2	0.022	<5	4	<2	>10,000
780584	579	1	0.03	12	0.068	1,230	4.09	<3	<5	41	<2	0.046	<5	23	<2	8,909
780588	1,097	2	<0.01	3	0.002	8,864	>10.00	22	<5	50	<2	0.005	<5	1	<2	>10,000
780593	1,443	2	0.01	2	0.002	413	>10.00	17	<5	38	<2	0.006	<5	<1	<2	>10,000
780597	1,353	<1	0.01	2	0.003	2,901	9.13	36	<5	52	<2	0.007	<5	<1	<2	>10,000
780600	101	1	<0.01	2	0.001	453	>10.00	<3	<5	2	<2	0.013	<5	<1	<2	3,453
780604	284	<1	0.07	7	0.042	28	0.14	<3	<5	8	<2	0.075	<5	26	<2	92
780607	171	1	0.01	1	<0.001	84	2.18	<3	<5	3	<2	0.023	<5	<1	<2	604
780608	118	1	<0.01	2	<0.001	224	6.23	<3	<5	1	<2	0.009	<5	<1	<2	5,576
780609	247	<1	0.01	1	0.001	67	4.59	<3	<5	3	<2	0.017	<5	<1	<2	1,451
780614	306	2	0.01	3	0.008	15	1.14	<3	<5	5	<2	0.011	<5	6	<2	518
780618	212	1	<0.01	2	0.001	7	0.86	<3	<5	2	<2	0.007	<5	<1	<2	378

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testing of McIlvenna Bay samples began in 2012, with a series of characterization tests completed by ALS Metallurgy in Kamloops. The work was followed by additional metallurgical testing at Base Metallurgical Labs ("Base Met Labs") in 2016 and again in 2019. Several mineralogical assessments by Terra Mineralogical Services were completed in parallel.

Metallurgical performance calculations incorporated into the current PFS study are based largely on the most recent (2019) metallurgical test program, but also made use of key findings from the previous mineralogical and metallurgical assessments.

Scope of work for the 2019 BL0351 testwork program "Feasibility Level Study of McIlvenna Bay", Base Met Labs Report #BL0351, July 28, 2019 (BL0351) was developed on the basis that ore from McIlvenna Bay would by processed either by HudBay' s 777 mill (via a toll milling arrangement) or by a new purpose-built processing facility at the McIlvenna Bay mine site. The program focused initially on the performance benchmarking of three representative ore types using the current 777 Mill flowsheet, but also considered several flowsheet optimization options. This allowed development of adequate design parameter definitions for the off-site milling option as well as an optimized on-site processing facility. Key program objectives were to identify:

- grade-recovery performance estimates for each major ore type
- if the massive sulphide ores could be co-processed with the copper stockwork ore, or whether separate circuits would be required, and the ores needed to be batched independently
- if the 777 concentrator flowsheet was appropriately configured for processing the McIlvenna Bay ore types
- if gravity pre-concentration processes such as dense medium separation (DMS) would be effective at offsetting trucking and processing costs
- flowsheet requirements with mass and water balances and indicative reagent consumptions
- estimates of grindability requirements for each ore type
- final concentrate and tailings characterizations

The general approach to testing was to prepare sizeable representative master composites for each ore type, then proceed through open circuit rougher and cleaner flotation tests to identify and optimize flowsheet conditions and reagent schemes. Locked cycle tests were then run on master composites to demonstrate the anticipated overall metallurgical performance within a continuous circuit. Sub-samples of individual drill hole material used to create the master composites were subsequently tested to provide variability data for feed grade and feed grade metal ratios – allowing assessment of flowsheet robustness and required reagent dosage ranges.

Tests were completed on blends of massive sulphide and copper stockwork ores to allow comparison with individual composite results and to assess the viability of co-processing the ore types.

Mineralogy assessments were undertaken to provide information to refine the grinding/flotation process during optimization, and to provide reasonable expectations for metallurgical performance versus mineral liberations and associations within each ore type. Samples from various open and locked cycle test products were used to provide characterizations of final concentrates and tailings.

This document highlights the most significant findings from the various studies while the individual study documents referred to should be consulted for further details. Findings from the three metallurgical studies agreed with one another with respect to mineralogical assessments, ore competency measurements and overall metallurgical performances.

13.2 Previous Metallurgical Test Programs

Two previous studies were undertaken with respect to McIlvenna Bay ore, namely:

- "Scoping Level Metallurgical Assessment of the McIlvenna Bay Project", Foran Mining Corporation, KM3125 ALS Metallurgy – G&T Metallurgical Services, May 28, 2012 (KM3125).
- "Phase II Metallurgical Assessment McIlvenna Bay Deposit", BL0103, Base Met Labs, December 23, 2016.

Terra Mineralogical Services Inc. (Terra) also completed two mineralogical assessments:

- "Initial Ore Characterization and Predictive Metallurgy Evaluation of Drill core Samples from the McIlvenna Bay VMS Deposit"; Saskatchewan, Giovanni Di Prisco, February 2012.
- "Mineral Characterization Of Mineralization Types And Predictive Metallurgy Evaluation Of Core Samples Used For Metallurgical Testing The McIlvenna Bay VMS Deposit"; Saskatchewan, Giovanni Di Prisco, January 2019.

13.2.1 Mineralogical Studies (Terra)

The McIlvenna Bay samples were mainly coarse to medium-grained and formed mostly intergrowths of non-opaque gangue and sulphide minerals. Non-opaque gangue was largely composed of carbonate and micaceous minerals in the massive sulphide lenses, and prevalently quartz and minor micas in the copper stockwork stringers. Platy micaceous minerals (sericite, muscovite, talc/anthophyllite), chlorite and biotite occurred pervasively and in abundant quantities sporadically. Iron oxides occurred locally in moderate amounts. Minor amounts of gahnite ($ZnAl_2O_4$) were also encountered. Gangue sulphides were chiefly pyrite and to a lesser degree pyrrhotite. Trace amounts of arsenopyrite were found in a few samples. Minor amounts of chalcopyrite were noted and determined to contain high concentrations of silver (up to ~1%). Trace to minor amounts of stannite, cassiterite, tetrahedrite, Bi-tellurides, and Bi-selenides were observed.

In the copper stockwork zone (CSZ) samples, chalcopyrite was the main economic sulphide with minor amounts of sphalerite. Other sulphides in low amounts included iron sulphides and trace amounts of magnetite, arsenopyrite, stannite and gahnite. Non-opaque gangue accounted for ~80% of the volume comprised chiefly of quartz with minor amounts of micas and carbonate. In the massive sulphides, non-opaque gangue generally constituted 40-50% of the ore by volume.

Silver was carried in discrete grains of electrum and silver sulfosalts, as well as substantial concentrations (~0.03wt%) in chalcopyrite. The bulk of the overall silver content in the ores was

expected to follow the chalcopyrite. Separation of chalcopyrite from sphalerite was expected to be the major metallurgical challenge for the McIlvenna Bay ores. Fine and complex mineral textures of chalcopyrite and sphalerite together with one another, and with iron sulphide gangues, were widespread in the massive sulphide ores. Fine to very fine regrinding was anticipated to ensure adequate liberation of the economic sulphides (target grinds were estimated from the samples to be 80% passing 60-65 µm for copper stringers, 80% passing 50 µm for massive sulphide primary grind and 80% passing 10-15 µm for concentrate regrinds). Table 13-1 summarizes the key middlings ratings for the various ore types.

Table 13-1: Middlings Ratings for the Main Ore Types (Terra, 2012)

Composite Name	Cu Min				Galena (Pb)				Sphalerite (Zn)			
	nop %	py/po %	ga %	sph %	nop %	py/po %	ga %	sph %	nop %	py/po %	ga %	sph %
Lenses 2/3	3.0	3.1	1.1	3.6	2.9	2.9	1.8	3.3	3.1	3.2	1.3	2.3
Upper West	3.5	3.3	0.3	3.4	1.0	3.3	3.4	5.5	3.3	3.4	0.3	3.5
Cu Stringer	3.2	2.8	0.0	1.8	0.0	0.0	0.0	0.0	3.0	2.8	0.0	3.3

The middlings ratings were scaled to denote the liberation complexity, where:

- 0 to 2.7 indicated liberation would be good to very good requiring little to no regrind
- 2.8 to 3.5 indicated liberation would be fair with moderate regrinding required
- 3.6 and higher indicated poor mineral liberation with significant regrinding (80% -30 µm and finer)

In Table 13-1, chalcopyrite grains as middlings with sphalerite in Leses2/3, and Upper West massive sulphides and sphalerite grains as middlings with chalcopyrite and pyrite were expected to dictate the regrind liberation requirements.

Table 13.2 through Table 13.4 summarize the minor elements measured within the sulphide minerals chalcopyrite, galena, and sphalerite, respectively. The average copper grade of chalcopyrite was 34.1% Cu, containing 118ppm Bi and 36ppm As. Galena contained 186ppm Bi and 63ppm Sb. Silver content in chalcopyrite was 29ppm Ag and 27ppm Ag in galena. Sphalerite zinc content in the grains analysed averaged 61.0%Zn, with an average iron content of 5.3%Fe. Sphalerite averaged 0.19%Cu, 2100ppm Cd and 20ppm Hg.

Table 13-2: Terra Microprobe Minor Element Contents in Chalcopyrite

	% Cu	% Fe	% Co	% As	% Bi	% Ag	% S	Total wt %
Average	34.1	30.6	0.0	0.0	0.0	0.0	35.0	99.8
std. dev	0.36	0.24	0.00	0.00	0.00	0.03	0.16	0.51
max	34.6	31.1	0.0	0.0	0.0	0.0	35.4	100.6
min	33.3	30.0	0.0	0.0	0.0	0.0	34.7	98.4
n	39.0							

Table 13-3: Terra Microprobe Minor Element Contents in Galena

	% Pb	% Cu	% Bi	% Sb	% Ag	% S	Total wt %
Average	86.9	0.2	0.0	0.0	0.0	13.3	100.4
std. dev	0.385	0.330	0.000	0.140	0.000	0.115	0.516
max	87.6	1.3	0.0	0.1	0.0	13.5	101.1
min	85.9	0.0	0.0	0.0	0.0	13.0	98.9
n	20.0						

Table 13-4: Terra Microprobe Minor Element Contents in Sphalerite

	Zn	Fe	Cu	Cd	Ag	Hg	In	S	Total wt %
Average	61.0	5.3	0.2	0.2	0.0	0.0	0.0	33.1	99.7
std. dev	1.68	1.63	0.23	0.07	0.01	0.03	0.02	0.26	0.42
max	65.1	7.2	0.8	0.4	0.0	0.1	0.1	33.6	100.5
min	59.1	1.7	0.0	0.1	0.0	0.0	0.0	32.5	98.7
n	47.0								

13.2.2 Historical Metallurgical Programs (ALS Metallurgy and Base Met)

KM3125 (by ALS) was an initial scoping study to characterize the metallurgical performance of samples of massive sulphide zone (Zone 2 and Upper West Zone) and copper stockwork lithologies with focus on ore hardness, mineralogical and chemical composition, plus flotation response. The study included limited optimizations using batch rougher and open cleaner flotation tests and completed locked cycle tests for each ore type with measurement of final concentrate minor element concentrations.

BL0103 (by Base Met) was described as a Phase II assessment and continued the metallurgical development program through an examination of the metallurgical response of likely mining zones from the deposit. Individual geological lithology composites were tested first, followed by two zone composites with a mix of copper stockwork and massive sulphide material. Table 13-5 showed the make-up of the composites used for the BL0103 study.

Table 13-5: BL0103 Composite Assemblies

Composite		Mass, kg
UWZ - Main	**Total**	**75.0**
HW-UWZ	Hanging wall dilution - Upper West Zone	6.2
MS-UWZ	Massive Sulphide - Upper West Zone	32.0
CSZ-UWZ	Copper Stringer - Upper West Zone	30.5
FW-UWZ	Footwall dilution - Upper West Zone	6.2
Z2 - Main	**Total**	**123.6**
HW-Z2	Hanging wall dilution - Zone 2	10.3
MS-Z2	Massive Sulphide - Zone 2	60
CSZ-Z2	Copper Stringer - Zone 2	43
FW-Z2	Footwall dilution - Zone 2	10.3

The average chemical composition of the three ore zone composites used for the KM3125 study is summarized within Table 13.6. Zinc Total assays were determined using a peroxide fusion method, whereas all other assays in the study were determined using standard aqua regia digestion. Zinc spinel (gahnite) was present in the ore in minor amounts and was digested in a standard aqua regia method and accounted for the additional zinc assayed as Zinc Total. Observed gahnite proportions were reported as per Table 13.6.

Table 13-6: Chemical Compositions for the Ore Zone Composites for KM3125

Element	Symbol	Units	CSZ	MS	UW-MS
Copper	Cu	%	1.45	0.30	1.61
Weak Acid Soluble Copper	$Cu_{(ox)}$	%	0.004	<0.001	0.008
Cyanide Soluble Copper	$Cu_{(CN)}$	%	0.02	0.02	0.06
Lead	Pb	%	0.02	0.43	0.16
Zinc (Aqua Regia)	Zn	%	0.16	7.05	3.71
Zinc Total	$Zn_{(t)}$	%	0.17	7.25	3.97
Zinc Oxide	$Zn_{(ox)}$	%	<0.001	0.03	0.01
Iron	Fe	%	7.40	28.30	17.80
Gold	Au	g/t	0.34	0.19	0.55
Silver	Ag	g/t	8	16	25
Magnesium	Mg	%	2.06	3.71	5.83
Sulphur	S	%	4.60	31.7	18.2

Table 13-7: Chemical Compositions for the Various Zones used to Build the Two Composites for BL0103

Composite	Analyte and Unit Symbol									
	Cu, %	Pb, %	Zn, %	Fe, %	As, %	Sb, %	Ag g/t	Au g/t	S %	C %
HW-WZ	0.72	<0.01	0.22	7.6	<0.01	<0.01	22	0.28	2.48	0.08
HW-Z2	0.11	0.05	0.25	4.3	<0.01	<0.01	3	0.03	1.68	0.08
FW-UWZ	0.55	<0.01	0.04	5.3	<0.01	<0.01	4	0.10	0.93	0.05
FW-Z2	0.38	<0.01	0.12	3.6	<0.01	<0.01	<3	0.05	1.22	0.10
CSW-UWZ	1.74	0.02	0.36	6.6	<0.01	<0.01	15	0.46	2.56	0.12
CSW-Z2	1.64	0.02	0.26	6.5	<0.01	<0.01	11	0.27	3.62	0.10
MS-UWZ	2.76	0.91	9.51	17.80	0.02	<0.01	78	1.91	22.5	0.84
MS-Z2	0.33	0.30	7.80	26.6	0.04	<0.01	20	0.20	32.0	2.36
UWZ-Main	1.73	0.41	4.21	11.2	<0.01	<0.01	39	1.23	10.9	0.40
Z2-Main	0.71	0.17	4.06	16.0	0.03	<0.01	14	0.20	17.3	1.26

Sulphur and iron levels were higher in the massive sulphide composites than the stockwork, and this is taken to mean more significant pyrite/pyrrhotite concentration in the MS. The lower levels of sulphide gangue in the copper stockwork composites would suggest superior performance as compared to the MS. Mineral contents for the three massive sulphide and stockwork zone composites used in KM3125 were shown as Table 13.8. Chalcopyrite included trace amounts of chalcocite, covellite and tennanite. UWZ was noted to contain elevated amounts of Talc.

Table 13-8: Mineral Compositions for the Ore Zone Composites for KM3125

Mineral	Mineral Content, %		
	CSZ	MS	UW-MS
Chalcopyrite	4.2	0.9	5.3
Galena	0	0.5	0.1
Sphalerite	0.2	10.7	5.8
Gahnite	0.1	0	1.2
Pyrite	4.4	50	25.8
Pyrrhotite	0.1	2.7	3.8
Quartz	62.6	3.3	7.3
Chlorite	13.3	5.5	20.1
Dolomite	0.3	13.5	5.7
Muscovite	8.6	0.8	2.2
Feldspars	2.4	1	4.7
Amphibole	1.1	1.3	4.7
Calcite	0.1	2.2	2.8
Iron Oxides	0.05	2.3	2.5
Talc	0.6	1.2	2.6
Serpentine	0.05	1.4	1.6
Others	1.9	2.7	3.8
Total	**100**	**100**	**100**

Table 13.9 summarized the mineral associations for the three ore zone composites (measured in 2D). CSZ showed about 63% liberation of chalcopyrite at a grind size of p80 of 95 µm, adequate for roughing. At similar grind size distributions, UW and Z2 massive sulphides indicated finer primary grinding requirements. Sphalerite liberations in all three composites were low (33-38% liberated), indicating finer primary grinding requirements.

Table 13-9: Mineral Associations for the Ore Zone Composites for KM3125

Mineral Class	CSZ, 95um K_{80}					CSZ, 95um K_{80}					CSZ, 95um K_{80}				
	Cs	Ga	Sp	Py	Gn	Cs	Ga	Sp	Py	Gn	Cs	Ga	Sp	Py	Gn
Liberated	63	39	35	62	93	25	18	38	59	77	42	22	33	43	73
Binary - Cs		-	21	7	3		1	5	1	1		3	18	6	9
Binary - Ga	-		-	-	<1	<1		1	<1	<1	<1		1	<1	<1
Binary - Sp	1	-		1	<1	30	18		13	9	10	23		7	2
Binary - Py	2	-	2		2	4	2	18		5	4	1	13		8
Binary - Gn	27	27	15	20		6	12	7	6		23	14	6	17	
Multiphase	7	33	26	10	1	34	50	30	20	7	21	37	29	27	8

Flotation testing for KM3125 culminated in several locked cycle tests (Table 13.10 for CSZ, 13.2.11 for Z2 and 13.2.12 for UWZ composites respectively). The CSZ locked cycle test was conducted at a primary grind of p80 95 µm and a regrind at p80 35 µm using lime, $ZnSO_4$, NaCN and 3418A as the main reagents.

Table 13-10: CSZ Locked Cycle Test Results KM3125

Product	Mass %	Assay - % or g/t					Distribution, %				
		Cu	Zn	S	Ag	Au	Cu	Zn	S	Ag	Au
Feed	100.0	1.6	0.16	4.5	8.0	0.38	100	100	100	100	100
Copper Conc	5.1	29.2	1.05	33.5	126	6.38	94.4	33.8	38.0	76.9	84.6
Copper 1st Clnr Tail	3.9	0.79	0.57	19.3	14.0	0.81	2.0	14.1	16.8	6.8	8.3
Copper Ro Tail	91.0	0.06	0.09	2.2	2.0	0.03	3.7	52.1	45.2	16.4	7.1

The locked cycle tests for the MS (Z2) composite were conducted at primary grind 80% passing 73 µm, copper/lead bulk regrind at 80% passing 12-21 µm and zinc regrind of 80% passing 14-16 µm using lime, $ZnSO_4$, NaCN and 3418A as the main copper (bulk) reagents, with lime, copper sulphate and SIPX used as the main zinc reagents.

Table 13-11: MS (Z2) Locked Cycle Test Results KM3125

Product	Mass %	Assay - % or g/t						Distribution, %					
		Cu	Pb	Zn	S	Ag	Au	Cu	Pb	Zn	S	Ag	Au
Test 17													
Bulk Feed	100.0	0.31	0.37	7.08	30.0	18	0.16	100.0	100.0	100.0	100.0	100.0	100.0
Prefloat	3.5	0.21	0.36	5.81	13.1	12	0.06	2.3	3.4	2.8	1.5	2.4	1.3
Bulk Conc	1.6	11.2	13.8	9.78	32.0	359	4.94	55.3	57.9	2.1	1.7	31.9	47.9
Zn Conc	10.3	0.77	0.51	53.8	33.0	45	0.23	25.2	14.1	78.4	11.3	26.4	15.0
Zn 1st Clnr Tail	9.7	0.24	0.24	3.23	28.0	17	0.13	7.5	6.4	4.4	9.0	9.4	7.8
Zn Rougher Tail	75.0	0.04	0.09	1.16	30.6	7	0.06	9.7	18.2	12.2	76.5	30.0	28.0
Test 20													
Bulk Feed	100.0	0.33	0.41	6.96	28.4	15	0.21	100.0	100.0	100.0	100.0	100.0	100.0
Prefloat	3.5	0.20	0.32	5.89	12.8	11	0.11	2.1	2.8	3.0	1.6	2.6	1.9
Bulk Conc	1.6	11.9	15.4	9.18	28.5	332	5.27	56.0	59.1	2.1	1.6	34.4	38.5
Zn Conc	10.8	0.63	0.46	55.0	32.1	38	0.29	20.5	12.2	85.4	12.2	27.3	14.6
Zn 1st Clnr Tail	17.8	0.16	0.15	0.96	38.6	10	0.15	8.4	6.5	2.5	24.1	11.6	12.7
Zn Rougher Tail	66.3	0.07	0.12	0.74	25.9	5	0.10	13.1	19.5	7.0	60.5	24.1	32.3

The locked cycle tests for the UWZ composite were conducted at primary grind of 80% passing 95 µm, copper/lead bulk regrind at 80% passing 10-17 µm and zinc regrind of 80% passing 13-22 µm using lime, $ZnSO_4$, NaCN and 3418A as the main copper (bulk) reagents, with lime, copper sulphate and SIPX used as the main zinc reagents.

Table 13-12: UW Locked Cycle Test Results KM3125

Product	Mass %	Assay - % or g/t						Distribution, %					
		Cu	Pb	Zn	S	Ag	Au	Cu	Pb	Zn	S	Ag	Au
Bulk Feed	100.0	1.75	0.18	4.02	17.3	26	0.66	100.0	100.0	100.0	100.0	100.0	100.0
Prefloat	2.8	0.86	0.13	1.44	4.0	23	1.16	1.4	2.1	1.0	0.6	2.5	4.9
Copper Conc	6.0	24.2	1.3	6.40	34.4	216	6.50	83.4	43.4	9.6	12.0	50.3	59.7
Zn Conc	5.6	1.87	0.24	54.3	32.5	63	0.81	6.0	7.5	76.3	10.6	13.6	6.9
Zn 1st Clnr Tail	7.7	0.69	0.14	2.39	21.4	22	0.53	3.0	6.0	4.6	9.5	6.6	6.2
Zn Rougher Tail	77.8	0.14	0.10	0.44	15.0	9	0.19	6.2	41.0	8.5	67.2	27.0	22.2

Table 13.13 summarizes the locked cycle test results for BL0103 for Z2 and UWZ composites under varying conditions. Improved results were obtained for Z2 with a primary grind of p80 of 75 µm versus p80 of 100 µm. Copper and zinc regrind sizing's were 80% passing 24 and 21 µm, respectively. UWZ composite responded better with the finer primary grind at 80% passing 75 µm, and with finer copper regrind at 80% passing 15 µm. Zinc regrind was 80% passing 21-22 µm.

Table 13-13: BL0103 Locked Cycle Test Results for Z2 and UWZ Composites

Product	Mass %	Assay - % or g/t						Distribution, %					
		Cu	Pb	Zn	S	Ag	Au	Cu	Pb	Zn	S	Ag	Au
BL103 Test 54, Z2-Main (75um grind)													
Feed	100.0	0.68	0.15	4.08	16.0	14	0.25	100.0	100.0	100.0	100.0	100.0	100.0
Copper Con	2.0	**23.4**	3.18	5.84	25.8	301	5.67	**58.7**	35.9	2.5	2.8	36.8	39.0
Zinc Conc	6.0	1.9	0.5	**51.5**	31.0	41	0.29	17.8	22.4	**79.7**	12.2	18.4	7.4
Zinc 1st Clnr Tail	13.0	0.83	0.24	4.0	25.6	18	0.17	16.0	20.7	13.0	21.0	16.9	9.5
Zn Rougher Tail	79.0	0.07	0.04	0.25	13.0	5	0.14	7.5	21.0	4.8	64.0	28.0	44.1
BL103 Test 55, UWZ Main (75um grind)													
Feed	100.0	1.93	0.40	4.00	8.7	45	1.17	100.0	100.0	100.0	100.0	100.0	100.0
Copper Con	7.0	**22.10**	3.98	9.90	28.4	361	13.0	83.8	73.1	18.2	23.9	59.1	81.7
Zinc Conc	5.0	2.3	0.9	**54.50**	30.2	122	1.23	6.3	12.3	71.9	18.3	14.3	5.6
Zinc 1st Clnr Tail	8.0	1.50	0.35	2.97	12.7	62	0.50	6.5	7.3	6.2	12.1	11.6	3.6
Zn Rougher Tail	79.0	0.08	0.04	0.19	5.0	9	0.14	3.4	7.3	3.8	45.7	15.0	9.2

Elemental ICP-MS scans for minor elements indicated fluorine and selenium concentrations in the KM3125 MS bulk concentrate of 506 and 430ppm respectively - levels that may attract penalties in certain smelters. Mercury levels in the KM3125 massive sulphide bulk and zinc concentrates varied from 80-155gpt and these are also in a range that can attract penalties in certain smelters. Concentrates in BL0103 were generally lower with respect to mercury versus the KM3125 results, but fluorine and selenium were again elevated in the copper concentrate for BL0103 concentrates.

13.3 Base Metallurgical Study (BL0351)

The BL0351 metallurgical testwork program was commissioned in late 2018 and was designed to support engineering and economic assumptions used in this PFS. A scope of work was defined to focus on optimization of the three main ore types (Copper Stockwork, Zone 2 and Upper West Zone) whilst considering two main processing options (toll processing in Flin Flon vs new on site facilities).

The final flowsheet developed at the locked cycle test level was a conventional sequential flotation process with grinding to a target 80% passing 75 µm and where a bulk final copper- lead concentrate was produced while depressing sphalerite with $ZnSO_4$/NaCN prior to floating zinc into a final concentrate. A secondary flowsheet was subsequently developed using sulphur dioxide gas (SO_2) or sodium metabisulphite (SMBS) for depression of galena and zinc as a useful and effective alternative to cyanide use.

13.4 Sample Characterization

13.4.1 Sample Preparation

Metallurgical composites created for the BL0351 program were prepared using a combination of fresh drill core from 2018 drilling and coarse rejects from the previous drilling campaign. Although the older coarse rejects were seen to be well packaged in oxygen-deficient conditions, a quality assurance step was introduced to confirm that this older material had not succumbed to surficial oxidation. The QA program subjected reject and drill core material from similar DDH intervals to comparative baseline flotation tests, the results of which showed similar metallurgical response. This testwork is described in more detail in Section 13.6.1 below.

The study assessed metallurgical performance by optimizing open circuit flotation conditions for the three main ore types, or "master composites" then verifying the new flowsheet with blended ratios of each master composite and 15 variability composites.

Samples were received in two shipments – August 31 and December 4, 2018. The first shipment was approximately 426kg of half NQ core, with the second shipment consisting of approximately 460kg of crushed and bagged coarse rejects. Each sub-composite and variability sample were submitted in duplicate for Cu, Pb, Zn, Fe and Ag by aqua regia; S and C by LECO, Au by fire assay and ICP for multi-elemental analyses. A summary of head assay measurements is given in Table 13.14 (ore type composites) and Table 13.15 (variability composites).

Table 13-14: BL0351 Composite Head Assay Summary

Composite	Assays							
	Cu, %	Pb, %	Zn, %	Fe, %	Ag, g/t	Au, g/t	S, %	C, %
Z2 Core Comp	0.30	0.51	7.50	21.0	19	0.22	25.7	2.09
Z2 Reject. Comp	0.30	0.50	7.90	21.0	20	0.19	26.3	2.10
UWZ Core Comp	1.95	0.20	4.55	23.6	24	1.20	25.8	1.49
UWZ Reject. Comp	1.75	0.20	4.50	24.1	22	1.21	27.2	1.40
Z2: Sub-Comp	0.33	0.44	6.65	20.1	16	0.20	25.2	1.93
UWZ: Sub-Comp	1.93	0.18	4.25	22.3	22	0.97	24.8	1.27
CSW: Sub-Comp	1.24	0.04	0.29	6.37	8	0.64	4.48	0.09
Blend 1	1.09	0.19	2.54	13.4	12	0.63	13.3	0.76
Blend 2	1.40	0.17	2.70	15.6	16	0.40	15.8	0.79
Blend 3	1.08	0.28	4.00	17.4	18	0.63	19.0	1.19
Blend 4	1.15	0.14	1.55	10.3	11	0.31	9.71	0.47

Table 13-15: BL0351 Variability Sample Head Assay Summary

Composite	Assays							
	Cu %	Pb %	Zn %	Fe %	Ag g/t	Au g/t	S %	C %
Z2-1	0.33	0.30	4.36	19.3	21	0.19	19.2	1.93
Z2-2	0.79	0.28	6.56	26.5	25	0.28	29.3	2.37
Z2-3	0.22	0.49	9.12	30.4	17	0.18	40.1	1.56
Z2-4	0.28	0.83	11.4	26.9	31	0.15	30.0	3.08
UWZ-1	0.47	0.36	4.20	16.1	23	0.64	16.1	3.48
UWZ-2	2.53	0.07	2.33	22.2	19	2.04	23.1	0.32
UWZ-3	0.40	0.30	6.25	25.7	21	0.55	26.0	1.16
UWZ-4	1.06	0.13	6.95	23.3	17	0.25	29.6	2.95
UWZ-5	5.76	0.82	4.25	17.6	104	3.17	18.8	0.24
CSZ-1	1.06	0.03	0.22	14.2	12	0.40	15.1	0.03
CSZ-2	2.07	0.05	0.56	20.5	17	0.43	21.3	0.03
CSZ-3	0.76	0.01	0.02	3.65	2	<0.01	1.42	<0.01
CSZ-4	1.53	0.01	0.06	5.15	10	1.06	3.21	0.02
CSZ-5	0.98	0.05	2.38	14.1	12	0.26	16.1	<0.01
Lens 3	0.34	0.18	4.84	21.0	12	0.15	25.7	0.81

13.4.2 Mineralogy

Representative samples of the master composites were submitted for detailed feed mineralogy to determine the major mineral species and occurrences. Samples were ground to the target grind size of 80% passing 75 µm, sized to fractions of +75, -75/+38 and -38 µm, polished sections created for each fraction and analysed by QEMSCAN to yield mineral abundance, liberation and associations of key sulphide minerals using Particle Mineralogical Analyses (PMA). Mineral abundances are summarized in Table 13.16, and liberation and exposure values are summarized in Table 13.17 and Table 13.18

The ores contained chalcopyrite, galena, sphalerite, pyrite and pyrrhotite almost exclusively, with only very minor occurrences of other sulphide minerals observed. Pyrite predominated over pyrrhotite in all ore types. For non-sulphide gangue, copper stockwork contained mostly quartz, mica, and chlorite, while massive sulphide ores contained more carbonates, iron oxides and talc and less quartz. Clay contents were generally low in all three ore type composites.

Of note, zinc deportment in the UWZ sample included 5% in gahnite – an unrecoverable zinc aluminium oxide mineral. The gahnite content in the CSZ sample was higher at 16% of the total zinc, but in this case the total zinc grades were generally too low in total to represent any significant zinc production in the overall deposit.

Table 13-16: BL0351 Mineral Distributions for the Three Main Ore Types

Normalized by Fraction (wt. %)	Composite		
	CSZ	UWZ	Z2
Pyrite	5.23	33.6	36.8
Pyrrhotite	0.49	2.34	1.00
Chalcopyrite	4.74	6.61	1.09
Sphalerite	0.70	7.28	12.6
Galena	0.02	0.13	0.48
Other Sulphides	0.01	0.11	0.04
Quartz	55.2	8.88	11.5
Feldspar	1.79	1.58	2.22
Amphibole/Pyroxene	0.74	2.93	1.28
Mica	10.5	2.04	4.34
Chlorite	17.3	18.4	11.7
Talc	0.60	3.18	2.06
Clays	0.96	0.33	0.22
Other Silicates	0.14	0.24	0.22
Fe Oxides	0.57	2.53	1.34
Zn-Al Oxide	0.22	0.63	0.07
Other Oxides	0.31	0.26	0.20
Carbonates	0.40	8.79	12.8
Apatite	0.08	0.09	0.08
Other	0.02	0.02	0.03

The mineral associations with the main economic minerals (chalcopyrite and sphalerite) are summarized in Table 13.17 and Table 13.18. Chalcopyrite liberation was highest in the CSZ composite at 86% and lowest in the Z2 composite at 62%. Sphalerite liberation was relatively consistent across all ore types at 72-74%. Liberations of both chalcopyrite and sphalerite increased with decreasing particle size.

Table 13-17: Normalized Mineral Associations: Chalcopyrite

Mass (% Chalcopyrite)	Composite		
	CSZ	UWZ	Z2
Pure Chalcopyrite	71.3	62.0	49.9
Free Chalcopyrite	8.61	7.24	6.35
Lib Chalcopyrite	5.99	8.61	5.83
Chalcopyrite:Pyrite	0.72	3.53	4.59
Chalcopyrite:Pyrrhotite	0.44	0.60	0.54
Chalcopyrite:Sphalerite	1.11	2.04	9.79
Chalcopyrite:Sphalerite:Pyrite	0.12	0.67	3.91
Chalcopyrite:Galena	0.00	0.01	0.14
Chalcopyrite:Qtz/Feld	3.51	0.57	1.50
Chalcopyrite:Mica/Chlor/Clays/Talc	2.62	3.86	1.87
Chalcopyrite:Other Silicates	0.00	0.37	0.29
Chalcopyrite:Carbonates	0.03	0.80	4.13
Chalcopyrite:Oxides	0.11	0.18	0.16
Complex	5.44	9.50	11.0
Total	100	100	100
Free and liberated	**85.9**	**77.9**	**62.1**

Table 13-18: Normalized Mineral Associations: Sphalerite

Mass (% Chalcopyrite)	Composite		
	CSZ	UWZ	Z2
Pure Sphalerite	59.2	55.3	55.8
Free Sphalerite	6.32	9.07	7.70
Lib Sphalerite	8.73	9.74	8.39
Sphalerite:Pyrite	2.52	8.26	11.6
Sphalerite:Pyrrhotite	0.15	0.39	0.51
Sphalerite:Chalcopyrite	5.25	2.16	0.82
Sphalerite:Chalcopyrite:Pyrite	0.35	1.13	0.72
Sphalerite:Galena	0.00	0.00	0.16
Sphalerite:Qtz/Feld	4.79	0.12	0.29
Sphalerite:Mica/Chlor/Clays/Talc	3.02	2.25	1.08
Sphalerite:Other Silicates	0.09	0.12	0.04
Sphalerite:Carbonates	0.23	1.42	2.76
Sphalerite:Oxides	0.01	0.14	0.08
Complex	9.34	9.90	10.1
Total	100	100	100
Free and liberated	**74.3**	**74.1**	**71.9**

Table 13.19 and Table 13.20 summarize mineral exposures and yielded an approximation for anticipated recoveries into final concentrates.

Table 13-19: Mineral Exposures vs Flotation Results: Chalcopyrite

Mass (% Chalcopyrite)	Composite		
	CSZ	UWZ	Z2
Exposed	83.0	75.1	60.0
>50-80% Exposed	8.07	11.2	13.4
>20-50% Exposed	4.67	8.32	12.8
<20% Exposed	3.88	5.01	12.9
Locked	0.40	0.30	0.93
Total	100	100	100
>20% Exposed	**95.7**	**94.7**	**86.2**
LCT Cu Ro Recovery (est.)	**96.7**	**89.4**	**80.9**
	LCT42	LCT44	LCT80
	Zn/CN	Zn/CN	MBS

Table 13-20: Mineral Exposures vs Flotation Results: Sphalerite

Mass (% Chalcopyrite)	Composite		
	CSZ	UWZ	Z2
Exposed	45.5	51.6	42.1
>50-80% Exposed	13.3	20.4	22.1
>20-50% Exposed	20.0	18.6	25.2
<20% Exposed	19.6	8.76	10.2
Locked	1.75	0.59	0.36
Total	100	100	100
>20% Exposed	**78.7**	**90.7**	**89.4**
LCT Zn Ro Recovery	**-**	**80.4**	**91.2**
	LCT42	LCT44	LCT80
	Zn/CN	Zn/CN	MBS

13.5 Grindability

13.5.1 Crushing Work Index

Crushing work indices were not determined in BL0351.

13.5.2 Bond Rod/Ball Mill Work Indices

Table 13.21 summarizes the ore Bond Rod/Ball Mill hardness test results for the three main ore composites fromKM3125.

Table 13-21: KM3125 Bond Rod and Ball Mill Grindability Test Results

Product	Bond Ball Mill Test		Bond Rod Mill Test	
	BWi_B kWh/t	P_{80}, micron	BWi_R kWh/t	P_{80}, micron
CSZ	16.1	80	17.0	882
MS	11.6	83	12.7	869
UW-MS	14.0	81	15.6	852

Table 13.22 summarized the ore Bond Ball Mill hardness test results for the BL0103 massive sulphide and copper stockwork composites.

Table 13-22: BL0103 Bond Ball Mill Grindability Test Results

Sample ID	BWI parameters				
	Grind, mesh	F80, um	P80, um	Gram/rev	Work Index, kWh/t
MS-Zone 2	150	2,351	80	1.82	11.3
MS UWZ	150	2,289	80	1.70	11.9
CSZ Zone 2	150	2,149	78	1.03	17.9
CSZ UWZ	150	2,400	78	0.98	18.4

Bond ball mill work indices (BWI) determined for the three composites in BL0351 were summarized in Table 13.23. The composite hardness ranged from 13.8 to 14.4kWh/t for the massive sulphide composites and 18.0 kWh/t for the copper stockwork composite, which spanned the medium to hard competency range.

Table 13-23: BL0351 Bond Ball Mill Grindability Test Results

Sample ID	BWI parameters				
	Grind, mesh	F_{80}, µm	P_{80}, µm	Gram/rev	Work Index, kWh/t
Zone 2 Master	150	1,684	80	1.50	13.8
UWZ Master	150	1,823	86	1.48	14.4
CSZ Master	150	2,013	79	1.04	18.0

Results from the two programs are quite consistent and show that the effective hardness of a blended feed to the mill will depend on the blend of CSZ:MS, with higher mill throughputs likely achievable when Massive Sulphide rich blends are processed. The PFS considers an average life of mine blend of roughly 33% CSZ and 67% MS, so the effective hardness will, on average be closer to the 12-14 kWh/t range than the 18 kWh/t point measured for CSZ.

13.5.3 SAG Mill Testwork

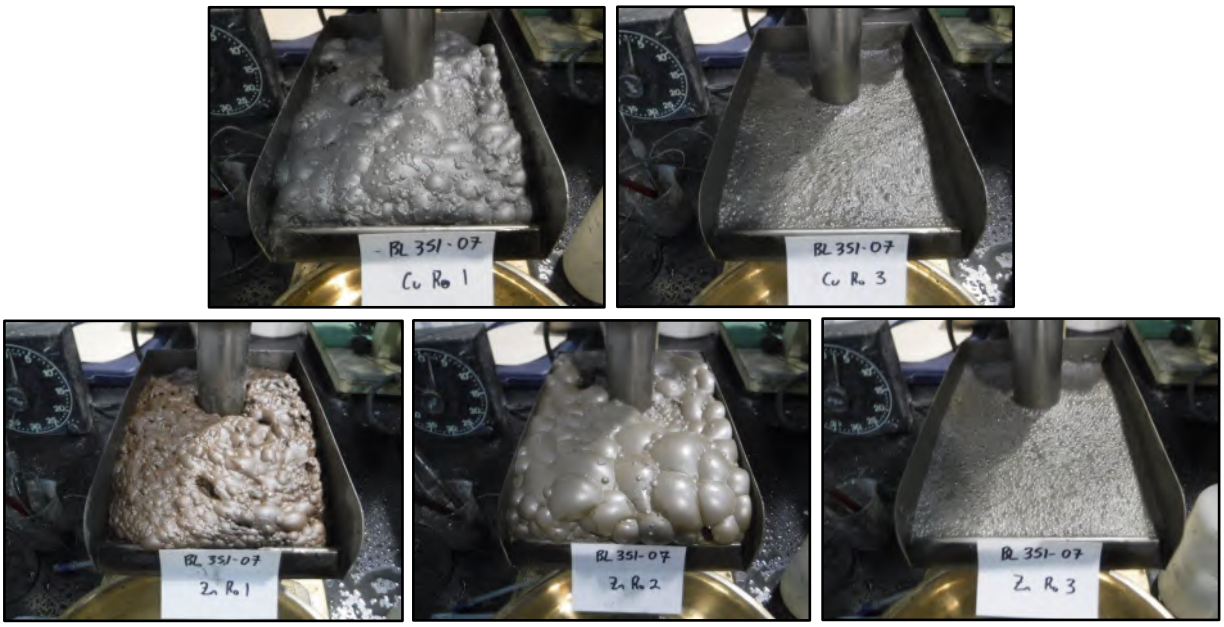
SAG Mill grindability testwork (DWi, SPI, SMC etc…) was not carried out in BL0351 as the proposed flowsheet (777 mill) uses rod mills and a ball mill. This will be addressed in future testwork.

13.6 Flotation Testing

The BL0351 flotation program consisted of open circuit rougher and cleaner tests, with locked cycle tests used towards the end of the program to provide robust recovery predictions. Flowsheet development was carried out on composites of CSW, Zone 2 Massive Sulphide and UWZ Massive Sulphide. Variability tests were completed on a variety of high- and low-grade samples, plus a program of blended composite testing (blends of CSZ and MS) was carried out.

The laboratory program BL0351 consisted of a total of 82 laboratory scale flotation tests, including 5 locked cycle tests. An example of the rougher flotation froth conditions for the Z2 composite is given in Figure 13-1 as an indication of typical performance.

Figure 13-1: Froth Conditions for Zone 2 Rougher Flotation



13.6.1 Sample Verification

Portions of the McIlvenna Bay ore deposit lie below areas of muskeg that are only accessible for drilling in winter months when the ground is frozen. With these seasonal drilling constraints and other scheduling issues in play, it was noted by the metallurgical team that insufficient massive sulphide (Z2 and UWZ) drill core mass was available for preparation of suitable composites for the 2018/19 metallurgical program.

A large quantity of coarse reject material had been bagged and stored at the primary assay lab in airtight containers from the previous drilling campaign and combined with similar reject mass from the latest metallurgical drilling campaign, more than sufficient mass was available for metallurgical testing. However, the use of this material raised concerns that that the mineralization within these coarse crushed samples may have suffered some degradation as a result of sulphide mineral surface oxidation etc.

Although visual examination of several samples gave no evidence of surface oxidation, the team decided to run some preliminary comparative flotation tests on small samples of core and the equivalent coarse reject (i.e. One half of core compared to the crushed/stored second half). This QA step was carried out prior to the main metallurgical program.

Two composites of massive sulphide corresponding to Zone 2 and UW Zone material were prepared. A series of five rougher kinetic tests using Z2 and UWZ massive sulphide materials evaluated performance of fresh drill core versus aged coarse assay rejects. Tests were conducted at 80% passing 100 µm primary grind and used optimized conditions from the previous BL0103 study.

Figure 13.1 and Figure 13.2 summarize the flotation response of each composite in terms of copper recovery versus mass pull to concentrate.

Figure 13-2: Copper Recoveries for Z2 Massive Sulphide Fresh Core versus Aged Assay Rejects

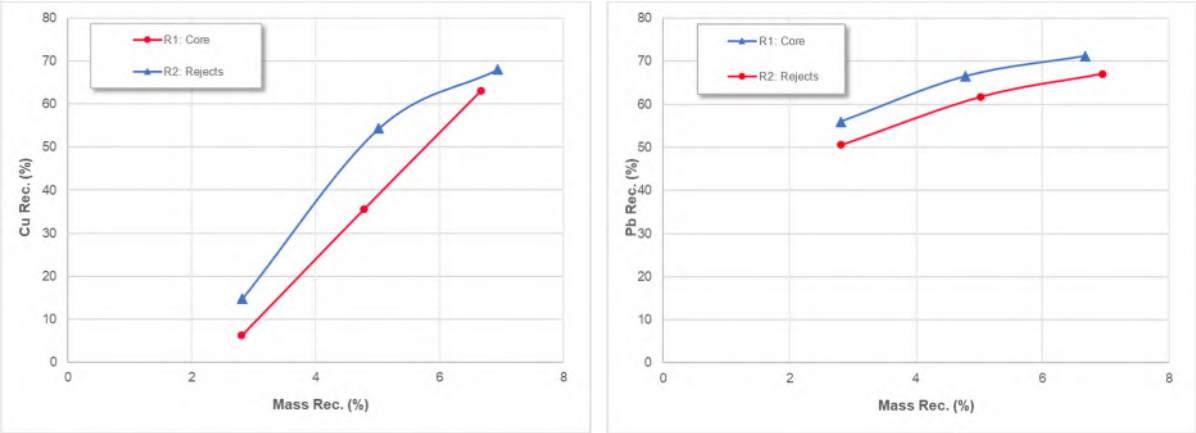
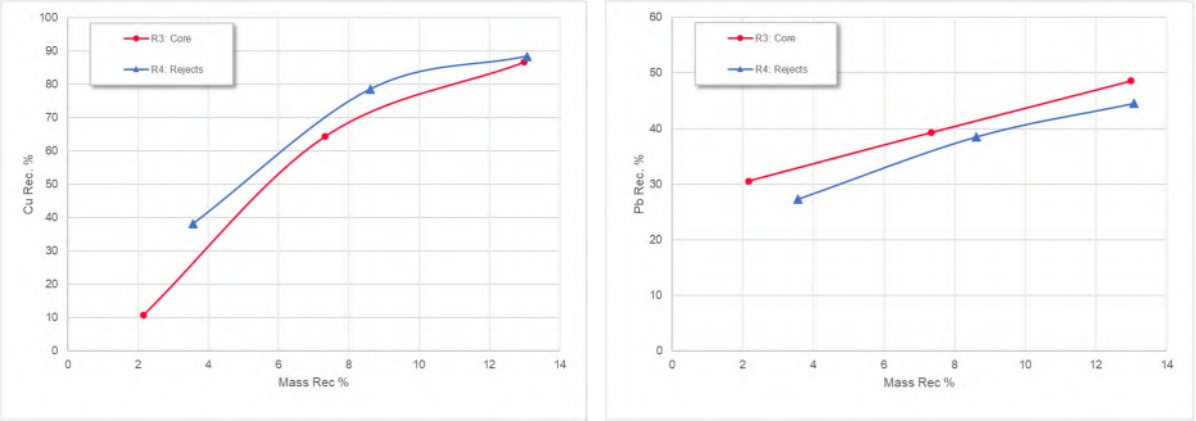


Figure 13-3: Copper Recoveries for UWZ Massive Sulphide Fresh Core versus Aged Assay Rejects



Results of the "aged" versus fresh samples raised little concern regarding sulphide mineral oxidation. A very small amount additional zinc was seemed to be recovering into the copper-lead bulk concentrate for the reject composites, but it is expected that movement of depressant addition into the primary grinding mill might help alleviate any mis-flotation of sphalerite. Copper recovery was slightly lower for the reject composites and it was noted that Eh out of the regrind mill was slightly lower in these tests. Again, a slight adjustment to conditions (i.e. a minor addition in aeration time, or

addition of secondary copper collectors such as Aero 5100) was expected to counter the issues observed.

With these results in hand, the metallurgical team was able to proceed with metallurgical testing using more representative composites of suitable mass and size distribution, using mixtures of fresh core and coarse reject material.

Composites of mineralization from the CSZ, Z2 and UWZ were assembled, using drill core halves and (for the two MS comps) coarse reject mass. The blending of core and rejects into the massive sulphide composites is summarized in Table 13-24 and Table 13-25 below.

Table 13-24: BL0351 Massive Sulphide Z2 Master Composites

Sample	Weight, kg	Cu, %	Pb, %	Zn, %
Core Comp	76.4	0.30	0.51	7.5
Rejects Comp	58.6	0.29	0.35	5.7
Additional Intervals	69.7	0.40	0.41	6.6
Total:	**204.6**	**0.3**	**0.43**	**6.7**

Table 13-25: BL0351 Massive Sulphide UWZ Master Composites

Sample	Weight, kg	Cu, %	Pb, %	Zn, %
Core Comp	105.3	1.95	0.20	4.55
Rejects Comp	85.9	2.08	0.15	4.02
Additional Intervals	101.5	1.53	0.22	3.74
Total:	**292.7**	**1.84**	**0.19**	**4.11**

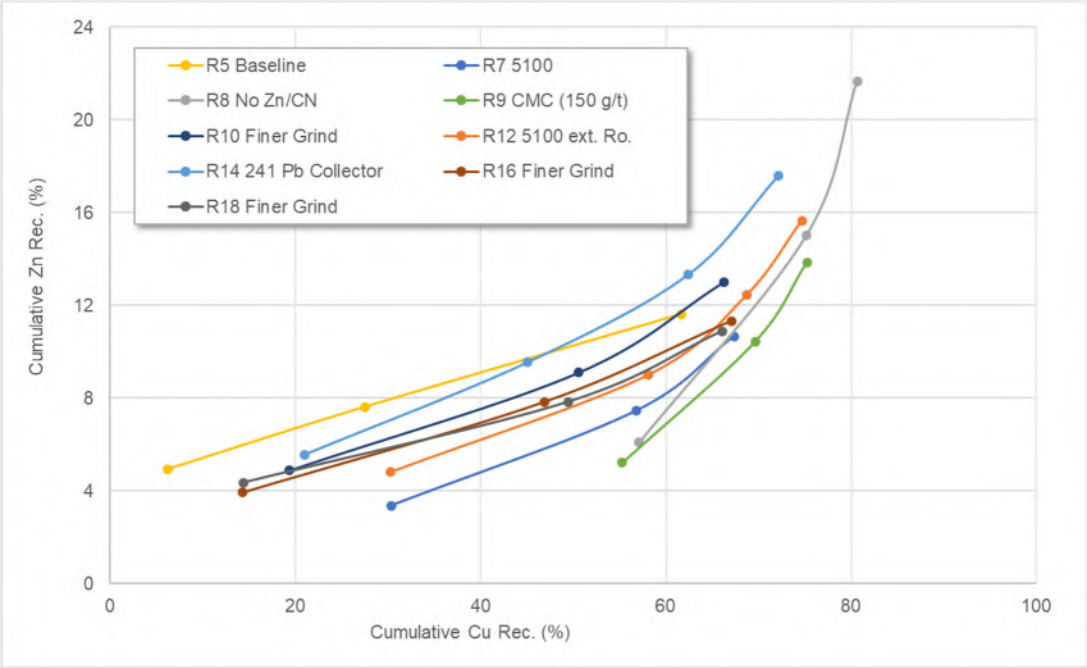
Each composite was stage-crushed to minus 6 mesh, thoroughly blended, and subsampled to make replicate 2-kg test charges. A head sample was split from a single test charge from each composite for assay determinations.

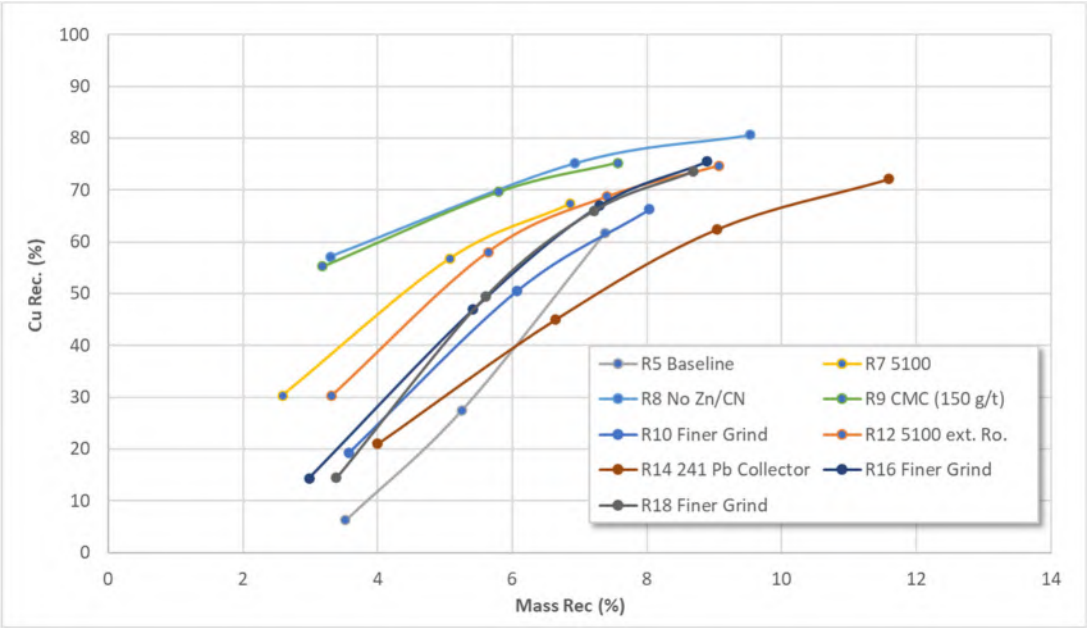
13.6.2 Rougher Flotation Testing

Previous test programs on samples of McIlvenna Bay mineralization have supported the use of a conventional grinding + sequential flotation flowsheet. As a starting point for this latest program of optimization, BL0351 rougher testing began with similar conditions from the previous BL0103 program locked cycle tests.

These initial (baseline) rougher performance profiles are shown for each of the three ore types via a performance comparison with the new various other BL0351 rougher results in the following figures – CSZ in Figure 13-4, UWZ in Figure 13-5 and CSZ in Figure 13-6.

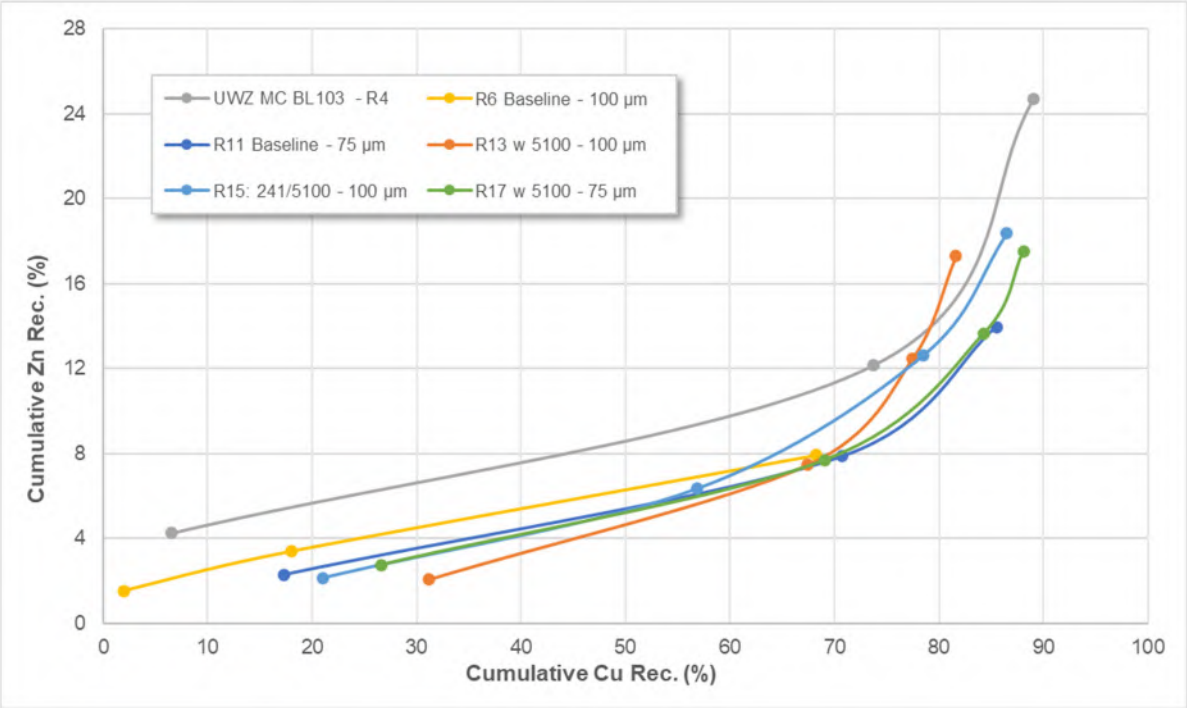
Figure 13-4: Rougher Kinetic Test Results for Zone 2 Comp – BL0351

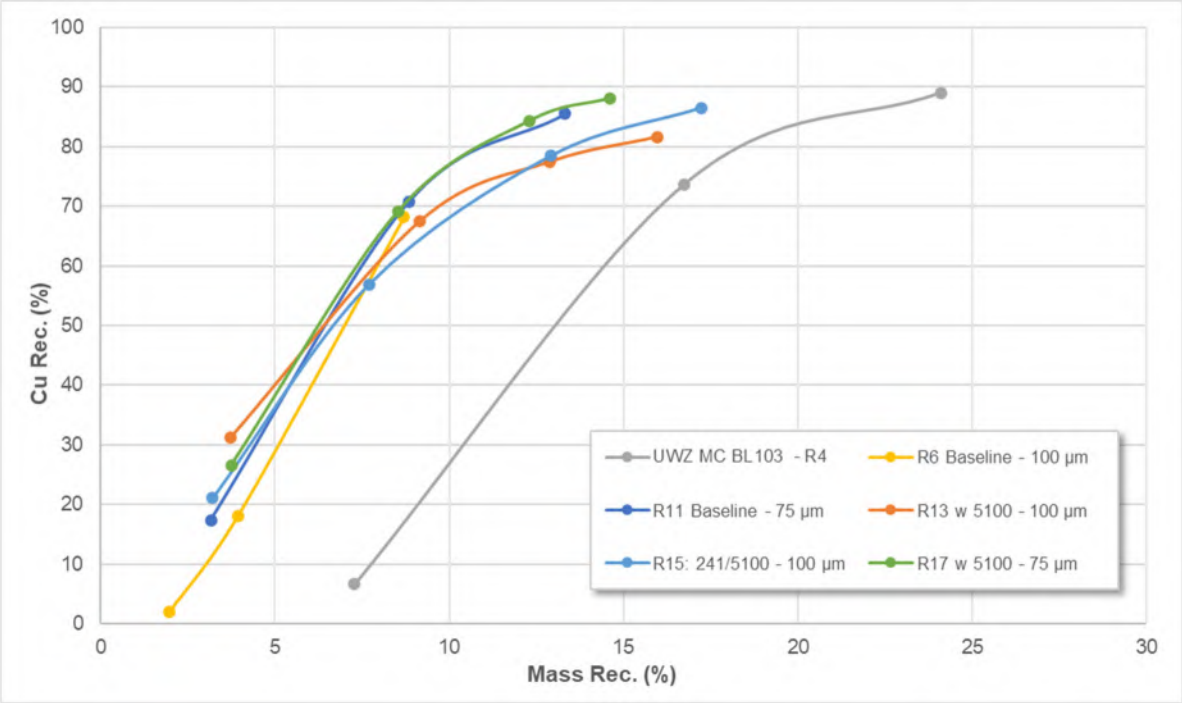




The majority of Z2 composite test results in the BL0351 program show an improvement over the baseline test using BL0103 conditions.

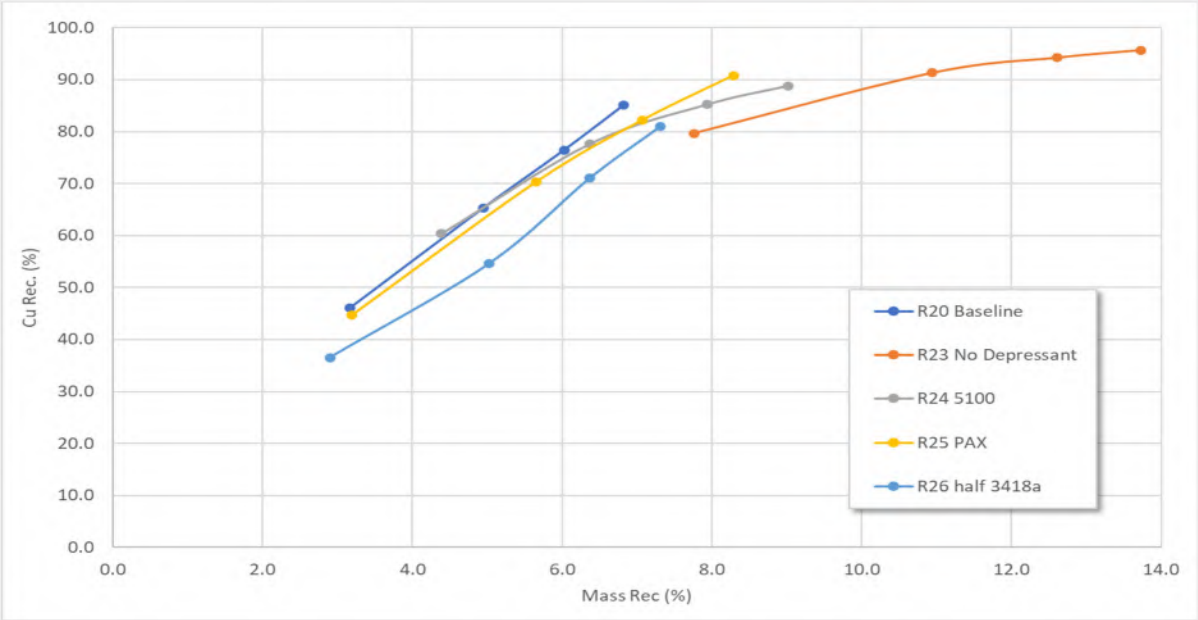
Figure 13-5: Rougher Kinetic Test Results for UWZ Comp – BL0351





Once more, the majority of BL0351 tests show improvement over the BL0103 conditions.

Figure 13-6: Rougher Kinetic Test Results for CSZ Comp – BL0351

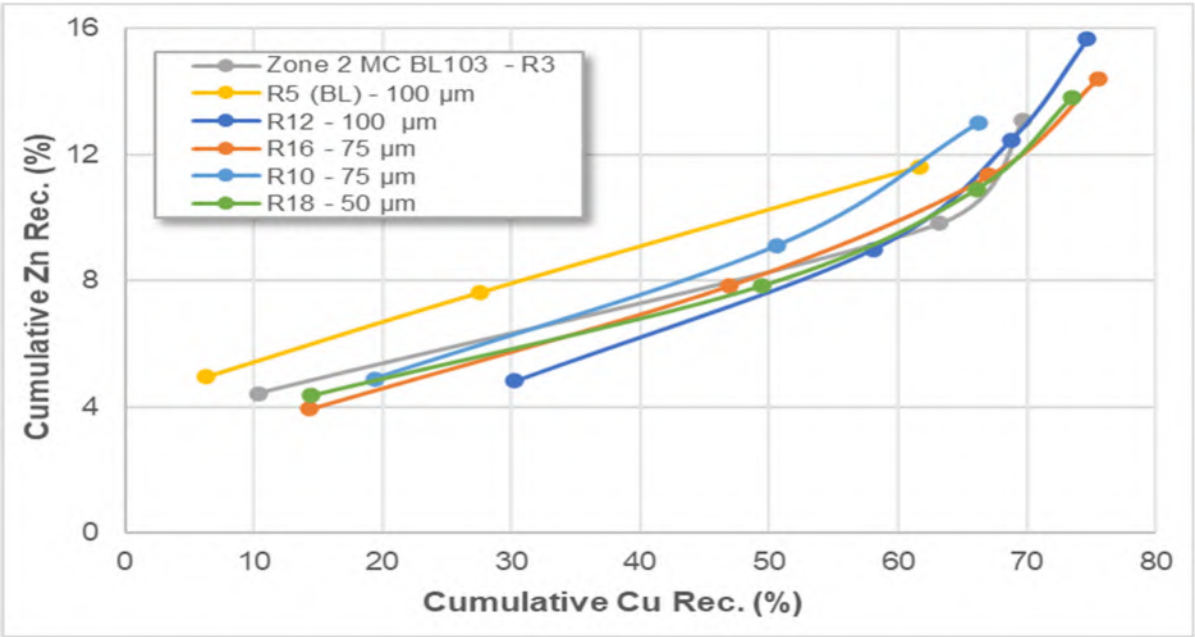


Again, results for three of the CSZ composite tests show an improvement over the baseline conditions.

Effect of Grind

The impact of primary grind was evaluated for the Zone 2 massive sulphide composite over a range of 80% passing 50 to 100 µm (Figure 13-7).

Figure 13-7: Effect of Primary Grind for Zone 2 Massive Sulphide Composite

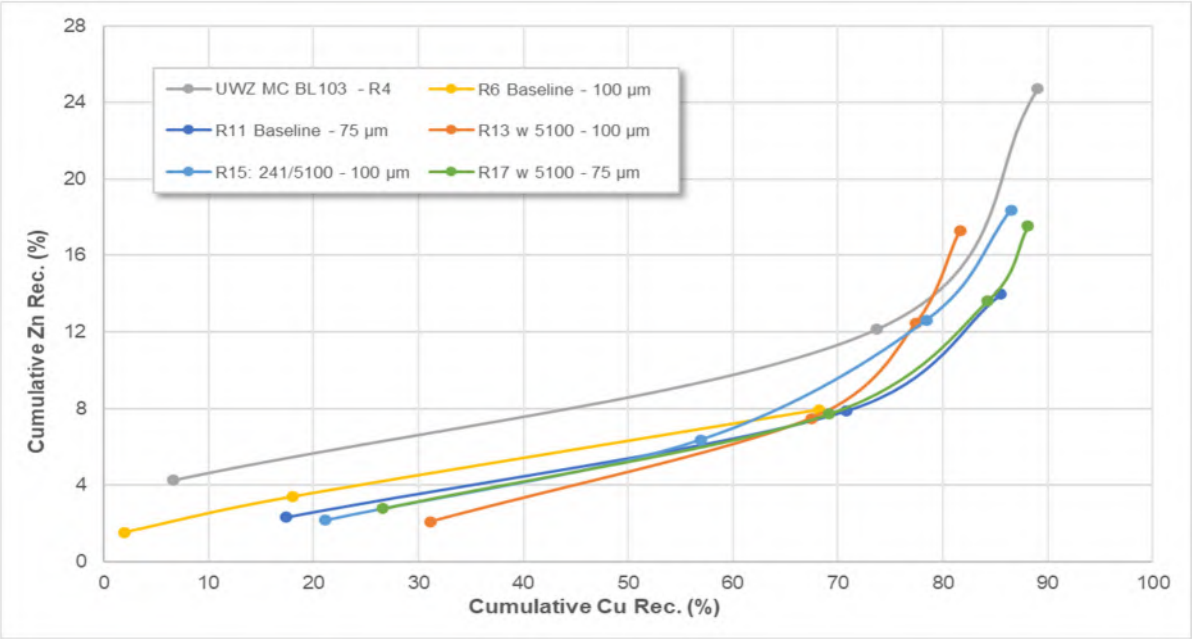


Optimal grind appeared to be in the range of 75-100 µm. The mass recovery relationship showed moderate diminishing performance of copper when grinding finer than 75 µm.

The primary grind size was similarly evaluated for UWZ for 80% passing 50 to 100 µm (Figure 13-8). Finer grind increased copper recovery and decreased zinc content in the copper-lead rougher concentrate at 75 µm but no further gains were made at 50 µm.

As with Z2, minor zinc recovery into the zinc rougher concentrate occurred at 50 µm versus 75 µm.

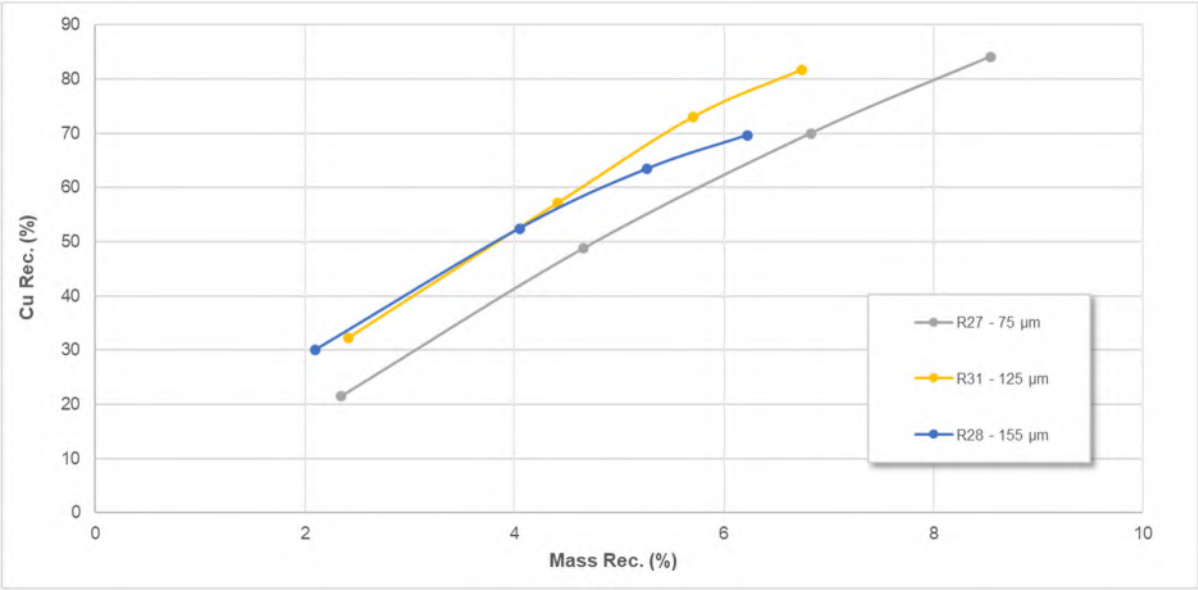
Figure 13-8: Effect of Primary Grind for UWZ Massive Sulphide Composite



The CSZ composite primary grind size was evaluated for 80% passing 75 to 155 µm (Figure 13-9). A grind size of 125 µm was superior to 155 µm, and also 75 µm (at the finer grind, it is possible that additional collector may have been required).

Importantly, if CSZ were to be batch processed for any reason, the test results indicate that a coarser primary grind than the MS would not be detrimental to recovery. This is convenient, as the CSZ has a higher work index than MS and for equivalent applied milling energy would have a coarser product.

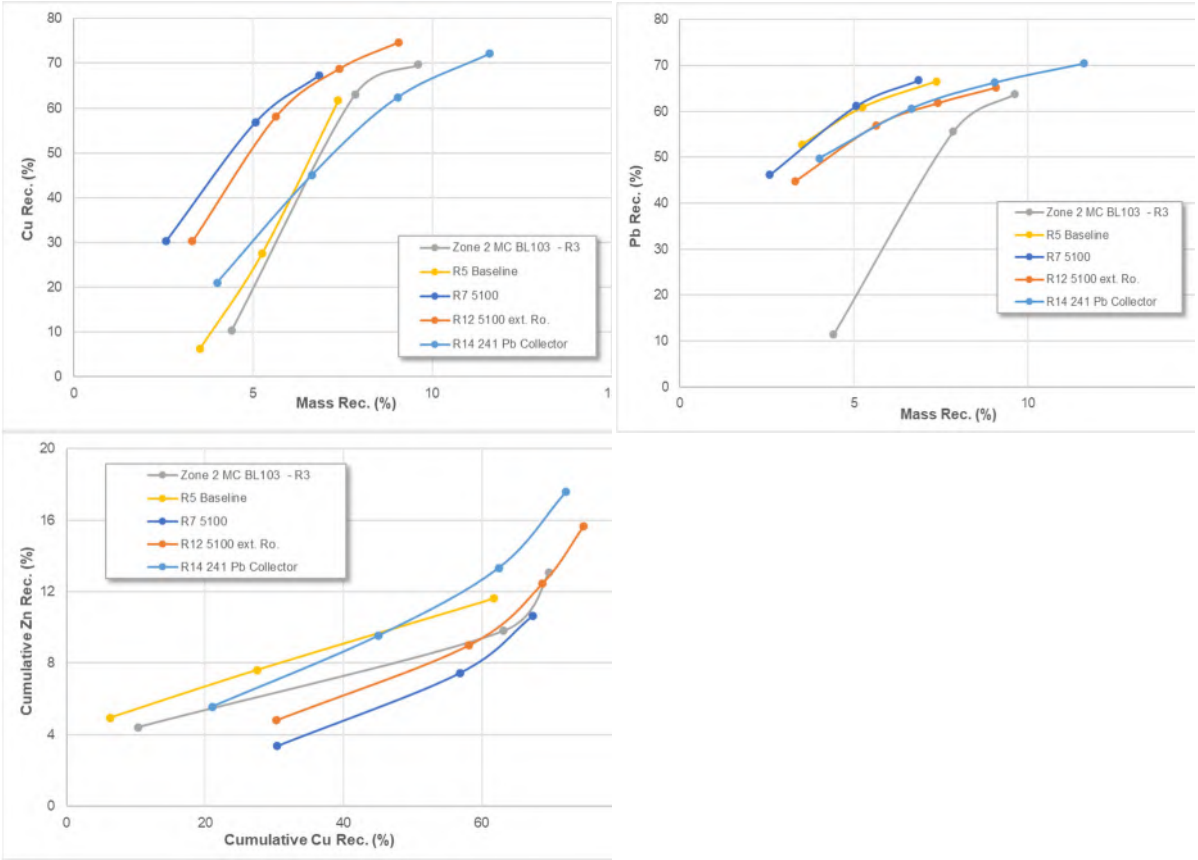
Figure 13-9: Effect of Primary Grind for CSZ Composite



Effect of Reagents

A series of rougher tests compared alternate collectors versus the baseline 3418A product. 3418A is an expensive dithiophosphinate collector, and it is always desirable to find cheaper alternatives. Fortunately, the use of Aero 5100 demonstrated good copper and lead recovery while being selective against sphalerite compared to the baseline test as shown in Figure 13.4. Collector selection had little impact on performance of UWZ mineralization in the copper/lead rougher. In addition to testing 5100 on the CSZ composite, PAX (potassium amyl xanthate) was also trialed. PAX was marginally more selective than 5100 with 3418A being the most selective for copper recovery versus overall mass pull.

Figure 13-10: Effect of Collector in the Zone 2 Copper-lead Roughers (3 charts)

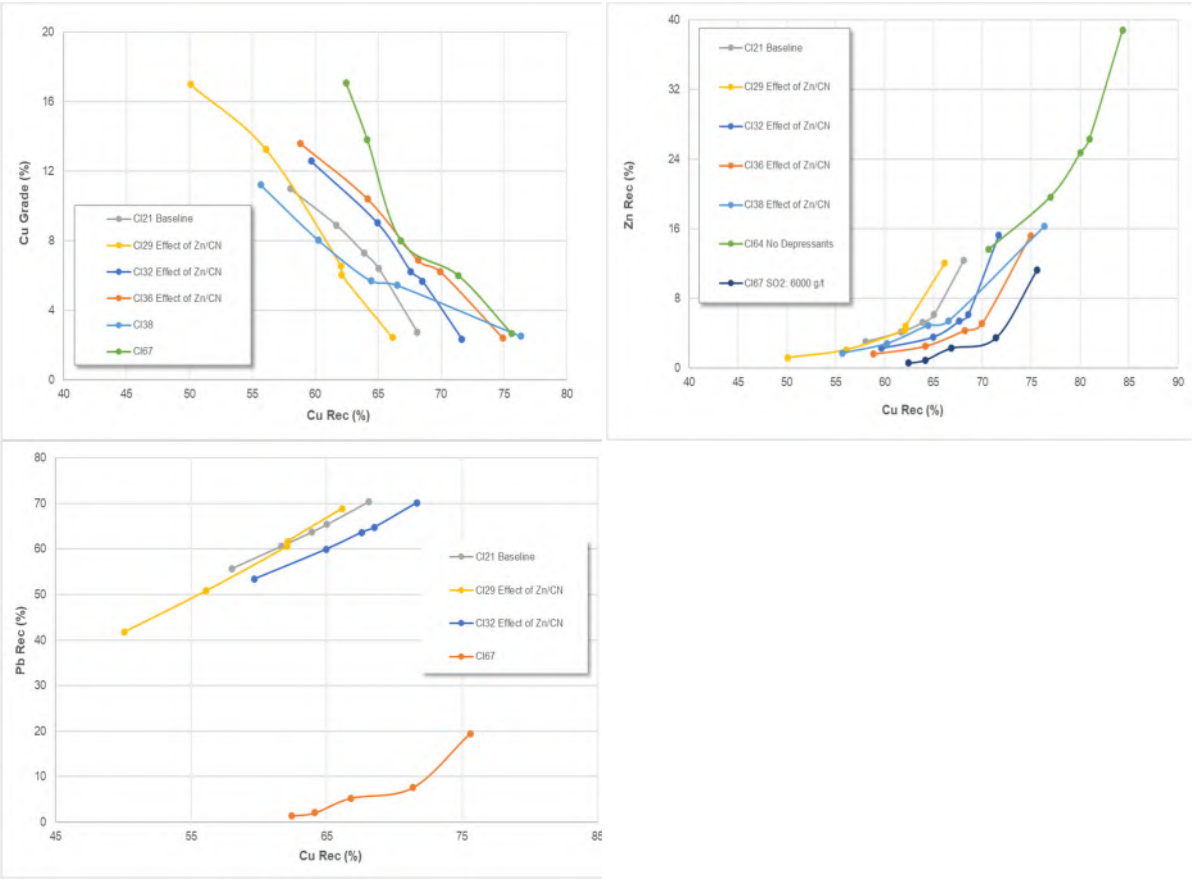


CMC (carboxymethyl cellulose) was also evaluated positively in the Z2 rougher for depression of non-sulphide gangue and was subsequently evaluated with additions moved to the cleaner circuit.

13.6.3 Cleaner Flotation Testing

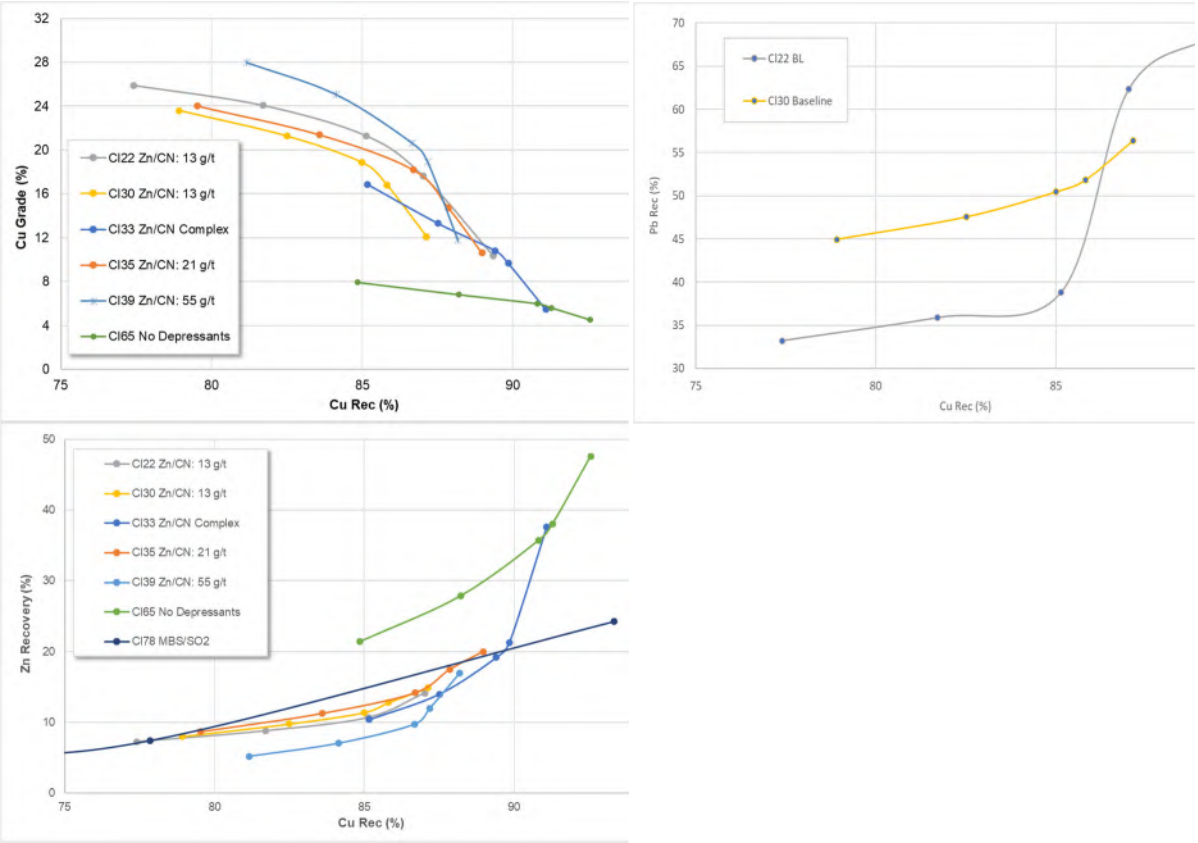
Open cleaner tests were conducted following the rougher tests. Results for the three composites are summarized graphically in Figure 13.5, Figure 13.6, and Figure 13.7. Each figure shows the copper grade vs recovery curves, the zinc recovery vs copper recovery curves, and the lead recovery vs copper recovery curves. Open cleaner testing of Z2 composite focused on optimization of cleaner collectors and dosages, depressant dosages and regrind size targets. Test Cl67 was the initial test with the SO_2/SMBS system, with improved zinc and more notably galena depression as shown in Figure 13.7.

Figure 13-11: Open Cleaner Test Results for Zone 2 Composite



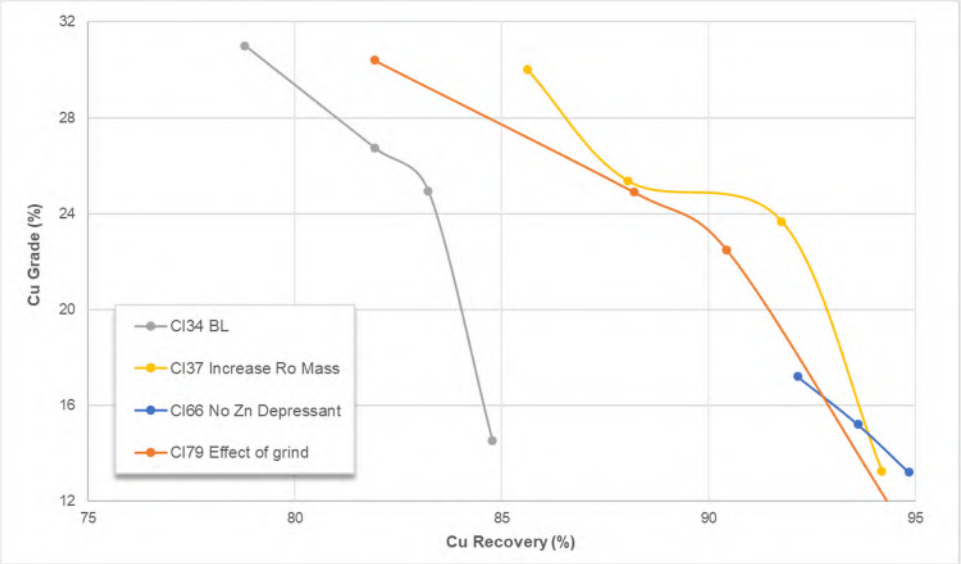
Open cleaner testing of the UWZ composite focussed on the optimization of depressant dosages and finer zinc regrind targets and also investigated the separation of lead from copper following the initial bulk copper-lead flotation step.

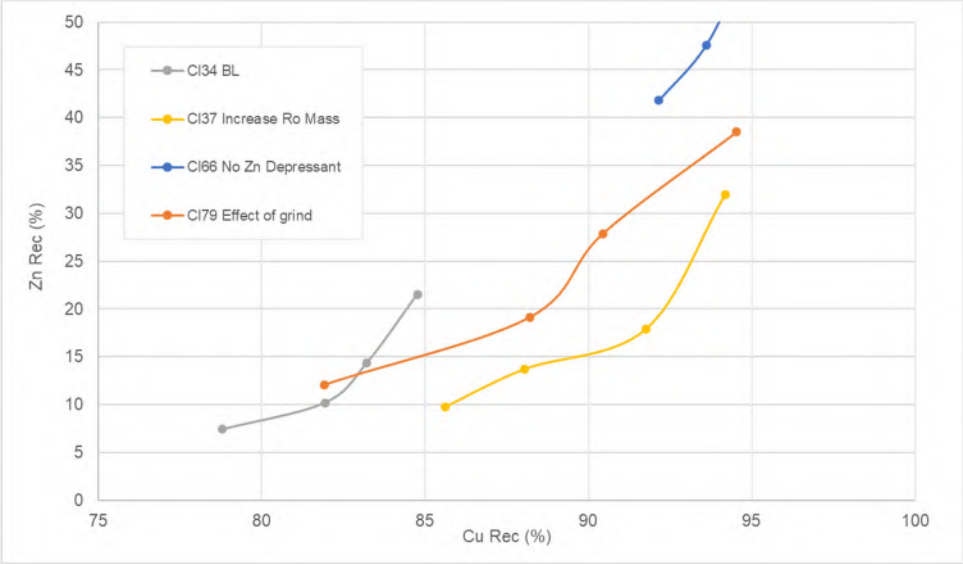
Figure 13-12: Open Cleaner Test Results for UWZ Composite



Open cleaner testing of CSZ attempted to optimize depressant dosages and the copper concentrate regrind size target.

Figure 13-13: Open Cleaner Test Results for CSZ Composite

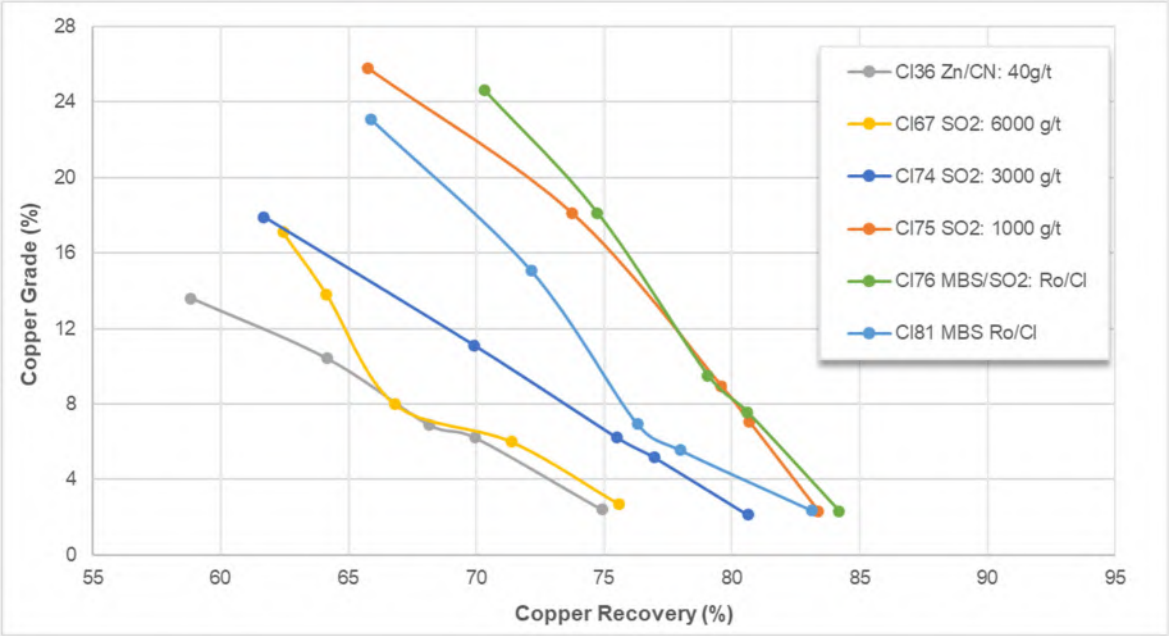


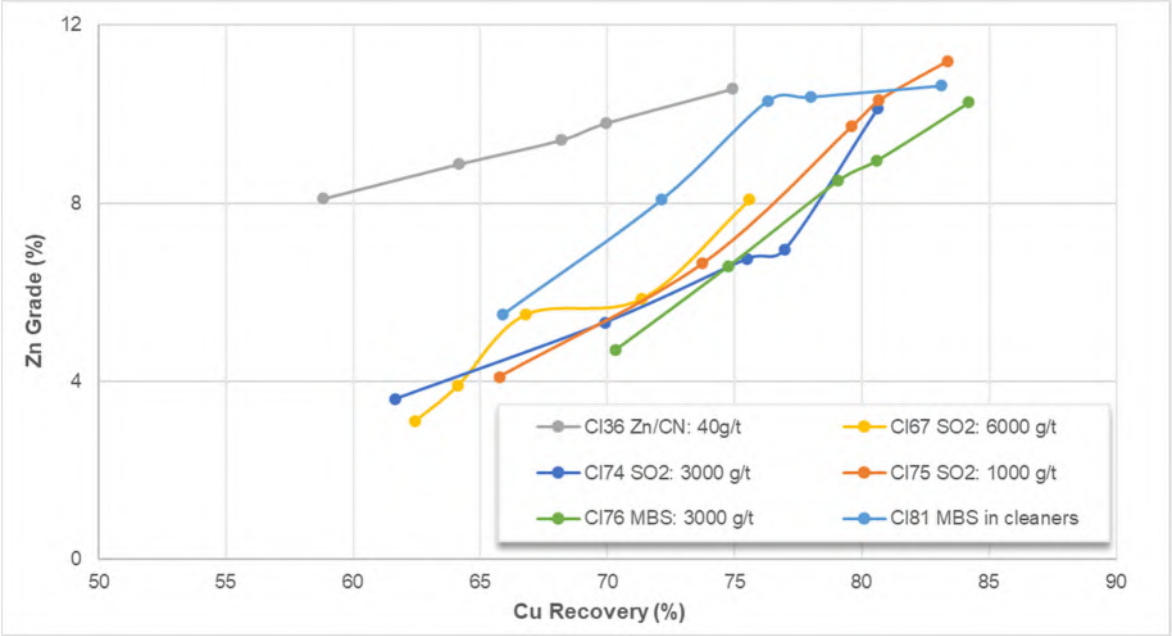


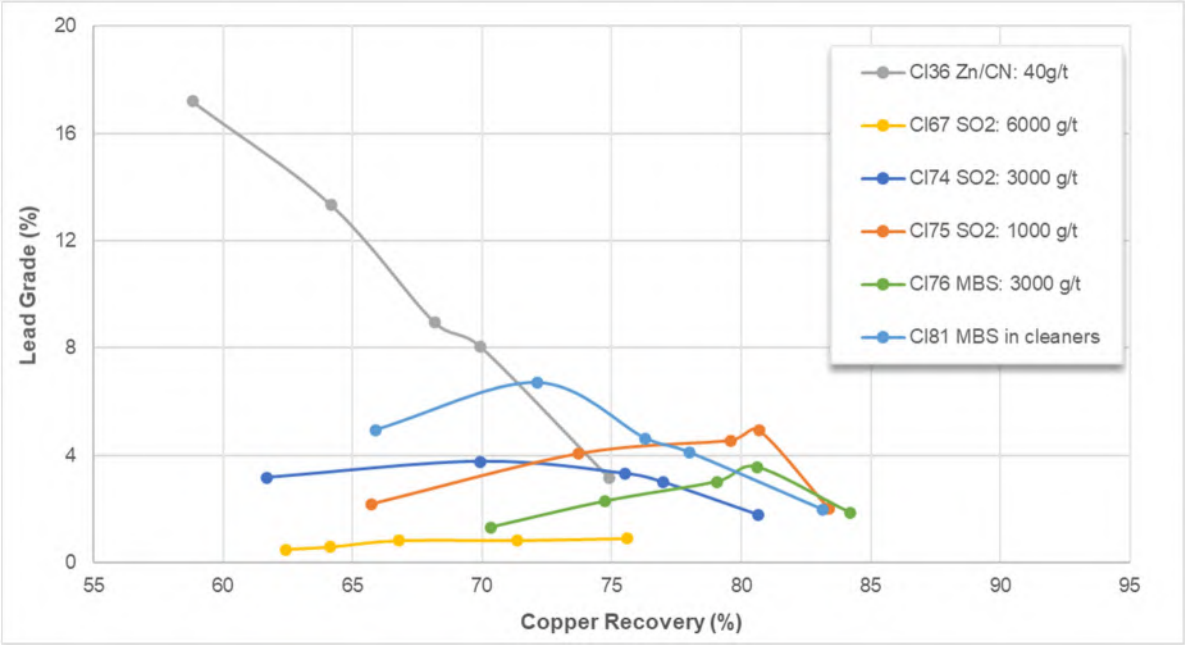
Alternative Depressants

A series of tests were run using gaseous sulphur dioxide and sodium metabisulphite as an alternative reagent scheme to zinc sulphate/cyanide in the event cyanide usage at site was not permissible. The selectivity of copper against pyrite, sphalerite and particularly was improved significantly when using either chemical in the rougher and cleaner circuits, as shown in Figure 13.8.

Figure 13-14: Open Cleaner Test Results for Z2 Composite- Cyanide Alternative (3 charts)





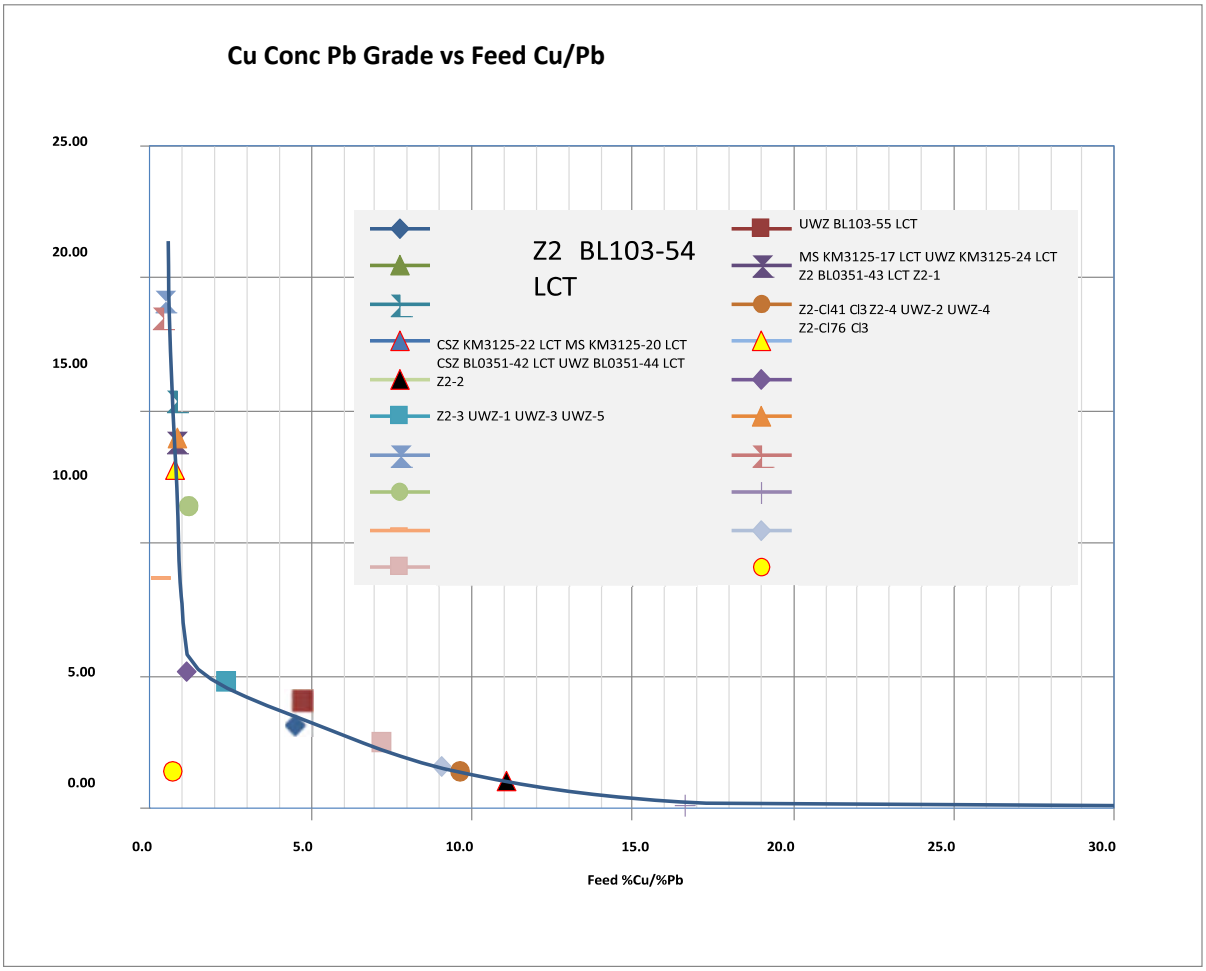


The presence of galena in the feed, particularly as measured with respect to the Cu/Pb feed ratio, influenced the copper performance in the initial separations. Figure 13.9 showed the results of copper concentrate lead content versus the feed Cu/Pb ratios from various open cleaner and locked cycle tests. A strong inflection point in the relationship occurred at around feed Cu/Pb ratio of 1.0-1.5, below which lead content in the copper concentrate increased dramatically with decreasing feed Cu/Pb. In an effort to improve the chalcopyrite-galena selectivity and reduce the unwanted galena flotation, various alternate reagent schemes were tested. Of these, SO_2 and SMBS achieved the best copper-lead

selectivity's. The test results for Z2-Cl76 (see Figure 13.8) demonstrated the effectiveness of the galena depression using SMBS.

Although SO_2 performed slightly superior to SMBS, given the relative ease of implementing an addition system in a concentrator, powdered SMBS would be much simpler and more cost effective to dose versus a gaseous SO_2 system, particularly given only sporadic additions should be required. The relationship in Figure 13.9 suggested SMBS additions would only be required at feed Cu/Pb ratios less than 2 if maximum allowable lead content in the copper concentrate were 5% Pb, for example.

Figure 13-15: Copper Concentrate Lead Content versus Feed Cu/Pb Ratio



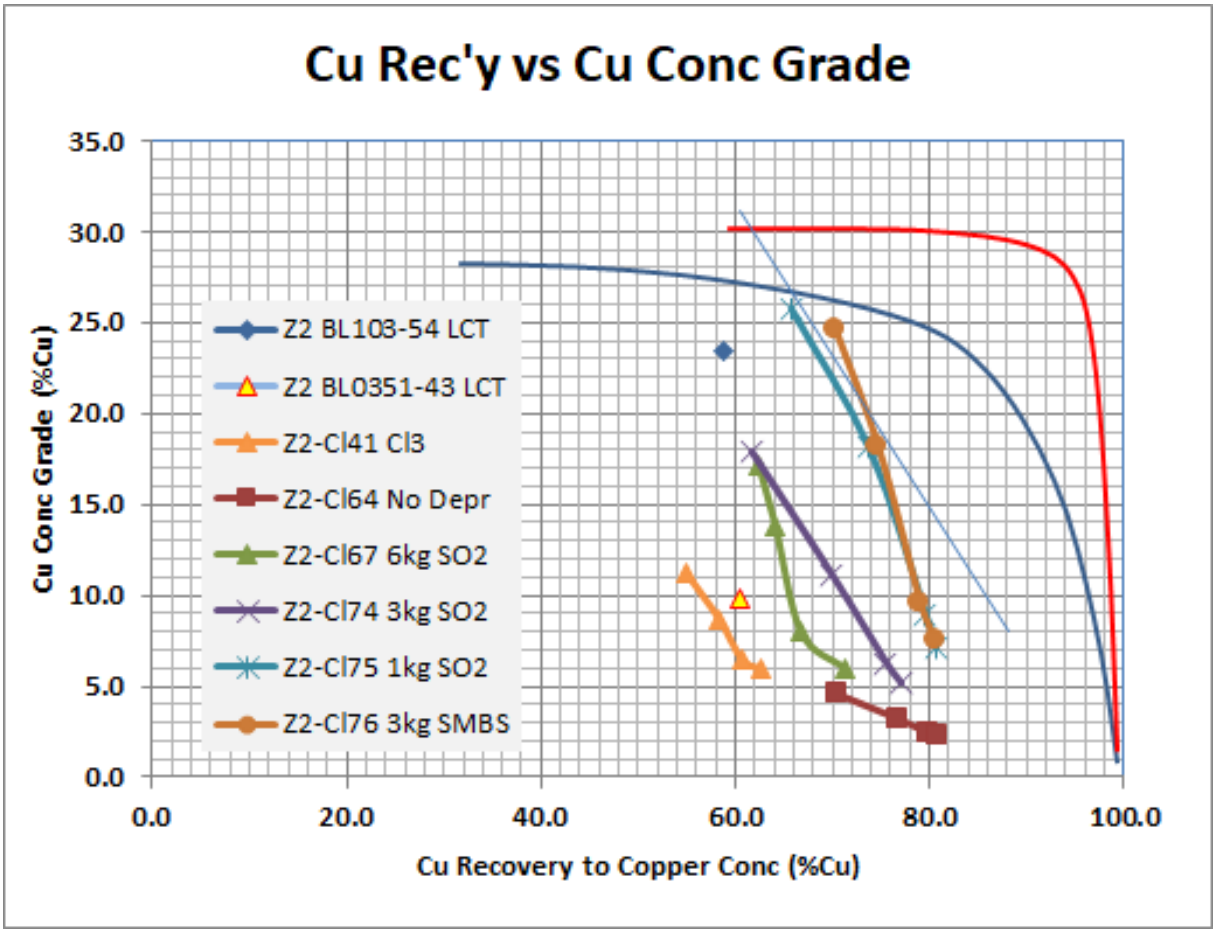
13.6.4 *Variability Cleaner Tests*

Following the development of the optimized grinding and flotation conditions for each of the composites, a series of open circuit cleaner tests were conducted on a set of smaller variability composites taken from each ore type sample set. Head Assays for the variability composites are given in Table 13-5.

The variability test samples were selected to give a reasonably wide range of feed grades and metal ratios from a variety of drill holes. With a wide-ranging set of results in hand, a methodology was developed to help bring the results to some form of comparative basis. Results as shown in Figure 13-16 (Zone 2 results) are given as an example; with the Z2 master composite grade-recovery results used as a guide, an "average" linear slope was chosen for the target grade recovery point on the developed curves. This estimated line was fit as close as possible to the results from each variability test, with the line at similar slope either interpolated or extrapolated to a constant grade for each test, thus allowing recoveries for each test to be compared at the same concentrate grade versus feed grade or feed grade metal ratio.

The data set developed from the variability tests was used to assist in fitting of curves for the metallurgical projections developed in section 13.6.6.

Figure 13-16: Copper Concentrate Grade-Recovery, Variability Test Results



13.6.5 Locked Cycle Testing

One locked cycle test was completed on the UWZ composite (LCT44), two on the Z2 composite (LCT43 and LCT80) and two on the CSZ composite (LCT40 and LCT42). The results of these tests are summarized in Table 13.31, Table 13.32, and Table 13.33.

The 2019 locked cycle flowsheet is illustrated in Figure 13-17 below. The Zinc circuit was not utilized for testing the CSZ composite due to low zinc head grades.

