



904 409 Granville St. Vancouver, BC V6C 1T2

Mcllvenna Bay Project



Technical Report on the Feasibility Study for the Mcllvenna Bay Project, Saskatchewan Canada

Prepared For:

Foran Mining Corporation

Prepared By:

Mr. Mark Hatton, P.Eng., Stantec Consulting Ltd.

Mr. William Lewis, B.Sc., P.Geo, Micon International Limited

Mr. Lyn Jones, P.Eng., Blue Coast Research

Mrs. Jocelyn Howery, M.Sc., P.Ag., Canada North Environmental Services

Mr. Michael Franceschini, P.Eng., Halyard

Mr. Alex McIntyre, P.Eng., Knight Piésold Consulting

Mrs. Kathy Kalenchuk, Ph.D., P.Eng. (ON, BC, SK & MB), PE (MT)

Date of Report:

April 14, 2022

Effective Date:

February 28th, 2022

Project Number:

169521520



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Acronyms and Abbreviations

AA	Atomic Absorption
AACE	Association for the Advancement of Cost Engineering
ADEX	Advance Development and Exploration Program
ALS	ALS Group
ASA	Aquatic Study Areas
Base Metallurgical	Base Metallurgical Laboratories Ltd.
BEV	Battery Electric Vehicle
BHEM	Borehole Electromagnetic Surveys
BHP Billiton	Billiton Metals Canada Inc.
BLE	Bluetooth Low Efficiency
BML	Base Metallurgical Laboratories
Bondar-Clegg	Bondar-Clegg & Company Ltd.
BP	Boreal Plain
BWI	Bond Ball Work Indices
Cameco	Cameco Corporation
Canmet	Canada Centre for Mineral and Energy Technology
CanNorth	Canada North Environmental Services
CEAA	Canadian Environmental Assessment Act
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CM	Construction Management
CMC	Carboxymethyl Cellulose
Copper Reef	Copper Reef Mining Corporation
CR	Crown Reserve
CSM	Conceptual Site Model
CSZ	Copper Stockwork Zone
DTM	Digital Terrain Model
EAB	Saskatchewan Environmental Assessment Branch
Eco-Tech	Eco-Tech Laboratories
EDC	Engineering Design Criteria
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EM	Electromagnetic
Esso	Esso Minerals Canada
ETP	Effluent Treatment Plant
FA-AA	Fire Assay Atomic Absorption
FFGB	Flin Flon Greenstone Belt
FID	Final Investment Decision
FS	Feasibility Study
FW	Copper Stockwork Footwall Zone
G&A	General and Administration



G&T	G&T Metallurgical Services Ltd.
Ga	Billion years ago
GCL	Geosynthetic Clay Liner
Geosight	Geosight Consulting Canada
Geospark	Geospark Consulting Inc.
HLEM	Horizontal Loop EM
HLJV	Hanson Lake Joint Venture
HudBay	HudBay Minerals Inc.
IAA	Impact Assessment Act
IBC	Isobutyl Carbinol
IRR	Internal Rate of Return
L3	Lens 3
LHD	Load-Haul-Dump
LNG	Liquified Natural Gas
LSA	Local Study Area
LTE	Long Term Evolution
MARS	Mineral Administration Registry Saskatchewan
MBS	Sodium Metabisulfite
MEL	Mechanical Equipment List
MEMS	Micro-Electro-Mechanical System
MIBC	Methyl Isobutyl Carbinol
MOE	Ministry of Environment
MORB	Mid-Ocean Ridge Basalt
M'Ore	M'Ore Exploration Services Ltd.
MS	Massive Sulphide
MSZ2	Massive Sulphide Zone 2
MTO	Material Takeoffs
MUP	Miscellaneous Use Permit
NPV	Net Present Value
NSR	Net Smelter Return
NTS	Canadian National Topographic System
OEM	Original Equipment Manufacturer
Parrex	Parrex Mining Syndicate
PAX	Potassium Amyl Xanthate
PBCN	Peter Balantyne Cree Nation
PCS	Process Control System
PDC	Process Design Criteria
PEA	Preliminary Economic Assessment
PFD	Process Flow Diagrams
PFS	Prefeasibility Study
PGNAA	Prompt Gamma Neutron Activation Analysis



PMF	Probable Maximum Flood
QP	Qualified Person
RQD	Rock Quality Designation
RSA	Regional Study Area
SAB	Semi-Autogenous Ball Mill
SAG	Semi-Autogenous Grinding
SARA	Species At Risk Act
SD	Standard Deviations
SG	Specific Gravity
SKMOE	Saskatchewan Ministry of Environment
SLD	Single Line Diagrams
SMDC	Saskatchewan Mineral Development Corporation
SOCC	Species Of Conservation Concern
SRC	Saskatchewan Research Council's Geoanalytical Services Laboratory
SSA	Site Study Area
Steward Group	Steward Holdings Group Limited
Strong Pine	Strong Pine Energy Services
SZ	Stringer Zone
TAWMP	Tailings Area Water Management Pond
TDEM	Time-Domain Electromagnetic Surveying
TEM	Time-Domain EM
Terra	Terra Mineralogical Services Inc.
TerraMin	TerraMin Research Labs Ltd.
TSF	Tailings Storage Facility
TSL	TSL Laboratories Inc.
UG	Underground
UWZ	Upper West Zone / Upper West Massive Sulphide
VGf	Vibrating Grizzly Feeder
VMS	Volcanogenic Massive Sulphide
VOD	Ventilation On Demand
VTEM	Versatile Time-Domain Electromagnetic
VTEMplus	Versatile Time-Domain Electromagnetic
Western Nuclear	Western Nuclear Mines Ltd.
XRAL	XRAL Laboratories Ltd.



1.0 SUMMARY

1.1 Introduction

The McIlvenna Bay Project is located in east-central Saskatchewan approximately 375 km northeast of Saskatoon, and 85 km west of Flin Flon, Manitoba. The site is accessible year-round via an 18 km long all-weather road connected to Saskatchewan Provincial Highway 106.

The Feasibility Study (FS) outlines a project designed to be a decline / shaft underground mining operation utilizing long-hole mining methods for ore extraction at a nominal 4,200 tonne per day (tpd). Ore is expected to be processed using conventional single stage crushing circuit with a semi-autogenous grinding (SAG) mill and ball mill design. Crushing will be followed by grinding and flotation circuits to produce both copper and zinc concentrates for transportation from site to Flin Flon for shipment by rail to Canadian smelters and/or offshore.

The Project envisaged by the FS includes a 4,200 tpd (nominal) underground mine, on-site crushing and mineral processing facilities, paste plant, filtered tailing storage facility, and other supporting project infrastructure such as water management / treatment facilities, offices, workshop, warehouse, mine dry, and first aid facilities.

The deposit at McIlvenna Bay includes several zones and two distinct styles of mineralization, typical of volcanogenic-hosted massive sulphide (VMS) deposits: Massive Sulphide (MS) mineralization, and stockwork-style mineralization in the Copper Stockwork Zone (CSZ). The MS is a continuous mineralized horizon which averages 3.5 m in thickness, while the CSZ averages a thickness of 12.0 m. The MS and the underlying CSZ are generally in contact throughout the deposit, giving the bulk of the deposit an average thickness of 15.5 m overall. The deposit plunges at approximately -40° from surface over a plunge length of approximately 2,000 m.

Pre-production capital costs for the project are C\$376M, followed by C\$473M of sustaining costs for a total capital cost of C\$849M. An additional C\$8.2M is required for mine closure. LOM average operating costs of C\$73.55/t ore processed are expected, translating to an overall operating cost of C\$91.94/t ore processed if the capitalized sustaining costs are included.

The project economics indicate a pre-tax net present value (NPV) at a 7.0% discount rate of C\$678M and a post-tax NPV of \$466M. The pre-tax internal rate of return (IRR) is 26% and the post-tax IRR is 22%. The Project generates a life-of-mine (LOM) undiscounted post-tax free cashflow of \$1,180M.

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.



Preparation of the FS included contributions from consulting firms listed below.

- Stantec Consulting Limited
- Micon International Inc. – Mineral Resource Estimate
- Blue Coast Research – Process Metallurgy
- Base Metallurgical Laboratories (BML) – Metallurgical testwork
- Canada North Environmental Services – Hydrology and Environmental Studies
- Knight Piésold – Tailings Storage Facility design and costs
- Halyard Inc. (Halyard) – Surface Infrastructure and Process Plant design and costs
- RockEng – Underground mine geotechnical designs
- MineFill Inc. – Design of underground backfill reticulation

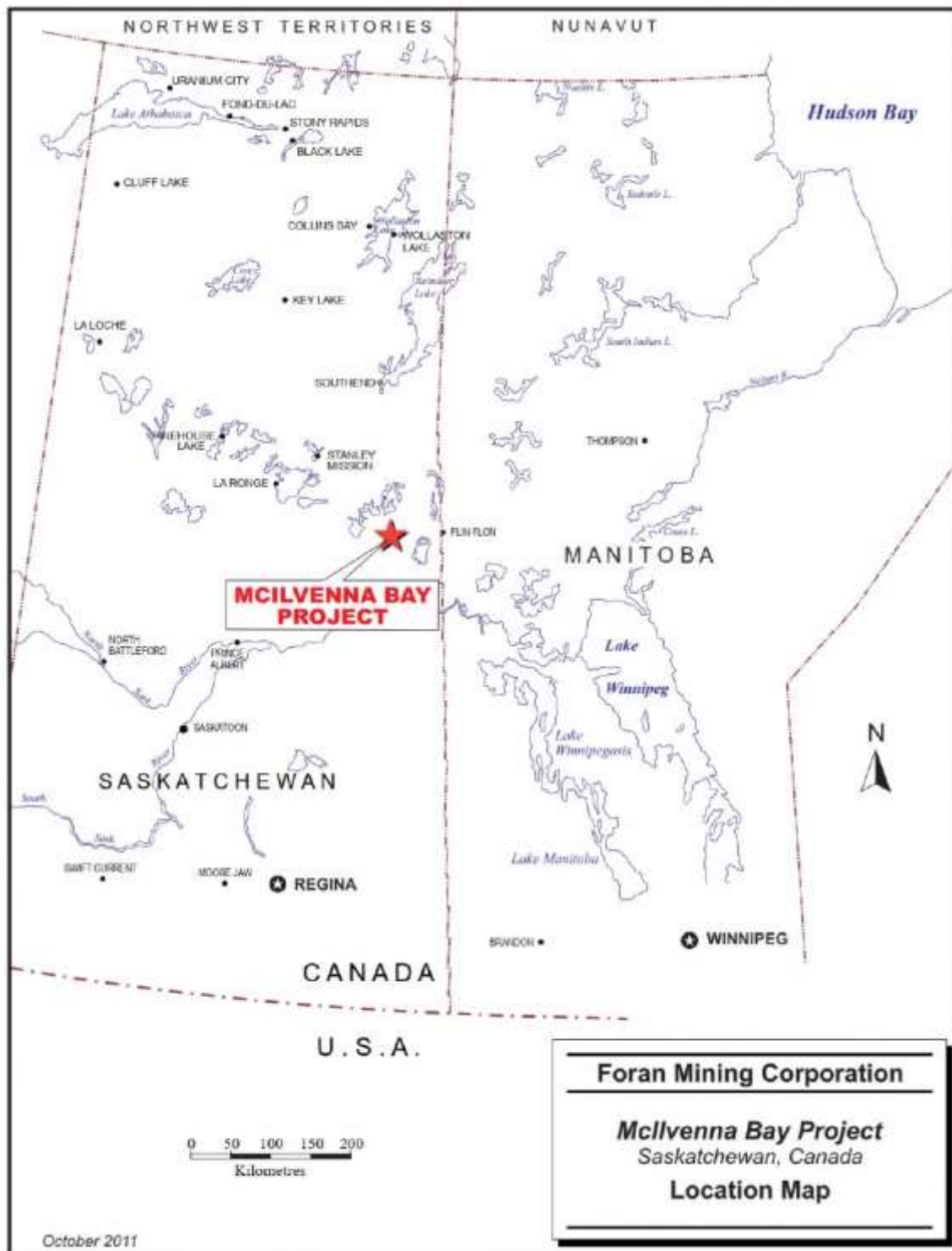
1.2 Project Description and Location

The McIlvenna Bay Project lies within Foran's McIlvenna Bay property, which is located approximately 1 km south of Hanson Lake, in northeastern Saskatchewan (Figure 1-1). The McIlvenna Bay property comprises 38 claims totaling 20,954 ha and is approximately 375 km northeast of Saskatoon and 85 km west of Flin Flon, Manitoba.

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 within the property boundaries.



Figure 1-1: Mclivenna Bay Project Location



1.3 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Mcllvenna Bay Project site is accessible via an 18 km all-weather gravel road that connects to Saskatchewan Provincial Highway 106, approximately 85 km west 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 will be generated on site until it can be supplied by SaskPower via overhead lines from the hydroelectric station at Island Falls, Saskatchewan.

Figure 1-2: Mcllvenna 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 beds, with cooking and eating amenities, along with an office, core shack, shop, and core storage facility. An all-season gravel road built through the property supports 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 has been brought back into service to support construction activities. Construction of a dedicated transmission line for the mine and mill will be required 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 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 350 mm of rain and 1,450 mm of snow. On average there are 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 20 m high. Soil thickness on the limestone ridges is minimal, with occasional rock exposure. Vegetation is dominated by large conifer and poplar trees. Below the cliffs are poorly drained muskeg swamps with scattered tamarack and black spruce. There are numerous lakes and ponds throughout the surrounding area.

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. This led to the development of the Hanson Lake (Western Nuclear) Mine, which 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 8 km 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 1,200 m long INPUT anomaly, striking east-southeast 1 km 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.

On acquisition of the property in 1998, Foran embarked on a diamond drilling program to test new targets, as well as to infill the existing drill pattern on the McIlvenna Bay Deposit. Drilling continued during the winter of 1999–2000, but exploration work on the



property ceased after 2000. Foran acquired a new option agreement in 2005 and resumed work. In 2018, Foran completed a large two-phase infill and expansion drill program designed to convert a significant portion of the known inferred resource base of the deposit into the Indicated category. A Prefeasibility Study for the Mcllvenna Bay Deposit was released in 2020.

1.4.1 Prior Mineral Resource and Reserve Estimations

Prior to the Mcllvenna Bay Project being optioned by Foran in 1998, mineral resource or reserve estimations had not been conducted on the property. Prior to this Technical Report, Foran has issued NI 43-101 Technical Reports containing mineral resource estimates for the Mcllvenna Bay Project.

The previous historical estimates will not be presented in this Technical Report, because the Micon QPs for this report have not reviewed any of the previous mineral resource estimates or assessed them for compliance with current CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014). Foran is not relying on the previous resource estimates which are superseded by the current estimate contained in Section 14.0 of this Technical Report. The current mineral resource estimate contained in Section 14.0 has an effective date of 06 September 2021. It was previously disclosed in a Technical Report dated 25 November 2021 and was amended on 31 January 2022.

1.4.2 Production from the Mcllvenna Bay Project

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

1.5 Geological Settings and Mineralization

1.5.1 Regional Geology

The Mcllvenna 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 is part of the Reindeer Zone, a subdivision of the Trans-Hudson Orogen which was created in a continental-scale tectonic event between 1.84 and 1.80 billion years ago (Ga) when the Superior and Hearne Archean Cratons collided. (Syme, et al. 1999)

The FFGB contains eight known geographically separate juvenile island arc volcanic assemblages (blocks), each being 20 km to 50 km across. From east to west, these are the Snow Lake, Four Mile Island, Sheridan, Flin Flon, Birch Lake, West Amisk, Hanson Lake, and Northern Lights blocks (Zwanzig and Schledewitz 1997, Maxeiner, et al. 1999). These blocks 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



subblocks, usually bounded by local to regional fault systems. Correlation of stratigraphy between subblocks 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 Wisconsin ice-flow indicators is south-southwest indicating the ice was flowing from a Keewatin dispersal centre. The resulting tills are thin and generally reflect local bedrock lithologies (McMartin, Henderson and Neilson 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 bounded 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 D2 structural event resulted in tight to isoclinal, southwest plunging F2 folds and local southwest verging mylonite zones. D3 deformation resulted in tight north trending folds followed by a brittle D4 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, because 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 and geophysical surveys.

The stratigraphy of the deposit area, divided into six formations, has been defined over a 2 km strike length by a total of 239 drill holes. The lowest formation intersected by drilling both structurally and stratigraphically is the Mcllvenna Bay Formation, the host of the Mcllvenna Bay Deposit. The Mcllvenna Bay Formation is overlain to the north by the Cap Tuffite Formation. The Mcllvenna Bay Formation and the Cap Tuffite Formation may be genetically related but are treated as separate units because the formations are temporally distinct. This is demonstrated by the positioning of the Mcllvenna Bay Deposit—an exhalative horizon and thus reflective of a period of clastic and volcanosedimentary quiescence—between these two units.

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 that grades laterally into iron formation. Capping the HW-A Formation is a thick unsorted bimodal package of mafic and felsic volcanics, 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 is the Davies Gabbro, represented by a number of sill-like plugs located 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 20 m and 30 m.

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

1.5.3 Mineralization

Mcllvenna Bay is a VMS consisting 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 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 Mcllvenna Bay Deposit includes five separate zones and two styles of mineralization that are mineralogically and texturally distinct, and typical of VMS deposits.

- 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, located between the Main Lens and Lens 3
 - the Copper Stockwork Footwall Zone (FW), which occurs as a separate lens underneath the CSZ for approximately 140 m of strike length and 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 Mcllvenna Bay Deposit. Phase I of this program commenced in December 1998 and was completed through the winter of 1998–1999. Fifty-five holes totaling 27,958 m were drilled during this program. 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 before being halted pending financing. Three holes totaling 2,938 m were completed in 2000, and an access road was constructed. The mineralization had been delineated to a maximum vertical depth of 1,230 m up to this period.

As of 31 May 2000, Foran had drilled an additional 59 holes totaling 33,350 m, 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 580 m, 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. This drilling was successful in extending the deposit an additional 300 m 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 150 m 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 27 November 2006 (Cook and Moore 2006). Seven diamond drill holes were completed for a total of 6,455 m. Drill holes were between 691.5 m and 1,298.4 m in length on sections 9400E through 9700E. The objective of the program was to tighten drill hole spacing and upgrade the Mineral Resources down plunge on MSZ2. A number of drill holes failed to intersect the deposit at depth, however Foran subsequently determined these holes 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 refinanced, 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 5,056 m. This program targeted a portion of the CSZ and was designed to infill and prove up the continuity over a portion of the zone in the central part of the deposit. 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, and RPA was retained to update the mineral resource estimate (Rennie 2011) for the CSZ. The zone was re-interpreted, using a nominal 0.5% Cu cut-off value and a minimum apparent thickness of 3 m. The other zones were largely unchanged, with the exception of Lens 4, which was incorporated into the footwall (FW). The re-inclusion of the CSZ resulted in a large increase in the total 2011 mineral resources when compared to the prior 2006 estimate.

Drilling resumed in August 2011 through to November 2011, with a total of 8,158 m completed in 18 holes. The purpose of the drill program was to infill the deposit to further increase the confidence in the resource, collect sample material for metallurgical testwork, 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 resurvey program was completed for all drill hole collars that could still be identified on the property. In addition, downhole gyroscopic surveys were conducted 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 versatile time domain electromagnetic (VTEM plus) and horizontal magnetic gradiometer (mag) over those areas of the McIlvenna Bay property not covered in 2007.



In 2012, Foran completed 3,825 m 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, to upgrade the classification and extend the known mineralization. Drilling was dominantly completed utilizing HQ-sized core to provide additional material for future metallurgical testwork. Geotechnical and hydrogeological studies were also conducted during the program.

Metallurgical testwork on the samples collected from the 2011 drilling was completed in June 2012. The work was completed by G&T Metallurgical Services Ltd. (G&T) of Kamloops, BC. Three composite samples, consisting of 516 kg of drill core, were created for each of three different mineralogical domains: the CSZ, MSZ2, and UWZ. The samples were then used in batch and locked cycle flotation testing, as well as determination of Bond Work Indices. 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 up-dip extension of the CSZ in the central part of the deposit that were accessible from the frozen muskeg.

Further exploration / drilling was not 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 this agreement Glencore contributed professional and technical services, assistance, guidance, and advice toward completing a Feasibility Study on the McIlvenna Bay Project, 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 resources as possible into Indicated categories which could then potentially be converted into reserves for the upcoming Feasibility Study.

In 2018, Foran conducted 26,827 m of drilling in 60 drill holes targeting the deposit. The program was completed in two phases, with 14,986.5 m in 32 drill holes (including several wedged holes) completed during the Phase I winter program and 11,840.5 m 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 that 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 the program included 3,733 m 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 (151.3 m) were drilled to characterize the proposed portal location, and four short vertical holes (104 m) were drilled for piezometer installations to quantify near-surface groundwater flow in the immediate deposit area.

Material for metallurgical testwork was collected from all Phase I and Phase II drill holes, with either a quarter or half of each sampled interval submitted for testing. Metallurgical work is being completed by Base Metallurgical Laboratories Ltd. (Base Metallurgical) of Kamloops, BC. A total of 1,441 g of drill core was provided from the 2018 drilling, supplemented with 712.4 kg of coarse rejects from assayed material from the 2018 program. Another 38.3 kg of core material from 2011 drilling was collected for HLS testing. Testwork currently under way is comprised of grind and flotation circuit tests, as well as DMS upgrading to maximize value.

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 using a drill on the pad and resurveyed using a True North Gyro.

To develop a larger library of rock density measurements across the deposit, Foran personnel completed 1,932 bulk density measurements both from 2018 drill holes, and core from 2007, 2011, and 2012, 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 1.0 g (0.5 g for skilled operators), and the larger sample sizes (often between 2 kg and 4 kg) minimized the error introduced by the 1.0 g precision. These bulk density samples are considered more representative of the actual density of the mineralized material in the ground, compared to 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 estimate which was the subject of an NI 43-101 Technical Report.

The Glencore Agreement was subsequently allowed to lapse, and further exploration was not conducted on the deposit until 2021. The 2021 exploration work and results are the subject of this report.



The 2021 winter drill program was focused on infill drilling, targeting the inferred portions of the resource to convert additional tonnes to the indicated category, as well as step-out drilling designed to expand the deeper parts of the deposit in the up-dip direction. The program was completed between January and July and consisted of 39 drill holes encompassing 27,298 m (including 11 wedges). The program included the use of oriented core technologies to better understand the structural setting of the deposit and all holes were surveyed using a gyro tool to provide accurate drill hole traces at depth.

The program was successful in significantly increasing the density of drill holes in the deeper part of the deposit and in expanding the mineralized horizons up-dip, along the upper plunge line of the deposit. The 2021 drilling continued to define a trend of thicker mineralization in the Copper Stockwork Zone in this area, which is often coupled with higher copper grades in the adjacent massive sulphide horizon. Ultimately, the drill program culminated in significant increases in the Indicated portion of the McIlvenna Bay resource.

During the 2021 program, Foran continued to collect bulk density measurements for all sample intervals within the deposit, to bolster the density database for the deposit. The measurements were made using the weight in air / weight in water method on complete sample intervals, similar to the process used in 2018.

During the winter of 2021, the immediate deposit area and projected extensions to the north-east were covered by ground EM surveys, in an effort to determine the potential for additional lenses and/or extensions of known lenses. The results of the survey have been modelled and testing of potential conductors is being planned for future programs.

1.7 Mineral Processing and Metallurgical Testing

Metallurgical testing of McIlvenna Bay samples was completed in several programs since 2012. Initial characterisation was completed at ALS Metallurgy with follow up programs at Base Metallurgical Laboratories (BML) in 2016, 2019, and 2021. Testwork has focused on two main mineralization styles: Copper Stockwork and Massive Sulphide, with the latter being further subdivided into Upper West and Z2 (MSZ2) zones. The Copper Stockwork Zone contains chalcopyrite as the main economic mineral, with minor amounts of sphalerite, and is considered moderate to hard in terms of grindability. The Massive Sulphide zone is of moderate grindability with primary sulphides as pyrite, sphalerite, and chalcopyrite, and lesser amounts of galena.

Testwork has focused on developing a flotation scheme to produce concentrate products from both zone composite material as well as blends of the zones. The flowsheet consists of a primary grind to an 80% passing size of 75 µm, sequential flotation of first the copper and then zinc minerals, regrinding of the rougher concentrates to 80% passing 20-25 µm, cleaner flotation, and then final concentrate dewatering.



The latest phase of the testwork, initiated in March 2021 at BML, was conducted to support the feasibility study and advance the metallurgy in four key areas.

- Development of the MBS depressant scheme to replace ZnSO₄/NaCN in the copper circuit
- Development of the pyrite flotation circuit to generate a low-sulphide tailings
- Further evaluation of blended composites including locked cycle testing
- Additional downstream testwork to support the feasibility level plant design.

Results of the testwork confirmed the flowsheet and further developed the understanding of the metallurgy using blended composites of copper stockwork and massive sulphide mineralization that are representative of the expected mill feed composition. In addition, final product samples of concentrate and tailings were used to evaluate settling and filtration characteristics and generate process design data.

Open circuit cleaner and locked cycle flotation test results from the metallurgical studies were used to develop feed grade-based models for copper, zinc, silver, and gold for both concentrate grade and recovery. 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 presented in Table 1-1.

Table 1-1: LOM Average Recovery Rates (%)

Circuit	Copper	Zinc	Gold	Silver
Massive Sulphide	82.4	84.4	82.9	54.6
Copper Stockwork	95.2	47.0	92.7	76.6
Blended Mill Feed	91.1	79.8	88.6	63.2

Over the LOM, the average copper grade in the copper concentrate is forecast to be 28% and the average zinc grade in the zinc concentrate is forecast to be 50.0%.

1.8 Mineral Resource Estimation

This section presents the updated mineral resource estimate for Foran's McIlvenna Bay Project in Saskatchewan. The updated mineral resource estimate is based on Foran's drilling database, which includes both previous drilling and Foran's drilling in 2021. Micon's QPs have conducted the mineral resource estimate for disclosure in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

The 2021 drilling program was designed not only to improve the confidence of the known inferred mineralization such that it could be upgraded to indicated, but also to potentially increase the mineral resources at depth. Previous iterations of the resource model have



been completed and published since 2010, with all previous resource estimations now superseded by the current 2021 estimate presented in this section.

The updated mineral resource estimate presented herein is summarized in Table 1-2. The effective date of this mineral resource is 06 September 2021, and the resource is reported using a Net Smelter Return (NSR) cut-off of US \$60/t.

Table 1-2: Mineral Resources for Mcllvenna 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	MS	10.75	1.01	6.17	0.41	0.53	26.56
	CSZ	22.74	1.31	0.38	0.02	0.37	9.14
	SZ	1.19	1.26	0.52	0.07	0.31	12.97
	L3	2.57	0.82	3.07	0.14	0.25	14.51
	FW	1.80	1.42	0.59	0.04	0.45	8.84
	Total	39.06	1.20	2.16	0.14	0.41	14.39
Inferred	MS	1.56	0.65	6.51	0.46	0.29	27.77
	CSZ	3.48	1.08	0.79	0.03	0.25	10.50
	Total	5.04	0.94	2.56	0.17	0.27	15.85

Notes:

- Effective date 06 September 2021; CIM definitions were followed for Mineral Resources.
- The mineral resource is estimated based on 240 diamond drill holes and a NSR cut-off of US\$60/t. NSR values were derived, and high-grade caps were applied as per the discussion in Estimation Methodology and Parameters and include provisions for metallurgical recovery and estimates of current shipping terms and smelter rates for similar concentrates. Metal prices used are US\$4.25/lb. Cu, US\$1.35/lb. Zn, US\$1,800/oz. Au, and US\$25.00/oz. Ag. Lead contributes no value.
- Rock density was interpolated for each block based on measurements taken from core specimens, with an average value of 3.59 g/cm³ for the main MS lens and 2.87 g/cm³ for the CSZ.
- Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- The block model grades were estimated using the Ordinary Kriging interpolation method, with search parameters derived from geostatistical analysis performed within the mineralization wireframes. Variogram ranges are from 65 m to 85 m for Au and Ag in the major axis and up to 100 m to 120 m for Cu and Zn.
- Micon has not identified any legal, political, environmental, or other factors that could materially affect the potential development of the mineral resource estimate.
- The mineral resource estimates are classified according to the CIM Definition Standards, which define a Mineral Resource as "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 characteristics of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge including sampling."
- The mineral resource was categorized based on geological confidence into inferred and indicated categories. An inferred mineral resource has the lowest level of confidence. An indicated mineral resource has a higher level of confidence than an inferred mineral resource. It is reasonably expected that the majority of the inferred mineral resources could be upgraded to indicated mineral resources with additional infill drilling.

1.9 Mineral Reserve Estimate

The Mcllvenna Bay Deposit Mineral Reserve estimate is based on indicated resource material identified in the block model provided by Foran (MCBAYBM_MICON2021NSR_ALL, issued on 09 September 2021). All resource material in the block model that was classified as inferred was assigned a 0% grade.



The Mineral Reserve estimate consists of selected portions of the Indicated Resource that are above a US \$90.24/tonne Net Smelter Return (NSR) cut-off value. This cut-off value was applied when generating the stoping shapes.

The Mineral Reserve is categorized as either Massive Sulphide or Copper Stockwork. It is assumed that both transverse and longitudinal retreat stope mining methods, as well as Avoca mining, will be used. The assumed mining rate is nominally 4,200 tonnes per day (t/d). A total dilution of approximately 22.2% is projected for the longhole stopes.

Extraction (mining recovery) is estimated at a combined 93.6% for longhole mining and ore development.

The Mineral Reserve for the Mcllvenna Bay Deposit is estimated at 25,703,062 tonnes, grading 2.39% Zn and 1.23% Cu. This Mineral Reserve estimate includes marginal tonnes, which is classified as marginal development or incremental stoping. This Mineral Reserve estimate also includes modifying factors including dilution and mining recovery.

Estimates of mineralization and other technical information included herein have been prepared in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price assumptions
- Changes in interpretations of mineralization geometry and continuity of mineralization zones
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions
- Input factors used to assess stope dilution and recoveries
- Assumptions the operation can obtain all required permits to operate
- Assumptions regarding social, permitting, and environmental conditions
- Additional infill or step-out drilling

Table 1-3 presents the estimated Mineral Reserve.



Table 1-3: Mineral Reserve Estimate

Classification	Recovered Ore Tonnes (tonnes)	Grade Zn (%)	Grade Cu (%)
Massive Sulphide	10,135,305	5.43%	0.99%
Copper Stockwork	15,567,756	0.41%	1.39%
Total	25,703,062	2.39%	1.23%

Notes:

9. Effective date 15 February 2022, CIM definitions were followed for Mineral Reserves.
10. Mineral Reserves include transverse, longitudinal, and Avoca stopes, as well as ore development, marginal development, and incremental stopes.
11. Stopes were estimated at a cut-off value of US \$90.24/tonne NSR.
12. Marginal tonnes were estimated at a cut-off value of US \$60.55/tonne NSR.
13. A minimum mining width of 3.0 m was applied for all stoping.
14. Numbers may not sum due to rounding.
15. NSR Reserve Prices (\$US); Cu \$3.50/lb, Zn \$1.20/lb, Ag \$22.50/oz, Au \$1600/oz

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, political, or other relevant factors that could materially affect the Mineral Reserve estimate.

1.10 Mining Methods

Underground (UG) access to the McIlvenna Bay Deposit will initially be via a ramp from surface. A 2.43 m × 7.32 m rectangular shaft will also be constructed, commencing in Year 5, to reduce the truck haulage distances and the time required for the workforce to access the lower levels. At the time of this report, Foran is constructing the portal and developing a ramp to the 0090 Level, along with a bulk sample and an exploration / diamond drill drift on the 0090 Level. This is called the Advance Development and Exploration Program (ADEX). The mine development and life of mine plan presented in the FS will commence at the completion of the ADEX program.

The capital costs for the ADEX program are excluded from the FS financials.

Thirty-five levels, spaced at 30 m intervals sill to sill, are planned for the McIlvenna Bay Deposit. Lateral development will be concentrated in the first four years to establish the initial production areas, UG infrastructure, and the permanent ventilation system.

The McIlvenna Bay Deposit will be extracted using conventional longhole mining methods including sublevel transverse and longitudinal stoping and Avoca stoping. The ore body geometry and rock characteristics indicate these mining methods are appropriate for safe and efficient production. Ore will be drilled using a top hammer drill, blasted, and then mucked using battery electric (BEV) load-haul-dump (LHD) vehicles.

Ore will be hauled to surface using BEV haul trucks early in the mine life and will be hauled to the rock breaker stations feeding the shaft in the latter stages of mine production. Waste will be either hauled to surface or hauled to an active production level where it will be used as backfill.

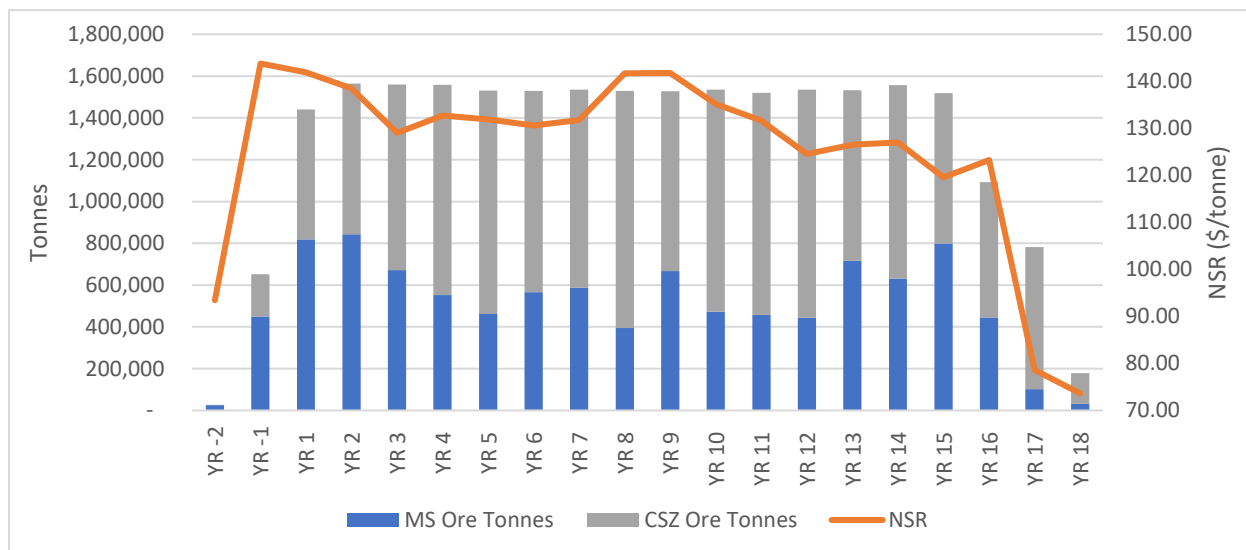


Mine dewatering will be completed using a multi-level clean water system. Main sumps will be located on the 0060, 0420, 0780, and 0960 Levels. The 0060 Level sump will be designed to collect surface water via the ramp and most of the water transmitted through the sandstone layer. This sump will feed water into the process water system and pump the surplus water to surface. The 0420, 0780, and 0960 Level sumps will collect any remaining surface water, plus the process water from mining activities. The run-of-mine water will decant through membranes; with the clean water being pumped to the next main sump (i.e., 0960 Level to 0780 Level, and 0780 Level to 0420 Level). The 0420 Level sump will feed water into the process water system and pump the surplus water to surface. The residual solids in the sumps will be collected and placed into a nearby stope for disposal.

Transverse stopes will be backfilled with paste fill using filtered tailings from the processing facility. Avoca stopes will be backfilled with waste rock generated from underground development. Conventional trackless mining equipment will be used to execute lateral development required to access the ore body. Ore will be produced at a nominal rate of 4,200 tonnes per day (tpd) with a mine life of approximately 20 years, including an initial ramp-up period of two years.

Figure 1-3 below outlines the annual mine production by ore zone. Both zones will be mined and processed together but are illustrated separately in Figure 1-3 because the metallurgical performance is dictated by the balance of tonnes / grades from the two zones.

Figure 1-3: Underground Production Profile with NSR



Massive Sulphide (MS) Ore tonnes include Marginal Ore Tonnes

The mine 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 determined for the operation identified a peak on site complement of up to 149 personnel.

The underground mobile fleet will be diesel equipment, except for the LHDs and haulage trucks. The fleet of mobile mining equipment will include LHDs, haulage trucks, drill jumbos, bolters, mechanical bolters, top hammer drills, a shotcrete unit and ancillary equipment. Some key mobile equipment units will be purchased through a lease-to-own program. Since the mine has access to surface, maintenance of the UG fleet will occur in the surface shop until the 0420 Level maintenance shop is constructed. It will service equipment for the life of the operation except for the haulage trucks, which will continue to be maintained in the surface shop.

The mine ventilation system is designed as a predominately negative or “pull” system and is designed using Canada Centre for Mineral and Energy Technology (Canmet) requirements for the described mobile fleet. It takes the use of BEVs into account, for a total airflow requirement of 250 m³/s. The design includes an exhaust system and a fresh air system complete with a heating plant located between the fresh air raise and a centrally located exhaust raise. The fresh air will be partially heated using a propane-fired heating system and by absorbing heat from the exhaust air using a heat exchanger. The ramp and internal fresh air raise will distribute the air throughout the mine. Each level will have two exhaust raises on the extremities. The return air will be collected on the 0060 Level and exhausted to surface. The upper portion of the ramp—from 0060 Level to surface—will be ventilated with air from the surface fresh air raise to provide heated air to the portal.

The ore and waste handling system will begin with LHD units hauling material from the active heading or stope to the nearest remuck.

Ore will be loaded into the haulage trucks at the remuck and hauled to surface or to the 0570 Level rock breakers, once the shaft is operational. The 0570 Level rock breakers will be equipped with a remotely operated rock breaker. The grizzly openings will be 500 mm by 500 mm. The sized ore will be loaded onto a conveyor on the 0600 Level and transferred to the shaft for skip loading.

Waste will be loaded into the haulage trucks at the remuck and hauled to surface or to available Avoca stopes or a secondary transverse stope in the backfill cycle.

1.11 Recovery Methods

Mill feed for the McIlvenna Bay Project will be processed in a purpose-built mineral processing facility to produce separate copper and zinc concentrates. The plant would provide on-site treatment of approximately 25.7 million tonnes of copper-zinc ore at a processing rate of 4,200 tpd (1.512 Mtpa), based on the current mine plan.

A key finding of the 2018–2019 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. Similar grind targets and reagent recipes are recommended for the processing of the two main mineralization types. Thus, a co-processing approach is considered a pragmatic method of processing as long as effective upstream blending strategies are implemented and managed.

Initially, ore will be hauled to surface using 42-tonne trucks and dumped into a surface crushing facility. As the mine development continues a shaft hoisting system will be constructed to feed minus 500 mm ore onto a new overland conveying system to feed the surface crushing facility.

Ore will be crushed to a nominal 100% passing 150 mm, (80% passing 100 mm) size. Conveyors will transfer coarse crushed material to one of two surface stockpiles in preparation for the SAB grinding circuit.

The selected grinding circuit consists of a SAG and Ball Mill combination designed to accommodate the variable hardness expected from blending 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.

Cyclone overflow slurry from the grinding area will be directed to the flotation section for sequential copper 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.

A single vertical pressure filter will be used to dewater both copper and zinc thickened concentrates in batches to provide two stockpiles of product filter cake suitable for transportation to toll smelters.

A desulphurization rougher flotation circuit will 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. Desulphurized flotation tailings will be used to produce a filtered cake suitable for stacking at the surface tailings facility.

1.12 Project Infrastructure

The McIlvenna Bay Project is a compact site with major features including the tailings storage facility, ore and waste dumps, water treatment infrastructure, and buildings such as administration and dry, gate house, and truck shop. The process facilities will include the primary crusher building, covered ore stockpiles, mill building, and paste plant building. The site will contain a 350-person camp with associated canteen, recreation facilities, and domestic water treatment infrastructure.

Foran has constructed infrastructure to support ongoing ADEX, including all-season camps with combined sleeping accommodation for 88, fuel storage, communications, domestic water, septic, canteen, core shack, construction office, and first aid / mine



rescue trailer. Site clearing and portal development have commenced. A temporary truck shop has been erected and laydown area prepared. The first phase of the lined waste rock dump is constructed, and Stage 1 of the contact water pond is in final permitting. The existing SaskPower distribution line has been energized and power is distributed via a containerized construction power e-house.

The existing gravel road between the mine site and SK-106 will be used for transportation of mine personnel and general supplies over 18 km. The road will require cleaning along its edges to recover the road to the width used during the quarrying operation.

A light vehicle parking lot to accommodate 156 vehicles will be located adjacent to the administration / dry complex and on-site camp. An additional 25-vehicle visitor parking will be located in front of the administration / dry complex.

A 35 m diameter helipad will be constructed adjacent to the mine site entrance and will be maintained over life-of-mine for emergency access.

A four-bay truck shop / warehouse will be housed within a 42 m W × 45 m L × 15 m H pre-engineered building. The mezzanine level will include offices, meeting rooms, washrooms, and kitchenette, as well as electrical and mechanical utility rooms and additional storage. Over the vehicle maintenance bays are two overhead cranes (20/5 t). The mezzanine level utility area is serviced by a 2 t hoist with access to the outdoor laydown area.

The administration office and mine dry are combined in a two-story modular building 18 m W × 36 m L. The lower level will accommodate 250 personnel and will include the men's and women's dries, canteen and lunchroom, mine rescue, laundry, and janitorial. The upper level will include reception, office units / open office area, meeting rooms, training room, kitchenette, washrooms, electrical, IT server, and mechanical utility rooms.

The existing gatehouse will remain in place during the construction and production phases. The gatehouse is located at the entrance of the mine site and controls access in and out of the site as well as the truck scale operation.

The temporary explosives storage magazines will be located north of the mine site a minimum of 650 m from adjacent structures, and with a partial earth berm enclosure. Emulsion will be stored in a 2.4 m W × 6 m L × 2.5 m H type 4 outdoor magazine.

A fuel storage and distribution facility will be located adjacent to the portal. This facility will store the diesel and low-sulphur diesel tanks, dispensing equipment, and an oil / water separator.



Waste from site such as plastic, paper, cardboard, scrap, and rubber will be collected and stored in the garbage disposal area. The waste will be picked up on a set schedule and sent offsite for disposal.

A 10,100 m² laydown area will be constructed of compacted granular material and located south of the truck shop. This laydown area will serve as the construction staging area during the surface and portal construction phases.

The headframe will provide access for personnel entering the mine in addition to hoisting ore to the surface. The headframe design includes backlegs, head sheaves, skip dumps, and ore bin. A deflector gate will direct hoisted material to a bunker on the ground. The tower of the headframe will stand 52.0 m tall and will house the head sheaves and skip-dumping equipment. The headframe will be constructed on site with structural steel. The collar house will be attached to the headframe. The hoist house will be constructed on site with structural steel.

The primary crushing circuit will be housed within a pre-engineered building. The building will be adjacent to a 6 m high concrete retaining wall to allow for dumping of underground haul trucks into the surge bin. The crushing building will be equipped with a 10 t overhead crane. A modular control room is located adjacent to the modular crushing plant.

The copper and zinc crushed ore stockpiles will be housed within a common fabric building. Each stockpile will have 6,000 t capacity and will be reclaimed using a vibratory feeder located in a corrugated arch multiplate steel tunnel.

The process plant will be housed within a pre-engineered building. Lean-to structures are attached for reagents and concentrate loadout. The process building will be equipped with two overhead cranes (20 t and 10 t). A modular control room will be located outside the process plant at the mill operating floor elevation. Operators will have direct access into the process building via covered walkway. The pebble transfer tower will be clad and space allocated for a possible future pebble crusher. Exterior process equipment will be located in a concrete containment bund.

A pre-assembled containerized assay lab will be located adjacent to the process plant building.

The dry stack tailings and paste plant will be housed within a pre-engineered building. A lean-to structure is attached for non-pyrite tails loadout to the Tailings Storage Facility (TSF). The paste building will be equipped with a 10 t overhead crane. A modular control room will be located on the second level. Exterior tailings slurry storage tanks will be located within a concrete containment berm.

The TMF storage shed is a fabric building and will provide overnight / inclement weather storage for dewatered non-acid generating tails.



The average power demand for the project will increase over the initial years of commercial production as the underground mine is developed, ranging from 13.5 MW to 17.7 MW. The existing 25 kV SaskPower distribution line has been re-energized and is providing 1.2 MVA capacity to the construction e-house. Phase I power will be supplied primarily through a power purchase agreement utilizing an onsite liquified natural gas (LNG) 19.4 MW power plant, supplemented by the existing 25 kV distribution line. The LNG power plant will be equipped with a heat recovery system for building heat. Phase II power will be provided from the Saskatchewan provincial grid via 77 km of new 138 kV transmission line from the Pelican Narrows substation. An onsite 138-13.8 kV substation will be constructed.

Power costs have been calculated based on a blend of energy consumption from the grid and onsite power generation and average \$115/MW.h over the life-of-mine. An additional rate rider of \$7.50/MW.h has been added to grid power to account for carbon emissions offset.

A lined ore and waste rock pad will be constructed in three stages in the vicinity of the underground mine portal and the process plant. Total storage capacity will be 1,700 kt.

The site water management strategy is to divert or deflect non-contact surface runoff water away from the project site to the maximum extent possible and collect and treat site-influenced contact water. Contact water will be collected, treated, tested, and reused for processing and mining operations. Surplus contact water will be treated through an effluent treatment plant and discharged north into Winn Bay (Hanson Lake). The primary and secondary containment ponds will be 115 m W × 155 m L × 2.5 m deep. Each pond will have a 41,000 m³ capacity providing settling and polishing of contact water.

Domestic wastewater from the administration and mine dry building, onsite camp, and process washroom facilities will be collected and channeled to a biological treatment system (sewage treatment plant). The effluent will be discharged through an on-site septic bed. A potable water treatment plant will be installed to treat water from an on-site well. Potable water will be used for showers, toilets, and sinks in the dry, camp, and office complexes.

The TSF is proposed as a filtered tailings (dry stack) facility located approximately 1 km north of the portal, at the site of the previous Preferred Sands quarry (now closed and rehabilitated). Approximately half of the tailings from the flotation plant will be desulphurized and dewatered to form a filter cake, which will be trucked to the TSF for placement and compaction. The other half of the mine tailings (including the higher sulphide tailings) will be placed underground as paste fill.



1.13 Market Studies and Contracts

McIlvenna Bay will produce two saleable concentrates from the mineral processing facility: a copper concentrate and a zinc concentrate.

Pricing for metals contained in the concentrate will be based on market price at the time of sales when the concentrate is received at the smelters.

The metal price assumptions used in this feasibility study are based on consensus pricing from a number of banking institutions to arrive at a reasonable long-term estimate. Metal price estimates are considered conservative based on supply and demand fundamentals.

The assumptions made for the purposes of this report include the following:

- The copper and zinc concentrates produced from the mineral processing facility will be sold to smelters. The transportation costs have been included in the Smelting and Refining costs (Section 22.4.3).
- 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.
- Zinc and copper are readily traded commodities and the sales terms for these are generally standard in nature. For the purposes of this study, it is assumed the products will be sold freely and at standard market rates.

The metal prices used in the economic evaluation of this project are summarized in Table 1-4.

Table 1-4: Consensus Price

Meta	Unit	Consensus Price
Copper	US\$/lb	\$3.50
Zinc	US\$/lb	\$1.20
Gold	US\$/oz	\$1,600
Silver	US\$/oz	\$22.50

Foran has not entered into any contracts for concentrate sales at the time of writing and has used reasonable treatment and refining charges based on standard commercial terms within the industry. Concentrate will be trucked from site to Flin Flon, MB where it will be loaded onto rail cars and can be transported to smelters in BC, QC, ON, or to ports.



1.14 Environmental Studies, Permitting, and Social / Community Impact

The Project area lies in the Boreal Plain (BP) 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 project 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. In 2012, a comprehensive environmental baseline program was completed by Canada North Environmental Services (CanNorth) for Foran that was designed to prepare the Project for future licensing and regulatory requirements. The program collected 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, chemistry, and 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 revisited to extend these datasets.

In 2021, Foran commissioned CanNorth to complete an environmental baseline program extending into 2022 and will provide updated information for the Environmental Impact Assessment (EIA). The 2021/2022 program used current best practices and the most up-to-date Saskatchewan Ministry of Environment (SKMOE) survey protocols.

The Project lies within the area traditionally occupied by the Peter Ballantyne Cree Nation (PBCN). It is located approximately 40 km southeast of the settlement of Deschambault Lake and approximately 50 km west of the community of Denare Beach. 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 provide updates regarding 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 discussions with PBCN with the objective of negotiating a collaboration agreement. The collaboration agreement will outline the responsibilities and requirements of each party, and focuses on areas of community engagement, environmental stewardship, workforce and business development, and community investment. Additionally, a detailed baseline socio-economic and land use study will be completed in early 2022 using primary and secondary information sources to inform the EIA process.

In February 2022, Foran submitted a Technical Proposal and Terms of Reference to the Saskatchewan Environmental Assessment Branch (EAB). The submission of this document initiated the EIA process. Throughout Q1 and Q2 2022 Foran will conduct,



prepare, and submit an Environmental Impact Statement (EIS) to the EAB for a technical review by experts, including those from other provincial ministries, and where required, to federal government reviewers. When all concerns are addressed, the Technical Review Comments from the EAB and the final EIS document will be placed in public review for a minimum of 30 days. Ultimately, the Project will require issuance of ministerial approval under Section 15 of the *Environmental Assessment Act* before proceeding with licensing and permitting.

1.15 Capital and Operating Costs

1.15.1 Capital Costs

The estimate meets the classification standard for a Class 3 estimate as defined by Association for the Advancement of Cost Engineering (AACE) International and has an intended accuracy of $\pm 15\%$. The estimate is reported in Q4 2021 Canadian dollars. Table 1-5 outlines the estimated capital cost for supplying, constructing, and pre-commissioning the Project, and excludes the ADEX program.

The capital costs primarily comprise the following areas.

- Underground mine lateral and vertical (including the shaft) development, and stationary mine infrastructure. Some of the mine mobile equipment is assumed to be purchased on a lease-to-own basis, with the costs incurred in the lease payments. Indirect costs include temporary construction facilities, construction services and supplies, construction management (CM) costs, construction equipment, freight, and Owner's costs
- Process plant costs include the construction of the entirety of the process plant facility, paste fill plant, and sustaining capital
- Other capital, including transmission line, tailings storage, pre-production costs, and site infrastructure costs which include site preparation, permanent camp, maintenance shop, fuel storage, administration and dry facility, water treatment systems, and site roads
- Contingency

At the time of this report, Foran is preparing an ADEX. This plan will include a portal, ramp to the 0090 Level, remucks, 0060 Level access, 0060 Level sump excavation, bulk sample on the 0090 Level, and a diamond drill drift on the 0090 Level in the hanging wall.



Table 1-5: Total Capital Cost Estimate

Description	Units	Cost
Project Capital		
UG Mine (includes indirect costs)	\$ million	95.9
Process Plant	\$ million	151.5
Other Capital	\$ million	90.0
Subtotal Project Costs	\$ million	337.4
Project Capital Contingency	\$ million	38.5
Total Project Capital	\$ million	375.9
Sustaining	\$ million	464.6
Closure	\$ million	8.2
Sustaining Capital Contingency	\$ million	9.0
Total	\$ million	857.7

Note: Totals may not sum due to rounding.

Sustaining capital incorporates all capital expenditures after the pre-production period of Year -2 and Year -1. Closure costs of \$8.2 million have been included in Year 19 through Year 21.

1.15.2 Operating Costs

Operating cost estimates were developed to present annual costs for production. Unit costs are expressed as \$/tonne processed. Operating costs were allocated to either mining, processing, tailings facility and paste plant, or general and administration (G&A). LOM operating costs are estimated to be \$1,769.8 million. LOM operating costs are summarized in Table 1-6.

UG mining occurs during Year -2 to Year 11 (note that in Year -2 and Year -1, UG mining costs are capitalized). UG mining begins with capital development in Year -2 and the capitalized development continues through the LOM.

Table 1-6: Operating Cost Estimate Summary (Year -1 to Year 18 inclusive)

Description	LOM Cost (\$ million)	Average Annual (\$ million)	Unit Cost (\$/t processed)
Mining / Paste Plant	1,060.6	60.4	41.26
Processing	615.9	35.2	23.96
Tailings Facility	39.7	9.9	1.54
General and administration	174.2	26.9	6.78
Total	1,890.3	132.4	73.55

Notes:

16. Totals may not sum due to rounding.

17. Average annual cost based on Year 1 through Year 17



G&A costs include labour, miscellaneous surface equipment, contractor costs for camp and catering, insurance premiums, allowance for training, consultants, and marketing and accounting functions.

1.16 Economic Analysis

The results of the economic analysis represent 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 here. Forward-looking statements in this report include, but are not limited to, statements with respect to future metal prices, currency exchange, estimation of Mineral Resources and Mineral Reserves, estimated mine production and metals recovered, estimated capital and operating costs, and estimated cash flows generated from the planned mine production.

Actual results may be affected by the following.

- Differences in estimated initial capital costs and development time from what has been assumed
- Unexpected variations in quantity of ore, grade, or recovery rates, or presence of deleterious elements that would affect the process plant or waste disposal
- Unexpected differences in geotechnical and hydrogeological conditions from what was assumed in the mine designs, including water management during construction, mine operations, and post mine closure
- Differences in the timing and quantity of estimated metal production, costs of future production, sustaining capital requirements, future operating costs, assumed currency exchange rate, requirements for additional capital, unexpected failure of plant, or equipment or processes not operating as anticipated
- Changes in government regulation of mining operations, environment, and taxes
- Unexpected social risks, higher closure costs and unanticipated closure requirements, mineral title disputes, or delays to obtaining surface access to the property

If additional mining, technical, and engineering studies are conducted, these may alter the project assumptions presented in this report and may result in changes to the calendar timelines and the information and statements contained in this report.

Production mine development and licensing approvals are currently not in place, and statutory permits, including environmental permits, are required to be granted prior to mine commencement.

The Project has been evaluated using discounted cash flow analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of project capital expenditures, sustaining capital costs, operating costs, taxes, royalties, and



commitments to other stakeholders. These are subtracted from revenues to determine the annual cash projections.

Cash flows are taken to occur at the midpoint of each period. To reflect the time value of money, annual cash flow projections are discounted to the Project valuation date using the annual discount rate. The discount rate appropriate to a specific project can depend on many factors, including the type of commodity, the cost of capital to the project, and the level of project risks (e.g., market risk, environmental risk, technical risk, and political risk) in comparison to the expected return from the equity and money markets.

The base case discount rate for the project is 7%. The discounted present values of the cash flows are summed to determine the Project's NPV. In addition to the NPV, the IRR and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project from the start of mineral processing.

Taxes and depreciation for the Project were modelled using input from Foran and were reviewed in the Prefeasibility Study (PFS) by EY LLP (2021).

On a pre-tax basis, the NPV at 7% is \$678.2 million, the IRR is 26%, and the payback period is 3.9 years. On a post-tax basis, the NPV at 7% is \$465.7 million, the IRR is 22% and the payback period is 4.5 years.

A summary of the LOM cashflow is provided in Table 1-7 and Figure 1-4. Table 1-8 summarizes the economic results with the NPV at 7%.

Table 1-7: LOM Cashflow Forecast Summary Table

Description	Units	Value
Gross Revenue	\$ million	4931.5
Less: Smelter and Refinery Costs	\$ million	-537.6
NSR	\$ million	4393.9
Less: Royalties	\$ million	-20.3
Less: Total Operating Costs	\$ million	-1890.3
EBITDA	\$ million	2483.3
Less: Capital Costs	\$ million	-848.7
Less: Closure Costs	\$ million	-8.2
Pre-Tax Cash Flow	\$ million	1626.4
Less: Taxes	\$ million	-447.0
Post-tax cash flow	\$ million	1179.5



Figure 1-4: Undiscounted After-Tax Cash Flow

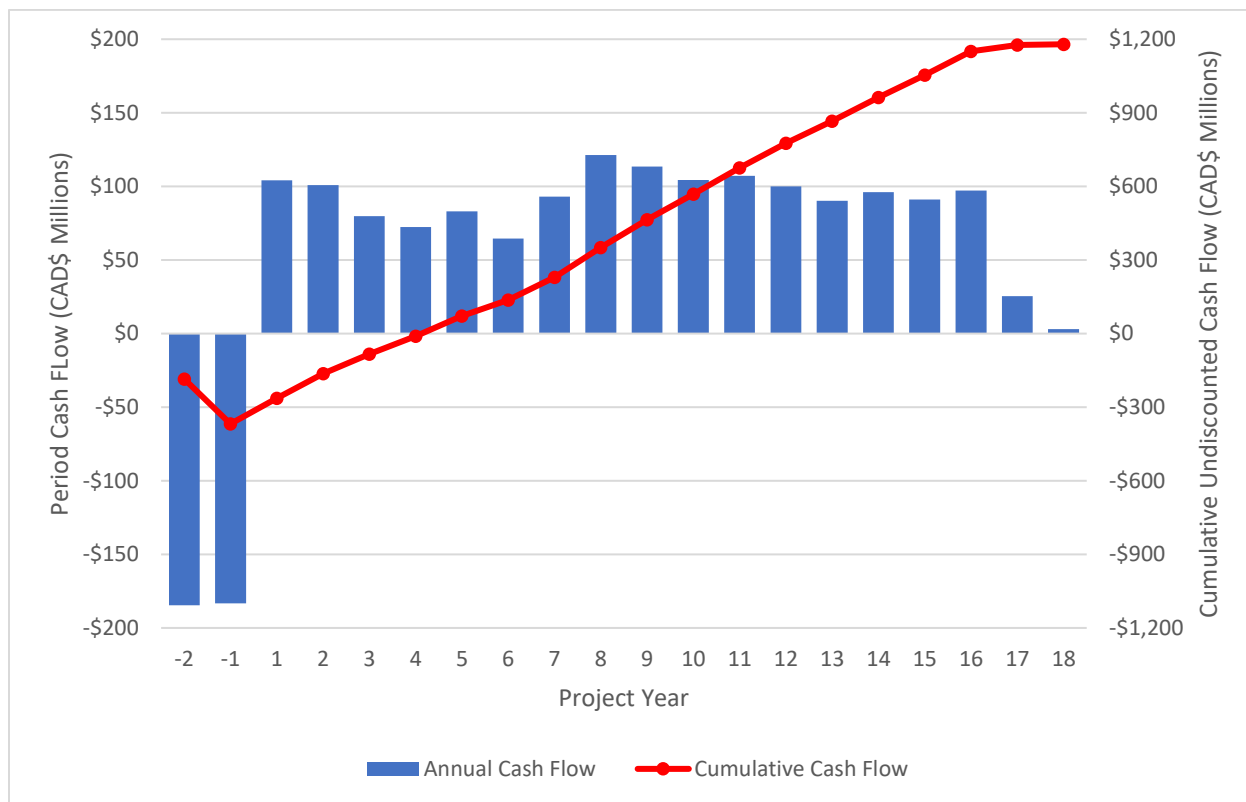


Table 1-8: Forecast Economic Results

Description	Units	Value
Pre-Tax		
NPV at 7%	\$ million	678.2
Internal rate of return	%	25.9%
Payback period	Years	3.9
After-Tax		
NPV at 7%	\$ million	465.7
Internal rate of return	%	21.8%
Payback period	Years	4.5

Note: Payback period is calculated from the start of mineral processing.

1.17 Sensitivity Analysis

The cash flow model was tested for sensitivity to variances regarding the following.

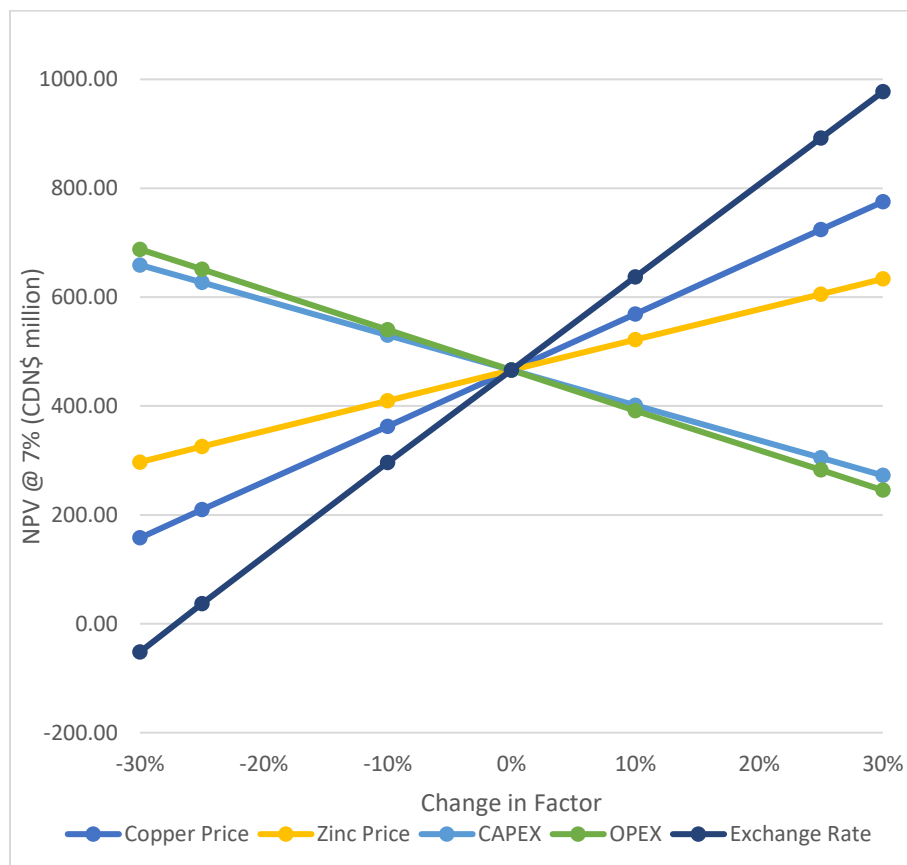
- Metal Prices (Copper and Zinc)



- Capital Expenditure (CAPEX)
- Operational Expenditure (OPEX)
- Exchange Rate

Figure 1-5 illustrates the results of the sensitivity analysis for varying metal prices. The anticipated Project cash flow is most sensitive to fluctuations in the exchange rate and the price of copper.

Figure 1-5: Metal Price Sensitivity Analysis



1.18 Interpretation and Conclusions

Under the assumptions presented in this report, the Project indicates positive economics. The anticipated Project cash flow is most sensitive to the United States dollar exchange rate and the price of copper.

1.19 Recommendations

Due to the positive, robust economics, it is recommended to advance the Mcllvenna Bay Project to the next phase of engineering. The recommended development path is to



continue to advance the environmental assessment and licensing efforts while completing the ADEX program to provide further project definition and reduce project execution timeline risks.

Engineering and field investigations should be advanced in support of increased certainty of costs and project timelines in preparation for permit approvals and a Final Investment Decision (FID).

The FS excludes pre-project approval expenditures associated with the previously announced pre-development program (surface preparation and exploration decline) as well early engineering costs. Based on latest estimates, Foran expects the totality of these pre-project approval expenditures to total \$45 million.

The following subsections list the programs that are recommended for the next phase of engineering work for the Mcllvenna Bay Project.

1.19.1 Site Investigations

It is recommended that Foran proceeds with site investigations to support detailed engineering, including the following.

- Detailed materials characteristics and quantification assessment to confirm borrow source locations and available volumes of aggregates
- Updates to hydrogeological studies based on results from the ADEX program—specifically the grouting program in the sandstone layer—to increase Foran’s understanding of the impact of groundwater on the UG mine and mine dewatering requirements
- Investigate near-surface and subsurface conditions in the area of the proposed surface infrastructure, shaft collar, and ventilation raise locations

The total estimated project cost for the geotechnical, hydrological, and surface material assessment is included in \$1.0-1.1 million.

1.19.2 Process Plant Optimizations

The following studies are proposed.

- Additional bench scale confirmation testing of zone and early year production composites from the advance exploration program currently underway.
- Collection of additional grindability data, specifically for SAG and regrind mills.
- Pilot flotation testwork on the bulk sample material to provide confirmation of the following:
 - Reagent dosages
 - Circuit configuration



- Effect of process water recycling
- Tailings desulphurization
- Investigate the potential to reject non-sulphide gangue and increase mill feed head grade using pre-concentration methods such as ore-sorting.
- Paste uniaxial compression test show comparable strength with addition of on-site quarry sand and reduced binder content. Examine availability of quarry sand and impact on backfill and TMF volumes.

The total estimated cost for this program is \$ 1.0–1.5 million.

1.19.3 Engineering

It is recommended that Foran proceeds to detail engineering, which will form the basis for procurement, construction, and commissioning work, and further provide Foran valuable information to finalize internal discussion and evaluation of the feasibility of the project. The work will include the following.

- Complete engineering to a 40–45% level of completion, in alignment with the AACE Class 2 estimate requirements.
- Develop Request for Proposal (RFP) packages for construction quotations.
- Fully define long-lead procurement items and initiate the procurement process for critical path items.
- Develop a Class 2 capital cost (CAPEX) estimate that will form the control budget for the Project.
- Develop a Level 4 Implementation Schedule for the Project.

The total estimated cost for detailed engineering is \$11–12.5 million, of that approximately \$4 million is estimated for 2023 and not carried in the total capital cost estimate for the FS.



2.0 INTRODUCTION

2.1 Terms of Reference

At the request of Mr. Dan Myerson, CEO of Foran, Stantec has completed a FS 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 FS on the Project that was completed on the Effective Date given in Section 2.3 below.

The user of this report is encouraged to ensure 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 the following.

- Data supplied by outside sources
- Information available at the time of preparation, and
- Qualifications and assumptions presented within this report

The FS 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 modelling prepared by Stantec, and geotechnical information from third party consultants
- Prepare a 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, paste plant, 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



- Prepare an economic model of the project, complete with discounted cashflows and sensitivity analyses
- Provide 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 Stantec do not consider these to be material.

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) Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014). The resource statement in this FS includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied that would enable these to be categorized as Mineral Reserves. The Mineral Reserve statement in this FS is the first disclosure of Mineral Reserves for the Project and does not include any Inferred Mineral Resources.

2.2 Site Visits and Scope of Personal Inspection

By virtue of their education, experience, and professional associations, the authors of this Technical Report are Qualified Persons 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.

None of the QP's responsible for the preparation of this report has any beneficial interest in Foran.



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 provided in Section 28.0 of this report.



Table 2-1: Details of QPs and Site Visits

Qualified Person	Title	Area of Responsibility	Site Visit Date
Mark Hatton, P.Eng.	Senior Project Manager. Stantec Consulting Ltd.	1.1, 1.2, 1.9, 1.10, 1.13, 1.15 to 1.19, 2.0, 3.2 to 3.4, 15.0, 16.1, 16.3 to 16.17, 18.5.1, 18.5.2, 19, 21.1.1, 21.1.2, 21.1.6, 21.1.7, 21.2.1, 21.2.3, 22.0, 24.0, 25.4, 25.5, 25.8, 25.10, 25.11, 26.0, and 27.0	August 2021
William J. Lewis, B.Sc., P.Geo	Senior Geologist, Micon International Ltd.	1.3 to 1.6, 1.8, 1.9, 3.1, 4.0 to 12.0, 14.0, 23.0, 25.1, 25.2, and 25.12	16–18 August 2018 17–19 November 2021
Lyn Jones, P.Eng.	Manager, Process Engineering, Blue Coast Research	1.7, 1.11, 13.0, 17.0, 25.3, and 25.6	None
Michael Franceshini, P.Eng.	Project Manager, Halyard, Inc.	1.12, 18.0 to 18.5, 18.6.3 to 18.6.7, 18.7 to 18.9, 21.1.3, 21.2.2, and 25.7.	October 2021
Jocelyn Howery, M.Sc., P.Ag.	Hydrology Division Manager, Canada North Environmental Services Ltd.	1.14, 20.1 to 20.4, 20.5.1 to 20.5.6, 20.5.8, 20.6, and 25.9	October 2014
Alex McIntyre, P.Eng.	Senior Engineer, Knight Piésold Ltd.	18.5.8, 18.9, 20.5.7, 20.5.9, 21.1.4, 21.1.5, and 21.2.4	October 2021
Kathy Kalenchuk, Ph.D., P.Eng.	President & Principal Consultant, RockEng Inc.	16.2	None

2.3 Effective Dates

- The effective date of the Resource Estimate is 06 September 2021.
- The effective date of the Reserve Estimate is 15 February 2022
- The effective date of the Feasibility Study is 28 February 2022.

2.4 Information Sources and References

Key sources of information for this report include the following.

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

Foran has previously filed the following technical reports for the Project.

Makarenko, M., D. Kennard, M. Bender, D. Rennie, J. Hull, L. Correia, and K. Major. 2015. Technical Report on the Preliminary Economic Assessment on the McIlvenna Bay Project Saskatchewan, Canada. Technical report prepared for Foran Mining Corporation. Effective date 21 January 2015

Lewis, W.J., A. J. San Martin, and L. Jones. 2021. Technical Report for the 2021 Mineral Resource Estimate on the McIlvenna Bay Project Saskatchewan, Canada. Technical report prepared for Foran Mining Corporation. Effective date 06 September 2021

Lewis, W.J., A. Holloway, S. Cole, D. Flood, M. Patel, J. Howery, and A. McIntyre. 2020: Technical Report on The Pre-feasibility Study on the McIlvenna Bay Project Saskatchewan, Canada. Technical report prepared for Foran Mining Corporation. Effective date 12 March 2020

Lewis, W.J., A. J. San Martin, and R. M. Gowans. 2019. Technical Report for the 2019 Mineral Resource Estimate on the McIlvenna Bay Project Saskatchewan, Canada. Technical report prepared for Foran Mining Corporation. Effective date 07 May 2019

Rennie, D.W. 2011. Technical Report for the 2011 Mineral Resource Estimate on the McIlvenna Bay Project Saskatchewan, Canada. Technical report prepared for Foran Mining Corporation. Effective date 09 December 2011



3.0 RELIANCE ON OTHER EXPERTS

3.1 Mineral Tenure, Surface Rights, and Encumbrances

In this Technical Report, discussions regarding royalties, permitting, taxation, and environmental matters are based on material provided by Foran staff and experts retained by Foran. The QPs and Micon are not qualified to comment on such matters and have relied on the representations and documentation provided by Foran for such discussions.

All data used in this report were originally provided by Foran. The QPs have reviewed and analyzed these data and have drawn their own conclusions therefrom.

The QPs and Micon offer no legal opinion as to the validity of the title to the mineral concessions claimed by Foran and, in that regard, have relied on information provided by Foran.

Information related to royalties, permitting, taxation, environmental matters and the validity of the title to the mineral concessions claimed by Foran were extracted from previous NI 43-101 Technical Reports and updated by Foran through personal communication with the QPs. Previous NI 43-101 Technical Reports, as well as other references, which were used in the compilation of this report are contained in Section 27.

3.2 Taxation

The QPs have relied upon, and disclaim responsibility for, information supplied by Foran staff and experts retained by Foran for information related to taxation as follows.

- EY L.L.P., 2021: Taxation audit email prepared by EY L.L.P. for Foran Mining Co., 15 September 2021, 6 pages (EY 2021).

This information is used in support of the economic analysis in Section 22.0, and the Mineral Reserve estimation in Section 15.0.

3.3 Royalties

The QP has relied upon, and disclaim responsibility for, information supplied by Foran staff for information related to royalties as follows.

- Royalties Agreement (see Cameco – Rio Algom – Foran [2015], Cameco – Rio Algom – Foran [2015], Rio Algom – BHP – Foran – Cameco [2015], Cameco – BHP – Foran [2005]).
- Foran-Copper Reef Asset Purchase Agreement, 17 October 2010 (Foran –Copper Reef 2010)



This information is used in support of the economic analysis in Section 22.0, and the Mineral Reserve estimation in Section 15.0.

3.4 Market and Metal Prices

The QP disclaims responsibility for the marketing and metal price forecast information included in this report as the QPs relied on experts retained by Foran for this information.

In preparing the project economic analysis, the QP has relied upon bank consensus pricing provided by Foran for long-term metal prices as forecasted within this report.

In preparing the project economic analysis, the QP has relied upon independent consultant Joel Adams (Balyasny Asset Management) for information related to concentrate sales market conditions in the following document.

- J. Adams – Balyasny Asset Management [Email], 22 February 2022: Commercial Assumptions: Foran McIlvenna Bay Concentrate Sales Market Condition (Balyasny Asset Management 2022).

Marketing and metal pricing information from these sources is referenced in Section 19.0 of this report. The information is also used in support of the Mineral Reserve estimate in Section 15.0, and the economic analysis in Section 22.0.



4.0 PROPERTY DESCRIPTION AND LOCATION

Portions of this section were extracted from previous Mcllvenna Bay Project Technical Reports and updated or edited where necessary.

4.1 General Description and Location

The Mcllvenna Bay property is located approximately 1 km south of Hanson Lake, Saskatchewan. The property is also approximately 375 km northeast of Saskatoon and 85 km west of Flin Flon, Manitoba (Figure 4-1). Mcllvenna 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 Mcllvenna Bay Deposit is located within the property boundaries.

4.2 Ownership, Land Tenure and Property Agreements

4.2.1 Ownership and Land Tenure

Foran owns 100% of the Mcllvenna Bay property.

The entire Mcllvenna Bay property comprises 38 claims totalling 20,954 ha (Figure 4-2). The relevant claim information is summarized in Table 4-1. The claims are listed in the name 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 designates.

4.2.2 Property Agreements

On 25 January 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 (HLJV)—which allowed Foran to acquire a 100% interest in the Mcllvenna Bay property, including the Mcllvenna Bay copper-zinc deposit. Foran would acquire 100% of the Mcllvenna Bay property by doing the following.

- Paying \$1,500,000 to the HLJV
- Paying a further \$2,000,000 to the HLJV before 31 May 2006
- Providing the HLJV 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

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 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.



Figure 4-1: McIlvenna Bay Project Location Map

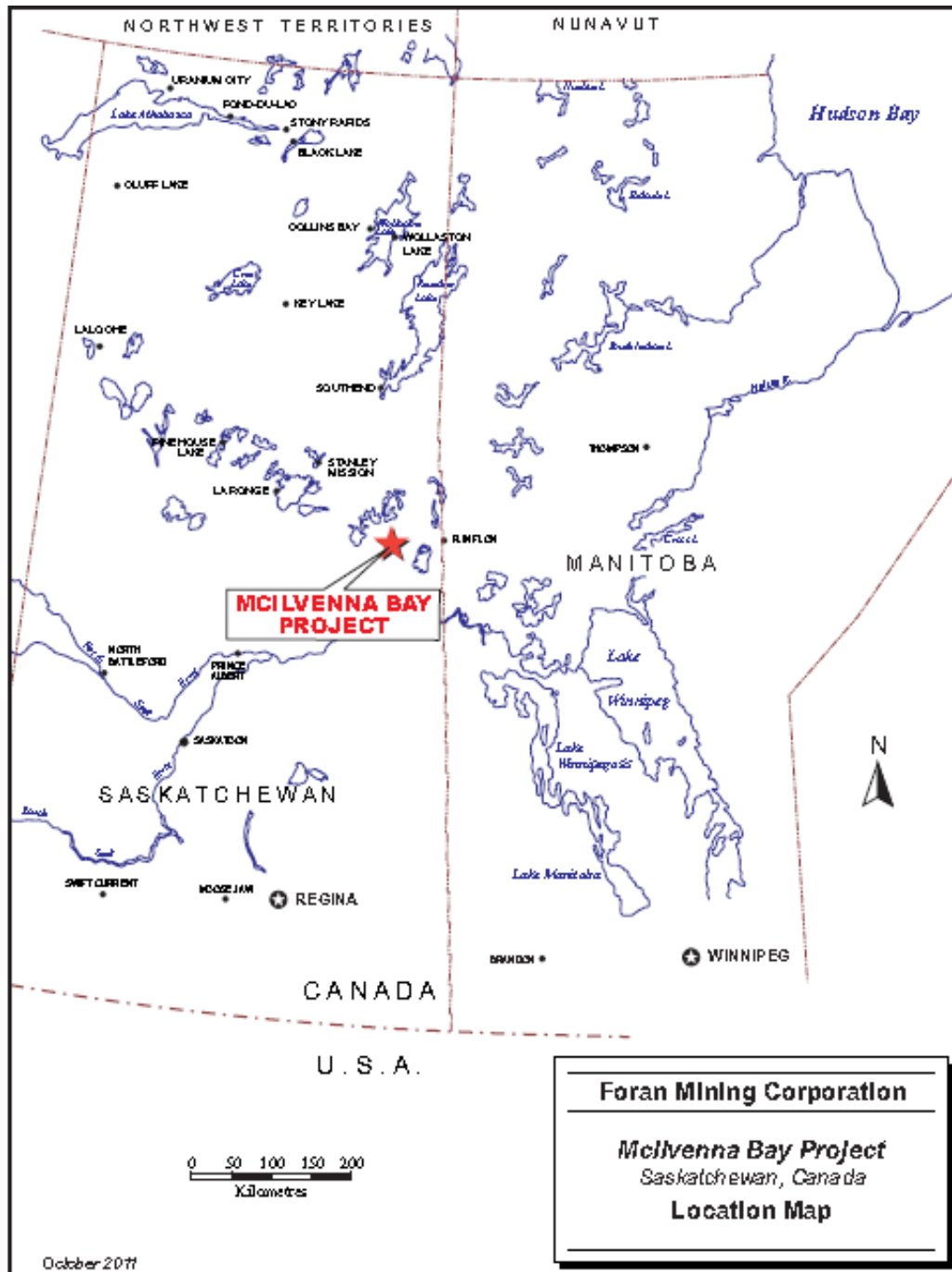


Figure extracted from the 2015 Technical Report. Figure originally from Foran, 2011.



April 2022
Project Number: 169521520

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

Property	Disposition No.	Owners ¹	Claim Staking Date	Claim Expiry Date	Hectares
Mcllvenna Bay	S-113791	Foran Mining Corporation	2011/03/21	2030/06/18	2,255.55
Mcllvenna Bay	S-113790	Foran Mining Corporation	2011/03/21	2030/06/18	571.124
Mcllvenna Bay	S-113789	Foran Mining Corporation	2011/03/21	2030/06/18	1,261.65
Mcllvenna Bay	S-113788	Foran Mining Corporation	2011/03/21	2033/06/18	1,107.29
Mcllvenna Bay	S-113787	Foran Mining Corporation	2011/03/21	2042/06/18	624.66
Mcllvenna Bay	S-113786	Foran Mining Corporation	1976/12/01	2042/02/28	518.836
Mcllvenna Bay	S-113785	Foran Mining Corporation	1976/12/01	2042/02/28	305.373
Mcllvenna Bay	S-113784	Foran Mining Corporation	1976/12/01	2042/02/28	157.614
Mcllvenna Bay	S-113783	Foran Mining Corporation	1976/12/01	2042/02/28	278.443
Mcllvenna Bay	S-101727	Foran Mining Corporation	1991/01/08	2031/04/07	5,283.66
Mcllvenna Bay	CBS 8460	Foran Mining Corporation	1988/03/14	2042/06/11	270.35
Mcllvenna Bay	S-95733	Foran Mining Corporation	1978/05/01	2032/07/29	12.63
Mcllvenna Bay	S-95734	Foran Mining Corporation	1978/05/01	2032/07/29	12.57
Mcllvenna Bay	S-95735	Foran Mining Corporation	1978/05/01	2032/07/29	10.58
Mcllvenna Bay	S-95736	Foran Mining Corporation	1978/05/01	2032/07/29	8.71
Mcllvenna Bay	S-95737	Foran Mining Corporation	1978/05/01	2032/07/29	11.42
Mcllvenna Bay	S-97903	Foran Mining Corporation	1990/06/12	2032/09/09	5.51
Mcllvenna Bay	CBS 3693	Foran Mining Corporation	1988/02/22	2042/05/22	107.65
Mcllvenna Bay	S-111933	Foran Mining Corporation	2011/03/21	2030/06/18	318.68
Mcllvenna Bay	S-100671	Foran Mining Corporation	1989/10/19	2031/01/16	102.71



Property	Disposition No.	Owners ¹	Claim Staking Date	Claim Expiry Date	Hectares
Mcllvenna Bay	S-112150	Foran Mining Corporation	2011/03/21	2035/06/18	434.06
Mcllvenna Bay	S-107931	Foran Mining Corporation	2006/06/12	2033/09/09	859.02
Mcllvenna Bay	S-95741	Foran Mining Corporation	1978/05/01	2032/07/29	17.15
Mcllvenna Bay	S-95742	Foran Mining Corporation	1978/05/01	2032/07/29	17.88
Mcllvenna Bay	S-95745	Foran Mining Corporation	1978/05/01	2032/07/29	18.79
Mcllvenna Bay	CBS 3692	Foran Mining Corporation	1989/06/20	2041/09/17	315.53
Mcllvenna Bay	S-100669	Foran Mining Corporation	1989/04/24	2031/07/22	683.88
Mcllvenna Bay	CBS 4909	Foran Mining Corporation	1977/04/14	2034/07/12	1,845.78
Mcllvenna Bay	CBS 9314	Foran Mining Corporation	1976/12/01	2033/02/28	587.28
Mcllvenna Bay	CBS 9315	Foran Mining Corporation	1976/12/01	2042/02/28	1,147.90
Mcllvenna Bay	CBS 9317	Foran Mining Corporation	1976/12/01	2035/02/28	675.21
Mcllvenna Bay	CBS 9318	Foran Mining Corporation	1976/12/01	2031/02/28	504.09
Mcllvenna Bay	S-95740	Foran Mining Corporation	1978/05/01	2032/07/29	16.29
Mcllvenna Bay	S-95743	Foran Mining Corporation	1978/05/01	2032/07/29	16.54
Mcllvenna Bay	S-95744	Foran Mining Corporation	1978/05/01	2032/07/29	17.84
Mcllvenna Bay	S-98827	Foran Mining Corporation	1986/04/07	2032/07/05	13.63
Mcllvenna Bay	S-98828	Foran Mining Corporation	1986/04/07	2041/07/05	15.21
Mcllvenna Bay	MC00011167	Foran Mining Corporation	2018/05/28	2041/08/26	543.25
Total					20,954.34

Notes: Claim data supplied by Foran, 2021. Foran owns 100% of the claims.

In a subsequent event, Foran and Copper Reef were in dispute regarding the assignment agreement concerning the Property Option Agreement for Mcllvenna Bay. This matter was resolved on 24 May 2006, and under that settlement, Foran made a payment of \$2,000,000 for Mcllvenna Bay. Foran's \$1,500,000 payment to the HLJV on behalf of Copper Reef (Foran contributed \$500,000 to Copper Reef for that payment on



25 January 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 HLJV remained the responsibility of the current Foran-Copper Reef joint venture.

On 03 November 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 CAD \$0.75/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 canceled, 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 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.

Most of the mineral claims that comprise the McIlvenna Bay Property have been in existence for more than 10 years, therefore most claims require \$25/ha of exploration



expenditures to keep them in good standing. Significant exploration work has been completed in the McIlvenna Bay area over the years by Foran and the company has built up a significant pool of exploration credits which can be used to maintain the claims in good standing when exploration is not occurring.

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 has 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 reclaimed 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 08 December 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.

Foran reports that, with the purchase of the overlapping 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 22 December 2014. Micon's 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 its 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 85 km east 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 (StatsCan 2012). The economy of the area is primarily based on copper and zinc mining, with additional contributions from tourism and forestry.

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. On 11 February 2022, a local Flin Flon newspaper, *The Reminder*, noted that the zinc processing plant in Flin Flon would shut down in May of 2022 followed by the mill and 777 mine in June (Westerhaver 2022). A number of employees will be transferred to the Snow Lake operations with a number of employees remaining to take care of the tailings facility on a care and maintenance schedule. Some of the administration buildings will also remain open and operational. This is an unfortunate occurrence for the community but, should Foran proceed with developing the Project, it will mean that, potentially, there will be a trained workforce available for the Project.

McIlvenna Bay lies within the traditional territory of the Peter Ballantyne Cree Nation (PBCN). As of 2019 the PBCN had approximately 12,000 members (PBCN n.d.) living on more than 36 reserves and/or settlements. The PBCN's traditional territory encompasses roughly 52,000 km², 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 challenges for PBCN members working to strengthen their local economies and personal economic well-being.

Foran has conducted consultation sessions for the Project in the nearby communities. 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, surveyed members of the PBCN communities 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 the mining sector does provide the potential for meaningful employment in mining, processing and skilled trades, it also has the potential for the creation of spin-off opportunities such as service business in site services, catering, janitorial, trucking, security, grocery and retail supplies, such development must be balanced with the continued reliance of PBCN members on the waters, lands, and forests for sustenance, livelihood, and spiritual support. As the Project proceeds, Foran will continue to engage the traditional users of the Project area to receive input on potential ways and means to mitigate 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, and numerous wetlands. Vegetation is 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, occurring predominantly in poorly drained peaty areas along with tamarack.

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 the 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. 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. 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 Mcllvenna Bay. Production from the site ceased in 2014 and the development area has been reclaimed. 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.



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

Portions of this section were extracted from previous McIlvenna Bay Project Technical Reports and updated or edited where necessary.

5.1 Accessibility

McIlvenna Bay is located 1 km south of Hanson Lake, Saskatchewan, and approximately 85 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) represent 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 of 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. Because of the swampy and muskeg ground conditions, most 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. The camp includes a 35-bed trailer camp with office, core shack, shop, and core storage facility. During the 2021 program, an additional 35-bed trailer camp and a second core shack were added to the camp to increase the capacity of the facilities. (Figure 5-1 and Figure 5-2). Also during 2021, the company applied for an advanced exploration permit which led to the addition of a 36 bed camp, and administration building.

A gravel road has been built through the property to support Foran's exploration programs, as well as an adjacent quarrying operation (subsequently reclaimed).

Foran's mineral concessions contain enough area for all necessary construction (e.g., tailings facilities, processing plant, waste disposal). The local region, primarily the community of Flin Flon, has enough capacity to house mining personnel. Power would be provided from SaskPower via a new or existing transmission line from Island Falls, Saskatchewan. Water for a mining / milling operation will be sourced from treated ground water inflows into the underground operations.



Figure 5-1: New 2021 Trailer Camp to Right of the Original Camp



Photograph taken during the November 2021 Micon site visit.

Figure 5-2: New 2021 Core Shack (Foreground, Left) and Exploration Office (Right)



Photograph taken in November 2021 during a Micon site visit.

6.0 HISTORY

Portions of this section were extracted from previous Mcllvenna Bay Project Technical Reports and updated or edited where necessary.

6.1 General Exploration History Prior to 1998

In 1957 the Parrex Mining Syndicate (Parrex) tested an 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 shows a historical view of the Hanson Lake mine.

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



Photograph from Copper Reef Mining Corporation Website, 2019.

In 1976, the Saskatchewan Mineral Development Corporation (SMDC), the provincial government exploration vehicle that eventually became Cameco, acquired a large exploration lease centred 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 to 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 VMS deposit located under the Paleozoic cover approximately 8 km southeast of Hanson Lake. This prompted Cameco to re-evaluate its existing airborne EM data between the new discovery and Hanson Lake, resulting in a Mark VI helicopter INPUT survey over the area south of Hanson Lake, with flight lines oriented northeast to southwest. The survey delineated a 1,200 m long INPUT anomaly, striking east to 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.

The previous estimates will not be discussed further in this Technical Report, because the Micon QPs for this report have not reviewed any of the previous mineral resource estimates or assessed them for compliance with current CIM Definition Standards (10 May 2014). Foran is not relying on the previous resource estimates which are superseded by the current estimate contained in Section 14.0 of 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.



There was a silica (fracking) sand quarrying operation near Mcllvenna Bay and there are quarry dispositions that overlap Foran mineral claims. The quarry dispositions that were overlapping part of the Mcllvenna 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 reclaimed.



7.0 GEOLOGICAL SETTING AND MINERALIZATION

Portions of this section were extracted from previous Mcllvenna Bay Project Technical Reports and updated or edited where necessary.

7.1 Regional Geology

The Mcllvenna Bay Property is located on the western edge of the Paleoproterozoic FFGB, which extends from north central Manitoba into northeastern 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), because of the collision between the Superior and Hearne Archean Cratons.

The FFGB is composed of structurally juxtaposed volcanic and sedimentary 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 Basalts (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 between 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 between 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.

As currently viewed, the FFGB contains eight geographically separate juvenile island arc volcanic assemblages (blocks), each being 20 km to 50 km across (Figure 7-1). From east to west, they are known as the Snow Lake, Four Mile Island, Sheridan, Flin Flon, Birch Lake, West Amisk, Hanson Lake, and Northern Lights assemblages (Zwanig and Schledewitz 1997, 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.

Figure 7-1: Regional Geology Map

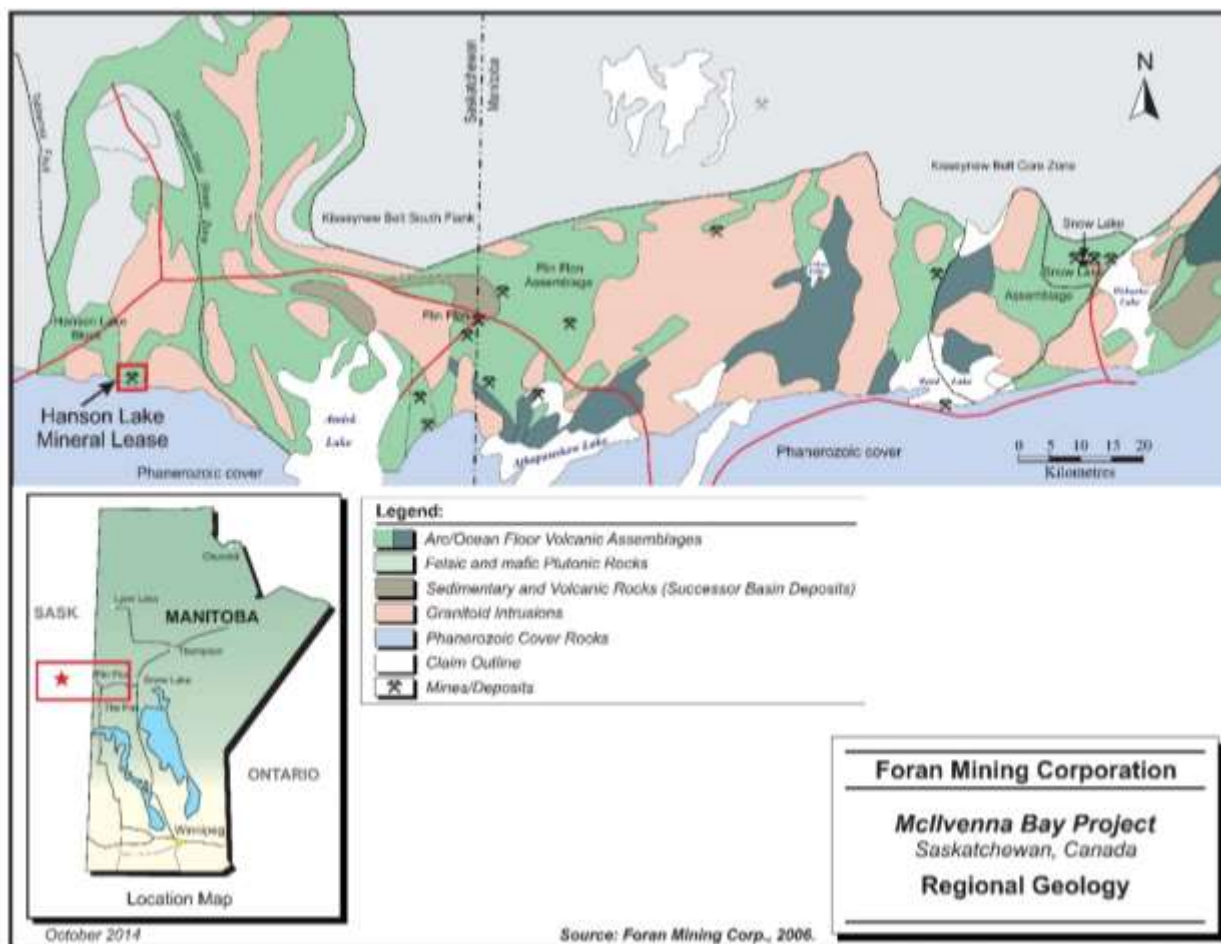


Figure taken from the 2015 Technical Report.

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 (Kiseynew 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 Mcllvenna 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, Henderson and Neilson 1999).

