PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

Preliminary Economic Assessment Technical Report McIlvenna Bay Project, Saskatchewan, Canada

Effective Date: November 12, 2014
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NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Foran Mining Corporation. The quality of information, conclusions and estimates contained herein are based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions and qualifications set forth in this report.

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1 EXECUTIVE SUMMARY

1.1 Introduction

JDS Energy & Mining Inc. (JDS) was commissioned by Foran Mining Corporation (Foran) to carry out a Preliminary Economic Assessment (PEA) of the McIlvenna Bay deposit (McIlvenna or McIlvenna Bay), a resource development base metal project owned by Foran Mining Corporation (Foran) located in the Flin Flon Greenstone Belt in east central Saskatchewan.

Two previous technical reports were prepared for McIlvenna Bay pursuant to Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 (collectively, "NI 43-101") and documenting exploration work completed by Foran on McIlvenna Bay in 2006, and 2011. Both technical reports were filed on SEDAR.

This technical report summarizes the results of the PEA study and was prepared following the guidelines of NI 43-101.

1.2 Property Description and Ownership

McIlvenna Bay occurs within Foran's McIlvenna Bay property located approximately 1 kmsouth of Hanson Lake, Saskatchewan. The property is also approximately 375 km northeast of Saskatoon and 65 km west-southwest of Flin Flon, Manitoba. McIlvenna Bay is located within 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^{o}50' W and 54^{o}38" N. McIlvenna Bay is located well within the property boundaries.

Foran owns a 100% interest in McIlvenna Bay, subject to a 1% Net Smelter Royalty (NSR) held by Cameco Corporation and BHP Billiton, with a buy-out provision in favour of Foran for the purchase of the whole NSR for $1,000,000 at any time and a Net Tonnage Royalty of $0.75/t of ore extracted with a right of first refusal in favour of Foran.

1.3 Geology and Mineralization

The Hanson Lake Block, the host terrain of McIlvenna Bay, is one of eight geographically separate juvenile island arc volcanic assemblages within the Paleoproterozoic Flin Flon Greenstone Belt (FFGB). It is bound to the east by the Sturgeon-Weir Shear Zone and to the west by the Tabbernor Fault Zone, and 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.

The deposit area is underlain by variably metamorphosed sedimentary and volcanic rocks of Proterozoic age, unconformably capped by Winnipeg Formation sandstones. Outcrops are scarce, and the stratigraphy has been defined over a two kilometre strike length by a total of 191 drill holes. From lowermost to uppermost, these units are: the McIlvenna Bay Formation, Cap Tuffite Formation, the Koziol Iron Formation, the Rusk Formation, the HW-A Formation, and the Upper Sequence. Sills and dykes of the Davies Gabbro intrude the Cap Tuffite formation.

The McIlvenna Bay Formation, the principal host of McIlvenna Bay, is known only to the extent 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, variably altered felsic volcanics, volcaniclastics, and/or volcanic-derived sediments of rhyolitic composition.

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.

Two phases of folding of the host stratigraphy have been observed in the drill core and are believed to correspond to the regional F2 and F3 folding events.

Evidence of faulting has been documented in drill core, but it is difficult to determine the orientation, scale, or continuity of most faults between drill holes with the present level of information.

McIlvenna Bay is a volcanogenic massive sulphide (VMS) deposit, comprising synvolcanic accumulations of sulphide minerals on or near the seafloor. The deposits consist of structurally modified, stratiform, volcanogenic, polymetallic massive sulphide mineralization and associated stringer zone mineralization. The sulphides contain copper and zinc, with comparatively low lead, silver, and gold values. The deposit has undergone strong deformation and upper greenschist to amphibolite facies metamorphism. The sulphide lenses are now attenuated down the plunge to the northwest.

McIlvenna Bay encompasses five different zones and includes three distinct styles of mineralization. The five zones identified are the Lens 2 massive sulphide (L2MS), Upper West (UW), Lens 3 (L3), Copper Stockwork Zone (CSZ) and Footwall Stockwork Zone (FW). The three different styles of mineralization are massive sulphides, semi-massive sulphides, and copper stockwork. Each style is mineralogically and texturally distinct.

1.4 History, Exploration and Drilling

Parrex Mining Syndicate first discovered zinc-lead massive sulphide mineralization in the Hanson Lake area in 1957. Hanson Lake Mine operated from 1967 to 1969, and produced 162,200 t grading 9.99% Zn, 5.83% Pb, 0.51% Cu, and 4.0 oz/t Ag.

Saskatchewan Mining Development Corporation (SMDC) acquired an exploration lease covering much of the Hanson Lake Block in 1976. SMDC, which eventually became Cameco Corporation (Cameco), conducted geological mapping, geophysical surveys, and diamond drilling in the area up to 1988. This work lead to the discovery of what is now McIlvenna Bay.

Cameco stopped work on the property in 1991, and it remained idle until 1998, when it was acquired by Foran. Since that time, Foran has conducted a series of diamond drilling campaigns, as well as geochemical sampling, geophysical surveys, and metallurgical test work. The total amount of drilling completed by all parties to January 2013 (the effective date of the Mineral Resource estimate) was 88,681 m in 191 holes.

1.5 Mineral Processing and Metallurgical Testing

KWM Consulting Inc. (KWM) was contracted by Foran in 2011 to define a scoping level metallurgical test program based on historical metallurgical testwork completed by Cominco and on 2011/2012 mineralogical analysis completed by Terra Mineralogical Services Inc (Terra).

ALS Metallurgical (formerly G&T) located in Kamloops were contracted to complete the metallurgical testwork. The metallurgical program was identified as KM3125.

A review of the metallurgical testwork indicates that the McIlvenna deposit has three distinct mineralized types. These have been identified as CSZ, L2MS, and UW-MS.

The three zones of mineralization from McIlvenna Bay will be extracted using underground mining methods. The testwork showed that the metallurgical properties of the three zones are very different providing a preliminary indication that the three zones will need to be processed independently in the process facility.

The analysis of the testwork results indicated that the CSZ is copper rich generating a copper concentrate product. The main sulphide minerals in the CSZ composite consisted of pyrite and copper sulphides. With a pyrite to copper sulphide ratio of about 1:1 a favorable flotation response was expected and observed.

The L2MS Composite contained about 50% by mass pyrite. The next dominant sulphide mineral, at about 11%, was sphalerite. The sample also contained about 1% Cu sulphide and 0.5% Galena. The flotation testwork resulted in the generation of a low grade combined Cu/Pb bulk concentrate and a high grade, high recovery Zinc concentrate.

The UW-MS Composite contained about 26% by mass pyrite. The copper sulphide and sphalerite were present in near equal masses of 5.3% and 5.8% respectively. The flotation testwork indicated that it was possible to generate marketable copper concentrate at 24% Cu and marketable zinc concentrate at 54% Zn.

An initial group of drill core samples from McIlvenna Bay were collected for a mineralogical evaluation by Terra. The scope of the Terra work was to carry out a characterization and predictive metallurgy study of a series of stacked sulphide zones occurring at McIlvenna Bay. The mineralogical evaluation was used to define the preliminary metallurgical testwork program.

A single locked cycle test was completed for the CSZ composite. The locked cycle test indicated that about 94% of the copper was recovered into a copper concentrate grading 29% Cu. The silver and gold recoveries were also high at 77% and 85% respectively.

Two locked cycle tests were completed on the L2MS Composite. The locked cycle tests indicated that a zinc concentrate ranging from 53% Zn to 55% Zn was achievable with recoveries in excess of 80% to the zinc concentrate. Approximately 55% of the Cu and Pb in the feed were recovered to a bulk concentrate.

A single locked cycle test was completed on the UW-MS Composite. The locked cycle test results indicated that approximately 83% of the feed copper was recovered to a final copper concentrate grading 24% Cu. About 10% of the zinc in the feed was recovered to the copper concentrate. Approximately 76% of the zinc in the feed was recovered in the zinc flotation circuit to a concentrate grading 54% Zn. The Au and Ag recoveries to the copper concentrate were 60% and 50% respectively.

A number of key design parameters have been identified in the test program. From the testwork it was determined that each mineralization type has different grinding parameters including work index, primary grind size and rougher regrind size. The flotation circuit design is also mineralization specific for reagents and flotation time.

Significantly more work is required to complete the metallurgical interpretation for the underground mine plan and mining schedule. None of the ancillary process testwork (concentrates and tailing thickening, concentrates filtering, etc.) was completed as part of the preliminary metallurgical testwork.

Future testwork programs should continue to optimize the flowsheets for the different mineralization types and evaluate blend mineralization types to simplify underground (UG) mine development and operations. Variability testing should look at determining feed grade vs. recovery relationships and develop metallurgical models for the various types. Ancillary process testing should be included in the metallurgical test program to complete the development of the process design criteria for finalizing the process flowsheet and the equipment selection.

A single flowsheet should be designed that can be used for all of the mineralization types. Not all of the components will be used for all of the mineralization types and the plant layout will need to consider the various operating conditions. The test work indicated that saleable concentrates can be generated for the three main mineralization types with the following operating results.

- CSZ: Copper concentrate grading 29% Cu, 94% Cu recoveries to the copper concentrate;
- L2MS: Bulk Concentrate grading 12% Cu/14% Pb, with recoveries of 56% Cu, 58% Pb, 33% Ag, 42% Au to the bulk concentrate;
- Zinc Concentrate grading 54% Znwith 81% Zn recoveries to the zinc concentrate; and
- UW-MS: Copper Concentrate grading 24% Cu with recoveries of 83% Cu, 50% Ag, 60% Au to the bulk concentrate; and a Zinc Concentrate grading 54% Zn with a recovery of 76% Zn to the zinc concentrate.

1.6 Mineral Resource Estimate

The Mineral Resource estimate was carried out in 2013 using a block model constrained by 3D wireframes of the mineralized zones. Values for Cu, Zn, Au, Ag, and Pb were interpolated into the blocks using Ordinary Kriging. The wireframe models of the mineralized zones were constructed from Foran geological interpretations on the 50 m cross sections. A nominal $45/t NSR cut-off value was used along with a minimum horizontal width constraint of two metres.

The database, comprising diamond drilling results collected over the entire history of the project, contained records for 178 diamond drill holes, with a total of 6,220 assay intervals. Of these assay intervals, 2,833 were eventually captured within the wireframe models used to constrain the estimate.

High-grade samples were capped at a range of values depending on the host zone, and then composited to 2-metre downhole lengths. A geostatistical analysis was conducted on the composites to define search and variogram model parameters for grade interpolation.

The Mineral Resource estimate was classified in a manner compliant with NI43-101. The Indicated classification was applied in the core of the deposit, where the nominal drill hole spacing is 65 m or less, and/or where the average distance to the nearest three drill holes is 45 m or less. All other blocks, estimated to a maximum distance of 250 m in the massive and semi-massive sulphide bodies and 190 m in the stockwork bodies, have been classified as Inferred Mineral Resources, with the exception of some of the stockwork mineralization in the lowermost extremity of the deposit.

RPA used an NSR value of $60/t for application of a cut-off to the block model. The NSR was estimated for each block using provisions for metallurgical recoveries, smelter payables, refining costs, freight, and applicable royalties. The smelter terms and freight costs were derived from a report prepared for Foran by a metals marketing consultant. Metal prices used for Mineral Resources were based on consensus, long term forecasts from banks, financial institutions, and other sources. The NSR calculation was done before the present study, and is therefore out of date. In RPA's opinion, the block model NSR value should be updated to reflect the results of this PEA.

Mineral Resources as of January 2013 are summarized in Table 1.1.

Table 1.1: McIlvenna Bay Mineral Resources - January 2013

Zone	Tonnes (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	CuEq (%)	ZnEq (%)	Cu (Mlb)	Zn (Mlb)	Au (oz)	Ag (oz)
Indicated												
L2 MS	3,390	0.31	7.15	0.24	23.7	81.55	1.51	10.19	23	534	25,700	2,580,000
L2 UW	2,150	1.66	4.1	0.88	30.7	150.00	2.79	18.75	78.7	194	61,000	2,120,000
L3 MS	760	1.23	2.55	0.3	14.5	96.25	1.79	12.03	20.5	42.4	7,310	353,000
CSZ	7,610	1.6	0.28	0.51	10.6	104.56	1.94	13.07	269	46.5	126,000	2,600,000
Total	**13,900**	**1.28**	**2.67**	**0.49**	**17.1**	**105.57**	**1.96**	**13.2**	**391**	**817**	**220,000**	**7,650,000**
Inferred												
L2 MS	2,800	0.5	7.13	0.38	26.1	96.29	1.79	12.04	31.1	439	33,800	2,350,000
L2 UW	2,910	1.62	3.68	0.51	19	132.95	2.47	16.62	104.3	236	47,800	1,780,000
L3 MS	124	1.61	2.67	0.51	17.7	124.13	2.31	15.52	4.39	7.3	2,040	70,300
CSZ	5,480	1.56	0.47	0.42	12.1	100.76	1.87	12.59	188	56.9	73,100	2,140,000
Total	**11,300**	**1.32**	**2.97**	**0.43**	**17.5**	**108.32**	**2.01**	**13.54**	**328**	**740**	**157,000**	**6,340,000**

Notes:

1. CIM definitions were followed for Mineral Resources
2. Mineral Resources are estimated at a cut-off of US$60/t.
3. NSR values, as well as CuEq and ZnEq grades, were calculated as per the description in this report and include provisions for metallurgical recovery.
4. Metal prices used for this estimate were US$3.25/lb Cu, US$1.10/lb Zn, US$1,400/oz Au, and US$25/oz Ag.
5. High-grade caps were applied as per the text of this report.
6. Specific gravity was estimated for each block based on measurements taken from core specimens.
7. CSZ includes the Copper Stockwork Footwall Zone.

Source: RPA

1.7 Mineral Reserve Estimate

Indicated and Inferred resources were used in the life-of-mine (LOM) plan and Inferred material represents 45% of the material planned for processing. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study (PFS) or a feasibility study (FS) of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at McIlvenna Bay.

1.8 Mining

McIlvenna Bay is proposed to be mined as an underground operation through long hole stoping and cemented paste backfill. Access will be from a shaft and ramp. The mine will operate at 5,000 tpd day and extract 23.7 million tonnes of mineralized material and 4.1 million tonnes of waste over its mine life.

The mine has been designed with 25 m level spacing, and major shaft levels every 100 vertical metres. Mineralized material will be hauled from the face to passes and fed through the underground crusher before being loaded into skips and hoisted to surface for processing. Waste will be stored in secondary stopes where possible, or hauled to waste passes and also crushed and skipped to surface for storage.

The mine will operate 365 days per year, running two 10 hour shifts per day. The shaft will be driven in two stages to defer upfront capital spending. While the first 750 m shaft is driven, a ramp will also be driven to access near surface material and provide a jump start on production. Once the shaft is complete, material will be hoisted through the shaft and the ramp will be reserved for man and material access. The shaft will be extended another 500 m in year six of operations to access the remainder of the resource.

Structural backfill in the form of cemented paste fill will be pumped underground from a surface paste facility to fill 73% of the opened stope voids. The remaining 27% will be filled with run of mine (ROM) waste as it is produced. Approximately 6.1M m^3 of paste fill will be pumped back underground, while 6.6 M m^3 of development waste are sent to a surface waste facility approximately 2.5 km north-west of the processing plant. The mine production schedule is shown in Table 1.2.

1.9 Recovery Methods

The metallurgical processing selected for the different mineralized material types, Copper Stockwork Zone (CSZ), Lens 2 Massive Sulphide (L2MS) and Upper West Massive Sulphide (UW-MS), were designed to produce copper concentrate, zinc concentrate or a bulk concentrate as final products depending on the material type fed to the plant.

The 5,000 tpd process plant flowsheet design follows conventional crushing, a semi-autogenous mill with a pebble crushing circuit, a ball mill grinding circuit using cyclones for classification followed by a talc pre-flotation step to remove detrimental talc prior to copper/zinc/bulk flotation. Conventional sequential flotation for the recovery of copper, zinc and bulk concentrates is utilized in this flowsheet. Rougher and scavenger flotation cells are used for zinc, while the copper and bulk circuits have only rougher cells. Regrinding prior to cleaning is required for all mineralization types. Each of the three mineralization types requires three stages of cleaning following regrind to produce final concentrates grades.

Concentrates will be shipped out via bulk trucks. Tailings from talc pre-float and the zinc flotation circuit will be sent to a common paste backfill system followed by tailings thickening. The thickener underflow will be sent for disposal in the tailings management facility.

1.10 Project Infrastructure

Project infrastructure at the McIlvenna Bay mine site will commence with upgrades to the existing 18 km access road. Eight kilometers of new light vehicles roads will be constructed to allow easy accessibility to all project areas. One of the major infrastructure upgrades will be a new 72kV overhead line that will supply site with the required power. Accompanying the new power line will be an onsite substation fed from the SaskPower grid and capable of handling the increased power demand of 25 kVA. During construction a temporary 200 person camp will be constructed.

Major building installations will include a 37,500 sq. ft. process plant, a 7200 sq. ft. truck shop warehouse, a bulk explosives storage facility and a 60,000 liter bulk fuel tank capable of supplying 50 gpm. A 400,000 l firewater tank will supply sufficient fire protection. Potable water and waste water treatment systems will be included in the temporary construction camp. These two facilities will remain in service after the construction camp is demobilized.

Table 1.2: Mine Production Summary

Parameter	Unit	Total	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09	Year 10	Year 11	Year 12	Year 13	Year 14
Production	Mt	**23.7**	0.91	1.37	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.38
Rate	ktpd	**4.6**	2.5	3.8	5	5	5	5	5	5	5	5	5	5	5	3.8
Au	g/t	**0.4**	0.3	0.4	0.5	0.5	0.4	0.3	0.4	0.5	0.4	0.4	0.4	0.3	0.4	0.4
Ag	g/t	**14.8**	14	18	15	15	17	12	14	12	11	19	19	15	13	13
Cu	%	**1.2**	1	1	1.2	1.3	1.2	1.2	1.1	1.2	1.3	1	1.1	1	1.2	1.5
Pb	%	**0.2**	0.1	0.2	0.2	0.2	0.2	0.1	0.2	0.1	0.1	0.2	0.2	0.1	0.1	0.1
Zn	%	**2.4**	2.7	3.3	2.2	2.1	2.1	1.7	1.9	1.8	1.4	3.3	3.4	3.8	2.2	1.3
NSR	CDN/t	**106**	99	108	113	116	107	97	94	100	105	103	110	110	104	111

1.11 Environmental Studies

McIlvenna Bay lies in the Boreal Plain Ecozone on the boundary of the Mid-Boreal Lowland and Churchill River Upland ecoregions. 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.

Terrestrial resources in the Project area include a number of vegetation species considered rare in the Province of Saskatchewan as well as 15 wildlife species of provincial and federal conservation priority. As such, additional mitigation and/or management consideration for species of provincial and federal conservation concern may be required for the Project. Of particular note, woodland caribou occur in and near the Project area. The boreal population of woodland caribou is listed as threatened under the federal Species at Risk Act. Adverse effect of a project on a listed wildlife species and its critical habitat must be identified, mitigated for, and monitored. In addition, a wetland habitat compensation plan will be required should any wetland habitat be lost as a result of the project.

The aquatic environment in the Project area consists of a number of lakes and streams, all of which ultimately flow into Hanson Lake, which then drains into the Sturgeon-Weir River, several large, lakes, the Saskatchewan River, and ultimately discharges into Hudson Bay through the Nelson River system. At least 15 species of fish are known to occur in the area including lake whitefish, northern pike, walleye, white sucker, and yellow perch. The tailings disposal site has the potential to have an effect on the aquatic environment in the area. Golder Associates Ltd. (Golder) completed a Tailings Management study which identified 12 potential sites within a 10km radius of McIlvenna Bay which would be suitable to store the estimated volume of tailings produced from McIlvenna Bay. The currently preferred option is Option 4 which utilizes the natural basin of Guyader Lake. Should Option 4 be selected as the preferred tailings disposal area, the aquatic habitat provided by Guyader Lake would be lost. Guyader Lake is a medium-sized headwater lake known to contain lake whitefish, northern pike, spottail shiner, white sucker, and yellow perch similar to many lakes in the area. As part of the EA, Foran would be required to prepare an assessment of alternatives for mine waste disposal for consideration, a fish habitat compensation plan, and participate in public and aboriginal consultations on the EA, including on possible amendments to the MMER.

From an environmental perspective, the next step for the project is to complete follow-up environmental studies in support of the Environmental Assessment process. This may include a quantitative aquatic habitat assessment of Option 4 in support of a fish habitat compensation plan, a quantitative wetland assessment in support of a wetland habitat compensation plan, and additional species at risk assessments.

1.12 Capital Costs

The initial capital cost estimate is $248.8M including a 20% contingency as summarized in Table 1.3**3**. Costs are expressed in Canadian dollars with no escalation (Q4 2014 dollars). Target estimate accuracy is +/-25%.

Table 1.3: Summary of Capital Costs

Capital Costs	**Pre-Production $M**	**Sustaining/ Closure $M**	**Total $M**
Site Development	0.9	0.0	0.9
Mining	72.9	119.6	192.5
Prim. Crushing & Coarse	5.8	0.0	5.8
Concentrator	53.8	0.0	53.8
Tailings & Waste Rock Management	3.1	4.9	7.9
On-Site Infrastructure	18.3	0.0	18.3
Off-Site Infrastructure	14.9	0.0	14.9
Project Indirects	18.8	0.0	18.8
Engineering & EPCM	15.8	0.0	15.8
Owner's Costs	3.0	0.0	3.0
Closure	0.0	10.0	10.0
Salvage	0.0	-9.3	-9.3
Subtotal	**207.3**	**125.2**	**332.5**
Contingency (20%)	41.5	25.0	66.5
Total Capital Costs	**248.8**	**150.3**	**399.1**

Source: JDS 2014

1.13 Operating Costs

Total operating cost over the life of mine amount to $1,211.3M. This translates to an average cost of $51.03/tonne processed over the life of mine. These costs are outlined in Table 1.44.

Table 1.4: Summary of Operating Costs

Operating Costs	**$/t milled**	**LOM C$M**
Mining	33.54	796.2
Processing - CSZ	12.91	174.9
Processing - UW	14.08	71.2
Processing - MS	13.94	71.6
G&A	4.10	97.4
Total	**51.03**	**1,211.3**

1.14 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to be more indicative of true investment value. Sensitivity analyses were performed for variations in metal price, head grades, operating costs, capital costs, and foreign exchange rate to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates and forecast of resulting cash flows as part of this study. The grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

1.15 Metal Prices

The metal prices and foreign exchange rate used in the economic analysis are spot metal prices and foreign exchange rate as at October 15, 2014. Table 1.55 summarizes the metal prices and exchange rate used in the economic analysis.

Table 1.5: Metal Prices and Exchange Rate used in the Economic Analysis

Commodity	Unit	Spot as at October 15, 2014
Copper Price	US$/lb	3.08
Lead Price	US$/lb	0.93
Zinc Price	US$/lb	1.06
Gold Price	US$/oz	1,238
Silver Price	US$/oz	17.00
Exchange Rate	US$:C$	0.89

Source: JDS 2014

1.16 Life of Mine Plan Summary

Recovered metals and concentrate production used in the economic analysis are provided in Table 1.66.

Table 1.6: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	13.7
Total Mined	**M tonnes**	**23.7**
Throughput Rate	tpd	4,761
Average Head Grade		
Cu	%	1.17
Zn	%	2.36
Pb	%	0.15
Au	g/t	0.42
Ag	g/t	14.82
Metal Production		
Cu Concentrate No. 1 (CSZ)	dmt	611,377
	dmtpa	44,759
Cu Concentrate No. 2 (UW)	dmt	253,809
	dmtpa	18,581
Zn Concentrate No. 1 (MS)	dmt	533,476
	dmtpa	39,056
Zn Concentrate No. 2 (UW)	dmt	244,582
	dmtpa	17,906
Bulk Concentrate (MS)	dmt	65,173
	dmtpa	4,771
Cu Payable	M lbs	513.7
	M lbs /yr	37.6
Pb Payable	M lbs	15.8
	M lbs /yr	1.2
Zn Payable	M lbs	804.7
	M lbs /yr	58.9
Au Payable	k oz	218.0
	k oz/yr	16.0
Ag Payable	k oz	5,437
	k oz/yr	398.0

Source: JDS 2014

1.17 Economic Results

Pre-tax and after-tax financial performance is summarized in Table 1.7. Pre-tax results provide a point of comparison with similar projects and are not intended to represent a measure of absolute economic value.

Table 1.7: Summary of Economic Results

Category	Unit	Value
LOM Pre-Tax Free Cash Flow	$M	894.6
Average Annual Pre-Tax Free Cash Flow	$M	65.5
LOM Taxes	$M	248.4
LOM After-Tax Free Cash Flow	$M	646.2
Average Annual After-Tax Free Cash Flow	$M	47.3
Discount Rate	%	7.0
Pre-Tax NPV	$M	381.7
Pre-Tax IRR	%	21.9
Pre-Tax Payback	Years	4.1
After-Tax NPV	$M	262.6
After-Tax IRR	%	18.9
After-Tax Payback	Years	4.1

Source: JDS 2014

1.18 Sensitivity Analysis

A sensitivity analysis was conducted on after-tax net present values (NPV) for individual parameters including metal prices, grades, operating costs, capital costs, and foreign exchange rate. The results are shown in Table 1.8. The project proved to be most sensitive to foreign exchange fluctuations and metal prices and showed least sensitive to change in capital costs.

Table 1.8: After-Tax $NPV_{7\%}$ Sensitivity Results

Factor	-10%	100%	+10%
Metal Price	129.4	262.6	394.5
Head Grade	156.3	262.6	368.0
OPEX	313.7	262.6	211.3
CAPEX	297.5	262.6	227.7
F/X Rate	378.7	262.6	166.9

Source: JDS 2014

The project was also evaluated using various discount rates to determine the effect on project NPV. Project NPV declined as the discount rate increase. Table 1.99 demonstrates the summary of the discount rate sensitivity test.

Table 1.9: Discount Rates Sensitivity Test

Discount Rate	Pre-Tax NPV $M	After-Tax NPV $M
0%	894.6	646.2
5%	490.2	344.4
7%	381.7	262.6
8%	335.5	227.5
10%	256.2	167.0
12%	191.2	117.2

Source: JDS 2014

1.19 Conclusions

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for the project. Standard industry practices, equipment and design methods were used in the PEA.

McIlvenna Baycontains a substantial base metal resource that can be mined by underground methods and recovered with conventional processing.

Based on the assumptions used for this preliminary evaluation, the project is economic and should proceed to the pre-feasibility or feasibility stage.

There is also a likelihood of improving the project economics by identifying additional mineral resources within the development area and other near-by deposits that may justify increased underground production or extend the mine life.

The most significant potential risks associated withMcIlvenna Bay are uncontrolled dilution, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

To date, the Qualified Persons (QP) are not aware of any fatal flaws for McIlvenna Bay.

1.20 Recommendations

It is recommended that McIlvenna Bay proceed to the feasibility study stage in line with Foran's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Foran's project development plans.

Prior to and/or concurrent with the studies and work programs designed to advance McIlvenna Bay towards Feasibility level studies, work should also be completed to validate and define several of the known additional sources of mineral resources which lie in close proximity. These potential resources consist of an historic deposit and an advanced exploration target, namely the historic Bigstone deposit and Thunder Zone prospect, which occur in proximity of McIlvenna Bay and may provide additional higher grade mill feed to the project early in the project life to reduce the payback period. Additional resources defined in these areas (assuming the completion of successful exploration programs) could substantially enhance the economics of development of McIlvenna Bay by adding additional higher grade mill feed early in the mine cycle and at the same time extend the life of any such operations.