Figure 13-17: Locked Cycle Flowsheet

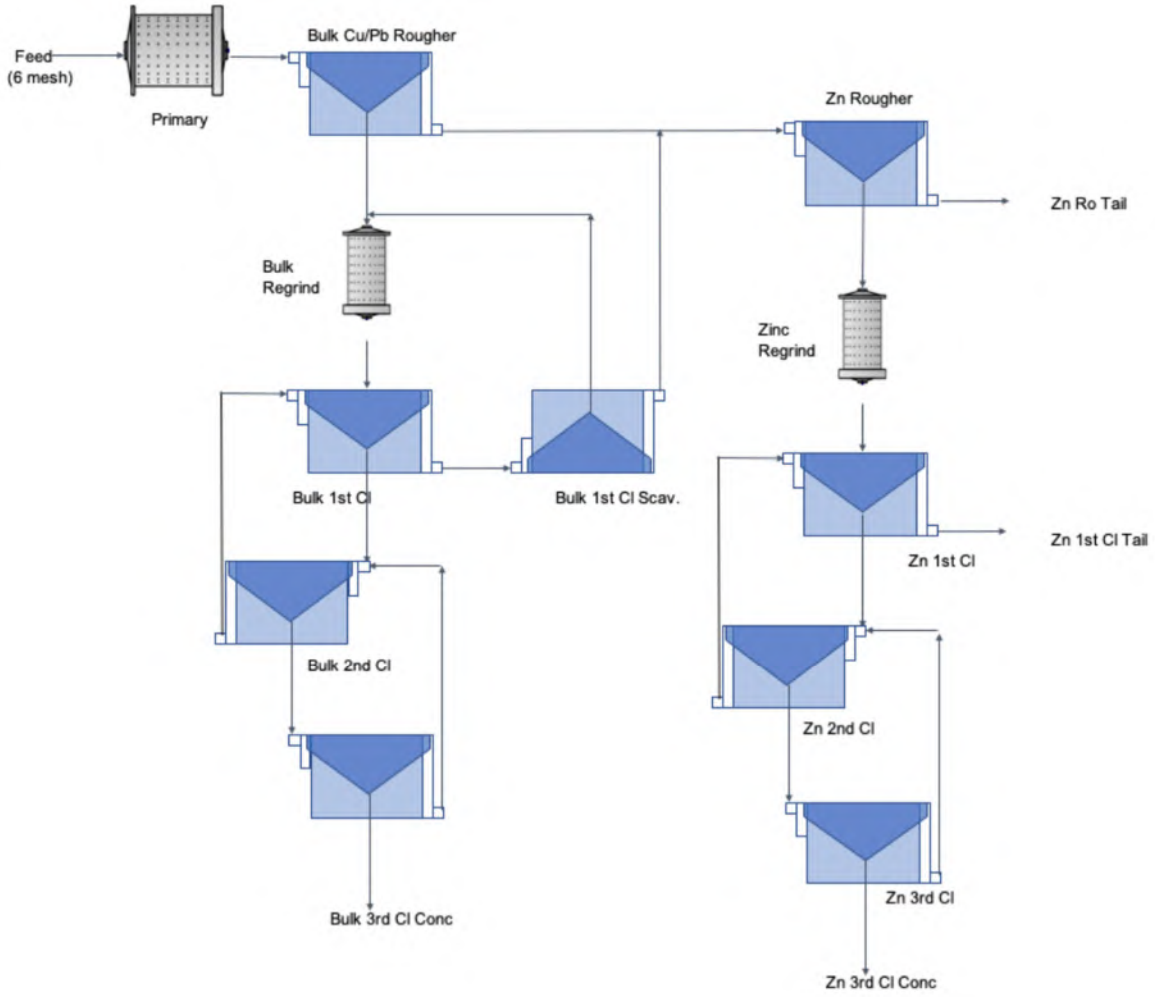


Table 13-26: Locked Cycle Test Results for UWZ Composite

Test	Product	Wt	Assay - percent or g/t						Distribution - percent					
		%	Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
LCT44	Cu/Pb Cl Conc	7.9	20.5	1.06	6.84	24.7	206	10.0	88.4	51.0	13.6	9.3	60.2	73.9
	Cu/Pb Ro Conc	**10.7**	**15.3**	**0.84**	**6.17**	**24.1**	**158**	**7.57**	**89.4**	**54.6**	**16.6**	**12.2**	**62.4**	**75.6**
	Zn Cl Conc	6.7	0.97	0.24	45.1	12.5	38	0.66	3.6	9.9	77.1	4.0	9.6	4.2
	Zn 1st Cl Tail	16.1	0.30	0.12	0.82	31.6	13	0.32	2.6	12.1	3.3	24.2	7.6	4.9
	Zn Ro Conc	**22.8**	**0.50**	**0.16**	**13.9**	**25.9**	**20**	**0.42**	**6.2**	**22.0**	**80.4**	**28.3**	**17.2**	**9.0**
	Zn Ro Tail	69.3	0.14	0.06	0.34	18.9	9	0.26	5.4	27.0	6.0	62.4	22.7	17.1
	Feed (calc.)	111	1.88	0.17	4.01	21.4	28	1.14	100	100	100	100	100	100

Table 13-27: Locked Cycle Test Results for Z2 Composite

Test	Product	Wt	Assay - percent or g/t						Distribution - percent					
		%	Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
LCT43	Cu/Pb Cl Conc	2.0	9.77	12.8	7.81	14.5	334	5.21	60.5	56.7	2.3	1.4	34.2	31.1
	Zn Cl Conc	10.2	0.72	0.54	52.7	8.85	41	0.31	23.2	12.5	80.9	4.5	21.9	9.7
	Zn 1st Cl Tail	13.5	0.10	0.18	4.67	28.3	15	0.28	4.2	5.4	9.5	18.9	10.6	11.7
	Zn Ro Conc	**23.7**	**0.36**	**0.33**	**25.3**	**19.9**	**26**	**0.30**	**27.4**	**17.9**	**90.4**	**23.3**	**32.5**	**21.4**
	Zn Ro Tail	74.4	0.05	0.15	0.65	20.4	9	0.21	12.2	25.4	7.3	75.2	33.2	47.5
	Feed (calc.)	100	0.32	0.44	6.57	19.9	19	0.29	100	100	100	100	100	100
LCT80														
	Cu/Pb Cl Conc	2.0	12.5	6.17	6.03	17.6	265	5.13	79.6	30.6	1.9	1.8	31.8	45.1
	Cu/Pb Ro Conc	**4.3**	**6.02**	**3.71**	**3.85**	**17.2**	**141**	**2.57**	**80.9**	**39.0**	**2.6**	**3.7**	**35.8**	**47.7**
	Zn Cl Conc	10.7	0.23	1.42	49.7	8.61	46	0.28	7.8	37.4	84.2	4.6	29.2	12.8
	Zn 1st Cl Tail	12.0	0.07	0.34	3.69	26.8	15	0.16	2.7	10.0	7.0	16.3	10.9	8.5
	Zn Ro Conc	**22.7**	**0.15**	**0.85**	**25.4**	**18.2**	**30**	**0.22**	**10.5**	**47.5**	**91.2**	**20.9**	**40.2**	**21.3**
	Zn Ro Tail	75.2	0.04	0.12	0.58	20.3	6	0.10	9.9	22.0	6.9	77.3	28.0	33.6
	Feed (calc.)	100	0.32	0.40	6.09	20.0	17	0.22	100	100	100	100	100	100

Table 13-28: Locked Cycle Test Results for CSZ Composite

Test	Product	Wt	Assay - percent or g/t						Distribution - percent					
		%	Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
LCT40														
	Cu 2nd Cl Conc	3.9	27.8	0.19	0.93	27.0	160	8.96	80.6	25.6	11.7	14.3	68.4	76.3
	Cu 1st Cl Tail	6.2	0.53	0.05	0.71	10.6	8.0	0.27	2.4	10.9	14.3	8.9	5.4	3.6
	Cu Ro Conc	**10.1**	**11.1**	**0.10**	**0.80**	**17.0**	**67**	**3.63**	**83.1**	**36.5**	**26.0**	**23.3**	**73.8**	**79.9**
	Cu Ro Tail	89.9	0.26	0.02	0.26	6.30	3	0.10	16.9	63.5	74.0	76.7	26.2	20.1
	Feed (calc.)	100	1.36	0.03	0.31	7.41	9.4	0.48	100	100	100	100	100	100
LCT42														
	Cu 2nd Cl Conc	4.6	26.7	0.18	1.26	26.9	129	8.31	93.8	23.5	18.7	17.1	71.0	82.6
	Cu 1st Cl Tail	7.1	0.54	0.06	0.86	11.1	9.0	0.24	2.9	11.5	19.6	10.9	7.7	3.7
	Cu Ro Conc	**11.6**	**10.8**	**0.10**	**1.01**	**17.3**	**56**	**3.40**	**96.7**	**35.0**	**38.3**	**28.0**	**78.7**	**86.2**
	Cu Ro Tail	88.4	0.05	0.03	0.22	5.85	2	0.07	3.3	65.0	61.7	72.0	21.3	13.8
	Feed (calc.)	100	1.32	0.03	0.31	7.21	8.4	0.47	100	100	100	100	100	100

For the Z2 composite, LCT80 was a repeat of LCT43 but with a variation on the zinc flowsheet whereby a small portion of the initial higher-grade rougher concentrate was bypassed to the 1st cleaner directly without any regrinding. This was an effort to assess if the initial high-grade portion of the rougher concentrate could be cleaned effectively, thereby enabling coarse sphalerite to be cleaned to final concentrate grade without imposing unnecessary regrinding on already liberated material available from the primary grinding stage.

For the CSZ composite, LCT42 was a repeat of LCT40, but with a finer grind of p80 of 75 µm in LCT42 versus 125 µm in LCT40.

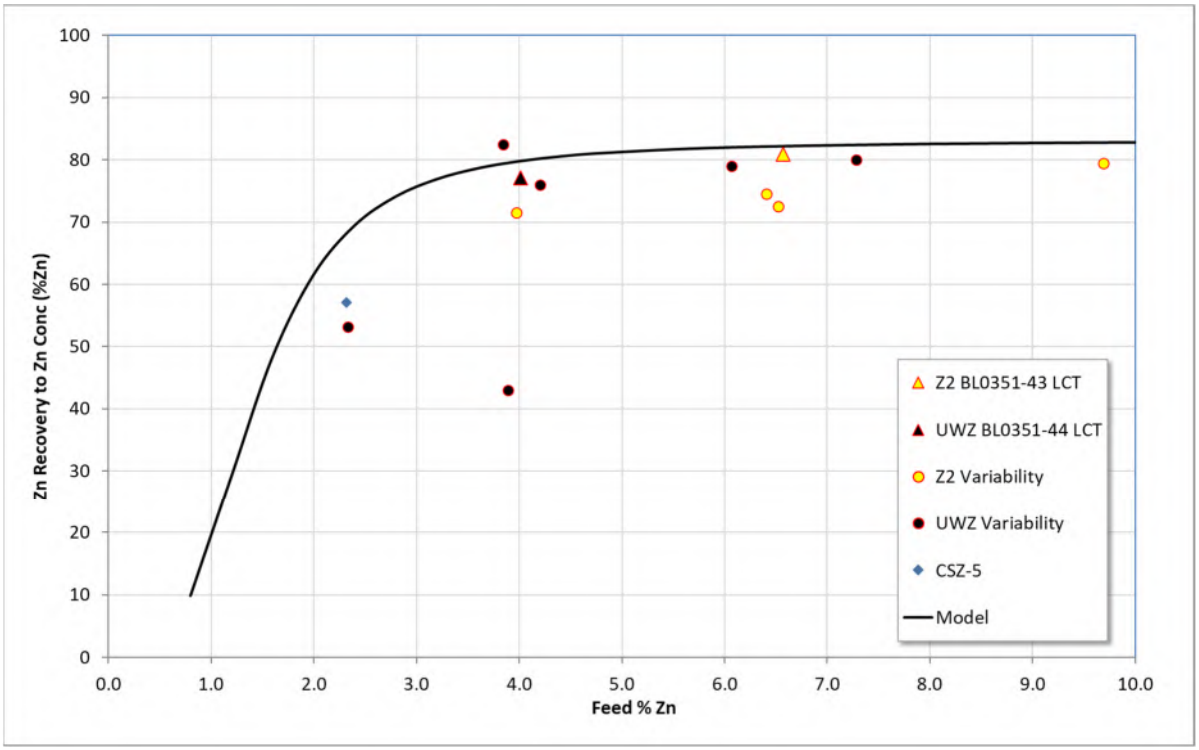
13.7 Recovery and Concentrate Grade Projections

Open circuit cleaner and locked cycle flotation test results from the three metallurgical studies described herein were used to develop feed grade-based models for copper, zinc, silver and gold concentrate grade and recovery.

In cases of stage recovery data for open cleaner tests, curves were parsed for recovery at a constant concentrate grade (in some instances the grade-recovery curve required extrapolation to derive values). The predictive values from the charts were then curve-fit to derive mathematical functions for use in PFS block models and/or financial models.

The following charts summarize the various performance projections. Figure 13-18 illustrates the zinc recovery vs zinc feed grade relationship.

Figure 13-18: Zinc Recovery for Massive Sulphides versus Zinc in Feed



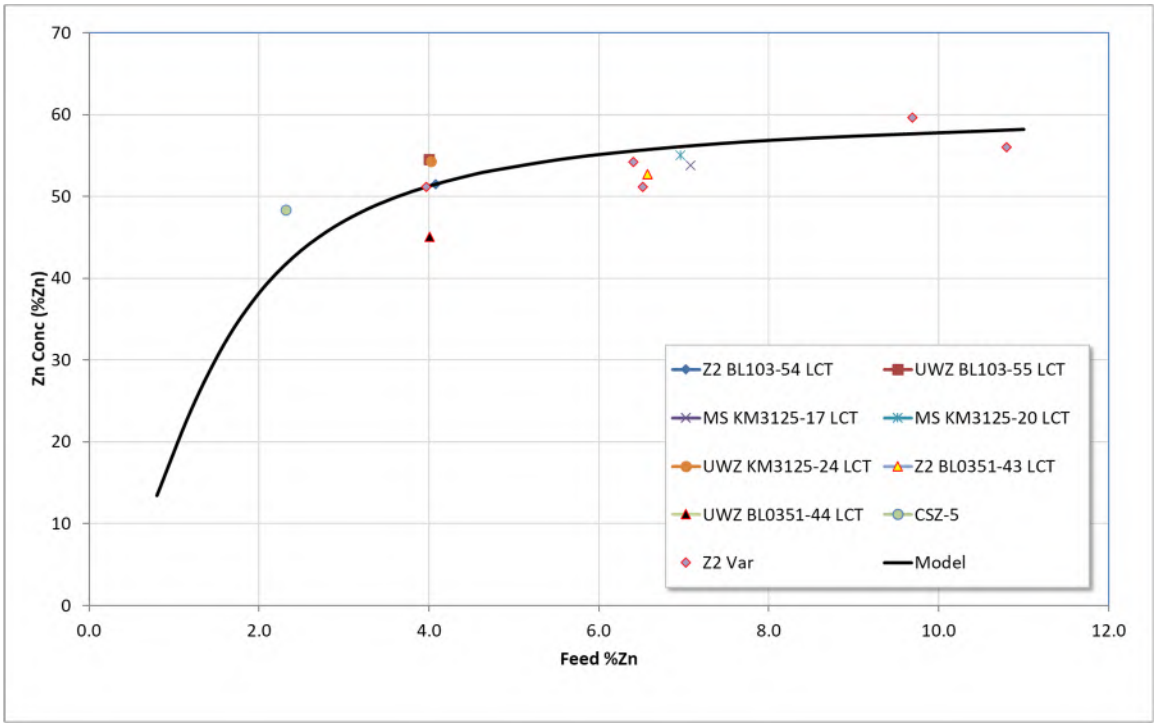
The zinc recovery vs head grade relationship is mathematically defined as:

$$\boldsymbol{Zn\ Recovery}(\%) = \frac{\mathbf{82.0} \times \boldsymbol{F}^{3.26} - \mathbf{3.745}}{\boldsymbol{F}^{3.255} + \mathbf{3.15}}$$

Where F is the zinc grade (%) in mill feed

Figure 13-19 describes the zinc concentrate grade (%) as a function of the % zinc in mill feed.

Figure 13-19: Zinc Concentrate Grade for Massive Sulphides versus Zinc in Feed



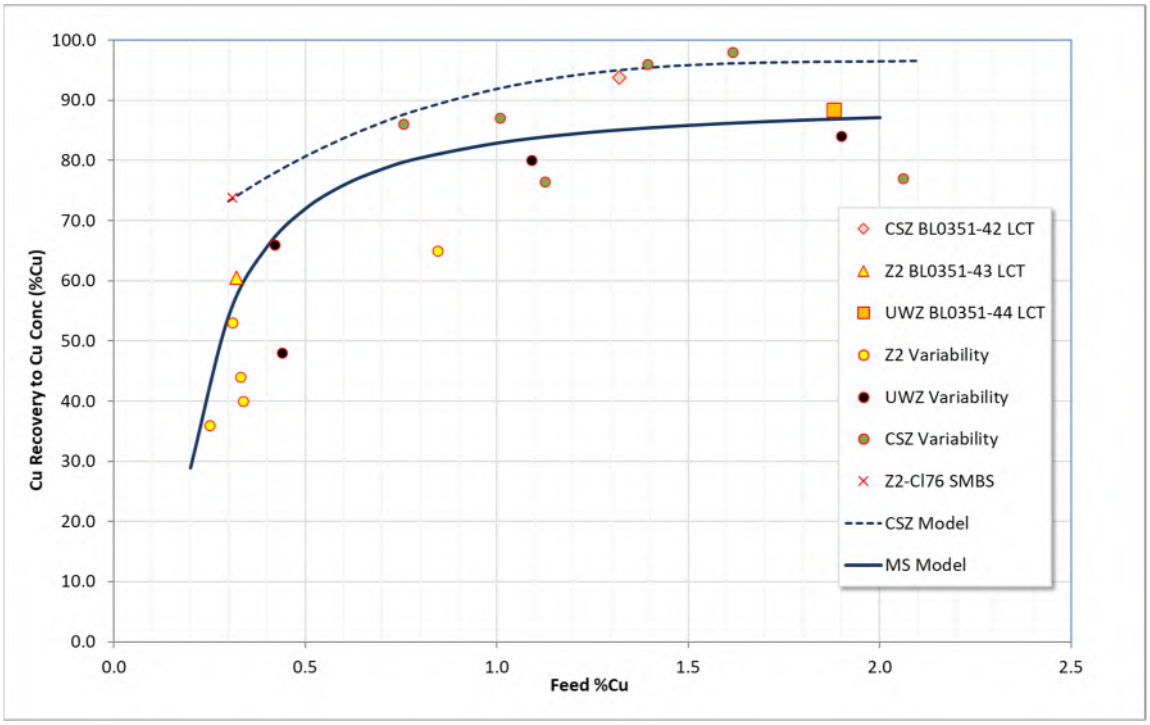
The zinc concentrate grade vs head grade relationship is mathematically defined as:

$$\boldsymbol{Zn\ Concentrate\ Grade}\ (\%) = \frac{\mathbf{55.0} \times \boldsymbol{F}^{\mathbf{2}} - \mathbf{2.65}}{\boldsymbol{F}^{\mathbf{1.97}} + \mathbf{1.78}}$$

Where F is the zinc grade (%) in mill feed

Figure 13-20 models % copper recovery as a function of % copper in feed for the different ore types, with the solid curves indicating the function for the MS and the dashed curve indicating the function for the CSZ.

Figure 13-20: Copper Recoveries for Massive Sulphides and Copper Stockworks versus Copper in Feed



The CSZ Cu recovery vs head grade relationship is mathematically defined as:

$$\boldsymbol{CSZ\ Cu\ Recovery}(\%) = (\mathbf{5.3} \times \boldsymbol{F}^{\mathbf{3.0}}) - (\mathbf{30.5} \times \boldsymbol{F}^{\mathbf{2}}) + (\mathbf{59.0} \times \boldsymbol{F}) + \mathbf{58.1}$$

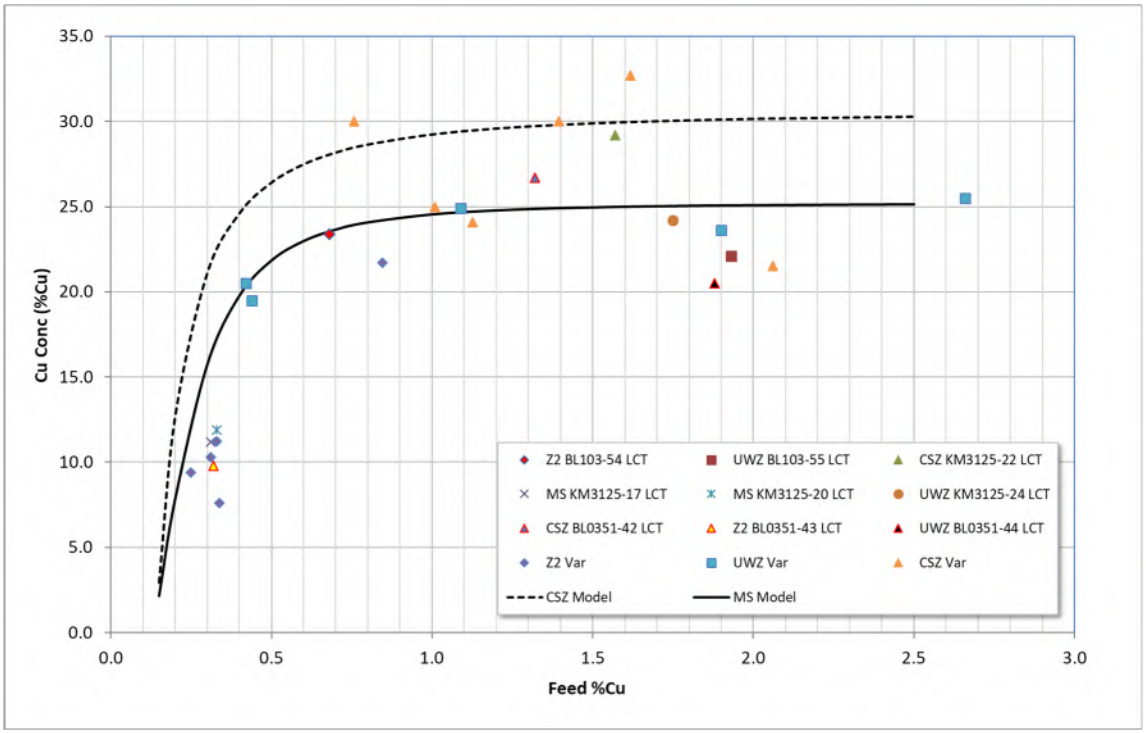
and for the Massive sulphide:

$$\boldsymbol{MS\ Cu\ Recovery}(\%) = \frac{\mathbf{89.3} \times \boldsymbol{F}^{\mathbf{1.433}} - \mathbf{5.76}}{\mathbf{0.0085} \times \boldsymbol{F}^{\mathbf{1.43}}}$$

Where F is the Cu grade (%) in mill feed

Figure 13-21 models copper concentrate % grade as a function of % copper in feed - with the solid curve illustrating the function for the MS, and the dashed curve illustrating the function for the CSZ.

Figure 13-21: Copper Conc. Grades for Massive Sulphides and Copper Stockworks versus Copper in Feed



The Copper concentrate grade vs head grade relationships are mathematically defined as:

$$\boldsymbol{CSZ\ Cu\ Concentrate\ Grade}\ (\%) = \frac{\mathbf{30.576} \times \boldsymbol{F}^{\mathbf{1.653}} - \mathbf{1.187}}{\boldsymbol{F}^{\mathbf{1.65}} + \mathbf{0.005}}$$

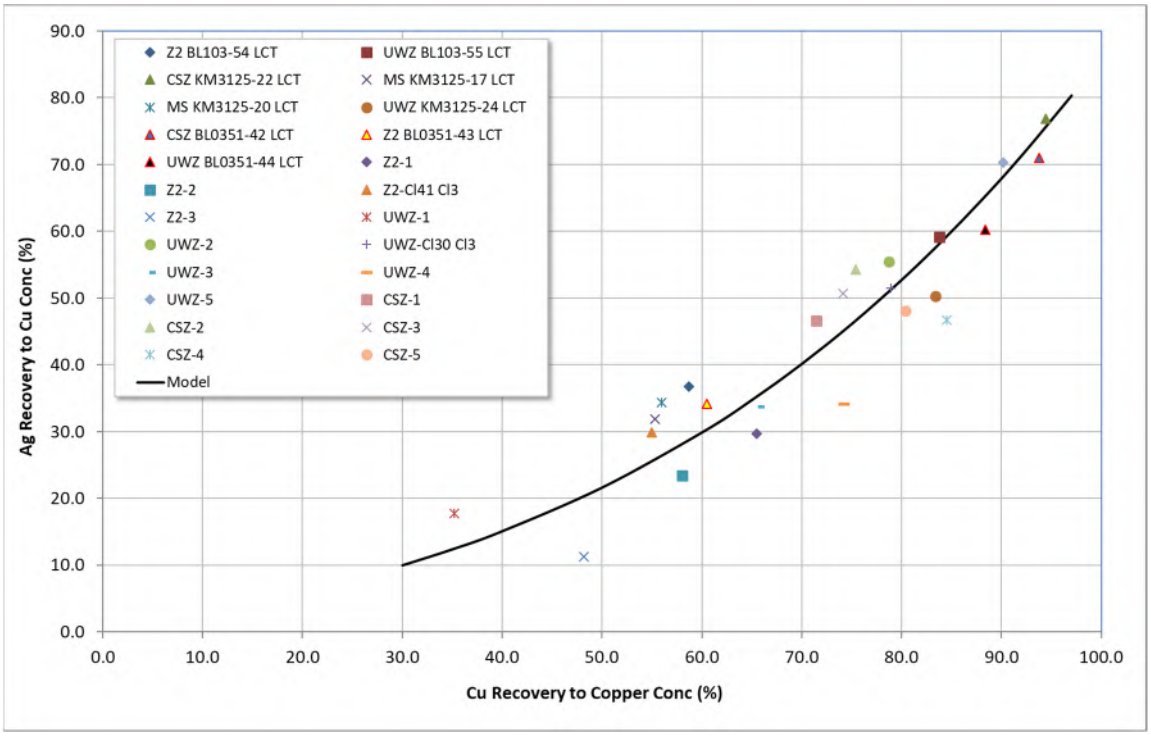
and for the Massive sulphide:

$$\boldsymbol{MS\ Cu\ Concentrate\ Grade}\ (\%) = \frac{\mathbf{25.2} \times \boldsymbol{F}^{\mathbf{2.47}} - \mathbf{0.17}}{\boldsymbol{F}^{\mathbf{2.47}} + \mathbf{0.02}}$$

Where F is the copper grade (%) in mill feed

Recoveries of silver and gold to the copper concentrate as functions of the copper recovery are summarized in Figure 13-22 and Figure 13-23

Figure 13-22: Silver Recovery to Copper Concentrate as a Function of Copper Recovery

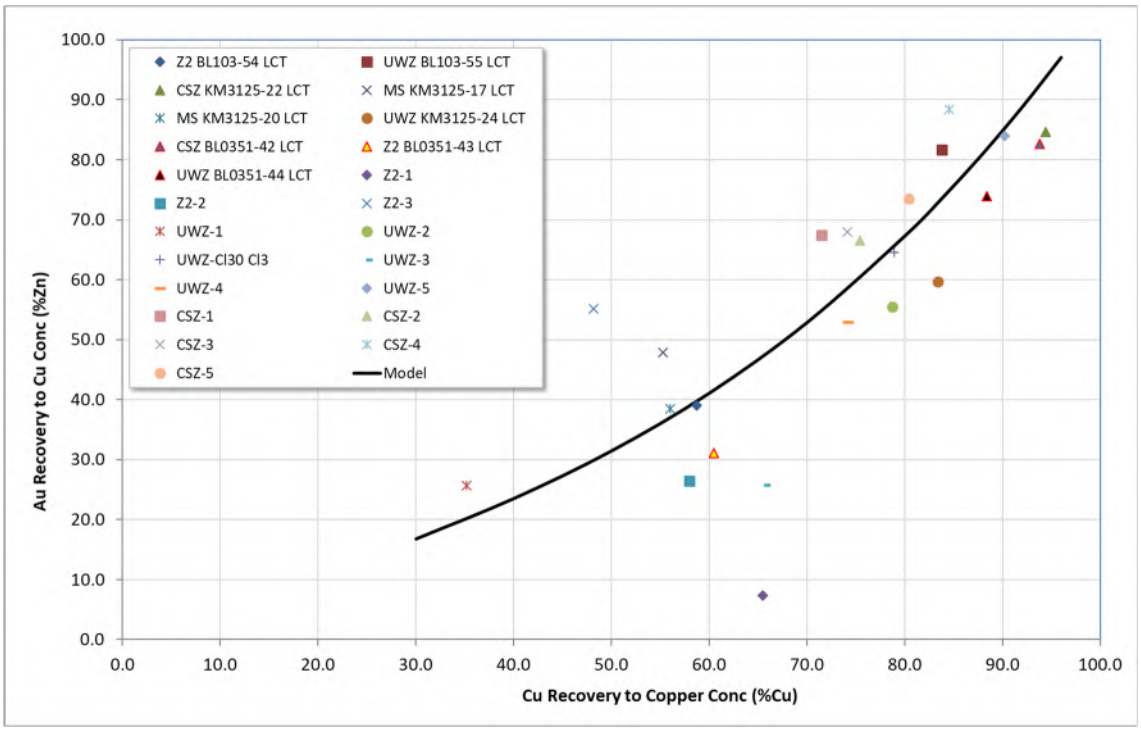


The silver recovery to copper concentrate vs copper recovery relationship is mathematically defined as:

$$\boldsymbol{Ag\ Recovery}\ (\%) = (\mathbf{0.00005} \times \boldsymbol{C}^{\mathbf{3}}) + (\mathbf{0.0011} \times \boldsymbol{C}^{\mathbf{2}}) + (\mathbf{0.25} \times \boldsymbol{C}) + \mathbf{0.16}$$

Where C is the copper recovery to copper concentrate.

Figure 13-23: Gold Recovery to Copper Concentrate as a Function of Copper Recovery



The gold recovery to copper concentrate vs copper recovery relationship is mathematically defined as:

$$\boldsymbol{Au\ Recovery}\ (\%) = (\mathbf{0.00008} \times \boldsymbol{C}^{\mathbf{3}}) + (\mathbf{0.0036} \times \boldsymbol{C}^{\mathbf{2}}) + (\mathbf{0.63} \times \boldsymbol{C}) - \mathbf{1.023}$$

Where C is the copper recovery to copper concentrate.

Note: Silver recovery to copper concentrate appeared to be very similar for both massive sulphides and copper stockwork materials.

13.8 Thickening and Filtration Tests

Settling tests were conducted on the BL0103 final tailings from locked cycle tests on the two main composites Z2 (test 54) and UWZ (test 55). A series of flocculants were tested, along with zero flocculant, with Magnafloc 350 deemed to be the most effective. Both composites tested comparatively, with settling rates between 0.23 and 0.39m/min with an average compacting density of 49-50% solids.

A program of static and dynamic settling work was also completed on CSZ and MS concentrate and tailing samples as part of the BL0351 program. Static testing showed that a number of flocculants were effective at dewatering the samples, with 52-68% solids being typical final densities. The addition of lime as a coagulant was seen to be helpful in certain instances.

Table 13-29: Static Settling Test Results

Sample	Flocculant		pH	Lime (g/t)	Density (%)		Free Settling Velo. (mm/s)	Clarity	
	Type	g/t			Initial	Final		Initial	Final
T84: Cu/Pb Conc (Blend 1)	MF10	10	11.0	386	14.5	60.4	7.08	Clear	Clear
	MF10	10	7.7 (nat)	0	14.5	60.4	6.34	Clear	Clear
	MF351	10	7.7 (nat)	0	14.5	60.4	7.18	Clear	Clear
	MF336	10	7.7 (nat)	0	14.5	60.4	4.75	Clear	Clear
	MF351	10	7.7 (nat)	0	13.7	62.1	7.63	Clear	Clear
T84: Zn Conc (Blend 1)	MF10	10	11.0	384	13.4	52.3	6.15	Slight Turbid	Clear
	MF10	10	7.7 (nat)	0	13.4	52.3	4.75	Turbid	Slight Turbid
	MF351	10	7.7 (nat)	0	13.4	52.3	6.15	Turbid	Slight Cloud
	MF336	10	7.7 (nat)	0	13.4	52.3	4.19	Turbid	Slight Cloud
	MF351	10	7.7 (nat)	0	13.7	72.6	7.10	Clear	Clear
T85: Final Tail (CSZ Comp)	MF10	10	9.0	315	14.5	60.1	3.46	Clear	Clear
	MF10	20	9.0	311	14.7	55.9	7.51	Clear	Clear
	MF10	40	9.0	312	14.6	54.3	12.8	Clear	Clear
T86: Final Tail (40% Z2 & 60% UWZ)	MF10	10	10.5	167	13.6	68.7	8.99	Clear	Clear
	MF10	20	10.5	167	13.6	64.3	11.8	Clear	Clear
	MF10	30	10.5	167	13.6	64.3	18.7	Clear	Clear

Dynamic settling tests on CSZ and MS tailings samples gave positive results, with underflow densities in the 60-70% range, reasonable overflow clarities (50-100 mg/l TSS) and acceptable loading rates of 12 t/m^2/day. Flocculant (Magnafloc 10) addition rates of 20-30 g/t were required to achieve the stated underflow densities.

13.9 Concentrate Quality

13.9.1 Minor Elements

Minor element ICP-MS scans were completed on the final copper and zinc concentrates produced from the various locked cycle tests. Table 13.29 summarizes the various assays.

Table 13-30: BL0351 Final Copper and Zinc Concentrate Minor Element Assays

Element	Unit	Zone2				UWZ		CSZ
		LCT43		LCT80		LCT44		LCT42
		Cu Conc	Zn Conc	Cu Conc	Zn Conc	Cu Conc	Zn Conc	Cu Conc
Hg	ppm	45.9	139	38.2	140	20.4	78.8	1.78
Al	%	0.22	0.08	0.21	0.09	0.32	0.28	0.54
As	ppm	147	125	87	105	237	306	85
B	ppm	20	30	20	20	20	20	30
Ba	ppm	7	7	7	4	7	6	25
Bi	ppm	163	10	55	20	530	56	290
Ca	%	0.55	0.6	0.3	0.73	0.25	0.73	0.16
Cd	ppm	226	1590	185	1550	246	1510	88
Ce	ppm	11.2	7.1	10.5	7.2	15.2	15	24.1
Co	ppm	9.9	9.8	16.3	10	35	22.5	25.8
Cr	ppm	100	140	300	120	160	220	230
Fe	%	15.4	9.31	20	9.16	27.6	13.3	27.2
Ga	ppm	4.6	2.9	4.2	2.9	4.4	6.2	4.7
In	ppm	12.6	54.7	14.5	52	71.2	70	91.2
La	ppm	5.4	3.5	5	3.6	6.7	7.5	10.5
Li	ppm	11	25	12	5	8	14	8
Mg	%	5.27	0.42	5.33	0.56	1.2	0.7	0.93
Mn	ppm	246	556	225	594	187	747	143
Mo	ppm	37	39	74	14	21	24	43
Ni	ppm	50	90	170	70	90	130	140
Pb	ppm	> 5000	> 5000	> 5000	> 5000	> 5000	1830	1510
Rb	ppm	2.1	1.7	1.9	1.8	1.9	2.3	3.8
S	%	21.3	> 25.0	24	> 25.0	> 25.0	> 25.0	> 25.0
Sb	ppm	385	31	301	41	91	26	19
Se	ppm	569	80.9	208	106	403	190	356
Si	%	8.82	0.45	9.03	0.57	2.24	0.73	4.54
Sn	ppm	126	57.4	129	55.5	144	104	764
Ta	ppm	0.3	0.3	0.3	0.3	0.4	0.4	0.5
Tb	ppm	0.2	< 0.1	0.1	< 0.1	0.1	0.1	0.3
Th	ppm	1.2	0.2	0.5	0.2	0.5	0.5	1.3
Tl	ppm	2.9	0.9	2.1	1	2.8	1.4	0.9
U	ppm	4.5	0.7	1.5	0.6	0.9	0.9	1
W	ppm	2.3	2	2.1	1.7	2.1	2.7	4.2
Y	ppm	3	1.3	2.9	1.2	3.5	3.2	7.3

Similar to previous studies, mercury content in zinc concentrates was elevated and could incur penalties at certain smelters. In BL0351, mercury ranged from 80 to 140ppm in the zinc concentrates, and from 20 to 46ppm in the massive sulphide copper concentrates. Overall, mercury in the copper concentrate will be blended down significantly with CSZ copper concentrate which was low (~2ppm) in mercury.

Iron content in zinc concentrate may incur penalties at around 9-10%. Iron penalties can be minimized by maintaining zinc concentrate grades around 54% Zn.

Selenium was elevated in copper concentrates for all ore types (190 to 570ppm).

Magnesium and silica were both somewhat elevated in the massive sulphide copper concentrates (approximately 5% Mg and 9% Si) but again, these will be lowered significantly with blending of copper concentrate from CSZ (0.7% Mg and 0.7% Si). The ratio of Si:Mg was quite similar to that found in talc and one could surmise that talc recovery is mainly responsible for the concentration of these elements.

Note that by installing additional residence time in the cleaner flotation circuit, these cells can be operated at lower pulp densities which will tend to reduce non-sulphide gangue entrainment. Further investigations into talc depressants and dispersants should be undertaken.

13.9.2 Concentrate Self Heating

A material Self Heating Evaluation (MASH) evaluation was completed on two composite samples of final concentrate (Cu/Pb Conc and Zn Conc). The samples were created from locked cycle test products for the different ore types.

Composite assays are summarized in Table 13-31

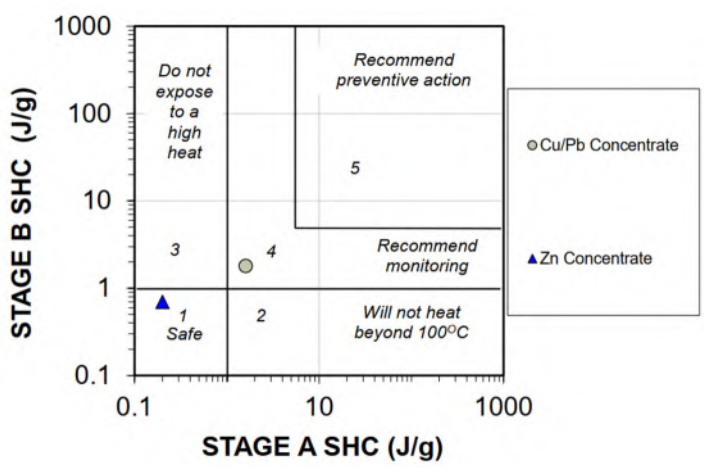
Table 13-31: MASH Test Head Assays

	Assay					
	% Cu	% Pb	%Zn	% Fe	g/t Ag	g/t Au
Cu/Pb 3rd Cleaner Conc	26.8	2.80	3.00	27.7	186	10
Zn 3rd Cleaner Conc	0.86	0.32	49.8	11.5	36.0	0.67

The samples were tested using standard protocol FR-2, which simulates self heating in stages, namely Stage A (70°C) and Stage B (140°C). Results, summarized in Figure 13-24, show that the copper concentrate is deemed moderately reactive (risk region 4) and the zinc concentrate is deemed to be safe (risk region 1) with no risk of self heating.

Further testwork is recommended as the project develops through feasibility level.

Figure 13-24: Self Heating Test Results – Copper and Zinc Concentrates



13.10 Dense Medium Separation

Dense medium separation (DMS) was proposed as a possible method of preconcentrating McIlvenna Bay mineralization prior to grinding. Thus, in the BL0103 testwork program, heavy liquid separations were performed on Z2 and UWZ composites to determine amenability to coarse gravity separations. Samples were crushed and screened into three size fractions (>3/4", <3/4">1/2", <1/2>1/4") and the fractions evaluated. The heavy liquid separations were conducted sequentially at 3.02 SG and the "float" particles subsequently split again at 2.83 SG.

For UWZ, greater than 95% of the zinc was retained in the sink at 2.83 SG with rejection of nearly half the mass. Copper performance lagged that of zinc significantly, with only about 85% of the copper reporting to the sink fraction. The performance of Z2 was less favourable under similar conditions. At 2.83 SG, only about one third of the mass was rejected to the float, with 97% of the zinc retained in the sink, but with only about 65% of the copper in the sink fraction.

The BL0103 work showed some promise, and it was recommended that testing continue in future programs. In addition, visual examination of drill core by metallurgical personnel continued to indicate the potential for DMS amenability – especially in the CSZ core samples. For this reason, heavy media separation testing of CSZ drill core was investigated as part of the BL0351 program – firstly using a laboratory scale ferro-silicon media based separator (Base Met Labs), then secondly using baths of organic heavy liquid (SGS Minerals, Lakefield) to provide more accurate separations. A sample of MS drill core was subsequently tested at SGS also. Samples tested included mineralization from a single hole, selected to be typical of the mineralization style, and with reasonable quantities of waste material at both ends of the hole ("shoulders") to represent mining dilution.

The initial methodology for DMS testwork was not satisfactory to Foran. The sulphide textures observed in CSZ drill core were noted to be relatively fine, so the selected crushing top sized (¾") was thought to be too coarse to achieve reasonable liberation. In addition, the vessel used for the heavy media separation was rather small (140mm diameter by 450mm high) and recirculation of medium at 7.5 Lpm resulted in high rise rates withing the vessel. The velocity and density of the rising medium is thought to have been sufficient to misplace "near density" particles from the finer size fractions. These two issues were determined to have caused a quantity of "sink" material to report to the "float" product, thereby impacting the accuracy of the first test. The work was therefore repeated using the more conventional method of using static baths of organic heavy liquid (methylene iodide) diluted with acetone. The sample topsize fraction was crushed to 100% passing ½" in order to improve liberation, and the sample submitted for testing was deslimed at 1mm prior to testing (the -1mm material being of sufficient grade to be combined with the DMS concentrate product).

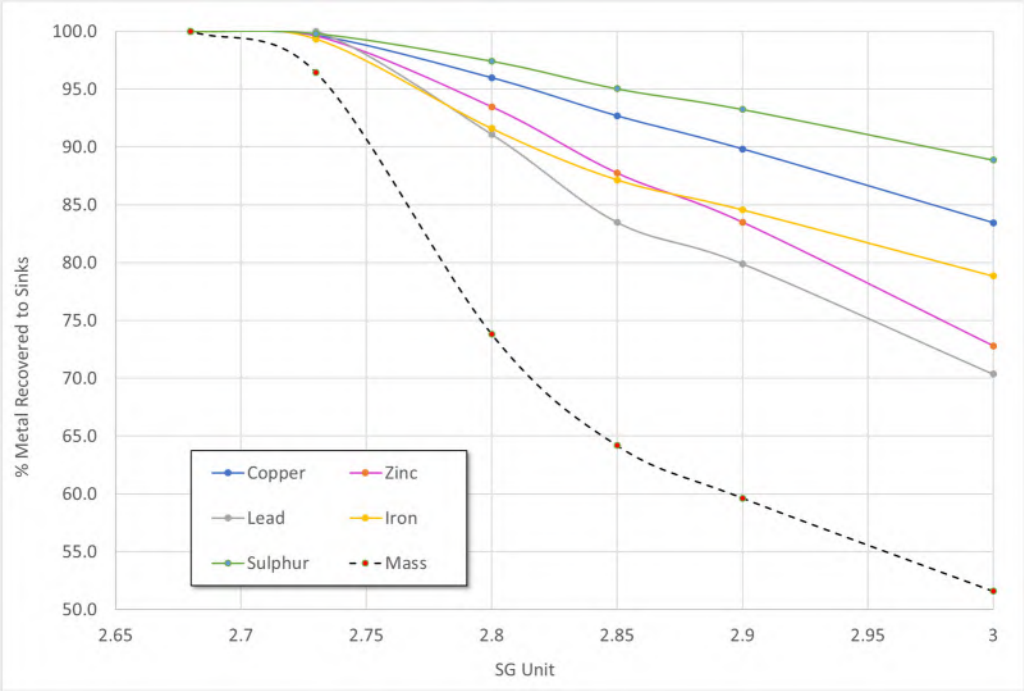
Table 13.39 and Figure 13.16 below summarize the results of the Heavy Liquid Test carried out at SGS on the CSZ core sample. With concentrates combined with the -1mm fraction to make an upgraded mill feed, the test gives 96.0%, 93.5%, and 97.4% of copper, zinc, and sulphur respectively, with 26.2% of the mass rejected to a "floats" product.

Table 13-32: HLS Test (SGS, 2018), Copper Stockwork Sample

Single Products	Mass		% Grade					% Recovery				
	gram	%	Cu	Zn	Pb	Fe	S	Cu	Zn	Pb	Fe	S
~1 mm	840.6	14.04	2.50	0.78	0.02	18	15.3	21.1	14.8	11.1	16.6	16.5
3.00 Sink	2248	37.55	2.77	1.14	0.04	25.2	25.1	62.4	58.0	59.3	62.2	72.4
2.90 Sink	482.4	8.06	1.32	0.98	0.03	10.8	7.07	6.4	10.7	9.5	5.7	4.4
2.85 Sink	272.1	4.55	1.05	0.69	0.02	8.68	5.11	2.9	4.2	3.6	2.6	1.8
2.80 Sink	576.2	9.62	0.57	0.44	0.02	7.0	3.23	3.3	5.7	7.6	4.4	2.4
2.73 Sink	1354	22.62	0.27	0.20	0.01	5.2	1.37	3.7	6.1	8.9	7.7	2.4
2.73 Float	213.9	3.57	0.17	0.08	0.00	2.87	0.71	0.4	0.4	0.0	0.7	0.2
Recon Head	5987.1		1.67	0.74	0.03	15.21	13.0	100.0	100.0	100.0	100.0	100.0
External Ref. Ave.	6000.0		1.70	0.81	0.03	16.25	13.85					
Call Factor	100%		98.1%	91.7%	101.4%	93.6%	94.0%					

Cumulative Results	Mass		% Grade					% Recovery				
	gram	%	Cu	Zn	Pb	Fe	S	Cu	Zn	Pb	Fe	S
~1 mm	841	14.0	2.50	0.78	0.02	18.00	15.30	21.1	14.8	11.1	16.6	16.5
~1 mm + 3.00 Sink	3088	51.6	2.70	1.04	0.03	23.24	22.43	83.4	72.8	70.4	78.8	88.9
~1 mm + 2.90 Sink	3571	59.6	2.51	1.03	0.03	21.56	20.36	89.8	83.5	79.9	84.6	93.3
~1 mm + 2.85 Sink	3843	64.2	2.41	1.01	0.03	20.65	19.28	92.7	87.8	83.5	87.2	95.0
~1 mm + 2.80 Sink	4419	73.8	2.17	0.94	0.03	18.87	17.19	96.0	93.5	91.1	91.6	97.4
~1 mm + 2.73 Sink	5773	96.4	1.72	0.76	0.03	15.66	13.48	99.6	99.6	100.0	99.3	99.8

Figure 13-25: Heavy Liquid Test on CSZ: Mass and Metal Recovery vs SG



This result was a significant improvement over past tests as it was not influenced by the poor liberation and physical constraints experienced in prior work. This encouraging result provided justification for further work on the massive sulphide samples of UWZ and Zone 2.

Figure 13-26 and Figure 13-27 below shows the result under similar conditions for representative samples of UWZ and Zone 2 drill core, crushed to -½" and with 1mm material excluded (then recombined with sinks product) as before.

Figure 13-26: Heavy Liquid Test on UWZ: Mass and Metal Recovery vs SG

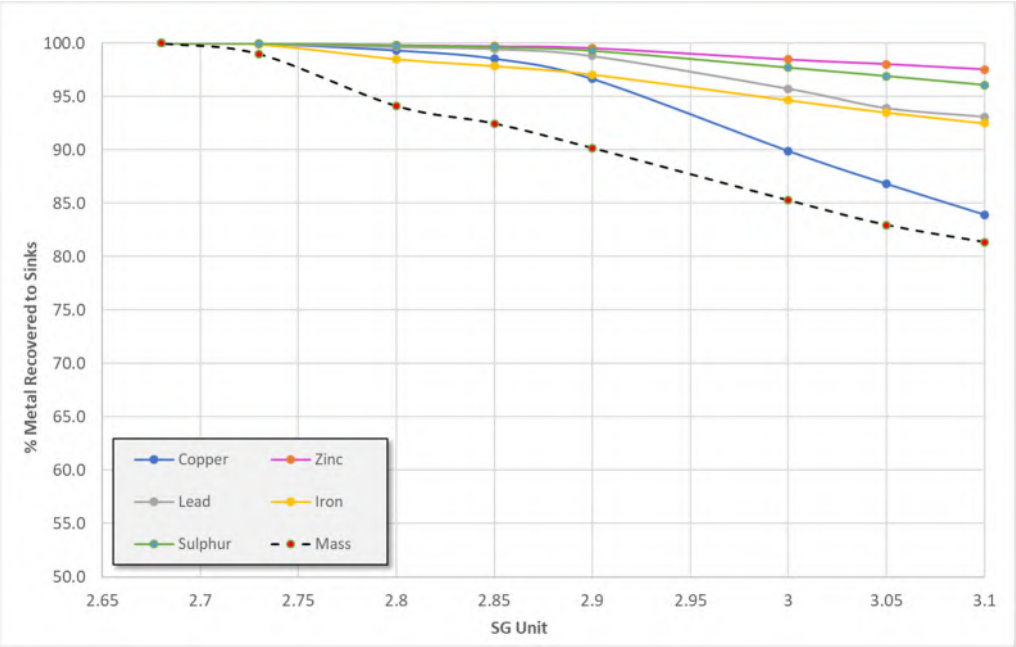
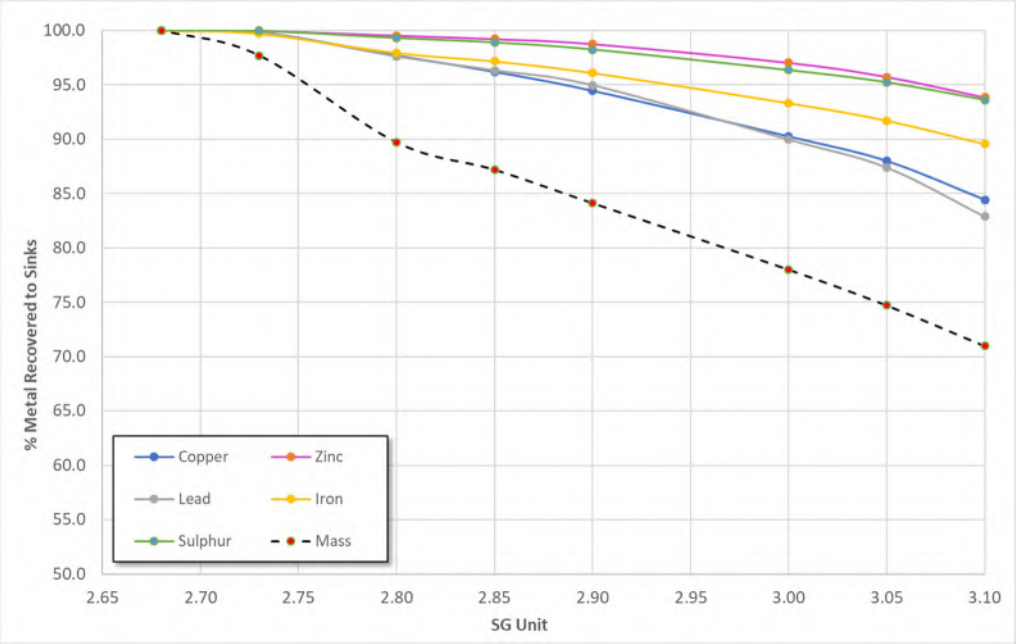


Figure 13-27: Heavy Liquid Test on Zone 2: Mass and Metal Recovery vs SG



The massive sulphide HLS results are not as encouraging as those noted for the CSZ. Densities required to achieve significant mass rejection are higher (to be expected given the higher sulphide concentrations) and the loss of copper to the floats (reject) stream is generally higher.

Upon analysis of these results, DMS separations are judged to be meaningful for the CSZ material, but less so for the UWZ and Z2 material. Given that:

1. the overall percentage of CSZ material in the mill feed is now calculated to be less than 40% of the overall mill feed tonnage, and
2. the process plant is now proposed as an on-site facility rather than 100-km away in Flin Flon

Then the economic impact of preconcentration is judged to be less impactful than at the onset of this study.

Preconcentration using modern sorting machines should still be considered as an alternative means of upgrading low-grade material prior to milling. Further testwork is certainly warranted.

13.11 BL0351 CSZ and Massive Sulphide Blend Testing

A series of tests were conducted to ascertain if there would be any metallurgical complications arising from the combination or blending of massive sulphides with copper stockwork materials prior to grinding and flotation. The fundamental premise of a blend performance was that if, for example, 60% of a massive sulphide composite was blended with 40% of CSZ, then the expectation of overall performance would be the mathematically weighted average of the two individual components (i.e. overall = 0.6 x MS + 0.4 x CSZ). The actual test results for the blend should be similar to the mathematically calculated results if no synergies or detrimental impacts occur. The blend ratios for the three ore types were shown in Table 13-33 below.

Table 13-33: BL0351 Blend Ratios for Blend Testing

	Blend Ratios		
	UWZ	**Z2**	**CSZ**
Blend 1	15	30	55
Blend 2	30	15	55
Blend 3	30	45	25
Blend 4	10	15	75

In Figure 13-28, with respect to the copper performance (left-hand column), Blend 1 which represented the average LOM masses had no indication of any issues with respect to ore type blending as the points for both the actual and calculated fell on the same trend line. Blend 4 with the high ratio of CSZ to MS performed better for actual versus calculated. Blends 2 and 3 showed some variation along the trend line, with some points for each of the actuals and calculated values on the trend line- Blend 2 actuals overall were slightly inferior to calculated and Blend 3 actuals were slightly better overall. In general, the endpoints of the curves at final concentrate grade were all close with respect to actual versus calculated results for all four blends.

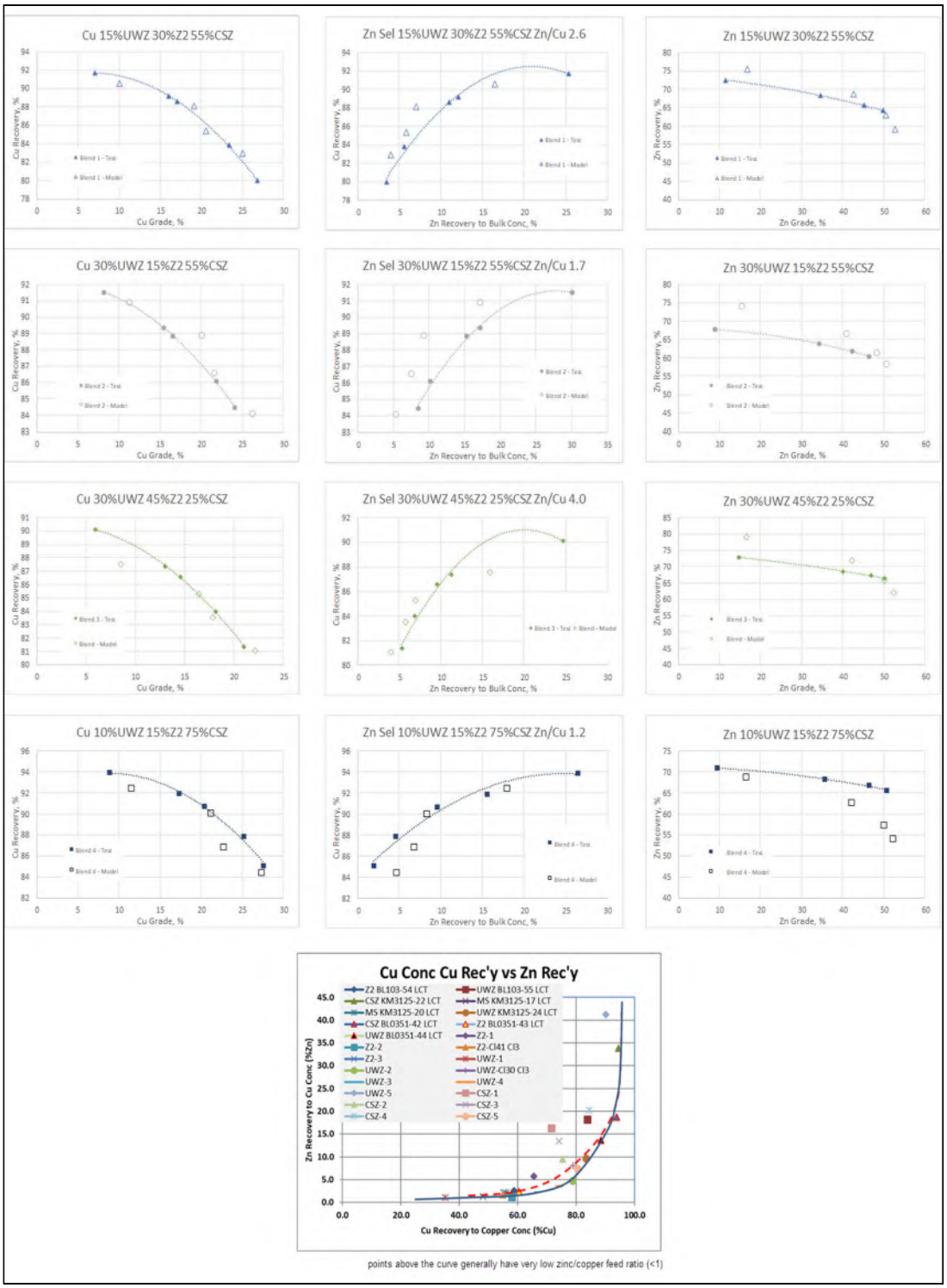
With respect to copper-zinc selectivity (center column), actual zinc recovery to copper concentrate was generally lower than the calculated values, particularly at the final copper recovery stage. The exception was Blend 4, where the actual zinc recovery was higher than calculated. On further investigation, there was a likely reason for this, not related to the fact the Blend 4 contained a high percentage of CSZ. In the bottom chart in Figure 13-28, additional copper-zinc selectivity was shown

for the overall test program results to date. The bulk of the data points fit quite closely to the curve as drawn, irrespective of ore type. However, there were a handful of points which exhibited poorer selectivity, sitting well above the curve. On inspection of what made these outliers different, the feed Zn/Cu ratio was clearly different- ore with Zn/Cu of <~1 resulted in higher zinc recovery into the copper concentrate. Blend 4 had a low Zn/Cu ratio of 1.2, whereas the other blend ranged from 1.7 to 4.0, and this likely contributed to the observation where the zinc recovery was higher in Blend 4 than the other blends.

With respect to zinc performance (right-hand column), actual versus calculated values followed different curves and at first glance actual results appeared to be inferior to calculated performances, with the exception of Blend 4. However, the more important comparison is the endpoint at final grade-recovery. At 50% concentrate grade, actual versus calculated results were quite comparable for Blends 1, 2 and 3, and superior for Blend 4.

Overall, there appears to be no significant concern with blending the ore types with respect to metallurgical performance. These results confirmed that the CSZ material would not have to be mined and processed separately from the MS material, as long as the copper and zinc flotation circuits are sized to accommodate peak copper and zinc feed grades, respectively.

Figure 13-28: Analyses of Blend Test Results (actual versus model)



13.12 BL0351 Tailing Desulphurization Testing

The Zinc flotation circuit tailing contains traces of copper and zinc minerals, but in addition carries a significant volume of sulphide gangue (mostly pyrite). Typically, sulphur grades in the tailing slurry will run between 2 and 8% depending on the blend of mineralization processed. Non sulphide gangue consists mainly of dolomite, and this mitigates the acid-generating qualities of the flotation tailing.

To further improve the acid-generating characteristics, a simple bulk sulphide flotation process utilizing PAX as a non-selective sulphide collector was proposed to treat the zinc flotation tailing, leaving the process plant tailing slurry with only traces of sulphide mineralization. This is proposed to mitigate any acid-generation risks associated with the on-site storage of tailing.

Two tests were run on a blended composite (Blend 1: 55% CSZ, 45% MS) to remove sulphides from the zinc tailing, which graded 3.48% sulphur. Results are shown in Table 13-34 overleaf.

A good quality pyrite concentrate (38% S) was recovered within 1 minute of flotation and after a second minute, the concentration of sulphur in the tailing slurry was reduced to <0.4%. This pyrite concentrate also contained residual copper/zinc sulphides – likely fine-grained material locked within larger pyrite particles.

Refinement of this process through further testing is certainly warranted, but the preliminary tests show that a simple flotation process can easily be added to the end of the copper/zinc flotation circuit to dramatically reduce the acid-generating potential of the tailing slurry. In practice, the pyrite concentrate will be cleaned, dewatered, and mixed with the paste backfill material. After mixing the small sulphide mass with barren tailing cake and portland cement, then storing underground as cemented backfill, any acid generating potential will be very effectively mitigated.

Geochemical testing of the low-sulphur tailings products is recommended as part of the tailing's facility design criteria development.

Table 13-34: Desulphurization Test – Blend 1

Product	Weight		Assay – percent or g/t							Distribution - percent						
	%	grams	Cu	Pb	Zn	Fe	S	Ag	Au	Cu	Pb	Zn	Fe	S	Ag	Au
Cu/Pb Ro Conc	9.1	183.1	10.2	1.25	6.20	20.1	26.8	78	4.43	90.5	60.4	19.3	13.9	17.7	59.5	80.6
Zn Ro Conc	20.6	413.9	0.26	0.15	10.0	33.0	43.4	13.0	0.26	5.2	16.4	70.4	51.6	64.7	22.4	10.7
Py Ro Conc 1	5.1	102.9	0.18	0.14	4.80	33.6	38.8	11.0	0.21	0.9	3.8	8.4	13.0	14.4	4.7	2.1
Py Ro Conc 1-2	8.3	166.9	0.25	0.16	3.17	25.6	26.6	11.0	0.15	2.0	6.8	9.0	16.1	16.0	7.7	2.5
Py Ro Tail	62.0	1246.1	0.04	0.05	0.06	3.92	0.38	2.00	0.05	2.3	16.4	1.3	18.4	1.7	10.4	6.2
Feed (calc.)	100	2010.0	1.03	0.19	2.92	13.2	13.8	11.9	0.50	100	100	100	100	100	100	100
Zn Ro Tail	70.3	1413.0	0.06	0.06	0.43	6.48	3.48	3.06	0.06	4.3	23.3	10.3	34.6	17.7	18.0	8.7

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

This section presents the updated Mineral Resource Estimate for Foran's McIlvenna Bay Project in Saskatchewan. This Mineral Resource Estimate is based upon Foran's drilling database which both the historical drilling and Foran's drilling up to the end of 2018. Micon's QPs have reviewed and audited the Mineral Resource Estimate and the estimate is presented here for disclosure as per NI 43-101 standards of disclosure for mineral projects.

The 2018 drilling program was designed to improve the confidence of the known mineralization, previously reported by Foran in 2013, and to potentially increase the Inferred resources at depth. Previous iterations of the resource model have been completed and published since 2010 including the latest iteration in 2013 by RPA. The last iteration in 2013 was used as the basis for the Preliminary Economic Assessment completed by JDS in 2014. All these previous iterations have now been superseded by the current 2019 estimate contained in this section.

14.2 CIM Mineral Resource Definitions and Classifications

If a company is a reporting Canadian entity, all resources and reserves presented in a Technical Report should follow the current CIM definitions and standards for Mineral Resources and Reserves. The latest edition of the CIM definitions and standards was adopted by the CIM council on May 10, 2014, and includes the resource definitions reproduced below:

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings, and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity, and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.3 Mineral Resource Database and Wireframes

14.3.1 Database

The basis for the Mineral Resource Estimate was a drill hole database provided by Foran on December 9th, 2018. The database and underlying QA/QC were validated by Foran prior to being used in the modelling and estimation. After a further visual validation of the database, it was decided to exclude two drillholes[1] from the resource estimate due to conflicting geological information. Table 14-1 summarizes the types and amount of data in the database and the portion of the data used for the Mineral Resource Estimate.

Table 14-1: McIlvenna Bay Project Database

Data Type	In Database	Used For 2019 Resource Estimate*
Collar	246	244
Survey	15,648	15,454
Assay	8,920	8,765

*Excludes two drillholes from the resource estimate due to conflicting geological information.

14.3.2 Wireframes

Jointly with Foran geologists, five mineralized domains were defined representing different areas and styles of VMS mineralization.

1. Massive Sulphide – Main mineralized lens with internal gradational boundaries. The lens was previously modelled as two separate zones (MS and Upper West), but contact plots show no justification for a hard boundary.
2. CSZ – Copper stockwork zone sitting stratigraphically below the massive sulphide
3. Stringer Zone – Copper and zinc stringer zone in the hangingwall above the massive sulphides
4. Lens 3 – Massive sulphide lens sitting in the hangingwall to the Stringer zone
5. FW – Small massive to semi-massive zone ore zone below the CSZ

[1] The excluded drillholes are MB-99-108 and MB-08-127. Drill hole 108 was removed due to an inaccurate collar location as confirmed by Foran and drill hole 127 was excluded due to conflicting mineralization intervals between drill holes 127 and 73 located within 3.3 m of each other. Drill hole 73 was selected based on Foran's review of the mineral intersections used for modelling the deposit.

Wireframes were generated based on a set of mineralized intercepts defined by Foran and validated. The wireframes for each of the five domains were validated against drill hole data and found to reasonably represent the mineralization and the host rock. All of the mineralization is hosted within the same lithological unit, the McIlvenna Bay Formation with minor local exceptions where the Lens 3 and Stringer mineralization can cross the hanging wall contact into the cap tuffite unit. The host rock package is of variably mineralized felsic and mafic volcanics, capped by a unit of mixed felsic tuff and cherty sediments locally mineralized and overlain by the Koziol Iron Formation.

Figure 14-1 is a screenshot showing the relationship between Lens 3 and the Copper Stockwork mineralized domains while Figure 14-2 shows the Massive Sulphide and Stringer mineralized domains and with Figure 14-3 showing all of the mineralized domains in relation to one another.

Figure 14-4 is a cross-section of the geological model showing all the grade shells hosted in the McIlvenna Formation with the figure looking towards the northwest.

Figure 14-1: Screenshot Showing Lens 3 and the Copper Stockwork Mineralized Domains

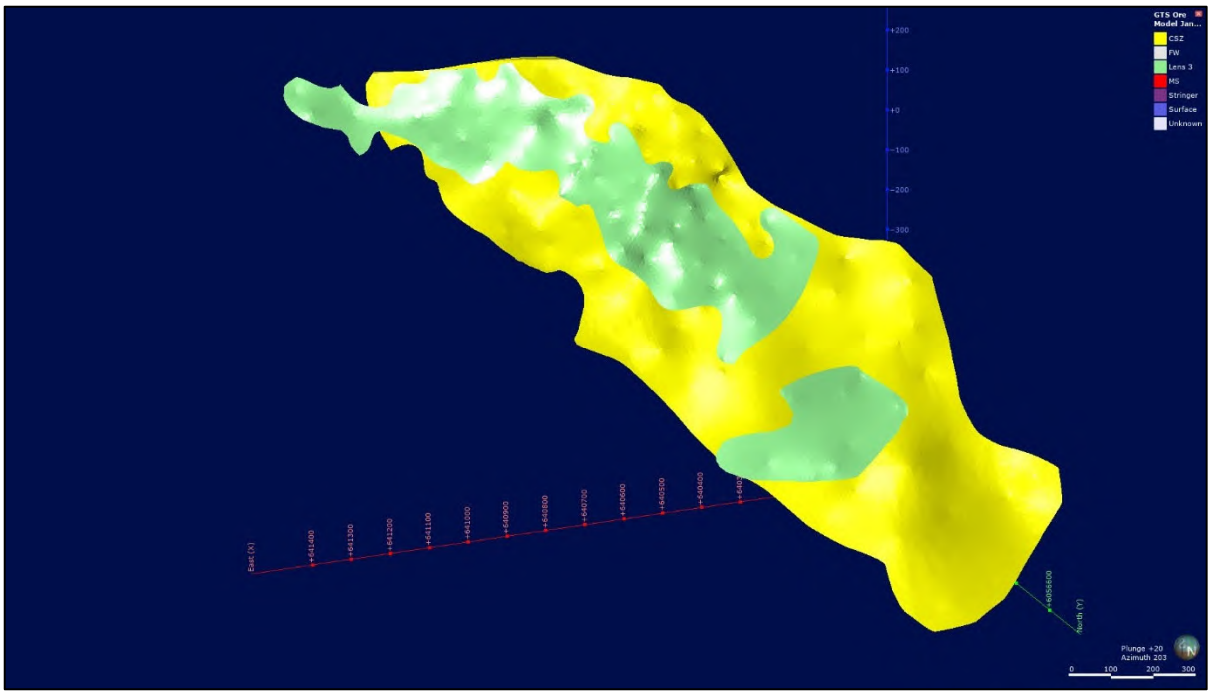


Figure 14-2: Screenshot Showing the Massive Sulphide and the Stringer Mineralized Domains

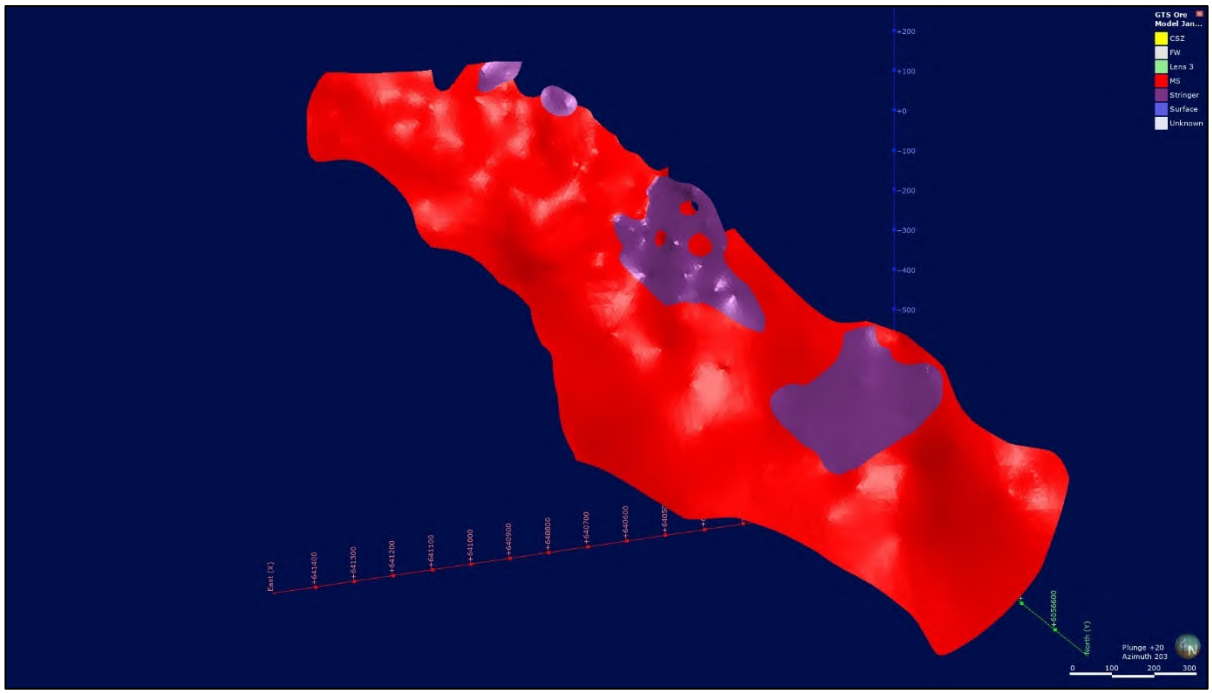


Figure 14-3: Screenshot Showing All of the Mineralized Domains

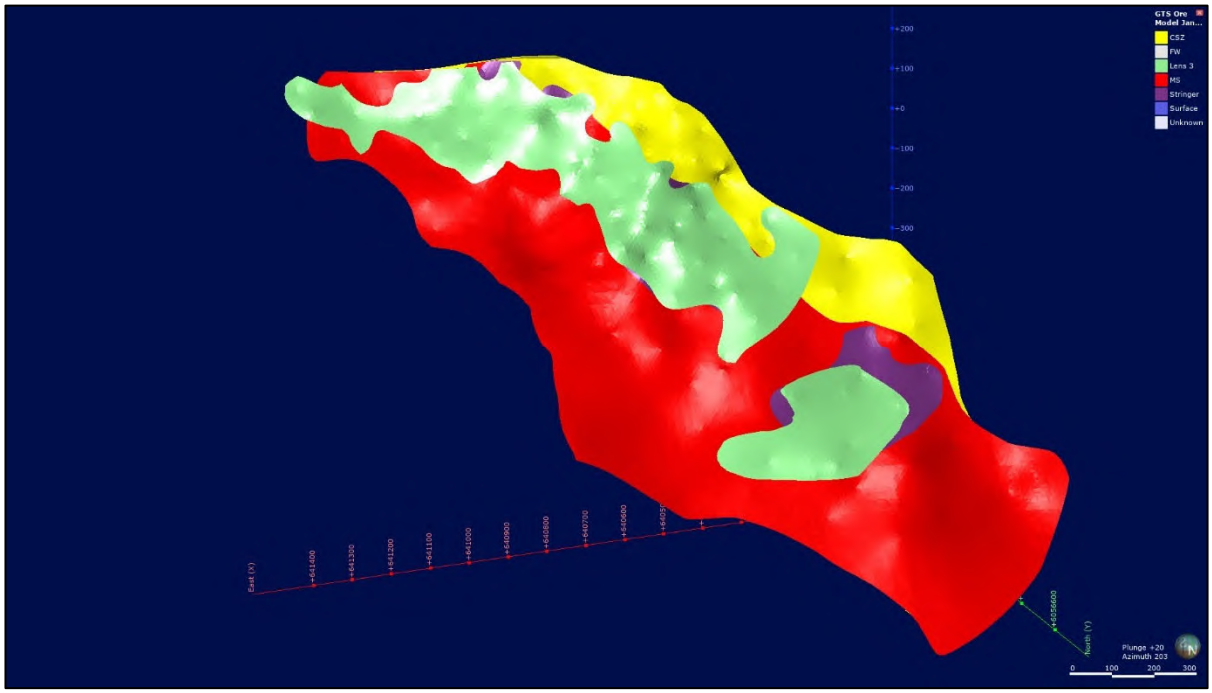
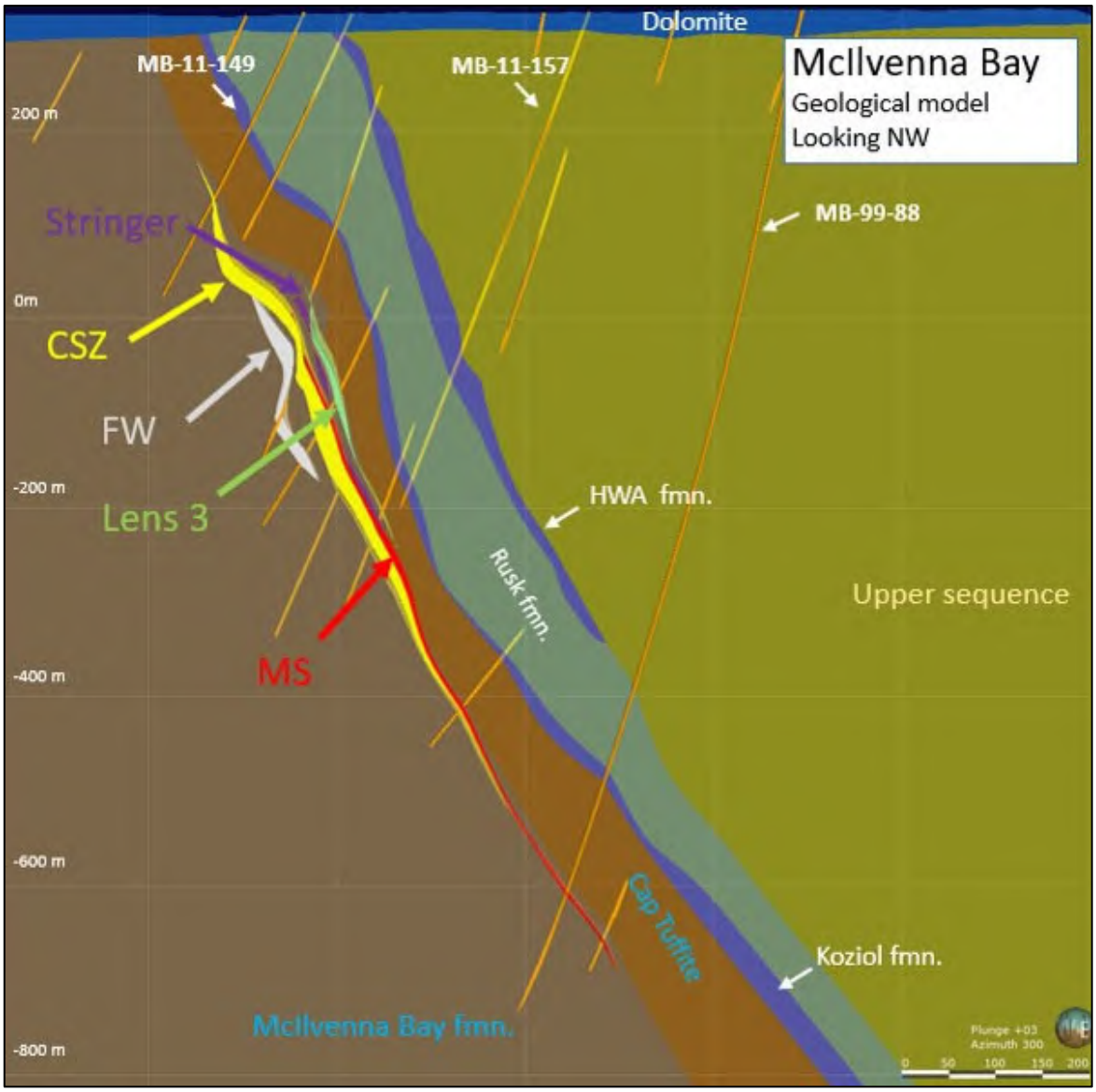


Figure 14-4: Cross-Section showing all Mineralized Grade Shells Hosted in the McIlvenna Bay Formation



A detailed geological and statistical analysis was performed to examine the grade variability and continuity within the MS and Upper West Zone (UWZ) areas. The principle justification for the merging of two domains is the grade transition across the two areas of mineralization (MS and UWZ). Contact plots illustrate that the transition is gradual and imposing a hard break between the two areas would misrepresent the grade transition and metal ratios.

Figure 14-5 is a contact plot across the old UWZ wireframe boundary depicting a Zn grade transition of all composites inside (left) and outside (right) Zone 2 (left). The gradual slope indicates a transitional contact. Figure 14-6 is a contact plot across the old UWZ wireframe boundary showing Zn grade transition of Zone 2 MS (left) and UWZ (right) the gradual slope indicates a transitional contact. Zn

grade transition of all composites inside (left) and outside (right) Zone 2 Massive Sulphide zone is shown in Figure 14-7. The gradual slope indicates a transitional contact. Figure 14-8 shows Zn grade transition of UWZ (left) and Zone 2 MS (right) the gradual slope indicates a transitional contact. Figure 14-24, later in this section, shows the visual appreciation of the transitional contact. This also reflects the nature of VMS deposits but the result on estimation when using a hard boundary would be to either over or under-estimate of the Zn or Cu along what is now a grade transition zone.

Figure 14-5: Contact Plot Showing all Composites Inside (Left) and Outside Lens 2 (Right)

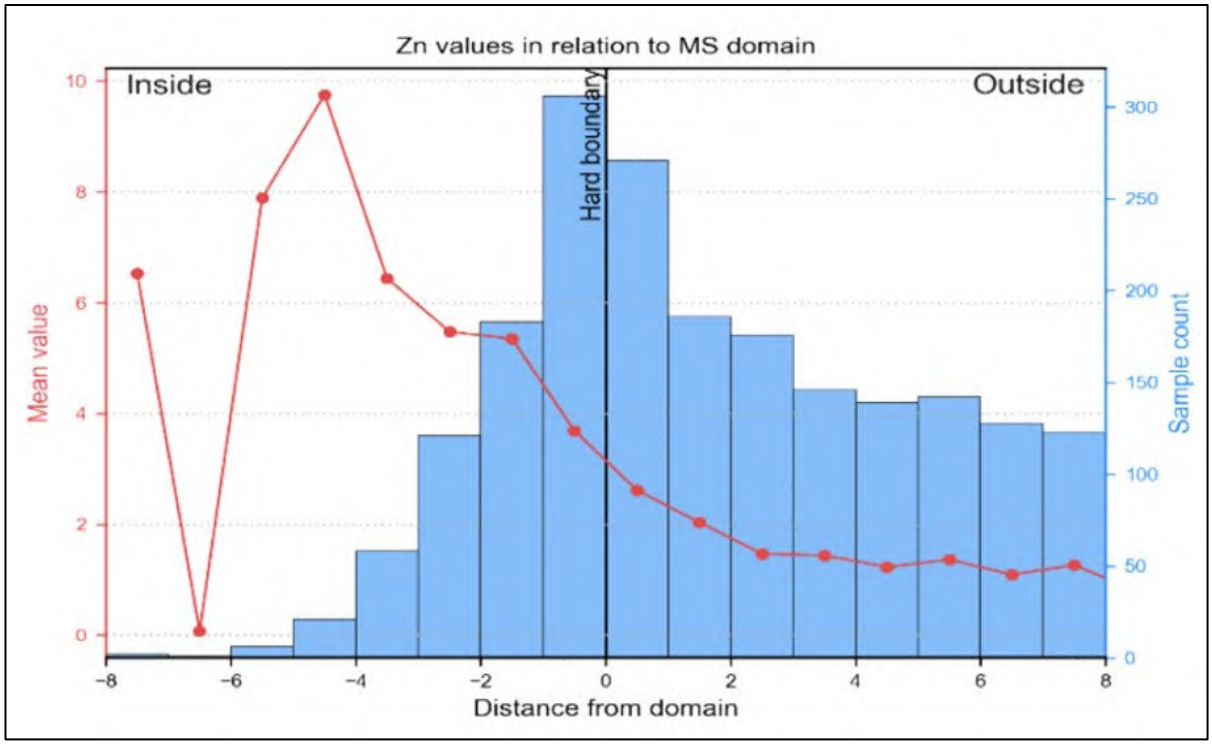


Figure 14-6: Contact Plot Showing Zn Grade Transition of Lens 2 MS (Left) and Lens 2 UWZ (Right)

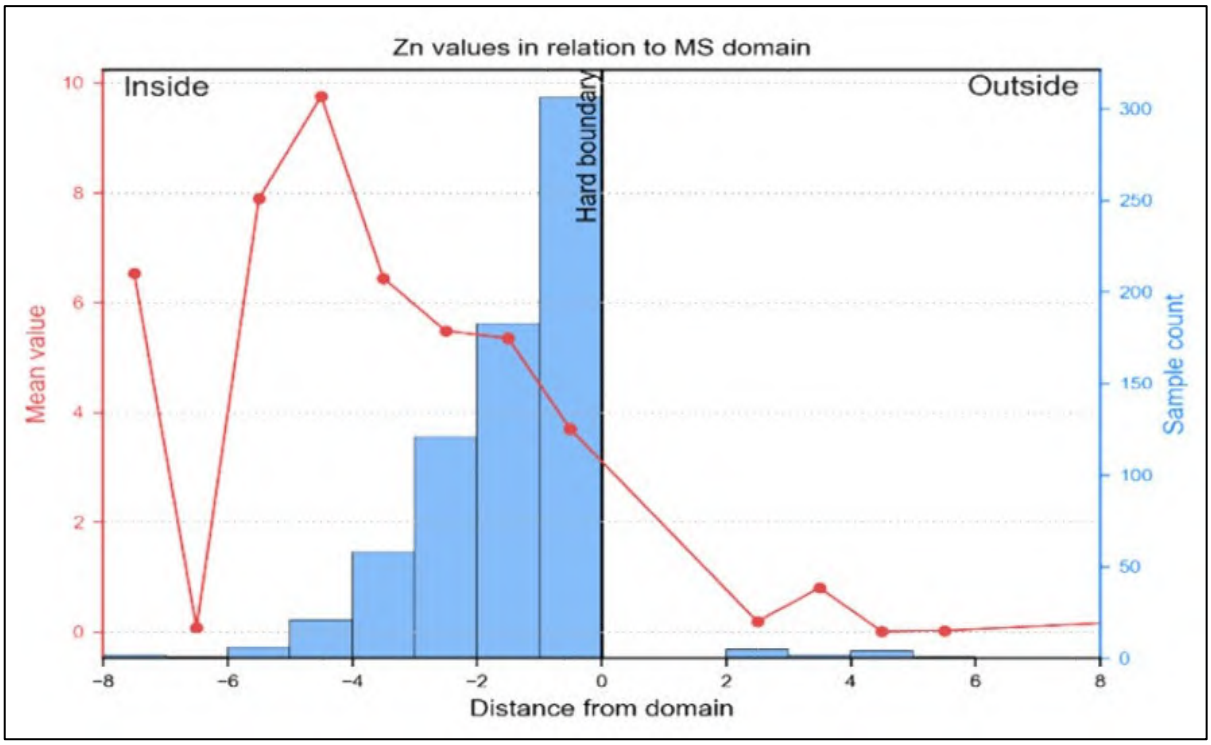


Figure 14-7: Contact Plot Showing Zn Grade Transition of all Composites Inside (L) and Outside (R) Lens 2 Semi-Massive Sulphide

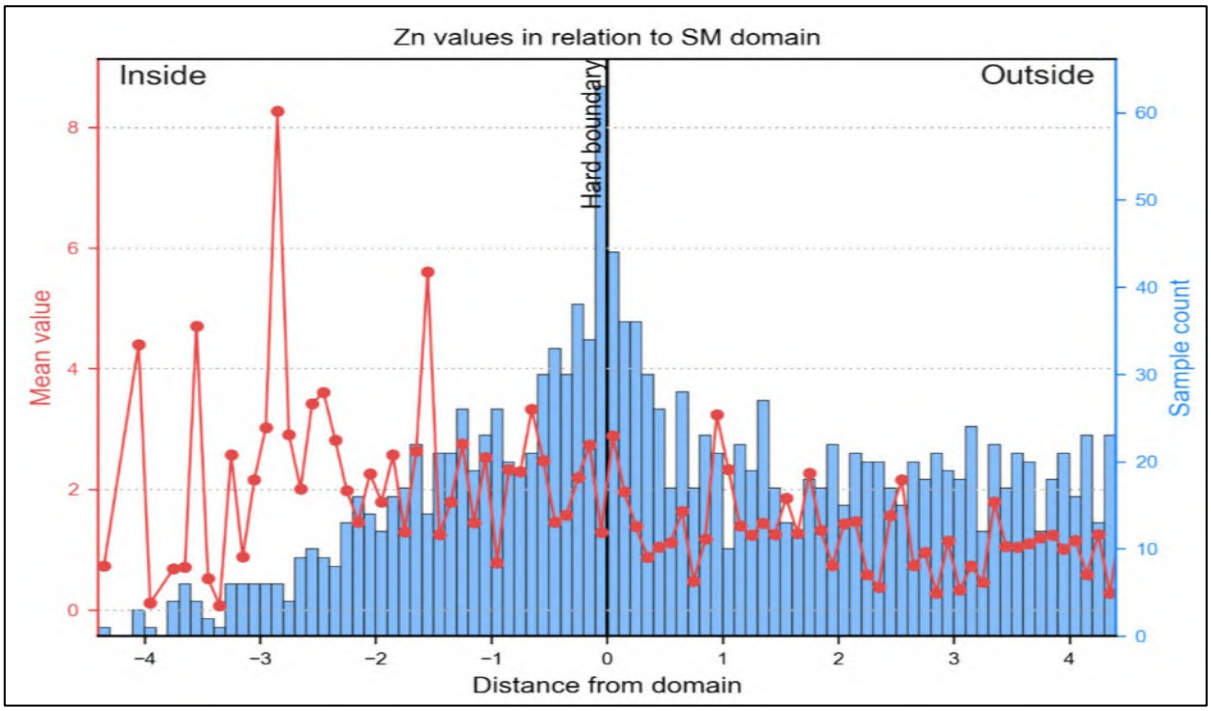
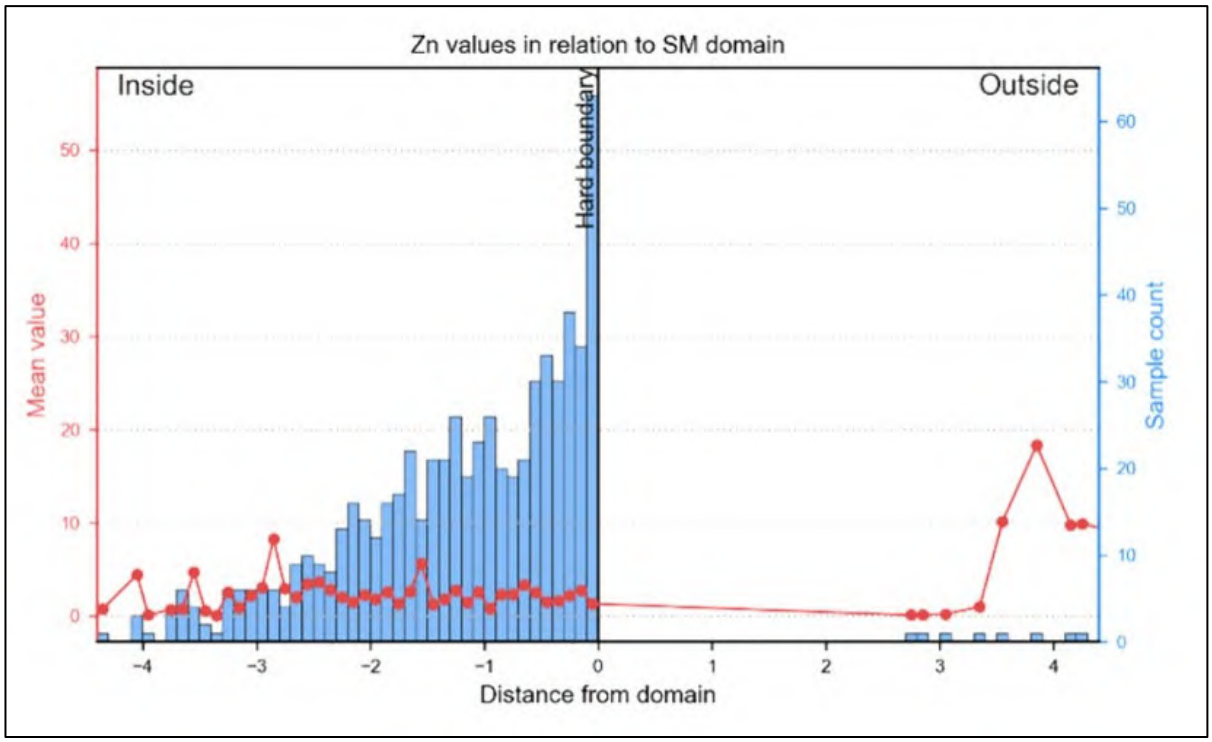


Figure 14-8: Contact Plot Showing Zn Grade Transition of Lens 2 UWZ (L) and Lens 2 SMS (R)



All diamond drill holes are properly snapped to the 3D wireframes to ensure that the volume to be estimated matches both the drilling and logging data collected on the deposit. Visual wireframe validation is presented in the cross-sections shown as Figure 14-9 and Figure 14-10 and indicates the wire frames respect the interval selection and are properly snapped to the drill hole data.

Figure 14-9: Type Cross-Section Showing Cu Assay Grade and the Modelled Wireframes

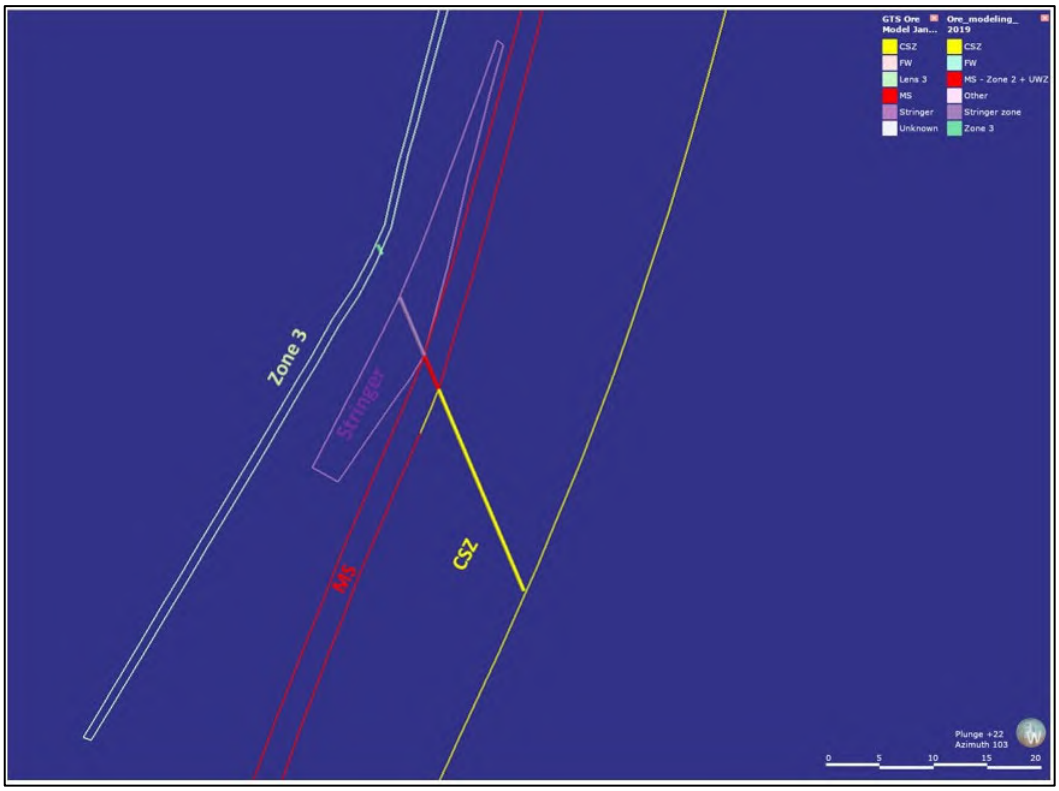
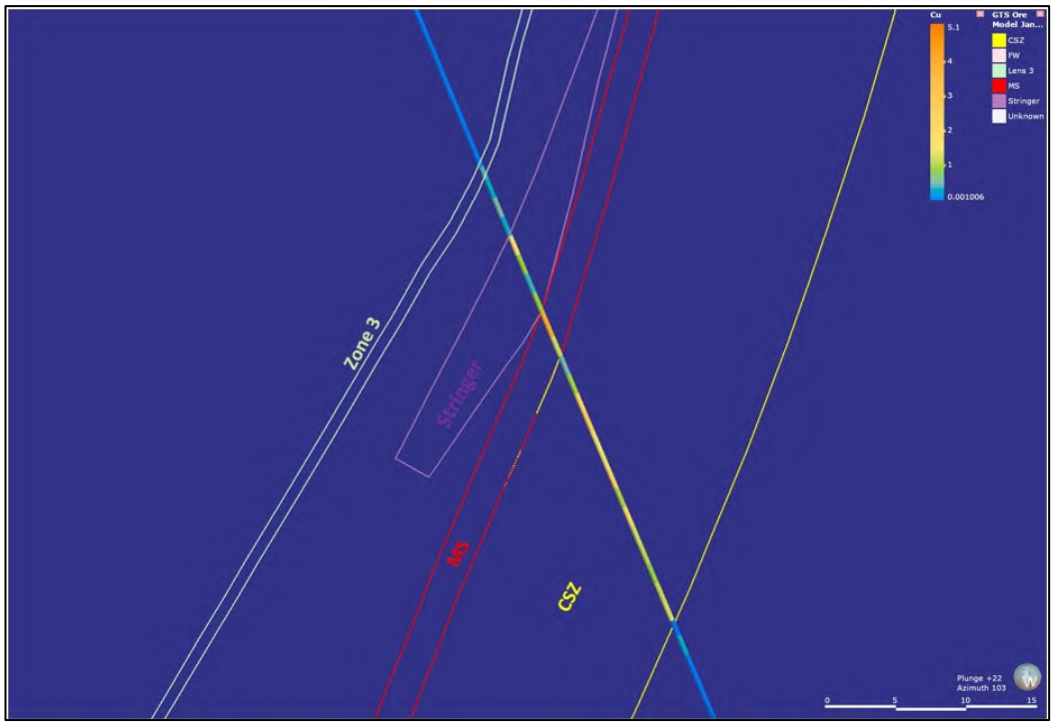


Figure 14-10: Type Cross-Section showing Mineralized Intercepts defined by Foran and Modelled Wireframes



14.4 Compositing And Variography

14.4.1 Compositing

Compositing was performed in Leapfrog Edge, with 1 m composites being used for all domains to honor the initial assay sample resolution and to fit the narrow width of the mineralized zones.

Figure 14-11 and shows the change of support from using the raw assays to a 1 m composite for the CSZ domain.

Figure 14-11: Graph Showing the Change of Support from Raw Assays to 1 m Composites for CSZ Domain

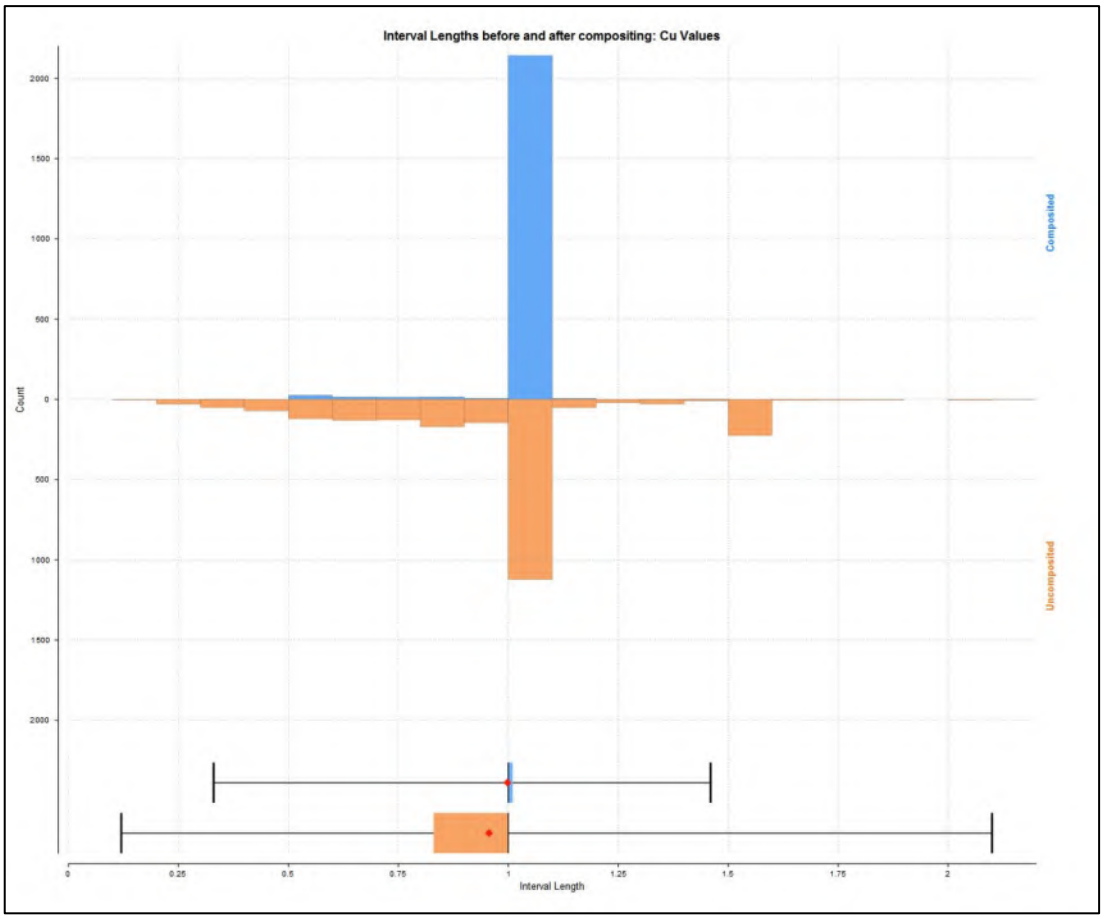
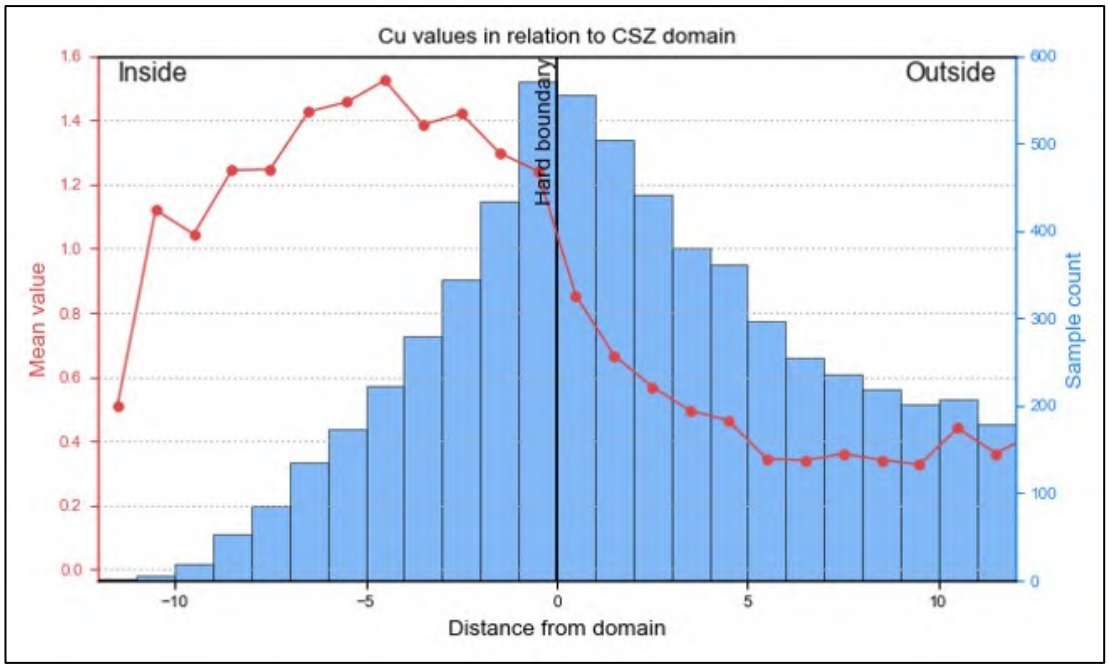


Table 14-2: Tabulation of the Change in Support from the Raw Assays to 1 m Composites for the CSZ Domain

Description	Composited Assays	Un-composited Assays
Count	2,288	2,326
Length (m)	2,223.7	2,224.5
Mean	0.998	0.956
SD	0.066	0.284
CV	0.067	0.297
Variance	0.004	0.080
Minimum	0.330	0.120
Q1	1.000	0.830
Q2	1.000	1.000
Q3	1.010	1.000
Maximum	1.460	2.100

Boundary analysis was performed on the composites to verify the nature of the contacts. While the grade within the wireframes are transitional to the external material, hard boundaries are used in all cases based on geological features which is standard practice in the mining industry. In Figure 14-12 the MS (outside) and CSZ (inside) domains are very close spatially particularly in the up-dip portion of the MS domain. Since both domains are somewhat enriched in Cu, the transition appears gradual, however since the mineralization style is very different, the data from each unit is not permitted to influence the other.

Figure 14-12: Boundary Analysis for Cu in the CSZ Domain



14.4.2 Variography

Variograms were calculated in Leapfrog Edge individually by metal and domain. The variograms produced with the 2018 drilling data demonstrated that the ranges of continuity for all domains decreased slightly relative to the January 2018 internal model but increasing understanding of the grade variability at shorter ranges.

Figure 14-13 to Figure 14-17, are dominant metal variograms for each domain, illustrating the typical behavior for grade continuity at McIlvenna Bay. Experimental variograms are well-modeled by nested nugget and two-structure models. As is typical with this style of sheet-like semi-massive to massive mineralization, the short-range variability is low, resulting in a nugget of approximately 10% of the total variance. Grade continuity along the major axis is generally over 100 m, whereas the semi-major and minor axes have ranges of 70 m to 80 m and 20 m to 30 m, respectively.