7.2 Local Geology

The Hanson Lake Block, the host terrain of Mcllvenna 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 Block are dominated by juvenile island arc, felsic to intermediate metavolcanic rocks, with subordinate amounts of mafic volcanics, minor intermediate volcanics, and graywackes. Oxide facies iron formations are not commonly exposed, but their presence has been confirmed by diamond drilling. Long continuous magnetic trends suggest that the distribution of iron formations is very widespread in the area south of Hanson Lake. The sequence has been intruded by various felsic intrusions, some of which are believed to be subvolcanic intrusions. Abundant diorite and gabbro plugs and dikes cut the sequence, as well as minor ultramafic intrusions (Koziol, Bell and Wasyliuk 1991). The supracrustal rocks generally dip moderately to steeply east to northeast. South of Hanson Lake, the Proterozoic sequence is poorly understood because of the unconformably overlying Paleozoic sedimentary rocks. Mcllvenna Bay Projects to subsurface under the sedimentary cover (Lemaitre 2000).

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 D2 structural event resulted in tight to isoclinal, southwest plunging F2 folds and local southwest verging mylonite zones. D3 deformation resulted in tight north trending folds followed by a brittle D4 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 Mcllvenna 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). U-Pb ages of supracrustal rocks in the block constrain the metamorphic event to between 1,808 Ma and 1,804 Ma (Maxeiner, et al. 1999). U-Pb age dating of a quartz-feldspar porphyry (a possible subvolcanic intrusion) which intruded the supracrustal sequence yielded a date of $1,888 \pm 12$ Ma.

7.3 Property Geology

The property geology map is shown in Figure 7-2. 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 discussion below is extracted from Lemaitre (2000).

The stratigraphy of the deposit area, which was divided into six formations (Figure 7-3), has been defined over a 2 km strike length by a total of 285 drill holes. The lowest formation intersected by drilling both structurally and stratigraphically is the Mcllvenna Bay Formation (Figure 7-4), the host of Mcllvenna Bay. The Mcllvenna Bay Formation is overlain to the north by the Cap Tuffite Formation. The Mcllvenna 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 Mcllvenna Bay Deposit between these two units, an obvious exhalative horizon (and hence a period of clastic and volcano-sedimentary 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 volcanic rocks. 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 volcanic rocks, 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 one or more 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 20 m and 30 m.

The Mcllvenna Bay Formation, the host formation of the sulphide deposit, is known only to the extent that it has been drilled below the footwall of the deposit. The formation is at least 200 m thick (true thickness) and comprises massive and semi-massive sulphides and copper-rich stringer zones that make up the Mcllvenna Bay Deposit, and



a succession of variably altered felsic volcanics, volcanoclastics, and/or volcanic-derived sediments of rhyolitic composition.

Figure 7-2: Project Geology Map

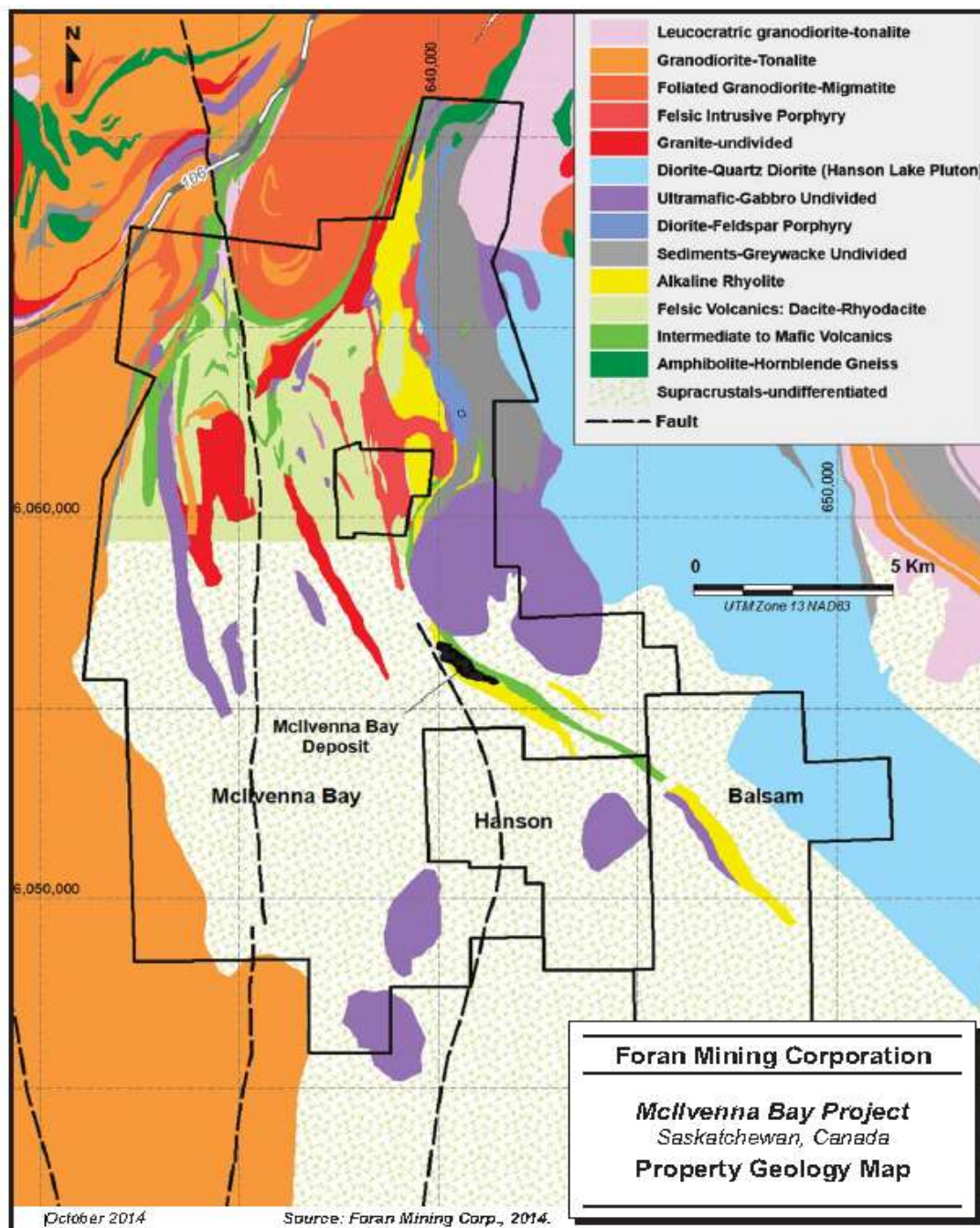


Figure taken from the 2015 Technical Report.



Figure 7-3: Stratigraphic Column for the Mcllvenna Bay Deposit Area

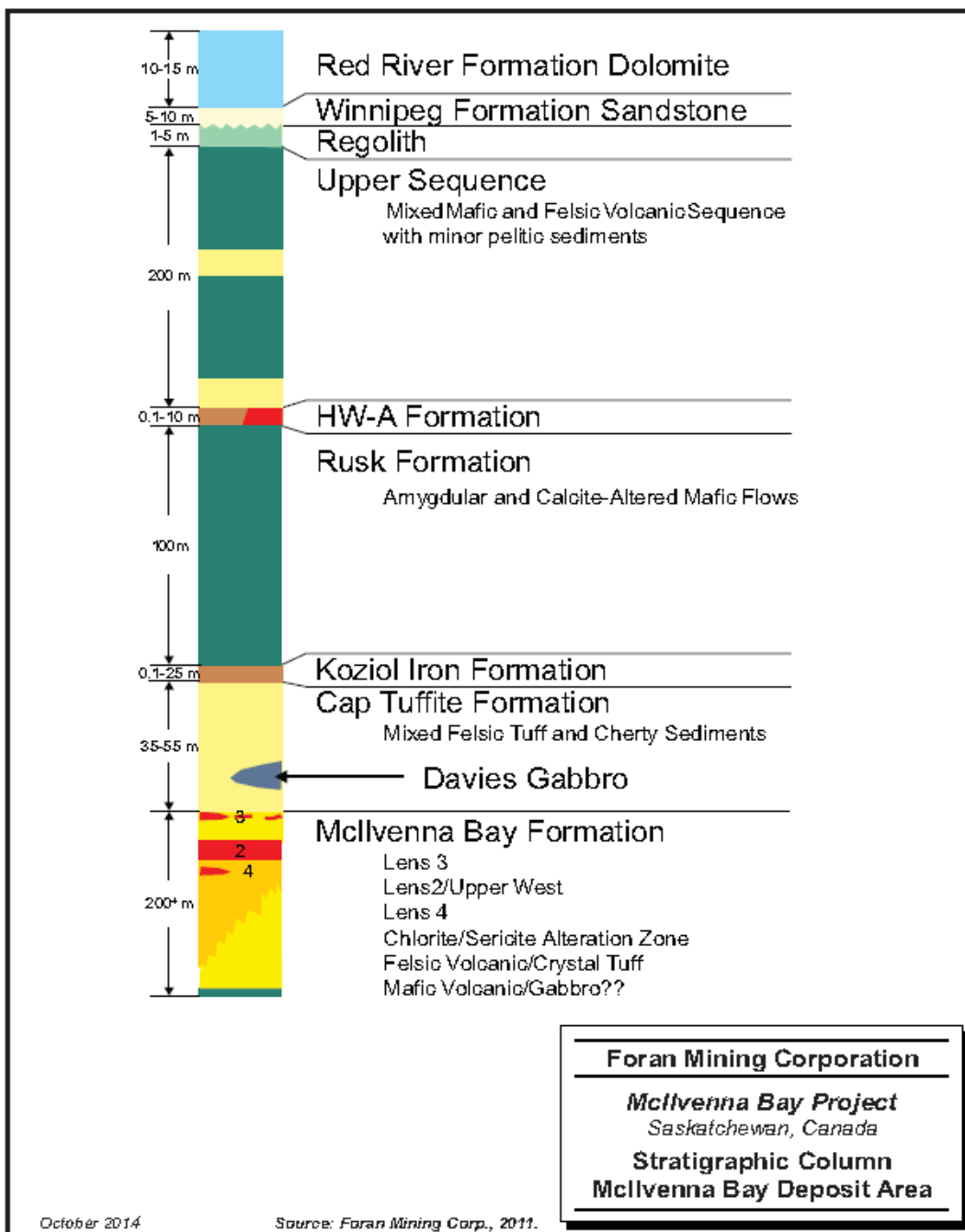


Figure taken from the 2015 Technical Report.



Figure 7-4: Cross-Section 9650E (Looking WNW)

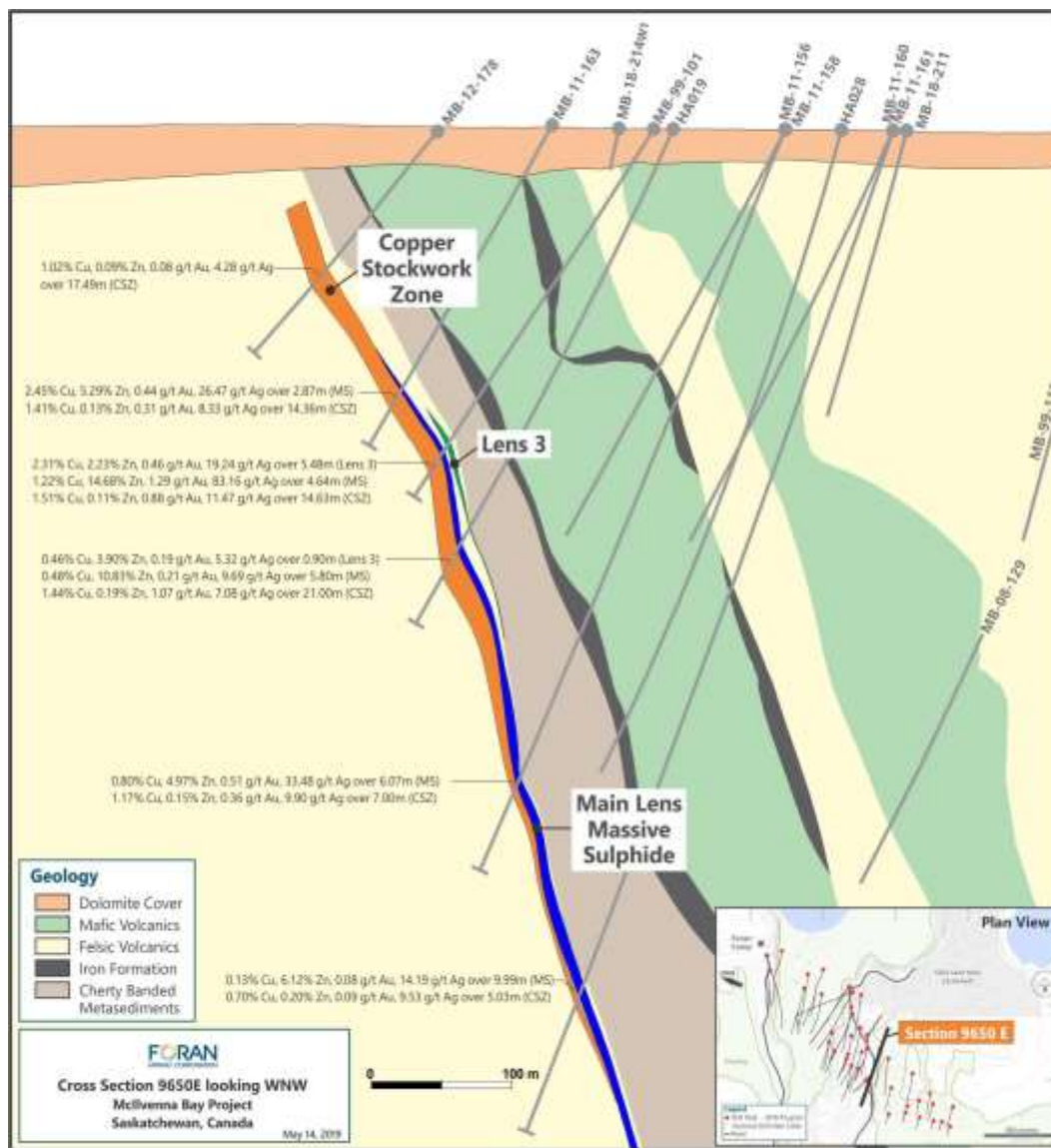


Figure supplied by Foran in June 2019.

Overlying the mineralized horizons of the Mcllvenna Bay Formation is the Cap Tuffite Formation, a sequence of intercalated felsic volcanic and cherty metasediments which have been intruded by sills and dikes of the Davies Gabbro (described below). The unit ranges from 35 m to 55 m thick, is finely banded to finely laminated, and ranges from white to cream to gray-green in color. Sections of the formation range from very finely laminated bleached chert to 1 cm to 10 cm 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 redeposited water-lain distal volcanoclastics and chert. An east to west



zonation is observed in the Cap Tuffite from cherty-dominated in the east to rhyolitic-dominated in the west.

Stratigraphically overlying the Cap Tuffite is the Koziol Iron Formation, a long, continuous exhalative horizon. It is traceable in drill core and by geophysics over several kilometres and, thus provides an excellent stratigraphic marker horizon. The unit is a true oxide-facies iron formation that ranges from 0.1 m to 25 m true thickness and is composed of 1 cm to 5 cm thick bands of fine-grained chert, interbedded with 1 mm to 50 mm massive magnetite bands and 1 cm to 1 m thick massive grunerite $\pm$ garnet $\pm$ magnetite $\pm$ chlorite bands. Occasional pyrite and/or pyrrhotite are observed in selected bands. Near the base of the iron formation is a ± 1 m thick graphitic shear / fault zone which is oriented sub-parallel to the stratigraphy and/or the S1 transposition foliation.

Overlying the Koziol Formation is the Rusk Formation, a thick package of massive and calcite altered mafic volcanic rocks that are approximately 100 m 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, however patchy 1 mm to 2 mm diameter white to pink rounded feldspar amygdules have been noted locally.

Topping the Rusk Formation is another exhalative horizon, the HW-A Formation which ranges from 1 cm to 5 m thick and shows a transition from west to east from oxide-facies iron formation to massive pyrite $\pm$ sphalerite.

Overlying the HW-A Formation is the +400 m thick Upper Sequence, a bimodal package of volcanic units that have been difficult to correlate from hole to hole. Approximately 45% of the unit is composed of aphanitic gray felsic volcanics and 50% fine-grained mafic volcanic rocks. Some of the mafic units may be gabbroic intrusions. Approximately 5% of the unit is composed of graywackes and at least two additional oxide-facies iron formation horizons. Individual members of the formation are difficult to trace between drill holes, as the existing drill holes that are collared far enough to the north to intersect the Upper Sequence are sparse and generally widely spaced. The Upper Sequence is not yet defined to the extent that it could be broken down into formational units. The down-plunge drilling program has discovered that the Upper Sequence may be the core of a regional synclinal structure and that the bimodal sequence may be structurally repeated by both folding and faulting (Lemaitre 2000).

The Davies Gabbro, a plug up to 100 m thick east of Mcllvenna Bay, extends westward toward the centre of the sulphide body, where it narrows into a series of thin dikes. The gabbro appears to be a series of sills that have intruded along the bedding planes of 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 dikes. It appears that the gabbro intruded along the bedding planes of the wet cherty banded sediments of the Cap Tuffite.



7.4 Structure

Stratigraphy in the deposit area strikes between 275° and 295° and dips to the north at 65° to 70°, although in selected areas it dips vertically. The deposit has the same orientation as the stratigraphy and also plunges at approximately 45° to the northwest. 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 Mcllvenna 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 F2 event (Lemaitre 2000). This isoclinal folding was responsible for the development of the dominant foliation (S1) in the deposit area, oriented at approximately 280°/65°, and resulted in the transposition of the original bedding into the plane of the S1 fabric, so that the stratigraphy is now 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 (possibly F3) 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 northwest or northeast. 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. It may be responsible for the broad warping of the stratigraphy observed in the magnetic maps between the Hanson Lake and the south end of Mcllvenna Bay (Lemaitre 2000).

Some evidence of faulting is 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. Where faulting is present, it often appears to be oriented sub-parallel to the stratigraphy and may represent discontinuities that helped to facilitate the transposition during deformation.

7.5 Mineralization

Mcllvenna Bay is a VMS deposit consisting 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. 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 Mcllvenna Bay Deposit includes five separate zones and two styles of mineralization that are mineralogically and texturally distinct and typical of VMS deposits.

- Massive to semi-massive sulphide mineralization in the Main Lens and Lens 3.
- Stockwork-style sulphide mineralization in CSZ directly beneath 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 (FW), which occurs as a separate lens underneath the CSZ for approximately 150 m of strike length and could represent a fault offset and repetition of the Main Lens and CSZ.

The Main Lens at Mcllvenna Bay is a large-massive to semi-massive sulphide horizon containing a metal zonation consisting of Cu- and Au-rich material near the upper plunge line of the deposit which. Down-dip it transitions into a more Zn and Ag dominant massive sulphide. In the 2013 resource estimate (Rennie 2013), the Main Lens was subdivided into the copper-rich Upper West Zone (UWZ) and the more zinc-rich Zone 2, based on these differences in mineralogy. However, statistical analysis of the assay grades within the lens suggests that there is a gradual transition between the two zones and that a hard boundary is not appropriate. Therefore, in the previous 2019 resource estimate (Lewis, San Martin and Gowans 2019), the Main Lens Massive Sulphide was reported as a single zone and this relationship continues in the current resource. The Main Lens Massive Sulphide is a continuous mineralized horizon which varies from 0.1 m to 18.0 m in thickness and averages 3.5 m overall (Figure 7-5).

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 CZS varies from 0.3 m to 26.0 m in thickness with an average thickness of 12.0 m.

The Main Lens Massive Sulphide and the underlying CSZ are generally in contact throughout the deposit, giving the bulk of the deposit an average thickness of 15.5 m overall. The mineralization in the deposit plunges at approximately -35° northwest from near surface, for a down-plunge length of approximately 2,000 m (Figure 7-6).



Lens 3 is a massive sulphide that sits approximately 10 m to 30 m in the hanging wall above the Main Lens and could represent a stacked massive sulphide lens within the deposit (Figure 7-4). This lens has been traced intermittently along a strike length of 1,600 m and plunges parallel to the underlying Main Lens and CSZ. Lens 3 ranges in thickness from 0.1 m to 8.0 m and averages 2.0 m.

The Stringer Zone (SZ) comprises a narrow, intermittent lens of stringer-style sulphide mineralization that occurs sporadically between the massive sulphides of the Main Lens and Lens 3. The zone has a strike length of 850 m and averages 4.5 m in true thickness through the deposit.

The Copper Stockwork Footwall Zone (FW) is a separate lens that underlies the SZ and has been intersected in nine drill holes over approximately 150 m of strike length in the shallow, central part of the deposit. The lens varies in thickness from 0.3 to 38 m, with an average thickness of 30 m. The FW 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 FW 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.

Figure 7-5: Longitudinal Section through the Main Lens Massive Sulphide

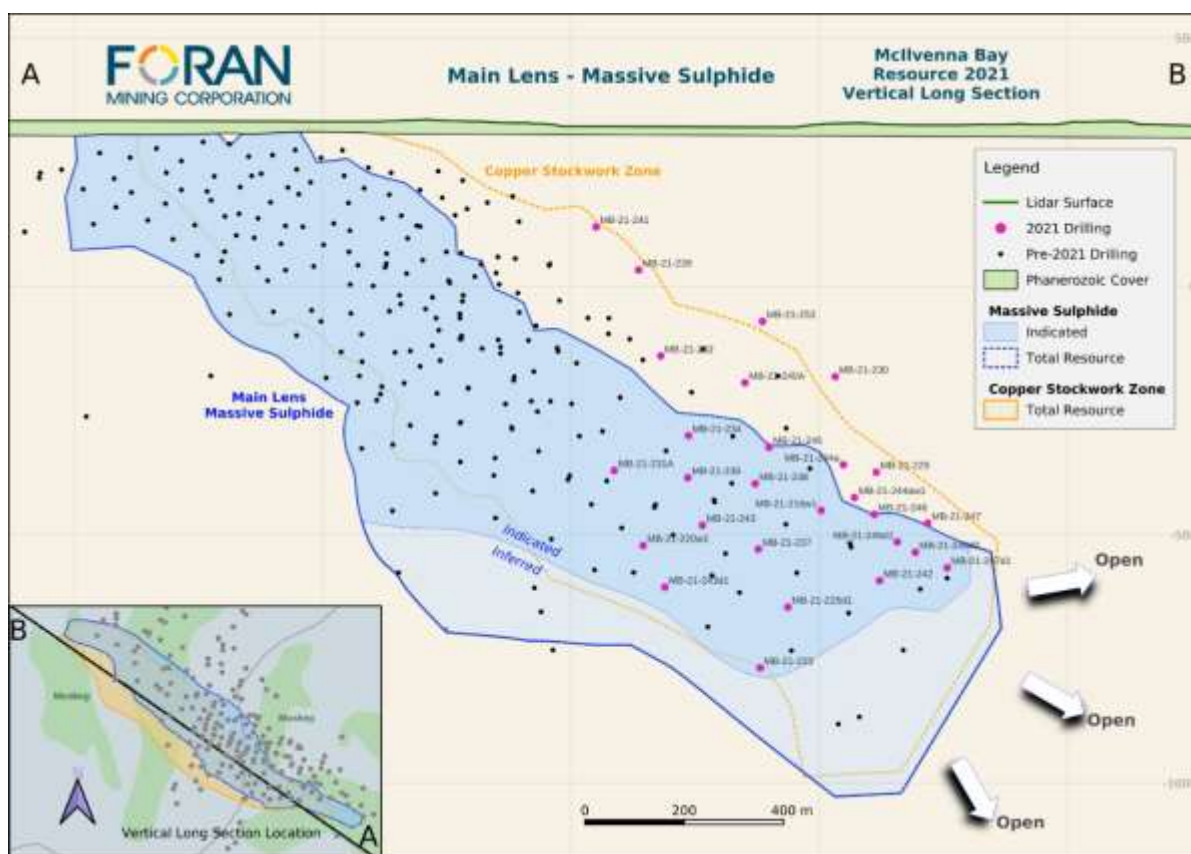


Figure provided by Foran and dated as of November 2021.



Figure 7-6: Longitudinal Section through the Copper Stockwork Zone

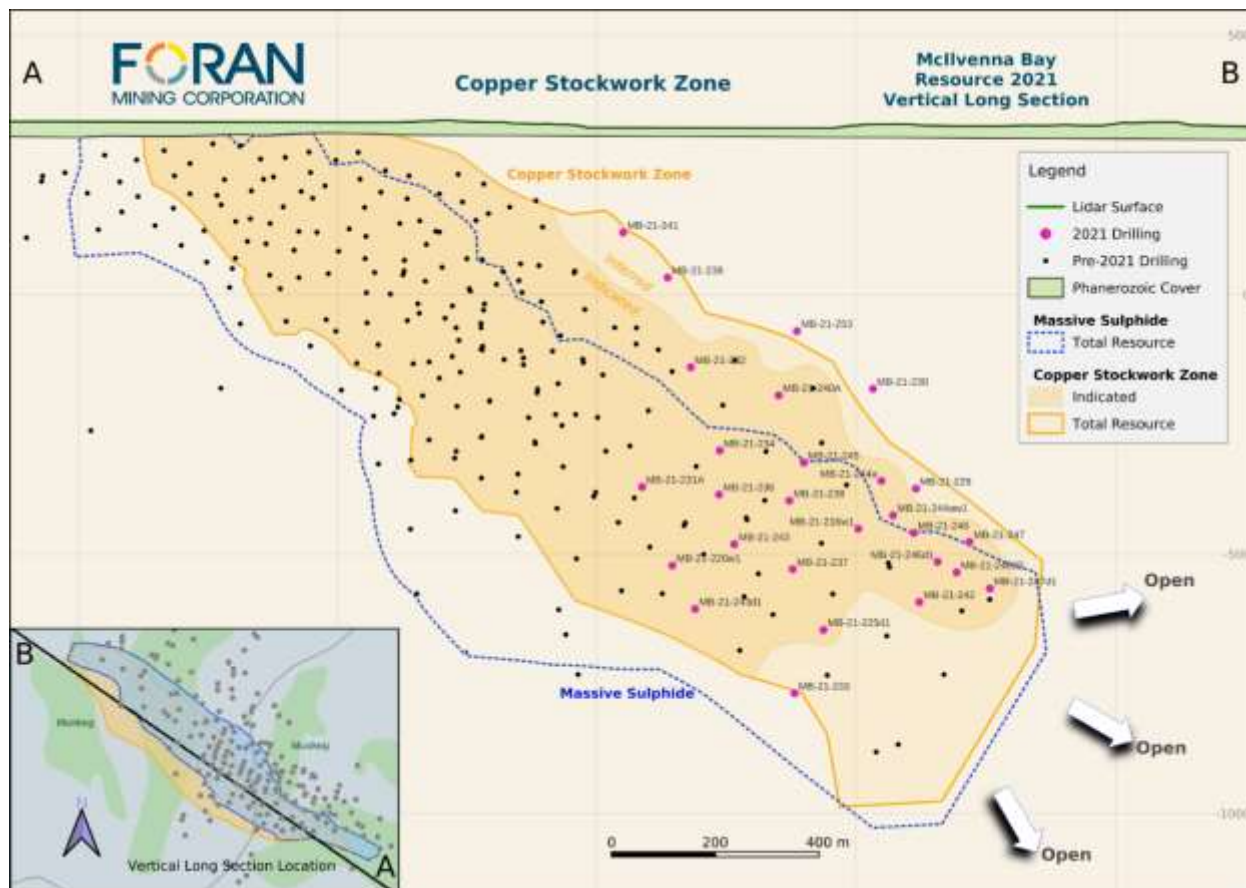


Figure provided by Foran and dated as of November 2021.

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 subrounded pyrite grains resembling “buckshot” in a fine-grained sphalerite-rich matrix. Sphalerite occurs 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 brown to medium brown in color. Faint banding of the massive sulphides is occasionally apparent. Up to 10% fine-grained gray quartz, and occasionally fine calcite, are also observed in the interstices. Subangular to subrounded inclusions or fragments of massive black chlorite ranging from 2 mm to 50 mm in diameter comprise 10% of the unit. Patchy but commonly rounded chert fragments ranging from 1 cm to 3 cm in diameter can constitute up to 20% of the unit locally. Such chert, when present, is often surrounded by zones 1–3 cm in thickness that are enriched in 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 a similar stringer style of mineralization has 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 as the “Stringer Zone” in the 2021 resource estimate (Lewis, San Martin and Jones 2021). 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. The chalcopyrite lines fine, hairline fractures within the strongly silicified host, occurring in 5–10 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 possible the FW zones, respectively overlying and underlying the Main Lens, is interpreted by Foran geologists to indicate the occurrences of smaller hydrothermal pulses at intervals along the timeline.

In the 2015 Technical Report (Makarenko et al. 2015), it was noted that “the UWZ, MSZ2, 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 Foran exploration programs subsequent to publication of the 2015 Technical Report, and the zones are currently still open down plunge.



8.0 DEPOSIT TYPES

Portions of this section were extracted from previous McIlvenna Bay Project Technical Reports 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 tholeiitic to transitional, bimodal, and submarine in origin (Galley, Hannington and Jonasson 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 metal-rich hydrothermal fluids. These deposits commonly occur in clusters that form a VMS camp. In many cases it can be demonstrated that the sub-seafloor fluid convection system was driven by large (15 km 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. 2006).

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). The lens 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.

The stockwork zone is thought to represent the near-surface channelways of a submarine hydrothermal system and the massive sulphide lens represents the accumulation of sulphides precipitated from the hydrothermal solutions onto 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. 2006).



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 $\pm$ Au. They may extend vertically below a deposit for several hundred meters or may continue above the deposit for tens to hundreds of meters as a discordant alteration zone (e.g., 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 (e.g., Amulet, Noranda, LaRonde, and Bousquet deposits), representing synchronous and/or sequential phases of mineralization formation during successive breaks in volcanic activity (Galley et al. 2006).

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 and continuous zone directly underlying the Massive Sulphide. The sulphides contain copper and zinc, with low lead, 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.



9.0 EXPLORATION

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 to infill the existing drill pattern on the McIlvenna Bay Deposit. Phase I of this program commenced in December 1998 and carried on through the winter of 1998-1999. A total of 55 holes were drilled during this program, totalling 27,958 m. 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 2,938 m 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 14 June 2000 (Galley et al. 2006). This block-model estimate was based on a total of 63,344 m of diamond drilling from 124 holes, of which 33,350 m 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 31 May 2000, Foran had drilled an additional 59 holes totalling 33,350 m, 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 580 m, 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 being successful in extending the deposit an additional 300 m vertically below the plunge of the previous resource base.

After 2000, exploration work on the property ceased, and the option agreement with the HLJV 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 an NI 43-101 Technical Report (Cook and Moore 2006). The mineral resources dropped significantly owing to an increase in the cut-off value used, which resulted in removal of much of the Copper Stockwork Zone (CSZ) as it was then termed.

In early 2007, Foran completed an airborne deep-penetrating Versatile Time-Domain Electromagnetic (VTEM) survey over portions of the Bigstone, Balsam and McIlvenna Bay properties. The program comprised 404.6 line-km on 150 m 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 16 November 2006 (Cook and Moore 2006). Seven diamond drill holes were completed for a total of 6,455 m. Drill holes were between 691.5 m and 1298.4 m 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 on MSZ2. 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 refinanced 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 5,056.0 m. This program targeted a portion of the CSZ and was designed to infill and prove up the continuity over a portion of the zone in the central part of the deposit, 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, and RPA was retained to update the mineral resource estimate (Rennie 2011) for the CSZ. The zone was re-interpreted, using a nominal 0.5% Cu cut-off value and a minimum apparent thickness of 3 m. The other zones were largely unchanged, with the exception of Lens 4, which was incorporated into the FW. The re-inclusion of the CSZ 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 8,158 m completed in 18 holes. The purpose of the drill program was to infill the deposit to further increase the confidence in the resource, collect sample material for metallurgical testwork, 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 resurvey program was completed for all drill hole collars that could still be identified on the property. In addition, downhole gyroscopic surveys were carried out in 39 of the historical 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 versatile time domain electromagnetic (VTM plus) and horizontal magnetic gradiometer (mag) over those areas of the McIlvenna Bay property not covered in 2007 (Figure 9-1).

In 2012, Foran completed 3,825 m 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 to upgrade the classification and extend the known mineralization. Drilling was predominantly completed utilizing HQ-sized core to provide



additional material for future metallurgical testwork. Geotechnical and hydrogeological studies were also conducted during the program.

Metallurgical testwork on the samples collected from the 2011 drilling was completed in June 2012. The work was carried out by G&T Metallurgical Services Ltd. (G&T), of Kamloops, BC. Three composite samples, consisting of 516 kg of drill core, were created for each of three different mineralogical domains: the CSZ, MSZ2, and UWZ. The samples were then used in batch and locked cycle flotation testing, as well as determination of Bond Work Indices.



Figure 9-1: Geophysical Surveys 2007 to 2014

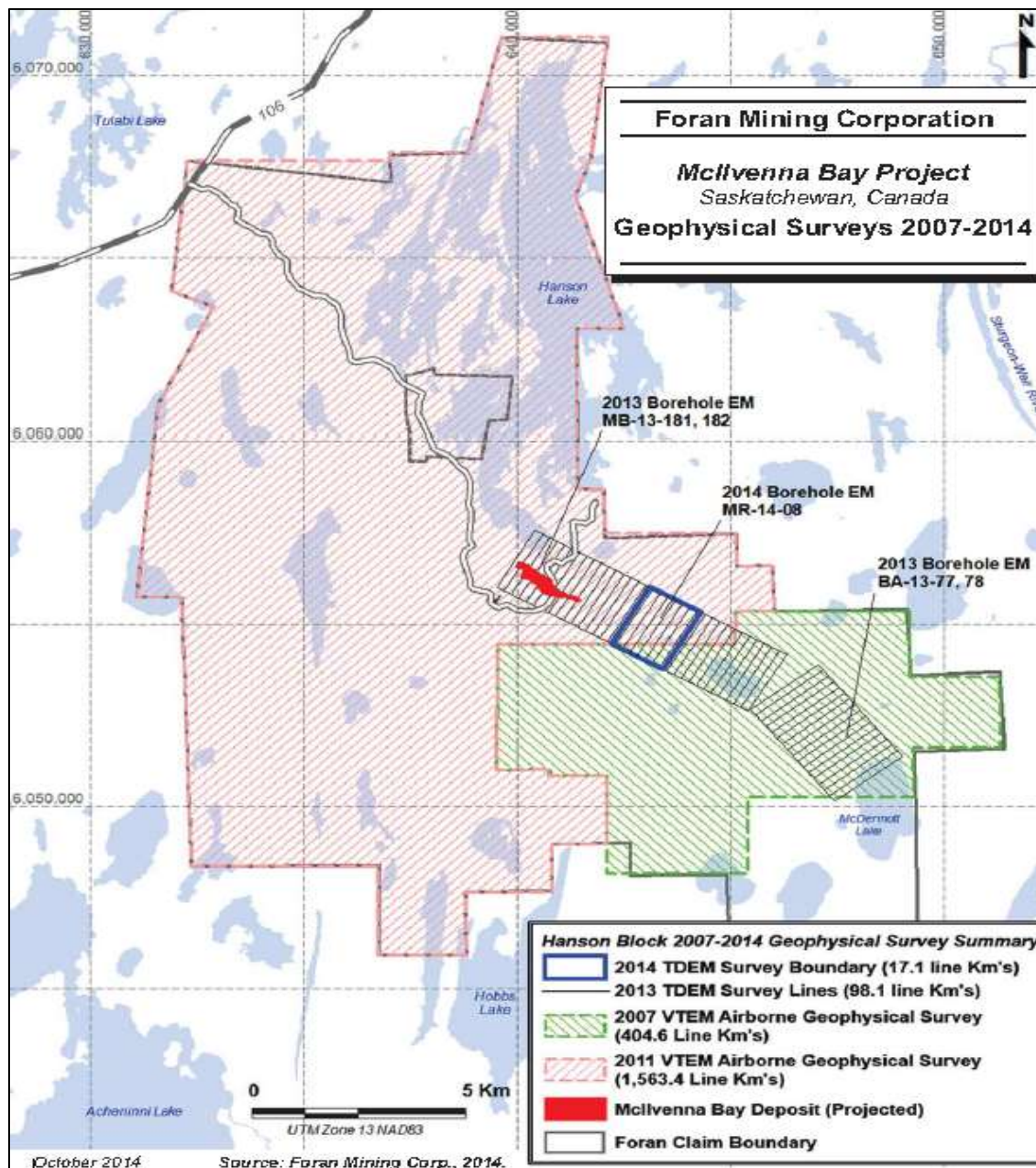


Figure taken from the 2015 Technical Report.

It is unknown whether G&T was ISO certified at the time of the work in 2012. However, the Steward Holdings Group Limited (Steward Group) purchased G&T in February 2011. On 29 June 2012, Campbell Brothers Limited (ALS Group [ALS]) announced its purchase of the Steward Group. While metallurgical processes are not usually covered by ISO certification, the assaying conducted by the laboratories usually is. ALS laboratories are ISO/IEC 17025 accredited.



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 preliminary economic assessment (PEA) completed by JDS and disclosed in the PEA Technical Report dated January 2015 (Makarenko et al. 2015).

Coincident with the update of the mineral resource estimate, Foran drilled four diamond drill holes totalling 2,243 m on the deposit in 2013. These holes were not incorporated into the 2015 estimate and a review by RPA concluded that the impact of these holes on the mineral resource estimate used in the 2015 Technical Report 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 of this report.

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

In addition to the work done on Mcllvenna 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 Mcllvenna Bay Deposit and the trend of the geology to the southeast into the Balsam area. The survey grid covered portions of the Mcllvenna 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 Mcllvenna Bay Deposit.

Foran has also drilled a number of holes on regional targets within the property boundary but outside of the immediate Mcllvenna 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 2,163 m on five separate regional targets in the southern portion of the property.

In 2013, nine holes, totalling 3,211 m were drilled in the Balsam / Thunder Zone area, located 5–7 km southeast of the Mcllvenna 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 be pod like in nature. Near the end of the program, a new 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



intersected 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.

The 2013 ground geophysical survey also identified a new large deep-seated EM conductor (Target A). In 2014, a short geophysical program comprising 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 Target A. 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 1,864 m 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. 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. Geological logging indicated that the stratigraphy was cut by a dike at that location of the conductor, so 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 the 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 will be included in a pending M.Sc. thesis report.

9.2 Foran Exploration 2015 to 2018

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 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 resource as possible into the indicated category which could then potentially be converted into reserves for the upcoming Feasibility study.



In 2018, Foran conducted 26,827 m of drilling in 60 drill holes targeting the deposit. The program was completed in two phases, with 14,986.5 m in 32 drill holes (including several wedged holes) completed during the Phase I winter program and 11,840.5 m 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 higher 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 testwork. In addition to converting inferred resources to the indicated category, other program components included geotechnical, hydrogeological and metallurgical testwork.



Figure 9-2: Regional Drilling Summary 2011 to 2014

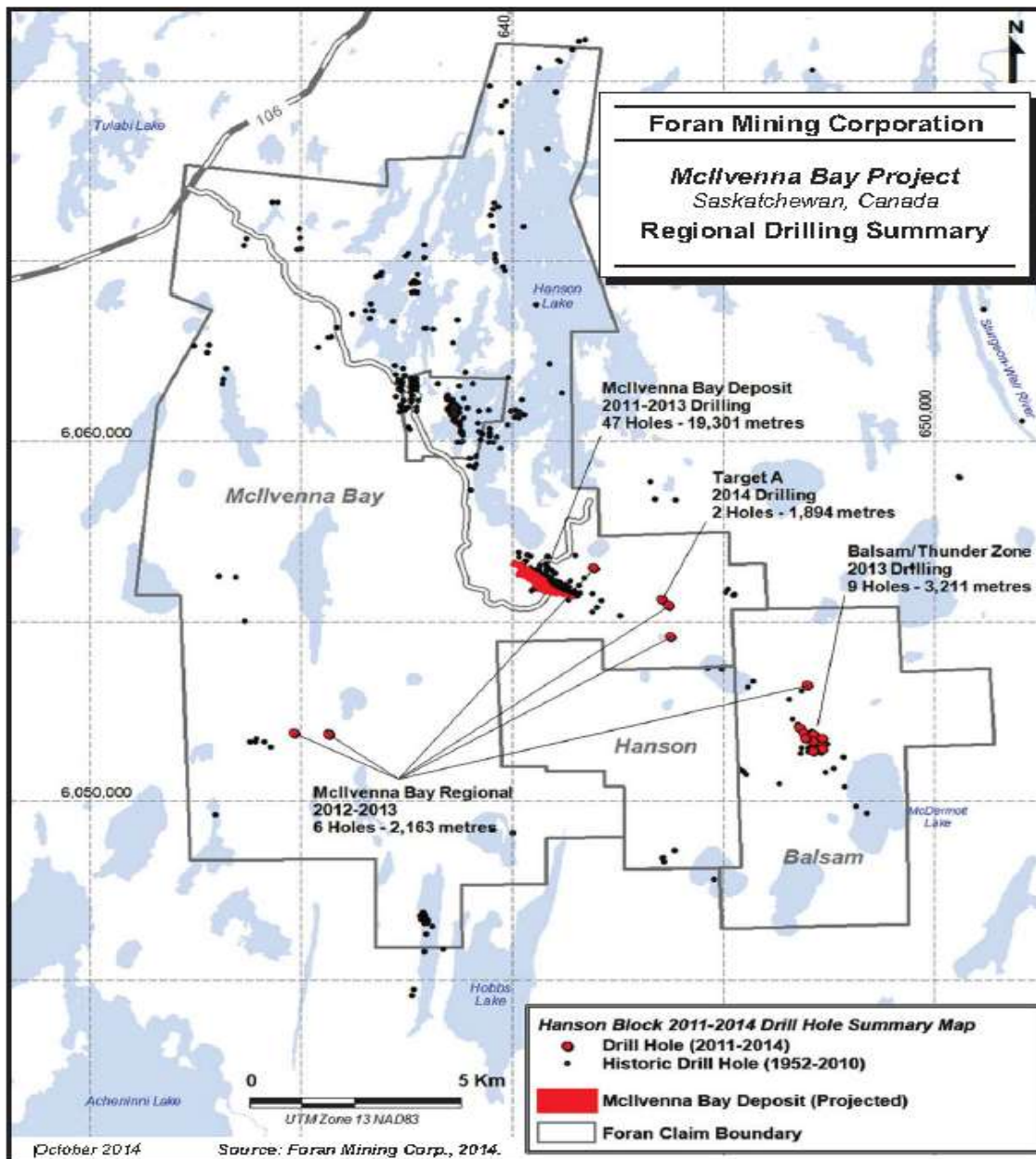


Figure taken from the 2015 Technical Report.



Figure 9-3: Lithogeochemical Sampling Surveys 2012 to 2014

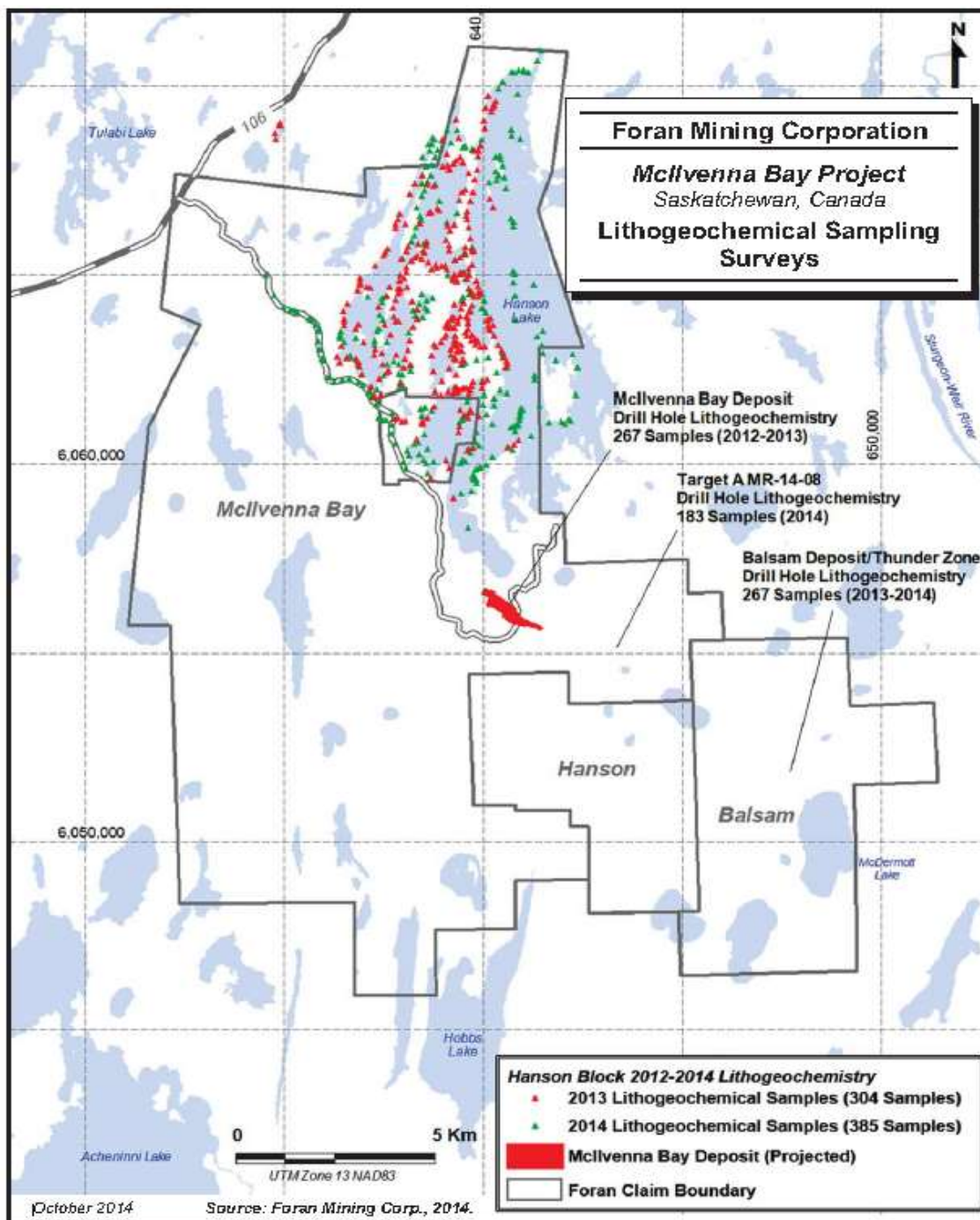


Figure taken from the 2015 Technical Report.

Geotechnical components of program included 3,733 m of detailed geotechnical logging on holes drilled at orientations amenable to both structural and resource studies. In addition, three short geotechnical holes (151.3 m) were drilled to characterize the proposed portal location and four short vertical holes (104 m) 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 drill holes, with either a quarter or half of each sampled interval submitted for testing. Metallurgical work is being carried out by Base Metallurgical Laboratories Ltd. (BML), of Kamloops, BC. A total of 1,440.96 kg of drill core was provided from the 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. Testwork currently under way is comprised of grind and flotation circuit tests, as well as DMS upgrading to maximize value.

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 historical downhole surveys beyond blockages were re-opened with a drill on the pad and resurveyed with a True North Gyro.

To develop a larger library of rock density measurements across the deposit, Foran employees performed 1,932 bulk density measurements both from 2018 drill holes, and from historical 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 scale with a precision of within 1 g (0.5 g for skilled operators). The larger sample sizes (often between 2 kg and 4 kg) minimized the error introduced by the 1.0 g precision. These bulk density samples are considered more representative of the actual density of the mineralized material in the ground, compared to 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 in the 2019 resource estimate which was the subject of an NI 43-101 Technical Report.

The Glencore Agreement was subsequently allowed to lapse and no further exploration was conducted on the deposit until 2021. This work and results are the subject of this report.

9.3 Foran Exploration 2021

No further exploration / drilling was conducted on the McIlvenna Bay Deposit until the winter of 2021. The 2021 drill program was focused on infill drilling, targeting the inferred portions of the resource to convert additional tonnes to the indicated category, as well as step-out drilling designed to expand the deeper parts of the deposit in the up-dip



direction. The program was completed between January and July and consisted of 39 drill holes encompassing 27,298 m (including 11 wedges). The program included the use of oriented core technologies to better understand the structural setting of the deposit and all holes were surveyed with a gyro tool to provide accurate drill hole traces at depth.

The program was successful in significantly increasing the density of drill holes in the deeper part of the deposit and in expanding the mineralized horizons up-dip, along the upper plunge line of the deposit. The 2021 drilling continued to define a trend of thicker mineralization in the Copper Stockwork Zone in this area, which is often coupled with higher copper grades in the adjacent massive sulphide horizon. Ultimately, the drill program culminated in significant increases in the indicated portion of the McIlvenna Bay resource.

During the 2021 program, Foran continued to collect bulk density measurements for all sample intervals within the deposit, to bolster the density database for the deposit. The measurements were made using the weight in air / weight in water method on complete sample intervals, similar to the process employed in 2018.

During the winter of 2021, the immediate deposit area and projected extensions to the northeast were covered by ground EM surveys in an effort to determine the potential for additional lenses and/or extensions of known lenses. The results of the survey have been modelled and testing of potential conductors is being planned for future programs.

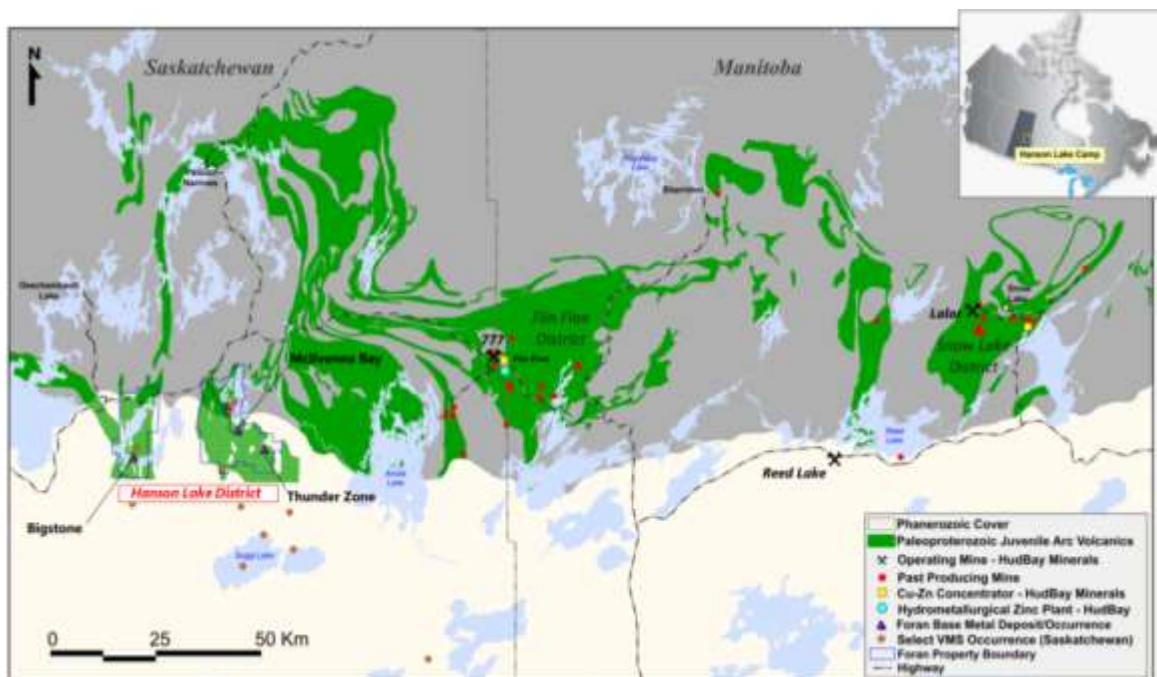
9.4 Exploration Potential on the Property

The following information has been extracted from Section 24 of the April 27, 2020, APG Mining Consultants Inc. (APG) Technical Report (AGP 2020) and updated where necessary.

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 225 km 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 (Mt) of production (Figure 9-4).



Figure 9-4: Flin Flon Greenstone Belt



Taken from the 27 April 2020 APG Technical Report.

There are numerous historical 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 Foran. Foran also holds a large claim block in the Bigstone Lake area, located 25 km to the west of McIlvenna Bay, which is host to the historical Bigstone deposit and a number of other compelling exploration targets.

During 2021, Foran has undertaken a review of the regional geology and geophysical datasets to better understand the potential of the project area and to develop new target areas for future exploration. As part of this review, several prospective target areas were identified from regional geophysical datasets and a number of areas were covered by ground EM surveys to define drill targets for further exploration.

As the McIlvenna Bay Deposit continues to advance toward production, the exploration focus in the area is transitioning to identifying additional regional target areas that could eventually develop into satellite mineralized zones, and be processed at a central processing facility adjacent to McIlvenna Bay. Foran is working on growing and advancing its pipeline of exploration targets and will continue to focus on regional targets in future programs.

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 historical Bigstone Deposit located on the Bigstone property, 25 km 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 9-6).

9.4.1 Thunder Zone

The Thunder Zone is located approximately 7 km 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 over the trend from McIlvenna Bay to the nearby 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 hole 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.

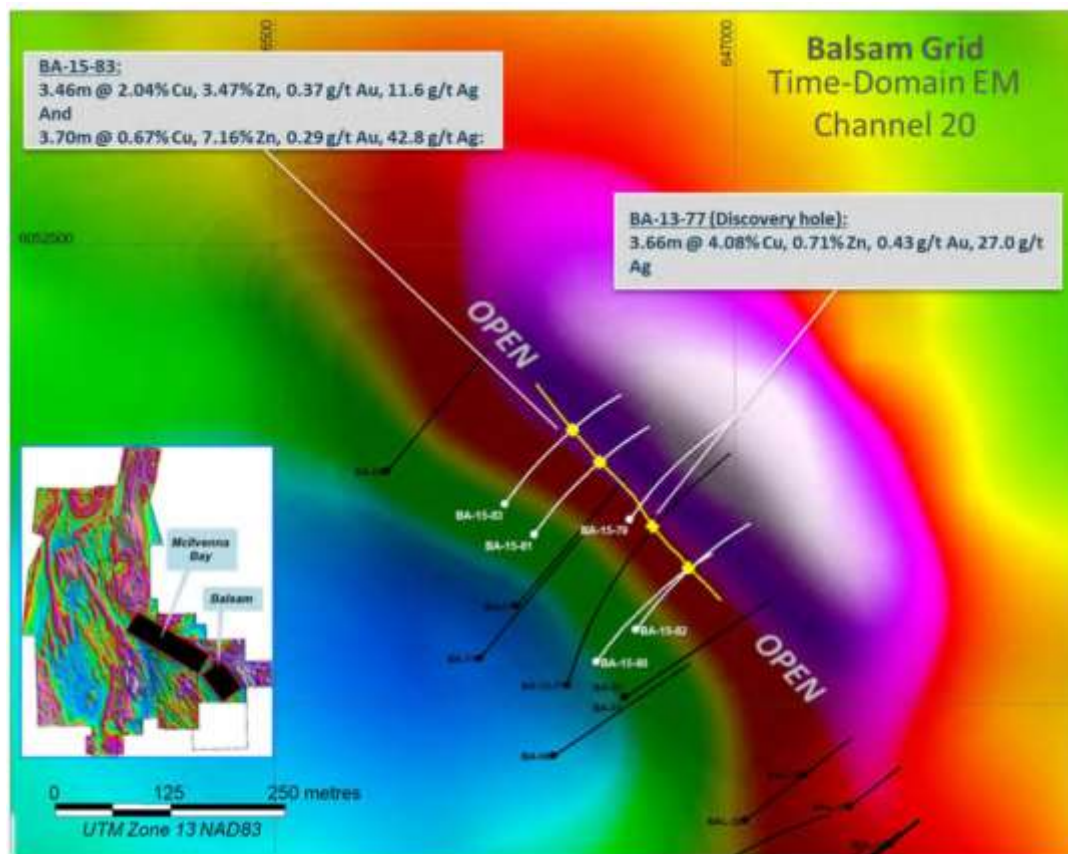
In 2015, Foran completed five drill holes totaling 1,914m at the Thunder Zone to follow up on the 2013 discovery. The program was successful in intersecting massive and stringer sulphide mineralization in four of the five holes drilled which partially defines a moderately dipping mineralized zone, hosted 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 drill program came from hole BA-15-83 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 downhole by a second zone containing 0.62% Cu, 3.41% Zn, 0.36 g/t Au, and 27.24 g/t Ag over 8.39 m (including an interval of 3.70 m grading 7.16% Zn). A map indicating a gridded profile of the EM response over the Thunder Zone with the drill hole superimposed on it is provided in Figure 9-5.

The Rosalind prospect is located ~1800m along strike to the SE of Thunder/Balsam and potentially represents an up-plunge extension. A surface EM survey has been conducted to better define targets in this area.



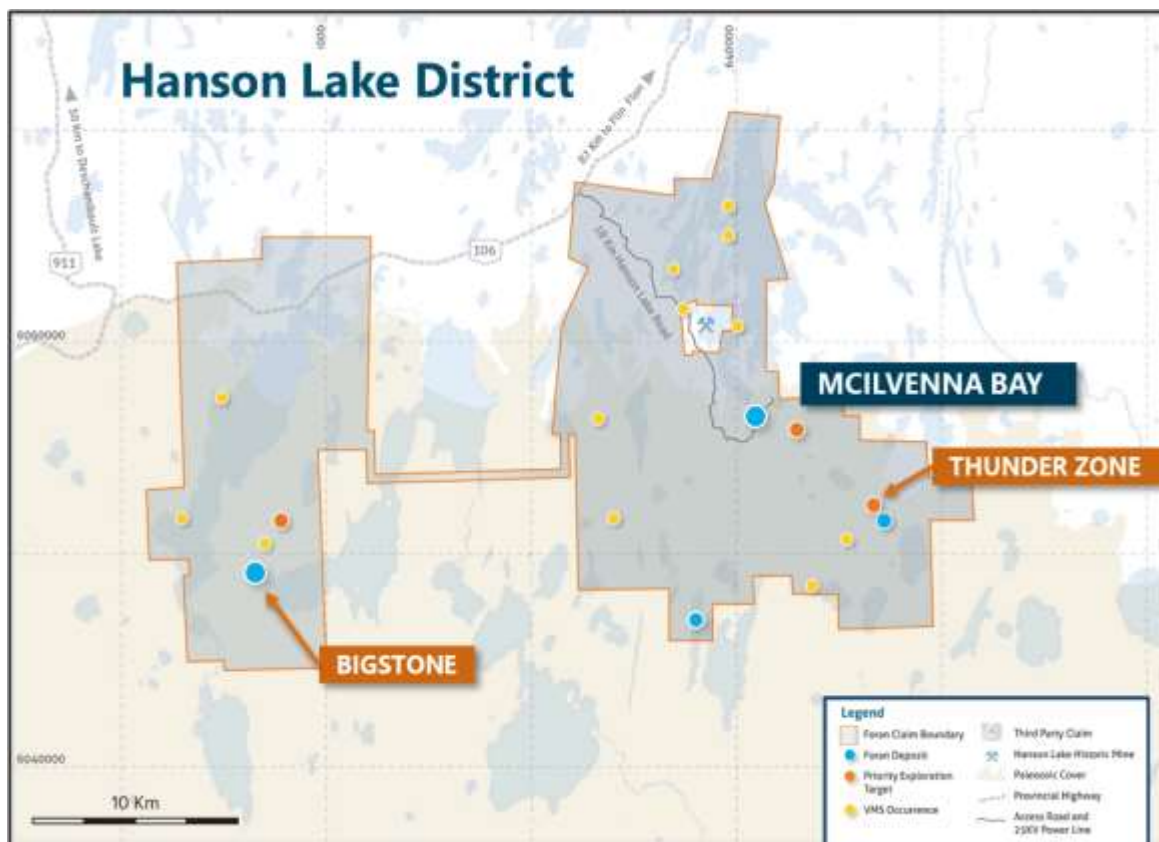
Figure 9-5: Thunder Zone – Gridded EM Response and Drilling



Note: Source Foran



Figure 9-6: Hanson Lake District Properties

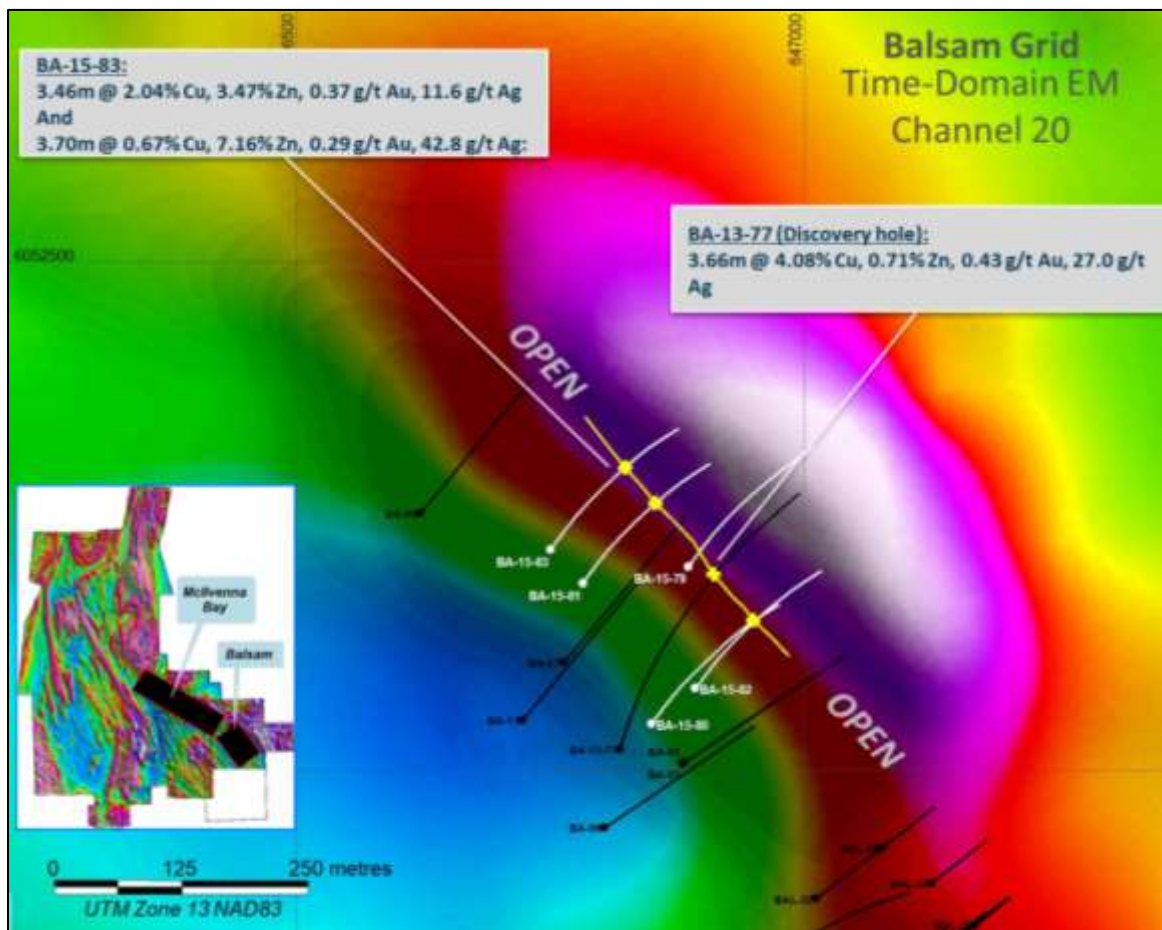


Taken from the 27 April 2020, APG Technical Report.

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, downhole, by a second zone containing 0.62% Cu, 3.41% Zn, 0.36 g/t Au and 27.24 g/t Ag over 8.39 m (including an interval of 3.70 m grading 7.16% Zn). A map showing a gridded profile of the EM response over the Thunder Zone with the drill holes superimposed on it is provided in Figure 9-7. 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 9-7: Thunder Zone – Gridded EM Response and Drilling



Taken from the 27 April 2020 APG Technical Report.

9.5 Micon QP Comments

The exploration programs conducted by Foran to date on the Project have continued to delineate additional mineralization at the Mcllvenna Bay Deposit. Further work will be needed to determine the full extent of the mineralization, both in the down-plunge direction and at depth. If the Project was put into production, though, the extent of the mineralization—either down-plunge or at depth—would most likely be more economically defined by underground drilling.

Further exploration programs will be necessary to identify the extent and tenor of the mineralization in the satellite zones which have been identified either historically or more recently by Foran. Further exploration will also be able to determine if these satellite zones of mineralization are economically viable for the purposes of exploitation along with the main Mcllvenna Bay Deposit.



10.0 DRILLING

Portions of this section were extracted from previous Mcllvenna Bay Project Technical Reports and updated or edited where necessary.

10.1 Drilling to 2014 (RPA Discussion)

Diamond drilling was started by Cameco in 1988. Cameco and their partners drilled 68 holes, of which 56 targeted the Mcllvenna Bay Deposit. All other drilling in and around the Project area has been completed by Foran. A summary of drilling within Mcllvenna Bay Deposit up to August 2014 is provided in Table 10-1.

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

Company	Year	Number of Holes	Meters 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 Technical Report.

RPA noted that the totals provided by Foran for the Cameco-era drilling do not match those contained in the database. 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 not possible to reconcile the drilled totals. The meters 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 relogged older drill core to update the records.



The 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 redrilled; 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 is not included in Mcllvenna 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 are recorded in logs and verifiable or have been re-acquired through logging of early core. In the opinion of the Micon QP these historical RPA concerns have been addressed during subsequent reviews of Foran's drill hole database by Micon QPs and no longer constitute any risk to the integrity of the database.

Cameco and Foran employed similar drilling procedures on Mcllvenna 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 75 m 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 ± 50 m in the east-west direction and ± 25 m 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 resurveyed using a Gyro tool from Reflex Instruments, which is not affected by magnetic minerals. There were significant differences found between the results for the two instruments. Based on this result, the gyro tool was deemed to provide the more accurate survey result and this tool was used for all subsequent downhole surveys. For all subsequent 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 resurveying was also conducted to relocate as many of the older drill collars as possible, to validate the historical 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. Micon's QP have subsequently reviewed the databases and note that any historical deficiencies noted by



RPA were corrected and the current drilling database is sufficient to support a mineral resource estimate.

10.2 Foran Diamond Drilling 2014 to 2018

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 the indicated category which could potentially be converted into reserves for the upcoming Prefeasibility Study.

The 2018 program consisted of 26,827 m of drilling in 60 drill holes which were completed in two phases, with 14,986.5 m in 32 drill holes (including several wedged holes) completed during the Phase I winter program and 11,840.5 m 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. A plan map showing the collar locations and hole traces is provided in Figure 10-2.

Drill hole collars were located in the field by a surveyor / geologist with a survey transit or differential GPS and two foresight 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 were 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 10-1: Drill Collar Locations to August 2014

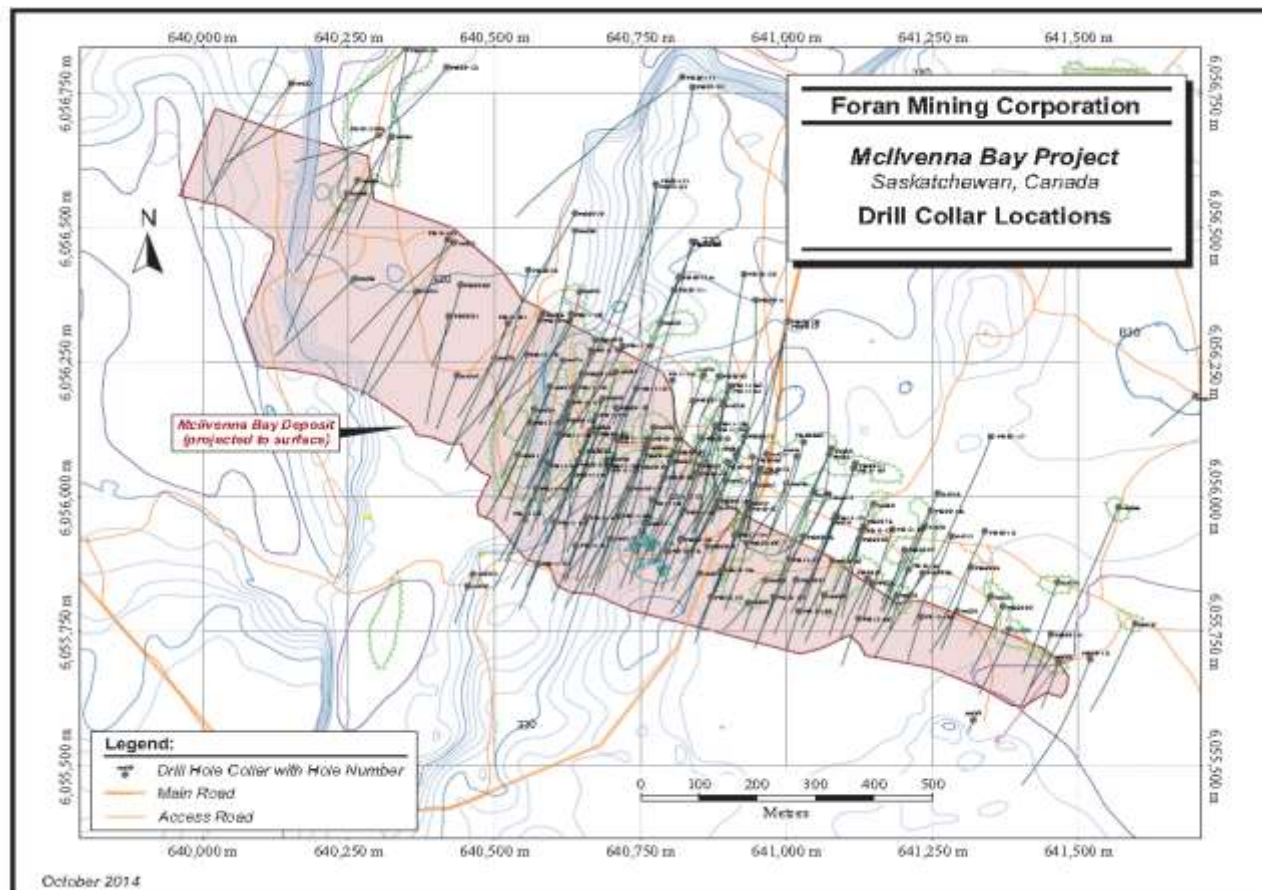


Figure taken from the 2015 Technical Report.

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

Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation (masl)	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



Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation (masl)	Azimuth from Total Station	Dip	Length (m)
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 Meters						14,986.50

Table supplied by Foran in June 2019.

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

Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation (masl)	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



Drill Hole	UTM NAD 83 Zone13 Easting	UTM NAD83 Zone13 Northing	Elevation (masl)	Azimuth from True North Gyro	Dip	Length Drilled (m)
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 Meters						11,840.5

Table supplied by Foran in June 2019.



Figure 10-2: Drill Collar Locations to September 2018

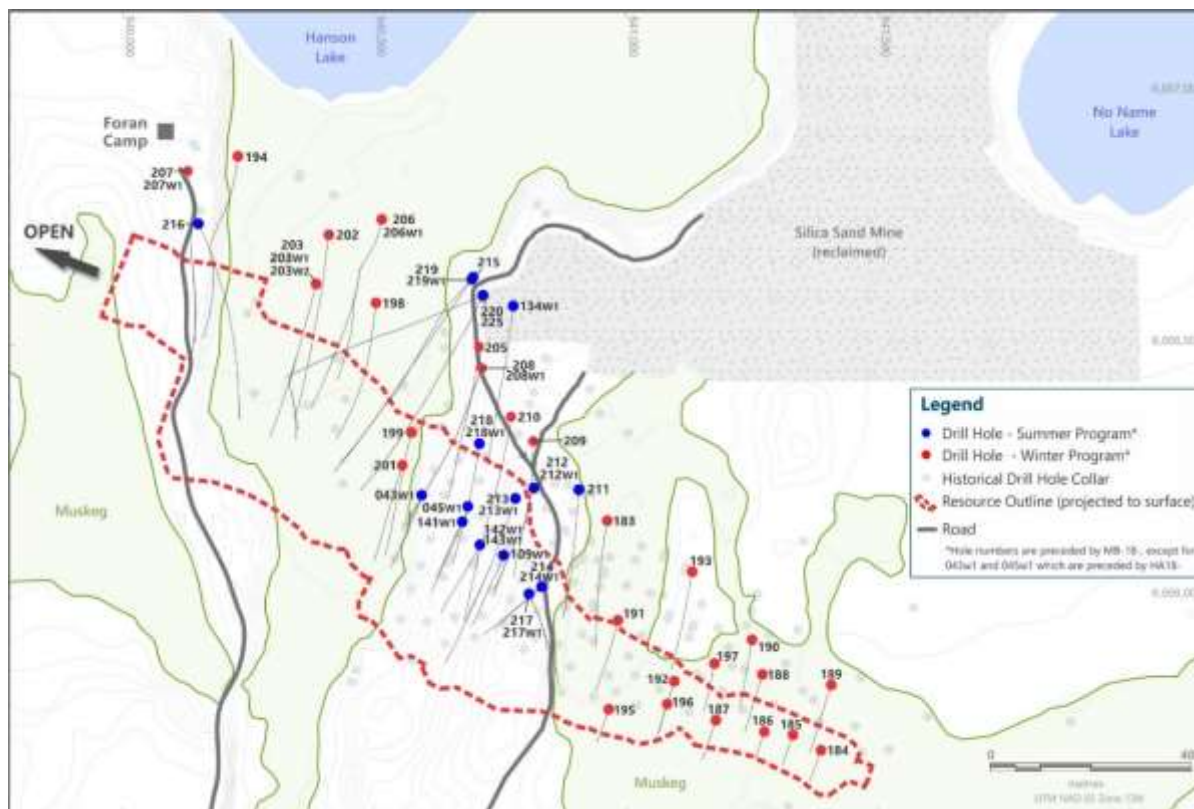


Figure supplied by Foran in June 2019.

During drilling, downhole survey readings were routinely collected by the drill crew at 50 m intervals as the holes progressed, utilizing an EZshot survey tool. 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 Mcllvenna Bay, the azimuth data for some readings from the EZshot tool may be unreliable, but they provide a backup of survey data for the hole if 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 were 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 and is not affected by magnetic rock units in the stratigraphy. Surveys were generally conducted from the bottom of the hole up, with measurements collected at 5 m or 10 m intervals throughout the hole. The survey data collected by the gyro tool are 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 hole's azimuth. Instead, the tool calculates the station azimuths independently.

Once the core was received at the McIlvenna Bay core shack, geological and geotechnical core logging were 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 that 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 and will greatly assist in the interpretation of the geology of the deposit.

10.3 Foran Diamond Drilling 2018 to Present

After 2018, no further drilling was conducted on the McIlvenna Bay Deposit until the winter of 2021, when Foran embarked on another large infill and expansion drill program at the deposit. The program was designed to convert as much of the existing deeper inferred deposit resources as possible into the indicated category, as well as to complete a series of closely spaced drill holes targeting an up-dip area in the deeper parts of the deposit where there appeared to be room to expand the deposit and grow the indicated category resource. This increase in the indicated category resource for the deposit was completed to support potential reserve growth for the upcoming Feasibility Study.

The 2021 program consisted of 24,893 m of drilling in 33 drill holes (including 10 wedged holes) completed between 16 January and 15 July. Table 10-4 provides detailed information on the drill holes from the 2021 program and a plan map showing the collar locations and hole traces is provided in Figure 10-3.



Table 10-4: Summary of the 2021 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)
MB-21-216w1	640150.44	6056743.87	329.10	155.73	-70.85	891.00
MB-21-220w1	640716.53	6056600.97	325.66	209.00	-68.00	993.00
MB-21-225d1	640715.85	6056600.97	325.58	246.63	-72.47	1130.60
MB-21-229	640082.04	6056730.19	327.72	194.53	-71.90	1016.00
MB-21-230	640047.79	6056646.22	327.58	190.83	-72.30	855.00
MB-21-231	640525.88	6056462.42	319.78	190.30	-72.90	183.50
MB-21-231A	640525.88	6056462.43	319.78	190.93	-72.90	798.00
MB-21-232	640383.74	6056360.74	320.16	190.03	-72.90	585.00
MB-21-233	640483.87	6056882.32	319.04	189.03	-73.10	1204.00
MB-21-233w1	640483.87	6056882.32	319.04	189.03	-73.10	689.00
MB-21-234	640399.64	6056470.57	320.15	191.03	-73.90	735.00
MB-21-235	640384.53	6056729.28	319.36	190.83	-73.90	272.00
MB-21-236	640435.55	6056542.17	319.62	190.83	-74.00	801.00
MB-21-237	640384.5	6056729.07	319.33	190.93	-72.70	938.50
MB-21-238	640321.51	6056649.39	318.95	190.93	-73.20	825.00
MB-21-238w1	640321.51	6056649.39	318.95	190.93	-73.20	437.90
MB-21-239	640388.37	6056195.61	320.70	194.83	-74.00	399.00
MB-21-240	640251.22	6056499.49	319.68	182.13	-69.00	57.00
MB-21-240A	640251.10	6056498.95	319.60	181.73	-69.10	654.80
MB-21-241	640422.98	6056068.15	321.05	193.64	-80.10	275.00
MB-21-242	640147.84	6056895.90	329.67	182.93	-76.10	1008.00
MB-21-243	640770.11	6056694.32	328.66	217.43	-71.10	1014.00
MB-21-243d1	640770.11	6056694.32	328.66	217.43	-71.10	1092.00
MB-21-244	640149.15	6056703.08	328.99	183.03	-71.10	29.00
MB-21-244a	640149.20	6056703.22	329.08	183.00	-71.00	795.00
MB-21-244aw1	640149.15	6056703.08	328.99	183.00	-71.48	864.00
MB-21-245	640155.14	6056600.72	328.20	153.03	-72.00	777.00
MB-21-246	640152.40	6056767.30	329.54	215.93	-78.00	900.00
MB-21-246d1	640152.40	6056767.30	329.54	215.93	-78.00	954.00
MB-21-246d2	640152.40	6056767.30	329.54	215.93	-78.00	963.00
MB-21-247	640153.74	6056738.94	329.28	218.23	-78.00	906.00
MB-21-247d1	640153.74	6056738.94	329.28	218.23	-78.00	1005.00
MB-21-253	640142.00	6056415.00	325.00	171.03	-67.00	845.50
Total Meters						24,892.80

Table supplied by Foran in September 2021.



Assay results from holes MB-21-230 and MB-21-253 were not available in time for the data cut-off for the 2021 resource estimate. Both holes are located outside the grade shells constructed for the 2021 resource estimate.

Figure 10-3: Drill Collar Locations for the 2021 Drilling Program

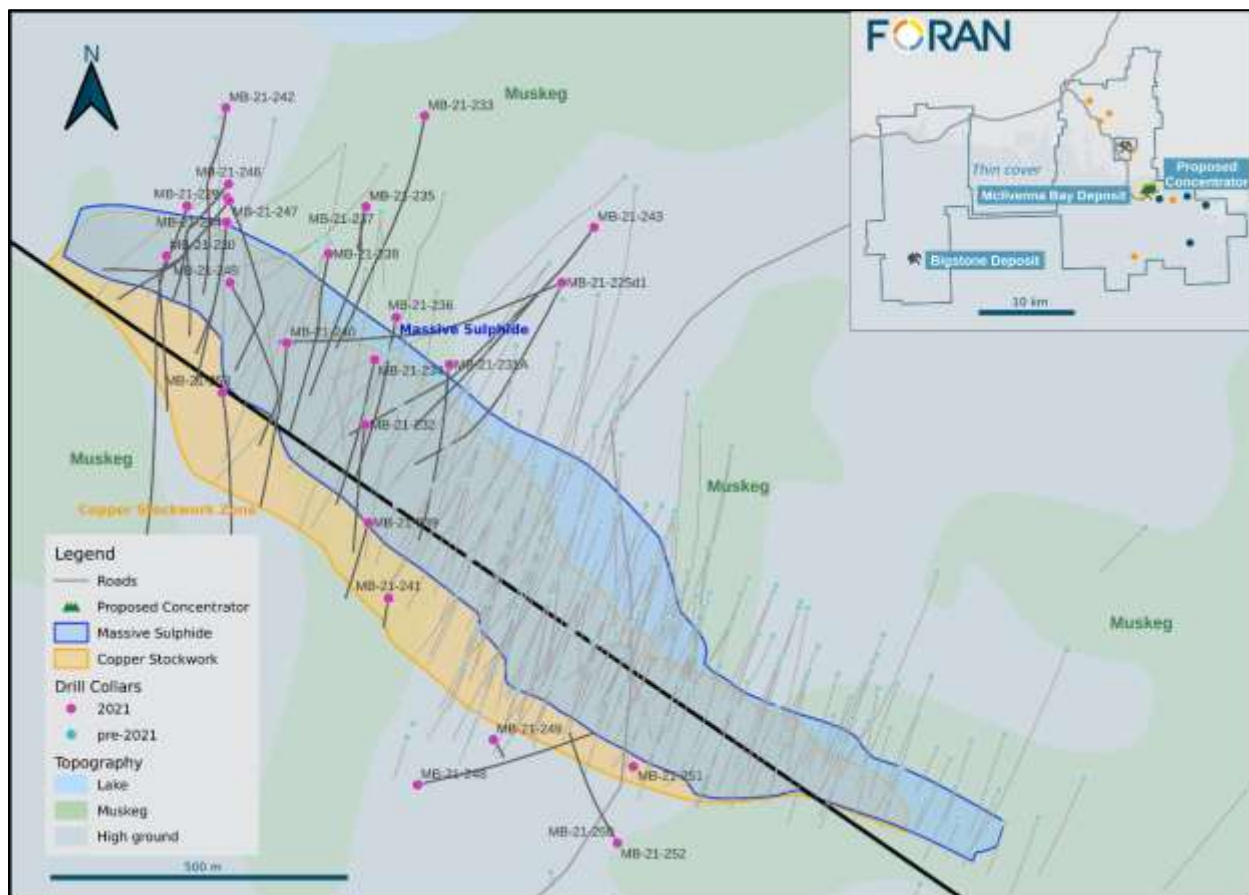


Figure supplied by Foran in November 2021.

Drill hole collars were located in the field by a Foran geologist utilizing a differential GPS and then aligned along the correct azimuth using a TN41 Gyrocompass from Reflex Instruments. 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 were drilled to depth, leaving the HQ rod string as casing. At the completion of the drilling, the hole was surveyed with a north-seeking Gyro tool to provide an accurate trace of the hole at depth.

For the 2021 drill program, all drill holes were completed using the ACT III digital core orientation system from Reflex Instruments to provide oriented drill core. This process provides valuable information on the true orientation of structures in the ground and greatly assists in the interpretation of the geology of the deposit.



Once the core was received at the McIlvenna Bay core shack and oriented (as described above), geological and geotechnical core logging were completed. GeoSpark core logging software (used under license from Geospark Consulting Inc.) was used to collect 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.

10.4 Micon QP Comments

Micon's QP has reviewed and discussed the drilling programs with Foran personnel, both during the 2018 site visit and at other times throughout its audit and review of the mineral resource estimate. Micon's QP believes that the programs have followed the best practice guidelines for exploration (CIM, November 2018).

A table of significant assays was not added to this report as there were no assays from the 2021 drilling program that materially impacted the resource model or were significantly greater than those seen in prior exploration drilling programs. A drill cross-section of the deposit on Section Line 9650E was included previously as Figure 7-4 during the discussion in Section 7.3 regarding the property geology. Long sections showing the pierce points of the drilling on the Main Lens Massive Sulphide and the Copper Stockwork Zone were included in Section 7.5 as part of the discussions related to the mineralization in the various lenses.

The drilling intersections down the drill holes varied greatly in width due to the mineral lens pinching and swelling both along strike and down dip and the angle at which the drill holes intersected the mineralization. Therefore, the true widths of the lenses vary as well but the variation in the true widths of the mineralized lenses as well as the average widths are noted in Section 7.5 as part of the discussions related to the mineralization.

In the opinion of Micon's QP for this section, Foran has achieved its objective of outlining the overall extent of the mineralization in the McIlvenna Bay Deposit with its diamond drilling programs. The deposit is still open at depth, both down plunge and down dip. The drilling programs on the McIlvenna Bay Deposit were sufficiently extensive to be used as the basis of a mineral resource estimate at the McIlvenna Bay Project.



11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Section 11.1 was extracted from the previous Technical Report for the McIlvenna Bay Project (Makarenko et al. 2015) and updated or edited by the Micon QP where necessary.

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)

For Cameco's exploration up to 1991, little information is available on the security measures employed, QA/QC procedures, and who 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. A total of 152 check assays were performed at TSL, Bondar-Clegg & Company Ltd. (Bondar-Clegg) in Ottawa, and TerraMin Research Labs Ltd. (TerraMin) in Calgary. Cameco was satisfied 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).

Eco-Tech, Bondar-Clegg, and TerraMin were independent laboratories that historically serviced the mining industry. TSL is still currently operating in Saskatoon. Discussions related to the independence and certification of these laboratories is covered in Section 11.2.

11.1.2 Foran (1998 to 2000)

The bulk of the assaying from the early 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 laboratory.

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 assaying. 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 its 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 believed 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 holes 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)

For the Foran drilling in 2007 and 2008, core was split using a diamond saw. Sampling was done on a range of intervals up to a maximum of 1.24 m, 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 laboratory. The site is fairly remote and, while not fenced, was continually supervised and relatively immune to incursions by 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 was 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, relogged the core, and did some resampling 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. Based on subsequent reviews of Foran's database it is the opinion Micon's QP that the issues noted by RPA regarding the historical database have been addressed through subsequent relogging, sampling and QA/QC programs. It is the Micon QP's opinion that Foran's subsequent QA/QC programs follow industry best practices.

11.1.4 Foran (2011 to 2013)

The initial winter 2011 program was managed under contract to Equity Exploration Consultants Ltd. After 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. At the time of the last RPA site visit, Foran was in the process of moving to a new building, constructed especially 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 meter, with breaks at lithological and mineralogical contacts. Routine bulk density measurements were taken from intact core specimens.

RPA inspected several sampled intervals and considered 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. Micon's QP has conducted extensive reviews of Foran's sampling practices, both during the site visits and during subsequent discussions with Foran personnel, and have found that the sampling practices are appropriate for both the deposit type and the mineralized intervals encountered.

Drill core from early programs was either stored in racks or cross-stacked boxes on site. Foran has collected the cross-stacked core, reboxed 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 laboratory 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 standards 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 (Rennie 2011)

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

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 limit provided by the manufacturer for the standard based on the results of round-robin testing.

Table 1-1 above and notes were extracted from the 2015 Technical Report (Makarenko et al. 2015) 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 very 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 rerun and the results of these reruns were used in the database.

Three standards and one blank failure were obtained in 2012, which resulted in the rerun of the affected batches. The results passed for all samples on rerun and the revised data



was incorporated into the database. One batch from the 2013 winter program was rerun owing to a standard failure.

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

Equity personnel relogged 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 relogging of portions of holes to help resolve complications in the geological interpretations.

Specific Gravity (SG) 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 to estimate block density from the interpolated zinc grades. This formula is as follows:

$$SG = (0.075 \times Zn) + 2.8124$$

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

Foran subsequently made many more density measurements, and by the end of 2013, there were 2,501 determinations in the database. RPA recommended that the regression formula be updated with this more recent data.

In RPA's opinion, Foran's logging, sampling, and assaying protocols were consistent with good industry practice. The QA/QC program, as designed and implemented by Foran was adequate, and the assay results within the database are suitable for use in a mineral resource estimate. In the opinion of the Micon QP, who further reviewed Foran's historical and current logging, sampling, and assaying protocols, these protocols remain consistent with good industry practice. The QA/QC program, as designed and implemented by Foran prior to 2013, was 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 Technical Report and agrees with RPA's opinion that it was



suitable for conducting mineral resource estimates. In the opinion of the Micon QP, who further reviewed Foran's historical and current logging, sampling, and assaying protocols, those protocols remain consistent with good industry practice. The QA/QC program, as designed and implemented by Foran prior to 2013, was adequate and the assay results within the database are suitable for use in a mineral resource estimate.

11.2.1 Notes Regarding Assay Laboratories

TSL's 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. (Bondar-Clegg) was an independent commercial assay laboratory company which was taken over by ALS Chemex Labs Ltd. in December 2001. No information is available regarding Bondar-Clegg certifications.

Eco-Tech Laboratories in Creighton, Saskatchewan was an independent commercial assay laboratory company, but was struck off the public company registry in Saskatchewan in 2002 (Saskatchewan Gazette 2002, p. 1486). There is no information available regarding Eco-Tech certifications.

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

XRAL Laboratories Ltd. (XRAL) was purchased by the SGS Group in 1988. XRAL was an anacronym that stood for X-Ray Assay Laboratories Ltd. There is no information available regarding XRAL certifications.

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 Foran or the other companies which conducted work on the McIlvenna Bay Project. Micon's QP believes that the laboratories applied best practices in undertaking their assaying techniques and obtained any certifications necessary to operate as independent laboratories serving the mineral industry. However, historically the independent mining laboratories were not



certified and maintained round robin studies¹ with other laboratories, as well as internal standards.

11.3 Sample Preparation Analysis and Security (2018 to Present)

For the 2018 through 2021 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 were 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 meter in length in homogeneous material, with a maximum of 1.5 m or a minimum of 0.20 m 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 laboratory 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 increase the number of duplicate analyses completed by the assay laboratory. 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. A list of the certified standards used for the program is 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 was created from the original reject for every eleventh sample as a prep duplicate. These samples were analysed in sequence 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 rerun 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-

¹ A round robin study, or interlaboratory study (ILS), is an experimental methodology to determine reproducibility of a "process" where tests are performed independently multiple times and the results are analyzed statistically to assess their variability.



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 rerun. All other instances occurred once the new QA/QC protocols had been established and, in these cases, a group of seven samples was rerun (three samples either side of the failure in the sample stream). In all cases, the standard material passed on rerun 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 QA/QC 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 rerun and the revised assay results for these samples were incorporated into the database.

For the 2021 program, sampling followed the revised QA/QC protocols established during the latter part of the 2018 program. For the 2021, program five certified standards were employed, as described in Table 11-3. 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 2007-2018 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 11-3: Reference Standards – 2021 Program

Standard	Au (ppb)		Ag (ppm)		Cu (%)		Pb (%)		Zn (%)	
	Mean	SD	Mean	SD	Mean	SD	Mean	SD	Mean	SD
CDN-ME-1410	542	48	69	3.8	3.80	0.170	0.248	0.012	3.682	0.084
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 3,338 samples (including all QA/QC materials) were analysed during the 2021 program and there were five standards, four blanks and one duplicate failure reported from the assaying. In all instances, a group of seven samples was rerun (three samples either side of the failure in the sample stream). In all cases, the standard material passed on rerun and the revised assay results for the rerun 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 were 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 laboratory. 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.

During the 2021 program, Foran continued the protocol of collecting 'bulk density' measurements for all assay samples collected during the program, using the same weight in water - weight in air method. Overall, a total of 1,861 samples were measured during the 2021 program and added to the project database.

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

Micon's QP was able to review the work conducted by Foran on its 2021 drilling programs and is of the opinion that the QA/QC programs have been conducted in line with CIM best practices. Also, Micon's QP reviewed the work that had been conducted during the prior exploration programs and, as noted in the prior 2019 Technical Report (Lewis, San Martin and Gowans 2019), those programs had been undertaken in line with CIM best practices. Therefore, Micon's QP believes that the work undertaken by Foran is suitable for use in as the basis of a mineral resource estimate on the McIlvenna Bay Project.



12.0 DATA VERIFICATION

12.1 Site Visits

12.1.1 2021 Site Visit

Micon's QP conducted a site visit from 17–19 November 2021. The site visit was undertaken to independently verify the geology and QA/QC programs. No samples were taken during the 2021 site visit since verification of the mineralization occurred during the 2018 site visit.

The QP notes that Foran's 2021 drilling program consisted of the following:

- The majority of the drilling was infill drilling to upgrade inferred resources into indicated resources at depth. This involved both drilling new holes from surface between existing holes and wedging off from existing drill holes at various depths and orientations.
- A smaller number of new drill holes were conducted to extend the deposit in the existing down dip and down plunge direction of the mineralized trend. These holes hit the targeted mineralization as predicted, demonstrating the continuation of the mineralization at depth along the down dip and down plunge direction.
- The 2021 drilling program was focussed on the main McIlvenna Bay Deposit and other known mineralized zones on the property were not drilled.

Foran and Micon jointly reviewed the best locations for the 2021 drilling infill and extensional drill holes, to maximize the potential coverage of the drilling program. Micon's assistance to Foran involved taking the current drilling database and deposit model and using that information to conduct the potential placement of either new drill holes or wedge holes collared at various depths off previous drill holes. Once the initial work was completed, this information was delivered to Foran so that it could make the final determination as to which drill holes they were going to complete.

During the site visit mineralized intervals for the following drill holes were reviewed:

- MS-21-231 A
- MS-21-243 D1
- MS-21-246 D2
- MD-21-251

The drill logs were found to be consistent with the mineralization observed in the drill core while. No drilling was being conducted on site during the 2021 site visit.



The 2021 site visit was conducted with the assistance of Brian Janser and Mike Schamborzki of Foran, who were in the camp at the time of the site visit.