An exploration program which encompasses diamond drilling and ancillary studies designed to produce 43-101 compliant resources and determine the suitability of the two target areas for mining is recommended. It is envisioned that exploration programs would be conducted in two phases. The phase I program should consist of 4,000 to 5,000 m of diamond drilling on the two target areas to establish that the required thresholds of potentially economic resources are present. It is estimated that the cost of the phase I program would be approximately $1.2-1.5M. Contingent on positive results from the initial program, a phase II program would be recommended consisting of additional diamond drilling of a density sufficient to support indicated resource classifications and initial metallurgical studies to determine potential recoveries, etc.

Assuming the successful completion of these proposed exploration programs, the additional satellite deposits should then be included in a revised PEA for the project and/or the deposits included in future feasibility studies.

It is estimated that a FS and supporting field work would cost approximately $13.0 million. A breakdown of the key components of the next study phase is as follows in Table 1.10.

Table 1.10: Cost Estimate to Advance McIlvenna Bay to Feasibility Stage

Component	Estimated Cost (M$)	Comment
Resource Drilling & Updated Resource	6.1	Conversion of inferred resources to indicated within and immediately adjacent to the proposed mine. Drilling will include holes for combined resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.5	Variability test work including expanded grinding testwork, evaluation of blending of mineralization types and testwork for ancillary processes (thickening and filtering)
Condemnation Drilling	0.4	Drilling under infrastructure and TMF to ensure no sterilization of resources
Geochemistry	1	ABA accounting tests and humidity cell testing to determine acid generating potential of all rock units and mitigation plans
Geotechnical/ Hydrology/Hydrogeology	0.5	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering & Paste Backfill Testing	4	FS-level mine, infrastructure, paste backfill & process design, cost estimation, scheduling & economic analysis
Environment	0.5	Other investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**13**	Excludes corporate overheads and future permitting activities

Source: JDS 2014

2 INTRODUCTION

2.1 Basis of Technical Report

This Technical Report was compiled by JDS for Foran. This technical report summarizes the results of the PEA study and was prepared following the guidelines of NI 43-101.

2.2 Scope of Work

This report summarizes the work carried out by the Consultants, some of which are associated or affiliated with Foran. The scope of work for each company is listed below, and combined, makes up the total PEA project scope.

JDS' scope of work included:

- Compile the technical report which includes the data and information provided by other consulting companies;
- Mine planning;
- Conduct optimal pit design and production schedule;
- Select mining equipment;
- Establish potentially mineable resources;
- Design required site infrastructure, identify proper sites, plant facilities and other ancillary facilities;
- Estimate Opex and Capex for the project;
- Prepare a financial model and conduct an economic evaluation including sensitivity and Project risk analysis; and
- Interpret the results and make conclusions that lead to recommendations to improve value, reduce risks.

Golder's scope of work included:

- PEA-level geotechnical assessment and estimate of appropriate stope sizes and development ground support; and
- PEA-level tailings management facility (TMF) evaluation design and construction cost estimation.

RPA Inc. (RPA) scope of work included:

- Project setting, history and geology description; and
- Mineral resource estimate.

Samuel Engineering, Inc. (Samuel) scope of work included:

- Develop a conceptual flowsheet, specifications and selection of process equipment;
- Estimate processing Opex and Capex; and
- Design processing to realize the predicted recoveries.

KWM Consulting Inc. (KWM) scope of work included:

- Implement and supervise the metallurgical testing program; and
- Establish recovery values based on metallurgical testing results.

Patterson & Cooke (P&C) scope of work included:

- PEA-level paste backfill plant and distribution system design and paste backfill capex and opex estimation.

Canada North Environmental Services (CanNorth) scope of work included:

- Review environmental and other permit requirements; and
- Summarize environmental results and concerns.

2.3 Qualifications and Responsibilities

The QP's preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

The following individuals, by virtue of their education, experience and professional association, are considered QP as defined in the NI 43-101 standard for this report, and are members in good standing of appropriate professional institutions. The QPs are responsible for specific sections as follows:

Table 2.1: Qualified Person Responsibilities

Author	Company	Report Section(s) of Responsibility
Mr. Michael E. Makarenko, P. Eng.	JDS	1 (except 1.3-1.6 and 1.9), 2,3, 15, 16 (except 16.2 and 16.12), 18 (except 18.19), 20,21, 23 (except 21.1.2, 21.2.2), 20, 22, 24, 25, 26, 27
Mr. Darren Kennard, P. Eng.	Golder	16.2
Mr. Matt Bender, P.E.	Samuel	1.9, 17, 21.1.2, 21.2.2
Mr. David Rennie, P. Eng.	RPA	4, 5, 6, 7, 8, 9, 10, 11, 12, 14
Mr. John Hull, P. Eng.	Golder	18.19
Mr. Leslie Correia, P. Eng.	P&C	16.12
Mr. Ken Major, P. Eng.	KWM	1.5, 13

Source: JDS 2014

2.4 Site Visits

- Michael Makarenko visited the site July 23, 2014;
- Darren Kennard has not visited the site;
- Matt Bender has not visited the site;
- David Rennie visited the site September 21-22, 2011;
- John Hull has not visited the site;
- Leslie Correia has not visited the site; and
- Ken Major has not visited the site.

2.5 Currency

Unless otherwise specified, all costs in this report are presented in Canadian Dollars (CA$ or $).

2.6 Units of Measure, Calculations & Abbreviations

All units in this report are based on the International System of Units (SI), except industry standard units, such as troy ounces for the mass of precious metals and pounds for the mass of base metals.

A list of main abbreviations and terms used throughout this report is presented in Table 2.2.

This report may include technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a slight margin of error. Where these occur, JDS does not consider them to be material.

2.7 List of Abbreviations

Units of measurement used in this report conform to the SI (metric) system. A complete list of abbreviations is shown in Table 2.2.

Table 2.2: Units of Measure & Abbreviations

°C	degree Celsius
°F	degree Fahrenheit
A	ampere
a	annum
Ag	silver
ASA	Aquatic Study Area
ASKI	ASKI Resource Management and Environmental Services
Au	gold
bbl	barrels
Btu	British thermal units
C$ or CA	Canadian dollars
cal	calorie
Cameco	Cameco Corporation
CanNorth	Canada North Environmental Services
CEA Agency	Canadian Environmental Assessment Agency
CEAA	Canada North Environmental Services
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
Cu	copper
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight ton
EA	Environmental Assessment
EC	Environment Canada
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMPA	Environmental Management and Protection Act
Foran	Foran Mining Corporation
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
G	giga (billion)
g	gram
g/L	gram per litre
g/t	gram per tonne
Gal	Imperial gallon

Golder	Golder Associates
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
HRIA	Heritage Resource Impact Assessment
in	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
l	liter
l/s	liters per second
LSA	Local Study Area
M	mega (million)
m	metre
μ	micron
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
MASL	metres above sea level
MBCA	Migratory Birds Convention Act
$\mu\gamma$	microgram
min	minute
mm	millimetre
MMER	Metal Mining Effluent Regulations
MOE	Saskatchewan Ministry of the Environment
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
opt, oz/st	ounce per short ton
oz	Troy ounce (31.1035g)
Pb	lead
PBCN	Peter Ballantyne Cree Nation

PEA	Preliminary Economic Assessment
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
RSA	Regional Study Area
s	second
SARA	Species at Risk Act
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
TOR	Terms of Reference
tpa	metric tonne per year
tpd	metric tonne per day
TSF	Tailings Storage Facility
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
VMS	Volcanogenic Massive Sulfide
W	watt
wmt	wet metric tonne
yd^3	cubic yard
yr	year
Zn	zinc

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by JDS for Foran. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to JDS at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report; and
- Data, reports, and other information supplied by Foran and other third party sources.

The Environmental Studies, Permitting, and Social or Community Impact Section was written by CanNorth. Michael Makarenko reviewed this section and assumed responsibility for its content.

For the purpose of this report, JDS has relied on ownership information provided by Foran. JDS has not researched property title or mineral rights for McIlvenna Bay and expresses no opinion as to the ownership status of the property.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party are at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

McIlvenna Bay occurs within Foran's McIlvenna Bay property located approximately 1 km south of Hanson Lake, Saskatchewan. The property is also approximately 375 km northeast of Saskatoon and 65 km west-southwest of Flin Flon, Manitoba (Figure 4.1). McIlvenna Bay is located within 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. McIlvenna Bay is located well within the property boundaries.

4.1 Land Tenure

The entire McIlvenna Bay property comprises 30 claims totalling 20,382 ha (Figure 4.2). A tabulation of the relevant claim information is listed in the RPA's 2011 resource report. The claims are listed in the name of Foran and are kept in good standing at the discretion of Foran. Foran has engaged an independent firm to track and maintain the claims in good standing. The information contained within this report was provided by Foran and/or its designates. RPA has not confirmed the validity of the mineral tenures but has no reason to doubt their validity.

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

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

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

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

On November 3, 2010, Foran announced the closure of an agreement for acquisition of Copper Reef's 25% interest in the McIlvenna Bay property. The deal included transfer to Foran of 3,000,000 Copper Reef shares, and the nearby North Hanson property. In exchange, Copper Reef received 4,000,000 Foran shares (to hold 8% on a non-diluted basis), $1,000,000 cash, a Net Tonnage Royalty of C$0.75/t on future ore produced from the property, and five Manitoba properties selected by Copper Reef from Foran's portfolio.

4.2 Permits and Authorization

Foran has acquired one Industrial Lease for the exploration camp (#303228) and one Miscellaneous Use Permit (MUP #603298) for the camp wastewater lagoon from the Ministry of Environment. These two 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 a silica sand quarrying operation near McIlvenna Bay and there are quarry dispositions that overlap Foran mineral claims. Some additional quarry staking took place west and northwest of McIlvenna Bay in January and February of 2012. On December 8, 2012 the Saskatchewan Ministry of Energy and Resources placed a Crown Reserve (CR #965) over McIlvenna Bay that restricts additional quarry staking in the deposit area.

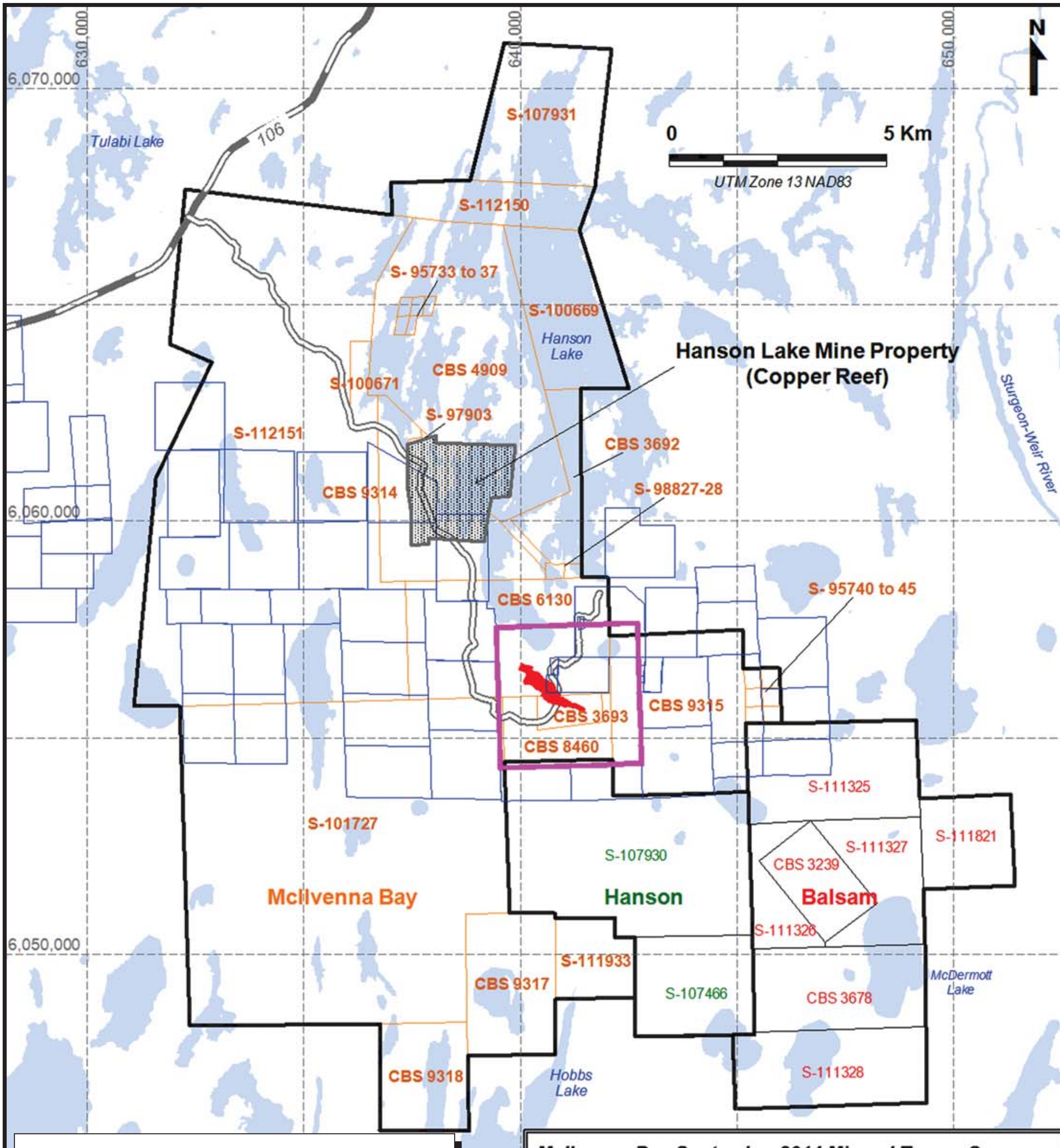
The company reports that a potential land-use conflict between the development of McIlvenna Bay and quarrying operations has been addressed. An area of exclusion from quarrying above McIlvenna Bay has been agreed upon by Foran, Preferred Sands of Canada ULC (Preferred), the Ministry of Environment, and the Ministry of the Economy. RPA is not aware of any other constraints on access rights to the property.

Foran Mining Corporation

McIlvenna Bay Project

Saskatchewan, Canada

Location Map



Foran Mining Corporation

McIlvenna Bay Project

Saskatchewan, Canada

Mineral Tenure Summary Map

October 2014 ***Source: Foran Mining Corp., 2014.***

4.3 Mining Rights in Saskatchewan

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

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

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

- Year two to year ten: $15/ha; and
- Thereafter: $25/ha.

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

- Assaying and testing; and
- Metallurgical, mineralogical, or other scientific studies.

Bulk sampling may be conducted, although any minerals recovered in the program remain the property of the Crown.

Leases are created through the conversion of a Mineral Claim and convey the right to extract, recover, or produce minerals from Crown lands. They are issued for a ten year term, which is renewable. Work expenditures are also required to maintain leases, although there are provisions for relief from these expenditure requirements where conducting exploration work would interfere with ongoing mining operations, or further exploration is no longer warranted.

Exploration and mining activities are regulated by the Ministry of Environment and the Ministry of Energy and Resources. Parties wishing to construct, operate, modify, temporarily close, or decommission and reclaim a mine must apply for approvals from the Environment Minister. The application process for a new mine would comprise an application for approval to construct the mine facility and attendant infrastructure.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following section is taken from the 2006 RPA Technical Report (Cook and Moore, 2006).

5.1 Accessibility

McIlvenna Bay is located 1 km south of Hanson Lake, Saskatchewan, and approximately 95 km by road west of Flin Flon, Manitoba (Figure 4.1). 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 neighbouring mining towns of Flin Flon, Manitoba/Creighton, Saskatchewan (population 7,100), represents the largest commercial/residential centre in the area. Flin Flon provides a railhead that connects the area to the North American railway system. Electrical power would be available from SaskPower at Creighton, Saskatchewan.

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 ecoclimate. 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.

5.3 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 supportive of mining.

5.4 Infrastructure

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

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

Water for a mining/milling operation could be drawn from one of the local lakes. Tailings impoundment areas would have to be constructed locally where there is the advantage of the natural alkaline buffering capacity of bedrock dolomite.

5.4.1 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.

6 HISTORY

The following section is largely taken from the 2006 RPA Technical Report (Cook and Moore, 2006), which was also quoted in the 2011 RPA Technical Report (Rennie, 2011).

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

In 1976, the Saskatchewan Mineral Development Corporation (SMDC), the provincial government exploration vehicle that eventually became Cameco Corporation, acquired a large exploration lease centered on Hanson Lake. The permit area covered much of the exposed portion of the Hanson Lake Block (see Item 7 Geological Setting) and extended several kilometres south of the present McIlvenna Bay Property. In 1977, SMDC flew an Aerodat helicopter-borne EM survey across much of the permit area with lines oriented east-west.

From 1978 to 1988, Cameco tested selected Aerodat EM anomalies with ground follow-up exploration programs consisting of grid establishment, geological mapping (in the exposed portions of the belt), and ground geophysical surveys which included Horizontal Loop EM (HLEM), Time-Domain EM (TEM), and Surface Pulse EM surveys. Diamond drilling led to the discovery of three new showings, the Miskat Zone (Cu), the Grid B occurrence (Zn), and the Zinc Zone (Zn).

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

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

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

7 GEOLOGICAL SETTING AND MINERALIZATION

The following section is taken from the 2006 RPA Technical Report (Cook and Moore, 2006).

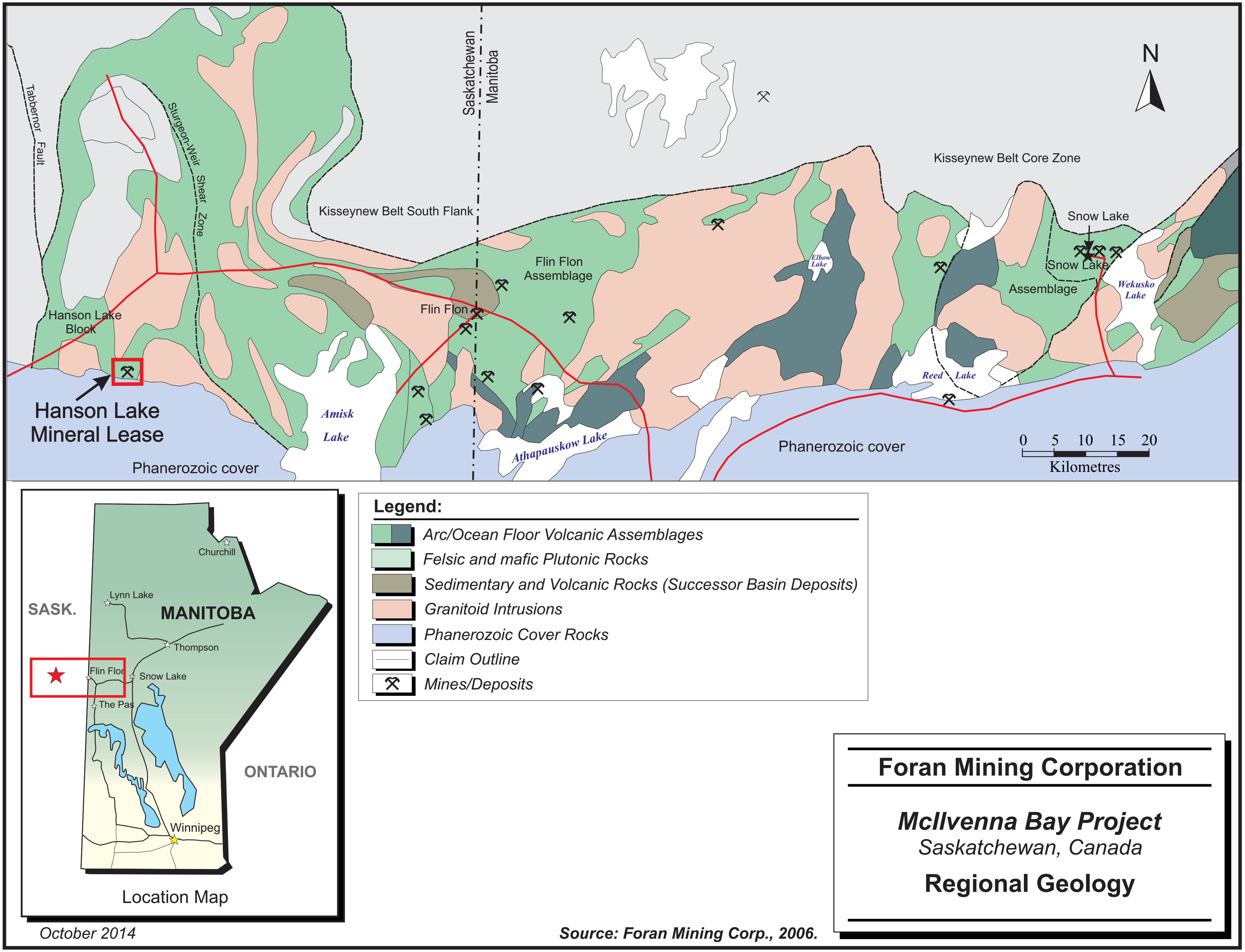
7.1 Regional Geology

McIlvenna Bay is located on the western edge of the Paleoproterozoic Flin Flon Greenstone Belt (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) as a result 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 Belts (MORBs) erupted in back-arc basins. Evolved arc assemblages and Archean crustal slices are present within the FFGB as minor components.

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

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, Sheridon, Flin Flon, Birch Lake, West Amisk, Hanson Lake, and Northern Lights assemblages (Zwanzig et al., 1997 and Maxeiner et al., 1999). These assemblages are separated by major structural features and/or areas of differing tectonostratigraphic origin. It is unclear whether the eight juvenile arc sequences represent different island arcs, or segments of a larger continuous arc (Syme et al., 1999). Within the belt, each tectonostratigraphic block has been broken into several sub-blocks, usually bounded by local to regional fault systems. Correlation of stratigraphy between sub-blocks is difficult to impossible to determine.


N
Kisseynew Belt Core Zone
Kisseynew Belt South Flank
Saskatchewan
Manitoba
Tabbernor Fault
Sturgeon-Weir Shear Zone
Flin Flon Assemblage
Snow Lake
Snow Lake Assemblage
Elbow Lake
Wekusko Lake
Hanson Lake Block
Flin Flon
Reed Lake
Amisk Lake
Athapauskow Lake
Hanson Lake Mineral Lease
Phanerozoic cover
Phanerozoic cover
0 5 10 15 20
Kilometres
Legend:
Arc/Ocean Floor Volcanic Assemblages
Felsic and mafic Plutonic Rocks
Sedimentary and Volcanic Rocks (Successor Basin Deposits)
Granitoid Intrusions
Phanerozoic Cover Rocks
Claim Outline
Mines/Deposits
SASK.
MANITOBA
ONTARIO
Churchill
Lynn Lake
Thompson
Flin Flon
Snow Lake
The Pas
Winnipeg
Location Map
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Regional Geology
October 2014
Source: Foran Mining Corp., 2006.

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

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

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

7.2 Local Geology

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

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 greywackes. 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 wide spread 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 dykes cut the sequence, as well as minor ultramafic intrusions (Koziol et al., 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. McIlvenna Bay projects to subsurface under the sedimentary cover (Lemaitre, 2000).

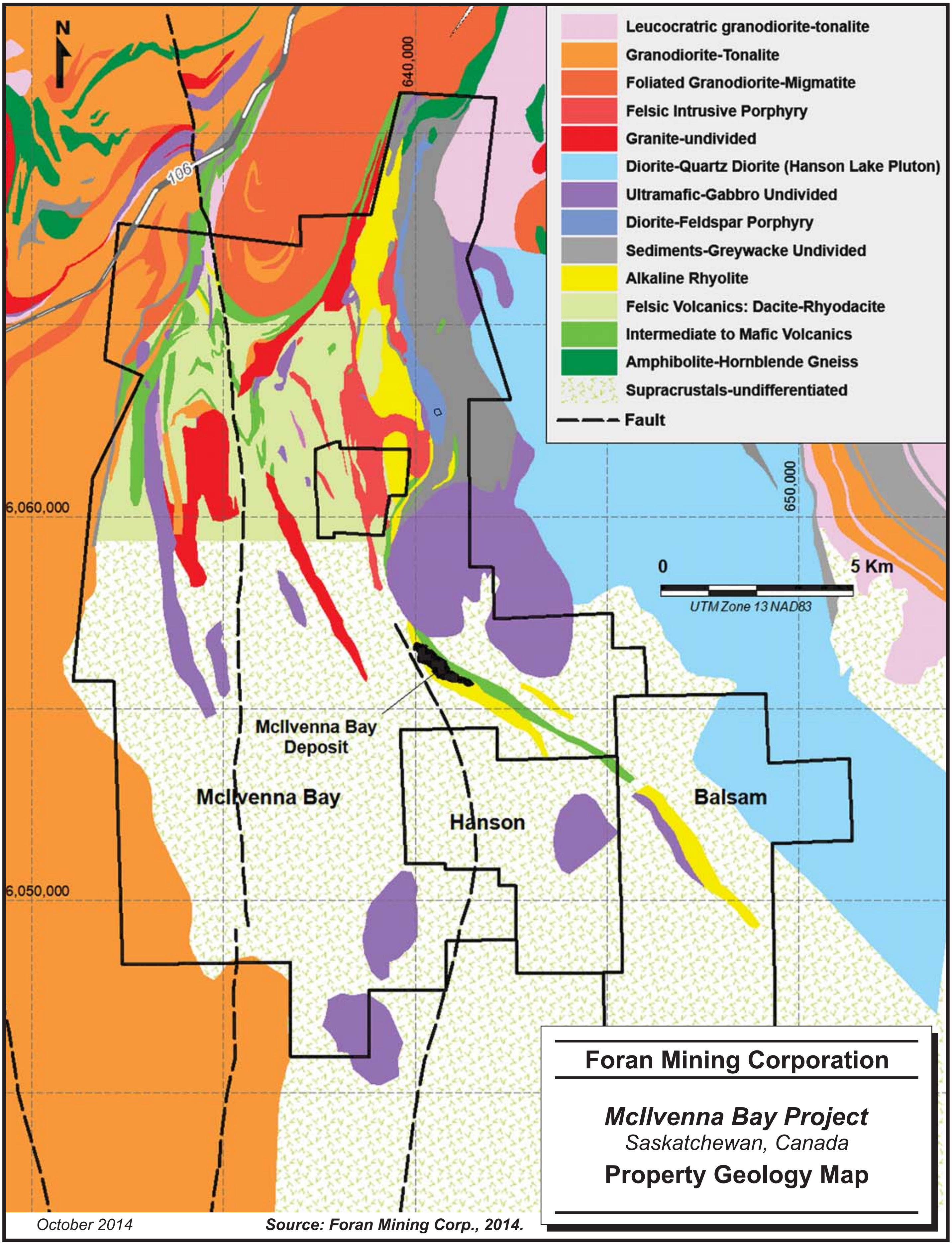
At least two distinct folding events, both having northerly trending fold axes, have influenced the stratigraphy in the Hanson Lake Area. The Hanson Block structural fabric is dominated by a north to northwest-southeast trending, upright regional transposition foliation. A protracted D2 structural event 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 McIlvenna Bay, the Proterozoic sequence exhibits a greenschist metamorphic facies as the deposit alteration assemblages are dominated by sericite and chlorite. The greenschist facies is probably a retrograde event after a previous amphibolite grade since relict cordierite, anthophyllite, garnet and andalusite are commonly observed in the VMS alteration package (Lemaitre, 2000). U-Pb ages of supracrustal rocks in the block constrain the metamorphic event between 1808 and 1804 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 1888 ± 12 Ma.