Figure 14-13: Copper Variogram Model and Fan Map for the CSZ Domain

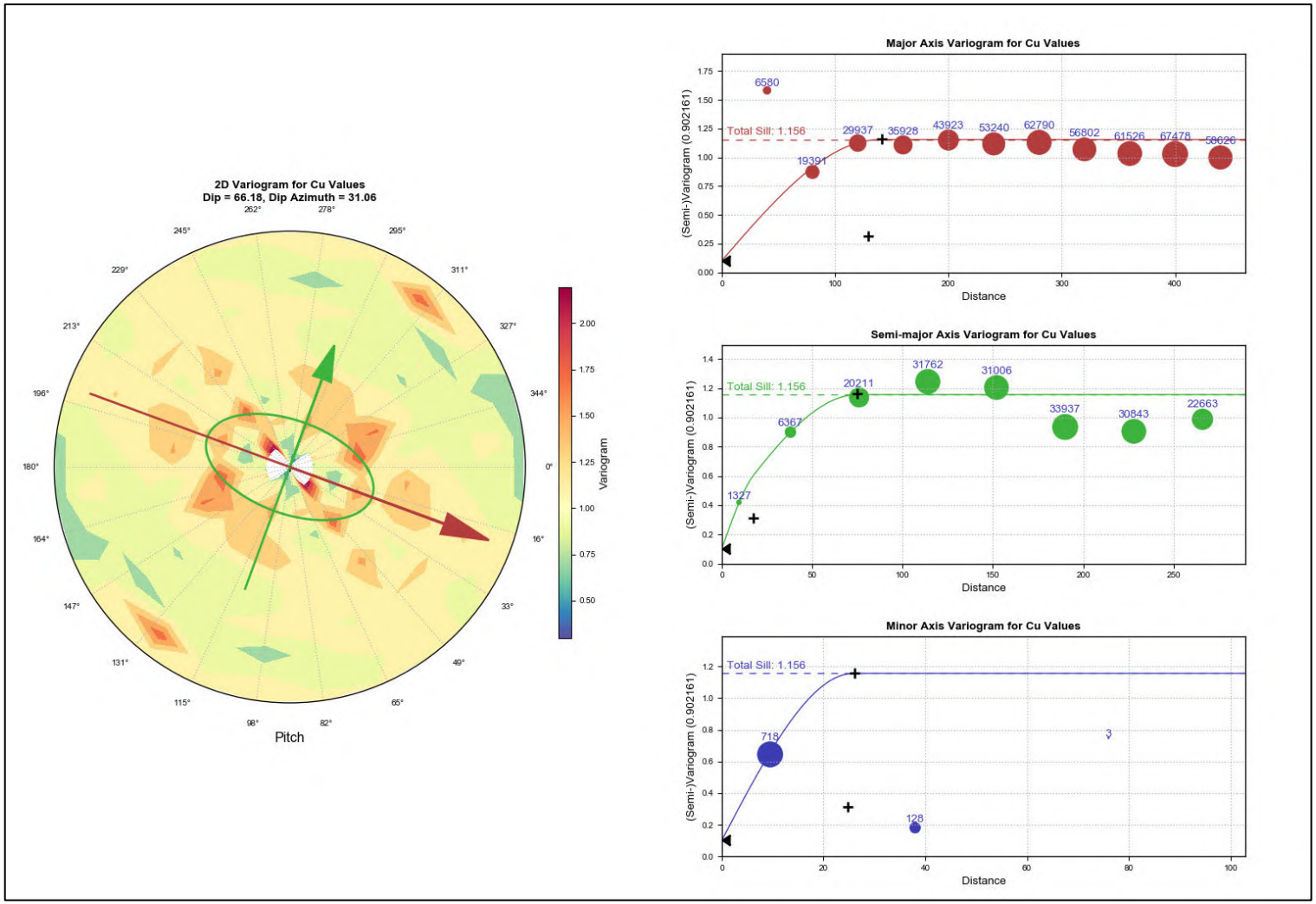


Figure 14-14: Copper Variogram Model and Fan Map for the FW Domain

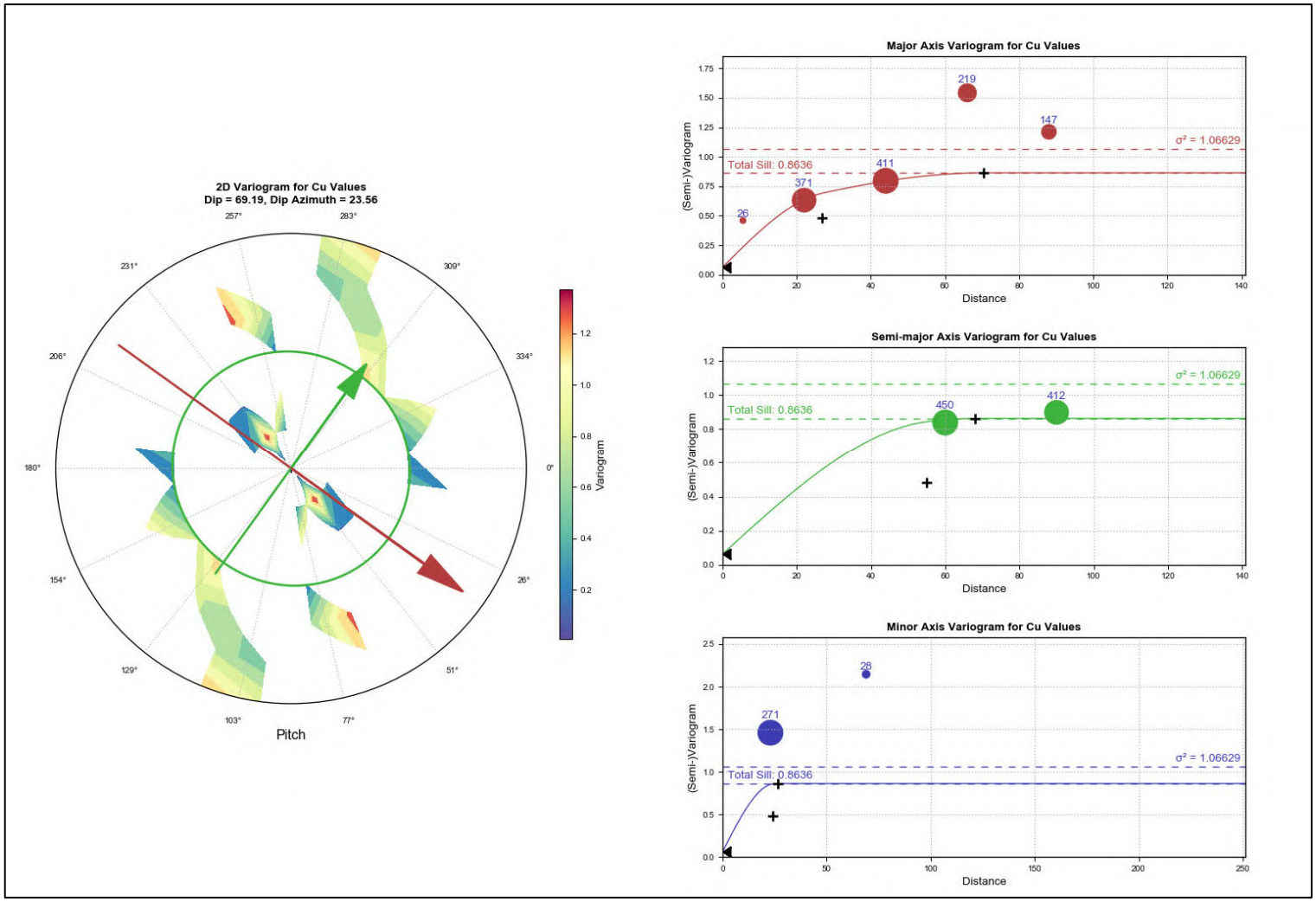


Figure 14-15: Zinc Variogram Model and Fan Map for the Lens 3 Domain

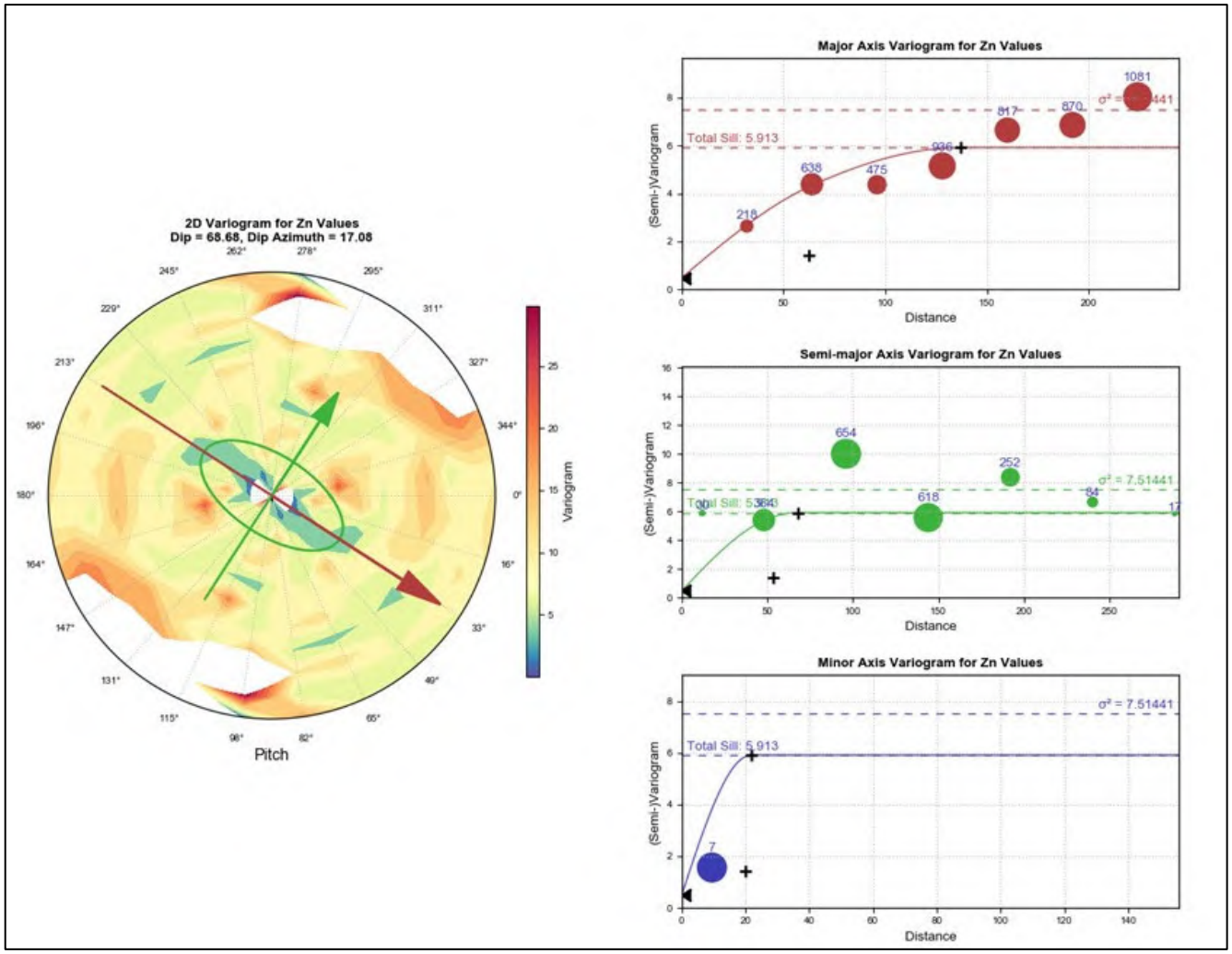


Figure 14-16: Zinc Variogram Model and Fan Map for the MS Domain

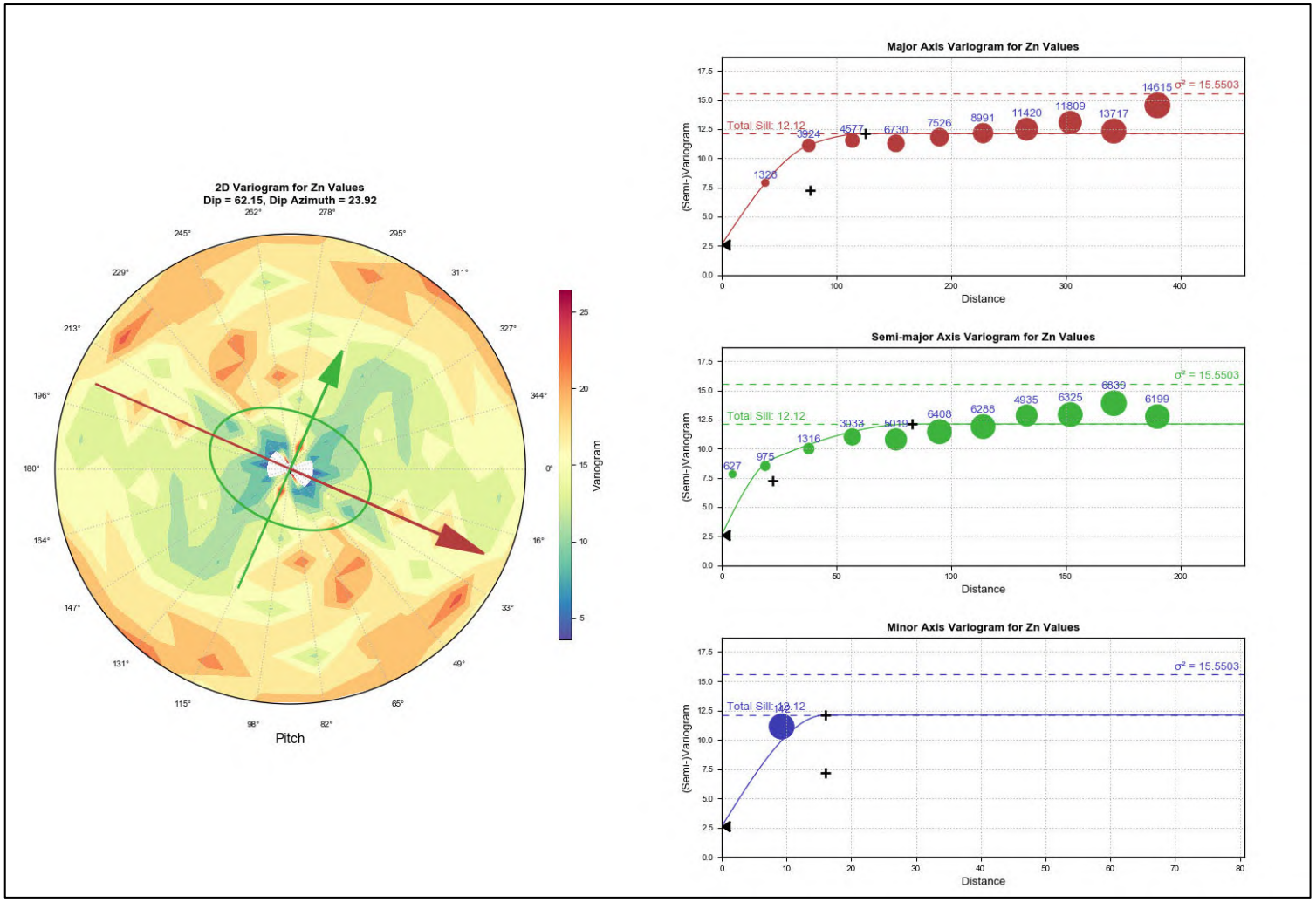
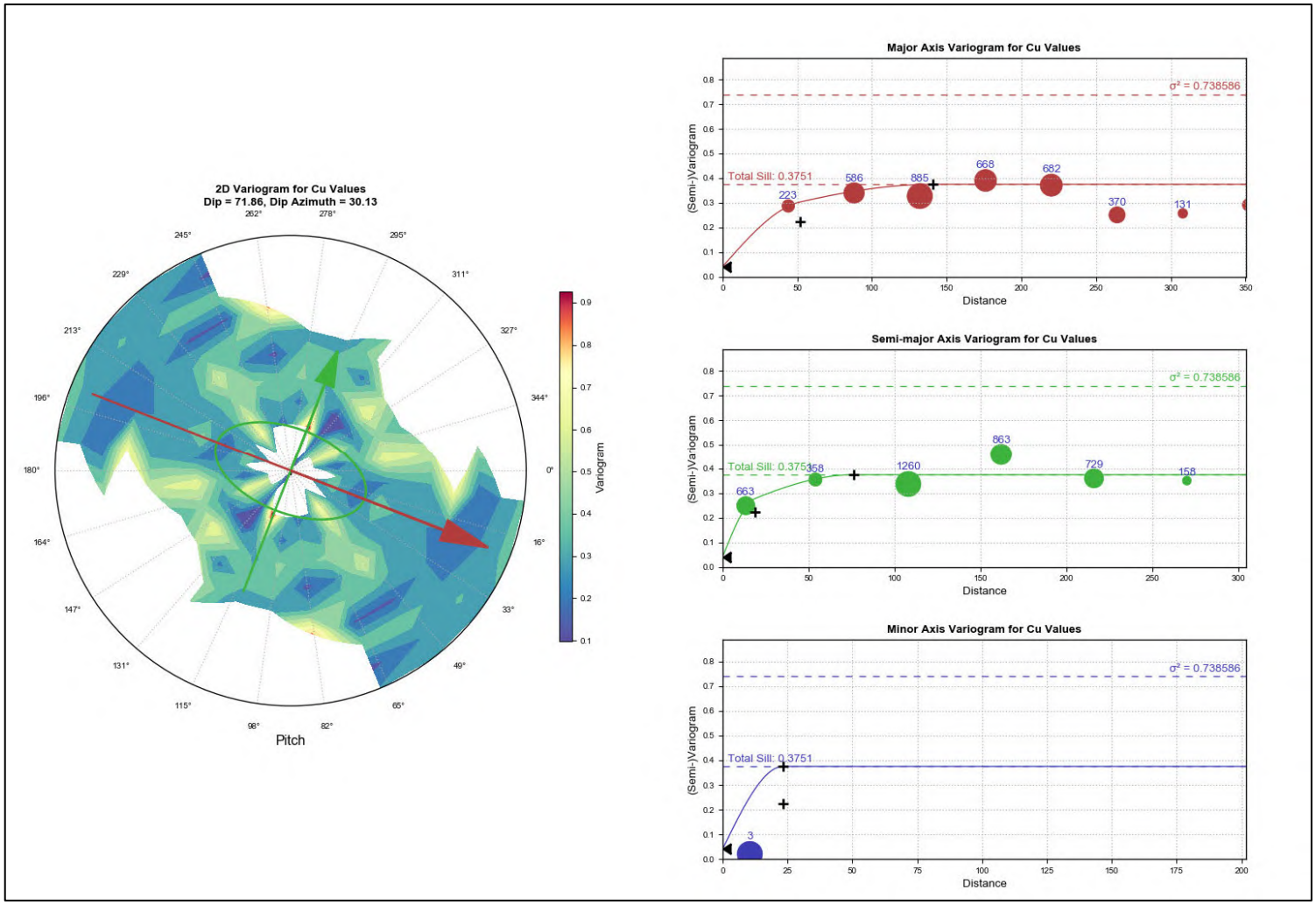


Figure 14-17: Copper Variogram Model and Fan Map for the Stringer Domain



14.4.3 Kriging Neighbourhood Analysis

Based on the modelled variograms, Kriging Neighbourhood Analysis (KNA) was performed on the McIlvenna Bay domains to determine the optimal Kriging parameters for the resource estimation (block size, minimum and maximum number of data points, discretization, and search ellipse size).

KNA uses the modeled variogram, and other criteria such as the maximum number of composites per hole, to conduct a local estimate and to assess the quality of the estimate based on the chosen parameters. In subsequent iterations, the block size, number of composites, size of the search ellipse and block discretization are varied. The Kriging Efficiency[2] and Slope of Regression[3] (or conditional bias), are reviewed for each set of parameters to measure the quality of the estimate. In the McIlvenna Bay study, KNA was used to determine the optimal estimation parameters for the first pass estimation.

The optimal search criteria were used in the interpolation of the grade into the parent blocks, which were later sub-blocked for accurate volume representation.

Figure 14-18 shows the sample KNA analysis for the MS domain used to determine the optimal Kriging parameters for the estimation of the zinc.

The search ellipsoids were configured with the major and semi-major axes parallel to the overall plane of mineralization for each domain. For each domain 3 passes were used with search ranges presented in Table 14-3. In the second and third passes, the interpolation would overwrite blocks estimated by the previous pass. The maximum search distances for each domain were based on the dominant metal per domain.

For the first and second passes, blocks required a minimum of 5 composites to generate an estimate, and the maximum number of composites per block was limited to 20. For the third pass a minimum of 2 composites and maximum of 20 composites was used. No more than 4 composites could be used from any one drill hole for all three passes. The same search parameters were used for all elements (i.e., copper, zinc, lead, gold, and silver) within each domain. Table 14-4 summarizes the optimal Kriging plans identified by KNA and employed in the block model estimation.

Block model grade estimation was completed in Leapfrog Edge. A single estimation method, ordinary kriging, was used for all metals, however validation estimates using ID[2] and NN were also performed.

[2] A measure of the effectiveness of the kriged estimate to reproduce the local block grade accurately.
[3] Summarises the degree of over-smoothing of high and low grades.

Figure 14-18: Sample KNA Analysis for the MS Domain for Determining Optimal Kriging Parameter for Zinc Estimation

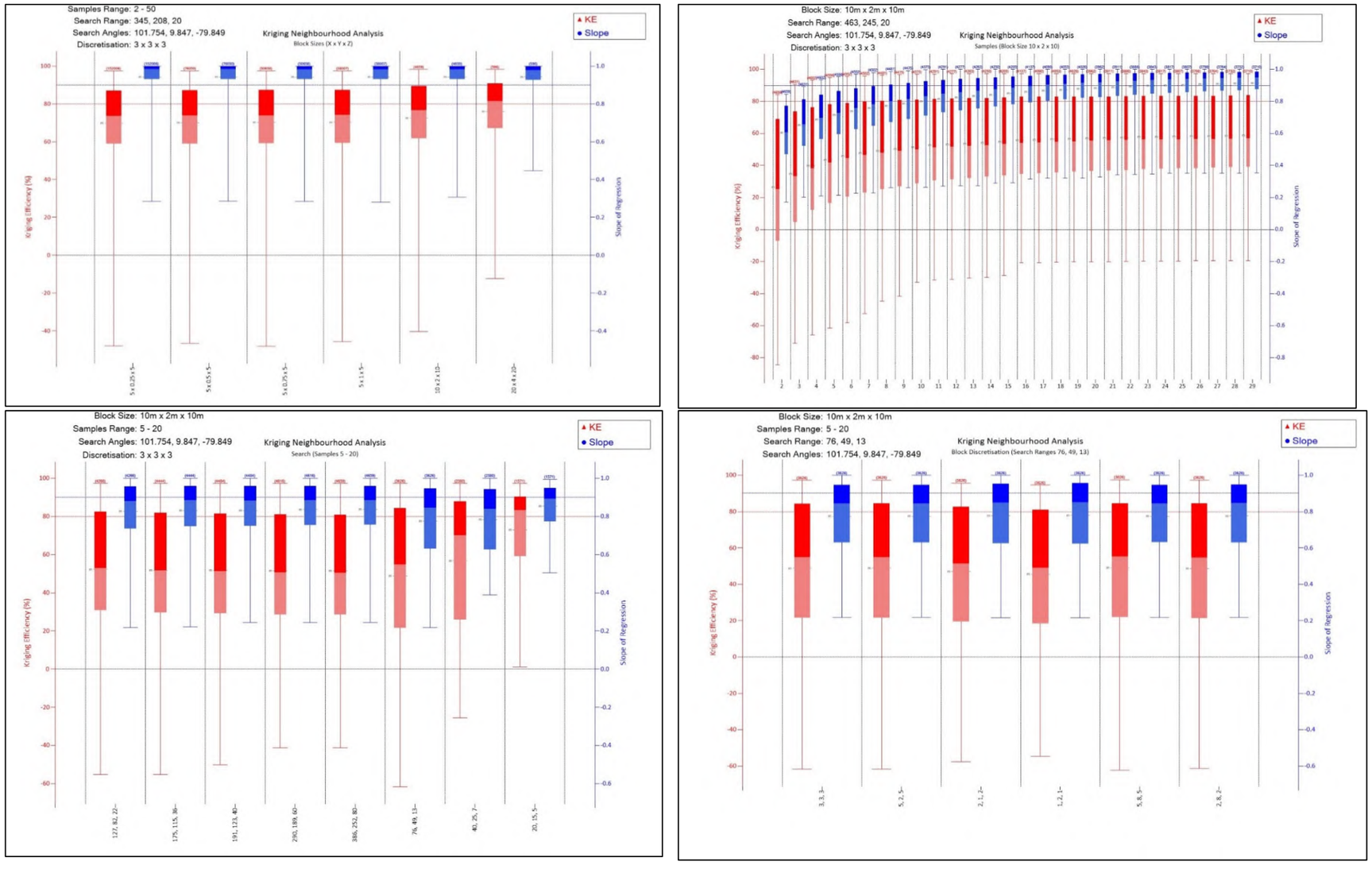


Table 14-3: Search Ellipse Ranges Obtained from KNA (Performed in Snowden Supervisor)

Domain	Pass	Major	Semi-Major	Minor
CSZ	1	55	45	12
	2	80	65	17
	3	170	140	35
FW	1	42	41	16
	2	70	68	26
	3	105	102	39
Len 3	1	75	50	20
	2	105	70	30
	3	210	140	50
MS	1	80	50	13
	2	120	70	20
	3	300	210	50
Stringer	1	75	70	10
	2	105	100	15
	3	210	200	30

Table 14-4: Summary of the Optimal Kriging Plans Identified by KNA and Employed in the Block Model Estimation

Pass Examples	Minimum Number of Samples	Maximum Number of Samples	Maximum Number of Samples/Hole
MS – Zn – OK – Pass 1	5	20	4
Stringer – Zn – OK – Pass 2	5	20	4
CSZ – Cu – OK – Pass 3	2	20	4
	Pass 1	**Pass 2**	**Pass 3**
Maximum Samples/Octant	5	5	-
Maximum Empty Octant	7	7	-

14.5 Capping

The influence of high-grade outliers on the overall grade estimates and contained metal is restricted by the use of top cuts applied to the composited data. Capping values were determined for each metal by domain using the Cumulative Distribution Function (CDF) and by examining the composite grade histograms. Figure 14-19 shows the CDF plot for determination of the copper outlier capping in the CSZ domain. Table 14-5 provides the capping values used for estimation per domain.

Figure 14-19: CDF Plot for Determining the Copper Outlier Capping in the CSZ Domain.

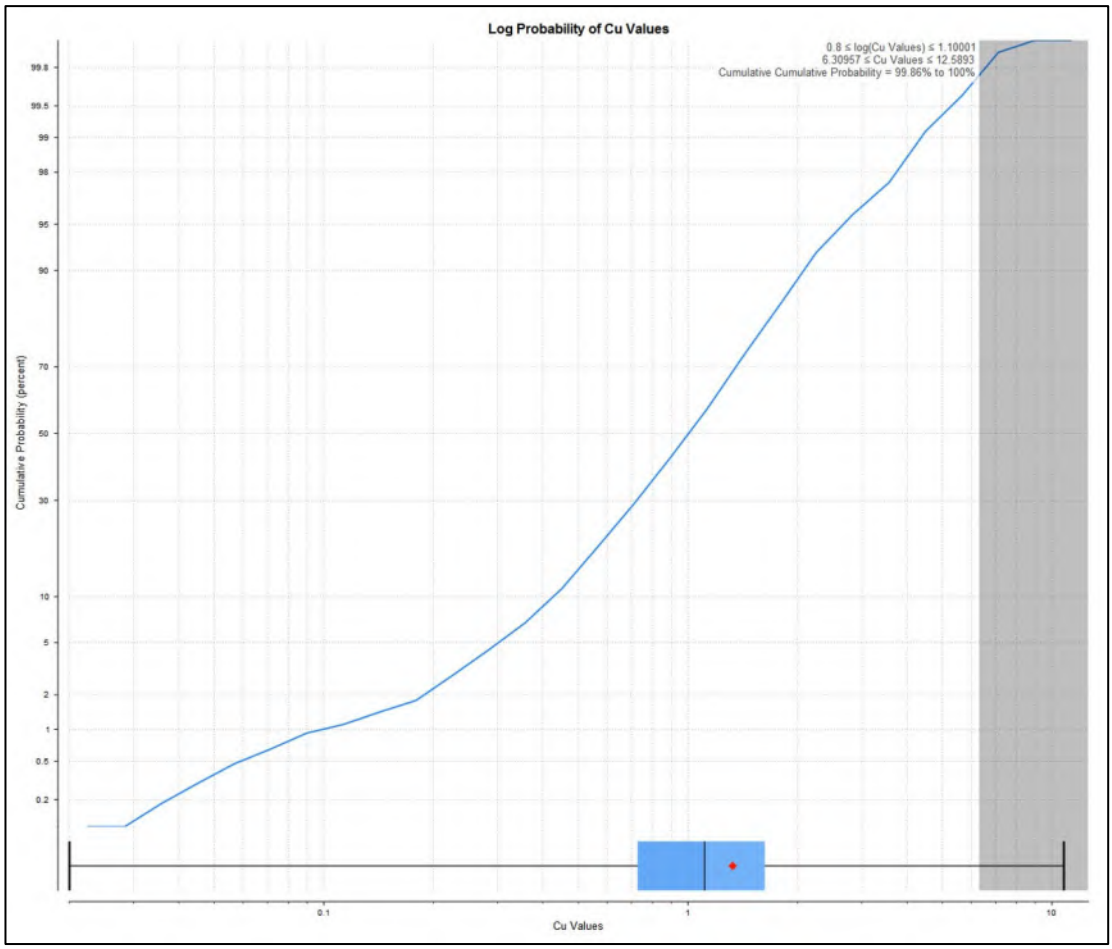


Table 14-5: Outlier Capping by Metal and Domain

Domain	Ag		Au		Cu		Pb		Zn	
	Capping Value (g/t)	Number of Samples Capped	Capping Value (g/t)	Number of Samples Capped	Capping Value (%)	Number of Samples Capped	Capping Value (%)	Number of Samples Capped	Capping Value (%)	Number of Samples Capped
CSZ	63	7	3	28	6.5	9	0.6	5	5	4
FW	50	1	3	3	5	3	---	---	8	5
Lens 3	50	2	2	2	56	4	0.6	9	8	16
MS	200	7	4	15	5	26	2.5	23	16	16
Stringer	50	5	1	6	2.5	15	0.5	3	6	2

14.6 Density

The basis for the density estimates in the previous block model was a set of 1,085 density samples, used to build a simple linear regression[4]. In the present update, a larger database was available which permitted the determination of density by means of interpolation.

Table 14-6 summarizes the density measurement database for the McIlvenna Bay Project.

Table 14-6: Summary of the Density Measurement Database for the McIlvenna Bay Project

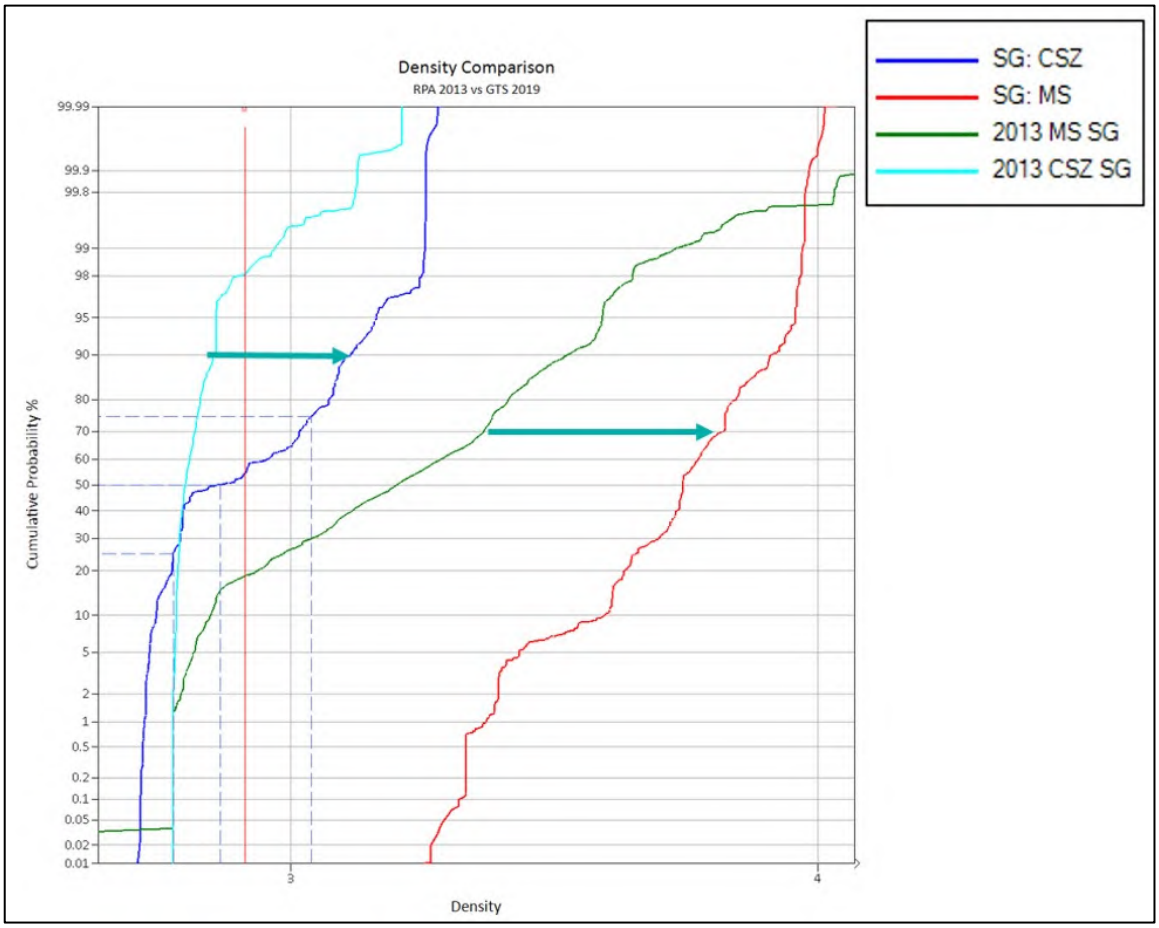
	Bulk Density				Specific Gravity			
Domain	Number	Minimum	Mean	Maximum	Number	Minimum	Mean	Maximum
CSZ	624	2.40	2.90	3.95	575	2.65	2.83	3.92
FW	286	2.74	3.66	4.39	242	2.67	3.33	5.64
Lens 3	53	2.73	3.41	4.10	96	2.33	3.09	5.36
MS	71	2.76	3.00	4.31	66	2.71	3.18	4.62
Stringer	38	2.76	2.99	4.00	30	2.72	2.75	2.78

Measurements of bulk density are considered the more robust, these data points (1,072, in total) were given precedence in the population of density in the block model. Where this data did not populate blocks, the specific density measurements were used (which do not consider bulk density). Comparison by means of a multilinear regression or stoichiometry was not possible since not all elements were available for analysis. The final block value was assigned using a rolling average (ID^0) for each domain, thus generating a smoother continuity of density.

Figure 14-20 shows a comparison of the final block model densities in the 2013 block model against the current 2019 update indicates that density was systematically underestimated in the past. Since the previous regression relied solely on Zn, the mass contributions from other dense minerals, such as pyrite and chalcopyrite, were not considered.

[4] SG = (0.075 x Zn) + 2.8124.

Figure 14-20: Density Comparison between the 2013 PEA Block Model and the 2019 Mineral Resource Block Model



14.7 Block Model

A rotated, sub-blocked model was set-up in Leapfrog Geo to capture the lithological and domain coding, grade estimates, density, and resource classification. Table 14-7 summarizes the parameters for the block model.

Table 14-7: Block Model Parameters

Block Model Setup			
	X	**Y**	**Z**
Parent Block size	10	2	10
Sub-Block count	2	8	2
	Az	**Dip**	
Rotation	45	0	
Model extent	X	Y	Z
Base point	639830	6056565	325
Boundary size	183000	532	1260

The geological model was evaluated on the sub-blocked model and block volumes were validated against wireframes. Table 14-8 summarizes the wireframe to block volume reconciliation.

Table 14-8: Wireframe to Block Volume Reconciliation

Name	Block Count	Volume	Mean	Std. Dev.	Coeff. Var.	Variance	Minimum	L. Quartile	Median	U. Quartile	Maximum
CSZ	795,561	7,794,612.5									
Zn	795,561	7,794,612.5	0.477	0.463	0.971	0.214	0.012	0.160	0.307	0.602	4.291
Cu	795,561	7,794,612.5	1.299	0.474	0.365	0.225	0.042	0.977	1.215	1.538	5.560
Pb	795,561	7,794,612.5	0.026	0.036	1.371	0.001	0.000	0.006	0.013	0.030	0.414
Ag	795,561	7,794,612.5	9.546	4.869	0.510	23.704	1.036	5.766	8.634	12.345	45.710
Au	795,561	7,794,612.5	0.374	0.297	0.793	0.088	0.000	0.159	0.295	0.492	3.041
SG	795,561	7,794,612.5	2.918	0.214	0.042	0.015	2.755	2.816	2.867	3.019	3.257
FW	24,117	297,787.5									
Zn	24,117	297,787.5	0.868	1.176	1.354	1.382	0.018	0.092	0.315	1.195	6.995
Cu	24,117	297,787.5	1.467	0.631	0.430	0.398	0.361	1.066	1.298	1.718	3.839
Pb	24,117	297,787.5	0.037	0.060	1.636	0.004	0.001	0.005	0.010	0.039	0.615
Ag	24,117	297,787.5	10.122	6.913	0.683	47.787	2.133	5.490	7.665	11.980	37.110
Au	24,117	297,787.5	0.472	0.274	0.581	0.075	0.044	0.258	0.436	0.633	1.803
SG	24,117	297,787.5	3.022	0.098	0.033	0.010	2.813	2.977	2.988	3.138	3.145
Lens 3	109,341	688,418.8									
Zn	109,341	688,418.8	2.905	1.746	0.601	3.050	0.021	1.357	2.995	4.141	7.746
Cu	109,341	688,418.8	0.823	0.520	0.632	0.271	0.000	0.467	0.733	1.037	4.712
Pb	109,341	688,418.8	0.122	0.133	0.924	0.013	0.004	0.039	0.076	0.177	0.563
Ag	109,341	688,418.8	13.976	5.439	0.389	29.581	1.879	10.133	13.035	16.648	39.530
Au	109,341	688,418.8	0.260	0.130	0.499	0.017	0.048	0.173	0.232	0.308	1.099
SG	109,341	688,418.8	3.473	0.166	0.048	0.027	3.106	3.331	3.541	3.588	3.659
MS	504,348	3,327,712.5									
Zn	504,348	3,327,712.5	5.957	2.561	0.430	6.560	0.249	3.833	6.295	7.796	14.557
Cu	504,348	3,327,712.5	1.004	0.980	0.976	0.960	0.024	0.234	0.616	1.556	5.201
Pb	504,348	3,327,712.5	0.374	0.287	0.768	0.083	0.000	0.160	0.321	0.513	2.500
Ag	504,348	3,327,712.5	25.548	15.957	0.625	254.626	3.803	17.107	21.389	28.703	177.546
Au	504,348	3,327,712.5	0.511	0.480	0.939	0.230	0.030	0.206	0.347	0.597	3.776
SG	504,348	3,327,712.5	3.717	0.147	0.040	0.022	3.224	3.626	3.716	3.807	4.042
Stringer	59,178	373,737.5									
Zn	59,178	373,737.5	0.585	0.625	1.069	0.391	0.070	0.260	0.424	0.624	4.592
Cu	59,178	373,737.5	1.177	0.378	0.321	0.143	0.145	0.883	1.144	1.465	2.409
Pb	59,178	373,737.5	0.039	0.041	1.050	0.002	0.004	0.017	0.028	0.040	0.438

Name	Block Count	Volume	Mean	Std. Dev.	Coeff. Var.	Variance	Minimum	L. Quartile	Median	U. Quartile	Maximum
Ag	59,178	373,737.5	12.640	4.034	0.319	16.277	5.700	10.174	11.745	13.520	42.281
Au	59,178	373,737.5	0.299	0.118	0.394	0.014	0.064	0.223	0.272	0.372	0.771
SG	59,178	373,737.5	2.995	0.111	0.037	0.012	2.828	2.936	2.941	2.983	3.296

Visual and numeric validation was performed to compare the results of the estimation methods and correlation to original input grades.

For increased confidence in the model estimate, and as a separate means of validation, a block model estimate was also generated in Datamine Studio RM. The validation estimate used the same domain wireframes and outlier top-cuts, but different variograms and search parameters. The Datamine block model was prepared as a validation tool, and both the visual and global means comparisons of the two models returned nearly identical results. This further validates both the accuracy and precision of the McIlvenna Bay Mineral Resource Estimate.

Table 14-9 summarizes the parameters used in the validation estimate.

Table 14-10 summarizes the comparison between the block model estimated in Leapfrog Edge (final model) and Datamine Studio RM (validation). The comparison was conducted at a zero-cut-off grade to compare the entire model and not just the potentially economic portion. This is because the economic portion of a resource model will vary due to changes in metal prices, mining, and processing costs, general and administrative (G&A) costs, regulatory changes, and taxes.

Table 14-9: Summary of the Parameters Used in the Validation Estimate

Maintained from Leapfrog Geo/Edge Model	Independently Generated Parameters Datamine Studio RM Model
Wireframes	Variograms
Composites	Search ellipse orientations
Capping	Search parameters
Block model prototypes	Sub-blocking
	Resource classification (no grooming or management for the spotted dog effect)

Table 14-10: Comparison Between Block Model Estimated in Leapfrog Edge (Final Model) and Datamine Studio RM (Validation)

Software	Cut-off	Tonnes (Mt)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Au (g/t)
Edge	0	39.52	2.37	1.17	0.14	14.90	0.41
Datamine	0	39.32	2.40	1.17	0.15	14.94	0.42

Figure 14-21 to Figure 14-23 show the grade intensity (heat) map of the zinc grades for the Leapfrog and Datamine models as a comparison and the grade intensity map for the copper within the MS domain in the resource model.

Figure 14-21: Grade Intensity (Heat) Map of Zinc Grades from Leapfrog Model

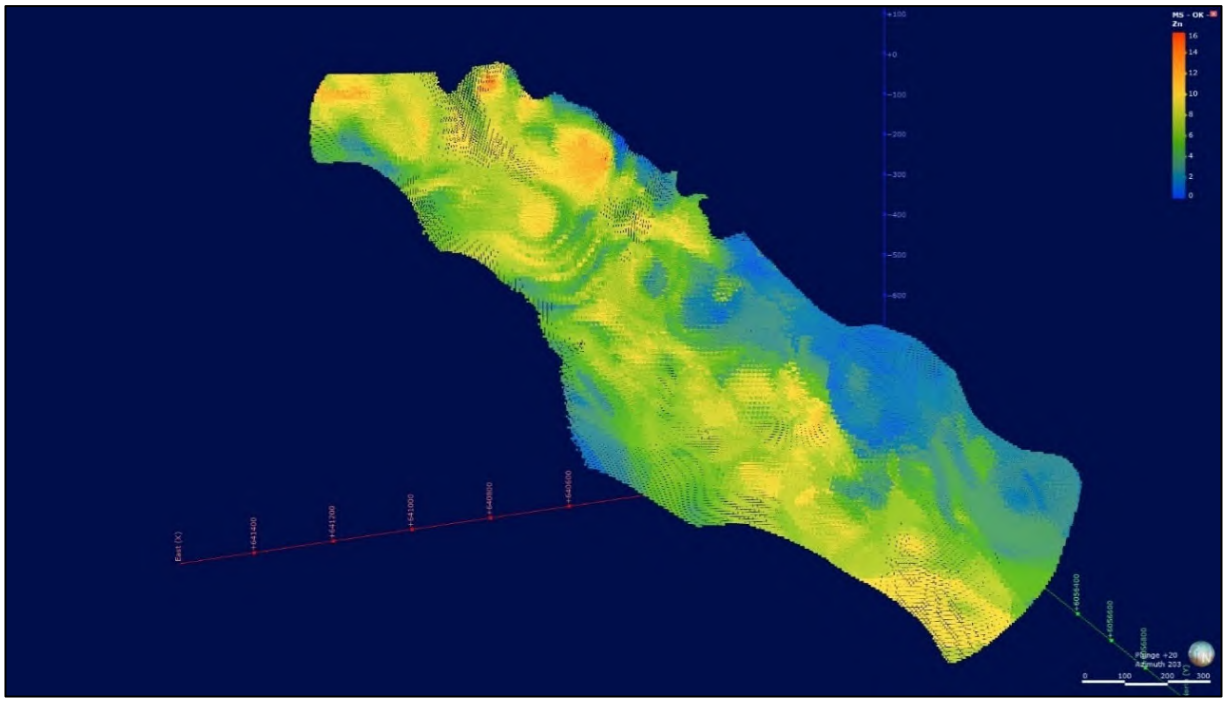


Figure 14-22: Grade Intensity (Heat) Map of Zinc Grades from Datamine Model

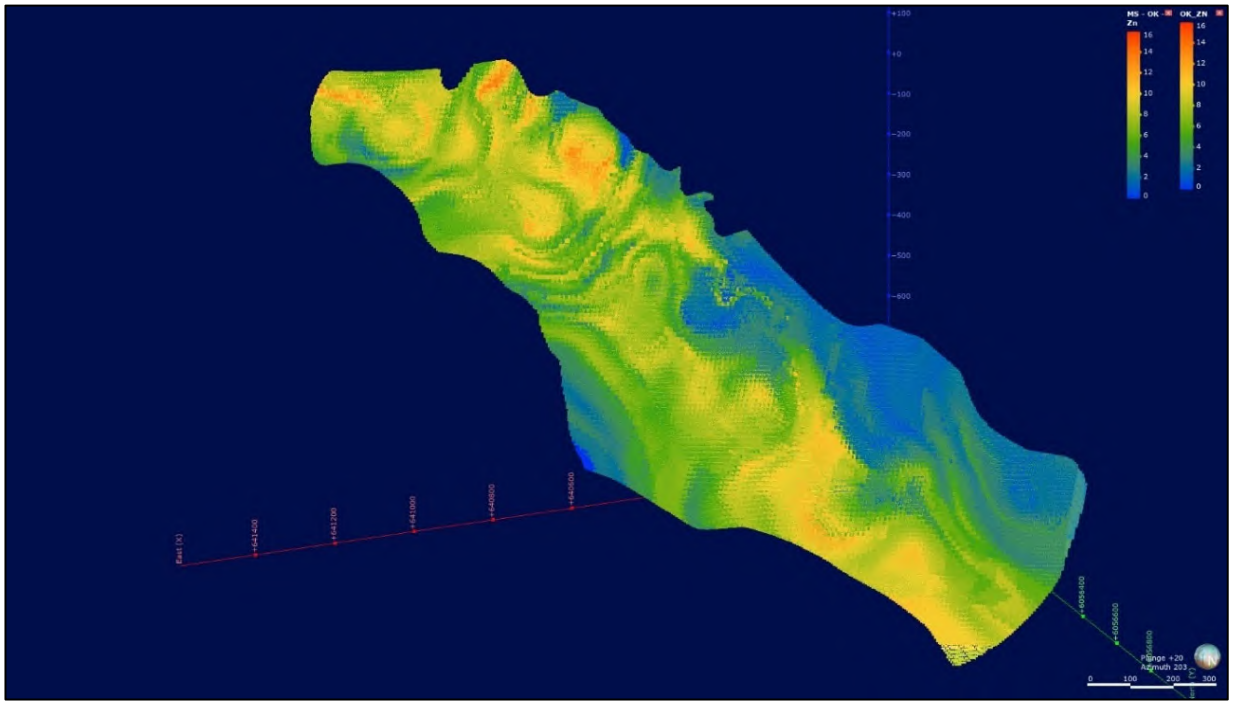
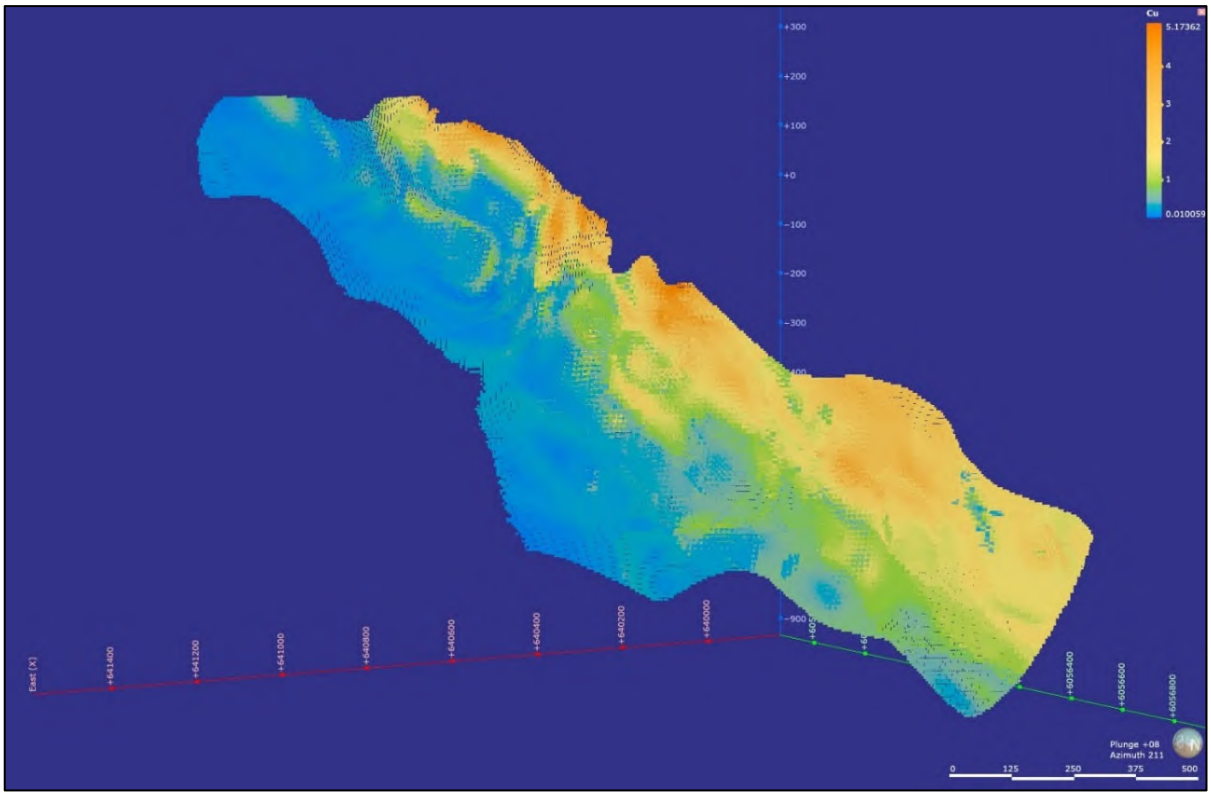


Figure 14-23: Grade Intensity (Heat) Map of Copper Grades for the MS Domain in the Resource Model



A visual validation was conducted which indicated there was a good correlation between the estimated grade and composite and assay values. The visual validation also indicated that the assays are properly snapped to domain as shown in Figure 14-24 and Figure 14-25.

Figure 14-24: General Cross-Section Showing the MS Domain Zinc Block Grade vs Composite Grade

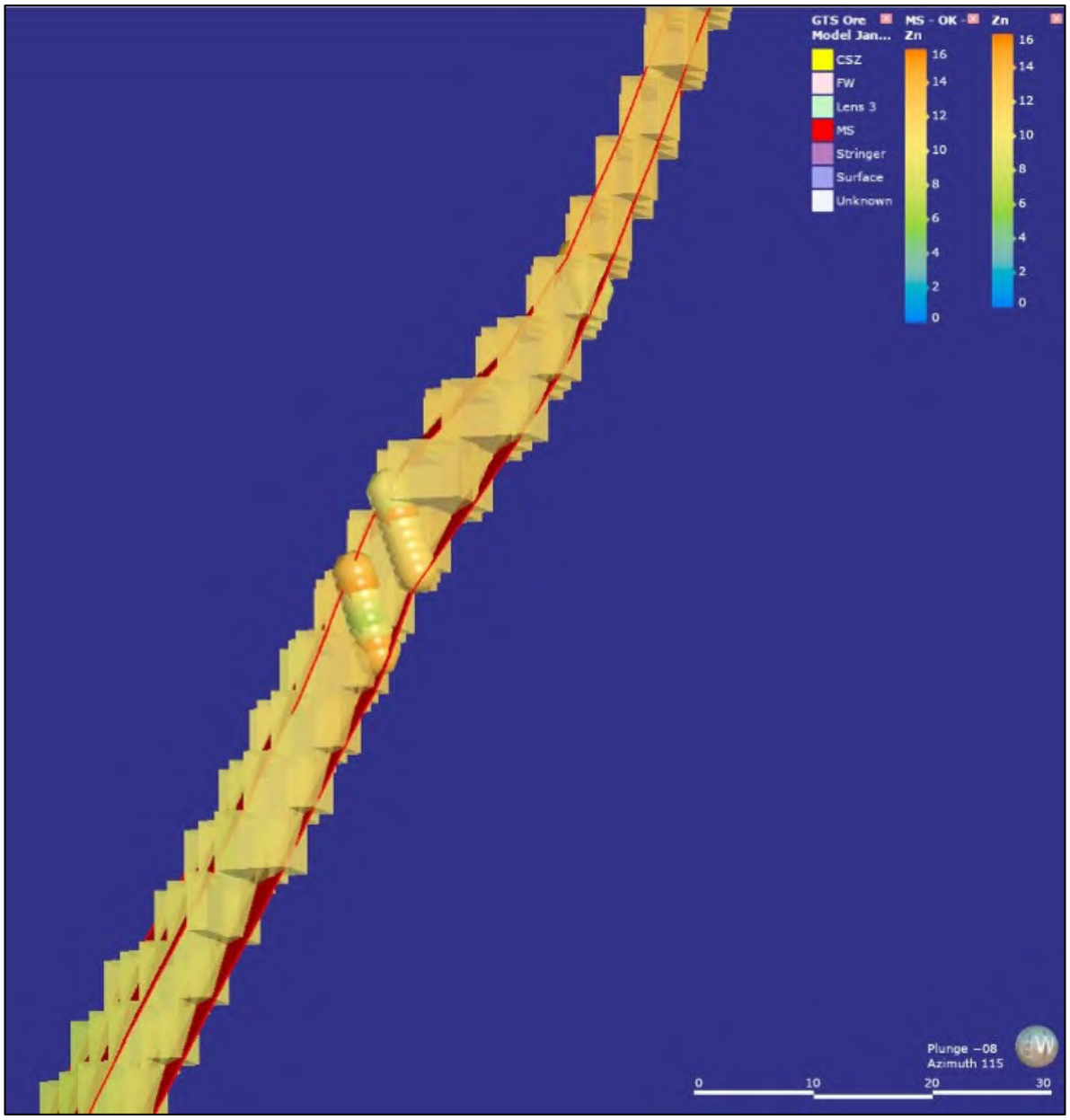
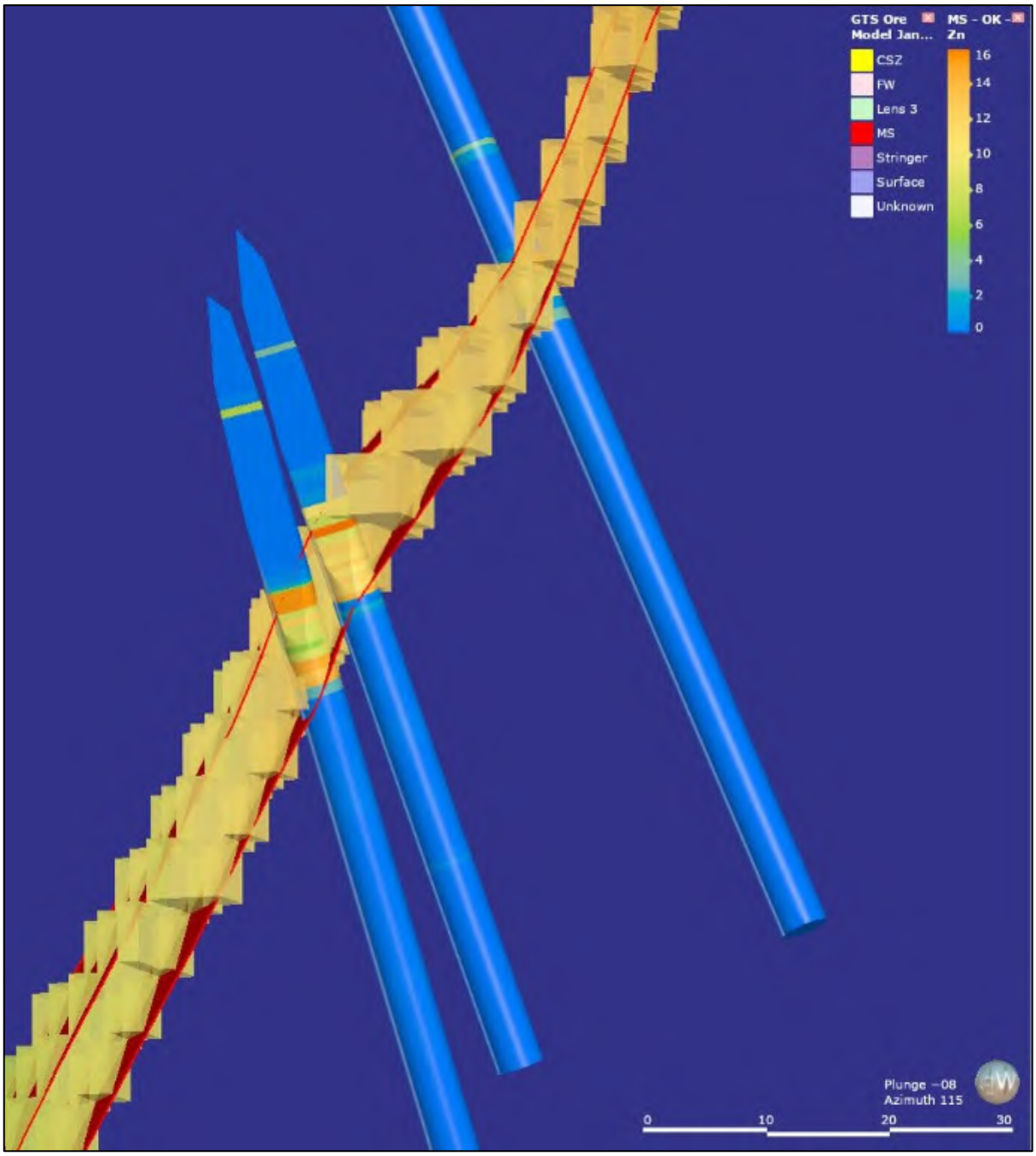


Figure 14-25: General Cross-Section Showing the MS Domain Zinc Grade vs Raw Assays



Swath plots are used as part of the validation process to show slices through the block model, usually in three different orientations (parallel, orthogonal, and horizontal). The plots illustrate the correlation of input composite values for a given metal, with the output block estimates for the estimation methods used, such as NN, ID and OK. The quantity of data is also illustrated in the swath plot as a histogram. Swath plots are useful for identification of possible over or underestimation, as well as the

degree of smoothing. Figure 14-26 presents the cross-sectional swath plot of the CSZ domain block model with Cu composite values, and the NN, ID^2 and OK block estimates. Figure 14-27 shows similar data for the MS domain using Zn composite values and estimation results. In both cases, swath plots across the block model show good correlation between the different estimation methods (ID^2, OK) and the declustered composite grades. There is no evidence of global or local over or underestimation, and the degree of smoothing is acceptable for the selected final OK estimate.

Figure 14-26: Northing Swath Plot of the Copper Estimate in the CSZ Domain

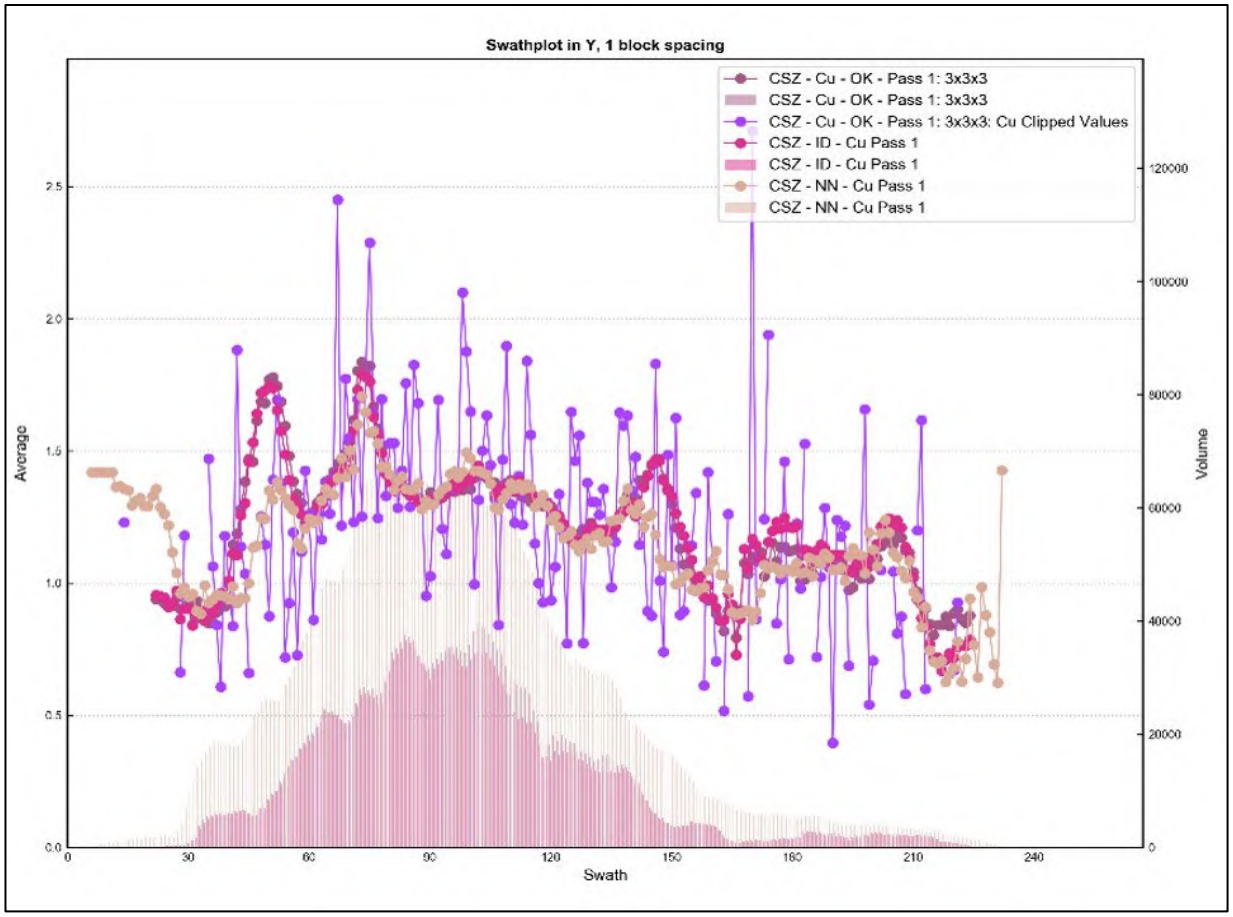
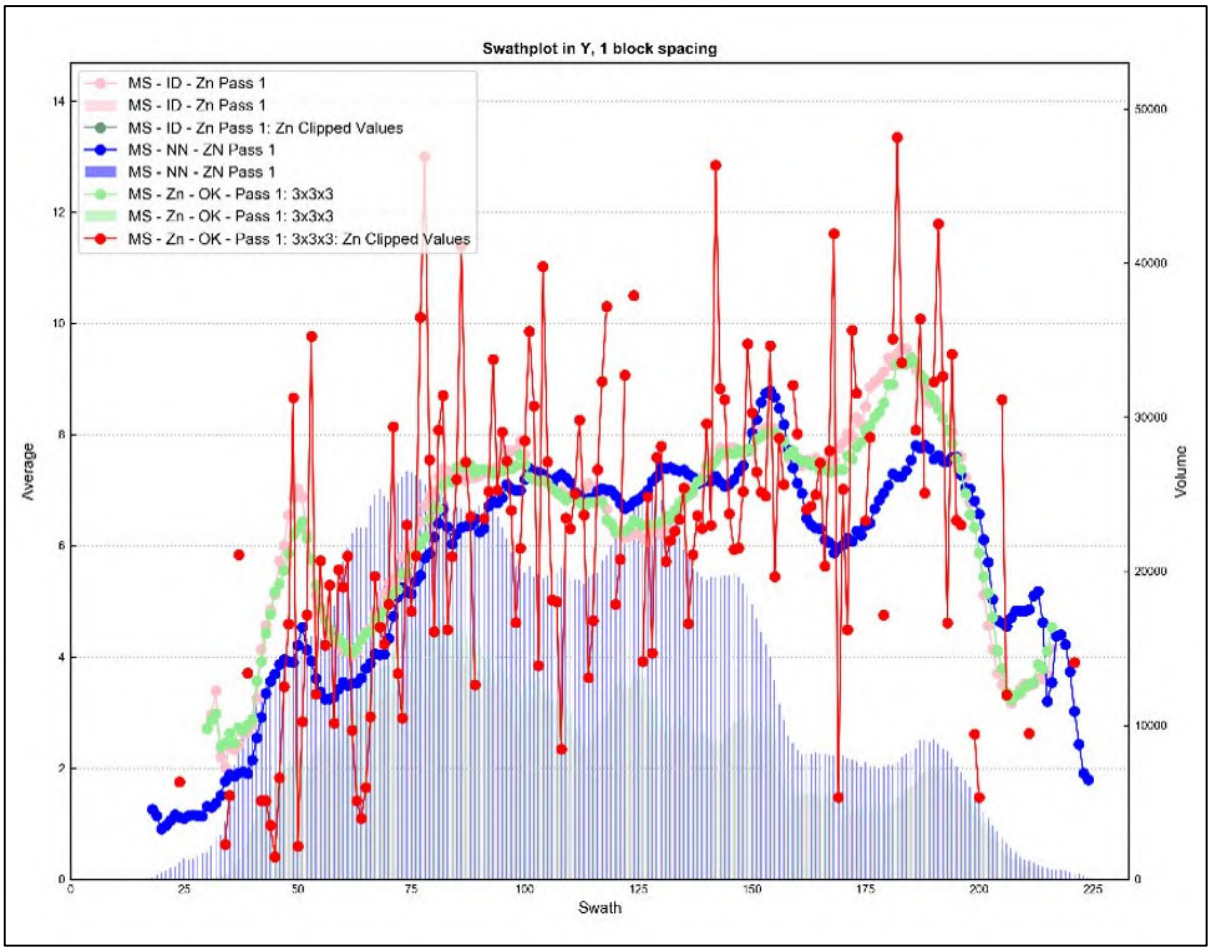


Figure 14-27: Northing Swath Plot of the Zinc Estimate in the MS Domain



14.8 Mineral Resource Classification

A preliminary assessment of the resource classification was generated by observing the integer field generated from the 3-pass search estimate process. Blocks estimated during each successively less stringent criterion were assigned either 1 (more stringent, highest confidence), 2 (moderate confidence) or 3 (least stringent, lowest confidence). The pattern generated by this automated process was visually reviewed against other information such as drill hole spacing and slope of regression.

Passes 1st and 2nd passes represent Indicated and 3rd pass Inferred resources. As the search ellipse passes produce a patchy distribution of blocks in various resource categories, the final classification is produced using hand-digitized shapes. A 3D polyline is drawn for each domain to encompass areas of contiguous material having the approximately the same measure of confidence based on statistical and geological criteria. The process was repeated for the Indicated and Inferred material to ensure that the classification is smooth and that extrapolation distances are reasonable.

Figure 14-28 through Figure 14-30 show the progression for the estimated blocks through Pass 1 and Pass 2 followed by the final groomed version of the resource classification.

Figure 14-28: Blocks Estimated During the Restrictive Pass 1 Estimation Process

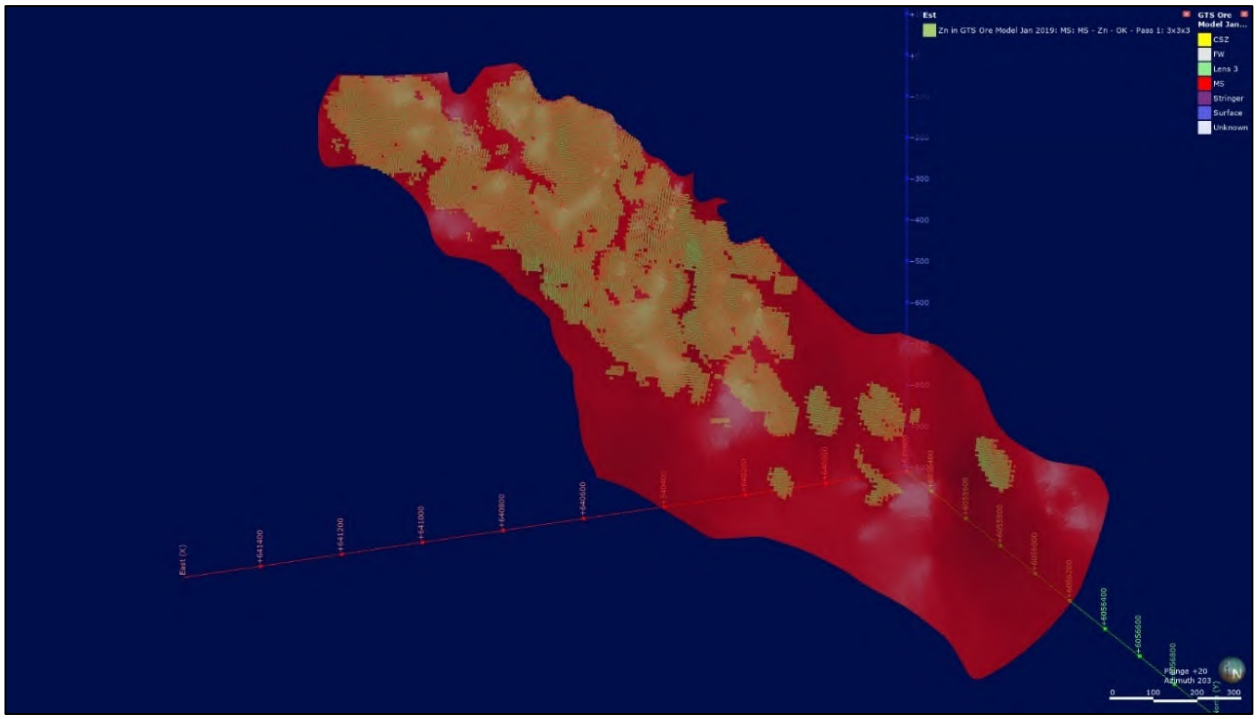


Figure 14-29: Blocks Estimated During the Pass 1 (Orange) and Pass 2 (Red) Estimation Process

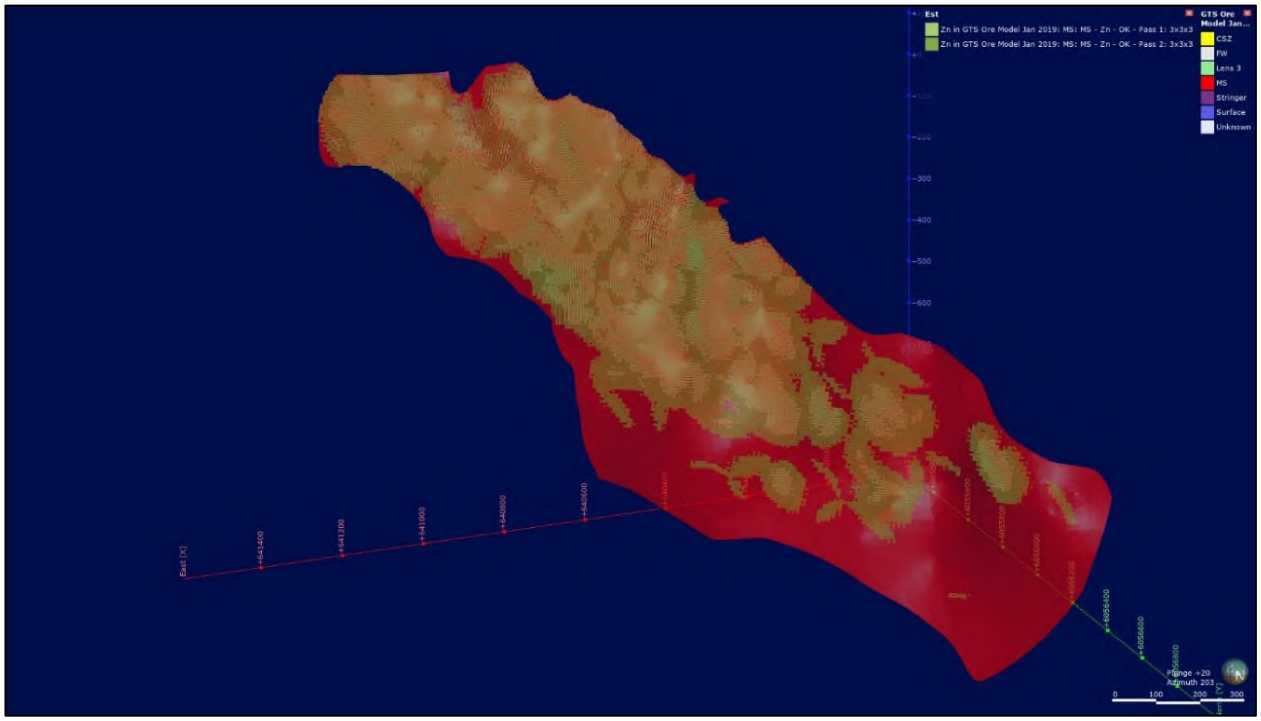
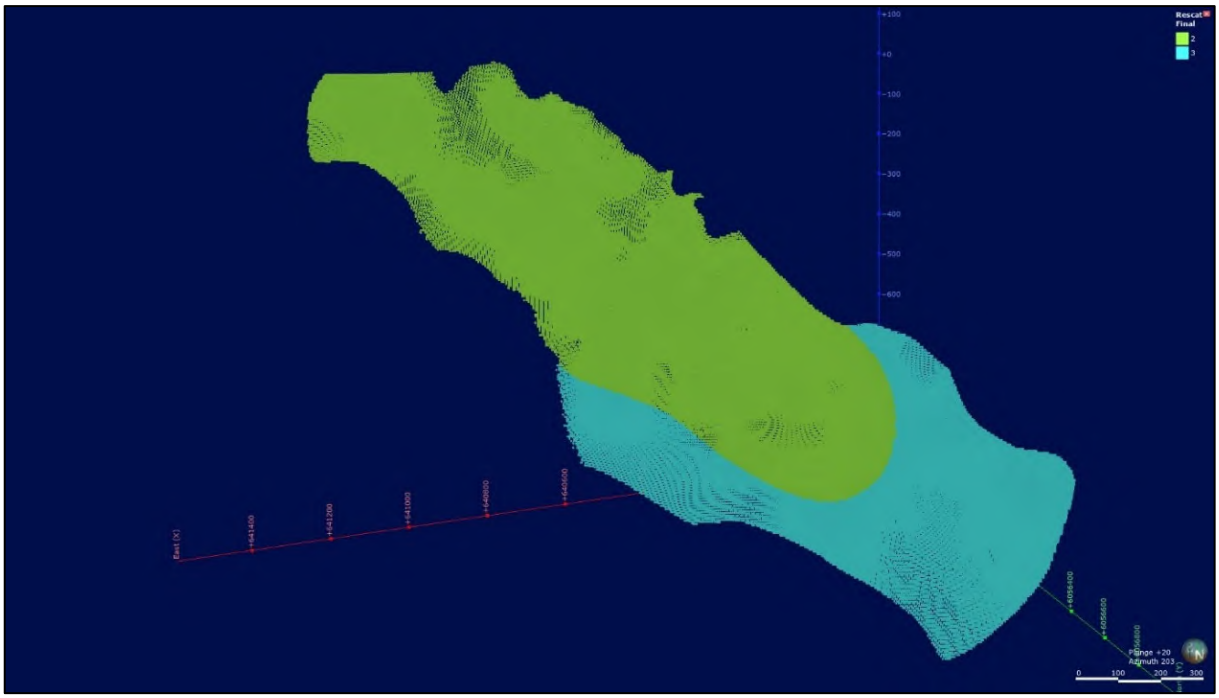


Figure 14-30: Final groomed Resource Classification: Indicated (Green) and Inferred (Turquoise)



14.9 Cut-Off Grade Criteria

Due to the multi-element nature of the of the McIlvenna Bay deposit an NSR value was used for the application of a cut-off to the block model. The NSR was estimated for each block using provisions for metallurgical recoveries, smelter payables, refining costs, freight, and applicable royalties (Table 14-11). Plant recoveries were based on the results of metallurgical test work conducted during the 2013 Preliminary Economic Assessment study. The smelter terms and freight costs were estimated by Foran. Metal prices used for the Mineral Resources were based on consensus, long term forecasts from banks, financial institutions, and other sources. The calculation was based on the assumption that two products, a copper, and a zinc concentrate, would be produced by a processing facility at site. The massive sulfide is split into Cu/Pb ratio greater than 1.2 and less than 1.2 as it is expected that Cu recovery will be significantly reduced where the ratio of Cu:Pb is less than 1.2.

Table 14-11: Mineral Resource Estimate NSR Parameter for the Cut-off Grade Assumption

	Descriptions	Metal	Domains		
			CSZ	MS Cu/Pb>1.2	MS Cu/Pb<1.2
Metallurgical Recoveries	Copper Conc	Copper	94%	83%	56%
		Zinc	34%	10%	2%
		Silver	77%	50%	34%
		Gold	85%	60%	39%
		Lead			59%
	Zinc Conc	Zinc		85%	85%
		Silver		27%	27%
		Gold		15%	15%
Metal Prices	Copper	US$/lb		$3.30	
	Zinc	US$/lb		$1.25	
	Silver	US$/oz		$16.20	
	Gold	US$/oz		$1,310	
	Lead	US$/lb		$1.00	
Smelting and Refining:	Copper	US$/dmt		$90.00	
	Zinc	US$/dmt		$215.00	
Transport	Copper	US$/dmt		$188.00	
	Zinc	US$/dmt		$97.00	

Cut-off was established from preliminary mining and operating cost. As a preliminary NSR calculation metal recovery were used to establish distinct metal multiplier for the CSZ and MS domain. Those same formulas were applied based on Zn and Cu content to the other domains.

The following NSR formulas were used for the MS, Lens 3, Stringer and FW domains:

- CSZ Domain
  - NSR = (Ag*0.34) + (Au*33.47) + (Cu*55.71)
- MS – Lens 3 – Stringer – FW Domains
  - NSR = Cu/Pb <= 1.2 -> NSR= (IF Zn >= 1.5% -> Zn*15.46 or IF Zn < 1.5% -> Zn *0) + (Ag*0.12) + (Au*15.47) + (Cu*2.46) + (Pb *10.5)
  - NSR = Cu/Pb > 1.2 -> NSR = (IF Zn >= 1.5% -> Zn*15.10 or IF Zn < 1.5% -> Zn *0) + (Ag*0.26) + (Au*26.56) + (Cu*46.69)

Foran has chosen to report the Mineral Resources at a higher cut-off value of US$60/t in order to be closer to the criterion used by other current and planned mining operations in the region.

14.10 Mineral Resource Estimate

14.10.1 Mineral Resource Estimate

The Mineral Resource Estimate reviewed and audited by Micon and its QPs is Summarized in Table 14-12. The effective date of this Mineral Resource is as of May 07, 2019 and is reported at using an NSR cut-off grade of US $60/t.

Table 14-12: Mineral Resources for the McIlvenna Bay Deposit, Reported at an NSR of US$ 60/t

NSR Cut-Off	Classification Category	Mineralized Domain (Zone)	Tonnage (Mt)	Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)
US $60/t (Base Case)	Indicated	Main Lens – Massive Sulphide	9.25	0.90	6.43	0.40	0.52	25.97
		Lens 3	1.99	0.85	3.29	0.14	0.27	14.71
		Stringer Zone	0.70	1.38	0.62	0.04	0.35	13.34
		Copper Stockwork Zone	10.30	1.43	0.28	0.02	0.40	9.30
		Copper Stockwork Footwall Zone	0.71	1.60	1.04	0.04	0.54	11.47
		Total	**22.95**	**1.17**	**3.05**	**0.19**	**0.44**	**16.68**
	Inferred	Main Lens – Massive Sulphide	2.97	1.29	4.79	0.29	0.47	23.58
		Copper Stockwork Zone	8.18	1.42	0.76	0.03	0.47	11.63
		Total	**11.15**	**1.38**	**1.83**	**0.10**	**0.47**	**14.81**

The Mineral Resources presented here were reviewed and audited by Micon's QPs using the CIM Definitions and Standards on Mineral Resources and Reserves as of May 10, 2014. Mineral Resources unlike Mineral Reserves do not have demonstrated economic viability. At the present time, neither Micon nor the authors of this report believe that the Mineral Resource Estimate is materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

14.10.2 Sensitivity Table

As part of its review and audit of Foran's 2019 Mineral Resource Estimate, Micon conducted a sensitivity to illustrate the sensitivity of the Mineral Resource to a higher and lower NSR. Table 14-13 summarizes the NSR sensitivity at US$75/t and US$45/t with the base case at US$60/t.

Table 14-13: Summary of the NSR Sensitivities at US$75/t, US$45/t with Base Case at US$60/t

NSR Cut-Off	Classification Category	Mineralized Domain (Zone)	Tonnage (Mt)	Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)
US$75/t	Indicated	Main Lens – Massive Sulphide	9.13	0.91	6.46	0.40	0.52	26.05
		Lens 3	1.62	0.87	3.60	0.15	0.28	15.26
		Stringer Zone	0.42	1.50	0.71	0.04	0.38	13.59
		Copper Stockwork Zone	7.33	1.59	0.30	0.02	0.47	10.29
		Copper Stockwork Footwall Zone	0.52	1.76	1.30	0.05	0.62	13.25
		Total	**19.02**	**1.21**	**3.58**	**0.22**	**0.48**	**18.44**
	Inferred	Main Lens – Massive Sulphide	2.92	1.30	4.81	0.29	0.47	23.60
		Copper Stockwork Zone	6.22	1.55	0.77	0.03	0.54	12.43
		Total	**9.14**	**1.47**	**2.06**	**0.11**	**0.52**	**16.01**
US$60/t (Base Case)	Indicated	Main Lens – Massive Sulphide	9.25	0.90	6.43	0.40	0.52	25.97
		Lens 3	1.99	0.85	3.29	0.14	0.27	14.71
		Stringer Zone	0.70	1.38	0.62	0.04	0.35	13.34
		Copper Stockwork Zone	10.30	1.43	0.28	0.02	0.40	9.30
		Copper Stockwork Footwall Zone	0.71	1.60	1.04	0.04	0.54	11.47
		Total	**22.95**	**1.17**	**3.05**	**0.19**	**0.44**	**16.68**
	Inferred	Main Lens – Massive Sulphide	2.97	1.29	4.79	0.29	0.47	23.58
		Copper Stockwork Zone	8.18	1.42	0.76	0.03	0.47	11.63
		Total	**11.15**	**1.38**	**1.83**	**0.10**	**0.47**	**14.81**
US$45/t	Indicated	Main Lens – Massive Sulphide	9.31	0.90	6.41	0.40	0.51	25.93
		Lens 3	2.23	0.84	3.07	0.13	0.27	14.31
		Stringer Zone	0.97	1.25	0.61	0.04	0.31	12.84
		Copper Stockwork Zone	12.12	1.34	0.27	0.02	0.36	8.74
		Copper Stockwork Footwall Zone	0.86	1.50	0.90	0.04	0.48	10.39
		Total	**25.49**	**1.14**	**2.79**	**0.17**	**0.41**	**15.72**
	Inferred	Main Lens – Massive Sulphide	3.05	1.26	4.74	0.30	0.46	23.48
		Copper Stockwork Zone	9.61	1.33	0.74	0.03	0.43	11.03
		Total	**12.66**	**1.31**	**1.70**	**0.09**	**0.44**	**14.03**

The 2013 estimation had two separate domains for what now comprises the MS domain. Historically UWZ and Lens 2 were always considered as a continuous geological unit with a Cu enrichment on the upper portion called UWZ. The analysis summarized in this report demonstrated that a hard internal boundary was not justified given the nature of VMS deposits to exhibit gradational metal zonation within the massive sulphide lenses. The use of a hard boundary within the massive area of the deposit, would have masked the transitional character of the grade variability, and falsely indicated a more clear-cut distinction between “Cu-rich” and “Zn-rich” mineralization.

15. MINERAL RESERVE ESTIMATES

This section summarizes the methodology and economic/technical parameters used in the preparation of a Mineral Reserve Estimate for the McIlvenna Bay Project, including commodity price assumptions, geotechnical considerations, NSR values, cut off grade calculation, mining recovery and dilution assumptions.

15.1 Introduction

The reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards, 2014. Mineral Reserves are sub-divided in increasing confidence into Probable and Proven Mineral Reserves, as described below:

- *A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve. The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.*
- *A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.*

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage and grade that is the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative, or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

The Reserve Estimate presented herein consists only of Probable Mineral Reserves and are those Resources in the Indicated category that have been calculated to be economically extractable after incorporation of the relevant Modifying Factors. The Reserve Estimate is effective as of February 17, 2020 and is a sub-set of the Mineral Resource Estimate (effective date of May 7, 2019) described in Section 14.

Table 15.1 below summarizes the Probable Mineral Reserve Estimate for McIlvenna Bay.

Table 15-1: Probable Reserve Estimate: McIlvenna Bay, March 2020.

	Probable Tonnes	Grade			
		Zn (%)	Cu (%)	Au (g/t)	Ag (g/t)
Massive Sulphide	7,773,176	5.71	0.88	0.51	25.24
Copper Stockwork Zone	3,566,067	0.31	1.70	0.60	11.65
Total	**11,339,243**	**4.01**	**1.14**	**0.54**	**20.97**

Notes:

1. Mineral Reserves have an effective date of February 17, 2020.
2. The Qualified Person for the estimate is Denis Flood, P.Eng.
3. The Mineral Reserves were estimated in accordance with the CIM Definition Standards for Mineral Resources and Reserves
4. The Mineral Reserves are supported by a detailed mine plan, based on a preliminary NSR cut-off value calculation. Inputs to the value calculation include:
    a. NSR Cut off value of US$100
    b. Metal prices of Zn US$1.25/lb, Cu US$3.30/lb, Au US$1310/oz and Ag US$16.20/oz
    c. Average total operating cost of $100/t, consisting of $62.5/t for mining, $31.0/t for processing and $6.5/t for G&A
    d. Metallurgical Recoveries of 81.1% Zn; 88.8% Cu:69.7% Au; and 56.8% Ag
    e. Smelter terms of US$90/t for Cu and US$215/t for Zn
    f. Concentrate transportation costs of US$188/t for Cu and US$97/t for Zn
5. The Mineral Reserve Estimate incorporates a mining recovery of 95% and dilution of 10% globally.
6. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.

15.2 Key Assumptions

A number of key assumptions are required to complete a mineral reserve statement for any mining project. For the McIlvenna Bay project, the key assumptions are as follows:

- continuity of the orebody between sublevels is consistent with the block model
- core samples selected for testing are representative of the rock quality and strength
- water inflows are consistent with hydrogeological modelling
- regulators will accept the assumption that Battery Electric Vehicle haulage trucks will require reduced airflow, compared to their diesel counterparts

15.3 Basis of Estimate

15.3.1 Dilution

Dilution, defined as subeconomic material that is mixed into the ore being mined during the mining process, is generally subdivided into internal and external dilution. Internal dilution is subeconomic material that must be mined with the ore because it is inside the designed stope boundaries and is the consequence of ore and stope geometry. External dilution is material which falls outside the immediate stope design and is a factor of geotechnical and mining conditions. The total dilution is the sum of the internal and external dilution.

For McIlvenna Bay, a Mine Stope Optimizer in Deswik was used to generate stope shapes based on the NSR cut-off above, the generated shapes are then interrogated over the block model encompassing the internal dilution. The primary stope optimizer parameters are outlined in Table 15-2 below.

The external dilution was calculated by expanding the optimised stope shapes into the hanging wall and footwall in accordance with recommendations generated by MD Engineering (now named RockEng). Hanging wall and footwall break assumptions were used to allow various levels of breakage according to ground strength and ground conditions. These expanded shapes were interrogated against the block model to calculate the minable inventory and the external dilution was back calculated.

Table 15-2: Stope Optimizer Parameters

Stope Parameter	Unit	Value
Length	m	20
Min. Width	m	3.8
Max. Width	m	25
Waste Pillar Width	m	5
Stope Height	m	30
Min. Stope Dip	degrees	55
Max. Stope Dip	degrees	135
Hanging Wall Dilution	m	0.7
Footwall Dilution	m	0.7

Dilution estimates were provided by MD Engineering and these are presented in Table 15-3 below. Economic trade off studies suggested that it was more economical to use HW cable bolting to manage dilution and to increase stability. The table below was used in the dilution assumptions for generating stopes.

For the reserve estimate, a global dilution of 0.7m was applied to all stopes to account for stope overbreak. This material is assumed to have zero grade. The result is that the dilution varies as a percentage on a stope by stope basis with the maximum being 18% and the global average being 10%.

Table 15-3: Dilution Analysis (MD Engineering, 2019)

ELOS Bin (m)		FW %	HW no Cables %	HW Cables %
Zone 1	2	2	15	11
	1	4	22	12
	0.5	21	28	30
	0	73	15	47
Zone 2	2	2	23	15
	1	15	32	20
	0.5	23	24	31
	0	60	21	34
Zone 3	2	15	53	26
	1	25	26	39
	0.5	27	19	26
	0	33	2	9

15.3.2 Mining Recovery

Mining recovery is defined as the portion of the optimised stope shapes that is recovered after allowing for blasting and mucking efficacy. Development, known as "T" drifts, were added to transverse stopes that have a thickness of less than 7.5m to increase mining recovery by facilitating better floors and space for LHD's to manoeuvre.

Recovery factors used in the reserve estimate are given in Table 15-4 below. For development, the recovery is 100% because the design drifts will overbreak, ensuring that the ore is extracted. The stoping recovery of 95% is based on good drill and blast technologies using computerized drills and electronic blasting caps for blasting. The stope geometries ensure good mucking recovery.

Table 15-4: Recovery Factors

Allowance Type	Factor
Stoping Recovery	95%
Development Recovery	100%

15.4 Cut-off Grade and NSR Calculation

A preliminary cut off value (COV) of US$100 per tonne was selected to determine the economic blocks within the resource model. Factors considered in determining the cut off value include:

- Metal Prices
- Payability (TC/RC)
- Operating cost estimate
- Metal recoveries

The costs and metal prices differ from those used within the final economic model for the project (described in detail within Section 21 and 22). This is due to refinements in operating philosophy that occurred after the commencement of mine planning activities. The operating costs used in the final financial analysis reflect the preferred operating scenarios and are generally lower than those used in the mine planning exercise.

Project revenues were calculated by coding an NSR to the block model. The NSR value was assigned in USD and was calculated using initial project provisions for metallurgical recoveries, smelter payables, refining costs, freight, and applicable royalties. These provisions, summarized in Table 15-5, were determined prior to finalization of economic parameters and in some cases differ slightly from the equivalent parameters used in the final project economic evaluation. These differences have been reviewed and are not considered to have a material impact on the overall project economics.

Metallurgical recoveries were based on initial performance models that in turn used the initial results of the 2019 metallurgical test work program. Provisional smelter terms and freight costs were estimated, and metal prices used for Mineral Reserves were based on consensus, long term forecasts from banks, financial institutions, and other sources. The calculation was based on the assumption that two flotation concentrate products (copper and a zinc concentrate), would be produced by a toll processing facility in Flin Flon.

Table 15-5: Mineral Reserve Estimate NSR Assumptions

Metallurgical Recoveries			
		Copper Stockwork	Massive Sulphide
Copper Conc	Copper	94%	83%
	Zinc	34%	10%
	Silver	77%	50%
	Gold	85%	60%
Zinc Conc	Zinc	-	85%
	Silver	-	27%
	Gold	-	15%

Metal Prices		
Copper	US$/lb	$3.30
Zinc	US$/lb	$1.25
Silver	US$/oz	$16.20
Gold	US$/oz	$1,310
Lead	US$/lb	$1.00

Smelting and Refining		
Copper	US$/dmt	$90.00
Zinc	US$/dmt	$215.00

Concentrate Transport		
Copper	US$/dmt	$188.00
Zinc	US$/dmt	$97.00

An NSR value was calculated for each block in the resource block model using the formulae in Table 15-6. To maximize project economics, a higher cut-off NSF value of US$100/t was selected by the QP's and Foran management. This approach avoids the inclusion of marginal (or near-marginal) blocks in the mine plan and tends to leave the reserve with lower LOM tonnes, but higher grades. The impact of higher mill fee grade includes higher metallurgical recovery and accelerated payback. Stopes were designed within the resource block model and the blocks with a collective value above or equal to the cut off NSR (US$100) were deemed economic and included in the mine plan.

Table 15-6: Mineral Reserve Estimate NSR Formulas

Domain	Formula
Copper Stockwork Zone (Cu > 0.175%)	56.60294 + (-112393000-56.60294) / (1+(Cu/0.000002296312)^1.318216) *Cu + 39.04+(10.13526-39.0435)/(1+(Cu/0.3796783)^1.767515)*Au + 0.3546588+(0.001517106-0.3546588)/(1+(Cu/0.2725457)^1.430114) *Ag
MS, Lens 3, Stringer FW (Cu > 0.175 and Zn > 1.5)	56.60294+(-112393000-56.60294)/(1+(Cu/0.000002296312)^1.318216)*Cu + 39.0435+(10.13526-39.0435)/(1+(Cu/0.3796783)^1.767515)*Au + 0.3546588+(0.001517106-0.3546588)/(1+(Cu/0.2725457)^1.430114)*Ag + 14.37575+(-4.398447-14.37575)/(1+(Zn/1.395387)^3.135888)*Zn

15.5 Stope Considerations

Stopes were designed in accordance with good practice and considerations were made for rock characteristics, orebody geometry, ore continuity, productivity, and cost. The longhole production methods proposed in the mine design are widely used in the industry and are well understood.

With a stope width of 20 meters, it was identified that LHD's would not be able to extract ore at the lateral extremities of narrow stopes due to the width of the LHD. For these stopes, "T" drifts are included along strike to ensure maximum mining recovery.

15.6 Conversion Factors from Mineral Resources to Mineral Reserves

15.6.1 Classification Criteria

Probable Ore Reserves are the part of Indicated, and in some circumstances, Measured Mineral Resources that can be mined in an economically viable fashion. It includes diluting material and allowances for losses which may occur when the material is mined. A Probable Ore Reserve has a lower level of confidence than a Proved Ore Reserve but is of sufficient quality to serve as the basis for decision on the development of deposit. In order for a stope to be considered Probable, more than 60% of the mineralized material must be Measured or Indicated. Probable Ore Reserves may also encompass areas which satisfy the requirements for Proven Ore Reserves in terms of the resource

category of contained mineralised material but are of lesser confidence due to any number of reasons including but not limited to metallurgical, geotechnical or economic uncertainty.

15.7 Factors That May Affect the Mineral Reserve Estimate

Areas of uncertainty or factors that could materially impact this Estimate of Mineral Reserves include, inter alia:

- changes to long-term metal price assumptions
- changes to CAD:USD exchange rate
- accuracy of block model
- accuracy of rock strength estimates
- variations in geotechnical, hydrogeological and mining assumptions
- changes to metallurgical recovery calculations
- variations in operating cost assumptions for mining, processing, and G&A
- local interpretations of mineralization geometry and continuity of mineralized zones
- marketability assumptions for the final product
- environmental, permitting and social license assumptions

16. MINING METHODS

The mine plan is predicated on the resource model described in Section 14.0 completed by Micon International on May 7, 2019. The mining method selected for the McIlvenna Bay deposit is a combination of transverse longhole stoping and Avoca longitudinal stoping. The orebody will be accessed by a decline, with trucks being used to haul ore to surface initially and a vertical conveyor to be commissioned later in the third year of production to transport ore to surface. The nominal production rate will be 3,600 tonnes per day, which is slightly lower than would be suggested by Taylor's law due to the narrow nature of the orebody. Backfill will be achieved with paste backfill utilizing filtered tailings. In the interest of minimizing stope cycle time, all transverse stopes are scheduled to be backfilled with paste, however secondaries could potentially be backfilled with development waste to reduce cement cost. Avoca stopes will be backfilled with development waste.

McIlvenna Bay will be an early adopter of Battery Electric Vehicles (BEV) haul trucks. At the time of writing, these trucks are a proven technology currently under use in at least two underground mines in Ontario with many companies investing in the technology.

16.1 Mine Design and Method

The mine design was optimized to maximize NPV by reducing waste development, delaying large capital items such as the vertical conveyor, and prioritizing early production. The level spacing of 30 meters reduces the overall development compared to a shorter sublevel interval.

16.1.1 Lateral Development

Lateral development will be achieved using conventional jumbo drill and blast methods. Typical ramp dimensions will be 5.0m wide by 5.5m high to accommodate the 50 tonne haul trucks. The round length will be 4m. The blasted material from development rounds will be mucked using LHD's and standard ground support will be installed using mechanized bolters. A summary of the development drift dimensions is shown in Table 16-1 below.

The footwall drifts will be offset 20m from the orebody with perpendicular cross cuts driven to access the ore. These cross cuts are driven every 20m, centre to centre, along strike to access transverse stopes. To access ore in the Avoca stopes, cross cuts have been designed every 60m along strike to ensure access for mucking, drilling, and filling Avoca fronts. The location of these accesses may vary based on the needs of the operation during production, a spacing of 60m was selected to ensure that they are included in the cost and schedule.

Table 16-1: Development Dimensions

Description	Height (m)	Width (m)
Cross Cut	4.5	4.5
Crusher	8.0	8.0
Electrical Substation	6.0	5.0
Footwall Drive	5.5	5.0
Fuel Bay	5.0	6.0
Level Access	5.0	5.0
Truck Loadout	7.0	5.0
Powder Magazine	6.0	5.5
Ore Drive	4.5	4.5
Ramp	5.5	5.0
Refuge Station	4.5	4.5
Remuck	6.0	5.0
Shop	6.0	6.0
Crane Bay	8.0	8.0
Storage	6.0	5.5
Sump	6.0	5.5
T drift	4.5	4.5
Truck Turnaround	5.0	5.0
Ventilation Access	4.5	4.5
Warehouse	6.0	8.0

16.1.2 Vertical Development

Vertical excavations will be driven by either raise bore or Alimak. Ventilation raises and the vertical conveyor raise will be raise bored and the secondary egress raise will be excavated using Alimak techniques.

Raise Bore

Raise bore will be used for exhaust raises at the eastern and western extremities of the orebody and for the vertical conveyor raise. Raise bore will be used as a pilot for ore silos, which will be slashed to achieve the final diameter. Exhaust raises will be excavated at an angle to reduce lateral development access drifts. Any excavations that intersect surface will be vertical to minimized cost and schedule impacts of excavating through muskeg. All capital raise bore excavations will be 3 m in diameter.

Alimak

Alimak will be used to excavate the second egress raise as this raise will need to be supported and a ladderway installed. The second egress raise will be used as an exhaust raise early in the mine life, before the permanent exhaust raises are available. It will then be converted to fresh air and will also be equipped with the propane burner for mine air heating during the winter.

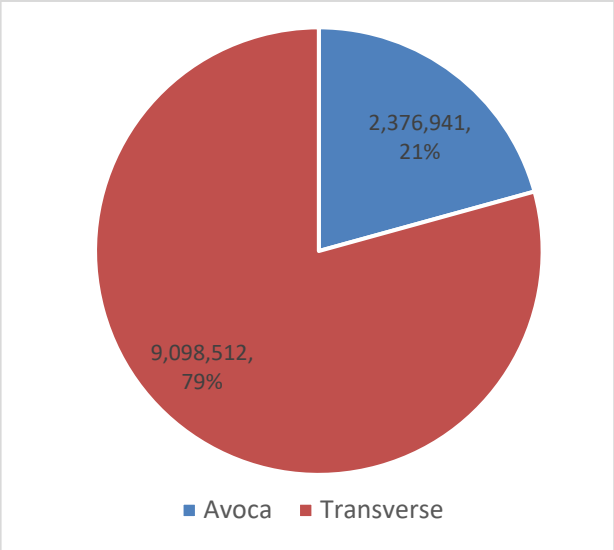
16.1.3 Level Spacing

A level spacing of 30m was selected as optimal as it reduces capital compared to a smaller sublevel interval and is within the range for conventional Tophammer drilling technology. The continuity and steepness of the orebody allows for a larger sublevel than many deposits.

16.1.4 Production Stoping

Ore at the McIlvenna Bay project will be produced by either longhole transverse stoping or longitudinal Avoca stoping, both longhole bulk mining methods. Most ore will be produced from the transverse stopes. The production breakdown by mining method is shown in Figure 16-1. The transverse sequence allows for a greater number of active workplaces resulting in a higher production rate whereas the Avoca fronts allow for earlier production from the plunge of the orebody on the western extremity. The cost for Avoca is lower than for transverse, largely due to the backfill method of CRF not requiring cement.

Figure 16-1: Breakdown of Production Tonnage by Mining Method

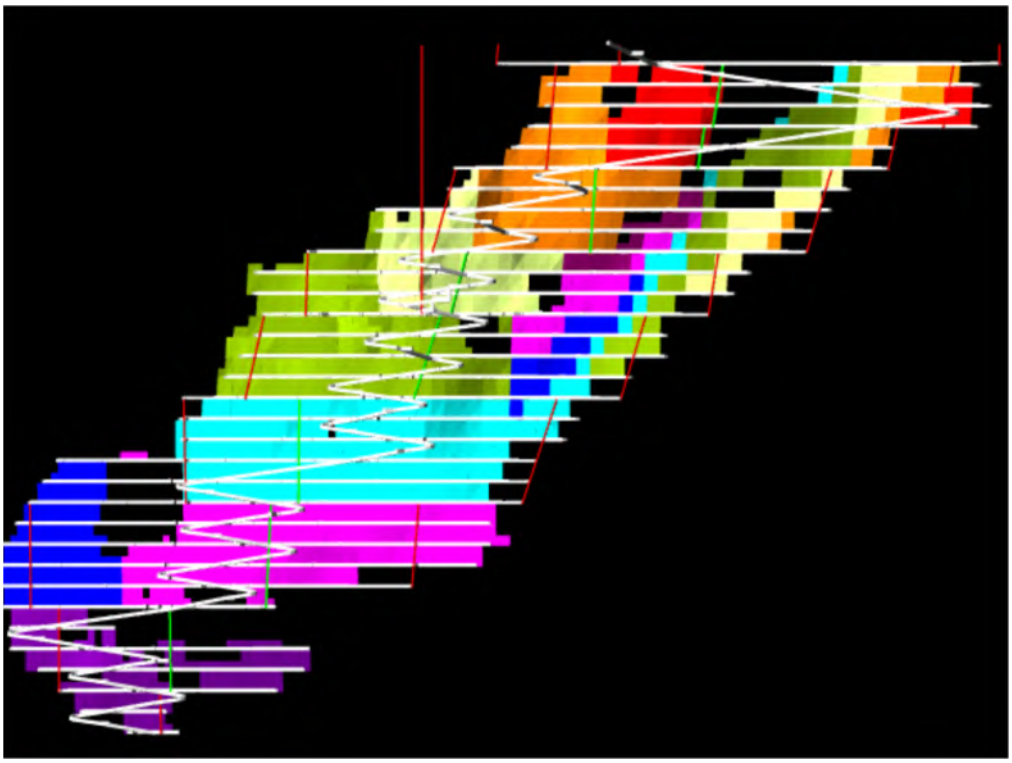


Horizontal sill pillars will be created to allow continuous mining at the name plate production rate such that new sills are established as previous horizons are depleted. The transverse production areas are divided into 8 distinct blocks and the Avoca stopes are mined such that two fronts are continuously maintained. The mine design is shown in Figure 16-2.

Transverse Longhole Open Stoping

The transverse longhole open stopes are access from the top crosscut for drilling, explosive loading, and backfilling and the bottom crosscut for mucking. The generic stope layout is shown in Figure 16-3. Stopes that are less than 7.5m thick will have a “T” drift excavated in the bottom crosscut to ensure LHD’s can muck around corners effectively. Initial void space will be created by excavating a slot raise using a raise bore machine and blastholes will be drilled using a top hammer drill rig. Blastholes will be loaded with emulsion.

Figure 16-2: Mine Design



The sequence for transverse stoping is generally from bottom up, starting in the centre and advancing up and to the west and east to form a pyramid. Some effort has been taken to start the sequence at each sill in the highest NSR material possible. The center, or lead panel, is where the sequence commences and as the lead panel advances upward, the next panel to the east and west can be mine, and so on, as shown in Figure 16-4.

Figure 16-3: Stope Layout

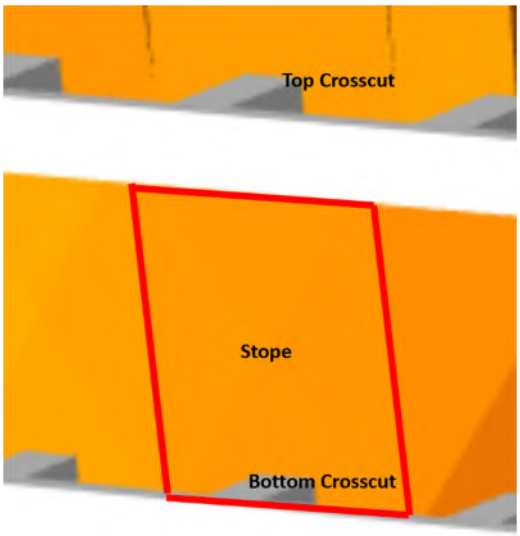
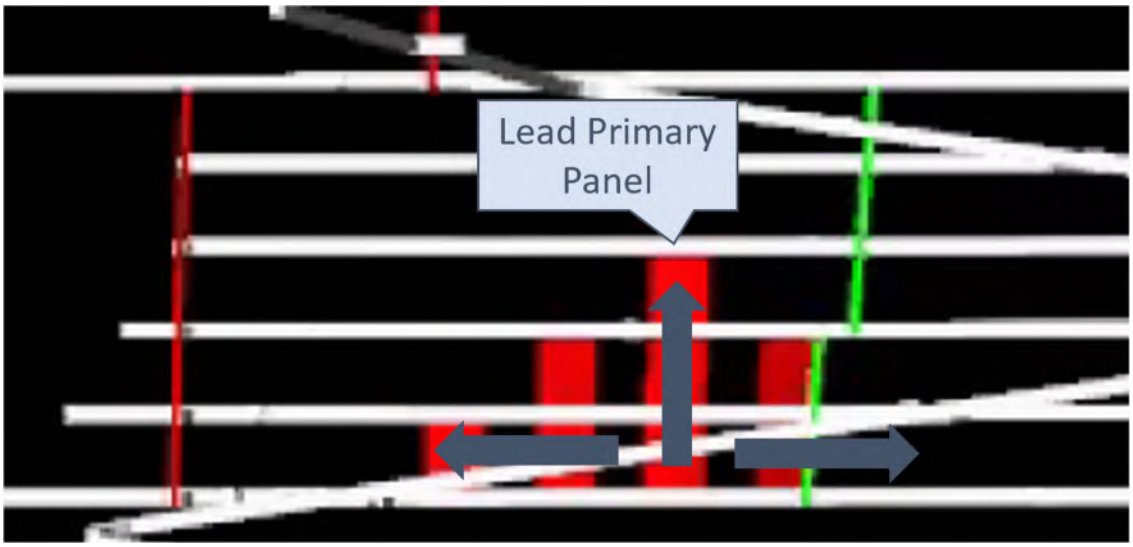
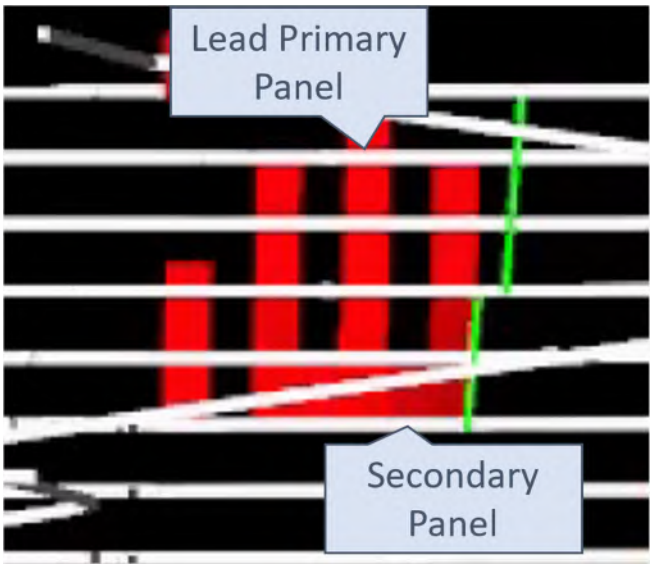


Figure 16-4: Primary Transverse Sequence



Once the primaries have advanced up three stopes for the lead panel and two stopes for the subsequent primary panel, the secondary sequence can commence, as shown in Figure 16-5.