Drill site drone footage of the drill sites was reviewed but the drill platforms have been reclaimed using the trees that were knocked down during drill pad construction. The trees have been redistributed over that area of the drill pads (Figure 12-1 and Figure 12-2). The first snowfall of the season occurred prior the QP arriving at the exploration camp, and it was deemed unsafe to visit the reclaimed drill pads due to the potential tripping hazards obscured by the masses of dead trees located on the pads.

Figure 12-1: Drone View of the Reclaimed Drill Hole Site for MB-21-230



Foran drone photograph, November 2021.

Figure 12-2: Drone View of the Reclaimed Drill Hole Site for MB-21-253



Foran drone photograph, November 2021.

12.1.2 2018 Site Visit

A previous site visit was conducted between August 16 and August 18, 2018, during which the Mcllvenna Bay property was inspected, and various aspects of the Project were discussed. The exploration programs for the Project were also discussed in detail. 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 12-3 shows the core storage area at Foran's Mcllvenna Bay camp during the 2018 site visit. This storage area holds both the historical core, as well as the core from Foran's previous drilling programs.

Figure 12-4 shows the buildings used to log core and prepare samples at Foran's Mcllvenna Bay camp.

Figure 12-5 shows one of the drills set up and drilling during the Micon site visit in August, 2018.

Figure 12-3: The 2018 Core Storage Area at Foran's Mcllvenna Bay Camp



Figure 12-4: Buildings Related to Logging and Sample Preparation at Foran's Mcllvenna Bay Camp in 2018



Figure 12-5: Drill Set-up and Drilling During the 2018 Micon Site Visit



After the 2018 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	Number 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	HNO ₃ -HF-NCIO ₄ -HCl/AA	1	1,000
	Cu	%	HNO ₃ -HF-NCIO ₄ -HCl/AA	0.01	80
	Pb	%	HNO ₃ -HF-NCIO ₄ -HCl/AA	0.01	80
	Zn	%	HNO ₃ -HF-NCIO ₄ -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 also 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 measurement.

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-HNO₃) liberates most of the metals noted in, Table 12-4 except those marked with an asterisk where the digestion will not be complete.



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

Sample Number	Micon Assay Results								Foran Original Assay Results				
	Au (ppb) ¹	Au (ppb) ¹	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Specific Gravity	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
780581	110			10.8	0.85	0.02	0.7	2.68	0.1	11.7	0.76	0.02	0.6
780583	140			20.5	2.29	0.02	2.23		0.11	19.6	1.92	0.01	1.83
780584	130			16.9	0.62	0.12	1.08		0.095	17.6	0.51	0.1	0.95
780588	620			45.7	0.77	0.88	10.3		0.56	45.5	0.73	0.75	9.98
780593	420			16.4	1.87	0.04	4.33		0.33	16.2	1.71	0.03	3.93
780597	>1,000	>1,000	1.37	34.2	3.25	0.27	2.57	3.08	1.23	34.2	3.06	0.23	2.63
780600	>1,000		7.27	44.8	5.36	0.05	0.38		6.86	39.9	5.02	0.04	0.4
780604	10			0.4	<0.01	<0.01	<0.01		0.005	0.9	0.005	0.005	0.005
780607	180			7.4	1.49	<0.01	0.06		0.14	3.3	1.38	0.005	0.05
780608	880			35.9	9.05	0.02	0.63		0.75	33.4	9.22	0.02	0.68
780609	320			6.5	1.5	<0.01	0.15	2.68	0.25	6.7	1.46	0.005	0.16
780614	150			3.6	0.66	<0.01	0.04		0.11	5.1	0.72	0.005	0.05
780618	35			2.2	0.65	<0.01	0.03		0.035	3.2	0.64	0.005	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						

1. 1 ppm = 1 g/t = 1,000 ppb = 0.0001%



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.

The TSL internal quality control system conforms to the requirements of ISO/IEC Standard 17025 guidelines and, in April 2004, TSL 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.

12.2 2019 Database Review

Micon received the earlier database on 07 January 2019, in which the data were 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 2021 Database Review

Micon's QPs received a number of updates to the Foran database in 2021, as information related to the drilling became available. The QPs verified that the information in the databases agreed with the original records a number of times as a result. No errors were found in the update to the Foran database. As a result, Micon's QP considers that the Foran database can be relied upon as the basis upon which to conduct an updated mineral resource estimate that, in turn, can be used as the basis for a Feasibility Study on the McIlvenna Bay Project.

12.4 2021 Metallurgical Review

Historical metallurgical testwork was carried out by ALS Metallurgy (ALS) of Kamloops, BC in 2012 and at Base Metallurgical Laboratories (BML), also of Kamloops, BC in 2016. The most recent testwork program was completed in 2019 at BML, and this work provides the basis for the metal recovery estimates included in Section 13.0.

Lyn Jones, P.Eng, is the QP for Section 13.0 and has reviewed the testwork reports from ALS and BML. Mr. Jones is of the opinion that the work performed is in accordance with the CIM Best Practice Guidelines for Mineral Processing.

12.5 Micon QP Comments

The Micon and its QPs 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 use in undertaking the updated mineral resource estimate on the McIlvenna Bay Project. The data provided by Foran are suitable to be used as the basis of a mineral resource estimate used as the foundation of Foran's Feasibility Study for the McIlvenna Bay Project.



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

Element	Ag	Al	As	B	Ba	Bi	Ca	Cd	Co	Cr	Cu
Units	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm
780581	10.1	4.00	10	28	16	24	0.57	22.9	27	35	8,172
780583	20.1	2.96	26	<20	63	6	0.30	80.3	48	62	>10,000
780584	14.8	3.30	30	<20	139	79	0.84	31.0	25	59	5,829
780588	42.0	0.84	202	24	10	64	5.98	302.6	51	34	6,831
780593	15.4	0.90	119	27	3	7	7.97	146.2	24	15	>10,000
780597	35.2	0.96	272	26	3	31	9.78	98.0	24	17	>10,000
780600	39.2	1.34	81	<20	30	72	0.05	18.2	87	59	>10,000
780604	<0.3	1.17	18	47	118	<3	0.33	<0.5	6	67	51
780607	6.1	2.15	27	35	49	28	0.07	3.4	27	61	>10,000
780608	33.0	1.40	140	27	22	42	0.03	33.5	77	77	>10,000
780609	5.9	2.14	73	39	40	32	0.06	5.4	53	61	>10,000
780614	2.2	2.74	29	<20	28	5	0.11	1.7	8	78	6,210
780618	2.0	2.15	5	23	9	4	0.04	1.3	13	70	6,216
Element	Fe	Ga	Hg	K	La	Mg	Mn	Mo	Na	Ni	P
Units	%	ppm	ppm	%	ppm	%	ppm	ppm	%	ppm	%
780581	10.53	25	<1	0.08	32	3.16	727	2	<0.01	2	0.003
780583	11.18	15	2	0.22	25	2.43	511	1	0.01	4	0.021
780584	8.57	16	<1	0.42	31	3.00	579	1	0.03	12	0.068
780588	18.64	34	27	0.02	14	4.24	1,097	2	<0.01	3	0.002
780593	16.53	21	7	0.01	16	6.42	1,443	2	0.01	2	0.002
780597	16.97	25	6	<0.01	20	6.33	1,353	<1	0.01	2	0.003
780600	14.76	13	2	0.21	12	1.06	101	1	<0.01	2	0.001
780604	1.98	6	<1	0.70	6	0.81	284	<1	0.07	7	0.042
780607	5.10	9	<1	0.40	14	1.79	171	1	0.01	1	<0.001
780608	14.80	15	1	0.14	9	1.14	118	1	<0.01	2	<0.001
780609	8.38	9	<1	0.23	16	1.66	247	<1	0.01	1	0.001
780614	4.84	14	<1	0.19	19	2.23	306	2	0.01	3	0.008
780618	3.79	10	<1	0.09	18	1.76	212	1	<0.01	2	0.001



Element	Pb	S	Sb	Sc	Sr	Th	Ti	Tl	V	W	Zn
Units	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm
780581	266	3.77	<3	<5	6	3	0.015	<5	<1	<2	6,393
780583	217	6.81	<3	<5	14	<2	0.022	<5	4	<2	>10,000
780584	1,230	4.09	<3	<5	41	<2	0.046	<5	23	<2	8,909
780588	8,864	>10.00	22	<5	50	<2	0.005	<5	1	<2	>10,000
780593	413	>10.00	17	<5	38	<2	0.006	<5	<1	<2	>10,000
780597	2,901	9.13	36	<5	52	<2	0.007	<5	<1	<2	>10,000
780600	453	>10.00	<3	<5	2	<2	0.013	<5	<1	<2	3,453
780604	28	0.14	<3	<5	8	<2	0.075	<5	26	<2	92
780607	84	2.18	<3	<5	3	<2	0.023	<5	<1	<2	604
780608	224	6.23	<3	<5	1	<2	0.009	<5	<1	<2	5,576
780609	67	4.59	<3	<5	3	<2	0.017	<5	<1	<2	1,451
780614	15	1.14	<3	<5	5	<2	0.011	<5	6	<2	518
780618	7	0.86	<3	<5	2	<2	0.007	<5	<1	<2	378



13.0 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 (ALS) in Kamloops. The work was followed by additional metallurgical testing at Base Metallurgical Labs (BML) in 2016, 2019, and 2021. Several mineralogical assessments by Terra Mineralogical Services were completed in parallel.

The approach to flowsheet development was to prepare 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 conducted on the 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.

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

The latest phase of the testwork, initiated in March 2021 at BML, was intended to support the feasibility study and advance the metallurgy in four key areas.

- Development and evaluation of the MBS depressant scheme to replace $\text{ZnSO}_4/\text{NaCN}$ in the copper circuit
- Development of the pyrite flotation circuit to generate a low-sulphide tailings
- Further evaluation of blended composites including locked cycle testing
- Additional downstream settling, filtration, and paste testwork to support the feasibility level plant design

This section highlights the most significant and relevant findings from the various studies.



13.2 Historical Metallurgical Test Programs

Two initial studies were undertaken with respect to Mcllvenna Bay samples.

- “Scoping Level Metallurgical Assessment of the Mcllvenna Bay Project,” Foran Mining Corporation, KM3125 ALS Metallurgy – G&T Metallurgical Services, 28 May 2012 (KM3125).
- “Phase II Metallurgical Assessment Mcllvenna Bay Deposit,” BL0103, BML, 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 Mcllvenna 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 Mcllvenna Bay VMS Deposit”; Saskatchewan, Giovanni Di Prisco, January 2019.

13.2.1 Mineralogical Studies

The Mcllvenna 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, sporadically in abundant quantities. 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 (~0.03%) 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 have a P80 (80% passing size) of 60–65 µm for copper stringers, a P80 of 50 µm for massive sulphide primary grind and a P80 of 10–15 µm for concentrate regrinds.

13.2.2 Metallurgical Testing

In 2012, ALS Metallurgical in Kamloops BC conducted an initial scoping study (KM3125) to characterize the metallurgical performance of samples of massive sulphide zone (Zone 2 and Upper West Zone) and copper stockwork lithologies with a focus on ore hardness, mineralogical and chemical composition, and 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.

The average chemical composition of the three ore zone composites used for the KM3125 study is summarized in Table 13-1. Zinc Total assays were determined using a peroxide fusion method, whereas all other assays in the study were determined using standard aqua regia digestion.

Table 13-1: Chemical Compositions for the Ore Zone Composites

Element	Symbol	Units	CSZ	MS (Z2)	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 13-2 summarizes the Bond Rod / Ball Mill hardness test results for the three main ore composites from KM3125.



Table 13-2: Bond Rod and Ball Mill Grindability Test Results

Product	Bond Ball Mill Test		Bond Rod Mill Test	
	BBWI kWh/t	P ₈₀ , micron	BRWI kWh/t	P ₈₀ , micron
CSZ	16.1	80	17.0	882
MS (Z2)	11.6	83	12.7	869
UW-MS	14.0	81	15.6	852

Flotation work for KM3125 culminated in locked cycle testing of the three zone composites (see Table 13-3 for CSZ, Table 13-4 for MS (Z2) and Table 13-5 for UW-MS composites, respectively).

Table 13-3: CSZ Locked Cycle Test Results

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

Table 13-4: MS (Z2) Locked Cycle Test Results

Product	Mass%	Assay (% or g/t)						Distribution (%)					
		Cu	Pb	Zn	S	Ag	Au	Cu	Pb	Zn	S	Ag	Au
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



Table 13-5: UW Locked Cycle Test Results

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

In 2016, Base Met Laboratories in Kamloops, BC conducted what was described as a Phase II assessment (BL0103) and continued the metallurgical development program. Individual geological lithology composites were tested first, followed by two zone composites with a mix of copper stockwork and massive sulphide material (UWZ-Main, Z2-Main). Table 13-6 indicates the make-up of the composites used for the BL0103 study, whereas Table 13-7 summarizes the head assays.

Table 13-6: BL0103 Composite Assemblies

Composite	Sub-component	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



Table 13-7: Chemical Compositions for the Composite Zones

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.

Table 13-8 summarizes the ore Bond Ball Mill hardness test results for the BL0103 massive sulphide and copper stockwork composites.

Table 13-8: Bond Ball Mill Grindability Test Results

Sample D	BWI Parameter				
	Grind, mesh	F80, µm	P80, µm	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

A total of six locked cycle tests were conducted on the Z2-Main and UWZ-Main 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 sizings were 80% passing 24 µm 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.

Elemental ICP-MS scans for minor elements indicated fluorine and selenium concentrations in the KM3125 MS bulk concentrate of 506 ppm and 430 ppm, respectively—levels that could attract penalties in certain smelters. Mercury levels in the



KM3125 massive sulphide bulk and zinc concentrates varied from 80–155 g/t and these are also in a range that can attract penalties in certain smelters. Concentrates in BL0103 were generally lower with respect to mercury compared to the KM3125 results, but fluorine and selenium were again elevated in the copper concentrate for the BL0103 concentrates.

13.3 2019 Base Metallurgical Laboratories

The scope of work for the 2019 BL0351 testwork program “Feasibility Level Study of McIlvenna Bay,” BML Report #BL0351, 28 July 2019 (BL0351) was developed on the basis that ore from McIlvenna Bay would be 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 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 the following:

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
- 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 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 ZnSO₄ / NaCN prior to floating zinc into a final concentrate. A secondary flowsheet was subsequently developed using sulphur dioxide gas (SO₂) or sodium metabisulphite (SMBS) for depression of galena and zinc as a useful and effective alternative to cyanide use.

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 a similar metallurgical response.

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: 31 August and 4 December 2018. The first shipment was approximately 426 kg of half NQ core, with the second shipment consisting of approximately 460 kg 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 13-9 (ore type composites) and Table 13-10 (variability composites).

Table 13-9: 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 Rej. 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 Rej. 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 13-10: 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

13.3.1 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, and sized to fractions of +75, -75/+38 and -38 µm. Polished sections were 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 13-11.

The ores contained chalcopyrite, sphalerite, pyrite, pyrrhotite and galena 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 13-11: 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

13.3.2 Grindability

Bond ball mill work indices (BWI) determined for the three composites in BL0351 are summarized in Table 13-12. The composite hardness ranged from 13.8–14.4 kWh/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 13-12: 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 MS-rich blends are processed.

13.3.3 Flotation Testing

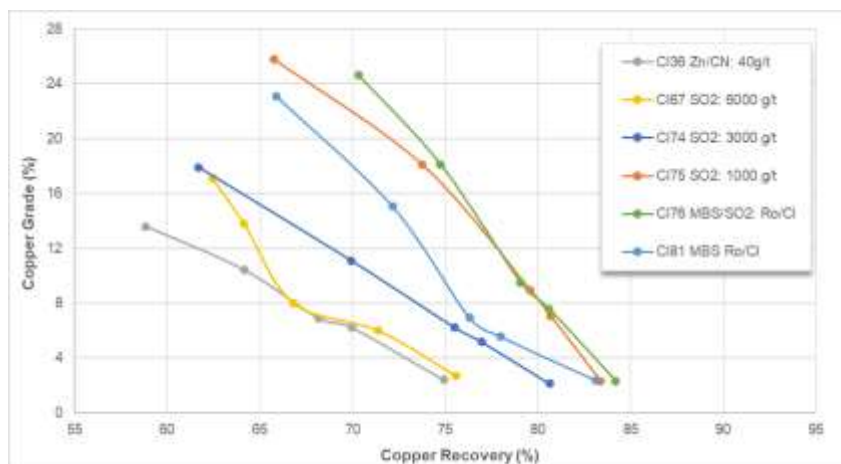
The BL0351 flotation program consisted of open circuit rougher and cleaner tests, with locked cycle tests used toward the end of the program to generate recovery predictions. Flowsheet development was carried out on composites of CSZ, Zone 2 Massive Sulphide, and UWZ Massive Sulphide. Variability tests were completed on a variety of high- and low-grade samples, and 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 five locked cycle tests.

Open cleaner testing of Z2 composite focused on optimization of cleaner collectors and dosages, depressant dosages, and regrind size targets. Cleaner testing of the UWZ composite focussed on the optimization of depressant dosages and finer zinc regrind targets, as well as the separation of lead from copper following the initial bulk copper-lead flotation step. Cleaner testing of CSZ attempted to optimize depressant dosages and the copper concentrate regrind size target.

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 that cyanide usage at site is not permitted. The selectivity of copper against pyrite, sphalerite, and particularly galena improved when using either chemical in the rougher and cleaner circuits (Figure 13-1).



Figure 13-1: Open Cleaner Test Results for Z2 Composite: Cyanide Alternative



Key findings of the cleaner flotation testwork on the three zone composites are summarized as follows.

- Z2 Composite: 13 cleaner circuit tests were conducted with open circuit copper recovery to concentrate achieving 70%, with stage recovery of zinc to the zinc concentrate of 87%.
- UWZ Composite: 7 cleaner circuit tests were conducted with open circuit copper recovery to concentrate achieving 84%, with stage recovery of zinc to the zinc concentrate of 79%.
- CSZ Composite: 4 cleaner circuit tests were conducted with open circuit copper recovery to concentrate achieving 92%.

13.3.4 Locked Cycle Testing

The results of the locked cycle tests are summarized in Table 13-13, Table 13-14, and Table 13-15. The locked cycle test flowsheet is illustrated in Figure 13-2 below. Note that the zinc circuit was not utilized for testing the CSZ composite due to low zinc head grades.



Figure 13-2: Locked Cycle Flowsheet

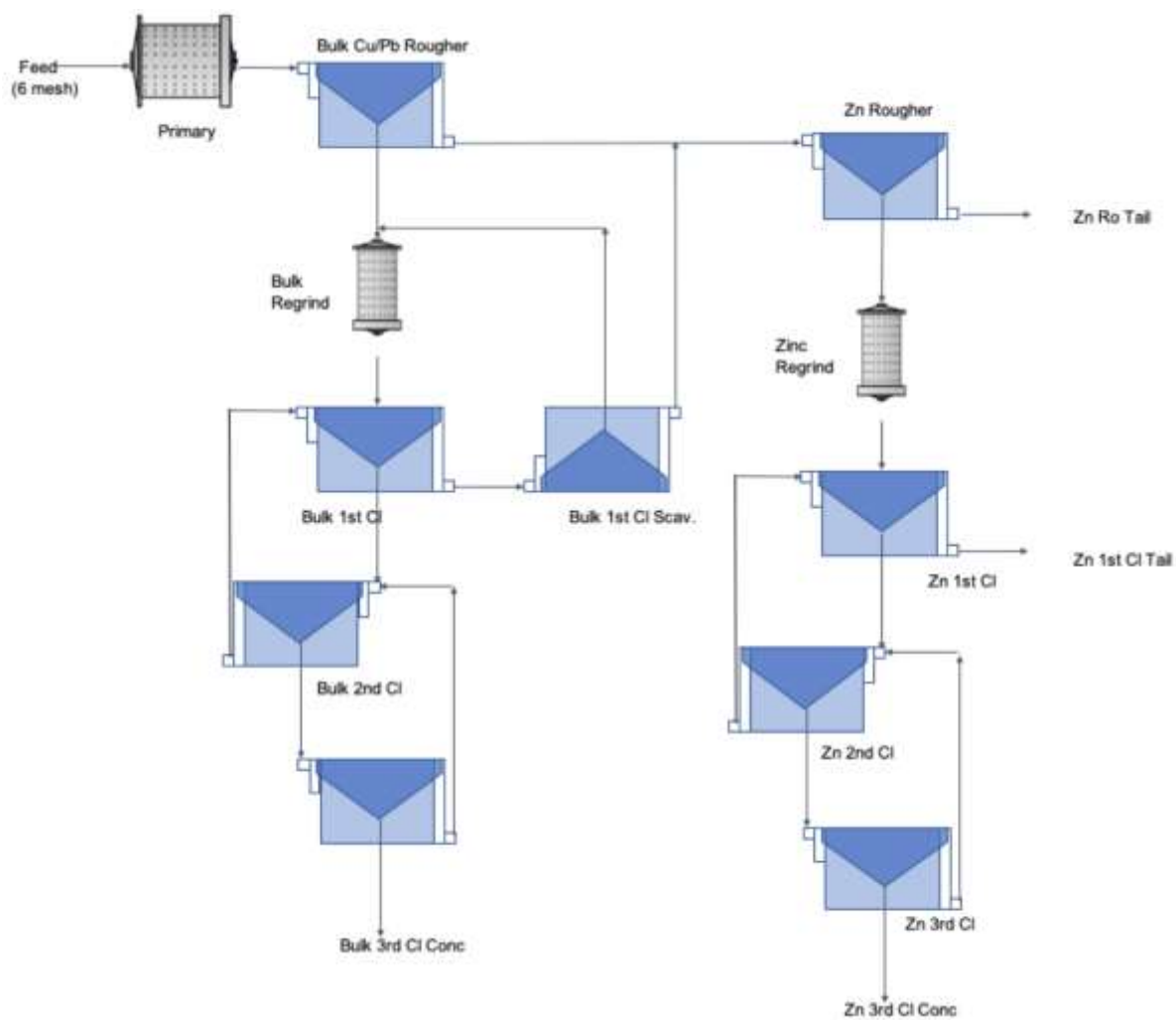


Table 13-13: Locked Cycle Test Results for UWZ Composite (LCT 44)

Product	Wt %	Assay (% or g/t)						Distribution (%)					
		Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
Cu/Pb CI 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 CI 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 CI 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 13-14: Locked Cycle Test Results for Z2 Composite (LCT80)

Product	Wt %	Assay (% or g/t)						Distribution (%)					
		Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
Cu/Pb CI 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 CI 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 CI 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 13-15: Locked Cycle Test Results for CSZ Composite (LCT42)

Product	Wt %	Assay (% or g/t)						Distribution (%)					
		Cu	Pb	Zn	Fe	Ag	Au	Cu	Pb	Zn	Fe	Ag	Au
Cu 2nd CI 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 CI 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

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 13-10, with results included in the estimates of concentrate grade and recovery presented in Section 13.5.



13.3.5 CSZ and Massive Sulphide Blend Testing

Tests were conducted to evaluate the metallurgical response of blending the massive sulphides with copper stockwork feed prior to grinding and flotation. The blend ratios for the three ore types are shown in Table 13-16.

Table 13-16: BL0351 Blend Ratios for Blend Testing

Blends	Blend Composition, %		
	UWZ	Z2	CSZ
Blend 1	15	30	55
Blend 2	30	15	55
Blend 3	30	45	25
Blend 4	10	15	75

In terms of copper recovery, the endpoints of the curves at final concentrate grade were all close with respect to actual versus calculated results for all four blends. Comparing copper-zinc selectivity, 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.

In terms of zinc recovery, actual versus calculated values at 50% concentrate grade were quite comparable for Blends 1, 2 and 3, and improved 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 will 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.

13.3.6 Concentrate Quality

Minor Elements

Minor element ICP-MS scans were completed on the final copper and zinc concentrates produced from the locked cycle tests. Table 13-17 summarizes the various assays.



Table 13-17: 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, ranging from 80 ppm to 140 ppm in the zinc concentrates, and from 20 ppm to 46 ppm in the massive sulphide copper concentrates. Overall, mercury in the copper concentrate will



be blended down significantly with CSZ copper concentrate which was low (~2 ppm) in mercury.

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

Magnesium and silica were both 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 is similar to that of talc, suggesting that talc recovery is mainly responsible for the concentration of these elements.

Concentrate Self Heating

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

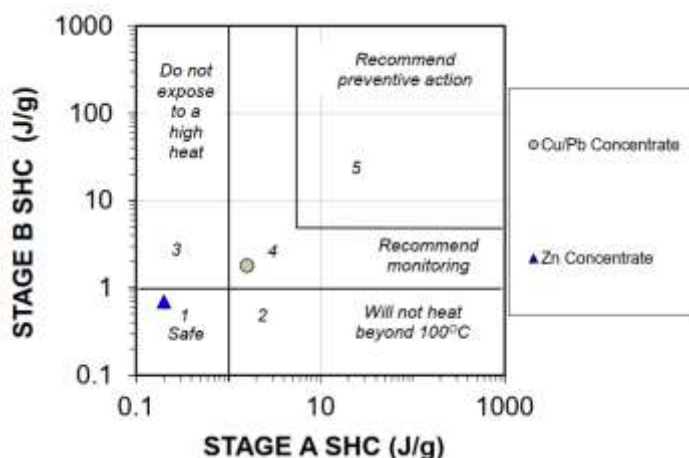
Composite assays are summarized in Table 13-18.

Table 13-18: MASH Test Head Assays

	Assay					
	% Cu	% Pb	% Zn	% Fe	g/t Ag	g/t Au
Cu 3 rd Cleaner Conc	26.8	2.80	3.00	27.7	186	10
Zn 3 rd 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 (Figure 13-3) 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.

Figure 13-3: Self Heating Test Results – Copper and Zinc Concentrates



13.4 2021 Testwork Program

In March 2021, a new testwork program was initiated at BML to provide additional data to support the feasibility-level process design (Metallurgical Study: McIlvenna Bay Project –BL0770, 01 April 2022). Samples selected for this work include the Z2, UWZ, and CSZ zone composites generated for the 2019 program, as well as new variability composites for hardness and flotation response characterisation. Key elements of the testwork program include the following.

- Additional hardness characterisation, including standardized Bond Ball Work Index, Bond Rod Work Index, SAG Mill Comminution, and Abrasion Index testing.
- Flotation optimization testwork focusing in three main areas:
 - Flotation collectors and dosages
 - Development of the SMBS depression scheme in the copper circuit
 - Pyrite flotation from the zinc circuit tailings to produce a low-sulphide tailings
- Additional variability testing for grindability and flotation
- Additional blended composite testwork to evaluate the effect of changes in mill feed composition on metallurgical response
- Third party / vendor testwork for settling and filtration for copper concentrate, zinc concentrate, non-sulphide tailings, and sulphide tailings; and paste backfill testwork on blended sulphide tailings

13.4.1 Sample Selection

The samples used for the 2021 testwork program consisted of the zone composites for CSZ, UWZ, and Z2 that were prepared in the BL0351 program. The test charges used had been freezer stored, and benchmark testing at the start of the program confirmed that the metallurgical response for the samples was consistent with the earlier work.

In addition to being tested individually, blended composites representing a range of potential mill feed conditions were also used. Blending was based on the ratios shown in Table 13-16 with an additional composite Blend 5 (40% CSZ, 30% UWZ, 30% Z2) generated to represent the lower CSZ component expected in the early years of production.

New variability composites were also generated for grindability and flotation testwork. Eight variability composites organized by zone and downhole depth were prepared and submitted for testwork. The head analysis of these samples is presented in Table 13-19.



Table 13-19: 2021 Variability Composite Head Assays

Comp	Zone	Downhole Depth [m]	Head Assays						
			Cu [%]	Zn [%]	Ag [g/t]	Au [g/t]	Fe [%]	Pb [%]	S [%]
CS-1	CSZ	-500	1.24	0.22	8.2	0.29	6.30	0.07	3.59
CS-2	CSZ	500-600	1.53	0.16	8.6	0.74	8.05	0.05	4.44
CS-3	CSZ	600-800	1.44	0.30	7.0	0.59	8.55	0.05	5.81
CS-4	CSZ	800+	1.25	0.84	13.0	0.74	11.7	0.05	10.2
UW-1	UWZ	-780	2.68	1.47	25.5	1.72	19.9	0.39	19.2
UW-2	UWZ	780+	2.47	3.50	22.2	1.55	25.1	0.13	27.9
Z2-1	Z2	-800	0.24	5.95	15.6	0.27	22.0	0.31	26.6
Z2-2	Z2	800+	0.41	7.90	22.0	0.30	21.6	0.45	25.7

In addition, another 16 core variability composites were generated from split core for SAG Mill Comminution (SMC) testing as summarized in Table 13-20.

Table 13-20: 2021 Variability Composites for SMC Testing

Comp	Zone	Hole Number	Core Size	From [m]	To [m]
SMC-1	CSZ	MB-18-207-W1	1/2 NQ	899.0	911.3
SMC-2	CSZ	MB-18-203-W1	1/2 NQ	798.0	806.5
SMC-3	CSZ	MB-18-207/8	1/4 NQ	785.6	916.0
SMC-4	Z2	MB-18-207-W1	1/2 NQ	887.8	892.9
SMC-5	Z2	MB-11-156	1/2 HQ	493.0	497.9
SMC-6	Z2	MB-21-220w1	1/4 NQ	922.7	928.1
SMC-7	Z2	MB-21-225d1	1/4 NQ	1076.6	1082.6
SMC-8	UWZ	MB-11-157	1/2 HQ	602.6	610.5
SMC-9	UWZ	MB-21-216w1	1/4 NQ	815.3	818.8
SMC-10	UWZ	MB-21-246d1	1/4 NQ	851.0	854.7
SMC-11	UWZ	MB-21-234	1/4 NQ	643.6	648.5
SMC-12	CSZ	MB-12-165	1/2 HQ	267.3	272.3
SMC-13	CSZ	MB-12-171	1/2 HQ	115.1	124.6
SMC-14	CSZ	MB-12-177	1/4 NQ	289.4	296.4
SMC-15	CSZ	MB-12-178	1/4 NQ	127.9	145.4
SMC-16	CSZ	MB-21-225d1	1/4 NQ	1082.6	1089.4



13.4.2 Grindability

Grindability testing was carried out at BML and consisted of SMC, BRWI, BBWI, and Ai measurements on variability composite samples. Table 13-21 presents a summary of the results of these tests. Stockwork zone samples are, on average, harder than those from the massive sulphide zone. Results are comparable to previous testwork with average BRWI and BBWI values of 15.8 kWh/t and 15.0 kWh/t, respectively.

Table 13-21: Summary of Grindability Results

Sample ID	Zone	RWI parameters				BWV parameters				Bond	
		Mesh of Grind	F80 µm	P80 µm	BRWI kWh/t	Mesh of Grind	F80 µm	P80 µm	BBWI kWh/t	Ai (g)	
CS-1	CSZ	14	9546	898	17.7	150	2221	78	16.6	0.267	
CS-2	CSZ	14	9491	912	17.7	150	2204	78	16.9	0.386	
CS-3	CSZ	14	10017	909	16.8	150	2167	78	15.8	-	
CS-4	CSZ	14	9370	891	15.8	150	2152	79	15.8	-	
UW-1	UWZ	14	9465	889	16.4	150	2067	78	15.5	-	
UW-2	UWZ	14	9800	870	14.7	150	1943	80	13.7	-	
Z2-1	Z2	14	9864	867	13.6	150	2155	80	12.7	0.139	
Z2-2	Z2	14	9657	873	13.7	150	2061	78	12.9	-	

To support a grinding circuit design study and sizing of a SAG mill for the project, a total of 16 additional composites were prepared, comprised of 8 composites from the CSZ zone and four each from the Z2 and UWZ zones. Table 13-22 provides a summary of these results. CSZ samples were found to be more competent, with Axb numbers as low as 26.4 (average value of 31.2). In comparison the Axb values for the Z2 and UWZ zones averaged 42.4 and 39.9, respectively.

Table 13-22: Summary of SMC Test Results

Sample ID	Zone	Relative Density	SMC Parameters				DWI (kWh/m ³)
			A	b	A x b	t _a	
SMC-1	CSZ	2.86	95.3	0.34	32.4	0.29	8.76
SMC-2	CSZ	2.96	91.4	0.35	32.0	0.28	9.34
SMC-3	CSZ	3.20	80.5	0.45	36.2	0.29	8.75
SMC-4	Z2	3.75	85.9	0.47	40.4	0.28	9.29
SMC-5	Z2	4.16	82.8	0.53	43.9	0.27	9.52
SMC-6	Z2	3.73	77.3	0.61	47.2	0.33	7.94
SMC-7	Z2	3.59	69.5	0.55	38.2	0.28	9.43
SMC-8	UWZ	2.96	62.3	0.78	48.6	0.43	6.08
SMC-9	UWZ	3.56	73.6	0.53	39.0	0.28	9.15



Sample ID	Zone	Relative Density	SMC Parameters				DWI (kWh/m ³)
			A	b	A x b	t _a	
SMC-10	UWZ	3.40	95.0	0.33	31.4	0.24	10.8
SMC-11	UWZ	3.52	67.8	0.60	40.7	0.30	8.59
SMC-12	CSZ	2.80	67.0	0.55	36.9	0.34	7.58
SMC-13	CSZ	2.84	87.9	0.30	26.4	0.24	10.6
SMC-14	CSZ	2.58	87.5	0.30	26.3	0.26	9.92
SMC-15	CSZ	2.94	71.3	0.38	27.1	0.24	10.9
SMC-16	CSZ	3.18	99.3	0.33	32.8	0.27	9.66

13.4.3 Flotation

Bench-scale batch flotation testwork was carried out to optimize and confirm the flowsheet. The primary focus of the initial testing was the development of the SMBS (sodium metabisulfite) reagent scheme that had demonstrated good potential to replace NaCN/ZnSO₄ in the earlier testwork program. As a result of the low zinc grade of the CSZ composite, the tests for the development of the SMBS flowsheet were conducted primarily on the Z2 and UWZ samples.

Figure 13-4 presents the results of Cu/Zn separation for the rougher flotation tests, with the reagent additions for these tests summarized in Table 13-16. The results indicate good separation between copper and zinc under a range of conditions. Lowering the SMBS addition to 1500 g/t in the roughers was not found to negatively affect the separation for either composite. In addition, replacement of Aerophine 3418A with a combination of AERO 3894/3501 similarly did not cause an adverse response.

Figure 13-4: Cu/Zn Separation in the Rougher Flotation Tests

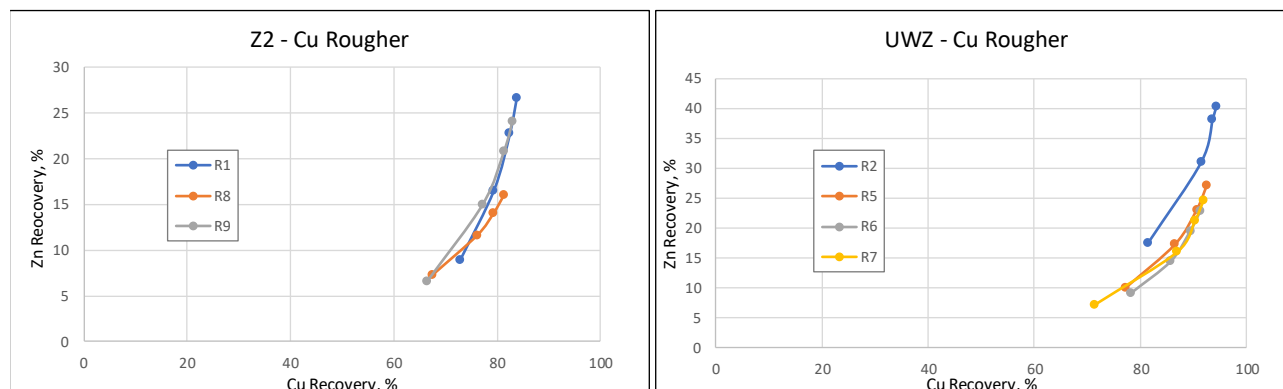


Table 13-23: Copper Circuit Reagent Additions for the Rougher Flotation Tests

Test #	Comp	Reagent Additions, g/t						
		Lime	SMBS	3418A	3894	3501	W31	MIBC
R1	Z2	350	3000	42	--	--	--	35
R2	UWZ	350	3000	--	35	15	70	--
R5	UWZ	350	2000	--	35	15	--	21
R6	UWZ	350	3000	--	22.5	10	--	28
R7	UWZ	350	1500	--	22.5	10	--	14
R8	Z2	350	1500	--	22.5	10	--	14
R9	Z2	350	1500	--	22.5	20	--	14

Zinc rougher flotation reagent additions and test results are summarized in Table 13-24 and Figure 13-5, respectively. Good zinc stage recoveries are observed for both composites with moderate amounts of collector AERO 5100 added.

Figure 13-5: Zinc Grade Recovery Curves for the Z2 and UWZ Composites

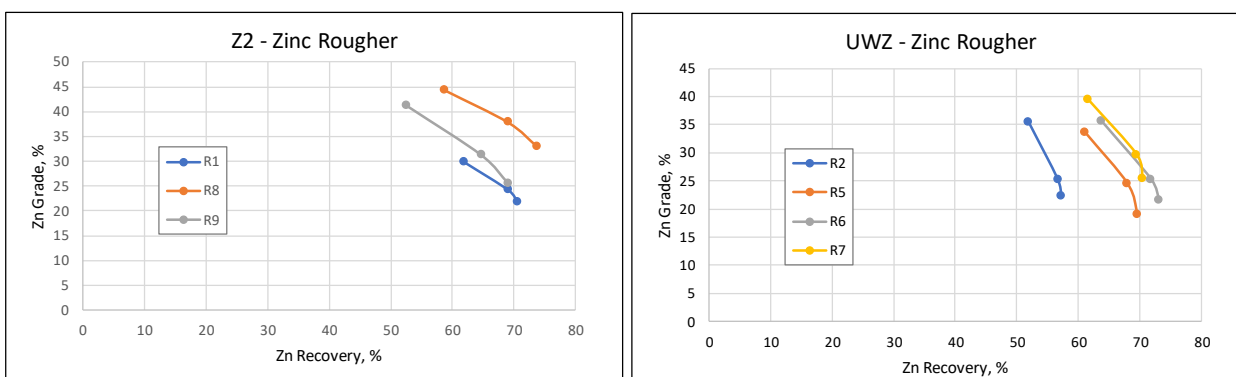


Table 13-24: Zinc Circuit Reagent Additions for the Rougher Flotation Tests

Test #	Comp	Reagent Additions, g/t					
		Lime	CuSO ₄	3418A	5100	W31	MIBC
R1	Z2	1635	500	30	--	40	
R2	UWZ	1585	300	--	7.5	60	--
R5	UWZ	1300	300	--	7.5	--	--
R6	UWZ	1400	300	--	7.5	--	--
R7	UWZ	1200	300	--	7.5	--	7
R8	Z2	1200	300	--	7.5	--	7
R9	Z2	1200	300	--	10	--	7

Additional cleaner flotation tests were conducted to optimize the sequential copper / zinc flowsheet using the new reagent scheme. A total of four tests were conducted on the Z2 composite, while five tests were conducted on the UWZ composite. A single test of the copper cleaner flowsheet was carried out on the CSZ composite to confirm the response in this domain. Key confirmation from the batch cleaner testwork included the following.

- Regrind size P₈₀'s of 20–25 µm in the copper and zinc circuits
- Copper cleaner circuit reagent additions of AERO 3894 and AERO 3501
- Zinc cleaner circuit reagent addition of AERO 5100
- Frother and depressant additions

Blend Testing

The flowsheet developed for the zone composites was evaluated using various blends intended to replicate different mill feed scenarios. For this work, three CS:MS (where MS = Z2 + UWZ) ratios were selected. Two of these blends were based on those from the previous program (summarized in Table 13-16) and representing ratios of 45:55 (Blend 1) and 75:25 (Blend 4). A third blend (Blend 5) was generated to represent a 40:60 ratio that is consistent with some periods during the earlier years of production. Table 13-25 presents the composition of the blend composites.

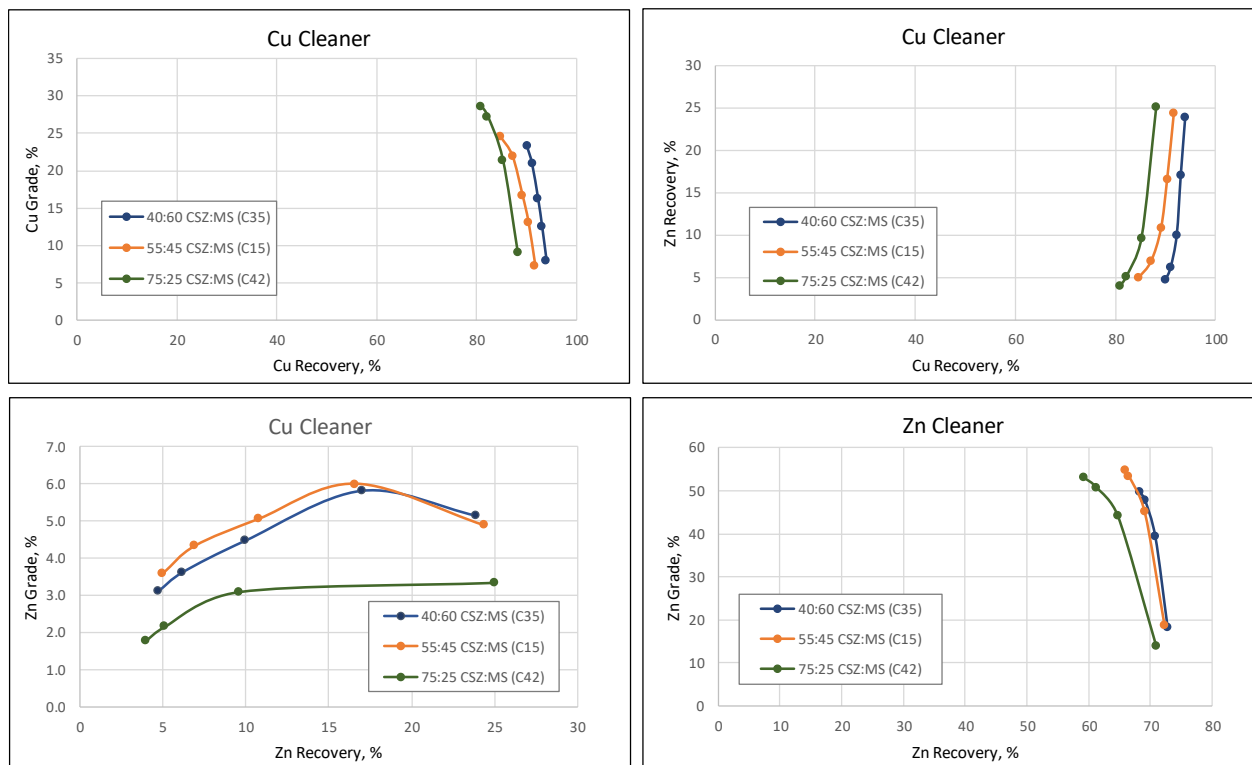
Table 13-25: Composite Blends Used for Flowsheet Confirmation

Blends	Blend Composition, %			Ratio
	CSZ	Z2	UWZ	CS:MS
Blend 1	55	30	15	55:45
Blend 4	75	15	10	75:25
Blend 5	30	30	40	60:40



Batch cleaner flotation tests were carried out on each of the three blend composites. Figure 13-6 presents the results of selected tests from each blend in both the copper and zinc circuits. In the copper circuit, good grades and open circuit recoveries were achieved in all samples, with slightly better selectivity between copper and zinc observed at higher MS ratios.

Figure 13-6: Blend Composite Cleaner Flotation Tests



Zinc circuit stage recovery was good for all composites, with improved performance at higher zinc head grades (40:60 and 55:45).

Variability Composites

Batch open circuit cleaner flotation testing was carried out on each of the eight variability met composites. A single flotation test was conducted on each composite using the reagent conditions from the development program. Flotation results for the copper stockwork and massive sulphide composites are presented in Figure 13-7 and Figure 13-8, respectively.



Figure 13-7: CSZ Variability Composite Cleaner Flotation Tests

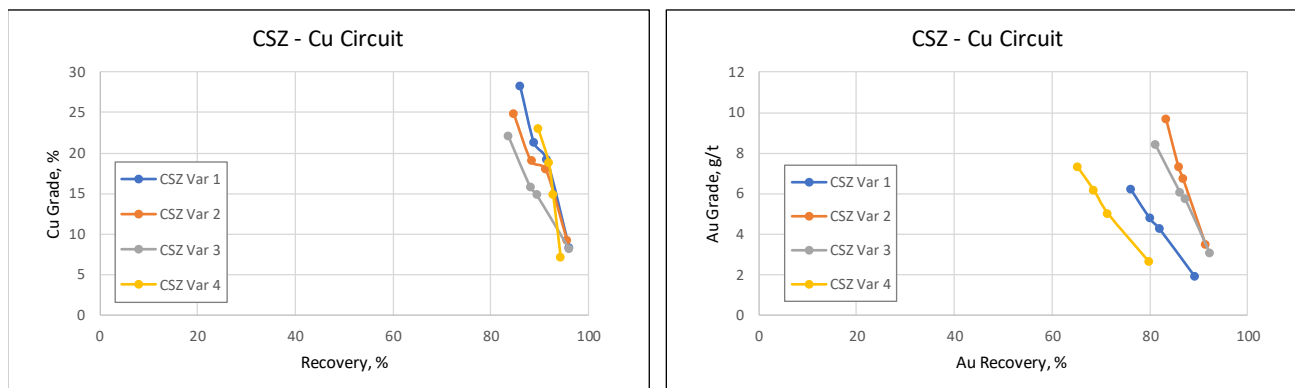
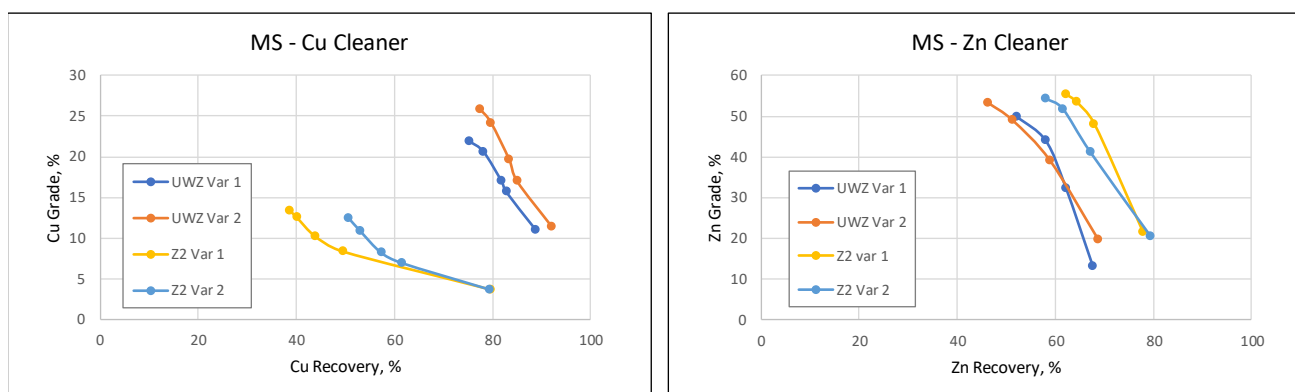


Figure 13-8: Z2 and UWZ variability composite cleaner flotation tests.



The variability flotation tests indicated comparable metallurgical performance to the CSZ, Z2, and UWZ zone composites. Lower copper recoveries observed with the Z2 composites are the result of high losses in the first cleaner stage that can be at least partially mitigated through adjustments to the depressant and collector additions.

Pyrite Flotation

Selected batch flotation tests included an add-on pyrite recovery circuit on the zinc rougher tailings to evaluate the production of a de-sulphurised tailings stream for surface disposal. The approach for this step was the addition of PAX (potassium amyl xanthate) collector to recover the remaining sulphides in the tailings stream. A summary of the test results is presented in Table 13-26.



Table 13-26: Pyrite Circuit Flotation Test Results.

Test #	Comp	PAX Addition [g/t]	PH	Float Time [min]	Sulphur Grade			S Stage Recovery [%]	NPR [NP:AP]
					Comp [%]	Py feed [%]	Tails [%]		
R5	UWZ	65	10.8	10	24.5	21	1.21	96.7	--
R6	UWZ	65	10.8	10	23.9	22.4	1.35	96.7	--
R7	UWZ	65	10.8	10	24.3	21.6	1.47	96.3	--
R8	Z2	65	10.8	10	25.2	23.8	1.81	96.0	--
R9	Z2	90	10.3	15	24.8	22.7	0.91	98.0	--
C10	UWZ	65	10.6	10	23.8	21.5	1.24	96.8	--
C11	UWZ	100	10.7	10	23.0	20.1	0.75	97.9	5.93
C12	CSZ	65	7.7	10	4.55	1.72	0.22	87.9	1.15
C13	Blend-1	100	10.9	10	13.5	9.18	0.44	96.1	3.41
C14	UWZ	100	7.5	10	24.3	21.5	1.19	96.8	6.72
C15	Blend-1	35	10.5	10	14.2	9.02	0.60	94.6	4.85
C16	Blend-1	35	10.9	10	14.2	8.93	0.73	92.8	--
C18	Z2	35	10.8	10	24.9	22.8	1.58	96.3	4.97

The initial dose of 65 g/t PAX was lowered to 35 g/t without a significant effect on sulphide flotation. Furthermore, lowering the pH of the zinc tailings with sulphuric acid was not found to improve the kinetics of pyrite flotation. Stage recovery of sulphur ranged from 88% for the CSZ zone to 98% in the massive sulphide areas.

For selected tests, a sample of the desulphurized tailings were submitted for standardized Acid-Base Accounting (ABA) analysis. The test compares the acid generating potential of the sulphides with the neutralizing potential of contained carbonate. The NPR is defined as the ratio of neutralizing potential to acid generating potential. NPRs for the samples submitted ranged from 1.15 for the CSZ composite to 6.72 for the UWZ composite. The Blend 1 composite (55% CSZ / 45% MS) was also tested, and at a tailings sulphur grade of 0.44% and 0.60% were found yield tailings that are expected to be non-acid generating (NPR >3).

13.4.4 Locked Cycle Testing

Locked cycle flotation tests were carried out on the blend composites. Each test consisted of five 2 kg batch flotation tests with the intermediate products of one cycle feeding to the corresponding stage of the next cycle, according to the flowsheet presented in Figure 13-2. Metallurgical projections based on the cycle test results are presented in Table 13-27.

In the first test, LCT-32, the Blend 1 composite was used, with test conditions based on the optimized conditions from an earlier batch test. Poor test stability was observed



however, as a large circulating load was observed to build up in the copper circuit. This test was repeated in LCT-34 with changes to the test conditions including adjustments to the collector addition and stainless steel grinding media. The changes resulted in an improvement in copper recovery as well as test stability.

Table 13-27: Summary of the Locked-Cycle Test Results on Blend Composites

LCT-32 Blend-1 Metallurgical Projection - Cycles

D - E

Product	Mass	Grade					% Distribution				
	%	Cu, %	Zn, %	Ag, g/t	Au, g/t	S, %	Cu	Zn	Ag	Au	S
Cu Cl Conc	3.2	26.1	2.82	174	10.0	34.1	81.6	3.4	44.1	71.9	8.1
Zn Cl Conc	4.9	2.05	45.8	54	0.92	32.2	10.0	86.5	21.4	10.3	11.9
Zn 1st Cl Tail	6.9	0.25	1.84	14	0.34	26.6	1.7	4.9	7.7	5.3	13.7
Zn Ro Tail	85.0	0.08	0.16	4.0	0.07	10.4	6.7	5.2	26.8	12.5	66.2
Head (calculated)	100	1.00	2.59	12	0.44	13.2	100	100	100	100	100

LCT-34 Blend-1 Metallurgical Projection - Cycles

D - E

Product	Mass	Grade					% Distribution				
	%	Cu, %	Zn, %	Ag, g/t	Au, g/t	S, %	Cu	Zn	Ag	Au	S
Cu Cl Conc	4.1	24.9	3.57	159	8.6	32.1	86.4	4.9	52.6	75.3	9.1
Zn Cl Conc	5.6	1.24	46.1	52	0.62	32.2	5.9	86.8	23.7	7.4	12.5
Zn 1st Cl Tail	7.5	0.22	1.15	10	0.30	24.2	1.4	2.9	6.0	4.9	12.7
Zn Ro Tail	82.8	0.09	0.19	2.7	0.07	11.4	6.3	5.3	17.7	12.4	65.7
Head (calculated)	100	1.16	2.92	12	0.46	14.2	100	100	100	100	100

LCT-37 Blend-5 Metallurgical Projection - Cycles

D - E

Product	Mass	Grade					% Distribution				
	%	Cu, %	Zn, %	Ag, g/t	Au, g/t	S, %	Cu	Zn	Ag	Au	S
Cu Cl Conc	6.2	17.9	4.75	142	8.3	29.0	90.8	9.9	62.3	75.4	11.1
Zn Cl Conc	4.6	0.83	50.8	44	0.92	33.4	3.1	78.4	14.4	6.2	9.4
Zn 1st Cl Tail	7.1	0.21	3.16	8	0.28	28.7	1.2	7.5	4.0	2.9	12.5
Zn Ro Tail	82.1	0.07	0.15	3.4	0.13	13.3	4.9	4.3	19.3	15.5	67.0
Head (calculated)	100	1.21	2.94	14	0.68	16.0	100	100	100	100	100

LCT-43 Blend-4 Metallurgical Projection - Cycles

C - D

Product	Mass	Grade					% Distribution				



	%	Cu, %	Zn, %	Ag, g/t	Au, g/t	S, %	Cu	Zn	Ag	Au	S
Cu Cl Conc	4.7	24.3	4.91	139	6.9	31.8	89.4	17.7	64.6	70.5	16.6
Zn Cl Conc	2.1	1.82	44.1	51	0.95	33.9	3.0	71.1	10.7	4.3	7.9
Zn 1st Cl Tail	10.6	0.28	0.71	8.7	0.62	39.2	2.4	5.9	9.2	14.5	46.8
Zn Ro Tail	82.6	0.08	0.08	1.9	0.06	3.10	5.2	5.4	15.5	10.7	28.7
Head (calculated)	100	1.27	1.30	10	0.46	8.92	100	100	100	100	100

Overall, final copper concentrate copper grades were lower than expected during locked cycle testing. Further refinements to the reagent additions, along with bulk sample bench scale, and potentially column flotation, are recommended to optimize concentrate grade.

Minor Elements

Final concentrate products from the last three locked cycle tests were submitted for detailed analysis to evaluate potential deleterious elements. Table 13-28 presents a summary of these analyses.

Table 13-28: Summary of Minor Element Analysis for the Copper and Zinc Concentrates

Element	Units	LCT 34 Blend 1		LCT 37 Blend 5		LCT 43 Blend 4	
		Cu Conc	Zn Conc	Cu Conc	Zn Conc	Cu Conc	Zn Conc
Cl	%	0.03	0.07	0.01	0.03	--	--
F	%	0.07	0.01	0.13	< 0.01	--	--
Al	%	0.28	0.27	0.57	0.17	0.25	0.2
As	ppm	59	135	119	110	56	105
B	ppm	< 10	< 10	< 10	< 10	< 10	< 10
Ba	ppm	9	10	9	7	< 10	< 10
Be	ppm	< 3	< 3	< 3	< 3	0.9	0.5
Bi	ppm	113	74	180	53	142	61
Ca	%	0.18	0.8	0.36	0.51	0.11	0.53
Cd	ppm	95	1340	143	1510	112	1271
Co	ppm	16.6	18.4	37.9	15.2	19	16
Cr	ppm	160	450	630	650	217	163
Fe	%	26.7	11.2	25	9.06	27.15	10.94
Ga	ppm	4.5	4.2	5.9	4.1	14	6
Hg	ppm	9.63	96.9	16	73	8	85
K	%	< 0.1	< 0.1	< 0.1	< 0.1	< 0.01	0.03
La	ppm	6.3	7.5	11.1	4.9	16	7
Mg	%	1.44	0.74	2.67	0.42	0.23	0.23



Element	Units	LCT 34 Blend 1		LCT 37 Blend 5		LCT 43 Blend 4	
		Cu Conc	Zn Conc	Cu Conc	Zn Conc	Cu Conc	Zn Conc
Mn	ppm	169	673	323	738	76	546
Mo	ppm	28	53	65	28	17	16
Na	%	--	--	--	--	0.002	0.002
Ni	ppm	100	310	390	230	111	97
P	%	--	--	--	--	0.003	0.007
Pb	%	0.83	1.36	1.13	0.84	0.80	0.98
Sb	ppm	58	39	93	30	45	24
Se	ppm	309	153	360	121	--	--
Si	%	2.97	1.36	5.38	0.6	3.94	0.03
Sn	ppm	467	150	373	108	--	--
Sr	ppm	18	19	17	18	2	3
Te	ppm	13	14	10	25	14	4
Th	ppm	0.4	0.6	0.9	0.4	< 5	< 5
Ti	%	< 0.01	< 0.01	0.01	< 0.01	0.001	0.001
Tl	ppm	1.4	1.1	1.8	1.1	< 2	< 2
U	ppm	0.8	0.9	1.3	0.9	< 10	< 10
V	ppm	< 5	5	8	7	5	3
W	ppm	2.7	4.8	6.3	4.8	2	1
Y	ppm	3.5	3.3	5.4	3.2	5	3

Results are consistent with those presented from earlier testwork (Table 13-17) and indicate elevated levels for mercury and cadmium in the zinc concentrates and selenium in the copper concentrates.

13.4.5 Settling & Filtration

Settling and filtration testing was carried out on test product samples from a bulk 150 kg flotation test at BML. Tailings rheology, settling, and filtration tests were conducted by FLSmidth, whereas settling and filtration testing on the concentrates were carried out by BML and Metso:Outotec.

Representative samples of pyrite tailings and desulphurized tailings were sent to FLSmidth's Solids-Liquid Separations Laboratory in Midvale, Utah. Scoping testwork indicated that good settling and overflow clarity could be achieved for both samples using the anionic polyacrylamide flocculant MF10. Dynamic settling tests with flocculant dosages of 17–20 g/t at feedwell densities ranging from 9–12% were found to yield underflow densities of greater than 60%, with overflow suspended solids of less than 100 ppm.



Filtration testwork was conducted at FLS on thickened samples of desulphurized tailings for surface disposal as well as blended pyrite / desulphurized tailings representing paste feed. Target moisture content for surface disposal was 18%, whereas optimal moisture content achieved during vacuum filtration testing was 14.5%. Filtration rate was found to vary with final moisture content from 384 kg/m²/hr at 14.5% moisture, to 1056 kg/m²/hr at 18% moisture. Similar rates were achieved for the blended tails (40% pyrite tailings / 60% desulphurized tailings). At a higher moisture content of 21%, typical of paste plant feed, the blended tailings filtration rate was measured at 1408 kg/m²/hr.

Testwork on dynamic settling of copper and zinc concentrate samples was undertaken at BML. Feed to the testwork consisted of final product copper and zinc concentrates generated during the bulk the flotation test with measured P₈₀'s of 26 µm and 29 µm, respectively. Good settling rates were achieved using MF10 flocculant at dosages of 20–30 g/t-concentrate. Measured suspended solids in the overflow ranged from 10 ppm to 36 ppm for the copper concentrate and from 14 ppm to 25 ppm for the zinc concentrate at loading rates varying from 0.5 to 1.0 t/m²/hr. Underflow densities reached as high as 59.8% for the copper and 64.6% for zinc.

Pressure filtration testwork was carried out on thickened copper and zinc concentrate samples by Metso:Outotec at the SGS Minerals Services lab in Lakefield, Ontario. Final concentrate moisture contents for pressure filtration were found to be 9.5% to 10.7% for the copper concentrate and 10.3% to 11.0% for the zinc concentrate. At the lowest moisture contents the filtration rates for copper and zinc concentrates were measured at 492 kg/m²/hr and 483 kg/m²/hr, respectively.

13.4.6 Paste Backfill

Samples of the pyrite tailings, desulphurized tailings, and process water from the 150 kg bulk flotation test were used by Minefill Services for head characterisation, size analysis, rheology, and paste mixing testwork to permit design of the paste plant. The program tested three different blends of pyrite / desulfurized tailings to determine paste recipes for high strength and low strength paste for underground storage.

To improve the strength of the paste, a binder was added consisting of 65% slag and 35% Portland Cement. A matrix of tests was conducted to evaluate the effect of binder addition from 2% to 8% on the change in unconfined compressive strength (UCS) over periods of 3, 7, 28, 56, and 112 days. The results were used to provide the design basis for the paste plant.

13.5 Recovery and Concentrate Grade Projections

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



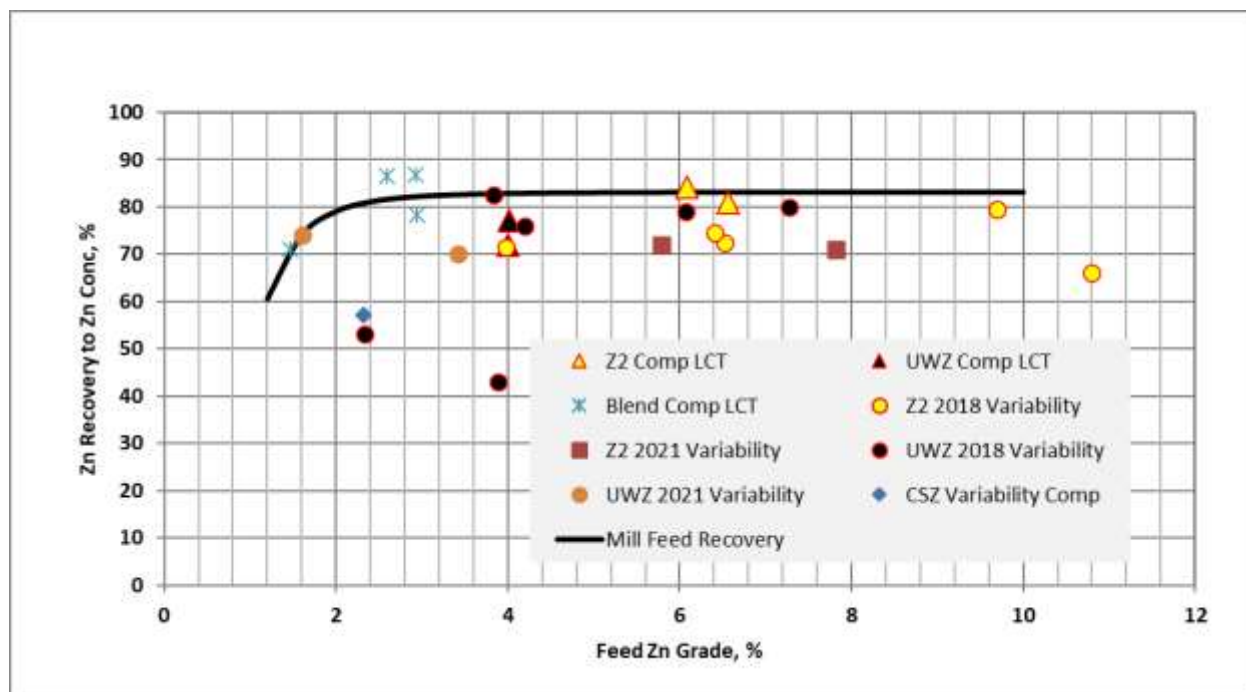
For open circuit cleaner tests, estimates were made for potential closed-circuit recovery at a constant concentrate grade. The predictive values from the testwork were then used to derive mathematical functions as described in this section. The following charts summarize the various performance projections. Overall, life-of-mine metal recovery estimates for the individual ore zones and the blended feed are presented in Table 13-29.

Table 13-29: Estimated Metal Recovery by Ore Type.

LOM Recovery Rates, %	Copper	Zinc	Gold	Silver
Massive Sulphide	82.4	84.4	82.9	54.6
Copper Stockwork	95.3	47.0	92.7	76.6
Blended Mill Feed	91.1	79.8	88.6	63.2

Figure 13-9 illustrates the relationship between zinc recovery to a zinc concentrate and the head grade of the blended feed. The model curve is used to estimate zinc recovery to final zinc concentrate based on the blended mill feed head grade. The Z2 and UWZ zones contain the bulk of the zinc units contained in the deposit, and testwork on composite samples from these zones provides the primary basis for zinc recovery. Blended composites of stockwork and massive sulphide have been used to confirm the response at lower zinc head grades.

Figure 13-9: Zinc Recovery as a Function of Mill Feed Zinc Head Grade

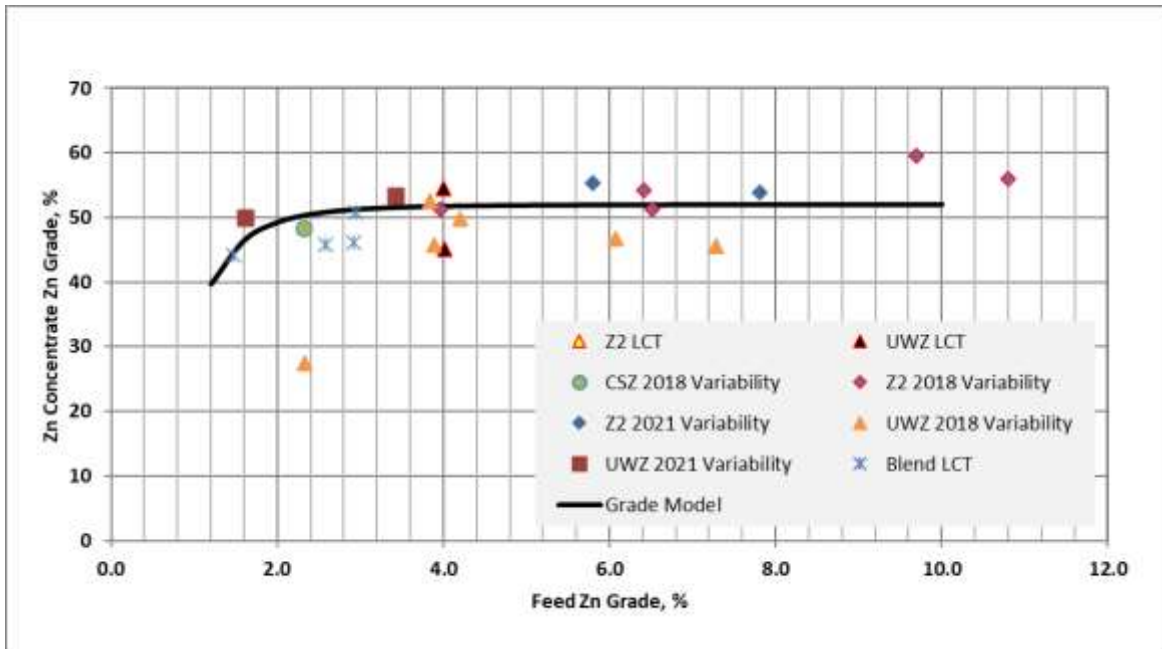


The zinc recovery vs head grade relationship is defined as:

$$\text{Zn Recovery}(\%) = \frac{83.2 \times F^{3.9} + 2.1}{F^{3.9} + 0.8}$$

Where F is the zinc grade (%) in mill feed. Figure 13-10 describes the zinc concentrate grade as a function of the % zinc in mill feed.

Figure 13-10: Zinc Concentrate Grade as a Function of Zinc Head Grade



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

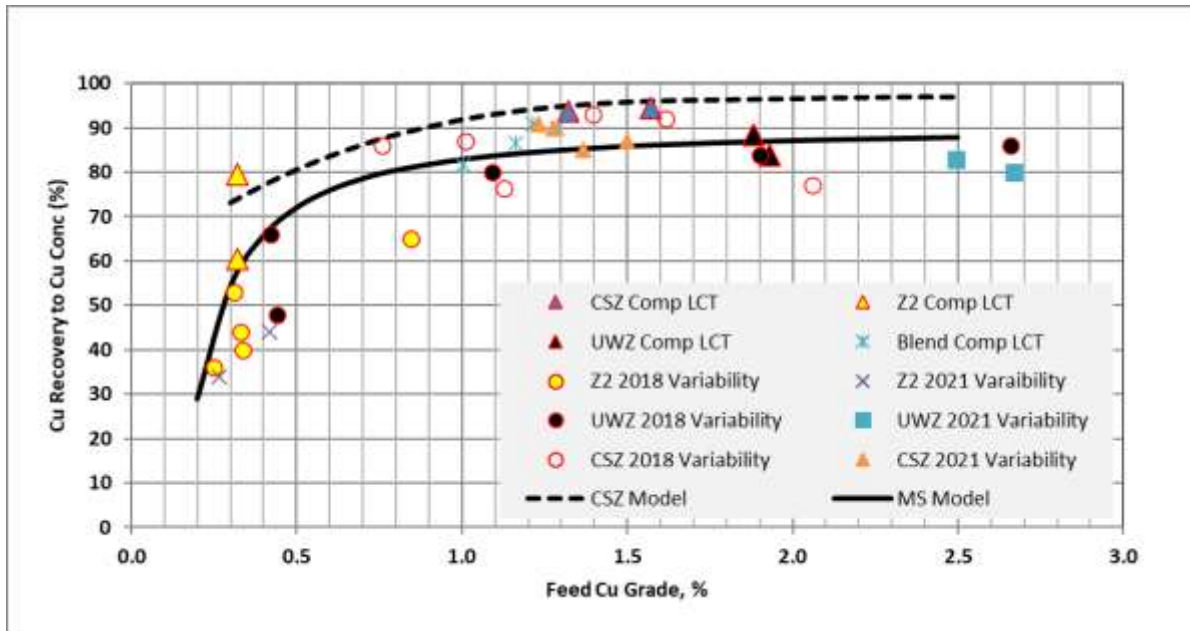
$$\text{Zn Concentrate Grade}(\%) = \frac{52.0 \times F^{3.4} + 2.9}{F^{3.4} + 0.65}$$

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

Figure 13-11 presents the models for copper recovery versus head grade developed for the CSZ and MS zones. Solid curves indicate the function for the MS and the dashed curve indicates the function for the CSZ.



Figure 13-11: Copper Recoveries for Massive Sulphide and Copper Stockwork Zones as a Function of Copper Head Grade



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

$$CSZ \text{ Cu Recovery}(\%) = (5.3 \times F^3) - (30.5 \times F^2) + (59.0 \times F) + 58.1$$

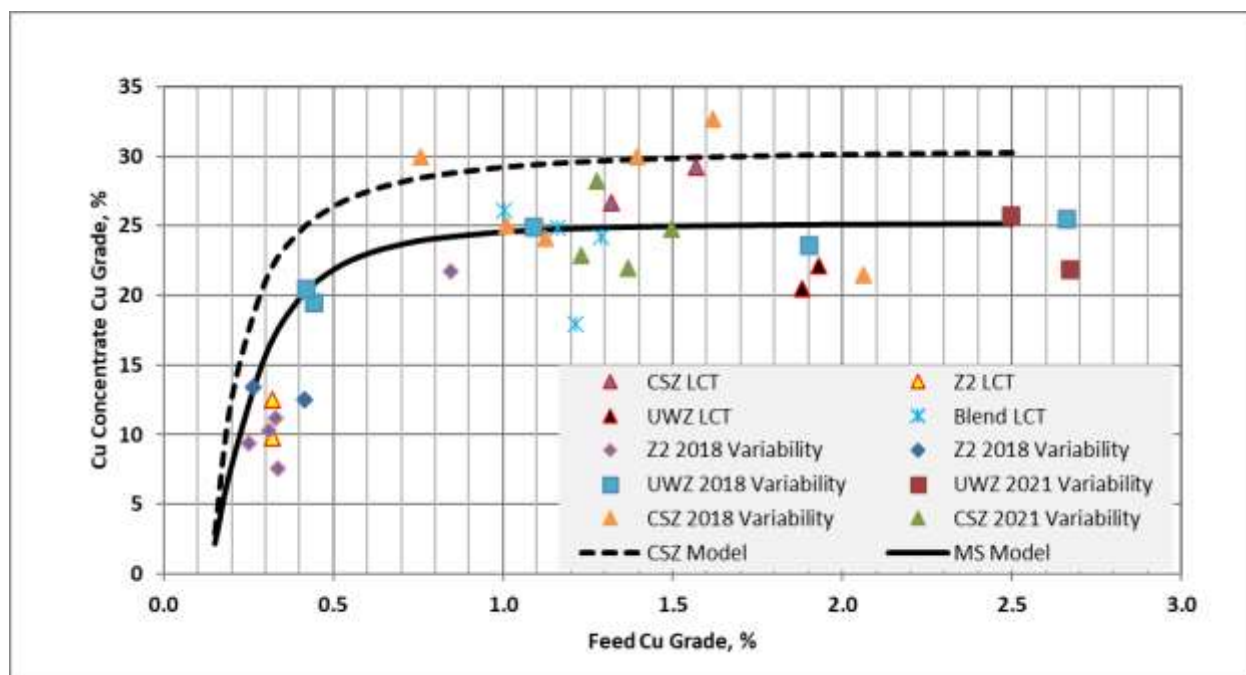
and for the massive sulphide:

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

Where F is the Cu grade (%) of CSZ or MS component of the mill feed.

Figure 13-12 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 13-12: Copper Concentrate Grade as a Function of Head Grade for CSZ and MS Zones



The copper concentrate grade vs head grade relationships are defined as:

$$CSZ \text{ Cu Concentrate Grade } (\%) = \frac{30.6 \times F^{1.65} - 1.18}{F^{1.65} + 0.005}$$

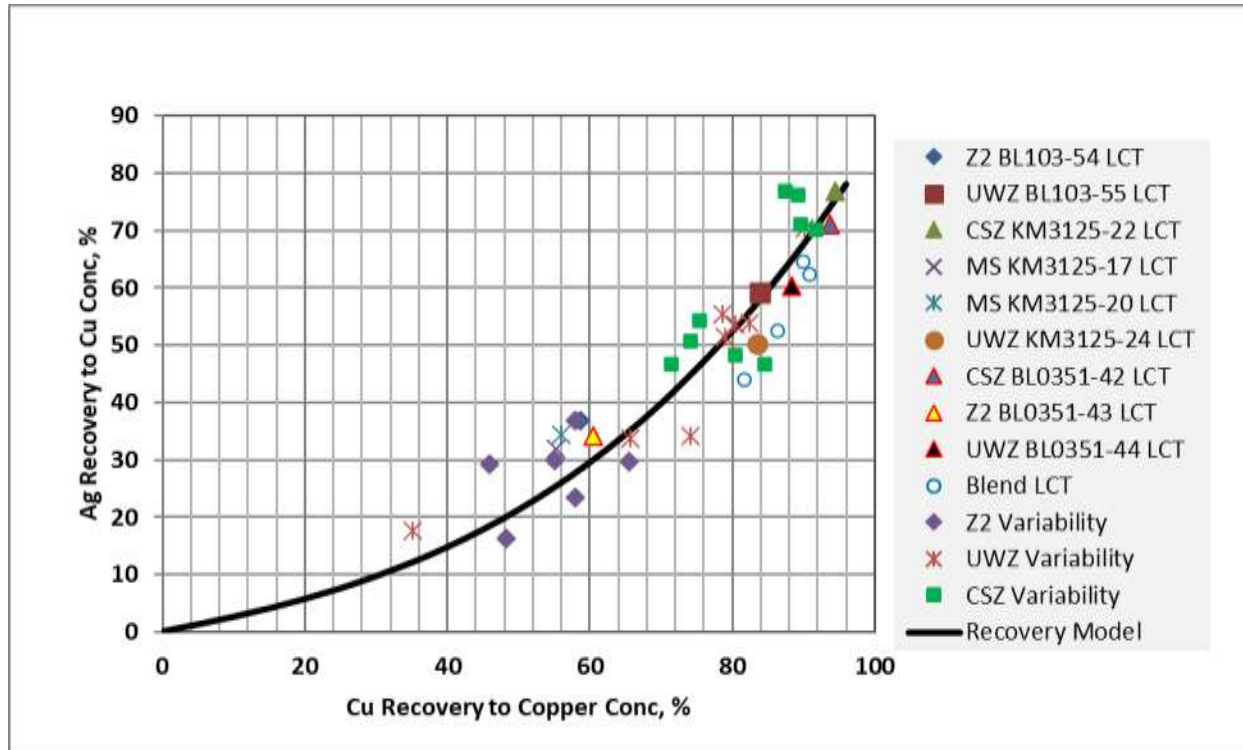
and for the massive sulphide:

$$MS \text{ Cu Concentrate Grade } (\%) = \frac{24.4 \times F^{2.47} - 0.17}{F^{2.47} + 0.02}$$

Where F is the copper grade (%) of the CSZ or MS component of the mill feed.

Recoveries of silver and gold to the copper concentrate as functions of the copper recovery are summarized in Figure 13-13 and Figure 13-14.

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

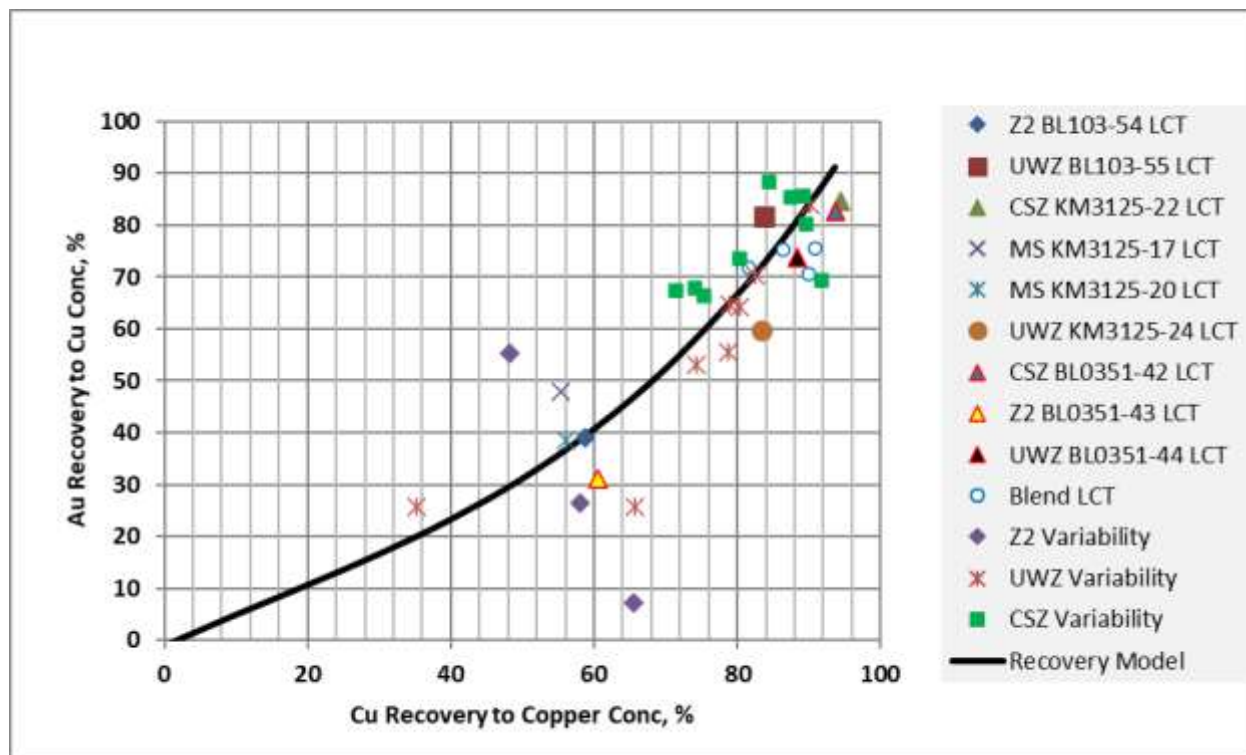


The silver recovery to copper concentrate vs copper recovery relationship is defined as:

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

Where C is the copper recovery to copper concentrate.

Figure 13-14: 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:

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

Where C is the copper recovery to copper concentrate. Note that gold recovery to the copper concentrate is capped at 92% based on the available data. Additional gold recovery to the zinc concentrate is estimated at 1% of the contained gold in the CSZ and 12% of the contained gold in the MS based on locked cycle test data.

14.0 MINERAL RESOURCE ESTIMATES

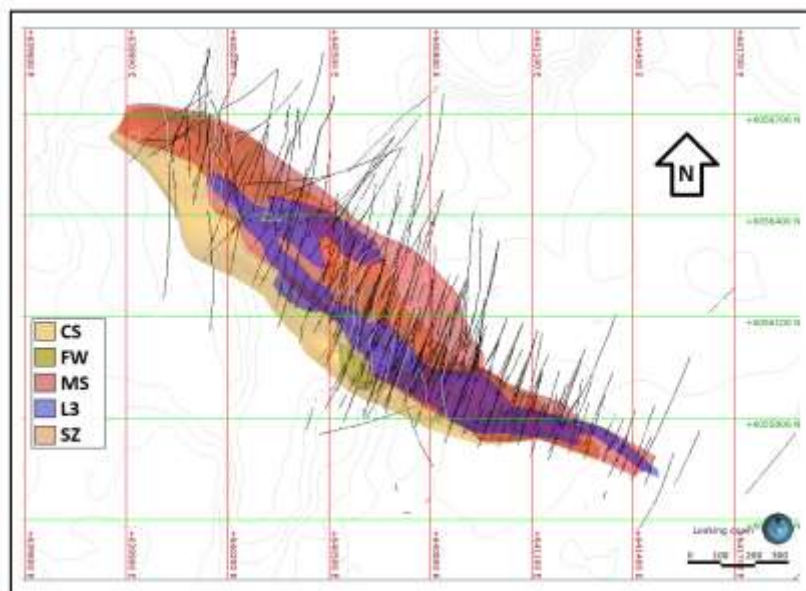
14.1 Introduction

This section discusses the updated mineral resource estimate for Foran's Mcllvenna Bay Project in Saskatchewan. The updated mineral resource estimate is based upon Foran's drilling database, which includes both the historical drilling and Foran's drilling in results of 2021. Micon's QPs have conducted the mineral resource estimate for disclosure under NI 43-101 standards.

The 2021 drilling program was designed not only to improve the confidence of the known inferred mineralization, such that it could be upgraded to indicated, but also to potentially increase the mineral resources at depth. Previous iterations of the resource model have been completed and published since 2010, with all of these previous resource estimations now superseded by the current 2021 estimate discussed in this section.

The Mcllvenna Bay mineral resources have been estimated using multiple tabular interpretations defined in five mineralization zones: Copper Stockwork (CSZ), Massive Sulphide (MS), Lens 3 (L3), Stringer Zone (SZ) and Copper Stockwork Footwall (FW). The five zones contain steep parallel, contiguous vein-type structures, disposed next to each other with similar bearings and dips. Figure 14-1 shows a plan view of the five interpreted zones defined by Foran and constructed by Micon. The mineral resources for the Mcllvenna Bay zones have been estimated assuming an underground mining scenario.

Figure 14-1: Plan View Foran Mcllvenna Bay Mineralized Zones and Drill Holes



Source: Micon 2021.



14.2 CIM Mineral Resource Definitions and Classifications

If a company is a reporting Canadian entity on Canadian mineral properties, all mineral resources and reserves presented in a Technical Report must follow the current CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014), which includes the mineral resource definitions reproduced below:

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.

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.

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.

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.

14.3 CIM Estimation of Mineral Resources Best Practice Guidelines

Micon and its QPs have used the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines which were adopted by the CIM Council on 29 November 2019 (CIM Council 2019), in estimating the mineral resources contained within of the McIlvenna Bay Project. The November 2019 guidelines supersede the 2003 CIM Best Practice Guidelines which were followed by Micon and its QPs when completing the previous resource estimations and audits for the Project.

14.4 Mineral Resource Database and Wireframes

14.4.1 Supporting Data

The basis for the mineral resource estimate was a drill hole database provided by Foran. The database and underlying QA/QC data were validated by Foran prior to being used in the modelling and estimation. After a further validation of the database, it was decided to exclude four drill holes² from the resource estimate due to: 1) an inaccurate collar survey for one hole; 2) one drill hole that was abandoned during the drilling program and 3) two drill holes because the assays had not been received by the time database was given to Micon. 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.

² The excluded drill holes are MB-99-108, MB-21-230, MB-21-251 and MB-21-253. Drill hole 108 was removed due to an inaccurate collar location as confirmed by Foran. Drill hole 251 was excluded because it was abandoned during the drilling program. Drill holes 230 and 253 were excluded because the assays for these holes were still pending at the time of the mineral resource estimate.



Table 14-1: McIlvenna Bay Project Database

Data Type	In Database	Used For 2021 Resource Estimate*
Drill Collar	285	240
Assay Samples	11,737	5,652
Core Metreage	152,130	4,658**

*Excludes four drill holes from the resource estimate.

**Actual meters used within the resource wireframes.

Topography

The project topography was provided by Foran as a digital terrain model (DTM) in DXF format. The DTM was of sufficient quality, although, given the underground extraction assumption, it was not used for the mineral resource estimate.

14.4.2 Wireframes

Foran and Micon jointly defined, five mineralized domains, representing different areas and styles of VMS mineralization using Leapfrog Version 2021.1.3.

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.

CSZ – Copper stockwork zone sitting stratigraphically below the Massive Sulphide.

Stringer Zone – Copper and zinc stringer zone in the hanging wall above the Massive Sulphides.

Lens 3 – Massive Sulphide lens sitting in the hanging wall to the Stringer Zone.

FW – Small massive to semi-massive zone ore zone below the CSZ.

Wireframes were generated based on a set of mineralized intercepts defined by Foran and validated by Micon. 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.

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.



14.5 Compositing and Variography

14.5.1 Compositing

The selected intercepts for the McIlvenna Bay Project were composited into 1.0 m equal length intervals, with the composite length selected based on the most common original sample length. Table 14-2 summarizes basic statistics for the composited data.

Table 14-2: Summary of the Basic Statistics for the 1.0 m Composites

Zone	Dataset	Element	Count	Length	Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max
CSZ	Uncapped	Cu (%)	2,791	2,779	1.26	0.93	0.74	0.86	0.01	0.66	1.07	1.60	10.78
		Zn (%)	2,791	2,779	0.33	0.52	1.59	0.27	0.01	0.06	0.15	0.37	8.72
		Pb (%)	2,791	2,779	0.02	0.05	2.98	0.00	-	0.01	0.01	0.01	0.91
		Au (g/t)	2,791	2,779	0.38	0.68	1.82	0.47	0.00	0.08	0.17	0.41	11.05
	Capped	Ag (g/t)	2,791	2,779	8.55	8.21	0.96	67.45	0.10	3.65	6.19	10.60	110.00
		Cu (%)	2,791	2,779	1.26	0.89	0.71	0.79	0.01	0.66	1.07	1.60	6.00
		Zn (%)	2,791	2,779	0.32	0.50	1.54	0.25	0.01	0.06	0.15	0.37	4.50
		Pb (%)	2,791	2,779	0.02	0.05	2.62	0.00	-	0.01	0.01	0.01	0.45
FW	Uncapped	Au (g/t)	2,791	2,779	0.36	0.54	1.50	0.29	0.00	0.08	0.17	0.41	4.00
		Ag (g/t)	2,791	2,779	8.38	7.12	0.85	50.76	0.10	3.65	6.19	10.60	40.00
		Cu (%)	217	217	1.38	1.03	0.75	1.07	0.12	0.69	1.09	1.69	6.07
		Zn (%)	217	217	0.50	1.71	3.40	2.92	0.01	0.04	0.08	0.20	13.22
	Capped	Pb (%)	217	217	0.03	0.10	3.42	0.01	0.00	0.01	0.01	0.01	0.83
		Au (g/t)	217	217	0.45	0.61	1.35	0.37	0.02	0.13	0.25	0.54	4.65
		Ag (g/t)	217	217	9.10	8.77	0.96	76.99	1.34	3.30	5.81	11.96	54.56
		Cu (%)	217	217	1.38	1.03	0.75	1.06	0.12	0.69	1.09	1.69	6.00
MSZ2	Uncapped	Zn (%)	217	217	0.36	0.92	2.59	0.85	0.01	0.04	0.08	0.20	4.50
		Pb (%)	217	217	0.03	0.08	3.16	0.01	0.00	0.01	0.01	0.01	0.45
		Au (g/t)	217	217	0.44	0.57	1.29	0.32	0.02	0.13	0.25	0.54	4.00
		Ag (g/t)	217	217	8.98	8.27	0.92	68.45	1.34	3.30	5.81	11.96	40.00
	Capped	Cu (%)	1,096	1,092	0.92	1.48	1.61	2.20	0.01	0.13	0.27	1.09	16.84
		Zn (%)	1,096	1,092	6.26	4.02	0.64	16.13	0.02	2.92	6.17	8.90	22.56
		Pb (%)	1,096	1,092	0.47	0.66	1.41	0.44	-	0.07	0.27	0.60	6.52
		Au (g/t)	1,096	1,092	0.55	1.03	1.88	1.07	-	0.12	0.24	0.55	14.30
L3	Uncapped	Ag (g/t)	1,096	1,092	28.08	32.76	1.17	1,073.00	1.84	12.61	18.73	29.32	342.51
		Cu (%)	1,096	1,092	0.90	1.36	1.51	1.85	0.01	0.13	0.27	1.09	7.00
		Zn (%)	1,096	1,092	6.26	4.01	0.64	16.07	0.02	2.92	6.17	8.90	21.00
		Pb (%)	1,096	1,092	0.47	0.64	1.38	0.41	-	0.07	0.27	0.60	5.00
	Capped	Au (g/t)	1,096	1,092	0.54	0.90	1.69	0.82	-	0.12	0.24	0.55	7.00
		Ag (g/t)	1,096	1,092	27.61	29.47	1.07	868.57	1.84	12.61	18.73	29.32	200.00
		Cu (%)	341	335	0.93	1.01	1.09	1.02	0.01	0.34	0.64	1.23	8.40
		Zn (%)	341	335	2.39	2.72	1.14	7.39	0.01	0.35	1.37	3.63	15.29
SZ	Uncapped	Pb (%)	341	335	0.11	0.18	1.72	0.03	0.00	0.02	0.04	0.11	1.30
		Au (g/t)	341	335	0.27	0.36	1.37	0.13	0.00	0.11	0.19	0.32	5.34
		Ag (g/t)	341	335	13.43	10.02	0.75	100.31	1.01	6.90	11.17	17.25	88.70
		Cu (%)	341	335	0.91	0.91	0.99	0.82	0.01	0.34	0.64	1.23	5.00
	Capped	Zn (%)	341	335	2.31	2.45	1.06	6.00	0.01	0.35	1.37	3.63	9.00
		Pb (%)	341	335	0.10	0.14	1.42	0.02	0.00	0.02	0.04	0.11	0.60
		Au (g/t)	341	335	0.25	0.20	0.81	0.04	0.00	0.11	0.19	0.32	1.00
		Ag (g/t)	341	335	13.18	8.76	0.66	76.77	1.01	6.90	11.17	17.25	45.00



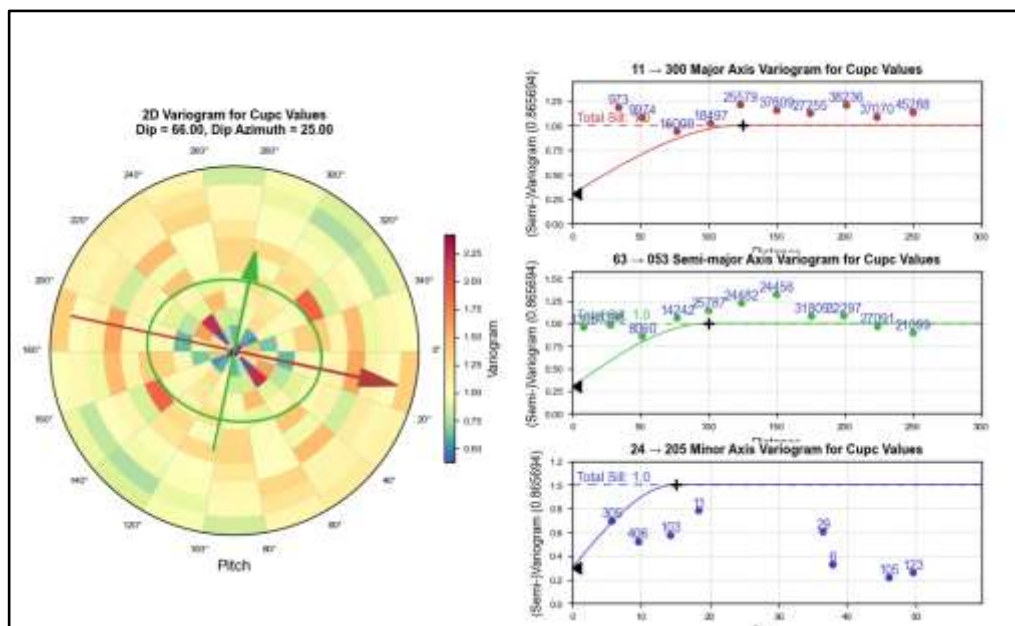
Zone	Dataset	Element	Count	Length	Mean	SD	CoV	Var	Min	Q1	Median	Q3	Max
		Au (g/t)	236	231	0.28	0.24	0.86	0.06	0.00	0.11	0.21	0.36	1.00
		Ag (g/t)	236	231	13.99	11.33	0.81	128.42	0.40	6.90	11.27	17.10	60.00

14.5.2 Variography

Variography is the analysis of the spatial continuity of grade for the commodity of interest. In the case of the Mcllvenna Bay Deposit, the analysis was done on each individual zone using down-the-hole variograms and 3D variographic analysis in order to define the directions of maximum continuity of grade and, therefore, the best parameters to interpolate the grades of each of the five zones.