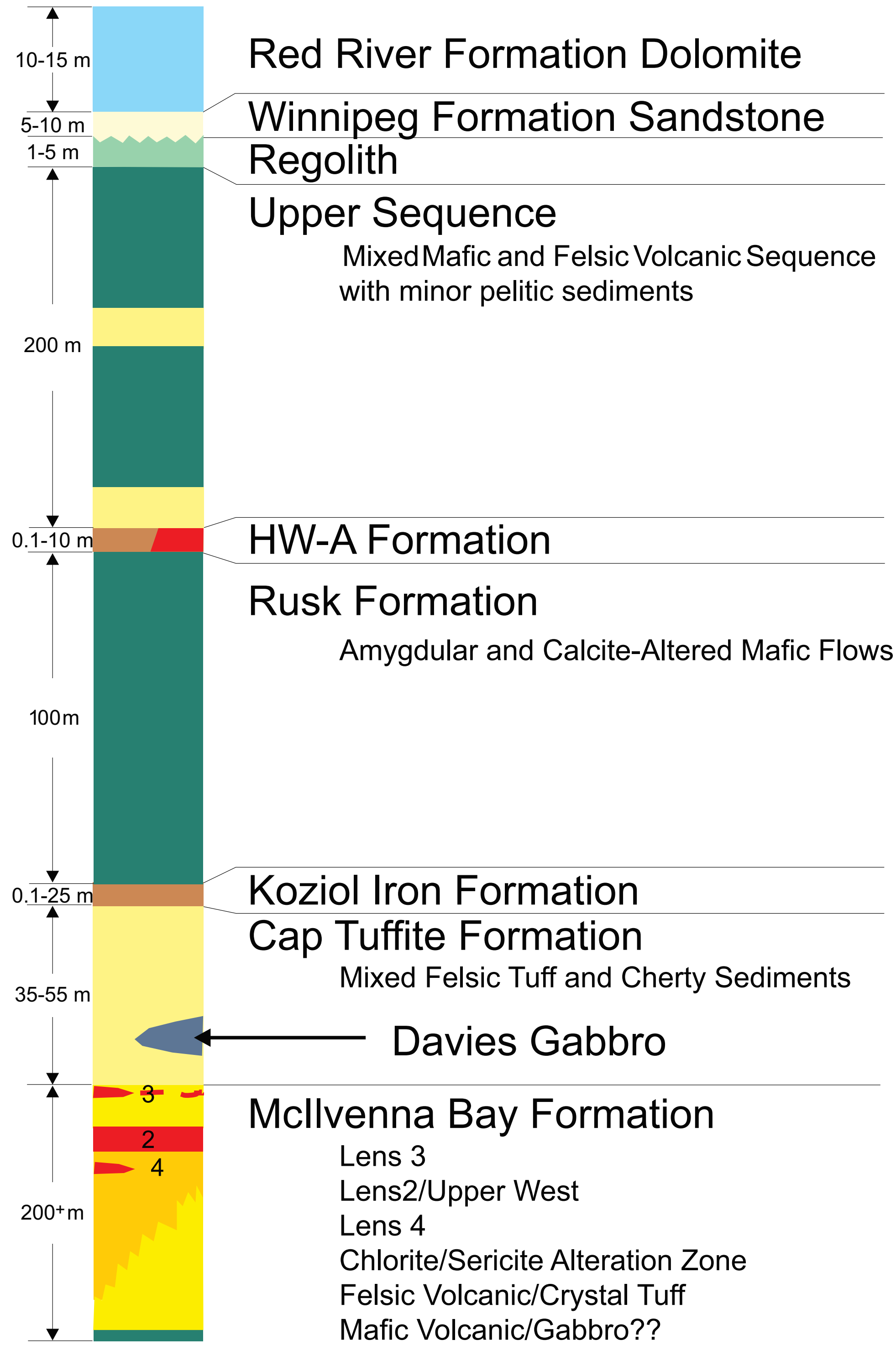
7.3 Property Geology

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, divided into six formations (Figure 7.3), has been defined over a two kilometre strike length by a total of 191 drill holes. The lowest formation intersected by drilling both structurally and stratigraphically is the McIlvenna Bay Formation (Figure 7.4), the host of McIlvenna Bay. The McIlvenna Bay Formation is overlain to the north by the Cap Tuffite Formation. The McIlvenna Bay Formation and the Cap Tuffite Formation may be genetically related but have been separated as they are temporally distinct, as demonstrated by the positioning of McIlvenna Bay 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 volcanics. The Rusk Formation in turn is overlain by the thin HW-A Formation, an exhalative massive sulphide horizon which grades laterally into iron formation. Capping the HW-A Formation is a thick unsorted bimodal package of mafic and felsic volcanics and mafic intrusions and minor iron formations tentatively called the Upper Sequence which may be thickened due to folding and faulting. The stratigraphic package has been cut by several different intrusions, the largest of which is the Davies Gabbro, a sill-like plug found within the Cap Tuffite Formation. The 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.



N
640,000
650,000
6,060,000
6,050,000
106
Leucocratric granodiorite-tonalite
Granodiorite-Tonalite
Foliated Granodiorite-Migmatite
Felsic Intrusive Porphyry
Granite-undivided
Diorite-Quartz Diorite (Hanson Lake Pluton)
Ultramafic-Gabbro Undivided
Diorite-Feldspar Porphyry
Sediments-Greywacke Undivided
Alkaline Rhyolite
Felsic Volcanics: Dacite-Rhyodacite
Intermediate to Mafic Volcanics
Amphibolite-Hornblende Gneiss
Supracrustals-undifferentiated
Fault
0
5 Km
UTM Zone 13 NAD83
McIlvenna Bay Deposit
McIlvenna Bay
Hanson
Balsam
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Property Geology Map
October 2014
Source: Foran Mining Corp., 2014.



Foran Mining Corporation

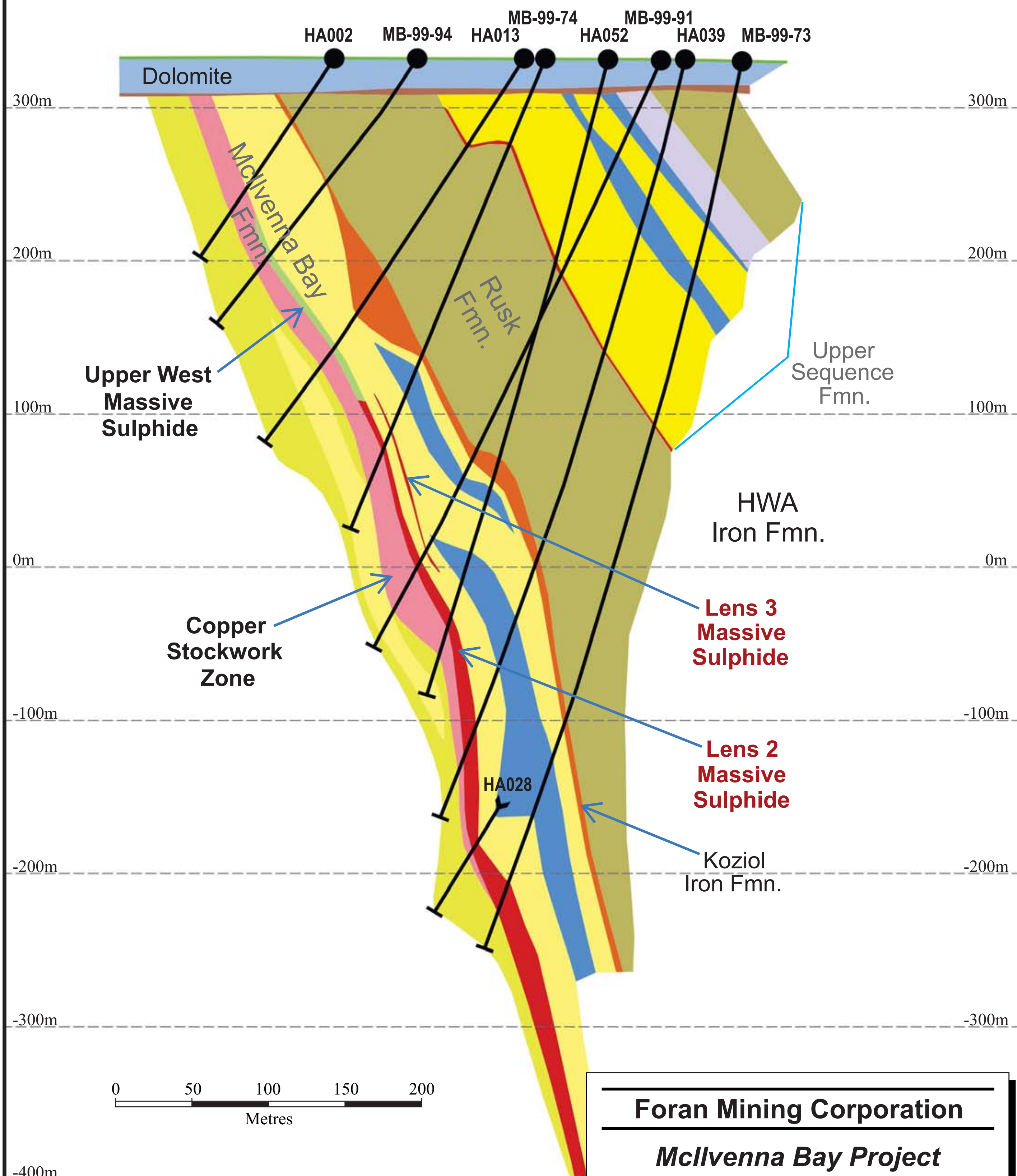
McIlvenna Bay Project

Saskatchewan, Canada

Stratigraphic Column
McIlvenna Bay Deposit Area

October 2014

Source: Foran Mining Corp., 2011.



Foran Mining Corporation

McIlvenna Bay Project

Saskatchewan, Canada

Cross Section 9700E (Looking WNW)

October 2014

Source: Foran Mining Corp., 2011.

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

Overlying the mineralized horizons of the McIlvenna Bay Formation is the Cap Tuffite Formation, a sequence of intercalated felsic volcanic and cherty metasediments which have been intruded by sills and dykes of the Davies Gabbro (described below). The unit ranges from 35 m to 55 m thick, is finely banded to finely laminated, and ranges from white to cream to grey-green in colour. Sections of the formation range from very finely laminated, bleached chert to 1 to 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 re-deposited, water-lain, distal volcaniclastics 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 traceable in drill core and by geophysics over several kilometres and, as such, an excellent stratigraphic marker horizon. The unit is a true oxide-facies iron formation that ranges from 0.1 m to 25 m true thickness and is composed of one to five centimetre 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 ± garnet ± magnetite ± chlorite bands. Occasional pyrite and/or pyrrhotite are observed in selected bands. Near the base of the iron formation is a one metre thick bed of graphitic chert.

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 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 sphalerite. From the centre of McIlvenna Bay and to the west, the HW-A Formation is an oxide-facies iron formation identical to that of the Koziol Formation. Overlying the iron formation is a one metre to 10 m thick massive mafic volcanic unit. From the centre of the deposit and to the east, the unit comprises either a thin pyrite band or massive sphalerite-pyrite from 10 cm to 75 cm thick. Overlying this portion of the unit is a thin 5 to 15 m thick massive, grey felsic volcanic unit.

Overlying the HW-A Formation is +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, grey, felsic volcanic, and 50% fine-grained mafic volcanic rocks. Some of the mafic units may be gabbroic intrusions. Approximately 5% of the unit is composed of greywackes 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 McIlvenna Bay, extends westward toward the centre of the sulphide body where it narrows into a series of thin dykes. The gabbro appears to be a series of sills that have intruded along the bedding planes of the Cap Tuffite Formation. The gabbro plug plunges along an axis parallel to the sulphide body and appears to exert some sort of control over the limits of mineralization along the bottom plunge line of the deposit. The unit ranges from fine-grained to very coarse grained; the grain size appears to be directly related to the unit thickness. Chilled margins have been observed on the thicker dykes. It appears that the gabbro intruded along the bedding planes of the wet, cherty banded sediments of the Cap Tuffite.

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.

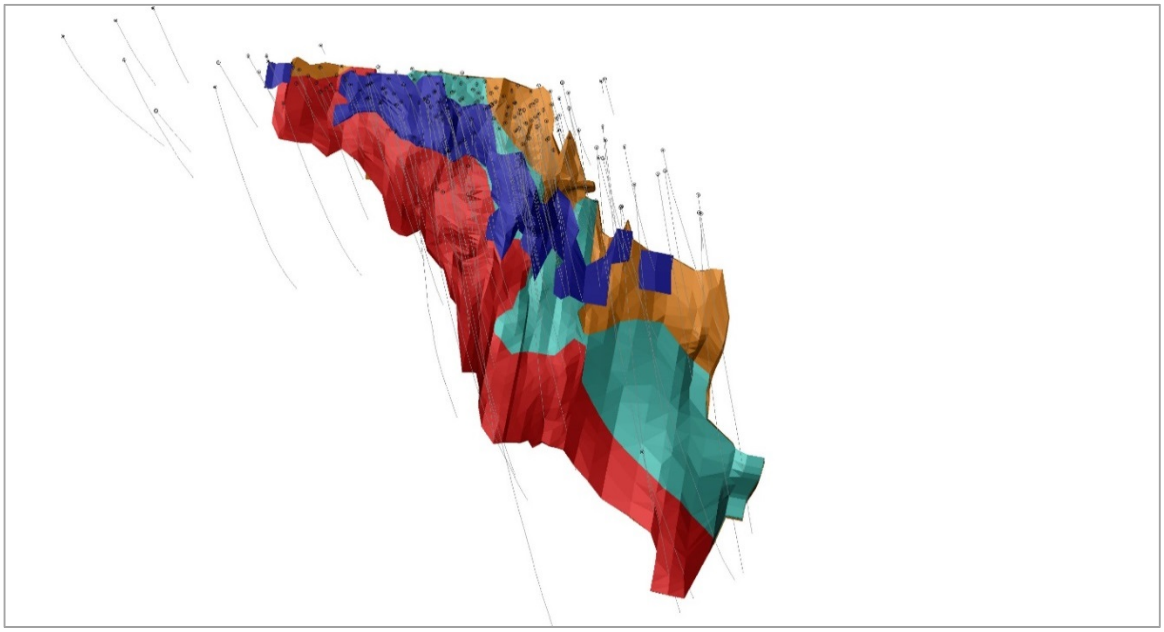
Two phases of folding of the host stratigraphy have been observed in the drill core and are believed to correspond to the regional F2 and F3 folding events. The first phase (regional F2) was responsible for the development of the dominant observed structural fabric, a foliation oriented at approximately 280°/65° (parallel to stratigraphy). The foliation is well developed in the least competent stratigraphic units, particularly the footwall altered rocks. Isoclinal folding of the iron formation, observed in several drill holes, has 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. A strong crenulation (regional F3 event) of the foliation is developed 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 and may be responsible for the broad warping of the stratigraphy observed in the magnetic maps between the Hanson Lake and the south end of McIlvenna Bay (Lemaitre, 2000).

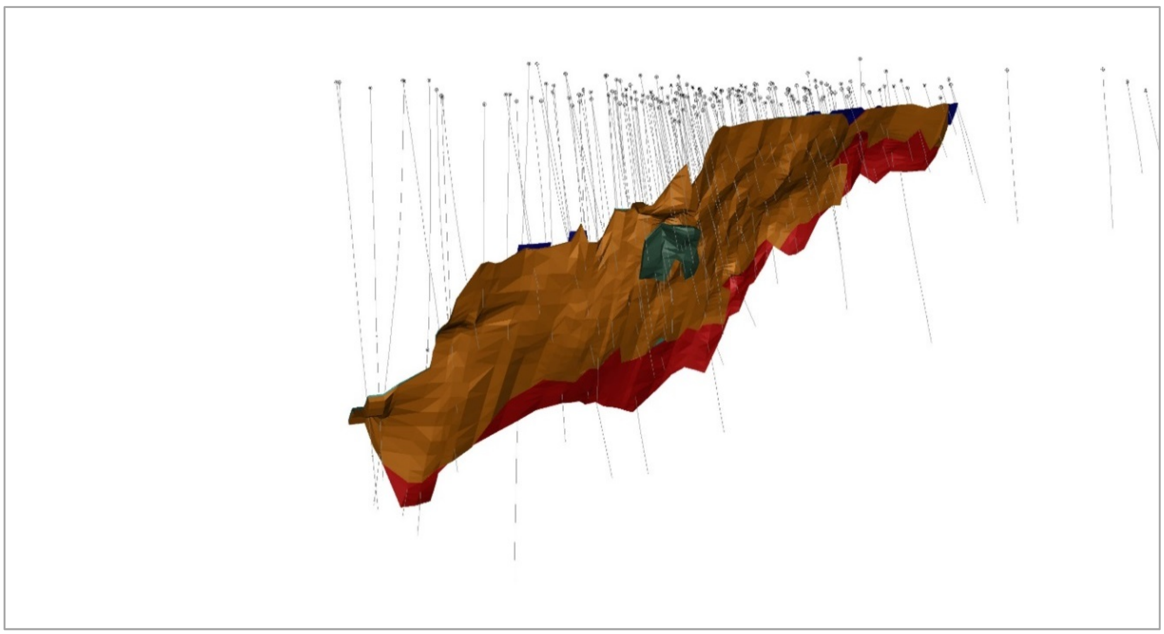
Evidence of faulting has been documented in drill core. However, it is difficult to determine the orientation, scale, or continuity of most faults between drill holes with the present level of information. The deep drilling program outlined a large fault structure that strikes east-northeast and dips steeply to the north. This fault appears to truncate the northern limb of the regional F2 synclinal structure discussed above. The fault is well to the north of the deposit and likely would not impact on the mineralized horizon above the 1,800 m vertical depth (Lemaitre, 2000).

7.5 Mineralization

McIlvenna Bay comprises five different zones and includes three distinct styles of mineralization. The five different zones identified are the Lens 2, the Upper West, Lens 3, and two separate Copper Stockwork Zones (see Figure 7.5). The three different styles of mineralization are massive sulphides, semi-massive sulphides, and stockwork. Each style is mineralogically and texturally distinct.

Figure 7.5: 3D Views of Mineralized Bodies





Source: RPA Inc.

The Lens 2 Massive Sulphide (L2MS) is by far the largest and most significant massive sulphide zone in the McIlvenna Bay deposit. As it is presently interpreted, it has a strike length of 400 m to 550 m, ranges in true thickness from 0.40 m to 16.75 m and has an average thickness of 5.6 m. The zone plunges approximately 45° to the north and strikes at 295° with an average dip of 68°. The L2MS extends between 35 m and 1,200 m below surface, along an overall plunge distance of approximately 1,880 m.

The Upper West Zone (UW-MS) is a relatively copper-gold-enriched semi-massive sulphide unit found as a long strip that lies parallel to and along the top of the plunge line of the L2MS. It is believed to be on the same stratigraphic horizon as, and laterally continuous with, the L2MS, with a 25 m to 50 m wide transition zone between the two. The UW-MS Zone has a strike length of 150 m to 300 m and has been delineated between the vertical depths of 35 m and 1,230 m. It varies from 2.80 m to 10.60 m true thickness and averages 4.8 m true thickness. The zone remains open down plunge below the 1,230 m level.

The Lens 3 Massive Sulphide (L3) is a discontinuous and comparatively thin massive and semi-massive sulphide horizon that is located 10 m to 30 m above the L2MS and UW-MS horizon. The zone has a strike length of up to 350 m and plunges parallel to the underlying mineralized zones. The true thickness of the zone ranges from 0.2 m to 6.7 m and averages 2.4 m. The zone is dominantly massive sulphides although semi-massive sulphides and typical copper stockwork mineralization have been observed in some holes. The zone is occasionally underlain by weak copper stockwork mineralization. The majority of the zone comprises sub-economic massive sulphides over widths of less than 3.0 m. Pods within the zone obtain threshold economic grades that exceed the minimum mining width, the largest of which has a strike length of 250 m and a dip extent of 50 m.

The Copper Stockwork Zone (CSZ) underlies and is in contact with the UW-MS and the western half of the L2MS. The zone is wedge-shaped with the blunt edge running parallel to the plunge of, and underlying, the UW Zone. The wedge terminates near the central axis of the L2MS. The zone is thickest where it underlies the UW-MS, and it is considered to be the proximal feeder zone for the hydrothermal system which deposited the massive sulphides. Stockwork mineralization in this area is hosted in chlorite-altered rock. To the east, immediately underlying the L2MS, the stockwork mineralization is hosted in fine fracture networks in silicified and sericitized rock for a horizontal distance of 1,750 m and a down-plunge distance of 1,950 m to an approximate depth of 1,100 m below surface. The interpreted wireframe of the main body measures approximately 460 m at its widest point, ranges up to 52.8 m in apparent thickness, and averages 9.5 m in true thickness. For most of its length, the CSZ extends upwards in elevation above the upper limit of the UW-MS. The zone remains open to the west and down plunge below the 1,100 m level.

A small body of stockwork-type mineralization occurs in the footwall of the primary body between eastings 9340 E and 9490 E and elevations 2,790 m and 3,115 m. This zone appears to be stratigraphically distinct from the main CSZ, however, fault repetition or folding has not been entirely ruled out. The drill pattern is too broad to allow for detailed structural interpretations. This zone is referred to as the Footwall Stockwork Zone (FW).

A very small semi-massive sulphide body occurs within the FW and was originally called the L4. It was interpreted to lie approximately 40 m to 50 m below the UW-MS, roughly in the centre of the deposit. Subsequent re-interpretation of the geology resulted in the L4 being included within the FW and it is no longer considered as a separate body.

Massive sulphides are typical of the L2MS and L3 horizons. This style of mineralization is composed of 70% to 80% medium-sized and subrounded pyrite grains resembling 'buckshot'. Sphalerite is hosted as fine-grained and sometimes feathery minerals located in the interstices of the pyrite grains ranging from 5% to 25% of the total unit. The sphalerite is generally dark to medium brown in colour. Faint banding of the massive sulphides is occasionally apparent. Up to 10% fine-grained grey quartz, and occasionally fine calcite, is also observed in the interstices. Subangular to subrounded inclusions or fragments of massive black chlorite ranging from two millimetre to 50 mm in diameter comprise 10% of the unit. Patchy but commonly rounded chert fragments ranging from 1 to 3 cm in diameter can constitute up to 20% of the unit locally. Such chert, when present, is often surrounded by one to three centimetre thick zones of enhanced, pale brown sphalerite.

The semi-massive sulphides are typical of the UW, and selected parts of the L3. 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 generally confined to the area below the UW-MS and L2MS, but has been observed underlying the L3. The nature of the stockwork zone mineralization varies according to the host rock alteration. Chlorite alteration-hosted copper stockwork mineralization comprises chalcopyrite and pyrrhotite, with occasional pyrite, and is found in veinlets and pods cutting the chlorite alteration. Sericite-quartz altered copper stockwork zones tend to comprise exclusively chalcopyrite which lines fine, hairline fractures within the strongly silicified host, and as five to ten centimetre long semi-massive pods containing angular to rounded host rock fragments. These pods and fractures appear to be late brittle features and suggest that the chalcopyrite was remobilized into fractured rock possibly during deformational events. This latter style of copper stockwork mineralization typically lies as a subordinate unit beneath the L2MS.

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 UW Zone. The location of the L3 and L4 zones respectively overlying and underlying the UW is interpreted by Foran geologists to indicate the occurrences of smaller hydrothermal pulses along different stratigraphic timelines.

The UW-MS, L2MS, and CSZ all remain open down plunge and, likely, both the zones and the plumbing system underlying them will continue at depth. In RPA's opinion, this is an important exploration target.

8 DEPOSIT TYPES

The following section was taken from the 2011 RPA Technical Report (Rennie, 2011).

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

VMS deposits are exhalative deposits, formed through the focused discharge of hot, metal-rich hydrothermal fluids. These deposits commonly occur in clusters which form a VMS camp. In many cases, it can be demonstrated that the sub-seafloor fluid convection system was apparently driven by large, 15 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., 2005).

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

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

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

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

McIlvenna Bay consists of structurally modified, stratiform, volcanogenic, polymetallic massive sulphide mineralization and associated stringer zone mineralization. The sulphides contain copper and zinc, with low lead and silver and gold values.

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

9 EXPLORATION

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. Phase I of this program commenced in December 1998 and carried out through the winter of 1998-1999. A total of 55 holes were drilled during this program, totalling 27,958 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. R. Lemaitre of M'Ore prepared a resource estimate which was released on June 14, 2000. 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. Cut-off grades of 1.5% Cu or 4.0% Zn were used. The area between L93+00E and L103+50E and above the 580 m vertical depth was deemed to have been drilled at sufficient intersection spacings to be classified as an Indicated resource. The remainder of the mineralization delineated to a maximum depth of 1,230 m vertical was classified as Inferred resources.

As of May 31, 2000, Foran had drilled 59 additional holes totalling 33,350 m into the property, with 57 holes directly testing the deposit. The first 44 holes were drilled with the objective of upgrading the quality of the resource 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 and extended 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. As described in the Land Tenure section of this report, Foran acquired a new option agreement in 2005, and resumed work. Scott Wilson RPA (a predecessor to RPA Inc.) was retained in 2006 to audit the Mineral Resource estimate and prepare a NI 43-101 Technical Report (Cook and Moore, 2006). The Mineral Resources dropped significantly owing to an increase in the cut-off grade used, which resulted in removal of much of the CSZ (then termed Copper Stringer Zone).

In early 2007, Foran completed an airborne deep-penetrating time-domain electromagnetic (VTEM) survey over 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 (Figure 9.1).

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

Exploration work underwent a hiatus until 2011, when Foran carried out a diamond drilling program consisting of 5,056.0 m in 10 holes conducted during the late winter and spring. Drill core from some of the earlier programs was also relogged and sampled.

RPA was retained to update the Mineral Resources estimate (Rennie, 2011) for the CSZ. The zone was re-interpreted, using a nominal 0.5% Cu cut-off grade 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 inclusion of the CSZ resulted in a large increase in the total Mineral Resources for the Project. The 2011 estimate is summarized in Table 9.1 and is superseded by the current Mineral Resource estimate contained in Section 14 of this report.

Table 9.1: McIlvenna Bay Mineral Resources - October 28, 2011

NSR Cut-off	Category	Zone	Kt	Cu %	Zn %	Ag (g/t)	Au (g/t)	NSR (US$)
$50	Indicated	Lens 2 Massive	4,760	0.27	7.26	23		75.33
		Upper West	1,340	2.64	4.77	42		79.25
		Lens 3	410	1.32	4.92	13		64.83
		Total	**6,510**	**0.82**	**6.60**	**26**		**75.48**
	Inferred	Lens 2 Massive	3,700	0.35	6.63	27		70.00
		Upper West	2,200	1.67	4.63	21		66.20
		Lens 3	100	0.39	6.47	29		69.00
		Total	**6,000**	**0.83**	**5.89**	**25**		**68.59**
CuEq Cut-off	**Category**	**Zone**	**Kt**	**Cu %**	**Zn %**	**Ag (g/t)**	**Au (g/t)**	**CuEq (%)**
1.10%	Indicated	Cu Stockwork	5,560	1.55	0.27	11	0.53	1.91
	Inferred	Cu Stockwork	3,570	1.48	0.43	10	0.35	1.81

Source: RPA Inc., Rennie, 2011

Notes:

1. CIM definitions were followed for Mineral Resources
2. Mineral Resources are estimated at a cut-off of $50/t for the Lens 2, UW, and Lens 3 zones, and 1.10% CuEq for the CSZ.
3. CuEq grades were calculated as per the description in this report and include provisions for metallurgical recovery.
4. Metal prices used for this update of the CSZ were US$2.75/lb Cu, US$1.00/lb Zn, US$1,300/oz Au, and US$21/oz Ag.
5. High-grade caps were applied in the CSZ as per the text of this report. Caps of 10.0% Cu and 25% Zn were used for the Lens 2, UW, and Lens 3 zones.
6. Specific gravity was interpolated into each block based on measurements taken from core specimens.
7. Totals may not add due to rounding.