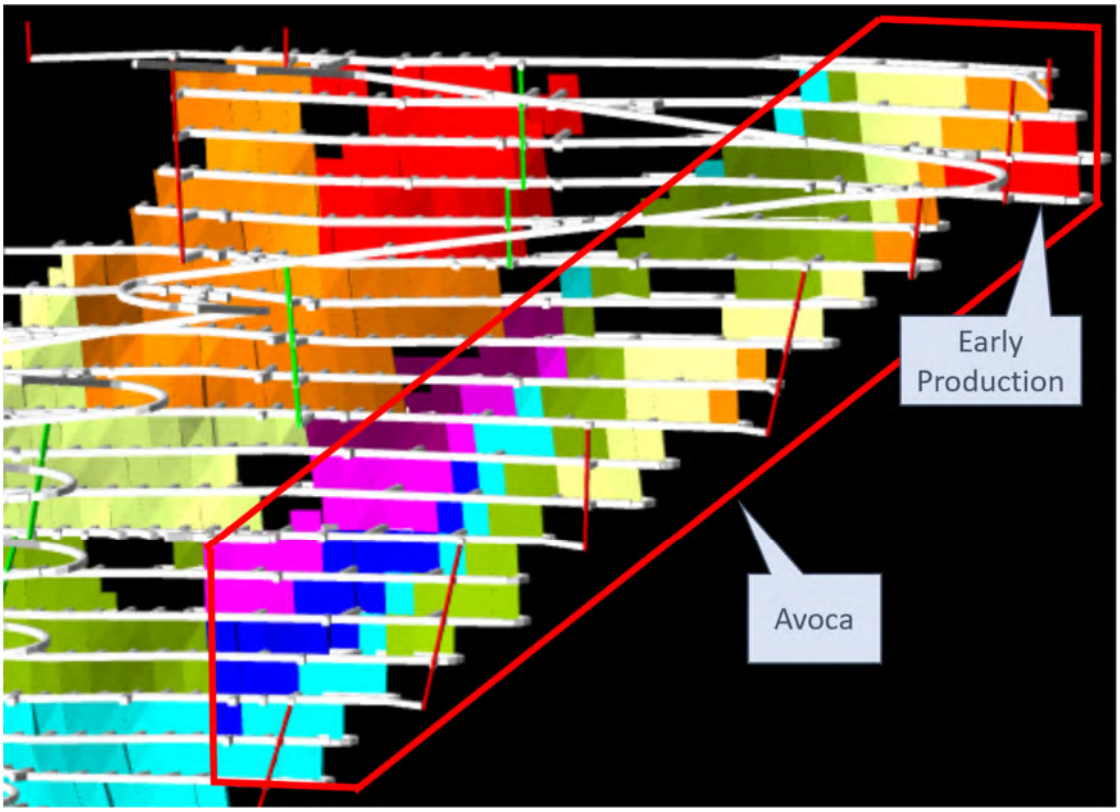
Figure 16-5: Secondary Sequence



Avoca Stoping

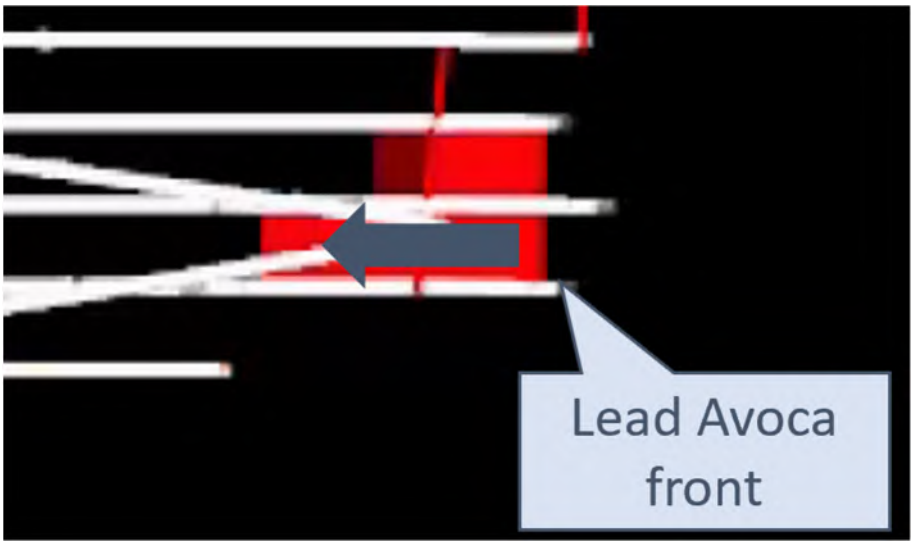
Avoca Stoping was selected for the eastern portion of the orebody where the sequence takes advantage of the plunge of the orebody to create an Avoca mining front very early in the mine life. The Avoca zone is shown in Figure 16-6. Avoca is a longitudinal retreat method where stopes are accessed from the east for drilling, explosive loading, and backfilling whilst mucking is accomplished from the west. In this way a front is created such that after each blast is taken the area will be filled, leaving a void at the angle of repose providing void for the next blast.

Figure 16-6: Avoca Zone



The Avoca front is initiated when the ramp provides access to the eastern plunge and the step downs required to maintain the advance is closely linked to the ramp advance. Since the Avoca stopes are filled with waste rock, a sill from which mining beneath can be done cannot be created.

Figure 16-7: Lead Avoca Front



The initial void for each Avoca front will be created using a Raisebore machine but due to the backfill/blasting cycle a raise is only required once per level. Drilling will be accomplished by a Tophammer drill and mucking will be done with diesel LHD.

16.1.5 Stope Optimization

Mine Planning for the McIlvenna Bay projects was completed using Deswik. Deswik Stope Optimizer (DSO) was used to create stope shapes based on the design parameters, shown in Table 16-2. Inferred resource was attributed with NSR = 0 to ensure that no stopes from Inferred resource were evaluated. The resulting optimized stope shapes were used to develop the mine plan.

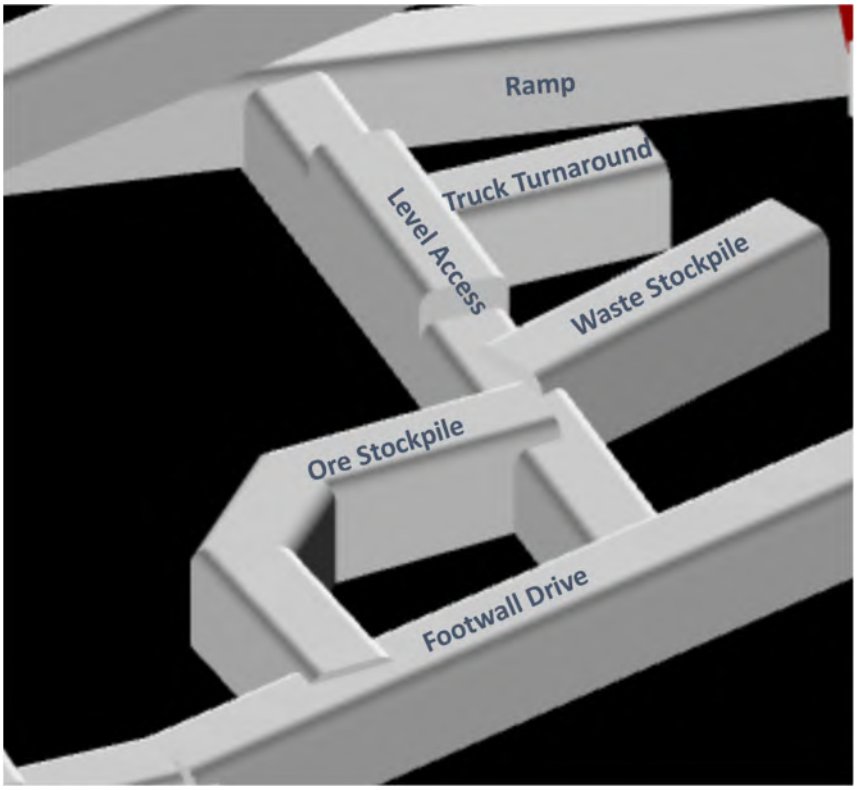
Table 16-2: DSO Parameters

Parameter	Value
Height	30m
Strike Panel Length	20m
Strike Sub-panel Length	10m
Uppers Height	10m
Minimum Planned Mining Width	3.8m
HW Dilution	0.5m
FW Dilution	0.5m
Minimum Diluted Mining Width	4.5m
Minimum Transverse Pillar Width	7m
Maximum Stope Thickness	25m
Minimum Dip	55 degrees

16.2 Material Handling

Initially, ore and waste will be transported to surface using BEV haul trucks. Each level has a waste stockpile, an ore stockpile, and a truck loading area as shown in Figure 16-8. Waste from development headings will be stockpiled in the waste stockpile until it can be loaded into a truck and transported to surface or to an Avoca stope that is being backfilled. Ore from stopes or development headings that are in ore will be stockpiled in the ore stockpile until it is loaded into a truck and transported to surface. Ore can be loaded into the stockpile from one side and be unloaded from the other side to separate the production LHD from the LHD dedicated to the truck loading to remove the potential for collisions.

Figure 16-8: Level Stockpiles and Truck Loading



Once the vertical conveyor is commissioned in year 3, trucks will dump at the underground crusher in preference to hauling to surface, although surface crushing facilities will remain available for periods in which the underground crushing system is unavailable.

The BEV trucks will be equipped with the ability to self-change batteries, so extensive infrastructure such as a crane is not required. This also allows battery charging stations to be moved as needed. During the first three years of the mine life all the batteries will be charged on surface. There will be two batteries for each truck and the charge time is less than the battery life so the trucks can dump the ore or waste at appropriate location on surface and change batteries before heading back underground. Once the vertical conveyor is commissioned, the charging stations will need to be relocated as required. These can be located in remuck bays that are no longer in use.

16.2.1 Haulage Productivity

Simulation data was provided by Sandvik based on the productivity for a single truck based on the longest haul distance (hauling waste from the bottom of the mine to surface). A summary of the simulation is shown in Figure 16-9.

Figure 16-9: Sandvik Haul Truck Simulation Summary

Mine Variables

Description		Value	Units
Mine speed limit	speed_limit	20	kph
Effective shift time / seat time	shift_hours	10	h
Fixed cycle time for loading, dumping, turning ε	idle_time	9	min
Mass of Load in the truck	load	45	tonnes
Shifts per day	shifts_per_day	2	shifts
Hauling target in single shift	shift_goal_tonnes	643	tonnes
Is the load being carried out or back	hauling_out	☑	
Availability	availability	85%	
Days per year	working days per year	352	days
Electricity Cost	electricity_cost	$0.07	$/kWh

Vehicle Parameters

Description	Name	Value	Units
Max Machine Speed	speed_max	20	kph
Charge Capacity of the vehicle's battery	battery_energy	353	kWh
Battery usable capacity upper limit	upper_lim	95	%
Battery usable capacity lower limit	lower_lim	10	%
Battery discharge power limit	discharge_lim	550	kW
Battery charge power limit	charge_lim	300	kW
Auxiliary Power Draw	power_aux	9.8	kW
The weight of the vehicle	vehicle_weight	48	tonnes
Time to swap a battery	swap_time	6	min

The cycle times used in the study were based on a speed limit of 10 km/h as 20 km/h is seen as too fast in a ramp that is being used by other traffic due to the increased potential and severity of collisions.

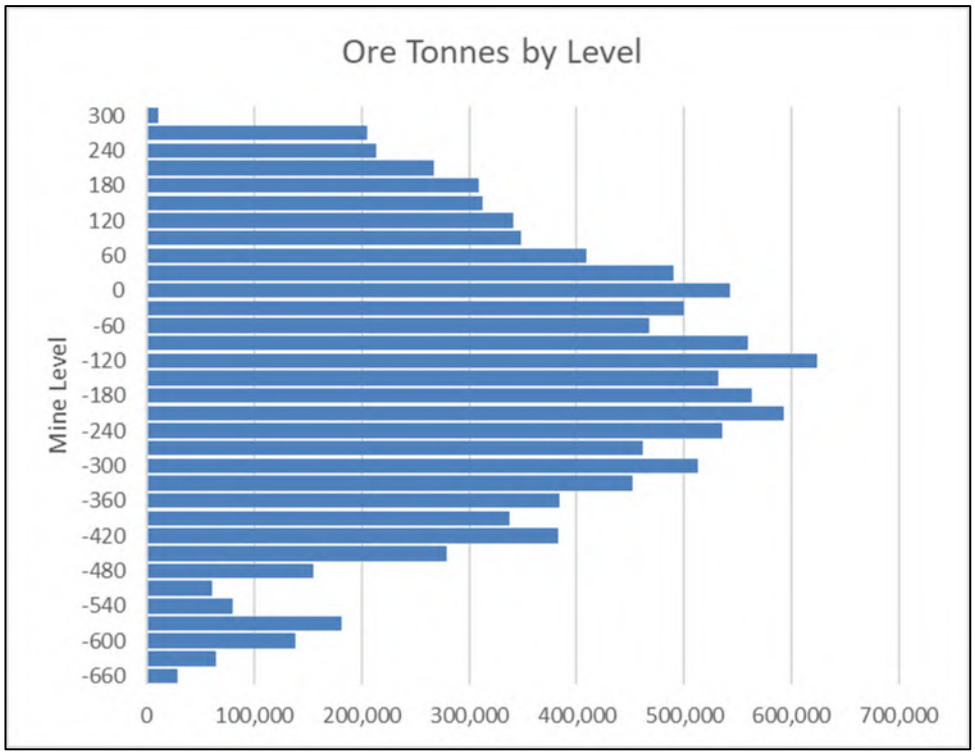
Using this data, and adjusting for, Utilization (79%), Congestion (20%), and contingency (30%) the fleet size was calculated for ore and waste movement. The resulting cycle time for ore haulage from the level stockpile to the dump point is shown in Table 16-3.

Table 16-3: Truck Haulage Cycle Times for Ore

Mine Level	Dump Location	Two Way Haul Distance (m)	Cycle Time (minutes)
300	Surface	1629	24.77
270	Surface	2057	27.34
240	Surface	2486	29.91
210	Surface	2914	32.49
180	Surface	3343	35.06
150	Surface	3771	37.63
120	Surface	4200	40.20
90	Surface	4629	42.77
60	Surface	5057	45.34
30	Surface	5486	47.91
0	UG rockbreaker	240	16.44
-30	UG rockbreaker	314	16.89
-60	UG rockbreaker	529	18.17
-90	UG rockbreaker	743	19.46
-120	UG rockbreaker	957	20.74
-150	UG rockbreaker	1171	22.03
-180	UG rockbreaker	1386	23.31
-210	UG rockbreaker	1600	24.60
-240	UG rockbreaker	1814	25.89
-270	UG rockbreaker	2029	27.17
-300	UG rockbreaker	2243	28.46
-330	UG rockbreaker	2457	29.74
-360	UG rockbreaker	2671	31.03
-390	UG rockbreaker	2886	32.31
-420	UG rockbreaker	3100	33.60
-450	UG rockbreaker	3314	34.89
-480	UG rockbreaker	3529	36.17
-510	UG rockbreaker	3743	37.46
-540	UG rockbreaker	3957	38.74
-570	UG rockbreaker	4171	40.03
-600	UG rockbreaker	4386	41.31
-630	UG rockbreaker	4600	42.60
-660	UG rockbreaker	4814	43.89

Applying the cycle time to the number of tonnes that are required to be moved from each level yields the trucks required. The ore by level is shown in Figure 16-10.

Figure 16-10: Ore Tonnes by Level



The Tonnes x Kilometers (TKM's) is a common metric used for ore trucking to show the overall truck requirements. The truck hours presented in Figure 16-11 do not include adjustments for utilization, availability, or congestion but provide an overview of the material movement over time. There is a dip in the TKM in years 5 and 6 after the vertical conveyor is commissioned and the haul distances become much shorter. This gradually increases as the mine gets deeper. The TKM's are based on the one-way haul distance.

The total material movement over the life of mine is shown in Figure 16-12. The figure shows the ore and waste that is hauled to surface, the ore that is hauled to the rock breaker, and the waste that is moved by trucks internally to be used as backfill in the Avoca stopes. The ore hauled to surface will be stockpiled at the mill for processing and the waste will be stockpiled. The ore hauled to the rock breaker will be transferred to the underground crusher and hoisted to the surface ore stockpile at the mill by the vertical conveyor. Ore being produced from levels above the rock breaker were assumed to be hauled to surface, even after the vertical conveyor commissioning, to avoid congestion. This can be validated through simulation and if not destructive to productivity the ore produced from these levels in years 4, 5, 7, and 8 should be hauled downramp to the rock breaker as this will allow the BEV haul trucks to perform better.

Figure 16-11: Underground Truck Hours and TKM Requirements

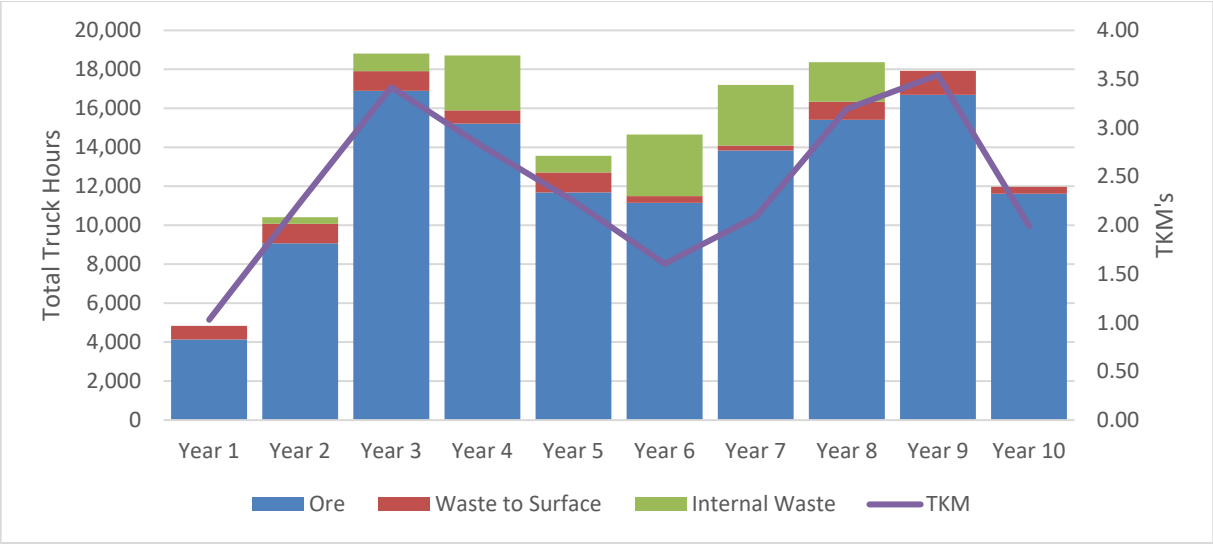
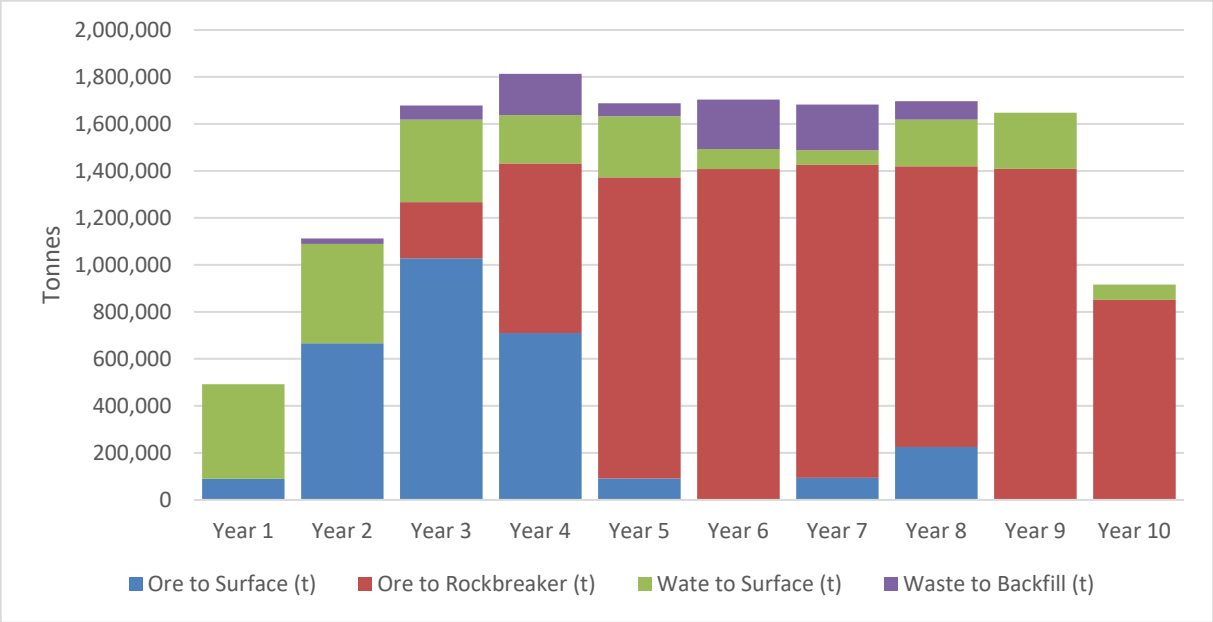


Figure 16-12: Total Material Movement



Underground waste that will be placed as backfill is not generally moved by trucks as sufficient waste is being generated on levels that are being backfilled and can be transferred by the LHD that is mucking the development headings. Where there is insufficient waste available on a level, waste will be trucked from another level by trucks. This internal waste movement is labelled as “waste to backfill” in Figure 16-12 and summarized in Table 16-4.

Table 16-4: Internal Waste Trucking

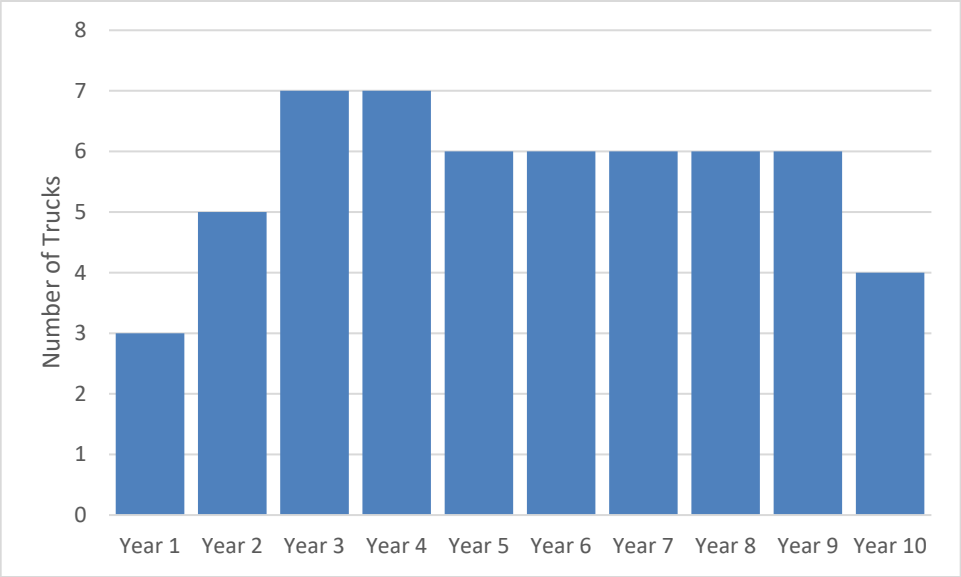
Year	Haul From	Haul To	Tonnes	Two Way Haul Distance (m)	Cycle Time (minutes)
Year 1	N/A	N/A	0	0	N/A
Year 2	30	300	23,054	4057	39.34
Year 3	-30	300	28,075	4914	44.49
Year 3	-30	270	14,308	4486	41.91
Year 3	-30	240	8,205	4057	39.34
Year 3	-30	210	1,495	3629	36.77
Year 3	-30	120	4,381	2343	29.06
Year 3	-30	90	2,550	1914	26.49
Year 3	-30	60	1,241	1486	23.91
Year 4	-150	180	47,359	4914	44.49
Year 4	-150	150	24,440	4486	41.91
Year 4	-150	120	2,982	4057	39.34
Year 4	-150	90	8,659	3629	36.77
Year 4	-150	60	17,351	3200	34.20
Year 4	-120	300	3,201	6200	52.20
Year 4	-120	270	8,732	5771	49.63
Year 4	-120	240	24,873	5343	47.06
Year 4	-120	210	37,733	4914	44.49
Year 5	-300	300	4,688	8771	67.63
Year 5	-300	270	5,315	8343	65.06
Year 5	-300	240	2,288	7914	62.49
Year 5	-300	150	4,670	6629	54.77
Year 5	-300	60	3,244	5343	47.06
Year 5	-300	-30	3,583	4057	39.34
Year 5	-300	-60	2,476	3629	36.77
Year 5	-300	-90	6,771	3200	34.20
Year 5	-300	-120	4,326	2771	31.63
Year 5	-300	-150	17,395	2343	29.06
Year 6	-450	-150	35,514	4486	41.91
Year 6	-450	-420	11,523	629	18.77
Year 6	-420	-120	61,001	4486	41.91
Year 6	-390	-60	19,724	4914	44.49
Year 6	-390	-90	51,410	4486	41.91
Year 6	-360	-60	11,271	4486	41.91
Year 6	-300	0	1,598	4486	41.91
Year 6	-300	-30	4,423	4057	39.34
Year 6	-300	-60	14,088	3629	36.77

(continued overleaf)

Year	Haul From	Haul To	Tonnes	Two Way Haul Distance (m)	Cycle Time (minutes)
Year 7	-450	-90	9,873	5343	47.06
Year 7	-420	-90	19,747	4914	44.49
Year 7	-390	-90	19,747	4486	41.91
Year 7	-360	-30	15,444	4914	44.49
Year 7	-360	-60	44,692	4486	41.91
Year 7	-360	-90	9,873	4057	39.34
Year 7	-330	0	39,562	4914	44.49
Year 7	-330	-30	15,444	4486	41.91
Year 7	-300	30	5,394	4914	44.49
Year 7	-270	60	9,532	4914	44.49
Year 7	-240	90	5,028	4914	44.49
Year 8	-600	30	14,585	9200	70.20
Year 8	-570	30	29,169	8771	67.63
Year 8	-570	60	8,553	9200	70.20
Year 8	-540	120	9,327	9629	72.77
Year 8	-540	90	16,952	9200	70.20
Year 9	N/A	N/A	0	0	N/A
Year 10	N/A	N/A	0	0	N/A

The total number of trucks required are shown in Figure 16-13. This includes hours to account for scheduled maintenance, availability, traffic congestion, and a BEV factor. Although BEV technology is proven and widely used in many applications around the world, the McIlvenna Bay project will be considered an early adaptor of BEV haul trucks in underground mining and therefore a "BEV factor" of 30% has been applied to the truck requirement calculations to account for any learnings that would result in lower productivity than indicated by the manufacturer.

Figure 16-13: Truck Requirements



16.2.2 Vertical Conveyor

During year 3, an underground crushing station and vertical conveying system will be installed and commissioned to allow transfer of ore directly to surface. Once commissioned, trucks will be able to dump ore into a coarse ore bin at level 0 m AMSL. The coarse ore bin will be protected with a static grizzly and hydraulic rock breaker for the handling of grizzly oversize material. This bin will be discharged in a controlled manner by a heavy duty vibrating grizzly feeder which feeds oversize (+125mm) material into the jaw crusher and bypasses undersize material onto the jaw crusher product conveyor.

Material discharged from the crusher, together with material passing through the vibrating grizzly feeder will be transferred a short distance into the fine ore bin by a heavy-duty conveyor, equipped with a self cleaning tramp removal magnet and oversized drive. The fine ore bin will be discharged in a controlled manner by a pair of heavy duty “brute force” pan feeders via lined chute work onto a transfer conveyor equipped with a static tramp metal magnet. The transfer conveyor will discharge crushed material onto the tail end of the vertical conveyor as shown in Figure 16-14.

The vertical conveyor will reduce the number of required haul trucks, and therefore ramp congestion. The capital cost of the vertical conveyor is significantly less than the cost of additional haul trucks that would be required to sustain production from deeper levels in the mine as well. The ore flow system from the underground truck dump to the vertical conveyor is shown in Figure 16-15.

Figure 16-14: Vertical Conveyor System Preliminary Layout

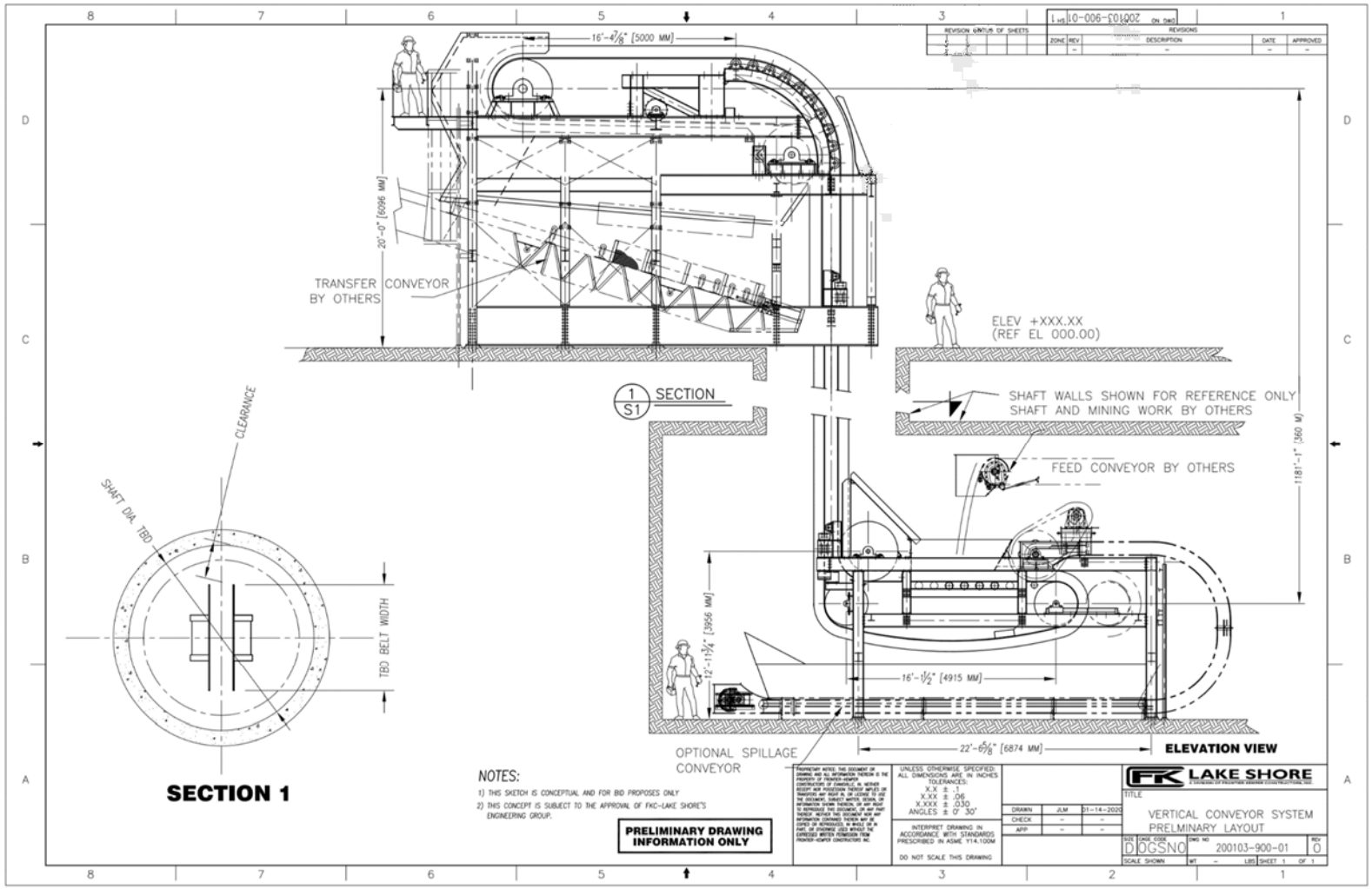
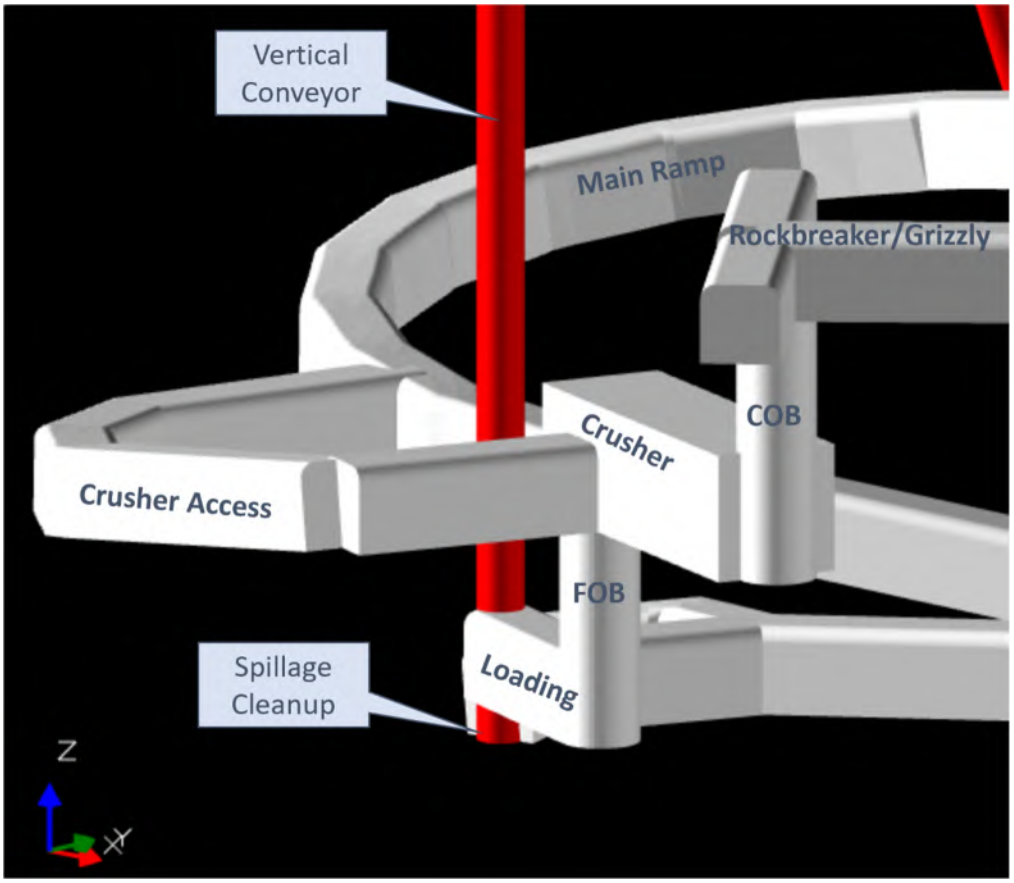


Figure 16-15: Underground Oreflow System Schematic



16.3 Geotechnical Considerations

RockEng (formerly known as MD Engineering) was engaged to provide a geotechnical analysis of the McIlvenna Bay deposit, and to assist with these aspects of the mine design.

16.3.1 Field Program

Two geotechnical core logging field programs were conducted by RockEng. The first field program was conducted in spring 2018 (March - April). This field program focused on collecting orientated geotechnical data from exploration drill holes. The second field program was conducted in the summer of 2018 (August – September). The second field program focused on collecting data to mitigate the blind zone, as well as collecting geotechnical data for the portal. In total 3,733m of core was logged, in which 151.2m was logged for the portal only.

16.3.2 Geotechnical Site Characterization

The McIlvenna Bay geotechnical site characterization has incorporated geological and geotechnical drill core logs and laboratory strength testing data. Detailed review and analyses of all available data has led to the following conclusions:

- There are in total, four apparent joint sets present:
  - Joint Set FO (foliation parallel jointing) – typically dipping 65° to 75 towards 020° (the orientation varies due to folding). This set is consistent throughout all domains except dolomite.
  - Joint Set B - 53°/125° (dip/dip direction). This is a weakly defined joint set.
  - Joint Set H – shallow dipping 5 to 25° with a dip direction ranging from south to west. In the dolomite this joint set is the bedding planes, and in all other rock units this set is interpreted to be associated with glacial relief and crenulation fractures.
- While no site-specific in situ stress measurements were carried out, in-situ stress conditions are inferred from other data, mainly Thompson Mine, Manitoba (Golder, 1981), and Ruttan Mine, Manitoba (Pakalnis, et al., 1985). $\sigma3=\sigma v$; $\sigma1=1.5\sigma v$ to $2\sigma v$, horizontally trending 070° (sensitivity testing has been conducted in this study due to uncertainty in far field stress orientation) and $\sigma2=1.0\sigma v$ to $1.5\sigma v$, horizontal trending perpendicular to $\sigma1$.
- To assign different material properties to the numerical model, spatial domains are defined based on lithological units in order to allocate separate material properties.
  - the ore body was divided into the MS and the CSZ
  - the Cap Tuffite Formation (HW), consisting of FT, CBM, MSED, and GAB
  - the McIlvenna Bay Formation (FW), consisting of Volcanics, and SCHT
  - note: The Koziol Fault is above the Cap Tuffite Formation and is typically greater than 50m away from the orebody, so it was deemed unnecessary to include in the numerical model; slip potential on this feature is evaluated separately
- A suite of rock mechanics laboratory tests was performed by the Queen's University Department of Mining Engineering rock mechanics laboratory on intact samples which were selected by the RockEng personnel and site geologist. The data was compiled with existing testing conducted with the 2012 program by Golder. The major lithologies were tested for unconfined compressive strength (UCS), indirect tensile strength or Brazilian tensile strength (BTS) and triaxial confined strength (TCS) tests. A summary of the laboratory test results is presented in Table 16-5

Table 16-5: Summary of Intact Properties

Lithology	Major/Minor Rock Types	Density (g/cm3)	Elastic Modulus (GPa)	Poisson's Ratio	UCS (MPa)
MS	Major	4.02	86	0.11	193±331
CSZ	Major	2.83±0.1	65±4	-	57±28
GAB	Minor-HW	2.95±0.08	65±4	0.19±0.04	121±28
Volcanic	Major-FW	2.95±0.16	67±15	0.19±0.06	123±16
Sil. Vol.	Minor-FW	2.79±0.17	56±19	0.24±0.12	102±851
CBM	Minor-HW	2.79±0.14	60±22	0.26±0.09	83±24
SCHT	Major-FW	2.71±0.04	51±16	0.19±0.04	91±26
REG	Minor	2.64±0.17	23±16	0.2±0.07	36±25
FT	Major-HW	2.72±0.12	56±20	0.17±0.05	155±33
DOL	Minor	-	105±63	-	247±182

16.3.3 Numerical Model

Three-dimensional numerical modelling has been used to estimate in situ stress conditions and the redistribution of stresses around mine excavations in order to investigate stress loading and/or changes in stress imposed on underground excavations, pillars, and geological structures over the mine life. Due to minimal volume of rock mass yield (and thus the minimal stress shedding around the yielded volume), elastic modelling is considered sufficient.

Mining Method and Stope Design

Longhole open stoping is a geotechnically sound mining method for McIlvenna Bay according to the RockEng (2020) assessment. Level spacing is 30m, resulting in stope heights ranging from roughly 35 to 40 m (depending on the HW incline). The current stope shapes have typical longitudinal HW to FW spans of 3.8 to 11.8 m and transverse stopes have HW to FW spans of roughly 5.8 to 21.5 m.

Stope sizing has been assessed using the Mathews/Potvin stability graph analysis method (Potvin, 1988). In this method, a Modified Stability Number (N') is plotted against the Hydraulic Radius (HR) to empirically asses the stability of each stope face. The modified number is calculated as follows:

$$N' = Q' * A * B * C$$

where Q' is the Modified Rock Quality index, A is the Rock Stress Factor, B is the Joint Orientation Factor, and C is the Gravity Adjustment Factor. A summary of N' parameters obtained in this study are shown in Table 16-6.

Table 16-6: Summary of N' Parameters

		A Factor		B Factor		C Factor		Q'		N'	
		min	max	min	max	min	max	min	max	min	max
Zone 1	HW/FW	0.1	1	0.3	0.3	4	8	4.2	11.5	0.5	28
	Lwall	0.1	1	0.35	0.45	5.5	7.8	12	50	2.3	176
	Rwall	0.1	1	1	1	7.8	8	12	50	9.4	400
	Back	0.1	1	0.2	0.2	2	2	12	50	0.5	20
Zone 2	HW/FW	0.1	1	0.3	0.3	4	8	4.2	11.5	0.5	28
	Lwall	0.1	1	0.35	0.45	5.5	7.8	12	50	2.3	176
	Rwall	0.1	1	1	1	7.8	8	12	50	9.4	400
	Back	0.1	0.7	0.2	0.2	2	2	12	50	0.5	159
Zone 3	HW/FW	0.1	0.4	0.3	0.3	4	8	4.2	11.5	0.5	12
	Lwall	0.1	0.2	0.35	0.45	5.5	7.8	12	50	2.3	32
	Rwall	0.1	0.2	1	1	7.8	8	12	50	9.4	72
	Back	0.1	0.2	0.2	0.2	2	2	12	50	0.5	3.9

The stope stability analysis for planned stope dimensions has concluded that stope performance is expected to be controlled by HW performance. For the current stope geometries, cable bolting can be utilized to minimize HW dilution. Stope sidewalls will be stable and stope back over-break can be controlled by ground support design. Once detailed mine plans are available and local ground conditions can be assessed in underground excavations, cable bolting requirements can be optimized based on local ground conditions.

For stope backs (Figure 16-16), the following should be considered:

- Zone 1 (<300 m depth): cable bolts required where HR exceeds 5 m
- Zone 2 (300 to 600 m depth): cable bolt required where HR exceeds 4 m
- Zone 3 (>600 m): cable bolts required where HR exceeds 3.2 m

The typical back HR for longitudinal stopes are in the order of 1.6 to 3.8 m, and transverse stopes are in the order of 2.2 to 5.4 m. As shown in Figure 16-16, a maximum HR of 5.7 (25 m HW to FW span) is expected to be stable with cable bolt support. It is noted that cable bolts in very large stopes may be ineffective when ground conditions are the absolute lower bound or when stress loading will negatively impact stope back performance due to high extraction ratio; this situation is not common.

- Design and costing studies should assume that cable bolting will be required for all stope HWs. Operationally there will be opportunity to optimize the application of cable bolting as follows:
    - upper bound stope conditions (sub-vertical HW faces, upper bound ground conditions) will not require cable bolting
    - average stope conditions (primarily driven by lower bound rock mass quality and inclined HWs) will require cable bolting
    - lower bound stope conditions for all zones are likely to incur some failure even if cable bolted when the HR is over 6 m. It is important to delineate areas with lower bound conditions (such as areas with a shallower HW incline), and mitigate the risks

associated with HW failure (dilution, oversize, equipment damage, etc.) by reducing panel width (along strike)

The implementation of cable bolts installed in stope HWs from the top and bottom cuts will reduce the unsupported HW HR (Figure 16-17). With this reduction in the unsupported span, the stope HWs are predicted to typically perform well. A reduction in stope strike length to 15 m has roughly the same effect as the use of cable bolting.

Stope sidewalls are expected to perform well (as shown in Figure 16-18). Maximum stope sizes (HW to FW spans in the order of 20 to 25 m) in transverse stopes may incur some sidewall overbreak in lower bound rock mass conditions, and the associated risks can be better quantified during later phases of study once detailed mine plans are available (for example, areas with maximum HW to FW spans may require multiple panels if lower bound rock mass conditions occur locally).

Figure 16-16: Empirical Stope Stability Analysis (back)

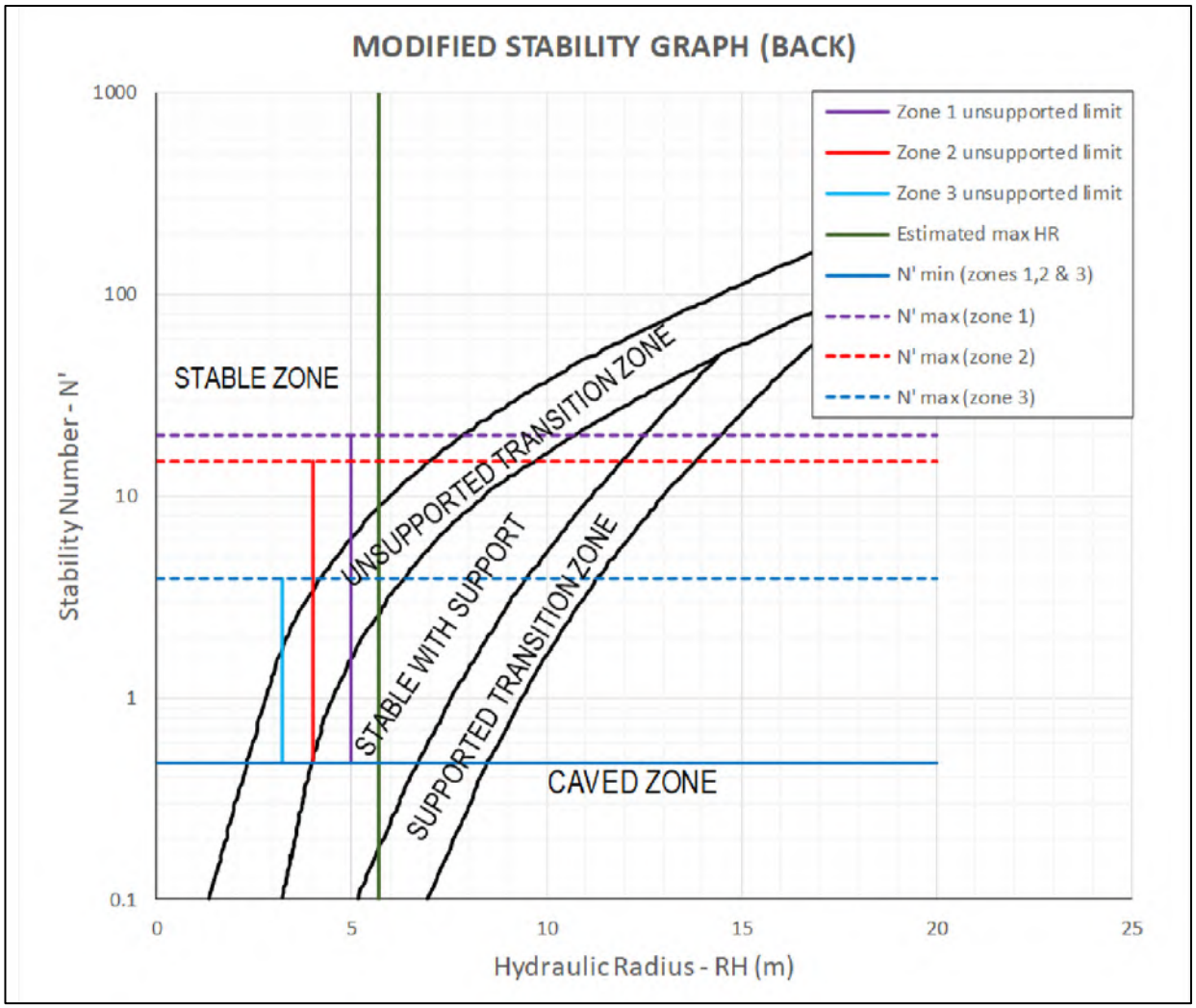


Figure 16-17: Empirical Stope Stability Analysis (HW)

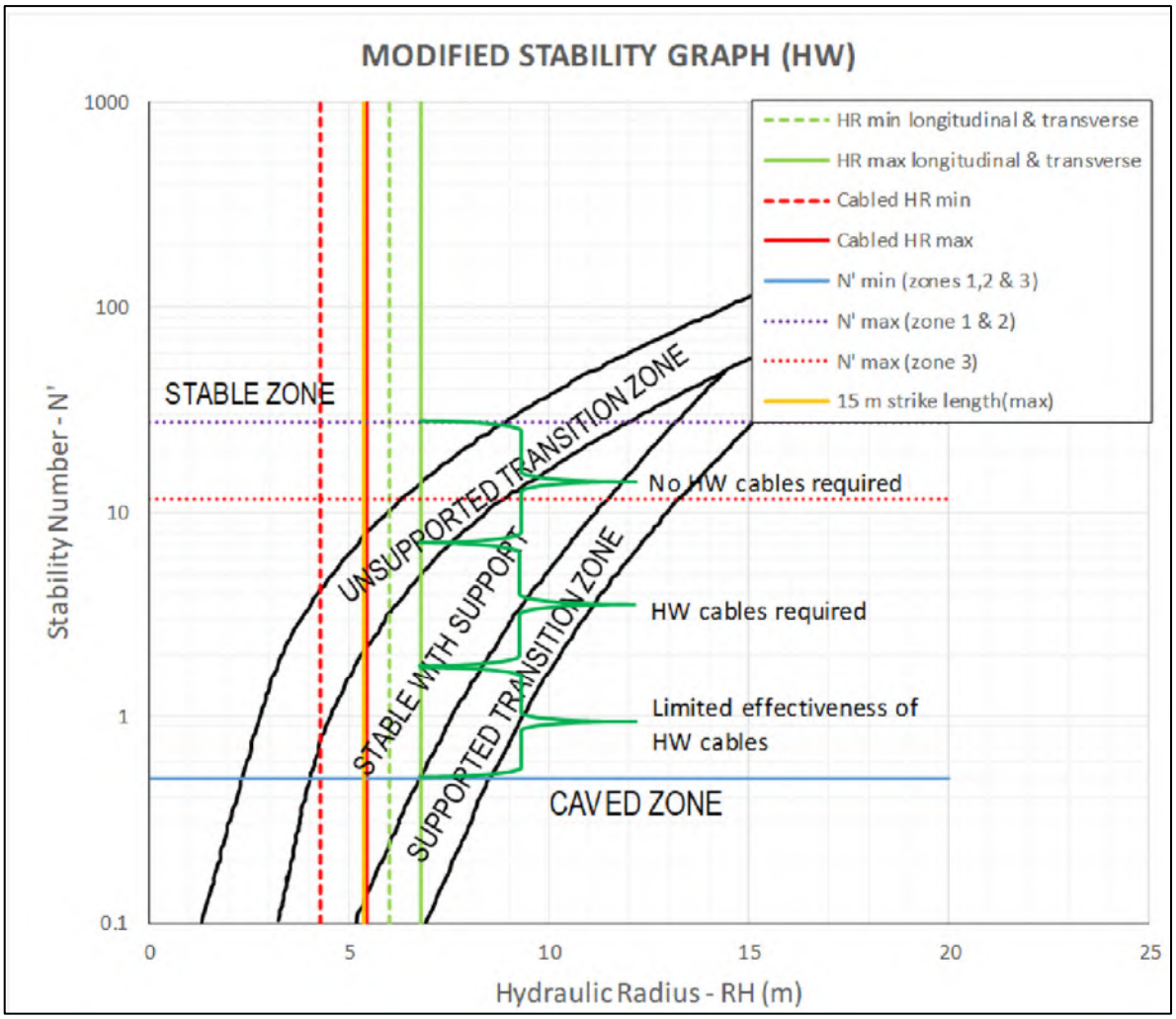
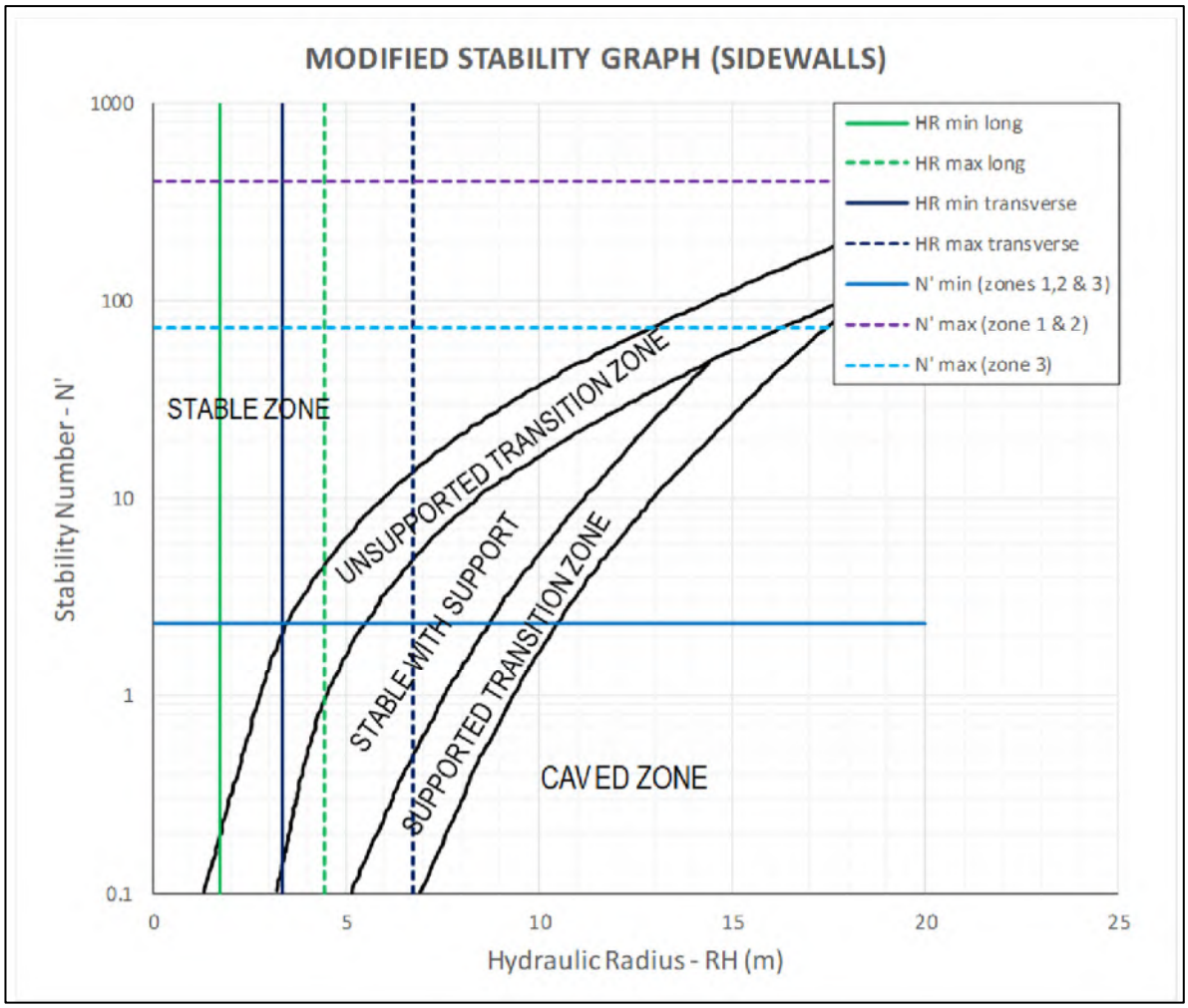


Figure 16-18: Empirical Stope Stability Analysis (sidewalks)



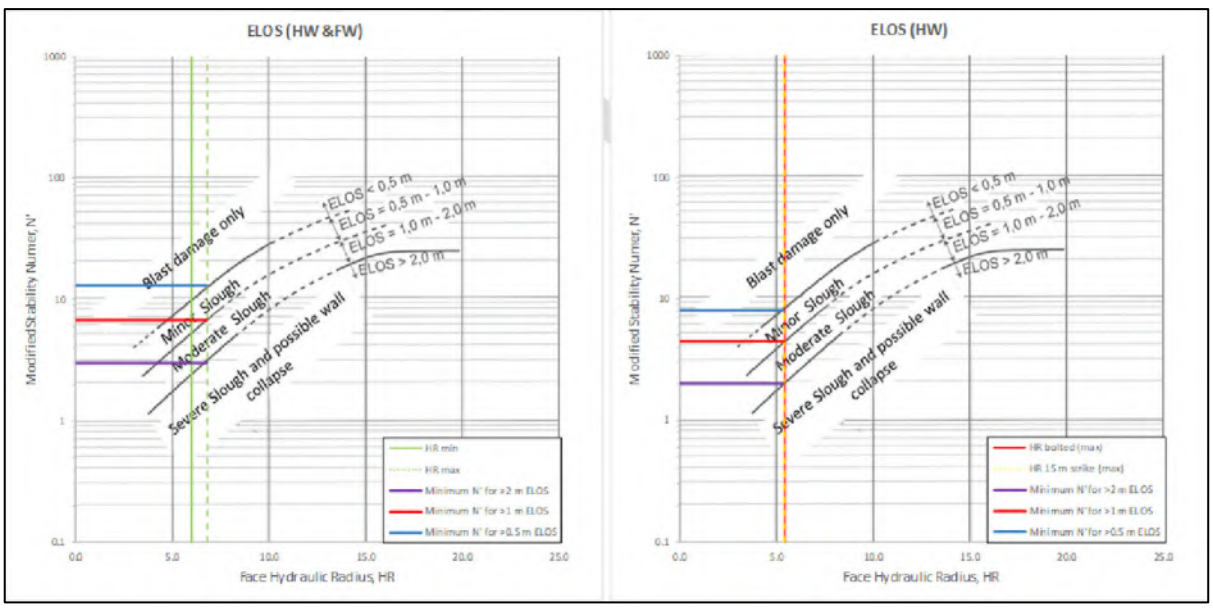
16.3.4 Dilution

The stope stability analysis for planned stope dimensions has concluded that stope sidewalls will be stable and stope back overbreak can be controlled by ground support design. This evaluation of dilution potential has focused on the HW and FW overbreak. Overbreak predictions have been made by using the statistical distribution of rock mass quality. Table 16-7 provides updated overbreak predictions from the empirical evaluation of ELOS bin limits for N' in Figure 16-19; the ELOS values (Clark, 1998) provide an estimate of equivalent linear overbreak for stope faces. These results demonstrate that overbreak will increase with depth, and the extent of overbreak is significantly reduced when the influence of cable bolting is factored into the performance of stope HWs. These total probable overbreak magnitudes should be interpreted as the mine-wide averages, and it must be understood that these are likely to vary from stope to stope. It is important to establish a routine for geotechnical data collection and a continuous analysis of the stope performance during operations to provide information needed for stope optimization.

Table 16-7: Predictions for Stope Overbreak Potential

Depth Zone	ELOS (m)	% of stopes within each zone with probable occurrence of ELOS for average geometry and stress conditions.		
		FW	HW (no cables)	HW (cables* or 15m strike length)
Zone 1	2	1	10	8
	1	3	5	5
	0.5	12	20	18
	0	84	65	69
Zone 2	2	1	10	8
	1	3	5	5
	0.5	12	20	18
	0	84	65	69
Zone 3	2	4	17	14
	1	24	33	19
	0.5	13	26	18
	0	59	24	49
*Assumed cables reduce unsupported stope height by 15 m				

Figure 16-19: Empirical Approximation of Stope Overbreak Potential (definition of minimum N' values for each ELOS bin)



16.3.5 *Sequencing*

Center-out primary secondary sequencing is expected to perform well. Production sequencing is defined so that stopes within each mining block are advanced vertically, and then laterally, to develop pyramids within each mining block, following a retreat from the center of the ore zones towards the

lateral edges of the orebody. This will favorably push stress concentrations to the mining abutments rather than unfavorably concentrating stresses in diminishing pillars. There are constraints in following this sequencing in areas with longitudinal stoping, which can be addressed by strategic risk mitigation methods, and tactical strategies.

16.3.6 Pillar Stability

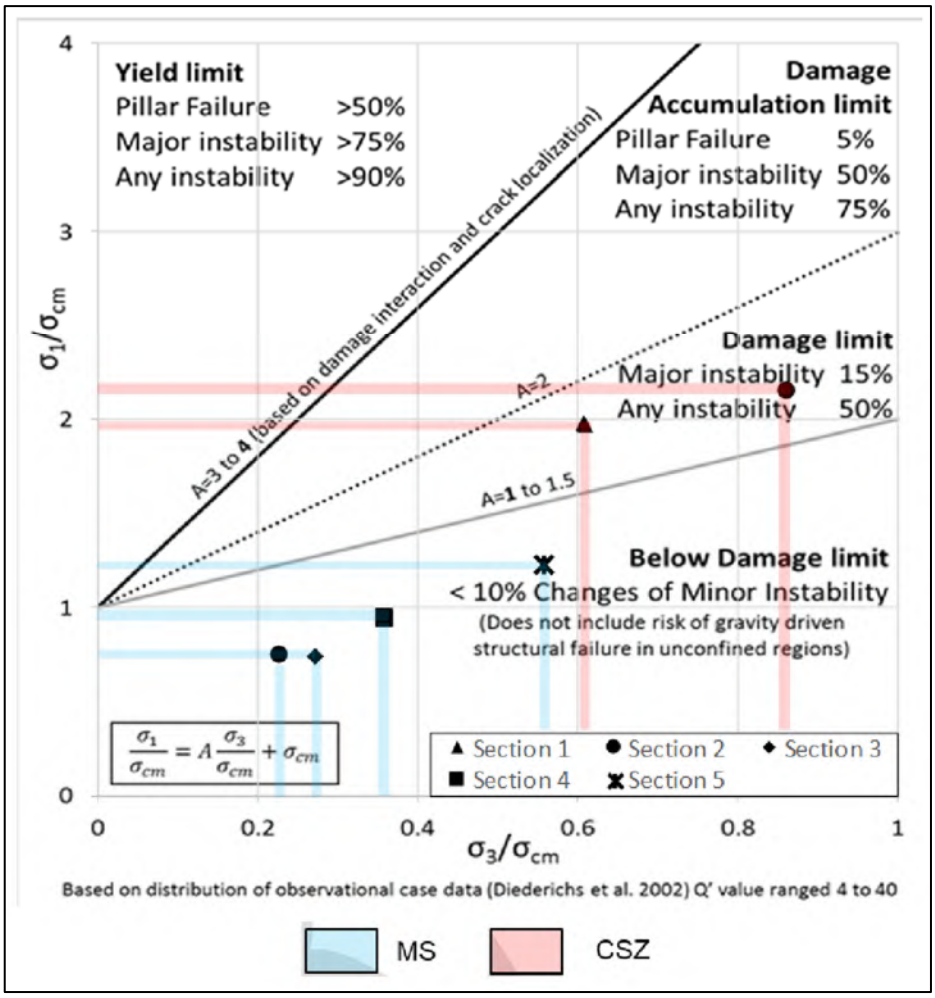
Stress models have been used to evaluate secondary stope pillars, the diminishing pillar between longitudinal and transverse mining fronts, inter-lens pillars and barren pillars based on the mine plan for McIlvenna Bay.

Secondary Stope Pillars

Secondary stope pillars have been assessed using empirical (Diederichs et al., 2002) and numerical methods. Secondary stope pillars were assessed for different extraction ratios, and for the CSZ and MS lithologies. The empirical assessment (Figure 16-20) utilizes stresses obtained from the numerical modelling. Additionally, numerical model results also provide predictions for the factor of safety (FS) within pillars for the CSZ and MS lithologies. In general, there is good agreement between the FS predictions and empirical predictions for damage accumulation. From the empirical and numerical evaluations, it is concluded that:

- Secondary stopes within the MS lithology are low risk. There is only a low probability that minor pillar damage will occur at high extraction ratios. Pillars with the MS ore lithology will perform better than those in the CSZ due to its higher estimated rock mass compressive strength (σcm).
- For the CSZ lithology, it is expected that stress induced damage will occur at moderate to high extraction ratios, and minor damage may occur at low extraction ratio in Zones 2 and 3. Damage in secondary stope pillars can generate sloughage from stope sidewalls, backfill dilution and damage in the secondary cross-cut associated with seismicity events. RockEng recommends that the lag of secondary stopes behind primary stopes be minimized.

Figure 16-20: Evaluation of Secondary Stope Pillars

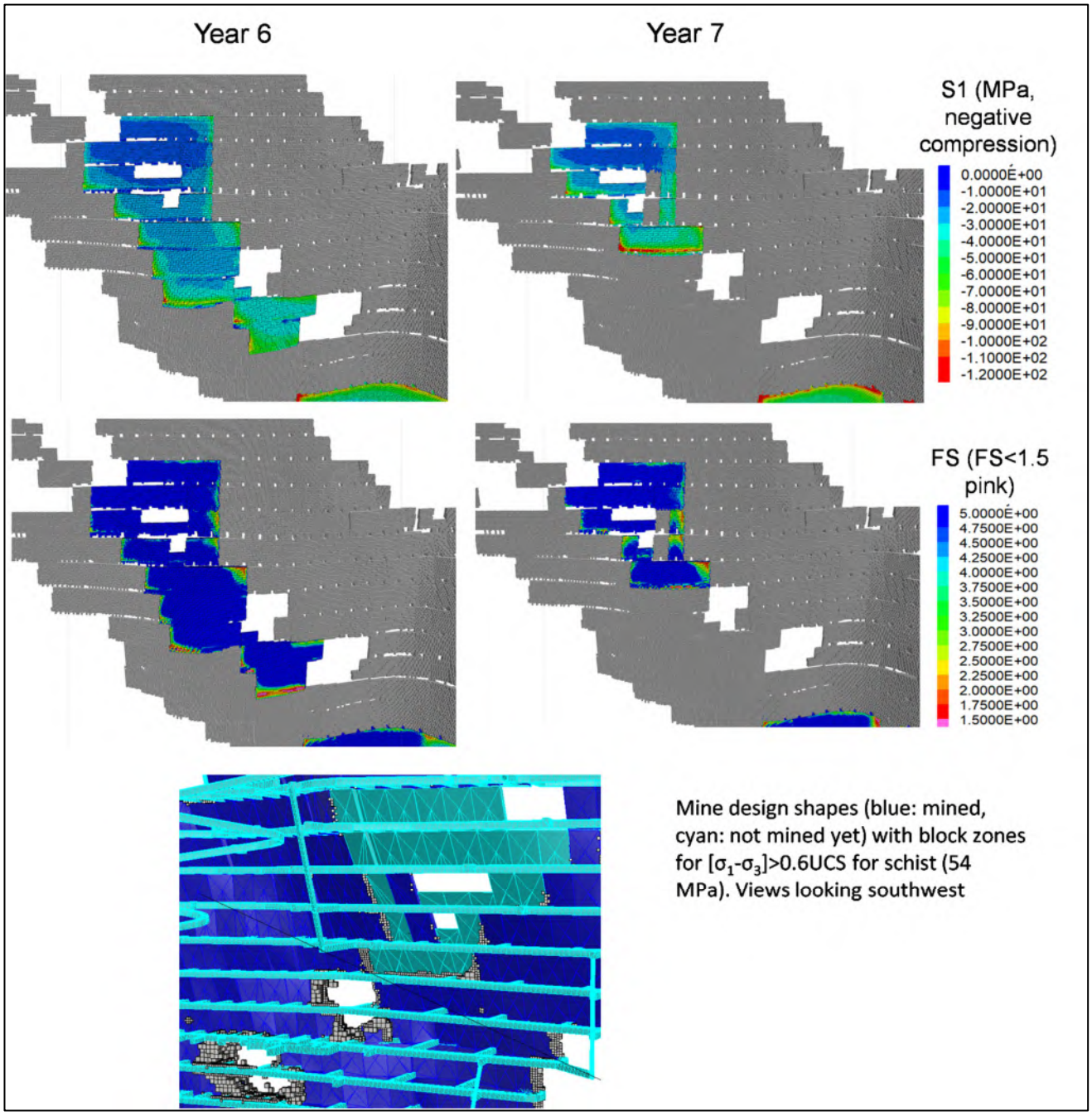


Diminishing Central Pillar

A diminishing central pillar occurs between the transverse and longitudinal mining fronts. Figure 16-21 illustrates the sequence of central pillar extraction during the years 6 and 7. Numerically, stress magnitudes indicate that the diminishing pillar will have no significant stress related risks to the operation. During years 6 and 7 there will be localized rock mass damage in the immediate abutments of advancing stopes within this pillar, however global pillar yield is a very low risk. It is reasonable to expect that stopes advancing at the pillar abutments may incur some drilling difficulties (blast hole squeezing) which may cause production delays.

As the diminishing pillar narrows to approximately 3 or 4 stope widths or less, stress loading will increase susceptibility to rock mass damage. Longitudinal development within the diminishing pillar will be higher risk than transverse development. It is recommended that the diminishing pillar be sequenced so that the final stopes (last 3 to 4 on each level) be extracted by transverse layouts to minimize risks associated with stress induced damage in longitudinal development.

Figure 16-21: Principal Major Stress and Factor of Safety Contoured on Stopes not yet Mined within the Diminishing Pillar at Years 6 and 7



Inter-lens Pillars

Inter-lens pillars will occur in Zone 2. Figure 16-22 and Figure 16-23 demonstrate the results of the inter-lens pillar analyses. These pillars are expected to lose confinement (very low or tensile Minor Principal stress and low Major Principal stress). It is reasonable to expect damage and yield within

these inter-lens pillars, however because yield is occurring due to loss of confinement (rather than stress loading), the failure mechanism is not likely to be violent (i.e. low seismic risk). Open stopes immediately adjacent to these pillars may incur exacerbated sloughage.

Figure 16-22: Inter-lens Section at Different Depths

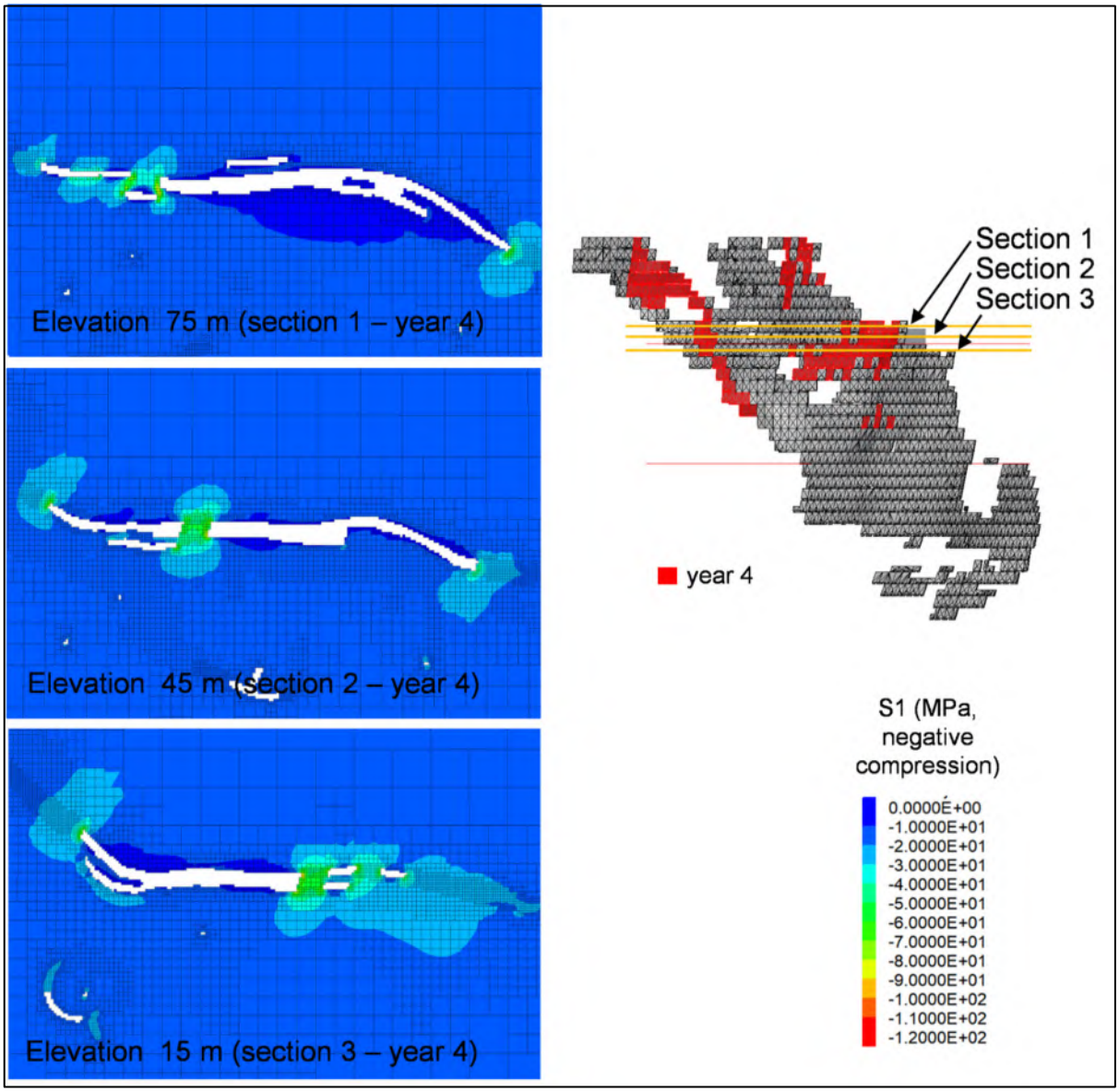
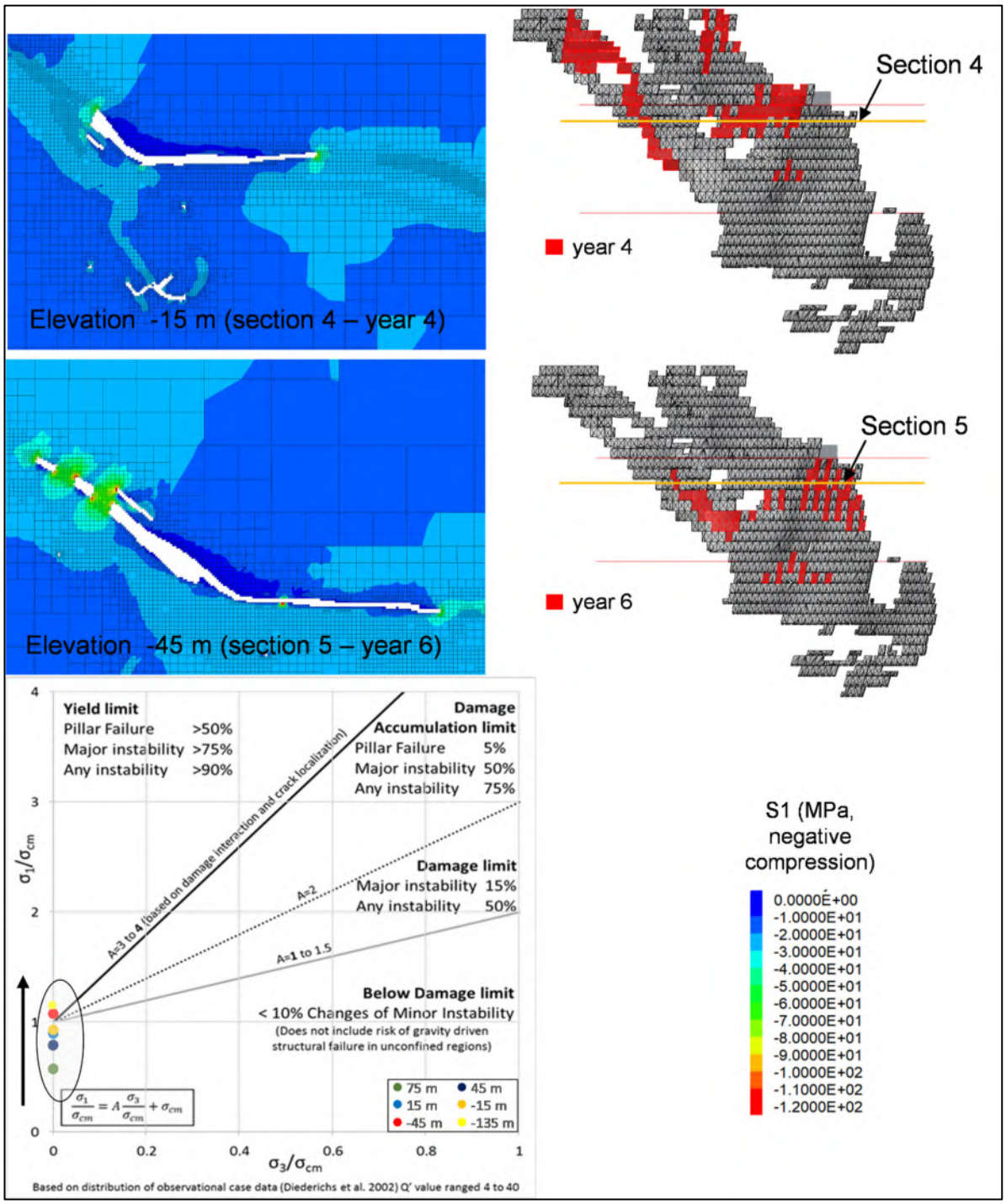


Figure 16-23: Inter-lens Section at Different Depths on the Numerical Model and Empirical Analysis of the Pillars



Barren Pillars

"Barren pillars" refers to regions of the orebody where stopes are not currently in the mine plan. Figure 16-24 illustrates the location of barren pillars and cross-sections used to demonstrated stress conditions within these pillar as predicted by the numerical models. Stress conditions within these pillars at the end of mine life are shown in Figure 16-25 and Figure 16-26. Small barren pillars (lower than 40m width measured along strike) are likely to incur high stress conditions at high extraction ratios. Stress loading and associated seismicity may adversely impact development and infrastructure in the vicinity. The risks associated with stress loading and seismicity can be mitigated by increasing standoff distances for development and/or adjusting the stope extraction sequencing to retreat away from barren pillars in a manner that ensures that FW development is not active during mining stages with elevated seismic risk.

FW development may need to be moved up to 10 m further from the orebody where it is adjacent to barren pillars to maintain the serviceability of the ventilation infrastructure. It is reasonable to implement this design detail at later stages of detailed engineering as stope shapes and infrastructure planning will continue to evolve. Further, during early production, engineering observation of pillar performance in shallow mining blocks can be used to define further needs for tactical strategies to manage the stress prior to mining in deeper mining blocks. Tactical strategies, if required, may include dynamic ground support and the use of strict re-entry protocols.

Figure 16-24: Sections Used to Assess Stresses in Barren Pillars

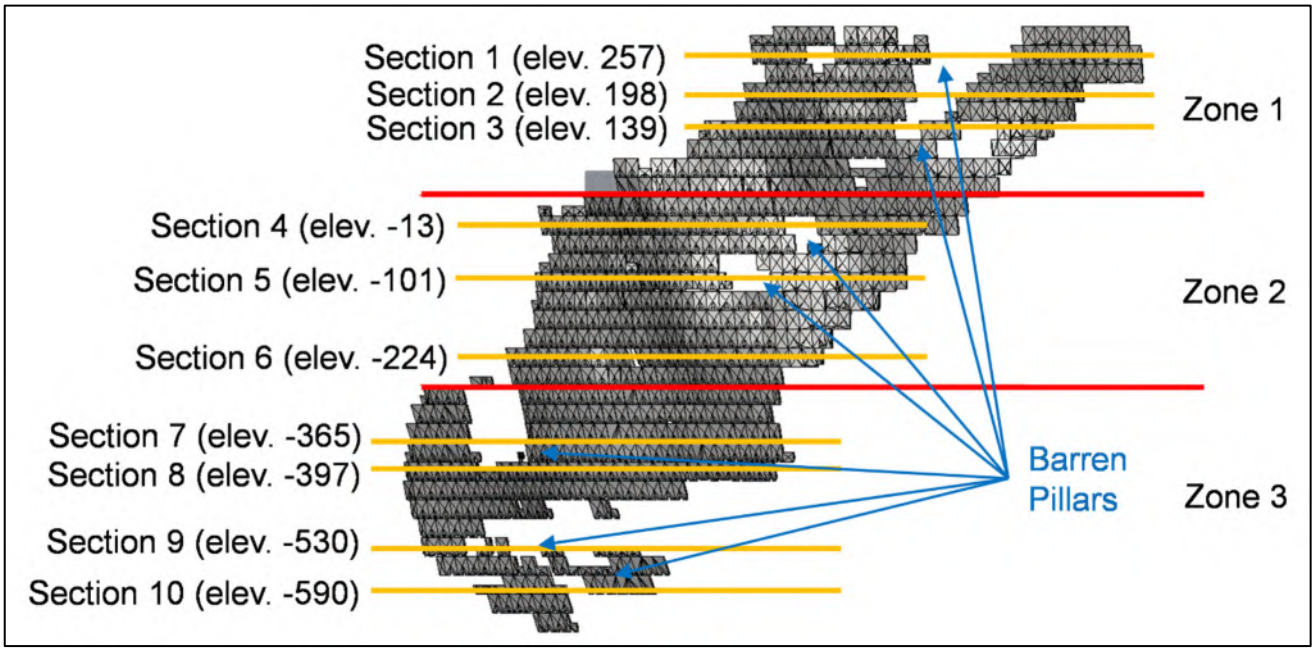


Figure 16-25: Plan View Section of Maximum Principal Stress at Different Depths

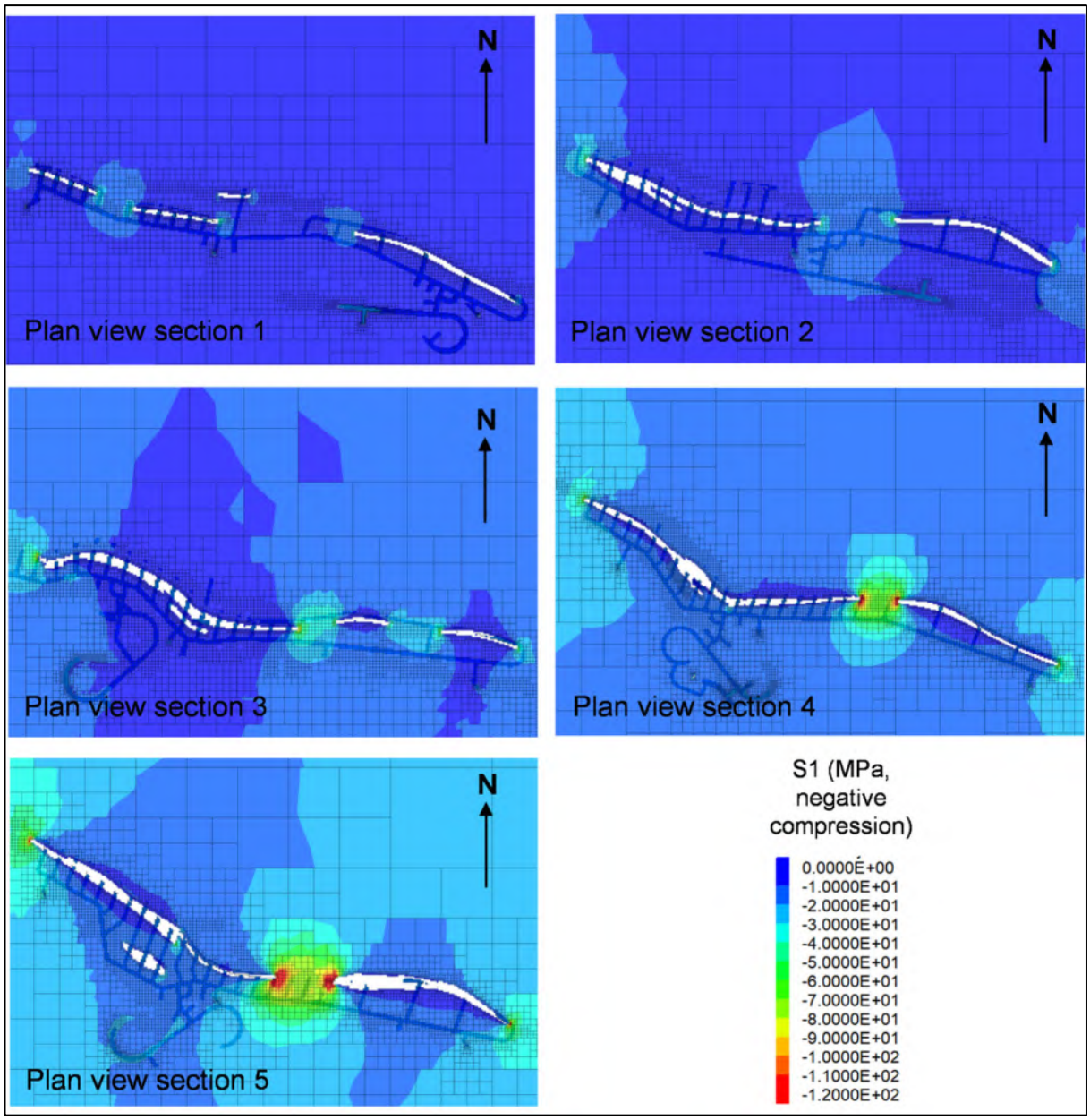
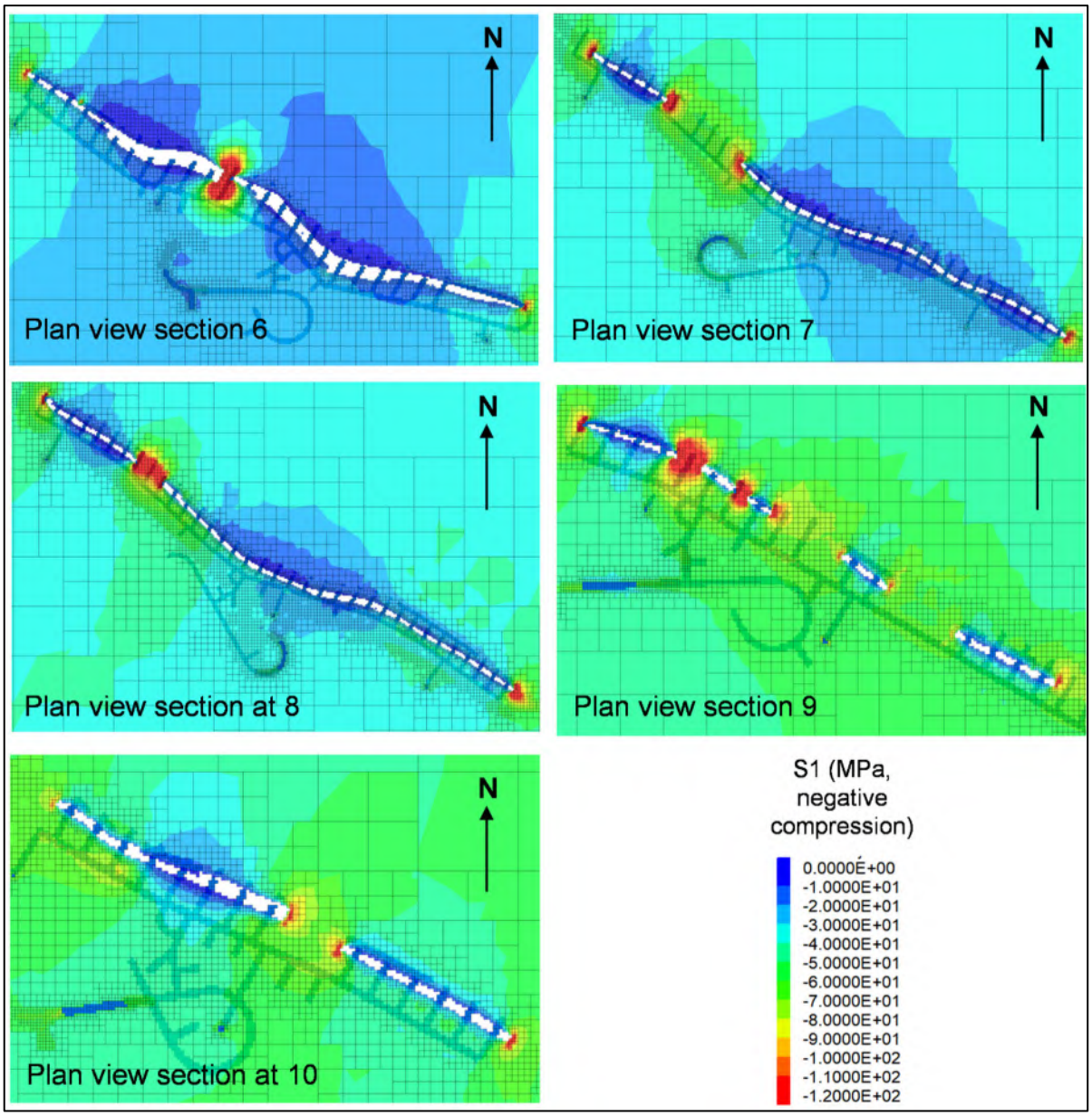


Figure 16-26: Plan View Section of Maximum Principal Stress at Different Depths



Infrastructure Siting

As a general rule, underground permanent infrastructure in underground mines should be sited outside of any zones where the risk of heavy mine induced seismicity and significant stress induced damage is expected. Numerical models have been utilized to identify regions where moderate and heavy damage may occur due to mine induced stress loading based on the following criteria:

- If the differential stress magnitude [σ1-σ3] is in the order of 0.4 UCS, it can be considered as the beginning of mine induced seismicity. In this condition, it can be expected to have a minor

stress damage (i.e. slabbing) near to the excavation (in unconfined low conditions). Standard ground support in good condition and correctly installed is typically adequate to minimize the damage and contain its propagation.

- If the differential stress magnitude [σ1-σ3] is in the order of 0.6 UCS, it is expected to initiate heavy seismicity (rock bursting can occur at low confinement) and a significant stress induced damage. In this condition, dynamic support should be used to manage severe rock burst prone ground.
- If the differential stress magnitude [σ1-σ3] is in the order of 0.8 UCS, significant yielding can occur resulting in a broken, static rock mass. In this condition, dynamic support should be required in order to mitigate safety and production risks associated with induced stresses (i.e. exposure to rock bursting and support rehabilitation). If the excavation is developed in the yielding zone, it is expected to have no significant stress damage due to relaxation of the rock mass prior to excavation. Under this condition, standard ground support should be required to control raveling. Excavations within or adjacent to the highly stressed rock surrounding this damage zone are at risk of rock bursting and/or seismic shakedown.

The FW region of the McIlvenna Bay deposit is predominantly schist (UCS = 91 MPa). The result in the numerical model are shown in Figure 16-27 to Figure 16-30 and summarized as follows:

- The main ramp is located outside of the heavy mine induced stress damage zones.
- Heavy induced stress damage, at high extraction ratios, is predicted in haulage drives close to barren pillars (Zones 2 and 3).
- Access drives to vertical development located below -440 m may be exposed to stress induced damage in proximity to lateral abutments of the mining areas. Assuming that these raises must be maintained for the life of mine, it is recommended that FW development be locally adjusted to increase the stand-off distance (by roughly 10 m).

Figure 16-27: Extent of High Stress Loading with Respect to Planned Development

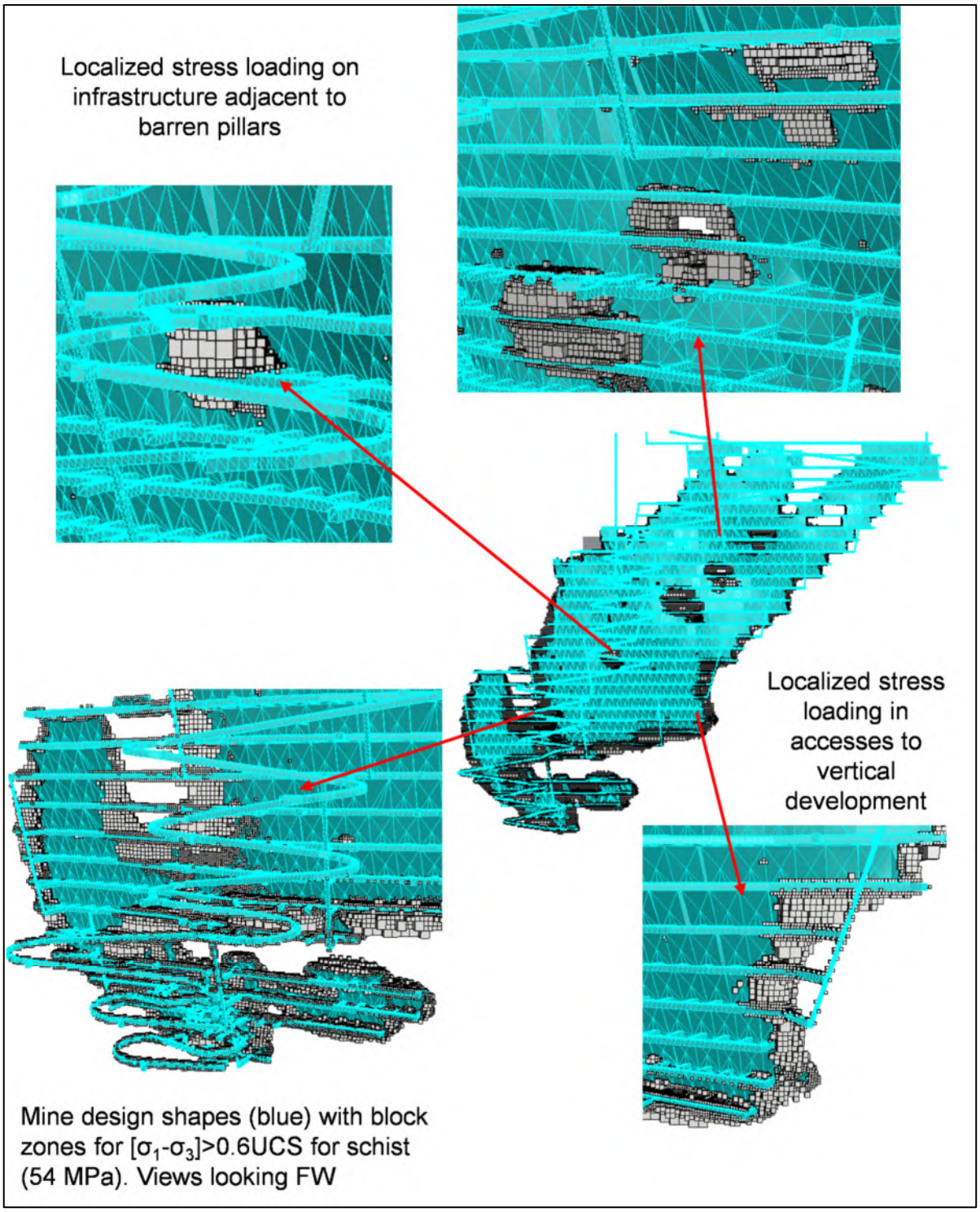


Figure 16-28: Extent of Minor Stress Loading with Respect to Planned Development

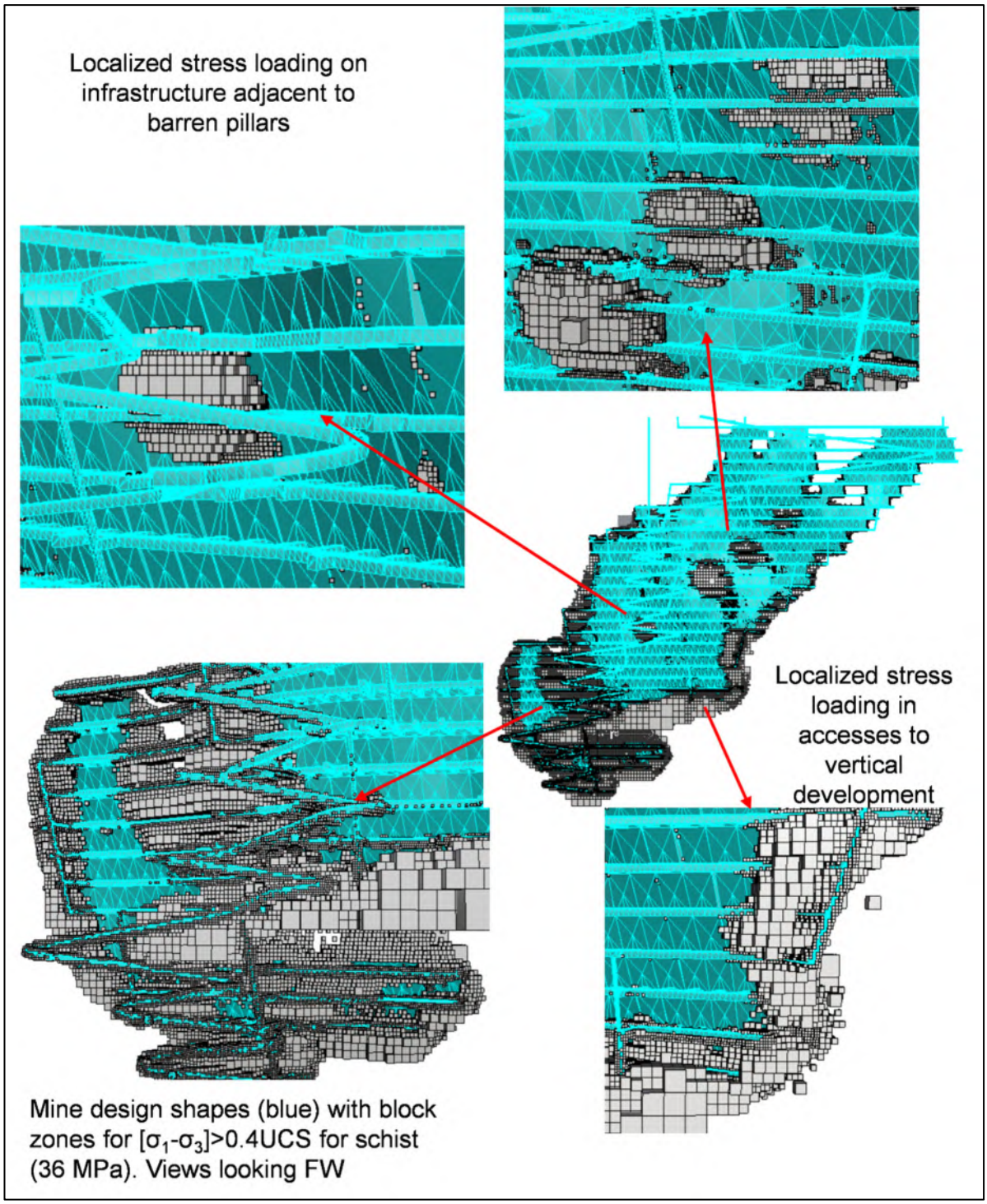


Figure 16-29: Contoured Differential Stress at Varying Elevations to Show Areas of Stress Loading on FW Infrastructure

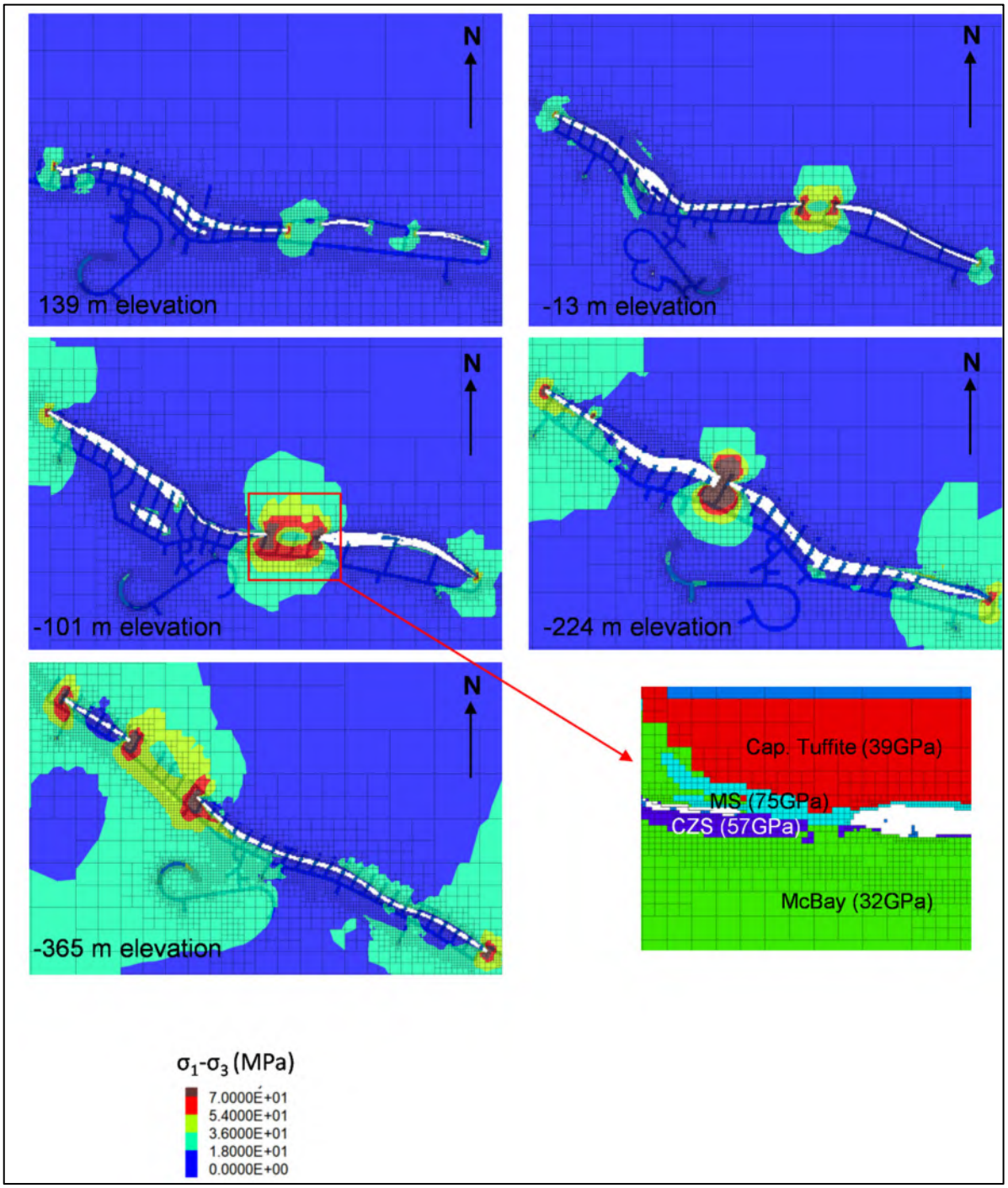
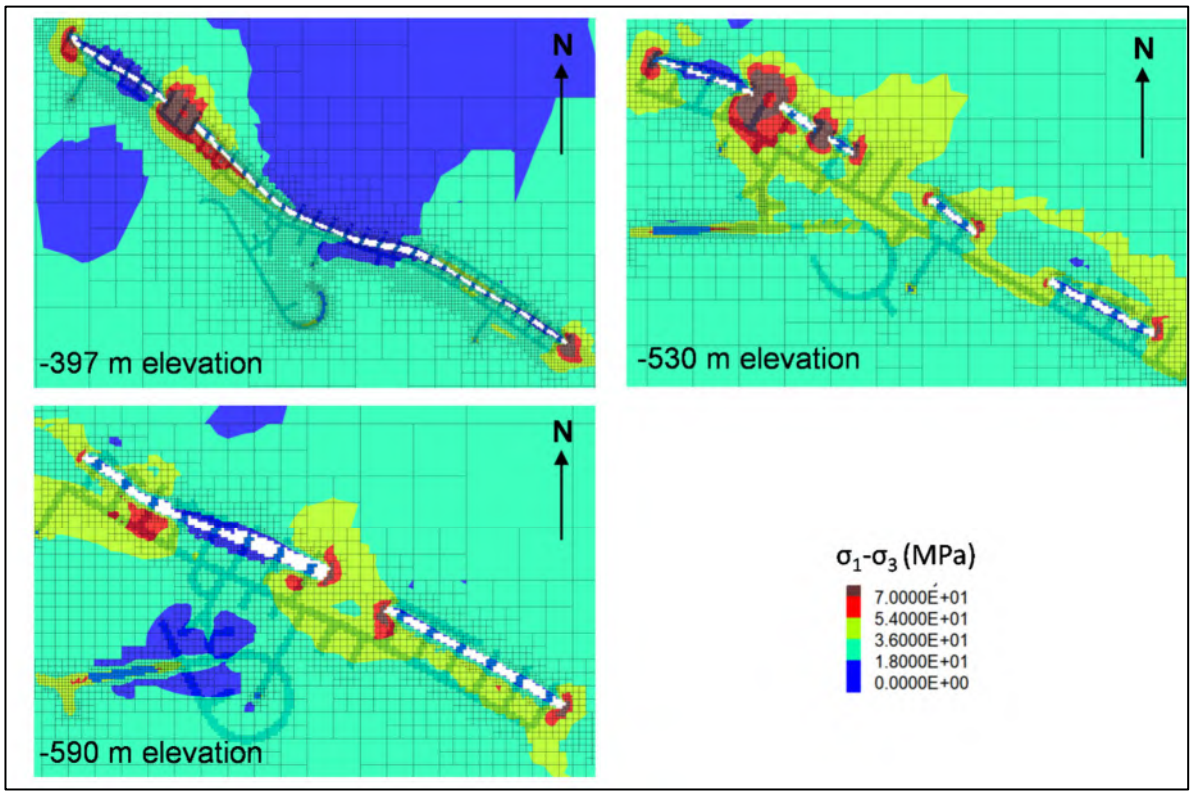


Figure 16-30: Contoured Differential Stress at Varying Elevations to Show Areas of Stress Loading on FW Infrastructure



16.3.7 Ground Support

Ground support guidelines provided by RockEng (2019) are based on empirical and kinematic analysis methods:

<u>Drift Development</u>

- 2.1m long, 19mm or #6 Grade 60
- #6 gauge welded wire screen installed with a primary bolting; consistent with industry best practice, 3 squares of overlap should be made between screen sheets, with a bolt through the centre square
- support pattern best suited for pinning and overlapping screen sheets of this size is a 3-2-3 pattern with a maximum bolt spacing of 1.2m x 1.2m (note: 3-2-3 pattern normally results in a slightly tighter bolt spacing depending on size of screen sheets)
- for adverse ground conditions 75 to 100mm of shotcrete should be applied to the walls and back

Intersections

- in addition to the standard primary support for drift development, 4m double strand bulbed cable bolts are recommended on a 2.5 x 2.5m pattern in intersections up to 8.5m span. Any spans beyond 8.5 m can assume cable bolt length equal to half the span
- as an alternative to double strand cable bolts, 25 tonne inflatable bolts ("Super Swellex") can be utilized on a 1.75 x 1.75m pattern

Main Ventilation Raises

- will be developed by raisebore (3.0m diameter), unsupported

Secondary Egress

- secondary egress raises will be developed between levels by Alimak raise method (2 m x 2 m profile)
- estimated ground support for man-way development are: 1.2 m rebar bolts with screen, two bolts per face per ring with 1 m ring spacing

16.4 Portal Development

The technical aspects related to the design of the portal and box cut development are based on the results of a geotechnical drilling program that was completed by Mine Design Engineering Inc. in 2018 to assess the subsurface conditions of the proposed portal location.