Variography must be performed on regular coherent shapes with geological continuity support. First, down-the-hole variograms were constructed for each vein, to establish the nugget effect to be used in the modelling of the 3D variograms. Figure 14-2 to show the variograms for the CSZ as an example.

Figure 14-2: CSZ Zone – 3D Variogram Summary for Copper



Source: Micon 2021.

Micon obtained good variogram models for all the five zones. They were sufficiently reliable to support the use of the Ordinary Kriging interpolation method. Major variogram ranges between 60 m and 125 m were modelled. Most ranges were in the range of 100 to 125 m for both copper and zinc. The variography results were used to support the search ranges and anisotropy directions.



Continuity and Trends

The McIlvenna Bay zones exhibit fairly stable strike and dip directions, with very mild variations. For the most part, both of the CSZ and the MS zones are contiguous with the remaining zones, running as parallel structures with well-defined geometries. Continuity of the zones is generally not only supported by geology but also by mineralization, with the regular drill hole intercepts giving sufficient confidence to the continuity of grade, both along strike and down dip. The general deposit bearings and dips are 315° strike direction and -68° dip, with a general plunge of -40° toward the northwest.

14.6 Grade Capping

All outlier assay values for copper, zinc, lead, gold, and silver were analyzed individually by zone, using log probability plots and histograms. It was decided to cap outlier assays based on the data grouped by zone.

To identify true outliers and reduce the effect of short sample bias, the data were reviewed after compositing to a constant length of 1.0 m. Table 14-3 summarizes the capping grades used.

Table 14-3: Selected Capping Grades on 1 m Composites

Zone	Element	Max. Grade	Capping Grade	Capped Composites	Total Composites
CSZ	Cu (%)	10.78	6.00	9	2,791
	Zn (%)	8.72	4.50	4	2,791
	Pb (%)	0.91	0.45	13	2,791
	Au (g/t)	11.05	4.00	16	2,791
	Ag (g/t)	110	40.00	27	2,791
FW	Cu (%)	6.10	6.00	1	217
	Zn (%)	13.22	4.50	8	217
	Pb (%)	0.82	0.45	4	217
	Au (g/t)	4.65	4.00	2	217
	Ag (g/t)	54.56	40.00	4	217
MS	Cu (%)	16.84	7.00	8	1,096
	Zn (%)	22.55	21.00	3	1,096
	Pb (%)	6.52	5.00	3	1,096
	Au (g/t)	14.30	7.00	9	1,096
	Ag (g/t)	342.50	200.00	8	1,096
L3	Cu (%)	8.40	5.00	4	341
	Zn (%)	15.29	9.00	12	341
	Pb (%)	1.30	0.60	10	341
	Au (g/t)	5.34	1.00	6	341
	Ag (g/t)	88.70	45.00	5	341



Zone	Element	Max. Grade	Capping Grade	Capped Composites	Total Composites
SZ	Cu (%)	5.26	4.00	4	236
	Zn (%)	11.20	2.00	19	236
	Pb (%)	6.05	1.50	3	236
	Au (g/t)	3.16	1.00	9	236
	Ag (g/t)	259.22	60.00	5	236

14.7 Rock Density

A total of 8,558 density measurements were delivered to Micon, from which average densities were calculated for each zone at the McIlvenna Bay Project. The overall average density value for the entire deposit is 2.96 g/cm³.

Table 14-4 summarizes the density measurement database for the McIlvenna Bay Project.

Table 14-4: Summary of the Density Measurements by Zone

Deposit Name	Count	Length (m)	Density Value (t/m ³)
Global (Entire Deposit)	8,558	3,315	2.96
CSZ	1,304	1,024	2.87
FW	74	57	2.97
MS	649	395	3.59
L3	174	109	3.28
SZ	140	99	3.00
Outside (Waste Rock)	6,209	1,632	2.83

14.8 Mineral Resource Estimate

The commodities of economic interest at the McIlvenna Bay Project are primarily copper and zinc, with secondary recoveries of gold and silver. The estimation of the deposit tonnage and grade was performed using Leapfrog Geo/EDGE software.

14.8.1 Block Model

A block model was constructed to represent the grades and densities within the five zones. A summary of the block model parameters is provided in Table 14-5.



Table 14-5: Block Model Information Summary

Description	Values Used
Model Dimension X (m)	2,000
Model Dimension Y (m)	550
Model Dimension Z (m)	1,450
Origin* X (Easting)	639,660
Origin* Y (Northing)	6,056,460
Origin* Z (Upper Elev.)	360
Clockwise Rotation (°)	35
Parent Block Size X (m) - Along Strike	10.0
Parent Block Size Y (m) - Across Strike	2.0
Parent Block Size Z (m) - Down Dip	10.0
Child Block Size X (m) - Along Strike	2.0
Child Block Size Y (m) - Across Strike	0.5
Child Block Size Z (m) - Down Dip	2.0

Note: Origin is the centroid of the block

The drill hole intercepts used to model the wireframes were flagged into the mineral envelope to which they belonged. Each zone was interpolated using only the composites within that zone.

Search Strategy and Interpolation

A set of parameters were derived from variographic analysis to interpolate the composite grades into the blocks. A summary of the McIlvenna Bay Project Ordinary Kriging (OK) interpolation parameters is provided in Table 14-6.



Table 14-6: Ordinary Kriging Interpolation Parameter Summary

Element	Zone	Pass	Orientation			Search Parameters					
			Dip (°)	Dip Az (°)	Pitch (°)	Range Major Axis (m)	Range Semi-Major Axis (m)	Range Minor Axis (m)	Minimum Samples	Maximum Samples	Maximum Samples per Hole
Cu	CSZ	1	Dynamic Anisotropy			125	100	15	9	18	3
	MS	1				120	100	10	9	18	3
	L3	1				100	80	10	9	18	3
	FW	1	66	25	12	125	100	15	9	18	3
	SZ	1	Dynamic Anisotropy			100	80	10	9	18	3
Zn	CSZ		Dynamic Anisotropy			100	100	10	9	18	3
	MS					120	120	10	9	18	3
	L3					90	80	10	9	18	3
	FW		66	25	30	100	100	10	9	18	3
	SZ		Dynamic Anisotropy			80	60	10	9	18	3
Pb	CSZ		Dynamic Anisotropy			80	80	15	9	18	3
	MS					100	80	10	9	18	3
	L3					90	80	10	9	18	3
	FW		66	25	30	80	80	15	9	18	3
	SZ		Dynamic Anisotropy			80	80	10	9	18	3
Au	CSZ		Dynamic Anisotropy			60	60	10	9	18	3
	MS					100	80	10	9	18	3
	L3					100	100	10	9	18	3
	FW		66	25	68	60	60	10	9	18	3
	SZ		Dynamic Anisotropy			80	80	10	9	18	3
Ag	CSZ		Dynamic Anisotropy			80	60	15	9	18	3
	MS					80	80	10	9	18	3
	L3					80	60	10	9	18	3
	FW		66	25	61	80	60	15	9	18	3
	SZ		Dynamic Anisotropy			80	80	10	9	18	3
All	All	2	Same as Pass 1			x 2	x 2	x 2	1	12	3

14.8.2 Prospects for Economic Extraction

The CIM Standards require that a mineral resource must have reasonable prospects for eventual economic extraction. The mineral resource discussed herein has been constrained by reasonable mining shapes, using economic assumptions appropriate for an underground mining scenario. The potential mining shapes are conceptual in nature, not stope designs, and are based on a US\$60.0/t net smelter return (NSR) cut-off value.

The metal prices and operating costs were provided by Foran and accepted by Micon's QP are considered appropriate to be used as the economic parameters for the mineral resource estimate.

Table 14-7 summarizes the underground economic assumptions upon which the resource estimate for the McIlvenna Bay Project is based.



Table 14-7: Summary of Economic Assumptions for the Mineral Resource Estimate

Description	Units	Value Used	Notes/Details
Metal Prices			
Copper Price	US\$/lb	\$4.25	
Zinc Price	US\$/lb	\$1.35	
Gold Price	US\$/oz	\$1,800	
Silver Price	US\$/oz	\$25.00	
Operating Costs			
Mining	US\$/t	\$41.00	From PFS
Processing	US\$/t	\$20.00	From PFS
G&A	US\$/t	\$8.40	From PFS
Royalty			
BHP Royalty (million)	CAD\$	\$1.0	From PFS
Copper Reef	CAD\$/t	\$0.75	From PFS
Marketing and Smelting Charges for Cu Concentrate			
Concentrate Moisture	%	8.0	
Payables:			
Cu	%	96.5	
Minimum deduction (units)	%	0.0	No minimum deduction
Au in Cu Con	%	96.0	
Ag in Cu Con	%	90.0	
Toll Charge	US\$/t	\$65.00	
Refining Charges:			
Cu	US\$/lb	\$0.065	
Au	US\$/oz	\$5.00	
Ag	US\$/oz	\$0.45	
Penalty for Impurities		\$0.00	No appreciable impurities - zero penalty
Transportation Cost	US\$/t	\$104.41	Assumes shipping to Sudbury
Marketing and Smelting Charges for Zn Concentrate			
Concentrate Moisture	%	9.0	
Zinc Payable	%	85.0	
Toll Charge	US\$/t	\$125.00	
Penalty for Impurities		\$0.00	
Transportation Cost	US\$/t	\$100.94	Assumes shipping to Trail

The economic parameters in Table 14-7 provided the foundation from which to develop NSR values for each block in the model.



NSR Calculation

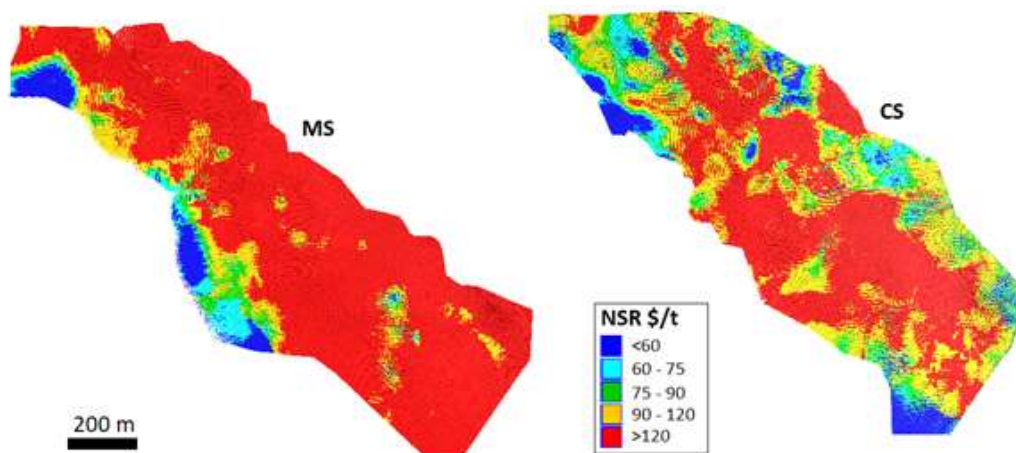
Using all the parameters shown in Figure 14-3, Micon proceeded to calculate the NSR values for each block in the model. The theoretical NSR formula shown in Figure 14-3 was used for the current Resource Estimate. Figure 14-4 shows the blocks by NSR value for the MS and CSZ zones.

Figure 14-3: NSR Theoretical Formula Used at Mcllvenna Bay

$$NSR(x_1, x_2, \dots, x_n) = x_1 r_1 p_1 (V_1 - R_1) + x_2 r_2 p_2 (V_2 - R_2) + \dots + x_n r_n p_n (V_n - R_n) - \frac{C_s}{K} - \frac{C_t}{K}$$

Variable	Definition
x	Grade of each metal in deposit
r	Process recovery of each metal
R	Refining cost of each metal
p	Smelting recovery of each metal
V	Market sale value of each metal
K	Tonnes of material required to produce one tonne of concentrate
C_s	Smelter cost per ton of concentrate
C_t	Transportation costs per ton of concentrate

Figure 14-4: MS and CSZ Zone Resource Blocks by NSR Value

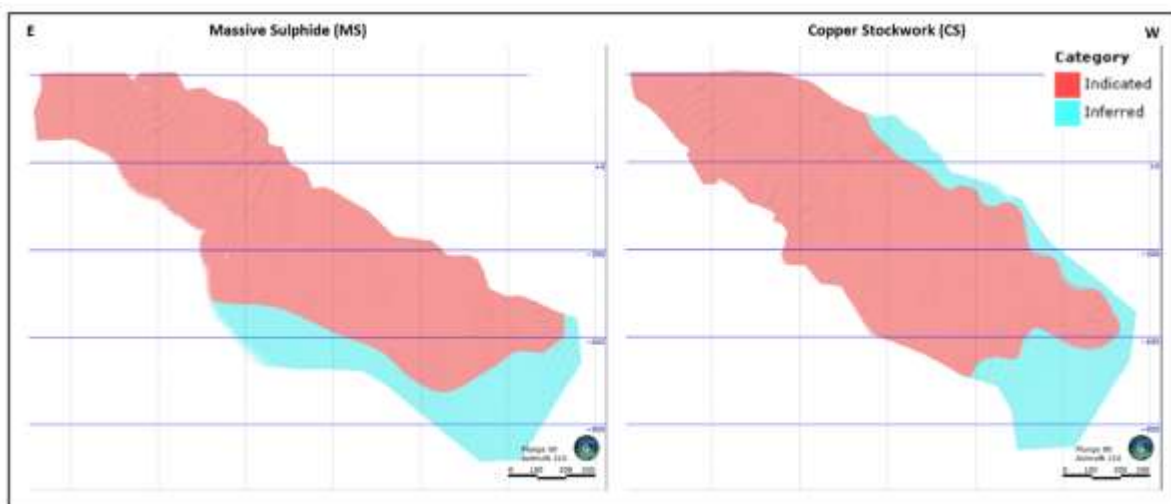


Mineral Resource Classification

Micon has classified the mineral resources at the Mcllvenna Bay Project in the indicated and inferred categories. No measured resource is declared at this time. The FW, L3 and SZ zones are entirely classified as indicated resources.

Micon categorized as indicated resource those blocks informed by at least 4 drill holes and within 100 to 120 m spacing, based on the ranges obtained in the variograms. The results were then smoothed out to remove isolated small blocks and produce coherent shapes of reasonable volume, eliminating the spotted dog effect. All other blocks were classified in the inferred category. Figure 14-5 shows the resource classification for the MS and CSZ zones.

Figure 14-5: Resource Classification for the MS and CSZ Zones, Looking in the SW Direction



14.9 Mineral Resource Estimate

14.9.1 Mineral Resource Estimate

The updated mineral resource estimate discussed herein is summarized in Table 14-8. The effective date of this mineral resource is 06 September 2021, and the resource is reported using an NSR cut-off of US \$60/t.

Table 14-8: Mineral Resources for the McIlvenna Bay Deposit, Reported at an NSR of US\$ 60/t

Category	Zone	Mass (Mt)	NSR (US\$/t)	Average Grades						Contained Metal				
				Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Zn (Mlb)	Pb (Mlb)	Au (Moz)	Ag (Moz)
Indicated	MS	10.75	198.8	1.01	6.17	0.41	0.53	26.6	3.13	238	1,462	98	0.18	9.2
	CSZ	22.74	127.1	1.31	0.38	0.02	0.37	9.1	1.60	659	190	10	0.27	6.7
	SZ	1.19	119.3	1.26	0.52	0.07	0.31	13.0	1.53	33	14	2	0.01	0.5
	L3	2.57	113.1	0.82	3.07	0.14	0.25	14.5	1.80	47	174	8	0.02	1.2
	FW	1.80	140.7	1.42	0.59	0.04	0.45	8.8	1.79	56	23	2	0.03	0.5
	Total	39.06	146.3	1.20	2.16	0.14	0.41	14.4	2.04	1,033	1,863	119	0.51	18.1
Inferred	MS	1.56	162.6	0.65	6.51	0.46	0.29	27.8	2.66	22	224	16	0.01	1.4
	CSZ	3.48	105.6	1.08	0.79	0.03	0.25	10.5	1.37	83	60	3	0.03	1.2
	Total	5.04	123.3	0.94	2.56	0.17	0.27	15.9	1.77	105	284	19	0.04	2.6

Notes:

1. Effective date September 6, 2021; CIM definitions were followed for Mineral Resources; CuEq = copper equivalent; NSR = Net Smelter Return.
2. The mineral resource is estimated based on 240 diamond drill holes and a NSR cut-off of US\$60/t. NSR grades values derived, and high-grade caps were applied as per the discussion in Estimation Methodology and Parameters and include provisions for metallurgical recovery and estimates of current shipping terms and smelter rates for similar concentrates. Metal prices used are US\$4.25/lb. Cu, US\$1.35/lb. Zn, US\$1,800/oz. Au, and US\$25.00/oz. Ag. Lead contributes no value.
3. Rock density was interpolated for each block based on measurements taken from core specimens, with an average value of 3.59 g/cm³ for the main MS lens and 2.87 g/cm³ for the CSZ.
4. Mineral resources which are not mineral reserves do not have demonstrated economic viability.
5. CuEq values were calculated from the NSR values for each zone, using both concentrate and recovery curves that were developed during Pre-Feasibility level metallurgical studies.
6. The block model grades were estimated using the Ordinary Kriging interpolation method, with search parameters derived from geostatistical analysis performed within the mineralization wireframes. Variogram ranges are from 65 m to 85 m for Au and Ag in the major axis and up to 100 to 120 m for Cu and Zn.
7. Micon has not identified any legal, political, environmental, or other factors that could materially affect the potential development of the mineral resource estimate.



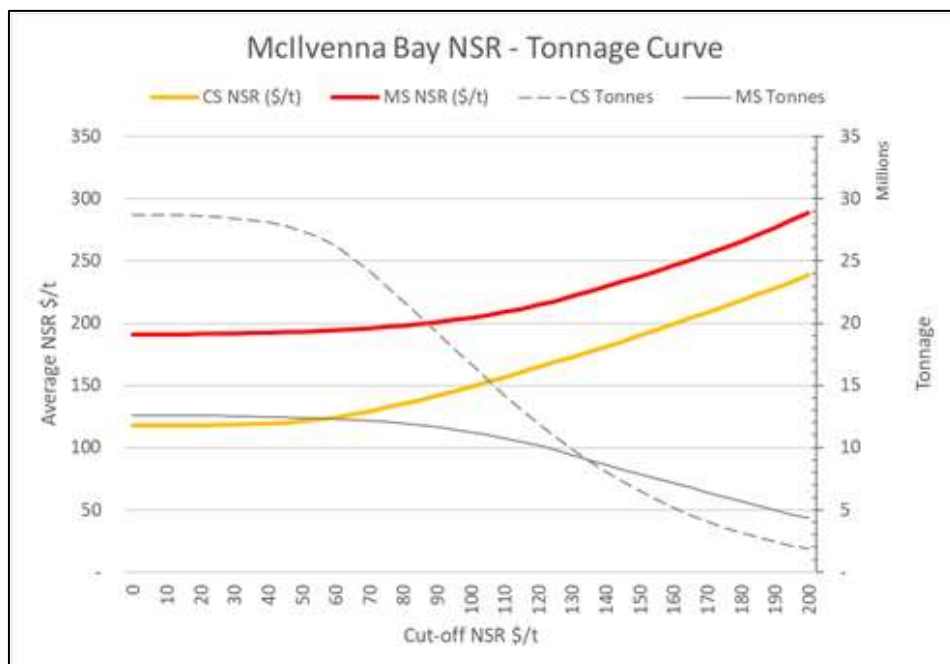
14.9.2 Sensitivity Analysis

As part of its update of Foran's 2021 mineral resource estimate, Micon examined the sensitivity of the mineral resource to a higher and lower NSR cut-offs. Table 14-9 summarizes the NSR sensitivity at US\$90/t, US\$75/t and US\$45/t, with the base case at US\$60/t. Figure 14-6 is a sensitivity graph which demonstrates the variation in tonnage and grade for the resource at different NSR cut-offs for the MS and CZ major zones. The QP has reviewed the NSR cut-offs used in the sensitivity analysis and it is the QP's opinion that they meet the test of reasonable prospects of economic extraction.

Table 14-9: Summary of the NSR Sensitivities at US\$90/t, US\$75/t, US\$45/t with Base Case at US\$60/t

Category	NSR Cut-off	Mass Mt	NSR US\$/t	Average Grades						Contained Metal				
				Cu %	Zn %	Pb %	Au g/t	Ag g/t	CuEq %	Cu Mlb	Zn Mlb	Pb Mlb	Au Moz	Ag Moz
Indicated	90	31.59	162.92	1.29	2.49	0.16	0.47	15.86	2.27	901.68	1,735.19	110.15	0.47	16.11
	75	35.68	153.71	1.24	2.31	0.15	0.43	15.02	2.14	977.97	1,813.87	115.24	0.50	17.23
	60	39.06	146.31	1.20	2.16	0.14	0.41	14.39	2.04	1,033.35	1,863.09	118.69	0.51	18.07
	45	40.92	142.07	1.17	2.09	0.13	0.40	14.08	1.98	1,058.43	1,886.50	120.69	0.52	18.52
Inferred	90	3.58	142.77	1.01	3.23	0.21	0.32	18.29	2.07	79.79	255.46	16.31	0.04	2.11
	75	4.32	132.54	0.98	2.86	0.18	0.29	16.92	1.91	93.23	272.68	17.55	0.04	2.35
	60	5.04	123.29	0.94	2.56	0.17	0.27	15.85	1.77	104.87	284.14	18.51	0.04	2.57
	45	5.35	119.22	0.93	2.44	0.16	0.26	15.41	1.71	109.18	287.99	18.89	0.04	2.65

Figure 14-6: MS and CSZ Zones Resource Sensitivity by NSR Value



14.9.3 Responsibility for Estimation

The updated mineral resource estimated discussed in this Technical Report has been prepared under the supervision of William J. Lewis, P.Geo., of Micon. Mr. Lewis is independent of Foran and is a Qualified Person within the meaning of NI 43-101.

14.10 Block Model Validation

In validating the block model and the resource estimate, Micon conducted a statistical comparison of the input 1 m composites, against output interpolated data in the block model. Table 14-10 shows the comparison of global means, by element, and Figure 14-7 to Figure 14-10 show the swath plots of Cu and Zn for the two major zones, MS and CSZ. All comparisons show good agreement between the input data and the output estimates.

Table 14-10: McIlvenna Bay Statistical Comparison: Composites (Input) vs Blocks (Output)

Element	1 m Composites		Block Model	
	Count	Mean	Block Count	Mean
Cu	4,681	1.15	5,382,466	1.14
Zn	4,681	1.87	5,382,466	1.84
Pb	4,681	0.13	5,382,466	0.12
Au	4,681	0.39	5,382,466	0.37
Ag	4,681	13.54	5,382,466	13.35



Figure 14-7: MS Zone – Cu Swath Plot

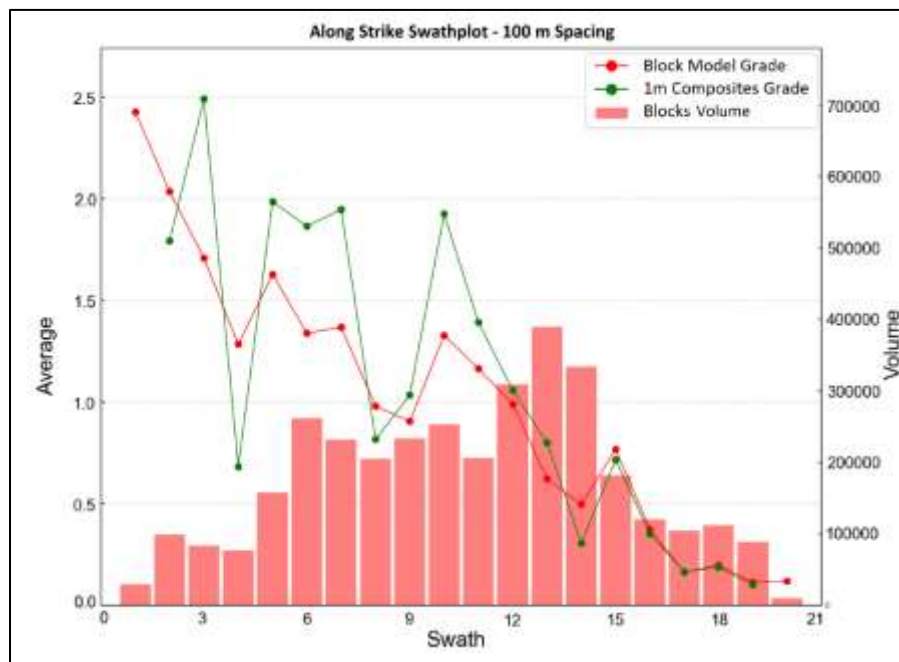


Figure 14-8: MS Zone – Zn Swath Plot

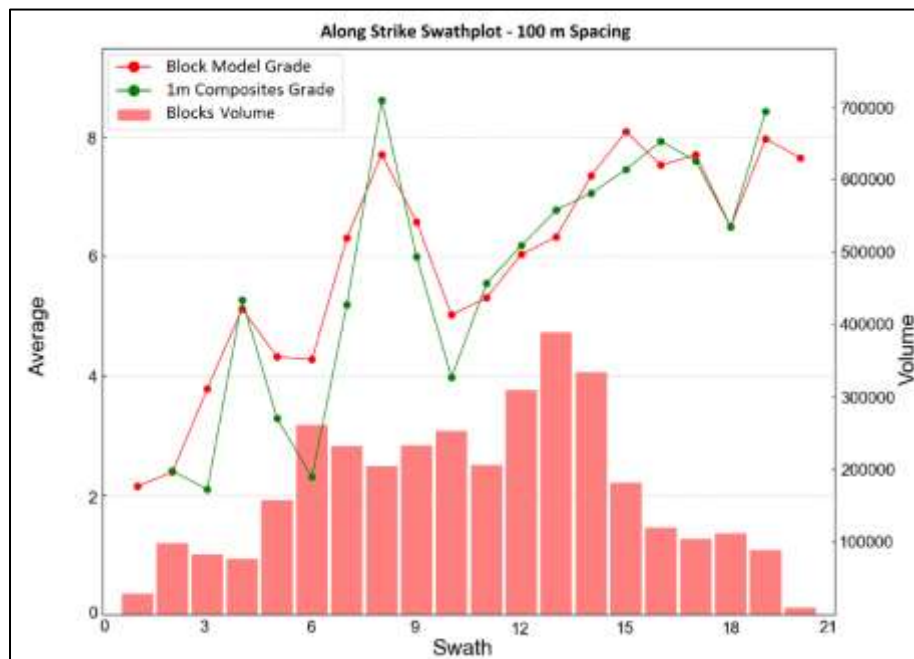


Figure 14-9: CSZ – Cu Swath Plot

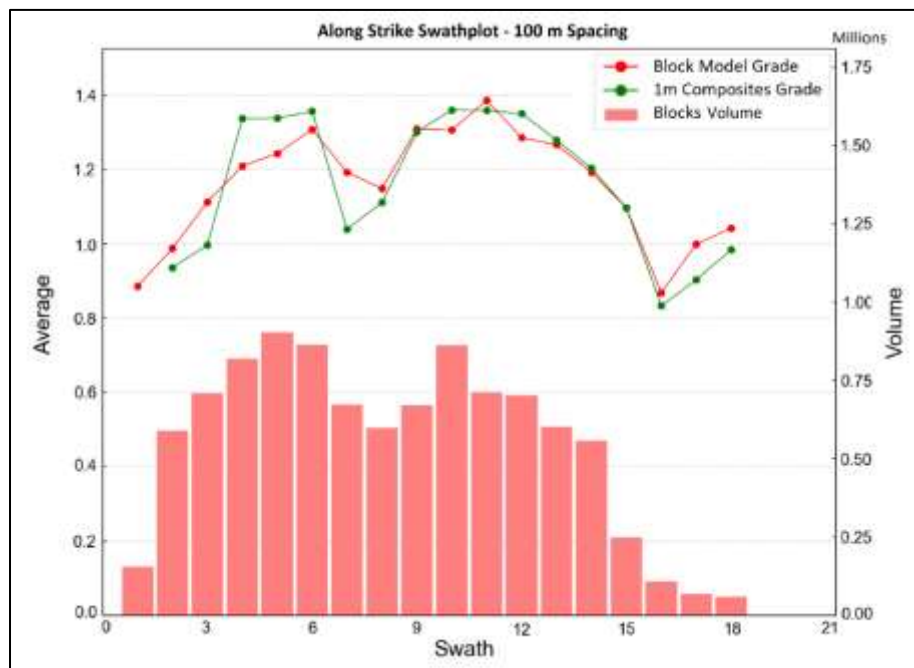
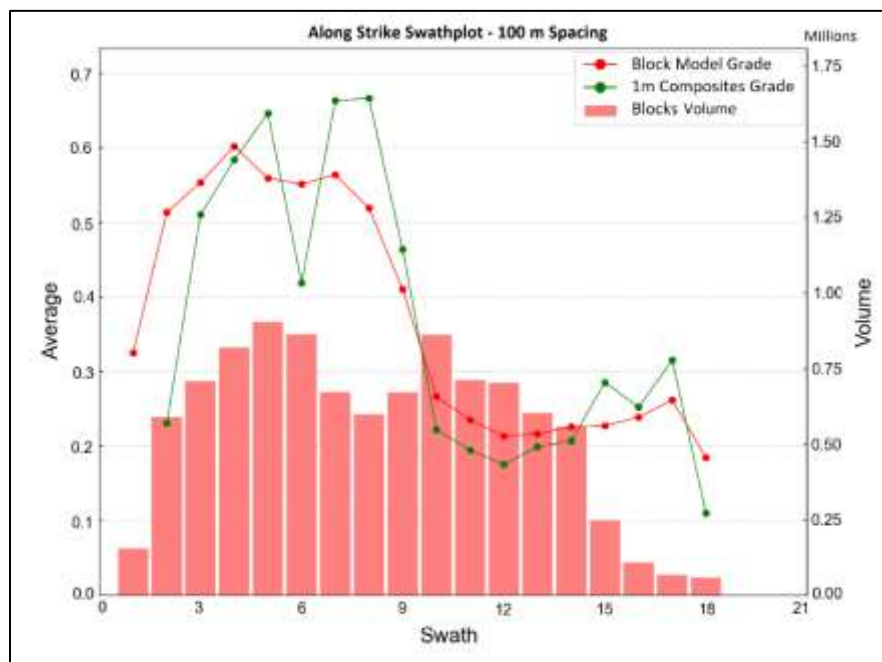


Figure 14-10: CSZ– Zn Swath Plot



15.0 MINERAL RESERVE ESTIMATE

15.1 Introduction

The Mineral Reserve estimate was prepared by Stantec Consulting Ltd. (Stantec). The McIlvenna Bay Deposit Mineral Reserve estimate is based on Indicated Mineral Resources identified in the block model provided by Foran (MCBAYBM_MICON2021NSR_ALL, issued on 09 September 2021). All mineral resource material in the block model that was classified as Inferred was assigned a zero grade.

The Mineral Reserve estimate consists of selected portions of the indicated mineral resource that are above a US \$90.24/tonne Net Smelter Return (NSR) cut-off value. This cut-off value was applied when generating the stoping shapes.

The Mineral Reserve is categorized as either Massive Sulphide or Copper Stockwork. It is assumed that both transverse stope and longitudinal retreat stope mining methods as well as Avoca mining will be used.

15.2 Mineral Reserve Statement

The Mineral Reserve for the McIlvenna Bay Deposit is estimated at 25,703,062 tonnes, grading 2.39% Zn, 1.23% Cu, 0.47 g/tonne Au and 15.4 g/tonne Ag. This Mineral Reserve estimate includes dilution, mining recovery factors, and marginal tonnes, which is classified as marginal development or incremental stoping.

Estimates of mineralization and other technical information included herein have been prepared in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

Table 15-1 presents the estimated Mineral Reserve.

Table 15-1: Mineral Reserve Estimate

Classification	Ore Tonnes	Zn (%)	Cu (%)	Au (g/t)	Ag (g/t)
Massive Sulphide	10,135,305	5.43	0.99	0.51	23.8
Copper Stockwork	15,567,756	0.41	1.39	0.45	9.9
Total	25,703,062	2.39	1.23	0.47	15.4

1. CIM definitions were followed for Mineral Reserves.
2. Mineral Reserves include transverse, longitudinal, and Avoca stopes, as well as ore development, marginal development, and incremental stopes.
3. Stopes were estimated at a cut-off value of US \$90.24/tonne NSR.
4. Marginal tonnes were estimated at a cut-off value of US \$60.55/tonne NSR.
5. A minimum mining width of 3.0 m was applied for all stoping.
6. Total dilution is 22.2%. Total mineable recovery is 93.6%
7. Numbers may not sum due to rounding.
8. NSR Reserve Prices (\$US); Cu \$3.50/lb, Zn \$1.20/lb, Ag \$22.50/oz, Au \$1600/oz



15.3 Underground Assumptions / Design Criteria

15.3.1 Throughput Rate and Supporting Assumptions

The assumed process plant capacity is 4,200 t/d. Additional information is provided in Section 16.0.

15.3.2 Stope Shape Design

The mine design is based on using a sublevel longhole mining method to extract the reserves. The mine design utilizes a combination of longitudinal, transverse, and Avoca stoping.

Mine stope shapes were prepared using the Deswik Stope Optimizer (DSO). The DSO parameters used to prepare the stope shapes are provided in Table 15-2.

Table 15-2: Deswik Stope Optimizer Parameters

Parameter	Value
Orientation of DSO (Z Rotation Axis)	-30°
Stope Width Along Strike (Transverse)	20 m
Stope Length Along Strike (Longitudinal / Avoca)	20 m
Stope Height	30 m
Minimum Stope Width HW to FW	3 m
Minimum Stope Dip Angle	55°

15.4 Modifying Factors

Modifying factors include internal dilution, external dilution, paste fill dilution, and mining recovery. These modifying factors are included in the mineral reserve estimate. The following sub-sections outline the modifying factors and the application of the factors to the mine design.

15.4.1 Dilution

Dilution consists of internal dilution, external dilution, and paste fill dilution. The total amount of dilution is estimated at 22.2%.

Internal Dilution

Mining methods such as longhole stoping typically capture material below the cut-off value within each stope. Internal dilution is classified as material below the US \$90.24/tonne NSR cut-off value that is contained within the stope shapes and mined with material above the cut-off value. This internal dilution is estimated at 14.7%.



External Overbreak Dilution

External dilution is material outside the planned stope shape but will be anticipated to overbreak and be mined with the stope. DSO utilized the external dilution factor and established mining shapes for hanging wall and footwall dilution. These shapes are added to the stope shape and the combined tonnage is required to be above the cut-off value.

Table 15-3 provides a summary of the external dilution factors used.

Table 15-3: External Overbreak Dilution Factors

	Dilution (m)
Footwall	0.2
Hanging wall	0.5
Total	0.7

The external dilution estimated is 6.6%.

Paste Fill Wall and Floor Dilution

Transverse secondary stopes will have side wall dilution from backfill as these stopes will be mined adjacent to paste fill walls from the primary stopes. Longitudinal stopes will be mined adjacent to a paste fill end wall from the previous stope and will have end wall dilution. Additional backfill dilution will be derived from mucking the paste fill floor as the stope sequencing progresses bottom-up.

An estimated 1.0% dilution from the paste fill was included in all longitudinal and transverse stopes. A zero grade was assigned to the paste fill dilution.

15.4.2 Mining Recovery

Mining losses account for Mineral Reserves that will be mined but will not be recovered due to losses that occur through the stoping process.

Mining losses in the excavation of the ore development drifts are assumed to be zero, as any unrecovered development ore will be extracted and included as part of the longhole stope.

Mining losses from longhole stopes were estimated based on mining methods and average stope size. Several factors influence mining losses such as mucking line of sight, depth of sight, possible hang ups on the footwall, and blast complications.

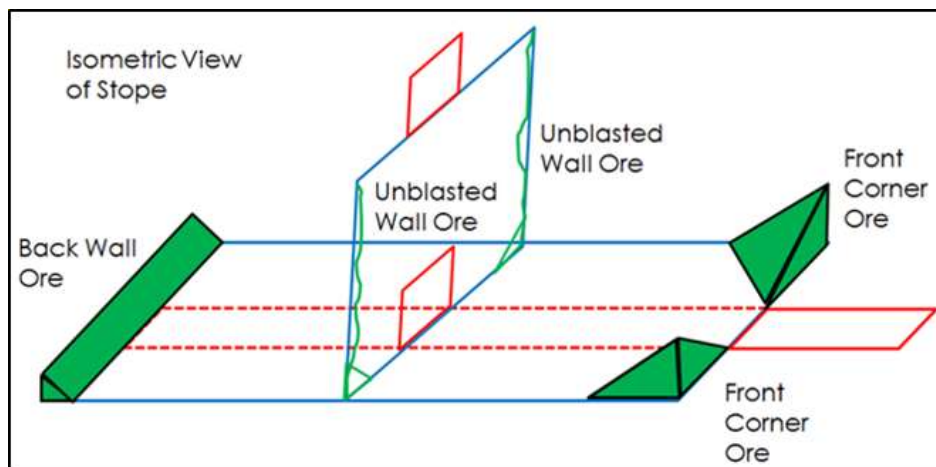
It is expected that some ore that is blasted will not be recovered. Line of sight and maneuverability will prevent the load-haul-dump units (LHDs) from accessing muck in



the front corners of the stope. The maximum angle at which the LHD will be able to operate from the draw point will be approximately 45°. Cleanup at the back of the stope will be difficult to gauge and may result in additional lost ore. Some of the unblasted ore in the side walls may be recoverable with the adjacent stope.

Figure 15-1 presents possible sources of mining losses in a stope.

Figure 15-1: Possible Mining Losses in a Stope



Design is another factor in determining how much ore is recovered from a stope. The designed blasted shape does not necessarily recover all the ore. Restrictions on the design drill and blast pattern may have a slight difference in shape, when compared with the planned stope, to ensure the stope shoulders stay in place for drift re-entry (refer to Figure 15-2).

Figure 15-2: Blasted Stope Outline

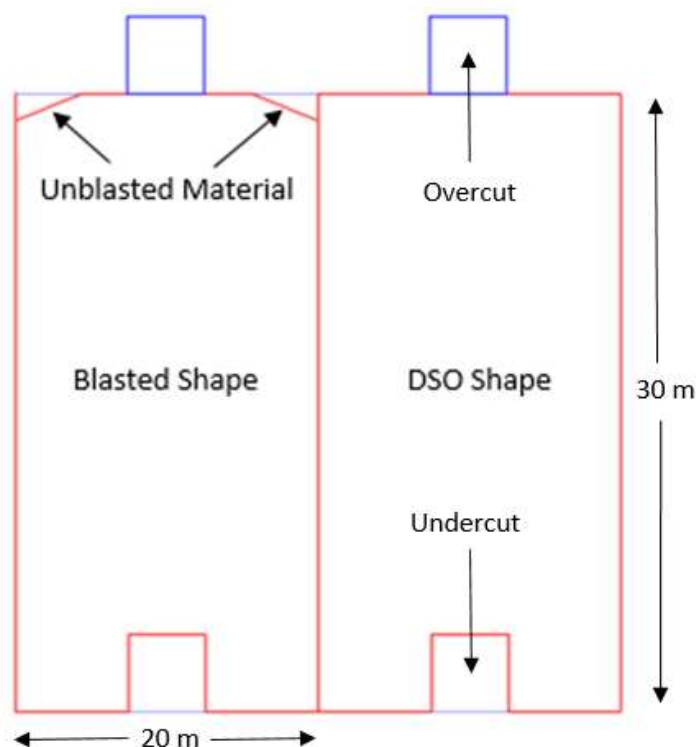


Table 15-4 provides a summary of the recovery factors used in the design.

Table 15-4: Recovery Factors

Mining Method	Recovery Factor (%)
Development (Ore and Marginal)	100.0
Longitudinal Stoping	95.0
Transverse Stoping	95.0
Avoca Stoping	95.0
Uppers < 30 m Height	90.0
Sill Pillar Recovery	85.0
Uppers > 30 m Height	85.0

Mining method will be further defined in Chapter 16.0.

15.5 Cut-off Value

To determine a mineral reserve for Mcllvenna Bay, a preliminary cut-off value using NSR was estimated. The parameters used to estimate the cut-off value were from the Prefeasibility Study (PFS).

The cut-off value for marginal development and incremental stoping excludes sustaining capital costs.

Table 15-5 presents the input parameters for the cut-off value calculation. All values are in Canadian dollars unless otherwise stated.

Table 15-5: Cut-off Value Calculation

Item	Value	Unit
Mine Operating Cost	\$47.37	\$/t
Mill Operating Cost (Processing)	\$22.48	\$/t
Infrastructure Operating Cost	\$3.24	\$/t
General and Administration Operating Costs	\$4.75	\$/t
Tailing Storage Facility Operating Costs	\$2.05	\$/t
Sustaining Capital Cost	\$34.34	\$/t
Cut-off Value \$C	\$114.23	\$/t
Cut-off Value \$US (0.79 exchange rate)	\$90.24	\$/t
Marginal Cut-off Value (No Sustaining Capital) \$C	\$76.65	\$/t
Marginal Cut-off Value \$US (0.79 exchange rate)	\$60.55	\$/t

15.6 Comments on Section 15

Mineral Reserves are reported herein according to the 2014 CIM Definition Standards.

Factors that may affect the Mineral Reserve estimate include the following.

- Commodity price assumptions
- Changes in interpretations of mineralization geometry and continuity of mineralization zones
- Existence of unrecognized fault structures that could affect mine design and underground access, ground control costs, and ore dilution
- Changes to geotechnical, hydrogeological, and metallurgical recovery assumptions
- Input factors used to assess stope dilution and recoveries
- Assumptions that the operation can obtain all required permits to operate
- Assumptions regarding social, permitting, and environmental conditions
- Additional infill or step-out drilling

The QP has reviewed the risks, opportunities, conclusions, and recommendation and is not aware of any conditions that would put the Mineral Reserve at a high risk level.



16.0 MINING METHODS

16.1 Overview

Initial access to the McIlvenna Bay underground (UG) copper zinc deposit will be via a ramp from surface to facilitate an Advanced Development and Exploration Program (ADEX). In Q4 2021 Foran began constructing the portal and will subsequently develop the ramp to the 0090 Level, along with an exploration drift on the 0090 Level. The capital costs for the ADEX program are funded by Foran and are excluded in the feasibility study (FS) financials.

In Year 5 of the project, construction of a 2.43 m × 7.32 m rectangular production and service shaft will begin. The shaft will extend from surface to the 0630 level and equipped with skip and cage conveyances.

There are 35 levels, spaced at 30 m vertical intervals planned for the McIlvenna Bay underground mine. Lateral development during the first four years of the project will be focused on establishing the initial production areas, excavating key UG infrastructure, and establishing the main permanent ventilation system.

The McIlvenna Bay Deposit will be extracted using conventional longhole mining methods including sublevel transverse and longitudinal stoping and Avoca stoping. Stopes will be drilled using a combination of ITH and top hammer drills, blasted using emulsion explosives, and mucked using battery electrical (BEV) load haul dump (LHD) vehicles.

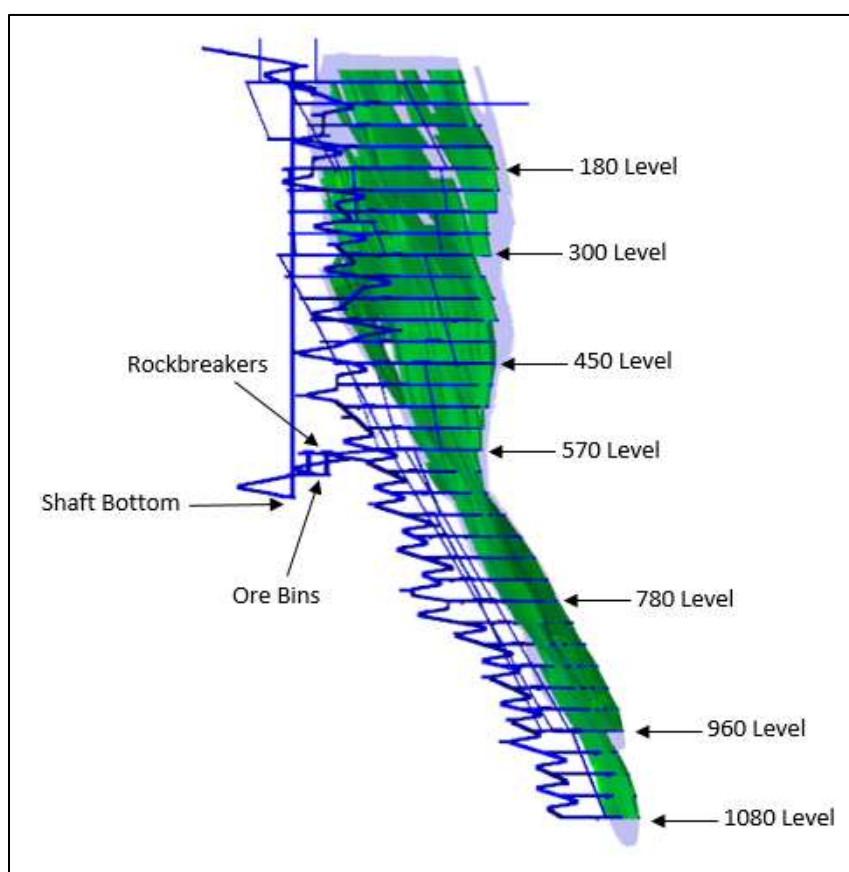
Ore will be hauled to surface using BEV haulage trucks early in the mine life. Once the shaft is commissioned, the BEV trucks will haul to the 0570 Level rock breaker stations feeding the shaft loading pocket. Waste rock will be hauled either to surface or to an active production level for use as stope backfill.

Mine dewatering will be achieved using a multi-level clean water system. Main sumps and pump stations will be located on the 0060, 0420, 0780, and 0960 Levels. The 0060 Level sump will be designed to collect surface run-off water that enters the mine via the ramp as well as most of the water transmitted through the sandstone layer. This sump will feed water into the process water system and pump the surplus water to surface. The 0420, 0780, and 0960 Level sumps will collect any remaining rock mass inflow water, plus the process water used during mining activities. The run-of-mine water will decant from the sumps through membranes. The clean water will be pumped to the next main sump (i.e., 0960 Level to 0780 Level, and 0780 Level to 0420 Level). The 0420 Level sump will feed water into the process water system and pump the surplus water to surface. The residual solids in the sumps will be mucked using an LHD and placed into a nearby stope for disposal.



Transverse and longitudinal longhole stopes will be backfilled with paste fill, using filtered tailings and pyrite concentrate from the processing facility and some secondaries will be filled with unconsolidated development waste. and Avoca stopes will be backfilled with waste rock generated from underground development. Conventional trackless mining equipment will be used to execute lateral development required to access the ore body. The targeted mine production rate is 4,200 tonnes per day (tpd) ore and the mine life will be approximately 20 years, including an initial ramp up period of two years. To achieve this target, the mine plan will include longhole production from multiple mining blocks, with multiple stopes available per block. Figure 16-1 presents a longitudinal projection of the underground mine.

Figure 16-1: Underground Mine Longitudinal Projection (Looking West)



Design criteria and parameters specific to each aspect of the mining method and design are presented in subsequent sections. The following were considered when determining the criteria and parameters during the mine design process:

- Develop and design the mine to achieve regulatory approvals
- Achieve buy-in from local stakeholders
- Health and safety for the workers, local communities, and the environment

- Company standards and specifications (or industry accepted practices where company standards and specifications were not available)
- Prevention through design concepts
- Minimizing risk to production
- Use of proven industry technology, equipment, and processes
- Use of automation to reduce worker exposure
- Operation flexibility
- Minimizing operating costs
- Mineral resource recovery (extraction rate)

16.2 Geotechnical Considerations

16.2.1 Summary

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 have led to the following conclusions:

There are three broad structural domains:

Dolomite Cap

- Sub-horizontal bedding plane joints

Ore Zone

- Foliation parallel jointing – typically 69° dip with 029° dip direction. The orientation of this set is variable in dip and dip direction.

FW zone

- Foliation-parallel jointing – typically 63° to 76° dip with 014° dip direction. The orientation of this set is variable in dip and dip direction.
- A weakly defined set dips of roughly 53° with dip direction of 125°.
- A sub-horizontal set with a dip of 13° to 22° with SSW to SSE dip direction.

Joint surface conditions are most commonly rough and joint surface alteration ranges from clean to coated with softening or clay minerals.

Intact strength and rock mass quality vary by lithological domains.



Table 16-1: Intact Rock Strength and Rock Mass Quality by Domain

Lithology Domain		Q Value		UCS (MPa)
Dolomite		23.4 to 50	Good to Very Good	247
Regolith		2.5 to 9.7	Poor to Fair	36
Volcanics	FV	4.1 to 12.2	Fair to Good	102 (not silicified) to 123 (silicified)
	MV	6.3 to 16.3	Fair to Good	
Ore	MS	18.5 to 37.5	Good to Very Good	193
	CSV	18.5 to 37.5	Good to Very Good	57
Gabbro		6.3 to 22.6	Fair to Good	121
CBM		8.2 to 25	Fair to Good	83
Schist		6.3 to 24.4	Fair to Good	96
FT		6.3 to 16.1	Fair to Good	155
Metasediments		4.2 to 25	Fair to Good	N/A

The only major fault structural feature delineated in the geological model is the Koziol fault which is located roughly 50 m into the HW (minimum 20 m from deposit). Geotechnical logs characterize this feature as a zone of rubble and gouge on the order of 1 m thick; the geology shape of this fault is wider than 1 m and likely captures a larger zone of influence. There is uncertainty as to whether other large-scale faults may exist near the deposit. Low RQD and fault intervals, as logged in the exploration database, indicate other possible faults; it is recommended that efforts be made to develop a fault model.

There is uncertainty in the far-field in situ stress tensor and so sensitivity testing has been conducted in this study. The base case stress tensor is assumed to be the following.

- $\sigma_1 = 2\sigma_v$, orientated horizontal on an azimuth of 160°
- $\sigma_2 = 1.5\sigma_v$, orientated horizontal on an azimuth of 70°
- $\sigma_3 = \sigma_v$, orientated vertically

16.2.2 Mining Method and Sequence

Longhole open stoping is a geotechnically sound mining method for the McIlvenna Project. Longhole stoping will include transverse, longitudinal and Avoca recovery. The general mining sequence in the current plan by mining block consist of the following steps.

1. Transverse stopes are extracted with a flat (horizontal) advance
2. Longitudinal stopes are extracted with a flat (horizontal) central retreat.



At relatively shallow depths, low stress conditions allow for operational flexibility with mine sequencing; the planned global sequencing strategies are geotechnically sound for the McIlvenna Project for mining Blocks 1 and 2. At depth, increasing extraction will lead to elevated stress conditions in some areas of the mine (particularly pillars and abutments); the strengths of the CSV and Schists are expected to contribute to stress-related geotechnical challenges at modest mining depths. The planned global sequence strategy for mining Blocks 5 to 7 is expected to see elevated stress conditions in the following areas.

- Flat diminishing sill pillars
- Diminishing central retreat pillars
- Transverse secondary stope pillars at moderate to high extraction

The risk associated with elevated stress conditions are as follows.

- Exposure of workers to high stress conditions (increased excavation vulnerability to damage and potential for induced seismicity).
- Elevated ground support and rehabilitation costs
- Potential for production delays due to challenging mining conditions

Options for risks mitigation include strategic and tactical means as follows.

Strategic

- Revise global sequencing strategies to allow for center-out retreat in longitudinal sills. This may require delayed transverse stoping, or directional retreat away from the transverse stoping blocks.
- Advance the mining fronts as steeply as possible to avoid flat horizontal advance toward the sill pillars.
- Transverse stope can be sequenced so that the secondary pillars yield, reducing the safety and production risks associated with high stress pillars (e.g., burst conditions where MS is exposed in stope cross cuts, damage accumulation and support loading where CSV is exposed, challenging drilling, production delays). At moderate to high extraction the necessary lag of secondary stopes behind primary stopes should be minimal. As a means of hazard mitigation, a seismic monitoring system can be used to quantify when the secondary pillars are aseismic (i.e., post peak) during operations.
- Avoidance of uphole stoping in sill horizons
- Schedule “just-in-time” development for secondary cross-cuts at depth to minimize rehabilitation costs.



Tactical

- Geotechnical monitoring (ground deformation monitoring in high-risk areas and/or seismic monitoring for mining at depth).
- Scheduling, which accounts for increased production delays at high extraction.
- Localized use of dynamic / yielding ground support (for example, in diminishing central access pillars if longitudinal stoping is not retreated center out).

It is recommended that sequencing adjustments be implemented at this phase of study to largely reduce reliance on tactical strategies.

16.2.3 Stope Design

The stope designs provided to RockEng are generally 20 m long (along strike), the vertical level spacing is 30 m, and stope HW to FW widths vary. The longitudinal stope HW to FW widths are typically in the order of 7 m to 8 m with a maximum of about 19 m. The transverse stope FW to HW spans range from roughly 13 m to a maximum of 25 m.

The performance of HW/FW stope faces will vary with depth and extraction ratio due to induced stresses. Expectations for HW performance are summarized as follows.

- Mining Blocks 1 and 2: cable bolts required rarely at early extraction (~10%), and for roughly one third of stopes at advanced extraction.
- Mining Blocks 3 and 4: cable bolts required rarely at early extraction (~10%), and for roughly half of the stopes at advanced extraction.
- Mining Block 5, 6 and 7: cable bolts required for roughly one third of stopes as early extraction, and for roughly half of stopes at advanced extraction.

Note: advanced extraction refers to secondary stope extraction in transverse areas and the later 50% of production in longitudinal mining areas.

If HW cable bolts are used as described above, stope HWs are expected to see minimal dilution.

Stope FW performance is expected to dictate maximum stope sizing (assuming that FWs are not cable bolted). To manage FW stability, stope sizing adjustments are recommended as follows.

- Mining Block 1 and 2: Current design is adequate
- Mining Block 3 and 4:
 - Primary stopes: Current design is adequate
 - Secondary Stopes: Maximum FW HR = 5.5 m



- Mining Block 5:
 - Primary Stopes: Maximum FW HR = 6 m
 - Secondary Stopes: Maximum FW HR = 5.5 m
- Mining Block 6 and 7:
 - Primary Stopes: Maximum FW HR = 6 m
 - Secondary Stopes: Maximum FW HR = 4.6 m
- If these HR adjustments are made, the cable bolting needs for stope HWs will be reduced.

16.2.4 Pillar Stability

There are 6 types of pillars that must be considered for the Mcllvenna Bay Project: sill pillars, diminishing central retreat pillars, secondary stopes, permanent pillars in the plane of the ore, inter-lens pillars, and the crown pillar.

Sill Pillars

There are six sill pillars in the Mcllvenna Bay mine plan. For each sill pillar, the transverse stopes are planned to be extracted first, followed by retreat of the longitudinal stopes. Recovery of sill pillar 1 is achieved by roughly an even split between longitudinal and transverse stoping methods. Sill pillars 2, 3 and 4 are recovered primarily by transverse stoping. Sill pillars 5 and 6 are recovered primarily by longitudinal stoping. Sill pillar performance will be complex and spatially variable, as conditions will be dictated by HW to FW stope spans, the dominance of MS vs CSV ore types, and extraction ratio.

The first and shallowest sill pillar is not considered a significant mining risk. CSV areas will be exposed to stress-induced damage at very high extraction. Damage accumulation is likely to manifest as spalling accumulation and some ground support loading. Drill hole squeezing may be experienced in CSV. It is reasonable to expect that these hazards can be largely managed by operations, and so production planning will have some flexibility in this sill.

For the remaining sills, numerical model results suggest that challenging conditions will be common at high extraction, particularly due to the low CSV strength. Rock mass damage in sill pillars can result in safety hazards, production delays, and added operational costs. The CSV domain is likely to incur heavy stress-induced damage (spalling, support loading, squeezing blast holes, etc.), and the MS may see localized bursting in some high-stress late extraction conditions. It is recommended that sequencing be adjusted to establish steeper mining fronts (accelerate vertical advance) with center-out or directional retreat. Secondary stopes should advance as close as possible behind the leading primaries. Strategic sequencing is likely to alleviate most stress related hazards during sill extraction at Mcllvenna Bay. Further tactical needs can



then be defined during operations as needed. These may include delayed secondary cross-cut development, local application of heavier support systems, and stress curtains to improve CSV production drilling efficiencies.

Sill pillar performance is largely influenced by HW to FW span. If possible, sill pillars could be positioned at horizons where the ore body is the thinnest. Advancing mining from wide to narrow areas may also be advantageous—for example, advance transverse then retreat longitudinally away from transverse areas.

Diminishing Central Retreat Pillars

Diminishing central retreat pillars are currently planned in mining block 4 and below. Where they exist below the sill pillars, they are expected to see elevated stresses resulting in challenging conditions during the last stope being extracted, necessitating elevated support systems. It is recommended that sequencing be adjusted to establish steeper mining fronts (accelerate vertical advance) with center-out or directional retreat.

Secondary Stopes

Transverse stoping is planned in areas where the ore zone has wider HW to FW thickness compared to the longitudinal areas. Secondary stope pillars generally consist of both MS and CSV (MS forming the HW portion of the stopes), and in some cases the CSV is quite wide forming a larger portion of the secondary stope pillars. Secondary stope performance is expected to be as follows.

- Mining Block 1: No significant stress related risks
- Mining Block 2: No significant stress-related risks at low extraction ratio, minor stress-induced damage in CSV at high extraction ratio.
- Mining Block 3-4: No significant stress related risks at low extraction ratio, minor to high damage accumulation at high extraction ratio in CSV, minor stress related hazard in MS.
- Mining Block 5-7: Minor stress-induced damage at low extraction ratios, moderate to high damage accumulation in CSV at high extraction ratios, localized moderate hazard in MS.

Damage accumulation in CSV may be expressed as sloughage of stope sidewalls during primary stoping (and associated potential for backfill dilution during secondary stoping), challenging production drilling (hole squeezing and cleaning demands), ground support loading and/or elevated support costs and rehabilitations demands. Stress accumulation in MS secondary stopes may result in local bursting hazard in secondary stope cross cuts at very high extraction ratio. At this stage of study, strategic sequencing is recommended to minimize stress-related risks. The lag of secondary stopes behind primary stopes should be minimized, with active secondary stopes being maintained



near mining front abutments, rather than delayed secondary extraction well behind primary stope advance.

Permanent Pillars in the Plane of the Ore

Permanent pillars exist where stopes are not planned for some areas within the plane of the ore. These pillars may be mineralized, but fall below the cut-off value, or may be left out of the mine plan to serve as a barrier pillar between mining areas. The most significant permanent pillar separates the Avoca and Longhole mining areas, where the stability of pillar is considered low risk.

A few additional permanent pillars occur in mining Blocks 5 and 6. Stress flow through these pillars may result in stress loading on the adjacent FW haulage drives. However, this is low exposure risk as these conditions occur during the last stages of mining. Localized stress loading predicted at this phase of study can be later mitigated by strategic sequencing and localized tactical means, including heavy support systems and monitoring / rehabilitation where needed. These pillars should be reviewed as mine planning advances and during operations.

Inter-Lens Pillar

Inter-lens pillars occur locally where HW and FW ore lenses are separated by barren or poorly mineralized ground. If the HW stope is mined first and the inter-lens pillar is narrow, the FW lens is expected to see poor HW performance. For longitudinal stoping areas, the FW lens should advance ahead of the HW lens, while the lag between the two advancing fronts is minimized as much as possible. For transverse stopes, HW to FW retreat is not expected to be problematic where the inter-lens pillar is greater than 20 m. If the inter-lens pillar is less than 20 m, consideration may be given to the following options to improve stope performance:

- Installing HW cables in the FW stope
- Mine the FW stope ahead of the HW stopes (this requires resilling through backfill).
- Revising stope designs to have near vertical walls adjacent to the pillar. This will introduce dilution.

Crown Pillar

The “crown pillar” referred to in this report refers to the basement rock pillar (including regolith) that is to be maintained below the sandstone layer. The sandstone is water-bearing and the regolith is expected to be semi-pervious (hydraulic conductivity between 10^{-4} to 10^{-5} cm/s, Stantec [2021]). The most significant hazard associated with crown pillar instability is perceived to be inflow of sand and/or water into the mine workings (which pose obvious safety and production hazards). For this study, it is assumed that less than 10% probability of failure is acceptable (short-term service life [5-10 years]), if



cemented backfill can be placed tightly after stope extraction to negate longer-term crown pillar deterioration. If tight backfill is not achievable, 5% probability of failure is required (medium-term service life [15-20 years]).

The following recommendations are made for mining in proximity to the crown pillar.

- Mining to the second-to-top planned level (i.e., an additional 30 m of crown pillar thickness) can be achieved with reasonably low risk (<5% PoF), therefore no additional study is required at this time. Stopes are still required to be backfilled. Tight backfill is not critical.
- To achieve extraction of the upper-most level the following will be required for detailed design:
 - Localized drilling will be required to specifically quantify ground conditions in the crown and the local thickness of each domain (host rock, ore, and regolith) as there is variability of crown pillar response based on lithology type.
 - Detailed numerical simulation (i.e., stope by stope extraction) to better predict crown pillar performance to provide better insight on timing of yield propagation and to identify critical crown pillar stopes.
 - Hydrogeology studies to quantify permeability of the host, ore, and regolith unit once yielded. (Establish a monitoring program during operations.)
 - Managing Avoca stope back cumulative HR and stability will be of critical importance.
- During operations it is expected that the following will be required:
 - A top cut will be required to support the stope backs and facilitate tight fill once the ore is extracted (i.e., uphole stoping in the crown pillar is not recommended).
 - Tight fill of each stope (in sequence, or as soon as possible). If cemented backfill cannot be placed tightly, then it is recommended that the upper most mining level not be mined until the end of the mine life.
 - Cemented backfill should be utilized, as uncemented rock fill may consolidate over time which can result in the development of unplanned voids.
 - Monitoring of critical stopes identified during the above mentioned detailed numerical analysis. This can be achieved by borehole extensometers installed through the crown pillar to monitor in situ deformations and achieve early detection of potential stope back instabilities.
 - Monitoring ground water conditions.

16.2.5 Infrastructure Siting

The following points summarize the interpretation and recommendations regarding infrastructure siting:



- The current ramp siting is adequate; it is not expected to be adversely impacted by mine-induced stress changes.
- The shaft siting is adequate; it is not expected to be adversely impacted by mine-induced stress changes. However, the access drives to the ore bins and conveyance drive at shaft bottom are predicted to be damaged by mine-induced stress (see haulage drive summary below). It is recommended that primary access to the shaft be achieved directly off the ramp to entirely mitigate project risks associated with potential damage to access for this critical infrastructure.
- Footwall haulage drives and on-level infrastructure are predicted to be impacted by stress-induced damage at depth. The following are offset adjustments (from the current stope shapes) are recommended for sill pillar horizons:
 - Sill pillar <4: No Change
 - Sill pillar 4: 40 m (+10 m to current plan)
 - Sill pillar 5: 60 m (+35 m to current plan)

These offsets are only required adjacent to the longitudinal stoping areas. The current haulage drive offsets for transverse stoping are adequate based on the current mining sequence. It is noted that these offset requirements may be largely (or entirely) mitigated if strategic sequencing is implemented to alleviate or delay stress loading on FW haulage drives and/or if heavier ground support systems are implemented.

- The following are offset adjustments (from the current stope shapes) are recommended for levels that are not sill pillar horizons:
 - Mining Block <5: No Change
 - Mining Block 6 – level below sill pillar: 30 m (+5 to current plan)
 - Mining Block 6 – remaining haulage drives: No change

These offset requirements may be largely (or entirely) mitigated if strategic sequencing is implemented to alleviate or delay stress loading on FW haulage drives and/or if heavier ground support systems are implemented.

- The vertical development locations are adequate, with the exception to the mining Block 7. Vertical development in mining Block 7 should be offset an additional 15 m (45 m from current stope shapes). This offset requirement may be largely (or entirely) mitigated if strategic sequencing is implemented (i.e., center-out or directional retreat)

16.2.6 Ground Support

Based on empirical and kinematic analyses the three support categories have been defined for the Mcllvenna Bay Project. The distribution of support categories has been



approximated based on the statistical distribution of rock mass quality in each of the relevant spatial domains.

Category 1 ($Q \geq 1$)

- 2.1 m long, 19 mm or #6 Grade 60 or better (assumed ultimate tensile strength of 18 tonnes).
- Screen (#6 gauge welded wire screen) installed with a primary bolting. Consistent with industry best practice, three squares of overlap should be made between screen sheets, with a bolt through the center square.
- Maximum bolt spacing of 1.2 m. Bottom row of rebar should not be higher than 1.5 m off the floor.
- In drifts trending parallel to foliation an additional row of bolts should be installed in the lower wall to mitigate planar failure and squeezing ground risks. This lowest row may be a split set rather than rebar.

Category 2 ($0.2 \leq Q < 1$)

For this category, two options may be considered:

- 75 mm of shotcrete should be installed over the Category 1 bolts and screen. Shotcrete should be brought down the walls to within 0.5 m the floor. To avoid loss of shotcrete in the lower portion of the walls it is recommended that screen and bolts be brought down as low as reasonably possible to the floor (realistically this is likely near 0.5 m).
- If 1.2 m spacing is achieved for Category 1 using a square bolting pattern, a dice bolt may be added to increase bolt pattern density. The lowers row of bolts should be brought down the wall as low as reasonably possible to the floor (realistically this is likely near 0.5 m).

Category 3 ($Q < 0.2$)

For this category, two options may be considered:

- 100 mm of shotcrete should be installed over the Category 1 bolts and screen. Shotcrete should be brought down the walls to within 0.5 m of the floor. To avoid loss of shotcrete in the lower portion of the walls it is recommended that screen and bolts be brought down as low as reasonably possible to the floor (realistically this is likely near 0.5 m).
- 75 mm of shotcrete using a dice bolt within the 1.2 m square bolting pattern. To avoid loss of shotcrete in the lower portion of the walls it is recommended that screen and bolts be brought down as low as reasonably possible to the floor (realistically this is likely near 0.5 m).



Intersections up to 8.5 m

In addition to the standard support listed above in Categories 1 to 3, 3 m spin cables or swellex are recommended on a 2.0 x 2.0 m pattern in intersections up to 8.5 m span. Any spans beyond 8.5 m can assume cable bolt length equal to half the span for costing study purposes, with detailed design conducted later as the project advances to detailed engineering and construction. As an alternative to spin cables, single strand cable bolts can be utilized.

In Stope Cable Bolting

It is recommended that 6 m long, double-strand cables be installed on a 25 by 2.5 m pattern. Frequencies are provided in RockEng Report 21016-107. For costing studies MS performance should be applied only where stopes are expected to be >2/3 MS. Otherwise, CSV predictions should be utilized. It is reasonable to expect that during operations there will be mixed cable bolting strategies, once ground reaction can be observed and quantified (e.g., only cable bolting the CSV portion of larger stopes).

Stope HWs should be supported by 7.5 m double-strand cables bolts installed from stope top and bottom cuts, fanned into the HW with 2 m x 2 m toe spacing.

Draw Point Support

The draw point support should consist of standard ground support with three rings of 4 m long spin cables or 25 tonne inflatable bolts with two holes per ring across the drift back.

Vertical Development

Escapeways will be developed between levels. It is assumed that these will be constructed by Alimak raise method (3 m x 3 m profile). Ground support estimates for escapeway development are recommended as follows: 1.2 m rebar bolts with screen, two bolts per face per ring with 0.9 m ring spacing.

16.2.7 Behavior of Large-Scale Geological Structures

The Koziol fault may be susceptible to fault-slip at depth (mining Block 6) where the fault offset from the stoping HW diminishes. This could generate large magnitude seismic events. Due to the relatively large distance between the fault and the mining areas (typically in the order of >70 m for FW development and the ramp) the risks associated with fault-slip seismicity is considered low at this time. Standard ground support in good condition is suitable to retain “shake-down.”

The Koziol fault may be water-bearing; no hydrological studies have intersected the fault. Numerical models indicate that the extent of yield will intersect the fault near the crown pillar, which may lead to increased water flows to mine workings. It is



recommended that future hydrological studies should consider the impact of mine-induced stresses on the hydraulic conductivity.

16.2.8 Backfill

Vertical Wall Stability

Paste backfill strength for vertical wall exposures varies depending on HW to FW span as summarized in Table 16-2.

Table 16-2: Backfill Strengths Required for Varying Spans and FS.

HW to FW span (m)	UCS (kPa)
5	139
10	243
15	324
20	389
25	442

Plug Strength

The minimum plug strength required to mitigate risk of barricade failure during the main pour is 186 kPa.

Cap Strength

The minimum 3 m cap strength required for tramming over paste is 300 kPa. An additional 30 cm of road bed or muck should be placed on top of the cemented paste backfill.

Undercutting Strength

Paste backfill strength for undercut exposures (i.e., sill pillar horizons) varies depending on HW to FW span as summarized in Table 16-3.



Table 16-3: Summary of Analytical and Empirical Design for Sill Spans. Safety Factor: 1.5

Sill thickness (m)	Span (m)	Required UCS (kPa)
5.5	5	1400
5.5	11	4900
7.5	15	5800
10	20	6500
12.5	25	6900
30	10	500
30	15	775* - 975**
30	20	1000* - 1500**
30	25	1350* - 1500**

Note: *based on HW dip of 65°, ** based on HW dip of 90°

Mining Through Paste

The minimum strength required to mine through backfill is 0.8 MPa, based on empirical design.

16.3 Hydrogeological Considerations

A hydrogeological model and analysis were completed for the McIlvenna Bay Project pre-feasibility study (PFS). As part of the FS, Stantec subsequently rebuilt the model incorporating new data collected since the PFS model construction. More refined shallow geological layering and geological features, such as faulting, were also incorporated in the FS model.

The understanding of site geology has not changed significantly since the construction of the PFS hydrogeological model. The 2021 geologic drilling activities onsite have confirmed that no major changes in the interpretation of stratigraphy or faulting have occurred. Therefore, this FS report will not reproduce the geological setting descriptions which were included in the PFS modeling report. Three 2021 geologic drill holes and one previously drilled geologic drill hole were selected for 2021 hydrogeologic testing of select geologic units. The FS hydrogeological model will focus on new data obtained since the PFS modeling and will outline how this data was incorporated into the FS groundwater modeling.

16.4 Model Domain and Layering

During the FS modeling, several refinements were made to the model to improve representation of the mine activities and surrounding geology. The refinements are provided in Table 16-4 for comparison between the PFS and FS models.

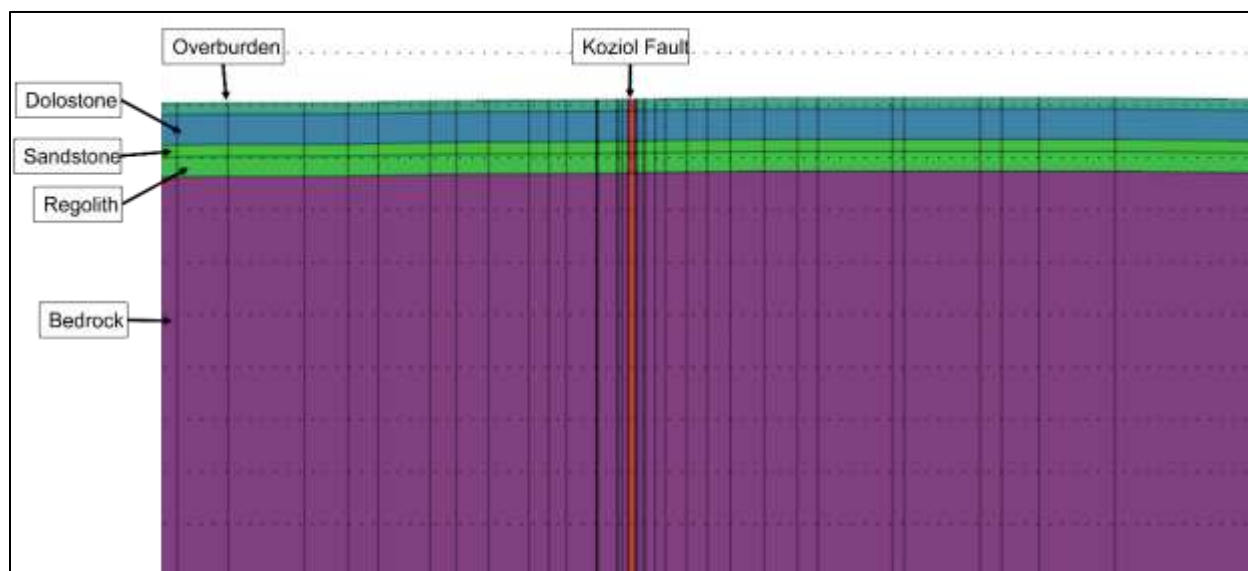


Table 16-4: Comparison of 2019 and 2021 Model Features

Model	Layers	Lithology	Mesh Type	Faulting	Anisotropy (Horizontal and Vertical)
2019 PFS Model	2	Overburden Bedrock	Structured Unstructured	No	No
2021 FS Model	5	Overburden Dolostone Sandstone Regolith Bedrock	Structured for all layers	Yes (Koziol fault)	Yes

The same model domain was used for the FS as in the PFS modeling as it encompassed a large enough area outside the mine to avoid boundary condition influences and included surface water bodies that favored boundary condition assignments. The two-layer PFS model was refined vertically to consist of five layers in the FS model: Overburden, Dolostone, Sandstone, Regolith, and Bedrock (Figure 16-3). This refinement was completed to better represent the local shallow geology. Starting with the PFS mesh, a structured mesh was applied to all layers; FEFLOW layer types were assigned with Layer 1 as Phreatic, Layers 2-4 as Dependent, and Layer 5 as Confined. A minimum thickness of 0.1 m was used for all layers, and the Koziol fault was simulated cutting across all the layers in the model (Figure 16-2). The fault location was determined from a 2018 Leapfrog model obtained from Rock Eng. Zones of higher hydraulic conductivity were used to represent the fault, and finer mesh discretization was conducted along the fault line to increase model precision over this area.

Figure 16-2: Model Layers and Fault Location



The data used to construct model surfaces for the mine project area, and areas more distant from the mine, originated from different sources. The topographic and formation surfaces in the mine project area were constructed using DXF files from the 2018 Leapfrog model. The topographic surface for the remaining model domain was constructed using DEM data obtained from the Government of Canada website. The DXF files from the mine project area were used to calculate the average thickness for each lithologic unit, and in areas where there were no formation data, the topographic DEM was projected downwards using the formation thicknesses to represent the subsurface stratigraphic. The lithologic data in the mine project area and the adjusted DEM was then stitched together to establish the model surfaces.

16.4.1 Predictive Simulations

The FS model simulation predicted a total of approximately 2,000 L/min of steady-state inflow into the mine workings, which is in general agreement with the value predicted by the PFS model (2,394 L/min). A sensitivity analysis was conducted by raising and lowering all model K values a half-order of magnitude, which resulted in an inflow range of approximately 450 L/min to 9,950 L/min. A second analysis was conducted by raising and lowering all K values except the Koziol fault, which resulted in a narrower inflow range of approximately 1,050 L/min to 5,500 L/min. A third analysis was conducted by only varying the Sandstone conductivity, which resulted in a narrower inflow range of approximately 2,000 L/min to 2,100 L/min. The sensitivity results demonstrate the inflow is most sensitive to the fault conductivity, for which there is no testing data.

16.4.2 Limitations and Recommendations

When geological surfaces from two sources are stitched together, interpolation artifacts at the stitching boundary are established. As such, the accuracy of the geologic interpretation at the stitching boundary may be limited. Additionally, the formation surfaces outside of the mine project area were projected based on the best available isopach data. Consequently, the model should not be relied upon to make detailed predictions outside of the mine project area or Leapfrog geologic model domain.

While the PFS model was updated with geological surfaces in the mine project area, the surfaces were prepared from 2018 data. As such, new drillhole and geotechnical data obtained since 2018 were not reflected in the Leapfrog model nor the FS model. An update to the 2018 Leapfrog model to incorporate new data would assist in increasing the accuracy and representativeness of the geological surfaces.

The quantity and seasonality of field-obtained hydrogeological property values were limited. Accordingly, only steady-state calibration of the model was attempted. Formation properties used in the model were determined from a limited amount of field data, and in some cases publicly available literature. Of particular note was the lack of any testing data associated with the Koziol fault. As noted from the sensitivity analysis



results, the mine inflows are quite sensitive to the modeled K values of the fault. A field-testing program targeting all lithologies, with longer testing durations, would increase the overall quality of data, and aid in transient model calibration.

16.5 Mine Design and Method

16.5.1 Lateral Development

All decline and lateral excavations will be developed using drill and blast methods. The mobile equipment required for development activities is as follows:

- Drill – 2-boom electric-hydraulic jumbo
- Blast – mobile explosives loader
- Muck – 18-tonne class LHD (BEV)
- Ground Support Installation – 2-boom electric-hydraulic jumbo
- Secondary Haulage –40 tonne underground haulage truck (BEV)

Lateral development will consist of all level access, footwall drives, ore sills, ramps, and excavations for infrastructure. The main development heading profiles for the underground workings are presented in Table 16-5.

Table 16-5: Main Development Heading Profiles

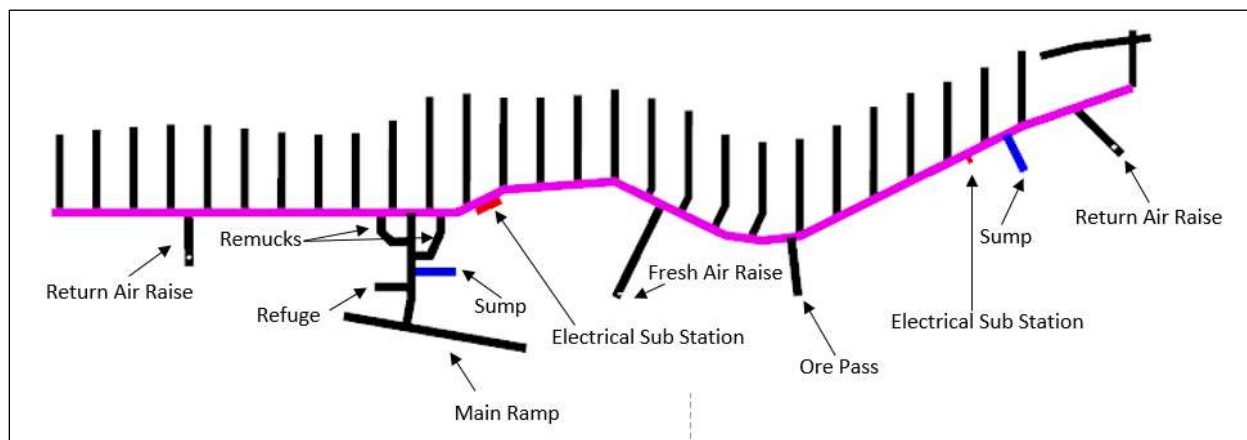
Heading Type	Width (m)	Height (m)
Footwall (FW) Drive (no truck access)	5.0	5.0
FW Drive (truck access)	5.0	5.5
Level Access	5.0	5.5
Ramp	5.0	5.5
Crosscuts and Sills	4.5	4.5
Infrastructure	varies	varies

16.5.2 Level Development

Levels will be developed at 30 m vertical intervals. Levels will have varying strike lengths, depending on the mineral resource at each elevation. A typical level is represented by the 0540 Level example in Figure 16-3.



Figure 16-3: Typical Level Development



16.5.3 Development Quantities

The 3D mine model includes ramp, level, and infrastructure development required to access and extract the reserves. Table 16-6 presents the lateral development quantities by excavation type. An allowance was added to the lateral waste development quantities to account for slashes, corners, and take-down backs.

Table 16-6: Lateral Development Quantities

Item	Meters
FW Drive (no truck access)	19,127
FW Drive (truck access)	4,421
Level Access	2,603
Ramp	7,077
Crosscuts and Sills	48,993
Infrastructure	11,075
Total	93,297

16.5.4 Vertical Development

Vertical development will consist of the shaft, ventilation raises, an ore pass, and bins. Return air raises will be excavated primarily using raise boring methods completed by a qualified contractor. Shorter exhaust raises may be developed via drop raise method. Fresh air raises will be excavated using an Alimak method to allow for installation of ground support and equipping of second egress. Ore pass and shaft development will also be excavated using an Alimak method. Fingers will be developed via drop raise method. Table 16-7 presents the vertical development quantities by excavation type.

Table 16-7: Vertical Development Quantities

Item	Meters
Ore Pass with Fingers	117
Ore Bins	49
Return Air Raises	1,958
Fresh Air Raises	1,129
Shaft (excluding collar)	600
Total	3,853

16.5.5 Production Stoping

For production mining, a combination of transverse and longitudinal longhole stoping approaches will be used for most of the ore. Avoca mining will be used.

Longhole stoping requires dividing the targeted mineral resource into individual stopes and establishing mining levels to access the stopes and position development to facilitate drilling, blasting, and extraction of the material. Once extraction of material within a stope is completed, the stope will be filled with either paste backfill or rockfill. Longhole is a non-entry mining method (i.e., during mining, personnel will be prohibited from entering the open portion of a stope). Material from within the stope is mucked remotely using LHDs to reduce worker exposure to potential ground hazards.

Table 16-8 presents the recovered ore tonnes by mining method. A significant portion of the mineral resource will be extracted via the transverse approach.

Table 16-8: Ore Tonnes by Mining Method

Mining Method	Recovered Ore Tonnes (million tonnes)
Transverse Longhole	17.47
Longitudinal Longhole	5.95
Avoca	0.63
Development	1.16
Marginal Development*	0.49
Total	25.70

*For definition of marginal development refer to 16.7.2.

Mining Blocks

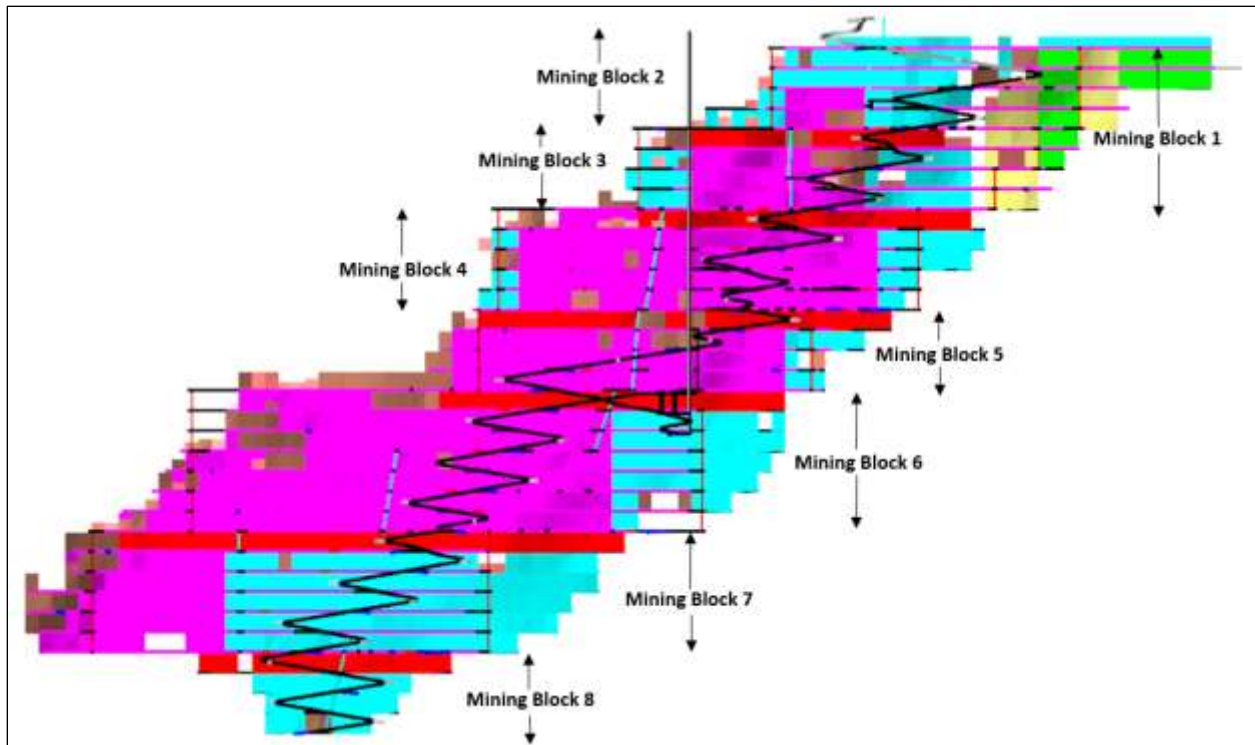
To achieve the planned production rate and allow flexibility in the schedule, production will be required concurrently from multiple mining blocks. Each mining front will be mined using the bottom-up approach.



The main vertical mining blocks are as follows and represented in Figure 16-4.

- Mining Block 1 (Avoca): 300 Level to 60 Level
- Mining Block 2: 180 Level to 60 Level
- Mining Block 3: 300 Level to 180 Level
- Mining Block 4: 450 Level to 300 Level
- Mining Block 5: 570 Level to 450 Level
- Mining Block 6: 780 Level to 570 Level
- Mining Block 7: 960 Level to 780 Level
- Mining Block 8: 1080 Level to 960 Level

Figure 16-4: Mining Blocks



Transverse Longhole Stopping

For transverse longhole stopping, a drift will be established in the FW and each stope will be accessed from the FW drift with a crosscut developed through the centre of the stope. Primary and secondary stopes will be defined at 20 m wide intervals along strike.

Longitudinal Longhole Stopping

Longitudinal stopes will generally be located at the end of the levels where the ore thickness is narrower. Access will be from the FW drift with a sill drift developed along the strike of the ore body. Stopping will start at the end of each sill and retreat toward the access. Each stope will be 20 m to 40m long. Once extracted, each stope will be paste-filled prior to mining the adjacent stope.

Avoca Stopping

The Avoca stopes will be accessed at each end from the FW drift. From the accesses, a sill drift connecting the two access will be developed along the strike of the ore body. As stopes are extracted from the access at one end of stope, rockfill will be placed into the open stope from the access at the opposite end of the sill.

16.6 Productivities

16.6.1 Effective Hours

The UG operations will consist of two 12-hour shifts per day, seven days per week. Time worked during these 12-hour shifts is expected to fall into one of two categories: effective or non-effective.

Non-effective time may result from necessary parts of the job (e.g., travel to the workplace, equipment refueling, and shift safety meetings) during which workers may not be actively conducting mining work (i.e., effective time).

The effective time calculation presented in Table 16-9 represents the available time underground that a worker will be able to perform tasks.

Table 16-9: Estimated Worker Effective Time per Shift

Activity	Time	Unit
Start of shift Lineup	15.0	min
Travel to workplace	38.0	min
Work site Assessment and Set-Up	5.0	min
Lunch	45.0	min
Breaks	30.0	min
Weekly Safety meeting (30 minutes per week)	2.1	min
Blasting Delays (60 minutes per week)	8.6	min
Supervisor Visits	10.0	min
Outside Interference (Engineer / Surveyor / etc. – 15 minutes per week)	4.3	min
Pre-shift workplace assessment	10.0	min



Activity	Time	Unit
Travel from workplace	33.0	min
Total Non-Effective Shift Time	191.0	min
Total Non-Effective Shift Time	3.2	hr
Total Shift Length	12	hr
Total Effective Shift Time	8.8	hr

To offset some of the non-effective time, certain mobile equipment with remote and/or automation capabilities will be included in the fleet to allow for underground activities during shift change and other non-effective time.

The automated equipment in the fleet includes the following.

- LHDs (tele-remote) – to muck from a draw point to a remuck, or to load haulage trucks
- Jumbos (automated) – facing drilling
- Longhole drills (single hole automated) – production drilling

This results in approximately 1.5 hours per shift increase in effective work time for mining activities related to these pieces of equipment.

For the equipment to operate safely on tele-remote or automated, controls will be placed at or near the workplace to ensure personnel will not enter the work area. If controls are by-passed the equipment will be shut down automatically.

16.6.2 Labour

The majority of the UG excavations and construction will be completed by Foran personnel. Contractors will be used for the initial year of development and construction in Year -2, and for ongoing raising activities throughout the life-of-mine. The shaft program will be completed by a contractor, which is scheduled to start in the Year 5.

The UG labour will peak at 149 personnel on-site. The total UG related labour (peak) by year is presented in Table 16-10. The UG labour is split between contractor and Foran personnel, separated into the following four main groups.

- Management
- Technical Services
- Mine Operations
- Mine Maintenance



Table 16-10: Total On-site Labour – Underground Only

Labour Type	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18
Contractor																				
Mine / Contract Management	4	0	3	3	0	0	3	3	3	0	0	0	0	0	0	0	0	0	0	0
Technical Services	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Operations	65	6	7	12	7	7	14	14	7	7	7	7	0	0	0	0	0	0	0	0
Mine Maintenance	12	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Foran Underground Labour																				
Mine Management	5	6	6	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	6	5
Technical Services	2	11	17	17	17	17	17	17	17	17	17	17	17	17	17	17	17	11	10	10
Mine Operations	1	71	82	82	90	87	79	75	73	73	73	73	65	67	67	67	67	49	41	29
Mine Maintenance	6	25	28	28	28	29	29	27	26	26	26	26	26	26	26	26	26	23	15	12
Total - On Site	96	119	143	149	149	147	149	143	133	130	130	130	115	117	117	117	117	90	72	56

Note: Numbers may not sum due to rounding.



16.6.3 Development

The lateral development advance rates were divided into the components of the drill-blast-muck-bolt cycle and estimated from first principles. Advance rates were developed for single and multiple heading scenarios. The rates reflect the advance each drill jumbo crew and its associated gear will achieve over extended periods of operation. The rates reflect long-term averages and include an efficiency allowance. The efficiency allowance accounts for interferences from other activities and conflicting priorities that occur during the operating period.

Table 16-11 presents the lateral development blended (multiple heading) advance rates for each heading type.

Table 16-11: Lateral Development Advance Rates

Heading Type	Multiple Heading (m/day)
Ramp (5.0 m W × 5.5 m H)	10.1
Lateral (5.0 m W × 5.5 m H)	10.1
Footwall (5.0 m W × 5.0 m H)	10.4
Cross-cut (4.5 m W × 4.5 m H)	712.1
Ore (4.5 m W × 4.5 m H)	11.7

The multiple heading rates for a mining complex varies, based on the productivities of the drill jumbos at 9.8 m/day and the number of available jumbos and faces. The quantity of remaining equipment required to support lateral development was calculated based on the total advance rate and the productivity of each piece of equipment.

The vertical development advance rate using the raise boring method will be approximately 2.2 m/day for a 3.0 m diameter raise. This advance rate includes setup and teardown (15 to 16 days) as well as pilot hole drilling (25 m/day) and reaming of the raise (10 m/day).

The vertical development advance rate using the Alimak method will be approximately 3.7 m/day for a raise with a 3.0 m × 3.0 m profile. This includes setup, drilling, blasting, mucking, bolting, installation of rails, and teardown.

16.6.4 Stope Productivity

Stope production rates were divided into the components of the drill-blast-muck (DBM) and backfill cycle and estimated from first principles. The time for each element of the DBM cycle for an average transverse stope (i.e., 20 m wide × 12.2 m long × 30 m high) is presented in Table 16-12. The 44.5 day cycle time averages an estimated 503 t/d for a typical 22,400 t stope.



Table 16-12: DBM Cycle of a Typical Transverse Stope

Item	Time (days)
Stope Preparation	2.0
Drilling	13.0
Blasting	2.5
Mucking	8.3
Backfill Barricade Installation	1.7
Backfilling (includes pour and cure, including the plug)	17.0
Total Cycle Time	44.5

Stoping DBM productivities were divided into the three main mining methods: transverse, longitudinal, and Avoca, with representative stope sizes and average productivities presented in Table 16-13.

Table 16-13: DBM Cycle for Representative Stope Sizes

Method	Stope Dimensions			Tonnes (excludes sill)	Cycle Time (Days)	Tonnes per Day
	Length (m)	Width (m)	Height (m)			
Transverse	12.2	20	30	22,400	44.5	503
Longitudinal	40	5.6	30	18,600	52.5	354
Avoca ¹	225	4.5	30	82,000	98.0	838

¹ Average stope size for entire level, include critical filling only.

16.7 Mine Production Schedule

16.7.1 Development Scheduling

All mine development and production scheduling has been prepared using Deswik scheduling software. The schedule is interactively linked to the 3D mine model. All development and production scheduled is based on dependencies linked within the mine model. All data are contained within the mine model and schedule.

The primary development focus will be to develop the ramp and establish the initial sills for each mining block. On each level, priority will be to establish the ventilation system as well as initial infrastructure, and ore development.

The total lateral development is separated into Capital and Operating and summarized annually in Table 16-14, and summarized by development type in

Table 16-15.