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

Foran also completed a helicopter-borne geophysical survey that comprised 1,587.4 line-km of time domain electromagnetic (VTEMplus) and horizontal magnetic gradiometer (mag) over those areas of the property not covered in 2007 (Figure 9.1).

In 2012, Foran completed 3,825 m of diamond drilling in 15 holes. The drilling was directed at near-surface projections of the deposit in order to upgrade the classification and extend the known mineralization. Drilling was dominantly completed utilizing HQ-sized core to provide additional material for future metallurgical test work. Geotechnical and hydrogeological studies were also conducted.

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., of Kamloops, BC. Three composited samples, consisting of 516 kg of drill core, were created for each of three different mineralogical domains: the CSZ, L2MS, and UW-MS. The samples were then used in batch and locked cycle flotation testing, as well as determination of Bond Work Indices.

In late 2012, RPA was engaged to prepare an updated Mineral Resource estimate for the Project, using drill results completed up to that time. The estimate update was completed in March 2013 (Rennie, 2013) and resulted in an increase of 15% in the Indicated tonnage and 18% in the Inferred tonnage. As this increase was not deemed to be material, a new NI 43-101 Technical Report was not triggered. However, the 2013 estimate is now considered to be current and forms the basis of the present study (Section 14, Mineral Resource Estimate).

Coincident with the update of the Mineral Resource estimate, Foran drilled four diamond drill holes totalling 2,243 m on the deposit. These holes have not, as yet, been incorporated in the Mineral Resource estimate. RPA reviewed the results from the 2013 drilling and concludes that the impact of these holes on the Mineral Resource estimate would be negligible. By extension, there should be no impact on the results of the present study by the inclusion of these holes in the estimate. RPA does recommend, however, that the estimate be updated with these holes as soon as is practical.

9.1 Work Done Outside of the Immediate Deposit Area

In addition to the work done on McIlvenna Bay proper, Foran has conducted exploration activity on the surrounding property area to look for additional deposits. Exploration work carried out in 2013 included 98.1 line-km of time-domain electromagnetic surveying (TDEM) on the southeast corner of the Hanson Block claims (Figure 9.1). Borehole electromagnetic surveys (BHEM) were carried out in two holes in the Thunder Zone/Balsam areas as well as two others at McIlvenna Bay. Geophysical surveys conducted in 2014 comprised 17.1 line-km of TDEM along strike to the southeast of McIlvenna Bay, as well as BHEM on one drill hole.

Foran has drilled a number of holes on targets within the property boundary but outside of the immediate McIlvenna Bay area. Figure 9.2 shows the location of these targets, and summarizes the amount of drilling done. In 2012 and 2013, Foran drilled six holes, totalling 2,163 m on five separate targets in the southern portion of the property. Nine holes, totalling 3,211 m were drilled in 2013 on the Balsam/Thunder Zone, located southeast of McIlvenna Bay.

The drilling targeted an electromagnetic anomaly along the same geological trend that hosts the deposit. Massive sulphide mineralization was intersected by drilling and 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. In 2014, Foran drilled 1,864 m in two holes on Target A, located just east of McIlvenna Bay (Figure 9.2).

Lithogeochemical sampling has been carried out on drill core from McIlvenna Bay, as well as at Thunder Zone/Balsam areas, and in surface exposures in a broad area surrounding Hanson Lake (see Figure 9.3). The work is being done jointly with the Saskatchewan government as well as a company-sponsored Master's thesis study. To date, a total of 1,406 samples have been taken in this program.

9.2 Exploration Potential

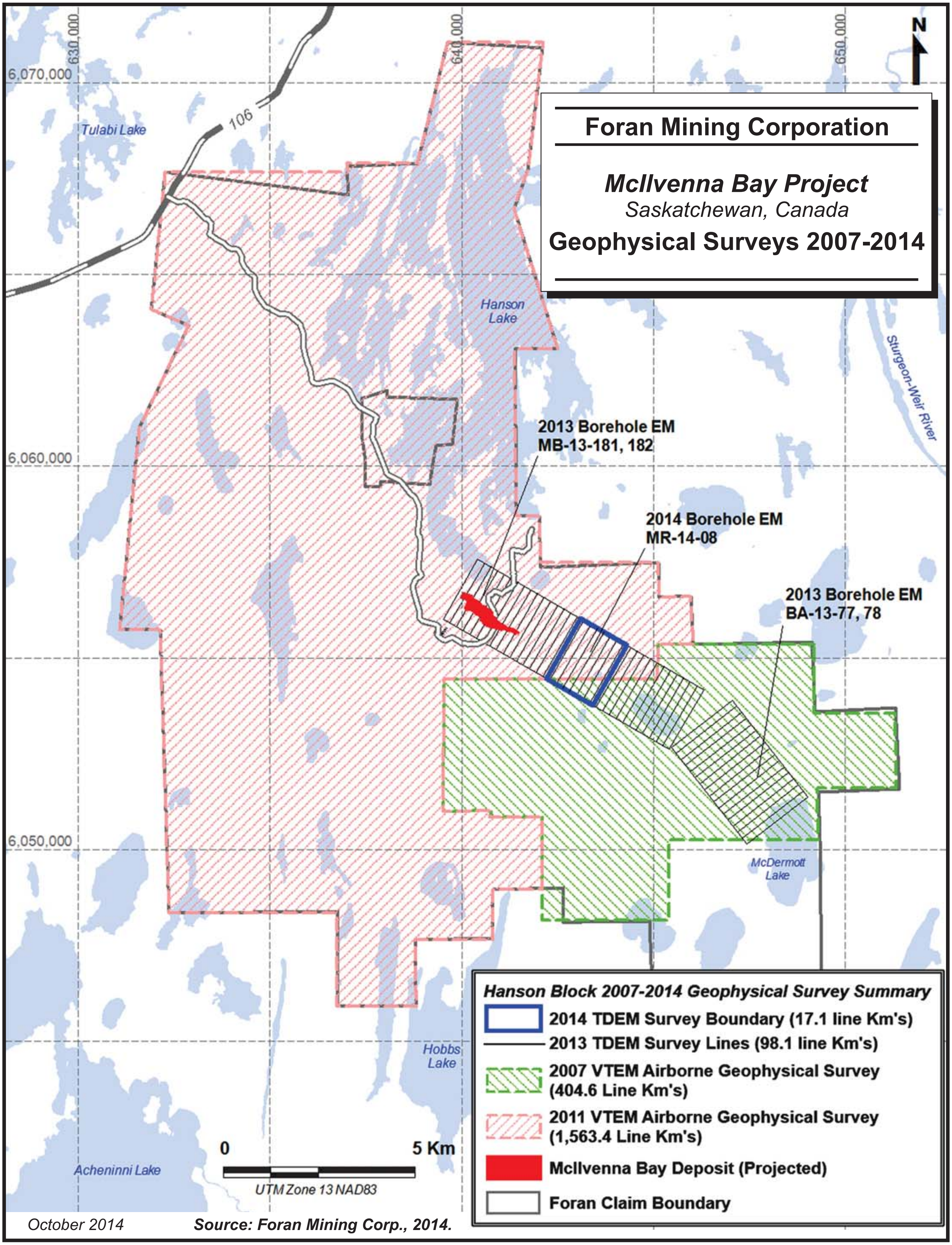
In RPA's opinion, there is significant potential for additional discoveries at McIlvenna Bay. There are opportunities to expand the present Mineral Resources through drilling along strike and down-plunge of the known deposit boundaries.

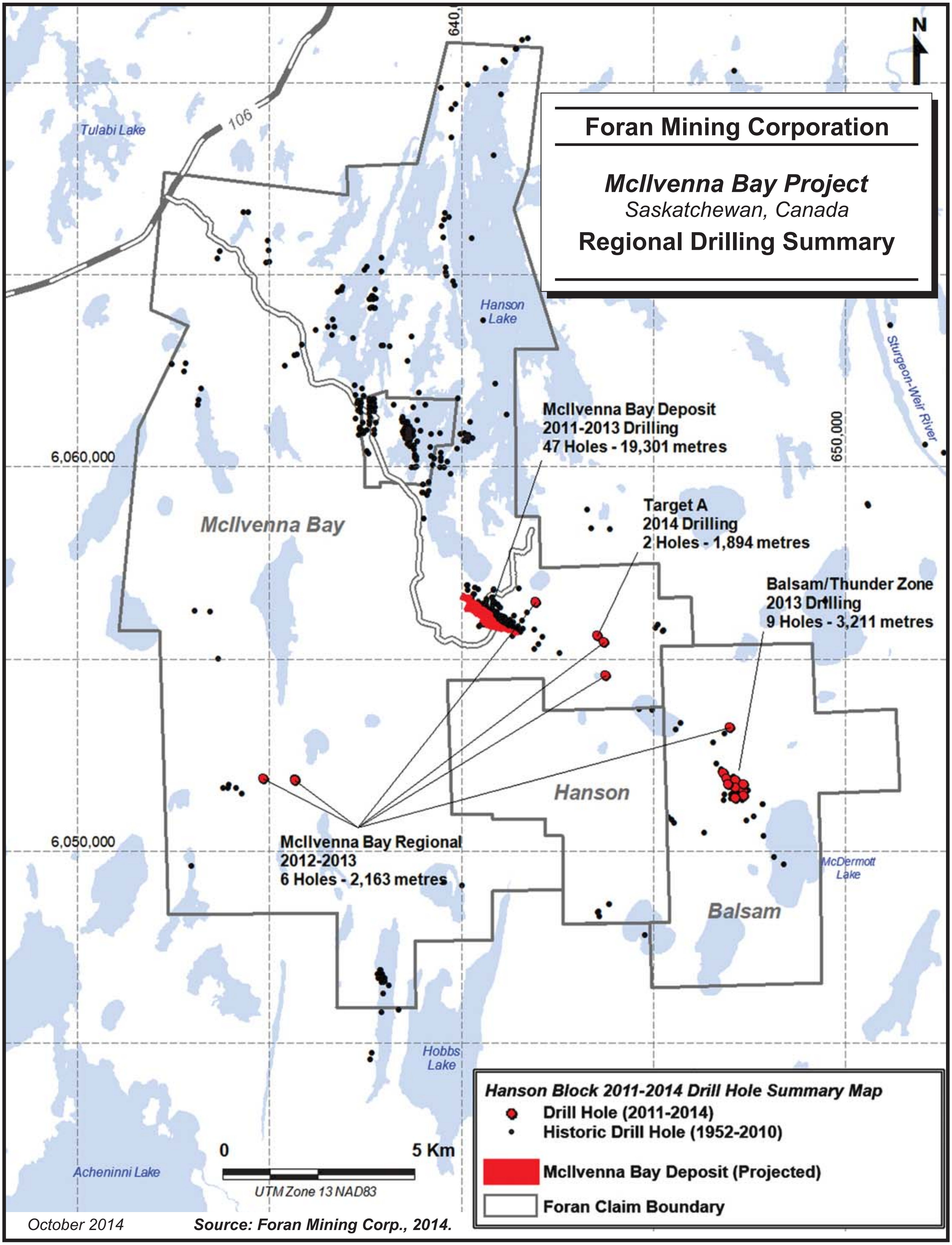
Comparatively high grade copper mineralization has been discovered at the Balsam/Thunder Zone (Figure 9.4). Foran geologists consider the zone to be a high priority target and estimate that it has the potential for high grade copper mineralization. Step-out drilling is required to upgrade this discovery to Mineral Resource status.

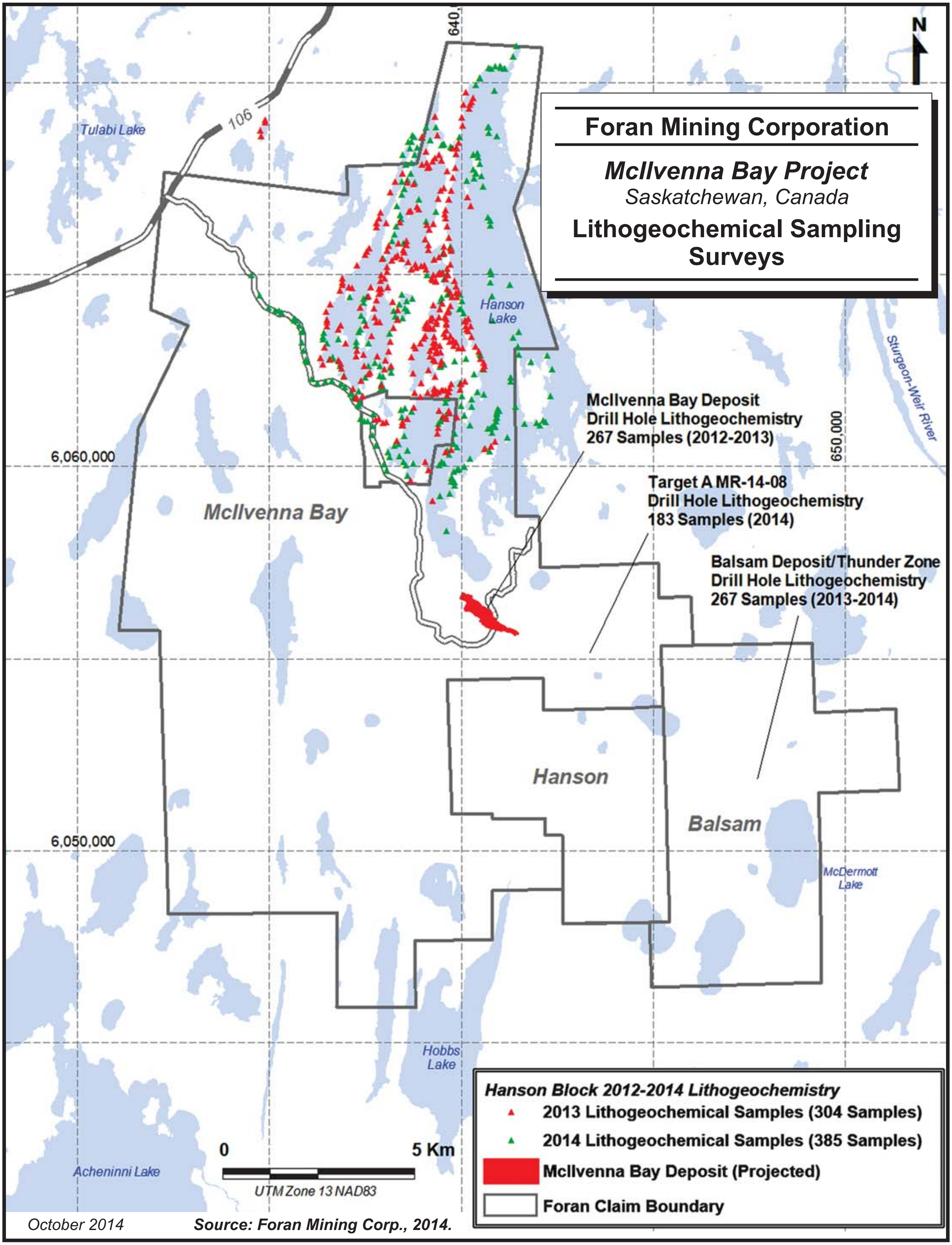
Target A is located two kilometres east and stratigraphically above McIlvenna Bay (Figure 9.4). A BHEM conductor was obtained on this target in hole MR-14-08, at a depth of 1,200 m.

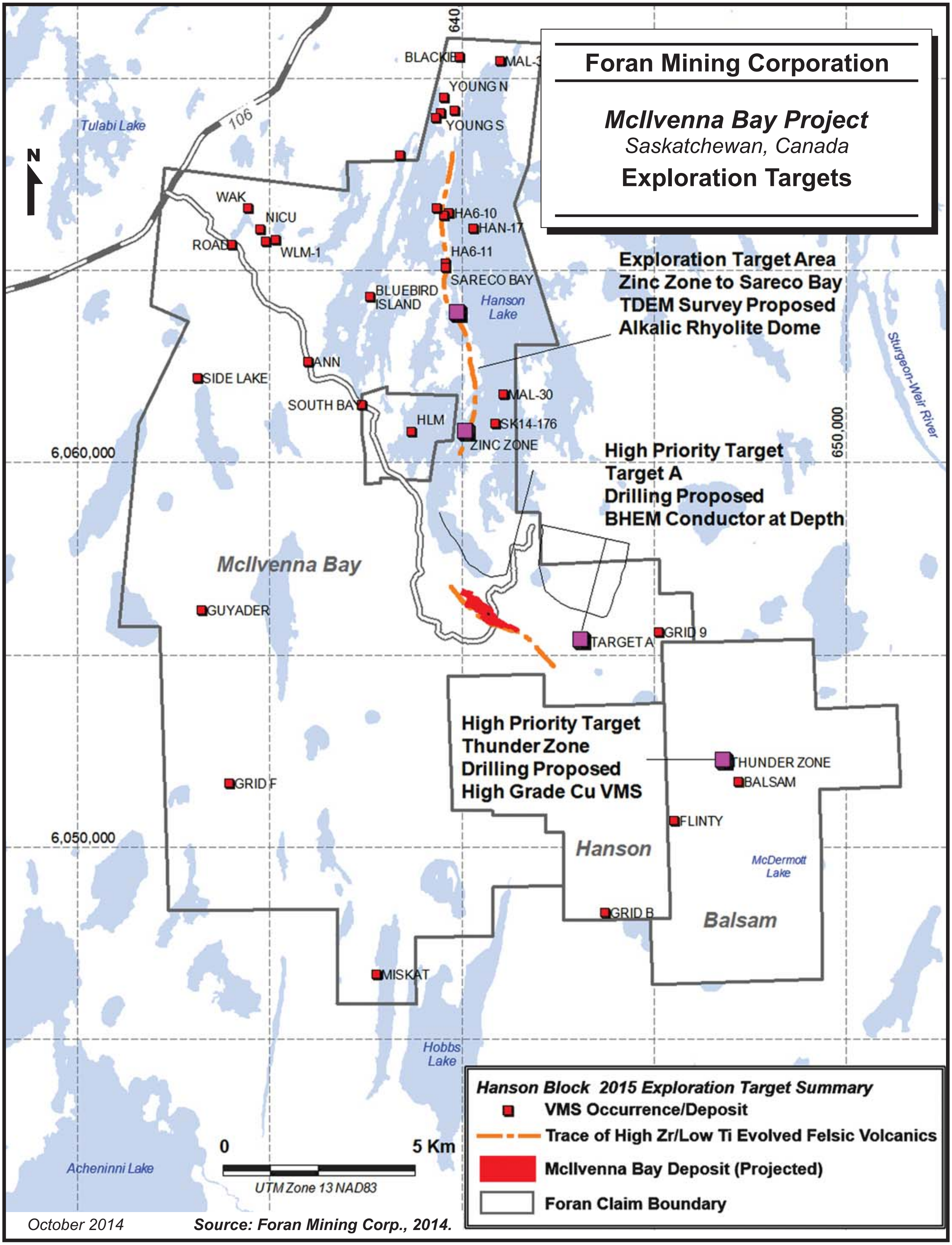
The Zinc Zone is an occurrence of zinc-rich exhalite mineralization hosted within favourable stratigraphy near a rhyolite dome north of McIlvenna Bay (Figure 9.4). Foran geologists consider this zone prospective for deep drill targets, and propose large loop TDEM to develop these targets.

Foran's present exploration strategy is to focus on the target areas outside of the immediate McIlvenna Bay area in order to discover additional Mineral Resources that could contribute to the overall project economics. At present, however, there are no specific plans to explore these targets, and a program budget has not been prepared.


Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Geophysical Surveys 2007-2014
630,000
640,000
650,000
6,070,000
6,060,000
6,050,000
N
106
Tulabi Lake
Hanson Lake
Sturgeon-Weir River
2013 Borehole EM
MB-13-181, 182
2014 Borehole EM
MR-14-08
2013 Borehole EM
BA-13-77, 78
McDermott Lake
Hobbs Lake
Acheninni Lake
0
5 Km
UTM Zone 13 NAD83
Hanson Block 2007-2014 Geophysical Survey Summary
2014 TDEM Survey Boundary (17.1 line Km's)
2013 TDEM Survey Lines (98.1 line Km's)
2007 VTEM Airborne Geophysical Survey (404.6 Line Km's)
2011 VTEM Airborne Geophysical Survey (1,563.4 Line Km's)
McIlvenna Bay Deposit (Projected)
Foran Claim Boundary
October 2014
Source: Foran Mining Corp., 2014.


Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Regional Drilling Summary
N
Tulabi Lake
106
640,
Hanson Lake
Sturgeon-Weir River
McIlvenna Bay Deposit
2011-2013 Drilling
47 Holes - 19,301 metres
650,000
6,060,000
McIlvenna Bay
Target A
2014 Drilling
2 Holes - 1,894 metres
Balsam/Thunder Zone
2013 Drilling
9 Holes - 3,211 metres
Hanson
6,050,000
McIlvenna Bay Regional
2012-2013
6 Holes - 2,163 metres
McDermott Lake
Balsam
Hobbs Lake
Hanson Block 2011-2014 Drill Hole Summary Map
Drill Hole (2011-2014)
Historic Drill Hole (1952-2010)
McIlvenna Bay Deposit (Projected)
Foran Claim Boundary
0
5 Km
UTM Zone 13 NAD83
Acheninni Lake
October 2014
Source: Foran Mining Corp., 2014.


Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Lithogeochemical Sampling Surveys
N
640,
106
Tulabi Lake
Hanson Lake
Sturgeon-Weir River
McIlvenna Bay Deposit
Drill Hole Lithogeochemistry
267 Samples (2012-2013)
650,000
6,060,000
McIlvenna Bay
Target A MR-14-08
Drill Hole Lithogeochemistry
183 Samples (2014)
Balsam Deposit/Thunder Zone
Drill Hole Lithogeochemistry
267 Samples (2013-2014)
Hanson
Balsam
6,050,000
McDermott Lake
Hobbs Lake
Hanson Block 2012-2014 Lithogeochemistry
2013 Lithogeochemical Samples (304 Samples)
2014 Lithogeochemical Samples (385 Samples)
McIlvenna Bay Deposit (Projected)
Foran Claim Boundary
0
5 Km
UTM Zone 13 NAD83
Acheninni Lake
October 2014
Source: Foran Mining Corp., 2014.



Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Exploration Targets
Exploration Target Area
Zinc Zone to Sareco Bay
TDEM Survey Proposed
Alkalic Rhyolite Dome
High Priority Target
Target A
Drilling Proposed
BHEM Conductor at Depth
High Priority Target
Thunder Zone
Drilling Proposed
High Grade Cu VMS
Tulabi Lake
106
640
650,000
6,060,000
6,050,000
BLACKIE
MAL-3
YOUNG N
YOUNG S
WAK
NICU
ROAD
WLM-1
HA6-10
HAN-17
HA6-11
SARECO BAY
BLUEBIRD ISLAND
Hanson Lake
ANN
SIDE LAKE
SOUTH BAY
HLM
MAL-30
SK14-176
ZINC ZONE
Sturgeon-Weir River
McIlvenna Bay
GUYADER
TARGET A
GRID 9
GRID F
THUNDER ZONE
BALSAM
FLINTY
Hanson
McDermott Lake
GRID B
Balsam
MISKAT
Hobbs Lake
Acheninni Lake
0
5 Km
UTM Zone 13 NAD83
Hanson Block 2015 Exploration Target Summary
VMS Occurrence/Deposit
Trace of High Zr/Low Ti Evolved Felsic Volcanics
McIlvenna Bay Deposit (Projected)
Foran Claim Boundary
October 2014
Source: Foran Mining Corp., 2014.

10 DRILLING

Diamond drilling has spanned a fairly broad period, starting with Cameco in 1988. Cameco (and partners) drilled 68 holes, of which 56 targeted McIlvenna Bay. All other drilling in and around the project area has been completed by Foran. A summary of drilling within McIlvenna Bay is provided in Table 10.1.

Table 10.1: Diamond Drilling Summary to August 2014

Company	Year	Number of Holes	Meters Drilled
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**

Source: Foran 2011

RPA notes that the totals provided by Foran for the Cameco-era drilling do not match what is in the database. The database contains 68 of these holes totalling 30,905.6 m of drilling versus 69 holes and 29,946.2 m of drilling as listed in Table 10.1. The apparent discrepancies are 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 really possible to reconcile the drilled totals. The metres from the lengthened holes are contained within the database as though they were drilled by Cameco but they should have been recorded as drilled by Foran. For some of the abandoned and lengthened holes, the records are not complete. Consequently, it is not possible to fully reconcile what is in the database, which is supported by logs, and what is reported. In some instances, Foran has re-logged older drill core to update the records.