The interpretation of the downhole geotechnical drilling data collected by MD Engineering's geotechnical site investigation of the portal location and forms the basis of the design criteria used for the feasibility engineering design of the box cut surface excavation and portal arrangement. In addition, it considers portal variables such as size, portal opening geometry, crown pillar thickness, excavation slope stability, ground support, and other risks at the portal area.

The proposed portal and box-cut arrangement have been located such that they satisfy environmental and air quality requirements as well as the optimal geometrical configuration of the portal.

16.4.1 Portal Site Characterization

Geotechnical Investigations

The assessment and interpretation of the subsurface site conditions at the portal area have been based on previous geotechnical downhole drilling programs completed by Golder Associates Ltd. (2011-2012) and Mine Design Engineering (2018).

This design largely relied on three down hole geotechnical drillhole logs and core photos at the portal location to assess the geotechnical conditions in support of the portal and box cut development (Figure 16-31).

Figure 16-31: MD Engineering (2018) and Golder (2012) Geotechnical Drillhole Locations

Stratigraphic Profile

Geotechnical logging of the portal drill holes describe the profile of the subsurface as a strong dolomite unit, overlying a consolidated to unconsolidated sandstone and regolith. Geotechnical drill records provided to BBA were not representative of true stratigraphic profile depths at the portal location due to the 60 degree inclination of the drillholes. BBA corrected the downhole portal drill records to reflect the true vertical depth of the underlying units at the portal area as shown in the Table 16-8 below.

Table 16-8: BBA's Corrected Values to Represent True Vertical Depth at Portal Location

Lithology	MB-18-22		MB-18-226		MB-18-227		Average Thickness
	from	to	from	to	from	to	
Overburden (m)	0	2.1	0	1.7	**0	1.9	1.9
Dolomite (m)	2.1	21.6	1.7	21.4	1.9	20.8	21.3
Sandstone (m)	21.8	26.4	21.4	26.8	21	26.2	5.0
Regolith (m)	26.8	31.2	26.8	*43.1	26.2	44.1	12.8
Host Rock (m)	31.8	43.3	n/a		n/a		

* Includes interval logged as REG/FLT

** Includes overburden

Rock Characterization

Review of available downhole drillhole logs and core photos identifies the dolomite as a non-to-slightly weathered competent rock mass with an overall field estimate rock strength of R5 (GAL, 2012), with the upper domain (0 to 2.5m) RQD averaging 30% and remaining lower domain averaging above 90%.

Figure 16-32: Indicative Dolomite Rock Cap - 50 km West of Snow Lake

Both the underlying sandstone and regolith are highly-to-slightly weathered with much of the sandstone rock material decomposed or disintegrated into soil. Strength of the sandstone ranged from R5 (extremely strong) to R0 (extremely weak), while the average strength of the regolith unit is R2 (weak).

BBA had identified the RQD, Jr, and Ja values for the Dolomite were not documented in the MD Engineering report and derived these values directly from the downhole drill logs at the portal area.

It was determined by BBA that the Dolomite be separated into two domains based on RQD values and recorded observations interpreted from the downhole drillhole logs:

- Upper Dolomite Domain RQD < 30% (DOL 1): 0 m to ~2.5m depth from top of Dolomite
- Lower Dolomite Domain RQD >90% (DOL 2): ~2.5m to top of Sandstone horizon.

Portal Design Criteria

The proposed portal is located in a poorly drained muskeg swamps with scattered tamarack and black spruce directly adjacent to the existing gravel access road at approximate elevation of 332m AMSL (Figure 16-33). Foran's reference tunnel design is generally oriented to the NE with an approximate portal access alignment of azimuth 047.

Figure 16-33: McIlvenna Bay Proposed Portal Site Location

The typical tunnel section provided by Foran is as follows:

- 5.5 m diameter D-shaped tunnel
- decline at 15% grade
- surface elevation at start of portal access ramp: 334.3 m

BBA has provided a typical plan (Figure 16-34), section (Figure 16-35) and profile (Figure 16-36) for the tunnel reference alignment, shown below. Reference 0+000 of the ramp alignment indicates the original location of surface excavation for portal ramp access at -15% grade that was provided to BBA as part of the overall underground infrastructure conceptual site plan.

BBA's surface elevations were based on 1m accurate Lidar provided by Foran. Tunnel profiles were modelled using Autodesk Civil 3D (2016) following the design criteria mentioned above.

Figure 16-34: Conceptual Portal Excavation and Location on Plan

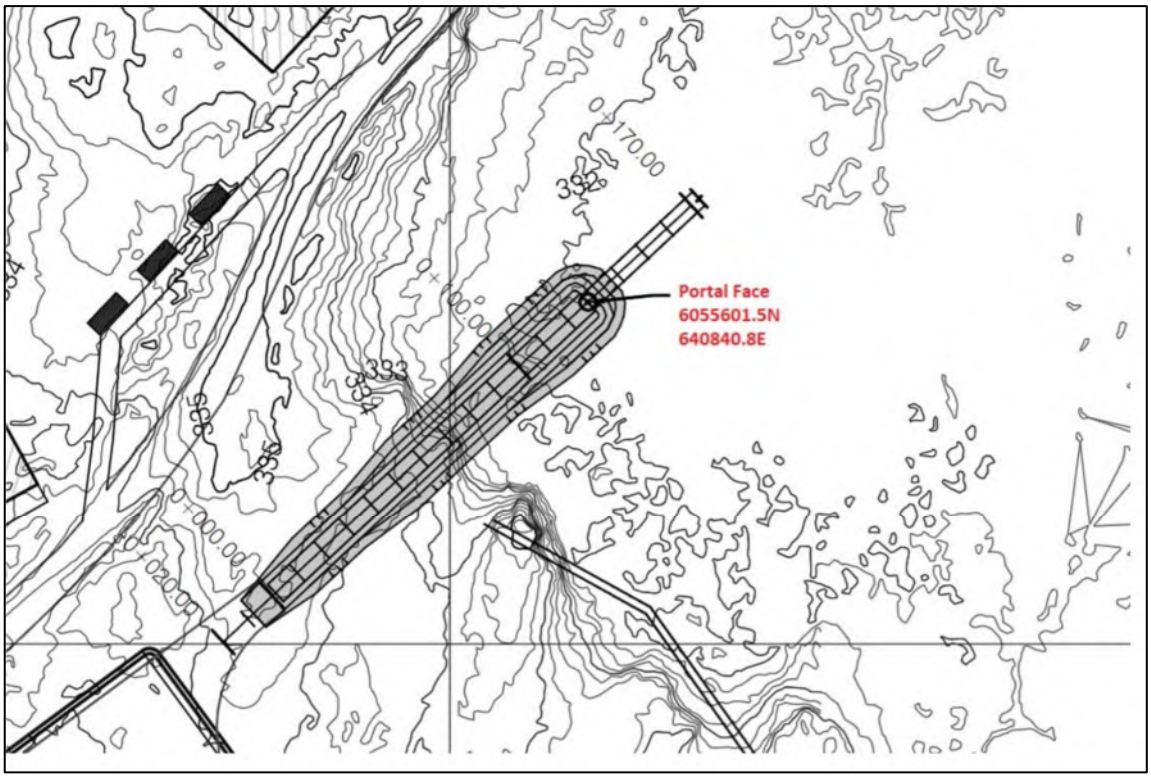


Figure 16-35: 5.5 m x 5.5 m Tunnel Portal Typical Section

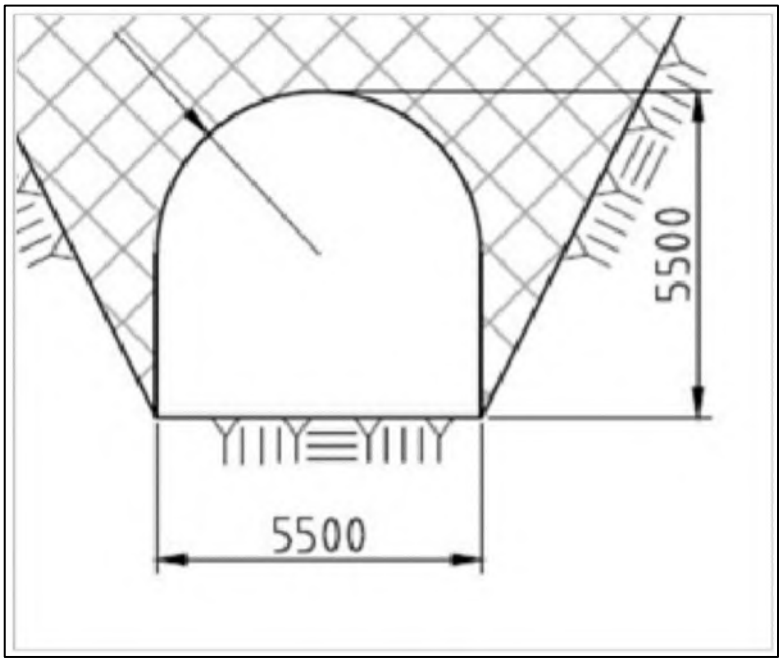
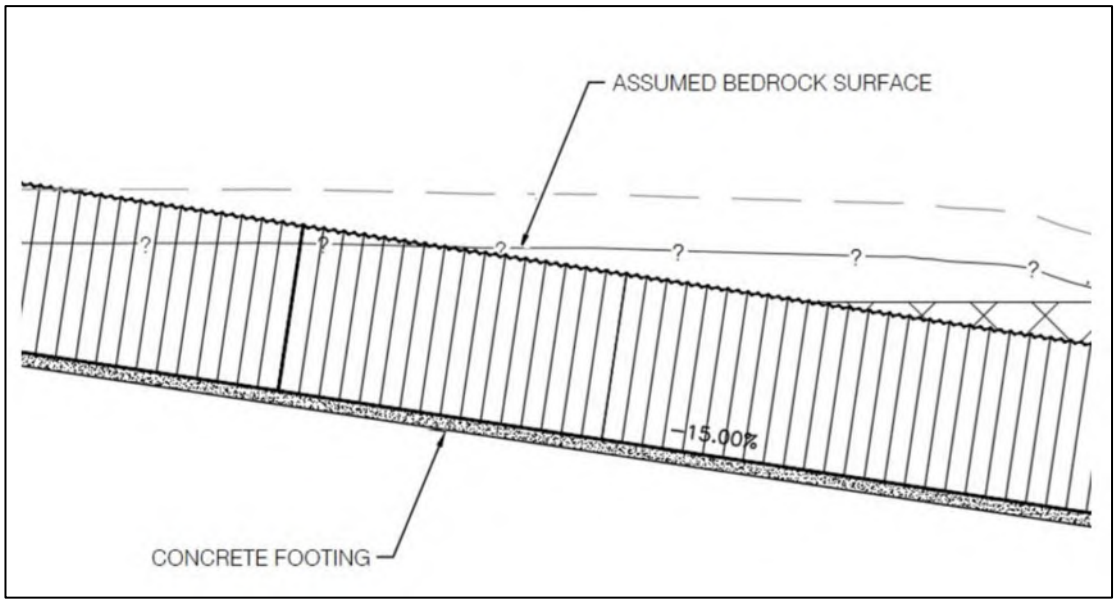


Figure 16-36: Tunnel Portal Typical Profile



Portal Arrangement

BBA determined that the following criteria are included for determining the location for the portal arrangement:

- starting invert elevation from laydown area is 334.3 m
- portal location is defined where minimum 10.5 m rock cover over top of portal is achieved
- avoid intersection of portal floor with underlying sandstone
- permanent rock cut slopes in Dolomite
- initial slope from portal site laydown area declined toward the tunnel portal with 15% slope and it continued with same trend to tunnel intersection
- incline length is assumed from defined from portal site laydown area to the tunnel portal
- maximum height of portal is prescribed 5.5 m

Permanent rock cut slopes have been selected as the preferred cut slope for the portal opening. A permanent rock cut slope is suitable at this location during both construction and operation due to the strong rock parameters and characteristics of the dolomite.

A single permanent rock cut slope also provides an advantageous economical solution in feasibility design over completing both a temporary and permanent rock cut slope. Modelled topography is based on 1m accurate Lidar data provide to BBA by Foran.

The assumed bedrock surface elevation has been determined from BBA's interpretation of the geotechnical core logs and photos using the corrected values of true vertical depth at the portal area.

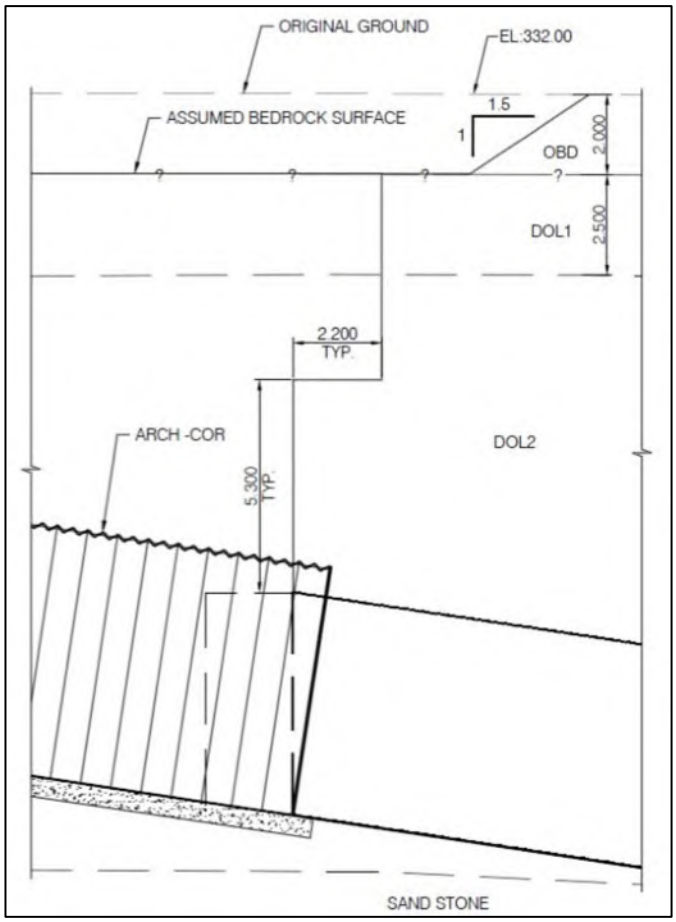
Design Basis

A summary of the portal location arrangement and design basis in Table 16-9 and portal profile arrangement shown in Figure 16-37.

Table 16-9: Preferred Portal Arrangement and Location

Item	Description	
Portal Width (min)	5.5m	
Portal Height	5.5m	
Coordinate System	UTM NAD83 Zone 13	
Coordinates	6055601.5N	640840.8E
Station Along Ramp Alignment	128m	
Rock Unit	Dolomite	
Overlying Dolomite Cover	10.5m	
Drop-cut Height	18m	
Bedrock Cover Width: Height	~2	
Overburden Thickness	~2m	
Ramp Grade	15%	
Portal Floor Elevation	314m	
Surface Elevation Above Portal Face	332m	
Surface Elevation at Portal Laydown	334.3m	
Portal Laydown Grade	2%	

Figure 16-37: Proposed Portal Profile Arrangement



Portal Design

BBA carried out engineering design of the portal and box cut development at the proposed portal location at the McIlvenna Bay project.

Detail review and interpretation of the available downhole drill logs, core photos, and information from previous geotechnical site investigations at the portal location provided model parameters for proposing best options of the portal and box cut surface excavation while determining the design for the portal arrangement.

Numerical Modelling

BBA assessment of typical dolomite slope configuration applied Rocscience RS2 numerical modelling software, a 2D finite element program for soil and rock applications. Various failure criteria including Mohr-Coulomb (Sandstone) and Generalized Hoek-Brown (Dolomite) were incorporated into the slope model to analyze and determine an appropriate inter-ramp-angle (IRA) and slope geometry for excavation design and stability of the portal trench.

Model Parameters

Geotechnical rock parameters that were used in the stability analysis include the unit weight of the material, friction angle (phi, "ϕ"), and cohesion (C) of the underlying bedrock excavation.

The model input data required for a full analysis of the portal box cut excavation using the Hoek Brown failure criterion for both the dolomite and regolith, and the Mohr-Coulomb failure criterion for the non-linear sandstone domain or the portal box cut excavation are shown in Table 16-10 below.

Undrained, dry water conditions have been assumed for this analysis.

A minimum required design factor of safety 1.5 has been used for the slope stability analysis for the design of the permanent rock cut slopes.

Table 16-10: Portal and Boxcut Model Parameters

Material	C (kPa)	ϕ (deg)	Unit Weight (kN/M^3)	Poissons Ratio	E (GPa)	UCS (Mpa)	m_b	s
DOL 1			26.4	0.25	8.5	200	0.715	0.0010
DOL 2			26.4	0.25	8.5	200	1.110	0.0030
Sandstone	0	18	26.4	0.30	0.1	0		
Regolith			26.4	0.30	1.0	36	1.525	0.0013

Boxcut Elevation Parameters

A base case design assumed that the box cut was excavated to a depth sufficient to create a portal within the competent dolomite while keeping the portal floor from intersecting the top of the sandstone, provided initial figures for comparison with alternative options. BBA analyzed the following three options for the box-cut design:

- 90° vertical slope in dolomite with maximum depth of 20m
- 70° slope in dolomite with maximum depth of 16m
- 68° slope in dolomite with maximum depth of 16m

The box-cut is sunk within the dolomite and is defined by planar and persistent vertical and horizontal jointing. BBA has determined the following slope geometry for the box cut design based on the results of the numerical modelling:

- Inter-Ramp Angle (IRA): 68m
- Bench Height (BH) : 5.3m
- Catch Benches (CB): 2.2m
- Total Height (TH): 16m

Portal excavation and fill volumes were calculated using AutoCAD Civil 3D (2016) for the preferred box-cut and portal location arrangement.

- soil + rock cut volume: 18,000 cu. m
- rock cut volume: 13,275 cu. m
- portal fill to elevation: 329.9 m
- fill volume: 8,920 cu. m

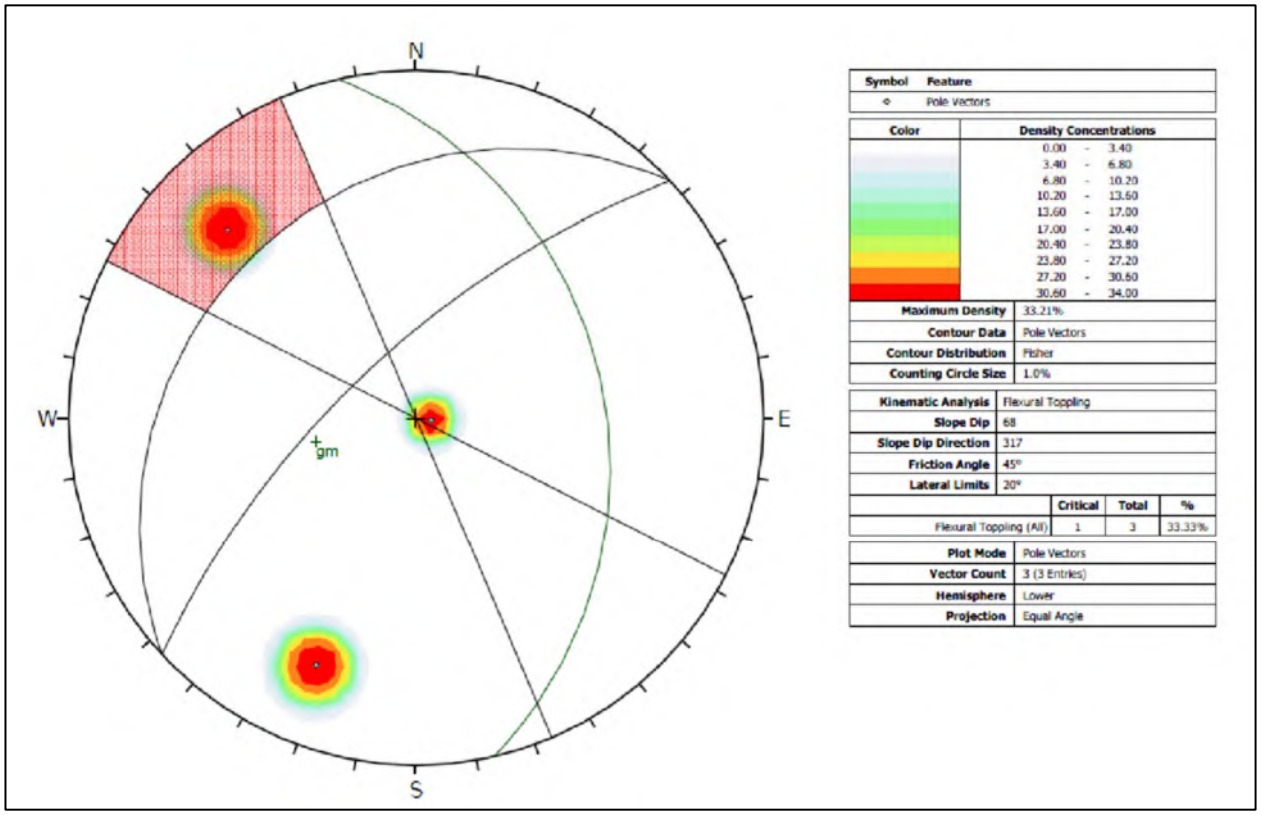
Kinematic Stability Analysis

BBA completed a kinematic analysis to evaluate the potential for planar, wedge, toppling, and flexural toppling failures in the box-cut excavation permanent rock cut slopes. Model parameters are based on the rock characteristics and design criteria of the dolomite domain outlined in this report.

The results of the kinematic analysis identified no distinct modes of planar, wedge, or toppling failures occurring in the side walls or the portal face based on BBA's preferred box cut and excavation design.

However, flexural toppling has been identified as a potential failure mode along the South dolomite rock cut slope face and the weakly defined sub-vertical joint set 2. A stereonet indicating the potential for flexural toppling along the South rock cut slope is shown in Figure 16-38 below:

Figure 16-38: Stereonet Flexural Toppling



BBA recognizes that the two (2) weakly defined sub-vertical joint sets that were documented in MD Engineering's report will have an effect on the results of the slope stability analysis as indicated in the by potential for flexural toppling. The purpose of including them in the kinematic analysis is to provide the model with enough parameters to perform an analysis and generate a baseline understanding of the potential slope failures at the box-cut and excavation cut-slope and portal area. Developing a better understanding of the failure mechanisms will allow us to mitigate these hazards with the appropriate ground support suitable for each unique scenario.

Filed verification and mapping of the dolomite domain during initial stages of the excavation is recommended to determine the presence of any structural features that were not captured in the geotechnical drilling investigation, and further define the two sub-vertical joint sets that were documented in MD Engineering's report.

Portal Arch Design

BBA has considered two primary portal arrangements for consideration:

- open box-cut and exposed permanent rock cut slopes
- an enclosed, backfilled multi-plate arch structure

Based on BBA's assessment of the available geotechnical information at the portal location, a backfilled multi-plate arch structure approach has been considered preferable for the portal arrangement.

Multi-Plate Arch

The Atlantic Industries Limited (AIL) provided the Super-Cor 39S High Profile Arch typical multiplate arch detail for the portal design options. The Super-Cor combines the advantages of lightweight construction with the superior strength and durability of deep-corrugated, galvanized steel for the corrugated metal structures. The larger, annular corrugations in Super-Cor provide nine times the stiffness of conventional structural plate and allowing it to withstand the heaviest of loads.

Below are some of the multi-plate arch system advantages:

- carrying extreme loads
- wide spans (more than 25 m)
- corrugation profile of 381 mm pitch × 140 mm depth
- available in different shapes including: Box Culverts; Standard, Low, Medium, or High Profile Arches; Rounds, and Ellipses
- environmentally friendly
- available with polymer coating

A typical section and profile of the proposed multi-plate arch system for the McIlvenna Bay portal arrangement are shown in Figure 16-39 and Figure 16-40.

Figure 16-39: Multi-plate Arch Typical Sections

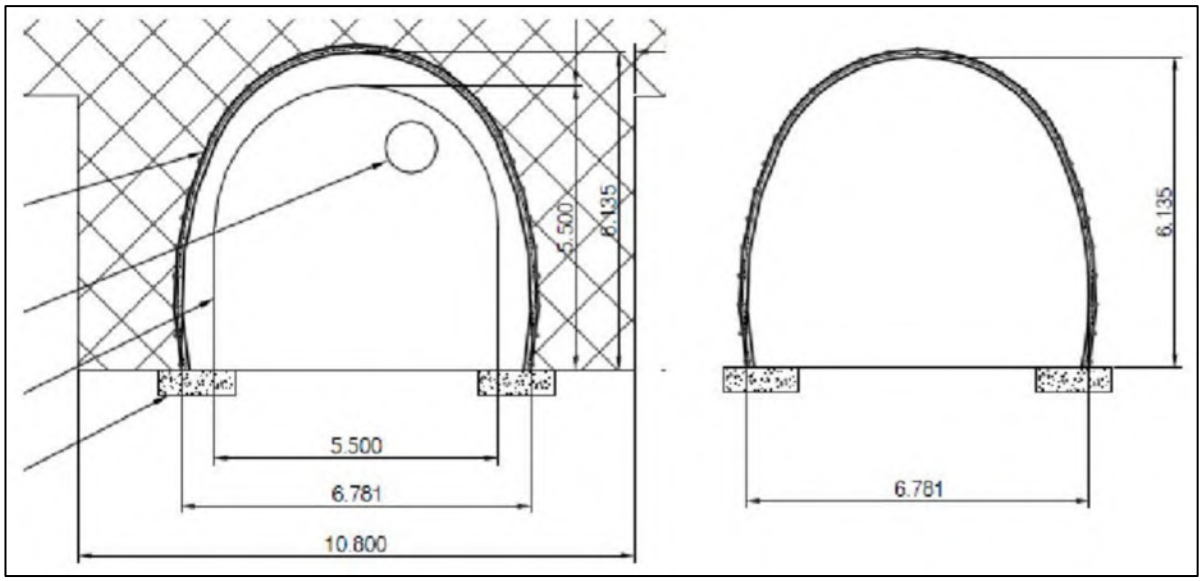
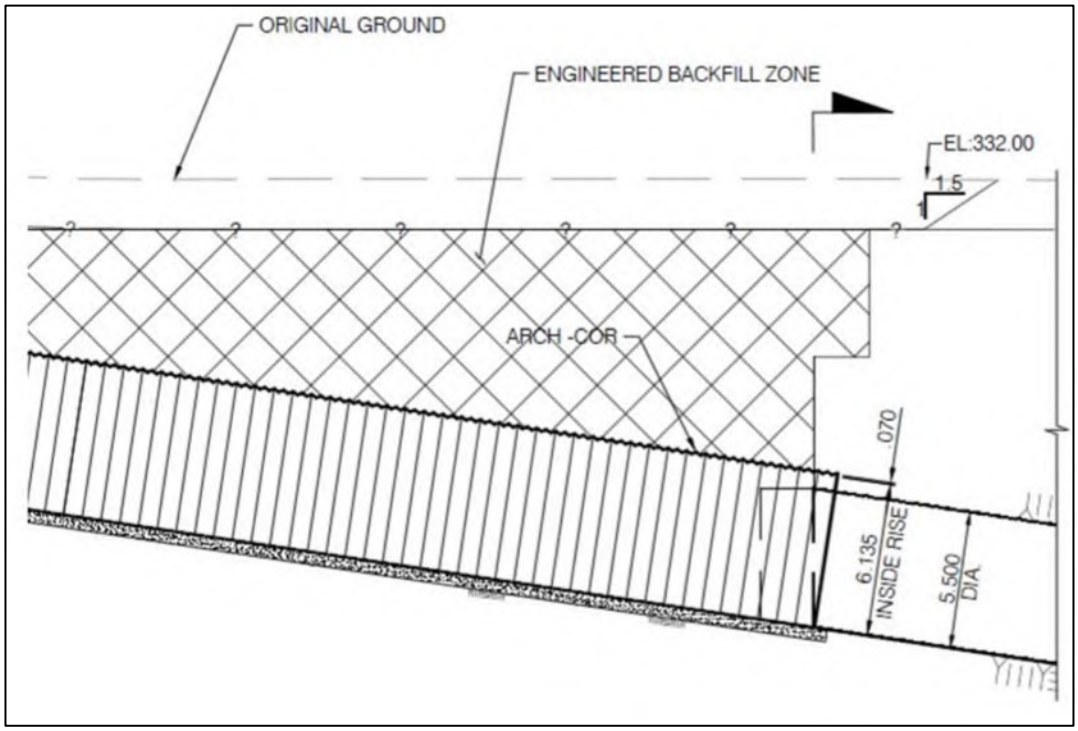


Figure 16-40: Multi-plate Arch-cor Typical Section



Portal Backfill

BBA has provided the following backfill specifications for consideration of the enclosed multi-plate arch structure:

- **Arch Bedding:** Free draining uniformly graded gravel placed at the base of the arch structure. Particles ranging from 50 mm to 3 mm in diameter. Placed and spread evenly to a maximum lift thickness of 300 mm. Compacted to a minimum of 95% of maximum dry density as determined by ASTM D1557 or method specification.
- **Structural Backfill:** Well graded granular material placed around the arch structure in lifts from the base to a minimum of 1000 mm cover above the overt of the arch. Maximum particle size does not exceed a diameter of 75 mm. with less than 10% of the material by mass is below the 0.002 mm diameter grain size. Placed and spread in lift with a maximum thickness of 500 mm, compacted to a minimum of 95% density as determined by ASTM D1557. Lifts both placed and compacted simultaneously on both sides of the arch or on alternating sides.
- **General Fill:** Bulk backfill placed above the structural fill to the maximum height of required backfill. Maximum particle size does not exceed a diameter of 500 mm. with less than 30% of the material by mass is below the 0.002 mm diameter grain size. Placed and spread in lift with a maximum thickness of 600 mm, compacted to a minimum of 93% density as determined by ASTM D1557 or method spec.

Portal Ground Support

BBA has summarized the ground support recommendations for the portal arch and permanent excavated dolomite cut-slopes at the portal and box-cut development in Table 16-11 below:

Table 16-11: Recommended Ground Support for Excavation Walls

Location	Ground Support
Drop Cut Face	2.4 m grouted rebar installed with #6 gauge, galvanized screen. Minimum bot pattern of 1.2 m x 1.2 m increasing to 3-2-3 in the upper dolomite domain
Drop Cut Walls	Bolt walls down to ~1.5 m above catch bench elevation. Drape the screening down lower wall. Any unravelling rock will accumulate at the base of the wall and remain on the catch bench.
Drop Cut Perimeter	Reinforce the perimeter around the drop cut face over the portal with two rings of 6 m-long grouted bar.
Bench Crest	Secure the mesh at the bench crest using horizontal rows of zero-gauge screen strap.
Brow	Strapping (0-gauge screen strap)
Screen Options	Chain-link; Weldmesh or double-twist (Maccaferri-style) mesh
Upper Ramp	2.1 m long 19 mm or #6 grade 60 or better
Rock Surface Protection	If required
Shotcrete	Shotcrete arches and pre-grouting of sand and regolith horizons (beyond portal scope)

Portal Summary

BBA has summarized the recommended design basis criteria for the McIlvenna Bay portal access and box-cut development in the Table 16-12, Table 16-13, Table 16-14, and Table 16-15 below.

Table 16-12: Portal Arrangement Details

Item	Detail
Portal Face Location (UTM NAD83 ZN13)	N6055601.5, E640840.8
Rock Unit	Dolomite
Portal Width (min)	5.5m
Portal Height	5.5m
Overlying Bedrock Thickness	10.5m
Drop-cut Height	18m
Bedrock Cover Width: Height over Portal	~2
Access Ramp Slopes	-15%
Portal Floor Elevation	314m
Surface Elevation above Portal Face	332m
Surface Elevation at Portal Laydown	334.3m
Portal Laydown Grade	2%

Table 16-13: Boxcut and Excavation Design

Item	Detail
Inter-ramp Angle (IRA)	68 deg
Bench Height (BH)	5.3m
Catch Benches (CB)	2.3m
Total Height (Top of Bedrock)	16
CB:BH Slope	~1H:2.5V
Overburden Slope at Surface	1.5H:1V

Table 16-14: Estimated Boxcut Material Quantities

Item	Volume (Cu. M.)
Soil + Rock Cut Volume	18000
Rock Cut Volume	13275
Soil Cut Volume	4725
Box-cut Fill to Elevation	329.9m
Fill Volume	8920

Table 16-15: Portal and Boxcut Ground Support

Location	Ground Support
Drop Cut Face	2.4 m grouted rebar installed with #6 gauge, galvanized screen. Minimum bolt pattern of 1.2m x 1.2m increasing to 3-2-3 in the upper dolomite domain
Drop Cut Walls	Bolt walls down to ~1.5m above catch bench elevation. Drape the screening down lower wall. Any unravelling rock will accumulate at the base of the wall and remain on the catch bench.
Drop Cut Perimeter	Reinforce the perimeter around the drop cut face over the portal with two rings of 6 m-long grouted bar (see support drawings in Appendix D)
Bench Crest	Secure the mesh at the bench crest using horizontal rows of zero-gauge screen strap.
Brow	Strapping (0-gauge screen)
Screen Options	Chain-link; Weldmesh or double-twist (Maccaferri-style) mesh
Upper Ramp	2.1 m long 19 mm or #6 grade 60 or better
Rock Surface Protection	If required
Shotcrete	Shotcrete arches and pre-grouting of sand and regolith horizons (beyond portal scope)

16.5 General Underground Infrastructure

Several ancillary items are required to be constructed underground to meet regulatory requirements as well as for efficient operations.

16.5.1 Permanent Refuge Station

Refuge stations have been designed at each level to ensure that workers can safely seek refuge during underground emergencies. Workers will be able to access the nearest refuge station within 15 minutes of detecting ethyl mercaptan gas walking at an average rate of 1.2m per second. Refuge stations will meet the following criteria:

- 5m minimum clearance from drift to start of the refuge station
- ground support installed in accordance with design specifications
- concrete floor with a negative gradient for drainage to a collection point
- two concrete walls equipped with steel doors to create an airlock
- 1” water line and 1” drain line
- air recirculation/scrubbing system
- ventilation ducting
- ventilation fan
- air conditioning unit
- sink for non-potable water, supplied with water heater
- electrical panel
- tables and seating

- refrigerator
- microwave
- water cooler and water bottles
- clay for sealing the door
- first aid supplies and stretcher

The refuge stations can also be used as lunchrooms and offices for supervisors.

16.5.2 Explosive Magazine

The explosive magazine will be used to store emulsion and ANFO. The magazine will be 5.5m high by 6.0m wide and 20m long. It will be equipped as following:

- concrete floor with negative drainage gradient
- ground support per geotechnical recommendations
- sufficient space for explosives
- wall and lockable gate
- concrete stop block at entrance
- signage indicating explosive storage
- emulsion loader parking area

16.5.3 Cap Magazine

The cap magazine will be used to store high energy explosives and initiators. It will be 6m wide by 10m long and will include:

- ground support per geotechnical recommendations
- shelving
- wall with lockable gate
- concrete stop block at entrance
- signage indicating explosive storage
- concrete floor with negative drainage gradient

16.5.4 Maintenance Shop

An underground maintenance shop will be required for maintaining equipment that does not go to surface such as drills, jumbos, and bolters. Light vehicles and haul trucks will be maintained on surface. The underground shop will have two small maintenance bays, a larger crane bay, and a wash bay.

16.5.5 Latrine

Sanitary washroom facilities are required throughout the mine for the workers. These facilities will consist of an enclosed electrical incineration toilet with a hand washing station suitable for both genders. An excavation of 3 x 3m is required with an appropriate barrier to prevent mobile equipment from contacting the facility. This can be achieved by either a barrier or use of an elevated concrete pad of sufficient height with a guardrail to act as a barrier.

16.5.6 Oil Storage

The mining plan will utilize mobile fuel and lubricant vehicles to support the operation of mobile equipment. Small oil storages will be located throughout the mine to store pails of oils for use by drilling equipment (jumbos and production drills) and large mobile equipment (LHD's and trucks) when small quantities are required. The storages will be limited in size (quantity) to eliminate the requirement of a permanent fire suppression system. A typical storage will be:

- limited to less than 1,000 litres of oil as per Section 17-17 Sub-Section (1) and (2) of the Saskatchewan Mine Regulations
- 4 m wide x 6 m deep
- have a concrete floor with collection tray to enable collection of any spillage
- a spill kit to collect spills
- limit access of equipment via concrete barrier
- installation of appropriate signage identifying the storage
- installation of fire extinguisher on both sides of the storage
- sufficient room for storage of absorbent material for spill collection

16.6 Ventilation

The McIlvenna Bay ventilation system is designed as a pull system with total airflow requirements of 200m^3/s. There will be two 2-fan exhaust fan systems installed over two perimeter located exhaust raises. The decline will be the primary intake route and will include a propane heater system located at the portal. In addition, a minimal amount of intake air will also enter the mine via the egress system.

Regulators will be used on production levels to control the flow of air across the levels between the intake ramp level accesses and the exhaust raises.

Ramp, level development and stope ventilation when required will rely on auxiliary ventilation by way of auxiliary fans and either lay flat or hard plastic duct to distribute the air.

16.6.1 Airflow Requirements and Mobile Equipment List

Table 16-16 below indicates the overall airflow requirements for the mine based on the Canmet quantities and load or utilization factors, as applied to the projected mobile equipment list supplied. The total airflow estimated for the mine is approximately 200m^3/s. This volume is mitigated using BEV's in the form of haulage trucks.

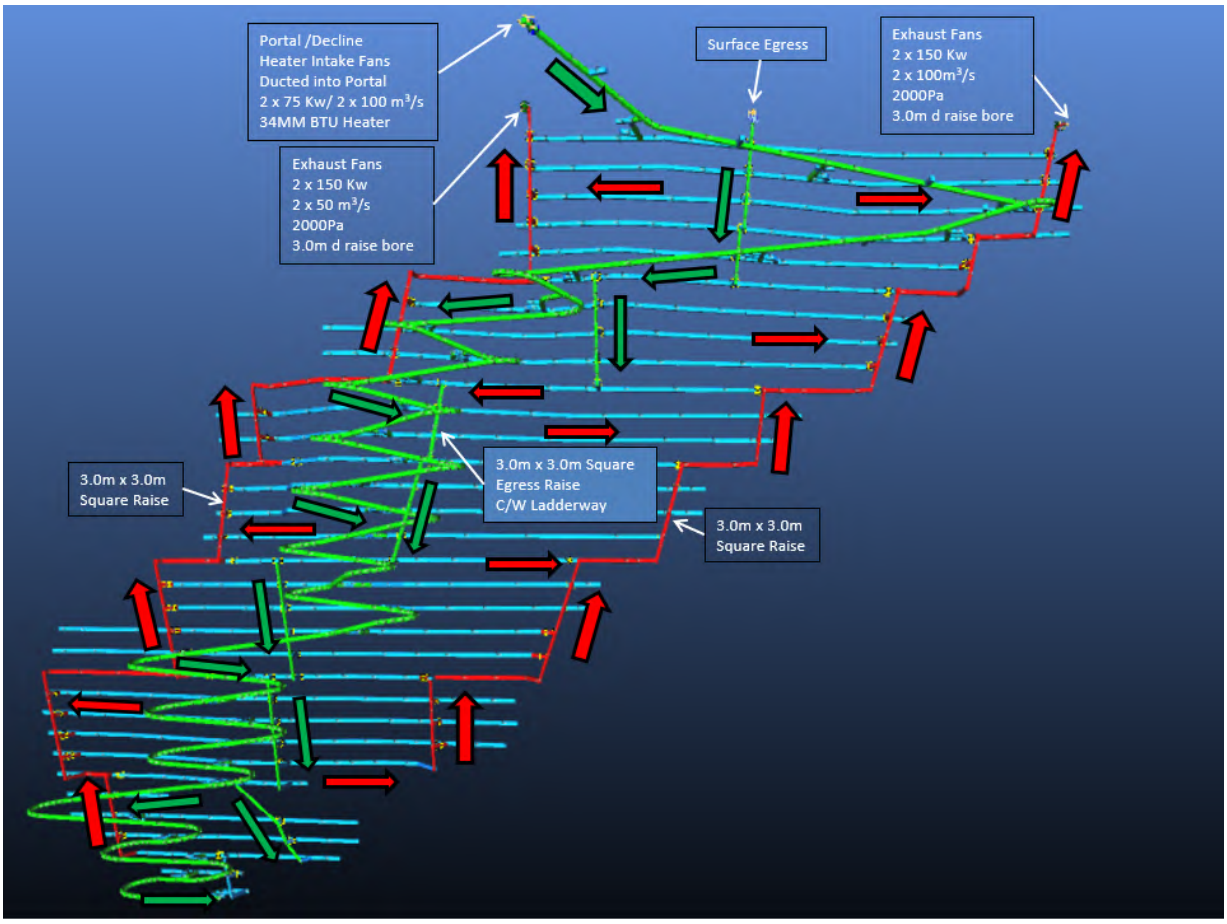
Table 16-16: Mobile Equipment and Ventilation Requirements

Equipment Description	Canmet Requirement (m^3/s)	Quantity	Load Factor	Total Airflow Required (m^3/s)
Jumbo 2 Boom	4.5	3	40%	5.4
LH Drill	3.0	3	40%	3.6
Bolter	1.7	3	25%	1.3
50 t Truck **	8.0	11	100%	88.0
3.7 cu.m LHD (LH307)	6.0	1	100%	6.0
7 cu.m LHD (LH517i)	12.0	7	100%	84.0
Scissor Lifts	1.7	3	25%	1.3
ANFO Loader	1.7	2	25%	0.9
Grader	2.4	1	35%	0.8
Boom Truck	2.4	1	35%	0.8
Mine Personnel Carriers	2.3	8	40%	7.5
			Total	**200**

16.6.2 Overall Ventilation

Figure 16-41 shows the ultimate overall ventilation schematic.

Figure 16-41: Ventilation Schematic



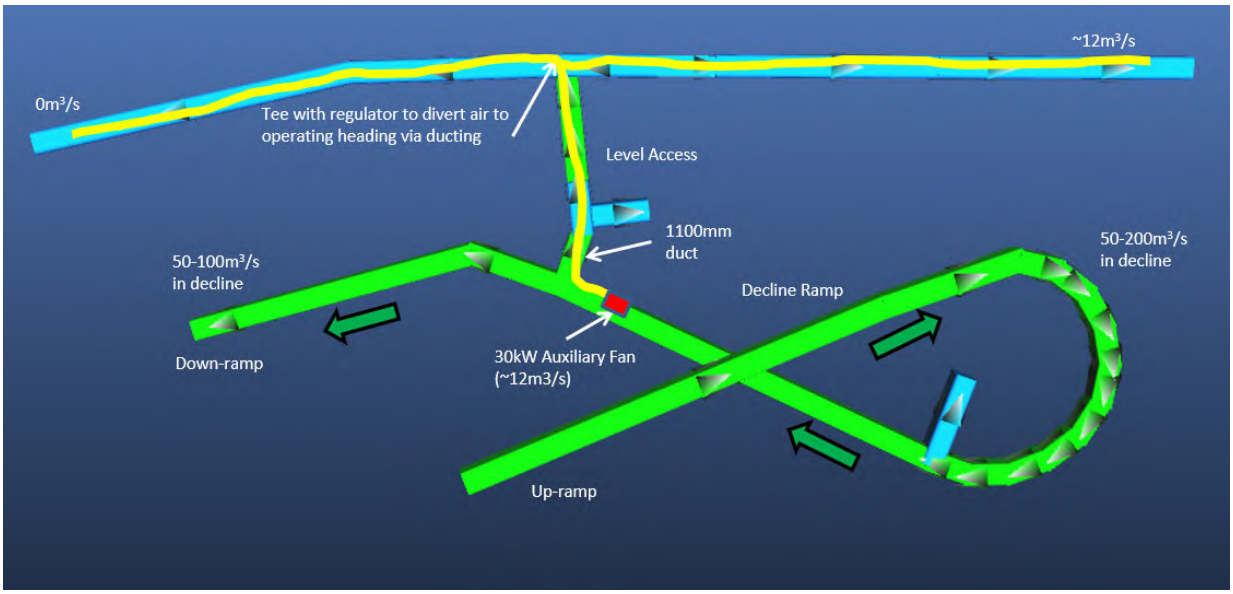
Decline Development Ventilation

Decline development will rely on auxiliary ventilation using either hard plastic duct or lay flat PVC supplying a minimum of 20m^3/s of air. The design is based on a LH517 LHD (12m^3/s) operating in the heading to load a 50-tonne battery electric truck (8m^3/s), For this purpose a 75kW axial fan was chosen connected to 1100mm ducting.

Level Development Ventilation

For level development fresh air will be pulled from the main decline into the active level development headings. A total of 12m^3/s will be provided for this development via a 30kW fan supplying a 1100mm duct. Air flow will be diverted to either end of the level via a regulator equipped "tee". This will allow for truck loading in the access using a LH517i LHD (12m^3/s) operating in the heading to load a 50-tonne BEV truck (8m^3/s) located in the ramp. Level development ventilation is shown below in Figure 16-42.

Figure 16-42: Level Development Ventilation

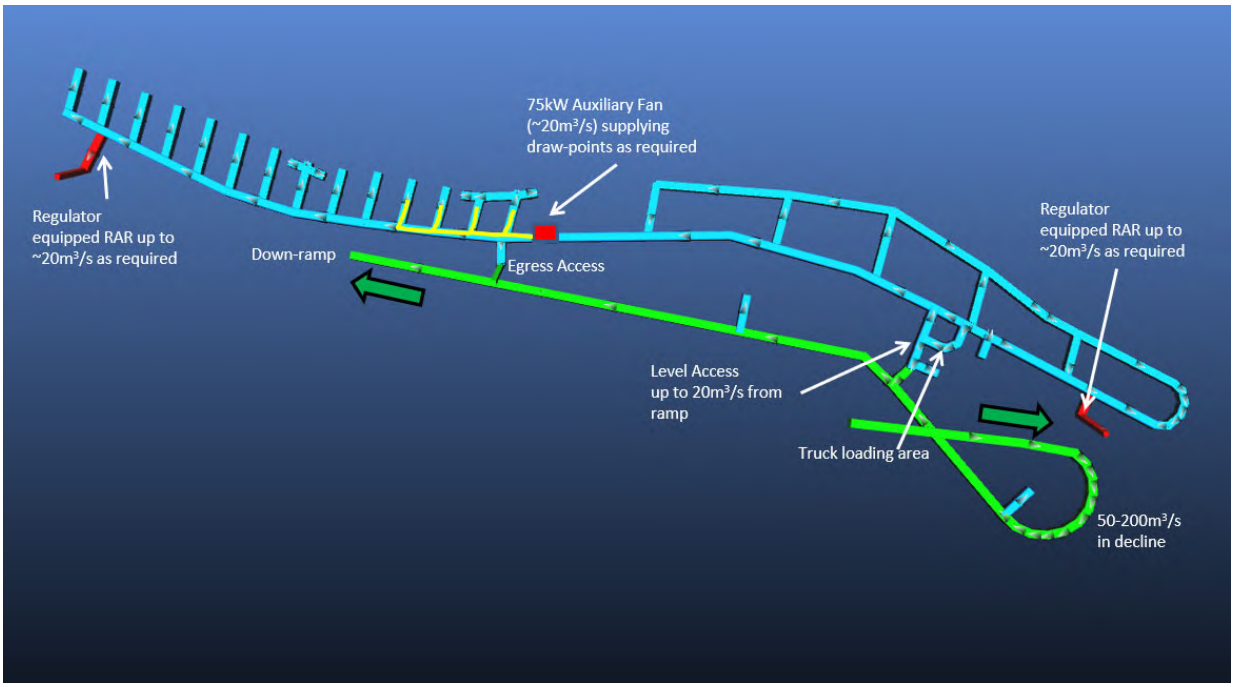


Production Ventilation

Production ventilation airflows will be regulated across each level using "drop-board" type regulators. These regulators will be located at each end of the level and set to regulate the flow based on the activities taking place on the level. Normally for a mucking sequence this flow will be set to approximately 20m^3/s. This will allow for a 50-tonne BEV truck and LH517 LHD loader to operate in this area. Truck loading will take place in the truck turn-a-round. Lesser amounts of air would be required for drilling or screening activities, once again these would be regulated based on the location of the activities on the level.

Ore accesses and captive development on these levels longer than 25m in length will require the use of auxiliary fans and ducting. Production level ventilation is shown below in Figure 16-43.

Figure 16-43: Production Ventilation



Main Surface Fans

Exhaust Fans

A two-fan parallel Arrangement-4 fan type installation was chosen for this project. Each of the two exhaust raises will have such an installation.

The total flow for these fans is 50m^3/s each for a total for 200m^3/s. These fans were specified in such a manner that a single fan operating, in the event of a failure, can supply 60% of the total flow or 30m^3/s. In addition, each fan will be the same model which minimizes the requirements for spares. The exhaust fan specifications are shown in Table 16-17.

Table 16-17: Exhaust Fans

McIlvenna Bay Exhaust Fan Installations BBA Requested Fan Spec	West RAR	East RAR
Exhaust Fan Volume (m^3/s) per fan (two fans per install)	50	50
Installation Volume (m^3/s) Total	100	100
Static pressure at collar (before added losses) (Pa)	1850	1850
Power (kW) per Fan	150	150
Power (kW) Total	300	300
Connection Opening	3.0m diameter concrete collar	3.0m diameter concrete collar

Each exhaust fan installation will come complete with an E-House to house all VFD's and switchgear.

Heater and Heater Fans

A propane-fired mine air heating system will be installed at the portal which will include two 75kW fans. This heater system will be rated at 30MM BTU and will be designed to raise the air from -40°C to +2°C. The specifications for the surface heater and associated fans are shown in Table 16-18.

Table 16-18: Intake Heater and Fans

McIlvenna Bay Intake Heater and Fan Installations BBA Requested Fan Spec	Portal Fresh Air
Portal Air Volume (m^3/s)	200
Temperature Rise (^{0}C)	-40 to +2
Total Heater BTU	30,000,000
Fan(s)	2 x 75 kW
Intake Air Ducting Size	2 x 1.8 m diameter
Portal Opening 5.5 x 5.5m (m^2)	30.25 m^2

This intake heater system will come complete with a heater control room, VFD's and stench emergency warning system.

Auxiliary Fans

To simplify the use of auxiliary fans only two fan sizes are specified for this project. One size is used for on level airflow requirements and the other is used for large equipment, decline development and drawpoint ventilation. The specifications for the auxiliary fans are shown in Table 16-19.

Table 16-19: Auxiliary Fans

McIlvenna Bay Intake Auxiliary Fans BBA Requested Fan Spec	Ramp Development	Stoping/Access Development
Fan Diameter (mm)	1100	910
Fan Volume (m^3/s)	75	30
Fan Motor Size (kW)	2250	1625
Fan Total Pressure (Pa) at Operating Point	2250	1625

These fans will be matched with plastic ducting to maximize the airflow distance and minimize power requirements.

16.7 General Underground Services

16.7.1 Compressed Air

Underground mobile equipment will be equipped with onboard compressors and mobile compressors will be purchased for other equipment. Refuge stations will be supplied with bottled air. For these reasons it is not anticipated that a central compressed air system will be required for the McIlvenna Bay project.

16.7.2 Service Water

The service water supply is used for the equipment requirements, dust suppression, washing, and other miscellaneous uses (mining systems). It may also be used for fire suppression. The service water requirements are listed in the Table 16-20 below. At maximum flow, 2,843 L/min is required for the 19 pieces of equipment and ancillary services. This value has been used for the pressure drop calculations.

Table 16-20: Service Water Consumers

Description	Units	Utilization	Flow Rate, m3/h		
			Min	Average	Max
Jumbos	3	75%	4	9	12
Bolters	5	75%	4	15	20
Production Drills	3	80%	12	20	36
Raise Machines	1	75%	5	8	11
Wash Bay	1	25%	-	5	1
Misc.	6	-	-	60	91
Total				127	171

Service water is supplied primarily from the two dewatering (MudWizard) systems located underground (Figure 16-44), which take general seepage water and waste service water and treat it within clarifier tanks to ensure adequate quality for consumers. Each MudWizard skid has a nominal capacity of 100 m^3/h and consists of static mixers, pumps, and settling/decantation cones. A backup water supply is available from the mill via pipes from surface.

Figure 16-44: MudWizard System

16.7.3 Potable Water

Potable water as part of the underground services is the water supply that is intended for human consumption, or for a human consumptive use as per Saskatchewan 'The Water Regulations, 2002'. At McIlvenna Bay, potable water will be supplied via portable jugs distributed throughout the mine. The jugs will have to be taken to surface regularly to be refilled from a potable water source on surface. There will be no piped distribution system for potable water.

16.7.4 Fire Water

Fire water systems provide an adequate water supply for firefighting in the event of a fire within the mine and may be active or passive. In this section, the need for fire water is analyzed in the broader context of fire protection requirements.

Chapter S-15.1 Reg 8 of The Saskatchewan Employment Act ('The Mines Regulations, 2018') governs the fire suppression requirements for the underground infrastructure.

Section 17-6(1) specifies the following:

> *An employer or contractor shall ensure that there are suitable and adequate portable fire extinguishers and other suitable and adequate firefighting equipment: (iv) on each vehicle and at each stationary diesel engine; (v) at every underground crusher station, electrical installation, pump station, shaft station, belt conveyor drive unit, service garage, fuel station, explosive storage area, flammable liquid storage area and hot work area; and (vi) at any other area that is designated as a fire hazard area...*

Accordingly, adequate portable fire extinguishers will be supplied at each pump station, repair bay, fuel storage area, explosive storage area, hot work area, and any other area designated as a fire hazard. No fire water system is required for these facilities.

Sections 17-8(1) and 17-8(3) state that:

> *A fire suppression system required pursuant to this section must: (a) include a sprinkler system, a dry chemical system, or any other system capable of suppressing the expected type and size of fire; […] an employer or contractor shall provide a fire suppression system that meets the requirements of subsection (1): (a) if reasonably practicable, on each piece of fixed equipment underground that contains more than 175 litres of diesel fuel, grease or oil…*

Fuel storage facilities will be equipped with their own automatic dry chemical fire suppression systems and no fire water system will be required for these facilities.

As a consequence of the above and in accordance with The Mines Regulations, no water-based fire suppression system is required for the underground infrastructure. In case of a fire however, the supply of water to the mining systems can be stopped, and the service water distribution system will be immediately available to support the firefighting efforts.

16.8 Backfill Distribution

Backfill to be used at the McIlvenna Bay project is paste fill for transverse stopes and waste rock for Avoca stopes.

16.8.1 Paste Backfill

Filtered tailings from the process plant backfill section will be used as aggregate for paste backfill. The backfill plant is incorporated into the process tailings dewatering circuit and includes mixing, batching, and delivery equipment. More information can be found in Section 17.0. Portland cement will be stored in a hopper, and accurately dispensed into paste batches before being pumped to the underground voids via the backfill piping system. The main paste backfill pipe will be installed in the ramp initially and in the vertical conveyor borehole in year 3. Once underground, the paste will be transported through a network of pipes to reach the mined-out stopes. The main trunk line will branch off into each level and subsequent branches will be installed in each crosscut. Crews will move the lateral piping to the crosscut being mined as part of the stope cycle.

Paste Backfill Strengths

Paste backfill requires a minimum strength to ensure wall stability in vertical exposures. The guidelines for cemented paste backfill (Factor of Safety = 1.5 for unit density of 20 kN/m^3) are summarized in Table 16-21 for varying HW to FW spans for stope heights of 30m.

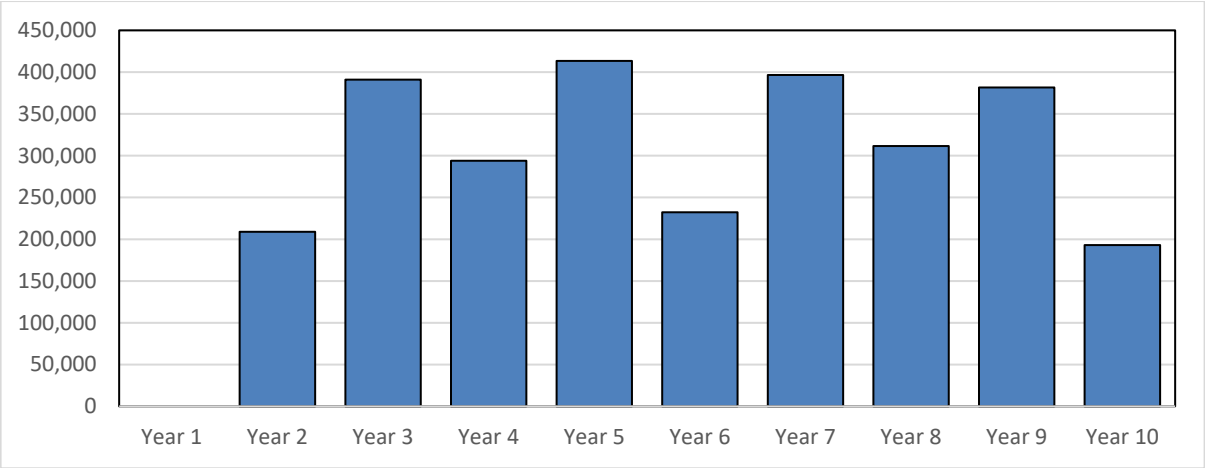
Table 16-21: Paste Backfill Strength Requirements for Varying Spans

HW to FW Span (m)	UCS (kPa)
5	131
10	233
15	315
20	382
25	438
30	485
35	525

Paste Backfill Quantities

The paste backfill schedule is shown in Figure 16-45. There is no paste backfill required during year 1 because all of the production is generated from Avoca stopes.

Figure 16-45: Paste Backfill Tonnes per Year



Paste Backfill Bulkheads

A bulkhead will be constructed at the drawpoint after each stope is empty. These bulkheads will prevent the paste from flowing into the crosscuts. The bulkheads are constructed by bolting a frame to the around the perimeter of the drawpoint drift and spraying it with shotcrete until it is at least 30 cm thick.

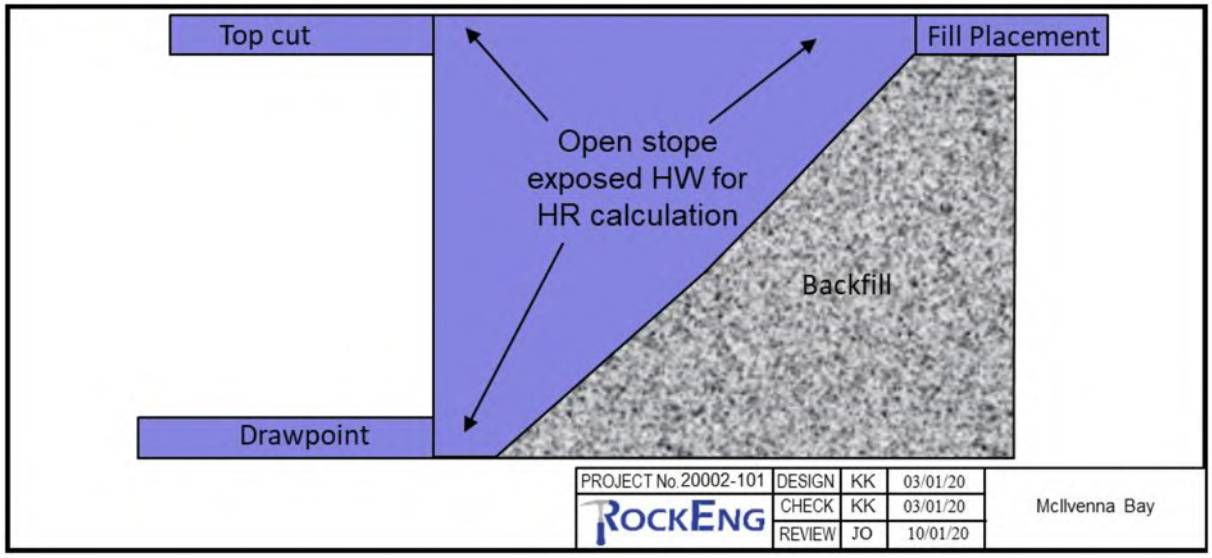
16.8.2 Waste Rockfill

Waste rock from lateral development headings will be used to fill Avoca stopes. LHD's will be used to transport waste from the waste stockpile to the stope along the footwall drift. The backfill schematic is shown in Figure 16-46. Backfill is placed from the east as the Avoca front advances to the west. The stopes are not tight filled, leaving void for production blasting. The backfill must advance at a rate sufficient to maintain the span of exposed ground below the desired threshold.

Waste Balance

Development waste will initially be stockpiled on surface and will be backhauled as required once production commences. In most cases, development waste being generated from underground headings will be mucked directly to the waste stockpile on the same level, and then hauled directly to the stope being backfilled to minimize re-handling.

Figure 16-46: Long Section View of Schematic Showing Avoca Backfill Sequence



16.9 Hydrogeological Considerations

North Rock Mining Solutions (NRMS) completed a hydrogeological analysis of the McIlvenna Bay Project in 2018. The objective of the work was to:

- complete gap analysis to optimize the scope of work
- conduct a sequence of DDH holes testing sequence (Profile tracer tests and Slug testing) to determine the conductivity and the water bearing zones
- conduct a sequence of seepage measurement in a nearby lake to understand surface/groundwater exchange
- prepare a numerical model to assess water inflow and environmental impacts.

Hydrogeological site investigations were undertaken by Golder Associates (2012) and most recently by NRMS in 2018. The 2018 program included:

- seepage testing on the shore of Hanson Lake
- Profile Tracer Tests (PTT) in 15 holes
- slug testing in the holes tested by PTT
- ground water sampling

Findings from the NRMS hydrogeological analysis are summarized in the following subsections.

16.9.1 Testing Procedures

The PTTs and the slug tests provide both K values of the formation around the holes. However, when the global concentration decreases during the PTTs, the results are more representative than the slug test as it provides a regional value of K. The slug test results however, allows us to obtain values when there is no dilution measured. It is critical to mention that the PTTs were done in order to isolate

contrasting flow along holes. Then the test durations were approximately a few hours. In a permeable environment, the dilution occurs usually within a few minutes and contrasting flow zones are obvious. When the dilution is very slow, it is not required to continue the measurements because no real flow zones have been identified and because the slug tests provides the average K values along the holes.

Along each hole, it was not possible to isolate any water bearing faults or features. That does not mean that those features do not exist. Among other things, the orientation of the tested holes is biased in a direction discordant with mineralization. None of the tested holes have been drilled parallel to the ore body, therefore water bearing structures could exist that are discordant with mineralization but have not been located.

To de-risk the Project, a drilling and testing campaign concordant with mineralization is recommended. However, there is no evidence of a permeable structure in the surrounding area with the current results.

16.9.2 Sedimentary Rock

The sedimentary rocks present at surface, such as the dolomite and the sandstone, clearly have higher K values (1.0e-7m/s). However, according to all the results, the K values do not seem to be significantly high to cause problems during the completion of the ramp. However, during this completion and once below the groundwater table, some pilot holes should be drilled ahead of the ramp within the sedimentary rock to finalize pre-excavation investigation and mitigation.

16.9.3 Effect of Hanson Lake

The results from simulations are not suggesting a significant effect of recharge from the Hanson Lake. In fact, even if the current exchange between surface and groundwater is significant at some places that does not mean a high relationship between the mine drainage and the lake. Rock permeability will control the flow first, and the lake will maintain the recharge. However, the sediments in the shoreline are relatively permeable and if an unknown permeable structure hydrologically connects the lake and the mine, inflow into the mine workings could be induced. There is no evidence of such a structure with current data.

16.9.4 Simulation Results

Simulation results suggests a relatively small inflow at final stage of operation. This is due to a low permeability rock, and to the fact that no contrasting water bearing zones have been located nor identified. The numerical model remains simple and convergence satisfactory, which make the model stable for prediction. Calibration is acceptable and generally agrees with the simulated head. Therefore, there is no data suggesting high groundwater inflow for the Project.

16.9.5 Inflow Mitigation

In a low permeability environment, the pressure of water will remain during most of the operation. Then water will evacuate through higher permeability features. Drainage or pumping is not efficient in this type of rock and operation. In order to minimize natural groundwater inflow, it will be better to grout during the operation than to drain specific features (at least in mine development such as the upper ramp).

16.9.6 Water Chemistry

Water chemistry does not respect the criteria in Saskatchewan for water discharge, and thus water treatment systems have been included in the infrastructure designs. Further work is required to enable measurement of more representative water flows.

16.10 Mine Dewatering

16.10.1 Overview

The purpose of the dewatering system is to remove the drainage water created from mining operations processes as well as the natural inflow of groundwater. This is achieved through a network of interconnected sumps, ditches, drain holes, pumps, and piping. The design is based on clear water pumping and includes solids separation devices. Table 16-22 gives the design basis for the dewatering system.

The dewatering system is split into three sections of the Access Ramp (Figure 16-47: Mine Dewatering Schematic). Water is transported by gravity to the nearest pump station that collects the water from the levels above (through cascading sumps and drain holes, Wilson sumps, and a slime retention system that separates the solids from the drainage water). Pumps cascade the clear water up to the next pump station, and ultimately to surface where it is stored for use as underground operations water.

Figure 16-47: Mine Dewatering Schematic

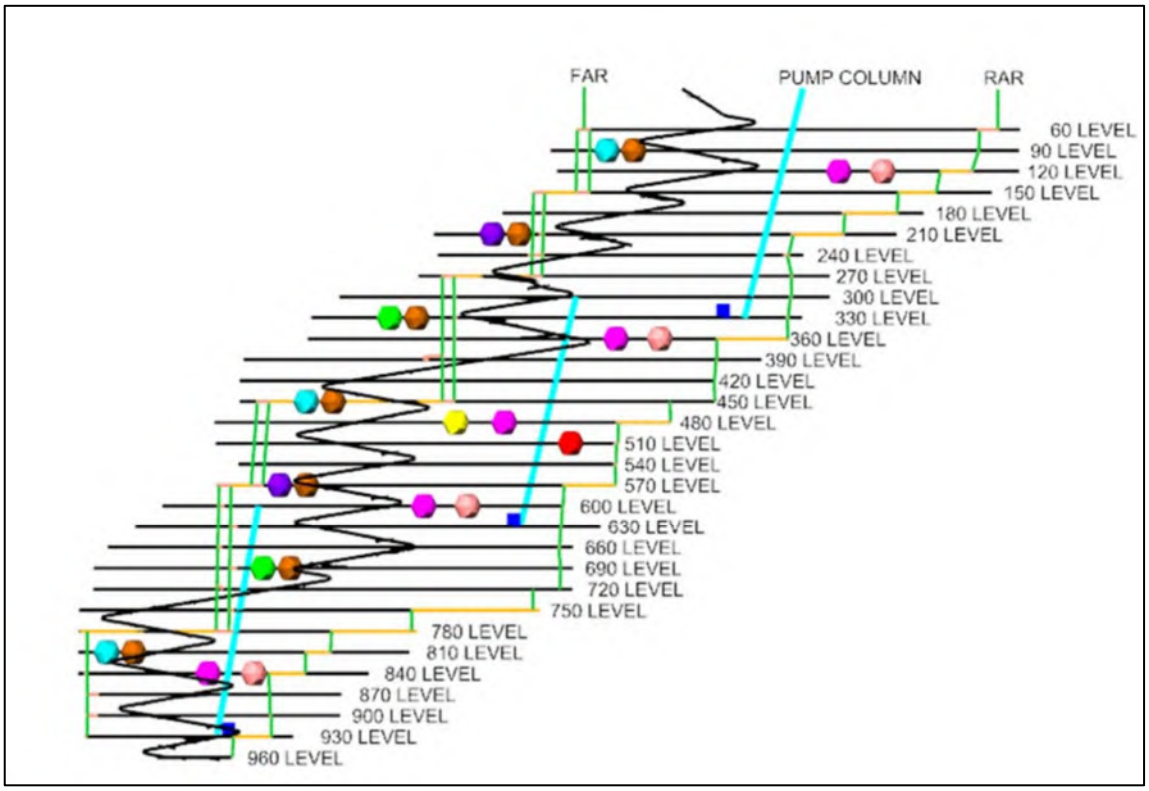


Table 16-22: Dewatering Design Specifications

Dewatering requirements		
Total equipment water	m^3/h	98.9
Water removed with muck		8.8
Total groundwater inflow		79.5
Total dewatering requirements		169.6
Pump Station #1		
Average Flow	m^3/h	169.6
Maximum pump duty per day	h	16
Design Flow	m^3/h	254.4
PS#1 Clear Water Sump		
Buffer time	h	3.42
Live volume	m^3	580.7
Pump Station #2		
Maximum Average Flow	m^3/h	144.0
Maximum pump duty per day	h	16
Design Flow	m^3/h	215.9
PS#2 Clear Water Sump		
Buffer time	h	3.09
Live volume	m^3	444.4
Pump Station #3		
Maximum Average Flow	m^3/h	118.3
Maximum pump duty per day	h	16
Design Flow	m^3/h	177.5
PS#3 Clear Water Sump		
Buffer time	h	2.93
Live volume	m^3	347.1

16.10.2 Pump Stations 1-3

Mine water from various levels (60 to 270 for PS#1, 360 to 570 for PS#2 and 660 to 870 for PS#3) is collected via a network of sumps and drains and is directed to one of the associated settling sumps. Clear decant water from the settling sumps flows by gravity to the related clear water sump, which for the upper two stations will also collect the clear water pumped from the lower pump station. Clear water is then pumped to the surface with one multistage centrifugal pump (one running, one standby).

16.10.3 Settling Sump

The solids settling system is identical for all pump stations. It consists of two decanting sumps (Catchment No.1 and Catchment No.2), each fenced with a Sturda Weir. Solids are separated from the water by the settling action and the filtration of water through the Sturda Weir membranes. The clear

water seeping through the weir is directed through a trench and drain hole to the Clear Water Sump. Two sumps are required as they see alternate use. While drainage water is sent to Catchment No.1, the slimes in Catchment No.2 are left to dry. Once dry, the Sturda Weir can be opened, and the slimes removed with an LHD. When the slimes level in Catchment No.1 gets close to the top of the weir (regular visual inspections required), drainage water is then directed to Catchment No.2 and the slimes in Catchment No.1 left to dry. When Catchment No.2 is full the cycle repeats.

16.10.4 Level Sump

Each mine level has a level sump that collects the mine drainage water on that level, as well as the water from the levels above by gravity flow through a drain hole. The end of the drain hole from the level above comes out above the sump; a short pipe spool connects the drain hole to the sump to avoid splashing. Unless located at a pump station level, the contents of the sump are transferred to the level sump below by gravity, through a drain hole located at the bottom of the sump.

16.10.5 Wilson Sump

When the extent of a mine level reaches a certain point, it may become impractical to drain all the water to the level sump (to be validated with final mine layout). Indeed, the grading of the level may not lend itself to having all the drainage water flow by gravity to the level sump, which is typically located in the vicinity of the ramp access. In such cases additional Wilson sumps will be added. These sumps collect the water from a given area, and then pump to the level sump (via the main drift) with a small submersible slurry pump. This will be a dirty water pumping application.

16.11 Underground Electrical and Power Distribution

Mine site electrical and power distribution has been divided into two distinct areas: surface electrical distribution and underground electrical distribution. The surface electrical distribution is described in Section 18.5.

16.11.1 Underground Power Supply

Underground electrical power supply originates at the surface mine services substation. Two primary 13.8kV feeders will supply the load of the underground mine plant. The first primary feeder will supply levels 60 through 630 and the second will supply levels 630 through 960.

A third feeder has been proposed to supply the above ground mine ventilation and will consist of an overhead feeder including two pumping stations. All three feeders will be egressed out of the mine services substation via an underground concrete encased ductbank (or direct buried), to ensure that the risk of contact with electrical plant is minimized in and around the mine surface stockpile area.

16.11.2 Underground Electrical Distribution

The underground electrical distribution system will be supplied by two primary feeders from the mine services power distribution switchgear on the surface; the first primary feeder will supply levels 60 through 630 and the second will supply levels 630 through 960.

Electrical Load Segregation

For increased operational reliability, the allocation of electrical loads on to separate electrical distribution systems should consider the mechanical design of the equipment in question. For example, where there are parallel mechanical systems, an effort should be made to have separate electrical distribution systems supplying each. Alternatively, where there is a standby unit, consideration should be given to supplying it from a separate system other than the one supplying the duty equipment. Good judgment shall be exercised in load segregation decisions, as the increased reliability may not be worth the additional costs.

Switchgear

Primary feeder design utilizes switchgear, allowing the primary feeder cable to be extended as the mine develops without taking an outage. Switchgear will consist of heavy-duty portable enclosures containing a disconnect switch to sectionalize the primary feeder and a fused disconnect switch to tie a power distribution center to the primary feeder.

Power Distribution Centers

The prefeasibility design takes into consideration a standard power distribution center for each distribution area (mine level), consisting of a heavy duty skid containing all necessary disconnect switches, step down transformers, circuit breakers, ground fault relays, surge arresters, neutral grounding resistors, lighting transformers and lighting distribution panels, which are necessary to energize mining plant, lighting, refuge stations, and lunchrooms. Exceptions to this rule have been made for the three pumping stations and the underground crushing station where appropriately sized power distribution centers will be specified.

Operability

The underground electrical distribution system design takes into account Canadian electrical safety codes and standards to deliver a state-of-the-art design that is safe to operate and maintain. Each level of the mine will have a power distribution center that functions independently of all other levels, allowing for operations to be planned independent of the work occurring elsewhere in the mine.

Communications will be provided to each power distribution center on the underground electrical distribution system, allowing for a shunt trip of the main circuit breaker and monitoring of the main protection devices contained within each power distribution center. This functionality will allow the operator to monitor power centers from the control room and determine which power centers should be disconnected to maintain supply in the event of an outage on the main service feed.

Reliability

Where practicable, elements of the underground electrical distribution system will be specified for underground mining duty with a minimum design life of twenty years.

The underground electrical distribution system will be designed such that a fault on a single power distribution center will not affect the primary feeder, thereby maintaining continuity of service for all unfaulted elements of the underground distribution system.

In the case of a fault on one of the underground primary feeders, a primary feeder tie will be made on levels 600 and 630 to provide an alternate (emergency) supply.

16.12 Automation and Instrumentation

The below listed underground mine systems are to be considered for implementation and at a minimum shall support the following functionalities.

16.12.1 Central Blasting System

The central blasting system will be used for remote detonation of development and production blasts from a central location on surface. The system will communicate from surface to various areas underground via analog telephone lines, Leaky Feeder radio system or the LAN infrastructure.

16.12.2 Micro-seismic

The micro-seismic system will monitor seismic activity underground. The system consists of accelerometers and geophones installed at specific locations within the mine to determine the hypocenter and magnitude of seismic events. The system will communicate either via dedicated fiber optics or via the LAN network supporting the IEEE 1588 precision time protocol.

16.12.3 Block Lighting System

The ramp traffic management will consist of traffic indication lights installed at all ramp intersection and truck turn around locations. The purpose of the system is to reduce and eliminate traffic congestion and ensure that loaded haulage trucks travelling up ramp have priority over other traffic on the ramp.

16.12.4 Mobile Equipment Data Loggers

Data loggers will be used on all production mobile equipment to collect production and maintenance data. Mobile equipment such as LHD and haul trucks will also collect cycle times and weights, eliminating manual entries. Data will be collected and stored onboard and uploaded via the wireless infrastructure when in range.

16.12.5 Radio Voice Communications

Two-way radio communications shall be installed within all lateral development drifts. This system will be used as a primary form of voice communications between underground personnel and surface personnel. The system will be configured to have the multiple channels to support operations including a global broadcast channel for emergency communications.

16.12.6 Fiber Optic and Ethernet Networks

A fiber optic network shall serve as the main communication backbone network for both surface and underground locations. Secondary systems/networks (e.g. control network, etc.) shall be supported by this network. The fiber optic network shall consist of fiber optic cables distributed throughout underground to key locations (e.g. substations stations, shops, etc.) via drifts. Additionally, fiber optic equipment such as fiber optic patch panels and splice panels shall be installed at key locations underground. A redundant fiber loop installed via vertical pathways (i.e. secondary egress or boreholes for electrical cables) should be considered to provide maximum uptime and a robust infrastructure.

An Ethernet (TCP/IP) will consist of network switches installed throughout the mine at key locations to ensure connectivity to end devices. The network will be supported by the fiber optic network and Ethernet network switches installed throughout the mine site.

The business and control network will share the same physical network and will be virtually segregated with the use of VLANs.

16.12.7 Digital Wireless Network

A wireless network shall be installed at key underground locations to support various mine systems such as wireless VoIP (secondary voice communications), wireless computer connections, connectivity for data loggers, etc. This network will consist of wireless access points connected to/interfacing with the Ethernet and fiber optic networks.

16.12.8 Tracking System

Tracking tags and supporting infrastructure for the location of personnel, equipment and assets for logistical and safety reasons shall be installed at key locations underground. This system shall be supported by the digital wireless network and the fiber optic network.

16.12.9 Air Quality Monitoring Station

Air Quality Stations will be installed in key locations underground (i.e. return air raises) to monitor gases for trending and potential ventilation control applications. This system will also be used for ensuring gases have cleared from the mine after blasting prior to workers re-entering.

16.12.10 Mobile Equipment

In the final selection of mobile equipment, OEMs will be asked to offer autonomous and tele-remote solutions. The primary objective of utilizing autonomous equipment is to increase utilization of the equipment and to remove personnel from underground or specific hazardous areas.

The following equipment should be considered for autonomous applications:

- Top hammer drills – single hole automation
- LHDs
- Haulage trucks.

16.12.11 Autonomous Systems

The control system network(s) shall provide the automation and remote operation for all fixed plant and underground systems. The control network manufacturer shall be the same throughout the mine site (i.e. surface and underground). HMIs shall be located in strategic underground locations, near process areas such as the dewatering pump rooms other locations where local control and monitoring is required. A Data Historian will be used a key component of the operation monitoring function of the site. The system is a single central application server, which includes the advanced calculation engine, analysis framework, and server database, to handle calculations based on collected data and to host custom interfaces.

16.12.12 Underground Messaging Screens

Digital messaging boards will be placed at key locations such as shaft stations, refuge stations and at the entrance to the ore zone on major levels to provide general information such as exclusion zones, production targets, KPIs, and air quality and will also provide mobile equipment location fed from the tracking system.

16.13 Mobile Equipment

16.13.1 Equipment Selection

Mobile equipment was selected that is appropriate for the mining methods employed and can support the production rate. Equipment fleet costs and performance are supported by OEMs and only commercially available products were considered. It is noted, however, that further electrification should be considered for the McIlvenna Bay project as OEM's demonstrate the efficacy of new equipment in the coming months and years. Key elements considered in the equipment selection include:

- mine design
- mining method
- productivities
- fleet standardization of key components
- maintainability
- economics

The equipment fleet composition allows completion of every mining process such as lateral development, production, secondary ground support, underground construction, and utility installation.

Table 16-23 below presents the complete equipment list including type, function, make and model.

Table 16-23: Equipment Selection

Equipment Type	Function	Make	Model
Jumbo	Development drilling	Sandvik	DD-421
ANFO Loader	Development explosive loading	MacLean	AC3
LHD	Transferring muck from drawpoint our heading to remuck or truck	Sandvik	LH-517
LHD - small	Miscellaneous	Sandvik	LH-307
Bolter	Primary ground support installation	Sandvik	DS-411
Scissor Truck	Utility support	MacLean	SL3
Cable Bolter	Secondary ground support installation	Sandvik	DS-421
Raise Bore	Initial void space for transverse stoping	Redpath	RB40
Top Hammer Drill	Production drilling	Sandvik	DL-311
Emulsion Loader	Production explosive loading	MacLean	CS3
Haul Trucks	Transfer muck from stockpiles to surface or underground crusher	Sandvik	Z50
Boom Truck	Material handling	MacLean	BT3
Fuel/Lube Truck	Provide fuel and lubricants to production equipment	MacLean	FL3
Transmixer	Transportation of shotcrete/concrete	MacLean	TM3
Telehandler	Material handling	JCB	509-23
Shotcrete Sprayer	Application of shotcrete	MacLean	SS2
Rock breaker	Reduce size of large rocks	Sandvik	XM500HD-HU55-3288
Personnel carrier_1	Crew transportation	Toyota	Land Cruiser - Side Seating Van
Personnel carrier_2	Staff transportation	Toyota	Land Cruiser - Pick Up Box
Personnel carrier_3	Service vehicle	Toyota	Land Cruiser - Flat Deck
Grader	Road maintenance	Miller	BG 100M

16.13.2 Equipment Operating Hours

Operating hours are based on the Deswik production schedule and are derived from first principles and industry benchmarks. Truck equipment hours were calculated using simulation data provided by Sandvik to estimate the total haulage requirements. BEV haul truck estimates assume waste is transported to an Avoca stope that is being filled if one is available, otherwise it is transported to surface. Some waste will be backhauled from surface to underground during the first year of production.

The operating hours were used in conjunction with the schedule, maintenance hours and planned downtime to estimate the complete equipment fleet. As the production schedule varies, the fleet size changes accordingly. Table 16-25 below presents the annual equipment in use for the main types of equipment. Table 16-26 shows the equipment procurement schedule based on the productivity of the equipment, the hours used, and the expected life span.

Table 16-24 below presents the yearly operating hours per equipment.

BEV haul truck estimates assume waste is transported to an Avoca stope that is being filled if one is available, otherwise it is transported to surface. Some waste will be backhauled from surface to underground during the first year of production.

The operating hours were used in conjunction with the schedule, maintenance hours and planned downtime to estimate the complete equipment fleet. As the production schedule varies, the fleet size changes accordingly. Table 16-25 below presents the annual equipment in use for the main types of equipment. Table 16-26 shows the equipment procurement schedule based on the productivity of the equipment, the hours used, and the expected life span.

Table 16-24: Mobile Equipment Hours

	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	
Jumbo	49,300	6,079	7,227	7,207	7,180	5,970	4,159	3,568	3,588	3,309	1,013	0
ANFO Loader	41,083	5,066	6,023	6,006	5,983	4,975	3,466	2,973	2,990	2,758	844	0
LHD	426,261	9,911	28,299	49,866	58,574	51,917	56,457	52,117	48,156	44,827	26,115	21
Bolter	115,032	14,184	16,863	16,817	16,752	13,931	9,705	8,325	8,371	7,722	2,363	0
Scissor Truck	14,086	1,737	2,065	2,059	2,051	1,706	1,188	1,019	1,025	945	289	0
Raise Bore	82,986	0	4,218	9,036	8,022	9,559	8,698	10,206	11,968	13,241	8,032	7
Top Hammer Drill	107,340	616	5,737	11,958	13,108	12,827	13,634	13,645	13,720	13,747	8,339	7
Emulsion Loader	24,903	150	1,334	2,777	3,071	2,981	3,189	3,170	3,164	3,153	1,912	2
Cable Bolter	38,049	1,771	3,261	4,699	4,985	4,573	4,298	4,122	4,121	4,035	2,182	2
Haul Trucks	273,055	11,475	23,733	32,685	31,943	29,592	31,181	31,068	31,259	31,295	18,810	16

Table 16-25: Active Equipment by Year

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	
Jumbo	5	6	6	6	5	4	3	3	3	1	0
Explosive loading	2	2	2	2	2	1	1	1	1	1	0
LHD	3	8	13	15	13	14	13	12	12	7	1
Bolter	7	8	8	8	7	5	4	4	4	2	0
Scissor Truck	1	2	2	2	1	1	1	1	1	1	0
Raise Bore	0	2	3	2	3	3	3	3	4	2	1
Top Hammer Drill	1	2	4	5	5	5	5	5	5	3	1
Loader (prod)	1	2	3	3	3	3	3	3	3	2	1
Cable Bolter	2	2	3	3	3	3	3	3	3	2	1
Haul Trucks	3	5	7	7	6	6	6	6	6	4	1

Table 16-26: Mobile Equipment Purchase Schedule

Underground Equipment	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Jumbo	**6**	5	1	0	0	0	0	0	0	0	0
ANFO Loader	**7**	5	2	0	0	0	0	0	0	0	0
LHD	**21**	3	5	5	2	0	0	3	3	0	0
Bolter	**8**	7	1	0	0	0	0	0	0	0	0
Scissor Truck	**5**	2	1	1	1	0	0	0	0	0	0
Raise Bore	**3**	0	2	1	0	0	0	0	0	0	0
Top Hammer Drill	**4**	1	1	1	1	0	0	0	0	0	0
Emulsion Loader	**2**	1	1	0	0	0	0	0	0	0	0
Cable Bolter	**2**	1	1	0	0	0	0	0	0	0	0
Haul Trucks	**27**	3	3	3	3	3	3	3	3	3	0
Boom	**4**	2	1	0	0	1	0	0	0	0	0
Lube	**1**	1	0	0	0	0	0	0	0	0	0
Transmixer	**1**	1	0	0	0	0	0	0	0	0	0
Telehandler	**1**	1	0	0	0	0	0	0	0	0	0
Shotcrete	**1**	1	0	0	0	0	0	0	0	0	0
Rockbreaker	**2**	1	0	1	0	0	0	0	0	0	0
Personnel Carrier 1	**15**	6	1	2	0	0	2	2	2	0	0
Personnel Carrier 2	**7**	3	0	2	0	0	1	1	0	0	0
Personnel Carrier 3	**2**	0	1	0	0	0	0	1	0	0	0
Grader	**1**	0	1	0	0	0	0	0	0	0	0
Electrical service	**1**	1	0	0	0	0	0	0	0	0	0
Haul truck battery	**54**	6	6	6	6	6	6	6	6	6	0
Charger	**27**	3	3	3	3	3	3	3	3	3	0
Top Hammer Rods	**100**	25	25	25	25	0	0	0	0	0	0

16.14 Mine Production Schedule

The mine production schedule was developed using Deswik Scheduler with a focus on prioritizing early production from higher NSR areas of the orebody.

16.14.1 Lateral Development

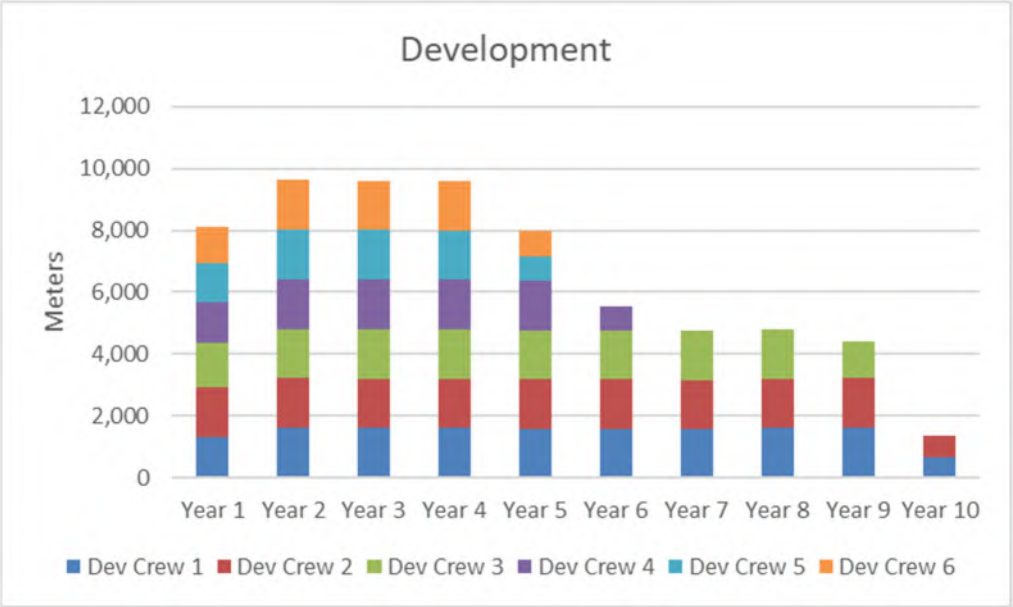
Cycle time for lateral development has been estimated for single heading and multiple heading advance and for 8 hour and 10.5-hour shift schedules. Crews will work an 8-hour schedule (three shifts per day) for the pre-production development period to maximize face time and transition to a 10.5-hour schedule (two shifts per day) once production commences. As shown in Table 16-27, development for single face headings are scheduled at 4.4 mpd and for multi-face headings they are scheduled at 6.1 mpd for the pre-production period and 3.9 mpd and 5.3 mpd, respectively, after production commences.

Table 16-27: Lateral Development Productivity

	8 Hour Shifts		10.5 Hour Shifts	
	Single Face	**Multi-Face**	**Single Face**	**Multi-Face**
hours / round	17.6	12.8	17.6	12.8
hours / day	19.5	19.5	17	17
metres / day	4.4	6.1	3.9	5.3

Six development crews will be required for the first four years of the mine life averaging 1,539 m/year each. During year 5 the number of development crews will drop to five, there will be four development crews in year 6, three in years 7 through 9, and two in year 10. A summary of the development schedule by crew is shown in Figure 16-48.

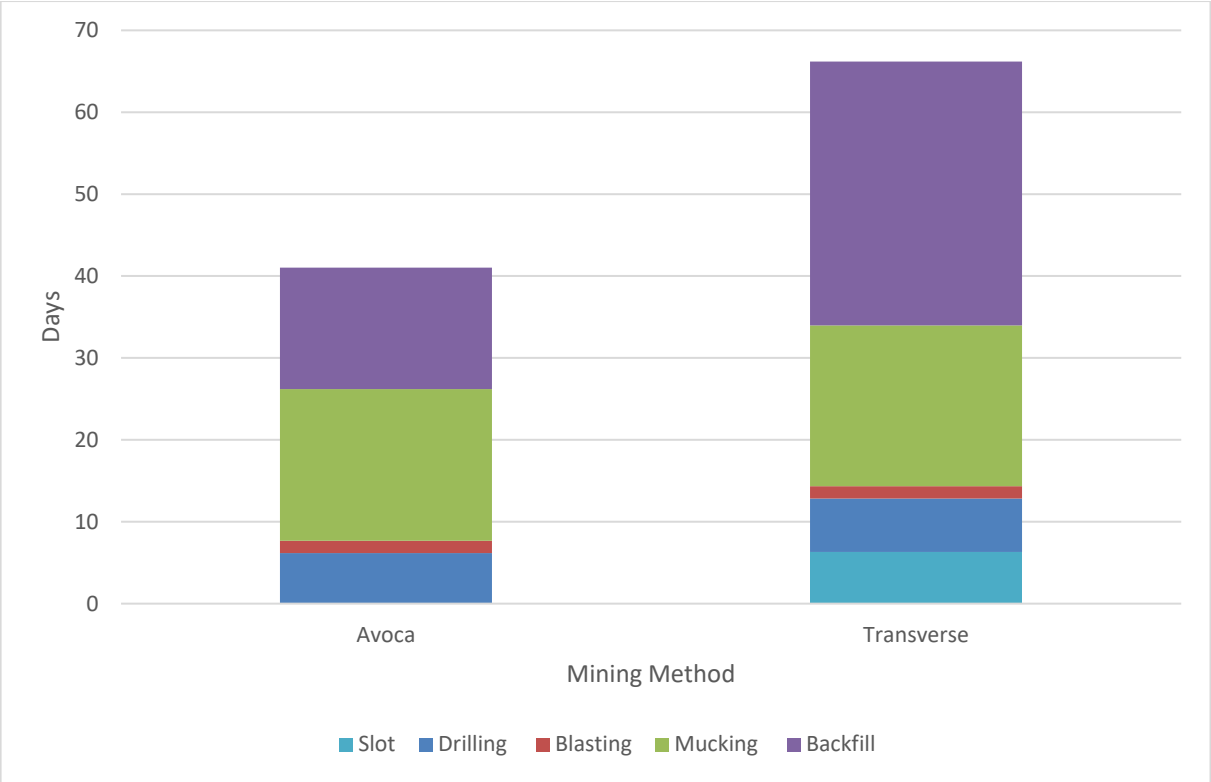
Figure 16-48: Development Requirements



16.14.2 Production

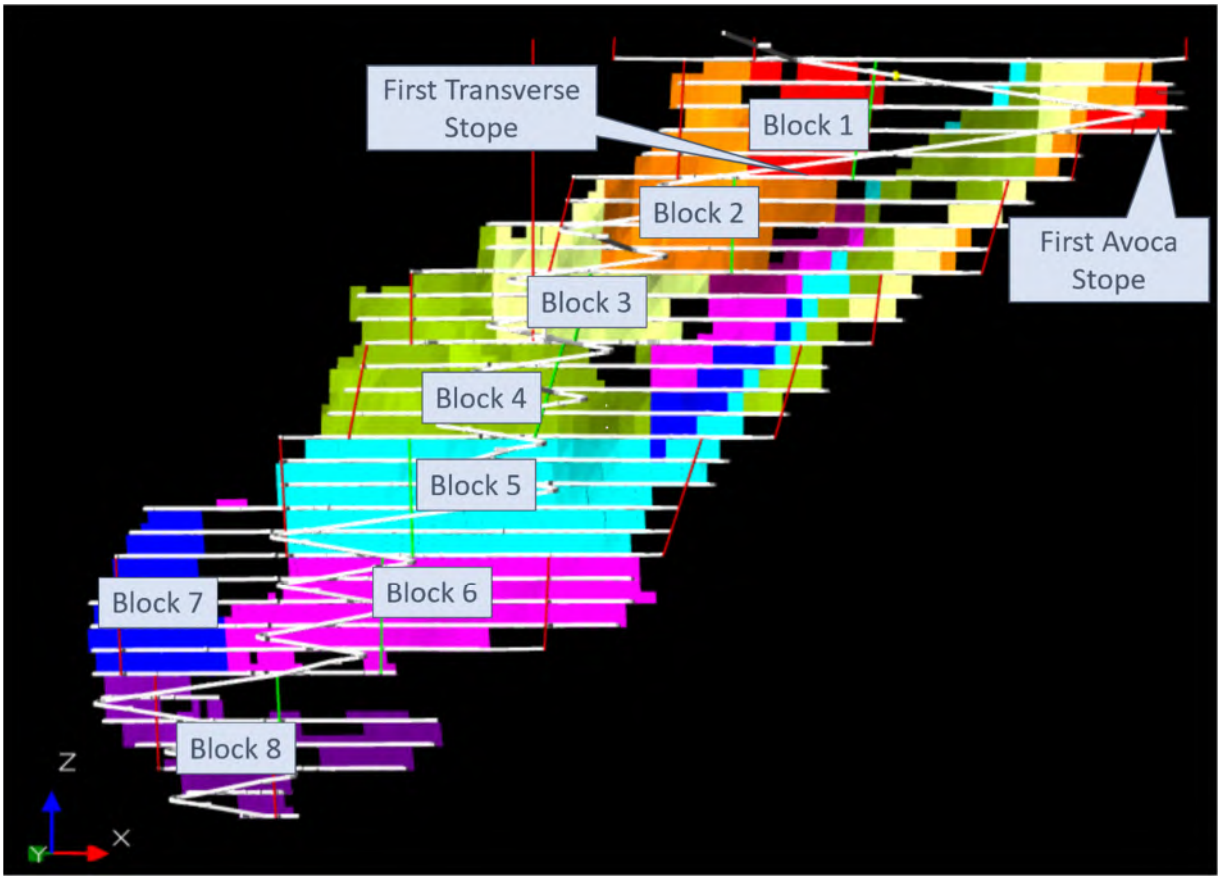
The nominal production rate of 3,600 tonnes per day was determined using Deswik scheduler and the shapes generated by the stope optimizer. Rates were calculated from first principles using equipment productivity and available work hours. Using this information, the average stope cycle time was estimated. Stope cycle time is a key factor in determining the production rate. The duration of each of the cycle steps were calculated from first principles for transverse and Avoca stoping. Many of the activities can be performed in parallel provided that there are sufficient workplaces available. In particular, activities such as slot drilling and production drilling can be done for one stope concurrently with the backfill cure time for a predecessor stope. The stope cycle times for transverse and Avoca stoping is shown in Figure 16-49. The backfill time for transverse stoping includes 28 days of cure time. The Avoca backfill is based on placing waste in the stope at a rate of 600 tonnes per day. The average stope size for transverse and Avoca stopes is 15,698t and 14,822t, respectively.

Figure 16-49: Stope Cycle Times



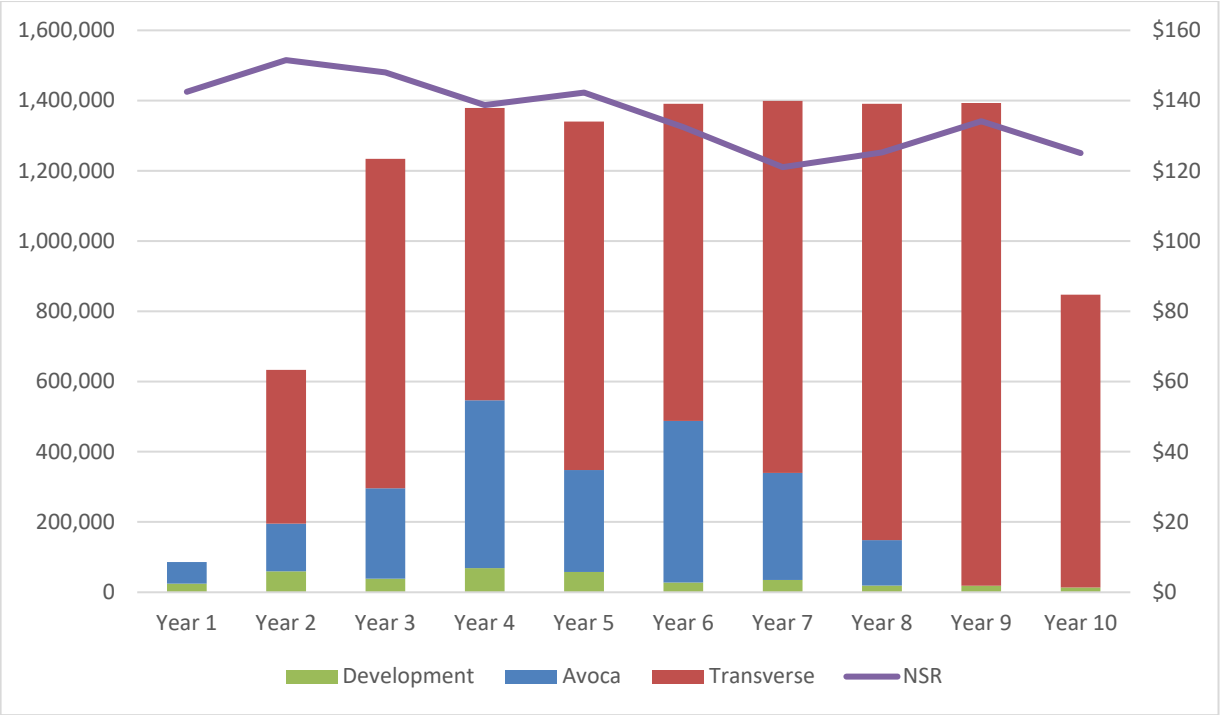
First production will occur from the Avoca zone where ore can be accessed earlier at a higher elevation. Production from the first block of transverse stopes will commence in year 2 from Block 1. Block 1 is a small high-grade block that was selected to accelerate production ramp up. Subsequent sill elevations were selected based on establishing a new sill before the previous block is depleted to maintain enough workplaces. The transverse mining zone is separated into eight blocks to provide sufficient workplaces to sustain the production rate of 3,600 tonnes per day. The transverse blocks are shown in Figure 16-50.

Figure 16-50: Transverse Stoping Blocks and Avoca Zone



The annual production rate will be 1.4 million tonnes per year, including transverse, Avoca, and ore development. Early production prioritizes higher NSR material where possible to maximize the net present value and the NSR gradually decreases over time. The production profile is shown in Figure 16-51.

Figure 16-51: Annual Production by Mining Method



16.15 Consideration of Marginal Cut-Off Grades

Marginal ore was not considered for the PFS mine plan. AGP recommends that this be incorporated into future studies.

16.16 Blasting and Explosives

Emulsion will be used for longhole blasting in transverse and Avoca stopes and ANFO will be used in lateral development headings. Blasting caps (electric and non-electric) and boosters will also be used where appropriate.

16.16.1 Lateral Development

To maximize productivity during the pre-production period, blasting for lateral development will be done at any time during the shift. Once production commences and there are crews doing a wider range of activities, it will become to difficult to execute mid-shift blasting and a central blasting system will be utilized to initiate blasting when there are no personnel underground.

The Sandvik DD-421 drill jumbo will drill a pattern of holes on the development face and ream 9 holes on either the left or right side of the face to a larger diameter. These 9 holes will serve as the initial free face for blasting and the location will alternate between left and right to avoid drilling into bootlegs.

ANFO will be loaded into the development blast holes using the MacLean AC3 ANFO loader where conditions are dry. If wet ground conditions are encountered a packaged emulsion product will be used. Non-electric (NONEL) blasting caps will be used to initiate the blast. The NONEL caps have a pre-set delay and are numbered in sequence. They correct delay must be placed in the appropriate hole to ensure that the round is successful. The caps will be connected to a blasting line which will subsequently be connected to an electric blasting cap. This electric blasting cap will initiate the blast.

16.16.2 Production

Blasting in the transverse and Avoca stopes is similar, given that these are both longhole mining methods. The production holes are 89mm in diameter and will be drilled on a 2.5 m x 2.5 m pattern using a Sandvik DL-311 top hammer drill rig. Most holes are breakthrough holes and will be plugged using packaged emulsion. The holes will then be loaded with emulsion by the MacLean CS3 emulsion loader and an IKON electronic blasting cap will be placed in a booster at 10-meter intervals in the hole. The timing for the initiation of each hole is programmed into the cap in a sequence designed by the engineering department for each blast.

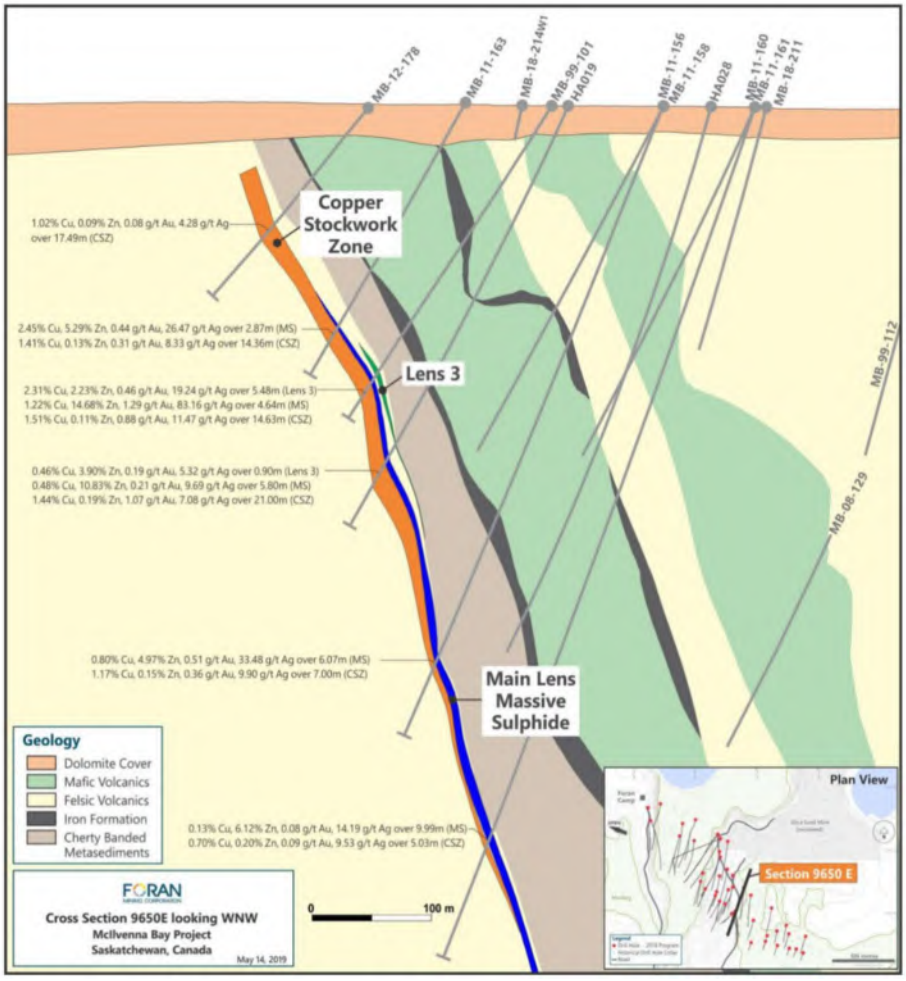
17. RECOVERY METHODS

17.1 Introduction

This section describes the parameters used to design a new mineral processing facility for the McIlvenna Bay project. The facility described herein will be designed to allow on site treatment of approximately 11.5 million tonnes of copper-zinc ore at an initial processing rate of 3,000 tpd (based on current mine plans). Ramp up to a peak processing rate of 3,800 tpd has been allowed in the design and can be achieved later in the mine life to match mine expansion through installation of additional equipment.