Table 16-14: Summary of Annual Lateral Development

Development Type	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8
Lateral Meters (Operating)	539	3,948	3,803	3,523	3,886	2,874	2,490	1,586	2,477	2,146
Lateral Meters Capital	2,675	6,069	5,181	4,962	4,694	5,409	3,474	4,038	3,201	3,472
Development Type	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18
Lateral Meters (Operating)	1,708	2,239	1,504	1,920	1,883	2,273	419	0	0	0
Lateral Meters Capital	3,396	2,401	1,668	1,330	1,340	767	0	0	0	0

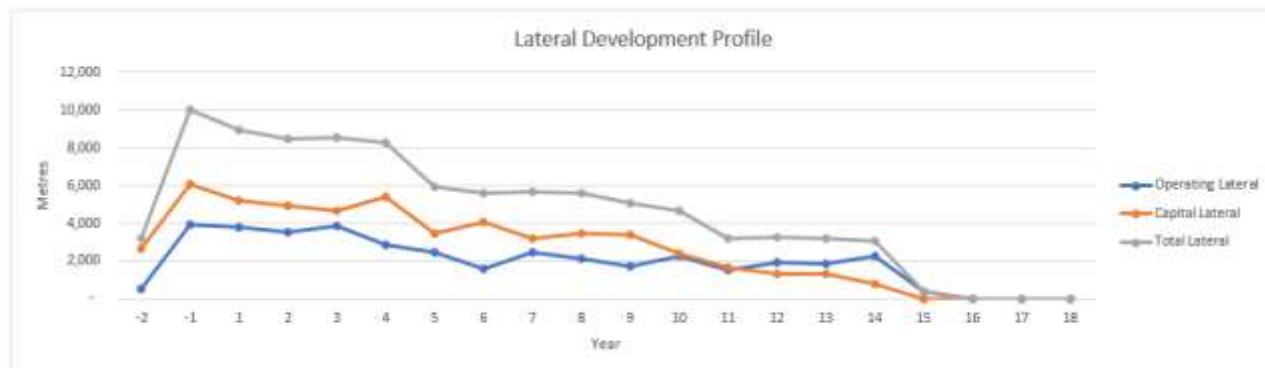
Table 16-15: Annual Lateral Development by Heading Type

Development Type	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8
FW Drive (no truck access)	349	1,420	1,241	1,996	1,883	1,947	1,195	2,098	1,057	1,235
FW Drive (truck access)	797	1,849	784	144	139	96	136	91	48	142
Level Access	302	316	380	13	280	131	224	130	113	263
Ramp	806	781	96	260	629	553	761	166	454	705
Crosscuts and Sills	580	4,435	4,528	4,752	50,48	3,732	3,122	2,593	3,196	2,897
Infrastructure	380	1,217	1,085	1,320	607	1,825	527	547	810	376
Development Type	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18
FW Drive (no truck access)	2005	53	902	687	880	180				
FW Drive (truck access)	50		101	44						
Level Access	62	361	28							
Ramp	183	812								
Crosscuts and Sills	2295	2,919	1,648	2,249	2,160	2,422	419			
Infrastructure	510	495	493	269	182	438				



Figure 16-5 presents a graph of the LOM development schedule.

Figure 16-5: Lateral Development Profile



16.7.2 Production Scheduling

The production rate for the Mcllvenna Bay Deposit is 4,200 tonnes per day. The production plan will focus on optimizing ramp-up and maximizing productivity. In the area of transverse stopes, a primary / secondary stope sequence will be used to maximize recovery. Primary stopes will be recovered first on the initial level, followed by primary stopes on the next vertical level above. Afterwards, secondary stope recovery will start on the initial level while still advancing primary stopes vertically.

Consideration of Marginal Cut-off Value

Marginal material is classified as either marginal development or incremental stoping. Marginal material is below the planned cut-off value but is economic to process. The cut-off value calculation for marginal material does not include sustaining capital or infrastructure costs and is US \$60.55/tonne NSR.

Marginal development occurs in operating crosscut and sill development. The majority of marginal development tonnes will be processed at the end of the mine life; however, 66,000 tonnes will be processed in Year -1 since the mill has capacity.

Incremental stoping utilizes the development that is planned to mine the above cut-off value material and occurs in the FW of stopes which have already been extracted. Incremental stoping will be taken independently and mined and milled in the last three years of the mine life.

16.7.3 Production Profile

Table 16-16, Figure 16-6 and Figure 16-7 present the life-of-mine production tonnes and profile.



Table 16-16: Life-of-mine Production Tonnes

Source	YR -2	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8
Development Ore Tonnes	10,970	126,178	104,832	71,706	73,926	84,061	60,544	49,393	75,753	71,094
Stoping Ore Tonnes	0	474,326	1,275,314	1,440,294	1,438,074	1,427,939	1,451,456	1,462,607	1,436,356	1,440,898
Incremental Stoping Ore Tonnes	0	0	0	0	0	0	0	0	0	0
Marginal Development Tonnes	14,954	51,073	59,621	52,844	47,925	46,093	19,323	16,328	23,659	16,748
Source	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18
Development Ore Tonnes	69,452	65,899	62,217	52,259	79,587	86,633	19,341	0	0	0
Stoping Ore Tonnes	1,442,548	1,446,101	1,449,783	1,459,741	1,432,413	1,425,367	1,379,568	875,665	78,501	
Incremental Stoping Ore Tonnes	0	0	0	0	0	0	113,091	217,385	702,793	178,972
Marginal Development Tonnes	15,045	23,918	8,558	23,585	20,005	43,991	6,354	0	0	0



Figure 16-6: Life-of-mine Production Profile – Ore Type

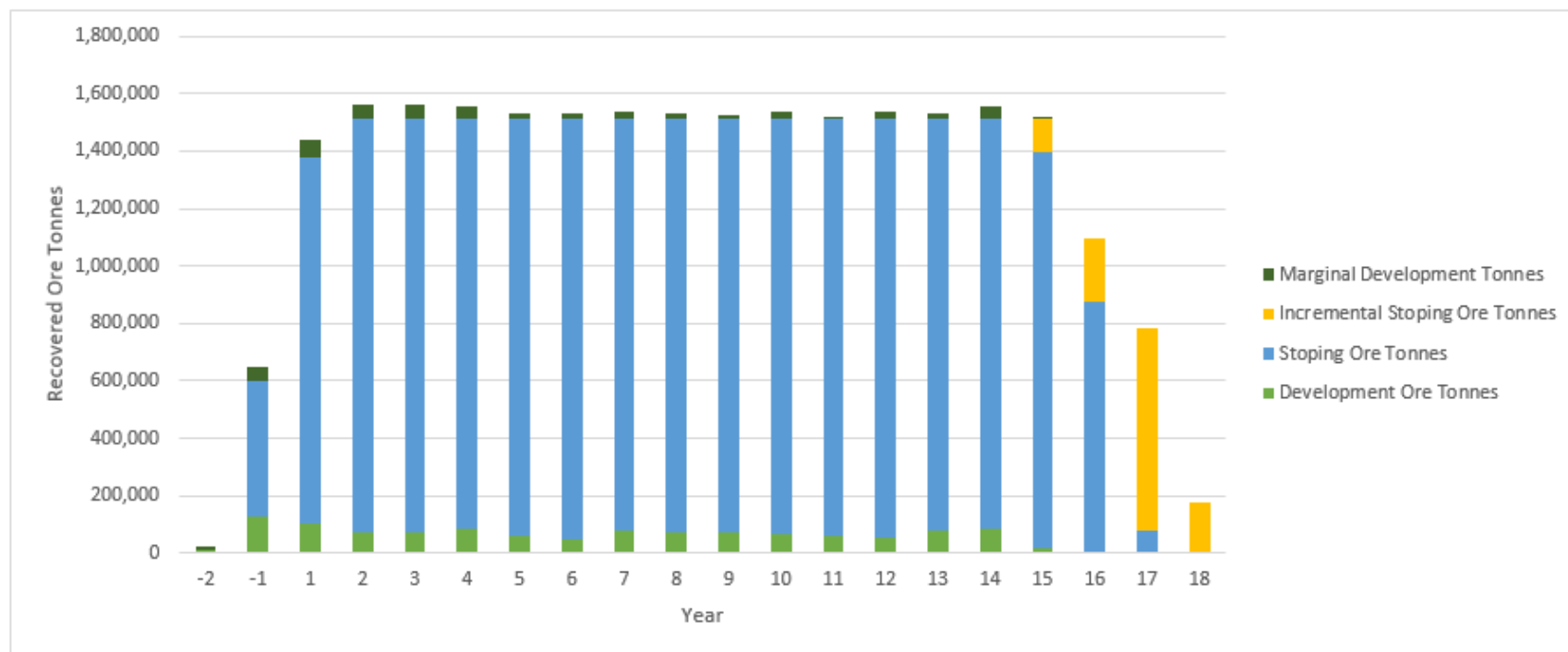
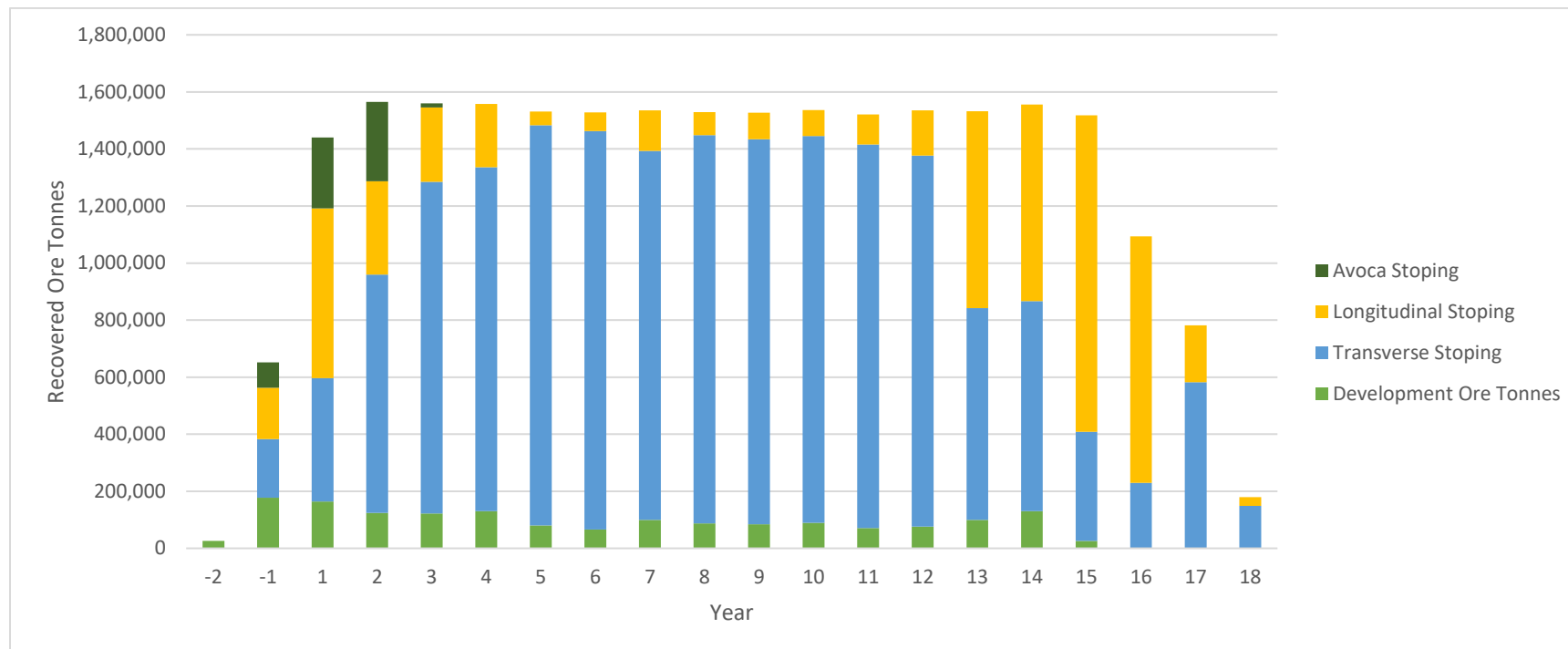


Figure 16-7: Life-of-mine Production Profile – Mining Method



16.8 Mobile Equipment Selection

The McIlvenna Bay Project will be developed using a high degree of automated equipment. The main production equipment will be capable of remote operation, and in some cases, autonomous operation. Mobile equipment was selected that is appropriate for the mining methods and to support the production rate. The automation backbone will not be fully installed during the project period (Years -2 and -1), but will be available during the production period (Year 1 onwards).

Equipment fleet costs and performance are supported by information from the original equipment manufacturers (OEMs) and only commercially available products were considered. It is noted, however, that further electrification should be considered as OEM's demonstrate the efficacy of new equipment in the coming months and years. The mine development contractor will use and maintain equipment provided by Foran and will return the equipment to Foran in proper working condition.

Key elements considered in the ultimate equipment selection will include:

- mine design
- mining methods
- productivities
- fleet standardization of key components
- maintainability
- economics

The equipment fleet includes the main development and production gear as well as auxiliary gear to support operations.

Following the completion of the ADEX Program (Year -2), there will be a single development crew on the 0090 Level until there are sufficient faces available to start the second crew. A third development crew will start shortly thereafter. Three development crews will be required until Year 5, then reducing to two crews until Year 11. After Year 11 one crew will be sufficient to maintain the lateral development requirements to support mining operations.

Each development crew will have a two-boom drill jumbo for face drilling and installing ground support, access to a shared ANFO loader for development headings, an LHD to muck the face and load trucks, and a scissor lift for installing services. The development crew will be supported by the haulage truck fleet.

Vertical development will be a combination of Alimak and raise boring and will be completed by contractors providing the equipment as required. The raising equipment is excluded from the equipment list.



Production will be largely completed by variations of longhole mining. Longhole ITH and Tophammer drills will be used to drill the blastholes, the blastholes will be loaded using a shared fleet of emulsion loaders, and an LHD will muck the stope to the nearest remuck or to a waiting truck.

The fleet includes two mechanical bolters for cable bolt drilling and other secondary ground support installation when required.

There will be approximately 53 pieces of equipment operating in the mine during peak production years, which includes 7 haulage trucks. This will occur prior to the shaft being operational, at which point the number of haulage trucks will reduce to a low of four trucks. As the mine progresses deeper in the later years, a fifth haulage truck will be required.

The mobile equipment fleet operating each year is summarized in Table 16-17 (production and development fleet) and Table 16-18 (auxiliary fleet).



Table 16-17: Production / Development Mobile Equipment Fleet

Equipment Type	Year																			
	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18
Production / Development Fleet	17	27	30	30	31	31	29	26	25	25	25	25	23	24	23	23	23	16	11	7
Two Boom Dro; Jumbo	3	3	3	3	3	3	2	2	2	2	2	2	1	1	1	1	1	0	0	0
BEV LHD's - Development	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
BEV LHD's - Production	1	3	4	5	4	4	4	4	4	4	4	4	4	4	4	4	4	3	2	1
Scissor Lift	3	3	3	3	3	3	2	2	2	2	2	2	1	1	1	1	1	1	0	0
Production Drill	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	1
ANFO Trucks	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Emulsion Loader	0	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Transmixer	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Shotcrete Sprayer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor Lift	1	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1	0	0	0
UG Forklifts	0	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
BEV 40 T UG Haulage Trucks	3	4	6	6	7	7	7	5	4	4	4	4	4	5	5	5	5	3	2	1



Table 16-18: Auxiliary Mobile Equipment Fleet

Equipment Type	Year																			
	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18
Auxiliary Equipment Fleet	12	18	22	22	22	22	21	21	21	21	21	21	20	20	20	20	20	18	14	11
Mechanical Bolter	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Block Holer	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
BEV LHD's - Utility	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Fuel Delivery Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Personnel Transport Vehicles	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	1
Supervisor's/Engineers Trucks	1	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	3
UG Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Leaders Truck with Backhoe	3	3	3	3	3	3	2	2	2	2	2	2	1	1	1	1	1	0	0	0
Mechanics Truck	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Electricians Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total UG Mobile Equipment	29	45	52	52	53	53	50	47	46	46	46	46	43	44	43	43	43	34	25	18



Equipment operating hours have been estimated from first principles. The requirement for equipment specific to development or production will fluctuate with changes to annual throughput, while some equipment will operate at a certain rate regardless of the tonnage or meters being developed.

A mobile equipment rebuild and replacement schedule was developed based on the operating hours required and the equipment available.

16.9 Ventilation

The underground mine ventilation system is designed to be a “push-pull” system, with the exhaust fans being the main driving fans and the intake fans used to pull fresh air through the heaters and into the intake raise. The portal will be upcasting with heated air to prevent freezing. The production shaft will also be upcasted to prevent freezing and to allow for safe air quality to travel within. The upcast ventilation will be provided through fans installed at the airlock doors underground to access the shaft on 620 Level. The ventilation system is designed to support a hybrid diesel and BEV fleet, with the main loading and haulage equipment being BEV.

The intake ventilation fans will have a maximum design capacity of 250 m³/s, and the exhaust fans will have a maximum design capacity of 240 m³/s. This ventilation capacity is based on the requirement to provide airflow for heat dilution, diesel particulate matter dilution, dust control, and to maintain adequate air changes through all active areas.

A heat recovery system is planned to recover the low-grade heat from the exhaust air to heat the intake ventilation air. Additional heating from propane heaters will be used when the exhaust heat recovery is insufficient to raise the intake air above freezing. Underground ventilation on demand (VOD) will be used to monitor and control the ventilation system to ensure adequate air quality is maintained on all working levels.

16.9.1 Airflow Requirements

The airflow requirement for the mine was based on the ventilation system capacity to maintain adequate working temperatures underground based on summer design temperatures, also ensuring that minimum velocities underground and airflow requirement for dilution of diesel particulate matter per CANMET are provided. The airflow requirement calculations are based on the main operating equipment requirements. The peak airflow requirement for the mine is approximately 240 m³/s, 220 m³/s required for heat dilution of the underground mobile equipment and 20 m³/s required to maintain some upcasting ventilation in the shaft.



16.9.2 Overall Ventilation

Stage 1

The initial ventilation stage will focus on the development of the main 5.0 meter diameter intake ventilation raise. This development will be supported by the 186 kW (250 hp) portal intake fan and heater used for the ADEX Program, with 1.52 m (60 inch) diameter rigid PVC ducting. A 22 kW (25 hp) auxiliary fan will direct ventilation air from the ramp and duct it to the development headings on the 60L. At this stage only the fresh air fans will be installed on surface.

Stage 2

The ventilation for the second stage (full production) will require the surface exhaust raise to be established and the surface exhaust fans to be installed, along with airlock doors on the 60L, between the intake raise (with escapeway) and exhaust raise (4.0 m diameter) locations. The airlock doors will allow the fresh air to be isolated from the exhaust air stream in the life-of-mine ventilation design. This stage will also include the development of internal exhaust raises on each end of the level footwall drifts.

The ventilation system will provide flowthrough ventilation along most of the levels, with auxiliary ventilation requirements to support production. The exhaust from the internal exhaust raises at each end of the level will be delivered to the 60L. This will be a dedicated exhaust transfer drift to direct the exhaust from the raises at the end of the level to surface. Automated regulators will be installed at the exhaust raise accesses on the level to control the level ventilation.

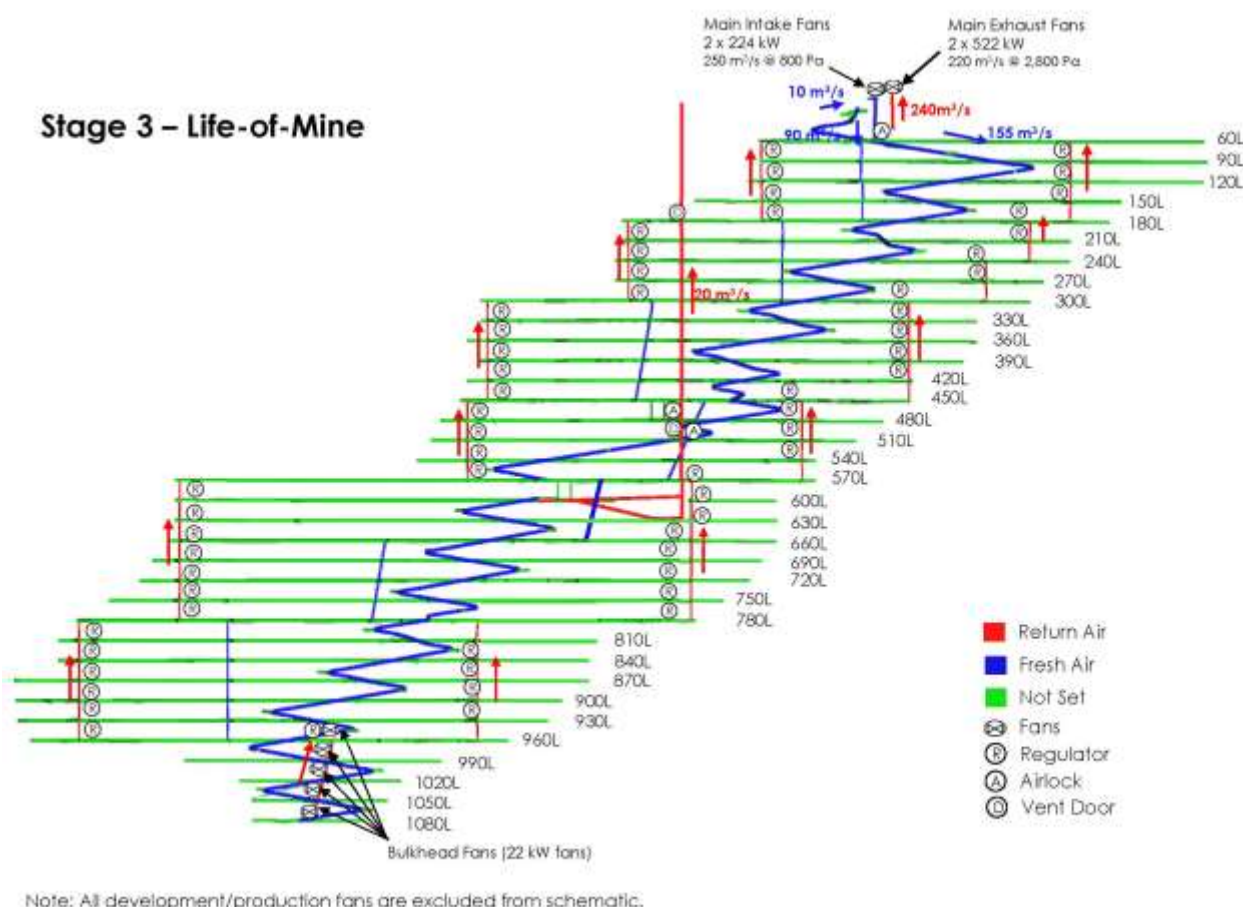
Stage 3

The life-of-mine ventilation system is an extension of the Stage 2 ventilation system, with additional internal exhaust raises being established at the end of the levels to support production in new levels and internal fresh air raises with escapeway established closer to the center of the level. Regulators will be installed at each of the internal exhaust raise accesses for control of ventilation on each of the mining levels. Air quality stations will be installed at each of the regulator stations and at intervals within the main ramp to monitor the air quality.

Figure 16-8 highlights the ventilation schematic for the final stage of the ventilation system. Additional accesses to the shaft will be established during life-of-mine. Doors will be installed at these accesses with 22 kW fans as required to maintain the shaft upcasting.



Figure 16-8: Stage 3 – Life-of-Mine, Ventilation Schematic



16.9.3 Ventilation Controls

System Controls, Operability, and Maintainability

Ventilation controls will be used to control the airflow throughout the mine to optimize the ventilation system performance and reduce health and safety risks from air recirculation and short circuiting. These controls will include airlocks, automated regulators with louvers, and underground fans in bulkheads.

Emergency Alert System

For the ventilation emergency alert system, ethyl mercaptan is planned to be used as a stench gas. It will be combined with R134a as a propellant to form a gas and then introduced into the ventilation stream. The system will be located at the surface ventilation intake raise. It will have redundancy injection and a cylinder package



complete with enclosures. The system will be sized to a capacity suitable for the intake air flows and required concentrations through the mine workings.

The stench gas can be released remotely via the human-machine interface located in the central control room, or manually via the panels located at the surface intake locations. The stench gas will be released on notification of fire or any other emergency underground. Cap lamp emergency warning will also be implemented for immediate emergency notification.

Ventilation on Demand

VOD is planned to be implemented to ensure adequate air quality underground and to avoid over-ventilating. The VOD system will employ a series of sensors distributed throughout the mine. The sensors will send real-time information to a central computer with specialized software that monitors the air quality, vehicle use, and personnel locations underground. This technology, combined with variable speed main fans and louver controls, will establish a highly adaptable ventilation system capable of substantial energy savings, while maintaining air quality standards on the working levels and improved blast clearance times.

16.10 Ore and Waste Handling System

16.10.1 Ore and Waste Handling Process

The ore and waste handling system will be a combination of truck haulage and a shaft hoisting system.

Prior to the shaft hoist being commissioned, ore will be mucked using LHDs from the sill headings and stopes and loaded into haulage trucks that will transport the ore via the ramp to surface. After the shaft loading pocket is commissioned, ore trucking to either surface or the 0570 Level rock breakers will be optimized based on travel distance and elevation change to take advantage of regenerative charging of the BEV haulage trucks.

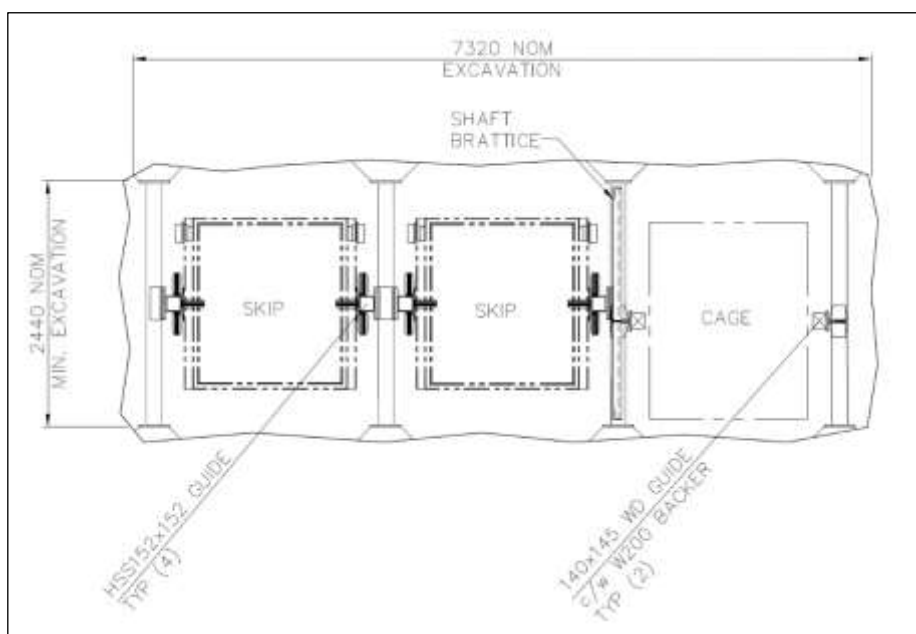
Once stoping begins, waste rock generated from development headings will primarily be used as backfill when there are available stopes in the backfilling cycle. The remaining waste will be trucked to surface. There are no plans to hoist waste to surface.

The shaft construction will commence in Year 5, in conjunction with production from the lower mining blocks. The shaft will be excavated from underground in two legs from 0630 Level to surface, breaking into the pre-constructed shaft collar. The legs will be driven at 2.4 m wide × 3.6 m long. Once the raises are complete, a shaft sinking stage will be set up in the collar. The pilot raise will be slashed to full length, with ground support installed and shaft equipping from the stage as advance progresses from



surface to the shaft bottom (nominally 0630 Level). Refer to Figure 16-9 for a plan view of the shaft.

Figure 16-9: Plan View of Shaft



Shaft layout will be rectangular with steel guides for the skips, wood guides for the cage, and unlined (see Figure 16-9). The production hoist will be a 3.66 m (12 foot) double drum hoist operating the two skips in balance. Each skip will be bottom discharge (arc gate type) and sized for approximately 10 tonne pay loads. The cage will use a 2.54 m, (8.3 foot) single drum hoist and used to transport personnel, materials, light equipment, and tools between surface and underground as required. As the ramp will provide drivable access from surface to all levels, the hoist will not be designed to lower large equipment.

The shaft loadout will consist of two rock breaker stations on 0570 Level, two 2,000 tonne bins with vibratory feeders onto the loadout conveyor on 0600 Level which in turn will feed the loadout transfer car. The loadout conveyor transfers the material to the loading pocket. The loading pocket receives material through the transfer car and directs it to one of two measuring flasks.

Table 16-19 presents the material handling system requirements.



Table 16-19: Material Handling System Requirements

Parameter	Value
Operating Days per Year	360
Shift duration	12 hours per shift, 2 shifts per day
Hoist Utilization	17 hours per day on average
Ore Production	4,200 tpd
Contingency	20%
Hoisting Design Capacity for Ore (including contingency)	5,040 tpd

The headframe will enclose the shaft and skip dump arrangement, and a bin house will enclose the surface bin and transfer point to the overland conveyor. Skips will bottom dump through a chute into a 165 tonne surface ore bin. A deflector gate will direct hoisted material to a bunker on the ground if the overland conveyor is not operating. The surface ore bin will discharge ore onto the overland conveyor via an apron feeder.

16.10.2 Rock Breaker Facilities

The rock breaker facilities will be located on 0570 Level. There will be two rock breaker stations fed directly from trucks or LHDs dumping material onto grizzlies.

Both rock breaker stations will have scalping bars and a sunken grizzly design with storage capacity of two truckloads. The sunken grizzlies will have 500 mm × 500 mm openings and be protected by wear liners.

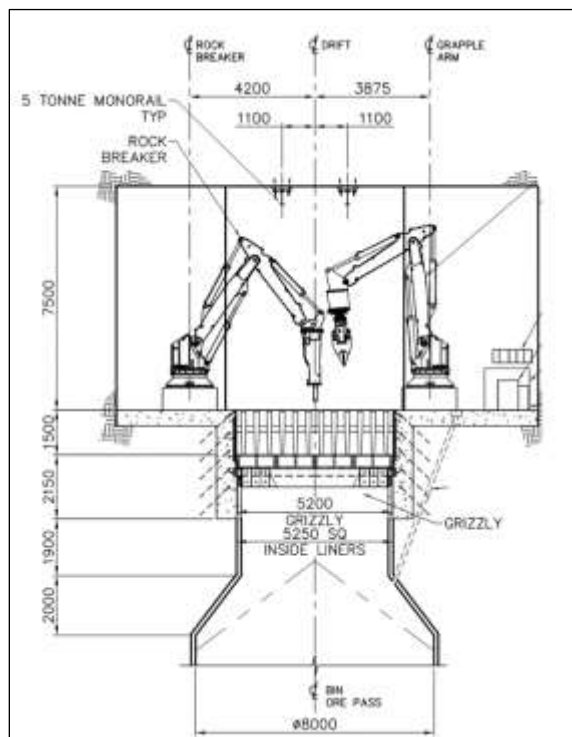
Each station will be equipped with two separate booms: one boom will be equipped with a hammer while the other boom will be equipped with a grappling arm. Respectively, these will be used to break oversized material to pass through the grizzly openings and to remove tramp metal. A hydraulic power unit (HPU) equipped with a self-contained fire protection system will be shared by the two booms.

The rock breaking stations will be equipped with closed-circuit television and controls to allow the rock breaker operator to work from the on-site control room on surface. This will remove the operators from exposure to noise and dust. A local operator station will also be available underground to continue the rock breaking operation in the event of a communications loss with the on-site control room.

Figure 16-10 presents a schematic of the rock breaker station with hammer and grapple booms



Figure 16-10: Schematic of a Rock Breaker Station



16.10.3 Vibratory Feeders and Loadout Conveyor

Hydraulic arc gates at the discharge of the bins will facilitate isolation of material flow for safe operations and maintenance activities. Once the gates have been opened, material will flow onto the deck of the vibratory feeders until the flow of material is stopped by the natural angle of repose of the feed material. Material from each bin will be fed onto a single 1,500 mm wide conveyor belt that transfers the material to the loading pocket. The width of the belt has been selected based on the ability to accept and handle the largest material that may pass through the grizzly opening.

Self-cleaning belt magnets will be used to facilitate the capture of tramp metal that was not removed at the grizzly.

The 0600 Level loadout conveyor will be equipped with variable speed drives and will operate at 1.0 m/s. This will allow maximum control over the feeding of the loading flasks to achieve consistent loading during skipping operations.



16.11 Mine Dewatering

16.11.1 Overview

The underground dewatering facilities are designed to receive water from three sources: ground water seepage, underground process operations, and flush water used to clean paste backfill lines. Ground water seepage will contribute approximately 80 m³/h, of which 50% is expected to be captured on the first mining level. Process water from operations which includes drilling, dust control, washing equipment, etc. will produce an average of 47 m³/h of water. Backfill line flushes will contribute on average 3 m³/h of water. Water from these three sources will be directed to different areas of the underground mine workings for a total combined average rate of 130 m³/h as presented in Table 16-20. Daily volumes will vary depending on the level of activities, backfill sequence, etc.

Table 16-20: Estimate Water Source Flows

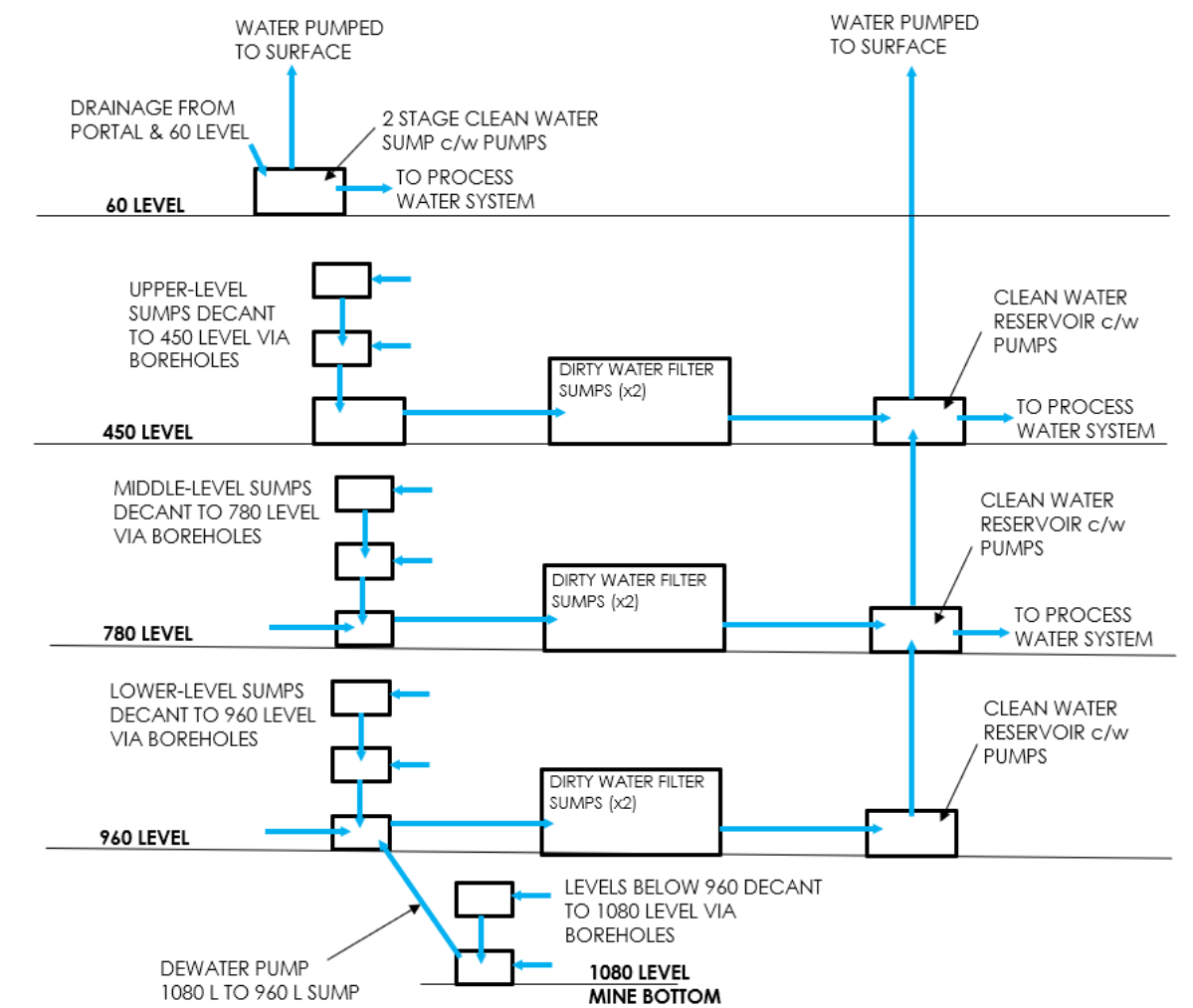
Mine Source	Average Flow (m ³ /h)
Groundwater	80
Process Water	47
Backfill Flush	3
Total	130

The dewatering system will be a cascading design that includes borehole sumps, permanent level sumps, and main pump stations. The 0060 Level permanent level sump will use a dammed decanting arrangement as the collected water will primarily be ground water, not requiring significant filtration. The main pumping stations, 0450 Level, 0780 Level, and 0960 Level will be clean water systems utilizing level sumps, three settling / filtering sumps, and clean water sumps and reservoirs.

Figure 16-11 presents a schematic process flow diagram that illustrates the water flow streams.



Figure 16-11: Schematic Dewatering PFD



Borehole sumps will consist of two drainage boreholes located in the floor at the front (or shallow end) of the sump, and intake boreholes from the back at the back end of the sump. Depending on the operating level, the sump may or may not be fed from a higher elevation level borehole sump. Process water tie-in points will be available at the entrance for every borehole sump for flushing and cleaning the sumps.

A typical level sump will consist of one or more submersible pumps located at the end of a walkway. The submersible pumps can be removed using a chain hoist on a monorail. Process water tie-in points will be supplied at the entrance of every level sump for flushing and cleaning purposes. A fan and duct will be installed to ventilate the sump.

Temporary dirty water level sumps will be commissioned away from the ramp, generally using existing cutouts with a negative gradient. These sumps will collect and pump water to the borehole sump using a single submersible pump.



16.11.2 Sump Locations and Roles in the Mine Dewatering System

Borehole sumps will be located on every level, nominally 30m vertical spacing, except on levels with permanent level sumps (0060 Level, 0450 Level, 0780 Level, 0960 Level, 1020 Level, 1080 Level).

Water collected in the level sumps on main pumping station levels will be pumped into one of three dirty water settling sumps. The settling sumps will be constructed on a base graded at a +3% slope to allow water to drain via gravity through a porous filter membrane (Sturda weir or equivalent). Based on an estimate of 1% solids by weight for run-of-mine water, the sump will take approximately 10 days to fill completely with solids. Once full, the dirty water will be directed to the subsequent settling sump allowing for a 20-day cycle to decant the excess water, remove the residual solids, and set up the weir to receive water again.

Settled and filtered water from the settling sumps will be directed to the clean water level sump through a concrete ditch. This sump will be at a -17% gradient and have a shallow excavation with a short holding time, pumping clean water at a rate of 197 m³/h to the clean water reservoir on the same level. The clean water reservoir will have a live capacity of 350 m³.

The clean water reservoir will be constructed in a cutout with a concrete dam to provide flooded suction for the pumps. There will be two horizontal multistage centrifugal pumps in the pump station, one operating and one standby, to pump the filtered water to the next pumping station or to surface for treatment. There will also be a pipe and valving to supply filtered water to the process water system two levels below, nominally 60 meters, from the clean water reservoirs on 0450L and 0780L, and the decanted water side of the 0060 Level permanent level sump.

16.12 Backfill Distribution

Backfill to be used at the Mcllvenna Bay Project is paste fill for transverse and longitudinal stopes, and uncemented waste rock for Avoca stopes. Waste rock will also be added to the secondary longhole stopes being filled with paste fill when available.

16.12.1 Paste Backfill

Two types of tailings material will be produced from the process plant and used in the production of paste backfill. A Non-Acid Generating Tailings will make up most of the tailings produced by the process plant along with a Pyrite Concentrate generated through a pyrite flotation circuit. The Non-acid generating Tailings will be thickened and sent to either a separate filtration plant and disposed in a conventional tailings disposal site, or to the paste plant to be sent underground as backfill. Due to the potential acid generating nature of the Pyrite Concentrate, it will be sent to either a temporary storage



tank and reclaimed, or directly to the paste plant to be used for backfill. The optimum blending ratio to meet the mine's process and geotechnical demands has been investigated through laboratory testwork. Binder was selected to be a slag-based cement with sulphate resistance. The selection was recommended due to the backfill high strength requirements and risk of strength degradation due to high sulphide content in the Pyrite Concentrate. Two recipes at 5% and 10% binder with a final solids content of 76.5% solids were selected. For testing, as low and high strength backfill options respectively.

For the initial years of mining the underground distribution system will require pumping the paste fill to reach the upper / shallow stopes at the furthest lateral extents of the ore body. For the remainder of the life-of-mine, the paste fill will be distributed via gravity to the lower levels. A combination of steel and steel reinforced corrugated pipe (SRCP) will be used for the paste reticulation system to meet a design pressure of 7.5 MPa. Steel pipe at Schedule 80 125 NB will be used in limited areas of expected high pressure that may exceed the maximum allowable operating pressure (MAOP) of SRCP. The SRCP has been sized at 160 NB and 7 MPa pressure class.

For the production levels below 0240L, burst pipes for line over-pressurization protection will be installed. Ancillaries such as automated dump and diversion valves, and monitoring instrumentation will be installed throughout the mine to provide a high level of operator visibility and control to the underground system during both normal and upset operating conditions.

The paste recipe design and fill strategy will be dictated by the ratio of different types of backfill exposure and associated strength requirements. Strengths for different exposures will vary by stope geometry and will be defined prior to backfill placement during operations. The void volume presents the amount of fill designated for each type of backfill in the life-of-mine and is presented in Table 16-21.

Table 16-21: Backfill Properties by Strength Requirement

Paste Type	Solid Content (wt%)	Binder Addition (wt%)	Cure Time (days)	Void Volume (%)
High Strength – Sill Exposure	76.5	6	56	18
Low Strength – Vertical Exposure	76.5	5	14	62
Low Strength – Secondaries	76.5	2	28	20

For primaries with only vertical exposure, a 3-day cure time on the plug pour and 14-day cure time for the main backfill pour is assumed.

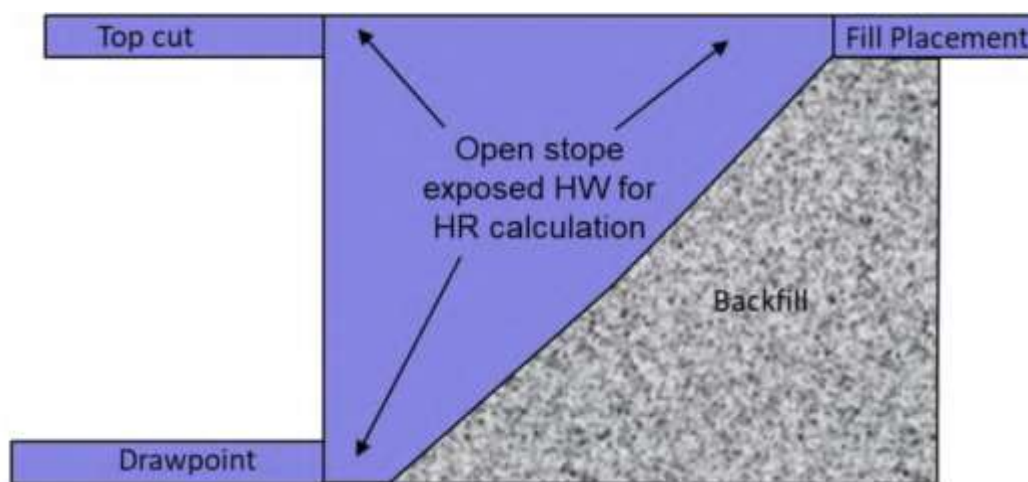


The McIlvenna Bay Project will use an arched shotcrete barricade formed from prefabricated steel frames. These barricades are designed to withstand between 150 to 300 kPa of lateral pressure. The barricade should be located well within the mucking drift. A general rule of thumb is to locate the midpoint of the arched barricade a distance of 1.5 times the drift height from the brow.

16.12.2 Waste Rockfill

Waste rock from lateral development headings will be used to fill Avoca stopes. LHDs will be used to transport waste from the waste stockpile to the stope along the footwall drift. Waste rock will be placed from one end of the stope as the Avoca front advances to the other end of the stope. The backfill placement must advance at a rate sufficient to maintain the span of exposed ground between the fill and the blasted ore muck pile below the required threshold. See Figure 16-12 for long section view through a typical Avoca stope, illustrating the backfill sequence.

Figure 16-12: Long Section View Showing Avoca Backfill Sequence



Source: foran_ni_43-101_technical_report-_pre-feasibility_study.

16.13 General Underground Infrastructure

16.13.1 Personnel and Material Movement

During steady state operations and prior to the installation of the shaft, all personnel and materials will be transferred to and from the underground workings via the mine ramp.

Once the shaft is commissioned, personnel will use either the cage in the shaft or continue to use the ramp, depending on the workplace location underground. All material will continue to be transported underground via the ramp.



During the early ramp development and during full mine production an additional means of emergency egress from the mine will be a ladder system located in the fresh air raises exiting through an airlock in the fresh air fan ducting on surface.

Personnel Movement

Personnel carriers will be used to move workers between the surface or shaft station and the underground workplaces at the start and end of each shift. Workers that operate mobile equipment will park in a designated area at the end of each shift.

Material Movement

Consumable materials, equipment, and maintenance parts will be delivered to designated underground storage locations via the mine ramp using cassette trucks, flatbed boom trucks, and forklifts.

16.13.2 Permanent Refuge Station

Permanent refuge stations will be located throughout the mine in the fresh air circuit, and the vertical interval between stations will not exceed 100 m. The refuge stations will serve the development, production, maintenance, and construction crews working in the area.

The refuge stations will be used primarily as an employee lunchroom but are designed for use during emergency situations.

Each refuge station will be constructed with materials that have at least a one-hour fire resistance rating, and will be equipped with the following:

- Boot and hand washing facilities at each entrance for use prior to entering the refuge station
- Racks to hang outer work gear prior to entering the refuge station
- Thirty-six hours of potable water and bottled respirable air
- Emergency food
- Lights
- First aid supplies
- Sanitation facilities
- Process water line
- Suitable firefighting equipment
- Sufficient seating for each worker



- Multiple means of communication with the surface
- A double door that creates an air lock when closed
- Means of tight sealing the door to prevent gases from entering

Two portable refuge stations will also be purchased. As mine development and production advances into new mining areas, these portable refuge stations will be placed in appropriate existing cutouts to establish a temporary refuge station.

A Rezplast portable sanitary system—or similar model—will be provided near each refuge station location.

16.13.3 Explosives and Detonators Storage Facilities

Underground storage magazines for explosives, detonators, and blasting accessories will be located on the 0120, 0390, and 0720 Levels, set a safe distance from the underground infrastructure and work areas. There will be separate magazines for explosives and detonators.

Explosive products used for development and production mining will be securely handled and stored in the magazines, including ANFO, emulsion, packaged explosives, and detonators.

All explosives will be stored, stacked, and labeled to facilitate a first-in / first-out inventory control system. Each magazine will be designed with a locking gate and include camera surveillance for security. The location of the explosive or detonator facility will be a minimum of 100 m from any work area or blasting area, and at least 25 m from the main travel way.

Explosive and detonator materials will be transported via truck from the surface via the ramp to the underground magazines and unloaded using boom trucks. Special trucks operated by trained and authorized individuals will be used to transport explosive materials from the underground magazines to the workplace.

16.13.4 Battery Charging and Changing Stations

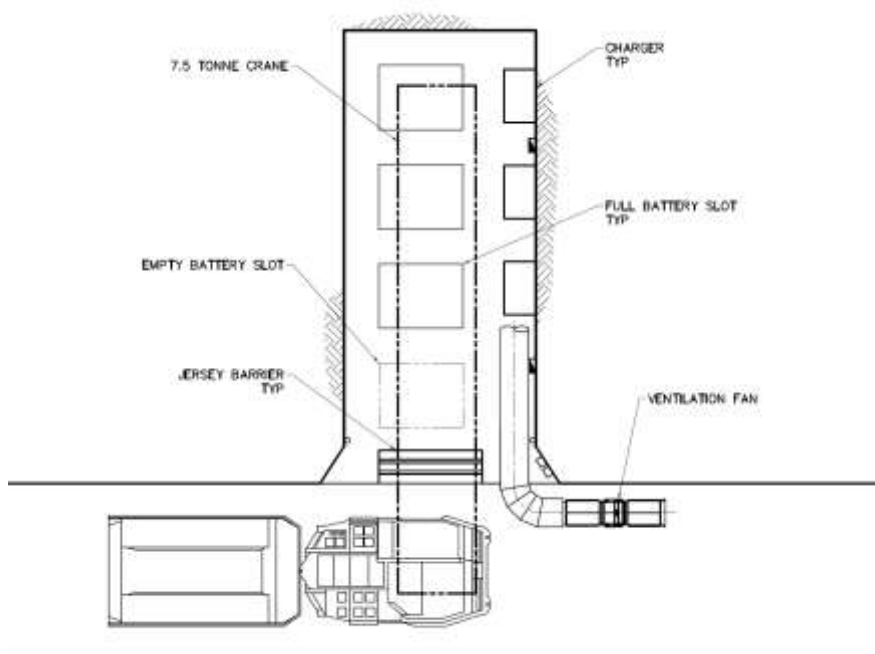
There will be two types of battery charging stations: the battery swap and charge stations and the small mobile charging stations. The battery swap and charge stations will initially be in the surface shop, and then the 0180 Level once the level is available. The first mobile charging stations will be located on the 0180 Level as well. As the mining progresses deeper, the uppermost stations will be moved down to a lower active level, with a maximum of two of each station type underground at any time. The battery swap and charge stations will be located on surface and the 0180, 0300, 0450, 0570, 0780,



and 0960 Levels. The small mobile charging stations will be located on the 0180, 0270, 0360, 0450, 0540, 0660, 0750, 0870, 0960, and 0150 Levels.

The charging philosophy for the battery swap and charge station works as follows: there are four battery slots, up to three will be in charge, and one will remain empty for the next battery to be dropped off. When a piece of equipment pulls up alongside the station a 7.5-tonne overhead crane will be used to remove the battery from the mobile equipment, place it in the empty charging slot, and then place a fully charged battery on the mobile equipment. There are three charging posts connected to one battery power cabinet. A plan view of the battery swap and charge station is presented in Figure 16-13.

Figure 16-13: Battery Swap and Charge Station Plan View



The battery swap and charge station will contain a fire alarm panel, electrical rack, fire extinguishers, and protective barriers at the entrance of the excavation and in front of the electrical racks. The station will be ventilated using a fan and silencers directed to the rear of the excavation.

There will be a similar set-up as the battery swap and charge stations, using three charging posts and one cabinet. However, the batteries will remain on the mobile equipment and will be charged in place while parked. The excavation is meant to allow traffic to pull to the side, away from traffic flow. Ventilation is not required as the depth of the station is shallow and can be classified as flow-through ventilation. The mobile charging stations will have fire detection and suppression.



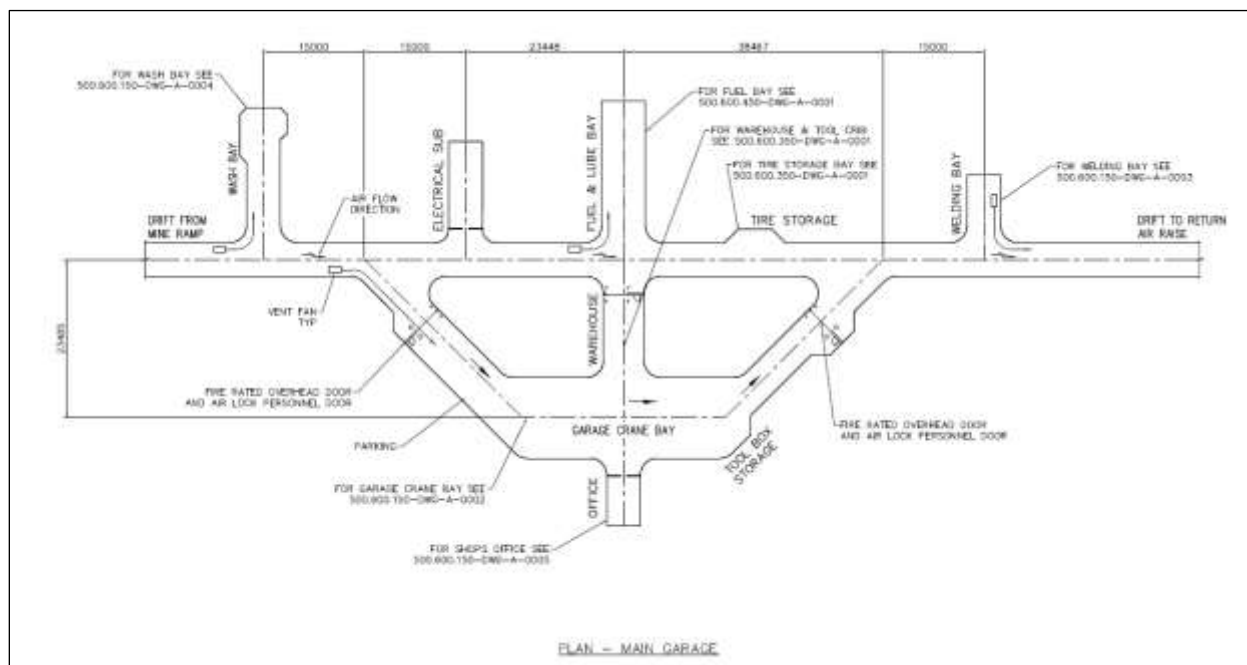
16.13.5 Maintenance Shop

The main workshop for the mine will be located on surface and mobile equipment will be driven up ramp for maintenance and repair. A workshop will be constructed on the 0420 Level in the underground mine to supplement the main workshop activities. The underground workshop will be a single-bay facility that can service two units of mobile equipment. The underground workshop area will include the following components:

- Combined crane and service bay
- Wash bay
- Welding bay
- Office
- Electrical substation
- Warehouse
- Tire storage

The key features of the underground workshop are presented in Figure 16-14.

Figure 16-14: Key Features of Underground Workshop



The combined crane and service bay will include the following equipment:



- One 25-tonne overhead crane
- 10 hp air compressor piped to hose stations within the shop
- Lube hose reels for equipment servicing (pipe from fuel and lube bay)
- Process water hose stations (piped from main access drift)
- Fire suppression sprinkler system (piped from main access drift)
- Fire extinguishers
- Work benches
- Tool boxes
- Sink
- Eye wash station

Ventilation for the crane and service bay will be flow-through to a nearby exhaust drift. Roll-up fire doors will be installed at the entrances to the workshop.

The welding bay will be located near the exhaust drift and will be equipped with welding tables, acetylene and oxygen bottles in a rack, 5-tonne monorail crane, work benches, storage cabinets, portable welding screens, 10 hp air compressor, and exhaust hood ducted to the exhaust drift.

The office will have tables and chairs, refrigerator, counter, teledyne oven, first aid supplies, desks, computers, and phone equipment for communication with the surface, as well as computer access to all necessary servers.

The warehouse located in the main workshop will have shelving along one wall for storage of equipment and small parts, as well as a tool crib. Pallets will be stored along the opposite wall. The warehouse will be sized for a forklift to access the pallets.

The tire storage bay will be designed to accommodate a minimal selection of tire sizes for the underground mobile equipment since major tire changes will be completed in the main workshop on surface.

The wash bay will be located adjacent to the workshop area for cleaning vehicles prior to maintenance. The wash bay will have a high-pressure washer with a water heater, soap cubes, and a high-pressure hose reel. Process water will be supplied via piping routed from the main drift. A fan with silencers will ventilate the wash bay area. A sloped concrete floor will promote gravity flow of wash water to a sump with a submersible pump. Water collected in the sump will be pumped to an oil / water separator.



16.13.6 Fuel and Lubricants

The underground mine will have a main fuel and lube bay on the 0420 Level and one satellite fuel bay on each of the 0180 and 0720 Levels. Fuel bladders will be transported via truck from surface via the mine ramp to the main and satellite fuel bays. Lubricants will be transported in bladders and pails via truck from surface via the mine ramp to the main fuel and lube storage bays.

The main underground fuel and lubricant bay will be centrally located on the 0420 Level near the underground workshop. Oils and lubricants such as glycol, transmission fluid, hydraulic fluid, and engine oil will be dispensed in the fuel bay as well as pumped to the crane and service bay. All fuel bays will accommodate one vehicle at a time to access diesel fuel.

16.14 General Underground Services

16.14.1 Compressed Air

Compressed air demand will be provided locally, without the use of a centralized compressor system or a network of distribution pipes. Mobile machinery will be equipped with onboard compressors, with two towable compressors used for the exploration diamond drills, and four towable compressors used for construction activities including Alimak raising.

The maintenance shop compressed air will be supplied by a free standing, electrically operated air compressor.

Compressed air bottles or oxygen bottles will be placed in the refuge stations to supply back-up breathable air.

16.14.2 Process Water

Process water is required for use in exploration, development, and production drilling, dust control, washing floors and equipment, preparing shotcrete and concrete mixes, fire protection, and other miscellaneous processes. The underground water distribution system will be supplied from a combination of a surface holding pond and underground recycled water. The clean water sumps on 0060 Level and 0450 Level, will recycle water into the process water line, connecting to process water line on the level below. The process water may need to be treated prior to entering the process water line. A detailed analysis will need to be completed during operations.

The system will initially be charged using a surface pump and then operate as a passive supply system. Water supplied from the surface holding pond or underground dewatering system will generate the pressure required by the elevation change



descending in the mine. Pressure reducing stations are located on specific levels to provide water between 500 kPa and 800 kPa to the level distribution for the mining activities.

The mine water distribution system will be comprised of the following:

- Process system supply piping
 - Surface piping from holding pond
 - Underground water distribution along ramp
- Pressure reducing stations
- Underground recycled water piping
- Level distribution piping

16.14.3 Potable Water

Potable water will be delivered underground via large plastic bottles. Potable water will be provided in workshops, permanent refuge stations, and dedicated bottle filling stations.

Personnel must fill appropriate water containers on surface or at a designated bottle filling station and carry a water supply to the work areas.

16.14.4 Fire Water

Underground fire-related services are required to meet NFPA requirements. Fire water must be available for underground fire suppression, hose reels, and sprinkler systems. The surface holding pond will meet the storage requirements and the process system piping will provide delivery to the various underground fire sprinklers.

Fire detection and suppression systems will interface with the emergency alarm system and will be included in areas at high risk of fires. These areas include the drive end of conveyors, main workshop (crane bay, service bays, welding bay, office, and lube storage bay), main fuel and lubricant storage and distribution area, and satellite workshop. Individual operating systems, such self-contained fuel and lube units and hydraulic power units, will be equipped with integral fire-suppression and will not require fire water.

16.15 Underground Electrical and Power Distribution

Power for the underground facilities is divided into three areas; underground portion of the mine, headframe and hoist house, and the surface ventilation fan plant (fresh and



return air). All power for these areas is provided from the surface substation E-House. Power will be distributed using a combination of overhead pole line and buried cable routings.

16.15.1 Underground Electrical Distribution

Power for the underground mine will be provided via two 15 kV feeders. Feeder #1 connects at the portal electrical building and provides power down the main ramp for early development and services (dewatering, ventilation, etc.). Feeder #2 connects at the fresh air raise at surface (access and inspected via the escapeway installed in the raise) and is routed to the 0180 Level electrical room.

The power distribution system design is based on providing power to two levels at a time. Junction boxes rated at 15 kV will be installed near the main electrical room on a level. From these junction boxes power will either continue down the mine or feed the electrical room.

From the main electrical room on the level, 13.8 kV power will be fed to development and production equipment, while a 600 V power distribution panel (PDP) will feed power to infrastructure facilities such as local ventilation fans and regulator stations, level sumps, shops, etc. For levels requiring two electrical rooms (due to the length of the level exceeding the recommended voltage drop), 13.8 kV power will be routed from the main electrical room to the second room.

A borehole will be used to provide 13.8 kV and 600 V power distribution from the main electrical room to the next level. For levels with two electrical rooms, the second room will also use a borehole to provide 13.8 kV power to the next level.

As the two main 15 kV feeders progress down the first portion of the mine, these will connect at the main-tie-main switchgear lineup at the Type 1 electrical room on the 0180 Level to provide the first point of redundancy in the underground power distribution system. Additional Type 1 electrical rooms with similar main-tie-main switchgear are planned on the 0420, 0780, and 0960 Levels.

The underground portion of the mine has been designed with five types of electrical rooms. Type 1 and 2 (A and B) electrical rooms are main power distribution rooms and house switchgear lineups. Type 3 rooms are small cutouts used to extend 13.8 kV power for development and production equipment, and 600 V power to infrastructure such as local ventilation fans and regulators, and level pump stations. Electrical rooms for the main pump stations will provide dedicated 13.8 kV power from the main electrical room on the level. This reduces the transformer size in the main electrical room and the cabling to the main pump station electrical rooms.

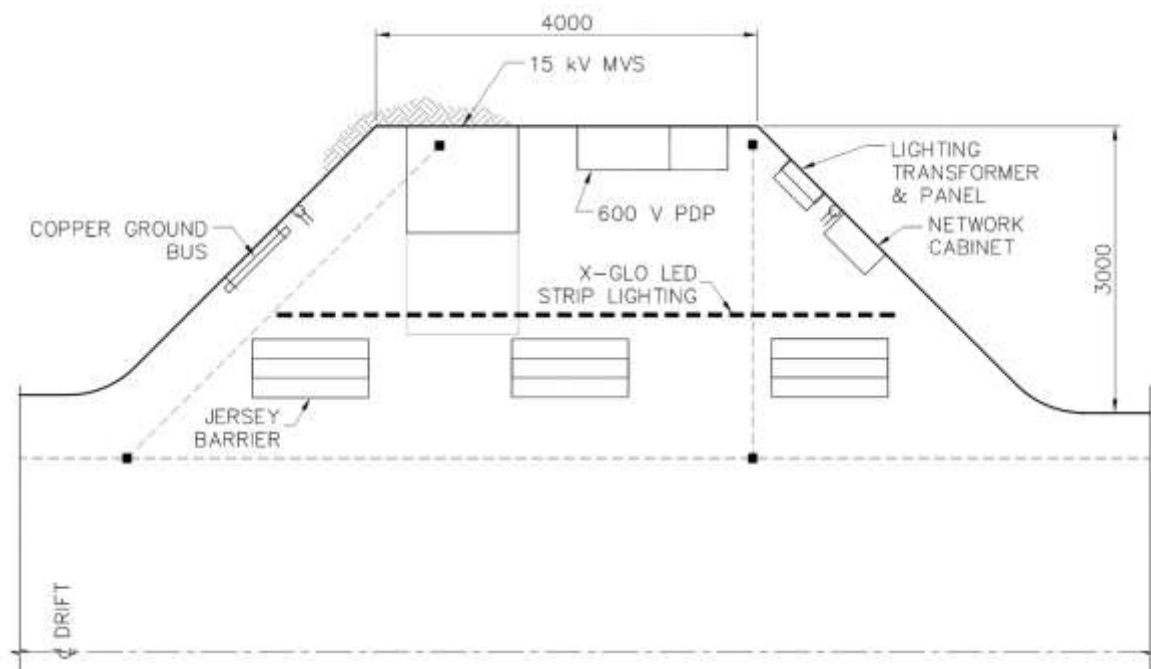


Type 1 electrical rooms include the switchgear lineup with battery charger and rack, 13.8 kV to 600 V transformer, 600 V power distribution panel, lighting transformers and panels, network cabinet for fiber optic and LTE communications, and a programmable logic controller (PLC) or remote input/output (RIO) cabinet when required.

Type 2 (A and B) electrical rooms are dual level power distribution rooms fed from high voltage junction boxes from the main underground feeder cables. A Type 2A room has a permanent 13.8 kV switchgear lineup while a Type 2B room has a temporary 13.8 kV switchgear lineup. Both types of rooms provide power to the level where the room is located and power to the level below. Type 2A rooms provide power to permanent infrastructure facilities (main pump stations, loading pockets, shop, etc.).

Type 3 electrical rooms are small satellite rooms that are either the secondary room on a level or the extension of a Type 1 or 2 electrical room on the level below. The purpose of these rooms is to provide a connection point for either 13.8 kV or 600 V power. Figure 16-15 presents the layout of a Type 3 electrical room.

Figure 16-15: Type 3 Electrical Room Layout



These Type 3 electrical rooms will operate for the life-of-mine to ensure communications are available on all levels.



Load List and Schedule for Mining Facilities (Underground and Surface)

Table 16-22 provides a load summary by area for underground and surface mining facilities.

Table 16-22: Mining Load Summary

Area	Demand Load (kVA)	Operating Load (kVA)	Peak Load (kVA)
Surface Ventilation	2,187	1,750	1,883
Headframe	261	209	209
Hoist House ⁽¹⁾	2,164	2,011	2,936 ⁽¹⁾
Total Loading – Surface	4,612	3,970	5,029
Mobile Equipment ⁽²⁾ (Development/Production)	1,585	792	792
UG Ventilation	1,024	806	806
UG Dewatering	1,081	791	791
UG Material Handling	331	256	256
UG Infrastructure	3,555	2,738	2,738
Compressed Air (Portable Compressors)	253	101	101
Total Loading – Underground	7,830	5,485	5,485
Loading for Mining Facilities	12,442	9,455	10,514

9. Peak load for the hoist house is determined by the hoists (166% of operating load—this can be mitigated using a battery bank, to be considered during detailed engineering).

10. Mobile equipment loading is based on the maximum pieces of equipment operating at the same time.

16.16 Underground Automation and Communication

The automation and communication systems planned for the underground facilities will use proven technologies. These systems will focus on safety, reliability, constructability, maintainability, and operability. The following are key considerations for the design:

- All equipment will be modern, standard (not proprietary), and reliable.
- All equipment selected will not be on a manufacturer's obsolete materials list.
- Commonality to reduce the spare parts required.



The following systems are planned as part of the underground communication and automation package:

- Fiber optic backbone and Ethernet networks
- Underground wireless network (LTE)
- Leaky feeder system (radio)
- Blasting system (via LTE)
- Safety systems such as proximity detection, equipment and personnel tracking, and safety curtains where tele-remote operations are being conducted

16.16.1 Fiber Optic Backbone and Ethernet Networks

The underground fiber optic backbone and ethernet networks will be shared between the business and process control networks. The design is based on a self-healing ring topology with redundancy on levels with main network cabinets.

The ring topology for the fiber optic backbone has been split into two parts.

Part one includes two 144-core fiber optic cables that are the backbone for the underground system. Part two includes a pair of two 48-core fiber optic cables that will be installed between the Type 1 electrical rooms and connect to the Type 2 electrical rooms in between.

16.16.2 Underground Wireless Network (Long Term Evolution – LTE)

The underground wireless network will be LTE using dedicated fibers from the fiber optic backbone to connect hardware throughout the mine. The LTE network will be the primary communication system underground to be used with telephones, tablets, and other peripheral devices. The LTE network will also be used for data collection (e.g., mobile equipment telemetry, equipment / personnel tracking), tele-remote operation, blasting system, and video system.

16.16.3 Leaky Feeder System (Radio Communication)

The leaky feeder system will be the primary method of communication in the ramp during development activities until the fiber optic backbone and LTE system is installed. The system's head end will be in the portal electrical building and will be connected to the local network cabinet.

Underground, the leaky feeder system will be routed along the main ramp and will only split onto levels with a refuge station. The leaky feeder system will continue on these levels to the refuge station.



16.16.4 Blasting System

The blasting system for underground development and production activities will be LTE-based, a single integrated system from blast initiators to data logger units and fit for purpose. A blast initiator unit will be located at the on-site control room in the office building, and this unit has the capability to connect with multiple blast initiator units underground. The underground units will be located a safe distance from where development and production activities are being conducted and can connect with up to six data loggers near the blast zone.

16.16.5 Safety Systems

For the McIlvenna Bay Project, the following safety systems will be included to support underground personnel, proximity detection, equipment and personnel tracking, and safety curtains for tele-remote operations.

The proximity detection system is a stand-alone system using frequency transceivers which are devices installed in mobile equipment (tablet, dashboard screen, or other) and on personnel cap lamps. These devices establish a peer-to-peer connection when in close proximity and provide notification to the user.

The equipment and personnel tracking system will operate over the LTE network using Bluetooth Low Efficiency (BLE) beacons. These BLE beacons initiate a periodic “ping” that is received by devices installed on cap lamps and on mobile equipment. These devices transmit data through the LTE network to the tracking software.

Areas planned for tele-remote operations (rock breakers, LHD units, and drills) will require safety curtains to ensure personnel do not enter these areas during operation. The tracking system will also be used to ensure these systems are not started from the on-site or remote-control centre when personnel are in the area. When one of these curtains are triggered, an alarm will be provided to the tele-remote operator and equipment will be safely shut down and disabled until the system can be reset.

16.17 Comments on Section 16

The mine plan will apply conventional mining techniques using conventional equipment. The installation of ground support using the drill jumbo is widely utilized in other mining jurisdictions.



17.0 RECOVERY METHODS

17.1 Introduction

This section describes the parameters used to design a new mineral processing facility for the McIlvenna Bay Project. The plant would allow on site treatment of approximately 25.7 million tonnes of copper-zinc ore at an initial processing rate of 4,200 tpd (1.512 Mtpa), based on the current mine plan.

The two most prevalent mineralization types feeding 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 minor amounts of zinc, and the latter containing higher zinc grades with additional copper units as well. In addition, the CSZ material is harder and contains less pyrite than the MS zone.

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. Similar primary and concentrate grind targets, plus reagent recipes are recommended for the processing of the two main 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.

Since the day-to-day mix of CSZ and MS tonnage is expected to vary, a practical blending strategy is proposed using online grade measurement equipment (PGNAA technology) to allow for the stockpiling of “copper-rich” and “zinc-rich” ore on surface, and selectively draw from these piles to stabilize the grade of the mill feed. 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. This grade management strategy will help to ensure that any unplanned mill feed grade deviations are minimized.

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 at Base Met Laboratories. This work is discussed in detail within Section 13.0 of the report.

17.2 Process Summary

17.2.1 Description

The proposed processing facility has been designed as a nominal 4,200 dmtpd concentrator plant. In the early years of production (Years 1-4), ore will be hauled to surface in 42-tonne trucks and dumped into a surface crushing facility. As the mine



development continues to depth and the number of trucks becomes difficult to manage, a mine hoist will be constructed to feed -500 mm ore onto a new overland conveying system to feed the surface crushing facility.

Run of mine ore will be crushed to a nominal 100% passing 150 mm, (80% passing 100 mm) size. Conveyors will transfer coarse crushed material to one of two surface stockpiles in preparation for the SAB grinding circuit.

The selected grinding circuit consists of a SAG and Ball mill combination designed to accommodate the variable hardness expected from blending 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 flotation testwork.

Cyclone overflow slurry from the grinding area will be directed to the flotation section for sequential copper 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 duty, with the conventional “trough” type cells used for cleaner flotation. The copper and zinc rougher concentrates will both be subjected to regrinding using vertical, stirred mills, with copper and zinc concentrates requiring a P80 of approximately 20-25 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 produce a 55% 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 rougher tailings slurry will pass through a desulphurization rougher flotation circuit. These cells will be similar in size/design to the copper and zinc rougher flotation cells, but with additional reagents added to 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 flotation circuit tailings slurry will be pumped to a tailings thickener for dewatering, with the water recovered returned to the process for re-use. Thickened underflow slurry (approximately 55% solids) will then be pumped to a storage tank at the paste plant.



At the paste plant, the desulphurized flotation tailings will be used to produce a filtered cake suitable for stacking at the surface tailings facility, and for inclusion in the 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 tailings thickeners will be located outdoors, and partially clad to ensure effective operation in cold climates. The tailings filtration and paste production plant will be located in a separate building.

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 promoters, 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 Process Design Criteria

The main process design criteria used for this study is summarized on Table 17-1.

Table 17-1: Main Process Design Criteria

Criteria	Units	Design Data
Annual Mean Throughput	dmtpa	1,512,000
Daily Mean Throughput	dmtpd	4,200
Operating Hours per Shift	h	12
Operating Shifts per Day	#	2
Operating Days per Week	d	7
Operating Days per Annum	#	365
Crushing Circuit Availability	%	70
Grinding Circuit Availability	%	92
ROM Moisture	%	4
Abrasion Index (Ai) - CSW	g	0.45
Abrasion Index (Ai) - MS	g	0.20
Rod Mill WI (Bond) - Metric - CSW	kWh/t	19.5
Rod Mill WI (Bond) - Metric - MS	kWh/t	12.5
Ball Mill WI (Bond) - Metric - CSW	kWh/t	18
Ball Mill WI (Bond) - Metric - MS	kWh/t	11.8
Plant Feed Blend of CSZ Ore - Average	%	50
Plant Feed Blend of CSZ Ore - Design	%	65 (70% grinding)



Criteria	Units	Design Data
Blended Plant Feed Grade Cu	%	1.29
Blended Plant Feed Grade Pb	%	0.21
Blended Plant Feed Grade Zn	%	3.01
Blended Plant Feed Grade Au	g/t	0.56
Blended Plant Feed Grade Ag	g/t	18.45
Blended Plant Feed Grade S	%	13.75
Crushing Circuit Feed F ₁₀₀	mm	500
Crushing Circuit Product P ₁₀₀	mm	150
Crushing Circuit Product P ₈₀	mm	100
SAG Mill Feed F ₁₀₀	mm	150
SAG Mill Feed F ₈₀	mm	100
SAG Mill Discharge % Solids	%	70
SAG Mill Pebble Extraction	%	10.4
Transfer Size	µm	560
Ball Mill Discharge % Solids	%	72
Ball Mill Circulating Load	%	350
Cyclone Overflow Product P ₈₀	µm	75
Cu Flotation Conditioning Time Required	min	7
Cu Rougher Res. Time Req.	min	20
Cu Rougher Mass Pull	%	15
Cu Regrind Mill Product P ₈₀	µm	20-25
Cu Cleaner 1 Res. Time Req.	min	15
Cu Cleaner Scav Res. Time Req.	min	5
Cu Cleaner 2 Res. Time Req.	min	15
Zn Flotation Conditioning Time Required	min	7
Zn Rougher Res. Time Req. (Plant)	min	15
Zn Rougher Mass Pull	%	12
Zn Regrind Mill Product P ₈₀	µm	20-25
Zn Cleaner 1 Res. Time Req.	min	12.5
Zn Cleaner 2 Res. Time Req.	min	7.5
Zn Cleaner 3 Res. Time Req.	min	7.5
Cu Rougher Res. Time Req.		12.5
Py Rougher Mass Pull	%	22.7
Cu Conc Thickener U/F Solids Content	%	55



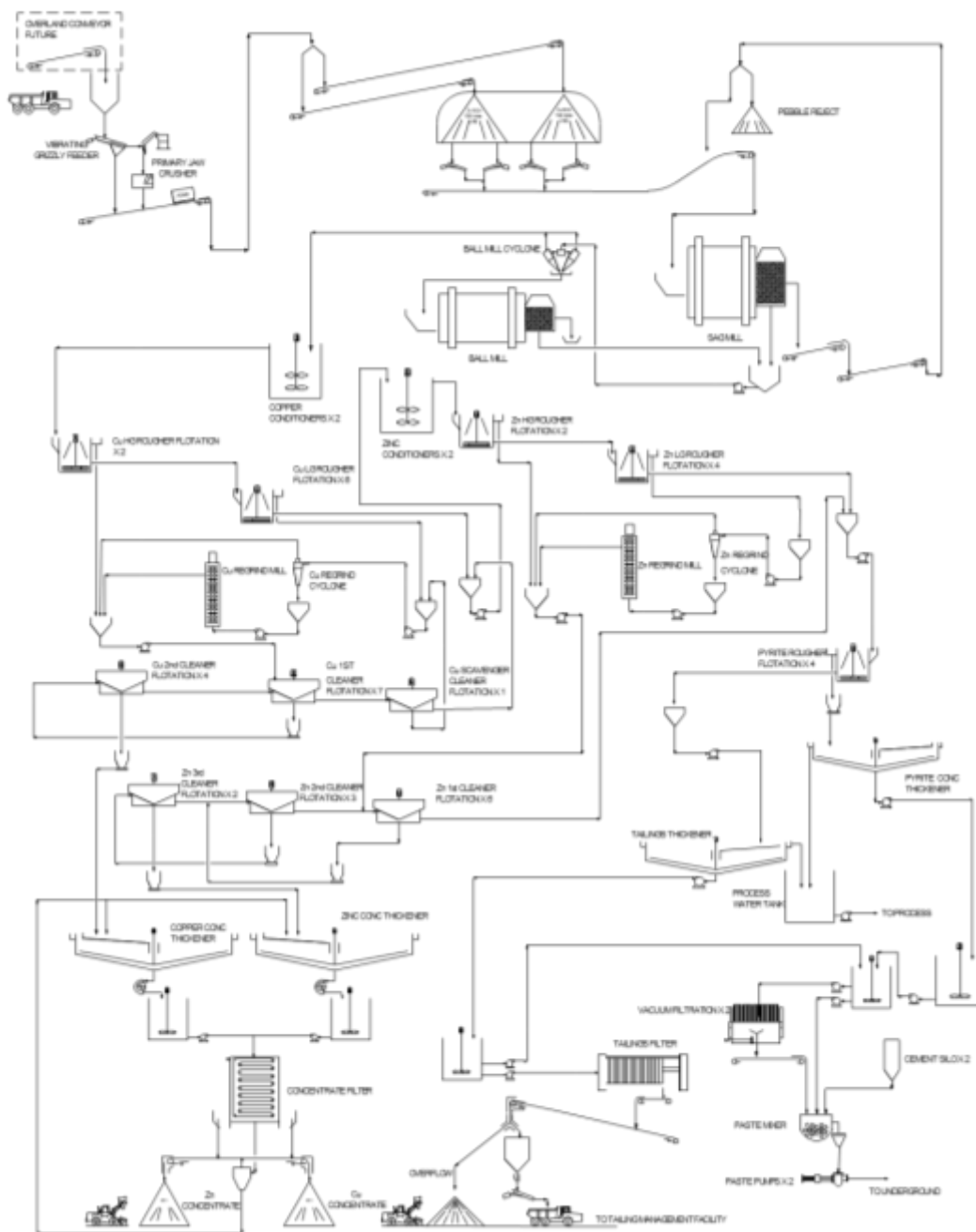
Criteria	Units	Design Data
Zn Conc Thickener U/F Solids Content	%	55
Cu/Zn Conc Cake Moisture	%	9
Py Conc Thickener U/F Solids Content	%	55
Py Conc Storage Time Required	d	3
Paste Plant Operating Schedule		5 days on 3 days off
Paste Plant Availability	%	62.5
Split of Mill Feed to Paste, design	%	50
Paste Cement Addition (%w/w)	%	4
Tailings Thickener U/F Solids Content	%	55
Tailings Average Cake Moisture	%	18

17.2.3 Summary Flowsheet

The planned process plant is described by a series of process flow diagrams. A summary process flowsheet for the full plant is shown below in Figure 17-1.



Figure 17-1: 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.



17.3.1 Primary Crushing – Area 100

In the initial years of mine life, ROM ore will be hauled to surface using 42-tonne capacity haul trucks and delivered from the portal either to the ROM stockpile or to the dump bin of the surface crushing facility. In later years, as the ramp development passes the 0600 level, the truck haulage system will be used in combination with a mine hoist lifting system. At this point, the hoist system will be able to feed the existing surface crushing circuit via a new overland conveyor.

Initial Operation (Year 1–4)

Trucks will deliver run of mine ore to the primary crusher dump bin with a nominal top size of 500 mm and an average rate of 247 dmtph. If the dump bin is approaching capacity, the truck will be diverted by lights to tip on the nearby ROM stockpile.

The stockpile and primary crusher dump bin arrangement are located in close proximity to the portal and will be designed to minimize truck travel and turnaround, thereby maximizing equipment utilization.

The primary crushing system consists of a dump bin (60 t capacity), vibrating grizzly feeder (VGF), rock breaker, 150 kW jaw crusher, various chutes and a sacrificial conveyor to facilitate the movement of material. The VGF oversize will report to the jaw crusher and undersize will bypass directly to the sacrificial conveyor.

VGF undersize and crushed ore (100% -150 mm) is moved to a transfer point by the sacrificial conveyor at a rate of 247 tph. This conveyor is equipped with a PGNAA analyzer to determine copper and zinc concentrations. 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 material stream. The composition of the rock determines the position of an actuated bifurcated chute (diverter gate) at the transfer point that diverts the crushed coarse ore material to the appropriate transfer conveyor and coarse ore stockpile; namely Zinc or Copper Coarse Ore Stockpile with a capacity of 1,500 live tonnes each. Both the Copper Coarse Ore Feed Conveyor and the Zinc Coarse Ore Feed Conveyor have weightometers for monitoring and recording the ore mass flow rate.

Later Operation (Years 7–18)

In Year 7 of the mine plan, a mine hoist lifting system will be commissioned to transport ore from the 0600 level to surface. Ore will be hauled from the lower levels of the mine to the hoist system using 42 t trucks. The truck haulage to surface will be retained to be used when required. The hoist system will discharge onto an overland conveyor and into the surface crushing system as described above.



The main items of equipment in Area 100 (Primary Crushing) are described in Table 17-2 below.

Table 17-2: Primary Crushing Circuit Main Equipment

Area 62100- EQ #	Description	Notes	kW
62100-RBR-02	Rock Breaker	Model: Rammer/Bretec B300	19
62100-BBA-04	Feed Hopper	37 m ³ , 60 tonnes	-
62100-FDA-08	Vibrating Grizzly Feeder	Model: SV1262 VGF	22
62100-CJA-14	Primary Crusher	Model: Sandvik CJ412	150
62100-FCV-18	Sacrificial Conveyor	54" W	19
62100-MAA-20	Cross Belt Self Cleaning Magnet	Model: SPM42 Magna Skin, 36" Belt	1.5
62100-PGN-54	On Line Grade Measurement	Geoscan-M, standard	

17.3.2 Coarse Stockpiles – Area 150

There will be two coarse ore stockpiles, both of which will be covered from the elements. One stockpile will contain mainly zinc-rich mineralization (MS), while the second coarse ore stockpile will contain mainly copper-rich mineralization (CSZ). The two coarse ore stockpiles will be drawn down in a controlled manner using vibrating feeders to feed the SAG Mill Feed Conveyor. 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 grinding circuit feed stream. This configuration allows for significant instantaneous grade fluctuations into the process plant to be attenuated prior to milling and flotation. The SAG Mill Feed Conveyor has a weightometer for monitoring and recording feed rate.

The main items of equipment in Area 150 are described in Table 17-3 below.

Table 17-3: Coarse Stockpile Area Main Equipment

Area 62150 - EQ #	Description	Notes	kW
62150-FCV-01	Overland Conveyor	1525 mm W × 182 m L	30
62150-FCV-06	Zn Coarse Ore Feed Conveyor	610 mm W × 54.4 m L, model 3/EP630	30
62150-FCV-10	Cu Coarse Ore Feed Conveyor	610 mm W × 54.4 m L, model 3/EP630	30
62150-FDB-20 & 24	Zn Vibrating Feeder #1 & #2	1 m × 2.5 m × 0.3 m, model: BW BFR 100 2500	7 × 4
62150-FDB-30 & 34	Cu Vibrating Feeder #1 & #2	1 m × 2.5 m × 0.3 m, model: BW BFR 100 2500	7 × 4
62150-FCV-38	SAG Feed Conveyor	610 mm W × 110 m L, model: 3/EP630	30



17.3.3 Grinding Circuit – Area 200

The grinding area is made up of a two-stage grinding circuit. The first stage includes a SAG Mill and two conveyors for pebble recycling or disposal. The blend of crushed ore sized at a P_{80} of 100 mm will be conveyed from the coarse ore stockpile (Area 150) to the mill building and into the SAG mill feed trunnion via the SAG mill feed conveyor and retractable loading chute. The SAG mill will be a 7.32 m diameter $\times$ 3.65 m EGL unit equipped with a 3,500 kW variable speed drive, a charge of 125 - 100 mm balls equivalent to ~10 to 12% of the chamber volume and a total load of ~25%. The SAG mill will be equipped with the necessary lubrication systems and inching drive to assist maintenance. It will be equipped with steel liners, and an internal discharge grate to permit the passage of slurry and finer particles from the milling chamber. Within the SAG mill, rock will be diluted with process water and ground to a coarse slurry (transfer size P_{80} of 0.56 mm).

Slurry will exit the SAG mill via a trommel screen with 8 $\times$ 24 mm slots. Trommel oversize will be recycled to the SAG mill feed chute, with option to be discharged to an outdoor pebble bunker. Trommel undersize will report to the mill discharge tank together with the ball mill discharge slurry from where it will be further diluted with process water and pumped to the ball mill cyclone pack for classification.

The second stage in the grinding area consists of a 4.7 m diameter $\times$ 7.4 m EGL overflow discharge ball mill equipped with a 2,700 kW variable speed drive and a charge of 65-50 mm balls equivalent to 30 – 32% of the chamber volume. The ball mill will be rubber lined and equipped with the necessary lubrication systems and inching drive to assist maintenance. Feed to the mill will consist of the cyclone pack underflow, with the mill discharge laundered to the mill discharge tank by gravity.

From the mill discharge tank, slurry will be diluted and pumped to the 6 $\times$ HC420 (four operating, two stand-by) hydrocyclone pack for classification. The feed to the hydrocyclone will have a densitometer, flowmeter and pressure transmitter to allow for monitoring and control of the feed density, flow and pressure. Hydrocyclone overflow slurry will gravitate to the copper flotation circuit, whereas the coarser cyclone underflow slurry will gravitate to the ball mill retractable feed chute as ball mill feed. The design circulating load in the cyclone underflow is expected to be between 320% and 350% of primary mill feed tonnage.

Grinding balls will be added to the SAG and Ball mills via a kibble bucket, hoisted by overhead crane. Outdoor storage bunkers located adjacent to the process building will hold 125-100 mm balls for the SAG mill, and 65-50 mm balls for the ball mill.

The main items of equipment in Area 200 are described in Table 17-4 below.



Table 17-4: Grinding Circuit Main Equipment

Area 62200 - EQ #	Description	Notes	kW
62200-MSA-12	SAG Mill	7.32 m D x 3.65 m EGL	3,500
62200-XUL-30	SAG Mill Bearing Lube System		7.5
62200-XID-29	SAG Ball Mill Inching Drive		37
62200-PCD-34	Grinding Area Spillage Pump	4x4-12 C1, Model: VHS	6.0
62200-FCV-22	SAG Pebble Transfer Conveyor	Width 610 mm x 36 m	15
62200-FCV-26	SAG Pebble Recycle Conveyor	Width 610 mm x 15 m	11
62200-XLH-20	Liner Handler	McLellan 3.5K	30
62200-MBA-52	Ball Mill	4.7 m x 7.4 m EGL	2,700
62200-TBA-54	Mill Discharge Tank	20 m ³ , Square tank, pyramid bottom 45 deg	
62200-PCB-56 & 58	Mill Discharge Pump & S/By	250XHD PF5, Model: XHDM	150 x 2
62200-YAA-60	Ball Mill Cyclone Pack	6 x HC420 (4 in op, 2 s/by)	
62200-XUL-70	Ball Mill Lube System		7.5
62200-XID-69	Ball Mill Inching Drive		37

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. Tailings 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 the first of two agitated and baffled conditioner tanks in series. 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 eight 20 m³, 30 kW tank cells. 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 flow by gravity to the regrind mill discharge pump box, thereby bypassing the regrind circuit. A lower grade rougher concentrate will be collected from the remaining six cells of the bank and laundered into the concentrate regrind circuit. The vertical stirred mill based copper regrind circuit consists of a cyclone feed tank and pump, a 2 × VV250 (one operating, one stand-by), de-sliming cyclone pack, a regrind mill feed tank and pump and the 400 kW vertical stirred regrind mill. The copper regrind circuit will reduce the P_{80} of the low-grade copper rougher concentrate to 20-25 μm .

Tailings slurry from the final cell in the bank of copper roughers will be pumped via a sampling station to the Copper Rougher Tailings Tank. Copper rougher area spillage will be collected in a dedicated bund area and pumped to the primary conditioner tank. High grade 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 seven 4.3 m³, 8 kW u-shaped flotation cells. Cleaner tailings slurry will be further processed with a 4.3 m³, 8 kW u-shaped cleaner scavenger flotation cell prior to sampling and gravitating to the Copper Rougher Tailings Tank. Combined rougher and cleaner-scavenger tails will be pumped via sampling station to the zinc flotation conditioner tanks.

First cleaner concentrate will be collected in integral launders and pumped to the second stage cleaners for further upgrading. Second cleaner concentrate will be collected from a bank of four 4.3 m³, 8 kW u-shaped flotation cells and pumped 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.

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.

The automatic metallurgical samplers installed at the Cu flotation feed line, combined Cu rougher and cleaner tailings line and Cu concentrate line will be connected to an on-stream analyzer to measure Cu, Zn and Fe content. Samples from the two additional automatic metallurgical samplers installed at the Cu rougher tailings line and Cu cleaner-scavenger tailings line will be collected and analyzed in the laboratory on site.

The main items of equipment in Area 300 are described in Table 17-5 below.



Table 17-5: Copper Flotation Circuit Main Equipment

Area 62300 - EQ #	Description	Notes	kW
62300-XXR-01	Flotation On-Stream Analyzer - 6 streams	Model: PX2200 and MUX	
62300-XDB-06	Cyclone O/F Primary Sampler	Model SAS300	0.5
62300-TBA-08	Cu Conditioner Tank #1	28 m ³ live vol, cylindrical tank, flat bottom.	
62300-XSA-10	Cu Conditioner Tank #1 Agitator	STX-85, single blade	12
62300-TBA-12	Cu Conditioner Tank #2	28 m ³ live vol, cylindrical tank, flat bottom.	
62300-XSA-14	Cu Conditioner Tank #2 Agitator	STX-85, single blade	12
62300-XCB-16 to 30	Cu Rougher Cell 1 to 8	20 m ³ Tank Cell, Model: nextSTEP-20RT INT	30 x 8
62300-XDB-38	Cu Rougher Tails Sampler	Model: SAS300 & MPS075	0.75
62300-TAA-40	Cu Rougher Tails Tank	6.2 m ³ live vol, cylindrical tank, conical bottom	
62300-PCB-42 & 42	Cu Rougher Tails Pump & S/By	200XHD PF4S, Model: XHDR	38 x 2
62300-TAA-46	Cu Rougher HG Concentrate Tank	1.4 m ³ live volume, cylindrical tank, conical bottom	
62300-PFB-48	Cu Rougher HG Concentrate Pump	5x4x14 Frame 2, Model: SRL	19
62300-TAA-50	Cu Rougher LG Concentrate Tank	2.7 m ³ live vol, cylindrical tank, conical bottom	
62300-PCB-52	Cu Rougher LG Conc Pump	5x4x14 Frame 2, Model: SRL	38
62300-YAA-54	Cu Rougher Concentrate Cyclones	2 x VV250-20-0/A-B/40 (1 in op, 1 s/by) D2-25/2 Duplex	
62300-TAA-56	Cu Regrind Mill Feed Tank	Cylindrical tank, conical bottom	
62300-PCB-58	Cu Regrind Mill Feed Pump	50XHD PF1, Model: XHD	4
62300-MLS-60	Cu Regrind Mill	Model: VXP 1000	400
62300-XCB-62 to 74	Cu 1st Cleaner Cell 1 to 7	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8 x 7
62300-XCB-76	Cu 1st Cleaner Scav Cell 1	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8
62300-XDB-80	Cu 1st Cleaner Scav Tail Sampler	Model: MPS075	0.25
62300-XCB-82 to 87	Cu 2nd Cleaner Cell 1 to 4	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8 x 4
62300-PFB-88	Cu 1st Cleaner Conc Pump	SALA VCTP6	15
62300-PFB-90	Cu 2nd Cleaner Conc Pump	SALA VCTP6	12

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 tailings slurry. Concentrate slurry from the zinc cleaner circuit will be pumped to the zinc concentrate dewatering section prior to shipment off site as product. Tailings slurry from the zinc roughers will be pumped to a pyrite flotation stage for tailings desulphurization.

Tailings slurry will be pumped from the copper circuit via a sample station into to the first of two agitated conditioner tanks in series. The first conditioner will be primarily for pH control, 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 zinc rougher cells consists of six 20 m³ 30 kW tank cells. 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.