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

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

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

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

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

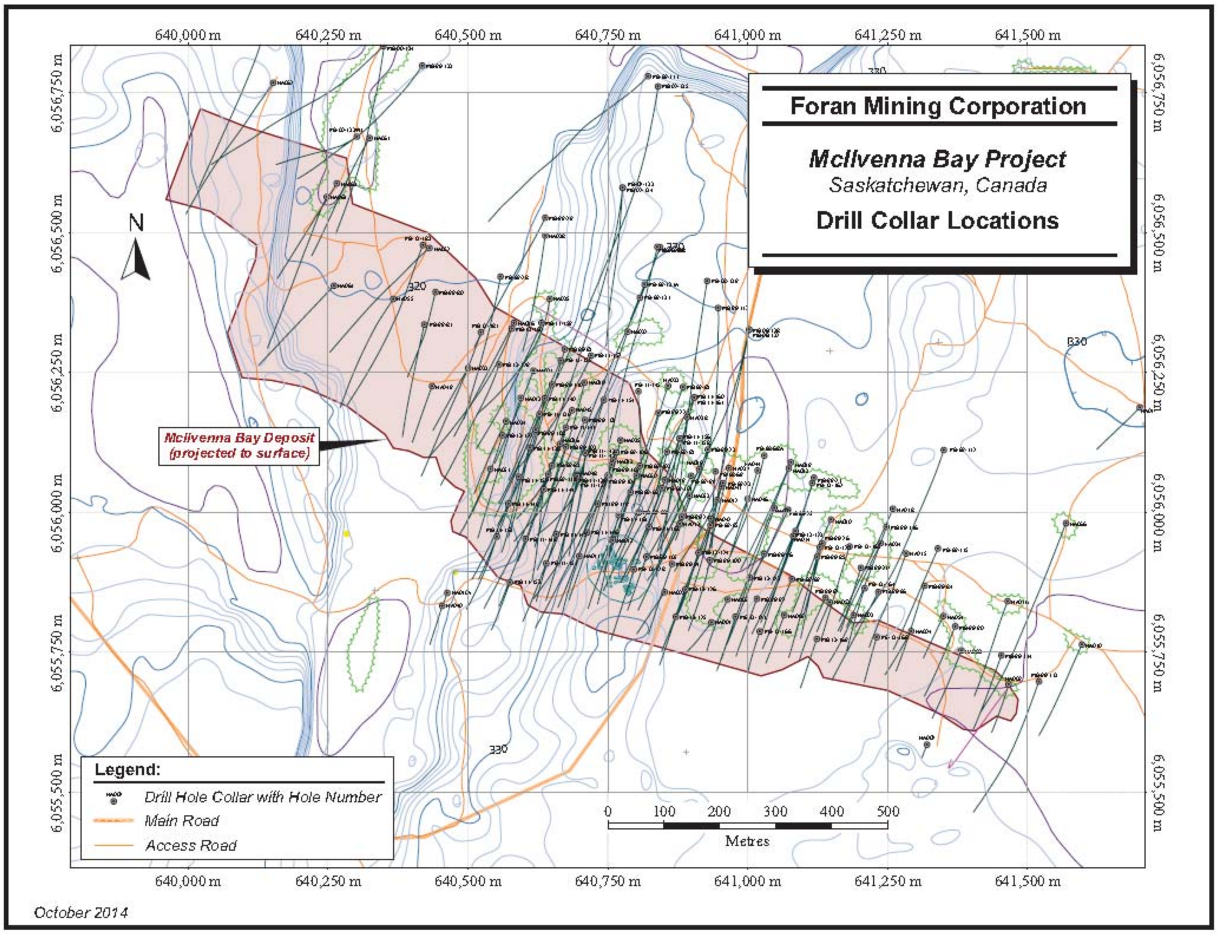
Initially, downhole surveying on the Foran holes was done using a combination of Tropari measurements and acid tests. Due to the presence of magnetic rocks in the stratigraphy, especially the iron formations, Tropari azimuths were sometimes inaccurate and were occasionally ignored in order to get reasonably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

Subsequently, the holes were surveyed initially with a Reflex EZ Trac instrument, which is another instrument based on magnetics. Holes MB-11-140 to -145, inclusive, were re-surveyed using a Gyro tool from Reflex Instruments, which is not affected by magnetics. There were significant differences found between the results for the two instruments. In 2011, re-surveys were conducted of many of the older drill collars, and where the casing could be found, downhole surveys were redone using the Gyro (or similar) 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 is provided in Figure 10.1 In RPA's opinion, the drilling and surveying conducted on the property has been done to industry standards and there are no apparent issues that would have a significant deleterious impact on the estimation of Mineral Resources.

Figure 10.1: Drill Collar Locations



11 SAMPLE PREPARATION, ANALYSES AND SECURITY

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

11.1 Cameco (1988-1991)

Little information is available for security measures employed, quality control and quality assurance (QA/QC) procedures, and who actually prepared the samples. The samples of sawn core were initially sent to TSL Assayers in Saskatoon (TSL). 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 gpt Au, the sample was re-analyzed. 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 in Creighton, Saskatchewan (Eco-Tech). A total of 152 check assays were performed at TSL, Bondar-Clegg (Ottawa), and Terramin Laboratories (Calgary). Cameco was pleased with the Eco-Tech results and believed that TSL returned somewhat lower values for zinc and, to a lesser extent, copper during check assays (MRDI, 1998).

11.2 Foran (1998-2000)

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

At TSL, each sample was crushed to a minimum of 60% passing –10 mesh and then split, with the rejects being stored at TSL. A split portion, approximately 250 g, was pulverized to 90% passing –150 mesh. All samples were analyzed for copper, zinc, lead, gold, and silver, while samples from holes MB-99-78 through 125 were also analyzed for iron and sulphur. All samples were also analyzed by a 31-element ICAP scan that was completed at the TSL laboratory in Vancouver, British Columbia. 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 in Saskatoon (SRC) for check assaying. In the case of a discrepancy between the original and check assay results, the sample was rechecked by XRAL Laboratories in Toronto to determine the most accurate result. In their signed assay reports, TSL included the analytical results of all internal repeat samples (duplicates) and TSL in-house or Certified Reference Material standard samples inserted into the assaying sequence. Foran's experience was that for most elements, TSL assayed very slightly lower (<10% difference) than the corresponding assay done at the SRC.

Generally, zinc, lead and silver assays were less than 10% lower at TSL than at SRC, copper assays were less than 5% lower, and gold results were comparable (Lemaitre, 2000).

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

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

11.2.1 Specific Gravity Determinations

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

11.3 Foran (2007-2008)

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

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

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

11.4 Foran (2011-2013)

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

Up until the latter part of the 2011 program, holes were logged in a dedicated facility established in an old office building. At the time of the last RPA site visit, Foran was in the process of moving to a new building constructed specially for core handling. This facility has been fully configured and is presently in use.

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

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

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

Assay QA/QC protocols were introduced 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 quarter-cores (field duplicates), as well as splits from pulps (preparation duplicates). The duplicates were taken at a rate of one in 20 samples; however, they alternated between field and preparation duplicates.

Material for the blanks consisted of locally obtained barren carbonate rock. The standards material comprised eight different commercially prepared reference standards, listed below in Table 11.1.

The samples were analyzed at TSL by multi-element ICP and AA following four-acid digestion, as described above. Over-limits were assayed by fire assay with both AA and gravimetric finish. 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 Program

Standard	Au (ppb)		Ag (ppm)		Cu (%)		Pb (%)		Zn (%)	
	Mean	SD	Mean	SD	Mean	SD	Mean	SD	Mean	SD
GBM909-11			25.5	1.7	0.5344	0.0195	0.2074	0.0103	1.9486	0.0591
GBM909-12			51.7	3	1.083	0.0339	0.4191	0.0141	4.0073	0.1348
GBM909-13			127.3	6.8	3.2093	0.1295	0.8513	0.0327	6.8362	0.2363
G310-4	430	30								
CDN-ME-11	1,380	100	79.3	6	2.44	0.11	0.86	0.1	0.96	0.06
CDN-ME-17			38.2	3.1	1.36	0.1	0.676	0.054	7.34	0.37
GLG307-1	2.86	1.7								
CDN-GS-P7B	710	70	13.4	1.6						

Source: RPA, Rennie, 2011

Notes:

1) 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. Common error limits used within the industry are two or more consecutive determinations outside of ±2 SD or a single determination outside of ±3 SD.

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. Standards failures were reportedly obtained during the 2011 program which resulted in re-assay of partial batches (20 samples ahead and behind in the sample stream).

In three cases, the failure was determined to have resulted from improper labelling of the standards packets. Two standards failures were obtained in 2012, which resulted in the re-run of the affected batches. One batch from the 2013 winter program was re-run owing to a standards failure.

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

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

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

11.4.1 Specific Gravity Determinations

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

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

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

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

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

An additional 184 bulk density determinations, using the water immersion method, were carried out. In RPA's opinion, Foran's present logging, sampling, and assaying protocols are consistent with good industry practice.

12 DATA VERIFICATION

On September 28, 2006, R. Barry Cook selected and marked out ten samples of sawn core for duplicate analysis. The specified intervals were quarter split by a technician under supervision by Mr. Cook, who then bagged, tagged, and sealed the samples in plastic bags. The bags of samples were packed in a box and shipped by courier to the RPA offices in Toronto. From there they were forwarded by courier to the SGS laboratory in Don Mills, Ontario. Table 12.1 indicates the relevant sample information and assay results. RPA concluded that the duplicate sampling compared reasonably well with the original assay results.

RPA also compared analyses as quoted on original assay certificates to the numbers listed in drill logs for specific assay intervals. The assays for copper, zinc, lead, gold, and silver for 162 samples from six different drill holes were checked without locating any serious errors in transcription. The few discrepancies noted were in fact only differences in the second decimal place.

RPA revisited the property in September 2011. The core storage, sampling and logging facilities were inspected along with representative sections of drill core. The assay results in the database were compared to the original certificates, and no errors were found.

During 2012, Foran personnel carried out a thorough verification of the database, going back to the original assay certificates. Typographical errors were found and corrected. Inconsistencies with regard to units, rounding, denoting of detection limits, and recording of duplicate or triplicate assays were resolved and standardized. As described in the Drilling section of this report, several of the holes were re-surveyed and their projected traces modified.

The revised and fully validated database was provided in its entirety to RPA. RPA imported the data and checked it by running the GEMS validation utility. No significant errors were found. RPA also compared the database against the original assay certificates for 10% of the samples collected in the 2011-1012 drilling and found no errors.

In RPA's opinion, the assay database is relatively free of errors and suitable for use in estimation of Mineral Resources.

Table 12.1: Assays of Samples collected from Drill holes

			RPA Sampling						Original Foran Results				
Drillhole	**From (m)**	**To (m)**	**Sample Number**	**Sample Description**	**Cu (%)**	**Zn (%)**	**Au (ppb)**	**Ag (g/t)**	**Sample Number**	**Cu (%)**	**Zn (%)**	**Ag (g/t)**	**Au (g/t)**
MB-99-101	266.69	267.62	71251	Quarter split core	2.49	8.61	896	16.8	0805	0.39	7.70	16	0.41
MB-99-101	286.00	287.00	71252	Quarter split core	0.6	0.03	132	2.9	0824	0.61	0.03	2.8	0.17
MB-99-98	245.68	247.47	71253	Quarter split core	5.6	1.03	542	81.4	0741	5.19	0.89	73	0.55
MB-99-98	250.84	251.46	71254	Quarter split core	5.33	0.61	549	34.7	0747	4.55	0.65	29	0.72
MB-99-97	114.20	115.38	71255	Quarter split core	0.3	0.11	69	2.6	0715	0.27	0.05	2	0.03
MB-99-97	151.28	152.00	71256	Quarter split core	1.84	5.08	1,020	82.6	0723	1.45	5.12	74	0.74
MB-99-86	167.00	168.13	71257	Quarter split core	0.2	10.2	83	4.3	0066	0.20	11	4	0.14
MB-99-86	150.53	151.92	71258	Quarter split core	0.26	4.82	105	4	0059	0.23	4.09	3.2	0.1
MB-99-78	753.00	754.00	71259	Quarter split core	0.26	2.01	81	3.4	0227	0.27	1.74	3.0	0.07
MB-99-78	746.31	747.06	71260	Quarter split core	2.43	0.42	1,580	17.8	0220	2.66	0.82	19	1.38

Source: Scott Wilson, RPA (Cook and Moore, 2006)

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Overview

Foran has completed a scoping metallurgical testwork program to evaluate the metallurgical properties for the project and to provide the preliminary process design criteria.

KWM was contracted by Foran in 2011 to define a scoping level metallurgical test program based on historical metallurgical testwork completed by Cominco and on 2011/2012 mineralogical analysis completed by Terra.

ALS Metallurgical (ALS) (formerly G&T) located in Kamloops were contracted to complete the metallurgical testwork. The metallurgical program was identified as KM3125.

13.2 Metallurgical Testwork

13.2.1 Overview

A review of the metallurgical testwork indicates that McIlvenna Bay has three distinct types of mineralization. These have been identified as CSZ, L2MS and UW-MS.

The three zones of mineralization from McIlvenna Bay will be extracted using underground mining methods. The testwork showed that the metallurgical properties of the three zones are very different providing a preliminary indication that the three zones will need to be processed independently in the process facility.

The analysis of the testwork results indicated that the CSZ is copper rich generating a copper concentrate product. The main sulphide minerals in the CSZ composite consisted of pyrite and copper sulphides. With a pyrite to copper sulphide ratio of about 1:1 a favorable flotation response was expected and observed.

The L2MS Composite contained about 50% by mass pyrite. The next dominant sulphide mineral, at about 11%, was sphalerite. The sample also contained about 1% Cu sulphide and 0.5% Galena. The flotation testwork resulted in the generation of a low grade combined Cu/Pb bulk concentrate and a high grade, high recovery Zinc concentrate.

The UW-MS Composite contained about 26% by mass pyrite. The copper sulphide and sphalerite were present in near equal masses of 5.3% and 5.8% respectively. The flotation testwork indicated that it was possible to generate marketable copper concentrate at 24% Cu and marketable zinc concentrate at 54% Zn.

Significantly more work is required to complete the metallurgical interpretation for the underground mine plan and mining schedule. None of the ancillary process testwork (concentrates and tailing thickening, concentrates filtering, etc.) was completed as part of the preliminary metallurgical testwork.

Future testwork programs should continue to optimize the flowsheets for the different types of mineralization and evaluate blending mineralization types to simplify underground mine development and operations. Variability testing should look at determining feed grade vs. recovery relationships and develop metallurgical models for the various types of mineralization. Ancillary process testing should be included in the metallurgical test program to complete the development of the process design criteria for finalizing the process flowsheet and the equipment selection.

13.2.1.1 Historical Testwork

COMINCO Engineering Services Ltd. (CESL)

CESL was contracted by Cameco's Mine Development Division to complete testwork for the Hanson Lake Project. CESL's report was issued in December 1990.

The testwork indicated that the Hanson Lake 'A' Zone sample contained about 12% sphalerite, 1% chalcopyrite and 65% pyrite. It was determined that the chalcopyrite needed to be recovered into a separate Cu concentrate to ensure saleable zinc concentrate.

The testwork showed that a 54% Zn concentrate with about 90% Zn recovery could be generated from the 'A' Zone samples.

Preliminary testing from the high-copper 'C' Zone of the deposit showed that a high grade copper concentrate could be produced at good recovery.

Hazen Research Inc.

Hazen Research Inc. (Hazen) completed and reported the results of small scale grinding tests for the Hanson Lake Project in August 1990.

Three mineralized samples were delivered to Hazen. Sample A represented the mineralization supply for the first five to seven years of operation. Samples B and C represented an additional five years of processing. Sample A was evaluated using the MacPherson 18" SAG mill test procedures and Bond ball mill work index methods. Samples B and C were evaluated using the Bond ball mill work index methods.

Sample A:

- AWi(c) = 11.50;
- RMWi = 11.00; and
- BMWi = 10.30.

Sample B:

- BMWi = 13.90

Sample C

- BMWi = 15.90

J.J. Hubregtse

Hubregtse submitted a report on the Mineralogical and Textural Analyses of the Cu-Zn Mineralization of McIlvenna Bay area in 1990.

The #2 massive sulphide lenses (L2MS) was the subject of the mineralogical evaluation. This lens included a massive sphalerite-pyrite zone and a chalcopyrite stinger zone.

For the massive sulphide zones the mineralogy and modal analysis indicated that a primary grind P80 < 100 microns will be required for liberation of the sphalerite and chalcopyrite from the gangue. A regrind P80 < 25 microns may be required to optimize mineral liberation and recovery. It was also observed that due to the occasional presence of fine-scale (<38 micron) chalcopyrite-sphalerite and pyrite-sphalerite intergrowths some Cu may report to the Zn concentrate and vice versa.

13.2.1.2 Mineralogy

An initial group of drill core samples from McIlvenna Bay were collected for a mineralogical evaluation by Terra Mineralogical Services (Terra). The scope of the Terra work was to carry out a characterization and predictive metallurgy study of a series of stacked sulphide zones occurring at McIlvenna Bay in northeastern Saskatchewan.

The present document reports on the mineralogy assemblages, mineral textures (middling ratings), and mineral chemistry of zone samples collected in drill cores intersecting mineralization throughout three main types referred to as: L2MS, UW-MS, and CSZ.

The McIlvenna Bay samples were identified as mainly coarse - to medium - grained, and mainly form intergrowths of non-opaque gangue and sulphide minerals. For L2MS and UW-MS types, the mineralization can be classified as semi-massive to massive sulphide ore, whereas CSZ is a typical low sulphide stringer material. The non-opaque gangue is mainly comprised of carbonate and micaceous minerals in both L2MS and UW-MS types, and prevalently quartz and minor micas in the CSZ material. Platy micaceous minerals (sericite, hydro-muscovite, talc/ anthophyllite), chlorite, and biotite occurred pervasively throughout the mineralized zones, yet were particularly abundant in specific area. Iron-oxides (mainly magnetite) occurred locally in moderate amounts and minor amounts of gahnite ($ZnAl_2O_4$) were also locally encountered. Gangue sulphides were chiefly pyrite and lesser pyrrhotite. Minor to trace amounts of arsenopyrite were also found in a few samples.

The main economic minerals in the three types of mineralization are sphalerite (main Zinc carrier), chalcopyrite (main Copper carrier), and galena (main Lead carrier). Minor amounts of tarnished chalcopyrite (referred to in the data tables as “blue chalcopyrite”) were commonly observed. These tarnished (bluish) chalcopyrite grains contained high silver concentrations (up to 1 wt% Ag; see probe analyses). Minor to trace amounts of stannite, cassiterite, tetrahedrite, bi-tellurides and bi-selenides were also locally observed. Finally, in a few samples native gold, electrum, and silver sulfosalt grains were identified. Electrum and native gold are predominately intergrown with chalcopyrite. Finally, silver is carried in discrete grains of electrum and rare silver sulfosalts and it is also contained in substantial concentrations in chalcopyrite (~ 0.03 wt% in untarnished chalcopyrite and ~ 0.5 wt% in tarnished chalcopyrite). Based on the mineralogical work it was believed the bulk of gold and silver would likely follow chalcopyrite and report to the copper concentrates.

Overall the bulk of the McIlvenna Bay mineralization is coarse - to medium - grained, yet fine - to very fine - grained textures of silicate gangue (mainly micaceous minerals) with chalcopyrite and sphalerite commonly present in all three zone types, forming complex jagged or blade-like mineral textures. These mineral textures will require a fine grind to achieve a sufficient primary liberation of chalcopyrite and sphalerite from micaceous silicate gangue. This represents one of the major challenges encountered in the McIlvenna Bay mineralization. The severity of this problem will be most pronounced for MS and UW massive sulphide mineralization, and will require a tight control of the removal of platy micaceous minerals, a large portion of which is magnesium-rich.

Fine and complex mineral textures of chalcopyrite and sphalerite with one another, and also with iron sulphide gangue are widespread in mineralization from L2MS and UW-MS mineralization types. Fine to very fine regrind would be necessary to break these bonds and achieve sufficient mineral liberation to avoid severe cross contamination of concentrates (Cu in Zn concentrate and vice versa) and iron-sulphide contamination.

Electron Microprobe analyses have shown that tarnished chalcopyrite (blue chalcopyrite) contains concentrations of silver up to 1 wt%. The average silver content in chalcopyrite from the analyses that have been completed on non-tarnished chalcopyrite grains stands at approximately 0.03 wt%, and at approximately 0.5 wt% for tarnished (”blue”) chalcopyrite grains. Finally, zinc and iron in sphalerite present a bi-modal distribution. One small population is comprised of low iron sphalerite, whereas the bulk of the McIlvenna Bay sphalerite possesses a zinc content of approximately 60% and iron content of approximately 6.5%.

McIlvenna Bay is not drastically different from many other VMS deposits occurring in the Flin Flon belt or from other VMS camps. From the Terra mineralogical work it was projected that the CSZ mineralization will be the least challenging to process while the massive sulphide lenses, particularly the UW-MS mineralization, will be the most challenging. The effective separation and removal of non-opaque gangue (chiefly the micaceous fraction) from the economic minerals, and the effective separation of chalcopyrite from sphalerite (Cu-Zn separation) are the two primary mineralogical challenges that need to be addressed to achieve a successful processing of the McIlvenna Bay mineralization. These challenges are most pronounced in the L2MS and UW-MS mineralization. The mineralogical report also identified that while CSZ mineralization could be successfully processed applying a primary grind of 60 - 65µm (possibly even coarser), processing of the L2MS and UW-MS mineralization would most likely require a substantially finer primary grind (~ 50 µm) and very fine regrind targets (10 - 15µm).

13.2.1.3 Metallurgical Testing 2011, ALS

Metallurgical Samples

Foran provided a list of drill core samples and sample rejects that were available to use in the scoping metallurgical test program. The list provided the length of the intercept interval; assay details for Cu, Pb, Zn, Au and Ag and the estimated weight. The samples were shipped to ALS in Kamloops for compilation into the respective composites for the metallurgical test program. Using the feed grades estimated for the resource at the time of the testwork KWM provided ALS a recipe for assembling each of the main composites.

Composites:

CSZ: Approximately 136 m of drill core sample;
Estimated sample feed grade, 1.44% Cu, 0.23% Zn;
Resource feed grade, 1.52% Cu, 0.33% Zn; and
Composite assay, 1.45% Cu, 0.16% Zn.

L2MS: Approximately 45 m of drill core sample;
Estimated sample feed grade, 0.37% Cu, 6.92% Zn;
Resource feed grade, 0.30% Cu, 6.98% Zn; and
Composite assay, 0.30% Cu, 7.25% Zn

UW-MS: Approximately 31 m of drill core sample;
Estimated sample feed grade, 1.49% Cu, 3.39% Zn;
Resource feed grade, 2.04% Cu, 4.68% Zn; and
Composite assay, 1.61% Cu, 3.97% Zn

Bond Work Index Tests (Grinding)

Bond rod mill and Bond ball mill work index tests were completed for each of the three composites. These results have been presented in Table 13.1.

Table 13.1: Bond Work Index Tests

Composite	Bond Mall Mill Grindability Test		Bond Rod Mill Grindability Test	
	Work Index kWh/tonne	µmP_{80}	Work Index kWh/tonne	µmP_{80}
CSZ	16.1	80	17	882
MS	11.6	83	12.7	869
UW-MS	14	81	15.6	852

Source: JDS 2014

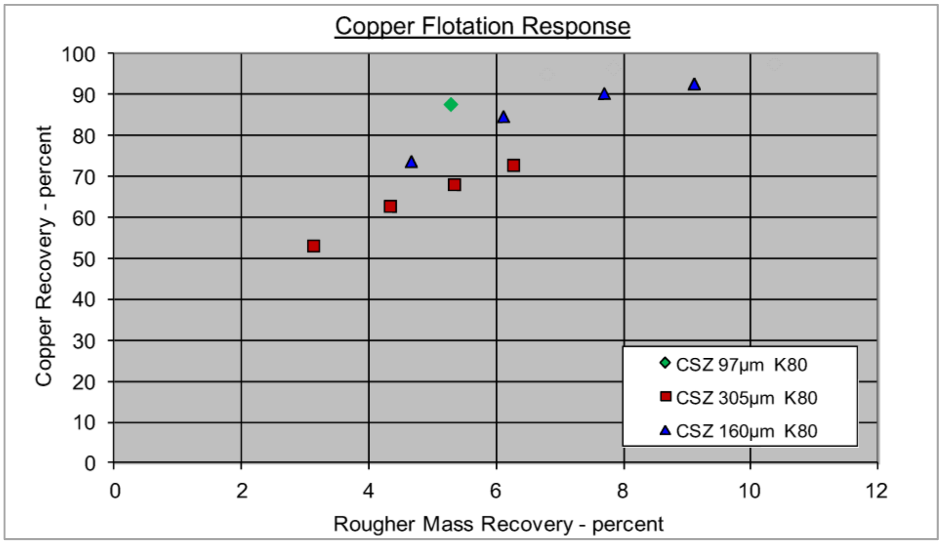
These results are consistent with the historical testwork and identified that the CSZ (copper stockwork) has the highest work index.

Flotation Test Results

CSZ

Three rougher flotation tests were completed on the CSZ Composite. Given the expectation that conditions for the depression of zinc would be required for the L2MS and UW-MS composites a flowsheet at natural pH incorporating zinc sulphide and sodium cyanide was used and a thiophosphinate collector was used. Due to the excellent recoveries at the nominal primary grind size P80, 100 microns coarser primary grind sizes of 160 and 305 microns were tested but the results deteriorated Figure 13.1.

Figure 13.1: CSZ Rougher Flotation Tests



Source: JDS 2014

A single locked cycle test was completed for the CSZ composite. The key observations were:

- About 94% of the copper was recovered into a copper concentrate grading 29% Cu. (Table 13.2);
- Silver and gold recoveries were also high at 77% and 85% respectively; and
- Rougher concentrate was reground to 35 microns prior to cleaner flotation.

Table 13.2: CSZ Locked Cycle Test Results

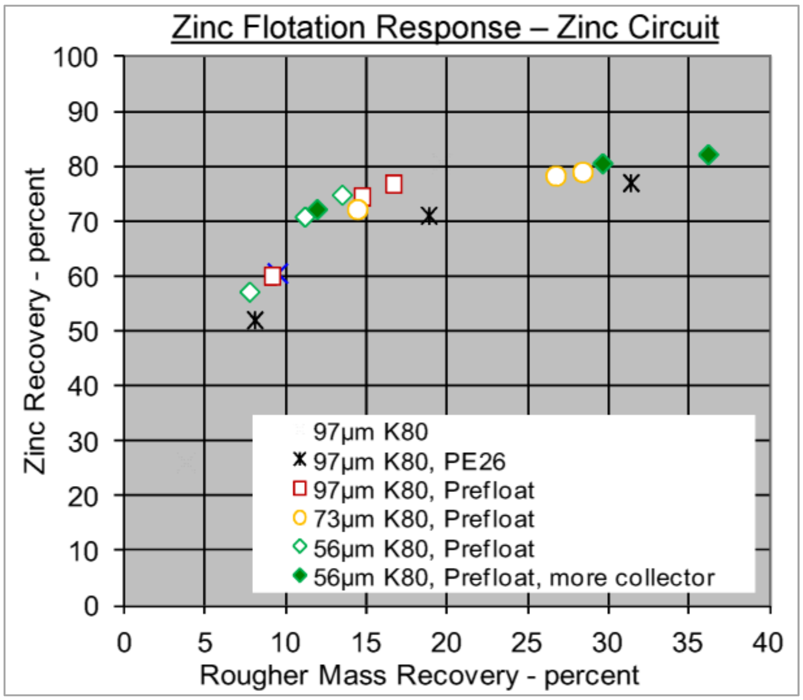
Product	Mass%	Assay - Percent or g/t						Distribution - percent					
		Cu	Zn	Fe	S	Ag	Au	Cu	Zn	Fe	S	Ag	Au
Feed	100	1.57	0.16	7.2	4.49	8	0.38	100	100	100	100	100	100
Copper Con	5.1	29.2	1.05	30.3	33.5	126	6.38	33.8	33.8	21.4	38	76.9	84.6
Copper 1st Clnr Tail	3.9	0.79	0.57	21.7	19.3	14	0.81	14.1	14.1	11.8	16.8	6.8	8.3
Copper Ro Tail	91	0.06	0.09	5.3	2.23	2	0.03	52.1	52.1	66.9	45.2	16.4	7.1

Source: JDS 2014

L2MS

A total of six kinetic rougher tests were completed on the L2MS Composite with primary grind sizes, P80, varied between 50 and 100 microns. Zn sulphate and sodium cyanide were used in the bulk circuit to depress pyrite and sphalerite. Copper sulphate was used in the zinc circuit to activate the zinc. The preflotation circuit was to remove talc from the primary flotation circuits. The bulk flotation circuit was to recover lead and copper and prevent contamination of the zinc concentrate in order to generate a saleable concentrate. Figure 13.2 illustrates the zinc rougher recovery at the various grind sizes and flotation conditions.

Figure 13.2: L2MS Zn Rougher Flotation



Source: JDS 2014

A total of three batch cleaner tests were completed on the L2MS Composite. These tests investigated the effects of zinc rougher concentrate regrinding on metallurgical performance in order to select the test parameters for the locked cycle tests. The batch cleaner tests indicated that a finer regrind size of 13 microns was detrimental to zinc flotation performance. A closed circuit cleaner test at a primary grind size of 73 microns and a regrind size of 20 microns resulted in zinc recoveries of 75% into a concentrate grading 56% Zn. About 2.6% of the zinc was lost to the pre-float and 11% was lost to the rougher tailings.