The McIlvenna Bay deposit consists of several mineralized lenses – each with different sulphide and gangue mineralogy plus varying metal contents and ratios. A typical section through the deposit illustrates this as shown in Figure 17-1 below.

Figure 17-1: Cross Section 9650E, Looking WNW



Very broadly, the two most prevalent mineralization types seen by the mill are Copper Stockwork Zone (CSZ), and Massive Sulphide (MS), with the former consisting mostly of copper sulphide minerals in stringer-type textures with significantly lower levels of zinc and lead, whilst the latter contains mostly zinc plus reasonable levels of copper and variable levels of lead.

The Massive Sulphide mineralization can be further broken down into discrete zones or lenses within the deposit, and these are described within the resource models and resource estimates, as shown in

Table 17-1 below.

Table 17-1: Indicated Resource Estimate (US$60/t Cut-off) – by Micon, 2019

Zone	Tonnage (Mt)	% Cu	% Zn	%Pb	g/t Au	g/t Ag
Main Lens – MS	9.25	0.90	6.43	0.40	0.52	25.97
Lens 3	1.99	0.85	3.29	0.14	0.27	14.71
Stringer Zone	0.70	1.38	0.62	0.04	0.35	13.34
Copper Stockwork Zone	10.30	1.43	0.28	0.02	0.40	9.30
Copper Stockwork Footwall Zone	0.71	1.60	1.04	0.04	0.54	11.47
Total	**22.95**	**1.17**	**3.05**	**0.19**	**0.44**	**16.68**

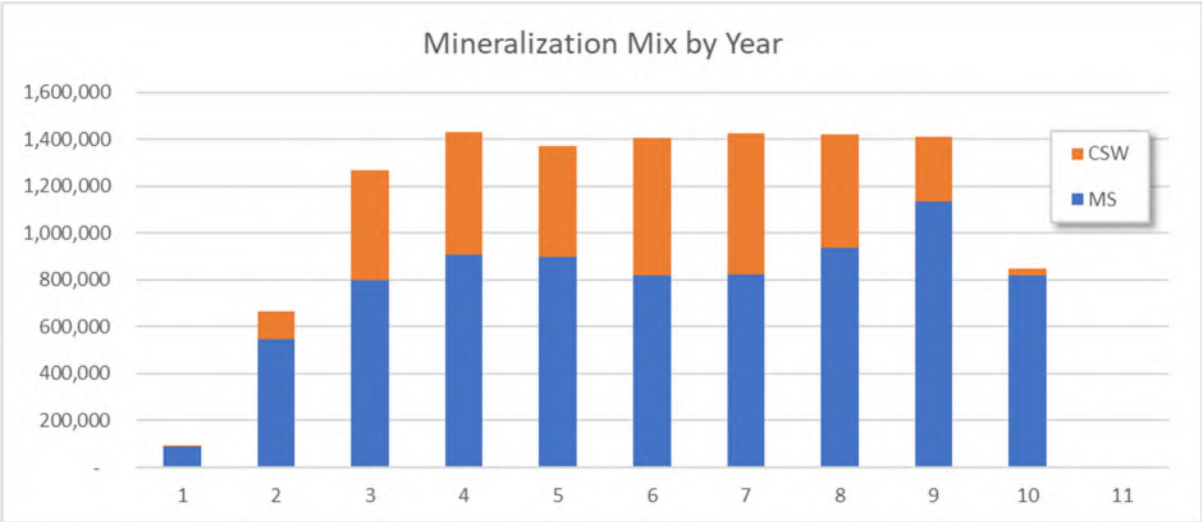
The tabled data shows that the Lens 3, Stringer Zone and Copper Stockwork Footwall Zone are relatively minor in terms of tonnage within the resource. Metallurgically, these zones behave as massive sulphide, and have been incorporated into this mineralization type.

Further to these minor components, the Main Massive Sulphide Lens was initially broken into two discrete zones, namely Zone 2 and Upper West Zone, with the latter being more copper rich. The BL0351 metallurgical program considers these two Main Massive Sulphide Lens mineralization types individually.

A key finding of the 2018/19 metallurgical testwork program was that the massive sulphide and copper stockwork mineralization types could be blended and processed without any noticeable effects on metallurgical response. This is described in Section 13. Similar primary and concentrate grind targets, plus reagent recipes are recommended for the processing of the two main different mineralization types, and thus a co-processing approach is considered to be a pragmatic method of processing – so long as effective upstream blending strategies are implemented and managed.

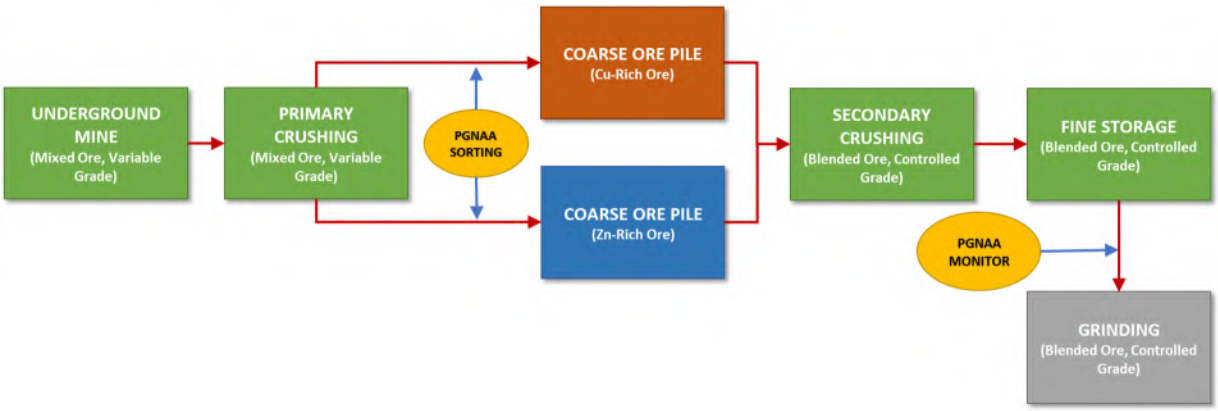
The mineral processing facility considered herein aims to treat a nominal blend of 33% CSZ and 67% MS mineralization over the life of mine, within a ±10% blend range accepted on a shorter-term basis. The planned mix of CSZ and MS in the mill feed is illustrated by year in Figure 17-2 below.

Figure 17-2: Mill Feed Mix by Year (PFS Mine Plan)



As the day to day mix of CSZ and MS tonnage could perhaps vary more significantly that shown above, a practical blending strategy is proposed - using online grade measurement equipment (PGNAA technology) to allow creation of "copper-rich" and "zinc-rich" surface stockpiles, and to measure the online grade of blended product as it passes through the crushing plant and onto the fine ore storage facility. This grade management ability, illustrated in Figure 17-3 below, will help to ensure that any unplanned mill feed grade deviations are minimized or completely eradicated.

Figure 17-3: Cu/Zn Blending Approach



The copper and zinc grades to the mill by ore type and by year are indicated in the following charts.

Figure 17-4: Copper Grade to the Mill by Year (PFS Mine Plan)

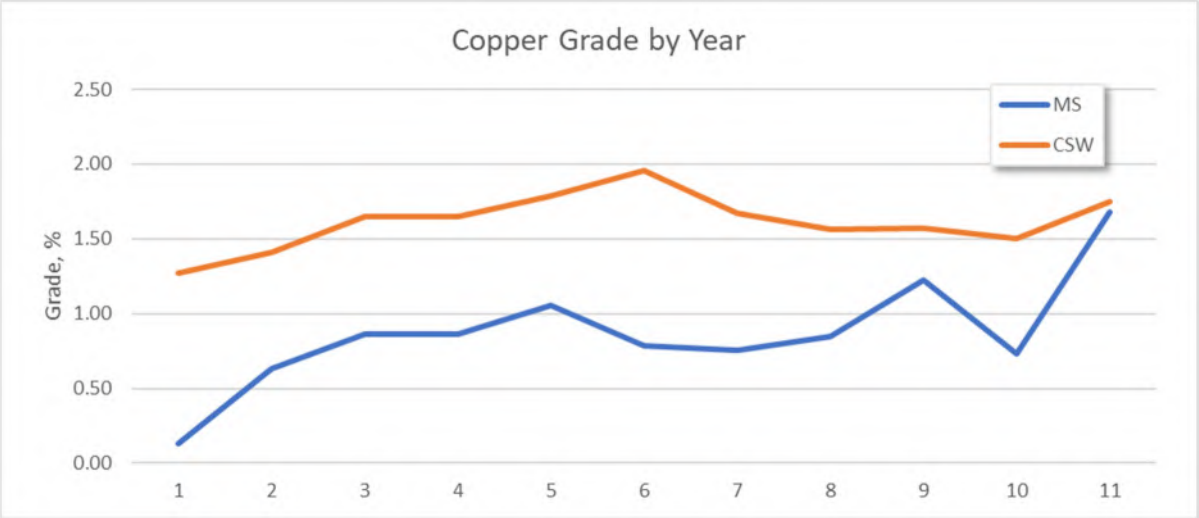
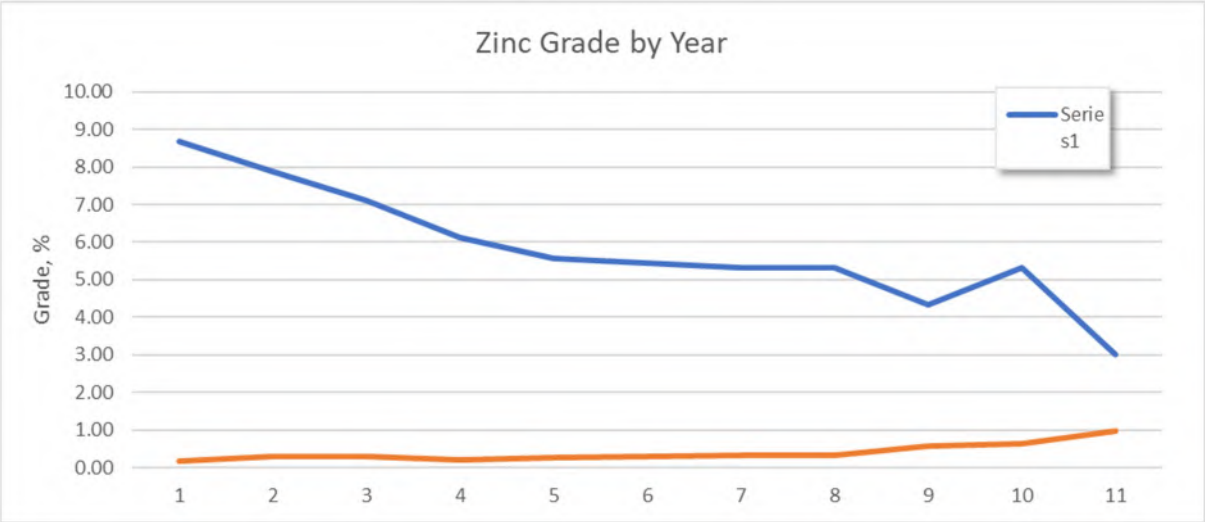


Figure 17-5: Zinc Grade to the Mill by Year (PFS Mine Plan)



The online grade measurement and control systems allow the creation of "copper-rich" and "zinc-rich" coarse ore stockpiles, which in turn allows greater control over blending and attenuation of any potential grade variability to the mill. The monitoring of Cu/Zn grade on the blended mill feed stream provides feedback regarding the effectiveness of the overall blending procedure, thereby improving efficiency.

As discussed in Section 13 of this report, the presence of higher lead concentrations within the MS lenses has been shown to be detrimental to metallurgy (specifically when the lead to copper ratio rises above unity), so the initial measurement of grade with PGNAA equipment will consider lead concentration also. High lead concentrations will be flagged automatically and addressed by metallurgists on an as-needed basis.

The process design criteria and process flowsheets described within this section and detailed in the appendices have been developed using the body of metallurgical testwork completed in recent years, including the most recent (feasibility level) work by Base Met. This work is discussed in detail within Section 13 of the report.

This proposed on-site milling facility will be designed very much with upfront capital efficiency in mind, but also with a view towards higher levels of automation and ease of operation, as a means to streamline the size of operations crews required to make the daily 100-km commute to site.

17.2 Process Summary

17.2.1 Description

The proposed processing facility has been designed as a nominal 3,700 dmtpd concentrator plant (i.e. slightly in excess of the nominal mine production), although the initial absence of standby equipment requires that the design availability of the process plant be practically limited to 85% for early mine life and thus no more 3,300 - 3,400 dmtpd effective capacity is initially expected. As the mine production rate will ramp up slowly over the initial production period, mill capacity is expected to always exceed the mine production rate.

Installation of standby equipment in year 3 is expected to improve plant availability to over 93%, at which time the daily throughput should peak at 3700 - 3800 tpd.

In the early years of production (years 1-3), ore will be hauled to surface in 50-tonne trucks and dumped into a surface crushing facility. As the mine development continues below 0m level, a new underground crushing station will be constructed to feed -150mm ore onto a new vertical conveying system (described more completely in Section 16). The surface crusher station will continue to operate as long as ore is being hauled to surface with trucks, and thereafter will remain available only as a standby/emergency crushing facility. The vertical conveyor will be tied into the surface crushing facility using a transfer conveyor.

Irrespective of crusher station location, run of mine ore will be crushed to a nominal 100% passing 150mm, (80% passing 80mm) size. Conveyors will transfer coarse crushed material to one of two surface stockpiles in preparation for secondary crushing. The secondary crushing circuit will consist of a cone crusher in closed circuit with a screen and is required to reduce the ore size in preparation for ball milling (-12mm).

The selected grinding circuit consists of a two-stage ball mill combination which is well suited to handling the variable hardness expected from mixtures of the high-silica copper stockwork ore and the softer massive sulphide ore. The grinding circuit is designed to reduce the particle size of flotation feed material to a nominal 80% passing 75 µm – as indicated by recent flotation testwork.

Cyclone overflow slurry from the secondary mill will be directed to the flotation section for sequential copper/lead and zinc concentrate recovery. The copper and zinc circuits will be similar in nature, with each circuit producing rougher concentrates prior to regrinding and multi-stage cleaning. Two saleable flotation concentrates – copper and zinc – will be produced separately. Tank cells are considered most appropriate for the rougher/scavenger duty in this plant, with the conventional “trough” type cells anticipated for cleaner duties. The copper and zinc rougher/scavenger concentrates will both be

subjected to regrinding using vertical, inert media (HIG) mills, with copper concentrates requiring a P_{80} of approximately 20-25 µm and zinc concentrates requiring a P_{80} of approximately 25-30 µm.

Final cleaner concentrate from the copper and zinc circuits will be pumped to the copper and zinc concentrate thickeners to recover water from the slurry and simultaneously produce a 50-60% solids (w/w) underflow stream suitable for pressure filtration. A single vertical pressure filter will be used to further dewater both copper and zinc thickened concentrates in batches to provide two stockpiles of product filter cake suitable for transportation to toll smelters.

Zinc scavenger tailing slurry will pass through a single stage desulphurization flotation cell. This cell will be similar in size/design to the zinc flotation cells, but with additional reagents added to non-selectively recover residual sulphide minerals to a sulphide concentrate. The sulphide concentrate produced will be dewatered separately and directed to the paste backfill circuit, for incorporation into the backfill mixture and safe storage underground.

Desulphurization cell tailing slurry will be pumped to a high compression tailings thickener for dewatering, storage and transfer across to the paste plant. Water will be recovered from the tailing slurry and returned to the process water tank for re-use. Thickened underflow slurry (approximately 65 to 70% solids) will then be pumped to storage tanks ahead of the paste plant.

The paste plant will treat 100% of flotation tailings and will make a filtered cake suitable for stacking on a nearby cake storage facility, or for mixing with water and Portland cement to make a paste backfill material for use underground.

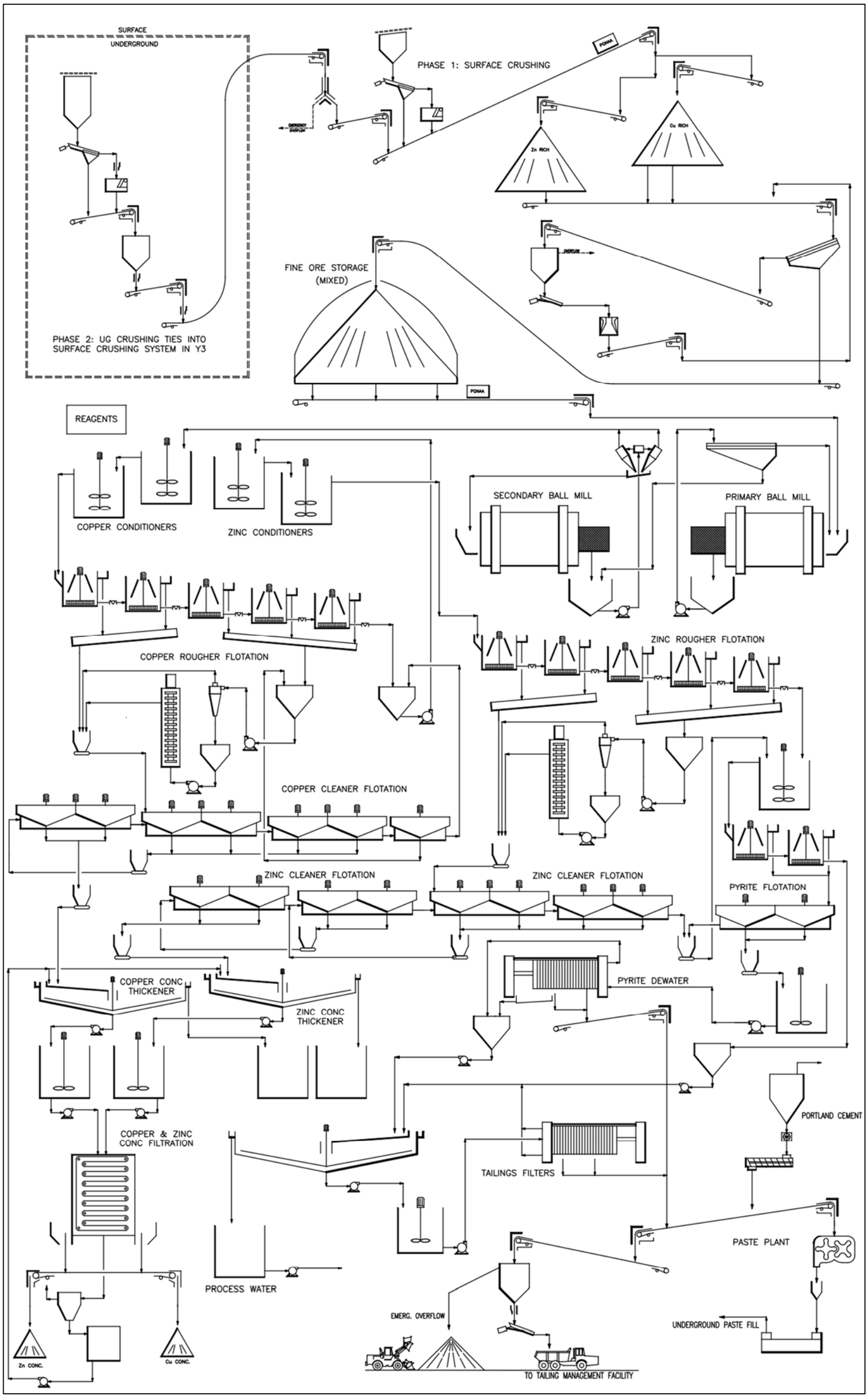
The plant will include various water reticulation and air services, in addition to HVAC and dust extraction modules. Most plant areas will be contained within one large mill building. The tailing thickener will be located outdoors but will be partially clad to ensure proper operation in the harsh winter temperatures.

A reagent storage area will be located alongside the process plant building. Reagents will be stored and transferred to the reagent day tank and dosing area within the main building. Reagents will include frothers, collectors and promotors, sulphide and talc depressants and flocculants. Accurate dosing of reagents into the process will be handled by plc-controlled peristaltic metering pumps.

17.2.2 Summary Flowsheet

The planned process plant is described by a series of process flow diagrams, given in Appendix A. A summary process flowsheet for the full plant is given below in Figure 17-6.

Figure 17-6: Process Plant Summary Flowsheet



17.3 Detailed Process Description

The process plant consists of several numbered areas, from Area 100 (crushing) to Area 900 (Services). These are described in detail below.

For all following sections, refer to Appendix A for detailed Process Flow Diagrams.

17.3.1 Primary Crushing – Area 100

In the initial years of mine life, ROM ore will be hauled to surface using 50-tonne capacity underground haul trucks and delivered from the portal either to the ROM stockpile or to the inload bin of the surface crushing facility. In later years, as ramp development passes the 0m level, the truck haulage system will be replaced by an underground crushing station and vertical conveyor system. At this point, the vertical conveyor will be tied into the surface crushing system by a transfer conveyor.

Initial Operation (Y1-4)

Trucks will deliver run of mine ore to the primary crusher inload bin (100-BAA-04), with a nominal top size of 500mm (~20”) and an average rate of 240 dmtph. If the inload bin is too full to safely accept a full load, then the truck will be directed by lights to tip on the nearby ROM stockpile.

The stockpile and inload bin arrangement is located in close proximity to the portal (Figure 17-7) and will be designed to minimize truck turnaround, thereupon maximizing equipment utilization.

The primary crushing system consists of an inload bin (100-BAA-04), a vibrating grizzly feeder (100-FDA-08), a 200Hp jaw crusher (100-CJA-14) and various chutes and conveyors to control the movement of material. A self-cleaning magnet will remove tramp metal prior to entering the secondary crushing section.

Crushed rock is moved to a transfer point by a sacrificial conveyor (100-FCV-18). This conveyor is equipped with the PGNAA unit to determine copper and zinc concentrations in the crushed rock. The composition of the rock determines the position of an actuated bifurcation chute (100-ZAA-26) and this in turn diverts the material to the appropriate transfer conveyor – 150-FCV-06 (to the copper-rich stockpile) or 150-FCV-10 (to the zinc-rich stockpile).

Later Operation (Y4-10)

Around Y4 of the mine plan, an underground crushing station and vertical conveying system will be commissioned to transport ore from the 0m level to surface. Ore will be hauled from the lower levels of the mine to the crushing station tip bin using 50t trucks, and as ore production moves to levels below the crusher station, then truck haulage to surface will be completely phased out.

The underground crushing station will consist of a grizzly feeder (110-FDA-08) and jaw crusher (110-CJA-14) and will deposit crushed ore (100% -150mm) onto a transfer conveyor (110-FCV-18) and into a surge bin (110-BBA-28). From the surge bin, crushed ore will be deposited onto the tail end of the vertical conveyor at a rate of 240 tph. The vertical conveyor will discharge onto a transfer conveyor (110-FCV-48) and into the surface crushing system as described above.

Figure 17-7: Process Plant and Stockpile Layout

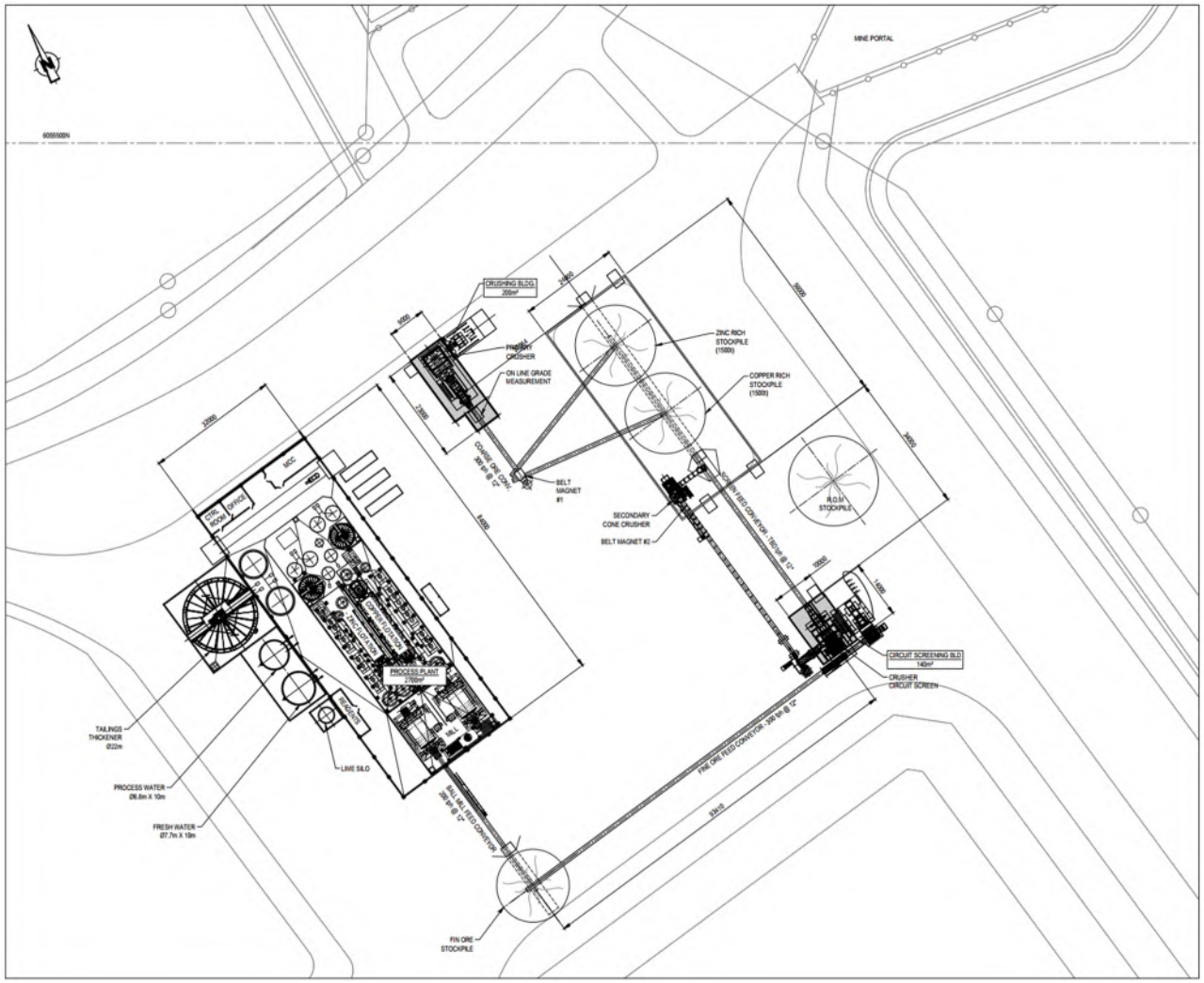


Figure 17-8: Typical PGNAA Installation

The main items of equipment in Area 100 (Primary Crushing, Surface) are described in Table 17-2 below.

Table 17-2: Area 100 Equipment (Surface Crushing)

EQ #	Description	Notes	kW
100-SSD-02	Static Grizzly	550mm x 500mm	-
100-BBA-04	Inload/Surge Bin	Steel Bin, 50t capacity; Lined	-
100-ZAA-06	BBA-04 discharge Chute	Lined AR400	-
100-FDA-08	Vibrating Grizzly Feeder	Model: UGS 1000 x 3350	18.75
100-CJA-14	Primary Crusher	TST1100 Single Toggle Jaw Crusher	150
100-FCV-18	Sacrificial Conveyor	24" x 39m	15
100-MAA-20	Self Cleaning Magnet	Trio CR36	7
100-AAF-52	Dust Extraction System	Centrifugal Fan and baghouse	11
100-XXX-04	Online grade measurement	determines Cu Rich vs Zn rich	3

The underground crushing station and vertical conveyor will be installed and commissioned in Year 3 of the production schedule. Equipment in this area is given in Table 17-3 below.

Table 17-3: Area 110 Equipment (U/G Crushing)

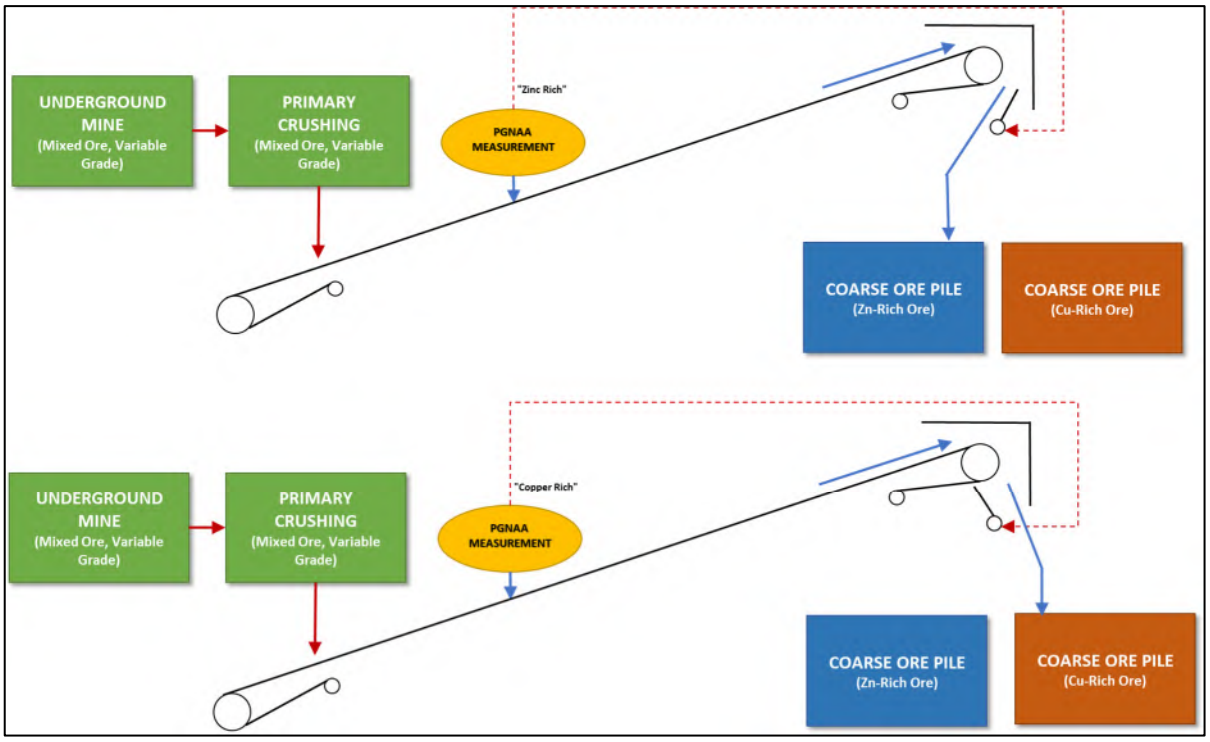
EQ #	Description	Notes	kW
110-SSD-02	Static Grizzly	500mm square	-
110-BBA-04	UG Surge Bin	500m3 capacity	-
110-FDA-08	Vibrating Grizzly Feeder	Model: UGS 1000 x 3350	7.5
110-CJA-14	Primary UG Crusher	TST1100 Single Toggle Jaw Crusher	110
110-ZAA-16	CJA-12 Discharge Chute	Lined AR400	-
110-FCV-18	Sacrificial Conveyor	30" X 50'	7.5
110-MAA-20	Self Cleaning Magnet		5.5
110-BBA-28	UG Surge Bin	200m3 capacity	-
110-FCV-32	Transfer Belt Feeder	240 tph	7.5
110-FCV-40	Vertical Conveyor	400' lift, pocket type	200
110-FCV-46	Emergency Stockpile Conveyor	includes small head chute	15
110-FCV-48	Transfer Conveyor	Transfers to existing crushing system	30
110-AAF-54	Dust Extraction System	Centrifugal Fan and baghouse	11

17.3.2 Secondary Crushing – Area 150

Crushed product (100% -150mm) will be conveyed from the primary crushing area (surface or underground) onto the sacrificial conveyor (100-FCV-18). From this point it will be assessed for composition by the PGNAA unit and diverted to one of two coarse ore stockpile feed conveyors (150-FCV-06 or 10) to feed onto a pair of 1,500 tonne coarse ore stockpiles. One stockpile will contain mainly zinc-rich mineralization and the second stockpile will contain mainly copper-rich mineralization. The split of copper and zinc need not be perfect, so long as the majority of copper and zinc is correctly placed. In addition to managing the blend of copper and zinc into the plant, the copper: lead ratio in mill feed will be carefully monitored as this has been shown to be detrimental to metallurgy below certain limits.

The PGNAA blending system is illustrated in Figure 17-9 below. The analyzer will be situated directly above the sacrificial conveyor belt and will penetrate the entire raw material cross-section, providing minute-by-minute, uniform measurement of the entire material stream, not just a sample.

Figure 17-9: Loading Zinc Ore vs Copper Ore



The two coarse ore stockpiles will be drawn down in a controlled fashion using vibrating pan feeders (150-FFB-20, 24, 30, 34) to feed a blended ore onto the secondary crusher screen feed conveyor (150-FCV-38). Estimates of coarse ore stockpile grade together with relative stockpile tonnages will be used to roughly meter the zinc and copper rich ore into the secondary crusher feed stream. In this manner, any significant instantaneous grade fluctuations into the process plant can be attenuated prior to milling and flotation.

The crusher screen (150-SVB-42) will be a double deck unit, with oversize material from both decks (+15mm) reporting to a 400 Hp cone crusher (150-CCB-68) via conveyor belt (150-FCV-56), small surge bin (150-BB-60) and vibrating feeder (150-FDB-64). The secondary cone crusher will crush the oversize material to roughly -12mm and will return this stream to the screen for re-classification. The circulating load of screen oversize is estimated to be around 80% on average. Screen undersize material (-15mm), suitable for grinding, will be conveyed (150-FCV-78) up to a covered fine ore stockpile with 2,000t live capacity.

The fine ore stockpile will be discharged in a controlled fashion using variable speed belt feeders (150-FBA-86, 92, 98). The variable speed feeders will discharge onto the primary mill feed conveyor belt (200-FCV-06) which will be equipped a second PGNAA unit (monitoring mill feed grades to allow reagent optimization) plus an accounting weigh-scale and belt speed sensor to provide tight control over mill feed rate.

The main items of equipment in Area 150 are described in Table 17-4 below.

Table 17-4: Area 150 Equipment

EQ #	Description	Notes	kW
150-FCV-06	Coarse Ore Conveyor - Cu	24" x 37m, 10m lift	15
150-FCV-10	Coarse Ore Conveyor - Zn	24" x 37m, 10m lift	15
150-FBA-20	Zn Vibrating Feeder #1	75 tph capacity	3
150-FBA-24	Zn Vibrating Feeder #2	75 tph capacity	3
150-FBA-30	Cu Vibrating Feeder #1	75 tph capacity	3
150-FBA-34	Cu Vibrating Feeder #2	75 tph capacity	3
150-FCV-38	Screen Feed Conveyor	24”x100m Lg, lift 15m @ 300MTPH. Material 125mm minus	30
150-SVB-42	Crusher Circuit Screen	"HONERT” Banana Screen 10’-0” Wide x 24’-0” Long Double Deck	30
150-ECB-50	Stationary Magnet		-
150-EEB-54	Metal Detector		0.5
150-FCV-56	Secondary Crusher Feed Conveyor	24”x36m Lg, lift 10m @ 300MTPH. Material 35mm minus.	15
150-BAA-60	Secondary Crusher Feed Bin	60m3 capacity, c/w overflow	-
150-FBA-64	Cone Crusher Feeder	300 tph capacity	5.5
150-CCA-68	Secondary Cone Crusher	Raptor 450	300
150-ZAA-70	CCA-22 Discharge Chute	Lined, AR400	-
150-FCV-72	Secondary Crusher Discharge Conveyor	24”x10m Lg, lift 2.5m @ 300MTPH. Material 25mm minus.	5.625
150-FCV-78	Stockpile Feed Conveyor	24”x98m Lg, lift 22m @ 300MTPH. Material 15mm minus	30
150-ZAA-80	FCV-78 Head Chute	Include inspection door	-
150-FBA-86	Belt Feeder #1	exact size TBA. 120 tph capacity	2.2
150-FBA-92	Belt Feeder #2	exact size TBA. 120 tph capacity	2.2
150-FBA-98	Belt Feeder #3	exact size TBA. 120 tph capacity	2.2
150-AAF-104	CCA-68 Dust Extraction System	Centrifugal Fan and baghouse	11
150-PFB-106	Crusher Dust Slurry Pump	VT40 - Tank Pump	5.5

17.3.3 Grinding – Area 200

A blend of crushed ore sized at 80% passing 10mm will be conveyed from the fine ore stockpile (Area 150) to the mill building and into the primary ball mill feed trunnion via the mill feed conveyor (200-FCV-06) and various chutework. The primary ball mill (200-MBA-12) will be a 14’ diameter x 22’ EGL unit equipped with a 2,200 kW variable speed drive and a charge of 70mm balls equivalent to ~33% of the chamber volume. The mill will be equipped with the necessary lubrication systems and barring gear to assist maintenance. It will be equipped with polymet (rubber/steel combo) liners, plus an internal discharge grate and pulp lifters to assist the removal of slurry and finer particles from the milling chamber into the mill discharge tank. Within the mill, rock will be diluted with process water and ground to a coarse slurry (80% -0.8mm).

Slurry will exit the primary mill discharge trunnion after passing through the mill into the mill discharge tank (200-TBA-14) from where it will be further diluted with process water and pumped (200-PCB-16) up to the mill circuit sizing screen (200-SVA-22) for initial classification. Screen oversize material (+0.55mm) will be returned back to the primary mill inlet, whilst material reporting to the screen undersize stream (-0.55mm) will gravitate down to the secondary ball mill discharge tank for further classification and grinding. The secondary ball mill (200-MBB-52) will be an overflow mill of the same dimensions and installed power as the primary unit. It will be designed to reduce the mill feed size down from 80% -325µm to 80% -75 µm and will be operated in closed circuit with a set of hydrocyclones.

From the secondary mill discharge tank (200-TBA-54), slurry will be diluted and pumped up to the hydrocyclone set (200-YAA-60) for classification. Cyclone overflow slurry will gravitate via launders and pipes over to the copper flotation circuit, whilst the coarser cyclone underflow slurry will gravitate via piping down to the mill inlet hopper as secondary mill feed. The design circulating load in the cyclone underflow is expected to be between 250 and 300% of primary mill feed tonnage.

Grinding balls will be added to the primary and secondary mills via loading chutework and ball-carrying skips. Approximately 1 or 2 drums of balls will be added to each mill each day to top up the charge. 70mm balls will be added to the primary mill, and 35mm balls to the secondary mill.

The main items of equipment in Area 200 are described in Table 17-5 below.

Table 17-5: Area 200 Equipment

EQ #	Description	Notes	kW
200-XXX-04	Ball Mill Feed Online Analysis	determines Cu Rich vs Zn rich	4
200-FCV-06	Ball Mill Feed Conveyor	24"x59m Lg, lift 12m @ 200MTPH. Material 15mm minus	18.75
200-MBA-12	Primary Ball Mill	14' diameter x 22' EGL	2200
200-TBA-14	Primary Mill Discharge Tank	10m3, Square tank, 45 deg. inclined bottom	-
200-PCB-16	Primary Mill Discharge Pump	millMAXUMD 8x6-25(M) MMB W/F-FF on DOH Assy.	75
200-ZAB-20	Screen Feed Box		-
200-SVA-22	Primary Mill Screen	8' x 20' horizontal deck vibrating screen	30
200-PCD-34	Primary Mill Spillage	All metal submersible pump	15
200-XLC-36	Primary Mill Ball loading winch		3
200-MBA-52	Secondary Ball Mill	14' diameter x 22' EGL	2200
200-TBA-54	Sec Mill Discharge Tank	20m3, Square tank, inclined bottom	-
200-PCB-56	Sec Mill Discharge Pump	slurryMAXXD 10x8-27 MMC W/F-FF on DOH Assy.	185
200-YAA-60	Sec Mill Cyclone Pack	3-Place gMax20-H Cyclone Manifold	-
200-ZAD-62	Cyclone Underflow Collection Box	Lined	-
200-ZAD-64	Cyclone Overflow Collection Box	Lined	-
200-PCD-66	Secondary mill Spillage Pump	millMAX 3x2-9 MMAA W/F-FF on DOH Assy.	7.5
200-XXX-72	Secondary Mill Gear Lube System	Air operated. Farval or equivalent	-

17.3.4 Copper Flotation – Area 300

The copper flotation circuit will consist of separate rougher and cleaner flotation equipment, with a concentrate regrinding mill installed to process the rougher concentrate prior to cleaner flotation. Concentrate from the copper cleaners will be pumped to the concentrate dewatering section prior to shipment as filter cake to toll smelters. Tailing slurry from the copper roughers will be pumped to the zinc conditioners for additional metal recovery.

Slurry will gravitate from the ball mill circuit (cyclone overflow) via a sample station into to a pair of agitated (300-XSA-10&14) and baffled conditioner tanks (300-TBA-08&12). The first tank will be used primarily as a pre-aeration vessel, and the second tank will provide time for proper reagent conditioning. From the second conditioner tank, slurry will overflow to the first cell in the copper rougher bank. Pulp short-circuiting will be prevented in these tanks using upcomer tubes.

The bank of copper rougher cells consists of five 70m^3 tank cells (300-XCB-16, 18, 20, 22 and 24). Flotation air to each cell will be supplied by flotation blowers (Area 900) via a low-pressure manifold and will be flow controlled by modulating valves and vent-captor type flow meters. Slurry level will be maintained in each cell by modulating dart valves. Froth surface area will be limited using static froth crowders.

High grade rougher concentrate will be collected from the first two cells and pumped (300-ZAC-26, PFB-50) directly to the first stage cleaner, thereby bypassing the regrind circuit. A lower grade rougher concentrate will be collected from the remaining three cells of the bank and laundered (300-ZAC-28) into the concentrate regrind circuit. The HIG-Mill based copper regrind circuit consists of cyclone feed

tank and pump (300-TAA-52, 300-PCB-54), a de-sliming cyclone (300-YAA-56), a regrind mill feed tank and pump (300-TAA-58, 300-PCB-60) and the regrind mill itself (MLS-62). The copper regrind circuit will reduce the 80% passing size of LG rougher concentrate to approximately 20-25 µm.

Tailing slurry from the final cell in the bank of copper roughers will be pumped (PCB-36/38) via a sampling station (ZAC-30, XDB-32) to the zinc flotation conditioner tanks.

Copper rougher area spillage will be collected in a dedicated bund area and pumped (PCD-40) to the primary conditioner tank.

Rougher concentrate bypass slurry, together with re-ground lower grade concentrate slurry will be received in the feed box on the first bank of cleaner flotation cells. First stage cleaner flotation Will be provided by a single bank of cells (XCB-64, 66, 68, 70, 72). Cleaner tailing slurry will be further processed with a cleaner scavenger flotation cell (XCB-74) prior to sampling (ZAC-76, XDB-78) and gravitating to the copper tailing tank.

Flotation air for cleaner cells will be supplied from a low-pressure manifold and will be flow controlled by modulating valves and vent-captor type flow meters. Pulp level will be maintained in each bank of cells by twin modulating dart valves. First cleaner concentrate will be collected in integral launders and pumped (PFB-86) to the second stage cleaners for further upgrading. Second cleaner concentrate will be collected from a bank of three cells (XDB-80, 82, 84) and pumped (PFB-88) as a final concentrate to the copper concentrate dewatering section. Tailings slurry from the second cleaner cells will gravitate via piping to the head of the bank of first copper cleaner cells. Copper cleaner area spillage will be collected separately and pumped (PPS-90) to the copper regrind mill circuit.

The main items of equipment in Area 300 are described in Table 17-6 below.

Table 17-6: Area 300 Equipment

EQ #	Description	Notes	kW
300-XDB-06	Cyclone O/F Primary Sampler	1000mm width	1.1
300-TBA-08	Conditioner Tank #1	30m3 @ 100% Level; 3m dia x 4.05m H	-
300-XSA-10	TAA-08 Agitator	AMX-750	7.5
300-TBA-12	Conditioner Tank #2	30m3 @ 100% Level; 3m dia x 4.05m H	-
300-XSA-14	TAA-08 Agitator	AMX-750	7.5
300-XCB-16-24	Cu Rougher Cell 1-5	nextSTEP™ -70RT INT	75 x 5
300-XDB-32	Cu Rougher Tails Sampler	1000mm width	3
300-TAA-34	Cu Rougher Tails Tank	7.5m3, 2.2m diameter	-
300-PCB-36	Cu Rougher Tails Pump	slurryMAXR 8x6-22 MMB W/F-FF	45
300-PCD-40	Cu Rougher Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
300-PFB-50	Cu Rougher HG Conc Pump	VF-150 Rubber, Open Impeller	11
300-TAA-52	Cu Rougher LG Conc Tank	3m3, 1.8m diameter	-
300-PCB-54	Cu Rougher LG Conc Pump	slurryMAXR 4x3-10-Rubber MMAA W/F-FF	15
300-YAA-56	Cu Rougher Conc Cyclones	2-Place gMAX10-3140 Cyclone Manifold	-
300-TAA-58	Cu Regrind Mill Feed Tank	3m3, 1.8m diameter	-
300-PCB-60	Cu Regrind Mill Feed Pump	slurryMAXXD 2x2-8 MMAA W/F-FF	5.5
300-MLS-62	Cu Regrind Mill	HIG 275/2000F	275
300-XCB-64-72	Cu 1st Cleaner Cell 1-5	nextSTEP™ 150UT	7.5 x 5
300-XCB-74	Cu Cleaner Scav Cell 1	nextSTEP™ 150UT	7.5
300-XDA-78	Cu Cleaner Scav Tail Sampler	750mm width	1.1
300-XCB-80-84	2nd Cleaner Cell 1-3	nextSTEP™ 150UT	7.5 x 3
300-PFB-86	Cu 1st Cleaner Conc Pump	VT-100 Rubber, Open Impeller	7.5
300-PFB-88	Cu 2nd Cleaner Conc Pump	VT-100 Rubber, Open Impeller	7.5
300-PCD-90	Cleaner Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5

17.3.5 Zinc Flotation – Area 330

The zinc flotation circuit will be similar in design to the copper circuit and consists of rougher and cleaner flotation circuits with a rougher concentrate regrind prior to cleaner flotation. The zinc circuit will be fed with combined copper flotation tailing slurry. Concentrate slurry from the zinc cleaner circuit will be pumped to the zinc concentrate dewatering section prior to shipment off site as product. Tailing slurry from the zinc roughers will be pumped to the tailing dewatering area for disposal (either deposited on the tailing management facility or used as part of the paste backfill mass used underground).

Slurry will be pumped from the copper circuit via a sample station into to a pair of agitated (XSA-08, XSA-12) conditioner tanks (TBA-06, TBA-10). The first conditioner will be primarily a pre-aeration vessel, and the second tank allows reagent conditioning. From the second conditioner, slurry will gravitate to the first cell in the zinc rougher bank.

The bank of copper rougher cells consists of five 20m^3 tank cells (XCB-16, 18, 20, 22 and 24). Flotation air to each cell will be supplied by flotation blowers via a low-pressure manifold and will be flow controlled by modulating valves and vent-captor type flow meters. Pulp level will be maintained in each cell by modulating dart valves.

Zinc rougher concentrate will be collected from the first two cells and pumped (ZAC-26, PFB-50) directly to the first stage cleaner, thereby bypassing the regrind circuit. A lower grade rougher concentrate will be collected from the remaining three cells of the bank and laundered (ZAC-28) into the zinc concentrate regrind circuit. The HIG-Mill based zinc regrind circuit consists of cyclone feed tank and pump (TAA-52, PCB-54), a de-sliming cyclone (YAA-56), a regrind mill feed tank and pump (TAA-58, PCB-60) and the regrind mill itself (MLS-62). The zinc regrind circuit reduces the 80% passing size to approximately 25-30 µm.

Tailing slurry from the final cell in the bank of zinc roughers will be pumped (PCB-36/38) via a sampling station (ZAC-30, XDB-32) to the tailing thickener for dewatering and paste production.

Zinc rougher area spillage will be collected in a dedicated bund area and pumped (PCD-40) to the primary zinc conditioner tank.

Zinc rougher concentrate bypass slurry, together with ground lower grade concentrate slurry will be pumped to the feed box on the first bank of cleaner flotation cells. First stage cleaner flotation will be provided by a single bank of cells (XCB-64, 66, 68, 70, 72 and 74) with integrated concentrate launders. Flotation air will be supplied from a low-pressure manifold and will be flow controlled by modulating valves and vent-captor type flow meters. Pulp level will be maintained in each bank of cells by twin modulating dart valves. First cleaner concentrate will be collected and pumped (PFB-88) to the second stage cleaners for further upgrading. Second cleaner concentrate will be collected from a bank of two cells (XCB-80 and 82) and pumped (PFB-90) to the third zinc cleaner circuit. Final stage zinc cleaning will be achieved via a pair of flotation cells (XCB-84 and 86) and zinc final concentrate will be collected in the integral concentrate launder prior to pumping (PFB-92) to the zinc concentrate dewatering section. Second and third cleaner tails slurry will gravitate via piping to the head of the bank of first and second zinc cleaner cells, respectively. Zinc cleaner area spillage will be collected separately and pumped (PCD-94) to the zinc regrind mill circuit.

The main items of equipment in Area 330 are described in Table 17-7 below.

Table 17-7: Area 330 Equipment

EQ #	Description	Notes	kW
330-XDB-04	Cyclone O/F Primary Sampler	1000mm width	3
330-TBA-06	Zn Circuit Conditioner Tank #1	30m3 @ 100% Level; 3m dia x 4.05m H	-
330-XSA-08	TBA-06 Agitator	AMX-750	7.5
330-TBA-10	Zn Circuit Conditioner Tank #2	30m3 @ 100% Level; 3m dia x 4.05m H	-
330-XSA-12	TBA-10 Agitator	AMX-750	7.5
330-XCB-16-24	Zn Rougher Cell 1-5	nextSTEP™-20RT INT	30 x 5
330-XDB-32	Zn Rougher Tails Sampler	1000mm width	3
330-PCD-40	Zn Rougher Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
330-PFB-50	Zn Rougher HG Concentrate Pump	VT-150 Rubber, Open Impeller	11
330-TAA-52	Zn Rougher LG Concentrate Tank	3m3, 1.8m diameter	-
330-PCB-54	Zn Rougher LG Conc Pump	slurryMAXR 4x3-10-Rubber MMAA W/F-FF	15
330-YAA-56	Zn Rougher Concentrate Cyclones	2 place radial manifold, gMAX10-3139	-
330-TAA-58	Zn Regrind Mill Feed Tank	3m3, 1.8m diameter	-
330-PCB-60	Zn Regrind Mill Feed Pump	slurryMAXR 4x3-10-Rubber MMAA W/F-FF	5.5
330-MLS-62	Zn Regrind Mill	HIG 350/2000F	350
330-XCB-64-74	Zn 1st Cleaner Cell 1-6	nextSTEP™ 150UT	7.5 x 6
330-XDB-78	Zn Cleaner Scav Tail Sampler	900mm width	1.1
330-XCB-80,82	2nd Cleaner Cell 1-2	nextSTEP™ 150UT	7.5 x 2
330-XCB-84,86	3rd Cleaner Cell 1-2	nextSTEP™ 150UT	7.5 x 2
330-PFB-88	Zn 1st Cleaner Conc Pump	VT-100 Rubber, Open Impeller	7.5
330-PFB-90	Zn 2nd Cleaner Conc Pump	VT-100 Rubber, Open Impeller	7.5
330-PFB-92	Zn Final Conc Pump	VT-80 Rubber, Open Impeller	7.5
330-PCD-94	Cleaner Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5

17.3.6 Tailing Desulphurization - Area 340

A pyrite flotation step has been added to the flotation process to ensure that the dry stack tailing facility contains material with no more than 0.5% sulphur on average. The sulphide gangue minerals that remain in the tailing slurry after copper and zinc flotation are easily floated into a concentrate that can be blended in with tailings cake and Portland cement prior to pumping underground for use as paste backfill in completed stopes.

Zinc scavenger and cleaner scavenger tailing slurries gravitate to the desulphurization (or Pyrite flotation) conditioner tank (340-TBA-02). In this conditioner, a non-selective bulk sulphide collector reagent (Potassium Amyl Xanthate, or PAX) will be added to encourage the flotation of the majority remaining sulphide minerals.

Two 20m^3 tank cells (340-XCB-06, 08) provide more than sufficient residence time for the pyrite flotation process, and these cells operate much as the copper and zinc units upstream in terms of control. The rougher concentrate from the two cells flows by gravity to the feed box of a small bank of cleaner cells (340-XCB-16) for further upgrading. A high sulphur flotation concentrate will be produced and pumped (340-PFB-20) to a small dewatering circuit for further treatment. The cleaner bank tailing slurry will be pumped (340-PFB-28) back to the conditioner tank as a recirculating stream.

Pyrite concentrate will be stored in an agitated (340-XSA-24) surge tank (340-TBA-22). From this tank, it will be pumped (340-PCB-26) in batches to the pyrite conc pressure filter (340-AAA-28) for dewatering. The press will operate continuously to give a throughput of approximately 40 tonnes cake per day, which will be conveyed to the cake stockpile for incorporation into paste backfill.

The main items of equipment in Area 340 are described in Table 17-8 below.

Table 17-8: Area 340 Equipment

EQ #	Description	Notes	kW
340-TBA-02	Pyrite Conditioner Tank	30m3 @ 100% Level; 3m dia x 4.05m H	-
340-XSA-04	TBA-02 Agitator	AMX-750	7.5
340-XCB-06, 08	Py Rougher Cell 1-2	nextSTEP™-20RT INT	30 x 2
340-TAA-10	Py Rougher Tails Tank	7.5m3, 2.2m diameter	-
340-PCB-12	Py Rougher Tails Pump	slurryMAXR 8x6-22 MMB W/F-FF	45
340-PCB-14	Py Rougher Tails Pump S/by	slurryMAXR 8x6-22 MMB W/F-FF	-
340-XCB-16	Py Cleaner Cell	nextSTEP™ 150UT	7.5
340-PFB-18	Py Cleaner Tailing Pump	VT-100 Rubber, Open Impeller	7.5
340-PFB-20	Py Cleaner Conc Pump	VT-80 Rubber, Open Impeller	4
340-TBA-22	Pyrite Conc storage Tank	$30m^3$ 3m dia x 4.05m H	-
340-XSA-24	TBA-02 Agitator	AMX-750	5.5
340-PCB-26	Py Conc Filter Feed Pump	millMAXe 2x1.5-7 EMAAA W/F-F	7.5
340-AAA-28	Pyrite Conc Filter		11
340-PCV-32	Transfer Conveyor - Py Conc	18" x 20' horizontal transfer	3

17.3.7 Tailings Disposal - Area 350

Tailing slurry will be pumped (340-PCB-12) from the pyrite rougher tails tank (340-TAA-10) to the final tailings sampling station (ZAC-02, XDB-04) to the final tailings thickener feed box (ZAC-06) and thickener tank (ACA-08).

Thickener overflow will gravitate from the thickener tank to the process water tank for re-use within the process (mainly the grinding circuit).

Tailing thickener underflow slurry will be pumped (350-PCB-10, 12) from the thickener cone as a high density slurry to the thickened tailings storage tanks (350-TBA-16, 20). Each tank will be agitated (350-XSA-18, 22) and will hold approximately 2.5 hours of thickened tailing production. The tanks will be filled continuously and discharged semi-continuously to meet paste demand schedules. Thickened slurry from the storage tanks will be pumped (PCB-24, 26) to the tailings filters (350-AAA-30, 32) for further dewatering. During periods when demand for backfill paste is low, then the filtered cake will be diverted via reversible conveyor (350-FCV-50) onto a product conveyor (350-FCV-56) and bin (350-BAA-60) prior to dumping into trucks for placement on the nearby dry-stack tailing storage facility.

The filtrate from the tailing filters will gravitate to the filtrate tank (350-TBA-40) from where it will be pumped (350-PCA-42, 44) back to the tailing thickener as feed dilution water.

Filter cake will discharge onto the reversible cake conveyor (FCV-50) and will be directed in batches to a pug-mixer (350-XXX-68), which will mix tailings cake, Portland cement (3-4%) and water to an exact recipe. From the mixing vessel, backfill slurry will either gravitate (in early years) or be pumped (later years, using 350-PPD-78, 80) down to the stopes via a network of paste pipes.

The paste production plant will be remote from the process plant and will be located within a separate building.

The main items of equipment in Area 350 are described in Table 17-9 below.

Table 17-9: Area 350 Equipment

EQ #	Description	Notes	kW
350-XDB-04	Plant Tailings Sampler	100mm linear crosscut	3
350-ACB-08	Tailings Thickener	22m diameter, High compression.	22.5
350-PCB-10, 12	Thickener underflow pumps	slurryMAXR 6x4-14-Rubber MMA W/F-FF	22x 2
350-TBA-16, 20	Thickener underflow surge tank #1 & #2	245m^3 cylindrical, 6.2m dia x 8m H	-
350-XSA-18, 22	TBA-16, 20 Agitators	AMX 3700	37x2
350-PCB-24, 26	Filter Feed Pumps	millMAX 3x2-9 MMAA M/S	2 x 18.5
350-AAA-30, 32	Tailings filters	ME2500 c/w 88 chambers	55 x 2
350-PCD-38	Tailings Filter Area Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
350-TBA-40	Filtrate Transfer Tank	30m^3 cylindrical 3m x 4.05m H	-
350-PCA-42, 44	Filtrate transfer pumps	millMAXe 2x1.5-7 EMAAA W/F-FF	7.5 x 2
350-FCV-50	Reversible Cake Conveyor	24" x 66'	11
350-FCV-56	Cake Transfer Conveyor (Stockpile)	24" x 101'	15
350-BAA-60	Cake Storage Bin	100m^3 capacity; 60 degree conical bottom	-
350-FCA-64	BAA-60 Discharge Feeder	Pan Feeder	3
350-PCD-66	Cake Area Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
350-XXX-68	Paste Mixer	2 x 30kW drives	60
350-BAA-70	Cement (Binder) Hopper	On load cells	0
350-XXX-72	Rotary Valve	design to dispense exact mass	3
350-FCS-74	Screw Feeder	closed system	7.5
350-BCA-76	Paste hopper	Allows sampling and ter fine tune	-
350-PPD-78, 80	Paste Pumps	Piston type, 55 m3/h	250 x 2
350-PCD-90, 92	Spillage Pumps	millMAX 3x2-9 MMAA W/F-FF	7.5 x 2

17.3.8 Copper and Zinc Concentrate Dewatering - Area 360

Copper and zinc final flotation concentrates are pumped from their respective cleaner flotation circuits to the two concentrate thickeners in this area.

Copper flotation concentrate will be pumped to the sampling launder (ZAC-02) and sampler (XDB-04) before entering the feed launder (ZAC-06) and concentrate thickener tank (ACA-08) for controlled dewatering. The copper concentrate thickener will be equipped with vendor-supplied rake lift, bed level detection and bed mass monitoring. Thickener overflow (water) will gravitate to the copper spray water tank (TBA-16) for re-cycling within the copper circuit, and thickener underflow will be withdrawn

from the cone by peristaltic type underflow pumps (PPA-10, PPA-12) and pumped forward to the storage tanks, or re-cycled to the thickener feed if of insufficient density.

Thickener underflow of the correct density will be directed forward to the two mechanically agitated (XSA-24, XSA-28) concentrate storage tanks (TBA-22, TBA-26), where the concentrate will be stored awaiting filtration. Storage capacity will exceed 24 hours.

Copper thickener area spillage will be recovered by pumping (PCD-14) back to the copper concentrate thickener.

Zinc flotation concentrate will be pumped to the sampling launder (ZAC-62) and sampler (XDB-64) before entering the feed launder (ZAC-66) and zinc concentrate thickener tank (ACA-68) for controlled dewatering. The zinc concentrate thickener will be also equipped with rake lift, bed level detection and bed mass monitoring. Thickener overflow will gravitate to the zinc spraywater tank (TBA-76) for re-cycling into the zinc circuit, whilst thickener underflow will be withdrawn from the cone by peristaltic type underflow pumps (PPA-70, PPA-72) and pumped forward to the storage section, or re-cycled to the thickener feed if of insufficient density.

Zinc thickener underflow of the correct density will be directed forward to the two mechanically agitated (XSA-82, XSA-86) concentrate storage tanks (TBA-80, TBA-84), where the concentrate will be stored awaiting filtration. Storage capacity will exceed 24 hours.

Zinc thickener area spillage can be recovered by pumping (PPD-74) back to the zinc concentrate thickener.

Concentrate stored in the copper and zinc storage tanks will be pumped in batches (PCA-30, 32) forward to the single pressure filter for dewatering. The pressure filter will be served by high pressure (10 and 16 bar), instrument grade air – allowing:

- inflation of the filter plate membranes to effect a pressure squeeze (16 bar air)
- further “air blow” dewatering of the compressed cake (10 bar air)

Filtrate from the pressure filter will be directed to a filtrate de-aeration chamber (ZAC-46) and into a filtrate/manifold flush tank (TAA-48). This tank overflows to the filtrate pump (PCB-50) which transfers filtrate back to the appropriate concentrate thickener for re-cycling. Automatic valves control the flow of filtrate to either thickener per the batch cycle. Manifold flush pumps withdraw filtrate from the filtrate tank and flush the filter feed manifold between cycles. Flushed manifold product will gravitate to the filtrate transfer pump and will be re-cycled to thickener feed. A cloth wash tank and pumps (vendor supplied) allow high pressure washing of the filter cloth at the end of each cycle. Clean water from the services area will be used as makeup.

Filter cake will be discharged from the press via cake discharge chutes (Vendor supply) onto a reversible cake transfer belt (FCV-54) which transports cake to the concentrate storage shed. A front-end loader serves the cake stockpiles and loads concentrate cake into side tipping trucks for transportation off-site. Trucks are weighed and auger-sampled at the weighbridge prior to dispatch.

The main items of equipment in Area 360 are described in Table 17-10 below.

Table 17-10: Area 360 Equipment

EQ #	Description	Notes	kW
360-XDB-04	Final Cu Conc Sampler	Rotary Vezin	1.1
360-ACB-08	Cu Concentrate Thickener	6m diameter HRT (C-Series)	4
360-PPA-10, 12	Cu Underflow Pumps	C65 Peristaltic	5.5 x 2
360-PCD-14	Cu Area Spillage Pump	HDSP65-QV-L1200	7.5
360-TBA-16	Cu Spraywater Tank	Unlined, 25m^3	-
360-PCC-18	Cu Spraywater Pump	millMAXe 2x1.5-7 EMAAA W/F-F	5.5
360-TBA-22, 26	Cu Thickener U/F storage Tanks	40m3 Capacity, unlined, 2 off	-
360-XSA-24,28	TBA-22&26 Agitators	AMX 550	5.5 x 2
360-PCA-30, 32	Cu Filter Feed Pumps	millMAX 3x2-9 MMAA M/S	18.5 x 2
360-AAA-34	Concentrate Filter	Larox PF 11/11 M1.6 1 60	22
360-xxx-42	Squeeze Water package	pumps x2 + small tank; included with filter	5.5
360-xxx-44	Compressed Air Package	Vendor Supplied	15
360-ZAC-46	Filtrate de-aeration tank	Vendor Supplied	-
360-TAA-48	Filtrate Transfer Tank	5m^3, 1.83m dia x 2.45m H	-
360-PCB-50	Filtrate Transfer Pump	Warman 1.5/1 BAH all metal	2.2
360-FCV-54	Cake Conveyor - Shuttle	Reversible, 600mm wide	5.5
360-PCD-60	Filtration Area Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
360-XDB-64	Final Zn Conc Sampler	Rotary Vezin	1.1
360-ACB-68	Zn Concentrate Thickener	6m diameter HRT (C-Series)	4
360-PPA-70, 72	Cu Underflow Pumps	C65 Peristaltic	5.5 x 2
360-PCD-74	Zn Area Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
360-TBA-76	Zn Spraywater Tank	Unlined, 25m^3	-
360-PCC-78	Zn Spraywater Pump	millMAXe 2x1.5-7 EMAAA W/F-F	5.5
360-TBA-80, 84	Zn Thickener U/F storage Tanks	40m^3 Capacity, unlined	-
360-XSA-82, 86	TBA-80,84 Agitators	millMAXe 2x1.5-7 EMAAA W/F-F	5.5 x 2
360-PCA-88, 90	Filter Feed Pumps (Zn)	millMAX 3x2-9 MMAA M/S	18.5 x 2

17.3.9 Reagents - Area 800/820

Reagents will be delivered to site by road and stored in a purpose-built storage building adjacent to the mill building, rather than in the main warehouse. From this storage area, reagents will be moved by forklift into the plant reagents area on an as-needed basis. Reagents required for the process plant are listed below:

Frother 1 - MIBC

Liquid MIBC will be delivered in 45gal drums or 1m^3 IBC totes to site by road and stored in the reagent area adjacent to the process plant building. Peristaltic hose pumps meter frother solution from a single IBC tote in the reagent area to several addition points throughout the plant. Roughly 1.8 tonnes of MIBC will be consumed per month.

Frother 2 – Polyfroth W31

Liquid W31 will be delivered in 45gal drums or 1m^3 IBC totes to site by road and stored in the reagent area adjacent to the process plant building. Peristaltic hose pumps meter frother solution from a single

IBC tote in the reagent area to several addition points throughout the plant. Roughly 1.8 tonnes of W31 will be consumed per month.

Promotor – Aero 5100

Liquid 5100 will be delivered in 45gal drums to site by road and stored near to the process plant. Peristaltic hose pumps meter promotor solution from a single drum in the reagent area to several addition points throughout the plant. Roughly 1.8 tonnes of 5100 will be consumed per month.

Collector – Aerophine 3418A

Liquid 3418A will be delivered in 45gal drums to site by tanker and stored near to the process plant. Peristaltic hose pumps meter collector solution from a single drum in the reagent area to several addition points throughout the plant. Roughly 8 tonnes of 3418A will be consumed per month.

pH Adjustment – lime

Quick lime powder will be delivered to site in bulk tankers and stored in the load cell-mounted bulk storage hopper. A vendor-package lime slaking plant mixes the quick lime with water on an as-needed basis and transfers the slaked lime solution to an agitated storage/dosing tank within the plant building.

The lime slaking batch sequence will be fully automatic and should occur twice daily on average. Lime will be pumped throughout the plant via a ring main distribution system, with flow-metered take-offs for each dosing point. Roughly 160 tonnes of quicklime will be consumed per month.

Reagent spillage will be pumped to the tailings tank for disposal on the tailings dam.

Depressant – Sodium Cyanide

Sodium Cyanide briquettes are delivered in 45gal drums to site by tanker and stored near to the process plant. Drums will be moved over to the process plant on an as needed basis for mixing and storage of solutions within the reagent area. Peristaltic hose pumps will meter collector solution from the storage tank to several addition points throughout the flotation plant. Roughly 1.8 tonnes of NaCN will be consumed per month.

Depressant – Zinc Sulphate

$ZnSO_4$ powder will be delivered in 50-kg bags to site by road and stored near to the process plant. Bags will be transferred over to the plant on an as needed basis, then mixed with water in a mixing tank within the reagent area. Peristaltic hose pumps will meter depressant solution from the storage tank in the reagent area to several addition points throughout the plant. Roughly 6.4 tonnes of $ZnSO_4$ will be consumed per month.

Activator - Copper Sulphate

$CuSO_4$ powder will be delivered in 500kg bags to site by road and stored near to the process plant. Bags will be transferred over to the plant on an as needed basis, then mixed with water in a mixing tank within the reagent area. Peristaltic metering pumps will dose activator solution from the storage tank in the reagent area to several addition points throughout the plant. Roughly 6.4tonnes of $CuSO_4$ will be consumed per month.

CMC Depressant – PE26

Depressant (CMC) powder will be delivered to site in bulk tankers and stored in the load cell-mounted bulk storage hopper. 3,000 kg of CMC powder will be metered into the batch storage hopper prior to mixing. From here, powder will be withdrawn by the depressant screw feeder and blown through a venturi to a wetting head located on top of the mixing tank. CMC solution will be then mixed further by re-circulation through a pump.

Once hydrated, the 2% solution will be fed forward to the required storage tank.

The CMC batch mixing sequence will be fully automatic and should occur daily on average. Depressant will be pumped throughout the plant via a ring main distribution system, with flow-metered take-offs for each dosing point.

The area will be served by strategically placed safety showers

Flocculant

Flocculant powder will be delivered to site in 1t bags and stored in the reagent storage area. Bags will be lifted by hoist and added to the flocculant powder hopper. Powder will 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 will be fed forward to the storage tank or recycled back into the mixing tank to aid mixing. Once mixed, the flocculant should be left for several hours to hydrate. The storage tank will provide sufficient volume for storage of flocculant whilst the mixed batch hydrates in the mixing tank. From the storage tank, flocculant will be pumped through a ring main for dosing at each of the three thickeners.

17.3.10 Water and Air Reticulation - Area 900

Water will be recovered from the concentrate thickeners into copper and zinc spray water tanks, from where it will be pumped to the respective flotation circuits as spray water.

Process water will be recycled from the tailing thickener and stored in an insulated tank located outdoors immediately adjacent to the process plant building. Process water will be distributed to the plant by process water pumps and pipe networks. The process water tank will be also used to feed the diesel-powered fire water pump from a separate (lower) offtake, thus guaranteeing water availability for emergencies.

Clean water will be piped into the plant from a nearby lake and stored in the plant clean/raw water tank. From the storage tank, water will be pumped around the plant for use as reagent mixing water, slurry pump gland seal water and as required for mill lubrication system cooling.

Plant and instrument air will be provided by three compressors. Air quality will be maintained by filters. Instrument air will be dried using a refrigeration drier. Receivers are provided for compressed and instrument air lines, to allow for surges in demand. Low pressure (flotation) air will be supplied to the flotation plant by two sets of blowers. Blowers will be fixed speed, with manifold pressure controlled by modulating valves on an exhaust lines. The area will be serviced by an electric hoist.

The main items of equipment in Area 900 are described in Table 17-11 below.

Table 17-11: Area 900 Equipment

EQ #	Description	Notes	kW
900-TBA-02	Process Water Tank	6.5m dia x 10m h; 330 m3	-
900-PCC-04	Process Water Pump	millMAXe 8x6-17 MMB C/S	75
900-PCC-06	Process Water Pump S/By	millMAXe 8x6-17 MMB C/S	-
900-PEA-08	Fire-water Pump	Diesel Pump; electrical jockey	-
900-PCE-10	Water Area Spillage Pump	millMAX 3x2-9 MMAA W/F-FF	7.5
900-TBA-12	Clean Water Tank	4.4m dia x 6.6m h; 100 m3	-
900-PCC-14	Clean Water Pump	millMAXHH 3x2-14 MMA C/S	45
900-PCC-16	Clean Water Pump S/by	millMAXHH 3x2-14 MMA C/S	-
900-HAC-18	Plant Air Compressor	GA50	55
900-HAC-20	Plant Air Compressor S/by	GA50	-
900-HAD-26	Instrument Air Drier		0.1
900-HFA-32	Instrument Air Receiver		-
900-HBB-34	Flotation Air Blower (Roughers)	Type 151A-07	300
900-HBB-38	Air Blower S/By	Type 151A-07	-
900-HBB-40	Flotation Air Blower (cleaners)	Type 151A-04	90
900-HBB-42	Air Blower S/By	Type 151A-04	-

17.4 Process Installation Requirements:

The following is a list of general process-specific installation requirements:

- Pipelines that transport slurry or water outside of the heated mill building will be lagged and heat traced, so as to handle the low temperatures and wind chill experienced on site during winter months.
- Regardless of location, slurry pipelines will be self-draining and equipped with flushing and drainage facilities, as necessary. Dead-legs will be avoided and the exploitation of gravity flow as a transport mechanism will be practiced wherever feasible.
- A slope of between 5° and 10° will be applicable in the concrete spillage containment area, where the floors will slope towards the spillage sumps.
- No individual flotation cell bypass facilities are required.
- A slope angle of between 12° and 15° is required for extended flotation concentrate launders. Spray water can be provided in the launders to assist with froth handling.
- Flotation concentrate launders should not be fully enclosed and should designed to facilitate manual sampling with hand-held sample cutters.
- Flotation cells are to be installed at a sufficient elevation to ensure that all process pump boxes are above ground level, taking due cognisance of launder slope angle and pump and cell mechanism maintenance.
- All mainstream slurry sumps/tanks will be sized to have a design residence time of 90 seconds or greater and a minimum valley angle of 45°.

- Slurry pipelines will be sized to ensure slurry velocities of between 1.5 and 2.0 m/s under normal operating conditions.
- The design of all equipment is to be such as to minimize spillage and maintenance and maximize availability.
- Final step heights between flotation cells will be determined by engineer and vendor in consultation with client's metallurgical personnel.

17.5 Control Systems and Instrumentation Levels

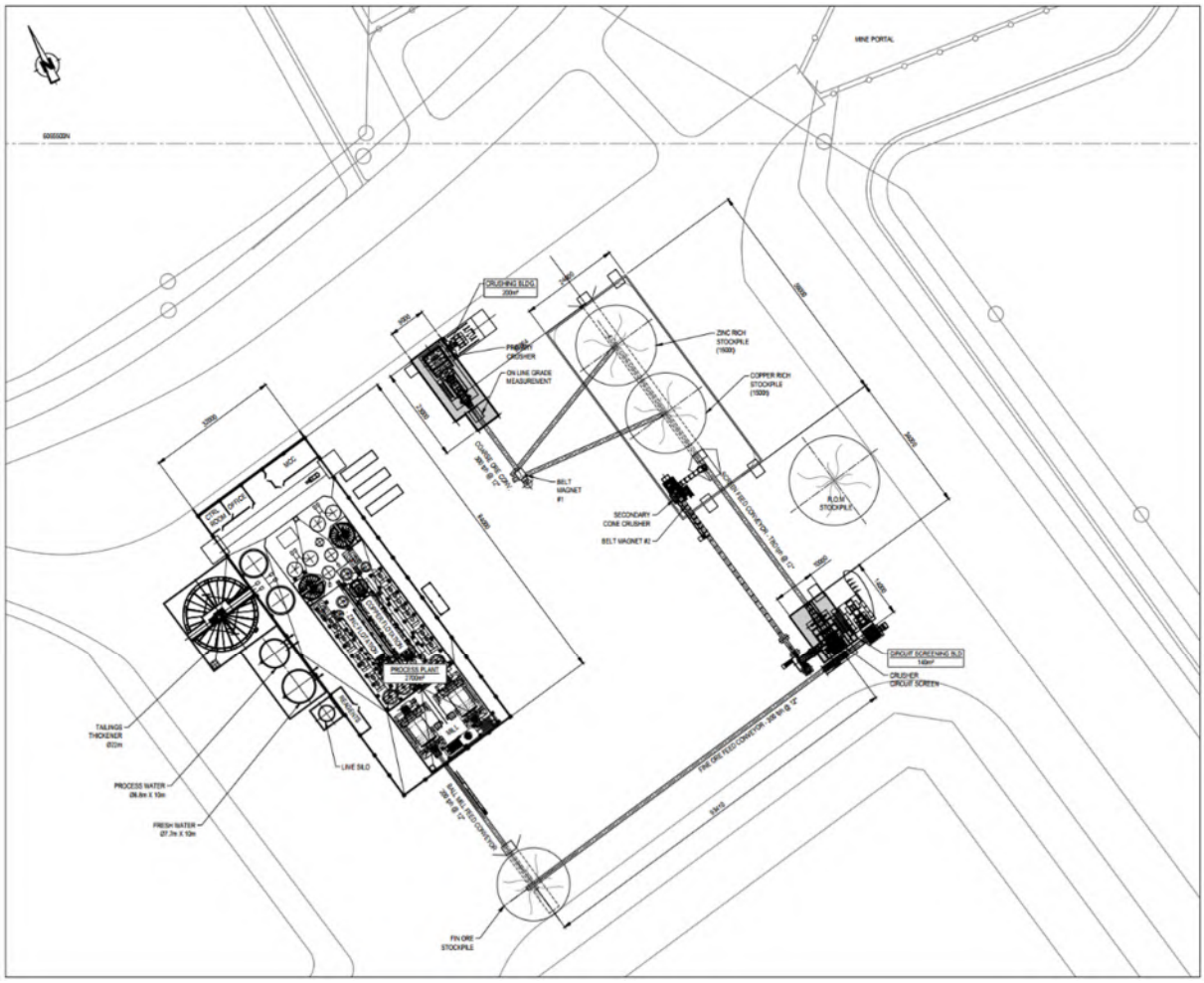
The mill will be equipped with modern process control systems, including remote operation (start/stop etc..) from a central control room. Process monitoring via HD camera systems mean that the control room may ultimately be located off site.

The majority of mill processes will be monitored and controlled through the addition of flow, density, pressure, temperature, pH controllers and actuated control valves. The level of instrumentation and control on the plant will in general be high, requiring lower levels of intervention by plant operators.

17.6 Mine Interfaces

The processing plant will be located immediately southeast of the mine portal with easy access for operators to the change room facilities, the workshop and spares storage areas. The current site layout is given in Figure 17-10 below.

Figure 17-10: Process Plant and Stockpile Layout



Site power will be achieved via a 34.5 kV powerline from Pelican Narrows.

Process plant make up water will be obtained largely from treated mine seepage, and during periods where that supply becomes unavailable, from a nearby ground well at a peak rate of 20 m^3/h . Average long term demand will be less than 10 m^3/h.

At times where the site water balance runs in surplus, excess water will be directed to the water treatment plant (described in the infrastructure package) prior to discharge. The water treatment plant will handle water from underground that is unsuitable for use as process water.

17.7 Expansion Planning

The processing facility is designed to allow expansion of throughput, should this be required. The initial facility (Year 1) will have an effective 3,300 – 3,400 tpd capacity due to the absence of standby equipment and lower equipment availability.

Process plant throughput can be increased incrementally, as follows:

- Provision of standby pumps and parallel pipelines for an improvement in operational availability to provide an increase to ~3,700 tpd capacity.
- Addition of a preconcentration facility ahead of the grinding circuit for a reduction in copper stockwork tonnage and overall hardness to the milling circuit (provides an effective 10% increase in front-end capacity).
- Additional crushing of ROM product, using a second cone crushing circuit. This will provide a potential increase in capacity in excess of 3,800 tpd should this be required.

The first option is budgeted in the sustaining capital budget. Other items could be implemented on an as-needed basis (e.g. metal price increases, and expansion of mine reserves) to match the mine production. However, these expansion items will be catered for in the initial designs, so as to ensure trouble-free implementation.

18. PROJECT INFRASTRUCTURE

The McIlvenna Bay project will include a relatively compact site, with major features including the tailings storage facility, ore and waste dumps, water treatment infrastructure, and buildings such as offices, workshops, dry, and the mill building.

The site will be connected to public roads via an existing gravel road and will receive hydropower via a tie-in to the existing grid. Supplies will be trucked to site via Flin Flon and/or Saskatoon, and flotation concentrate products will be shipped to market via road to Flin Flon.

18.1 Existing Infrastructure

The McIlvenna Bay project is located in close vicinity to a previous (reclaimed) quarrying operation and benefits from an existing access road, cleared laydown areas, and a deactivated SaskPower distribution line. In addition to this, Foran has constructed various infrastructure to support ongoing exploration activities, including an all-season exploration camp (Figure 18-1) complete with sleeping accommodation for 35, fuel storage, communications, toilet and showering facilities, canteen, core shack, septic, potable water supply and treatment system.

Figure 18-1: Existing Exploration Camp

18.2 Project Site Preparation

The proposed McIlvenna Bay site layout is presented on Figure 18-2. The site layout is designed to be compact and centralized around the mine portal with the objectives to minimize the environmental footprint, and to minimize underground to surface haul distances. Prior to construction, the site will be cleared and stripped of vegetation with organics stockpiled for future surface reclamation. There is generally 0 to 2m of overburden on site underlain by dolomite rock.

Upon engineering design completion and construction mobilization, the first area to be developed will be the mine laydown area. Clearing this area will give contractors room to set-up trailers and stage equipment during the early phases of construction. The site is to be cleared and grubbed to support the construction of the surface infrastructure as well as preparation for light and heavy vehicle traffic.

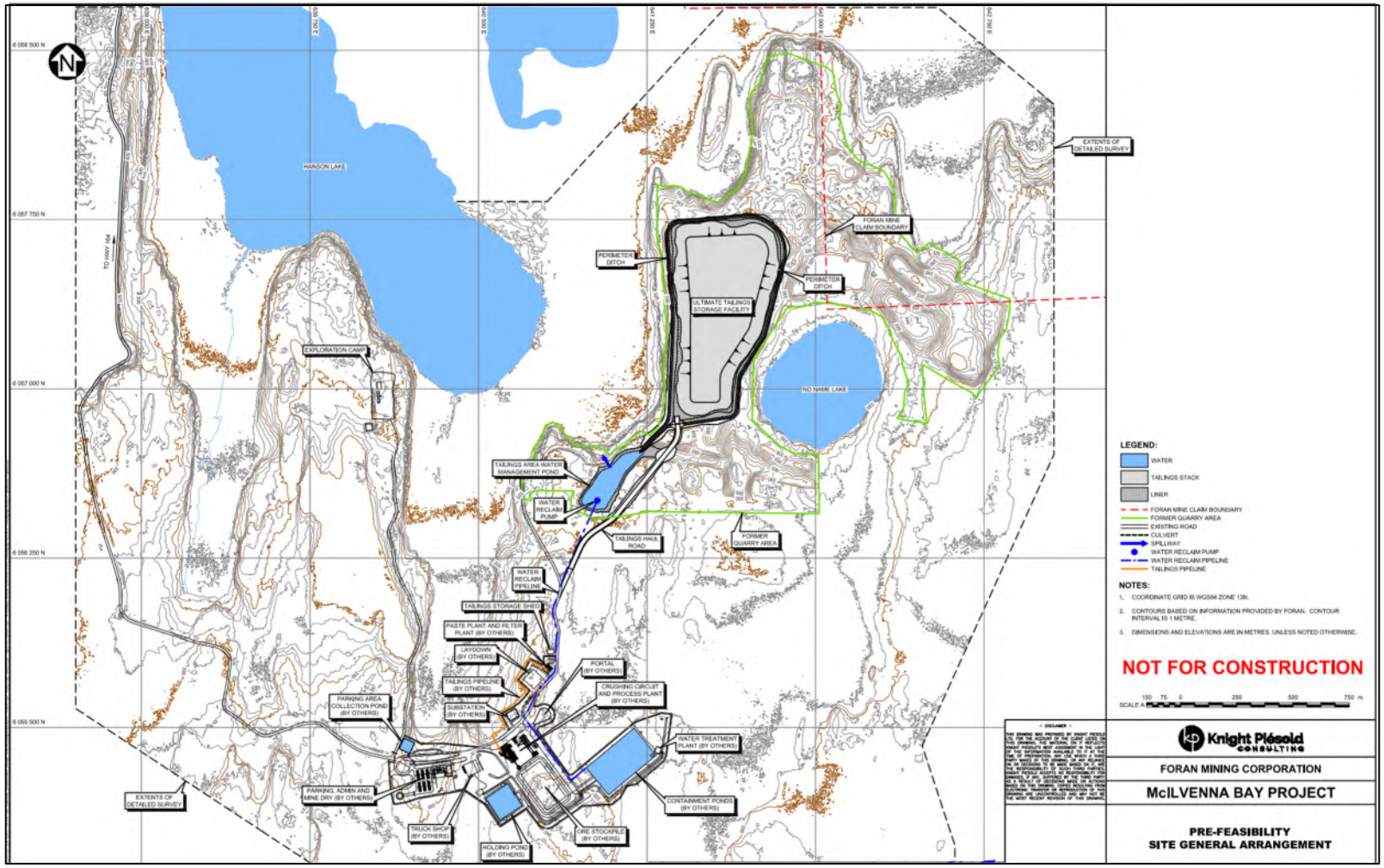
Clearing and grubbing will include the cutting and disposing of trees, timber, brush, roots, and stumps. The general areas around infrastructure will be roughly graded prior to construction.

The existing hydro line will be recommissioned to provide adequate power for construction activities.

18.2.1 Soil Storage

Topsoil removed from the site will be stored in the designated soil storage area for reclamation. The actual volume of stripped topsoil stored for reclamation will be determined with a complete site investigation and geotechnical assessment. The base and top of the soil storage area will be lined to mitigate contamination and erosion of the soil.

Figure 18-2: Proposed Site Layout (Knight Piésold, 2020)



18.3 Project Access

18.3.1 Mine Road

The existing gravel road between the mine site and Highway 106 will be used for transportation of mine personnel and general supplies over a distance of approximately 18km. The road will require cleaning along its edges where organics have grown in since the closure of the sand quarry. The intent is to recover the road to the width it was during the quarrying operation.

There is a bridge located on the mine access road at coordinates 636320E, 6062173N (Figure 18-3). The span of the haul road from the bridge (including the bridge) is under the jurisdiction of the Saskatchewan Ministry of Highways and Infrastructure (SK MH&I). The bridge was previously used for haul traffic during the former sand quarry and therefore it is assumed that the bridge will not require upgrading. Future detailed inspection and consultation with the SK MH&I will be required to confirm the condition, specifications and load ratings of this structure.

Figure 18-3: Single Lane Bridge on Access Road

18.3.2 Highway 106

Traffic on public Highway 106 from the intersection at the haul mine road to Flin Flon, MB is under the control of the SK MH&I and will be maintained, including the removal of snow by the SK MH&I. To maintain safe/reasonable traffic levels on this highway, the majority of Operations and Admin

personnel will be bussed to/from site on a daily basis, using contract transportation services. New road lighting at the intersection of the mine access road and Highway 106 is recommended to assist with visibility, given that mine personnel busses will be maneuvering in this area at night.

18.3.3 Mine Service Roads and Site Parking

A light vehicle parking lot for mine personnel, contractors and visitors will be located adjacent to the office complex, just inside the guard house gate but separated from heavy underground haul truck traffic. The parking lot will be partially equipped with receptacles for block heaters as well as a walkway towards the office complex and warehouse. The parking lot will accommodate up to 84 light vehicles

18.3.4 Helipad

A 20 m diameter helipad will be constructed adjacent to the mine site entrance and will be maintained over life of mine for emergency access. The helipad can be utilized in case of an emergency that would benefit from medivac.

18.4 Buildings and Ancillary Items

18.4.1 Temporary Construction Facilities

Temporary construction facilities required during the construction phase will be the responsibility of the site contractor(s) as well as contractor mobilization and demobilization. It is assumed that there will be no construction camp facilities provided by Foran, and that construction crews will make use of off-site accommodation.

Transportation to and from the site will be the contractors' responsibility and no on-site catering services will be provided by Foran.

18.4.2 Mill Building

The process plant will be housed within an 84 x 32 x 22m high pre-engineered building. The building will be complete with metal cladding, insulation, framed openings for equipment and HVAC, snow guards, crane runway beams and several man doors and overhead doors for mobile equipment access.

The pre-engineered building will be installed on concrete walls around the perimeter of the building. The walls will be backfilled and compacted on the interior and exterior of the building. There will be concrete floor slabs in the process plant along with sloped floors which are directed towards sumps in areas in which there is potential for spillage. The sumps will be equipped with sump pumps which will direct spillage to specific parts of the process.

The process building will be equipped with a 7.5t crane along the span of the building. Jib cranes or hoists will be used in partitioned areas such as the reagents store and thickener bridge.

On the north end of the building, the control room, instrumentation room and two offices will be located for operations and supervisors. Two levels of MCC rooms will be located to the east of the offices, complete with double doors to allow for the installation and removal of MCC tiers and drives. A second means of egress will also be included for each MCC room.

A drive-through bay on the north end of the building will allow for the maneuvering of equipment in and out of the plant.

The ore will enter the process plant via a mill feed conveyor on the south end of the building. The process will generally flow from the south towards the northern end of the building where the concentrate will be discharged.

There will be several pieces of equipment on the exterior of the process plant, such as the thickener, process and freshwater tanks, lime silo and storage containers. Where practical, the concrete for these exterior pieces of equipment will be separate from the concrete works of the pre-engineered building. This will allow for the pre-engineered building to be erected independently which is advantageous since this permits construction to continue within the building during the winter season.

A plan of the process plant building is given in Figure 18-4 overleaf.

18.4.3 Warehouse and Truck Shop

A 2,250 m^2, heated, pre-engineered building will be constructed for the warehouse and truck shop to allow for sheltered space to perform maintenance on heavy and light vehicles as well as storage space for surface and underground parts and consumables (Figure 18-5). A corridor between the warehouse and truck shop maintenance area will allow for access throughout the building without having to exit the building.

The truck shop will be equipped with 20t and 5t cranes (1 each). The shop will be fitted with 12 x 9m and 4.2 x 4.2m overhead garage doors to allow for mobile equipment to drive into the bays. An oil/water separator will be included to collect wastewater prior to being pumped to the containment pond. The truck shop will contain two large vehicle bays initially with room for future expansion when mine development reaches peak production and more trucks are in circulation.

The warehouse will have 1,500m^2 of open storage space in the main floor and additional space for lockable storage, tool crib, and electrical storage on the first floor. A fire-rated wall will separate the 180 m^2 lubrication storage facility.

18.4.4 Office Complex

A modular, 800 m^2 office complex will accommodate general management and admin staff and includes the following:

- office space for management, technicians, safety personnel, administration and other staff,
- space for the underground operations control room,
- conference room,
- kitchen,
- washrooms, and
- mechanical and IT room.

A modular facility was chosen to minimize concrete works as well as the ease of piping from the kitchen and washrooms to the sewage treatment plant.

Figure 18-4: Mill Building Layout (Halyard, 2020)

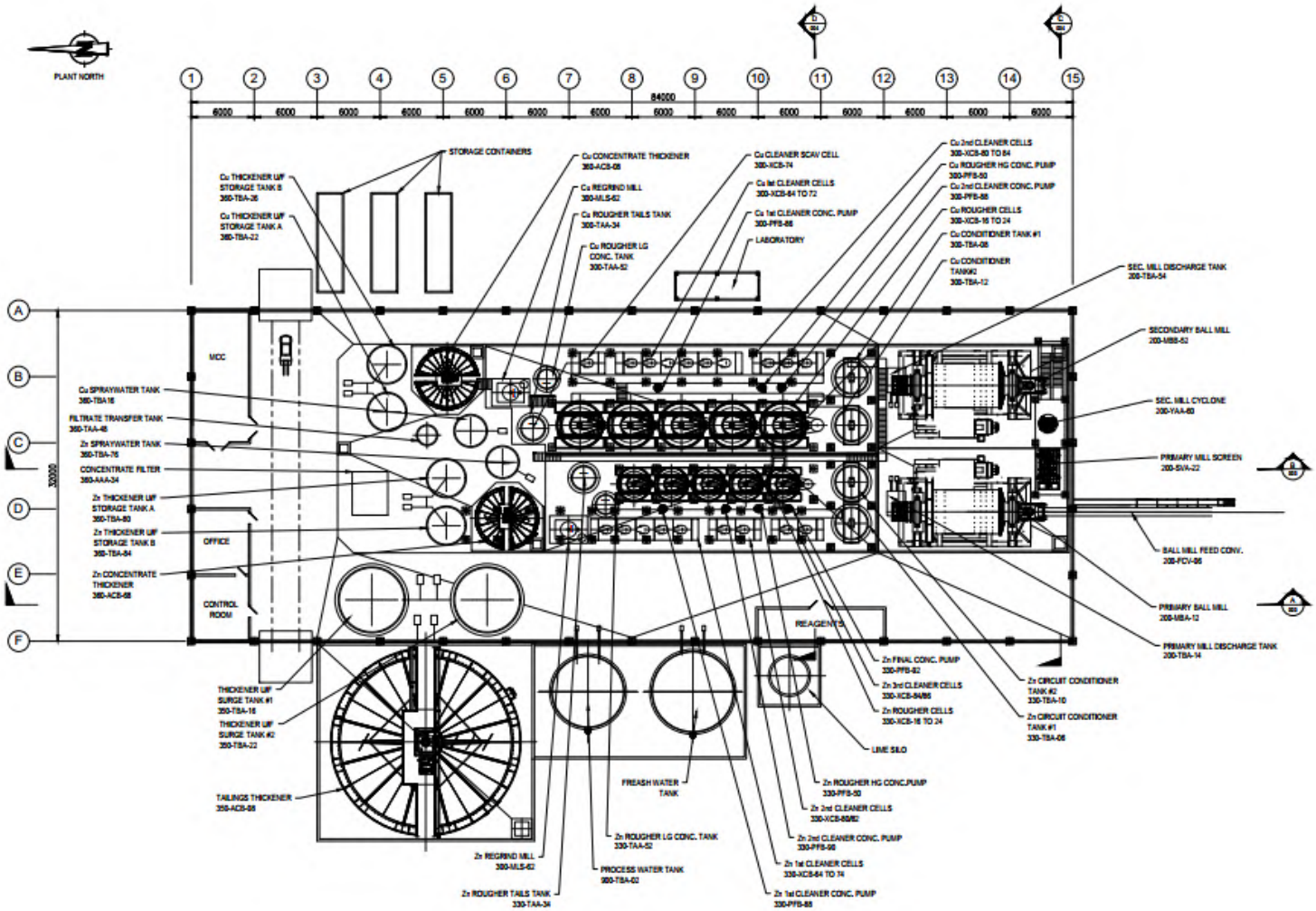
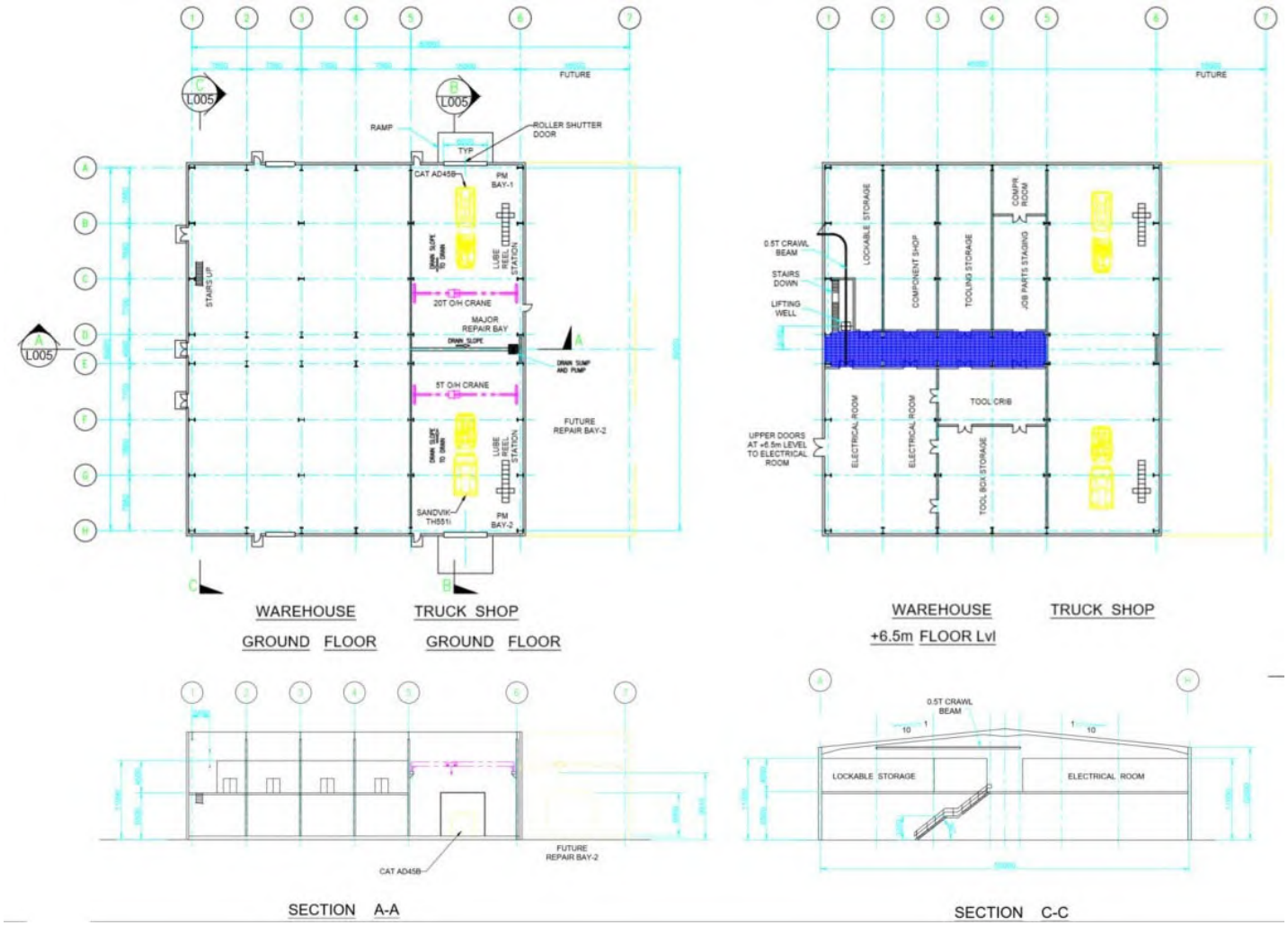


Figure 18-5: Truck Shop and Warehouse Layout (Halyard, 2020)



18.4.5 Mine Dry

A modular, 540m^2 dry will accommodate 150 personnel on site at any time, with a total of 300 mine personnel. The contents of the dry are as follows:

- 300 total lockers
- 35 shower heads
- 17 urinals/stalls
- 15 sinks
- benches
- mechanical room and 3 hot water tanks

18.4.6 Truck Wash Facility

A truck wash facility will be constructed to the south of the truck shop. The facility will be equipped with water hoses for manual washing of trucks. Platforms on both sides of the facility will allow for operators to reach the top of the trucks. An oil/water separator will be installed in the facility for the spent water prior to pumping to the containment pond.

18.4.7 First Aid

A first aid trailer will be constructed adjacent to the office complex.

18.4.8 Security Gate House And Truck Scale

A gate house will be constructed at the entrance of the mine site. The gate house will control access in and out of the site as well as the truck scale operation. The weigh scale will be required to provide accurate measurement of concentrate shipment weights. The weigh scale indicator and ticket printer will be located in the gate house.

18.4.9 Explosives Storage

The explosives storage magazines will be located north of the mine site, towards the old sand pit. Emulsion will be stored in a 20'x8'x7' type 4 magazine as per the Natural Resources Canada – Explosives Regulatory Division's requirements. The emulsion magazine will have the capacity to store 12,000kg of product.

A separate storage magazine cabinet, away from the emulsion magazine, will be used to store the detonators. This magazine will have a capacity of 500kg. Each of the magazines will be complete with lockable doors.

18.4.10 Mine Rescue Trailer

A 67 m^2 mine rescue trailer will be constructed adjacent to the office complex.

18.4.11 Fuel Storage And Distribution

A fuel storage and distribution facility will be located at the northeast section of the mine site. This facility will store the diesel and low sulphur diesel tanks, dispensing equipment and an oil-water separator.

Low Sulphur and Diesel Storage Distribution

Storage and dispensing equipment for low sulphur diesel will be constructed for use by the underground mobile equipment. The low sulphur diesel will be delivered to site in tanker trucks every 2 days. The storage facility will be equipped with a high flow pump to fill the service truck that will be used to fill the underground mobile equipment.

Regular Diesel Storage and Distribution

Storage and dispensing equipment for diesel will be constructed for use by the surface generators, as well as filling mobile surface equipment (excluding haul vehicles). Diesel for the generators will be gravity fed into the local tank within the generator enclosure.

18.4.12 Waste Disposal

Waste from site such as plastic, paper, cardboard, scraps, rubber, etc. will be collected and stored in the garbage disposal area. The waste will be picked up on a set schedule and sent away for disposal. Hazardous materials such as spent oil, batteries, paint, compressed gas cylinders, corrosive products, etc. will be disposed of adequately as per Foran's Environmental Department's Standard Operating Procedure. Foran's Health and Safety Department will ensure signage, SDS's, and Standard Operating Procedures are available and current for all hazardous materials that will be used on site.

Hazardous materials will be transported to and from site by contractors. Foran's Health and Safety Department will ensure the drivers of the vehicles transporting hazardous goods are trained and equipped to deal with accidental spills.

18.4.13 Mine Laydown Area – Cold Storage

A 3,600 m^2 laydown area will be constructed at the north end of the mine site. This laydown area will serve as the construction staging area during the surface and portal construction phase.

18.5 Power Supply and Distribution

Flin Flon and its surrounds have historically been well served with hydroelectric power, due to the presence of the now decommissioned Flin Flon copper smelter. Excellent hydroelectric infrastructure remains and is available to power a mining project such as McIlvenna Bay.

18.5.1 Power Supply

Long term power will be supplied to McIlvenna Bay by the Saskatchewan provincial grid via lines from the Pelican Narrows substation (which in turn is powered by the Island Falls Hydroelectric Station). Power will be supplied to the McIlvenna Bay site via two parallel overhead distribution lines:

- an existing 25 kV overhead distribution line, previously used at the sand pit operation
- a new dedicated 34.5 kV dual overhead distribution line

The existing 25 kV distribution line to the mine site is a remnant from the closed sand quarrying operation. The existing line is a shared line with approximately 1.2 MVA of available capacity. This line will be recommissioned in time to provide power for all construction activities. Recommissioning maintenance activities will be required in specific areas to replace missing and damaged parts, as well

as a sweep of the line to ensure trees and vegetation are not obstructing the line. Modifications to the existing equipment will be required at the project site to suit the site conditions and planned project infrastructure.

Site power will be supplied by a new dedicated 34.5 kV distribution/transmission line from Pelican Narrows. The routing of this line will parallel the existing 25kV line, which could be de-commissioned once the 34.5 kV supply is in service. The study assumes that portions of the existing 25 kV line will be re-used for site power distribution.

18.5.2 Power Distribution

The incoming 34.5 kV overhead lines will supply a new on-site substation, to be constructed adjacent to the mine portal. Two power transformers will supply underground mining operations and surface infrastructure as follows:

- a liquid filled transformer will step down the 34.5 kV distribution voltage to 13.8 kV for the underground mining operations; dual 13.8 kV feeds will provide power to the mine via the portal.
- a 34.5-4.16 kV liquid filled transformer will supply the process plant and surface infrastructure facilities.

Two 4.16 kV overhead power lines will supply facilities such as the crushing area, administration building, air raises, diesel fuel farm, effluent treatment plant, and explosives storage. The routing of the 4.16 kV overhead lines will endeavour to re-purpose as much of the existing overhead lines infrastructure as possible.

Power factor correction equipment will be provided at the main substation in order to meet SaskPower's minimum power factor requirements as well as reduce electrical system losses.

Additional details of the underground power distribution to mine service areas are described in Section 16.12 of this report.

The 250kW 600V self-contained, skid mounted diesel generator currently used to power the exploration camp will be provided at the new office complex to supply the critical power loads in the event of a hydropower interruption/outage. The generator will be fitted with a 12 hr sub-base fuel tank. This generator will be wired to provide backup power to critical equipment such as life and safety systems, communications equipment, security equipment and control systems. The generator loads and size will be finalized during the detailed design.