High grade zinc rougher concentrate will be collected from the first two cells and flow by gravity to the regrind mill discharge pump box, thereby bypassing the regrind circuit. A lower grade rougher concentrate will be collected from the remaining four cells of the bank and laundered into the zinc concentrate regrind circuit. The vertical stirred mill based zinc regrind circuit consists of cyclone feed tank and pump, a 2 x VV250 (1 operating, 1 stand-by), de-sliming cyclone pack, a regrind mill feed tank and pump and the 337 kW vertical stirred regrind mill. The zinc regrind circuit reduces the P₈₀ of the low grade zinc rougher concentrate to 20-25 µm.

Tailings slurry from the final cell in the bank of zinc roughers will be pumped via a sampling station to the first pyrite flotation rougher cell for tailings desulphurization. Zinc rougher area spillage will be collected in a dedicated bund area and pumped to the primary zinc conditioner tank.

High grade zinc rougher concentrate bypass slurry, combined 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 two banks of three 4.3 m³, 8 kW u-shaped cells 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 modulating dart valves. First cleaner concentrate will be collected and pumped to the second stage cleaners for further upgrading. Second cleaner concentrate will be collected from a bank of three 4.3 m³, 8 kW u-shaped cells and pumped to the third zinc cleaner circuit. Final stage zinc cleaning will be achieved via a bank of two 4.3 m³, 8 kW u-shaped flotation cells and zinc final concentrate will be collected in the integral concentrate launder prior to pumping 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.



Automatic metallurgical samplers installed at the Zn rougher tailings line, Zn cleaner tailings line and Zn concentrate line will be connected to an on-stream analyzer to measure Cu, Zn and Fe content.

The main items of equipment in Area 330 are described in Table 17-6 below.

Table 17-6: Zinc Flotation Circuit Main Equipment

Area 62330 - EQ #	Description	Notes	kW
62330-XDB-04	Copper Tails Primary Sampler	Model: PPV250	
62330-TAA-06	Zn Conditioner Tank #1	28 m ³ live vol, cylindrical tank, flat bottom	
62330-XSA-08	Zn Conditioner Tank #1 Agitator	STX-85, single blade	12
62330-TAA-10	Zn Conditioner Tank #2	28 m ³ live vol, cylindrical tank, flat bottom	
62330-XSA-12	Zn Conditioner Tank #2 Agitator	STX-85, single blade	12
62330-XCB-16 to 25	Zn Rougher Cell 1 to 6	20 m ³ Tank Cell, Model: nextSTEP-20 RT INT	30 x 6
62330-XDB-32	Zn Rougher Tails Sampler	Model: SAS300	1
62330-TAA-40	Zn Rougher HG Concentrate Tank	1.0 m ³ live vol, cylindrical tank, conical bottom	
62330-PFB-50	Zn Rougher HG Concentrate Pump	5x4x14 Frame 2, Model: SRL-S	19
62330-TAA-52	Zn Rougher LG Concentrate Tank	1.8 m ³ live vol, cylindrical tank, conical bottom	
62330-PCB-54	Zn Rougher LG Conc Pump	3x3x10 Frame 1, Model: SRL-S	19
62330-YAA-56	Zn Rougher Concentrate Cyclones	2 x VV250-20-0/A-D/30 (1 in op, 1 sby), D2-25/2 Duplex	
62330-TAA-58	Zn Regrind Mill Feed Tank	0.2 m ³ live vol, cylindrical tank, conical bottom	
62330-PCB-60	Zn Regrind Mill Feed Pump	40L XHD PF0, Model: XHDM	4
62330-MLS-62	Zn Regrind Mill	Model: VXP1000	337
62330-XCB-64 to 74	Zn 1st Cleaner Cell 1 to 6	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8 x 6
62330-XDB-78	Zn 1st Cleaner Tails Sampler	Model: MPS090	0.25
62330-XCB-80 to 83	Zn 2nd Cleaner Cell 1 to 3	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8 x 3
62330-XCB-85 & 86	Zn 3rd Cleaner Cell 1 & 2	4.3 m ³ Cell, Model: nextSTEP™ 150UT	8 x 2
62330-PFB-88	Zn 1st Cleaner Conc Pump	SALA VCTP6	12
62330-PFB-90	Zn 2nd Cleaner Conc Pump	SALA VCTP6	12
62330-PFB-91	Zn Final Conc Pump	SALA VCTP4	8



Area 62330 - EQ #	Description	Notes	kW
62330-TAA-92	Zn Rougher-Cleaner Tails Tank	5.9 m ³ live vol, cylindrical tank, conical bottom	
62330-PFB-93 & 97	Zn Rougher-Cleaner Tails Pump & S/By	200XHD PF4S, Model: XHDR	30 x 2

17.3.6 Pyrite Flotation and Pyrite Concentrate Thickening – Area 340

The pyrite flotation step ensures that the dry stack tailings facility contains material with no more than 0.5% sulphur, on average. The sulphide gangue minerals that remain in the tailings slurry after copper and zinc flotation are easily floated into a concentrate that can be blended with tailings cake and Portland cement prior to pumping underground for use as paste backfill in completed stopes.

Zinc rougher and cleaner tailings slurry is pumped to the pyrite rougher flotation circuit. A non-selective bulk sulphide collector reagent (Potassium Amyl Xanthate [PAX]) will be added to encourage the flotation of the remaining sulphide minerals.

The bank of pyrite rougher cells consists of four 20 m³ 30 kW tank cells. 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. The rougher concentrate from the four cells is pumped via sampling station to the 9 m diameter Pyrite Concentrate Thickener for dewatering and paste production. Tailings slurry from the final cell in the bank of pyrite roughers will be pumped via a sampling station to the tailings thickener for dewatering and dry stack tails production.

Pyrite concentrate obtained at the pyrite thickener underflow will be pumped at 55% solids concentration to the Paste Plant located 235 m away and stored in an agitated surge tank designed for 72 hours of retention time to accommodate the 5 days on, 3 days off, operating schedule anticipated for the Paste Plant. The pyrite concentrate overflow will be pumped to the process water tank.

Samples obtained from the two automatic metallurgical samplers installed at the pyrite rougher concentrate and tailings lines will be collected and analyzed in the laboratory on site.

The main items of equipment in Area 340 are described in Table 17-7 below.



Table 17-7: Pyrite Flotation and Pyrite Concentrate Thickening Main Equipment

Area 62340- EQ #	Description	Notes	kW
62340-XCB-02 to 08	Py Rougher Cell 1 to 4	20 m ³ Tank Cell, Model: nextSTEP-20 RT INT	30 x 4
62340-TAA-10	Py Rougher Tails Tank	4.4 m ³ , cylindrical tank, conical bottom	
62340-PCB-12 & 14	Py Rougher Tails Pump & S/By	150XHD PF4S, Model: XHDR	23 x 2
62340-PFB-18	Py Rougher Conc Pump	SALA VCTP6	19
62340-ACB-26	Py Conc Thickener	Ø9m x 2.2m SWH – HRT	4
62340-HPU-27	Py Conc Thickener Lift Drive		2
62340-PCB-28 & 30	Py Conc Thickener Underflow Pump & S/By	50XHD PF1, Model: XHDM	19 x 2
62340-TAA-32	Py Overflow Stand Pipe	1.3 m ³ , cylindrical tank, flat bottom	
62340-PCB-34	Py Overflow Pump	3x4-8 MTi, Model: 3196	4
62340-XDB-40	Py Concentrate Sampler	Model: MPS090	0.25

17.3.7 Tailings Dewatering and Disposal – Area 350

Tailings slurry will be pumped from the pyrite rougher tails tank via final tailings sampling station to the final tailings thickener. Thickener overflow will gravitate to the process water tank for re-use within the process. The tailings filtration and paste production plant will be remote from the process plant and will be located within a separate building.

Tailings thickener underflow slurry will be pumped from the thickener cone at 55% w/w solids concentration to the thickener underflow surge tank located outside the Paste Plant 235 m away from the process plant building. This tank will be agitated and will hold approximately 6 hours of thickened tailings production. The surge tank will be filled continuously and discharged semi-continuously to meet the tailings filter press cycle times and the paste demand schedules.

Thickened slurry from the storage tank will be pumped to the tailings filter for further dewatering. Cake obtained from the tailings filter will be discharged via the tailings cake conveyor and tailings transfer cake conveyor into the cake storage bin prior to dumping into trucks for placement on the nearby dry-stack tailings storage facility. The filtrate from the tailings filter will gravitate to the return water tank from where it will be pumped back to the tailings thickener or pyrite concentrate thickener as feed dilution water.

During periods when the paste plant is running, thickened non-acid generating tailings and thickened pyrite concentrate will be pumped to the paste plant feed tank for further vacuum filtration and mixing with Portland cement (4%), slurry and water to a specific



recipe to produce backfill paste. Backfill paste production will be pumped to the stopes via a network of paste distribution pipelines.

The Paste Plant area consists of one paste plant feed agitated tank, one Portland cement storage and addition system, and two 200 kW GEHO pumps for underground distribution (one duty, one standby).

Slurry within the paste plant feed tank is pumped to two cloth vacuum disc filters equipped with 200 kW vacuum pumps and 15 kW filtrate pumps. Cake from the disc filters is weighed on the disc filter discharge conveyor, which then reports to the paste mixer. The filter feed pump speed is controlled by the cake weight according to the paste recipe. The filtrate from the disc filter gets pumped to the return water tank to be re-used at the process plant.

The paste plant has two cement slurry systems that operate alternately. The cement slurry system consists of two 130 m³ (187 t) cement silos equipped with rotary valves. The rotary valves will discharge into the cement screw conveyors which feed into the colloidal tanks, outfitted with weightometers, where the cement is then mixed with clean water. While one cement slurry system is making a cement slurry batch, the other system will be dispensing cement slurry to the paste mixer via a screw feeder pump. This pump speed is controlled by flow according to the paste recipe.

The paste mixer is a 60 kW twin-shaft continuous paste mixer that will mix cake, cement slurry and water as per the designed recipe. The variable speed tailings dilution pump and top-up water line controlled by a water modulating valve are used to adjust the paste solids concentration at the mixer. The paste mixer feeds the paste hopper, which is outfitted with a level transmitter. The paste hopper feeds one of two 200 kW piston paste pump used for underground distribution. The paste pumps will be in a duty/standby arrangement.

The main items of equipment in Area 350 are described in Table 17-8 below.

Table 17-8: Tailings Dewatering and Disposal Main Equipment

Area 62350 - EQ #	Description	Notes	kW
62350-XDB-04	Plant Tailings Sampler	Model: PPV200 & MPS075	0.25
62350-ACB-08	Tailings Thickener	Ø22m x 2.4m SWH – HRT	8
62350-PCB-10 & 12	Thickener Underflow Pump & S/By	100XHD PF3, Model: XHDR	30 x 2
62350-TBA-16	Thickener Underflow Surge Tank	833 m ³ live vol, cylindrical tank, flat bottom	
62350-XSA-18	Non-sulphide Tailings Storage Tank Agitator	LHX-5160, single impellor	45



Area 62350 - EQ #	Description	Notes	kW
62350-TBA-19	PY Concentrate Storage Tank	2,830 m ³ live vol, cylindrical tank, flat bottom	
62350-XSA-20	PY Concentrate Storage Tank Agitator	LHX-6110, double impellor	94
62350-PCB-21	Paste Plant Feed Pump	125XHD PF3, Model: XHDR	15
62350-TBA-22	Paste Plant Feed Tank	144 m ³ , cylindrical tank, flat bottom	
62350-XSA-23	Paste Plant Feed Tank Agitator	STX-87, double impellor	12
62350-PCB-24	Tailings Filter Feed Pump	100XHD PF3, Model: XHDR	373
62350-PCB-26	Tailings Transfer Pump	100XHD PF3, Model: XHDR	23
62350-AAA-30	Tailings Filter	Model: M1500FBM-90-PP-RP-HS-225-50MM-FWASH	38
62350-TBA-40	Return Water Tank	96 m ³ , cylindrical tank, flat bottom	
62350-PCA-42	Return Water Pump & S/By	4x6-10G Mti, Model: 3196	12
62350-FCV-50	Tailings Cake Conveyor	48" W x 24m L	15
62350-FCV-52	Cake Transfer Conveyor (Stockpile)	30" W X 22m L	19
62350-BAA-54	Cake Storage Bin	130 tonne, cylindrical mass flow bin, 55 deg	-
62350-FCA-56	Cake Storage Bin Discharge Feeder	3' x 20' vibrating feeder	30
62350-FCV-15	Paste Plant Cake Conveyor	48" W x 30m L	22
62350-PCB-58 & 43	Disc Filter 1 & 2 Feed Pumps	Model: HM100	22 x 2
62350-AAA-62 & 68	Disc Filter 1 & 2		
62350-PVA-65 & 71	Disc Filter 1 & 2 Vacuum Pumps	Model: Nash 2BE440	200 x 2
62350-HAC-72 & 73	Paste Plant Compressor 1 & 2	Model: 7bar	110 x 2
62350-TBA-79 & 11	Colloidal Tank 1 & 2	Model: VM1000 Duo	30 x 2
62350-XXM-80	Paste Mixer	Model: MDC 501 L twin shaft mixer	60
62350-BAA-81 & 97	Cement Silo 1 & 2	130 m ³ (187 t) - 5 mD x 9.5 mH, 60 deg cone, bolted	
62350-FBD-83 & 84	Cement Screw Feeder 1 & 2	273 mm diameter	6 x 2
62350-FBD-85	Cross-feed Screw Conveyor	273 mm diameter	6
62350-PPA-90	Binder Pump	Model: 4S-22-15, 4-stage progressive cavity.	22
62350-BAA-85A & B	Paste Hopper A & B	silo, 453 kg	
62350-PPD-86	Paste Pump & S/By	30 bar, Model: GEHO 21000	200 x 2

17.3.8 Copper & Zinc Concentrate Dewatering – Area 360

Copper flotation concentrate will be pumped to a sampling station before entering the 8 m diameter concentrate thickener for controlled dewatering. The copper concentrate thickener will be equipped with vendor-supplied rake lift, bed level detection and bed



mass monitoring. Thickener overflow will gravitate to the copper spray water tank for recycling within the copper circuit, and thickener underflow will be withdrawn from the cone at 55% solids concentration by peristaltic pumps and sent to the agitated copper thickener underflow storage tank or recycled to the thickener feed if the density is too low. The Copper Thickener U/F Storage Tank surge capacity will be approximately 12 hours. Copper thickener area spillage will be recovered by pumping back to the copper concentrate thickener.

Zinc flotation concentrate will be pumped to a sampling station before entering the 8 m diameter concentrate thickener for controlled dewatering. The zinc concentrate thickener will be equipped with rake lift, bed level detection and bed mass monitoring. Thickener overflow will gravitate to the zinc spray water tank for recycling into the zinc circuit, and thickener underflow will be withdrawn from the cone at 55% solids concentration by peristaltic pumps and sent to the agitated zinc thickener underflow storage tank or recycled to the thickener feed the density is too low. The Zinc Thickener U/F Storage Tank surge capacity will be approximately 12 hours. Zinc thickener area spillage can be recovered by pumping back to the zinc concentrate thickener.

Concentrate stored in the copper and zinc storage tanks will be pumped in batches to a single pressure filter for dewatering. The pressure filter will be served by high pressure (12 and 16 bar), instrument-grade air, allowing the following:

- 1.0 Inflation of the filter plate membranes to generate a pressure squeeze (16 bar air);
- 2.0 Further “air blow” dewatering of the compressed cake (12 bar air).

Filtrate from the pressure filter will be directed to a filtrate de-aeration chamber and into a filtrate / manifold flush tank. This tank overflows to the filtrate pump which transfers filtrate back to the appropriate concentrate thickener for recycling. 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 recycled to thickener feed. The 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 onto a reversible cake transfer belt which transports cake to the concentrate loadout area. 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 17-9 below.



Table 17-9: Copper & Zinc Concentrate Dewatering Main Equipment

Area 62360 - EQ #	Description	Notes	kW
62360-XDB-04	Final Cu Conc Sampler	Model: PPV150	
62360-ACB-08	Cu Concentrate Thickener	Ø8m x 2m SWH – HRT (Bolted)	4
62360-PPA-10 & 12	Cu Underflow Pump & S/By	Albin, ALHX80, NR Hose	12 x 2
62360-TBA-16	Cu Spraywater Tank	23.3 m ³ live vol, cylindrical tank, flat bottom	
62360-PCB-18	Cu Spraywater Pump	3x4-8G MTi, Model: 3196	6
62360-TBA-22	Cu Thickener U/F Storage Tank	143 m ³ , Cylindrical tank, flat bottom, 7157 kg	
62360-XSA-24	Cu Concentrate Storage Tank Agitator	STX-87, double impellor	12
62360-PCA-30	Cu Filter Feed Pump		10
62360-AAA-34	Concentrate Filter	Larox PF 72/96 M60 1 45	105
62360-TBA-37	Flush Water Tank		
62360-PCB-38	Cloth Wash Pump	Model: Grundfos CR 32-9-2, IEC	19
62360-HAC-40	Pressing Air Compressor	Model: Atlas Copco GR110	110
62360-HEA-41	Pressing Air Filter	Max. pressure: 20 bar	
62360-HFA-49	Pressing Air Receiver Type	5 m ³ , 30 Bar	
62360-HAC-42	Air Compressor	10 bar, Model: Atlas Copco GR110	355
62360-HFA-45	Drying Air Receiver	16 bar, 30 m ³	
62360-ZAC-46	Filtrate De-Aeration Tank		
62360-TAA-48	Filtrate Transfer Tank	XX m ³ , cylindrical tank, flat bottom,	
62360-PCB-50	Filtrate Transfer Pump	23.50m ³ /h, TDH 15.08m, 2x3-8 MTi, Model: 3196	3
62360-FCV-54	Cake Conveyor - Shuttle	42" W x 10m L, reversible	11
62360-XDB-64	Final Zn Conc Sampler	Model: PPV150	
62360-ACB-68	Zn Concentrate Thickener	Ø8m x 2m SWH – HRT	4
62360-PPA-70 & 72	Zn Underflow Pump & S/By	Albin ALHX80, NR Hose	12 x 2
62360-PCD-74	Zn Area Spillage Pump	3x3-10 V3 , Model: VRS	8
62360-TBA-76	Zn Spray water Tank	16 m ³ , cylindrical tank, flat bottom	
62360-PCB-78	Zn Spray water Pump	3x4-7 MTi, Model: 3196	4
62360-TBA-80	Zn Thickener U/F Storage Tank	143 m ³ , cylindrical tank, flat bottom	
62360-XSA-82	Zn Concentrate Storage Tank Agitator	STX-87, double blade	12
62360-PCA-88	Filter Feed Pump (Zn)	2 x 1.5 Warman AH Slurry Pump	8



17.3.9 Reagents – Area 800/820

Reagents will be delivered to site by road and stored in the warehouse. Reagents will be moved to the plant reagents area on an as-needed basis. The speed of the reagents dosing pumps can be adjusted remotely from the control room. The operator can adjust reagents based upon viewing the cameras and automated flow measurement. Reagents required for the process plant are listed below:

Frother 1 – MIBC

Liquid MIBC will be delivered in 900 kg IBC totes to site by road. Peristaltic hose pumps meter frother solution from a single IBC tote in the reagent area to several addition points throughout the plant. Approximately 37.8 tonnes of MIBC will be consumed per year.

Frother 2 – Polyfroth H57

Liquid H57 will be delivered in 1,000 kg IBC totes to site by road. Peristaltic hose pumps meter frother solution from a single IBC tote in the reagent area to several addition points throughout the plant. Approximately 151.2 tonnes of H57 will be consumed per year.

Promotor – Cytec Aero 5100

Liquid 5100 will be delivered in 950 kg IBC totes to site by road. Peristaltic hose pumps meter promotor solution from a single tote in the reagent area to several addition points throughout the plant. Approximately 24.2 tonnes of 5100 will be consumed per year.

Collector – Cytec AERO 3894

Liquid 3894 will be delivered in 1,000 kg IBC totes to site by road. Peristaltic hose pumps meter collector solution from a single drum in the reagent area to several addition points throughout the plant. Approximately 56.7 tonnes of 3894 will be consumed per year.

Collector – Cytec AERO 3501

Liquid 3501 will be delivered in 1,100 kg IBC totes to site by road. Peristaltic hose pumps meter collector solution from a single drum in the reagent area to several addition points throughout the plant. Approximately 30.2 tonnes of 3501 will be consumed per year.

Collector - Potassium Amyl Xanthate (PAX)

PAX briquettes will be delivered in 1,000 kg supersacks to site by road. Bags will be mixed with water in a mixing tank and transferred to a holding tank within the reagent area. Peristaltic metering pumps will dose collector solution from the holding tank in the reagent area to several addition points throughout the plant. Approximately 52.9 tonnes of PAX will be consumed per year.



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 automated. Lime will be pumped throughout the plant via a ring main distribution system, with flow-metered take-offs for each dosing point. Approximately 2,437 tonnes of quicklime will be consumed per year.

Reagent spillage will be pumped to the tailings tank for disposal in the tailings dam.

Activator – Copper Sulphate

CuSO_4 powder will be delivered in 1,000 kg supersacks to site by road. Bags will be mixed with water in a mixing tank and transferred to a holding 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. Approximately 491 tonnes of CuSO_4 will be consumed per year.

Depressant – Sodium Metabisulfite

$\text{Na}_2\text{S}_2\text{O}_5$ powder will be delivered in 1,000 kg supersacks to site by road. Bags will be mixed with water in a mixing tank and transferred to a holding tank within the reagent area. Peristaltic metering pumps will dose depressant solution from the storage tank in the reagent area to several addition points throughout the plant. Approximately 4,158 tonnes of $\text{Na}_2\text{S}_2\text{O}_5$ will be consumed per year.

CMC Depressant – PE26

Depressant (CMC) powder will be delivered to site in 900 kg bags by road. Bags will be lifted by hoist and added to the depressant powder hopper. 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 1% solution will be fed forward to the required holding tank.

The CMC batch mixing sequence will be fully automatic. 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.

Approximately 49.9 tonnes of PE26 will be consumed per year.



Flocculant – Magnafloc 10

Flocculant powder will be delivered to site in 800 kg supersacks by road. 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 holding tank or recycled back into the mixing tank to aid mixing. Once mixed, the flocculant should be left for several hours to hydrate. The holding tank will provide sufficient volume for storage of flocculant whilst the mixed batch hydrates in the mixing tank. From the holding tank, flocculant will be pumped through a ring main for dosing at each of the four thickeners.

Approximately 30.2 tonnes of flocculant will be consumed per year.

Portland Cement

Portland Cement powder will be delivered to site in bulk tankers and stored in two load cell-mounted bulk storage hoppers. Cement powder will discharge into the cement screw conveyors which feed into the colloidal tanks, where the cement is mixed with water. A screw pump will dose the cement slurry to the paste mixer.

Approximately 33,264 tonnes of Portland cement will be consumed per year.

17.3.10 Water and Air Recirculation – 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 tailings and pyrite concentrate thickeners 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.

Clean water will be recovered from the effluent treatment plant and piped to the fresh/fire 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 mechanical cooling systems. The fresh/fire water tank will retain a dedicated feed the diesel-powered fire water pump from a separate (lower) nozzle, thus guaranteeing water availability for emergencies.

Plant and instrument air will be provided by two compressors. Air quality will be maintained by filters. Instrument air will be desiccated using a drier. Receivers are provided for plant and instrument air, and will be fitted with automatic condensate drains and pressure safety relieve valves. 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 line.

The main items of equipment in Area 900 are described in Table 17-10 below.

Table 17-10: Water and Air Recirculation Main Equipment

Area 62900 - EQ #	Description	Notes	kW
62900-TBA-02	Process Water Tank	370 m ³ live vol, cylindrical tank, flat bottom	
62900-PCB-04 & 06	Process Water Pump & S/By	6x8-13XLTi, Model: 3196	56 x 2
62900-PEA-08	Fire-Water Pumping skid	5.600 Lpm electric & diesel fire pumps 60 Lpm jockey	150
62900-TBA-12	Clean/Fire Water Tank	276 m ³ live vol cylindrical tank, flat bottom	
62900-PCC-14 & 16	Clean Water Pump & S/By	3x4-8 MTi , Model: 3196	6 x 2
62900-PCC-13	Gland Water Pump & S/By		3 x 2
62900-HAC-18 & 20	Process Plant Air Compressor S/By		75
62900-HBB-34 & 38	Cu/Zn Flotation Blower Roughers & S/By		132 x 2
62900-HBB-40 & 42	Cu/Zn Flotation Blower Cleaners & S/By		45 x 2
62900-PCD-44	Contact Water Pond Pump		37



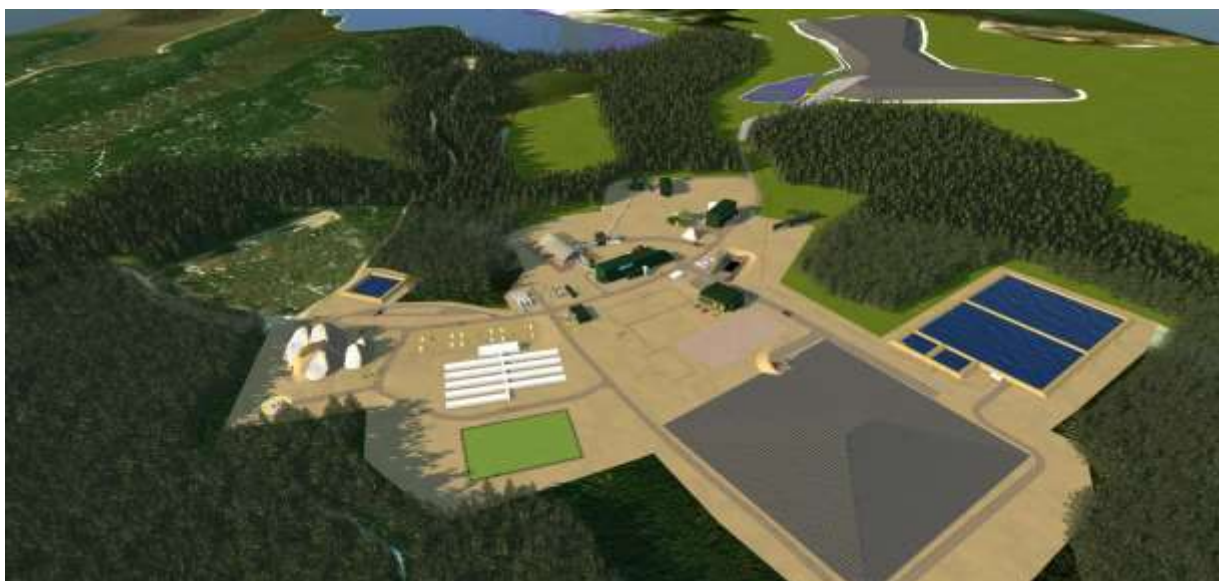
18.0 PROJECT INFRASTRUCTURE

The McIlvenna Bay Project will include a relatively compact site (Figure 18-1) with major features including the tailings storage facility, ore and waste dumps, water treatment infrastructure, and buildings such as admin and dry, gate house, and truck shop. The process facilities will include primary crusher building, covered ore stockpile, mill building and paste plant building. The site will contain a 350-person camp with associated canteen, recreation, and domestic water treatment infrastructure.

The site is connected to public road SK-106 via an existing all-season 18 km gravel road. 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, with transfer to rail network.

Construction power will be received by tie-in to existing grid and supplemented during operations by on-site power generation, eventually replaced by grid connected transmission line.

Figure 18-1: Rendering of McIlvenna Bay Surface Infrastructure (Halyard, 2022)



18.1 Control System and Instrumentation Levels

The control system allows for a high degree of automation to minimize operator intervention. The main control room will be located adjacent to the main process building and house the DCS hardware, with small local control rooms at the crushing and paste plants.



An estimated I/O count of approximately 600 was determined to size of the control system. The main characteristics of the control philosophy consist of the following:

- Control of process equipment via PCSs and manual operation
- Hardwired interlocks for personnel safety
- The system is intrinsically safe in that all stop buttons are active no matter what mode the equipment is in
- Monitoring at all times of select operating conditions on the SCADA system and recording selected information for data logging or trending
- Alarm inputs to the PCS that are fail safe in operation unless otherwise specified, e.g. the signal reverts to zero for the fault condition to exist

18.2 Existing Infrastructure

The McIlvenna Bay Project is in close proximity to a previous (reclaimed) quarrying operation and benefits from an existing access road, cleared laydown areas, and a SaskPower distribution line. Foran has constructed various infrastructure to support ongoing advanced exploration and exploration activities (ADEX), including all-season camps with combined sleeping accommodation for 88, fuel storage, communications, process water supply, septic, canteen, core shack, construction office, mine dry and first aid / mine rescue trailer (Figure 18-2).

Figure 18-2: Existing Exploration Camp



18.2.1 Advanced Exploration (ADEX)

A significant portion (16.2 ha) of the site (shown in Figure 18-4) has been cleared and grubbed as part of ADEX development. The box cut excavation and jet grouting of the sandstone layer has commenced and the contractor for portal development has been mobilized to site. A temporary truck shop has been erected and laydown area prepared. The first phase of the lined waste rock dump is constructed, and stage 1 of the contact water pond is in final permitting, along with treated water cells and associated water treatment plant. The main haul road and minor access roads are built. The existing SaskPower distribution line has been energized and power is distributed via a containerized construction power e-house. McIlvenna Bay ADEX site plan is presented in Figure 18-3 and depicts surface infrastructure completed or currently under construction.

Figure 18-3: Existing Infrastructure for ADEX (Foran, 2022)



Box cut for portal development



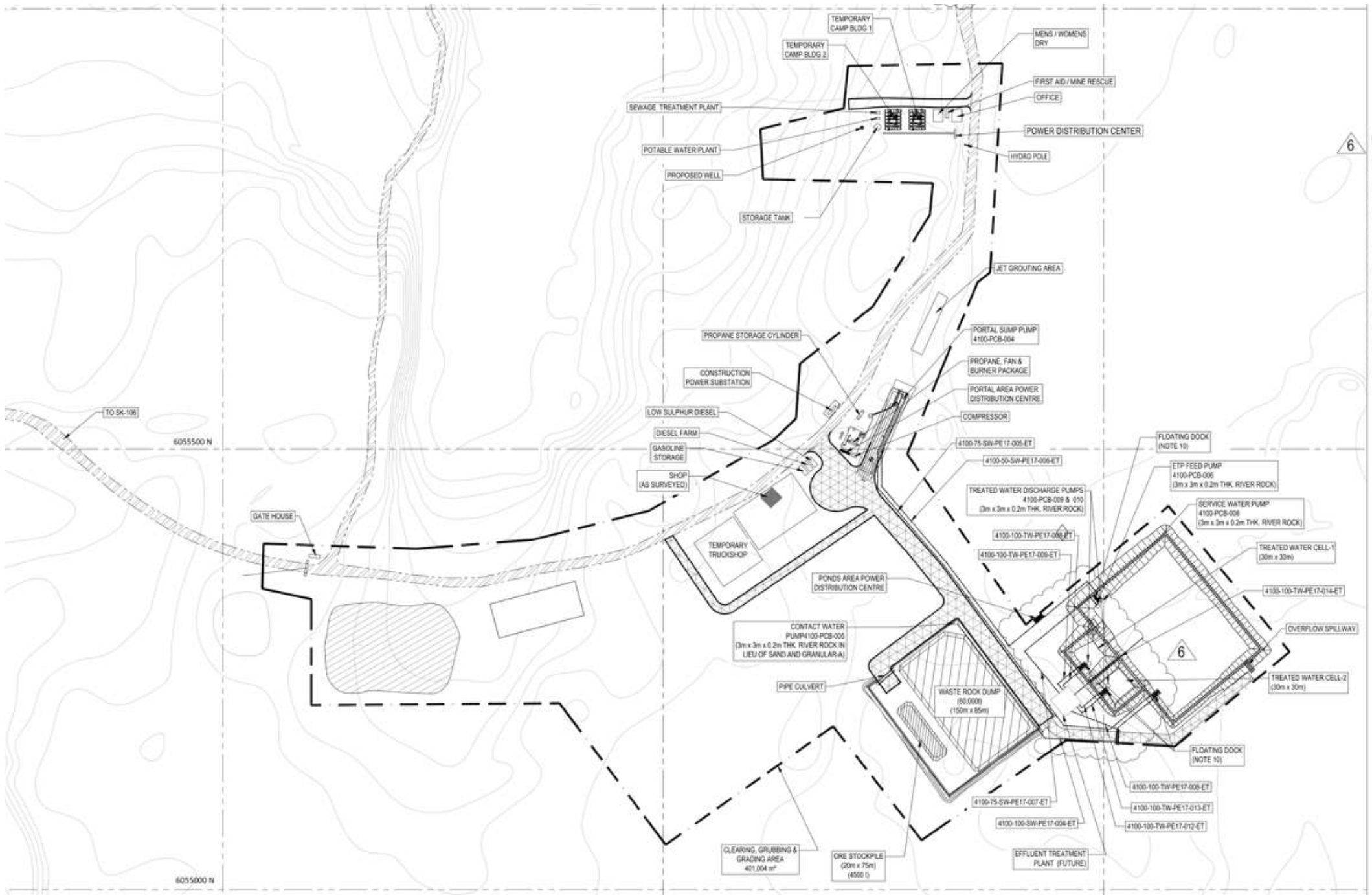
Temporary truck shop



Stage 1 waste/ ore pad



Figure 18-4: ADEX Site Plan (Halyard, 2022)



18.3 Project Site Preparation

The proposed McIlvenna Bay site layout is presented on Figure 18-5. The site layout is designed to be compact and centralized around the mine portal, and to utilize infrastructure developed during the ADEX program, with the objectives of minimizing the environmental footprint, and minimizing underground to surface haul distances. Prior to construction, the balance of site (32.2 ha) will be cleared and grubbed, and soil and organics will be stockpiled for future surface reclamation. There is generally 0 m to 2 m of overburden on site underlain by dolomite rock.

The contractor and mine laydown areas and temporary truck shop will be utilized for contractor site establishment. During this time the balance of site, mainly north of the main access road, can be cleared and grubbed in preparation for the process building erection once the necessary work authorizations are received.

Clearing and grubbing will include the cutting and disposing of trees, brush, roots, and stumps. The area around infrastructure will be roughly graded prior to construction. Merchantable timber will be harvested and sold by *Mee-Toos*, a wholly owned subsidiary of *PBCN*.

18.3.1 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.



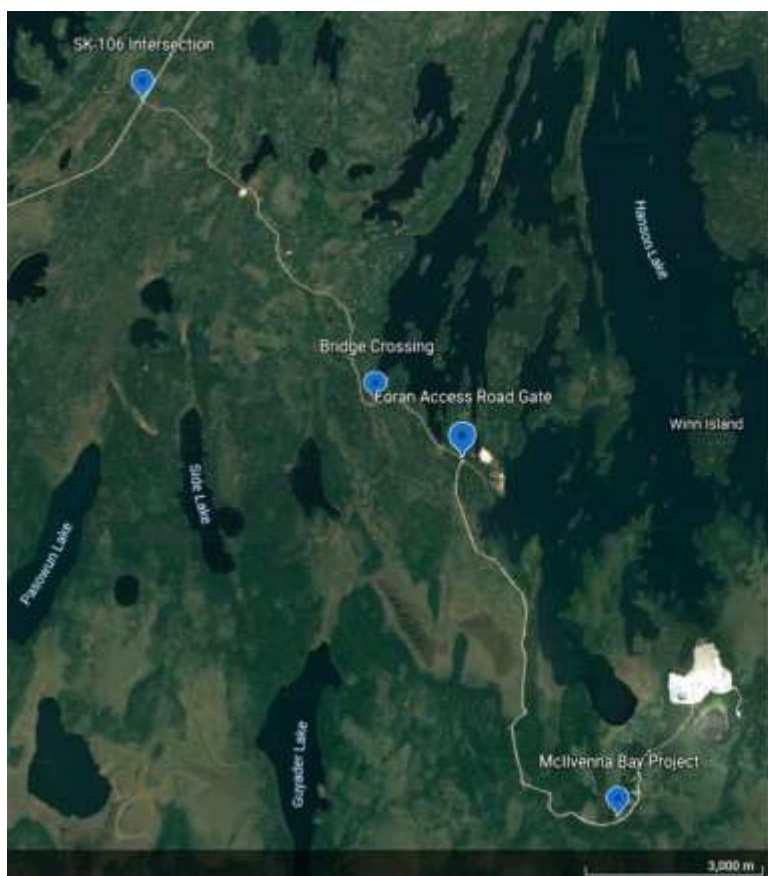
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18.4 Project Access

18.4.1 Mine Road

The existing gravel road between the mine site and SK-106 (Figure 18-6) will be used for transportation of mine personnel and general supplies over 18 km. 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.

Figure 18-6: Mclivenna Bay Site Access Road (Google Earth, 2022)



SK-106 intersection



Foran Access Road Gate



There is a bridge located on the mine access road at coordinates 54.689 °N, 102.886 °W (Figure 18-7). The 9 km span of road between SK-106 and the Foran access gate, 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 has been in use during the ADEX operation. SK MH&I has confirmed the bridge was constructed in 2004 and received SEAL-750 (load rating exceeds provincial capacity requirements). Provincial safety inspections are schedule every 2 years per provincial regulation. The 2019 route survey confirmed the bridge is suitable for it intended use without restriction.

Figure 18-7: Mcllvenna Bay Site Access Road Bridge



18.4.2 Highway 106

Traffic on public Highway SK-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. Operations and administration personnel will be bussed to/from site on a rotational basis, using contract transportation services. New turn in/turn out lanes and road lighting at the intersection of the mine access road and Highway SK-106 is recommended to assist with visibility, given the increased level of traffic that will be maneuvering in this area at night.

18.4.3 Mine Service Roads and Site Parking

A light vehicle parking lot for mine personnel and contractors will be located adjacent to the administration / dry complex and on-site camp. It will be inside the guard house gate but separated from heavy underground haul truck traffic. The parking lot will be equipped with receptacles for block heaters. The parking lot will accommodate up to 156 light vehicles.



A small visitors parking lot directly in front of the administration / dry complex will accommodate 25 light vehicles.

18.4.4 Helipad

A 35 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.

18.5 Buildings and Ancillary Items

18.5.1 Temporary Construction Facilities

Mobilization, demobilization, temporary construction facilities and mobile equipment required during the construction phase will be contractor supplied. Foran will provide construction power, water, on-site accommodation, and catering in the 350-person camp.

18.5.2 Truck Shop / Warehouse

The truck shop / warehouse will be housed within a 42 m W × 45 m L × 15 m H pre-engineered building. A 7 m W × 28 m L × 11 m H lean-to structure is attached at the east end of the building for oil / waste oil storage. The 21 m W × 45 m L mezzanine level will house offices, meeting rooms, washrooms, and kitchenette, as well as electrical and mechanical utility rooms and additional storage. The building will be complete with metal cladding, insulation, HVAC and snow guards. The ground floor has four (4) truck bays with overhead doors designed to accommodate mining mobile equipment (haul trucks and drills). The balance of the ground floor (880 m²) consists of a compressor room, tool rooms, EV battery storage and charging, welding stations, wash area, and warehousing.

Over the vehicle maintenance bays are two (2) overhead cranes (20/5 t). The mezzanine level utility area is serviced by a 2 t hoist with access to outdoor laydown area.

The pre-engineered building will be installed on a concrete perimeter wall with footings under each of the pre-engineered building's columns. Footings will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. The slab-on-grade, slopped in the maintenance area, will contain and direct spillage toward in-ground sumps equipped with pumps. The wash area will be isolated by low curb walls and will contain dedicated in-floor sump pumps with oil / water separator.



18.5.3 Administration Office / Mine Dry

The administration office and mine dry are combined in a two-story modular building 18 m W × 36 m L. The building is constructed of a wood frame with steel cladding and will be elevated on a precast concrete block foundation atop a compacted granular pad to allow crawl space utility access. The building will be shop assembled to the greatest extent practical for shipping and comes furnished with the following.

- Two internal staircases
- HVAC units and internal ducting
- Interior finishes including flooring, painting, lighting / emergency lighting, internal partition walls, interior and exterior doors and glazing
- Fire detection and protection system
- Internal plumbing and water filtration system
- Electrical and communications wiring

Mine Dry

The lower level (670 m²) will house the men's and women's dry, canteen and lunchroom, mine rescue, laundry and janitorial. The dry will accommodate approximately 250 personnel with locker, washroom and shower facilities.

Administration Office

The upper level (670 m²) will house reception, office units / open office area, meeting rooms, training room, kitchenette, washrooms, electrical, IT server, and mechanical utility rooms.

18.5.4 First Aid

A first aid office will be incorporated into the mine dry complex.

18.5.5 Security Gatehouse and Truck Scale

The existing gatehouse will remain in place during the construction and production phases. The gatehouse is located at the entrance of the mine site and controls 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 in the gatehouse.



18.5.6 Temporary Explosives Storage

The temporary surface explosives storage magazines will be located north of the mine site a minimum of 650 m away from adjacent structures with a partial earth berm enclosure. Emulsion will be stored in a 2.4 m W × 6 m L × 2.5 m H type 4 outdoor magazine as per the Natural Resources Canada – Explosives Regulatory Division's requirements. The emulsion magazine will have the capacity to store 12,000 kg of product. This facility will be decommissioned once the underground magazine operational.

A separate smaller storage magazine cabinet will be attached to the emulsion magazine to be used for detonator storage. This magazine will have a capacity of 500 kg. Each of the magazines will be complete with lockable doors.

18.5.7 Mine Rescue

A 27 m² mine rescue office will be incorporated into the mine dry complex on the lower level.

18.5.8 Fuel Storage and Distribution

A fuel storage and distribution facility will be located adjacent to the portal. 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 two 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 EV haul vehicles). Diesel for the generators will be gravity fed into the local tank within the generator enclosure.

18.5.9 Waste Disposal

Waste from site such as plastic, paper, cardboard, scraps, and rubber will be collected and stored in the garbage disposal area. The waste will be picked up on a set schedule and sent offsite for disposal. Hazardous materials such as spent oil, batteries, paint,



compressed gas cylinders, and corrosive products will be disposed of adequately as per Foran's Environmental Department's Standard Operating Procedure. Foran's Health and Safety Department will ensure signage, SDSs, 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.

18.5.10 Mine Laydown Area

A 10,100 m² laydown area will be constructed of compacted granular material and located south of the truck shop. This laydown area will serve as the construction staging area during the surface and portal construction phases.

18.6 Process Buildings

18.6.1 Head Frame / Collar House Building

The headframe will provide access for personnel entering the mine, in addition to hoisting ore to the surface. The design includes a backlegs, collar house, head sheaves, skip dumps, and ore bin. A deflector gate will direct hoisted material to a bunker on the ground.

The collar house will have sufficient room to stage personnel to be sent UG. There will be suitable clearances for removing the skips for service and maintenance on the east side of the collar house.

The tower of the headframe will stand 52.0 m tall and will house the head sheaves and skip-dumping equipment. The headframe will be 13 m W × 23 m L and will be constructed on site with structural steel (approximately 430 tonnes). The headframe will be clad with 75 mm thick RSI 4.0 insulated metal panel.

The headframe will be supported primarily by the shaft collar, which will consist of a concrete liner tied into the basement rock below the sandstone and regolith layers. The remainder of the headframe will be installed on a concrete perimeter wall with footings under the columns. The footings will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. The headframe will be supported by back legs. The back legs will be anchored to footing which will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required.

The collar house will be installed on a concrete perimeter wall with footings under the columns of the collar house and will be tied to the headframe. The footings will be



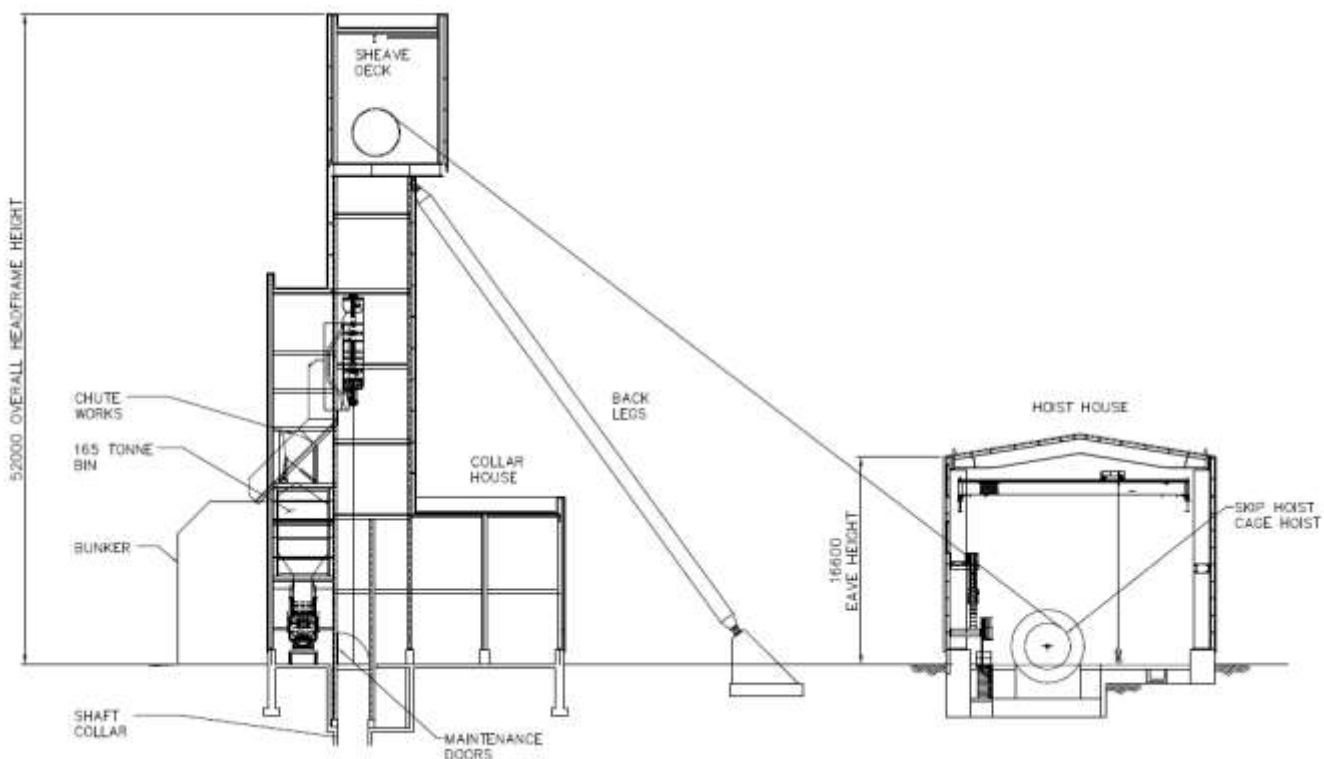
founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. A concrete slab-on-grade in the collar house will facilitate removal of the skips and cage, and access to the shaft. Layout drawings of the headframe and hoist house are shown in Figure 18-8.

18.6.2 Hoist House Building

Similar to the headframe, the hoist house will be constructed on site with structural steel. This hoist house will be 20 m W × 39 m L × 16.6 m H. The hoist house will be clad with 75 mm thick RSI 4.0 insulated metal panel.

The hoist house will be installed on a concrete perimeter wall with footings under the columns. The footings will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. The production and service hoists will be anchored to concrete bases. A concrete slab-on-grade will be installed in the hoist house. Layout drawings of the hoist house and headframe are shown Figure 18-8.

Figure 18-8: General Arrangement of Headframe and Hoist House



18.6.3 Primary Crushing Building

The primary crushing circuit will be housed within a 14 m W × 15 m L × 17 m H pre-engineered building which is intended to keep the equipment from being exposed to the elements. The building will be adjacent to a 6 m high concrete retaining wall to allow for dumping of underground haul trucks into the surge bin. The building will be complete with metal cladding, snow guards, and dust collection. Overhead doors will accommodate mobile equipment access into the building.

The crushing building will be equipped with a 10 t overhead crane running the span of the building to service the crushing equipment and laydown area.

The pre-engineered building will be installed on a concrete perimeter wall with footings under the columns of the pre-engineered building. The footings will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. A concrete slab-on-grade in the crushing plant will facilitate cleanup of spillage.

At the south end of the building is the elevated control room located adjacent to the modular crushing plant. Operators will have line of sight to the jaw crusher and rock breaker and have direct access into the crushing structure.

A prefabricated electrical room will house MCCs and switchgear for process equipment. The e-room will be elevated for bottom cable entry, equipped with HVAC and double doors for maintenance access.

The ore will enter the process plant via a large opening in the east wall sized for haul truck dumping. For future consideration, space has been designated for a feed conveyor to enter the north side of the building to allow for discharging into the surge bin. The discharge conveyor exits from the west building wall.

The crushed ore transfer tower will be located adjacent to the crushing building and be covered by a steel-clad roof.

18.6.4 Stockpile Cover & Reclaim Tunnel

The copper and zinc crushed ore stockpiles will be housed within a common fabric building 34 m W × 62 m L × 15 m H. The building will be complete with fire retardant polyethylene fabric cladding over galvanized tubular truss frame, with roof openings for feed conveyor discharge. The building does not require insulation or heat and will have passive ventilation. The fabric is translucent, so lighting is not required. The building will be founded on concrete footings with a 1.5 m high interior perimeter wall to protect the fabric from rolling ore on the stockpile. The end wall will be equipped with overhead door to accommodate mobile equipment.



Each stockpile will have 6,000 t capacity and will be reclaimed by vibratory feeder located in a corrugated arch multiplate steel tunnel. The reclaim and conveyor portions of the tunnel will be 6 m diameter × 66 m long with a concrete strip footing and slab-on-grade. The escape tunnel will be round multiplate steel 3 m diameter × 16 m long with slab-on-grade to form a level floor. Both tunnels will be backfilled around the exterior, with minimum 1.5 m cover above the tunnel roof. The base of the tunnels is located at bedrock surface to limit rock blasting and allow the tunnel to be self-draining.

18.6.5 Process Plant Building

The process plant will be housed within a 24 m W × 107 m L × 23 m W pre-engineered building. A 24 m W × 24 m L × 13.5 m H lean-to structure is attached at the east end of the building for concentrate loadout and storage. A second 10 m W × 27 m L × 9 m H lean-to structure is attached at the north side of the building for reagent storage / dosing and plant utilities. The building will be complete with metal cladding, insulation, HVAC, and snow guards. Overhead doors accommodate mobile equipment access to the grinding and reagents areas, as well as drive-through concentrate load-out.

The process building will be equipped with two (2) overhead cranes (20 t and 10 t) along the span of the building to service the grinding, flotation, and filtration areas. The cranes will share common runway beams and rails. Jib cranes or hoists will be used in partitioned areas such as the reagents store, and under the concentrate filter and thickener bridge.

The pre-engineered building will be installed on a concrete perimeter wall with footings under the columns of the pre-engineered building. The footing will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. A sloped concrete slab in the process plant will contain and direct spillage toward sumps to facilitate clean up. A drive-in sump in the grinding area will accommodate a skid steer loader. The reline floor will consist of an elevated slab-on-deck to allow maneuvering of the reline machine and forklifts. Discrete process areas are isolated by low curb walls and will contain dedicated in-floor sump pumps.

At the west end of the building is the modular control room located outside the process plant at mill operating floor elevation. Operators will have direct access into the process building via covered walkway.

Two (2) prefabricated electrical rooms will house MCCs and switchgear for process equipment. The grinding e-room is located below the control room, and the concentrate e-room along the north building wall. Both e-rooms are elevated for bottom cable entry, equipped with HVAC and double doors for maintenance access.



The ore will enter the process plant via a mill feed conveyor on the west end of the building. The process will generally flow from the west toward the east end of the building where the concentrate will be discharged.

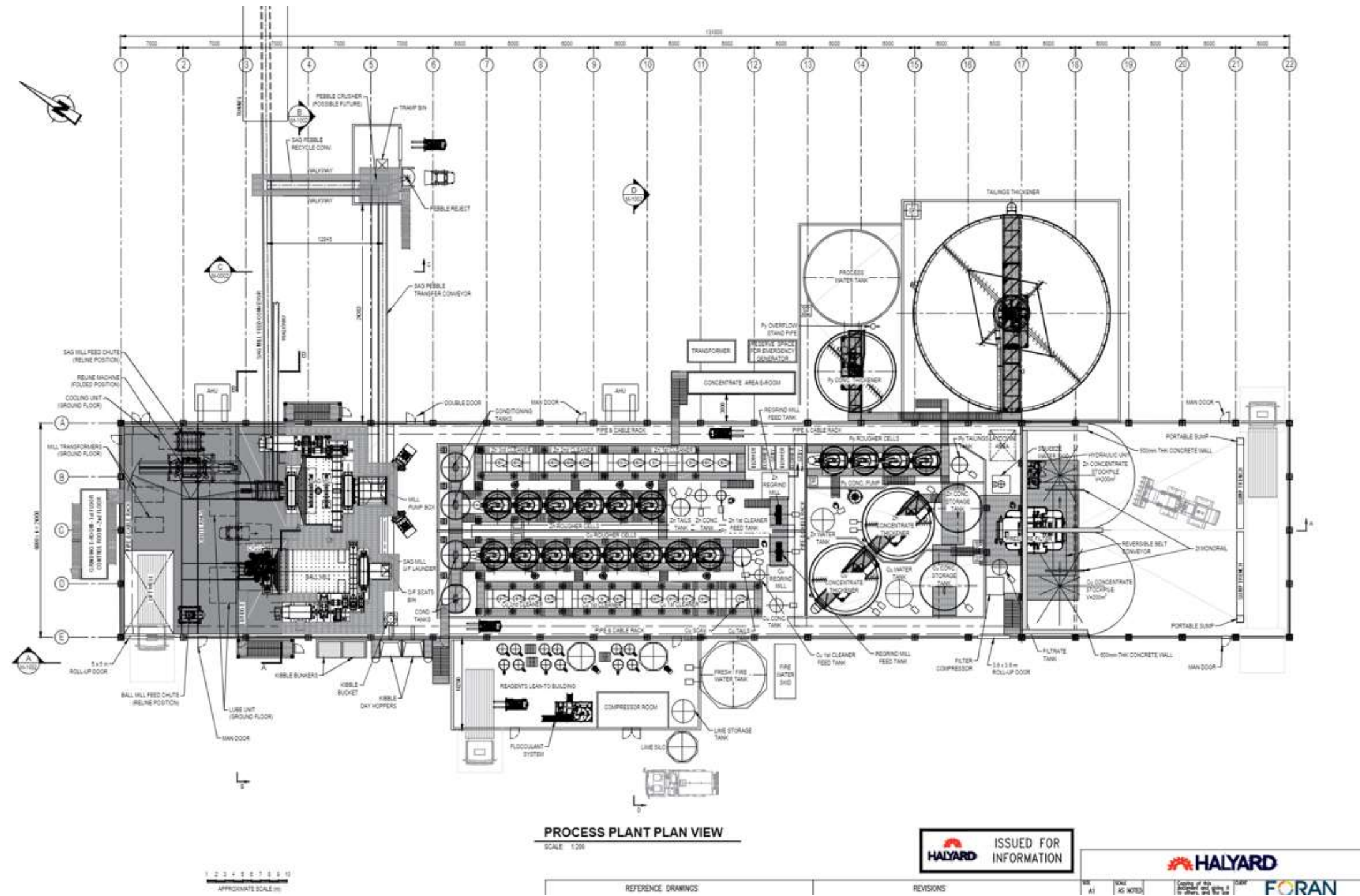
A small enclosure will house the pebble transfer tower and there is a space allowance for a pebble crusher if required in future. This building has metal cladding, a slab-on-grade, and 5 t maintenance hoist.

There will be several pieces of equipment on the exterior of the process plant, such as thickeners, process and freshwater tanks, lime silo, and grinding media storage. The process water tank and thickeners will be housed within a concrete containment berm sized for 110% of the largest vessel's volume. The concrete for these exterior pieces of equipment will be separate from the concrete works of the pre-engineered building to allow for independent erection of the pre-engineered building. Construction activities will continue through the winter season inside the building, utilizing the overhead cranes.

Layout drawings of the process plant building are shown in Figure 18-9.



Figure 18-9: Process Plant Building Plan (Halyard, 2022)



18.6.6 Assay Lab

A containerized assay lab will be located adjacent to the process plant building. The 2.4 m W × 12.2 m L × 2.6 m H building will come complete with HVAC, dust extraction and fume hoods, power and plumbing, lighting, doors, and windows. The lab will be constructed within a modified sea container and will be elevated on a precast concrete block foundation on a compacted granular pad.

Lab equipment and workstations suitable for sample preparation, fire assay, and wet chemical analysis will be included.

18.6.7 Paste Plant Building

The dry stack tailings and paste plant will be housed within a 24 m W × 72 m L × 26 m H pre-engineered building. A 7 m W × 19 m L × 7 m H lean-to structure is attached at the west end of the building for non-pyrite tails loadout. The building will be complete with metal cladding, insulation, HVAC, and snow guards. Overhead doors accommodate drive-through access to the paste and tailings areas.

The paste building will be equipped with a 10 t overhead crane running the span of the building to service the tailings and pyrite concentrate areas. Jib cranes or hoists will be used in partitioned areas such under the filter floor.

The pre-engineered building will be installed on a concrete perimeter wall with footings under the columns of the pre-engineered building. The footing will be founded on bedrock, utilizing rock anchors to optimize concrete quantities where required. The foundation will be backfilled and compacted on the interior and exterior of the building. A slopped concrete slab-on-grade in the paste plant will contain and directed spillage toward sumps and facilitate clean up. The pressure and vacuum filters are elevated on a steel / check plate floor with access levels underneath for cake conveying and paste equipment. Discrete process areas are isolated by low curb walls and will contain dedicated in-floor sump pumps.

At the north end of the building is the modular control room located inside the paste plant on the second level. Since the paste plant is located away from the main process plant, washroom facilities are provided and will be plumbed to septic holding tank for pump out. A laboratory space is provided for monitoring of paste recipe and strength.

Below the control room is the electrical room which houses MCCs and switchgear for process equipment. The e-room is equipped with HVAC and exterior double doors for maintenance access.

There will be tailings slurry storage tanks on the exterior of the paste plant, which will be housed within a concrete containment berm sized for 110% of the largest vessel's



volume. The concrete for these exterior pieces of equipment will be separate from the concrete works of the pre-engineered building to allow for independent erection of the pre-engineered building. Construction activities will continue through the winter season inside the building, utilizing the overhead cranes.

18.6.8 TMF Storage Shed

The TMF storage shed is a fabric building 30 m W × 36 m L × 15 m H and will provide overnight / inclement weather storage for dewatered non-acid generating tails. The building will be complete with fire retardant polyethylene fabric cladding over galvanized tubular truss frame. The building does not require insulation or heat and will have passive ventilation and lighting. The building will be founded on concrete footings with a 1.5 m H interior perimeter wall to protect the fabric mobile equipment. The end wall will be equipped with overhead door to accommodate mobile equipment such as a loader.

18.7 Power Supply and Distribution

The McIlvenna Bay site is currently service by the Saskatchewan provincial grid via existing 25 kV distribution line. Additional capacity will initially be met by onsite cogeneration LNG power plant, ultimately phased out and replaced by a SaskPower 138 kV transmission line. The region is well served with hydroelectric power from Island Falls Hydroelectric Station.

18.7.1 Power Demand Estimate

The average power demand for the project increases over the first few years of commercial production as the underground mine is developed. Table 18-1 shows the connected load and average load calculated considering demand and utilization factors by year.

Table 18-1: Connected and Average Load by Year

	Year	1	2	3	4+
Connected Load (MW)	Mineral Processing & Surface Infrastructure	14.7	14.7	14.7	14.7
	Mining	7.0	9.6	10.5	13.5
	Total	21.7	24.3	25.2	28.2
Average Load (MW)	Mineral Processing & Surface Infrastructure	9.0	9.0	9.0	9.0
	Mining	4.5	6.2	6.8	8.7
	Total	13.5	15.2	15.8	17.7

18.7.2 Existing 25 kV SaskPower Distribution Line

The existing 25 kV SaskPower distribution line to the mine site is a remnant from the closed sand quarrying operation. This line has been re-energized and is providing 1.2



MVA capacity to the construction e-house in support of ADEX and future construction activities.

During commercial production, the 25 kV line will be repurposed to power non-process infrastructure such as the administration/dry complex, truck shop, gatehouse, and camp at an average demand of 1 MW.

18.7.3 Onsite Power Generation (Phase 1)

During years 1 to 4 of commercial production, power for will be supplied primarily through a power purchase agreement utilizing an onsite liquified natural gas (LNG) power plant, supplemented by the existing 25 kV distribution line.

The power plant will operate on an N+2 operating philosophy, whereby eight (8) internal combustion generators will generate power to meet average and peak power demand, one (1) unit will be in hot standby, and one unit will be in maintenance. The availability of the power plant in this configuration is considered to be 100%. Each generator provides electrical output of 2.42 MW at voltage of 13.8 kV and frequency of 60 Hz. The design capacity of the plant under the N+2 operating philosophy is 19.4 MW. This will be increased during the ramp up phase to steady state.

Total available electrical power from both power plant and distribution line will be 20.4 MW. The LNG power plant will be equipped with a heat recovery system. A water / glycol (50%/50%) mixture will be heated to 85 °C at the power plant heat exchanger for distribution through a district heat loop via insulated piping. HVAC systems for the administration/dry complex, camp, process plant, and paste plant will utilize warm glycol as a heat source.

18.7.4 Transmission Line (Phase 2)

Long term power supply (year 5 and beyond) will be provided from the Saskatchewan provincial grid via a new 138 kV transmission line from the Pelican Narrows substation (which is supplied from the Island Falls Hydroelectric Station). The proposed routing of this line is depicted in Figure 18-10, and includes 77 km of new 138 kV transmission line and reconfiguration of 10.7 km of existing 25 kV line. The new and upgraded transmission line will be constructed by Foran and operated by SaskPower. Foran will construct a 138-13.8 kV substation with metering on the McIlvenna Bay site.

Upon commissioning of the 138 kV site substation and connection to the provincial grid, the onsite power generation plant will be decommissioned. The LNG storage infrastructure from the power plant will remain in service to fuel a glycol boiler providing a heat source for the glycol district heating loop.



Figure 18-10: Proposed 138 kV SaskPower Transmission Line (SaskPower, 2021)



18.7.5 Power Cost

Power costs have been calculated based on a blend of energy consumption from the grid and onsite power generation. Onsite power generation rates are inclusive of delivered fuel, operation and maintenance. The grid power costs are based on SaskPower schedule E22 and E24 for phases 1 and 2 respectively, and are inclusive of consumption-based energy, demand and carbon charges, and monthly connection fees. An additional rate rider of \$7.50/ MW.h has been added to grid power to account for carbon emissions offset. Table 18-2 summarizes power cost with energy mix associated with both phases.

Table 18-2: McIlvenna Bay Energy Cost

	Provincial Grid	Site Generation	Combined Average
Onsite Power Generation (Phase 1)	\$CAD 90/ MW.h	\$CAD 220/ MW.h	\$CAD 209/ MW.h
Transmission Line (Phase 2)	\$CAD 90/ MW.h	-	\$CAD 90/ MW.h
LOM Average	-	-	\$CAD 115/ MW.h



18.7.6 Power Distribution

During Phase I the site will be powered from an onsite power generation plant. When switching to grid power (Phase II), the transmission line will be connected to a 138 kV substation which will have two 20/25 MVA, 138-13.8 kV transformers in a main-tie-main arrangement. The transformers are sized such that in the event a transformer is taken out of service the remaining transformer can supply the site power requirements. The power plant and substation are both located adjacent to the process plant to minimize cable runs for the largest loads.

Site power for process areas will be distributed by 13.8 kV overhead pole lines or buried duct bank to remote e-rooms. Four 13.8 kV feeds are dedicated to the underground mine operations. The process plant and paste plant have two feeders each to provide 100% redundancy.

The power factor will be corrected to 0.95 minimum at each process facility to satisfy SaskPower's minimum power factor requirements and reduce electrical system losses.

The process plant and paste plant will be provided with standby diesel generators to provide back-up power in case of a loss of main incoming power. The standby generators will supply power for the critical process loads, life safety systems, communications equipment, security equipment, and control systems.

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
- 138 kV substation

18.8 Ore And Waste Rock Management

The ore and waste rock pad will be constructed in three (3) stages in the vicinity of the underground mine portal and the process plant (Figure 18-5). Stage 1 (226 kt capacity) has been constructed as part of the ADEX scope. Phase 2 (426 kt capacity) will be constructed during the pre-production period. Phase 3 (1,057 kt capacity) will follow in Year 2 of commercial production. Total storage capacity will be 1,700 kt creating a



280 m W × 265 m L × 30 m H stockpile. Waste rock will be stacked in lifts at an average slope of 4H:1V.

The pad and perimeter earthen berm footprint will be cleared down to bedrock. The subbase will be constructed of crushed and compacted granular sloping toward the north-east corner for contact water collection. The pad and 1 m deep collection ditch will be lined with GCL and HDPE liners on a bedding of granular A. Above the liners, 100 mm sand and 300 mm of granular A will be placed to form a drivable surface for mobile equipment.

Contact water will be pumped to the primary containment pond via submersible pumps and overland HDPE pipe. The pipe will be insulated, heat traced, and supported via sleepers on grade. A culvert will be used where the pipe passes under the haul road.

At closure the remaining material on the ore and waste rock pad will be encapsulated with an engineered cover to minimize any long-term environmental impacts. The engineered cover will include a sand and gravel bedding layer, GCL, LLDPE, sand and gravel liner protection layer, riprap erosion channels, and organic surface soils. The surface of the pile would then be re-vegetated with native species. Given that the pile will be constructed at a 4H:1V slope during operations, no significant re-grading or sloping of the pile would be expected for closure.

18.8.1 Ore Stockpiling

ROM ore that cannot be delivered directly to the primary crusher in a load bin (located on surface) will be stored temporarily in a stockpile adjacent to the ROM pad, or on a dedicated portion of the lined ore / waste rock pad.

Following the primary crusher, an online analyzer working in conjunction with conveyors and actuated chute work will split the ore stream into two separate piles— copper-rich ores and zinc-rich ores—according to grade. Each ore stockpile will have a 6,000 t capacity (1,520 t live) and utilize a reclaim system to meter and blend feed to the SAG mill feed conveyor (Figure 18-11).

Figure 18-11: Ore Stockpile Capacity (Halyard, 2022)



18.8.2 Waste Rock Dump

Waste rock storage will be on the lined ore / waste rock pad described in Section. The geometry of the stockpile has been designed to accommodate the portion of waste rock to be stored on surface, mainly during initial mine development. The shallow slope has been selected for pile stability and so the stockpile can be capped in place with minimal regrading.

18.9 Water Management

The site water management strategy is to divert or deflect non-contact surface runoff water away from the project site to the maximum 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.

Most of the water that will need to be managed at the site comes from the dewatering of underground workings. The next most abundant source is 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 ore and waste rock pad, 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 in Figure 18-12.



[illegible]

18.9.1 Non-Contact Water Management

Since the project site is relatively flat, perimeter deflection berms and diversion ditches will be constructed around the north and south sides of site to minimize site catchment area and to facilitate drainage of non-contact water away from the site.

18.9.2 Contact Water Management

Contact water is defined as 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 and mining operations. Surplus contact water will be treated through an effluent treatment plant and discharged north into Winn Bay (Hanson Lake) via overland HDPE pipeline.

Surface contact water collection will be accomplished by utilizing lining under ore and waste stockpiles, site grading, and ditching to allow the water on site to collect in the lined collection pond. The pond is complete 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.

Four ponds will be constructed on surface to manage contact and process water.

Containment Ponds

The primary and secondary containment ponds will be 115 m W × 155 m L × 2.5 m deep and share a common central berm wall. Each pond will have 41,000 m³ live capacity and provide settling and polishing of contact water for reuse as process or underground service water. The containment ponds are equipped with emergency spillways sized for the Probable Maximum Flood (PMF) theoretical storm event (10,000-year Type II 24-hour storm event consisting of 349 mm precipitation). Two (2) 2,000 m³ treated water cells are located adjacent to the primary containment pond for temporary storage of treated water for reuse as process makeup water or service water. Excess treated water will be sampled and discharged to the environment.

The containment ponds will be constructed as gravity earth berms, 2H:1V slope, composed of compacted granular material and lined with GCL and HDPE liner system to minimize infiltration of pond water and erosion of the inboard berm wall.

The primary containment pond, including both treated water cells, is currently in permitting and will be constructed as part of the ADEX program.



Collection Pond

The collection pond is in the northwest corner of the site and is approximately 5,000 m³. Runoff from the plant site will be conveyed to the collection pond by ditching. Water collected within the collection pond will be pumped to the containment pond.

18.9.3 Tailings Area Water Management Pond (TAWMP)

An existing topographical depression (approximately 100 m L × 300 m W × 4 m D) remaining from closure of a former quarry will be utilized as a surface water runoff management pond for the TSF. A 2 m tall berm will be constructed around the perimeter of the pond to provide additional containment for extreme storm events and provide a containment capacity of 206,000 m³. The pond and berm will be lined with an HDPE liner to minimize infiltration and seepage. Water from the pond will be pumped to the Primary Containment Pond using a submersible pump and pipeline. A spillway has been included in the pond to convey events over and above the environmental design storm.

18.9.4 Effluent Treatment Plant (ETP)

The ETP will be a containerized plant complete with HVAC, piping, and electrical located adjacent to the containment water pond. The facility will utilize settling ponds and multi-media filtration to reduce heavy metals and suspended solids. Treated water quality will be sampled and analyzed to confirm compliance with Metal and Diamond Mining Effluent Regulations (SOR/2002-222) before being pumped to Winn Bay via overland heat traced pipeline for discharge.

The capacity will be staged and increase with mine water volumes. Initially, a 700 m³/d facility will be installed as part of the ADEX program and will be expanded to 1,500 m³/d in pre-production, and ultimately to 4,000 m³/d in Year 10.

18.9.5 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 4,700 m³/d.

18.9.6 Sewage Handling

Domestic wastewater from the administration and mine dry building, onsite camp, and process washroom facilities will be collected and channeled to a biological treatment system (sewage treatment plant). The treatment system will be in a climate-controlled, fully enclosed building and will include oil separation, UV disinfection chamber, pump out chamber, and effluent pumps. The effluent will be discharged through an on-site septic bed located south of the camp.



18.9.7 Domestic Water

A fully enclosed and climate-controlled domestic water treatment plant will be installed to treat water from an on-site well for non-potable use (showers, toilets, and sinks in the dry, camp, and office complexes). Potable water for consumption will be bottled.

18.10 Tailings Storage Facilities

The current mine production plan indicates that approximately 23.8 Mt of tailings will be produced from the on-site mineral processing plant over the life-of-mine.

To mitigate the risk of complications arising from generation of acid within the surface 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.

Approximately 5.4 Mt of the tailings will be separated into a higher-sulphur pyrite concentrate tailings stream that will be mixed with cement binder and utilized as underground backfill. In addition, 6.0 Mt of desulphurized tailings will also be used in the backfill. The remaining 12.4 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.

A plan and sections of the TSF are given in Figure 18-13 through Figure 18-15.



Figure 18-13: Tailings Storage Facility General Arrangement (Knight Piésold, 2022)

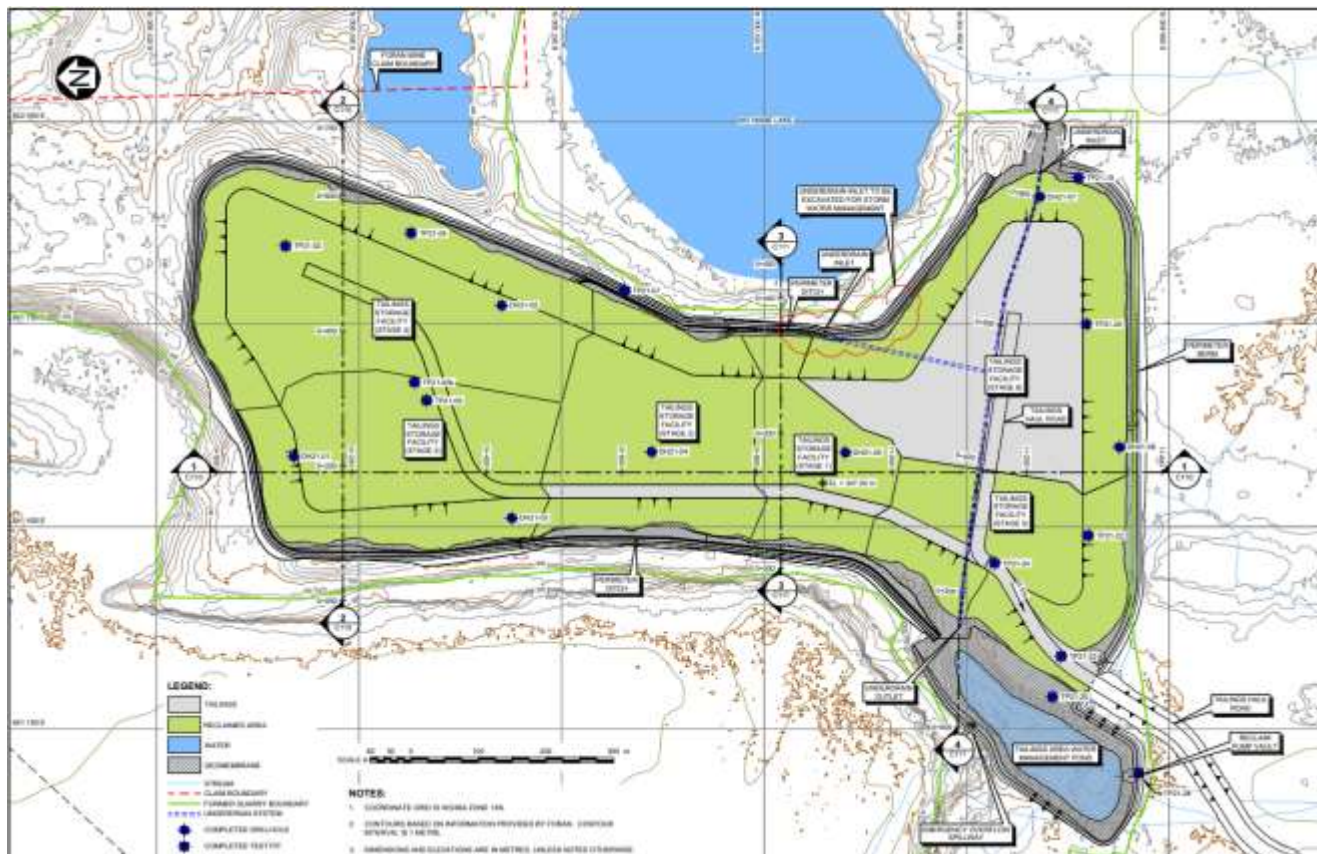
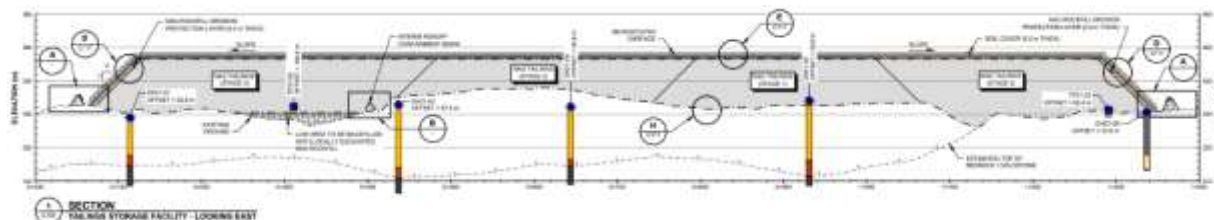


Figure 18-14: Tailings Storage Facility Cross Section 1 (Knight Piésold, 2022)



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The tailings stack will be compacted during placement. This 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 to minimize effort required at closure.

The side slopes of the TSF will be lined with a GCL then covered with NAG rockfill and a soil cover progressively during operations. As stages of the TSF are completed, the top of the TSF will be covered with an HDPE liner and surface soil. These covers will serve as the final closure cover for the TSF and will minimize potential environmental impacts such as dusting, and tailings contact runoff water from the pile.

19.0 MARKET STUDIES AND CONTRACTS

McIlvenna Bay will produce two saleable concentrates from the mineral processing facility, a copper concentrate and a zinc concentrate.

Pricing for metals contained in the concentrate will be based on market price at the time of sales when the concentrate is received at the smelters.

The metal price assumptions used in this feasibility study are based on consensus pricing from a number of banking institutions to arrive at a reasonable long-term estimate. Metal price estimates are considered conservative based on supply and demand fundamentals.

The assumptions made for the purposes of this report include the following:

- The copper and zinc concentrates produced from the mineral processing facility will be sold to smelters in Canada. The transportation costs have been included in the Smelting and Refining costs.
- 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.
- 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 metal prices used in the economic evaluation of this project are summarized in Table 19-1 and spot prices for 27 February 2022 are summarized in Table 19-2.

Table 19-1: Consensus price

Metal	Unit	Consensus Price
Copper	US\$/lb	\$3.50
Zinc	US\$/lb	\$1.20
Gold	US\$/oz	\$1,600
Silver	US\$/oz	\$22.50



Table 19-2: Spot Price (27 Feb 2022)

Metal	Unit	Consensus Price
Copper	US\$/lb	\$4.53
Zinc	US\$/lb	\$1.64
Gold	US\$/oz	\$1,936
Silver	US\$/oz	\$24.52

19.1 Supply and Demand Forecasts

McIlvenna Bay is expected to produce approximately 316 thousand tonnes of copper and approximately 614 thousand tonnes of zinc over 18 years of mine life

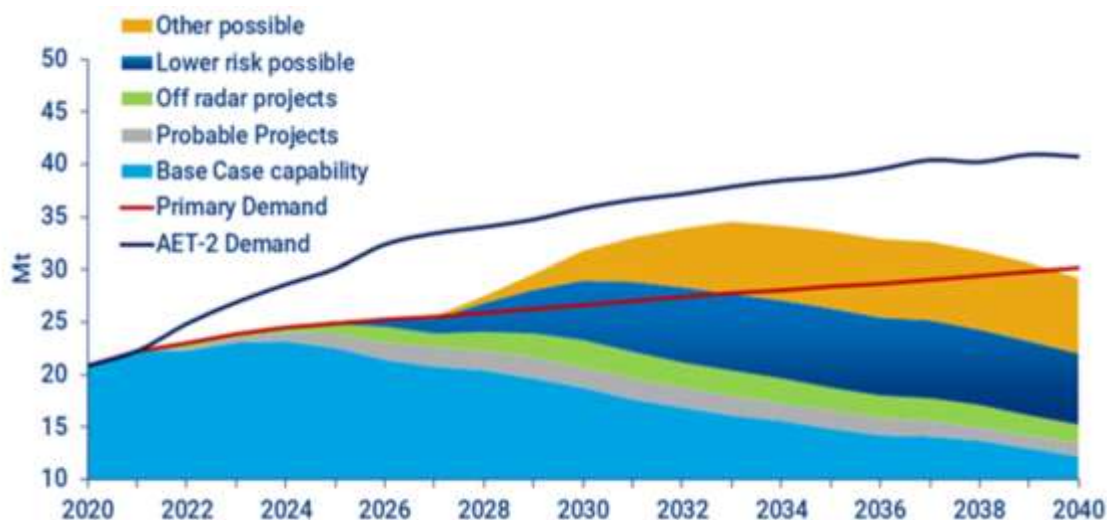
19.1.1 Copper

Copper market analysts are predicting a copper supply shortfall which is expected to drive increased copper prices through 2040. Copper demand is expected to increase due to large investments in clean energy and battery electric vehicles. The electrical conductivity of copper makes it an important material in the decarbonization of electrical grids across the globe. In a report in May 2021, Goldman Sachs wrote "Copper will be crucial in achieving decarbonization and replacing oil with renewable energy sources, and right now, the market is facing a supply crunch that could boost the price by more than 60% in four years." Goldman estimates that copper demand will increase by up to 8.7 million tonnes by 2030.

On the supply side, S&P Global Market Intelligence anticipates copper demand outpacing supply due to a lack of projects that can be brought into production. They predict copper global copper production to increase from 21.16 million tonnes in 2020 to 26.14 million tonnes in 2025. In March 2021, Wood Mackenzie (2021) forecasted that copper demand required to achieve an accelerated energy transition to keep global temperature increase at 2° C (AET-2 scenario). Figure 19-1 shows forecasted mine supply to be unable to keep up with demand by a large margin, even when including projects that have not been approved for construction.



Figure 19-1: Primary Copper Demand Scenarios versus Mine Supply Potential

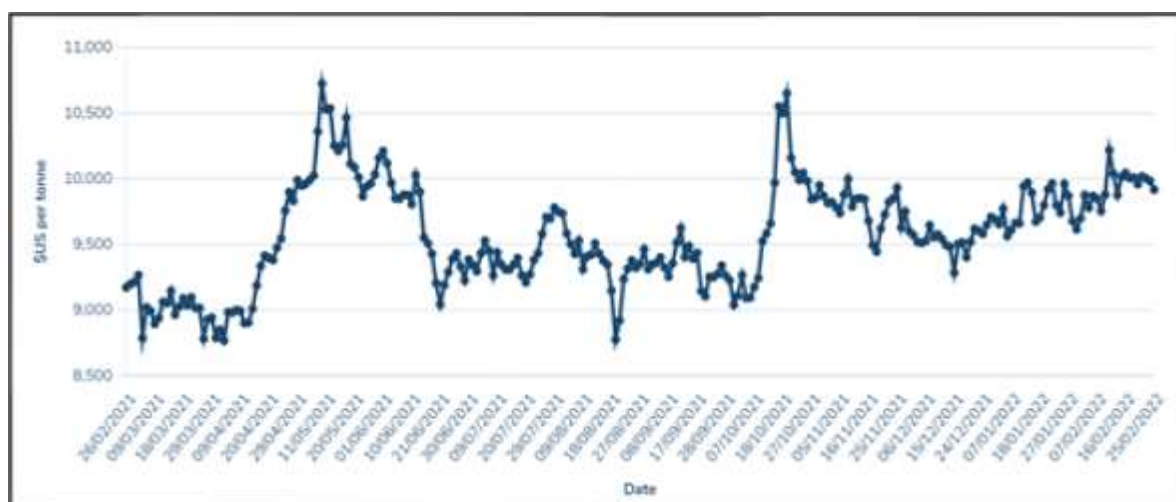


Source: Wood Mackenzie (2021)

Copper reserves at existing mines are being depleted and there has been a lack of investment in new project development for the past 20 years. CRU Group estimates that over 200 copper mines will close by 2035, without enough new discoveries and new mines to be able to fill the supply gap.

The supply and demand fundamentals are expected to underpin sustained elevated copper prices over the coming years, with copper price hitting an all time high in 2021 as shown in Figure 19-2.

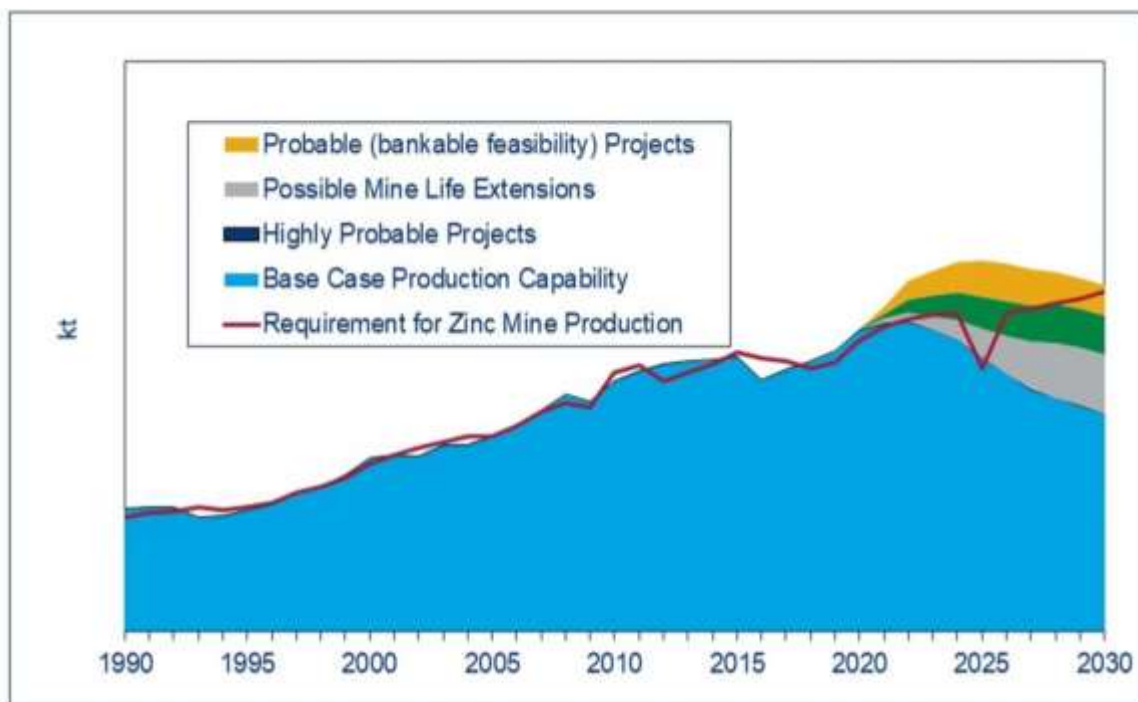
Figure 19-2: LME Copper Official Price Graph



19.1.2 Zinc

Zinc consumption is expected to remain strong in 2022 due to increased demand for steel. Supply is expected to increase and put downward pressure on Zinc price in 2023 and beyond. According to Fitch Solutions galvanized steel production is expected to increase by 3.6% during 2022 and 2.2% during 2023. Anticipated and historical supply and demand can be seen in Figure 19-3.

Figure 19-3: Sources of Future Zinc Mine Production

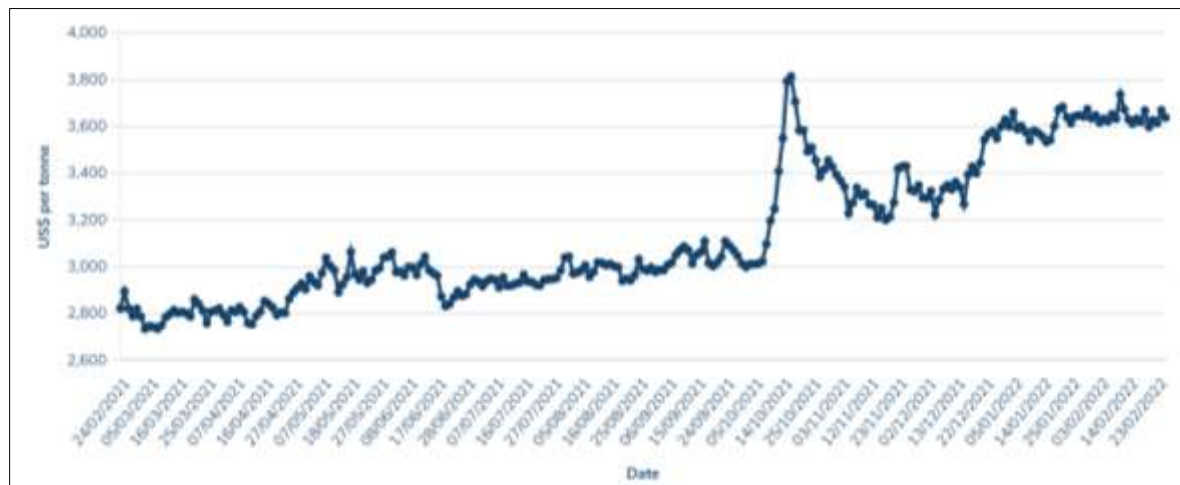


Source: Wood Mackenzie (2019)

Zinc prices have been steadily increasing since early 2021 and remain buoyant, as shown in Figure 19-4. High energy costs in Europe have led smelters there to reduce production and more stringent environmental regulations in China have also led to reduced production, supporting elevated zinc price.



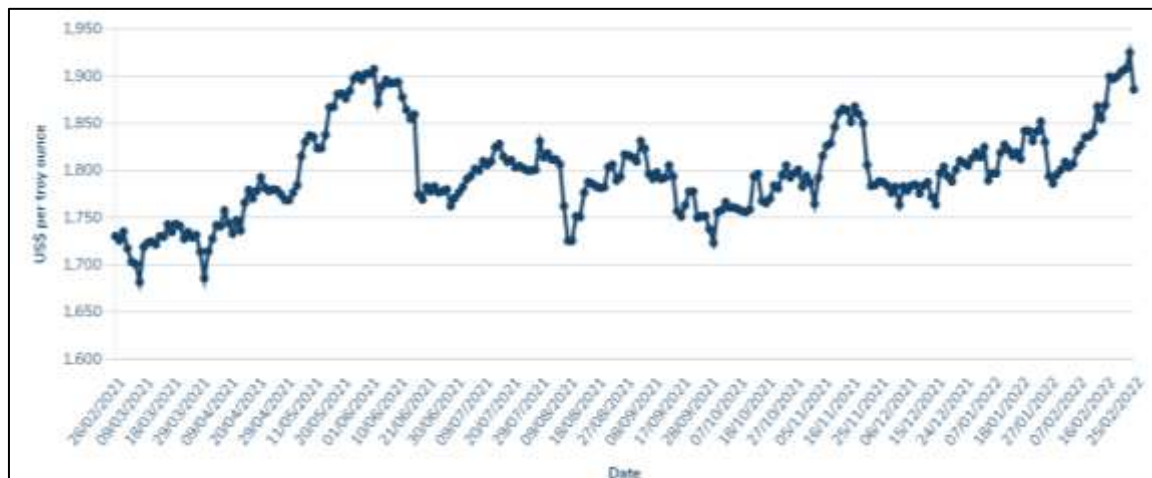
Figure 19-4: LME Zinc Official Prices Curve



19.1.3 Precious Metals

Foran expects to receive revenue from precious metals contained in the concentrates. Gold prices have been relatively high based on inflation concerns and geopolitical instability. Gold price is expected to weaken in 2022 as the US federal reserve increases interest rates.

Figure 19-5: LME Gold Closing Prices Graph



Silver price is expected to remain strong as it has some safe-haven demand and also has significant industrial use, including in renewable energy such as wind and solar power. As with copper, silver demand will increase with investment in decarbonization.



Industrial applications account for 51% of silver consumption, largely driven by renewable energy production. This proportion is expected to increase over time.

Silver price has been consistent for the twelve months leading up to the effective date of this report, as shown in Figure 19-6, and is forecasted to increase in the coming years.

Figure 19-6: Silver Closing Prices Graph



19.2 Concentrate Sales

Foran has not entered into any contracts for concentrate sales at the time of writing and has used reasonable treatment and refining charges based on standard commercial terms within the industry. Concentrate will be trucked from site to Flin Flon, MB, from where it will be loaded onto rail cars and can be transported to smelters in BC, QC, ON, or to ports. The average LOM mine concentrate grades are shown in Table 19-3.

With several Canadian mines forecasted to cease operations in 2022 and 2023 there are several smelters in Canada projected to have significant capacity available for Canadian sourced concentrate. Mines forecasted to close include Glencore's Kidd Mine and Mattagami Mine, Hudbay's 777 Mine, and Trevali's Caribou Mine. As such, smelters will be eager to source feed from McIlvenna Bay. Canadian smelters with available capacity include Teck Resources' Trail Smelter (BC), Vale's Copper Cliff Smelter (ON), and Glencore's Horne Smelter (QC).

Penalties are not expected to be levied against settlements. Penalties could be levied for several reasons, including deleterious elements, concentrate grades that are out of the agreed specification range and concentrate moistures that are outside of the agreed specification range. Assays of the concentrate quality specifications for typical elements that may occur a penalty are shown in Table 19-4.



Table 19-3: Average LOM Concentrate Grade by Metal Type

Metal	Unit	Concentrate Grade	
		Copper	Zinc
Cu	%	28.0%	
Zn	%		50.0%
Ag	g/t	242	
Au	g/t	10	0.70

Table 19-4: Assay of the Concentrate Quality – Typical Penalty Elements

Analyte	Unit	Cu Conc	Zn Conc
Cl	%	0.02	0.05
F	%	0.10	0.01
As	ppm	78	117
Hg	ppm	11.21	85.63
Sb	ppm	65	31
Se	ppm	335	137
Si	%	4.10	0.66



20.0 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, mineral processing facility, and installation of associated surface amenities including a warehouse and truck maintenance shop, fuel storage and distribution, a paste backfill 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 year-round via an 18 km long all-weather gravel road which connects to Highway 106 (the Hanson Lake Road) approximately 85 km west of the neighboring towns of Flin Flon, MB and Creighton, SK.

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, Padbury and Stushoff 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, Padbury and Stushoff 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, Padbury and Stushoff 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, Padbury and Stushoff 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 Mining Corporation (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 Volcanogenic Massive Sulphide (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 Bertum 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 Tailings Storage Facility (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

The Project area does not represent undisturbed baseline conditions as it has a long history of past exploration and mining activities as detailed above. Historical environmental studies conducted specific to Foran's Project have included aquatic investigations and water sampling programs completed between 1989 and 1992, and rare plant, raptor, and boreal woodland caribou (*Rangifer tarandus caribou*) habitat studies completed along access road route options for Foran in 1999 (TAEM 1990a, 1990b; Conor Pacific 1999).