Two locked cycle tests were completed on the L2MS Composite. The results have been illustrated in Table 13.3.

Table 13.3: L2MS Locked Cycle Flotation Tests

Product	Mass%	Assay - % or gpt									Distribution - %								
		Cu	Pb	Zn	Fe	S	Ag	Au	Mg	C	Cu	Pb	Zn	Fe	S	Ag	Au	Mg	C
Test 17 - Cycles IV and V																			
Bulk Feed	100	0.31	0.37	7.08	27.7	30.0	18	0.16	3.30	2.19	100	100	100	100	100	100	100	100	100
Prefloat	3.5	0.21	0.36	5.81	11.1	13.1	12	0.06	9.81	1.78	2.3	3.4	2.8	1.4	1.5	2.4	1.3	10.3	2.8
Bulk Concentrate	1.6	11.2	13.8	9.78	22.5	32.0	359	4.94	1.21	0.20	55.3	57.9	2.1	1.3	1.7	31.9	47.9	0.6	0.1
Zn Concentrate	10.3	0.77	0.51	53.8	8.7	33.0	45	0.23	0.33	0.19	25.2	14.1	78.4	3.2	11.3	26.4	15.0	1.0	0.9
Zn 1st Cleaner Tail	9.7	0.24	0.24	3.23	29.1	28.0	17	0.13	3.57	2.61	7.5	6.4	4.4	10.1	9.0	9.4	7.8	10.5	11.5
Zn Rougher Tail	75.0	0.04	0.09	1.16	31.0	30.6	7	0.06	3.41	2.47	9.7	18.2	12.2	84.0	76.5	30.0	28	77.7	84.6
Test 20 - Cycles IV and V																			
Bulk Feed	100	0.33	0.41	6.96	26.7	28.4	15	0.21	3.78	2.27	100	100	100	100	100	100	100	100	100
Prefloat	3.5	0.2	0.32	5.89	11.7	12.8	11	0.11	9.82	1.66	2.1	2.8	3.0	1.6	1.6	2.6	1.9	9.2	2.6
Bulk Concentrate	1.6	11.9	15.4	9.18	21.1	28.5	332	5.27	1.45	0.28	56.0	59.1	2.1	1.2	1.6	34.4	38.5	0.6	0.2
Zn Concentrate	10.8	0.63	0.46	55.0	7.8	32.1	38	0.29	0.22	0.17	20.5	12.2	85.4	3.2	12.2	27.3	14.6	0.6	0.8
Zn 1st Cleaner Tail	17.8	0.16	0.15	0.96	38.7	38.6	10	0.15	1.83	1.35	8.4	6.5	2.5	25.7	24.1	11.6	12.7	8.6	10.6
Zn Rougher Tail	66.3	0.07	0.12	0.74	27.5	25.9	5	0.10	4.61	2.93	13.1	19.5	7.0	68.3	60.5	24.1	32.3	81.0	85.8

Source: JDS 2014

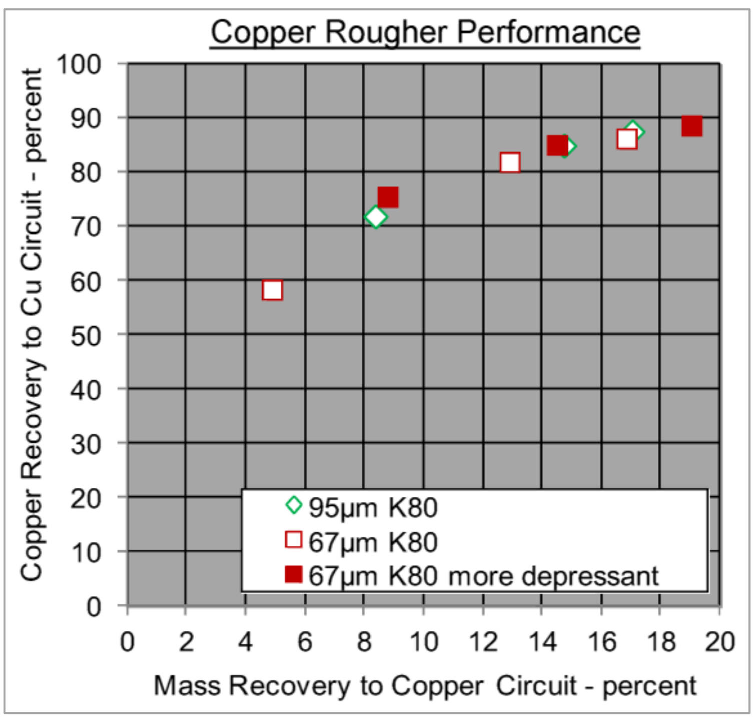
The locked cycle tests indicated that a zinc concentrate ranging from 53% Zn to 55% Zn was achievable with recoveries in excess of 80% to the zinc concentrate. Approximately 55% of the Cu and Pb in the feed were recovered to a bulk concentrate. No test work has been completed to determine if separate marketable concentrates can be generated. The results also showed that 30% of the silver and 40% of gold in the feed would report to the bulk concentrate.

UW-MS

A total of three kinetic rougher tests were completed on the UW-MS Composite with primary grind sizes, P80, varied between 60 and 100 microns. Zn sulphate and sodium cyanide were used in the bulk circuit to depress pyrite and sphalerite. Copper sulphate was used in the zinc circuit to activate the zinc. The preflotation circuit was to remove talc from the primary flotation circuits.

Copper flotation circuit performance was not impacted by a finer grind as shown in Figure 13.4. On average copper from the feed was 87% recovered into a rougher concentrate containing about 18% of the feed mass.

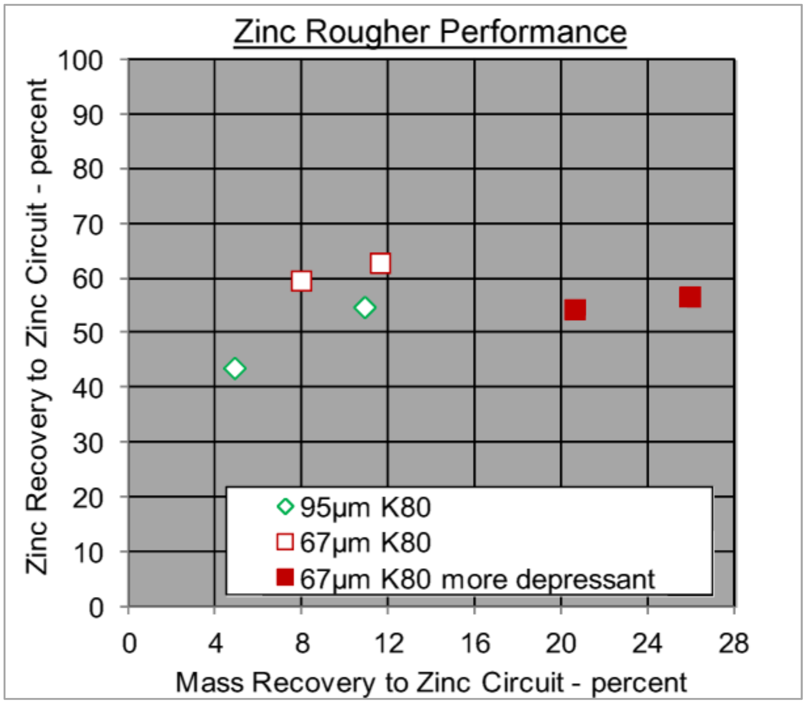
Figure 13.3: UW-MS Copper Rougher Flotation



Source: JDS 2014

Zinc rougher performance for the UW-MS appeared to improve at a finer primary grind size as illustrated in Figure 13.4.

Figure 13.4: UW-MS Zn Rougher Flotation



A total of four batch cleaner tests were completed on the MS-MS Composite. These tests investigated the effects of zinc rougher concentrate regrinding on metallurgical performance in order to select the test parameters for the locked cycle tests. Finer regrinds in the copper flotation circuit generally resulted in higher concentrate grades. Increasing the pH in copper cleaner flotation to 10 also improved the metallurgical performance of the copper circuit. Similar to the MS results, a finer regrind in the UW-MS cleaner flotation was detrimental to zinc flotation performance

A single locked cycle test was completed on the UW-MS Composite. The results have been illustrated in Table 13.4.

Table 13.4: UW-MS Locked Cycle Flotation Tests

Product	Mass%	Assay - % or gpt									Distribution - %								
Cycles IV and V		Cu	Pb	Zn	Fe	S	Ag	Au	Mg	C	Cu	Pb	Zn	Fe	S	Ag	Au	Mg	C
Bulk Feed	100	1.75	0.18	4.02	17.7	17.3	26	0.66	6.09	1.05	100	100	100	100	100	100	100	100	100
Prefloat	2.8	0.86	0.13	1.44	6.6	4.0	23	1.16	12.2	0.40	1.4	2.1	1.0	1.0	0.6	2.5	4.9	5.5	1.0
Copper Concentrate	6.0	24.2	1.3	6.40	27.2	34.4	216	6.50	0.46	0.12	83.4	43.4	9.6	9.3	12.0	50.3	59.7	0.5	0.7
Zn Concentrate	5.6	1.87	0.24	54.3	8.0	32.5	63	0.81	0.25	0.10	6.0	7.5	76.3	2.5	10.6	13.6	6.9	0.2	0.5
Zn 1st Cleaner Tail	7.7	0.69	0.14	2.39	24.7	21.4	22	0.53	5.74	1.06	3.0	6.0	4.6	10.8	9.5	6.6	6.2	7.3	7.8
Zn Rougher Tail	77.8	0.14	0.10	0.44	17.4	15.0	9	0.19	6.77	1.21	6.2	41.0	8.5	76.4	67.2	27.0	22.2	86.5	90.0

Source: JDS 2014

The locked cycle test results indicated that approximately 83% of the feed copper was recovered to a final copper concentrate grading 24% Cu. About 10% of the zinc in the feed was recovered to the copper concentrate. Approximately 76% of the zinc in the feed was recovered in the zinc flotation circuit to a concentrate grading 54% Zn. The Au and Ag recoveries to the copper concentrate were 60% and 50% respectively.

13.3 Metallurgical Projections

There has been insufficient testwork completed to generate metallurgical recovery models for the various types of mineralization. There is sufficient information available from the testwork to complete preliminary design criteria and subsequently generate a scoping study assessment of the mill design.

A number of key design parameters have been identified in the test program. From the testwork it was determined that each type of mineralization has different grinding parameters including work index, primary grind size and rougher regrind size. The flotation circuit design is also specific for reagents and flotation time for each type of mineralization. A single flow sheet should be designed that can be used for all of the types of mineralization. Not all of the components will be used for all types of mineralization and the plant layout will need to consider the various operating conditions. (Example: the grinding circuit designed for one type of mineralization will result in different operating conditions [mill feed rate] than for another based on work index and product grind.)

The test work indicated that saleable concentrates can be generated for the three main mineralization types with the following operating results.

- CSZ: Copper concentrate grading 29% Cu with 94% Cu recoveries to the copper concentrate;
- L2MS: Bulk Concentrate grading 12% Cu/14% Pb, with recoveries of 56% Cu, 58% Pb, 33% Ag, 42% Au to the bulk concentrate;

Zinc Concentrate grading 54% Zn with 81% Zn recoveries to the zinc concentrate;

- UW-MS: Copper Concentrate grading 24% Cu with recoveries of 83% Cu, 50% Ag, 60% Au to the bulk concentrate; and
- Zinc Concentrate grading 54% Zn with a recovery of 76% Zn to the zinc concentrate.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

As discussed in the section of this report entitled Exploration, RPA prepared the most recent Mineral Resource estimates for McIlvenna Bay. The most current NI 43-101 Technical Report for Mineral Resources on the Project was issued in December 2011 (Rennie, 2011). Foran retained RPA in 2013 to carry out an update of the Mineral Resource estimate, and as the difference from the 2011 estimate was not deemed to be material, no Technical Report was triggered. The 2011 Mineral Resource estimate is summarized in Table 9.1 in Section 9 of this report, and the current, January 2013, estimate is summarized in Table 14.1.

The 2013 estimate was carried out using a block model constrained by 3D wireframes of the mineralized zones. Values for Cu, Zn, Au, Ag, and Pb were interpolated into the blocks using Ordinary Kriging. The models were constructed using GEMS (Gemcom) software, which is an off-the-shelf commercial package commonly used within the industry.

The work was conducted by David Rennie, P. Eng., Principal Geologist for RPA. Mr. Rennie is a geological engineer with 35 years of experience in mining and mineral exploration, most of which has been spent on evaluation and estimation of Mineral Resources and Mineral Reserves. Both RPA and Mr. Rennie are independent of Foran as defined by NI 43-101.

Table 14.1: McIlvenna Bay Mineral Resources - January 2013

Zone	Tonnes (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	CuEq (%)	ZnEq (%)	Cu (Mlb)	Zn (Mlb)	Au (oz)	Ag (oz)
Indicated												
L2MS	3,390	0.31	7.15	0.24	23.7	81.55	1.51	10.19	23	534	25,700	2,580,000
UW-MS	2,150	1.66	4.1	0.88	30.7	150	2.79	18.75	78.7	194	61,000	2,120,000
L3	760	1.23	2.55	0.3	14.5	96.25	1.79	12.03	20.5	42.4	7,310	353,000
CSZ	7,610	1.6	0.28	0.51	10.6	104.56	1.94	13.07	269	46.5	126,000	2,600,000
Total	**13,900**	**1.28**	**2.67**	**0.49**	**17.1**	**105.57**	**1.96**	**13.2**	**391**	**817**	**220,000**	**7,650,000**
Inferred												
L2MS	2,800	0.5	7.13	0.38	26.1	96.29	1.79	12.04	31.1	439	33,800	2,350,000
UW-MS	2,910	1.62	3.68	0.51	19	132.95	2.47	16.62	104.3	236	47,800	1,780,000
L3	124	1.61	2.67	0.51	17.7	124.13	2.31	15.52	4.39	7.3	2,040	70,300
CSZ	5,480	1.56	0.47	0.42	12.1	100.76	1.87	12.59	188	56.9	73,100	2,140,000
Total	**11,300**	**1.32**	**2.97**	**0.43**	**17.5**	**108.32**	**2.01**	**13.54**	**328**	**740**	**157,000**	**6,340,000**

Source: RPA 2014

Notes:

1. CIM definitions were followed for Mineral Resources
2. Mineral Resources are estimated at a cut-off of US$60/t.
3. NSR values, as well as CuEq and ZnEq grades, were calculated as per the description in this report and include provisions for metallurgical recovery.
4. Metal prices used for this estimate were US$3.25/lb Cu, US$1.10/lb Zn, US$1,400/oz Au, and US$25/oz Ag.
5. High-grade caps were applied as per the text of this report.
6. Specific gravity was estimated for each block based on measurements taken from core specimens.
7. CSZ includes the Copper Stockwork Footwall Zone.

14.2 Database

The database comprised diamond drilling results collected over the entire history of the project. The database contained records for 178 diamond drill holes, with a total of 6,220 assay intervals. Of these assay intervals, 2,833 were eventually captured within the wireframe models used to constrain the estimate.

Surveys for the project work are based on the local UTM grid. This is a change from the earlier 2011 database, which was based on a local property grid. The change-over was carried out by Foran personnel, during the 2012 database review work.

14.3 Wireframe Models

As discussed in the Mineralization section of this report, the principal zones interpreted to date include the Lens 2 Massive Sulphide (L2MS), Upper West Zone (UW-MS), Lens 3 Massive Sulphide (L3), the Copper Stockwork Zone, and the Copper Stockwork Footwall Zone (both included as CSZ). Wireframe models for each of the mineralized zones were constructed from the Foran geological interpretations on the 50 m cross sections. A nominal $45/t NSR cut-off value was used along with a minimum horizontal width constraint of two metres. Due to the dip of the zones, the two metre horizontal width resulted in a minimum true width of approximately 1.6 m.

The interpreted digital outlines of the zones were drawn on cross-section and then adjusted on level plan views in a series of iterations to produce a coherent and reasonably smooth solid. Profile lines were pinned to the drill holes in 3D in order to remove inaccuracies resulting from holes being off the section planes. The interpreted envelopes were allowed to extend without limit between drill holes, and to a maximum of 50 m beyond the outermost intercepts.

RPA notes that where zones shared a contact with one another, there were often minor overlaps between the wireframes. Every effort was made to remove or reduce the size of these overlaps, and in RPA's opinion, they are not significant. There was no significant effect on the tonnage reported from the model as a result of these overlaps.

The wireframe models for all zones are shown in Figure 7.4.

14.4 Sample Statistics

RPA conducted statistical analyses on the raw sample data contained within the wireframe models. These statistics are summarized in Table 14.2.

Table 14.2: Length-Weighted Sample Statistics

Element	Number	Mean	SD	CV	Median	Maximum	Minimum	Zeroes
Lens 2 Massive Sulphide								
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**	**Zeroes**
Ag	644	24.931	38.661	1.551	15.896	550	0.1	0
Au	641	0.247	0.341	1.381	0.14	4.54	0.003	3
Cu	644	0.269	0.393	1.342	0.16	4.52	0.002	0
Pb	638	0.478	0.686	1.436	0.32	6.53	0.003	6
Zn	644	6.219	4.007	0.644	6.571	56	0.006	0
Upper West								
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**	**Zeroes**
Ag	340	29.795	44.817	1.504	19	1,144.00	0.9	0
Au	340	0.875	1.447	1.654	0.44	14.78	0.003	0
Cu	340	1.747	1.777	1.017	1.308	19.6	0.005	0
Pb	339	0.351	0.841	2.397	0.042	8.1	0.002	1
Zn	340	3.279	4.838	1.475	0.89	26.4	0.01	0
Lens 3								
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**	**Zeroes**
Ag	217	12.749	9.888	0.776	10.394	68.6	0.2	0
Au	217	0.309	0.497	1.61	0.2	5.66	0.003	0
Cu	217	0.966	1.053	1.091	0.67	8.08	0.005	0
Pb	216	0.078	0.163	2.105	0.03	1.77	0.001	1
Zn	217	1.924	2.741	1.425	0.502	13.9	0.02	0
Copper Stockwork								
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**	**Zeroes**
Ag	1,445	9.069	11.547	1.273	5.897	180	0.1	1
Au	1,445	0.422	0.819	1.939	0.19	10.34	0.003	1
Cu	1,445	1.364	1.14	0.836	1.12	12.28	0.005	1
Pb	1388	0.031	0.199	6.345	0.005	5.74	0.001	58
Zn	1,444	0.328	0.878	2.681	0.115	17.19	0.005	2
Copper Stockwork - FW								
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**	**Zeroes**
Ag	186	10.221	9.58	0.937	7.07	61	1.2	0
Au	186	0.509	0.681	1.339	0.31	5.07	0.003	0
Cu	186	1.542	1.207	0.783	1.141	6.07	0.03	0
Pb	186	0.024	0.083	3.425	0.005	0.6	0.001	0
Zn	186	0.496	1.768	3.565	0.09	13.5	0.01	0

Notes:

1. Means are weighted by sample length.
2. Analyses include only non-zero values.

Source: RPA 2014

14.5 Top Cuts

The grade distributions were observed to be moderately to weakly skewed, with occasional high grade outliers. Overestimation of block grades can occur when the data are positively skewed, owing to the disproportionate effect that the highest grade assays exert on the overall mean of the distribution. In order to ameliorate this potential bias, it is common practice to cap high grades, or limit the distance over which the highest grade composites can be extrapolated. For this estimate, top cuts were applied to the samples prior to compositing. The cap values used are shown in Table 14.3.

Table 14.3: Top Cuts

Zone	Code	Ag (g/t)	No. Cut	Au (g/t)	No. Cut	Cu (%)	No. Cut	Pb (%)	No. Cut	Zn (%)	No. Cut
L2MS	20	250	9	1.5	12	2	6	3	15	20	3
UW-MS	21	150	5	7.5	8	7.5	4	3	9	20	6
CSZ	22/23	75	12	5	14	7	8	0.5	17	5	19
L3	30	40	6	1.5	4	5	3	0.5	3	10	4

Source: RPA, 2014

14.6 Composites

Samples were composited to 1.5 m downhole intervals across the zones subtended by the wireframe models. The compositing was configured to start at the point at which a drill hole entered a wireframe and progress in 1.5 m increments to the exit point. Invariably, a short composite was created at the exit point owing to the fact that the distance from hanging wall to footwall was rarely an exact multiple of 1.5 m. Remnant composites with a length of less than 0.75 m were discarded from the database. Composite values were generated for Cu, Zn, Au, Ag, and Pb. Composites whose centroids were contained within a wireframe (i.e., for which the majority of their lengths were within the wireframe) were then tagged with an integer code corresponding to that wireframe. This provided the means for limiting the grade interpolations to those composites contained within the zones.

Table 14.4: Composite Statistics

Lens 2 Massive Sulphide							
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**
Ag	427	24.406	29.158	1.195	16.493	236.798	0.403
Au	427	0.235	0.221	0.942	0.152	1.403	0.011
Cu	427	0.285	0.336	1.181	0.17	2	0.002
Pb	427	0.453	0.475	1.048	0.357	3	0
Zn	427	6.232	3.415	0.548	6.467	16.825	0.006
Upper West							
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**
Ag	214	28.469	26.848	0.943	19.556	150.001	2.2
Au	214	0.861	1.181	1.372	0.496	7.5	0.023
Cu	214	1.697	1.261	0.743	1.378	7.135	0.059
Pb	214	0.317	0.58	1.831	0.077	3	0.003
Zn	214	3.223	4.044	1.255	1.4	18.785	0.043
Lens 3							
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**
Ag	142	12.408	7.99	0.644	10.641	40	0.216
Au	142	0.281	0.258	0.906	0.21	1.5	0.008
Cu	142	0.951	0.904	0.951	0.763	5	0.017
Pb	142	0.069	0.093	1.347	0.034	0.5	0.001
Zn	142	1.888	2.246	1.19	0.86	10	0.027
Copper Stockwork							
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**
Ag	1,021	8.884	8.823	0.993	6.135	75.001	0.259
Au	1,021	0.404	0.565	1.4	0.215	5	0.003
Cu	1,021	1.353	0.896	0.662	1.171	6.916	0.013
Pb	1,021	0.021	0.06	2.897	0.005	0.5	0
Zn	1,021	0.301	0.562	1.865	0.128	5	0.005
Copper Stockwork - FW							
Element	**Number**	**Mean**	**SD**	**CV**	**Median**	**Maximum**	**Minimum**
Ag	129	10.205	8.593	0.842	8.033	56.701	1.587
Au	129	0.51	0.53	1.04	0.354	3.664	0.006
Cu	129	1.532	1.015	0.662	1.239	5.212	0.046
Pb	129	0.023	0.068	2.959	0.005	0.454	0.001
Zn	129	0.347	0.86	2.478	0.087	5	0.014

Source: RPA 2014

14.7 Bulk Density

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

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

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

14.8 Geostatistics and Search Criteria

14.8.1 Geostatistics

RPA carried out a geostatistical analysis for copper, zinc, lead, gold and silver to assist in deriving search and kriging parameters for block grade interpolations. The analysis was conducted using Sage, and GEMS software. The variogram models are summarized in Table 14.5. The data were grouped into two broad classes: sulphides (L2MS, UW-MS, and L3) and stockwork, in order to maximize the number of composite pairs but still keep mineralogically distinct zones separate. Reasonably coherent variogram models, oriented in a manner consistent with the known geological trends, were achievable for most components. In general, spherical models seemed to fit best for the sulphides, and exponential models for the stockwork zones. The overall interpretation of the variogram models was hampered to some degree by the lack of pairs within a range of 30 m, which is a reflection of the drill spacing.

Note that in this document the terms semi-variogram, variogram, and correlogram are used interchangeably.

The downhole semi-variograms also tended to be reasonably coherent, albeit with fairly short ranges (in the order of 10 m to 20 m). The nugget effects interpreted from these diagrams were typically less than 15% of the total sill. Gold in the sulphide zones had the highest nugget, at 31.2% of the sill. Stockwork zinc was also relatively high, at 23.5% of the sill (Table 14.5).

Table 14.5: Variogram Results

Zone	Type	Nugget	C1	Sill	Axis	Ranges (m)	Orientation (Az/Plunge)
Silver							
L2/L3	Spherical	0.105	0.895	1	Major	117.8	205/56
					Semi	117.8	115/00
					Minor	12.5	205/-34
Stockwork	Exponential	0.076	0.924	1	Major	82.7	107/-30
					Semi	29.1	014/-05
					Minor	19.9	095/59
Gold							
L2/L3	Spherical	0.312	0.688	1	Major	45.6	126/42
					Semi	20.2	171/-38
					Minor	4.5	060/-25
Stockwork	Exponential	0.073	0.928	1.001	Major	82.4	172/10
					Semi	30	264/11
					Minor	11	220/-75
Copper							
L2/L3	Spherical	0.133	0.867	1	Major	49.4	124/-16
					Semi	49.4	357/-65
					Minor	7.3	040/19
Stockwork	Exponential	0.028	0.972	1	Major	43.4	070/-28
					Semi	32.5	306/-47
					Minor	9.8	358/30
Zinc							
L2/L3	Exponential	0.066	0.934	1	Major	272.7	115/00
					Semi	225.2	205/67
					Minor	32.3	040/19
Stockwork	Spherical	0.235	0.765	1	Major	192.1	112/00
					Semi	114.1	203/66
					Minor	45.9	201/-24
Lead							
L2/L3	Spherical	0.043	0.957	1	Major	111.4	121/19
					Semi	53.8	250/61
					Minor	8.7	203/-21
Stockwork	Exponential	0.172	0.828	1	Major	68.4	111/00
					Semi	36.1	200/-69
					Minor	20.1	021/-21

Silver variograms were fairly well-formed and interpretable for both the sulphides and the stockwork. The model for the sulphides was aligned parallel to the strike and dip of the L2/L3 horizons. There was no anisotropy apparent in the plane of the mineralization. The nugget effect was 10.5% of the total sill for the sulphides, and only 7.6% for the stockwork. For the stockwork zones, the model was aligned close to the overall plunge of the deposit.

The variograms for gold were the least coherent of all five metals, with the shortest ranges. For the sulphide horizons, the gold model was aligned with major and semi-major axes parallel to the plane of the mineralization, but the major axis was perpendicular to the deposit plunge. The model for the stockwork zones did not fit an orientation for any known geological feature.

In the sulphide zones, the variograms for copper were quite clear and interpretable, with the major and semi-major axes parallel to the plane of the mineralization. As with silver, there was no apparent anisotropy within this plane. In the stockwork zones, the model was oriented subparallel with the trend of the main bodies of mineralization, but with the major axis perpendicular to the plunge. Ranges were similar in both zones. This model also had a very low nugget effect, measuring 2.8% of the total sill.

The zinc variograms in the sulphides had the longest ranges of any of the metals (see Table 14.6). The orientation of the major and semi-major axes was parallel to the sulphide horizons, although the major axis was horizontal and not parallel to the plunge. The model for the stockwork zones was similar in that it was parallel to the mineralized zones, with the major axis oriented horizontally. The ranges for the stockwork zones were shorter than those in the sulphides, with a higher anisotropy ratio. The ratio between the major and semi-major axis ranges in the sulphides was 1.01, compared to 1.68 in the stockwork.