Provision for the connection of a portable diesel generator will be provided for the following areas:

- Potable Water Treatment Plant
- Sewage Treatment Plant
- Containment Pond pumps and heat tracing
- Holding Pond pumps and heat tracing
- Collection Pond pumps and heat tracing
- First Aid Trailer

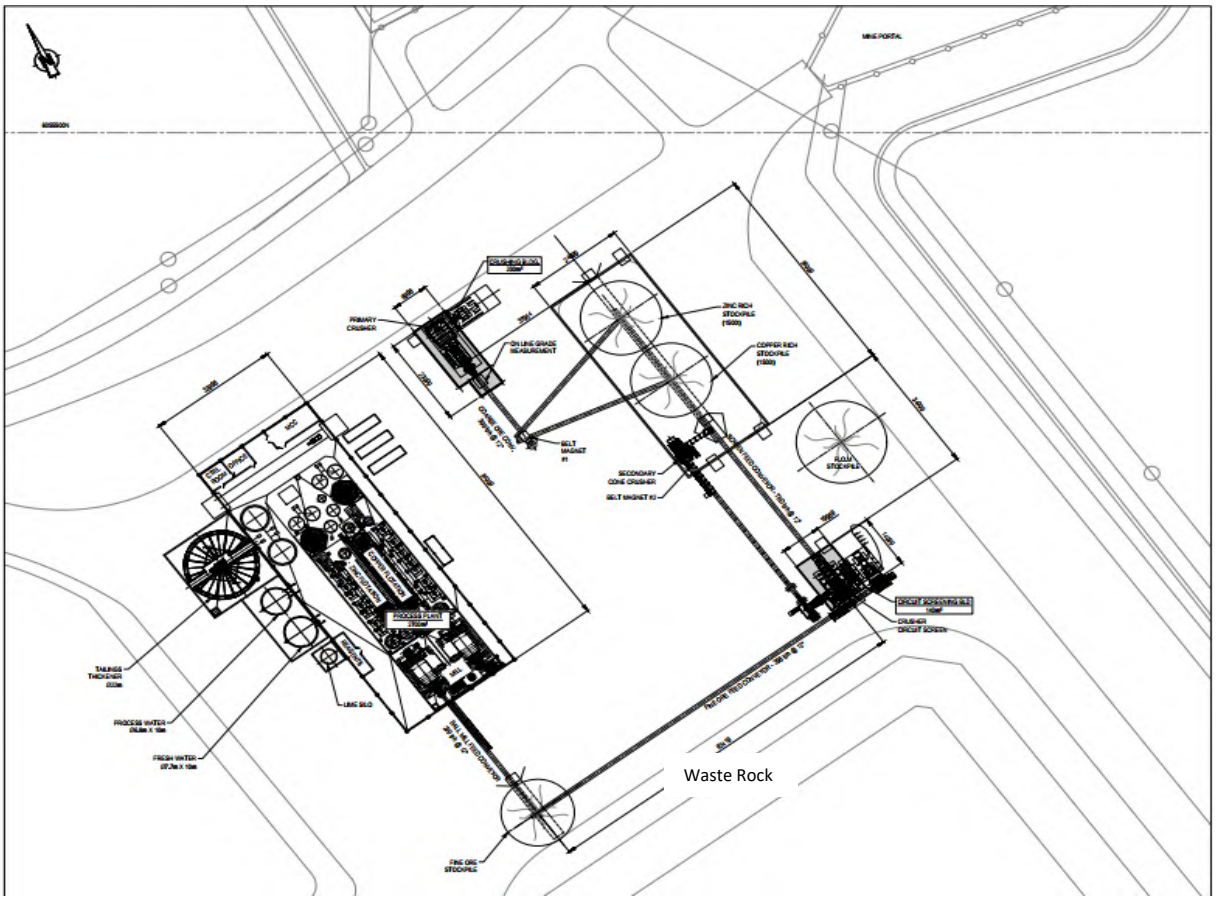
18.6 Ore And Waste Rock Management

Ore and waste rock stockpiles will be constructed in the vicinity of the underground mine portal and the process plant (Figure 18.7). The waste stockpile is a temporary facility.

It is currently assumed that the stockpile and waste dump foundations will be graded and lined with a Coletanche liner to minimize any seepage from the waste rock dump and ore stockpiles. Collected contact runoff will be diverted and/or pumped to the mine effluent collection pond for treatment.

No mined materials (waste rock and ore) will be left on surface at the end of mining operations, and liners from the stockpiles and dump will be removed as part of the closure activities.

Figure 18-6: Ore Stockpiles and Waste Rock Dump Layout



18.6.1 Ore Stockpiling

ROM ore that cannot be delivered directly to the primary crusher in load bin (located on surface) will be stored temporarily to the southeast of the crushing plant via a 2,000t capacity stockpile (Figure 18-6).

Following the primary crusher, an online analyser together with conveyors and actuated chute work will split the ore stream in two separate piles according to grade: namely copper-rich and zinc-rich. The

footprint and height of the stockpiles have been designed within the ground bearing capacity of the area and load bearing capacity of the liners.

The copper-rich and zinc-rich stockpile material will be blended through the secondary crushing process. The secondary crushed product will be conveyed to a covered fine ore stockpile ahead of the milling process.

18.6.2 Waste Rock Dump

Waste rock storage will be located to the south of the ore storage stockpiles. The footprint and height of the waste stockpile has been designed within the ground bearing capacity of the area and load bearing capacity of the liners namely: 170,000m^3 at 6.5m height.

18.7 Water Management

The site water management strategy is to divert or deflect non-contact surface runoff water away from the project site to the extent possible and to collect and treat site-influenced contact water. Where applicable, best management practices for sediment and erosion control will be utilized including the use of check dams, surface roughening, and hydro-seeding to minimize erosion potential.

The majority of water that will need to be managed at the site comes from the dewatering of underground workings, followed by surface run-off around site, especially during major precipitation events (such as 1 in 100-year storms). Additional aspects of water management around this site will include collection of interstitial water from the tailings facility and the ore and waste rock pads, collection of water from ditches around site and treatment of water for potable and process uses. The nominal water management approach on-site is illustrated by Figure 18-7.

18.7.1 Non-Contact Water Management

As the project site is relatively flat, perimeter deflection berms and diversion ditches will be constructed around the site to minimize site catchment area and to facilitate drainage of non-contact water away from the site.

18.7.2 Contact Water Management

Contact water is water that has been in contact with mining activities, mined material, and/or underground mine infiltration. Contact water will be collected, treated, tested, and reused for processing, mining operations, or at times where a large surplus is forming, to the environment.

Surface contact water collection will be undertaken by lining ore stockpiles and waste pad and by site grading and ditching the site towards a lined collection pond with transfer works (pumps and pipelines) to a containment pond. Underground contact water collection and mine dewatering will be managed through a series of pumps and sumps that transfer water to the containment pond as described in Section 16. Three ponds will be constructed on surface to manage contact and process water. Details of the ponds and effluent treatment plant are summarized below:

- **Containment Pond:** A lined containment pond with a capacity of approximately 72,000 m^3 will be constructed to contain and provide settling and polishing for surface contact and underground mine dewatering. The pond has two sides separated by a berm with an overflow spillway to allow for primary settling of suspended solids, and secondary polishing side. An

effluent treatment plant will be located at the polishing pond side. The pond will also contain an emergency overflow spillway to provide an allowance for controlled overflow if the design storm were surpassed. The containment pond can provide a residence time of seven days to settle the suspended solids. If an increased residence time is required, an option of adding baffles within the pond to prevent short circuiting can be undertaken.

- **Surface Runoff Collection Pond:** this is located in the northwest corner of the site and is approximately 5,000 m^3. During storm events, the collection pond will be pumped to the mine effluent pond or process water holding pond. If water quality within the collection pond is inadequate for process usage, the water can be diverted directly to the containment pond for treatment. This will be done via manual valves and reversing the flow of the pipe from the effluent treatment plant to the holding pond.
- **Holding Pond:** The holding pond stores treated water from the effluent treatment plant as well as runoff water from within the site. The water is intended for use underground for services and for the truck wash facility. The holding pond is designed to hold 10 days of water for mine operations (25,000 m^3).

18.7.3 Effluent Treatment Plant (ETP)

The ETP will be housed within a 25 x 25 m heated enclosure adjacent to the containment pond. The ETP is capable of treating 2,500 m^3/day which includes inflow from mine dewatering, tailings, sewage treatment plant, truck wash and the surface runoff collection pond. Treated water from the effluent treatment plant will be pumped to the holding pond for use as service water and/or into adjacent swamps as treated environmental discharge.

18.7.4 Mine Dewatering

The underground mine dewatering system is described in more detail within Section 16 of this report. The peak underground net flow at full mine development plus a factor of safety is expected to be approximately 4,700m^3/day.

18.7.5 Sewage Handling

Domestic wastewater from the mine dry, the mill and the office complex will be collected and channeled to a biological treatment system (sewage treatment plant). The treatment system will be located in a climate-controlled, fully enclosed building including oil separation, UV disinfection chamber, pump out chamber and effluent pumps. The effluent will be pumped to the truck wash facility and combined with wastewater from the truck wash prior to being pumped to the containment pond.

18.7.6 Potable Water

A fully enclosed and climate controlled potable water treatment plant will be installed to treat water from a nearby well. Treated water will be used for showers, toilets and sinks in the dry and office complexes. Water from the treatment plant is not intended for consumption and bottled drinking water will be supplied throughout the site for drinking.

Figure 18-7: Site Water Balance (Halyard, 2020)

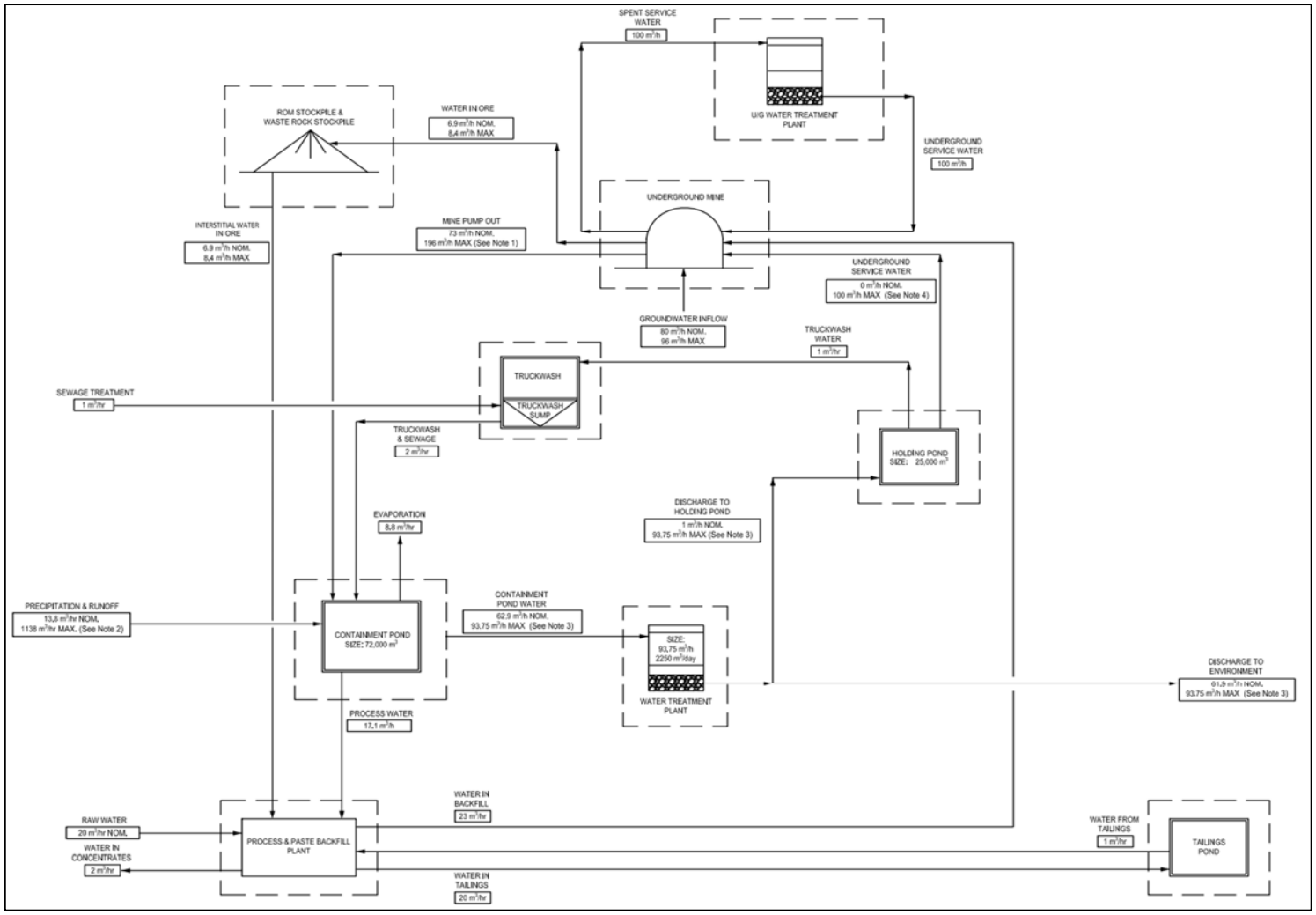


Figure 18-8: TSF Site Plan (Knight Piésold, 2020)

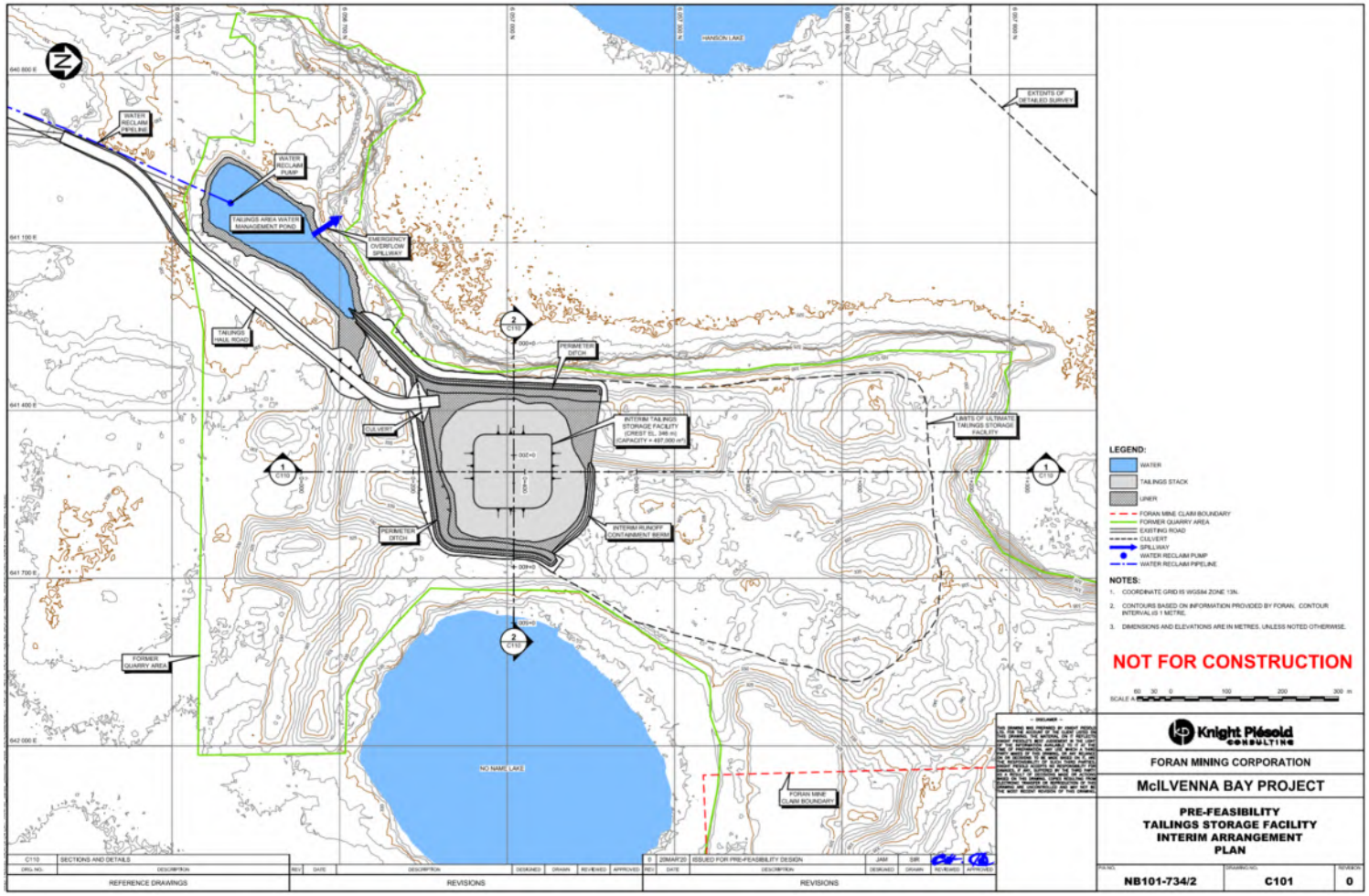
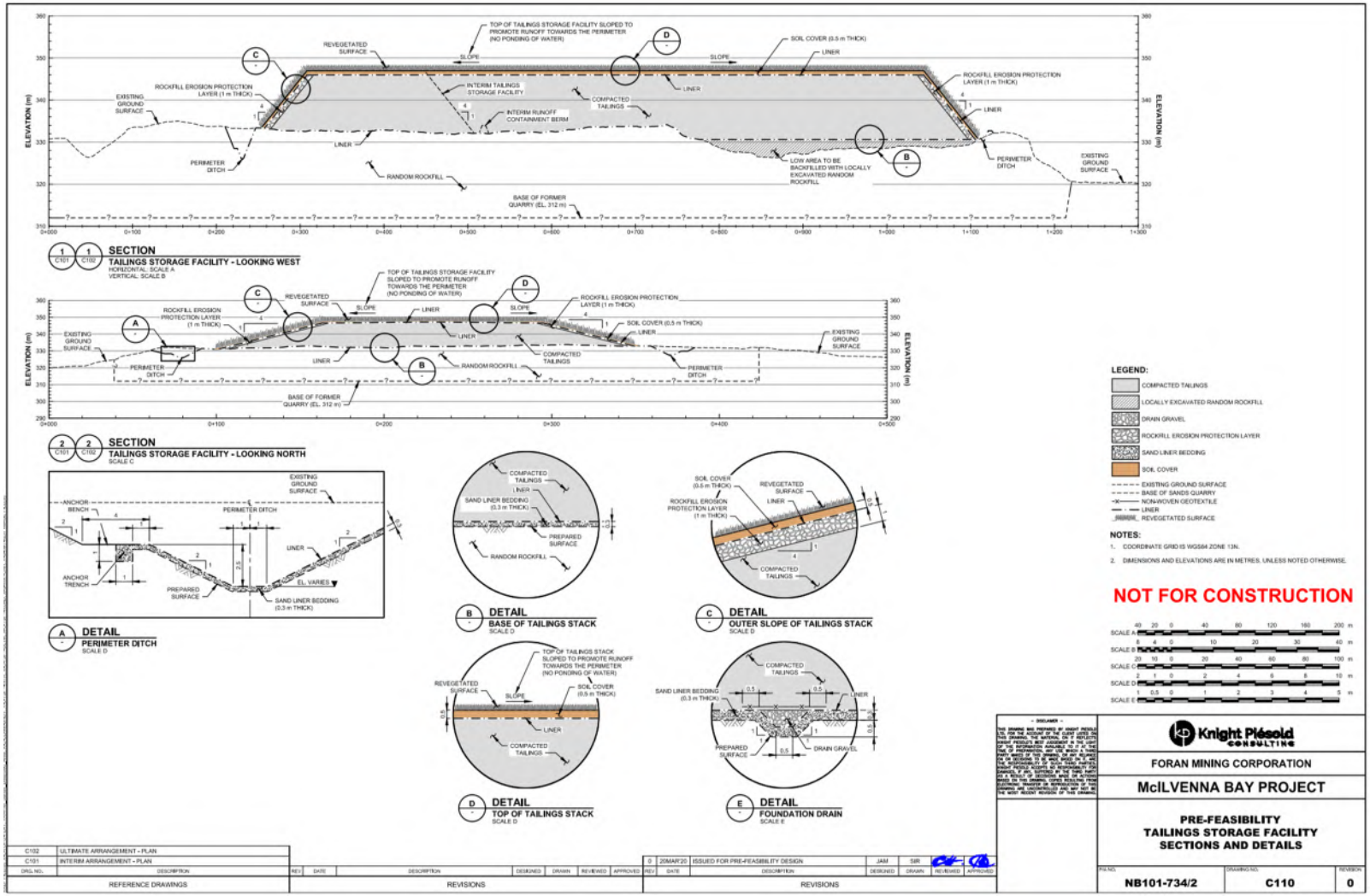


Figure 18-9: TSF Details



18.8 Tailings Storage Facilities

The current mine production plan indicates that approximately 11.4 Mt of filtered flotation tailings will be produced from the on site mineral processing plant over the life of mine. Approximately 50% of this mass will be mixed with binder and transported underground for use as paste backfill. The remaining 5.7 Mt of tailings filter cake will be placed and compacted in an engineered Tailings Storage Facility (TSF) that will be constructed on a recently closed sand quarry previously owned and operated by Preferred Sands Ltd.

An aerial view of the project area (from 2012) showing the TSF setting is provided in Figure 18-10. A plan and sections of the TSF are given in Figure 18-8 and Figure 18-9.

Figure 18-10: Aerial View of TSF Site (Looking Southwest), CanNorth 2017

In order to mitigate the risk of complications arising from generation of acid within the TSF, the sulphur content of the material to be stored on surface will be reduced within the process plant via a simple bulk sulphide flotation processing step. The small volume of sulphide flotation concentrate from this process will be bled into the underground paste backfill volume within the paste plant. Further details of the sulphur removal process are given in Section 17.0 of this report, with laboratory flotation test results for the proposed flotation process given in Section 13.0.

The main tailings related infrastructure expected to be constructed as part of the Project includes the following:

- **Tailings Storage Facility:** Half of the tailings from the process plant will be stored in a filtered TSF located to the south of Hanson Lake, approximately 1 km north of the mine site. The filtered tailings will be hauled, placed and compacted in the TSF. A perimeter ditch will be utilized to convey surface runoff from the TSF into the Tailings Area Water Management Pond.
- **Tailings Haul Road:** A 1 km long haul road will be constructed from the plant site to the TSF which will allow truck haulage of the filtered tailings.
- **Tailings Area Water Management Pond (TAWMP):** An existing topographical depression remaining from closure of a former quarry will be utilized as a surface water runoff management pond for the TSF. The pond will be lined with a geomembrane and water levels will be managed using a submersible pump and pipeline.
- **Tailings Storage Shed:** A 20 x 30 m fabric covered building will be erected adjacent to the tailings filter presses to allow for temporary storage of tailings during inclement weather and overnight production.

The TSF will be comprised of a tailings storage pad, a perimeter runoff and seepage collection ditch, and water management pond, all of which will be lined with a conventional polyethylene geomembrane to prevent seepage reporting to the environment. The tailings stack will be compacted during placement which will increase stability and minimize infiltration of precipitation. The outer slopes of the facility will be constructed at a shallow angle of 4H:1V up to an approximate height of 16 m above original ground in order to minimize effort required at closure.

19. MARKET STUDIES AND CONTRACTS

Two saleable products from McIlvenna Bay are envisaged – namely a copper flotation concentrate and a zinc flotation concentrate.

As is normal industry practice, the revenue that will be generated from the sale of these concentrates will be as the result of contracts that include pricing related to metal prices on the open markets such as the LME and COMEX.

The metal prices used for the purposes of this report are three-year trailing averages with a base date of 20 January 2020. This methodology is in line with US Securities and Exchange Commission (SEC) guidelines for disclosure of information relating to minerals and mining projects. The reader is cautioned that past performance is no guarantee of future results.

The use of three year trailing average prices will remove some of the volatility when compared to using current prices; however, in a rising market this method understates Mineral Resources and Mineral Reserves and project returns, and in a falling market it overstates Mineral Resources and Mineral Reserves and project returns. In periods of rapid metal price changes, the differences may be significant.

Alternate metal price metrics have been provided in this section to add context to the three-year trailing averages used, and the project's economic sensitivity to changing metal prices is discussed in Section 22 of this report.

The assumptions made for the purposes of this report include the following:

- The copper and zinc concentrates will be sold to separate smelting facilities. The transportation costs have been included in the Smelting and Refining costs used in this study.
- Copper and Zinc credits will be payable as metal credits as per normal industry practice. The percentage payables and refining costs have been accounted for in the Smelting and Refining costs for this project.
- Gold and Silver will be payable as metal credits as per normal industry practice. The percentage payables and refining costs have been accounted for in the Smelting and Refining costs for this project.
- There will be penalties levied against settlements for a number of potential reasons, including penalty elements, concentrate grades that are out of the agreed specification range and concentrate moistures that are outside of the agreed specification range. Potential penalties for deleterious elements have been calculated for the project and applied to financial models.
- Zinc and copper are readily traded commodities and the sales terms for them are generally standard in nature. For the purposes of this study, it is assumed that the products will be sold freely and at standard market rates.

The commodity prices used for the purposes of economic evaluation of this project are summarized in Table 19-1 and spot prices for 20 January 2020 are summarized in Table 19-2.

Table 19-1: Three Year Trailing Average for Commodity Prices

Commodity	Unit	3 Year Trailing Average
Zinc	US$/lb	1.26
Copper	US$/lb	2.82
Gold	US$/oz	1,312
Silver	US$/oz	16.30
Exchange Rate	C$/US$	1.30

Table 19-2: Spot Prices (20 Jan 2020) for Commodity Prices

Commodity	Unit	Spot Prices (20 Jan 2020)
Zinc	US$/lb	1.17
Copper	US$/lb	2.83
Gold	US$/oz	1,560
Silver	US$/oz	18.01
Exchange Rate	C$/US$	1.31

Figure 19-1 to Figure 19-4 illustrate the long term spot prices and three year trailing averages for Zn, Cu, Ag and Au.

Figure 19-1: Historical Prices for Zn

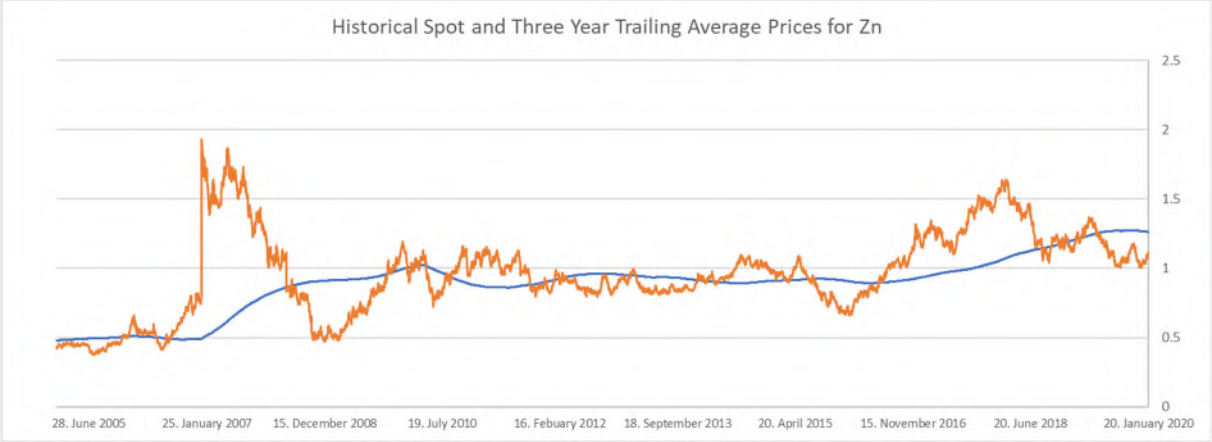


Figure 19-2: Historical Prices for Cu

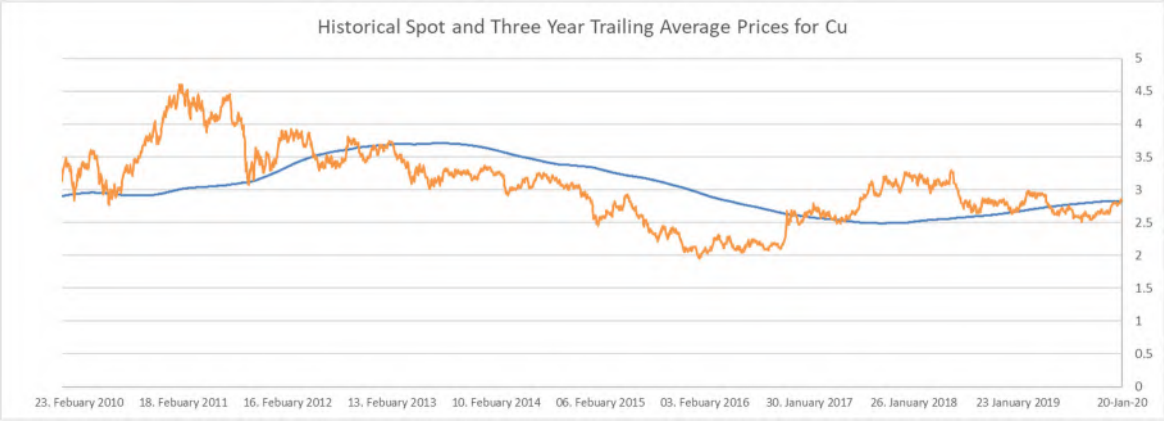


Figure 19-3: Historical Prices for Au

Figure 19-4: Historical Prices for Ag

19.1 Supply and Demand Forecasts

McIlvenna Bay is expected to produce approximately 800 million lbs. zinc and approximately 250 million lbs. copper over nine years of mine life.

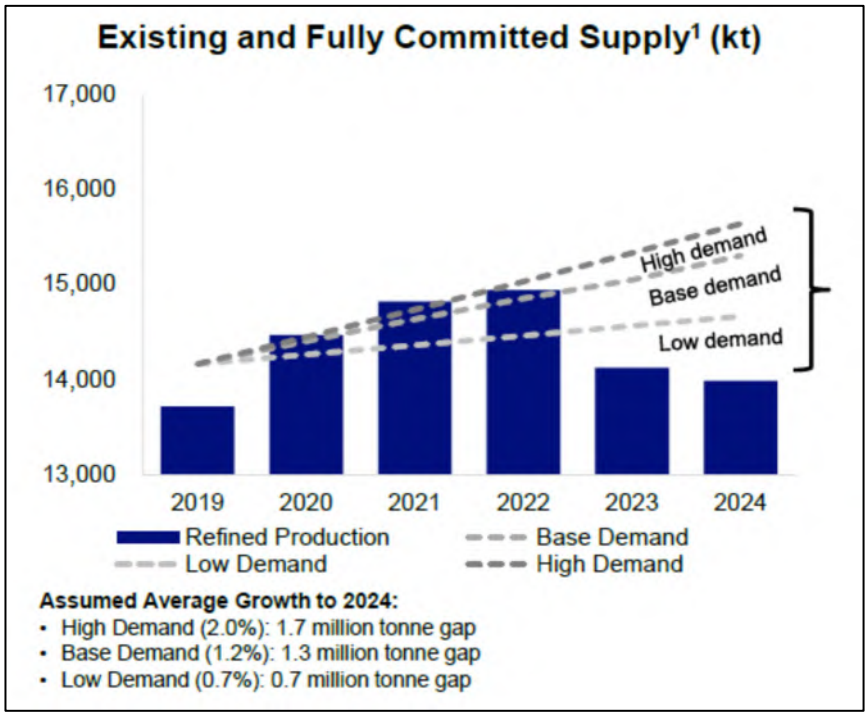
19.1.1 Zinc

Present metal inventories are well below long term averages, however Chinese zinc production has increased as zinc smelters are responding to higher treatment charges. Smelters in China were recently subject to stricter environmental standards resulting in decreased capacity and there has subsequently been a period of adjustment. RBC estimated cost support at $0.80 – $0.90/lb if demand remains soft in 2020. Inventory levels will take time to recover however, and RBC projects a support price above $1.00/lb. It is noted that a decline in auto production has coincided with the fall in zinc demand and corresponding zinc prices.

The global concentrate market is expected to be in surplus for the next few years, with mine supply outpacing smelter capacity and rebuilding concentrate inventories over this period. Miners such as Glencore, Hindustan Zinc, Korea Zinc and Nexa Resources are all either re-starting shuttered operations or bringing new projects on-line, thereby increasing zinc supply in the near term. High cost producers are under pressure and seeing closures resulting from price and treatment charges, while trade tensions are undermining various commodity prices, including zinc.

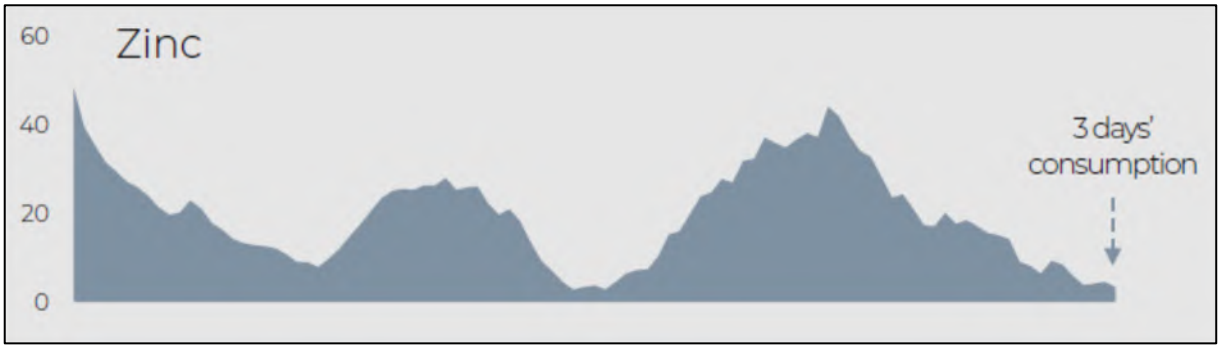
However, this near-term surplus is not expected to outpace projected demand after 2022, with companies such as Teck and Glencore forecasting a supply deficit at this time.

Figure 19- 1: Historical and Fully Committed Zn Supply



Source: Teck Base Metals Presentation, December 4, 2019

Figure 19- 2: Zinc Global Visible Inventory December 2019, days consumption



Source: Glencore Investor Update, December 3, 2019

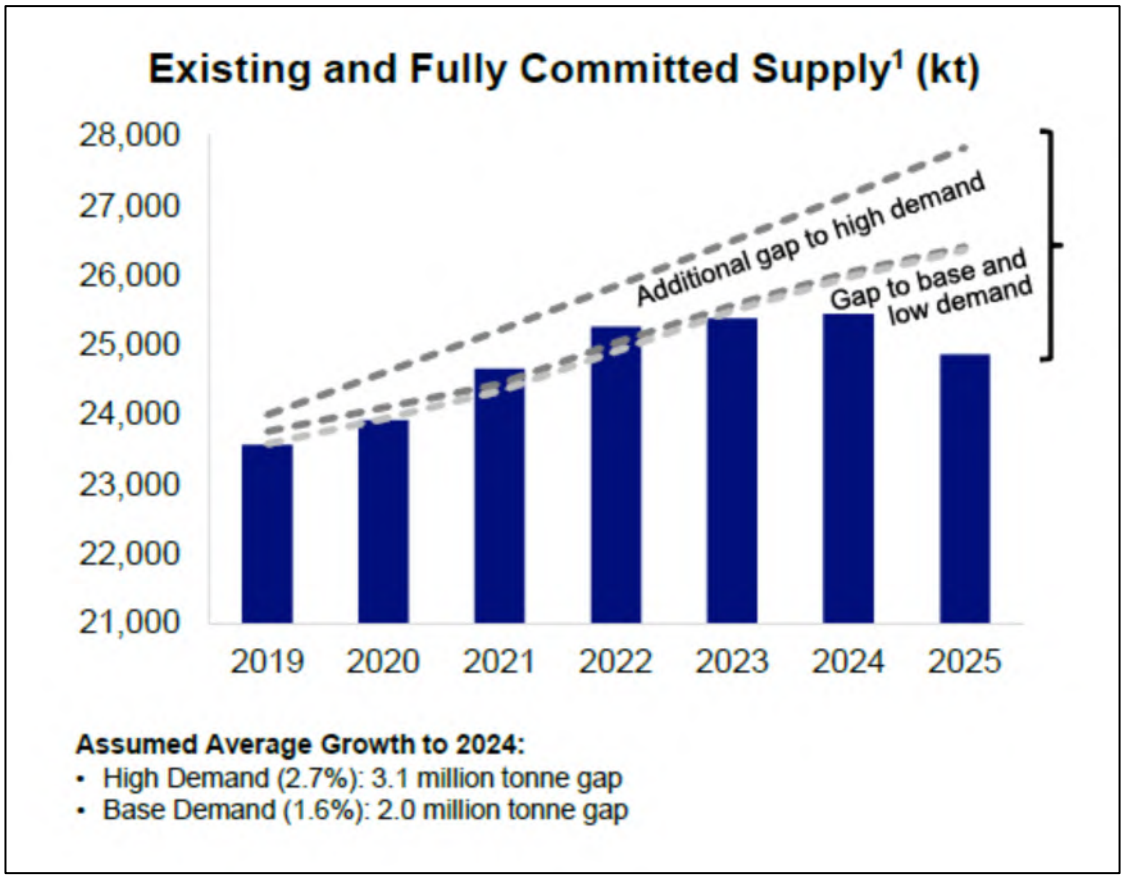
19.1.2 Copper

Market analysts at leading investment banks and research institutions predict a supply shortfall, exacerbated by increasing demand in the coming years. The supply shortfall is expected due to a current lack of investment in greenfield and brownfield projects, as well as increased production costs and declining grades in key developed mining jurisdictions such as the USA and Chile.

According to the International Copper Study Group, on a regional basis, mine production was estimated to have increased by around 4% in North America, 1.5% in Latin America and 6% in Oceania but declined by 6.5% in Asia, 2% in Africa and 2% in Europe, resulting in overall world mine production declining by about 0.4% in the first nine months of 2019 (International Copper Study Group, December 20, 2019). RBC noted in its Base Metals Outlook (December 2, 2019) that they expect copper demand growth in 2020 of 1.8% vs. 0.5% in 2019. This increase is attributed partially to improvement in the global economy and to Chinese electrical grid spending. RBC forecasts that new projects are expected to balance the market from 2021 – 2024, with deficits projected from 2025 onwards. These forecasts are subject to execution risk, as the new projects are sanctioned but not completed.

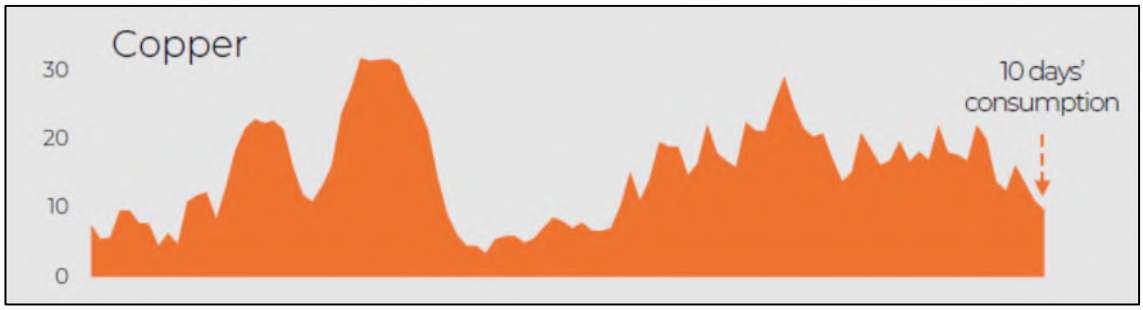
In the past 5 years, Toll Charges and Refining Charges have declined, which is largely seen as a result of smelting capacity growing faster than new mine supply.

Figure 19- 3: Copper Demand Projection



Source: Teck Base Metals Presentation, December 4, 2019

Figure 19- 4: Copper Global Visible Inventory, December 2019, days consumption



Source: Glencore Investor Update, December 3, 2019

19.1.3 Precious Metals

Gold prices rose above US$1,400/oz in July 2019 and have remained until the time of writing.

Silver has underperformed compared to gold, likely due to continued worry about global economic prospects and trade tensions.

19.2 Concentrate Sales

Foran entered into a Technical Services Agreement (TSA) with Glencore Canada Corporation (Glencore) in December 2017, and this TSA included provision for off-take agreements for the McIlvenna Bay metal concentrates. Pricing used in the PFS was based on standard commercial terms available at the time of writing.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The McIlvenna Bay Project (the Project) involves the development and operation of an underground mine, process plant and installation of associated surface amenities including a truck shop, fuel storage and distribution, a paste and filter plant, a crushing circuit, a water treatment plant, mineral waste storage (tailings and waste rock), temporary camp accommodations, administration buildings, and other miscellaneous infrastructure. The Project and deposit area are accessible via an 18km long all-weather gravel road which connects to Highway 106 (the Hanson Lake Road). It is expected this road will need minor upgrading for heavy machinery delivery and concentrate trucking; however, road condition is still to be determined.

20.2 Site Setting

The Project area lies in the Boreal Plain Ecozone on the boundary of two Ecoregions: the Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion and the Flin Flon Plain landscape area of the Churchill River Upland Ecoregion. The boundary between these two ecoregions passes through McIlvenna Bay on Hanson Lake, such that the northern part of the study area lies in the Churchill River upland, and the southern part lies in the Mid-Boreal Lowland.

The Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion is characterized by a gently undulating to nearly level landscape, featuring deciduous and coniferous forests with numerous wetlands (Acton et al. 1998). Vegetation is generally influenced by landscape and soil types. Peatlands, which comprise approximately one third of the ecoregion, typically consist of tamarack and black spruce interspersed with wet meadows (Acton et al. 1998). The Flin Flon Plain landscape area of the Churchill River Upland Ecoregion lies in eastern Saskatchewan's southernmost stretch of Precambrian Shield. Bedrock predominates in this area, with thin deposits of sandy glacial till or glaciolacustrine silt and clay (Acton et al. 1998). Vegetation of the Flin Flon Plain landscape is characterised by mixed wood forests. Black spruce is the most common tree species and is largely found in poorly drained peaty areas along with tamarack; however, black spruce is not as abundant as it is in other landscape areas of the boreal shield (Acton et al. 1998).

Extensive mining and exploration activities associated with other metal and frac sand mining projects have occurred in the Project area; therefore, the area does not represent undisturbed baseline conditions. Exploration of the McIlvenna Bay deposit began in 1988, when it was discovered by Cameco Corporation (Cameco) and Esso Minerals Canada. Cameco suspended exploration in 1991. The Project was optioned by Foran in 1998. A number of drill programs focused on McIlvenna Bay were completed between 1998 and 2000, 2011 and 2013, and a larger program in 2018.

The site of the past-producing Hanson Lake Mine, operated by Western Nuclear Mines Ltd., lies approximately 5 km north of the McIlvenna Bay deposit on the western shore of Bertrum Bay of Hanson Lake. The mine operated between 1966 and 1969 and mined a high-grade copper/zinc/lead VMS deposit. A natural basin north of the mine site was dammed for tailings containment, and runoff from the tailings area originally reported to Bertrum Bay; however, surface flows from the former site

currently enter both Bertrum Bay and Mine Bay of Hanson Lake. A number of remediation efforts have been completed for the Saskatchewan Ministry of Environment (SKMOE) regarding this abandoned mine.

A silica sand mine previously operated by Preferred Sands Ltd. is located in the immediate vicinity of the Project and is shown in operation in 2012 in Figure 20-1 below. This recently closed sand quarry is where the TSF will be constructed to reduce the amount of disturbed area at the project site for the mine development. Decommissioning and reclamation of this site began in 2015 and was completed in 2017 with activities including blasting vertical dolomite quarry faces, contouring remaining materials to achieve slopes no greater than 4H:1V, and spreading clayey topsoil and organics over the footprint to promote re-vegetation.

Figure 20-1: Preferred Sands Site in 2012 (Looking Southwest), CanNorth 2017

20.3 Environmental and Heritage Baseline Studies

Comprehensive environmental baseline studies for the McIlvenna Bay Project were completed by Canada North Environmental Services (CanNorth) in 2012. The baseline program was designed to prepare the Project for future licensing and regulatory requirements, and included collection of a full suite of environmental data including climate and meteorology; noise; surface water hydrology; water and sediment quality; plankton, benthic invertebrate, and fish communities; fish habitat; fish chemistry; fish spawning; ecosite classification; vegetation communities; wildlife communities; species at risk; and heritage resources (CanNorth 2013). Additionally, in 2018 and 2019, the hydrological and meteorological stations were re-visited to extend these datasets. A limited amount of additional

baseline environmental investigations will be required at the proposed mine site and will focus on the footprint of mine and related infrastructure and Hobbs Lake and the muskeg and creek flowing into the lake. The following sections describe the results of the baseline investigations completed to date.

20.3.1 Aquatic Resources

The Aquatic Study Area (ASA) included a number of lakes and streams, all of which ultimately flow into Hanson Lake, which drains into the Sturgeon-Weir River. The Sturgeon-Weir River then flows through several large lakes (Amisk Lake, Namew Lake, and Cumberland Lake) to join the Saskatchewan River near Cumberland House. The Saskatchewan River forms part of the Nelson River system, which ultimately discharges into Hudson Bay.

At least 15 species of fish are known to be present in the Project ASA, including lake whitefish, northern pike, walleye, white sucker, and yellow perch. None of these species are considered to be of conservation concern. Unnamed Pond is the only waterbody in the Project ASA which does not contain fish. Aquatic habitat mapping indicated a variety of habitat types are present in the McIlvenna Bay Project ASA, with suitable habitat for fish spawning, rearing, feeding, and overwintering provided by most waterbodies. Evidence of spawning (i.e. eggs) by northern pike and yellow perch was abundant throughout most of the ASA, and the Bad Carrot River was found to be an important spawning migration route/area for white sucker, walleye, northern pike, and yellow perch. The ASA did not include Hobbs Lake or its inflows and outflow. If, as currently proposed, the treated effluent is discharged into a muskeg to the south/southeast of the site, additional aquatic and hydrological investigations (surface water hydrology; water and sediment quality; plankton, benthic invertebrate, and fish communities; fish habitat; fish chemistry; fish spawning) will need to be completed in the Hobbs Lake drainage.

20.3.2 Terrestrial Resources

Terrestrial species that are considered to be Species of Conservation Concern (SOCC) are defined as federally and provincially legislated species that are identified on federal and provincial tracking lists and activity restriction guidelines. This includes:

- species listed under Schedule 1, Schedule 2, or Schedule 3 of the federal Species At Risk Act (SARA) as endangered, threatened, or special concern (SARPR 2020)
- species listed by the Committee on the Status of Endangered Wildlife in Canada (COSEWIC) as endangered, threatened, or special concern that are not listed under SARA (COSEWIC 2020)
- species listed as endangered, threatened or vulnerable in The Wildlife Act (GS 1998)
- species listed as ranking of S1, S2, or S3 or a combination of rankings (including an applicable rank modifier such as “M” for migrant) by the Saskatchewan Conservation Data Centre (SKCDC) (SKCDC 2020a, b)
- species included in the Saskatchewan Activity Restriction Guidelines for Sensitive Species (SKMOE 2017)

A number of vegetation SOCC were identified in the Project Local Study Area (LSA) and Regional Study Area (RSA), with conservation rankings ranging from S1 to S3S4 (rare to uncommon). It should be noted that none of these were federally protected species. The provincial Activity Restriction Guidelines for Sensitive Species apply to vegetation species with conservation rankings between S1 and S3, thus,

mitigation for these species may be required (SKMOE 2017). Additionally, 63 of the plant species observed within the Project LSA and RSA have documented traditional uses by the Cree and/or Dene people of northern Saskatchewan (Marles 1984; Marles et al 2008, Moerman 2010), although many of these plants are common and widely distributed in the Mid-boreal Lowland and/or Churchill River Upland ecoregions.

A total of 15 wildlife SOCC were observed during wildlife field surveys and incidentally in the Project LSA and RSA. Seven of these species are listed federally as species at risk, including common nighthawk (threatened), olive-sided flycatcher (threatened), rusty blackbird (special concern), barn swallow (threatened), horned grebe (special concern), northern leopard frog (special concern), and boreal woodland caribou (threatened). Other observed species that are not federally listed but are considered sensitive in Saskatchewan include bald eagle, Franklin's gull, osprey, American white pelican, double-crested cormorant, common tern, and Canadian toad. The McIlvenna Bay Project LSA and RSA are considered to provide a moderate to high amount of suitable habitat for the species listed above based on field data and supervised satellite image habitat classification.

20.3.3 Heritage Resources

One previously unrecorded heritage resource, GdMq-1, was discovered during the Heritage Resource Impact Assessment (HRIA) conducted in the Project LSA during the baseline program. GdMq-1 was found to be of significance due to the discovery of a quartz biface, which is a stone cutting tool or knife that has been flaked on both sides and may have been hafted to a handle (Kooyman 2000). Additionally, upon further investigation of GdMq-1, three deeply incised dolomite rock crevices were observed in a shelter bay that were large enough to conceal a person, suggesting that this area may have been used as a hunting blind or temporary shelter during the winter.

20.4 Environmental Risks/Issues

This is a preliminary examination of the potential impacts to the environment from the Project. The main impacts identified during this pre-feasibility assessment are impacts to aquatic and terrestrial habitat and biota, losses of habitat and impacts to local users. The preliminary project design appears to mitigate these impacts where necessary; however, this will be further examined in the Environmental Impact Assessment (EIA) when a detailed risk assessment will be completed. The following sections address some of the identified environmental risks.

20.4.1 Species at Risk

Additional mitigation and/or management consideration for vegetation and wildlife SOCC may be required for the Project.

As noted previously, a number of vegetation SOCC were identified in the Project LSA and RSA. The SKMOE recommends a 30m setback distance for high level disturbance (e.g., road building, blasting) for all SOCC rated S1 to S3 (SKMOE 2017); therefore, mitigation for any vegetation SOCC within the construction footprint may be required.

A number of wildlife SOCC were observed during baseline surveys in the Project LSA and RSA. Of particular note, woodland caribou (boreal population; Suggi-Amisk-Kississing management unit) occur in and near the Project LSA and RSA. The boreal population of woodland caribou in Saskatchewan is

listed on SARA Schedule 1 as Threatened (SARPR 20120) and is ranked as rare to uncommon (S3) by the SKCDC (2020a); however, there are currently no activity restriction guidelines pertinent to woodland caribou in Saskatchewan. A federal recovery strategy for the boreal population of woodland caribou has been published by Environment and Climate Change Canada (EC 2012). The long-term goal of the strategy is to achieve or maintain self-sustainability in as many of the local populations as possible, and to stabilize the remaining populations. Currently, it is thought that the population is self-sustaining in the management unit (EC 2012).

The SKMOE has established a technical committee to deliver data to inform the woodland caribou range plans under the federal recovery strategy. The McIlvenna Bay deposit is located near the northern boundary of the SK2-East range plan area and the results of these initiatives will also apply to this management unit and is expected in 2020 (EC 2012). It is expected, in this range plan, that SKMOE will create tiered Caribou Habitat Management Areas to focus caribou conservation (Tier 1 - areas primarily comprised of high and moderate habitat potential; Tier 2 - areas primarily comprised of upland ecosites with moderate habitat potential; Tier 3 – areas with the highest proportion of low habitat potential). Because of extensive historical activities in the area, including a historical mine, the operation of the frac sand facility, and associated roads (linear disturbance) over the past 20 years or more, significant surface disturbance to the terrestrial habitat already exists such that the area is likely not considered high or moderate habitat potential for woodland caribou.

Proponents have an obligation to notify the competent minister or ministers of a project if the project is likely to affect a listed wildlife species or its critical habitat under SARA. Additionally, adverse effects of the project on a listed wildlife species and its critical habitat must be identified, and, if the project is carried out, those effects must be mitigated and monitored (i.e. the development of a ""management plan"). These obligations are in addition to the requirements set out in IAA for an assessment of the environmental effects of the project, including in particular any change it may cause to a listed wildlife species, its critical habitat or the residences of individuals of that species as those terms are defined in SARA.

Finally, a number of provincially and federally listed avian SOCC were also observed within the Project LSA and RSA during baseline field surveys, thus, it is possible that disturbances to breeding migratory birds may occur during construction or operation of the Project via habitat destruction and/or disruption of breeding and/or nesting activities. Under the Migratory Birds Convention Act (MBCA) (GC 1994), destruction of birds, nests, and eggs is prohibited. In the case of SOCC, additional guidelines may apply (SKMOE 2017; EC 2009).

20.4.2 Tailings Storage Facility

The TSF will be constructed on a recently closed sand quarry previously owned and operated by Preferred Sands Ltd. Decommissioning and reclamation of this site began in 2015 and was completed in 2017 with activities including blasting vertical dolomite quarry faces, contouring remaining materials to achieve slopes no greater than 4H:1V, and spreading clayey topsoil and organics over the footprint to promote vegetation (Knight Piésold Ltd. 2020). This site was chosen as a preferred site for the TSF given that trees and natural vegetative species have not fully re-established and that the ground has been previously worked and contoured.

Approximately 5.7Mt of the 11.4Mt of tailings produced will be filtered and stored on surface in the TSF (Foran 2020), the remainder will be utilized for underground paste backfill. The tailings tonnage

reporting to the TSF will also have been subjected to a sulphide mineral reduction process, to ensure that risks associated with the generation of acid and leaching of metals within the TSF stack are well mitigated. Regardless of this, the TSF design includes a geomembrane lined pad upon which the compacted tailings will be stored. A geomembrane lined perimeter ditch, sized to manage a peak flow resulting from a 1-in-100 year rainfall event over 24 hours, will surround the pad to collect and direct runoff and seepage to flow by gravity to the TAWMP. An underdrain system comprised of several finger drains will be included above the geomembrane liner below the placed tailings. Water collected in the TAWMP will be pumped to the containment pond within the mine area for recycling and/or treatment.

20.4.3 Mine Site Water Management

Contact water is considered water that has come in contact with mining activities, mined material, and/or underground mine infiltration and will therefore require handling and treatment. All contact water at the Project site will be collected, treated, tested, and reused for mining/processing operations or discharged to the environment, as described in Section 18.7.

In Saskatchewan, the Approval to Operate a Pollutant Control Facilities (license) to operate a mine requires that as much clean surface water as reasonably possible shall be diverted away from any areas at the facilities or works where that water may become contaminated, including but not limited to waste management areas, waste rock piles and ore stockpiles. In order to minimize the Project site catchment area and facilitate the interception and diversion of clean water (non-contact water) from the site, perimeter deflection berms and diversion ditches will be constructed around the site.

Peak mine dewatering flow is anticipated to be approximately 4,700m^3 per day (corresponding to full mine development plus a safety factor) and will be pumped to a lined containment pond with a capacity of approximately 72,000m^3 constructed to contain and provide settling and polishing for surface contact and underground mine dewatering.

Four ponds will be constructed on surface to manage contact and process water:

Containment Pond: a lined pond with a capacity of approximately 72,000m^3 constructed to contain and provide settling and polishing for surface contact and underground mine dewatering. The containment pond will provide a residence time of seven days primarily to settle the suspended solids.

Holding Pond: to store treated water from the effluent treatment plant as well as runoff water from within the site. The water is intended for use underground and for the truck wash facility. The holding pond is designed to hold 10 days of water for mine operations (25,000m^3).

Tailings Area Water Management Pond: located in an existing topographical depression in the former quarry, this pond will be lined with a geomembrane and water levels will be managed using a submersible pump and pipeline. Approximately 556 m^3/day will be pumped to the water treatment plant from the TAWMP. This pond is large enough to contain the 1 in 100 year 24-hour rainfall with an additional operating allowance of 61,000 m^3 during summer operations.

Parking Area Collection Pond: 5,083 m^3 at 2.5 m depth.

The ETP will be housed within a heated enclosure adjacent to the containment pond and will be capable of treating 2,250 m^3/day which includes inflow from mine dewatering, truck wash and precipitation and surface runoff. Treated water from the ETP will be pumped to the holding pond for re-use as service water and/or discharged to the environment. The exact discharge location has not

been determined, but effluent will be discharged to a muskeg to the south/southeast of the site in a manner that meets the effluent discharge criteria identified in the Saskatchewan Mineral Industry Environmental Protection Regulations,1996 and the federal Metal and Diamond Mining Effluent Regulations (MDMER). Drainage to the south of the site is towards Hobbs Lake. This lake was not included in the ASA in the baseline investigations completed in 2012. If this pathway is chosen for effluent discharge, additional environmental investigations will need to be completed within the Hobbs Lake drainage.

20.5 Environmental Assessment Process and Permitting

20.5.1 Provincial Environmental Assessment Process

The Environmental Assessment Act requires that a proponent receives the approval of the Minister of Environment before proceeding with a development that is likely to have significant environmental implications. Since the Project is likely to be considered a development under the Act, Foran will be required to submit a technical proposal to the Environmental Assessment (EA) Branch of the SKMOE for review or screening. It is assumed that the EA Branch will determine that the Project is a development under the Act and therefore Foran will be required to prepare a Terms of Reference (TOR) and Indigenous and Public Consultation Plan which will identify the key impacts to be studied. The initial Environmental Impact Statement (EIS) submission will then be circulated by the EA Branch for a technical review by experts (including those from other provincial ministries) and, where required, to federal government reviewers.

The Environmental Assessment Act requires the Minister to give notice that an EIA is being conducted. Additionally, where the Minister's decision on a development leads to actions that have the potential to adversely impact Treaty and Aboriginal rights and the pursuit of traditional uses, the province has a duty to consult with First Nations and Métis communities in advance of the decision.

Following the completion of the technical review, the EA Branch advises the proponent and the Minister notifies the public that the EIS will be made available for review and comment by the public. Following the end of the period provided for public review and comment, the Minister will decide whether to approve or deny the project as proposed. If approved, the Minister may also require the proponent to implement additional environmental protection measures as a condition of an approval. Once ministerial approval is received, the proponent has 'cleared' the EA process and may proceed to obtain any other provincial approvals.

20.5.2 Federal Environmental Assessment Process

On August 28, 2019, the new federal Impact Assessment Act (IAA) came into force. The IAA created the new Impact Assessment Agency of Canada and repealed the Canadian Environmental Assessment Act (CEAA), 2012. Under the IAA, an EA focuses on potential adverse environmental effects that are within federal jurisdiction including fish and fish habitat, species at risk, migratory birds, changes to the environment on federal lands, including First Nation reserve lands, changes to the environment in a province other than the one where the project is taking place or outside of Canada (e.g. greenhouse gas emissions), and environmental effects arising from federally regulated project types such as nuclear, rail, ports, airports, interprovincial pipelines and offshore energy activities.

As currently envisaged, the McIlvenna Bay Project is sized below the proposed Project List criteria (new metal mine, other than a rare earth element mine or placer mine, with an ore production capacity of 5,000 t/day or more) and therefore is not likely to trigger a comprehensive assessment under the new federal IAA.

20.5.3 Environmental Permitting

The Project will require a number of approvals, permits, and authorizations during all stages following approval pursuant to the provincial EA process in accordance with various standards outlined in legislation, regulations, and guidelines. Foran will also be required to comply with any other terms and conditions issued by regulatory agencies associated with approval under the EA process. A preliminary list of permits, approvals, and authorizations that may be required for the Project is presented in Table 20.1, subject to confirmation with the responsible agencies. Permits and authorizations may also be required from other jurisdictions, such as municipalities, if any are affected.

Table 20-1: Potential Permits, Approvals, and Authorizations Anticipated to be Required

Permit, Approval, or Authorization	Issuing Agency
Provincial	
Environmental Assessment Process	Saskatchewan Environmental Assessment Branch
Approval to Construct and Operate Waterworks (Surface Water Withdrawal and Groundwater Withdrawal)	Water Security Agency
Water Rights License	Water Security Agency
Approval to Construct and Operate Drainage Works	Water Security Agency
Approval to Construct and Operate Sewage Works	Water Security Agency
Aquatic Habitat Protection Permit	Water Security Agency
Temporary Work Camp Site Permit	SKMOE
Forest Product Permit	SKMOE
Miscellaneous Use Permit	SKMOE
Construction Permit	SKMOE
Approval to Construct/Alter Highways Approach	SK Ministry of Highways and Infrastructure
Approval to Construct and Operate an Industrial Effluent Works	SKMOE
Approval to Construct and Operate a Storage Facility (Hazardous Materials and Waste Dangerous Goods)	SKMOE, Industrial Branch
Approval to Operate Pollutant Control Facilities	SKMOE
Sand and Gravel Surface Lease	SKMOE
Federal	
Fisheries Act Authorization	Department of Fisheries and Oceans Canada
Species at Risk Permit	ECCC
Designation of a Tailings Impoundment Area	ECCC
Aquatic Environmental Effects Monitoring Program	ECCC
License to Store, Manufacture, or Handle Explosives	Natural Resources Canada

20.5.4 Operating and Post Closure Requirements and Plans

Plans to address requirements for environmental mitigation and monitoring during all stages of the Project will be developed based in part on the results of the EA process. These plans may include (but are not limited to) the following:

- Environmental Protection Plan
- Environmental Contingency Plan
- Emergency Spill Response Plan
- Waste Management Plan
- Emergency Response Plan
- Tailings Management Plan
- Rehabilitation and Closure Plan

Additional plans will be developed in response to conditions of release of the EA process, as well as to address compliance monitoring requirements pursuant to applicable legislation. Monitoring may be required during construction and operation of the Project depending on the conditions of release issued by governments. Monitoring and follow-up is the responsibility of the proponent to demonstrate that the project is carried out according to the regulatory conditions and authorizations issued, to determine the accuracy of the environmental effects predicted in the EIS, and to evaluate the effectiveness of the mitigation measures.

Proponents must comply with the terms and conditions set forth in any Approval to Operate Pollutant Control Facilities issued by the SKMOE pursuant to the Environmental Management and Protection Act (EMPA), 2010 and the regulations there under. The Approval to Operate covers all areas of a pollutant control facility and can include general conditions around management of wastes, discharges and air emission controls and limits, and monitoring and reporting of effectiveness of the pollutant control facilities. Development of operating approvals and environmental monitoring plans is a collaborative process between the SKMOE and the proponent to ensure a mutual understanding of the contents of the Approval and that appropriate site-specific monitoring and controls are in place. The SKMOE regularly inspects mining and milling operations and reviews monitoring reports and other reports required to ensure that the mining company is in compliance with the applicable regulations and its operating approval.

Environmental monitoring programs must be conducted in accordance with the conditions outlined in the Approval to Operate and may include monitoring of effluent quality, surface water quality and quantity, groundwater quality and quantity, sediment quality, and aquatic biota. The frequency of each type of monitoring program will vary depending on the conditions set forth by the SKMOE. Environmental and security inspections of the mine and mill facilities must also be conducted in accordance with the Approval to Operate. The results of the monitoring programs and inspections are included in quarterly and/or annual monitoring reports which are submitted to the SKMOE in compliance with the Approval. These reports must interpret the data/information collected and discuss what, if any, impacts to the environment have occurred or may potentially occur, and what mitigation measures have and/or will be implemented to reduce or eliminate those impacts.

Effluent monitoring and an aquatic Environmental Effects Monitoring (EEM) program may be required for the Project. The MDMER are triggered when metal mines exceed an effluent flow rate of 50 m^3 per day at any time, based on the effluent deposited from all the final discharge points of the mine, and deposit a deleterious substance in any water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water. Deleterious substances are defined in the MDMER as: arsenic, copper, cyanide, lead, nickel, zinc, suspended solids, and radium 226. If the MDMER are triggered, monitoring must be initiated. Twelve months after triggering MDMER, a study design for an EEM Program must be submitted to ECCC and an EEM field study and interpretative report must be submitted to ECCC no later than 30 months after triggering the MDMER.

Prior to decommissioning of the McIlvenna Bay Project mine, mill, and ancillary facilities, a number of approvals must be obtained from the SKMOE, including:

- Approval to Decommission Pollutant Control Facilities
- Release from Decommissioning and Reclamation
- Approval of Custodial Transfer to Institutional Control

Transition phase monitoring will be completed beginning at the start of approved decommissioning and reclamation activities to determine the recovery of the impacted areas and any impacts as a result of the shutdown of operations. As part of transition monitoring, a set of site-specific performance indicators should be developed to measure progress in meeting the decommissioning and reclamation criteria. The monitoring of these environmental indicators will show whether the ecological processes that will lead to successful rehabilitation are trending in the right direction. This action will also identify and enable early intervention where trends are not positive.

During the transition phase monitoring period, Foran will be required to continue monitoring and maintaining the site at their own expense as per the requirements in the decommissioning and reclamation plan, as well as maintain an assurance fund of sufficient value to cover the cost of the remaining obligations outlined in the decommissioning and reclamation plan and any monitoring and maintenance requirements for the balance of the transitional period as well as a negotiated contingency for any unexpected occurrences.

20.5.5 Post Performance or Reclamations Bonds

A proposal for an assurance fund to ensure the completion of decommissioning and reclamation activities at the mining site must be approved by the Minister prior to approval and/or operation of a pollutant control facility, mine, or mill as per The Mineral Industry Environmental Protection Regulations, 1996. The application must be made in writing to the Minister and include a proposal for an assurance fund to ensure completion of the decommissioning and reclamation plan, including provisions for the management and administration of the assurance fund and details in respect to the release of all or portions of the assurance fund during the decommissioning and reclamation of the mining site.

The assurance fund is to be in an amount and form approved by the Minister and may consist of cash, cheques and other similar negotiable instruments, government bonds, guarantees, irrevocable letters of credit, performance bonds, security interests, or similar financial assurances as outlined in The Mineral Industry Environmental Protection Regulations, 1996.

20.5.6 Mine Closure

A conceptual plan for decommissioning and reclamation of the Project site is required as part of the EA of a mining development in Saskatchewan. As such, Foran will prepare a conceptual decommissioning and reclamation plan for inclusion in the EA, including details related to the predicted impacts of the project on the surrounding ecosystems; a description of how the impacts will be mitigated and what the residual impacts, if any, will be; a general overview on how the site will be decommissioned (i.e. buildings removed; pits filled in, etc.); and the final decommissioning objective, which will in part be based on the residual impacts of the project. Foran will also adhere to Saskatchewan's Mineral Industry Environmental Protection Regulations, 1996, Mine Regulations 2018, and Reclaimed Industrial Site Regulations; the objectives specified by SKMOE's Guidelines for Northern Mine Decommissioning and Reclamation (November 30, 2008 – Version 6 or as updated); the Reclamation Guidelines for Sand and Gravel Operators, SKMOE, May 2003; and Environment Canada's Environmental Code of Practice for Metal Mines (2009) throughout all decommissioning and reclamation activities.

During all decommissioning activities, Foran will maximize opportunities to recycle and reuse materials wherever possible. Furthermore, Foran intends to implement passive decommissioning and reclamation strategies at the site whenever possible. The intent of these strategies are to minimize, the use of engineered containment structures during closure as such structures will likely require long term care and maintenance after the post-closure custodial transfer of the property to the province. It is anticipated that after decommissioning, reclamation and transition phase monitoring are completed, the Project will be in a state that will allow for unrestricted access and for a land use similar to that which existed before the development of the site. In general, areas disturbed by operations will be reclaimed to an ecological (i.e., physical and biological) condition that will be similar to that which was observed in the area prior to disturbance.

Each of the physical works on the project site will be decommissioned and reclaimed to a standard that will ensure their long-term stability (i.e. resistance to erosion, reestablishment of drainage, etc).

Reclaimed areas will be similar to undisturbed areas in the following ways:

- soil infiltration rates and groundwater movement (pathways and rates) will be similar to nearby undisturbed areas
- vegetation communities will have similar species at densities that are comparable to the natural range of variation in nearby ecosystems
- when the disturbed areas of the site have been restored, it is expected that local animals will immigrate into and use the reclaimed areas
- mine rock piles, if present, will be shaped as much as possible to blend in with the local topography
- lake shorelines and creek banks will be reclaimed to their pre-disturbed condition
- surface water quality will be within the natural range of variation for the area

20.5.7 Closure Work Packages

Work packages are logical groupings of relatively contiguous tasks aimed at achieving a particular step in the overall site closure process. For example, the removal of a specific facility component (including

its decontamination, disassembly and delivery to a waste segregation area) would constitute a single work package. Work packages form the basis of the decommissioning cost estimates and detailed project schedule. The number and scope of individual work packages is dependent on the physical complexity of a facility and on the nature of the hazards present.

The following provides an anticipated list of the closure work packages to complete the decommissioning, reclamation and transition phase monitoring of the McIlvenna Project site.

- submission of final decommissioning and closure plan
- submission of site surface and underground mine plans
- completion of a crown pillar assessment
- pre-closure grid inspection of entire Mineral Surface Lease area
- processing facility and related infrastructure closure
- decline and raise closure
- TSF closure
- support buildings and infrastructure
- freshwater infrastructure
- power infrastructure
- exploration core
- generators, propane and fuel storage areas
- hydrocarbon impacted soils
- secondary containment areas
- administration facilities
- sewage infrastructure
- foreign material collection and disposal
- overburden replacement and scarification
- site re-vegetation (if required)
- borrow areas closure
- access road closure
- post-closure transition phase monitoring and inspections
- transfer of the property to the Province of Saskatchewan Institutional Control Registry

20.5.8 TSF Closure

The Government of Saskatchewan 2008 guidelines on mine decommissioning highlights that the primary objective of tailings management facility closure is to minimize the potential for that facility to serve as a source of contaminants. To this end, the guidelines recommend that design should seek to optimize the following objectives:

- consolidation of the tailings
- control of infiltration into the tailings
- enhancement of runoff

- control of erosion
- re-vegetation where appropriate with plants that do not have a tendency to uptake contaminants

In addition to these generalized objectives, overall public safety is considered paramount to any design. The general closure strategy for the McIlvenna Bay TSF includes:

- Compaction of tailings during placement with full time quality control as part of operational monitoring; this is expected to result in over consolidation of the tailings and minimize permeability.
- The tailings surface will be graded for drainage during operations, and again prior to final closure to promote runoff.
- Rockfill will be placed on the outer tailings slopes during operations in order to minimize erosion. Riprap lined channels will be incorporated into the final grading to control erosion in the cover layer.
- A geosynthetic liner will be placed over the final tailings surface to control infiltration into the tailings over the long-term.
- A cover layer of clayey organic soils will be place over the geosynthetic liner and seeded for re vegetation. The liner will minimize the potential for contaminant intake into the cover species.
- Shallow side slopes (4H:1V max) and relatively low height (20 m max) will aid in reducing the potential for stability over the long-term.
- Exposed geomembrane liners, pumps, pipeline, sheds, etc. will be removed, recycled, and or disposed in a permitted landfill once post closure monitoring period has been completed and water collection is no-longer required.

20.6 Considerations of Social and Community Impacts

The Project is located near Hanson Lake in east-central Saskatchewan, approximately 375 km northeast of Saskatoon, Saskatchewan. The closest communities include Creighton, Saskatchewan and Flin Flon, Manitoba, which are located approximately 65 km west-southwest of the Project. Creighton and Flin Flon have a combined population of approximately 6,200 residents, with 4,800 living in Flin Flon and the remainder in Creighton (Statistics Canada 2017a, 2017b). The economy of the area is primarily based on copper and zinc mining, while tourism and forestry are also of some importance.

HudBay operates several mines in the Flin Flon/Snow Lake area as well as a mill and zinc processing plant in Flin Flon. The 777 Mine located in Flin Flon is nearing the end of its operating life and HudBay recently announced that the mine will reach the end of its reserve life in Q2 2022 (HudBay 2020). At this time, it is unclear what the future plans are for the company or the operations in the area. This is potentially an unfortunate occurrence for the communities of Creighton and Flin Flon; however should Foran proceed with developing the Project, it presents the opportunity to recruit a trained and knowledgeable workforce from the nearby area.

The Project lies within the area traditionally occupied by the Peter Ballantyne Cree Nation (PBCN), which is made up of approximately 9,000 members living on more than 36 reserves and/or settlements. The PBCN's traditional territory encompasses roughly 52,000km^2, from the

Saskatchewan/Manitoba border west to the west end of Trade Lake, north to Reindeer Lake, and south to Sturgeon Landing (ASKI 2012). The Project is located approximately 40km southeast of the settlement of Deschambault Lake and approximately 50km west of the community of Denare Beach. Approximately 1,500 PBCN members reside in these communities (ASKI 2012).

The isolated nature of these communities creates special circumstances for PBCN members working to strengthen their local economies and personal economic well-being. Although rich in natural resources, this sparsely populated region is challenged by infrastructure, education levels, and average income when compared to the rest of the province (ASKI 2012).

Foran has conducted engagement sessions for the Project in the communities of Deschambault Lake and Denare Beach commencing in 2012. Foran also initiated a Traditional Land Use/Knowledge Inventory Study which was completed by ASKI Resource Management and Environmental Services in 2012 (ASKI 2012). During the study, members of the PBCN communities surveyed clearly articulated their continuing reliance on large game, fish, and waterfowl as well as innumerable plant species, to provide for the physical, social, and spiritual needs of the boreal forest inhabitants. While most acknowledged that the mining sector does provide the potential for employment and to create spin-off opportunities such as service businesses in catering, janitorial, trucking, security, grocery and retail supplies, such development must be tempered against the continued reliance of PBCN members on the waters, lands, and forests relied on for sustenance, livelihood, and spiritual support (ASKI 2012).

More recently, Foran has entered into discussion with the PBCN with the objectives of negotiating a Memorandum of Understanding that outlines the responsibilities and requirements of each party, focused on areas of community engagement and environment stewardship, workforce and business development, and community investment. As the Project proceeds, Foran will continue to engage the traditional users of the Project area in order to receive input on potential ways and means to minimize, to the extent possible, negative impacts on the traditional use of the lands in the vicinity of McIlvenna Bay site while at the same time maximizing the economic, social, and cultural opportunities that the Project will provide.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

Overall project capital costs are estimated to be $600 million (including contingency), with $261 million scheduled as initial (pre-production) costs and $339 million as sustaining costs. The total capital amount includes $70 million of contingency allowances.

Capital Costs have been compiled for this study using information provided within detailed estimates in each of the project disciplines, including:

- Process Plant and Infrastructure - Halyard Inc.
- Tailings and Tailings Closure - Knight Piésold Ltd.
- Mine Infrastructure - BBA
- Mining - AGP Mining Consultants Inc.

The costs given within this section reflect the current project configuration and scope of work described in this report. All costs are presented in Canadian Dollars unless otherwise stated and are based on a mixture of Q4 2019 or Q1 2020 pricing. The overall capital estimate has been prepared with a target accuracy of +/- 25% as per AACE International Level 4 study recommendations.

A summary of the overall project capital cost estimate is presented in Table 21-1 below with additional detail of the basis of estimate and estimate inclusions presented in the sub-sections that follow.

Table 21-1: Capital Estimate Summary

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Mine	72.7	273.9	346.6
Process Plant	100.6	7.2	107.8
Infrastructure	50.8	-	50.8
General and Admin	0.7	-	0.7
Tailings	5.9	11.8	17.6
Closure	-	6.4	6.4
Subtotal	230.7	299.3	530.0
Contingency	30.6	39.3	70.0
Total	**261.3**	**338.6**	**600.0**

21.1.1 Methodology - General

The capital estimate has been built up from first principles in the majority of areas, with supplier budget quotations obtained for major equipment items and database costs used for minor items. For surface works, general arrangement drawings have been used to estimate bills of quantity by area and contractor-supplied rates for buildings, structural steel, civil and earthworks supplies and installation labour.

21.1.2 Mine Capital Costs

Underground mine capital costs are estimated to be $390 million, with $83 million scheduled as initial (pre-production) capital and $307 million as sustaining capital. The total capital amount is inclusive of contingency allowances totalling almost $44 million. Mine capital costs have been broken down into mining equipment and portal costs, lateral and vertical development costs and early (pre-production) stoping costs. These are summarized in Table 21-2 below and described in more detail in the sub-sections that follow.

Table 21-2: Mine Capital Estimate

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Mining Equipment and Portal	38.97	152.34	191.31
Mine Development:			
Lateral Development	22.33	106.35	128.68
Vertical Development	2.75	15.20	17.95
Capitalized Stoping	8.66	-	8.66
Sub-total	**72.71**	**273.89**	**346.6**
Contingency	10.45	33.33	43.78
Total	**83.16**	**307.22**	**390.38**

Mining Equipment and Portal Costs

Mining equipment purchases and other fixed asset capital costs include $42.7 million of preproduction expenditure, plus a further $161 million of sustaining capital. A total LOM contingency allowance of $12.7 million is included. Costs are summarized in Table 21-3 below.

Table 21-3: Mining Capital Equipment and Portal Costs

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Mobile Equipment Purchases	9.49	12.68	22.17
Mobile Equipment Leased	13.74	97.32	111.07
Dewatering Pumps and Clarifiers	-	3.70	3.70
Ventilation and UG Bulkheads	0.92	4.24	5.16
UG Crusher, Vertical Conveyor, Rock breaker	-	10.25	10.25
Portal Box Cut	3.00	-	3.00
Electrical Distribution / Automation	9.07	14.49	23.56
Misc Infrastructure and Gen. Services	2.00	5.00	7.00
UG Paste Distribution Systems	0.75	4.65	5.40
Sub-total	**38.97**	**152.34**	**191.31**
Contingency	3.70	9.02	12.72
Total	**42.67**	**161.36**	**204.03**

Mobile equipment purchase costs and lease costs are based on mine production schedules, operating hour estimates, productivity benchmarks and OEM-supplied unit cost estimates. Fleet costs and performance are supported by OEMs and only commercially available products were considered. Details of purchased and leased fleet are given in Section 16.14.

The leased equipment cost schedule, and subsequent allocation to initial or sustaining capital budgets, is based on quoted lease-to-own financing terms with a 15% down payment and a 60-month lease period with an effective interest rate of 6.9% p.a. The equipment replacement schedule was developed based on operating hours required in the mine schedule for activities directly linked to production numbers (trucks, scoops, drills) and average annual hour consumptions for auxiliary equipment (grader, support LHD, scissor truck, etc). The replacement age for each piece of equipment was based on OEM supplier recommendations and operational benchmarking.

Underground infrastructure capital costs include estimates for mine portal construction, dewatering systems (pumps, sumps, clarifiers), ventilation (including surface-located heaters), underground power distribution, automation and communications and general mine infrastructure and underground services. These are described within Section 16.0 of this report.

Electrical cost estimates assume that a maximum of 19 levels would be equipped at any time over LOM, and that large components of electrical infrastructure such as switchgear and transformers will be re-used as the mine matures. Components that are installed in the upper levels will be relocated to lower levels as the upper levels are depleted and deeper levels are accessed.

Capital costs for the underground crushing station were developed by AGP with the assistance of Halyard for crushing equipment. Crusher station costs were developed in a similar fashion to those for the surface crushing installations, with mechanical equipment, structural steel, civil and electrical/instrumentation supply and installation cost components. The vertical conveyor costs are turnkey costs quoted by a US-based supplier with equivalent installations elsewhere worldwide.

Mine Development Capital Costs

Mine development capital expenditures include all development costs to drill, blast and excavate mine development works during the 24-month pre-production period, plus any capitalized (pre-production) labour costs. Development quantities are derived from a detailed mine development model and schedule that was developed using Deswik® mine planning software with unit rates benchmarked from similar operations within Canada.

Mine development capital costs are derived from the detailed mine development schedule and unit costs for items such as fuel, power and maintenance. The development costs are broken down into horizontal (lateral) development, vertical development, and pre-production stoping costs, as detailed in

Table 21-4, Table 21-5 and Table 21-6 below. Sustaining capital development is defined as development the is greater than 20 meters from the orebody. This includes ramps, maintenance shops, explosive magazines, level accesses, raises, and truck loading stations. Footwall drives and crosscuts as well as any ore development is considered operating expenditure.

Table 21-4: Lateral Development Costs

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Power	0.24	1.02	1.26
Fuel	0.12	0.53	0.65
Maintenance	2.34	10.11	12.45
Direct Labour	4.00	18.35	22.35
Indirect Labour	6.01	29.41	35.42
Consumables	8.70	37.16	45.86
Indirect Power	0.92	9.77	10.69
Sub-total	**22.33**	**106.35**	**128.68**
Contingency	4.47	21.27	25.74
Total	**26.80**	**127.62**	**154.42**

Table 21-5: Vertical Development Costs

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Alimak Cost	1.24	7.50	8.74
Raisebore Cost	1.51	7.70	9.21
Sub-total	**2.75**	**15.20**	**17.95**
Contingency	0.55	3.04	3.59
Total	**3.30**	**18.24**	**21.54**

Table 21-6: Capitalized Stoping (Pre-production) Costs

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Pre-production Stoping	8.66	-	8.66
Sub-total	**8.66**	**-**	**8.66**
Contingency	1.73	-	1.73
Total	**10.39**	**-**	**10.39**

21.1.3 Process Capital Costs

Process Plant capital costs are estimated to be $119.5 million, with $111.5 million scheduled as initial (pre-production) costs and $8.0 million as sustaining costs. These capital amounts include $11.7 million of contingency. The process plant capital cost was estimated using information within the following detailed design documents:

- Process design basis:
  - process design criteria
  - mechanical equipment list
  - process flow diagrams
- Site general layouts
- Equipment budget quotes from vendors
- Halyard's database for fabrication and installation rates of structural steel, mechanical and electrical equipment

A summary of plant capital cost is given in Table 21-7 below. Costs are broken down by discipline and activity (i.e. supply or installation). The initial capital totals include the costs associated with pre-production activity, whilst the sustaining capital cost estimate includes the items required to equip the process plant with standby equipment and to tie in the underground crushing system in year 3. Note that the expected life of this process plant (10-years) means that no major capital replacement (mill motors, buildings etc.) has been included in the sustaining cost schedules.

Table 21-7: Process Plant Capital Cost Breakdown

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Materials and Supply			
Mechanical Equipment (Incl Plate)	25.88	1.63	27.51
Conveyors	2.88	0.12	3.00
Buildings and Tunnels	4.83	0.14	4.97
Structural Steel	5.63	0.53	6.16
Platework	3.20	0.49	3.69
Piping and Valves	4.91	-	4.91
Electrical and Instrumentation	10.28	0.34	10.62
Freight and Transport	2.27	0.12	2.39
Installation			
Civil Construction	13.02	0.28	13.3
Mechanical Installation	3.57	0.45	4.02
Conveyors Installation	0.64	0.23	0.87
Buildings and Tunnels Construction	2.95	0.06	3.01
Structural Steel Installation	1.16	0.36	1.52
Platework Installation	0.58	0.61	1.19
Electrical Installation	4.07	0.49	4.56
Piping and Valves Installation	2.44	-	2.44
Indirects			
Engineering and Management	8.52	0.59	9.11
First Fills	0.79	-	0.79
Construction Indirects	2.96	0.76	3.72
Sub-total	**100.58**	**7.20**	**107.78**
Contingency	10.94	0.75	11.69
Total	**111.52**	**7.95**	**119.47**

21.1.4 Infrastructure Capital Costs

Infrastructure capital costs are estimated to be $57.6 million (inclusive of $7.5 million contingency), with all amounts scheduled as initial (pre-production) cost.

Surface infrastructure designs, quantities and cost estimates were developed by Halyard. Off-site hydro power (supply) costs were developed as a result of various communications with the provincial utility company, SaskPower. Estimate quantities are based on design drawings or appropriate allowances, and costs are based on supplier quotation, contractor budgetary pricing, and/or Halyard reference databases for minor items. Mechanical and electrical spares are included within the CAPEX estimates where required. A summary description of the scope of estimated costs is given in the subheadings below, and total estimated costs are presented in Table 21-8 below.

Table 21-8: Infrastructure Capital Cost Summary

Capex Item	Cost, $ Million
Site General	10.45
Waste and Ore Pads	2.62
Fuel Storage (Diesel and Propane)	0.80
Water Management	11.51
Communications	0.34
Power Supply and Distribution	13.71
Site Access	2.43
Buildings	7.31
Freight	0.92
Capitalized OPEX	0.70
Sub-total	**50.80**
Contingency	7.52
Total	**58.32**

Indirect Mining Costs

This includes civil works related to the ventilation raises – collars and fan foundations.

Site General

This includes bulk earthworks to clear, grub, strip, and grade the mine site for surface infrastructure and laydown areas. In addition, costs for drainage related items such as culverts, manholes and general surface lighting are also included in site general costs.

Waste and Ore Pads

This includes civil costs to prepare and compact the subgrade below the waste rock dump and ore stockpile locations. This includes compaction, placement of bedding layers and a liner to control seepage.

Water Management

Water management costs include civil works and mechanical/electrical costs. Civil works costs include all earthworks to excavate, shape, and line three water management ponds around site including a treated water holding pond, surface runoff collection pond, and a mine effluent treatment pond. Mechanical and electrical costs include pumps, pipelines, valves and heat tracing to transfer water on surface around site. Water treatment plant installations include a potable water treatment, mine effluent treatment, and a sewage treatment plant.

Communications

This includes costs for a communications room in the administrative complex including computers, hardware, security system, wiring, and a communications tower including foundation works.

Propane and Diesel Fuel Storage

This includes costs to construct a concrete pad and protective barriers such that rental propane tanks, vaporizers and ancillary equipment can be parked and used for mine operations (building heating and mine heating).

The diesel fuel storage depot costs include the supply and installation of two diesel storage tanks, protective jersey barriers, accompanying electrical switches, breakers, lighting, and construction of a perimeter containment berm out of earthen material.

Power Supply

This includes costs for the reactivation of an existing SaskPower distribution line as well as the construction of a new 34.5 kV SaskPower line from Pelican Narrows to the site substation. Various discussions with SaskPower and budgetary costs estimates were used to derive a capital budget for this project.

Costs include allowances to supply, install and construct a new mine services substation including surface preparation, foundation works, fencing, power distribution equipment and buried TECK distribution cabling.

Buildings

This includes the cost of buildings on surface. Buildings will be pre-fabricated or pre-engineered structures to be erected onsite and include those listed below:

- truck shop, warehouse facility, and truck wash
- administrative office complex
- mine dry, lockers, and shower facilities
- first aid trailer
- mine rescue trailer
- lunchroom
- explosives storage

Site Access

This includes costs to recover the edges of the 18km gravel mine access road where organics have grown in since the closure of the sand quarry. In addition, site access includes earthworks to construct/upgrade light vehicle access roads around site and construct an 84 stall light vehicle parking lot complete with block heaters, and an emergency-use helipad.

21.1.5 Tailings

Tailings capital costs are estimated to be $22.9 million, with $7.6 million scheduled as initial (pre-production) costs and $15.3 million as sustaining costs. These capital amounts include $5.3 million of contingency. The capital cost estimate for an on site TSF was provided by Knight Piésold and is based on the design and scope of work given in Section 18.8 of this report. Capital expenditures include:

- mobilization of earthworks and geosynthetics contractors

- all earthworks associated with the TSF including, site preparation, TSF Pad construction, TAWMP preparation, and haul road construction
- structural and concrete works for erection of a tailings storage shed and reclaim pump shed
- geosynthetics associated with lining of the TSF, perimeter ditch and TAWMP
- pipeworks associated with the TAWMP reclaim system
- instrumentation, including several monitoring wells and piezometers
- indirects such as engineering and construction QA/QC

The estimated capital costs are given in Table 21-9 and are based on a combination of supplier quotes, contractor estimates for similar scopes of work, plus experience with other similar projects. The costs tabled here exclude any capital costs associated with the removal of sulphur from, and filtering of the plant tailings material; these items are included within the process plant capital budgets. The cost estimate includes a 30% contingency to account for minor items excluded from the estimate and is reflective of the relatively early stage of estimating data.

Table 21-9: Tailings Capital Estimate (Operating Phase)

Capex Item	Cost, $ Million		
	Initial	Sustaining	Total
Mobilization	0.21	0.41	0.62
Earthworks	2.92	6.26	9.18
Structural and Concrete	0.28	-	0.28
Geosynthetics	1.82	4.04	5.86
Pipeworks and Appurtenances	0.11	-	0.11
Instrumentation	0.02	0.02	0.04
Indirects	0.51	1.03	1.54
Sub-total	**5.87**	**11.76**	**17.63**
Contingency	1.76	3.53	5.29
Total	**7.63**	**15.29**	**22.92**

21.1.6 General and Administrative Capital Costs

G&A capital costs are estimated to be $0.7 million and consist of the pre-production operating costs (years 1 and 2) that have been capitalized. These costs are detailed under the Operating Costs section below.

21.1.7 Site Closure Costs

Closure capital costs are estimated to be $8.20 million, including a contingency amount of $1.78 million, as detailed in Table 21-10 below.

Table 21-10: Closure Capital Estimate

Capex Item	Cost, $ Million
Surface Closure	1.00
Tailings Closure	5.42
Sub-total	**6.42**
Contingency	1.78
Total	**8.20**

Surface Closure

This budget includes costs for the rehabilitation of underground and surface works (excluding the tailings facility) including various demolition and removal activities, decommissioning and remediation, surface rehabilitation, engineering, project management and 5 years of post closure monitoring. These rehabilitation costs, totalling $3.5 million, are partially offset by the assumed net proceeds of salvaging process equipment and surface buildings. Conservative salvage values (7.5% of initial cost for equipment, and 10% for buildings) have been assumed, or a total of $2.5 million.

The surface closure capital allowance tabled above is therefore the surplus amount required after salvage proceeds are taken into account.

Tailings Closure

This budget includes costs for the rehabilitation of dry stack tailings facilities, and includes allowances for earthworks, TSF capping with a geosynthetic membrane, removal of pump stations, pipelines and monitoring wells, plus post-closure stability monitoring activities.

21.1.8 Contingencies

An overall project contingency of $70 million has been calculated and included within the capital cost allowances. Contingency is defined as the capital costs added to the base estimate to account for unexpected items and unforeseen activities and requirements not anticipated in the cost estimate. Where other information is not available, a 15% factor on the total direct and indirect cost estimates is used, but in many cases within the capital estimate, specific factors have been used to account for QP's knowledge of the underlying information. For example, certain quoted process plant items require only 10% contingency, whilst various civil and earthworks budgets might attract a 30% contingency due to geotechnical uncertainty etc. A summary of the allowances within each discipline is given in Table 21-11 below, split into initial and sustaining capital budgets.

Table 21-11: Contingencies Summary

Contingency Area	Cost, $ Million		
	Initial	Sustaining	Total
Mine	10.45	33.33	43.78
Mill	10.94	0.75	11.69
Infrastructure	7.52	-	7.52
Tailings	1.76	3.53	5.29
Closure	-	1.78	1.78
Total	**30.66**	**39.38**	**70.04**

21.1.9 Owner/Corporate Capital Costs

Corporate costs have not been included in these project capital estimates.

21.2 Operating Cost Estimates

Total LOM operating expenditures of $788 million are estimated for the project. These expenditures are non-capital costs that begin once the pre-production period is complete and continue through until mine closure. Pre-production operating costs totaling $10.1 million have been capitalized in the economic models for the project.

The operating cost estimate is broken down into five broad project areas:

- mine
- processing
- infrastructure
- general and administrative
- tailings

The operating costs detailed herein are based on mine and process plant design criteria and engineering and unit cost information derived from various sources. Sources for the mine site operating costs include vendor quotations, historical data or data from similar projects, Cost Mine information and empirical factors. All costs are estimated in Q1 2020 Canadian dollars and unless otherwise stated, are referred to as "$". All operating costs are estimated to within -15%/+25% accuracy unless specified otherwise.

A summary of the estimated LOM operating costs are presented in Table 21-12 below and details of the cost breakdowns are summarized in the sub-sections that follow:

Table 21-12: LOM Operating Cost Estimate

Operating Cost Item	LOM Cost, $ million	Cost, $/tonne ore
Mine	467.1	41.19
Processing	221.7	19.55
Infrastructure	32.0	2.82
General and Administrative	46.9	4.13
Tailings	20.2	1.78
Total	**787.9**	**69.48**

21.2.1 Mine Operating Costs

Mine operating costs have been built up from first principles, using detailed mine development and production schedules together with a variety of unit costs (quoted by suppliers, or using benchmark data and QP operational experience). The costs have been scheduled by period and built up by type for each basic mining activity (i.e. lateral development, transverse stoping, Avoca stoping and general mine/indirect activities). Operating costs in the first year (i.e. Prior to achievement of commercial production) total $8.7 million and have been capitalized.

A summary of overall mine operating costs is given in Table 21-13 below.

Table 21-13: Mine Operating Cost Summary

Operating Cost Item	LOM Cost, $ million	LOM Cost, $/tonne ore
Direct Power	13.6	1.20
Fuel	17.4	1.53
Maintenance	85.1	7.50
Direct Labour	100.7	8.88
Consumables	121.7	10.73
Paste	62.3	5.49
Indirect Labour	39.7	3.50
Indirect Power	12.4	1.10
UG Dewatering System	0.5	0.04
Mine Rehabilitation	7.2	0.63
Ventilation	15.2	1.34
Capitalized Opex (preproduction)	-8.7	-0.76
Total	**467.1**	**41.19**

<u>Power</u>

The estimates of direct and indirect power costs assume a unit MWh rate of $75 and use detailed mine production schedules, vendor supplied power consumption data for equipment, and estimated equipment utilization to calculate costs per production period.

Fuel

The estimate of fuel cost assumes a diesel cost of $1.10 per litre, and uses detailed mine production schedules, vendor supplied fuel consumption data for equipment, and estimated equipment utilization to calculate costs per production period.

Maintenance

Maintenance accounts for $7.50 per tonne ore averaged over LOM. Costs are derived from the calculated equipment usage data and uses detailed benchmark maintenance data for equipment to calculate costs per production period.

Labour

Direct and indirect labour accounts for $12.38 per tonne ore averaged over LOM. Labour budgets are built up using detailed mine production/development schedules, together with labour productivity estimates and project labour rates.

Mine labour complements vary by production period and is expected to peak in year 4. A summary of peak mine labour complements is given in Table 21-14 below.

Table 21-14: Mine Labour Cost Breakdown

Labour Category	No.	Peak Annual Cost, $000's			
		Gross Earnings	Fringe Benefits	Tax and Insurances	Total
Salaried (Staff)					
Mine Operations (Direct)	134	12,782	800	624	14,207
Mine Operations (Indirect)	15	1,960	113	188	2,261
Training	3	310	19	33	362
Maintenance	4	476	24	47	548
Technical Services	17	1,673	106	180	1,959
Hourly					
Mine Maintenance	12	971	23	55	1,093
Surface Operating	16	1,116	82	73	1,270
Total	**201**	**6,506**	**411**	**575**	**7,493**

Notes:
Labour costs are built up from a base salary or hourly rate with allowances made for vacation pay.
An average profit sharing allowance of 10% of base salary has been assumed.
Company benefits were calculated to include group benefits (healthcare etc..), company pension contributions and training allowances.
Taxes and Insurances were added, including statutory holiday allowances, CPP, EI and WCB contributions.
All labour costs are presumed to be fixed – i.e. independent of throughput rate.

Consumables

Consumable budgets are based on calculated costs for items such as:

- ground support consumables (rebar, mesh, plates, cable bolts etc.)
- explosives (ANFO and emulsion), detonators, B-line and boosters

- drill bits and drill steel
- backfill bulkhead material
- ventilation piping
- leaky feeder wire
- concrete, tools and PPE

Unit rates for the various consumables per metre (for lateral development) or per tonne mined (for production stoping) were applied to the detailed production schedules to arrive at consumable budgets by activity and by year.

Paste

Paste backfill cost allowances include the cost of operating the paste plant plus the cost of producing paste for distribution to transverse stopes underground. Costs are made up of fixed ($ per year) and variable ($ per tonne) costs. Fixed costs include plant maintenance and labour costs, whilst variable costs include power, Portland cement binder and miscellaneous consumables. Paste volume requirements were calculated as part of the detailed mine design schedules, and these were used together with a preliminary paste recipe selected to give a cured UCS value of 0.4-0.5 MPa. The life of mine average cost of paste production (excluding underground placement) is $5.49 per tonne processed, or $10.42 per tonne of backfill used underground.

Dewatering

An allowance of $450,000 p.a. has been made for underground dewatering system maintenance spares and flocculant/coagulant use within the underground mine water clarifiers.

Rehabilitation

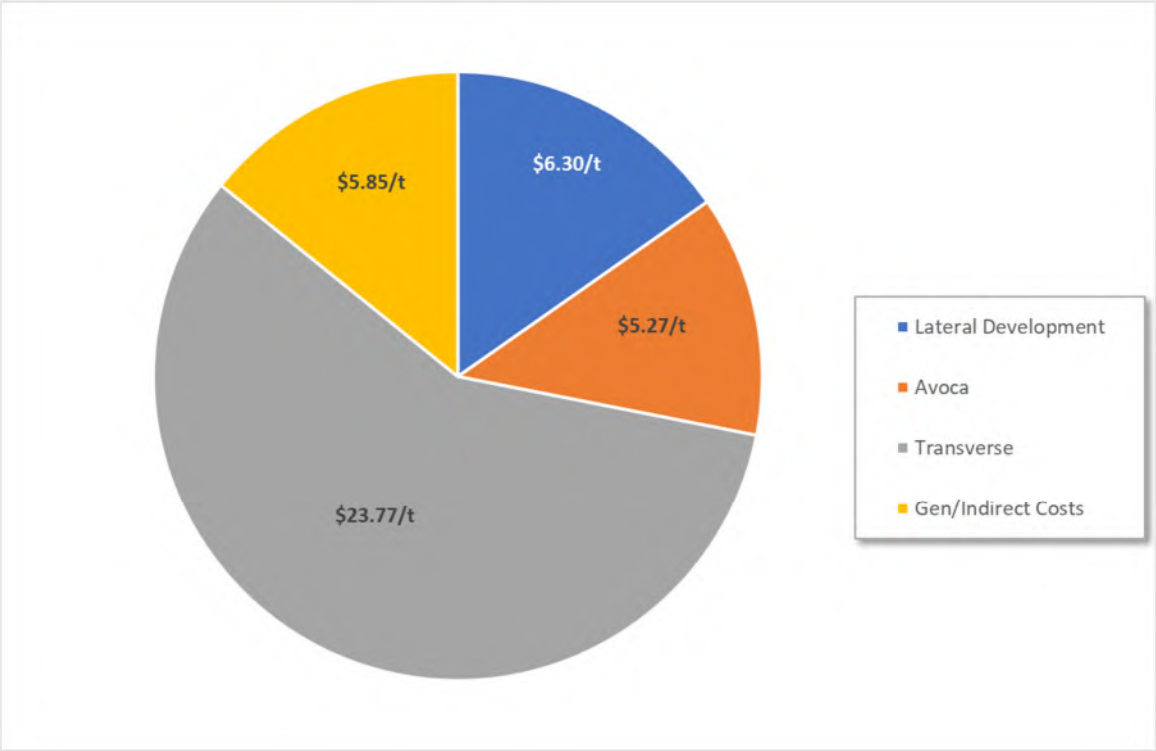
Mine rehabilitation costs were built up from first principles, using a schedule of equipment (bolter) hours, consumables use, fuel and power consumption, maintenance and labour requirements by period. These costs average approximately $500 per metre.

Ventilation

Ventilation costs were calculated by BBA, and were based on the ventilation system design, plus modelled airflow rates and fuel consumption rates for standard outside ambient temperature profiles. The costs within this category consist mainly of propane and power.

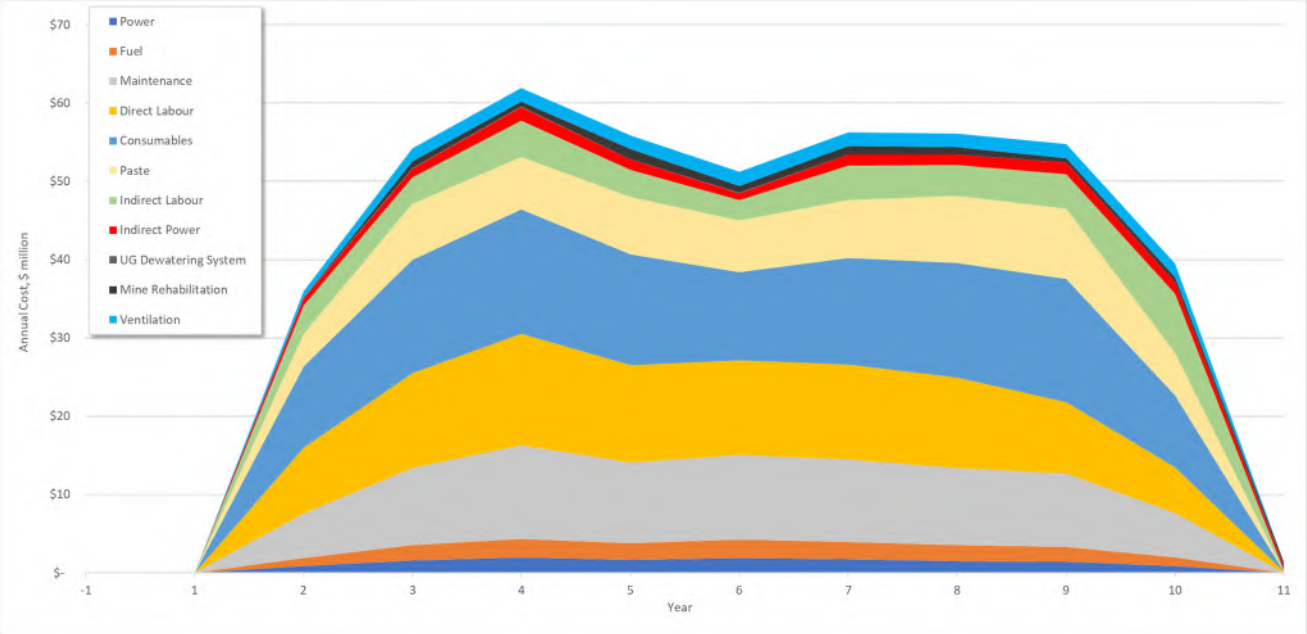
Mine operating costs (on a cost per tonne basis) are broken down by mining method in Figure 21-1 below.

Figure 21-1: Mine Opex Breakdown



Mine operating cost (OPEX) varies by year, as development metres and production tonnages vary. The annual total mine OPEX is shown for each year of mine life in Figure 21-2 below.

Figure 21-2: Mine Opex by Year



21.2.2 Process Operating Costs

Process plant operating cost budgets have been estimated from first principles, using mechanical equipment specifications for estimation of power consumption, metallurgical testwork for reagent and grinding media consumption estimates, and labour schedules and cost build-ups for process labour costs. Quotations for consumables such as reagents, steel/rubber liners and grinding media were obtained from suppliers.

Transportation of all consumable items was calculated from point of supply to site using quoted transportation costs.

All cost items are categorized as either fixed or variable costs (see Table 21-15 below).

Table 21-15: Process Plant Operating Cost Summary (LOM)

Cost Item	LOM Cost, $000's	LOM Cost, $/tonne ore
Fixed Operating Costs	78,941	6.96
Variable Operating Costs	142,797	12.59
Total	**221,738**	**19.55**

Annual fixed costs are summarized in Table 21-16 below. This budget consists mainly of labour costs but also includes components of the maintenance, supplies and contracts

Table 21-16: Process Plant Fixed Operating Costs

Cost Item	Cost, $000's p.a.
Process Labour	6,759
Tools/Equipment/Safety Supplies	198
Maintenance parts	755
Contracts – support and maintenance	350
Training (Plant Specific)	245
Power	153
Assay/General Laboratory	309
Total	**8,769**

Variable operating costs are throughput based and calculated on a per tonne basis, as summarized in Table 21-17 below.

Table 21-17: Process Plant Variable Operating Costs

Operating Cost Item	Cost, $/tonne ore
Power	3.01
Tailings	0.31
Reagents	2.69
Grinding Media	2.44
Liners (crushers/mills)	2.01
Maintenance Parts	0.81
Piping Replacement	0.07
Lubricants	0.05
Contracts	1.10
Other Variable	0.10
Total	**12.59**

The application of fixed and variable costs to the annual production schedule gives rise to a range of plant operating costs from $18.72/t ore to over $24/t ore, with an average over LOM of $19.55/t ore.

Each of the cost categories are described in more detail within the following sub-sections.

Process Labour

A total labour complement of 72 persons has been estimated for the process plant, using QP's operating experience and consultation with local labour unions. Labour unit costs are based on 2019 Manitoba labour cost survey data, with adjustments made as a result of benchmarking similar operations and after consultation with local labour union representatives. A summary of labour costs used within the process plant OPEX calculation is given in Table 21-18 below.

Table 21-18: Process Plant – Labour Cost breakdown

Labour Category	No.	Annual Cost, $000's			
		Gross Earnings	Fringe Benefits	Tax and Insurances	Total
Salaried Staff					
Operations	2	232.1	13.9	20.8	266.8
Training	2	134.2	10.0	15.7	159.9
Maintenance	4	459.8	27.6	41.7	529.1
Technical Services	2	247.5	14.5	21.7	283.7
Hourly					
Mill Ops	38	3047.0	210.3	175.0	3432.3
Maintenance	12	944.7	65.7	55.3	1065.7
Laboratory	12	902.1	64.1	54.9	1021.1
Total	**72**	**5,967.4**	**406.1**	**385.1**	**6,758.6**

Notes:
Labour costs are built up from a base salary or hourly rate with allowances made for vacation pay.
An average profit sharing allowance of 10% of base salary has been assumed.
Company benefits were calculated to include group benefits (healthcare etc.), company pension contributions and training allowances.
Taxes and Insurances were added, including statutory holiday allowances, CPP, EI and WCB contributions.
All labour costs are presumed to be fixed – i.e. independent of throughput rate.

Power

Total power costs have been calculated using estimated power draw data and unit cost estimates. Rated/installed capacity for all equipment was specified as part of the mechanical equipment listing (Section 17.0) and estimated equipment utilization (% of full load current when running, multiplied by average running time) was also calculated.