In 2012, a comprehensive environmental baseline program was completed by CanNorth for Foran that was designed to prepare the Project for future licensing and regulatory requirements. The program collected 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, chemistry, and 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.

Foran recently commissioned CanNorth to complete an environmental baseline program commencing in 2021 and extending into 2022 to provide updated supporting information for the Environmental Impact Assessment (EIA) and to investigate additional study areas



for Project planning. The 2021/2022 program used current practices and the most up-to-date SKMOE survey protocols.

20.3.1 Aquatic Resources

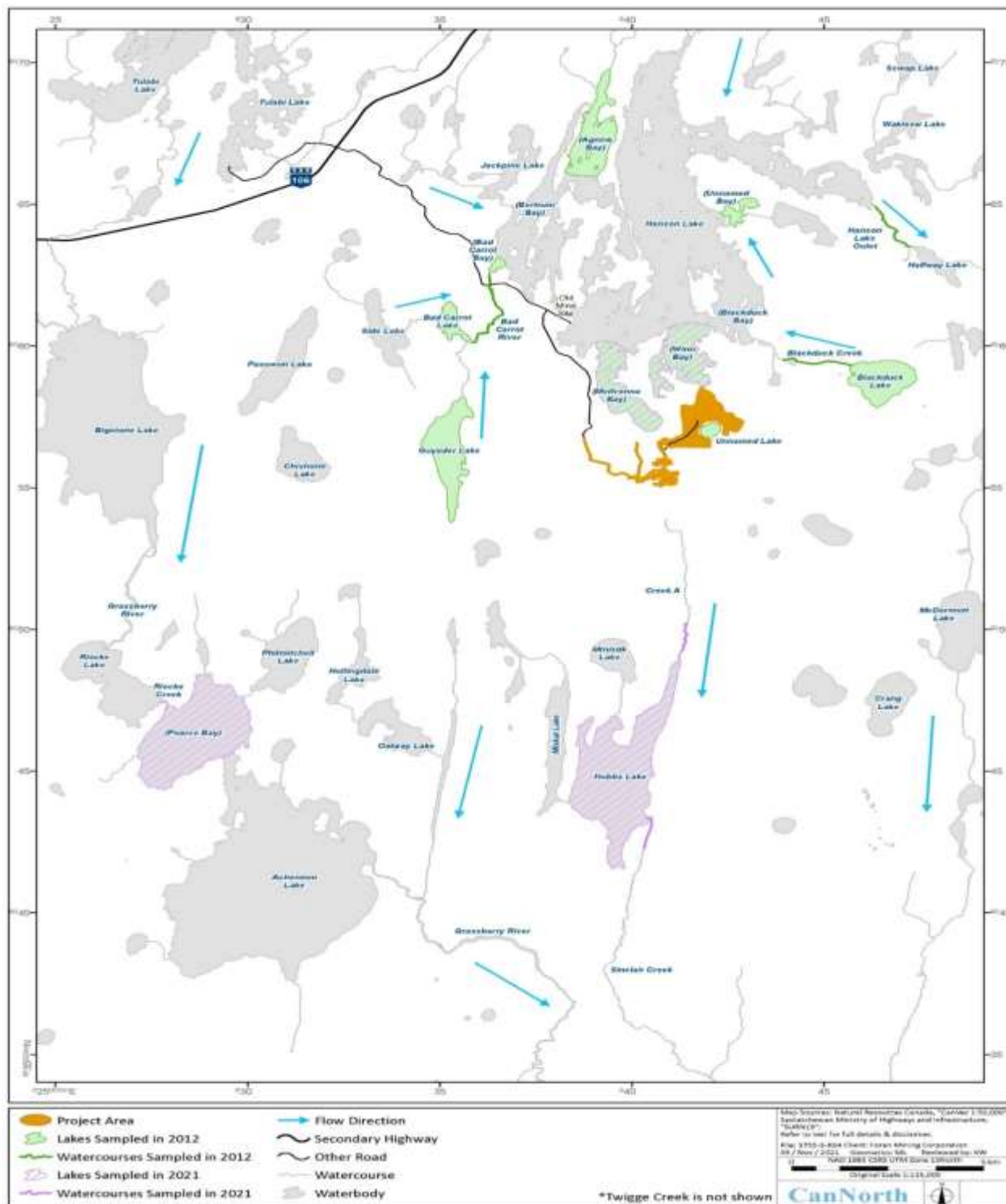
The Aquatic Study Areas (ASA) for the 2012 and 2021 studies were chosen to include numerous lakes and bays located in the vicinity of the Project, as well as various options for the treated effluent discharge location. The 2012 studies were conducted in five bays on Hanson Lake (Agnew Bay, Bad Carrot Bay, McIlvenna Bay, Unnamed Bay, and Winn Bay), as well as Bad Carrot River, Bad Carrot Lake, Blackduck Lake, Guyader Lake, and Unnamed Lake (Figure 20-2). Agnew and Unnamed bays were included in the program as they have the potential to function as future reference areas once the mine is operational.

The 2021 ASA included McIlvenna and Winn bays on Hanson Lake to provide updated information, the Hobbs Lake drainage because it was as an option for treated effluent discharge, and waterbodies located in the region that could act as future reference areas for the Hobbs Lake drainage (Figure 20-2). An attempt was made to resample Unnamed Lake in 2021; however, the unconsolidated sediment adjacent to the lake made access unsafe. A comprehensive suite of information was obtained from the Hobbs Lake drainage; however, it was subsequently determined to be an unsuitable location for treated effluent discharge due to the extensive wetland area extending between the Project and Hobbs Lake that would have negatively affected effluent dispersion and dilution and potentially wetland function. After a review of options, it was decided that the most suitable location for treated effluent discharge is into Winn Bay of Hanson Lake.

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 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 southern portions of McIlvenna and Winn bays both contain shallow areas with predominantly silt/clay/organic substrate and numerous areas with emergent vegetation. These habitat types create an abundance of northern pike and yellow perch spawning habitat, and eggs were located in these bays during the 2012 spring spawning survey (CanNorth 2013).



Figure 20-2: Aquatic Study Areas from the 2012 and 2021 Baseline Sections

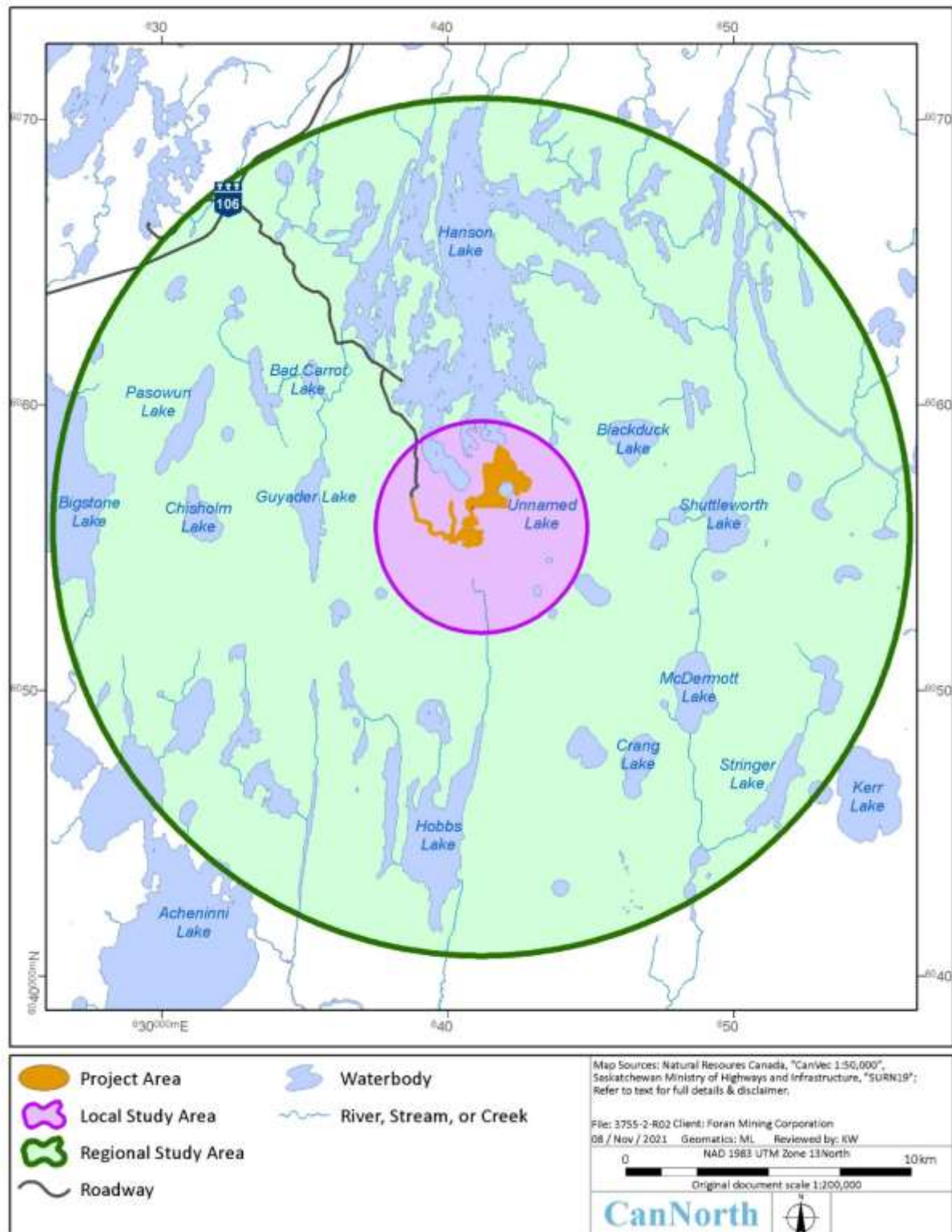


20.3.2 Terrestrial Resources

The terrestrial baseline program included a combination of database searches for rare, at-risk, or sensitive plant and wildlife species, along with intensive field programs to assess the vegetation, wildlife, and wildlife habitat present within the Project study area. A comprehensive program was conducted in 2012 that included vegetation and landcover classification, mapping, and accuracy assessment; ecosite classification; rare plant surveys; a winter tracking survey; an ungulate aerial survey; a breeding waterbird and raptor nest aerial survey; a breeding bird survey; an amphibian acoustic survey; an ungulate pellet group/browse survey; a small mammal survey; a semi-aquatic furbearer shoreline survey; and fall bird migration surveys. The 2012 terrestrial baseline studies were conducted in the site study area (SSA), local study area (LSA; 3.7 km radius of the deposit), and regional study area (RSA; 15 km radius of the deposit), with studies primarily focused in the SSA/LSA (Figure 20-3).



Figure 20-3: Terrestrial Local and Regional Study Areas from the 2012 Baseline Studies



Terrestrial species 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 the following:

- 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 2021);
- 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, 2020b); and
- Species included in the Saskatchewan Activity Restriction Guidelines for Sensitive Species (SKMOE 2017).

A total of 12 vegetation SOCC were identified in the LSA and 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. Many of these plants are common and widely distributed in the Mid-boreal Lowland and/or Churchill River Upland ecoregions. Vegetation inventory studies are being repeated in 2022 to provide current information and to ensure methodologies are in accordance with updated SKMOE guidance (SKMOE 2019).

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.



In 2021 and 2022, updated site-specific wildlife data will be obtained through a variety of methods including the deployment of automated recording units (ARUs), bat detectors, and trail cameras, and multiple types of species detection surveys in order to provide updated information for the Project area. Field survey methodologies will meet standards published by the SKMOE for species detection survey protocols. Studies commenced in March and December of 2021 with winter tracking surveys and will extend throughout the spring and summer of 2022. Small mammal trapping to obtain community and chemistry information as well as a shoreline semi-aquatic mammal sign survey were also completed in September 2021.

20.3.3 Heritage Resources

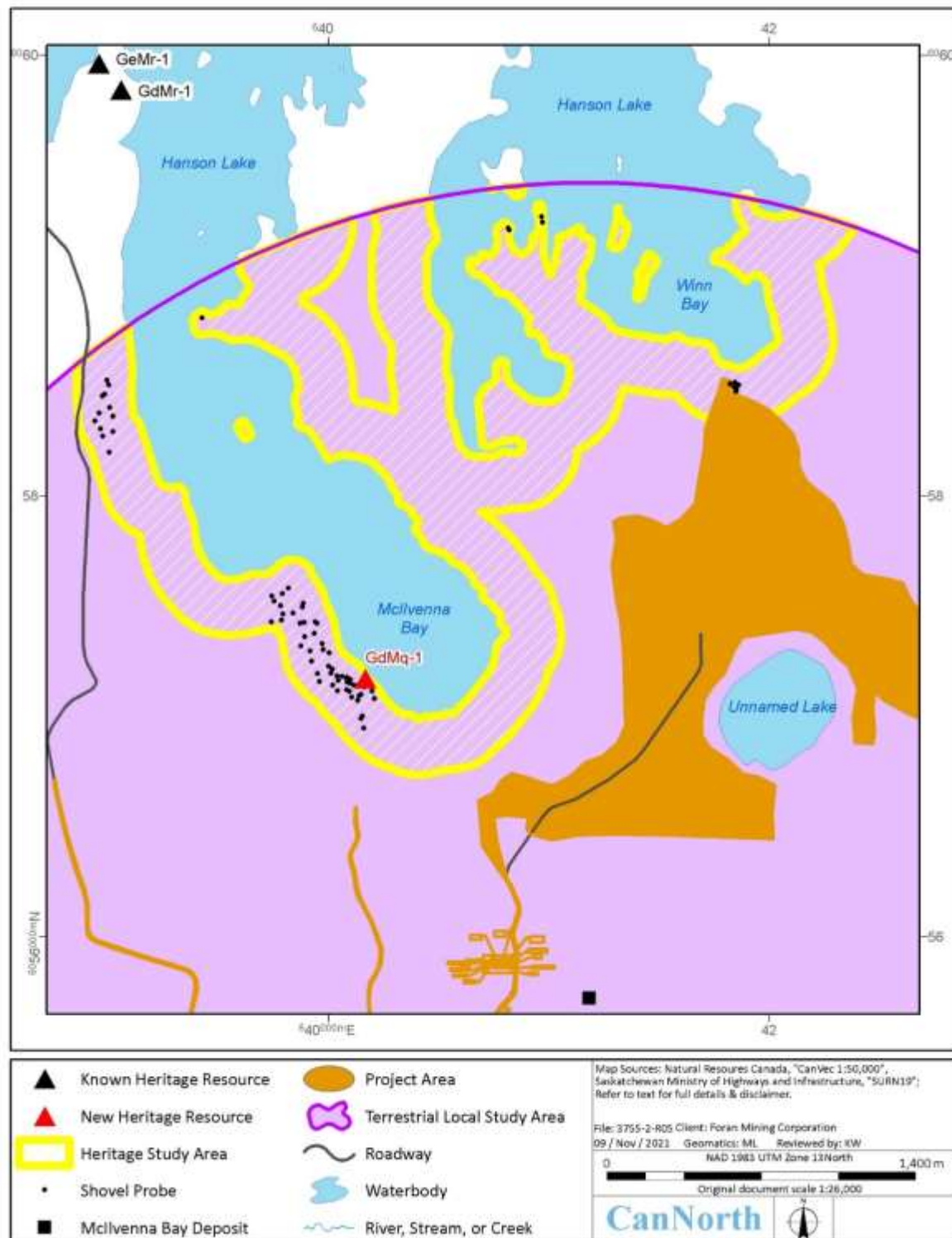
The Heritage Resource Impact Assessment (HRIA) conducted in 2012 was focused in the SSA, which included the Project footprint at the time (CanNorth 2013). CanNorth archaeologists initially reviewed the development area for heritage sensitivities and submitted the Project to the Saskatchewan Heritage Conservation Branch (HCB) to determine HRIA requirements. The HCB determined that a HRIA was required for areas that are within 250 m of Hanson Lake (HCB File No. 12-1072). Therefore, to adequately assess the shorelines of Hanson Lake's McIlvenna Bay and Winn Bay and to identify areas considered to have the greatest heritage potential, a canoe was paddled along the shorelines. All areas considered to have moderate to high heritage potential were assessed using a combination of pedestrian reconnaissance, the excavation of shovel probes, and examining surface exposures including tree throws and cut lines. Dolomite outcroppings (e.g., cliff faces) were also closely inspected for rock art.

In total, approximately 18.5 ha were assessed on foot including the excavation/inspection of 68 shovel probes/surface exposures and one new heritage resource, GdMq-1, was discovered (Figure 20-4). 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.

Foran's updated project footprint includes several changes as compared to what was previously assessed and surveyed under Archaeological Investigation Permit No. 12-141. This updated footprint was submitted to the HCB in 2021 and it was determined that no additional work was required to be completed in 2022.



Figure 20-4: Heritages Resources in the Terrstrial LSA and 2012 Baseline Study Search Locations



20.4 Environmental Risks/Issues

Provided below is a high-level summary of the potential effects (positive and negative) anticipated to occur during all phases (construction, operation, decommissioning, and post-decommissioning) of the Project, including accidents and malfunctions. As part of the EIA, a Conceptual Site Model (CSM) that details the predicted Project inputs, potential receptors, and linkages between them will be developed. This information will then be fed back to the design team to allow for optimization of the mitigation strategies.

A preliminary risk assessment has been completed based on the current Project plans. The objective was to identify potential effects pathways and mitigation measures on potential valued components from various activities that could result from the Project. The results of the preliminary risk assessment are provided in Table 20-1.

20.5 Environmental Assessment Process & Permitting

Based on the nature of the Project and the provincial and federal legislative criteria, Foran expects the Project will be subject to a provincial EIA before the Project can proceed. The Project will also require provincial licenses, approvals, and permits for various aspects and phases, and will need to meet federal legislation requirements. The regulatory framework and associated approvals process for the Project are outlined in the following sections.

20.5.1 Environmental Assessment

Federal Impact Assessment

On 28 August 2019, the federal *Impact Assessment Act* (IAA) came into force, which created the Impact Assessment Agency of Canada and repealed the Canadian Environmental Assessment Act (CEAA) 2012. Projects subject to the IAA include:

- projects described in the Physical Activities Regulations (the Project List) (GC 2019);
- projects designated by the Minister of Environment and Climate Change that are not described in the Project List; and
- projects on federal lands and outside Canada assessed by federal authorities before decisions are made.



Table 20-1: Preliminary Risk Assessment and Mitigation Measures for the McIlvenna Bay Project

Activity/Disturbance	Potential Impact/Effects Pathway	Component	Preliminary Mitigation
Dust Generation	Increased release of dust due to excavation, earth-moving, vehicle travel, waste rock pile, and ore stockpile.	Atmospheric Environment	Use of dust control measures to minimize dispersion during construction and along roads. Use of engineered controls to minimize releases from waste rock pile and ore stockpile.
	Uptake of contaminants by plant roots from soil, deposition of dust onto plant leaves can affect plant health, consumption of contaminated plant material can affect wildlife.	Terrestrial Environment	Construction in the majority of the study area will occur under frozen conditions which will reduce dust deposition onto plants. Erosion control practices will be implemented to limit dust production and deposition in surrounding areas and to limit erosion to exposed soils. Monitoring of plant chemistry will be planned for the operational phase.
	Changes in ambient air quality and associated deposition may have direct and/or indirect effects on surface water quality.	Aquatic Environment	Effects of dust on the terrestrial and aquatic environment will be considered in the EIA including the use of air dispersion models developed in accordance with the Saskatchewan Air Quality Modeling Guideline (SKMOE 2012).
	Inhalation of dust.	Human Environment	Development and implement a comprehensive health and safety program for all phases of the Project.
	Increased emissions from vehicles, mine air exhaust, air conditioners, generators, the mineral processing facility, and explosives could affect air quality and contribute greenhouse gases to the atmosphere.	Atmospheric and Human Environment	Use of best available technology in design of mine, mill, and supporting infrastructure to reduce emissions to air. Foran is aiming to have the McIlvenna Bay Project be carbon neutral. Tying into provincial power grid from the Island Falls hydroelectric station will reduce greenhouse gas emissions associated with the Project. Maximize use of battery electric vehicles, where possible. Conduct regular maintenance on all vehicles and machinery to ensure optimal performance. Conduct follow-up monitoring to ensure air quality standards are met.
Noise	Noise related to vehicles, heavy equipment, blasting, and mining/milling activities can increase ambient noise levels disrupting wildlife.	Terrestrial and Atmospheric Environment	Given the nature of the noise propagation, noise emissions are expected to be local in extent. Equipment will be selected to minimize noise generation and maintained regularly; this could include the use of electric vehicles where possible. Noise effects will be further quantified in the EIA with further potential mitigation options provided.
Dewatering	Underground mine dewatering could change groundwater flow patterns.	Hydrogeological Environment	Hydrogeological flow models to predict effects will be utilized in the EIA and further mitigation options will be provided if required.
	Underground mine dewatering could lead to a lowering of groundwater levels causing surface water drawdown, subsequent water level alterations, and changes in natural streamflow.	Aquatic Environment	Project planning will aim to have the lowest possible impact on groundwater flow and levels. Development and implementation of a Groundwater management and monitoring Plan.
	Mine water release could impact water quality and quantity in the receiving waterbody (Hanson Lake).		Mine water will be collected and treated in the mill prior to release.
Seepage from ore stockpile, waste rock, or TSF	Seepage of contaminants could affect groundwater quality.	Hydrogeological Environment	Use of engineered barriers to prevent seepage. A cemented tailing paste backfill system will be used for backfilling stopes; desulphurized tailings from the mineral processing facility will be thickened and filtered to reduce the moisture content and trucked to the TSF. TSF management program and design standards will be applied (e.g., Mining Association of Canada Tailings Guidance). Groundwater monitoring program will be designed and implemented.
Water supply requirements	Freshwater use requirements may cause lowered water levels and volumes from the source waterbody potentially impacting aquatic habitat and could lead to fish entrainment at the water intake pipe.	Aquatic Environment	Foran expects to use treated water from the treatment plant as the fresh/makeup water source for the process plant and not to draw from a local waterbody for processing. Potable water will be taken from McIlvenna Bay, but the quantities will be insignificant.
Site run-off and treated effluent release	Site run-off and treated effluent release may result in changes in volume, flow, and water and sediment quality of receiving waterbody from addition of treated effluent. This in turn could impact aquatic biota.	Aquatic Environment	On-site freshwater will be diverted and all potentially contaminated surface water run-off will be collected and treated prior to release. Treated effluent release will be designed to reduce erosion. Treated effluent volume will be minimized by recycling process water as much as possible. Estimates of treated effluent quality and quantity will be used in the EIA to predict impacts to the aquatic environment and mitigation measures will be developed where necessary along with follow-up monitoring program. Prior to treated effluent release, parameter concentrations will meet or be lower than provincial and federal requirements. A monitoring program for the aquatic environment will be developed and implemented.
Accidental Spills	Hazardous waste spills or leaks from equipment/vehicles can affect waterbodies, soils, vegetation, and wildlife.	Aquatic and Terrestrial Environments	An Emergency Response Plan will be developed to provide quick detection, control, and management of any spill during all phases of the Project and to ensure proper disposal of hazardous waste. A Hazardous Materials Management Plan will be developed. Hazardous materials and fuel storage facilities will be properly designed and licensed. Surface facilities will be designed to allow for the collection of spills. Best management practices will be followed when working on or near water.
Project footprint construction	Direct loss and fragmentation of wildlife habitat and movement caused by the Project footprint.	Terrestrial and Aquatic Environments	Project footprint will be partially located on existing brownfield site and disturbance of native vegetation will be minimized as much as possible.
	Potential loss of vegetation, wetlands, and wildlife/fish habitat as a result of water level changes during operation of the Project.		Potential impacts to the terrestrial environment will be assessed in the EIA and mitigation measures will be developed where necessary along with follow-up monitoring programs.



Activity/Disturbance	Potential Impact/Effects Pathway	Component	Preliminary Mitigation
	Sensory disturbance of wildlife.		Construction during sensitive wildlife windows will be avoided.
	Direct loss or alteration of native vegetation from construction of the Project infrastructure.		Development and implementation of an Environmental Protection Plan, Erosion and Sediment Control Management Plan, Wildlife Management Plan, Vegetation Management Plan, and Caribou Management Plan.
	Direct loss of soil via erosion, reduction in soil quality/productivity, changes to terrain stability, and alteration of local slope profiles.		Detailed designs will be tailored to minimize and balance the cut and fill needs of the Project footprint.
	Introduction of prohibit, noxious, nuisance, and/or invasive plants.		Topsoil removed during the site preparation process will be placed on a lined pad near the entrance to site, graded, and suitably covered to prevent contamination and erosion. The topsoil will be reclaimed during future progressive and final closure activities.
Increased traffic	Increased traffic may lead to collisions with wildlife.	Terrestrial Environment	Prevention of the introduction and spread of prohibited, noxious, nuisance, and invasive plants will be addressed in management plans.
	Sensory disturbance to wildlife, in particular SOCC.		Development and implementation of a Closure Plan that leaves reclaimed areas with vegetation communities comparable to the natural range of variation in nearby ecosystems and site terrain shaped as much as possible to blend with local topography.
Waste generation	Wildlife may be attracted to the waste generated by the Project which may increase the human-wildlife interactions and mortality and risk to wildlife.	Terrestrial Environment	Development and implementation of an Environmental Protection Plan, Wildlife Management Plan, and Caribou Management Plan.
Health and safety of workers	Potential for increased exposure to workers at the Project to contaminants.	Human Environment	Enforce on-site speed limits to reduce collisions with wildlife.
Disturbance of heritage resources in Project footprint	Potential disturbance of heritage resources during land clearing and construction activities.	Human Environment	Development and implementation of a Waste Management Plan.
Changes in traditional and other land use	Potential loss or loss of access to traditional hunting, fishing, gathering grounds.	Human Environment	Waste will be regularly trucked off site.
Socio-economic benefits to local communities	Potential for increased employment, training, economic development, and community services.	Human Environment	Assessment of potential health effects in the EIA and development and implementation of a comprehensive health and safety program.
			Follow-up Heritage Resource Impact Assessments may be required when the Project plans are finalized.
			Impacts will be mitigated as required by the Heritage Conservation Branch.
			Potential impacts will be assessed in the EIA and management plans and monitoring will be developed and implemented.
			Continued engagement with local stakeholders and rights-holders to ensure concerns are heard and adequately addressed.
			A socio-economic assessment will be completed in the EIA and Project interactions will be evaluated with mitigation strategies developed.
			Training and employment of local community members during construction and operation of the Project will occur to the extent possible.
			Continued engagement with local stakeholders and rights-holders to ensure community involvement in the Project is maximized.



It is not anticipated that the Project will require a federal Impact Assessment (IA) because the Project falls below the applicable criteria listed in the Physical Activities Regulations (GC 2019):

- a new metal mine, other than a rare earth element mine, placer mine, or uranium mine, with an ore production capacity of 5,000 t/day or more; and
- a new metal mill, other than a uranium mill, with an ore input capacity of 5,000 t/day or more.

Provincial Environmental Assessment

The Saskatchewan *Environmental Assessment Act* requires that a proponent receive the approval of the Minister of Environment before proceeding with a project that is defined as a development under the Act (SKMOE 2014a). Foran is self-declaring the Project to be a development and; therefore, a provincial EIA is required. The submission of the joint Technical Proposal (TP) and Terms of Reference (TOR) for review by the Environmental Assessment & Stewardship Branch (EAB) initiated the EIA process. Foran will conduct, prepare, and submit an Environmental Impact Statement (EIS) to the EAB for a technical review by experts, including those from other provincial ministries and, where required, to federal government reviewers. Once there are no concerns remaining, the Technical Review Comments from the EAB and the final EIS document will be placed in public review for a minimum of 30 days. Ultimately, the Project will require issuance of ministerial approval under Section 15 of the *Environmental Assessment Act* before proceeding with licensing and permitting.

20.5.2 Federal and Provincial Legislation

Table 20-2 provides a list of potentially applicable federal and provincial Acts and Regulations. Foran recognizes that other legislation may apply and it will be the responsibility of Foran to comply with all applicable legislation throughout the life-cycle of the Project.

Table 20-2: Potentially Applicable Federal and Provincial Legislation

Act	Regulation
Federal:	
Fisheries Act	Metal and Diamond Mining Effluent Regulations
	Deposit Out of the Normal Course of Events Notification Regulations
	Wastewater Systems Effluent Regulations
Canadian Environmental Protection Act	Environmental Emergencies Regulations
	Federal Halocarbon Regulations
	Greenhouse Gas Reporting Regulations



Act	Regulation
Transportation of Dangerous Goods Act	Transportation of Dangerous Goods Regulations
Navigation Protection Act	No specific regulations related to this act
Species at Risk Act	No specific regulations related to this act
Canadian Wildlife Act	Wildlife Area Regulations
Migratory Birds Convention Act	Migratory Birds Regulations
	Migratory Birds Sanctuary Regulations
Explosive Act	Explosive Regulations
Provincial:	
The Environmental Assessment Act	No specific regulations related to this act
The Environmental Management and Protection Act, 2010	The Environmental Management and Protection (Saskatchewan Environmental Code Adoption) Regulations
	The Mineral Industry Environmental Protection Regulations, 1996
	The Waterworks and Sewage Works Regulations
	The Hazardous Substances and Waste Dangerous Goods Regulations
	The Municipal Refuse Management Regulations
The Wildlife Act, 1998	The Wildlife Regulations
	The Wild Species at Risk Regulations
The Wildlife Habitat Protection Act	Wildlife Habitat Lands Disposition and Alteration Regulations
The Fisheries Act (Saskatchewan), 1994	The Fisheries Regulations
The Groundwater Conservation Act	The Groundwater Protection Regulations
The Water Security Agency Act	The Water Security Agency Regulations
	The Withdrawal from Allocation Regulations
The Forest Resources Management Act	The Forest Resources Management Regulations
The Wildfire Act	The Wildfire Regulations
The Provincial Lands Act, 2016	Saskatchewan Wetland Conservation Corporation Land Regulations
	Crown Resource Land Regulations
	Provincial Lands Regulations
The Heritage Property Act	The Heritage Property Regulations
The Crown Minerals Act	The Crown Mineral Royalty Regulations
	The Crown Resource Land Regulations
The Mineral Resources Act	The Quarrying Regulations
The Natural Resources Act	The Resource Protection and Development Service Regulations
The Pest Control Act	The Pests Declaration Regulations



Act	Regulation
	The Pest Control Products Amendment Regulations, 2012
The Weed Control Act	The Weed Control Regulations
The Management and Reduction of Greenhouse Gases Amendment Act	The Management and Reduction of Greenhouse Gases (General and Reporting) Regulations
The Northern Municipalities Act, 2010	The Northern Municipalities Regulations
The Saskatchewan Employment Act	The Occupational Health and Safety Regulations, 1996
	The Mines Regulations, 2018
The Boiler and Pressure Vessel Act	The Boiler and Pressure Vessel Regulations
The Technical Safety Authority of Saskatchewan Act	No specific regulations related to this act
The Electrical Inspection Act	The Electrical Inspection Regulations
The Gas Inspection Act	The Gas Inspection Regulations
The Public Health Act	The Food Safety Regulations
	The Plumbing and Drainage Regulations
	The Public Sewage Works Regulations
	The Public Accommodations Regulations
The Passenger and Freight Elevator Act	The Passenger and Freight Elevator Regulations, 2017
The Reclaimed Industrial Sites Act	The Reclaimed Industrial Sites Regulations
Treaty Land Entitlement Act	No specific regulations related to this act

20.5.3 Guidelines, Policies, Standards

In addition to regulatory requirements, Foran will apply a number of other guidelines, policies, and standards to the Project. Foran will consider the following guidelines, policies, and standards when completing the EIA, although the list is not exhaustive:

- Federal Policy on Wetland Conservation (GC 1991).
- Guidelines for Northern Mine Decommissioning and Reclamation (SKMOE 2008).
- Environmental Monitoring Guidelines for Mining/Industrial Operations (SKMOE 2014b).
- The Saskatchewan Environmental Code and attendant standards.
- Health Canada guidance for environmental assessments for air, water, noise and human health risk assessment (Health Canada 2016a, 2016b, 2017a, 2017b, 2019).
- Saskatchewan Air Quality Modelling Guideline (SKMOE 2012).
- Consultation with First Nations and Métis in Saskatchewan Environmental Impact Assessment (SKMOE 2014c).



20.5.4 Licensing and Permitting

Following approval pursuant to the provincial EIA process, the Project will require a number of permits, approvals, and authorizations 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 EIA process. A preliminary list of permits, approvals, and authorizations that may be required throughout the life of the Project is presented in Table 20-3, subject to confirmation with the responsible agencies.

Table 20-3: Potential List of Permits, Approvals, or Authorizations Required by the Project

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
Forest Product Permit	SKMOE
Miscellaneous Use Permit	SKMOE
Construction Permit	SKMOE
Environmental Protection Plan for Industrial Sources	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
Approval to Decommission Pollutant Control Facilities	SKMOE
Release from Decommissioning and Reclamation	SKMOE
Provincial Acceptance of Decommissioned and Reclaimed Site into Institutional Control Program	SKMOE
Federal	
Fisheries Act Authorization	Department of Fisheries and Oceans Canada



Permit, Approval, or Authorization	Issuing Agency
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

20.5.5 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 provincial 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.

The Project commits to preparing and fulfilling management and offset plans as required. As Project plans are currently being finalized, it is not known if fish habitat offset plans or wetland offset plans will be required. It is anticipated that a caribou management and offset plan will be needed, and this will be developed during the EIA. The caribou mitigation and offsetting plan developed during the Strong Pine EIS (SPES 2019) will be taken into consideration during preparation of Foran's plan to ensure high level objectives for caribou protection are met. 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 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 m³ 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 Project mine, mill, and ancillary facilities, a number of approvals must be obtained from the SKMOE, including the following:

- 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.

20.5.6 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.

20.5.7 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 (e.g., buildings removed; pits filled in); 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 MOE'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.

For the purposes of this feasibility study a conceptual closure strategy has been developed which includes the following key activities:

- All shafts, raises, portals, and other openings will be capped and sealed. Larger openings such as shafts and raises will be sealed with an engineered reinforced concrete cap. Smaller openings such as wells will be sealed with bentonite or grout as appropriate.
- The mine waste rock stockpile will be capped with a composite cover comprising geosynthetic clay liner (GCL), Low Density Polyethylene Liner (LLDPE), sand and gravel protection layers, and revegetated surface soil. The cap will be constructed to minimize percolation and mitigate the potential for ARD issues post closure.
- Ponds and other water management structures will be breached and graded into the surrounding landscapes. Synthetic materials such as liners will be removed and disposed of.
- Pumps and pipelines will be removed and disposed of.
- All buildings will be demolished, with steel and other rubble being removed and disposed of off site. Concrete footings will be broken down to surface and buried. Concrete pads will be broken up and buried.
- Stationary surface equipment along with mobile equipment will be dismantled as required and removed off site to a licensed scrap yard.
- Fixed underground equipment will be drained of hazardous materials (oil and fuel) and will remain underground post closure.
- Potentially contaminated soils from around the site (i.e., soils from around fuel storage areas) will be tested and hauled off site to appropriate disposal sites as required.
- General grading of the entire site will be carried out to restore natural drainage patterns. Surface soils stripped during construction activities will be utilized to cap the disturbed areas. Revegetation will take place including tree planting in all areas with exception of the waste rock pile and tailings areas, which will be vegetated with grasses on account of their geomembrane caps.

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 is 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.

20.5.8 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 dependant 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 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 & raise closure
- Tailings storage facility closure
- Support buildings & 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 & 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

20.5.9 Tailings Storage Facility Closure

The Government of Saskatchewan 2008 guidelines on mine decommissioning highlights that the primary objective of tailings management 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 Project 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 closest communities to the Project include Creighton, SK, Flin Flon, SK, and Flin Flon, MB, which are located approximately 85 km east of the Project. Creighton and Flin Flon have a combined population of 6,587 residents, with 4,982 living in Flin Flon, MB 1,402 in Creighton, SK, and the remaining 203 in Flin Flon, SK (StatsCan 2017).



The Project is located on the Traditional Territory of Peter Ballantyne Cree Nation (PBCN), which has a registered population of 11,536, living on more than 36 reserves and/or settlements, as well as off-reserve (INAC 2021). PBCN's traditional territory encompasses roughly 52,000 km², 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 40 kilometers southeast of the settlement of Deschambault Lake and approximately 50 kilometers west of the community of Denare Beach. These communities have a combined population of 1,795 and are largely made up of PBCN members.

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). The largest employer in PBCN communities is the Band itself through programs such as Band education, health center, Band office, and child and family services (PBCN Health Services 2021). The local economy in PBCN communities includes hunting and trapping, commercial fishing, guiding, mining, forestry, and recreational and cultural opportunities (PBCN 2021). The opening of a new mine in the area would increase employment opportunities for all community members.

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) has a large presence in the Project area, operating 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 expected to transition to care and maintenance in mid-2022 (HudBay 2021). The transition of the mine may impact employment rates in surrounding communities. The mine currently employs approximately 785 individuals with plans to reduce to 50 positions in 2023. As the 777 mine transitions, activity at the Snow Lake Mine located 198 km by road from Flin Flon will increase and approximately 225 jobs will be created (Westerhaver 2021). This transition will result in a loss of jobs in the Project area; however, should Foran proceed with developing the Project, it presents the opportunity to recruit a trained and knowledgeable workforce from the nearby area. As currently planned, the Project has the potential to employ 350 people directly at their mine site in addition to creating economic growth in sectors of the economy that provide services to the mine (e.g. trucking and service industry).

Foran has been actively engaging with nearby communities since 2011; however, to date engagement efforts have focused primarily on rights-holder interests and concerns. Foran has developed a comprehensive engagement strategy to guide engagement with both rights-holders and stakeholders going forward. Foran has conducted consultation sessions for the Project in the communities of Deschambault Lake and Denare Beach commencing in 2012. Engagement has taken the form of meetings (both in person and virtual), site tours, notification letters, email correspondence, and phone calls. In 2012,



Foran offered support to PBCN to conduct 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 PBCN surveyed clearly articulated their continuing reliance on large game, fish, and waterfowl as well as innumerable plant species, to provide for their physical, social, and spiritual needs. Most acknowledged that the mining sector does provide the potential for employment, including creating spin-off opportunities in service related businesses; however, it was stated the benefits of mining developments must be tempered against the continued reliance of PBCN members on the waters, lands, and forests for sustenance, livelihood, and spiritual support (ASKI 2012). More recently, Foran has entered into discussion with the PBCN with the objectives of negotiating a Collaboration Agreement 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 part of the comprehensive engagement strategy, Foran has met with the leadership of several local stakeholder and rights-holder communities and will continue to actively engage the leadership and community members going forward.

As the Project proceeds, Foran will seek to minimize, to the extent possible, negative impacts on the 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. In order to do this Foran will undertake several studies as part of the larger EIS. This includes continued and increased rights-holder and stakeholder engagement, a socio-economic impact assessment, and an updated Traditional Land Use Study.



21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

Capital cost estimates were prepared by Stantec, Halyard, and Knight Piesold, with contributions from Foran.

Total capital expenditures of \$848.67 million are estimated for the LOM. The capital cost estimate is divided into three broad scope areas:

- Mining
- Processing
- Other

A summary of the estimated LOM capital costs is presented in Table 21-1 and details of the cost breakdowns are summarized in the sub-sections that follow:

21.1.1 Methodology – General

The capital cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International with an approximate accuracy of $\pm 15\%$. All costs included in the estimate are reported in Q1 2022 Canadian dollars.

The capital cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope is described and quantified within material takeoffs (MTOs) in a series of line items.

The project period capital costs are defined as all costs incurred from Year -2 to the end of Year -1, while sustaining capital costs are incurred from Year 1 through to the end of mine life. Pre-production capital costs have been divided into the following:

- Non development mining cost (e.g., equipment, construction, UG infrastructure)
- Mining development cost, including vertical development
- Mineral Processing and Paste Plant
- Surface Infrastructure
- Transmission line (initial install)
- Pre-production costs (Operating cost that occur prior to the mill operating)
- Tailings
- Contingency



Sustaining capital costs are related to the following:

- UG mine mobile equipment
- Development
- Exhaust raise and fan plant
- Shaft collar construction / Shaft Sinking
- UG Infrastructure construction
- Transmission line and substation
- Process plant maintenance
- Paste plant maintenance
- Surface and UG Infrastructure maintenance
- Mine closure
- Contingency

The FS outlines an initial pre-production capital cost estimate of \$367.98M, including a contingency of \$38.46 million and net of pre-commercial production credits of \$7.89 million. A summary of the estimated LOM capital costs is presented in Table 21-1 and details of the cost breakdowns are summarized in the sub-sections that follow:

Table 21-1: Capital Cost Estimate (\$ Millions)

Description	Project Period Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining			
Mining (non-development)	68.39	296.34	364.73
Development	27.51	122.35	149.86
Total Mining	95.89	418.69	514.58
Processing			
Mineral Processing and Paste Plant	151.46		151.46
Processing Sustaining		8.79	8.79
Total Processing	151.46	8.79	160.25
Other			
Infrastructure	32.41		32.41
Transmission line	10.00	15.00	25.00
Pre-production costs ³	36.34		36.34
Tailings	11.30	21.29	32.59



Description	Project Period Capital Cost	Sustaining Capital Cost	Total Capital Cost
Total Other	90.05	36.29	126.34
Total before Contingency	337.40	463.77	801.17
Contingency	38.46	9.04	47.50
Total	375.87	472.81	848.68
Closure		8.19	8.19
Pre-Commercial Production Revenue	(54.46)		(54.46)
Pre-Commercial Production Costs ⁴	46.57		46.57
Net Capital Costs	\$367.98	481.00	848.98

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Totals may not sum due to rounding as required by reporting guidelines.
3. Pre-production costs are operating costs that occur prior to the mill operating.
4. Pre-commercial production costs include fees for smelting and refining, operating costs in Year -1, and royalties.

21.1.2 Mining Capital Costs

The project period mine capital costs primarily include lateral and vertical mine development, and purchase and installation of stationary mine infrastructure. Mine mobile equipment used for the Project will likely be a combination of purchased on a “lease to own” basis and purchased outright.

For estimating the direct costs, it has been assumed there will be two main contracts associated with the access and development of the mine: UG Lateral Development and Installations Contract, and a Shaft and Headworks Construction Contract (which will be started in Year 5 as part of sustaining capital costs).

The UG lateral development and installations contractor will be responsible for the UG ramp, lateral and vertical (raising) development (in waste-rock only) starting from the completion of the ADEX program, and construction/installation of stationary mine infrastructure from Year -2, Month 6 to Year -2 Month 12. Starting in Month 1 of Year -1, Foran labour workforce will assume all underground development and construction activities.

All UG ore development and pre-production mining will be executed by Foran’s labour force. The mining (non-development) capital cost estimate is presented in Table 21-2 and the development capital cost estimate is present in Table 21-3.



Table 21-2: Mining (Non-Development) Capital Cost Estimate (\$ Millions)

Description	Pre-Production Capital Cost	Sustaining Capital Cost	Total Capital Cost
Mining (non-development)			
Engineering ²	8.13		8.13
Fixed Plant Procurement	13.70	41.82	55.52
Mobile Fleet Procurement	24.64	27.72	52.35
Mobile Fleet Rebuild/Repurchase		116.03	116.03
Surface Construction	2.04	17.34	19.39
Shaft and Collar		21.00	21.00
Ventilation Collars (FAR and RAR)	1.60	1.55	3.15
UG Construction	10.10	53.35	63.45
Contractor Indirects	6.01	13.01	19.02
Construction Indirects (Heating/Hoarding/Site Job/Tool Box)	0.13	0.95	1.08
Spares	0.77	1.39	2.16
First Fills	0.12	0.09	0.22
Freight	1.15	2.09	3.24
Subtotal Mining (non-development)	68.39	296.34	364.43
Contingency	7.91	4.41	12.32
Total Mining (non-development)	76.30	300.75	377.05

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.
2. Engineering costs for all consultants begin in Month 1, Year -2. Foran have committed to alternative funding for all consultant engineering prior to Month 1, Year -2.

Table 21-3: Development Capital Cost Estimate (\$ Million)

Description	Pre-Production Capital Cost	Sustaining Capital Cost	Total Capital Cost
Development			
Lateral Development	23.87	105.70	129.56
Vertical Development Mobilization/Demobilization	0.22	0.22	0.44
Vertical Development	3.42	16.43	19.85
Subtotal Development	27.51	122.35	149.86
Contingency	4.73	3.48	8.21
Total Development	32.24	125.83	158.07

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



21.1.3 Processing & Surface Infrastructure Capital Costs

Processing and surface infrastructure capital costs are summarized in Table 21-4 and Table 21-5. Project period capital is estimated to be \$219.7M, inclusive of \$30M indirect and \$25.8M contingency. Sustaining capital is estimated to be \$24.9M, inclusive of \$0.3M indirect and \$1.15M contingency.

The process plant capital cost was estimated using engineering documents developed during the Feasibility Study, mainly:

- Process design basis
 - Process Design Criteria (PDC)
 - Mechanical Equipment List (MEL)
 - Process Flow Diagrams (PFD)
- Tender Packages (equipment and contract packages)
 - Site Conditions
 - Engineering Design Criteria (EDC)
 - Scopes of work and datasheets
- Drawings
 - General site layout
 - Process and non-process building/ equipment layouts
 - Conveyor profiles
 - Power distribution and single line diagrams (SLD)
- Material Quantities
 - Cut, fill and rock excavation from Civil3D with input from geotechnical investigation report
 - Steel and concrete takeoff using major equipment and building loads
 - Platework takeoff from drawings/representative historic models
 - Major piping takeoff from layout drawings
 - Electrical bulks takeoffs from layout drawings
 - Valve & instrumentation estimate from PFD
 - Preliminary I/O count for control system



These costs represent site preparation and roads, process buildings complete with equipment, pipework, instrumentation and controls, waste rock pad, water management, power supply and distribution and ancillary non-process buildings and facilities as described in Section 18.0.

Costs for the mine development and surface infrastructure (hoist and ventilation) and the tailings storage facility are captured in Sections 21.1.2 and 21.1.4 respectively. Engineering consulting fees for the design are carried in Table 21-2 and construction management of the surface infrastructure are carried in the Owner's costs (Table 21-11) in Section 21.2.



Table 21-4: Process and Surface Infrastructure Capital Cost (\$ million)

WBS Description		Capital Cost (\$CAD, millions)		
		Project Period	Sustaining	Total
Process Cost	Process Building	24.44		24.44
	Primary Crushing (surface)	5.71	0.87	6.58
	Coarse Ore Stockpile	9.02		9.02
	Grinding	27.82		27.82
	Copper Flotation	6.75		6.75
	Zinc Flotation	5.31		5.31
	Pyrite Flotation	2.20		2.20
	Tailings Dewatering	22.07		22.07
	Concentrate Dewatering	5.86		5.86
	Reagents	1.48		1.48
	Additional Reagents	2.38		2.38
	Services	3.33		3.33
	Backfill Plant	10.00		10.00
	Operational Spares		0.23	0.23
	Process Area Indirect	25.09	0.17	25.26
	Process Subtotal	151.46	1.26	152.72
Surface Infrastructure Cost	Site Power supply / distribution	2.18	3.90	6.08
	Process Water	0.50		0.50
	Sewage Treatment	0.18		0.18
	Water Treatment	1.70	0.96	2.66
	Overall Site Plan	6.47	2.49	8.97
	Admin, Dry and Parking	6.97		6.97
	Permanent Camp	0.10		0.10
	Bulk Explosive Magazine Storage	0.07		0.07
	Truck Shop	9.32		9.32
	Bulk Fuel Storage	0.01		0.01
	Infrastructure Indirect	4.89	0.17	5.06
	Infrastructure Subtotal	32.41	7.52	39.93
	Transmission Line	10.00	15.00	25.00
	Contingency	25.82	1.15	26.97
Total Process and Surface Infrastructure		219.69	24.94	244.62

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



Processing & Surface Infrastructure Project Period Costs

Table 21-5 shows project period cost breakdown by discipline/activity. Costing inputs to the estimate were derived from:

- Civil (earthworks) – contractor supply and install unit rates (multiple budget bids)
- Concrete & structural steel - contractor supply and install unit rates (multiple budget bids)
- Platework - fabricator unit pricing for tankage and platework (multiple budget bids)
- Mechanical equipment – vendor pricing (multiple budget bids)
- Piping, valves & instruments – vendor unit rates and historic pricing
- Electrical equipment – vendor pricing (multiple budget bids)
- Electrical bulks – supplier unit rate pricing (multiple budget bids)
- Control system and programming – vendor pricing (multiple budget bids)
- Buildings & stockpile cover – contractor supply and install pricing (multiple budget bids)
- Foran / SaskPower transmission line agreement

Table 21-5: Processing and Surface Infrastructure CAPEX by discipline (\$ millions)

Discipline	Materials & Equipment	Labour & Contractor	Total
Civil (earthworks)	4.63	6.72	11.35
Concrete	6.49	7.70	14.19
Structural Steel	8.98	6.15	15.13
Platework	4.82	0.80	5.62
Mechanical	44.05	4.21	48.26
Piping	3.92	4.34	8.27
Electrical	9.95	2.45	12.40
Instrumentation	3.17	1.80	4.98
Architectural	1.72	31.97	33.69
138 kV Transmission Line		10.00	10.00
Direct Subtotal	87.73	76.15	163.88
Indirect			29.99
Contingency			25.82
Total			219.69

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



Processing & Surface Infrastructure Sustaining Costs

Sustaining capital costs (Table 21-6) are associated with the expansion of the effluent treatment plant to treat the expected increase in mine water with ongoing development, construction of the overland conveyor to coincide with commissioning of the mine hoist and waste rock pad expansion aligning with the production schedule.

The cost of the transmission line will be shared with SaskPower, with Foran contributing an estimated \$15M over production years 1 to 4. Prior to commissioning of the transmission line, a substation will be built by at the McIlvenna Bay site.



Table 21-6: Processing and Surface Infrastructure Sustaining Capital Cost

Description	(\$CAD, millions)										
	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Effluent Treatment Plant										0.96	0.96
Operational Spares		0.23									0.23
Overland Conveyor					0.87						0.87
110 kV Transmission Line	3.75	3.75	3.75	3.75							15.00
110 kV Substation				3.90							3.90
Waste Rock Dump Expansion		2.49									2.49
Direct Subtotal	3.75	6.47	3.75	7.65	0.87	0.00	0.00	0.00	0.00	0.96	23.45
Indirect				0.07	0.17					0.10	0.33
Contingency		0.35		0.55	0.12					0.13	1.15
Total	3.75	6.82	3.75	8.26	1.16	0.00	0.00	0.00	0.00	1.19	24.94

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



21.1.4 Tailings

Tailings Area pre-production capital costs are summarized in Table 21-7 and estimated to be \$13.0M, inclusive of \$2.9M mobile equipment purchase and \$1.7M contingency. The Tailings Area cost was estimated using engineering documents developed during the feasibility design, mainly:

- FS Drawings
- Mine Mass Balance
- Processing Schedule
- Geotechnical Site Investigations

Materials and quantities were derived based primarily on the following:

- Cut and fill excavations developed from Civil 3D with input from geotechnical investigations
- Neat line takeoffs for geosynthetics (Geomembranes, pipes, etc.)
- Estimated construction schedules

For estimating the direct costs, it has been assumed that all pad construction, lining, and site grading will be carried out under the same contract and timeframe as the plant site bulk earthworks. Costing inputs to the estimate were derived from:

- Earthworks – contractor supply and install unit rates (multiple budget bids)
- Structural and Concrete & structural steel - contractor supply and install unit rates (multiple budget bids)
- Geosynthetics - contractor supply and install unit rates (multiple budget bids)
- Mechanical equipment – vendor pricing (multiple budget bids)
- Pipework's, valves & instruments – vendor unit rates and historic pricing
- Mobile Equipment – vendor pricing and historic pricing
- Instrumentation – vendor pricing



Table 21-7: Tailing Capital Cost Summary

Description	Pre-Production Capital Cost	Sustaining Capital Cost	Total Capital Cost
Tailings			
Earthworks	3.68	11.18	14.86
Structural and Concrete	0.53	-	.53
Geosynthetics	2.65	7.46	10.11
Pipework's (Foundation drains)	0.15	-	0.15
Instrumentation	0.06	.03	0.09
Mobile Equipment	2.91	-	2.91
Indirects	1.32	2.62	3.94
Subtotal Tailings	11.30	21.29	32.59
Contingency	1.70		
Total Tailings	13.00	21.29	34.29

21.1.5 Site Closure Costs

Closure costs for the site are summarized in Table 21-8 and estimated to be \$8.2M. The closure costs were estimated based on engineering documents and layouts developed during the feasibility design. Closure costs are inclusive of all surface infrastructure (tailings and the process plant site, anticipated monitoring requirements and anticipated maintenance).

Materials and quantities for the closure costs are based on the following:

- Neat line estimates
- Conceptual closure details
- Monitoring durations for similar projects

For estimating the direct costs, it has been assumed that all pad construction, lining, and site grading will be carried out by an independent contractor. Costing inputs to the estimate were derived from the following:

- Earthworks – contractor supply and install unit rates (multiple budget bids)
- Demolition - historic contractor pricing from similar works
- Geosynthetics - contractor supply and install unit rates (multiple budget bids)
- Monitoring and Maintenance – recent pricing based on similar closure plans



Table 21-8: Closure Costs (\$ Millions)

Description	Progressive Rehabilitation	Active Closure	Total Cost
Tailings			
Earthworks	2.62	1.02	3.64
Geosynthetics	6.69	0.53	7.22
Decommission / Disposal	-	0.44	0.44
Monitoring and Maintenance	-	0.43	0.43
Subtotal	9.31	2.42	11.73
Process Plant			
Site Revegetation (incl. Stockpiles)	-	1.17	1.17
Openings	-	0.21	.21
Mine Waste Rock Stockpile	-	3.92	3.92
Ponds and Water Management Infrastructure	-	0.27	.27
Building Demolition	-	1.60	1.60
Equipment Removal	-	0.34	.34
Monitoring and Maintenance	-	0.69	.69
Subtotal	-	8.19	8.19
Total	-	10.62	10.62

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Totals may not sum due to rounding as required by reporting guidelines
3. The table excludes indirects

Note that progressive rehabilitation costs (\$9.31M and active closure costs (\$2.42M) are carried within the tailings OPEX (see Table 21-24).

21.1.6 Contingencies

Contingencies applied to the estimate were determined based on cost and schedule risk analysis conducted against the capital cost estimate report. Each main area within the CAPEX estimate costs were assessed on an individual basis and assigned a potential percentage increase applied against the estimate totals. Total weighted contingency costs were then calculated as provided in the cost summary tables above.

21.1.7 Pre-Production Operating Costs

The pre-production capital costs include operating costs incurred within the project period, prior to the process plant operating. The capitalized operating costs are presented in Table 21-9.



Table 21-9: Pre-Production Operating Costs

Item	Total (\$ millions)
Operating Development	\$5.5
Stoping Costs	\$1.1
Haulage	\$3.3
Power	\$2.6
Fixed Plant Maintenance	\$0.7
Indirect Equipment Operating	\$1.8
Owner's Team	\$10.2
G & A	\$11.2
Total	\$36.3

21.2 Operating Cost Estimates

Stantec, Halyard, and Knight Piesold developed operating cost estimates to determine the annual production costs. Unit costs are expressed in total LOM and \$/tonne processed. All costs included in the estimate are reported in Q1 2022 Canadian dollars.

Total LOM operating expenditures are estimated at \$1,926.7 million. Expenditures that are non-capital costs that occur prior to processing plant becoming operational in Year -1 are classified as Pre-production operating costs. Pre-production operating costs totaling \$36.3 million have been capitalized in the economic models for the project (refer to Section 21.1.7). The total operating cost expenditure less the pre-production operating cost is \$1,890.3 million (summarized in Table 21-11).

The operating cost estimate is divided into four broad areas:

- Mining
- Processing
- General and Administrative (G&A)
- 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.

The operating cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International with an approximate accuracy of $\pm 15\%$. All costs included in the estimate are reported in Q1 2022 Canadian dollars.



The operating cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope was described and quantified within material takeoffs (MTOs) in a series of line items.

A summary of the estimated LOM operating costs is presented in Table 21-10 and details of the cost breakdowns are summarized in the sub-sections that follow:

Table 21-10: Operating Cost Estimate

Operating Cost Item	LOM Total (\$ millions)	Cost \$/tonne Processed
Mining	1,060.6	41.26
Processing	615.9	23.96
G & A	174.2	6.78
Tailings	39.7	1.54
Total	1,890.3	73.55

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

21.2.1 Mining 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 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, maintenance, and general mine/indirect activities). Mine operating costs in the years prior to the processing of ore have been capitalized.

A summary of overall mine OPEX is presented in Table 21-11.

Table 21-11: Summary of Mine Operating Costs – Post Process Plant Operations

Operating Cost Item	LOM Total (\$ millions)	Cost \$/tonne Processed
Operating Development	61.6	2.40
Stoping Costs	135.9	5.29
Paste - Binder	173.4	6.74
Haulage	122.1	4.75
Power	90.7	3.53
Fixed Plant Maintenance	88.4	3.44
Paste Plant - Operations	12.9	0.50
Indirect Equipment Operating	49.6	1.93



Operating Cost Item	LOM Total (\$ millions)	Cost \$/tonne Processed
Owner's Team	326.1	12.69
Total	1,060.6	41.26

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

Owner's Team

The Owner's team operating cost listed in Table 21-11 is comprised of Management, Technical Services, Operating (non-direct), and Maintenance personnel. The non-direct operating includes underground general foreman, shift boss', hoist operators and cage tenders, rock breaker operators, grader operators, etc.

The total cost the Owner's team by labour category is presented in Table 21-12. The portion of the Owner's team costs prior to the mill operating (Year -2 and a portion of Year -1) is included in the pre-production capital cost.

Table 21-12: Owner's Team by Category

Labour Category	# of Positions Peak Year (On-Site)	LOM Total (\$ millions)	Capitalized (\$ million)	OPEX (\$ million)	Cost \$/tonne Processed
Management	7	37.4	1.6	35.8	1.39
Technical Services	17	50.0	0.8	49.1	1.91
Operating (non-direct)	15	88.7	1.6	87.1	3.39
Maintenance	27	158.6	4.7	154.0	5.99
Total	66	334.8	8.7	326.1	12.69

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.
2. \$/tonne processed consists of mine OPEX only.

Power

The total power demand for the UG operations has been calculated using connected load and adjusting for utilization (operating hours), demand factor (% of nameplate draw), drive efficiency, and system losses.

Power costs were calculated for Year -2 to Year 4 and Year 5 to Year 18 to reflect the varying mix of provincial grid and site power generation as detailed in Section 18.1. A summary of the resultant power consumption and cost is presented in Table 21-13.



Table 21-13: UG Power Consumption and Cost

Facility	Site Generated Power (Years -2 – 4)		Provincial Grid (Years 5 – 18)	
	Electricity Usage (MWh/Year)	Cost (\$ millions)	Electricity Usage (MWh/Year)	Cost (\$ millions)
Surface Ventilation	39,564	8.70	137,262	12.41
Headframe	0	0.00	15,003	1.36
Hoist House	0	0.00	121,584	10.99
UG Mobile Fleet (Development and Production)	24,275	5.34	44,949	4.06
UG Ventilation	23,162	5.10	72,959	6.60
UG Dewatering	3,768	0.83	44,720	4.04
UG Material Handling	0	0.00	12,072	1.09
UG Infrastructure Facilities	53,227	11.71	225,166	20.36
UG Compressed Air	1,520	0.33	4,266	0.39
Total	145,516	32.01	677,980	61.29

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

The portion of the power costs prior to the mill operating (Year -2 and a portion of Year -1) are included in the preproduction capital cost. A summary of the total underground power costs, the power cost carried in the Pre-production operating cost and the power cost carried in OPEX is presented in Table 21-14.



Table 21-14: Underground Power Cost by Period

Facility	LOM Total (\$ millions)	Capitalized (\$ million)	OPEX (\$ million)	Cost \$/tonne Processed
Surface Ventilation	21.11	0.54	20.57	0.80
Headframe	1.36	0.00	1.36	0.05
Hoist House	10.99	0.00	10.99	0.43
UG Mobile Fleet (Development and Production)	9.40	0.62	8.78	0.34
UG Ventilation	11.69	0.46	11.24	0.44
UG Dewatering	4.87	0.05	4.82	0.19
UG Material Handling	1.09	0.00	1.09	0.04
UG Infrastructure Facilities	32.06	0.93	31.14	1.21
UG Compressed Air	0.72	0.02	0.70	0.03
Total	93.30	2.62	90.68	3.53

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.
2. \$/tonne processed consists of mine OPEX only.

21.2.2 Processing Operating Costs

Processing operating cost are associated with mineral processing including crushing, ore storage, grinding, flotation, tailings and concentrate dewatering, reagents and utilities. Also included are non-process surface infrastructure costs including water management, domestic water and sewage treatment.

Process plant operating costs 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 salary build-ups for process labour. Maintenance costs were estimated as a percentage of mechanical equipment supply cost.

Quotations for consumables such as reagents, lime, binder, and grinding media were obtained from suppliers inclusive of transportation to McIlvenna Bay's site. Major consumables for mechanical equipment including crusher liners, SAG/ball mill liners and filter cloths/bags were sourced from vendor equipment quotations.

Annual and life-of-mine operating cost for the process and surface infrastructure are shown Table 21-15.



Table 21-15: Process and Surface Infrastructure Operating Costs Summary

Operating Cost	Site Generated Power		Provincial Grid		LOM	
	Annual (\$000's)	Per tonne milled	Annual (\$000's)	Per tonne milled	LOM (\$M's)	Per tonne milled
Fixed	14,765	9.77	13,357	8.83	235.6	9.17
Variable	28,368	18.76	20,359	13.46	380.2	14.79
Total	43,134	28.53	33,716	22.30	615.9	23.96

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

Fixed costs are itemized in Table 21-16 and consist of mainly labour, power for non-process infrastructure (water treatment administration / dry and truck shop), fuel, maintenance, non-process consumables (water / sewage treatment).

Table 21-16: Process and Surface Infrastructure Fixed Operating Costs

Operating Cost - Fixed	Site Generated Power		Provincial Grid		LOM	
	Annual (\$000's)	Per tonne milled	Annual (\$000's)	Per tonne milled	LOM (\$M's)	Per tonne milled
Labour	8,731	5.77	8,731	5.77	150.2	5.85
Power	2,519	1.67	1,111	0.73	24.9	0.97
Fuel	498	0.33	498	0.33	8.6	0.33
Maintenance	2,302	1.52	2,302	1.52	39.6	1.54
Consumables	77	0.05	77	0.05	1.3	0.05
Lubrication	100	0.07	100	0.07	1.7	0.07
Safety Equipment & Training	235	0.16	235	0.16	4.0	0.16
Assaying Cost	302	0.20	302	0.20	5.2	0.20
Total Fixed Operating	14,765	9.77	13,357	8.83	235.6	9.17

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

Variable costs are itemized in Table 21-17 and consist of throughput-based costs including reagents, process power and consumables/wear items for process equipment.



Table 21-17: Process and Surface Infrastructure Variable Operating Costs

Operating Cost – Variable	Site Generated Power		Provincial Grid		LOM	
	Annual (\$000's)	Per tonne milled	Annual (\$000's)	Per tonne milled	LOM (\$M's)	Per tonne milled
Reagents	8,270	5.47	8,270	5.47	140.6	5.47
Power	14,328	9.48	6,318	4.18	141.6	5.51
SAG/Ball Mill Grinding Media & Liners	4,862	3.22	4,862	3.22	82.7	3.22
Regrind Mill Grinding Media & Discs	607	0.40	607	0.40	10.3	0.40
Crusher Liners	138	0.09	138	0.09	2.4	0.09
Filter Clothes	162	0.11	162	0.11	2.8	0.11
Total Variable Operating	28,368	18.76	20,359	13.46	380.2	14.79

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

Labour

A total labour complement of 68 persons (Table 21-18) has been estimated for the process and paste plants, including operators, maintenance, laboratory technicians and management. Labour rates have been benchmarked against similar operations in Saskatchewan and Manitoba.

Table 21-18: Annual Process Labour Positions and Cost

Labour Category	Number of Positions	Annual Cost (\$CAD) \$000s
Operations - Management	7	1,030
Operations - Process Plant	14	1,644
Operations - Paste Plant	4	467
Metallurgy & Laboratory	11	1,263
Maintenance (mechanical)	18	2,435
Maintenance (electrical)	14	1,891
Total	68	8,731

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

Power

Total power demand has been calculated using connected load and adjusting for utilization (operating hours), demand factor (% of nameplate draw), drive efficiency and system losses for process equipment, infrastructure and buildings.



Power costs were calculated for years 1 to 4 and year 5+ to reflect the varying mix of provincial grid and site power generation as detailed in Section 18.7.5.

A summary of the resultant power consumption is included in Table 21-19.

Table 21-19: Power Consumption and Cost

Facility	Electricity Usage (MWh/Year)	Annual Electrical Cost (Years 1-4)		Annual Electrical Cost (Years 5+)	
		\$CAD, 000's	\$CAD/t milled	\$CAD, 000's	\$CAD/t milled
Admin/Dry	1,214	249	0.16	110	0.07
Truck Shop/Warehouse	2,907	596	0.39	263	0.17
Gate House	76	16	0.01	7	0.00
Camp	4,599	943	0.62	416	0.27
Overland Piping Heat Tracing	237	49	0.03	21	0.01
Effluent Treatment Plant	140	29	0.02	13	0.01
Sewage & Potable Water Treatment	350	72	0.05	32	0.02
Parking lot & Area Lighting	126	26	0.02	11	0.01
Process & Paste Plants	72,076	14,776	9.77	6,516	4.31
Electrical System Losses	2,640	541	0.36	239	0.16
Total	84,365	17,295	11.44	7,627	5.04

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

Reagents

The cost of the various reagents added to the process are summarized in Table 21-20. Costs are calculated using average LOM consumption rates scaled up from metallurgical testwork and multiplied by unit rates from reagent supplier's budget quotations which include transportation to the McIlvenna Bay site.



Table 21-20: Reagent Consumption and Cost

Reagent	Reagent Cost	Consumption		Consumption Cost	
	Cost (\$/kg)	(g/t milled)	(tonnes/yr)	Annual (\$CAD 000's)	(\$CAD/t milled)
NaOCl	1.60	-	20	36	-
Anthracite (1.2 to 2.5 mm)	1.60	-	7	12	-
Microsand (0.5 to 1 mm)	1.00	-	7	8	-
Subtotal Water Treatment				56	
Quick Lime	0.39	1,612	2,437	955	0.63
Sodium Metabisulfite	0.68	2,750	4,158	3,110	2.06
Copper Sulphate CuSO ₄	3.83	325	491	2,068	1.37
CYTEC AERO 3894 (totes or drums)	7.32	38	57	477	0.32
CYTEC AERO 3501 (totes or drums)	3.08	20	30	107	0.07
CYTEC AERO 5100 (totes or drums)	10.53	16	24	293	0.19
Xanthate, PAX	2.25	35	53	131	0.09
Organic gangue depressant PE26	3.22	33	50	177	0.12
Frother MIBC	5.23	25	38	217	0.14
Frother H57	7.76	100	151	1,291	0.85
Magnafloc 10	4.75	20	30	158	0.10
Subtotal Flotation				8,984	5.94
Total				9,040	5.94

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



Liners and Grinding Media

The cost and estimated consumption rate of the various steel/rubber wear liners, ceramic discs and steel / ceramic grinding media required for crusher and mill circuits are summarized in Table 21-21 and Table 21-22. Unit costs are quoted from local suppliers and include delivery to the McIlvenna Bay site.

Table 21-21: Grinding Media Consumption and Cost

Grinding Media	Unit Cost (\$/kg)	Consumption		Annual Consumption Cost	
		(Kg/t milled)	(tonnes/yr)	(\$CAD 000's)	(\$CAD/t milled)
SAG Mill (forged steel)	1,702	0.540	816	1,390	0.92
Ball Mill (forged steel)	1,633	0.760	1,149	1,876	1.24
Cu Regrinding Mill (ceramic)	21,357	0.120	15	330	0.22
Zn Regrinding Mill (ceramic)	21,357	0.085	7	146	0.10
Total				3,742	2.48

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

Table 21-22: Major Wear Component Consumption and Cost

Liners & Discs	Set Cost	Consumption	Consumption Cost	
	(\$/Set)	(set/yr)	Annual (\$CAD 000's)	(\$CAD/t milled)
Grizzly	5,000	2.5	13	0.01
Liner - Fixed Jaw	13,827	5.0	69	0.05
Liner - Moving Jaw	11,355	5.0	57	0.04
SAG Mill (Steel)	1,045,044	1.3	1,393	0.92
Ball Mill (Rubber)	118,188	1.7	203	0.13
Cu Regrinding Mill (Discs)	38,216	1.7	66	0.04
Zn Regrinding Mill (Discs)	38,216	1.7	66	0.04
Total			1,865	1.23

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

Maintenance

The estimated cost of spare parts and consumables used for routine process plant maintenance are factored from mechanical equipment supply costs. Factors vary from 4% to 6% of mechanical supply cost annually, depending on the plant area. Electrical maintenance budget is calculated as 1% of mechanical supply cost for all areas. Due to the highly automated nature of the process plant and remote operating capability, 5% of instrument/controls supply cost has been nominated for annual maintenance. Combined



annual budgets for maintenance parts and consumables is estimated at \$2,302,000 per annum.

Assay Lab

Assay lab 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 \$302,462 per annum.

21.2.3 General and Administrative Operating Costs

G&A costs include labour, miscellaneous surface equipment, contractor costs for camp and miscellaneous G&A items, such as catering, insurance premiums, allowance for training, consultants, and marketing and accounting functions.

All G&A costs are summarized in Table 21-23. G&A costs that occur prior to the process plant operating are included in the Pre-production capital costs (refer to Section 21.1.7).

Table 21-23: G&A Cost Estimate

Operating Cost Item	LOM Total (\$ millions)	Capitalized (\$ million)	OPEX (\$ million)	Cost \$/tonne Processed
Labour	87.6	3.82	87.65	3.41
Propane	33.3	1.48	33.30	1.30
Fuel (for surface equipment)	10.4	0.51	10.40	0.40
Services	5.6	0.29	5.56	0.22
Surface Support Equipment	3.3	0.19	3.25	0.13
Camp & Flights	2.1	3.94	2.11	0.08
G&A Misc. Items	31.9	0.94	31.88	1.24
Total	174.2	11.17	174.16	6.78

Note:

1. All figures are rounded to reflect the relative accuracy of the estimate. Totals may not sum due to rounding as required by reporting guidelines.
2. \$/tonne processed consists of mine OPEX only.

21.2.4 Tailings Operating Costs

Tailings operating costs have been estimated from first principles for haulage and placement of tailings using mine owned equipment. Progressive rehabilitation costs which are inclusive of lining and covering of the tailings. Both are summarized in Table 21-24.

Maintenance costs were estimated as a percentage of mechanical equipment supply cost and are included in the haulage and placement costs



Table 21-24: Tailing OPEX Estimate

Operating Cost Item	LOM Total (\$ millions)	Cost \$/tonne Milled
Tailings Haulage and Placement	30.3	1.18
Progressive Rehabilitation	9.4	0.36
Total	39.7	1.54



22.0 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 \$1,179.5 million, after-tax NPV (7%) of \$465.7 million, and after-tax IRR of 22%. The Project is most sensitive to the exchange rate and the copper price.

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 (post ADEX program) 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 inflation
- exploration / ADEX program costs are deemed outside of the Project
- any additional project development costs, including feasibility studies, or permitting activities, are excluded from the analysis



- annual gross revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year
- constant commodity pricing was used for the economic analysis
- a constant exchange rate assumption of US\$1.00 = C\$1.26 (or C\$1.00 = US\$0.79) 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 completed by estimating the overall value of all future cash flows (incoming and outgoing), and then discounting these 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 payback period is calculated from the commencement of mineral processing.

The model was developed using Microsoft Excel spreadsheet software.

22.4 Financial Model Parameters

22.4.1 Mineral Resource, Mineral Reserve, and Mine Life

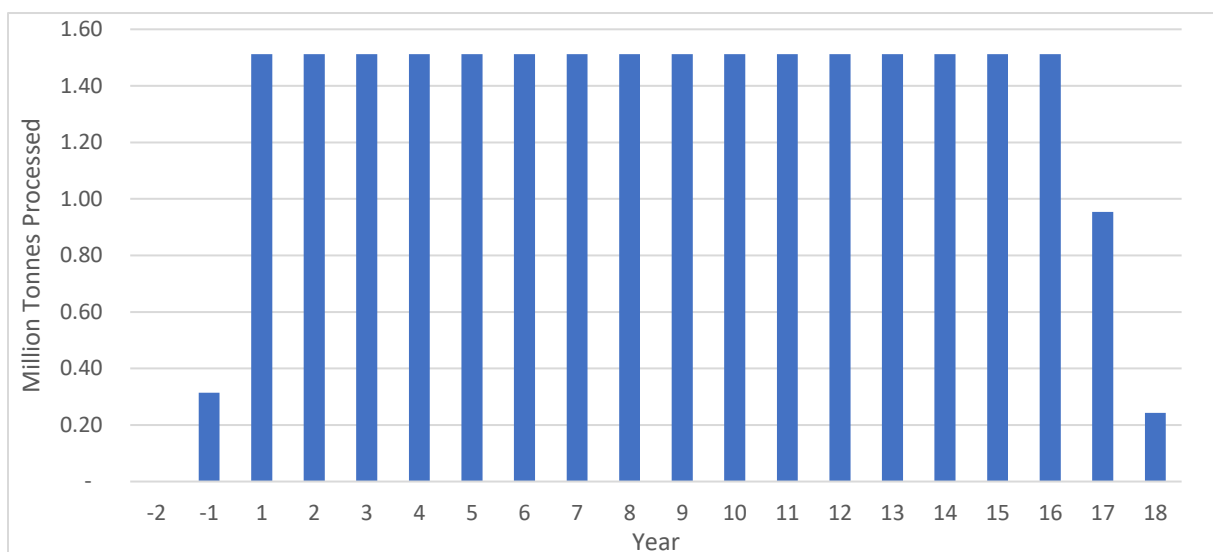
The mine plan presented in Section 16.0 of this report was used as the basis for revenue and cost estimates in the DCF, and also represents the current Mineral Reserve estimate. The mine plan excludes all material from the inferred resource category.

The stated Probable Mineral Reserve is 25.7 Mt at 1.23% Cu, 2.39% Zn, 0.47 g/t Au and 15.35 g/t Ag, resulting in a 2.51% copper equivalent grade (Cu. Eq).

The mine plan envisaged a 20-year LOM. The mill processing profile is presented in Figure 22-1.



Figure 22-1: LOM Processing Profile (Tonnes)



The production profile for payable Cu and Zn 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)

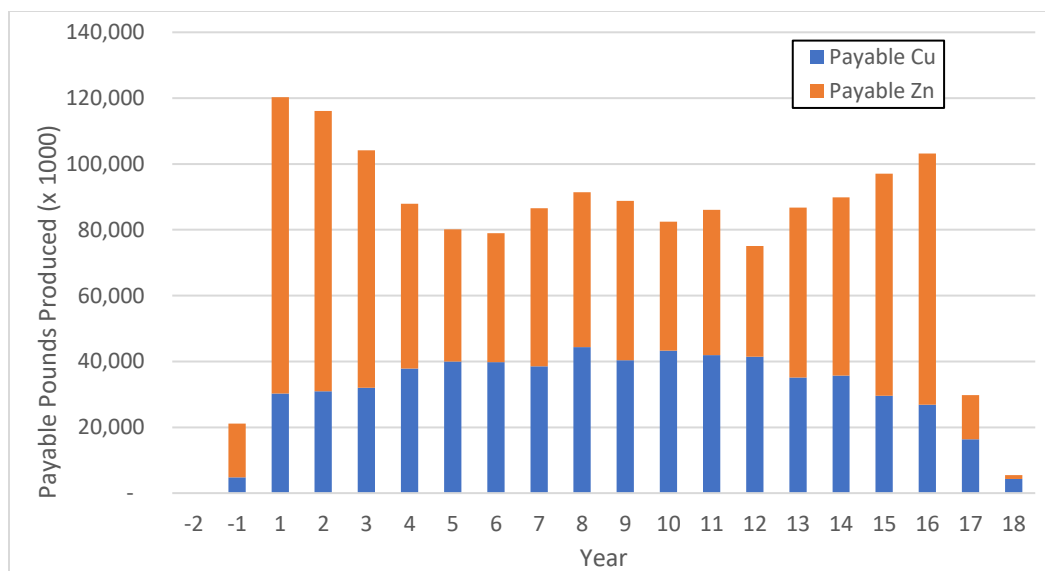
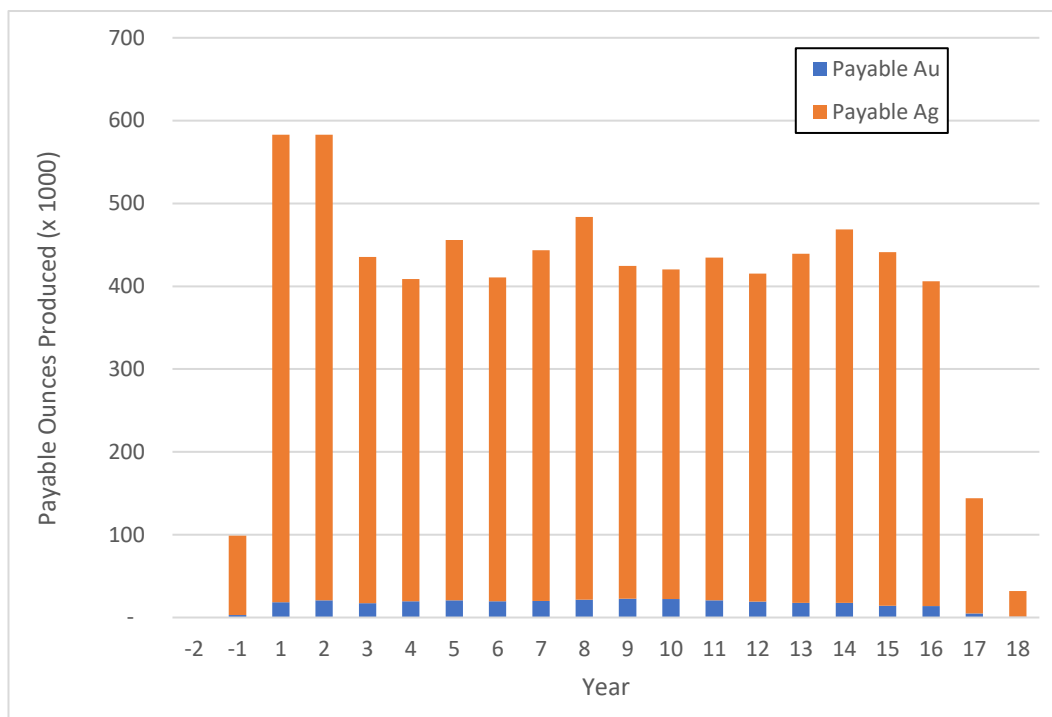


Figure 22-3: LOM Production Profile (Payable Au and Ag Produced)



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 versus recovery formulas were used to calculate recovery metrics on an annual basis within the cashflow model. A detailed review of metallurgical recoveries is provided in Section 13.0 of this report.

Table 22-1: LOM Average Metallurgical Recoveries

Reserve Type	Copper	Zinc	Gold	Silver
Massive Sulphide	82.4%	84.4%	82.9%	54.6%
Copper Stockwork	95.2%	47.0%	92.7%	76.6%
Blended Mill Feed	91.1%	79.8%	88.6%	63.2%

22.4.3 Freight, and Smelting and Refining

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, smelting and refining charges, and transportation. There no expected penalty fees related to out of specification minor elements.



Over the life-of-mine, the model anticipates production of 980,498 dry metric tonnes (dmt) of zinc concentrate with an average zinc grade of 49.9% and 1,028,448 dmt of copper concentrate with an average copper grade of 28.0%. Charges and penalties attributable to the zinc and copper concentrates over LOM are summarized in Table 22-2.

Table 22-2: LOM Toll, Smelting and Refining Charges, and Transportation,

Charge	Units	Zn Concentrate	Cu Concentrate	Total
Toll	\$ M	84.2	154.4	238.7
Smelting and Refining	\$ M	6.1	50.2	56.3
Transportation	\$ M	137.0	105.6	242.7
Total	\$ M	227.3	310.2	537.6

Note: numbers may not sum due to rounding

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 economic valuation were consensus, long term with a base date of January 2022. Metal prices used are summarized in Table 22-3.

Table 22-3: Metal Price Assumption

Metal	Units	Price
Copper	US\$ / lb	3.50
Zinc	US\$ / lb	1.20
Gold	US\$ / oz	1,600
Silver	US\$ / oz	22.50

A detailed review of metal prices is provided in Section 19.0 of this report.

22.4.5 Capital Costs

A summary of pre-production and sustaining capital costs is provided in Table 22-5. A detailed description of the capital costs is provided in Section 21.1 of this report.



Table 22-5: Capital Cost Summary (\$ Millions)

Description	CAPEX	SUSEX	Total
Mining	95.89	418.69	514.59
Processing	151.46	8.79	160.25
Infrastructure	32.41	-	32.41
Transmission Line	10.00	15.00	25.00
Pre-production Costs	36.34	-	36.34
Tailings	11.30	21.29	32.59
Closure	-	8.19	8.19
Total capital before contingency	337.40	471.96	809.37
Contingency	38.46	9.04	47.50
Total	375.87	481.00	856.87

22.4.6 Operating Costs

A summary of LOM operating costs is presented in Table 22-4. A detailed description of operating costs is provided in Section 21.2 of this report.

Table 22-4: LOM Operating Cost Summary

Operating Cost Item	Cost (\$/tonne Processed)
Mining	41.26
Processing	23.96
G & A	6.78
Tailings	1.54
Total	73.55

22.4.7 Royalties

Two royalties have been taken into account for this economic analysis:

- **BHP Royalty:** it has been assumed this royalty will be purchased for \$1.0 million as per the buy-out provision in the royalty agreement.
- **Copper Reef Royalty:** it has been assumed this royalty will be paid at the agreed rate of \$0.75 per tonne of ore mined.



22.4.8 Working Capital

Allowances for working capital beyond metals in inventory at the mine and mill have been excluded 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 \$8.2 million and represent closure costs for the entire project (surface infrastructure and tailings). Closure plan details are provided in Section 20.5 and costs are detailed in Section 21.1.5 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 continue after project closure.

22.4.11 Financing

Project level financing has not been considered 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 vendor for a portion of the mining equipment fleet. The financing terms were based on a lease-to-own basis with a 15% down payment and an effective financing rate of 0.79% over a 36-month period.

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 are illustrated in Table 22-4, Table 22-5, and Figure 22-4.



Table 22-4: Summary of Economic Metrics for the Project

Description	Units	Value
Pre-Tax		
NPV at 7%	\$ million	678.2
Internal rate of return	%	25.9%
Payback period	Years	3.9
After-Tax		
NPV at 7%	\$ million	465.7
Internal rate of return	%	21.8%
Payback period	Years	4.5

Note: Payback period calculated from start of mineral processing.

Table 22-5: Calculation of Post-tax Cash Flow

Description	Units	Value
Gross revenue	\$ million	4,931.5
Less: Smelter and refinery costs	\$ million	-537.6
NSR	\$ million	4,393.9
Less: royalties	\$ million	-20.3
Less: total operating costs	\$ million	-1,890.3
EBITDA	\$ million	2,483.3
Less: capital costs	\$ million	-848.7
Less: closure costs	\$ million	-8.2
Pre-tax cash flow	\$ million	1,626.4
Less: taxes	\$ million	-447.0
Post-tax cash flow	\$ million	1,179.5



Figure 22-4: Undiscounted After-Tax Cash Flow

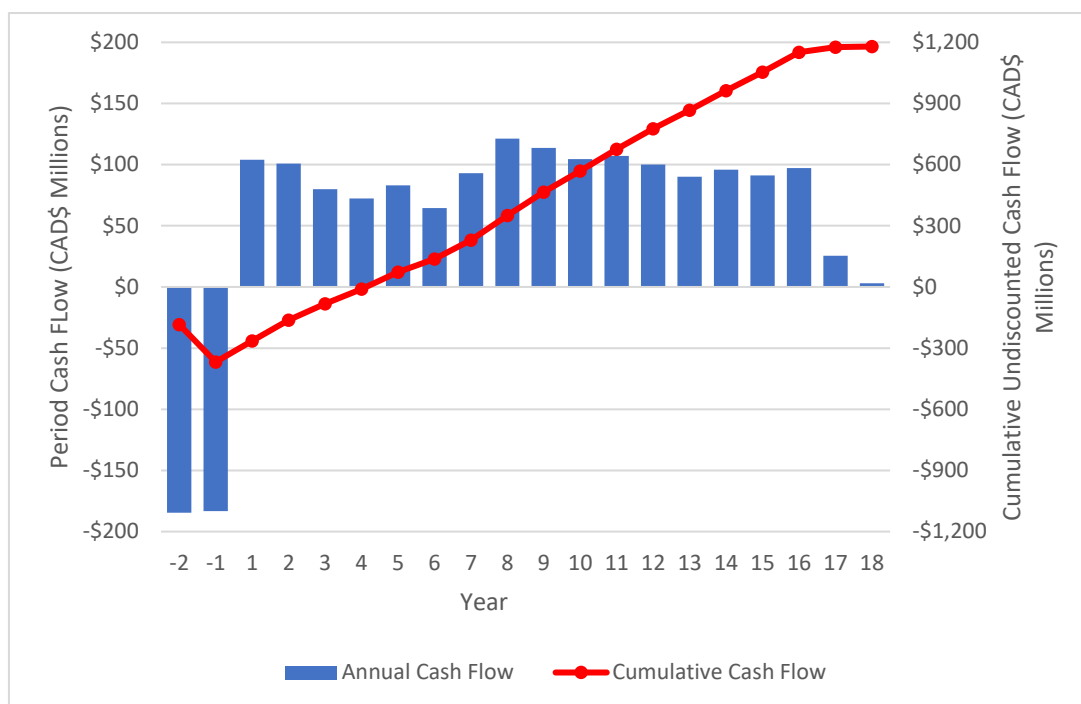


Table 22-6: Summary of Cash Flows

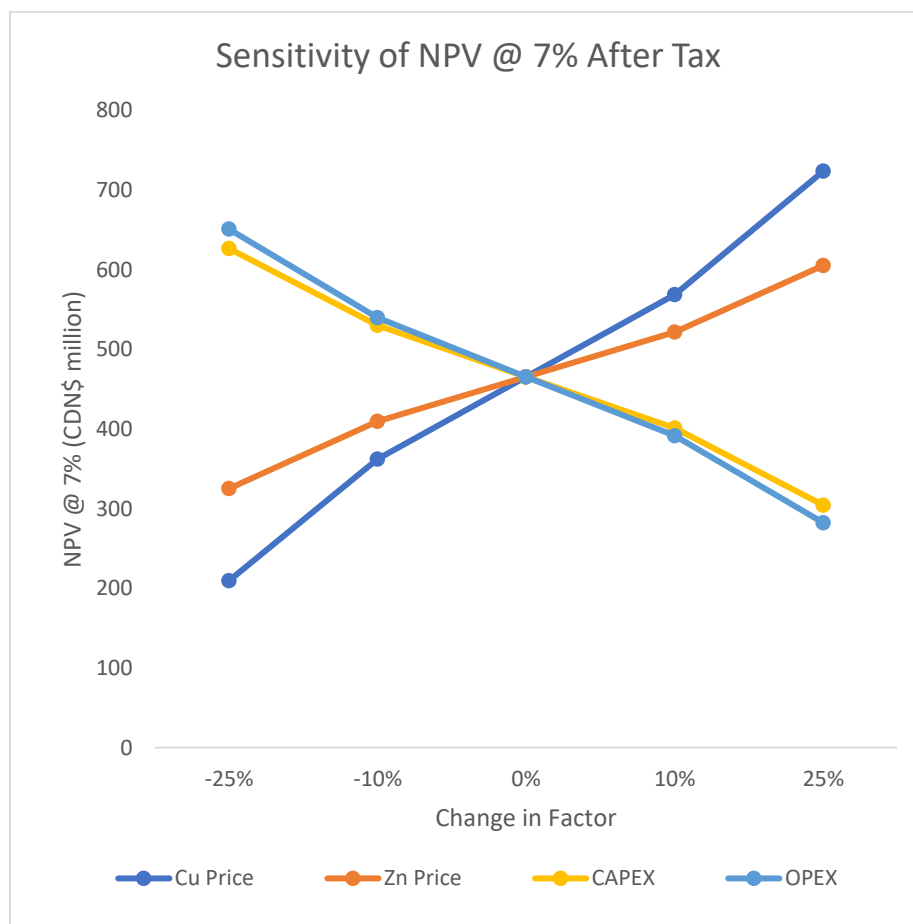
	Total	Year																			
		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Revenue																					
Gross Revenue																					
Copper	2,704.8	0.0	21.1	133.2	136.4	141.2	166.8	176.4	175.3	170.0	195.4	178.0	191.1	184.7	182.7	154.9	157.6	130.4	118.5	71.9	19.2
Zinc	204.5	0.0	2.7	16.0	15.9	11.8	11.0	12.3	11.1	12.0	13.1	11.4	11.3	11.7	11.2	11.9	12.8	12.1	11.1	3.9	0.9
Gold	634.8	0.0	6.0	36.8	42.1	34.9	39.3	41.6	39.3	40.6	43.3	46.0	44.6	42.2	39.0	35.7	35.4	28.4	27.5	9.7	2.2
Silver	1,387.4	0.0	24.6	136.1	128.8	109.1	75.8	60.7	59.3	72.6	71.2	73.3	59.2	66.8	50.9	78.0	81.9	102.0	115.4	20.3	1.7
Smelter Cost																					
Toll	(238.7)	0.0	(3.3)	(19.0)	(18.2)	(16.2)	(13.5)	(12.4)	(12.2)	(13.4)	(14.0)	(13.7)	(12.8)	(13.3)	(11.8)	(13.4)	(13.9)	(15.2)	(16.3)	(5.0)	(1.0)
Refining	(56.3)	0.0	(0.5)	(2.9)	(3.0)	(3.0)	(3.4)	(3.7)	(3.6)	(4.0)	(3.7)	(3.9)	(3.8)	(3.7)	(3.2)	(3.3)	(2.8)	(2.5)	(1.4)	(0.4)	
Transportation	(242.7)	0.0	(3.2)	(18.4)	(17.8)	(16.0)	(13.8)	(12.9)	(12.8)	(13.8)	(14.7)	(14.2)	(13.5)	(13.9)	(12.5)	(13.7)	(14.1)	(15.0)	(15.9)	(5.3)	(1.1)
Penalties	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Net Revenue	4,393.9	0.0	47.4	281.9	284.3	261.8	262.2	262.0	256.4	264.5	290.2	277.1	276.0	274.4	255.8	250.3	256.4	240.0	237.7	94.1	21.5
OPEX																					
Mine	(1,060.6)	0.0	(22.8)	(61.5)	(65.9)	(68.7)	(69.2)	(62.0)	(61.1)	(60.6)	(59.3)	(60.6)	(62.0)	(60.0)	(61.7)	(62.2)	(63.6)	(60.5)	(51.0)	(37.4)	(10.6)
Mill	(615.9)	0.0	(12.0)	(43.1)	(43.1)	(43.1)	(43.1)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(33.7)	(21.3)	(5.4)
G&A	(174.2)	0.0	(3.6)	(10.6)	(9.8)	(10.3)	(10.4)	(10.4)	(10.4)	(10.1)	(10.2)	(10.2)	(10.1)	(10.0)	(9.9)	(10.1)	(10.2)	(10.1)	(8.8)	(7.3)	(1.7)
Tailings	(39.7)	0.0	(0.6)	(2.4)	(2.3)	(2.4)	(2.4)	(3.0)	(2.4)	(2.2)	(3.0)	(2.0)	(2.1)	(2.9)	(2.0)	(2.1)	(3.0)	(1.8)	(1.8)	(0.7)	(0.5)
Total OPEX	(1,890.3)	0.0	(39.0)	(117.6)	(121.0)	(124.5)	(125.2)	(109.1)	(107.6)	(106.7)	(106.2)	(106.5)	(108.0)	(106.6)	(107.3)	(108.2)	(110.5)	(106.1)	(95.3)	(66.7)	(18.2)
EBITA	2,483.3	(0.0)	7.9	162.2	162.1	136.1	135.8	151.8	147.7	156.7	182.9	169.5	166.8	166.6	147.3	140.9	144.7	132.8	141.6	26.8	3.2
CAPEX																					
Mining	(514.6)	(49.3)	(46.6)	(50.1)	(37.4)	(34.9)	(31.8)	(41.3)	(57.9)	(31.2)	(23.5)	(18.8)	(19.3)	(19.1)	(11.5)	(12.2)	(13.1)	(8.7)	(7.8)	0.0	0.0
Processing	(160.2)	(66.5)	(84.9)	0.0	(2.7)	0.0	(4.0)	(1.0)	0.0	0.0	0.0	0.0	(1.1)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Infrastructure	(32.4)	(29.8)	(2.6)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Transmission line	(25.0)	(5.0)	(5.0)	(3.8)	(3.8)	(3.8)	(3.8)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Pre-production costs	(36.3)	(10.8)	(25.6)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings	(32.6)	0.0	(11.3)	(0.1)	(3.8)	(0.1)	(4.0)	(0.1)	(0.1)	(4.5)	(0.1)	(0.1)	(3.6)	(0.1)	(0.1)	(4.3)	(0.0)	(0.0)	(0.0)	0.0	0.0
Closure	(8.2)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	(8.2)
Contingency	(47.5)	(23.3)	(15.2)	(4.1)	(2.1)	(1.5)	(0.5)	(0.6)	0.0	0.0	0.0	0.0	(0.1)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total CAPEX	(856.9)	(184.7)	(191.2)	(58.1)	(49.7)	(40.2)	(44.1)	(43.0)	(58.1)	(35.7)	(23.6)	(19.0)	(24.1)	(19.2)	(11.6)	(16.6)	(13.1)	(8.8)	(7.8)	0.0	(8.2)
Royalty																					
BHP	(1.0)	0.0	0.0	(1.0)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Copper Reef	(19.3)	(0.0)	(0.5)	(1.1)	(1.2)	(1.2)	(1.2)	(1.1)	(1.1)	(1.2)	(1.1)	(1.1)	(1.2)	(1.1)	(1.2)	(1.1)	(1.2)	(1.1)	(0.8)	(0.6)	(0.1)
Total Royalty	(20.3)	(0.0)	(0.5)	(2.1)	(1.2)	(1.2)	(1.2)	(1.1)	(1.1)	(1.2)	(1.1)	(1.1)	(1.2)	(1.1)	(1.2)	(1.1)	(1.2)	(1.1)	(0.8)	(0.6)	(0.1)
Cash Flow (Pre Tax)	1,626.4	(184.7)	(183.3)	104.1	112.4	95.8	91.8	108.7	89.7	121.0	157.6	147.7	138.9	147.4	135.6	124.3	131.6	124.0	133.7	26.8	3.2
Taxes	(447.0)	0.0	0.0	0.0	(11.4)	(16.0)	(19.4)	(25.6)	(25.1)	(28.0)	(36.3)	(34.1)	(34.5)	(40.3)	(35.7)	(34.1)	(35.6)	(32.9)	(36.4)	(1.3)	(0.2)
Cash Flow (Post Tax)	1,179.5	(184.7)	(183.3)	104.1	100.9	79.8	72.3	83.1	64.6	93.0	121.3	113.6	104.4	107.1	100.0	90.2	96.0	91.1	97.3	25.5	3.0



22.6 Sensitivity Analysis

A DCF calculation was prepared for the Project based on the various cost and revenue inputs. A 25% increase and decrease for metal prices, capital costs, and operating costs was calculated, and the results are illustrated in Figure 22-5.