Lead in the sulphide horizons yielded a variogram model with the major and semi-major axes parallel to the overall trend of mineralization, and an anisotropy ratio of roughly two to one. The plunge of the major axis was somewhat shallower than the apparent plunge of the zones. The stockwork model was not as clearly aligned with the geology. The major axis of this model was subparallel with that of the massive/semi-massive (MS/SM) zones, but the semi-major axis was perpendicular to the overall strike and dip of the stratigraphy. Ranges in the stockwork zones were shorter than for the sulphide horizons. The variograms were somewhat less coherent as well, and required a pairwise relative transformation to attenuate the noise and yield an interpretable curve. Nugget effects were low in the sulphide zones and moderate in the stockwork zones.

14.8.2 Search Strategy

The mineralized bodies are observed to strike approximately 110° to 115°, with a dip of 68° to the north-northeast and a plunge of -45° to the west-northwest. The search ellipsoids were configured with the major and semi-major axes parallel to the overall plane of mineralization. The minor axes, by definition, were perpendicular to the dip plane. Within each domain, three interpolation passes were run, each with progressively smaller ranges. In the second and third passes, the interpolation would overwrite blocks estimated by the previous pass. The maximum search distances for each domain were based on the range of the zinc variograms (see Table 14.5).

Search radii in the sulphide zones were 250 m by 250 m by 40 m for the first pass, 125 m by 125 m by 20 m for the second, and 50 m by 50 m by 10 m for the third. In the stockwork zones, the respective searches were 200 m by 200 m by 50 m, 100 m by 100 m by 25 m, and 50 m by 50 m by 15 m.

For all passes, blocks required a minimum of two composites to generate an estimate, and the maximum number of composites per block was limited to 12. No more than two composites could be used from any one drill hole. The same search parameters were used for all elements (i.e., copper, zinc, lead, gold, and silver) to ensure that uniform coverage was achieved for all components in all blocks.

14.9 Block Model

The block model for the 2013 estimate was an array of blocks 10 m by 10 m by 10 m oriented roughly parallel to the strike of the deposit (approximately 110°). The model geometry is summarized in Table 14.6.

Table 14.6: Block Model Geometry

Block Size:		
	X	10
	Y	10
	Z	10
Origin (Uppermost SW Corner):		
	X	639172.071 E
	Y	6055570.727 N
	Z	350 m el
Extents:		
	Columns	300
	Rows	170
	Levels	122
Rotation (positive counter-clockwise):		-19.712 deg

Source: JDS 2014

The model was constructed in GEMS, an off-the-shelf mining software package. Grades for copper, zinc, lead, gold, and silver were interpolated into the blocks using ordinary kriging.

14.10 Block Model Validation

The block grade interpolations were validated using the following methods:

- Visual inspection in section views of the block grades and comparison with drill hole composite grades;
- Cross-validation (comparison of each data point with an estimate of that data point made from surrounding data); and
- Comparison between global composite means and block means.

In RPA's opinion, the visual inspection demonstrated that the block grade interpolations honoured the drill hole composites reasonably well.

Cross-validation is a process whereby each composite is sequentially removed from the data set and its value is interpolated from the surrounding composites. The real and interpolated values are then compared to check for biases in the kriged results. Table 14.7 shows the results of this comparison. In RPA's opinion, cross-validation demonstrates that there was no significant bias introduced by the kriging.

Table 14.7: Cross-validation Results

Copper Stockwork - FW					
	Silver (g/t)	**Gold (g/t)**	**Copper (%)**	**Lead (%)**	**Zinc (%)**
Mean - Comps	10.205	0.510	1.532	0.023	0.347
Mean - Est	10.244	0.522	1.482	0.021	0.279
%Difference	0.4%	2.4%	-3.3%	-8.7%	-19.6%
L2MS					
Mean - Comps	24.406	0.235	0.285	0.453	6.232
Mean - Est	24.137	0.233	0.294	0.442	6.212
% Difference	-1.1%	-0.9%	3.2%	-2.4%	-0.3%
UW-MS					
Mean - Comps	28.469	0.861	1.697	0.317	3.223
Mean - Est	28.629	0.845	1.679	0.350	3.381
%Difference	0.6%	-1.9%	-1.1%	10.4%	4.9%
L3 MS					
Mean - Comps	12.408	0.281	0.951	0.069	1.888
Mean - Est	12.893	0.279	0.957	0.073	2.004
% Difference	3.9%	-0.7%	0.6%	5.8%	6.1%
CSZ					
Mean - Comps	8.884	0.404	1.353	0.021	0.301
Mean - Est	8.901	0.409	1.366	0.019	0.285
% Difference	0.2%	1.2%	1.0%	-9.5%	-5.3%

RPA, 2014

The global mean grades of the composites were compared with the mean block grades for Pass 3 (i.e., the shortest radius search). Table 14.8 shows the results of this comparison. In RPA's opinion, with the exception of lead in the Stockwork zones, the mean grades compare very well and do not suggest that there is any bias in the interpolations. The reason for the apparent bias in lead is unknown; however, RPA notes that the lead grades are not significant. Very small absolute differences in grade will show as large percent differences, but the overall impact on the resource estimate will be negligible.

Table 14.8: Comparison of Block and Composites

Copper Stockwork - FW					
	Silver (g/t)	**Gold (g/t)**	**Copper (%)**	**Lead (%)**	**Zinc (%)**
Composites	10.205	0.510	1.532	0.023	0.347
Blocks	9.329	0.460	1.331	0.040	0.436
%Difference	-8.6%	-9.8%	-13.1%	72.6%	25.6%
L2MS					
Composites	24.406	0.235	0.285	0.453	6.232
Blocks	24.376	0.259	0.322	0.407	6.190
%Difference	-0.1%	10.2%	13.0%	-10.2%	-0.7%
UW-MS					
Composites	28.469	0.861	1.697	0.317	3.223
Blocks	27.374	0.844	1.695	0.309	3.533
%Difference	-3.8%	-2.0%	-0.1%	-2.5%	9.6%
L3 MS					
Composites	12.408	0.281	0.951	0.069	1.888
Blocks	12.181	0.287	0.941	0.064	1.890
% Difference	-1.8%	2.1%	-1.1%	-7.2%	0.1%
CSZ					
Composites	8.884	0.404	1.353	0.021	0.301
Blocks	9.915	0.370	1.310	0.029	0.344
Pct Difference	11.6%	-8.3%	-3.2%	38.1%	14.3%

Source: RPA, 2014

14.11 Classification

Blocks within the nominal range of the zinc variogram (i.e., 250 m in the massive and semi-massive sulphides and 190 m in the stockworks) were assigned the Inferred classification. A block interpolation was carried out wherein the average anisotropic distance to the nearest three drill holes was calculated. The results were plotted and inspected on a longitudinal projection view to determine where the drill pattern was dense enough to upgrade the Mineral Resources to Indicated. Much of the upper portion of the deposit has been drilled to a nominal spacing of 65 m or less. For a regular rectangular pattern of holes, this is roughly equivalent to a maximum distance of about 45 m to the nearest holes. A cluster of blocks within this distance constraint was outlined on the long section and a wireframe was created to capture these blocks so that they could be upgraded to Indicated.

A portion of the CSZ near the lowermost portion of the deposit was excluded from the Mineral Resources. Most of this material is outside of the 190 m distance constraint applied for the Inferred classification and, in RPA's opinion; the geometry of this portion of the zone also appears to be somewhat different than the rest of the deposit. Consequently, it was excluded from the Mineral Resources pending more drilling to improve the confidence level of the geological interpretation.

In summary, the Mineral Resources were classified using the following criteria:

- Indicated classification was applied in the core of the deposit, where the nominal drill hole spacing is 65 m or less, and/or where the average distance to the nearest three drill holes is 45 m or less;
- All other blocks, estimated to a maximum distance of 250 m in the massive and semi-massive sulphide bodies and 190 m in the stockwork bodies, have been classified as Inferred Mineral Resources, with the exception of some of the stockwork mineralization in the lowermost extremity of the deposit; and
- In RPA's opinion, the Mineral Resources are classified in a manner that is consistent with CIM Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM definitions) as incorporated into NI 43-101 regulations and guidelines.

14.12 Cut-Off Criteria

RPA used an NSR value for application of a cut-off to the block model. The NSR was estimated for each block using provisions for metallurgical recoveries, smelter payables, refining costs, freight, and applicable royalties (Table 14.9). Plant recoveries were based on the results of metallurgical test work conducted by Foran. The smelter terms and freight costs were derived from a report prepared for Foran by a metals marketing consultant. Metal prices used for Mineral Resources were based on consensus, long term forecasts from banks, financial institutions, and other sources. The calculation was based on the assumption that two products, a copper and a zinc concentrate, would be produced by a processing facility at site.

Table 14.9: NSR Cut-off Grade Assumption

Exchange Rate: US$1.00 : C$1.10		
Metallurgical Recoveries:		
Copper Conc	Copper	85.0%
	Zinc	5.0%
	Gold	62.0%
	Silver	63.0%
Zinc Conc	Copper	6.0%
	Zinc	75.0%
	Gold	11.0%
	Silver	21.0%
Metal Prices:		
Copper	US$3.25/lb	
Zinc	US$1.10/lb	
Gold	US$1,400/oz	
Silver	US$25.00/oz	
Smelting and Refining:		
Copper Conc	US$75.00/dmt conc	
Zinc Conc	38% of Zinc	
Transport:		
Copper Conc	US$220.00/dmt conc	
Zinc Conc	US$99.00/dmt conc	

A $45/t cut-off value was derived by RPA based on comparable projects in North America, taking into account provisions for milling, general and administration (G&A), and direct mining costs (i.e., no development). Foran has chosen to report the Mineral Resources at a higher cut-off value of $60/t in order to be closer to the criterion used by other current and planned mining operations in the region. Table 14.10 shows the Mineral Resources reported at a range of cut-off values, starting at the base case. The percent difference between the Mineral Resources at $45/t and $60/t is also shown. The impact of increasing the cut-off value was reduction of the tonnage and increase in the grades. The increase in grades did not fully offset the reduction in tonnage, which resulted in a reduction of overall metal contents of between 5% and 10%. In RPA's opinion, the more conservative cut-off value is not overly punitive and provides a margin of safety with respect to costs and metal prices. RPA concurs with the use of the $60/t cut-off value.

RPA notes that the NSR calculation was done before the present study, and is therefore out of date. In RPA's opinion, the block model NSR value should be updated to reflect the results of this PEA.

Table 14.10: Mineral Resources at a Range of Cut-offs

Cut-off (US$/t)	Tonnes (kt)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	NSR ($/t)	Cu (Mlb)	Zn (Mlb)	Au (oz)	Ag (oz)
Indicated										
75	10,200	1.47	2.71	0.59	18.7	119.56	332	610	194,000	6,120,000
65	12,700	1.33	2.73	0.52	17.7	109.44	371	765	211,000	7,220,000
60	**13,900**	**1.28**	**2.67**	**0.49**	**17.1**	**105.57**	**391**	**817**	**220,000**	**7,650,000**
55	14,900	1.24	2.59	0.47	16.7	102.42	408	852	227,000	8,000,000
45	**16,300**	**1.19**	**2.48**	**0.45**	**16.1**	**97.96**	**428**	**890**	**235,000**	**8,430,000**
Inferred										
75	9,500	1.4	3.19	0.47	18.5	115.91	294	667	144,000	5,640,000
65	10,500	1.35	3.1	0.45	18	111.92	313	717	152,000	6,070,000
60	**11,300**	**1.32**	**2.97**	**0.43**	**17.5**	**108.32**	**328**	**740**	**157,000**	**6,340,000**
55	12,000	1.28	2.86	0.42	16.9	105.04	339	757	161,000	6,530,000
45	**13,100**	**1.23**	**2.72**	**0.4**	**16.3**	**100.64**	**356**	**785**	**167,000**	**6,880,000**
Diff Ind	-14.70%	7.20%	7.60%	9.80%	6.40%	7.80%	-8.60%	-8.20%	-6.40%	-9.30%
Diff Inf	-13.70%	7.00%	9.20%	9.00%	6.80%	7.60%	-7.70%	-5.80%	-6.00%	-7.80%

Source: RPA 2014

14.13 Changes from the Last Estimate

Table 14.11 shows the differences between the current estimate and the 2011 estimate. In RPA's opinion, there are many factors contributing to the changes, some of which complement each other while others conflict. The principal influences on the Mineral Resource changes are listed below:

- Additional drilling information;
- Revised downhole and collar surveys;
- Updated geological interpretations and wireframe models of the mineralized zones;
- Estimation of gold grades for the first time in much of the deposit;
- Increased metal prices;
- Revised NSR calculation for application of the cut-off;
- Revised variogram parameters; and
- Revised classification, particularly in the CSZ.

Table 14.11: Current vs. Previous Estimate

January 2013 Estimate									
	Tonnes (kt)	**Cu (%)**	**Zn (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu (Mlb)**	**Zn (Mlb)**	**Au (oz)**	**Ag (oz)**
Indicated	13,900	1.28	2.67	0.49	17.1	391	817	220,000	7,650,000
Inferred	11,300	1.32	2.97	0.43	17.5	328	740	157,000	6,340,000
October 2011 Estimate									
Indicated	12,070	1.16	3.68	0.24	19.2	308	980	94,700	7,470,000
Inferred	9,570	1.08	3.8	0.16	19.3	227	801	49,400	5,940,000
% Difference									
Indicated	15.20	10.30	-27.60	101.60	-11.00	27.00	-16.70	132.30	2.40%
Inferred	18.10	22.50	-21.80	169.40	-9.60	44.70	-7.60	217.80	6.70%

Source: JDS 2014

Gold content appears to have increased quite significantly, which is somewhat misleading, because in 2011, gold was not estimated for many of the zones. Consequently, the reported gold content for the 2011 estimate was artificially low.

The revised metal prices, along with the updated NSR calculation, resulted in an overall enlargement of all the zones. This was due to the fact that with increased metal prices and the contribution of gold to the calculation, additional lower grade material was captured by the cut-off criteria used. This would tend to increase the tonnage at the expense of the grade. A reduction in grade occurred for zinc and silver, but not for copper. This suggests that different factors are influencing the copper grades.

Copper grades are observed to have dropped fairly significantly in the UW-MS, and increased in the L2MS. The reason for this is not known but may have resulted from the following factors. The re-surveying of the holes introduced a number of quite significant modifications in the locations of some of the drill holes and in their downhole traces. The revisions to the geological interpretations had some significant impacts on the block grades, the extent of which are not fully understood. Zone boundaries were changed and some intercepts were re-assigned to different zones. In RPA's opinion, these influences would certainly result in changes to the local block grades, but it is not clear what effect this would have on the global grades.

The increase in copper could also be partially due to a change in the classification of the Mineral Resources in the CSZ. RPA relaxed a constraint placed in 2011 on some of the material in the lowermost extremity of the zone, which allowed the inclusion of material comparatively high in copper. This had the effect of increasing both the tonnes and grade of the Inferred Mineral Resources in the CSZ.

Much of the more recent drilling was done primarily in the central and upper portions of the deposit, and affected the L2MS, UW-MS, and CSZ zones. In RPA's opinion, the zinc grades in these holes appear, at least in some areas, to be lower than the zinc grades in the earlier generations of holes. This resulted in a reduction in block zinc grades, particularly in the upper portions of the UW-MS, which may explain why the overall reduction in zinc grade was greater than that for the silver.

15 MINERAL RESERVE ESTIMATE

Indicated and Inferred resources were used in the life-of-mine plan with Inferred resources representing 43% of the material planned for processing. Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no mineral reserves at McIlvenna Bay.

16 MINING METHODS

16.1 Proposed Mining Method

McIlvenna Bay dips to the north at 65° to 70°, although in selected areas it dips vertically. The average strike length of the economic deposit is 600 m with an average thickness of approximately 5.6m. In areas where mineralized zones run parallel to each other, the distance between the host rock hangwall and footwall can exceed 50 m.

It has been identified that due to distinctly different metallurgical properties between the different mineralization zones, it is economically beneficial to mine them separately.

JDS selected sub-level long hole (LH) stoping with cemented paste backfill as the principal mining method at McIlvenna Bay due to its high productivity, low cost, selectiveness, and successful history of application for deposits of this nature, such as the Hudbay Mineral Inc. nearby 777 operation, which produces 4,330 tpd from their underground operation (Hudbay, Oct 2012).

16.1.1 Mine Method Description

LH stoping is a semi-selective and productive underground mining method, and well suited for steeply dipping deposits of varying thickness. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralization. In the planned LH for McIlvenna, a top and bottom drift delineate the stope and a dedicated long hole drilling machine drills blast holes between the two drifts. The drill holes are loaded with explosives and the stope is blasted, with broken material falling to the bottom drift for extraction. Once the stope has begun blasting phase it cannot be accessed by personnel. For this reason a tele-remote load haul dump machine (LHD) is required to remove the blasted material from the stope.

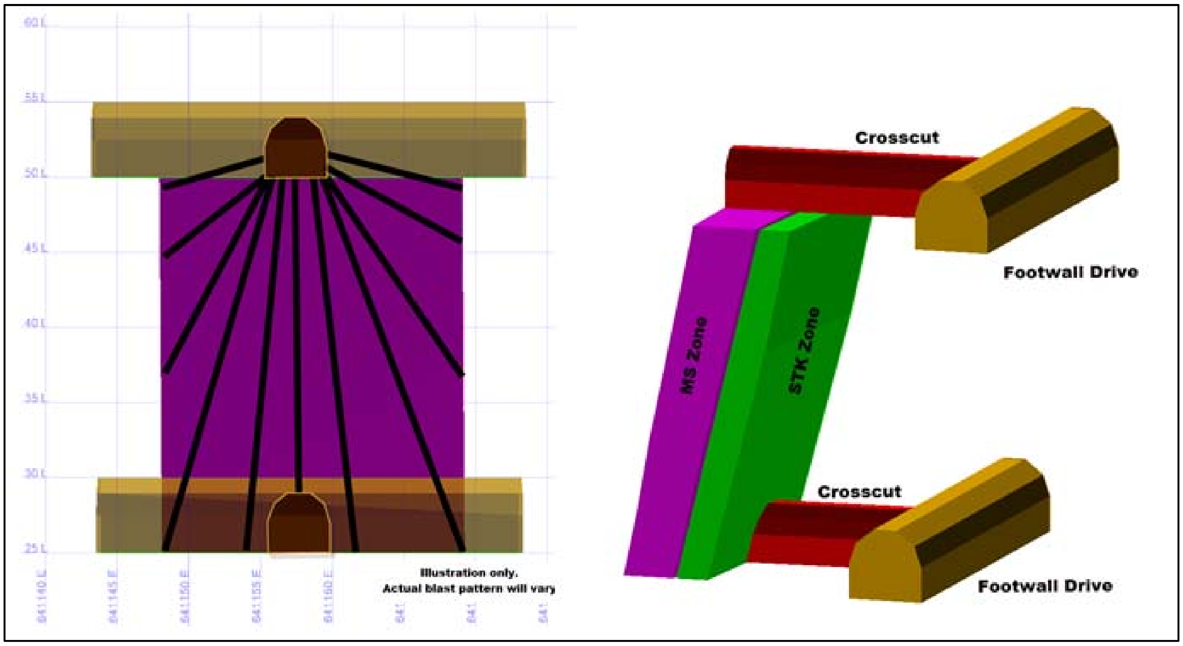
One of the limitations with LH stoping is that the dimensions of the stope should not exceed a long hole drilling machine's effective range which, for top hammer drill rigs, is generally 30 m. Another limitation with LH stoping is the stopes must remain open long enough to remove the mineralized material and fill with an engineered backfill material (if pillars are not used). These limitations generally restrict level spacing to 30 m or less, and subject stope strike lengths to geotechnical review.

Two methods of LH stoping are considered for McIlvenna Bay. Transverse stoping will be the primary method, whereby crosscuts will cut through the stope perpendicularly, and long hole fans will drill off the strike length of the stope.

This method is beneficial for production rates as multiple stopes can be in operation at once on a level. It is also beneficial for mining parallel mineralization zones, as clean separation between the mineralization types is possible with careful fan drilling.

The shortfall of transverse long hole mining is that a footwall drift is required outside the mineralized zone for the entire strike length, and crosscuts must be driven long enough to maintain a safe distance between the footwall drift and the zone, which depending on geotechnical constraints can exceed 25 m. The method is shown in Figure 16.1 and Figure 16.2.

Figure 16.1: Transverse Long Hole Stoping (Oblique View)



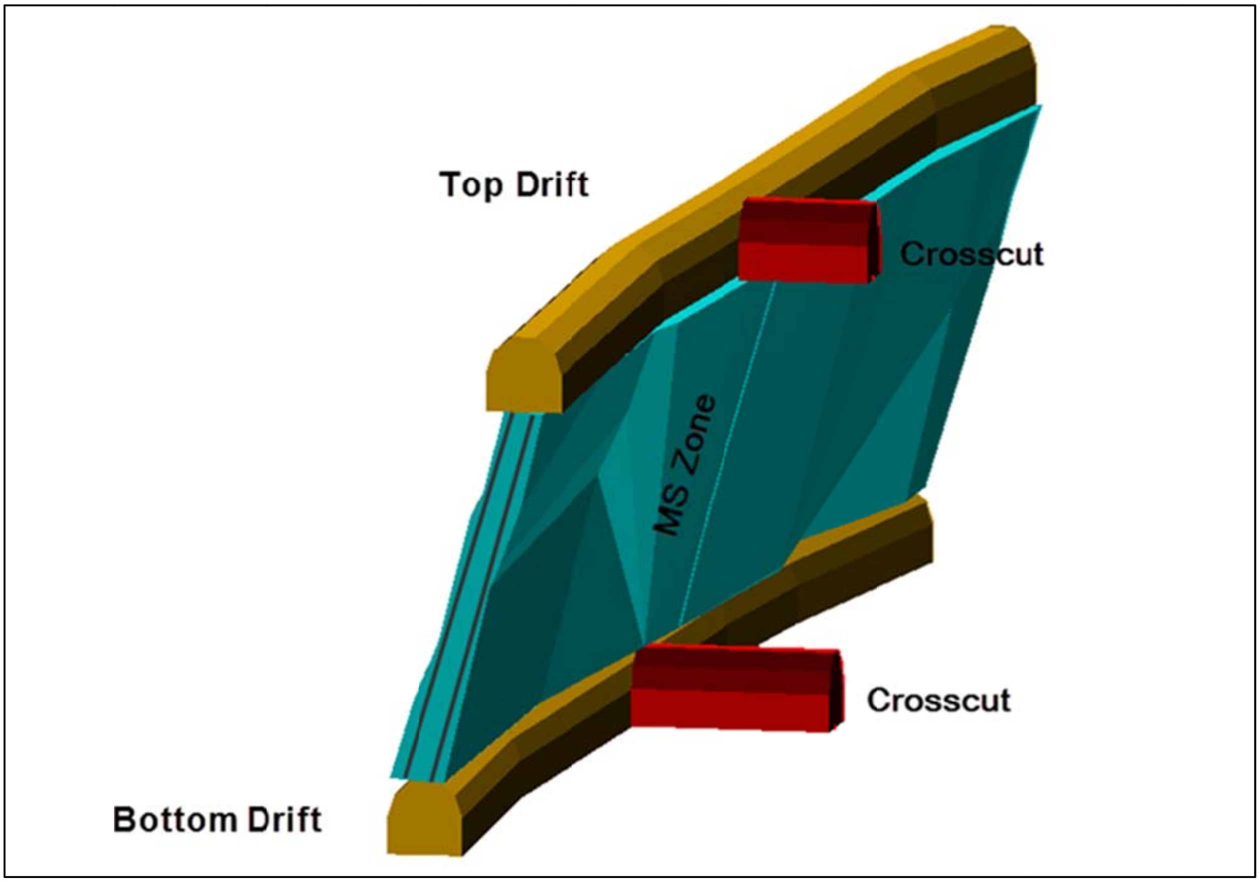
Source: JDS 2014

Figure 16.2: Transverse Long Hole Stoping (Plan View)

Source: JDS 2014

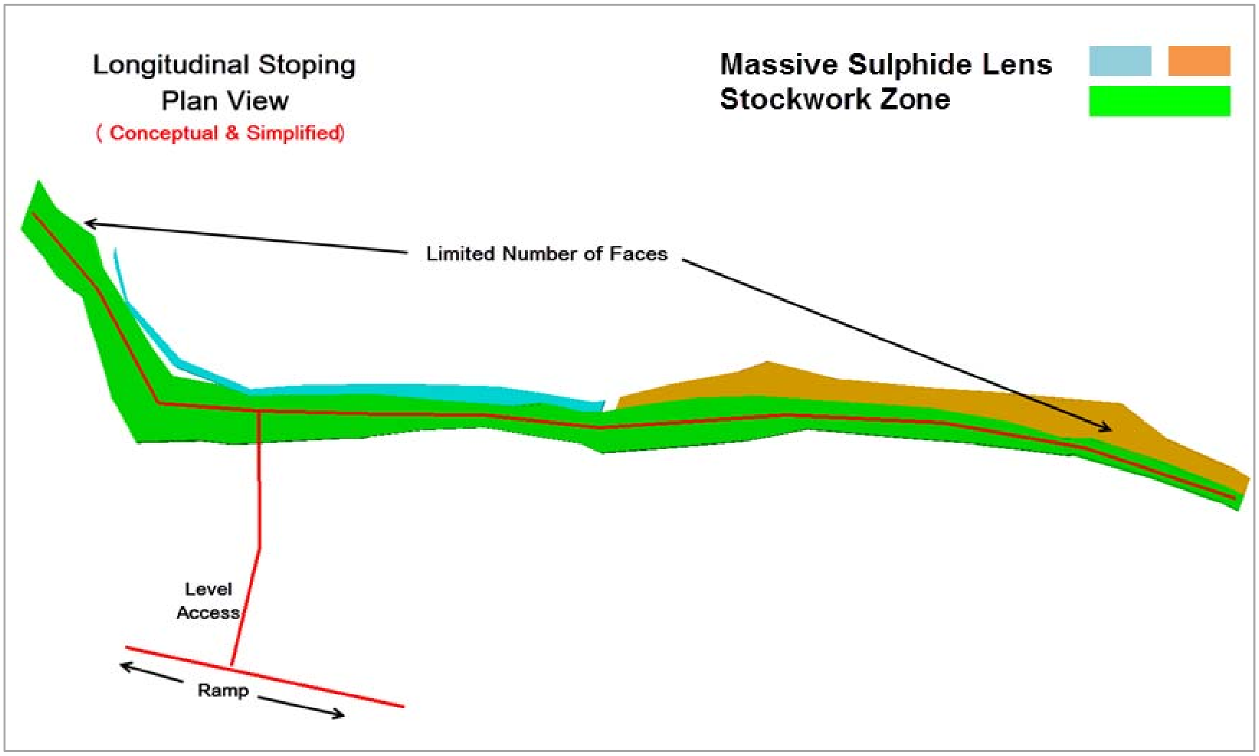
Longitudinal LH stoping may also be applied at McIlvenna Bay for isolated zones of narrow vein mineralization. In this method, the top and bottom drifts are driven through the vein, rather than perpendicularly to it, and long holes are drilled along the strike of the stope. This method is much more cost efficient than transverse mining as it does not require a footwall drift or multiple crosscuts into the mineral zone. The shortfall of this method is that it is much less productive than transverse mining, as the entire strike length of the level must be driven first, and then mined in retreat, one stope at a time. The method is also limited to vein width, as the entire bottom sill must be developed (silled out). In man-entry sub level drifts the silled out width may be limited by geotechnical constraints. The method is shown in Figure 16.3 and Figure 16.4.