A summary of the resultant power consumption estimate by plant area is included in Table 21-19 below:

Table 21-19: Process Plant Power Costs

Plant Area	Daily kWh	kWh/t ore
Crushing	9,994	2.60
Grinding (2 stages)	88,334	22.94
Copper Flotation	15,274	3.97
Zinc Flotation	13,320	3.46
Pyrite Flotation	2,842	0.74
Tailing Dewatering (Incl. Paste Plant)	9,946	2.58
Concentrates Dewatering	3,019	0.78
Reagents	3,024	0.79
Services (water, air)	6,636	1.72
Laboratory	5,520	1.43
Total	**157,908**	**41.0**

This corresponds to an instantaneous power consumption of 7.0MW, and with an average cost of $76.00/MWh for power, this amounts to $4.38 million per annum for power. Power costs have a small fixed component, meaning that if the plant stops, then a small percentage of the above calculated power draw remains. This is typically in the 3-5% range.

Reagents

The cost of the various reagents added to the process are summarized in Table 21-20 below. Costs are calculated using average LOM consumption rates (generally in grams per tonne) scaled up from metallurgical testwork and production schedules, together with unit costs obtained from reagent suppliers budget quotations. Transportation costs for delivery to site have been included in this budget.

Table 21-20: Process Plant Reagent Costs

Reagent	Consumption, t.p.a.	Unit Cost, $/tonne	Annual Cost, $ 000's
Xanthate (PAX)	27	2,600	70.2
Danafloat 233 (Aero 3418A)	34	5,850	197.3
Polyfloat 24301 (Aero 5100)	35	10,296	364.6
Polyfloat 2979 (Aero 3894)	10	10,361	104.8
Polyfroth W31 (Dow 250)	57	5,800	328.5
Lime	1485	221	328.2
MIBC (Frother)	56	3,200	179.2
NaCN (Depressant)	15	3,185	48.9
PE26 Depressant	26	4,500	116.4
Sodium Metabisulphite	556	806	448.5
Copper Sulphate	287	3,575	1,024.8
Zinc Sulphate	47	2,145	99.9
Flocculant, Anionic	48	4,500	215.0
Transportation		120	248.8
Total	**2683**		**3,775.1**

Reagent costs are 100% variable, i.e. this budget has no fixed cost component.

Liners and Grinding Media

The cost of the various steel/rubber wear liners and steel/ceramic grinding media required for crusher and mill circuits are summarized below (Table 21-21 and

Table 21-22Table 21-22 below). Costs are calculated using average LOM consumption rates derived from metallurgical testwork and production schedules, together with unit costs obtained from vendor's budget quotations. The cost of transportation to site has been included in the budget.

Table 21-21: Process Plant: Grinding Media Costs

Item	Consumption, t.p.a.	Unit Cost, $/tonne	Annual Cost, $000's
Balls: Primary Ball Mill	857	1,560	1,337.2
Balls: Primary Ball Mill	1012	1,560	1,578.4
Media: Copper Regrind Mill	13	6,305	79.7
Media: Zinc Regrind Mill	13	6,305	79.7
Transportation	1894	190	359.9
Total			**3,435**

Table 21-22: Process Plant: Liner Costs

Item	Consumption, t.p.a.	Unit Cost, $/tonne	Annual Cost, $000's
Liners: Crusher Circuit	104	7,020	732.9
Liners: Mills, Steel	23	7,722	179.0
Liners: Mills, Rubber	110		1,874.0
Transportation	238	190	45.1
Total			**2,831**

Maintenance Costs

The estimated cost of spare parts and consumables used for routine process plant maintenance are factored from mechanical equipment supply costs. Factors vary from 3% to 8% of mechanical supply cost annually, depending on the particular plant area. Annual budgets for maintenance parts and consumables are given by area in Table 21-23 below.

Table 21-23: Process Plant Maintenance Costs

Plant Area	Annual Cost, S000's
Crushing	210.0
Grinding	396.0
Flotation	252.0
Concentrate Dewatering	48.0
Tailing Dewatering	264.0
Reagents	34.4
Services (water, air etc..)	66.0
Mobile equipment	32.0
E&I	412.0
Transportation	171.4
Total	**1,886**

It is assumed that approximately 40% of the maintenance budget is a fixed budget, i.e. independent of plant throughput rates. The remaining 60% of cost is directly related to throughput, corresponding to a cost per tonne milled.

Assay Lab Consumable Costs

This budget covers the cost of running and maintaining the assay lab (excluding labour). The budget is calculated based on number of samples assayed and includes allowances for various supplies. The total cost allowance is $309,114 per annum.

21.2.3 Infrastructure Operating Costs

Operating costs associated with various infrastructure items are all assumed to be fixed costs with a total of $32 million over LOM. The annual infrastructure budget used in the PFS amounts to a LOM average $2.82 per tonne processed and is summarized in Table 21-24 below.

Table 21-24: Infrastructure OPEX Summary

Plant Area	Total Annual Cost, $	LOM $/t ore
Power	1,098,887	
Reagents	886,584	
Lubrication	29,553	
Maintenance	478,957	
Consumables	77,400	
Personnel Transportation	949,000	
Total	**3,520,381**	**2.82**

Labour

Labour costs have been included within the G&A budget.

Power

The cost of power for infrastructure items is included in this budget. The power drawn by infrastructure areas (workshop, offices etc.) has been calculated and applied to the Saskpower unit costs.

Reagents

Reagents for the water treatment plant

Transportation

Allowance has been made for transportation of employees to/from the MineSite using contract transportation services.

21.2.4 General and Administrative Operating Costs

General and administrative (G&A) costs have been built up using labour schedules and operational benchmarking. The G&A labour cost is the largest component of this budget and is calculated using labour costs and schedules similar to those for the mine and process plant. The G&A labour force include 10 staff positions and 18 hourly positions.

The annual G&A budget used in the PFS amounts to a LOM Average $4.13 per tonne processed and is summarized in Table 21-25 below.

Table 21-25: General and Administrative Operating Costs

Plant Area	Annual Cost, S 000's	LOM $/t ore
Administrative Staff (site)	30,781	
Admin Vehicles	2,900	
Contract Services	7,200	
Other G&A Costs	3,250	
Software Licensing	1,115	
IT & Communications Costs	2,350	
Total	**47,596**	**4.13**

21.2.5 Tailings Operating Costs

Tailings operating costs total $20.2 million over LOM and consist of transportation (from paste plant to TDF) and compaction, pumping operations and indirect costs including annual inspections and 3rd party reviews.

Over the life of mine, the average operating cost is calculated to be $1.78/t milled, or $3.55/t deposited on the TDF.

21.2.6 Owner (Corporate) Operating Costs

Corporate "Head Office" costs have not been included in this operating cost estimate.

22. ECONOMIC ANALYSIS

An economic model based on various engineering estimates was prepared for the Project to estimate annual cash flows and assess the sensitivity of the Project to certain economic parameters.

The Project indicates an after-tax cash flow of $365 million, after-tax NPV (7.5%) of $147 million, and after-tax IRR of 19.2%. The Project is most sensitive to commodity prices and currency exchange rates.

22.1 Cautionary Statement

The economic analysis results are based on forward-looking information that is subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented herein. Forward-looking statements in this section include, but are not limited to, statements with respect to:

- future commodity prices (copper, zinc, gold, and silver)
- currency exchange rate fluctuations, primarily the CAD:USD rate
- estimation of Mineral Reserves
- realization of Mineral Reserve Estimates
- estimated costs and timing of capital and operating expenditures, including labour, materials, consumables, supplies and services

22.2 Assumptions

The cash flow estimate includes revenue, costs, taxes, and other factors applicable to the Project from the beginning of final design/construction to the end of mine life. Corporate obligations, financing costs, and taxes at the corporate level are excluded.

The model was prepared using mining schedules that were estimated on an annual basis. The cash flow model was based on the following assumptions.

- all costs are reported in Canadian dollars (CAN$) and referenced as '$', unless otherwise stated
- 100% equity ownership
- no provision for effects of inflation
- constant 2020 dollar analysis
- exploration costs are deemed outside of the Project
- any additional project development costs, including feasibility studies, or permitting activities, have not been included in the analysis
- annual gross revenue is determined by applying estimated metal prices to the annual recovered metal estimated for each operating year factored by the payable assumption
- constant commodity pricing was used for the economic analysis
- a constant exchange rate assumption of US$1.00 = C$1.30 (or C$1.00 = US$0.77) was used in the economic analysis

22.3 Methodology Used

The economic analysis for the Project was undertaken using Discounted Cash Flow (DCF) analysis, where the analysis is carried out by estimating the overall value of all future cash flows (incoming and outgoing), and then discounting them by an assumed cost of capital to find a present value of that cash. The Net Present Value (NPV) is the sum of all discounted future cash flows. The Internal Rate of Return (IRR) is the discount rate at which the NPV calculation returns a zero value and is a measure of the potential profitability of the project.

The model was developed using Microsoft Excel spreadsheet software.

22.4 Financial Model Parameters

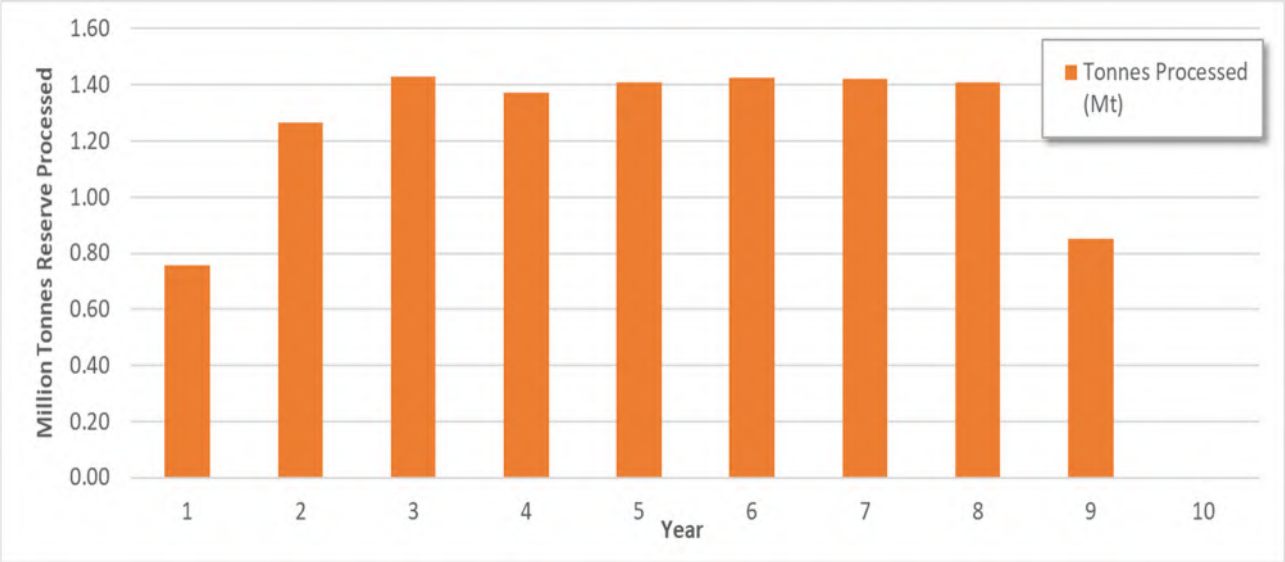
22.4.1 Mineral Resource, Mineral Reserve, and Mine Life

The mine plan presented in Section 16 of this report was used as the basis for revenue and cost estimates in the DCF, and also represents the current Probable Mineral Reserve estimate. The mine plan does not include any material from the Inferred resource category.

The Probable Mineral Reserve stated was 11.34 Mt at 4.01% Zn, 1.14% Cu, 0.54 grams per tonne ("g/t") Au and 20.97 g/t Ag.

The mine plan envisaged a nine-year LOM. The production profile in presented in Table 22-1.

Figure 22-1: LOM Production Profile (tonnes milled)



The production profile for payable Zn and Cu tonnage is presented in Figure 22-2. The production profile for payable Au and Ag is presented in Figure 22-3.

Figure 22-2: LOM Production Profile (payable Cu and Zn produced)

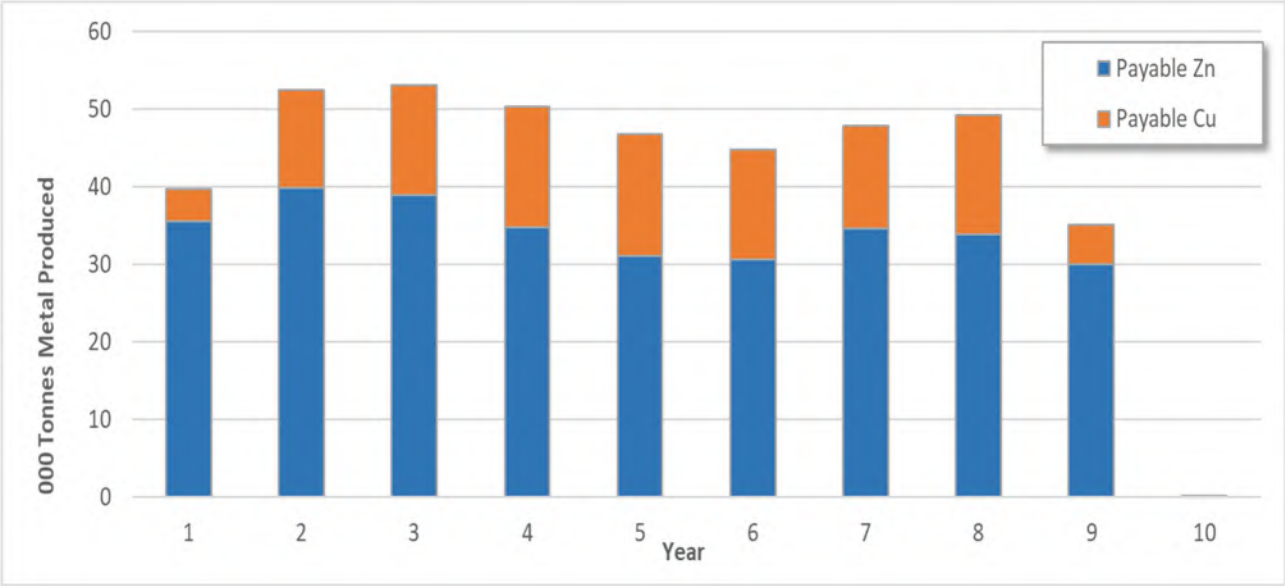
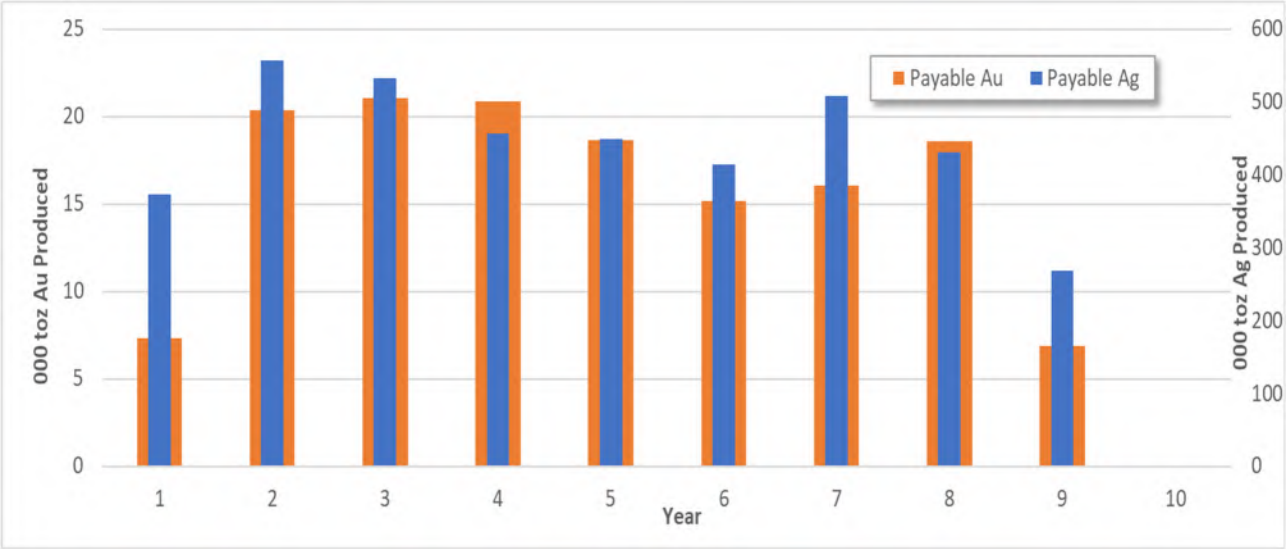


Figure 22-3: LOM Production Profile (payable Au and Ag produced)



22.4.2 Metallurgical Recoveries

The average metal recoveries that have been used for the economic evaluation are summarized in Table 22-1. Various grade vs recovery formulas were used to calculate recovery metrics on an annual basis within the cashflow model. A detailed discussion on metallurgical recoveries is available in Section 13.7 of this report.

Table 22-1: LOM Average Metallurgical Recoveries

Parameter	Units	Copper	Zinc	Gold	Silver
Massive Sulphide Recovery	%	80.9	81.8	68.8	53.7
Copper Stockwork Recovery	%	96.2	10.0	97.5	78.5
Average Blend Recovery	%	88.2	80.0	79.1	58.0

22.4.3 Freight, Smelting and Refining Terms

Typical industry rates and terms for smelting and refining have been applied when calculating the DCF. The cashflow accounts for payable metal rates, toll treatment charges, refining charges and penalties related to out of specification minor elements.

Over the life of mine, the model anticipates production of 666 dmt of zinc concentrate with an average zinc grade of 54.7% and 423 dmt of copper concentrate with an average copper grade of 26.8%. Charges and penalties attributable to the zinc and copper concentrates over LOM are summarized in Table 22-2 below.

Table 22-2: LOM Freight, Smelting/Refining charges and Smelter Penalties

Metal	Units	Zinc Concentrate	Copper Concentrate	Total
Smelter Charges	$ M	217	64	281
Smelter Penalties	$ M	9	13	22
Transportation Costs	$ M	73	69	142
Total	**$ M**	**299**	**146**	**445**

The rates used in the model do not reflect or imply that Foran has entered into any agreements relating to these contracts.

Freight of concentrate products to market has been estimated using quantities calculated from production schedules, together with quoted rates from logistics companies, assuming sale to domestic smelters.

22.4.4 Metal Prices

The metal prices used for the purposes of the economic valuation were three-year trailing averages with a base date of January 20, 2020. Metal prices used are summarized in Table 22-3 below.

Table 22-3: Metal Price Assumptions

Metal	Units	Price
Copper	US$/lb	$2.82
Zinc	US$/lb	$1.26
Gold	US$/oz	$1,312
Silver	US$/oz	$16.30

A detailed discussion relating to metal prices is available in Section 19 of this report.

22.4.5 *Operating Costs*

A summary of LOM operating costs is presented in Table 22-4 below. A detailed discussion on operating costs is available in Section 21.2 of this report.

Table 22-4: LOM Operating Cost Summary

Cost Area	LOM Average $ per tonne ore
Mine	41.19
Mill	19.55
Infrastructure	2.82
G&A	4.13
Tailings Placement	1.78
Total	**69.48**

22.4.6 *Capital Costs*

A summary of pre-production and sustaining capital costs is provided in Table 22-5 below. A detailed description of the capital costs is available in section 21.1 of this report.

Table 22-5: Capital Cost Summary

Cost Area	Pre-Production $M	Sustaining $M	Total $M
Mine	72.7	273.9	346.6
Mill	100.6	7.2	107.8
Infrastructure	50.8	0.0	50.8
G&A	0.7	0.0	0.7
Tailings Placement	5.9	11.8	17.6
Closure	0.0	6.4	6.4
Contingency	30.7	39.4	70.0
Total	**261.3**	**338.6**	**600.0**

22.4.7 *Royalties*

Two royalties have been taken into account for this economic analysis:

- **BHP Royalty:** it has been assumed that this royalty will be purchased for $1.00 Million as per the buy-out provision in the royalty agreement.
- **Copper Reef Royalty:** it has been assumed that this royalty will be paid at the agreed rate of $0.75 per tonne of ore mined.

22.4.8 *Working Capital*

No allowances for working capital beyond metals in inventory at the mine and mill have been made for the purposes of this economic analysis.

22.4.9 Taxes

A taxation model has been developed for the project based on current federal and provincial tax rates and including the present Mineral Property Tax Pools associated with the Project. It should be noted that the tax data used in the economic model are accurate at the current time, but the actual tax rates that will be levied on the Project will be influenced by many factors including, but not limited to, the timing of capital expenditures, changes in taxation policies at a federal or provincial level and any investment incentives that may be available to Foran.

22.4.10 Closure Costs and Salvage Value

Closure costs were estimated to be $6.4 million and represent closure costs for the entire project (surface infrastructure and tailings). Closure plan details are discussed in Section 20 and costs are detailed in Section 21 of this report.

The bonding strategy for the project has not been developed at this stage and costs are assumed to be incurred during the last two years of production and then to continue after project closure.

22.4.11 Financing

No project level financing has been allowed for in this economic evaluation (i.e. 100% equity ownership is assumed), however it should be noted that the economic analysis does take into account financing terms offered by the supplier for a portion of the mining fleet. The financing terms were based on a lease-to-own basis with a 15% down payment and a 60-month lease period with an effective interest rate of 6.9%.

22.4.12 Inflation

Inflation has not been included in the economic analysis.

22.5 Financial Results

A summary of the economic parameters for the project is given in Table 22-6 below.

Table 22-6 Summary of Economic Metrics for the Project

Pre-Tax NPV (7.5%) Pre-Tax IRR	$218.6 M 23.4%
Post-Tax NPV (7.5%) Post-Tax IRR	$147.1 M 19.2%
Undiscounted After-Tax Free Cash Flow (LOM, before pre-production capital deductions)	$626 M
Undiscounted After-Tax Free Cash Flow (LOM, Net of pre-production capital)	$365.4 M
Payback Period from start of processing (undiscounted, after-tax cash flow)	3.8 years
Pre-Production Capital Expenditures (rounded)	$261.3 M
LOM Sustaining Capital Expenditures (including closure)	$338.6 M
LOM Cash Cost (net of by-products) per lb. Zinc LOM Cash Cost (net of by-products) per lb. Copper	US$ 0.41 US$ 0.44

Table 22-7: Summary of Project Cash Flows

			MINE YEAR					
		TOTAL	-2	-1	1	2	3	4
REVENUE								
Gross Revenue	CAD	$ 2,337,067,688	$ -	$ -	$ 182,368,779	$ 292,377,059	$ 302,354,887	$ 296,299,566
Copper	CAD	$ 887,911,581	$ -	$ -	$ 33,347,293	$ 101,864,318	$ 114,466,245	$ 125,583,316
Zinc	CAD	$ 1,117,426,241	$ -	$ -	$ 128,600,539	$ 144,028,072	$ 140,701,177	$ 125,389,875
Gold	CAD	$ 247,163,987	$ -	$ -	$ 12,511,476	$ 34,695,788	$ 35,910,431	$ 35,634,998
Silver	CAD	$ 84,565,879	$ -	$ -	$ 7,909,470	$ 11,788,881	$ 11,277,035	$ 9,691,377
Smelter Costs	CAD	$ 451,274,120	$ -	$ -	$ 39,565,089	$ 54,910,383	$ 56,722,637	$ 54,886,814
Toll Smelt and Refining	CAD	$ 280,662,699	$ -	$ -	$ 26,656,362	$ 34,547,966	$ 35,219,548	$ 33,326,293
Concentrate Transportation	CAD	$ 142,231,688	$ -	$ -	$ 10,906,650	$ 17,000,465	$ 17,920,313	$ 17,916,866
Smelter Penalties	CAD	$ 28,379,732	$ -	$ -	$ 2,002,077	$ 3,361,951	$ 3,582,776	$ 3,643,655
Net Revenue	**CAD**	**$ 1,885,793,568**	**$ -**	**$ -**	**$ 142,803,690**	**$ 237,466,676**	**$ 245,632,251**	**$ 241,412,752**
OPEX								
Mine	CAD	$ 467,054,051	$ -	$ -	$ 35,987,879	$ 54,187,102	$ 61,881,639	$ 55,879,831
Mill	CAD	$ 221,738,405	$ -	$ -	$ 18,288,509	$ 24,725,217	$ 26,794,478	$ 26,053,027
Infrastructure	CAD	$ 31,976,791	$ -	$ -	$ 3,520,381	$ 3,520,381	$ 3,520,381	$ 3,520,381
G&A	CAD	$ 46,876,663	$ -	$ -	$ 6,020,787	$ 5,105,787	$ 4,955,787	$ 5,005,787
Tailings	CAD	$ 20,225,000	$ -	$ -	$ 1,348,344	$ 2,259,993	$ 2,553,068	$ 2,448,054
Total Opex	**CAD**	**$ 787,870,910**	**$ -**	**$ 0**	**$ 65,165,900**	**$ 89,798,479**	**$ 99,705,352**	**$ 92,907,080**
EBITA		**$ 1,097,922,658**	**$ -**	**-$ 0**	**$ 77,637,790**	**$ 147,668,197**	**$ 145,926,898**	**$ 148,505,672**
CAPEX								
Mine	CAD	$ 346,596,074	$ 7,657,627	$ 65,054,622	$ 45,424,584	$ 58,967,827	$ 37,004,493	$ 36,952,026
Mill	CAD	$ 107,771,154	$ 33,816,344	$ 66,760,388	$ -	$ 7,194,422	$ -	$ -
Infrastructure	CAD	$ 50,805,624	$ 10,077,036	$ 40,728,588	$ -	$ -	$ -	$ -
G&A	CAD	$ 719,579	$ -	$ 719,579	$ -	$ -	$ -	$ -
Tailings	CAD	$ 17,628,000	$ -	$ 5,868,000	$ -	$ 5,880,000	$ 5,880,000	$ -
Closure	CAD	$ 6,419,000	$ -	$ -	$ -	$ -	$ -	$ -
Contingency	CAD	$ 70,042,811	$ 3,999,628	$ 26,661,895	$ 5,796,514	$ 10,641,291	$ 6,154,306	$ 4,327,484
Total Capex		**$ 599,982,242**	**$ 55,550,635**	**$ 205,793,072**	**$ 51,221,098**	**$ 82,683,540**	**$ 49,038,798**	**$ 41,279,510**
Royalty								
BHP	CAD	$ 1,000,000			$ 1,000,000			
Copper Reef	CAD	$ 8,504,432		$ 67,383	$ 499,584	$ 950,307	$ 1,073,542	$ 1,029,385
Total Royalty		**$ 9,504,432**	**$ -**	**$ 67,383**	**$ 1,499,584**	**$ 950,307**	**$ 1,073,542**	**$ 1,029,385**
Free Cash Flow (Pre Tax)	CAD	**$ 488,435,983**	**-$ 55,550,635**	**-$ 205,860,455**	**$ 24,917,108**	**$ 64,034,350**	**$ 95,814,558**	**$ 106,196,777**

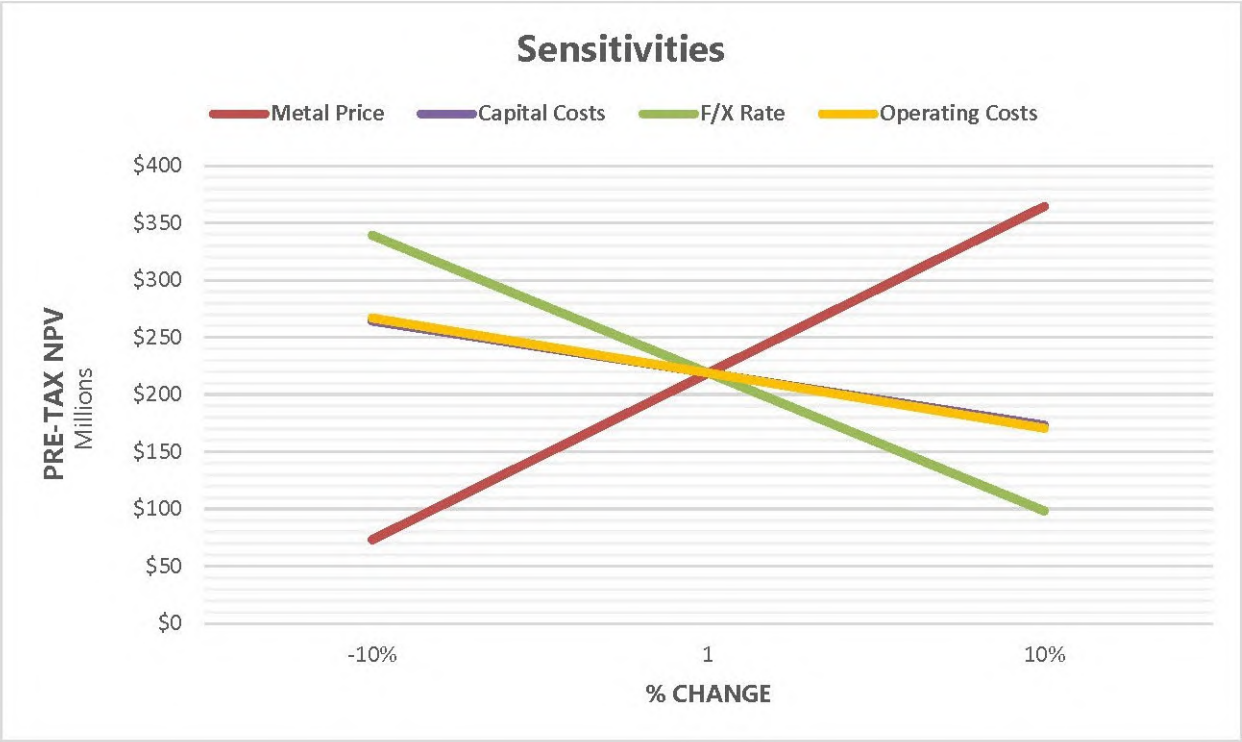
(continued overleaf…)

		TOTAL	MINE YEAR 5	6	7	8	9	10
REVENUE								
Gross Revenue	CAD	**$ 2,337,067,688**	$ 280,174,657	$ 259,549,028	$ 270,012,530	$ 286,996,876	$ 166,777,362	$ 156,943
Copper	CAD	**$ 887,911,581**	$ 126,802,578	$ 114,314,403	$ 106,811,222	$ 123,889,439	$ 40,754,259	$ 78,508
Zinc	CAD	**$ 1,117,426,241**	$ 112,101,017	$ 110,566,116	$ 125,019,816	$ 122,308,387	$ 108,662,525	$ 48,718
Gold	CAD	**$ 247,163,987**	$ 31,759,128	$ 25,891,491	$ 27,400,076	$ 31,657,563	$ 11,678,241	$ 24,794
Silver	CAD	**$ 84,565,879**	$ 9,511,934	$ 8,777,018	$ 10,781,417	$ 9,141,487	$ 5,682,338	$ 4,923
Smelter Costs	CAD	**$ 451,274,120**	$ 51,068,992	$ 48,881,025	$ 52,068,899	$ 55,985,232	$ 37,155,804	$ 29,246
Toll Smelt and Refining	CAD	**$ 280,662,699**	$ 30,722,614	$ 29,670,518	$ 32,186,079	$ 33,873,642	$ 24,442,856	$ 16,820
Concentrate Transportation	CAD	**$ 142,231,688**	$ 16,886,104	$ 15,962,474	$ 16,560,874	$ 18,371,556	$ 10,696,120	$ 10,267
Smelter Penalties	CAD	**$ 28,379,732**	$ 3,460,273	$ 3,248,033	$ 3,321,946	$ 3,740,034	$ 2,016,828	$ 2,159
Net Revenue	**CAD**	**$ 1,885,793,568**	**$ 229,105,665**	**$ 210,668,003**	**$ 217,943,631**	**$ 231,011,644**	**$ 129,621,558**	**$ 127,696**
OPEX								
Mine	CAD	**$ 467,054,051**	$ 51,180,523	$ 56,257,061	$ 56,099,628	$ 54,747,568	$ 39,515,189	$ 1,317,631
Mill	CAD	**$ 221,738,405**	$ 26,496,146	$ 26,719,354	$ 26,644,986	$ 26,503,998	$ 19,480,998	$ 31,693
Infrastructure	CAD	**$ 31,976,791**	$ 3,520,381	$ 3,520,381	$ 3,520,381	$ 3,520,381	$ 3,520,381	$ 293,365
G&A	CAD	**$ 46,876,663**	$ 5,045,787	$ 5,095,787	$ 5,045,787	$ 5,095,787	$ 4,995,787	$ 509,579
Tailings	CAD	**$ 20,225,000**	$ 2,510,814	$ 2,542,428	$ 2,531,895	$ 2,511,926	$ 1,517,240	$ 1,239
Total Opex	**CAD**	**$ 787,870,910**	**$ 88,753,651**	**$ 94,135,011**	**$ 93,842,676**	**$ 92,379,660**	**$ 69,029,595**	**$ 2,153,506**
EBITA		**$ 1,097,922,658**	**$ 140,352,014**	**$ 116,532,993**	**$ 124,100,955**	**$ 138,631,984**	**$ 60,591,963**	**-$ 2,025,810**
CAPEX								
Mine	CAD	**$ 346,596,074**	$ 29,379,350	$ 19,473,226	$ 20,551,386	$ 14,412,591	$ 4,927,337	$ 6,791,006
Mill	CAD	**$ 107,771,154**	$ -	$ -	$ -	$ -	$ -	$ -
Infrastructure	CAD	**$ 50,805,624**	$ -	$ -	$ -	$ -	$ -	$ -
G&A	CAD	**$ 719,579**	$ -	$ -	$ -	$ -	$ -	$ -
Tailings	CAD	**$ 17,628,000**	$ -	$ -	$ -	$ -	$ -	$ -
Closure	CAD	**$ 6,419,000**	$ -	$ -	$ -	$ 1,250,000	$ 2,200,000	$ 2,969,000
Contingency	CAD	**$ 70,042,811**	$ 3,798,631	$ 2,099,591	$ 2,865,503	$ 2,217,268	$ 695,000	$ 785,700
Total Capex		**$ 599,982,242**	**$ 33,177,981**	**$ 21,572,817**	**$ 23,416,889**	**$ 17,879,858**	**$ 7,822,337**	**$ 10,545,706**
Royalty								
BHP	CAD	**$ 1,000,000**						
Copper Reef	CAD	**$ 8,504,432**	$ 1,055,775	$ 1,069,068	$ 1,064,639	$ 1,056,243	$ 637,986	$ 521
Total Royalty		**$ 9,504,432**	**$ 1,055,775**	**$ 1,069,068**	**$ 1,064,639**	**$ 1,056,243**	**$ 637,986**	**$ 521**
Free Cash Flow (Pre Tax)	CAD	**$ 488,435,983**	**$ 106,118,258**	**$ 93,891,108**	**$ 99,619,427**	**$ 119,695,884**	**$ 52,131,640**	**-$ 12,572,037**

22.6 Sensitivity Analysis

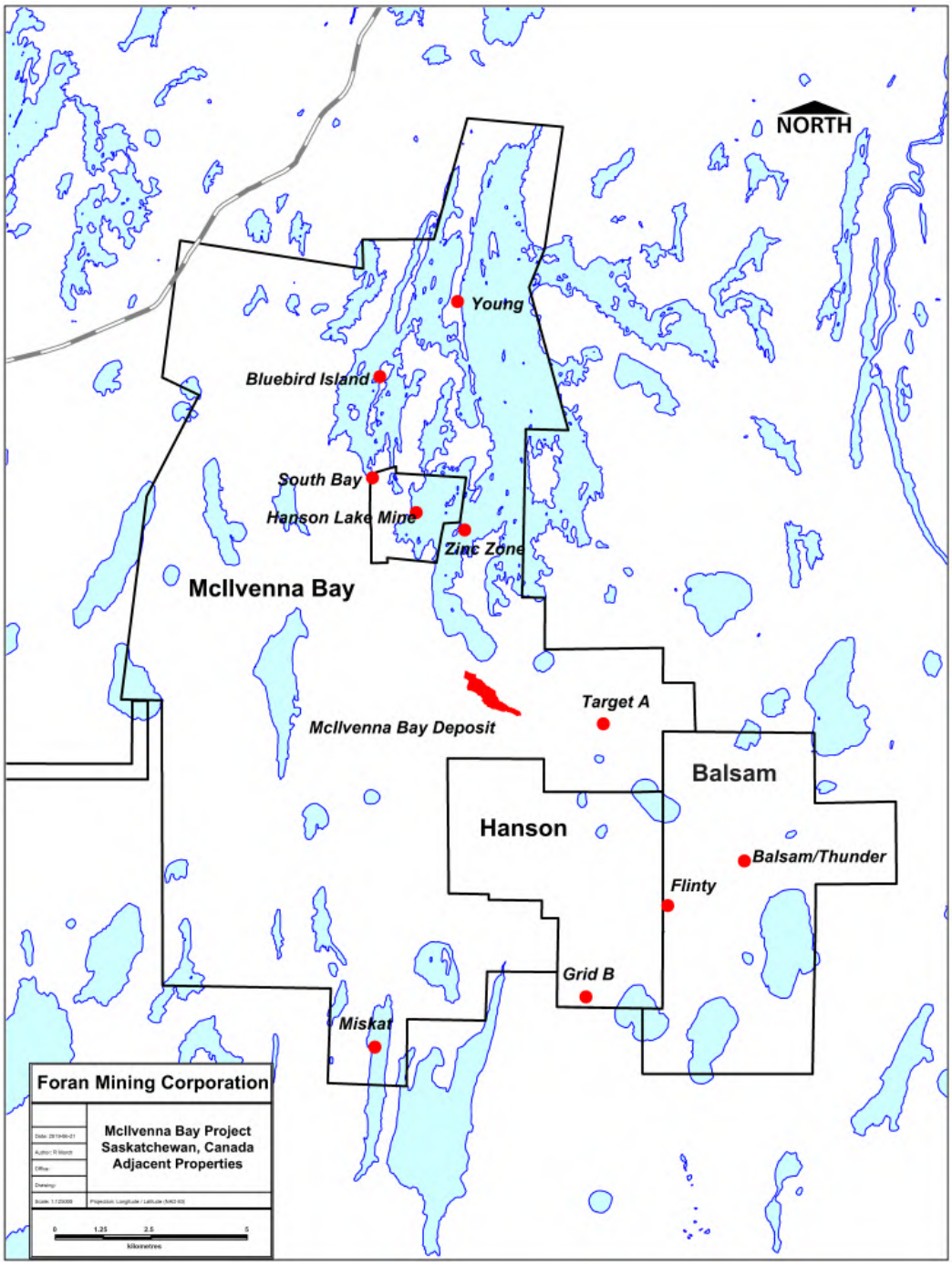
A DCF calculation was calculated for the Project, based on the various cost and revenue inputs. A 10% increase and decrease for metal prices, exchange rate, capital costs and operating costs was calculated, and the results are illustrated in Figure 22-4.

Figure 22-4: Sensitivity Analysis of NPV to Major Inputs



In January 2020, Copper Reef Mining announced the commencement of a 4,300 metre diamond drill program, including a small 5-hole program at the Hanson Lake property. At the time of writing, no results have been reported.

Figure 23-2: Adjacent Properties



23.2 Fracking Sand

Preferred Sands was the operator of a past producing silica sand (frac sand) quarry located immediately east of McIlvenna Bay. The quarry was operated as an open pit mine where up to 25m of dolomite cap rock was blasted and removed, accessing three to five metres of silica sand (Figure 23-3). The sand was mined, washed and sorted into various size factions and marketed throughout western Canada and the US where it was used as a proppant for hydraulic fracturing ("fracking"). In 2014, Preferred Sands shutdown operations and the site was subsequently re-claimed by pushing the waste rock and remaining sand back into the pits and re-contouring the landscape.

Figure 23-3: View of Preferred Sands Quarry, July 2011

The sand quarry leases overlie Foran's mineral tenure in the area (first claims originally acquired in 1986) and were held by Preferred Sands and its predecessor companies since 1998 with additional leases in the area acquired in 2006. When the new management group took over operations for Foran in 2011, it was brought to the attention of the Saskatchewan Government that a potential conflict existed due to the granting of overlapping tenure. In order to protect the McIlvenna Bay deposit area from further conflict, a Crown Reserve was established by the Government over the deposit to remove this area from further staking. Subsequently, the regulations around sand quarry staking in the province were amended to remove areas of existing mineral tenure from availability for quarry staking. When Preferred Sands shutdown operations, Foran acquired five existing quarry leases from them that were in the vicinity of the McIlvenna Bay deposit to ensure that there was no further direct conflict with the deposit.

Hanson Lake Sand Corp. (now Strong Pine Energy Services, or "Strong Pine") also has sand quarry dispositions in the McIlvenna Bay area that were staked prior to the changes in the staking regulations discussed above. No production has taken place for these dispositions. In late 2019 Strong Pine presented an Environmental Impact Statement (EIS) and proposals to local communities regarding the operation of a new fracking sand mine in the vicinity of the McIlvenna Bay deposit (between the existing access road and Guyander Lake). However, the EIS and proposals do not disclose the economic parameters necessary to outline the financial viability of such an operation. Should Strong Pine bring this mine into operation, the main impact to the McIlvenna Bay project would be increased traffic on the mine access road. It is assumed at this point that an agreement would be reached between Strong Pine and Foran to safely manage this increase.

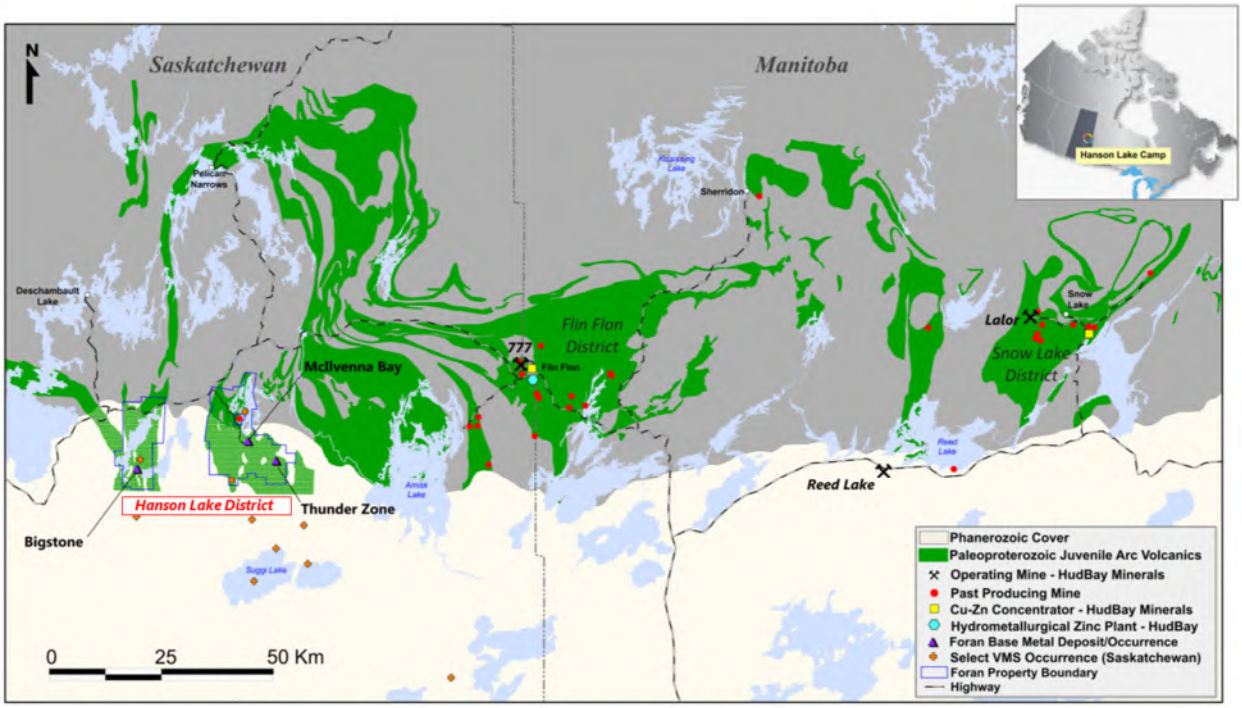
There has been no effort by Foran to attempt to establish a fracking sand resource estimate for the Project, although the same silica sand layer that was mined in the Preferred Sands pits does appear to extend over the McIlvenna Bay deposit. As such, no value has been taken or is implied from frac sand by this study.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Exploration Potential

Exploration potential exists for the discovery and delineation of additional VMS deposits on Foran's land holdings in the Hanson Lake District. The Hanson Lake District represents the two western most volcanic assemblages which form part of the prolific Flin Flon Greenstone Belt that extends over 225km from Snow Lake in Manitoba to the Bigstone Lake area in west-central Saskatchewan and is host to 29 past and present producing mines representing over 170 million tonnes of production (Figure 24-1).

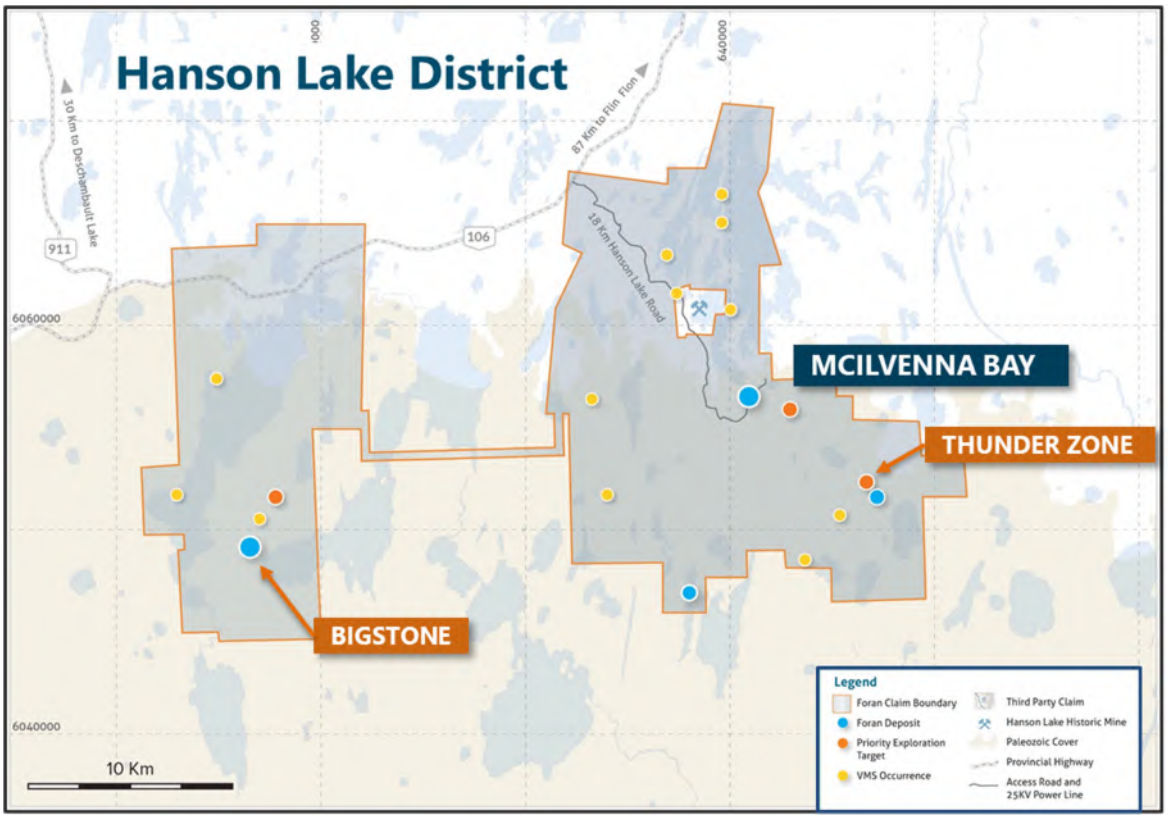
Figure 24-1: Flin Flon Greenstone Belt



As briefly mentioned in Section 23 and illustrated in Figure 23-2 above, there are numerous historic occurrences of VMS style mineralization and a number of high priority geophysical exploration targets in the Hanson Lake area, several of which have been the focus of recent work by the company. Foran also holds a large claim block in the Bigstone Lake area located 25km to the west of McIlvenna Bay which is host to the historic Bigstone Deposit and a number of other compelling exploration targets.

Currently, the more advanced of these prospects that could potentially represent satellite mill feed for a central processing plant at McIlvenna Bay, subject to continued successful exploration, are the historic Bigstone Deposit located on the Bigstone property, 25km to the west of McIlvenna Bay; and the recently discovered Thunder Zone massive sulphide prospect, located seven kilometres to the southeast of the McIlvenna Bay deposit along the Balsam Trend (Figure 24-2).

Figure 24-2: Hanson Lake District Properties



24.1.1 *Thunder Zone*

The Thunder Zone is located approximately seven kilometres from McIlvenna Bay along a trend of prospective stratigraphy that extends to the southeast from the deposit.

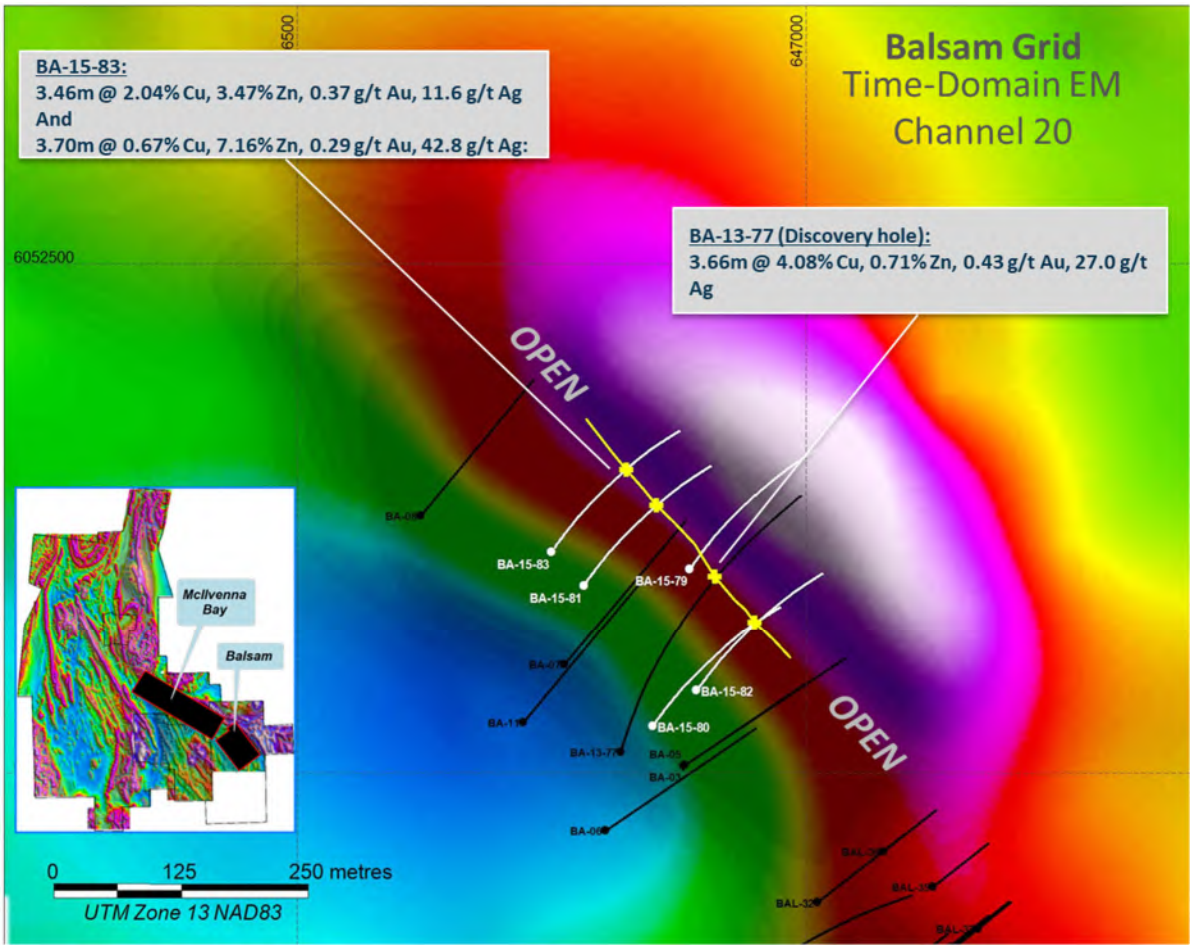
The Thunder Zone was initially discovered in 2013 when massive sulphide mineralization was intersected in a drill hole designed to test a new electromagnetic (EM) conductor identified as part of a ground geophysical program conducted that year over the trend from McIlvenna Bay to the near by Balsam area. One of the last drill holes of the program at Balsam tested this new anomaly and was successful in intersecting a new zone of mineralization, termed the Thunder Zone along the same geological trend that hosts the McIlvenna Bay deposit. Massive sulphide mineralization was intersected in BA-13-77 which included a 3.66 m intercept grading 4.08% Cu, 0.43 g/t Au, and 27.0 g/t Ag.

In 2015, Foran completed five drill holes encompassing 1,914m at the Thunder zone to follow up on the new discovery from 2013. The program was successful in intersecting massive and stringer sulphide mineralization in four of the five holes drilled which partially defines a moderately dipping mineralized zone, hosted by moderately to strongly altered felsic volcanic rocks. Drilling to date has defined the zone over approximately 300 metres along strike. The zone remains open for expansion.

The best result from the 2015 program came from hole BA-15-83 drilled at the northwest edge of the known horizon which intersected two zones, an upper zone containing 2.04% Cu, 3.47% Zn, 0.37 g/t

Au and 11.6 g/t Ag over 3.46 m followed down hole by a second zone containing 0.62% Cu, 3.41% Zn, 0.36 g/t Au and 27.24 g/t Ag over 8.39m (including an interval of 3.70m grading 7.16% Zn). A map showing a gridded profile of the EM response over the Thunder Zone with the drill hole superimposed on it is provided in Figure 24.3 below. Additional exploration is warranted at the Thunder zone to further define the extent of the mineralized zone and if significant, to prepare a 43-101 compliant Mineral Resource estimate.

Figure 24-3: Thunder Zone – Gridded EM response and drilling



24.1.2 *Bigstone*

The historic Bigstone Deposit is located in rocks of the Northern Lights Assemblage, near the western margin of the Flin Flon Greenstone Belt. The Bigstone property has been the subject of significant exploration since the 1980's with work conducted by several operators over the years. The Bigstone deposit itself was originally discovered in 1982 and has been the subject of a number of drill campaigns since that time.

The deposit is hosted in a north to northeast trending, west facing, vertical to steeply west dipping succession of volcanic to volcaniclastic units with lesser associated sediments and local iron

formations. The bulk of the mineralization in the deposit is hosted in a copper-rich zone consisting of disseminated, semi massive to massive and stringer styles of pyrrhotite-pyrite-chalcopyrite +/- magnetite associated with strong chlorite alteration in rocks of intermediate to felsic affinity. This mineralization is thought to represent a sub-seafloor replacement body located stratigraphically below a generally narrow zinc-rich massive sulphide horizon.

A historic (Non – 43-101 compliant) Mineral Resource estimate was prepared by past operators in 1990 which estimated a Mineral Resource at Bigstone at a number of different cut-off grades. At a 1.0% Cu cut-off the Copper Zone at Bigstone contained an estimated 3.75Mt grading 2.03% Cu, 0.33 g/t Au, 9.3 g/t Ag. A sensitivity analysis of various Cu cut-offs for the resource is presented in Table 24-1.

Table 24-1: Bigstone Copper Zone Historic Resource Estimate Sensitivity Analysis (Cameco, 1990).

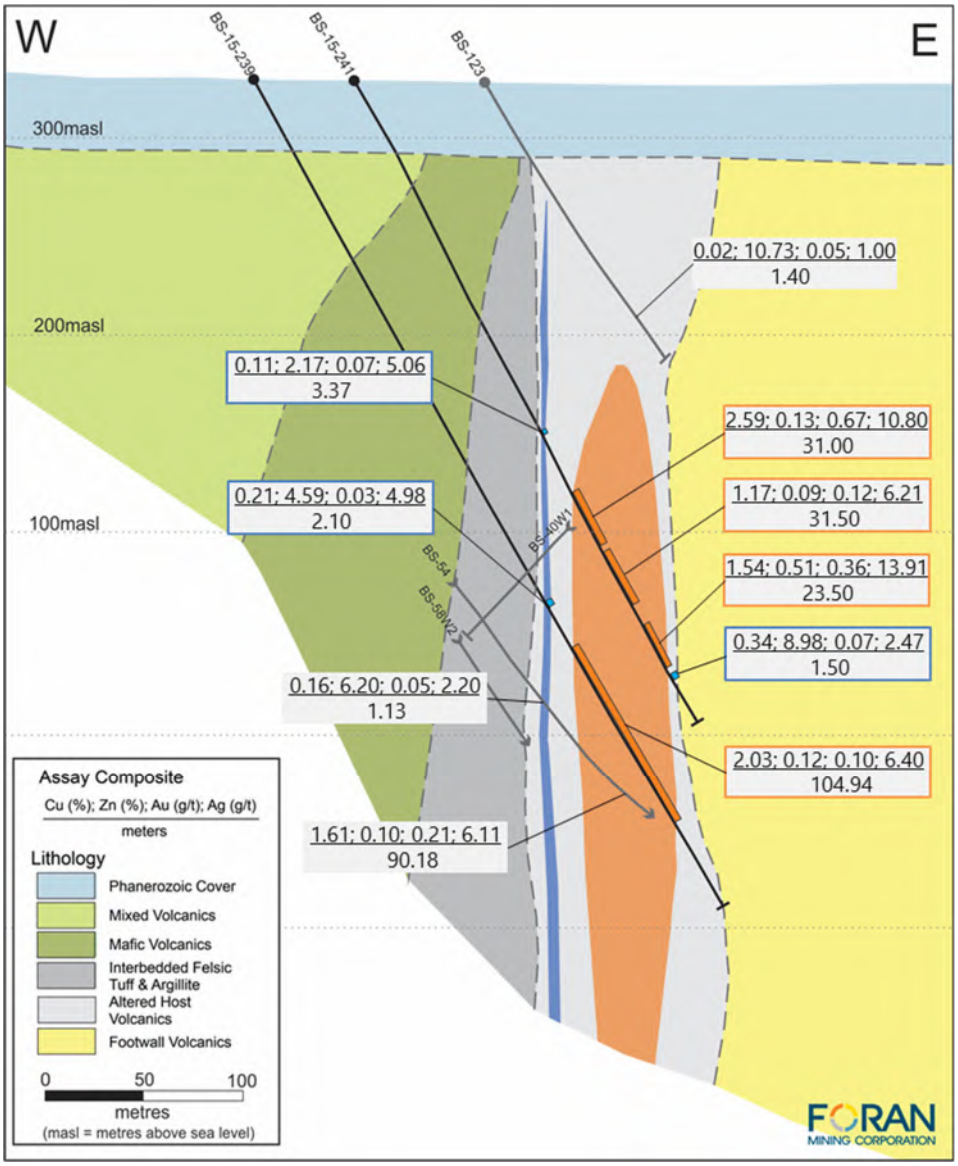
Cu cut-off (%Cu)	Tonnage	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)
1.0	3,747,500	2.03	0.14	0.33	9.3
1.5	3,136,600	2.26	0.15	0.36	9.9
2.0	1,983,600	2.57	0.17	0.48	11.3
2.5	1,199,300	3.11	0.20	0.61	13.5

The Company is not treating the historic estimates as current as a Qualified Person within the meaning of NI 43-101 has not completed sufficient work to classify the historic estimate as current; additional work, including re-surveying, re-logging and drill core QA/QC would be required to verify and upgrade the historic estimate to current.

An additional 525,300 t of material grading 9.62% Zn, 0.34 g/t Au and 15.9 g/t Ag (5.0% Zn cut-off) was also estimated for the zinc-rich massive sulphide zone that is located stratigraphically above the Copper Zone.

In 2015, Foran completed a six-hole 2,545m drill program designed to validate historic drill hole locations, geology and assay results at the Bigstone Deposit. The program was successful in its objectives, returning a number of thick intersections of high grade copper, including 2.03% Cu over 104.94 metres starting at a downhole depth of 327.56 metres in hole BS-15-239. Included within this composite were two higher grade intervals of 4.11% Cu over 20.35 metres at 333.70 metres and 3.16% Cu over 19.00 metres starting at 363.00 metres downhole. A type cross-section through the Bigstone Deposit, which includes the results from two of the 2015 drill holes, is provided in Figure 24-4 below.

Figure 24-4: Bigstone Deposit - Typical Cross-Section (1675N)



At this time, it appears that additional drilling will be required to validate and fully define the Bigstone deposit before a 43-101 resource estimate can be completed. Bigstone is a priority exploration target for future follow up work since the deposit is located within trucking distance of McIlvenna Bay and could be a future source of additional mill feed for a potential mine development.

25 INTERPRETATION AND CONCLUSIONS

Foran Mining Corporation is satisfied that the work that is summarized within this Technical Report has been completed to established standards for prefeasibility studies, and that is meets the requirements of National Instrument 43-101 Standards for Disclosure for Mineral Projects.

The subsections that follow provide QP's concluding remarks for the main sections of the report.

25.1 Geology and Mineralization

Since there is little outcrop at the Project site, the Foran exploration group has developed a detailed geological model using data obtained from the extensive diamond drilling programs. As a result of the drilling, a very a good understanding of geological processes that formed the McIlvenna Bay deposit has been developed which provides sufficient information to support the declaration of Indicated resources.

25.2 Exploration

Since acquiring the mining permits comprising the McIlvenna Bay Property, Foran has completed a number of economic studies as well as exploration and drilling programs on both the McIlvenna Bay deposit and a number of secondary targets or zones. Foran has managed to outline potentially economic mineralization in the upper portion the McIlvenna Bay deposit but it remains open both down plunge and at depth.

The upper portion of the mineralization has seen sufficient drilling to confidently classify a portion of the mineralization as Indicated according to the current CIM guidelines, adopted on May 10, 2014. The mineralization encountered in the deeper portions of the deposit continues to be classified as Inferred, at this time. It is believed that future drilling programs will be able to upgrade the Inferred material to Indicated. Further drilling should be able to outline the extent of further mineralization at depth and down plunge of those contained within the current Mineral Resource Estimate. At this time, no further drilling of the deposit from surface has been outlined by Foran as it believes that it would be more cost effective to drill the deeper portions of the deposit from underground in the future. The QP concurs with this approach for future drilling programs.

25.3 Mineral Processing and Metallurgical Testing

McIlvenna Bay is located in a region of Canada that is well known for VMS mineral deposits, and from work completed so far appears to be similar metallurgically to those processed historically in the region. Metallurgical testing of samples from McIlvenna Bay began in 2012 and has continued, along with several programs of mineralogical characterization, through to the most recent work completed in 2019.

Metallurgical work has focussed on the characterisation of two primary ore types, namely Copper Stockwork (CSZ) and Massive Sulphide (MS). These ore types, though mineralogically dissimilar, can be processed as a blended feed into a conventional crushing/grinding/flotation process, with separate

copper and zinc concentrates generated. The most recent work has been conducted on composite samples that the QP understands to be adequately representative of the specified mineralization styles. Variability composites have allowed the examination of a wide range of performance data, suitable for the prefeasibility level of study.

The results of flotation testwork carried out on master composites, master composite blends, and a series of variability composites have been used to prepare metallurgical performance predictions in the form of grade vs recovery relationships and metal 1 recovery vs metal 2 recovery relationships.

Physical and chemical characterisation of metallurgical process products has been carried out to a reasonable standard, and data has been used to support downstream cost/performance parameters.

25.4 Mineral Resource Estimate

25.4.1 *General Notes*

The Mineral Resource Estimate described within this technical report is unchanged from the Resource disclosed in Micon's Technical Report dated July 10, 2019 which has been filed by Foran on SEDAR.

Due to the multi-element nature of the of the McIlvenna Bay deposit an NSR value was used for the application of a cut-off to the block model. The NSR was estimated for each block using provisions for metallurgical recoveries, smelter payables, refining costs, freight, and applicable royalties. Metallurgical recoveries were based on the results of laboratory testwork conducted during the 2013 Preliminary Economic Assessment study. The smelter terms and freight costs were estimated by Foran. Metal prices used for the Mineral Resources were based on consensus, long term forecasts from banks, financial institutions, and other sources. The calculation was based on the assumption that two products, a copper and a zinc concentrate, would be produced by a processing facility at site. The massive sulphide is split into Cu/Pb ratio greater than 1.2 and less than 1.2 as it is expected that Cu recovery will be significantly reduced where the ratio of Cu:Pb is less than 1.2

The cut-off was established using preliminary mining parameters and operating costs. For the preliminary NSR calculations metal recoveries were applied to establish distinct metal multipliers for the CSZ and MS domains. Those same formulas were applied to the other domains based on Zn and Cu content. Foran has chosen to report the Mineral Resources at a cut-off value of US$60/t in order to be closer to the criterion used by other current and planned mining operations in the region. Table 25-1 summarizes the 2019 Micon Mineral Resources which have an effective date of May 07, 2019.

Table 25-1: Mineral Resources for the McIlvenna Bay Deposit, Reported at an NSR of US$60/t

Classification Category	Mineralized Domain (Zone)	Tonnage (Mt)	Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)
Indicated	Main Lens – Massive Sulphide	9.25	0.90	6.43	0.40	0.52	25.97
	Lens 3	1.99	0.85	3.29	0.14	0.27	14.71
	Stringer Zone	0.70	1.38	0.62	0.04	0.35	13.34
	Copper Stockwork Zone	10.30	1.43	0.28	0.02	0.40	9.30
	Copper Stockwork Footwall Zone	0.71	1.60	1.04	0.04	0.54	11.47
	Total	**22.95**	**1.17**	**3.05**	**0.19**	**0.44**	**16.68**
Inferred	Main Lens – Massive Sulphide	2.97	1.29	4.79	0.29	0.47	23.58
	Copper Stockwork Zone	8.18	1.42	0.76	0.03	0.47	11.63
	Total	**11.15**	**1.38**	**1.83**	**0.10**	**0.47**	**14.81**

The Mineral Resources presented here were reviewed and audited by Micon's QPs using the CIM Definitions and Standards on Mineral Resources and Reserves as of May 10, 2014. Mineral Resources unlike Mineral Reserves do not have demonstrated economic viability. At the present time, neither Micon nor its QPs believe that the Mineral Resource Estimate is materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

25.5 Mineral Reserve Estimate

The Mineral Reserves estimate for the McIlvenna Bay project has been completed in accordance with accepted good practice and meet the CIM Definition Standards for Mineral Resources and Reserves (2014).

Many factors may affect the Mineral Reserves Estimate such as dilution; metal prices; metallurgical recoveries and the geotechnical characteristics of the orebody and host rock; and capital and operating cost assumptions. These modifying factors have been taken into account subject to the information and means available to QPs at the time of study, as described in this report.

The reserves were estimated by designing mineable stope shapes in Deswik using Deswik Stope Optimizer (DSO). The dimensions of the shapes were determined using geotechnical modelling and modifying factors such as dilution and mining recovery were applied. The shapes were then interrogated against the resource block model and included in the mine plan where the NSR exceeded the cut off. The result was the reserve estimate on which the mine plan was based.

As with the resources, the multi-element nature of the of the McIlvenna Bay deposit requires that an NSR value was used for the application of a cut-off to the block model. The NSR was estimated for each block using preliminary metallurgical recovery formulas, smelter charges and payable metal values, refining costs, freight, and applicable royalties. Smelter terms and freight costs were market-based. Metal prices used for the Mineral Reserves were based on consensus, long term forecasts from banks, financial institutions, and other sources. The calculation was based on the assumption that two products, a copper and a zinc concentrate, would be produced by a processing facility at site. The

massive sulphide is split into Cu/Pb ratio greater than 1.2 and less than 1.2 as it is expected that Cu recovery will be significantly reduced where the ratio of Cu:Pb is less than 1.2

Table 25-2 Mineral Reserves for the McIlvenna Bay Deposit, Reported at an NSR of US$100/t

	Probable Tonnes	Grade			
		Zn (%)	Cu (%)	Au (g/t)	Ag (g/t)
Massive Sulphide	7,773,176	5.71	0.88	0.51	24.24
Copper Stockwork Zone	3,566,067	0.31	1.70	0.60	11.65
Total	**11,339,243**	**4.01**	**1.14**	**0.54**	**20.97**

25.6 Mining

25.6.1 *Mining Methods*

The McIlvenna Bay deposit is a narrow, steeply deep orebody hosted in competent ground conditions and is highly amenable the proposed mining methods of Sublevel Longhole Transverse Stoping and Longitudinal Avoca Stoping.

The project will be an early adaptor of Battery Electric Vehicles (BEV) in the form of the underground haul trucks. This technology has been proven in other industries and has been in use in underground mining since at least 2012, although it has yet to be widely adopted as the benefits are limited in operating mines because the cost of existing ventilation infrastructure has already been incurred. The capital cost for a new project needs to weigh the higher initial cost for BEV haul trucks (compared to diesel haul trucks) against the savings enjoyed through lower ventilation capital costs and lower operating costs. In the case of McIlvenna Bay, the BEV option was seen to give a better financial result.

The inclusion of underground crushing and vertical conveying to surface facilities enables reduced truck haulage costs and provides a low cost and efficient means of tramming ore from the deeper levels to the surface processing facilities with a minimum of re-handling.

25.6.2 *Geotechnical*

The 2019 characterisation work by MD Engineering in 2019 led to the following conclusions:

- Four joint sets are apparent
  - Joint Set FO (foliation parallel jointing) – typically dipping 65° to 75° towards 020° (the orientation varies due to folding). This set is consistent throughout all domains except dolomite.
  - Joint Set B - 53°/125° (dip/dip direction). This is a weakly defined joint set.
  - Joint Set H – shallow dipping 5 to 25° with a dip direction ranging from south to west. In the dolomite this joint set is the bedding planes, and in all other rock units this set is interpreted to be associated with glacial relief and crenulation fractures.
- Far field in situ stress is assumed to be:
  - $\sigma 3=\sigma v$
  - $\sigma 1=1.5\sigma v$ to $2\sigma v$, horizontally trending 070° (sensitivity testing has been conducted in this study due to uncertainty in far field stress orientation)

- σ2=1.0σv to 1.5σv, horizontal trending perpendicular to σ1
- Most rock units at McIlvenna Bay are very strong (100 to 250 MPa), with the exception of Schist (SCHT), Copper Stockwork Zone (CSZ) and Cherty Banded Metasediments (CBM) which are both considered strong (50 to 100 MPa) and the regolith (REG) which is medium strong (25 to 50 MPa).
- Geotechnical domaining has demonstrated that, for feasibility study design purposes, the rock mass can be spatially categorized into HW/FW/ORE. The rock mass quality is, on average, fair to good
- Subsequent geomechanical studies by the same group (renamed "RockEng" in 2020) have evaluated the Project mine plans based on the PFS design geometries and production sequencing. The most recent geotechnical update verified that the mine design and global sequencing strategies for McIlvenna Bay are geotechnically sound.
- There are no major changes in expectations for stope performance, and minor updates to dilutions predictions have been made.
- Minor stope sequencing adjustments have been recommended to improve performance of secondary stopes (secondary stopes should not lag more than 1 lift behind primary stope advance).
- Localized geotechnical risks associated with pillar stability have been identified.

25.6.3 *Hydrogeological*

A 2019 Hydrogeological study of the project area collected site data from a selection of existing drillholes and performed numerical modelling of data to assist the prediction of mine dewatering requirements at the different stages of development over the life of mine. The study concluded that:

- Along each hole, it was not possible to isolate any water-bearing faults or features, although this alone does not mean that those features do not exist.
- The sedimentary rocks present at surface such as the dolomite and the sandstone have higher K values, although this does not seem to be high enough to cause problems during the completion of the ramp.
- The simulation results suggest that there is not a significant effect of recharge from Hanson Lake.
- Simulation results suggest that a relatively small inflow of water will be present at the final stage of operation, due to a low permeability rock and also due to the fact that no contrasting water bearing zones were identified.

25.7 Recovery Methods

The QP's interpretation of metallurgical testwork results has been used to prepare a detailed process design criteria for the plant. This information was used together with mass and water balances to prepare plant specifications and equipment listings that then formed the basis for capital cost estimation. A conceptual plant layout has been developed that provides sufficient detail for the appropriate estimation of steel, civil and earthwork capital costs.

The selected process utilizes conventional mineral processing techniques and is similar to other VMS operations in the vicinity. Surface and/or underground primary crushing operations are followed by coarse ore storage, ore blending (to deal with potential copper and zinc grade variability), secondary crushing, fine ore storage, grinding, sequential selective flotation and flotation products dewatering. Copper and zinc flotation concentrates do contain minor concentrations of certain penalty elements, but in general can be considered to be high quality with 26.8% Cu in the copper concentrate and 54.7% Zn in the zinc concentrate (both life of mine average grades).

A pyrite flotation circuit at the back-end of the flotation plant ensures that tailings destined for the filtered tailing storage facility include very low concentrations of sulphur and can be treated as non-acid generating.

Approximately half the processed tonnage is mixed with binder and used underground as a paste backfill medium.

25.8 Project Infrastructure

McIlvenna Bay is located in one of Canada's oldest mining districts, in relatively close proximity to the town of Flin Flon. Flin Flon and its surrounds enjoy very well established infrastructure for road and rail transport, hydropower and mining-related service providers.

The site is accessed by an existing gravel mine road that was used historically to serve a large fleet of haul trucks for a fracking sand mine. An existing, decommissioned hydropower line suitable for provision of construction power, runs alongside the access road to site.

On site infrastructure is relatively compact in extent and is quite conventional in nature, consisting of offices, workshops, a processing plant, storage yard, fuel storage facilities, water storage and treatment facilities.

A design has been completed for a lined filtered tailings storage facility, containing approximately 5.7Mt of de-sulphurized plant tailings filter cake and located on the old Preferred Sands (rehabilitated) mine site. Whilst the PFS design is considered preliminary in nature, the overall concepts for disposal and long term storage represent current best practises and are considered to be the most effective mitigation of associated risks.

25.9 Market Studies and Contracts

A market study for copper and zinc concentrates was not conducted as part of the PFS, as these products are considered to be commonly traded at prevailing market rates. NSR calculations and economic modelling of the project used generic commercial terms as described within this report.

25.10 Environmental Studies, Permitting and Social Impact

Current exploration activity is fully permitted and in good standing. The baseline environmental work completed for the Project has been conducted over a period of several years by qualified independent consultants and site monitoring activities will continue as the project develops.

The baseline work completed for the Project described a typical northern Saskatchewan setting and did not identify any issues that could materially impact the Project, assuming that proper planning, permitting, and mitigations are incorporated into the project design. Such mitigations may include, but are not limited to, habitat compensation for any fish habitat impacted by the project, possible mitigation strategies for vegetation and wildlife SOCC impacted by the project, and consultation with local First Nations and communities.

25.11 Capital and Operating Costs

Capital and operating cost estimates completed as part of this study have been prepared generally in accordance with practices recommended by the AACE for a Class 4 Study. The overall estimating accuracy range is considered to be ±25% which is within the normal limits for a prefeasibility study cost estimate.

The Capital estimate has been built up using bills of quantity and specifications that are supported by appropriate engineering or the industrial experience of relevant QP's. Supplier quotations have been obtained for all significant cost items, and engineers database costs, or benchmark data from similar projects has been accepted for minor cost items.

Operating costs have been estimated from first principles throughout the project, using information quoted by suppliers, staffing levels typical for the stated operations, and consumable consumption rates indicated by testwork, industry benchmarking, and QP's experience in relevant operations.

25.12 Economic Analysis

A discounted cashflow analysis model has been developed for the project using Microsoft Excel®, with revenues and costs input on an annual basis, commencing on a nominal date that corresponds to the commencement of detail design and construction.

The model indicates that the Project as described by this prefeasibility study, including all stated assumptions, is economically viable. The post-tax NPV (7.5% discount rate) is calculated to be $147 million with an IRR of 19.2%.

Pre-production capital expenditures total $261 million, including a $30 million contingency allowance.

The project economics are sensitive to commodity prices (copper, zinc, gold and silver) and the CAD:USD foreign exchange rate.

25.13 Adjacent Properties

The QP has not verified the information discussed within Section 23 regarding the other base metal properties or deposits and is of the opinion that the information is not necessarily indicative of mineralization within the McIlvenna Bay deposit, which is the subject of this Technical Report.

The QP is not qualified to comment on the technical aspects of Fracking Sand deposits. However, the material is not indicative of the mineralization within the McIlvenna Bay deposit, which is the subject of this Technical Report.

25.14 Risks And Opportunities

25.14.1 *Risks*

A Risk Register has been established for the project and this is being maintained as the project progresses. Some of the key risks identified for the project are summarized in Table 25-3 below.

Table 25-3: Identified Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Metal Prices	Commodity prices are sensitive to many external variables, such as global economic conditions and fundamental supply/demand balances. Lower copper, zinc, gold or silver prices will negatively impact the project economics.	Market studies
FX Rates	CAD : USD exchange rates have a direct influence on project revenues and costs.	
Resource Model Accuracy	All Mineral Resource Estimates carry some risk and are one of the most common issues with project success. If LOM tonnes or grades are lower than currently assumed, the project economics would be negatively impacted.	Peer review: Infill drilling from underground may generate information to provide a greater level of confidence in the resource.
Dilution	Higher than expected dilution has a negative impact on project economics. The mine must ensure accurate drilling and blasting practices are maintained to minimize dilution from wall rock backfill and other mineralized zones, minimize secondary breaking, and optimize extraction. The ability to segregate higher-grade material, early in the mine life, is important to project economics. If LOM metal grades are lower than currently modelled, then project economics would be negatively impacted.	A well-planned and executed grade control plan is necessary immediately upon commencement of mining. The use of sorting technology could help discard dilution before haulage and processing.
Geotechnical Risk	Mining with backfill and/or pillars may increase dilution and overall mine recovery.	Overall geotechnical stability of the mine needs to be assessed in more detail at the feasibility level.
Metallurgical Performance	Changes to metallurgical recovery assumptions could lead to reduced metal recovery, increased processing costs, and/or changes to the processing circuit design. If LOM metal recovery is lower than assumed, the project economics would be negatively impacted.	Additional sampling and test work is recommended at the next level of study.
CAPEX and OPEX	The ability to achieve the estimated capital and operating costs are important elements of project success. Operating costs in particular, effectively increase the NSR cut-off would increase and, all else being equal, the size of the economic resource would reduce yielding fewer tonnes in the mine plan.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Permit Acquisition	The ability to secure all the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits could stop or delay the Project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.
Development Schedule	The Project's development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the Project economics.	If an aggressive schedule is to be followed, FS level field work should begin as soon as possible.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the Project. High turnover or the lack of appropriate technical and management staff at the Project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries and benefits identify, attract, and retain critical people.
TSF Foundation Conditions	The geotechnical data available for the foundations of the TSF are based on text descriptions by various parties, photos, and discussions with Foran. Given the lack of available data, unsuitable foundation conditions may be present below the proposed TSF footprint.	Depending on the actual foundation conditions and density of the 20 to 25 m thick layer of loose rockfill and sand, additional foundation excavation, changes to the lining system to accommodate settlement, or relocation of the tailings footprint, may be required. Further geotechnical studies will help to define this.
Availability of borrow material for TSF construction	It has been assumed based on discussions with CPI (the contractor who decommissioned the previous quarry) that some sand stockpiles remain on site. There is insufficient data available to estimate the quantity and suitability of the sand for use as a TSF Liner Bedding material. It is recommended that test pit investigations, lab testing and surveys be carried out to evaluate the availability and suitability of sand for construction.	Test pit investigations, lab testing and surveys should be carried out to evaluate the availability and suitability of sand for construction.
TSF : Clay Layer Thickness	-The former quarry area under the TSF footprint has been covered in a 1 m thick layer of clay and organics, however the true thickness of this layer is unknown	Geotechnical investigations should be carried out to quantify the thickness of this layer
TSF : Geochemistry	A key assumption in the TSF design is that desulphurization of the tailings will reduce the potential for acid generation and metal leaching to a level that can be easily managed during operations and following closure.	Geochemical testing is ongoing, and more is recommended. Test results should be used to evaluate the potential seepage water quality from the TSF over the long-term. Additional pre-treatment of the tailings may be required during the process to fully neutralize the tailings prior to storage.
Delays to Power Supply	Power is readily available in the area, but specification, design, construction and permitting of a suitable powerline for the project may be slower than anticipated.	An existing line is adequate for construction power. Continued liaison with Saskpower and other stakeholders is recommended.

25.14.2 *Opportunities*

Several opportunities for project improvement have been identified and are summarized in Table 25-4 below.

Table 25-4: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of the Mine	Conversion of Inferred to Indicated resources may result in an increase to reserves.	higher throughput rate or a longer mine life – both of which would improve the project economics
Increased Production	Increased production may be possible in some areas of the mine. There is an opportunity for the mine to produce more tonnes for short durations on the high tonnage levels of the mine.	Reduced unit operating costs and increased revenue.
Optimize Mine Plan	Optimize the mine plan and stope sequence.	Decrease ramp-up duration and potentially higher grades earlier in the mine life.
BEVs and Automation	Early adoption of technology is a core principle of Foran's project development plan; BEV implementation could be expanded, and autonomous equipment can improve efficiencies underground.	Continue engaging specialists and vendors to examine the viability of BEV technology, vertical conveying, autonomous equipment technologies.
Mineral Resources	Further definition drilling and sampling could increase the volume of the resource. Areas within the mine plan could convert from waste to material above cut-off grade, reducing strip ratios and increasing cash-flow.	Further exploration of McIlvenna Bay will likely only happen once underground drilling access is granted.
Satellite Deposits	Potential exists for additional resources at nearby deposits or prospects. These deposits are discussed in Section 24.	Additional mill feed (especially at higher grade) could improve the project economics by speeding up project payback and/or extending the mine life.
Metallurgical Performance	Changes to metallurgical performance predictions may improve estimates of recovered metal.	Increased revenue stream leading to improved economics.
Preconcentration	Implementation of sorting technology, or density separators prior to milling can reject low/no grade material.	Increased grade/reduced tonnage to the process plant can lead to lower cost per tonne treated, smaller TSF and improved economics.
Tailings lift Thickness and Compaction Effort	It is currently assumed that all tailings stored in the TSF will be spread and compacted in 0.3 m lifts. It is likely that the entirety of the TSF stack does not need to be compacted to stringent criteria to achieve a suitably stable facility	Additional geotechnical testing for tailings strength properties, and additional stability analyses could be used to optimize the tailings placement and compaction specifications. It is also recommended that thorough testing be completed during the initial period of operations to optimize the placement and compaction procedures
TSF : In situ Clay	The current design includes a sand layer and geotextile cushion below the LLDPE liner for added protection. A composite liner (i.e. LLDPE in intimate contact with clay) may be achievable if the in situ clay can be suitably prepared.	An evaluation of the in situ clay is recommended to determine if it can be worked into a suitable liner subgrade surface
TSF : Closure Liner	It is currently assumed that an LLDPE cover liner will be required for closure of the TSF. This is based on an assumption that the clay recovered from foundation preparation will not provide a sufficient barrier to infiltration.	If the clay on site is of suitable quantity and quality, there may be justification to replace the LLDPE liner with clay to establish the closure cover.

26 RECOMMENDATIONS

26.1 General

The work completed as part of the Prefeasibility Study and summarized in this Technical Report outlines a project that is economically feasible given the stated inputs and assumptions. The work conducted over recent months has compared various operational scenarios for the project, including alternate mining methods, production rates and processing strategies.

AGP recommends that the preferred project configuration, described within this report, should be studied in more detail via a full feasibility study. Estimates of cost and schedule for this are outlined in Section 26.9 below.

26.2 Further Exploration

It is the QPs understanding that Foran will look at conducting further exploration programs on the McIlvenna Bay deposit from underground in order to gain further knowledge regarding the true extent of the base metal mineralization. In that context, the QP makes the following additional recommendations:

- Micon recommends that any future exploration drilling on the McIlvenna Bay deposit should be conducted from underground.
- Micon recommends that Foran continue to conduct exploration on the secondary deposits on the McIlvenna Bay property so that it may be able to outline secondary deposits which may contribute to mining production in the future.

26.3 Mining

Several recommendations have been identified for the mining of the McIlvenna Bay deposit:

- Further optimization of the mine plan and stope sequence to identify and realize higher NPV.
- Further optimization of the cut off NSR to increase resource conversion and extend the mine life.
- Evaluate contracting for mining development or production, as part of the risk mitigation strategies to current market limitations of skilled employees, and particularly maintenance and mechanics.
- Investigate expanding electrification of underground mining equipment to include the LHD fleet. At time of writing BEV LHD equipment was determined to not be demonstrated technology, however it could be demonstrated in the near future and would likely be able to be implemented as the project matures.
- Automate the haulage fleet to allow production to continue during shift change. The PFS has not considered automation of the haulage fleet because there is a single decline contemplated, meaning automated equipment and operator driven equipment would be occupying the same

space. There are 4 hours available during shift change where automated trucks would be able to transport additional material, resulting in a lower capex requirement for mobile equipment.

- Optimize backfill to displace pastefill with development waste where possible. There are many secondary transverse stopes that could potentially be filled with waste rather than pastefill to reduce binder cost, however this would increase stope cycle time for those stopes.
- Include incremental ore into the mine plan to increase metal production, particularly during production ramp up when the mill feed is lower. Incremental ore is material that must be mined to access ore, such as mineralization that is extracts in a stope access. Since this material must be mined, the mining costs can be discounted from the economic evaluation so if the cost to process the material is value accretive, it should be processed unless it is displacing higher value ore.

26.3.1 *Hydrogeology*

Water chemistry results obtained in the PFS program did not respect the criteria in Saskatchewan for water discharge, triggering the inclusion of water treatment facilities in the PFS designs and cost estimates. In additional to further site sampling exercises similar to those carried out for the PFS, It is recommended that a dedicated pumping well be established for water sampling. It is considered likely that sampling water in open holes or piezometers might induce sediments and turbidity, which could lead to higher results, even with water filtration. A well, with long term pumping (even at low flow), would reduce the dissolved components and would provide a more representative water sample to bolster the existing data set.

26.3.2 *Geotechnical/Geomechanics*

MD Engineering were retained in 2019 to provide Geotechnical Characterization of McIlvenna Bay drill core. Their study included the following recommendations for future work:

- Refine the Geological Model. More detailed definition of lithological domains within each major formation will improve the spatial delineation of rock mass variability based on rock type. This will help to improve certainty in anticipated ground conditions during the construction of FW development. Further to refining the lithological model, ongoing geological work should also aim to improve the site structural model to understand spatial changes in the orientation and intensity of foliation. All designs provided in this study account for the presence of horizontal jointing, there is an opportunity to optimize design if the spatial distribution of this set can be associated with folding in a structural geology model.
- Geotechnical Mapping: As underground construction and operations ramp up geotechnical scanline mapping should be conducted underground to better quantify joint persistence and spacing. Routine geotechnical mapping should continue over the life of mine to maintain an up to date geotechnical model.
- Laboratory Testing: To better quantify anisotropic material strength, samples which cut foliation fabric on different orientations should be tested to better define anisotropic strengths.
- Verify Hydrogeology Conditions: If groundwater or meteoric inflows are encountered (or anticipated based on any future hydrogeology work), geotechnical designs should be reviewed accordingly. Assumptions regarding water conditions will have to be verified once hydrogeology data is available.

A follow up report by the same group in 2020 (now named "RockEng") provided additional guidance and recommendations regarding the mining methods and production schedules prepared by AGP's engineers for the PFS:

- It is recommended that the diminishing pillar between the longitudinal and transverse mining fronts be sequenced so that the final stopes (last 3 to 4 on each level) be extracted by transverse layouts to minimize risks associated with stress induced damage in longitudinal development.
- Where FW infrastructure is located near barren pillars, stress induced damage can be expected. Ventilation infrastructure is located at the lateral abutments of the mining levels, and so, FW development will have to be maintained for the full life of mine. It is recommended, adjacent to barren pillars, the FW development be moved (roughly 10m) further from the ore body. It is reasonable to implement this design detail at later stages of detailed engineering as stope shapes and infrastructure planning will continue to evolve.

26.3.3 *Mine Portal*

BBA were retained in 2019 to provide detailed designs and cost estimates for underground infrastructure at McIlvenna Bay. Their study included the following recommendations for future work related to the design of the mine portal:

- Additional geotechnical drilling at the proposed portal face is recommended prior to box-cut excavation to confirm and verify true vertical depth of the underlying units at the portal location (i.e. top of sandstone).
- The existing ground condition at the portal should be reviewed during portal excavation to optimize the box-cut excavation design with required mitigation measures and support system for the long-term stability of the portal.
- Field observations and detailed mapping of the dolomite during excavation to verify and confirm MD Engineering downhole drill record data (e.g. verify two sub-vertical joint sets that were weakly defined in the downhole drill logs).
- Update box-cut excavation design kinematic analysis with results of field mapping of the dolomite excavation.
- The sandstone horizon underlying the dolomite is poorly consolidated and weak. To avoid potential box cut instabilities associated the sandstone, such as lateral wall creep, it is recommended that the box remains in dolomite and above the sandstone.
- Probe drilling into the box cut floor, starting at ~20m below rock surface contact is recommended to confirm the depth of the underlying sandstone horizon.
- A DDH near the portal location would be useful to determine the true extents of the DOL 1 and DOL 2 layer. If DOL 1 domain is extensive, then pre-grouting of the sandstone will be required.
- Grouting ahead of the main ramp development from the lower box cut development is key due to the poor consolidation of the underlying sandstone layer. Additional systematic grouting ahead of the advancing ramp face will be required until the ramp has successfully been established beyond and below the sandstone horizon.

26.3.4 *Other Underground Infrastructure*

For the mine dewatering system, water inflow is assumed to be uniformly distributed between all levels. It is recommended that further hydrology testing be completed to gain a further understanding of water flows at lower elevations.

During the feasibility study or the detail engineering phase, it is recommended that a trade-off study be completed to determine the most cost effective way to meet the intent of IEEE519 (system guidelines for setting limits on voltage and current distortion) at the point of common coupling. The outcome of this study should be approved by Foran as it will have an immediate and long term cost implication.

As part of future studies of the underground electrical systems, the following trade off work is recommended:

- determine the suitability of using Aluminum conductors vs. Copper conductors for the 13.8 kV primary feeders. The trade-off study should compare the weight, cost, and the current carrying capacity
- determine the suitability of utilizing a 500MCM Cu cable vs. a 350MCM Cu cable; the 500MCM cable was selected for the current study due to uncertainty related to the electrical load list
- determine the suitability of using ventilation shaft or pumping shaft to tie electrical vaults
- determine the suitability of using Aluminum windings vs. Copper windings for the 13.8/0.6 kV step-down transformers; the trade-off study should compare the weight, cost, and the current carrying capacity

26.4 Metallurgy and Process

Metallurgical testwork has been reasonably consistent throughout the project's history and results are generally consistent with performance of other VMS deposits in the area. The representativity of samples tested in recent work was found to be quite acceptable for the level of study contemplated in this report. Despite this, the QP recommends additional testwork, with a focus on the following areas:

- grindability testwork, particularly related to SAG milling
- flotation fine tuning, including:
  - assessment of composites selected to represent Y1-3 mine production
  - assessment of additional variability composites, to build confidence in the performance envelope
  - reagent optimization, with a focus on simplification and/or selection of more cost-effective chemicals
  - further desulphurization testing
- further characterisation of products, including tailings and concentrate dewatering, paste production and minor element analysis
- geochemical testing of tailings

The PFS on-site process plant design is well advanced and uses no unconventional equipment or design criteria. Significant flowsheet changes are therefore considered unlikely as the project advances to

feasibility level and detail design. The additional grindability testwork recommended to accompany feasibility studies may indicate that SAG milling is a more cost effective option for the process plant grinding circuit, and this would require additional process design effort in the crushing and milling areas. Minor changes to flotation and dewatering sections may be expected.

In any case, the process plant and paste plant designs should continue to advance, in terms of detailed engineering (drawings, specifications etc.) to support feasibility level cost estimates. Detailed plant water balance information should be prepared moving forward to complement the additional hydrogeology work. Together, this additional information would be used to support the preparation of a more thorough overall site water balance and design of water treatment and storage facilities.

26.5 Infrastructure

In support of a feasibility study, additional site works are required to further refine the designs and costs associated with the site infrastructure:

- complete a thorough geotechnical investigation prior to detailed engineering to optimize the design of the foundations and the exact location of all infrastructure including ponds and ore pads. The geotechnical investigation shall include a combination of test pits and drilling along with laboratory testing and reporting
- complete additional on site hydrogeology studies in the immediate area around the site as well as the area to be mined
- survey and/or generate a point cloud of the site to assist with refining earthworks quantities and other measurable construction quantities
- in consultation with the Saskatchewan Ministry of Highways and Infrastructure, complete a full inspection of the bridge located on the mine access road at coordinates 636320E, 6062173N

For the tailings facility, a number of gaps should be addressed to enable development of designs and operational plans to a feasibility study level of detail. These include:

- **Foundation Conditions:** The geotechnical data available for the foundations of the TSF are based on text descriptions by various parties, photos, and discussions with Foran. Given the lack of available data, unsuitable foundation conditions may be present below the proposed TSF footprint. Depending on the actual foundation conditions and density of the 20 to 25 m thick layer of loose rockfill and sand, additional foundation excavation, changes to the lining system to accommodate settlement, or relocation of the tailings footprint, may be required.
- **Borrow Material Availability:** It has been assumed based on discussions with CPI (the contractor who decommissioned the previous quarry) that some sand stockpiles remain on site. There is insufficient data available to estimate the quantity and suitability of the sand for use as a Liner Bedding material. It is recommended that test pit investigations, lab testing and surveys be carried out to evaluate the availability and suitability of sand for construction.
- **Clay Layer Thickness:** The former quarry area under the TSF footprint has been covered in a 1 m thick layer of clay and organics, however the true thickness of this layer is unknown. Geotechnical investigations should be carried out to quantify the thickness of this layer.
- **Geochemistry**: A key assumption in this TSF design is that desulphurization of the flotation tailings will reduce the potential for acid generation and metal leaching to a level that can be

easily managed during operations and following closure. It is understood that additional studies are ongoing to further evaluate the geochemistry of the tailings. It is recommended that the results of this testing be used to evaluate the potential seepage water quality from the TSF over the long-term. Additional pre-treatment of the tailings may be required during the process to fully neutralize the tailings prior to storage.

- **Weather and Upset Operating Conditions**: The TSF design includes a tailings storage shed to allow for storage of up to 4 days of tailings production without haulage to the TSF. However, there is a potential for more then 4 days of continuous poor weather, or potentially a major breakdown of the filter presses, that would result in wetter tailings being placed within the TSF that cannot be properly compacted, impacting the operations and stability of the TSF. Additional sensitivity analyses should be carried out once additional data becomes available on the tailings in order to further define the range of allowable operating criteria and develop suitable mitigation measures and procedures.

26.6 Market Studies and Contracts

Additional metallurgical testwork is recommended to allow more accurate and detailed predictions of copper and zinc concentrate composition. This information can be used, together with feasibility level production schedules to begin project-specific discussions with smelters. Domestic smelters and overseas (Asian) smelters would likely be considered.

Long term pricing forecasting should be sought from established independent commodity forecasting consultants (e.g. Roskill) to support a feasibility study.

A logistics study to assess options more completely for movement of concentrates to market should be considered.

26.7 Environment and Permitting

The environmental baseline data collected to date can be used for the EIA and to provide a comparison for future monitoring programs; however, if, as currently proposed, the treated effluent is discharged into a muskeg to the south/southeast of the site, additional aquatic and hydrological investigations (surface water hydrology; water and sediment quality; plankton, benthic invertebrate, and fish communities; fish habitat; fish chemistry; fish spawning) will need to be completed in the Hobbs Lake drainage.

Continued geochemical characterization of metallurgical products, particularly of tailings, is prudent for both operations and assurance of post-closure environmental performance.

It is also recommended that consultation with the local First Nations and communities in the Project area continue.

26.8 Project Economics

The economic modelling conducted, together with the productivity assumptions therein are deemed to be reasonable for this prefeasibility study. For the feasibility study, AGP recommends that monthly cashflows are determined and that a basic monte-carlo simulation series be completed.

It is recommended that the next stage of work should consider in more detail the link between mining activities and address cycle times, on a stope by stope analysis, and the dependencies to achieve those. It is recommended that work continue on optimization of the mine design and scheduling, as part of continued optimization of the project economics.

26.9 Further Studies

The prefeasibility study summarized in this report describes an economic project that should be advanced to feasibility level. The PFS is a suitable foundation for this advancement. The following steps are recommended to further de-risk the project and to allow development of more detailed (feasibility level) designs and cost estimates:

- mine plan optimization
- more detailed investigations into mine automation and BEV technology
- continuation of metallurgical programs
- tailings facilities and water balance – detailed investigations and designs
- development of surface infrastructure designs
- detailed geotechnical work (site buildings and tailings area)
- further field studies to define geotechnical and hydrogeological parameters
- baseline data collection to advance environmental permitting processes
- continue to engage with local communities and First Nations regarding the project development

The estimated cost of completing a feasibility study for the project is summarized in Table 26-1 below:

Table 26-1: Future Studies: Estimate of Cost

Feasibility Activity	Est. Cost ($)
Direct Feasibility Study Costs	
Underground Mining Designs	$770,000
Geomechanical (incl. Site Studies)	$340,000
Hydrogeology (inl. Site Studies)	$120,000
Geochemistry	$120,000
UG Infrastructure and Cost Estimating	$280,000
Metallurgical Testwork	$160,000
Process Plant Designs and Cost Estimates	$330,000
Surface Infrastructure	$110,000
Tailings: Site Investigations	$170,000
Tailings: Design	$200,000
Hydrology	$75,000
Financial model and Marketing Study	$100,000
Logistics/Transportation study	$30,000
Subtotal:	2,805,000
Other Feasibility Costs	
QP Services / Review / NI 43-101 Reporting	$100,000
Feasibility Study Management	$360,000
Environmental and Permitting	
Permitting	$75,000
Baseline and Other (Closure planning)	$220,000
Community Engagement and Social License	$120,000
Subtotal:	945,000
Total:	**3,680,000**

This excludes owners' costs and corporate overheads that would be required over the duration of the Feasibility Study.

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28 CERTIFICATE OF QUALIFIED PERSON

28.1 William J. Lewis, B.Sc., P.Geo.

I, William J. Lewis, B.Sc., P.Geo. am employed as a Senior Geologist with Micon International Limited located at #900–390 Bay Street, Toronto, ON, M5H 2Y2 Canada. This certificate applies to the technical report NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "Technical Report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of a number of Associations such as: The Association of Professional Engineers and Geoscientists of Manitoba, membership #1450, Association of Professional Engineers and Geoscientists of British Columbia (Membership # 20333), Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories (Membership # 1450), Professional Association of Geoscientists of Ontario (Membership # 1522) and The Canadian Institute of Mining, Metallurgy and Petroleum (Member # 94758)
- I graduated from The University of British Columbia with a B.Sc. (Geology) in 1985.
- I have practiced my profession continuously since 1985.
- I have sufficient relevant experience having worked as an exploration geologist, an underground mine geologist and as a surficial and consulting geologist on precious and base metal deposits and industrial minerals. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I visited the McIlvenna Bay Project from August 16 to 18, 2018.
- I am responsible for Sections 1.2 to 1.6, 1.8, 4.0 - 12.0, 14.0, 25.1 to 25.4, and 26.2.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have previous involvement with the project as a co-author of the previous 2019 Micon Technical Report for the McIlvenna Bay project with an effective date of 07 May 2019 and a report date of 10 July 2019.
- 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: April 27, 2020

"Signed and sealed"

William J. Lewis, B.Sc., P.Geo.

28.2 Andrew Holloway, P. Eng.

I, Andrew Holloway, P.Eng. am employed as a Principal Process Engineer with AGP Mining Consultants Inc. located at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7 Canada. This certificate applies to the technical report titled NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "technical report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of Professional Engineers of Ontario, membership #100082475.
- I graduated from The University of Newcastle upon Tyne, England, B.Eng. (Hons), 1989.
- I have practiced my profession in the mining industry continuously since graduation.
- My relevant experience with respect to metallurgy, process engineering and mining project management includes 30 years' experience in the mining sector covering mineral processing, process plant operation, design engineering, and operations and project management. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I have visited the McIlvenna Bay Project site on September 17, 2018 and again on January 16, 2019.
- I am responsible for Sections1.1, 1.7, 1.11, 1.15, 1.17, 1.18, 2.0, 3.0, 13.0, 17.0, 21.1.1, 21.1.6 – 21.1.9, 21.2.2, 21.2.4, 21.2.6, 23.0, 24.0, 25.7, 25.11, 25.13, 25.14, 26.1, 26.4, and 27.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with the McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Andrew Holloway, P. Eng.

28.3 Stephen Cole, P.Eng.

I, Stephen Cole, P.Eng. am an independent Consultant, with a duly authorized Certificate of Authorization (Licence No:100539980) issued by Professional Engineers of Ontario to practice Engineering in Ontario. The address on file for the Certificate of Authorization is 25 Albert St, Lakefield Ontario,K0L 2H0, Canada. This certificate applies to the technical report titled NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "technical report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of Professional Engineers of Ontario, membership #100043291.
- I graduated from University of the Witwatersrand, South Africa, B.Eng., 1994.
- I have practiced my profession in the mining industry continuously since graduation.
- My relevant experience with respect to metallurgy, process engineering and mining project management includes 30 years' experience in the mining sector covering mineral processing, process plant operation, design engineering, and operations and project management. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I have not visited the McIlvenna Bay Project.
- I am responsible for Sections 1.13, 1.16, 19.0, 22.0, 25.9, 25.12, 26.6, and 26.8.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with the McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Stephen Cole, P. Eng.

28.4 Denis Flood, P. Eng.

I, Denis Flood, P.Eng. am employed as an Associate Mining Engineer with AGP Mining Consultants Inc. located at #246-132K Commerce Park Dr., Barrie ON L4N 0Z7 Canada. This certificate applies to the technical report titled NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "technical report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of Professional Engineers of Ontario, membership #100082766.
- I graduated from Dalhousie University in Halifax, Nova Scotia, B.Eng., 2004.
- I have practiced my profession in the mining industry continuously since graduation.
- My relevant experience with respect to mining includes 16 years' experience in the mining sector covering underground mine engineering, operations, construction, and project management. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I have visited the McIlvenna Bay Project site on April 20, 2020.
- I am responsible for Sections 1.9, 1.10, 15.0, 16.0, 21.1.2, 21.2.1, 25.5, 25.6, and 26.3.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with the McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Denis Flood, P. Eng.

28.5 Manoj Patel, P.Eng.

I, Manoj Patel, P.Eng. am employed as a Project Manager with Halyard Inc. located at #501-212 King St. W., Toronto, ON M5H 1K5 Canada. This certificate applies to the technical report titled NI 43-1-1 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "Technical Report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of Professional Engineers of Ontario, membership #100190815.
- I graduated from McMaster University, Hamilton, ON, in 2006 with a Bachelor of Engineering (Mechanical).
- I have practiced my profession in the mining and mineral processing industry for 13 years since graduation.
- My relevant experience includes having been directly involved in mechanical engineering, mineral processing plant design, infrastructure design, project management, and cost estimating in the mining and mineral processing industry. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I have visited the McIlvenna Bay Project on January 16, 2019.
- I am responsible for Section 1.12, 18.0 (except 18.8), 21.1.3, 21.1.4, 21.2.3, 25.8, and 26.5.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with the McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Manoj Patel, P.Eng.

28.6 Jocelyn Howery, M.Sc., PAg

I, Jocelyn Howery, M.Sc., PAg am employed as the Hydrology Division Manager with Canada North Environmental Services Ltd. located at 211 Wheeler St, Saskatoon, SK S7P 0A4 Canada. This certificate applies to the technical report titled NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "Technical Report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing with the Saskatchewan Institute of Agrologists, membership #99931.
- I graduated from The University of Alberta, Edmonton, AB, M.Sc., 2010.
- I have practiced my profession as a hydrologist continuously since graduation.
- I have sufficient relevant experience having been directly involved as a project manager and senior hydrologist in numerous environmental baseline assessments, environmental performance reports (EPRs), and environmental monitoring programs (EMPs) for the mining sector. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43–101.
- I have visited the McIlvenna Bay Project in May 2013, September 2013, and October 2014.
- I am responsible for Section 1.14, 20.0, 25.10, and 26.7.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Jocelyn Howery, M.Sc., PAg

28.7 Alex McIntyre, P. Eng.

I, Alex McIntyre, P.Eng. am employed as a Senior Engineer with Knight Piésold Ltd. located at 1657 Main Street West, North Bay ON P1B 8G5 Canada. This certificate applies to the technical report titled NI 43-101 Technical Report, Pre-feasibility Study for the McIlvenna Bay Project (the "technical report") with an effective date of 12 March 2020 and a report date of 27 April 2020 and I hereby certify that:

- I am a member in good standing of Professional Engineers of Ontario, membership #100125386.
- I graduated from Carleton University, Ottawa ON., B.Eng., 2007.
- I have practiced my profession in the mining industry continuously since graduation.
- My relevant experience with respect to tailings management, geotechnical design, and earthworks construction includes 12 years' experience in the mining sector. As a result of my experience and qualifications, I am a Qualified Person as defined in NI 43-101.
- I am responsible for Tailings related items described in Sections 18.8, 21.1.5, and 21.2.5.
- I am independent of Foran Mining Corporation as described by Section 1.5 of the instrument.
- I have had no previous involvement with the McIlvenna Bay project.
- 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: April 27, 2020

"Signed and sealed"

Alex McIntyre, P. Eng.