Figure 22-5: Sensitivity Analysis



23.0 ADJACENT PROPERTIES

The following section has been extracted from the previous Foran Technical Reports for the Mcllvenna Bay Project and updated or edited where necessary.

23.1 Base Metals

There are no producing metal mines adjacent to the Mcllvenna Bay property

Other VMS-style prospects are known to exist on Foran's claims and on adjacent ground (Figure 23-1). The more significant of these include the Balsam/Thunder Zone, located southeast of Mcllvenna Bay, the Miskat Zone, located in the southernmost extremity of the property and the historic Bigstone deposit located on an adjacent property, 25 km to the west.

The past producing Hanson Lake Mine is located approximately 5 km to the northwest of Mcllvenna Bay. The mine operated between 1967 and 1969 and produced 162,200 tons of ore 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.

On 23 January 2020, Copper Reef Mining Corporation (Copper Reef) announced in a press release the commencement of a 4,300 m drilling program in the Flin Flon Camp of Manitoba and Saskatchewan (Copper Reef 2020a). Copper Reef started with a 630 m program of 5 holes at its 100% owned Hanson Lake Property in Saskatchewan where Copper Reef has outlined two horizons each 2.5 km long.

Copper Reef's press release stated further:

The immediate drilling starting today will concentrate on the 2.5 km long Hanson Lake Mine horizon which was host to the former small but very rich Hanson Lake mine. The mine operated by Western Nuclear Mines between 1967 and 1969, produced 147,000 t containing 10% Zn, 5.8% Pb, 0.5% Cu and 137.0 g/t Ag. Although gold was noted, it appears to have not been recovered. Interesting is that there is no deep drilling below the mine where the existing workings did not extend below the 200 m level This leaves the horizon wide open for deep exploration. Copper Reef flew a deep penetrating VTEM airborne survey over the property, which showed that the Hanson Lake deposit extended significantly southward (700 m) beyond the mined area. In 1986, SMDC drilled a hole south of the mine that intersected 21.9% Zn, 1.6% Cu, 10.1% Pb and 28oz/ton Ag (960g/t Ag) and 0.89 oz/t Au (30.51 g/t Au) over 1.2 m just below the bottom of the lake. This high-grade intersection was not followed up with further drilling. Copper Reef intends to step back 20 meter and re-test this horizon at the 45 and 100 m levels and again from a separate set up 50 m south along strike of the first setup at similar depths.



Copper Reef will then step back and drill the horizon with a fifth drill hole at the 200 m level.

On 14 August 2020, Copper Reef announced that it has changed its name to Voyageur Mineral Explorers Corp. (Voyageur; Copper Reef 2020b).

Figure 23-1: Adjacent Properties

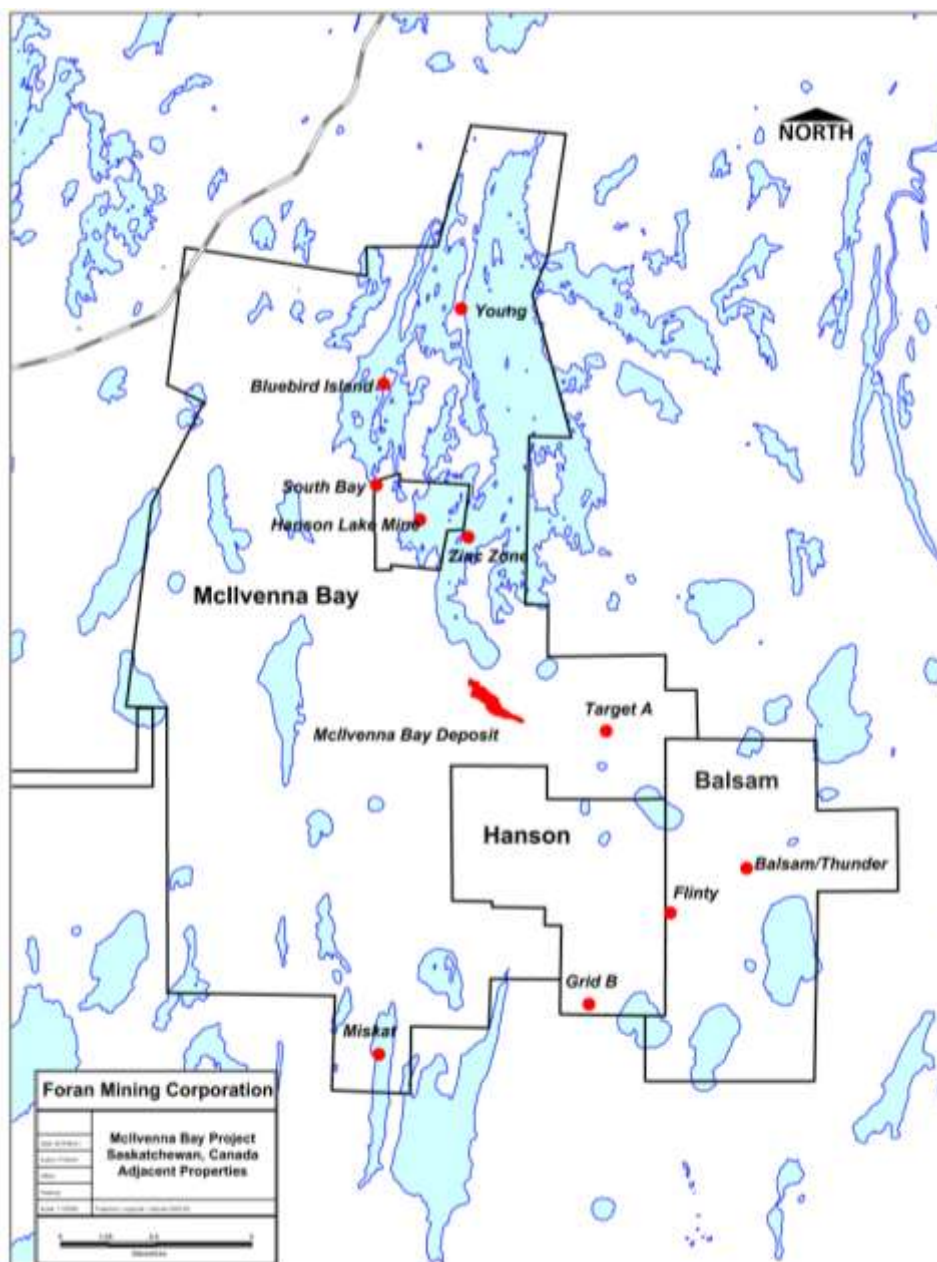


Figure provided by Foran dated June 2019.

There were no further press releases by Copper Reef or Voyageur regarding the results of the drilling at its Hanson Lake Property in 2020 or 2021.



23.2 Micon QP Comments

The Micon QP for this section does not consider that the information disclosed regarding the other base metal properties or deposits is indicative of mineralization within the McIlvenna Bay Deposit.



24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Risks

The Project team has identified risks associated with the technical and cost assumptions used. The risks have been classified as low/moderate/high and comment are provided on risk mitigation in the risk evaluation plan. In all cases, the level of risk refers to the subjective assessment as to how the identified risk could affect the achievement of the Project objectives.

24.2 Risk Analysis Definitions

The following definitions have been employed in assigning risk levels to the various aspects and components of the Project:

- **Low Level Risk** – Risks that could or may have a relatively insignificant impact on the character or nature of the deposit and/or the economics. Generally, can be mitigated using normal management processes combined with minor cost adjustments or schedule allowances.
- **Moderate Level Risk** – Risks that are considered to be average or typical for a deposit of this nature. These risks are generally recognizable, and through good planning and technical practices, can be minimized so the impact on the deposit or the economics is manageable.
- **High Level Risk** – Risks that are largely uncontrollable, unpredictable, unusual, or are considered not to be typical for a deposit of a particular type. Good technical practices and quality planning are no guarantee of successful exploitation. These risks can have a major impact on the economics of the deposit including significant disruption of schedule, significant cost increases, and degradation of physical performance. Included in this category may be environmental/social non-compliance, particularly in regard to Equator Principles and IFC Performance Standards.

In addition to assigning risk levels, the probability of the risk occurring during the Project has been considered. The following definitions have been employed in assigning probability of the risk occurring:

- **Low Probability** – The risk is unlikely to occur during the Project.
- **Moderate Probability** – There is an increased probability that the risk will occur during the Project.
- **High Probability** – The risk is likely to occur during the Project.

A summary of the Project related risks identified is presented in Table 24-1.



Table 24-1: Project Risk Evaluation

Project Element	Risk Description	Risk Level Classification	Risk Probability Classification	Existing / Proposed Strategies / Actions for Mitigation
Mineral Resources	There is a risk the resource tonnes and grade estimates are less than expected.	Low	Moderate	The core area of the deposit is drilled to the Indicated Mineral Resource category.
Construction / Operations	Ability to attract and retain competent and experienced personnel.	Low	Moderate	Compensation packages and site conditions favourable to attract and retain quality construction and operations personnel.
Mining	There is a risk of higher than anticipated ground water inflows from the sandstone layer near surface.	Moderate	Moderate	Update the hydrogeological model based on results from the portal grouting program. Use updated model to redesign underground dewatering system.
Mining	Collar construction through the sandstone/regolith layers for the FAR could be more challenging than estimated.	Moderate	Moderate	Review results of grouting program for the portal excavation, and use this information to develop the collar excavation methodology.
Mining	There is a risk that certain elements of the material handling system, including rock breaker through put, underground bins, skips, and surface bin, have not been sized to provide steady state and peak operations, including allowances for downtime.	Moderate	Low	It is recommended that material handling simulations be undertaken to identify critical pinch points.
Mining	Assumptions in the study is PAG material can be kept separate from clean waste rock.	Moderate	Moderate	Need to work closely with the geology team to develop methodology for defining PAG vs NAG.
Processing	There is a risk that metal recovery of the process plant is less than expected.	Low	Low	Testwork supports recovery assumptions. Additional testwork completed to provide optimization of flowsheet.
Processing	Copper concentrate grade lower than predicted, increased shipping costs	Moderate	Low	additional testwork to refine conditions



Project Element	Risk Description	Risk Level Classification	Risk Probability Classification	Existing / Proposed Strategies / Actions for Mitigation
Processing - Tailings	Concerns over stackable tailings in cold climates, limited bench marking examples	Moderate	Moderate	Review current mining operations in more detail, site visits, etc. to fully understand any potential issues
Regulatory	There is a risk the project will encounter environmental permitting delays	Moderate	Moderate	Continue to hold regular stakeholder and regulatory meetings.
Cost Estimation	There is a risk the cost of key materials and supplies will increase.	Moderate	Moderate	Continue to refine cost estimate, and identify key vendors.
Procurement	Lead times for certain equipment are highly variable. Some long-lead items could cause overall project delays if not procured in a reasonable timeframe.	Moderate	Low	Ongoing lead-time awareness of certain key pieces equipment should be prioritized, with the refinement of a long-lead register, and other procurement initiatives.

24.3 Opportunities

Foran and its lead consultants completed a review of the opportunities that should be explored both as the project is advanced through the next level of study, and into eventual construction and operations. Some of the key opportunities include the following.

- **Resource Expansion Potential** – The deposit extends down-plunge approximately 2km and remains open at depth and along strike.
- **Exploration Potential** – Foran is turning its exploration focus to advancing regional targets and deposits (see Section 24.4 for more detail).
- **Automation** – Foran will continue to evaluate emerging technology that will increase automation in the operations to improve safety and efficiency. This could include more autonomous production equipment, as well as support equipment, such as boom trucks for transporting materials underground.
- **Remote Operations Centre** – Foran is considering locating a control room offsite, where semi-autonomous equipment could be operated from. Certain other non-production functions could also be conducted from this site which would reduce camp and travel costs and potentially assist Foran to further attract and retain top-tier talent.
- **Reduced Backfill Costs** – Foran will investigate adding locally sourced sand (dolomite) in the backfill to reduce binder cost. While further testwork is required, the geochemical characteristics of the dolomite, combined with the cement used in the



paste, creates a stronger product compared to using tailings only. This would result in a lower binder requirement.

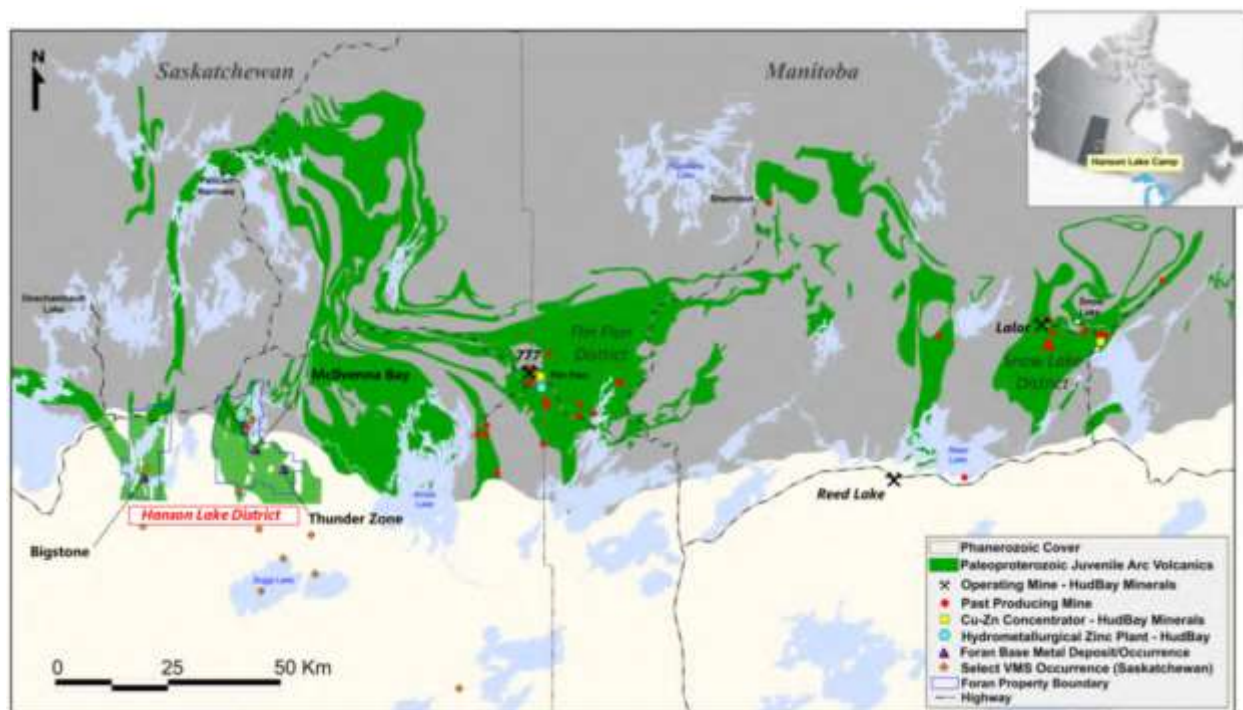
- **Ore Sorting** – Initial testwork conducted on core samples indicated amenability to using sorting technology. More testwork will be undertaken, using material from the planned bulk sample, to investigate the use of sorting in the flowsheet. The benefit would be upgrading the ore before it is milled, reducing costs.
- **Potential Expansions** – The processing plant is being designed as an initial 4,200 tpd plant that could accommodate future potential phased expansions as Foran advances and delineates additional near-mine deposits across the Hanson Lake District.
- **Alternative Power** – Investigate the use of renewable energy such as wind power as an addition to baseload power generation, to reduce greenhouse gas emissions, and reduce overall net power costs.
- **Project Execution Plan** – Continue to develop the project execution plan to reduce overall project risk, and identify areas of the initial capital cost outlay, such as the power plant, acid plant, and camp, that could be deferred through the use of specialized service providers.
- **Procurement** – Develop the supply chain network, including the establishment of key vendors, warehouse locations and function, and enterprise resource planning systems to streamline eventual project construction.

24.4 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 district represents the two western most volcanic assemblages which form part of the prolific Flin Flon Greenstone Belt that extends over 225 km from Snow Lake in Manitoba to the Bigstone Lake area in west-central Saskatchewan. This area is host to 29 past and present producing mines representing over 170 million tonnes of production (Figure 24-1).



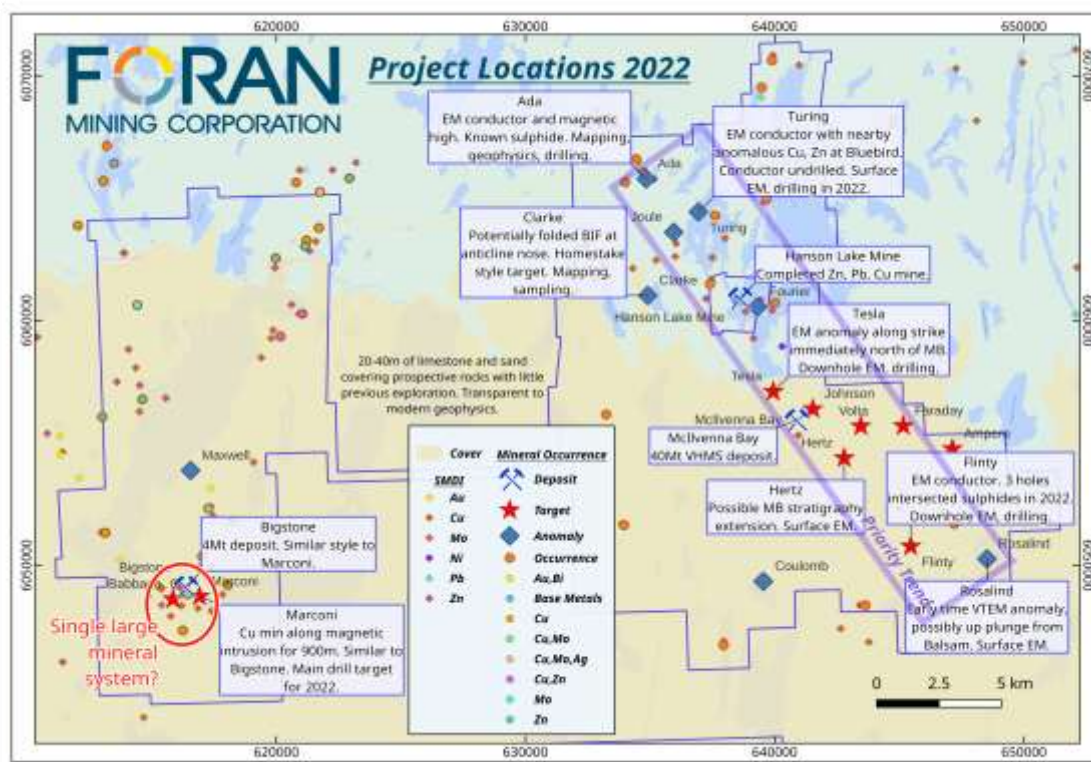
Figure 24-1: Flin Flon Greenstone Belt



There are numerous historic occurrences of VMS style mineralization and a number of high priority geophysical exploration targets in the Hanson Lake District, 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 25 km to the west of McIlvenna Bay which is host to the historic Bigstone Deposit and a number of other compelling exploration targets.

Subject to continued successful exploration, the more advanced of these current prospects that could potentially represent satellite mill feed for a central processing plant at McIlvenna Bay are the historic Bigstone Deposit located on the Bigstone property, 25 km to the west of McIlvenna Bay; Tesla located immediately north of the McIlvenna Bay deposit; Flinty, located approximately 7km to the SE of McIlvenna Bay; and Marconi, located 500m east of Bigstone (Figure 24-2).

Figure 24-2: Hanson Lake District Properties



Note: Source Foran

24.4.1 Bigstone

The historical 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 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 volcanoclastic 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.

In 2015, Foran completed a six-hole 2,545 m drill program 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 depth, and 3.16% Cu over 19.00 metres starting at 363.00 metres depth. A typical cross-section through the Bigstone Deposit, which includes the results from two of the 2015 drill holes, is provided in Figure 24-4 below.

Based on the 2015 drilling, A NI 43-101 compliant Mineral Resource estimate was prepared by RPA in 2021, which estimated a Mineral Resource at Bigstone an estimated resource of 1.98 million tonnes (“Mt”) in the Indicated category grading 2.22% copper equivalent (“CuEq”) and an additional 1.88 Mt of Inferred resources grading 2.14% CuEq based on 55 drill holes.

Table 24-2: Bigstone Resource Estimate (RPA, 2022)

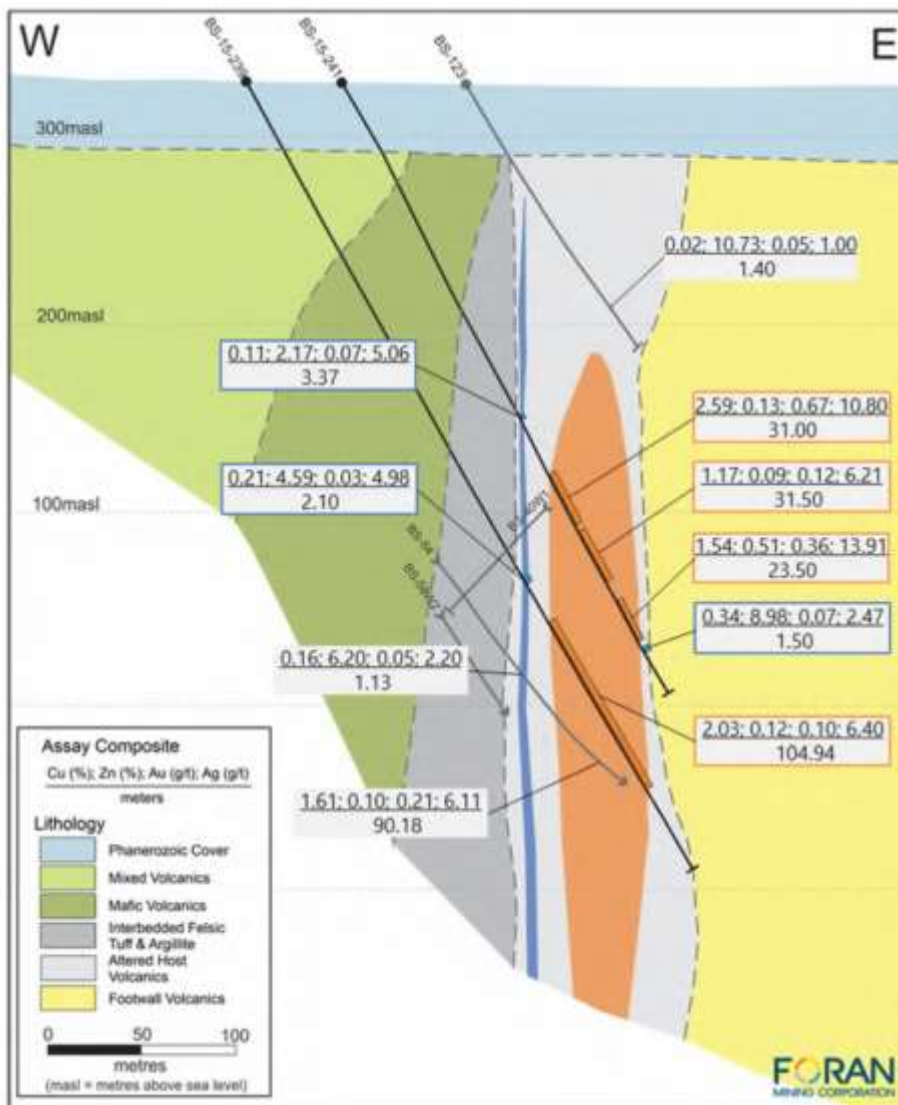
Category	Tonnes	Grade				Contained Metal				
		CuEq	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag
	(kt)	(%)	(%)	(%)	(g/t)	(g/t)	(Mlb)	(Mlb)	(000 oz)	(000 oz)
Indicated	1,979	2.22	1.88	0.92	0.25	9.5	81.9	40.2	16	603
Inferred	1,884	2.14	1.35	2.75	0.32	12	55.9	114.4	19	729

Note:

1. Table Source Technical Report on the Bigstone Project, East Central Saskatchewan, Canada
2. CIM (2014) definitions were followed for Mineral Resources
3. Mineral Resources are estimated at average long-term metal prices of Cu: US\$3.75/lb; Zn: US\$1.35/lb; Au: US\$1,650/oz; and Ag: US\$21.00/oz
4. Mineral Resources are constrained using underground mining shapes for reporting
5. Mineral Resources were estimated at a cut-off Net Smelter Return (NSR) value of US\$65/t
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
7. Copper equivalent (CuEq) is based on metallurgical recoveries and smelter terms by zone, long-term metal prices, and off-property costs. Copper in the Copper Zone is the basis, while contributions from other metals and copper in other zones are converted based on equivalent net value.
8. Numbers may not add due to rounding



Figure 24-3: Bigstone Deposit – Typical Cross Section (1675N)



Note: Source Foran NI 43-101_technical_report_pre-feasibility_study

The Bigstone Mineral Resource estimate demonstrates the prospective nature of the stratigraphy in the area to host potentially economic concentrations of mineralization. VMS deposits typically occur in clusters. Past geophysical surveys have identified numerous geophysical conductors and anomalies and there remains good potential to identify additional occurrences on the Property with continued drilling and exploration (RPA, 2021).



24.4.2 Tesla

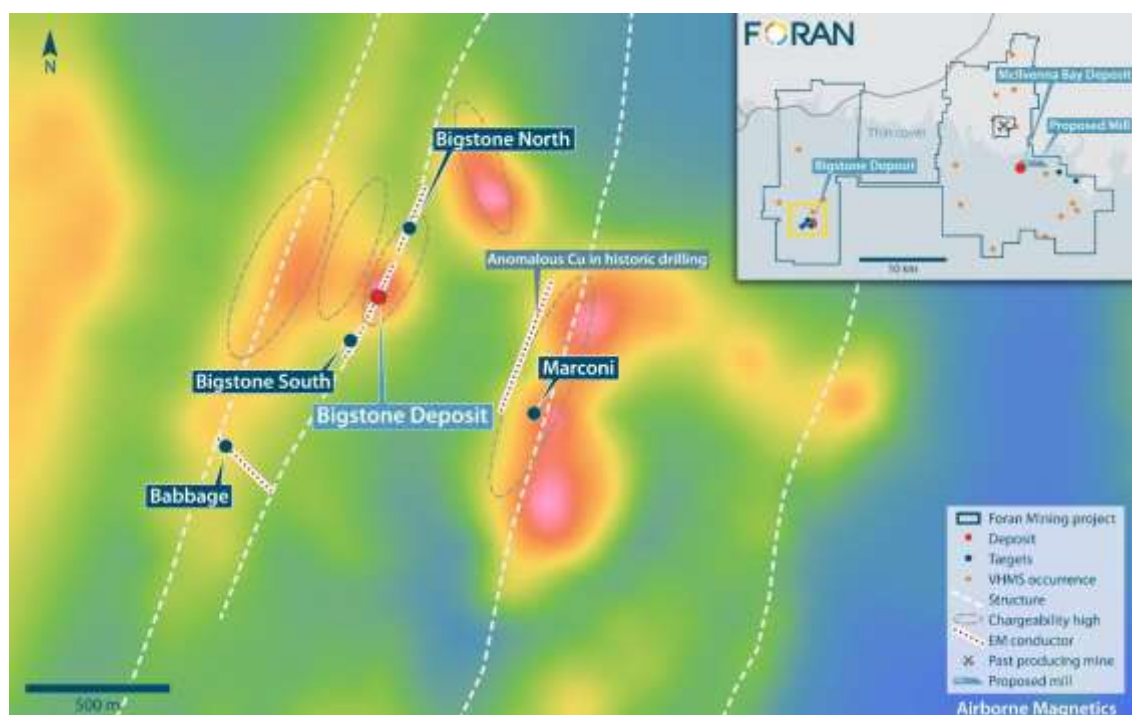
The Tesla prospect is a 800m long electromagnetic (EM) anomaly immediately north of Mcllvenna Bay. A single hole was drilled in late 2021 and down hole EM shows a strong conductor nearby the initial target. Additional exploration is warranted to validate and fully explain the anomaly due to its location and potential association with the Mcllvenna Bay resource.

24.4.3 Marconi

The Marconi prospect is 25km west of the Mcllvenna Bay deposit and 500m East of the Bigstone Deposit. Historic drilling has outlined anomalous copper mineralization extending 900m along strike and remains open in all directions.

Both Marconi and the Bigstone deposit, located 500m to the west of Marconi are interpreted to have a strong association with magnetite and similarities between the two suggest they may have, in part formed through similar processes and be locally bound by controlling structures.

Figure 24-4: Marconi – Airborne Magnetics



Re-interpretation of airborne electromagnetic work by Foran in 2021 has identified a magnetic low anomalous region similar to the Bigstone Deposit. This zone of reduced magnetic amplitude along the main magnetic ridge is also a target as this would be consistent with hematite (non-magnetic) alteration rather than magnetite.

Additional exploration work is warranted to follow up on the anomalous copper in historic drilling, and to test structural controls and the potential relationship between Marconi and Bigstone.

24.4.4 Flinty

Three historic holes were drilled at Flinty and intersected stockwork sulphides. Down hole EM was completed at the time with a conductor being interpreted between and below the 3 holes. Foran completed a surface EM survey in early 2021 and a conductor is modelled in a position consistent with the historic down hole EM. A second conductor is modelled from Foran's surface EM survey approximately 400m to the south. Both these targets were tested during 2022 with 3 drill holes.

All conductors were intersected close to the modelled depth and there were significant sulphides in all holes.

A fixed loop EM survey has been executed to cover 4km² from the Flinty holes across to an early-time VTEM conductor 350m to the NE and 2300m in strike length. This has confirmed the presence of the conductor and will provide the basis for modelling and follow-up drilling once final data is received.

A second fixed loop EM survey has been completed along strike from Flinty to the SE to cover anomalous features in VTEM. This is envisioned as a target for 2023.

24.5 Frac 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 25 m of dolomite cap rock was blasted and removed, accessing three to five meters of silica sand. 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 shut down operations and the site was subsequently reclaimed by pushing the waste rock and remaining sand back into the pits and re-contouring the landscape. Figure 24-5 is a view of the Preferred Sands quarry in July 2011.



Figure 24-5: View of Preferred Sands Quarry, July 2011



Picture taken from the 2020 APG Prefeasibility Technical Report

The sand quarry leases overlie Foran's mineral tenure in the area (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. 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. When Preferred Sands shutdown operations in December 2014, Foran acquired the five quarry leases from Preferred Sands that were in the vicinity of the McIlvenna Bay Deposit to ensure that there was no further potential for conflict. In 2021, the Saskatchewan government entered into a transfer agreement with Foran to transfer the surface lease to Foran upon completion of the environmental assessment.

Hanson Lake Sand Corp. (now Strong Pine Energy Services [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 Mcllvenna 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 Mcllvenna Bay Deposit. No value has been taken or is implied from frac sand in this study.



25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineralization

25.1.1 General Notes

Previous iterations of the resource model have been completed and published since 2010, with all these previous resource estimations now superseded by the current 2021 estimate discussed herein.

The McIlvenna Bay mineral resources have been estimated using multiple tabular interpretations defined in five mineralization zones, Copper Stockwork (CSZ), Massive Sulphide (MS), Lens 3 (L3), Stringer Zone (SZ) and Copper Stockwork Footwall (FW). The five zones contain steep parallel, contiguous vein-type structures disposed next to each other, with similar bearings and dips. The mineral resources for the McIlvenna Bay zones have been estimated assuming an underground mining scenario.

25.1.2 Supporting Database

The basis for the mineral resource estimate was a drill hole database provided by Foran. The database and underlying QA/QC data were validated by Foran prior to being used in the modelling and estimation. Table 25-1 summarizes the types and amount of data in the database and the portion of the data used for the mineral resource estimate.

Table 25-1: McIlvenna Bay Project Database

Data Type	In Database	Used For 2021 Resource Estimate*
Drill Collar	285	240
Assay Samples	11,737	5,652
Core Metreage	152,130	4,658**

*Excludes four drill holes from the resource estimate.

**Actual meters used within the resource wireframes.

The digital terrain model (DTM) was of sufficient quality, although, given the underground extraction assumption, it was not used for the mineral resource estimate.

25.1.3 Wireframes and Other Modelling Parameters

Wireframes

Foran and Micon jointly defined five mineralized zones, representing different areas and styles of VMS mineralization:

- Copper Stockwork Zone (CSZ) - sitting stratigraphically below the Massive Sulphide.



- Massive Sulphide (MS) - main mineralized lens with internal gradational boundaries. The lens was previously modelled as two separate zones (MS and Upper West), but contact plots indicate no justification for a hard boundary.
- Lens 3 (L3) – A massive sulphide zinc lens located in the hanging wall of the Stringer Zone.
- Stringer Zone (SZ) - Copper and zinc stringer zone in the hanging wall above the Massive Sulphide.
- Copper Stockwork Footwall (FW) - Small massive to semi-massive ore zone below the CSZ.

The wireframes for each of the five domains were validated against drill hole data and determined to reasonably represent the mineralization and the host rock.

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 for the deposit.

Compositing

The selected intercepts for the Mcllvenna Bay Project were composited into 1.0 m equal length intervals, with the composite length selected based on the most common original sample length.

Variography

Micon obtained good variogram models for all five zones. The models were sufficiently reliable to support the use of the Ordinary Kriging interpolation method. Major variogram ranges between 60 m and 125 m were modelled. Most variograms were in the range of 100 m to 125 m for both copper and zinc. The variography results were used to support the search ranges and anisotropy directions.

Continuity and Trends

The Mcllvenna Bay mineralized zones exhibit fairly stable strike and dip directions, with mild variations. For the most part, both CSZ and MS zones are contiguous, with the remaining zones running as parallel structures with well-defined geometries. Continuity of the zones is supported not only by geology, but also by mineralization, with the regular drill hole intercepts providing sufficient confidence to the continuity of grade both along strike and down dip. The overall deposit bearings and dips are 315° strike direction and -68° dip, with a general plunge of -40° northwest.

Capping

All outlier assay values for copper, zinc, lead, gold, and silver were analyzed individually by zone using log probability plots and histograms. Outlier assays were capped based



on the data grouped by zone. To identify true outliers and reduce the effect of short sample bias, the data were reviewed after compositing to a constant interval length of 1.0 m.

Density

Foran has taken density measurements as the exploration programs continue, generating a large database of measurements that permitted the determination of density by means of interpolation.

Measurements of bulk density are the more robust method, and these data points (1,072, in total) were given precedence in the population of density in the block model. The final block model value was assigned using a rolling average (ID0) for each domain, thus generating a smoother continuity of density.

25.2 Mineral Resource Estimate

The commodities of economic interest at the McIlvenna Bay Project are primarily copper and zinc, with secondary recoveries of gold and silver. The estimation of the deposit tonnage and grade was completed using Leapfrog Geo/EDGE software.

25.2.1 Block Model

A block model was constructed to represent the grade and densities within the five zones. The drill hole intercepts used to model the wireframes were flagged into the corresponding mineral envelope. Each zone was interpolated using only the composites within that zone.

25.2.2 Mineral Resource Classification

Micon has classified the mineral resource at the McIlvenna Bay Project in the indicated and inferred categories. A measured resource is not declared at this time. The FW, L3, and SZ zones are classified entirely as indicated resources.

Indicated resources were restricted to those blocks informed by at least four drill holes and within 100 m to 120 m spacing based on the ranges obtained in the variograms. The results were then smoothed to remove small, isolated blocks and produce coherent shapes of reasonable volume, eliminating the spotted dog effect. All other blocks were classified in the inferred category.

25.2.3 Mineral Resource Estimate

Micon's updated mineral resource estimate is summarized in Table 25-2. The effective date of this mineral resource estimate is 06 September 2021, and the estimate is reported using an NSR cut-off of US\$60/t.



Table 25-2: Mineral Resources for the McIlvenna Bay Deposit, Reported at an NSR of US\$ 60/t

Category	Zone	Mass (Mt)	NSR (US\$/t)	Average Grades						Contained Metal				
				Cu (%)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu (Mlb)	Zn (Mlb)	Pb (Mlb)	Au (Moz)	Ag (Moz)
Indicated	MS	10.75	198.8	1.01	6.17	0.41	0.53	26.56	3.13	238	1,462	98	0.18	9.2
	CSZ	22.74	127.1	1.31	0.38	0.02	0.37	9.14	1.60	659	190	10	0.27	6.7
	SZ	1.19	119.3	1.26	0.52	0.07	0.31	12.97	1.53	33	14	2	0.01	0.5
	L3	2.57	113.1	0.82	3.07	0.14	0.25	14.51	1.80	47	174	8	0.02	1.2
	FW	1.80	140.7	1.42	0.59	0.04	0.45	8.84	1.79	56	23	2	0.03	0.5
	Total	39.06	146.3	1.20	2.16	0.14	0.41	14.39	2.04	1,033	1,863	119	0.51	18.1
Inferred	MS	1.56	162.6	0.65	6.51	0.46	0.29	27.77	2.66	22	224	16	0.01	1.4
	CSZ	3.48	105.6	1.08	0.79	0.03	0.25	10.50	1.37	83	60	3	0.03	1.2
	Total	5.04	123.3	0.94	2.56	0.17	0.27	15.85	1.77	105	284	19	0.04	2.6

Notes:

1. Effective date 06 September 2021; CIM definitions were followed for Mineral Resources; CuEq = copper equivalent; NSR = Net Smelter Return.
2. The mineral resource is estimated based on 240 diamond drill holes and a NSR cut-off of US\$60/t. NSR grades values derived, and high-grade caps were applied as per the discussion in Estimation Methodology and Parameters and include provisions for metallurgical recovery and estimates of current shipping terms and smelter rates for similar concentrates. Metal prices used are US\$4.25/lb. Cu, US\$1.35/lb. Zn, US\$1,800/oz. Au, and US\$25.00/oz. Ag. Lead contributes no value.
3. Rock density was interpolated for each block based on measurements taken from core specimens, with an average value of 3.59 g/cm³ for the main MS lens and 2.87 g/cm³ for the CSZ.
4. Mineral resources which are not mineral reserves do not have demonstrated economic viability.
5. Cu.Eq values were calculated from the NSR values for each zone, using both concentrate and recovery curves that were developed during prefeasibility-level metallurgical studies.
6. The block model grades were estimated using the Ordinary Kriging interpolation method, with search parameters derived from geostatistical analysis performed within the mineralization wireframes. Variogram ranges are from 65 m to 85 m for Au and Ag in the major axis and up to 100 m to 120 m for Cu and Zn.
7. Micon has not identified any legal, political, environmental, or other factors that could materially affect the potential development of the mineral resource estimate.
8. The mineral resource estimates are classified according to the CIM Definition Standards
9. The mineral resource was categorized based on geological confidence into inferred and indicated categories. An inferred mineral resource has the lowest level of confidence. An indicated mineral resource has a higher level of confidence than an inferred mineral resource. It is reasonably expected that most of the inferred mineral resources could be upgraded to indicated mineral resources with additional infill drilling.

25.2.4 Conclusion

Foran's exploration activities have been successful in increasing the confidence in the geological interpretation of the McIlvenna Bay deposit, as well as expanding upon the previous mineral resource estimates. Micon and its QPs consider the current mineral resource estimate is robust.

25.3 Mineral Processing and Metallurgical Testing

Based on the metallurgical testwork completed to date, the following conclusions are drawn.



- Copper in the Stockwork and Massive Sulphide zones is mainly present as chalcopyrite, with minor amounts of secondary copper sulphides. Copper minerals are mainly associated with pyrite, and to a lesser extent pyrrhotite and sphalerite.
- Zinc is present in the Copper Stockwork Zones in minor quantities, primarily as sphalerite. Higher zinc grades are observed in the Massive Sulphides, with the majority of zinc present as sphalerite and minor amounts of the zinc oxide mineral gahnite.
- Grindability testwork has indicated the Copper Stockwork Zones is hard in terms of SMC and BBWI. The Massive Sulphide zone is of moderate hardness by comparison.
- Flotation testwork on samples from the Mcllvenna Bay Deposit have indicated that a primary grind size P80 of 75 µm followed sequential flowsheet consisting of copper flotation followed by zinc activation and flotation can achieve good recoveries to separate copper and zinc concentrates.
- Regrinding of both the copper and zinc rougher concentrates to a P80 of 20–25 µm is sufficient to achieve good sulphide liberation and yield typical copper and zinc concentrate grades.
- Locked cycle testing on zone and blended composites has confirmed the flowsheet and reagent additions, as well as grades and recoveries of the final concentrates. Minor element analysis has indicated the zinc and copper concentrates may contain elevated concentrations of mercury and selenium, respectively, that may attract a smelter penalty.
- Sulphide flotation tests on zinc rougher tailings has indicated that a desulphurized tailings can be generated, achieving up to 98% stage recovery of the remaining sulphides.

25.4 Mineral Reserve Estimate

The mineral reserves estimate for the Mcllvenna Bay Project has been completed in accordance with accepted practice and meets the CIM Definition Standards for Mineral Resources and Reserves (CIM Council 2014).

Many factors may affect the mineral reserves estimate such as dilution, metal prices, metallurgical recoveries, geotechnical characteristics of the deposit and host rock, and the capital and operating cost assumptions. These factors have been taken into account subject to the information and means available to QPs at the time of this 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 value exceeded the cut-off value. The result was the reserve estimate on which the mine plan was based.



As with the resources, the multi-element nature of the McIlvenna Bay Deposit requires that an NSR value was used for the application of a cut-off grade to the block model. The NSR was estimated for each block using 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 using a processing facility at site. The mineral reserve estimate is shown in Table 25-3.

Table 25-3: Mineral Reserve Estimate – Reported at an NSR Cut-off of US\$90.24/t

Classification	Recovered Ore Tonnes	Grade Zn (%)	Grade Cu (%)
Massive Sulphide	10,135,305	5.43 %	0.99 %
Copper Stockwork	15,567,756	0.41 %	1.39 %
Total	25,703,062	2.39 %	1.23 %

1. CIM definitions were followed for mineral reserves.
2. Mineral reserves include long hole transverse and longitudinal, and Avoca stopes, as well as ore development, marginal development, and incremental stopes.
3. Stopes were estimated at a cut-off value of US \$90.24/tonne NSR.
4. Marginal tonnes were estimated at a cut-off value of US \$60.55/tonne NSR.
5. A minimum mining width of 3.0 m was applied for all stoping.
6. Numbers may not sum due to rounding.
7. NSR reserve prices (\$US); Cu \$3.50/lb, Zn \$1.20/lb, Ag \$22.50/oz, and Au \$1600/oz

25.5 Mining

The McIlvenna Bay Deposit is a narrow, steeply dipping ore body hosted in competent ground conditions and is highly amenable to the proposed mining methods of sublevel longhole transverse and longitudinal stoping and Avoca stoping.

Initial access to the McIlvenna Bay Deposit will be via a ramp from surface to the underground to facilitate an Advanced Development and Exploration Program (ADEX). In Q4 2021, Foran began constructing the portal and will subsequently develop the ramp to the 0090 Level, along with an exploration drift on the 0090 Level

The project will be an early adaptor of Battery Electric Vehicles (BEV) in the form of the underground haulage trucks and LHDs. This technology has been proven in other industries and has been in use in underground mining since 2012. BEVs have yet to be widely adopted in existing operating mines as the benefits are limited because the cost of existing ventilation infrastructure has already been incurred. The capital cost for a new project must weigh the higher initial cost for BEV equipment compared to diesel equipment against the savings achieved through lower capital and operating costs for the ventilation system.



In Year 5 of the project, construction of a production and service shaft will begin. The shaft will extend from surface to the 0630 Level and be equipped with skip and cage conveyances.

The McIlvenna Bay Deposit will be extracted using conventional longhole mining methods including sublevel transverse and longitudinal stoping and Avoca stoping. Stopes will be drilled using a combination of ITH and top hammer drills, blasted using emulsion explosives, and mucked using battery electrical (BEV) load haul dump (LHD) vehicles.

Ore will be hauled to surface using BEV haulage trucks early in the mine life. Once the shaft is commissioned, the BEV trucks will haul to the 0570 Level rock breaker stations feeding the shaft loading pocket. Waste rock will be hauled either to surface or to an active production level for use as stope backfill.

Mine dewatering will be achieved using a multilevel clean water system. Main sumps and pump stations will be located on the 0060, 0420, 0780, and 0960 Levels. The 0060 Level sump will collect surface run-off water that enters the mine via the ramp as well as most of the water transmitted through the sandstone layer. This sump will feed water into the process water system and pump the surplus water to surface. The 0420, 0780, and 0960 Level sumps will collect any remaining rock mass inflow water, plus the process water used during mining activities. The run-of-mine water will decant from the sumps through membranes. The clean water will be pumped to the next main sump (i.e., 0960 Level to 0780 Level, and 0780 Level to 0420 Level). The 0420 Level sump will feed water into the process water system and pump the surplus water to surface. The residual solids in the sumps will be mucked using an LHD and placed into a nearby empty stope for disposal.

Transverse and longitudinal longhole stopes will be backfilled with paste fill, using filtered tailings and pyrite concentrate from the processing facility. Some secondary stopes will be filled with unconsolidated development waste. Avoca stopes will be backfilled with waste rock generated from underground development. Conventional trackless mining equipment will be used to execute lateral development required to access the ore body. The targeted mine production rate is 4,200 tonnes per day (tpd) ore and the mine life will be approximately 20 years, including an initial ramp-up period of two years. To achieve this target, the mine plan will include longhole production from multiple mining blocks, with multiple stopes available per block.

The inclusion of the production shaft to hoist material to surface reduces truck haulage costs and provides a low cost and efficient means of transferring ore from the deeper levels to the surface processing facilities with a minimum of rehandling.



25.5.1 Geotechnical

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 have led to the following conclusions:

There are three broad structural domains:

Dolomite Cap

Sub-horizontal bedding plane joints.

Ore Zone Domain

Foliation parallel jointing – typically 69° dip with 29° dip direction. The orientation of this set is variable in dip and dip direction.

FW Zone Domain

- Foliation-parallel jointing – typically 63° to 76° dip with 14° dip direction. The orientation of this set is variable in dip and dip direction.
- A weakly defined set dips of roughly 53° with dip direction of 125°.
- A sub-horizontal set with a dip of 13° to 22° with SSW to SSE dip direction.

Joint surface conditions are most commonly rough and joint surface alteration ranges from clean to coated with softening or clay minerals.

The only major structural feature delineated in the geological model is the Koziol fault which is located approximately 50 m into the HW (minimum 20 m from deposit). Geotechnical logs characterize this feature as rubble and gouge on the order of 1 m thick; the geology shape of this fault is wider than 1 m and likely captures a larger area of influence. It is uncertain whether other large-scale faults exist near the deposit. Low RQD and fault intervals, as logged in the exploration database, indicate other possible faults. It is recommended that efforts be made to develop a fault model.

25.5.2 Hydrogeological

A hydrogeological model was prepared in the pre-feasibility study (PFS) and was rebuilt in the FS. More refined shallow geological layering and geological features such as faulting were also incorporated in the FS model.

The understanding of site geology has not changed significantly since the completion of the PFS hydrogeological model. The 2021 geologic drilling activities onsite have resulted in no major changes in the interpretation of stratigraphy or faulting.



During the FS modeling, several refinements were made to the model to improve representation of the mine activities and surrounding geology.

The two-layer PFS model was refined vertically to consist of five layers in the FS model: Overburden, Dolostone, Sandstone, Regolith, and Bedrock. This refinement was completed to better represent the local shallow geology. A minimum thickness of 0.1 m was used for all layers, and the Koziol fault was simulated cutting across all the layers in the model. Zones of higher hydraulic conductivity were used to represent the fault, and finer mesh discretization was conducted along the fault line to increase model precision over this area.

The FS model simulation predicted a total of approximately 2,000 L/min of steady-state inflow into the mine workings, which is in general agreement with the value predicted by the PFS model (2,394 L/min).

When geological surfaces from two sources are stitched together, interpolation artifacts at the stitching boundary are established. As such, the accuracy of the geologic interpretation at the stitching boundary may be limited. Additionally, the formation surfaces outside of the mine project area were projected based on the best available isopach data. Consequently, the model should not be relied upon to make detailed predictions outside of the mine project area or Leapfrog geologic model domain.

25.6 Recovery Methods

The 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. Primary surface 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. Since the day-to-day mix of CSZ and MS tonnage is expected to vary, a practical blending strategy is proposed using online grade measurement equipment (PGNAA technology) to allow for the separate stockpiling of “copper-rich” and “zinc-rich” ores on surface, and selectively draw from these piles to stabilize the grade of the mill feed. 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. This grade management strategy will assist to ensure that mill feed grade deviations are minimized.



Copper and zinc flotation concentrates contain minor concentrations of certain penalty elements, but in general can be considered high quality with 26% Cu in the copper concentrate and 50% Zn in the zinc concentrate (both life-of-mine average grades).

A pyrite flotation circuit at the back-end of the flotation plant circuit ensures that tailings destined for the filtered tailing storage facility include low concentrations of sulphur and can be treated as non-acid generating. Thickened underflow slurry (approximately 55% solids) will then be pumped to a storage tank at the paste plant. At the paste plant, the low concentrated sulphur flotation tailings will be used to produce a filtered cake suitable for stacking at the surface tailings facility, and for inclusion in the paste backfill material for use underground. Approximately half the processed tonnage is mixed with binder and used underground as a paste backfill.

25.7 Project Infrastructure

The McIlvenna Bay Project is located in one of Canada's oldest mining districts, and near the town of Flin Flon, Manitoba. Flin Flon and its surrounds have well established infrastructure for road and rail transport, hydropower, and mining-related service providers.

The site is connected to public road SK-106 via an existing 18 km all-season gravel road. Supplies will be trucked to site via Flin Flon and/or Saskatoon, Saskatchewan. Flotation concentrate products will be shipped to market via road to Flin Flon.

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 administration and dry, gate house, and truck shop. The process facilities will include primary crusher building, covered ore stockpiles, mill building, and paste plant building. The site will include a 350-person camp with associated canteen, recreation facilities, and domestic water treatment infrastructure.

Power for site construction will be received by tie-in to existing grid and supplemented during operations using on-site power generation, eventually replaced by a grid connected transmission line.

The current mine production plan indicates that approximately 23.8 Mt of tailings will be produced from the on-site mineral processing plant over the life-of-mine.

To mitigate the risk of complications arising from generation of acid within the surface 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.

Approximately 5.4 Mt of the tailings will be separated into a higher-sulphur pyrite concentrate tailings stream that will be mixed with cement binder and utilized as



underground backfill. In addition, 6.0 Mt of desulphurized tailings will also be used in the backfill. The remaining 12.4 Mt of tailings filter cake will be placed and compacted in an engineered Tailings Storage Facility (TSF) that will be constructed on the site of the closed sand quarry previously owned and operated by Preferred Sands Ltd.

25.8 Market Studies and Contracts

McIlvenna Bay will produce two saleable concentrates from the mineral processing facility: a copper concentrate and a zinc concentrate. A market study for copper and zinc concentrates was not conducted as part of the study, as these products are commonly traded at prevailing market rates. NSR calculations and economic modelling of the project used generic commercial terms as described within this report.

Pricing for metals contained in the concentrate will be based on market price at the time of sales, when the concentrate is received at the smelters.

The metal price assumptions used in this feasibility study are based on consensus pricing from a number of banking institutions to arrive at a reasonable long-term estimate. Metal price estimates are considered conservative based on supply and demand fundamentals.

25.9 Environmental Studies, Permitting, and Social Impact

Current exploration and construction activities are 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. 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.

25.10 Capital and Operating Costs

Capital cost estimates were prepared by Stantec, Halyard, and Knight Piesold, with contributions from Foran.

The capital cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International with an approximate accuracy of $\pm 15\%$. All costs included in the estimate are reported in Q1 2022 Canadian dollars.



The capital cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope is described and quantified within material takeoffs (MTOs) in a series of line items.

The operating costs are based on mine and process plant design criteria, and engineering and unit cost information derived from various sources. The operating cost estimate meets the criteria to be classified as a Class 3 estimate, as defined by AACE International with an approximate accuracy of $\pm 15\%$. All costs included in the estimate are reported in Q1 2022 Canadian dollars.

The operating cost estimate reflects a detailed bottom-up approach based on key engineering deliverables that define the project scope. This scope was described and quantified within material takeoffs (MTOs) in a series of line items.

25.11 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 continuation of detail design and construction after the completion of the ADEX program.

The model indicates the Project as described by this feasibility study, including all stated assumptions, is economically viable. The post-tax NPV (7.0% discount rate) is calculated to be \$465.7 million with an IRR of 21.8%.

Pre-production capital expenditures total \$375.9 million, including a \$38.5 million contingency allowance.

The project economics are sensitive to commodity prices (copper and zinc) and the metallurgical recovery.

25.12 Adjacent Properties

There are no producing metal mines adjacent to the McIlvenna Bay property

Other VMS-style prospects are known to exist on Foran's claims and on adjacent ground. The more significant of these include the Balsam / Thunder Zone, located southeast of McIlvenna Bay; the Miskat Zone, located in the southernmost extremity of the property; and the historic Bigstone deposit located on an adjacent property 25 km to the west.

The past producing Hanson Lake Mine is located approximately 5 km to the northwest of McIlvenna Bay. The mine operated between 1967 and 1969 and produced 162,200 tonnes of ore 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.



On 23 January 2020, Copper Reef Mining Corporation (Copper Reef) announced in a press release the commencement of a 4,300 m drilling program in the Flin Flon Camp of Manitoba and Saskatchewan (Copper Reef 2020a). Copper Reef started with a 630 m program of five holes at its 100% owned Hanson Lake Property in Saskatchewan where Copper Reef has outlined two horizons each 2.5 km long.



26.0 RECOMMENDATIONS

26.1 General

The work completed as part of the Feasibility 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 ore and waste handling scenarios, surface crushing alternatives, primary power source, etc.

Stantec recommends that the preferred project configuration, described within this report, should be studied in more detail via Detailed Engineering. Estimates of cost and schedule for this are outlined in Section 26.6.

26.2 Further Exploration

Since acquiring 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 outlined potentially economic mineralization at the McIlvenna Bay Deposit, which continues to remain open down dip and plunge at depth.

There has been sufficient drilling to classify a large portion of the mineralization as indicated, according to the current (2014) CIM guidelines. The mineralization encountered at depth within the deposit continues to be classified as inferred, at this time. It is believed that future underground drilling programs will be able to upgrade at least a portion of the inferred material to indicated and to define further mineralization, both down dip and down plunge of the current mineral resource estimate. While Foran conducted the 2021 drilling from surface, it is believed that all further drilling of the deposit will be conducted from underground, as it would be more cost effective to drill the deeper parts of the deposit in this manner. Micon concurs with this approach for future drilling.

Micon's QPs make the following 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, since these may contribute to mining production in the future.



26.3 Mining

26.3.1 Hydrogeology

Stantec QP suggest the hydrogeological model prepared for the FS be updated with information from the current ADEX program, specifically related to the grouting of the sandstone layer. If the grouting program is successful in reducing the flow of water through the sandstone layer, this could significantly reduce the predictive ground water inflows into the underground operations.

26.3.2 UG Mining

Stantec's QP has identified various opportunities that have the potential to further improve project economics:

- Further optimization of the mine plan and stope sequence to identify and realize higher NPV.
- Further optimization of the cut off NSR, based on the final FS costs, to increase resource conversion and extend the mine life.
- Based on the results of the results of the ADEX program, review the methodology for sinking and support the collars for shaft and the ventilation raises. The FS methodology is based on the sandstone and regolith layers being of very poor quality, requiring a combination of concrete and steel lining to support, If the grouting program in the ADEX is successful, this methodology may be significantly reduced.
- 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 other underground mobile equipment. At time of writing BEV equipment was determined to not be economically beneficial to the project, but as the BEV technology improves, the cost of the equipment is becoming more competitive with the standard diesel equipment.
- Continue to evaluate emerging technology that will increase automation in the operations to improve efficiency and safety. This could include more autonomous production equipment, as well as support equipment, such as boom trucks for transporting materials underground.
- Consider installing a control room located offsite, where semi-autonomous equipment could be operated from. Certain other non-production functions could also be performed from this site which would reduce camp and travel costs and potentially help the company to attract and retain top-tier talent.
- Foran will investigate adding locally sourced sand (dolomite) in the backfill to reduce binder cost. While further test work is required, the geochemical characteristics of the dolomite, combined with the cement used in the paste, creates a stronger



product compared to using only tailings. This would result in a lower binder requirement.

26.3.3 Shaft / Hoisting Configuration

The hoisting system carried in FS is a vertical shaft, which is proposed to be Alimak raised in two legs. The shaft consists of three compartments, two for skips and one for the cage, along with two hoists and the related infrastructure. Foran should investigate the potential to raisebore the shaft. Foran should also consider options to reduce the number of hoists, by either eliminating the cage or considering a cage/skip combination in one compartment.

26.4 Metallurgy and Process

Based on the testwork completed to date and the results achieved, the following recommendations are made.

- Additional bench scale confirmation testing of zone and early year production composites from the advance exploration program currently underway.
- Collection of additional grindability data, specifically for SAG and regrind mills.
- Pilot flotation testwork on the bulk sample material to provide confirmation of the following:
 - Reagent dosages
 - Circuit configuration
 - Effect of process water recycling
 - Tailings desulphurization
- An opportunity may potentially exist to reject non-sulphide gangue and increase mill feed head grade using pre-concentration methods such as ore-sorting.
- Paste uniaxial compression test show comparable strength with addition of on-site quarry sand and reduced binder content. Examine availability of quarry sand and impact on backfill and TMF volumes.

26.5 Infrastructure

The following are potential opportunities, related to surface infrastructure, that should be investigated as new technology or information become available. Since these recommendations are consider long term, and will could rely upon information from other sources, there have not been any cost estimates associated with this work.

- Continue effluent water quality modelling to optimize effluent discharge limits to Winn Bay per Metal and Diamond Mining Effluent Regulations (SOR/2002-222) as design input to permanent effluent treatment plant (ETP).



- Initial test work conducted on core samples indicated amenability to ore sorting technology. More test work will be undertaken, using material from the planned bulk sample, to investigate sorting in the flowsheet. The benefit would be upgrading ore before it is milled, reducing costs.
- The processing plant is being designed as an initial 4,200tpd plant that could accommodate future potential phased expansions as Foran advances and delineates additional near-mine deposits across the Hanson Lake District.

26.6 Further Studies

It is recommended that Foran proceeds to detailed engineering, which will form the basis for procurement, construction, and commissioning work, and further provide Foran valuable information to finalize internal discussion and evaluation of the feasibility of the project. The work will include the following.

- Complete engineering to a 40–45% level of completion, in alignment with the AACE Class 2 estimate requirements.
- Develop Request for Proposal (RFP) packages for construction quotations.
- Fully define long-lead procurement items and initiate the procurement process for critical path items.
- Develop a Class 2 capital cost (CAPEX) estimate that will form the control budget for the Project.
- Develop a Level 4 Implementation Schedule for the Project.

The total estimated cost for basic engineering is \$12–14 million.

The FS excludes pre-project approval expenditures associated with the previously announced pre-development program (surface preparation and exploration decline) as well early engineering costs. Based on latest estimates, Foran expects the totality of these pre-project approval expenditures to total \$45M.



27.0 REFERENCES

27.1 Technical Reports, Papers and Other Sources

- ALS Metallurgy – G&T Metallurgical Services. 2012. "Scoping Level Metallurgical Assessment of the Mcllvenna Bay Project," Foran Mining Corporation, KM3125, 28 May 2012.
- Acton, D. F., G. A. Padbury, and C. T. Stushoff. 1998. The Ecoregions of Saskatchewan. Regina, Saskatchewan: Canadian Plains Research Centre.
- AGP Mining Consultants Inc. 2020 NI 4-3101 Technical Report, Pre-feasibility Study for the Mcllvenna Bay Project, Saskatchewan, Canada, Effective Date 12 March 2020, Report Date 27 April 2020, and Amended 14 July 2021. 436 p.
- ASKI Resources Management and Environmental Services LP (ASKI). 2012. Mcllvenna Bay Project Tradition Land Use/Knowledge Inventory. Prepared for Foran Mining Corporation, Vancouver, British Columbia.
- Balyasny Asset Management [J. Adams]. 2022. Commercial Assumptions: Foran Mcllvenna Bay Concentrate Sales Market Condition [Email]. Received by D. Myerson, 22 February 2022.
- Base Metallurgical Laboratories (BML). 2016. Phase II Metallurgical Assessment Mcllvenna Bay Deposit. Report # BL0103, 23 December 2016.
- Base Metallurgical Laboratories (BML). 2019. Feasibility Level Study of Mcllvenna Bay. Report # BL0351, 28 July 2019.
- Base Metallurgical Laboratories (BML). 2022. Metallurgical Study: Mcllvenna Bay Project – BL0770. Report # BL0770, 01 April 2022.
- Cameco – BHP – Foran. (2005). Property Option Agreement. Between Cameco Corporation, Billiton Metals Canada Inc., and Foran Mining Corporation. Effective date: 24 January 2005.
- Cameco – Rio Algom – Foran. 2015. Notice of Assignment of Royalty Interest: Cameco Corporation, Rio Algom Limited, and Foran Mining Corporation. Effective date: 24 April 2005.
- Canada North Environmental Services (CanNorth). 2013. Mcllvenna Bay Project Environmental Baseline Assessment. Prepared for Foran Mining Corporation, Vancouver, British Columbia.



- CIM Council. 2014. CIM Definition Standards for Mineral Resources and Mineral Reserves. Prepared by the CIM Standing Committee on Reserve Definitions, Canadian Institute of Mining, Metallurgy and Petroleum.
- CIM Council. 2019. CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines. Prepared by the CIM Mineral Resource & Mineral Reserve Committee, Canadian Institute of Mining, Metallurgy and Petroleum.
- Committee on the Status of Endangered Wildlife in Canada (COSEWIC). 2021. COSEWIC status reports. <https://cosewic.ca/index.php/en-ca/statusreports>.
- Conor Pacific. 1999. Upper and Lower Link Lakes Fisheries Investigation.
- Cook, R. B., and C. M. Moore. 2006. Technical Report on Mcllvenna Bay, Saskatchewan, 2006: NI 43-101 Technical Report Prepared for Foran Mining Corporation. Report Prepared by Scott Wilson RPA. Effective Date 16 November 2006.
- Copper Reef Mining Corporation (Copper Reef). 2020a. "Copper Reef to Begin Large Drill Program in The Flin Flon Belt," 23 January 2020. <https://voyageurexplorers.com/news/copper-reef-to-begin-large-drill-program-in-the-flin-flon-belt/>
- Copper Reef Mining Corporation (Copper Reef). 2020b. "Copper Reef Changes Name To Voyageur Mineral Explorers Corp.," 14 August 2020. <https://voyageurexplorers.com/news/copper-reef-changes-name-to-voyageur-mineral-explorers-corp/>
- Di Prisco, G. 2019. Mineral Characterization of Mineralization Types and Predictive Metallurgy Evaluation of Core Samples Used for Metallurgical Testing the Mcllvenna Bay VMS Deposit, Saskatchewan.
- Ernst & Young LLP (EY) [H. Pandya]. 2021. Life of Mine Model : High Level Review on the Tax Logic [Email]. Received by A. Karbovskiy, 15 September 2021.
- Environment Canada (EC). 2009. Petroleum Industry Activity Guidelines for Wildlife Species at Risk in the Prairie and Northern Region. Canadian Wildlife Service, Environment Canada, Prairie and Northern Region, Edmonton Alberta. 64p.
- Environment Canada (EC). 2011a. Guidelines for the Assessment of Alternatives for Mine Waste Disposal. Environment Canada, September 2011.
- Environment Canada (EC). 2011b. Recovery Strategy for the Woodland Baribou, Boreal Population (*Rangifer tarandus caribou*) in Canada [Proposed]. Species at Risk Act Recovery Strategy Series. Environment Canada, Ottawa. vi + 55 pp.



- Foran – Copper Reef. 2010. Asset Purchase Agreement. Between Foran Mining Corporation and Copper Reef Mining Corporation. 17 October 2010.
- Galley, A. G., M. D. Hannington, and I. R. Jonasson. 2006. Volcanogenic Massive Sulphide Deposits. Ottawa: Geological Survey of Canada.
https://explorationinsights.com/site/assets/files/4062/vms_deposits.pdf.
- Golder Associates, Ltd., B. Wickland, and D. Winston. 2012. Technical Memorandum on Phase 1 of TSF Siting Study of McIlvenna Bay Project, Saskatchewan.
- Golder Associates, Ltd. 2013. Report on Preliminary Study on Mine Waste Management, McIlvenna Bay Project.
- Golder Associates, Ltd., M. Shang, B. Wickland, and J. Hull. 2014. Technical Memorandum on Tailings Storage Facility Cost Estimate, PEA, McIlvenna Bay Project, Saskatchewan.
- Golder Associates, Ltd., and D. Winston. 2014. Technical Memorandum on Design Basis for Preliminary Study on Mine Waste Management, McIlvenna Bay Project, Saskatchewan, Canada.
- Government of Canada (GC). 1991. The Federal Policy on Wetland Conservation.
<https://publications.gc.ca/site/eng/100725/publication.html>
- Government of Canada (GC). 1994. Migratory Birds Convention Act. <http://laws-lois.justice.gc.ca/eng/acts/M-7.01/>.
- Government of Canada (GC). 2019. Impact Assessment Act.
<https://laws.justice.gc.ca/eng/acts/I-2.75/FullText.html>.
- Government of Saskatchewan (GS). 1998. The Wildlife Act.
<http://www.publications.gov.sk.ca/details.cfm?p=938>.
- Health Canada. 2016a. Guidance for evaluating human health impacts in Environmental Assessment: air quality. Ottawa.
- Health Canada. 2016b. Guidance for evaluating human health impacts in Environmental Assessment: drinking and recreational water quality. Ottawa.
- Health Canada. 2017a. Guidance for evaluating human health impacts in Environmental Assessment: Noise. Ottawa.
- Health Canada. 2017b. Guidance for evaluating human health impacts in Environmental Assessment: country foods. Ottawa.



- Health Canada. 2019. Guidance for evaluating human health impacts in environmental assessment: human health risk assessment.
- HudBay. 2021. Hudbay releases first quarter 2021 results. TMX, NYSE - HBM 2021 No. 8. https://s23.q4cdn.com/405985100/files/doc_news/2021/HBMNR8_AnnualReserveResourceUpdate_FINAL.pdf.
- Indigenous and Northern Affairs Canada (INAC). 2021. First Nations Profile : Peter Ballantyne Cree Nation. <https://www12.statcan.gc.ca/census-recensement/2016/dp-prof/prof/details/page.cfm?Lang=E&Geo1=CSD&Code1=4718052&Geo2=CD&Code2=4718&SearchText=FlinFlon&SearchType=Begins&SearchPR=01&B1=All&TABID=1&type=0>.
- Kociumbas, M. 2000. WGM Review of Resource Estimate for Mcllvenna Bay. Unpublished Internal Report to Foran Mining Corp. 7p.
- Kooyman, B. P. 2000. Understanding Stone Tools and Archaeological Sites. University of Calgary Press, Calgary, Alberta.
- Koziol, M., D. Bell, and K. Wasyluk. 1991. Hanson Lake Project, Winter 1991 Diamond Drilling Program Groups 44830 & 44925, Northern Mining District, Saskatchewan, NTS 63L/10. Cameco Corporation Internal Report, May 1991, 3v.
- Lemaitre, R., 2000, Resource Estimates of Mcllvenna Bay, July 2000: Hanson Lake, Saskatchewan; internal report by M'Ore Exploration Services Ltd, Foran Mining Corporation. 51 p., plus 6 Appendices Lewis, W. J., A. J. San Martin, and L. Jones. 2021. Technical Report for the 2021 Mineral Resource Estimate on the Mcllvenna Bay Project Saskatchewan, Canada. Technical Report Prepared for Foran Mining Corporation. Effective Date 6 September 2021.
- Lewis, W., A. J. San Martin, and R. M Gowans. 2019. Technical Report for the 2019 Mineral Resource Estimate on the Mcllvenna Bay Project, Saskatchewan, Canada. Technical Report Prepared for Foran Mining Corporation. Effective Date 7 May 2019. 174p.
- Lewis, W.J., A., Cole, S. Holloway, D. Flood, M. Patel, J. Howery, and A. McIntyre. 2020. Technical Report on The Pre-feasibility Study on the Mcllvenna Bay Project Saskatchewan, Canada. Technical Report Prepared for Foran Mining Corporation, Effective Date 12 March 2020.
- Lydon, J. W. 1990. Volcanogenic Massive Sulphide Deposits Part 1: A Descriptive Model. Vol. 3, in Ore Deposits Models, edited by R. G. Roberts and P. A. Sheahan. Geoscience Canada Reprint Series.
- Makarenko, M., D. Kennard, M. Bender, D. Rennie, J. Hull, L. Correia, and K. Major. 2015. Technical Report on the Preliminary Economic Assessment on the Mcllvenna Bay



Project Saskatchewan, Canada. Technical Report Prepared for Foran Mining Corporation. Effective Date 21 January 2015.

- Marles, R. 1984. The Ethnobotany of the Chipewyan of Northern Saskatchewan (Unpublished). M.Sc. Thesis, Department of Biology, University of Saskatchewan, Saskatoon, Saskatchewan.
- Marles, R., C. Clavelle, L. Monteleone, N. Tays, and D. Burns. 2008, Aboriginal Plant Use in Canada's Northwest Boreal Forest. Natural Resources Canada, Edmonton, Alberta.
- Masun, K. M. and D. W. Rennie. 2021. NI 43-101 Technical Report on the Bigstone Project, East Central Saskatchewan, Canada. 139 p.
- Maxeiner, R. O., T. I. Sibbald, L. S. William, L. M. Heaman, and B. R. Watters. 1999. Lithogeochemistry of Volcano-Plutonic Assemblages of the Southern Hanson Lake Block and Southeastern Glennie Domain, Trans-Hudsonian Orogen: Evidence for a Single Island Arc Complex. Canadian Journal of Earth Sciences 36: 209-225.
- McMartin, I., P. J. Henderson, and E. Neilson. 1999. Impact of a Base Metal Smelter on the Geochemistry of Soils in the Flin Flon Region, Manitoba, and Saskatchewan. Canadian Journal of Earth Sciences 35: 141-160.
- Moerman, D. 2010. Native American Plant Foods. Timber Press, Portland, Oregon.
- MRDI. 1998. Hanson Lake Due Diligence and Scoping Study L313-2A000-01 for Foran Mining Corporation. Report prepared by MRDI, Saskatoon, Saskatchewan, April 1998, 164 p.
- Peter Ballantyne Cree Nation (PBCN). n.d.. History: Population. <https://pbcn.ca/administration.html/> Accessed 13 April 2021.
- Peter Ballantyne Cree Nation (PBCN). 2021. Communities: Deschambault Lake. <https://www.peterballantyne.ca/deschambault-lake/> Accessed 03 November 2021.
- Peter Ballantyne Cree Nation Health Services (PBCN Health Services). 2021. Communities: Deschambault Lake. <http://www.pbcnhealthservices.org/index.php/communities/deschambault-lake/> Accessed 03 November 2021.
- Rio Algom – BHP – Foran – Cameco. 2015. Deed of Assumption and Assignment. Between Rio Algom Limited, BHP Billiton Royalty Investments Pty Ltd, Foran Mining Corporation, and Cameco Corporation.
- Rio Algom – BHP. 2015. Sale and Purchase Deed. Between Rio Algom Limited, BHP Billiton Royalty Investments Pty Ltd.



- Rennie, D. W. 2011. Technical Report for the 2011 Mineral Resource Estimate on the Mcllvenna Bay Project Saskatchewan, Canada. Technical Report prepared for Foran Mining Corporation. Effective date: 09 December 2011.
- Rennie, D. W. 2013. Mcllvenna Bay Project Mineral Resource Estimate Update, internal letter report to Foran Mining Corporation, April 16, 2013. Report Issued by RPA, 18 p.
- Rennie, D. W. 2014. Revised sections of the 2011 RPA Technical Report for Inclusion in the 2014 JDS Technical Report. Undated Internal Draft Sections.
- RockEng. 2022. Geomechanical Support for the Mcllvenna Bay Feasibility Study - March 2022.
- Sangster, D. F., and S. D. Scott. 1976. Precambrian Stratabound Massive Cu-Zn-Pb Sulphide Ores of North America. In Handbook of Stratabound and Stratiform Ore Deposits, edited by K. H. Wolf, 130-221. Amsterdam: Elsevier Scientific Publishing Co.
- Species at Risk Public Registry (SARPR). 2011. Species Profile: Woodland Caribou Boreal Population. Environment Canada.
http://www.sararegistry.gc.ca/species/speciesDetails_e.cfm?sid=636
- Species at Risk Public Registry (SARPR). 2014. Species List. Environment Canada, Ottawa, Ontario. http://www.sararegistry.gc.ca/species/default_e.cfm.
- Species at Risk Public Registry (SARPR). 2020. Species List. Environment and Climate Change Canada. <https://species-registry.canada.ca/index-en.html#/species?sortBy=commonNameSort&sortDirection=asc&pageSize=10>.
- Saskatchewan Conservation Data Centre (SKCDC). 2020a. Taxa list for vertebrates. <http://www.biodiversity.sk.ca/SppList/verts.pdf> (accessed 06 June 2018).
- Saskatchewan Conservation Data Centre (SKCDC). 2020b. Saskatchewan vascular plant taxa list. Regina, Saskatchewan. <http://www.biodiversity.sk.ca/SppList.htm> (accessed 20 March 2018).
- Saskatchewan Gazette. 2002. Struck Off the Register Pursuant to Section 290. December 27, 98(52): 1486.
- Saskatchewan Ministry of Environment (SKMOE). 2008. Northern Mine Decommissioning and Reclamation Guidelines. EPB 381.
https://pubsaskdev.blob.core.windows.net/pubsask-prod/96788/96788-Northern_Mine_Decommissioning_and_Reclamation_Guidelines.pdf.



- Saskatchewan Ministry of Environment (SKMOE). 2012. Saskatchewan Air Quality Modelling Guideline. March.
- Saskatchewan Ministry of Environment (SKMOE). 2014a. Environmental Assessment in Saskatchewan. A High-Level Overview of the Environmental Assessment Process for Developments within Saskatchewan under the Environmental Assessment Act. Environmental Assessment Branch, Regina, SK.
- Saskatchewan Ministry of Environment (SKMOE). 2014b. Environmental Monitoring Guidelines for Mining / Industrial Operations (Draft). Version 4.3, November 5, 2014.
- Saskatchewan Ministry of Environment (SKMOE). 2014c. Consultation with First Nations and Métis in Saskatchewan Environmental Impact Assessment.
- Saskatchewan Ministry of Environment (SKMOE). 2014d. Activity Restriction Guidelines for Sensitive Species. February 2014. Fish and Wildlife Branch, Ministry of Environment, Regina, Saskatchewan.
- Saskatchewan Ministry of Environment (SKMOE). 2017. Saskatchewan Activity Restriction Guidelines for Sensitive Species.
<https://publications.saskatchewan.ca/#/products/79241>.
- Saskatchewan Ministry of Environment (SKMOE). 2019. Rare vascular plant survey protocol.
<http://www.publications.gov.sk.ca/details.cfm?p=79238>.
- Strong Pine Energy Services (SPES). 2019. Strong Pine Energy Services Environmental Impact Statement; Silica Sand Project, Hanson Lake, Saskatchewan.
- Stantec. 2021. Groundwater Model Update: Foran Mining Corporation, McIlvenna Bay, Underground Mine Feasibility Study, Project 169521520. Issued 19 November 2021.
- Statistics Canada (StatsCan). 2012. Flin Flon, Manitoba (Code 0282) and Northwest Territories (Code 61) (table). Census Profile. 2011 Census. Statistics Canada Catalog No. 9-3816-XWE. Ottawa.
- Statistics Canada (StatsCan). 2017. Census Profile. 2016 Census. Statistics Canada Catalog no. 98-316-X2016001. Ottawa. Released 29 November 2017.
<https://www12.statcan.gc.ca/census-recensement/2016/dp-pd/prof/index.cfm?Lang=E>.
- Syme, E. C., S. B. Lucas, A. H. Bailes, and R. A. Stern. 1999. Contrasting Arc and MORB-like Assemblages in the Paleoproterozoic Flin Flon Belt, Manitoba, and the Role of Intra-Arc Extension in Localizing Volcanic-Hosted Massive Sulphide Deposits. Canadian Journal of Earth Sciences 36: 1767-1788.



- Terra Mineralogical Services Inc. 2012. Initial Ore Characterization and Predictive Metallurgy Evaluation of the Drill Core Samples from the McIlvenna Bay VMS Deposit.
- Terrestrial and Aquatic Ecosystem Managers (TAEM). 1990a. Aquatic biological resources of the Hanson Lake project area, Saskatchewan. Prepared for Cameco Corporation, Saskatoon, Saskatchewan.
- Terrestrial and Aquatic Ecosystem Managers (TAEM). 1990b. Aquatic biological resources of the Hanson Lake Project Area, Spring Spawning Investigations. Prepared for Cameco Corporation, Saskatoon, Saskatchewan.
- Wood Mackenzie. 2019. "Global zinc long-term outlook." June 26.
<https://www.woodmac.com/reports/metals-global-zinc-long-term-outlook-q2-2019-319195>
- Wood Mackenzie. 2021. "Will a lack of supply growth come back to bite the copper industry?" March 23. <https://www.woodmac.com/news/opinion/will-a-lack-of-supply-growth-come-back-to-bite-the-copper-industry/>
- Westerhaver, Eric. 2021. "Hudbay starts plan to shift jobs to Snow Lake, announces update for Snow Lake area." The Reminder, April 7. <https://www.there minder.ca/local-news/hudbay-starts-plan-to-shift-jobs-to-snow-lake-announces-updates-for-snow-lake-area-4210827>
- Westerhaver, Eric. 2022. "Zinc plant to shut down in May, 777 and mill in June, about 275 layoffs expected, shop and admin to stay open: Hudbay." The Reminder, February 11. <https://www.there minder.ca/local-news/hudbay-set-to-shut-down-zinc-plant-in-may-777-and-mill-in-june-about-275-layoffs-expected-shop-and-admin-to-stay-open-hudbay-5051581>
- Zwanzig, H. V., and D. C. Schledewitz. 1997. Geological Compilation of Parts of the Flin Flon Terrane and Kiseeynew Gneisses, Trans Hudson Orogen, Manitoba. GAC-MAC Annual Meeting Abstract Volume 22: A163.

27.2 Internet Sources

- Foran Mining Corporation website: www.foranmining.com, (2014 to 2021) Press Releases.
- Hudbay Minerals Inc. website: www.hudbayminerals.com, (2018 and 2019) Press Releases.
- Kitco website: www.kitco.com
- SEDAR website: www.sedar.com



Voyageur Mineral Explorers Corp. website: www.voyageurexplorers.com, (2020 and 2021 Press Releases)



28.0 QUALIFIED PERSON CERTIFICATES





CERTIFICATE OF QUALIFIED PERSON

1-1760 Regent Street
Sudbury, ON P3E 3Z8
Canada

I, Mark Hatton, P.Eng. am employed as a Mining Engineer by Stantec Consulting Ltd. of 1-1760 Regent St, Sudbury, ON, Canada P3E 3Z8.

This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada", dated 13 April 2022 (the "technical report").

I am a member in good standing of the Association of Professional Engineers of Ontario, registration number 90433939. I graduated from Queen's University in Kingston Sudbury Ontario, with a Bachelor of Science Degree in Mining Engineering, in 1994.

I have practiced my profession continuously since 1994 and have experience in mining operations and consulting. I have worked as a Mining Engineer with Stantec for 27 years working on studies for most commodities in many different countries, with 4 years of on-site operations experience at nickel, gold, potash and zinc mines in Canada, Tanzania, and Turkey. My relevant experience includes underground mine design, surface and underground related infrastructure, capital and operating cost estimates, and cash flow modelling and financial analysis for gold, nickel, zinc, copper, and uranium mines in Canada, and Mexico.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I visited the Foran McIlvenna Bay project site on 23 August 2021.

I am the co-author of the Report and am responsible for Sections 1.1, 1.2, 1.9, 1.10, 1.13, 1.15 to 1.19, 2.0, 3.2 to 3.4, 15.0, 16.1, 16.3 to 16.17, 18.5.1, 18.5.2, 19, 21.1.1, 21.1.2, 21.1.6, 21.1.7, 21.2.1, 21.2.3, 22.0, 24.0, 25.4, 25.5, 25.8, 25.10, 25.11, 26.0, and 27.0

I am independent of Foran Mining Corporation. as independence has been described in section 1.5 of NI 43-101.

I have read NI 43-101 and the sections of the technical report for which I am responsible and have confirm that the technical report has been prepared in compliance with that Instrument.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Sudbury, ON: 14 April 2022.

"Signed and Sealed"

Mark Hatton, P.Eng.

Certificate of Qualified Person

I, William J Lewis I am employed as a Senior Geologist by, and carried out this assignment for, Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario M5H 2Y2, tel. (416) 362-5135, e-mail wlewis@micon-international.com

This certificate applies to the technical report titled “Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada”, dated 14 April, 2022 (the “technical report”).

I hold the following academic qualifications:

- B.Sc. (Geology) University of British Columbia 1985

I am a registered Professional Geoscientist with the Association of Professional Engineers and Geoscientists of Manitoba (membership # 20480); as well, I am a member in good standing of several other technical associations and societies, including:

- 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)
- The Canadian Institute of Mining, Metallurgy and Petroleum (Member # 94758).

I have worked as a geologist in the minerals industry for over 36 years

I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 4 years as an exploration geologist looking for gold and base metal deposits, more than 11 years as a mine geologist in underground mines and 21 years as a surficial geologist and consulting geologist on precious and base metals and industrial minerals.

I have read NI 43-101 and this Technical Report has been prepared in compliance with the instrument.

I conducted my original site visit to the McIlvenna Bay Project between August 16 and 18, 2018 to review the drilling programs on the property and discuss the ongoing QA/QC program. I conducted a second site visit in conjunction with this report between November 17 and November 19, 2021.

I have co-authored a number of previous Technical Reports for the mineral property that is the subject of this Technical Report.

I am independent Foran Mining Corporation and its subsidiaries according to the definition provided in NI 43-101 and the Companion Policy 43-101 CP. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I am the co-author of the Report and am responsible for Sections 1.3 to 1.6, 1.8, 1.9, 3.1, 4.0 to 12.0, 14.0, 23.0, 25.1, 25.2, and 25.12.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Toronto, ON: 14 April 2022.

“William J. Lewis” {Signed and Sealed as of the Report Date}

William J. Lewis, B.Sc., P.Geo.

Senior Geologist, Micon International Limited

CERTIFICATE OF QUALIFIED PERSON

Lyn Jones, P.Eng.

To accompany the technical report entitled “Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada” dated April 13th, 2022, with an effective date of February 28th, 2022 (the “Technical Report”).

I, Lyn Jones, P.Eng., do hereby certify that:

- I am the Manager, Process Engineering with Blue Coast Research with a business address at 2-1020 Herring Gull Way, Parksville, British Columbia. .
- I graduated from the University of British Columbia with a Bachelor’s of Applied Science in 1996, and a Master’s of Applied Science in 1998.
- I am registered as a Professional Engineer in the province of Ontario (PEO licence #100067095).
- I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 24 years working on base and precious metals projects in the mining sector with experience including metallurgical testwork, flowsheet development, process engineering, and plant commissioning. I have read the definition of “qualified person” set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
- I am independent of the issuer, Foran Mining Corporation, as defined in Section 1.5 of NI 43-101.
- I am responsible for Sections 1.7, 1.11, 13.0, 17.0, 25.3, and 25.6 of the Technical Report and accept professional responsibility for those sections of the Technical Report.
- I have had previous involvement with the project as a QP for the report entitled, “Technical Report for the 2021 Mineral Resource Estimate on the McIlvenna Bay Project, Saskatchewan Canada”, dated February 11, 2022.
- I have not visited the McIlvenna Bay Project site.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
- I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Dated this 14th day of April 2022, in Peterborough, Ontario.

“signed electronically”

Lyn Jones, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

501-212 King Street West
Toronto, ON M5H 1K5
Canada

I, Michael Franceschini, P.Eng. am employed as a Project Manager by Halyard Inc. of 501-212 King St W, Toronto, ON, Canada M5H 1K5.

This certificate applies to the technical report titled “Technical Report on the Feasibility Study for the McIlvanna Bay Project, Saskatchewan Canada”, dated 14 April 2022 (the “technical report”).

I am a member in good standing of the Association of Professional Engineers of Ontario, registration number 100113722. I graduated from Queen’s University in Kingston Ontario, with a Bachelor of Science Degree in Mechanical Engineering, in 2006.

I have practiced my profession continuously since 2006 and have experience consulting in the design of mineral process plants. I have worked as a Project Manager with Halyard for 1 year. My relevant experience includes mineral process plant and related surface infrastructure design, capital and operating cost estimates for various non-ferrous metals, primarily gold, copper and zinc. I have held various engineering, project/ construction management and commissioning roles in Canada, and Dominican Republic.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I visited the Foran McIlvanna Bay project site on 20 October 2021.

I am the co-author of the Report and am responsible for Sections 1.12, 18.0 to 18.5, 18.6.3 to 18.6.7, 18.7 to 18.9, 21.1.3, 21.2.2, and 25.7.

I am independent of Foran Mining Corporation, as independence has been described in section 1.5 of NI 43-101.

I have read NI 43-101 and the sections of the technical report for which I am responsible and have confirm that the technical report has been prepared in compliance with that Instrument.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Toronto, ON: 14 April 2022.

“Signed and Sealed”

Michael Franceschini, P.Eng.



CERTIFICATE OF QUALIFIED PERSON

Canada North Environmental Services
211 Wheeler Street
Saskatoon, SK S7P 0A4
Canada

I, Jocelyn Howery, P.Ag. am employed as a Senior Hydrologist and Environmental Scientist by Canada North Environmental Services of 211 Wheeler Street, Saskatoon, SK S7P 0A4, Canada.

This certificate applies to the technical report titled "Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada", dated 13 April 2022 (the "technical report").

I am a member in good standing of the Saskatchewan Institute of Agrologists, member number 99931. I graduated from the University of Alberta, with a Master of Science Degree in Water and Land Resources, in 2010.

I have practiced my profession continuously since 2010 and have experience in the design and management of environmental baseline and monitoring programs throughout Saskatchewan. I have worked as a Project Manager and the Hydrology Division Manager at CanNorth for 11 years.

As a result of my experience and qualifications, I am a Qualified Person as defined by the Saskatchewan Environmental Code.

I visited the Foran McIlvenna Bay project site most recently in October of 2014.

I am responsible for Sections 1.14, 20.1 to 20.4, 20.5.1 to 20.5.6, 20.5.8, 20.6, and 25.9.

I am independent of Foran Mining Corporation. as independence has been described in section 1.5 of NI 43-101.

I have read NI 43-101 and the sections of the technical report for which I am responsible and have confirm that the technical report has been prepared in compliance with that Instrument.

At the effective date of the Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated at Saskatoon, SK: 14 April 2022.

"Signed and Sealed"

Jocelyn Howery, P.Ag.

CERTIFICATE OF QUALIFIED PERSON

I, Alex McIntyre, P.Eng. am employed as a Senior Engineer by Knight Piesold Ltd. Of 1650 Main Street West, North Bay, On, Canada, P1B 8G5, do hereby certify that:

1. This certificate applies to the Technical Report entitled Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada with an effective date of February 28, 2022 (the "Technical Report") prepared for Foran Mining Corporation
2. I am a graduate of the Carleton University, with a Bachelor of Engineering in Civil Engineering, in 2007. I have practiced my profession continuously since 2007. My experience includes tailings facility design, earthworks construction quality assurance and control, surface water hydrology design in projects located in Canada, United States of America, Mexico, Cuba.
3. I am a Professional Engineer in good standing with Association of Professional Engineers and Geoscientists of Saskatchewan (No. 64017).
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the McIlvenna Bay project site on October 20, 2021.
6. I am responsible for Sections 18.5.8, 18.9, 20.5.7, 20.5.9, 21.1.4, 21.1.5, and 21.2.4 of this Technical Report.
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have had no involvement with the property that is the subject of this Technical Report.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 28, 2022

Signing Date: April 14, 2022

"Signed and Sealed"

Alex McIntyre, P.Eng.

Certificate of Qualified Person

To accompany the Report entitled “Technical Report on the Feasibility Study for the McIlvenna Bay Project, Saskatchewan Canada”, dated April 13, 2022 (the “Technical Report”), prepared for Foran Mining Corp (“Foran” or the “Corporation”).

I, Kathy Kalenchuk, P. Eng., PE, do hereby certify:

1. I am President and Principal Consultant, with RockEng Inc. located at 920 Princess St., Suite 310, Kingston, Ontario, Canada.
2. I am a graduate of the University of Alberta, Edmonton, Alberta, Canada in 2004 with a Bachelors degree in Mining Engineering. I am a graduate of the Queen’s University, Kingston, Ontario, Canada in 2007 with a Master’s degree in Geomechanical Engineering and 2010 with a Ph.D. in Geomechanical Engineering.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100164661), the Province of British Columbia (Reg. # 189685), the Province of Saskatchewan (Reg. # 66741), the Province of Manitoba (Reg. # 44426), and the State of Montana (Reg. #PEL-PE-LIC-59819). I have worked as a Geotechnical Engineer for a total of 12 years continuously since my Ph.D. graduation.
4. My relevant work experience includes:
 - Due Diligence Reviews: Review of mine plans and designs for investors as well as third-party review for technical quality and validity of geomechanical studies.
 - Underground Mine Design: All geomechanical components of underground mine design such as, mining method, mine sequencing, stope sizing, excavation stability, static and dynamic ground support, hazard identification and risk management, infrastructure siting, construction method evaluations.
 - Operational Geomechanics and Ground Control: Routine site support for operating mines, including audits and inspections, incident investigations, regular design reviews, support evaluations, instrumentation data interpretation, development and review of Ground Control Management Plans, and provision of ground control training to site personnel.
 - Site Characterization: Core logging, geotechnical mapping, damage mapping, data analysis, domain delineation, material parameterization, in situ stress.
 - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Section 16.2.
8. I have not personally visited the property that is the subject to the Technical Report.
9. I have had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 14th day of April 2022.

“Original Signed and Sealed on file”

Kathy Kalenchuk, Ph.D., P.Eng., PE
President & Principal Consultant