Figure 16.3: Longitudinal Longhole Stoping (Oblique View)



Source: JDS 2014

Figure 16.4: Longitudinal Longhole Stoping (Plan View)



Source: JDS 2014

The mine plan presented in this report focuses on the application of transverse LH mining, which was selected to sustain high production rates and selectivity between the mineralized zones. JDS has utilized longitudinal stoping at the fringes of each level where the method is more cost effective and does not hinder production rates.

Drift and fill mining may also be used at McIlvenna Bay for areas of the deposit which fall below an allowable dip for LH stoping; however, this report does not include costs or equipment for driving drift and fill stopes.

16.2 Geotechnical Criteria

A geotechnical study was prepared by Golder in July 2013 in order to provide geotechnical input to this report. The report endorsed JDS's level spacing and provided recommendations on the following:

- Maximum stope strike lengths;
- Horizontal mineralized width break-point where when SLOS should change from longitudinal (man-entry) to transverse (non-man-entry) stoping;
- Backfill requirements including strength and estimated required cement content;
- Sill and crown pillar dimensions; and
- Ground support.

Golder designed stope strike lengths for transverse and longitudinal mining shown in Table 16.1 below. Golder recommends that transverse stopes not have strike lengths greater than 25 m, and at depths below 1000m should be reduced to 20 m. Longitudinal stopes share the same strike limitations, but are also limited to a maximum of 10 m stope width with additional support (w.r.t. to standard development support) since it will be a man-entry operation. Generally transverse stopes are mucked with remote LHDs and therefore there were no restrictions placed on stope width. Golder provided estimates of dilution for individual stopes using the Equivalent Linear Ore Slough (ELOS) approach for individual stopes as outlined in Table 16.1 below.

Table 16.1: Maximum Stope Strike Length (Golder, July 2013)

SLOS Orientation	Maximum Stope Strike Length (m)	Total Hangingwall ELOS (m)
Longitudinal	25	1.5
Transverse above 1,000 m	25	1.5
Transverse below 1,000 m	20	1.5

Source: JDS 2014

Golder recommended that in areas with a dip of the orebody is less than 50°, overhand drift and fill mining should be used in place of SLOS mining.

Where en-echelon stope pillars exist, sequencing should proceed from the footwall lodes to the hangingwall lodes to avoid hangingwall pillar instability with the exception of where the thickness of the en-echelon pillar exceeds 10 m (measured as true thickness).

Ground support requirements are anticipated to be typical of other hard-rock mining situations in the general area, e.g. HudBay's 777 mine.

Based on Golder’s experience, benchmark strength requirements for the indicated cemented paste backfill configurations, suitable for cost estimation purposes are as follows:

- Vertical walls (up to 30 m high) – cemented paste that meets a minimum specification of approximately 1.0 MPa (28 day UCS); and
- Underhand mining up to 7 m span – cemented paste that meets a minimum specification of approximately 1.5 MPa (28 day UCS).

16.3 Net Smelter Return

Net Smelter Return (NSR) is the net revenue generated from mining, processing, and selling metal concentrates to a smelter. It is the total revenue of the product less the cost of production, and does not include capital costs.

JDS performed NSR calculations on the McIlvenna Bay resource model as a method of determining cut-off grade requirements. As each mineralized zone has unique metallurgical properties, it is important to accurately model the NSR performances of each zone separately. Three separate formulas were prepared for McIlvenna Bay.

- Copper Stringer: Copper concentrate NSR for the CSZ and CSZFW;
- Semi Massive Sulphide: Copper and zinc concentrate NSR for the UW-MS) zone; and
- Massive Sulphide: Zinc and bulk concentrate NSR for the L2MS and L3.

NSR results are converted to grade multipliers, which are applied to the resource grades to quickly estimate the resource economic potential. Details of the NSR formula are shown in Table 16.2 to Table 16.6. It should be noted that NSR parameters may not reflect the identical parameters used in economic modeling of the production plan.

Metal prices and royalty assumptions are shown in Table 16.7 and Table 16.8.

Table 16.2 Copper Concentrate 1 NSR Parameters for Mineable Resource

Copper Concentrate No. 1 - Copper Stockwork		
NSR Parameters	**Unit**	**Value**
Recovery to Cu Concentrate No. 1		
Cu Recovery	%	94.4
Zn Recovery	%	33.8
Pb Recovery	%	0.0
Au Recovery	%	84.6
Ag Recovery	%	76.9
Concentrate Grade		
Cu	%	29.2
Zn	%	1.1
Pb	%	0.0
Au	g/t	6.38
Ag	g/t	126.00
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	100.0
Min. Cu deduction	% Cu/tonne	1
Zn Payable	%	0
Au Payable	%	95
Min. Au deduction	g/t concentrate	0
Ag Payable	%	90
Min. Ag deduction	g/t concentrate	0
Treatment & Refining Costs		
Cu TC	$US/dmt concentrate	75.00
Cu RC	$US/payable lb	0.08
Au RC	$US/payable oz	15.00
Ag RC	$US/payable oz	1.00
Calculated Penalties	$US/dmt	3.50
Transport Costs	$US/dmt	198.48

Source: JDS 2014

Table 16.3 Copper Concentrate 2 NSR Parameters for Mineable Resource

Copper Concentrate No. 2 - Upper West		
NSR Parameters	**Unit**	**Value**
Recovery to Cu Concentrate No. 2		
Cu Recovery	%	83.4%
Zn Recovery	%	9.6%
Pb Recovery	%	43.4%
Au Recovery	%	59.7%
Ag Recovery	%	50.3%
Concentrate Grade		
Cu	%	24.2%
Zn	%	6.4%
Pb	%	1.3%
Au	g/t	6.50
Ag	g/t	216.00
Moisture Content	%	8%
Smelter Payables		
Cu Payable	%	100.0%
Min. Cu deduction	% Cu/tonne	1%
Zn Payable	%	0%
Au Payable	%	95%
Min. Au deduction	g/t concentrate	0
Ag Payable	%	90%
Min. Ag deduction	g/t concentrate	0.0
Treatment & Refining Costs		
Cu TC	$US/dmt concentrate	75.00
Cu RC	$US/payable lb	0.08
Au RC	$US/payable oz	15.00
Ag RC	$US/payable oz	1.00
Calculated Penalties	$US/dmt	9.12
Transport Costs	$US/dmt	198.48

Source: JDS 2014

Table 16.4: Zinc Concentrate 1 NSR Parameters Used for Mineable Resource

Zinc Concentrate No. 1 - Massive Sulphide		
NSR Parameters	**Unit**	**Value**
Recovery to Zn Concentrate No. 1		
Cu Recovery	%	0
Zn Recovery	%	85.40
Pb Recovery	%	0
Au Recovery	%	14.60
Ag Recovery	%	27.30
Concentrate Grade		
Cu	%	0.00%
Zn	%	55.00
Au	g/t	0.29
Ag	g/t	38
Moisture Content	%	8
Smelter Payables		
Zn Payable	%	85
Min. Zn deduction	% Zn/tonne	0
Au Payable	%	80
Min. Au deduction	g/t concentrate	1
Ag Payable	%	70
Min. Ag deduction	g/t concentrate	93.3
Treatment & Refining Costs		
Zn TC	$US/dmt	215
Zn RC	$US/Pay lb Zn	0
Au RC	$US/payable oz	0
Ag RC	$US/payable oz	0
Calculated Penalties	$US/dmt	2.5
Transport Costs	$US/dmt	97.27

Table 16.5: Zinc Concentrate 2 NSR Parameters Used for Mineable Resource

Zinc Concentrate No. 2 - Upper West		
NSR Parameters	**Unit**	**Value**
Recovery to Zn Concentrate No. 2		
Cu Recovery	%	0.0
Zn Recovery	%	76.3
Pb Recovery	%	0.0
Au Recovery	%	6.9
Ag Recovery	%	13.6
Concentrate Grade		
Cu	%	0.0
Zn	%	54.3
Au	g/t	0.81
Ag	g/t	63.00
Moisture Content	%	8
Smelter Payables		
Zn Payable	%	85
Min. Zn deduction	% Zn/tonne	0
Au Payable	%	80
Min. Au deduction	g/t concentrate	1
Ag Payable	%	70
Min. Ag deduction	g/t concentrate	93
Treatment & Refining Costs		
Zn TC	$US/dmt	215
Zn RC	$US/Pay lb Zn	0.00
Au RC	$US/payable oz	0.00
Ag RC	$US/payable oz	0.00
Calculated Penalties	$US/dmt	2.50
Transport Costs	$US/dmt	97.27

Table 16.6: Bulk Concentrate NSR Parameters Used for Mineable Resource

Bulk Concentrate - Massive Sulphide		
NSR Parameters	**Unit**	**Value**
Recovery to Bulk Concentrate		
Cu Recovery	%	56.0
Zn Recovery	%	2.1
Pb Recovery	%	59.1
Au Recovery	%	38.5
Ag Recovery	%	34.4
Concentrate Grade		
Cu	%	11.9
Zn	%	9.2
Pb	%	15.4
Au	g/t	5.27
Ag	g/t	332.00
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	25
Min. Cu deduction	% Cu/tonne	2
Zn Payable	%	100
Min. Zn deduction	% Zn/tonne	5
Pb Payable	%	100
Min. Pb deduction	% Pb/tonne	3
Au Payable	%	90
Min. Au deduction	g/t concentrate	1
Ag Payable	%	90
Min. Ag deduction	g/t in concentrate	93.3
Treatment & Refining Costs		
Bulk Conc TC	$/dmt	325
Zn RC	$US/dmt concentrate	0.00
Au RC	$US/payable oz	0.00
Ag RC	$US/payable oz	0.00
Calculated Penalties	$US/dmt	35.20
Transport Costs	$US/dmt	198.48

Table 16.7: NSR Metal Prices Used for Mineable Resource

Metal Prices in Use		
Cu	$US/lb	3.25
Zn	$US/lb	1.10
Pb	$US/lb	1.00
Au	$US/oz	1,400
Ag	$US/oz	25.00
CAD/USD Exchange	$/$	1.00

Source: JDS 2014

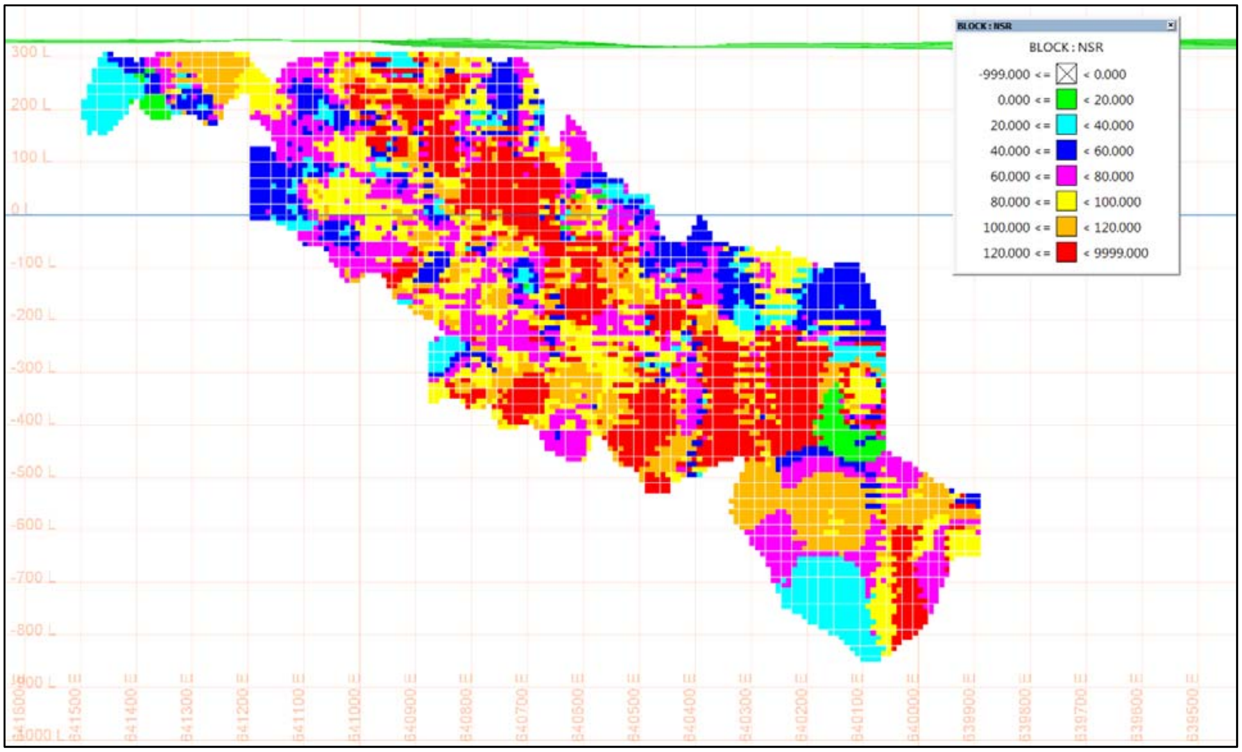
Table 16.8: NSR Royalties Used for Mineable Resource

Royalties		
Cameco/BHP Royalty	% of NSR	1%
Copper Reef Royalty	$/tonne ore	$0.75

Source: JDS 2014

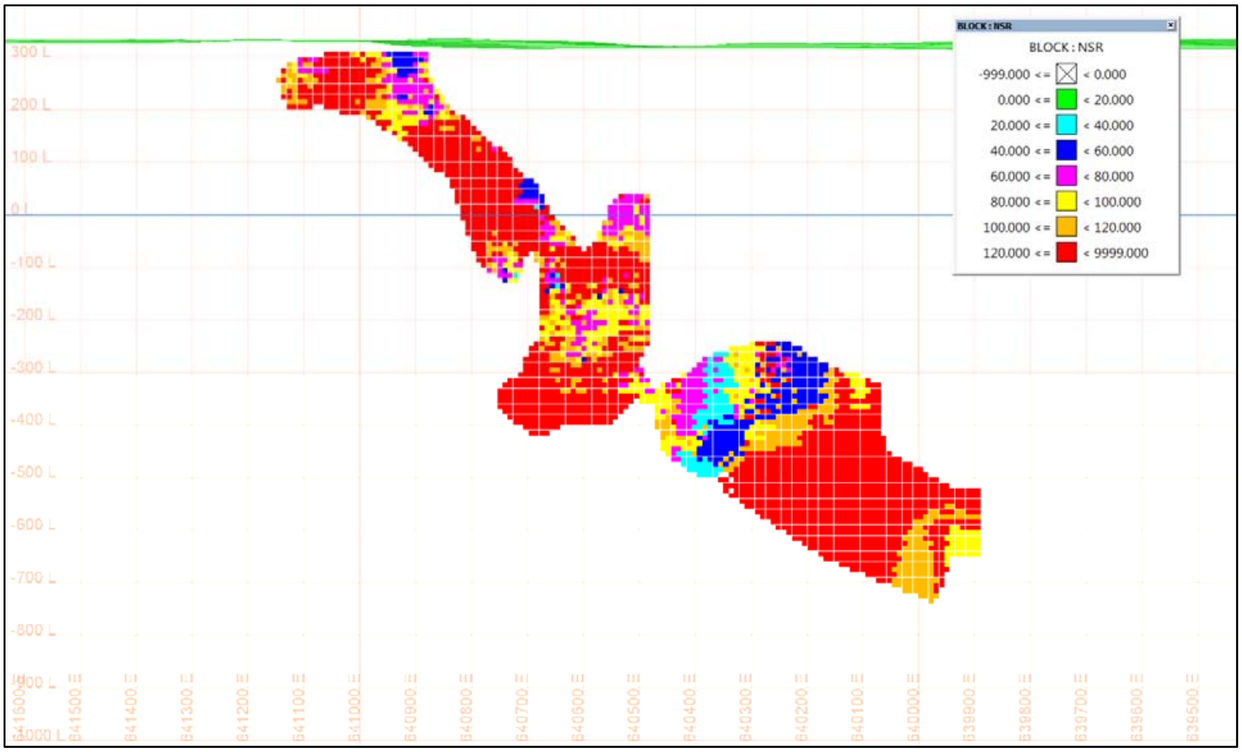
The above information was used to establish NSR formulae for each main mineral zone at McIlvenna. Figures 16.5 to 16.7 outline the NSR blocks by value ($/t) through each three NSR groupings.

Figure 16.5: 3D View of Copper Stockwork Zones by NSR



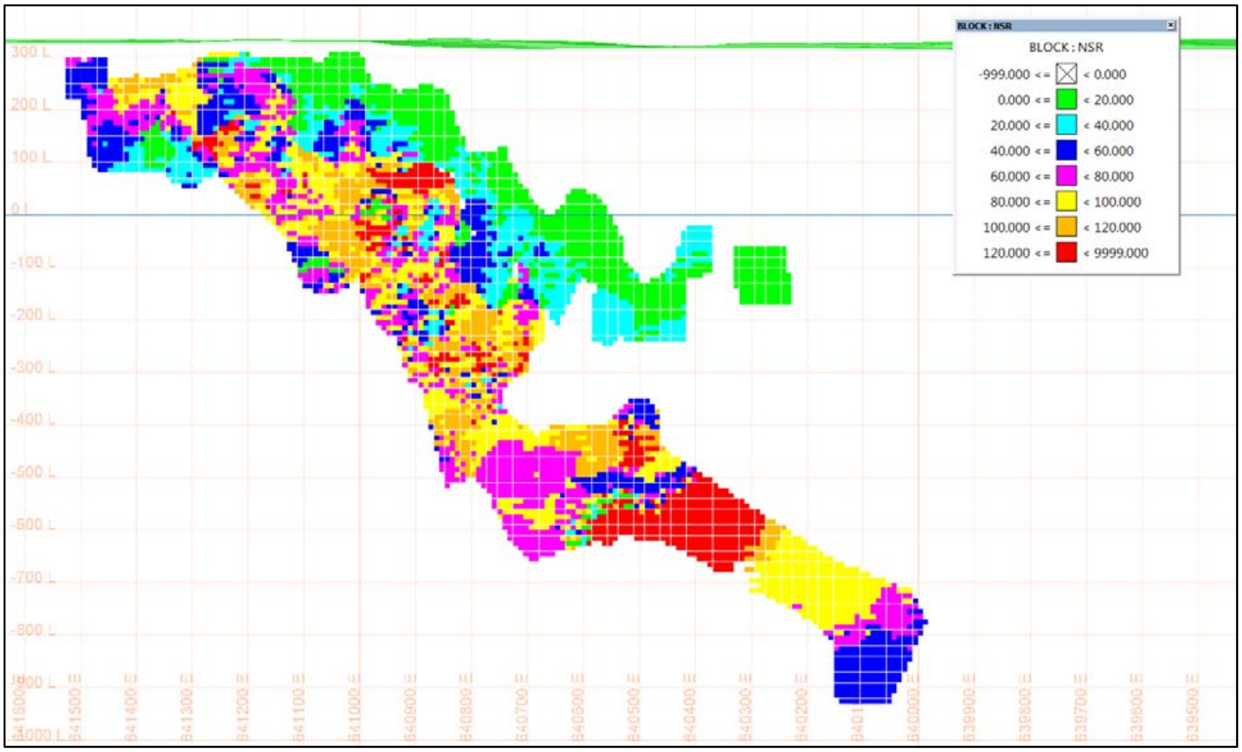
Source: JDS 2014

Figure 16.6: 3D View of Upper West Zone by NSR



Source: JDS 2014

Figure 16.7: 3D View of Massive Sulphide Zones (Zone 2 & Lens 3) by NSR



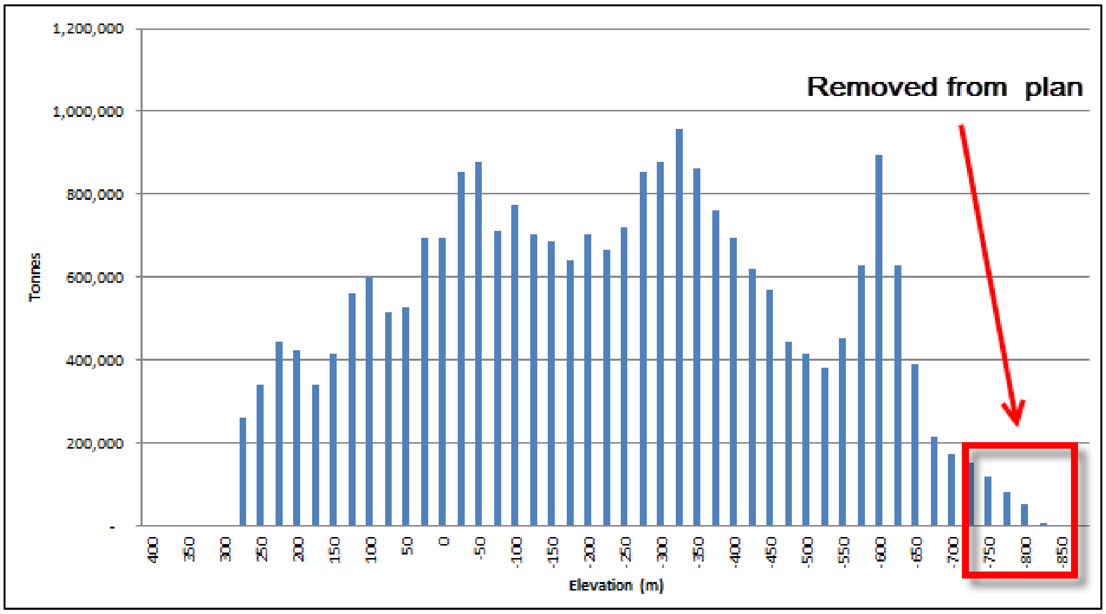
The three figures suggest that the highest grade material follows a central chute through the resource. The information above suggests that the McIlvenna Bay resource is best suited for a low cost, high tonnage bulk operation as opposed to a high grade, low tonnage operation. The reason for this is that the high grade is stretched out over a large vertical distance, which would provide low tonnage per vertical metre and a high capital cost per tonne in a high grade, low tonnage scenario.

16.4 Cut-off Grade Evaluation

Fully diluted and recovered stopes were assessed for NSR values. An NSR cut-off of US$65/tonne was selected as the economic cut-off, which was based on iterative trial and error mine plan designs. All diluted stopes falling below the NSR cut-off were removed from the mine plan, leaving only stopes that could pay for their own operating costs.

Elevation limitations were also applied to the resource, eliminating resources that could not cover capital development. The bottom of the McIlvenna deposit narrows to a point at 825 m, with a strike length of only 115 m, and tonnage of 155 tonnes per vertical metre (TPVM). The economic depth was set to -725 m where there are 6,800 TPVM, which is sufficient to pay for the capital costs of development. Figure 16.8 depicts the economic tonnes at elevations throughout the deposit.

Figure 16.8: Economic Tonnes by Elevation



Source: JDS 2014

The resource is open at depth and it is likely that exploration drilling will expand the lower resource and these lost tonnes will be reintroduced to the mine plan.

16.5 Mineable Resource

The mineable resource JDS has identified meets mine planning criteria and passes an NSR cut-off of US$65 /tonne.

Table 16-9 below outlines this mineable resource.

Table 16.9: Mineable Resource

Zone	Tonnage (kt)	Copper (%)	Zinc (%)	Lead (%)	Gold (g/t)	Silver (g/t)	NSR ($/t)
Indicated							
Main Lens - Upper West Zone	2,326	1.47	3.58	0.33	0.77	26.57	136.45
Main Lens - Zone 2	3,157	0.23	6.59	0.42	0.20	21.36	89.06
Lens 3	76	0.37	5.41	0.09	0.17	9.58	71.69
Copper Stockwork Zone	7,025	1.39	0.22	0.02	0.46	9.26	97.79
Copper Stockwork Footwall	968	1.50	0.39	0.03	0.49	9.34	104.55
Total Indicated	**13,552**						
Inferred							
Main Lens - Upper West Zone	2,729	1.45	3.33	0.11	0.44	16.91	121.22
Main Lens - Zone 2	1,901	0.33	6.91	0.43	0.28	21.58	93.49
Lens 3	0	0.00	0.00	0.00	0.00	0.00	0.00
Copper Stockwork Zone	5,555	1.38	0.44	0.04	0.39	10.90	95.32
Copper Stockwork Footwall	0	0.00	0.00	0.00	0.00	0.00	0.00
Total Inferred	**10,185**						

16.6 Mine Design

16.6.1 Dilution & Mining Recovery

16.6.1.1 Stope Design

Stopes were designed based on mine method and geotechnical restrictions discussed above:

- Height: 25 m;
- Length: 20 m; and
- Width: Mineralization Width (variable).

The mineralization wireframes LMS, L3, UW-MS, CSZ, and the CSZFW were portioned into stope dimensions of 25 m tall by 20 m long, and were left to run the full mineralized zone width. Each stope was measured for height, width, and length, and queried through their respective block models for tonnage, grade, and class.

16.6.2 Dilution

Dilution parameters were assigned to each stope to estimate over-break dilution experienced during mining operations. Several of the mineralization zones sit in parallel to each other, one footwall being another's hangingwall, and it was ensured that only hanging walls and footwalls backing onto host rock were diluted for over break. A minimum stope width of 2.0 metres was also applied to the resource, along with a 90% mine recovery factor. It has been assumed that 10% of mineralized material will either remain in the hangingwall, be mistaken for waste, or be mined as dilution into a different zone and fed through a mill circuit which yields no concentrate (milling will be campaigned for three different zones).

Average dilution experienced during operations using the parameters in the Table 16.10 below is estimated at 10%.

Table 16.10: Long Hole Dilution Parameters

Dilution Parameter	Units	LH Stoping Estimate
Overbreak HW/FW	m	0.5
Overbreak Stope Ends	m	0.5
Overbreak Floor	m	0.1
Minimum Width	m	2.0
Mine Recovery	%	90

16.7 Mine Production Criteria

16.7.1 Mine Access

McIlvenna Bay extends over 1.3 km in depth. To facilitate large scale mining it is recommended that a shaft be installed. As the economically mineable resource starts very near surface, it is also recommended that a ramp be driven to begin production while the shaft is constructed. Shaft operations typically have ramps linking working levels to allow mobile equipment to move from level to level, and this ramp will have the added convenience of starting from surface. A multitude of benefits are present when a mine has both ramp and shaft access, several of which are listed below.

Ramp Benefits:

- Quick access to mineral zones;
- Drive in and out capabilities;
- Alternate haulage route during shaft downtimes;
- Walk out secondary egress; and
- Fresh air source.

Shaft Benefits

- Lower haulage operating costs;
- Higher productivity than trucks; and
- Quicker access to the face at depth (higher availabilities).

It is proposed that the ramp will provide access to mill feed at a reduced throughput while the shaft is constructed. Once the shaft is fully operational all material will be crushed underground and hoisted to surface.

16.7.2 Production Rate Selection

The McIlvenna Bay mine plan is sized for a 5,000 tpd operation. Lower tonnage, high grade plans have not shown successful in mine plan iterations due to the spatial distribution of high grade material over long vertical extents. McIlvenna Bay has the bulk tonnage to support a large scale operation, and mineralization is very continuous at NSR cut-offs of US$65 /tonne and less.

16.7.3 Production Rate Justification

In order to achieve a production rate of 5,000 tpd, over 90 vertical metres of economic resource must be mined annually. JDS prepared production and cycle time calculations to estimate the productivity of one 100 m tall mining block consisting of four 25 m levels. It was found that one mining block containing just over one million tonnes could be developed in one year, and mined in just under 400 days, at an average mine rate of 2,700 tpd. In order to sustain a production rate of 5,000 tpd, at least two mining blocks are required per year.

A ramp operation does not have the capacity to drive level development fast enough to sustain this kind of production rate. However, with the shaft operation, multiple levels can be developed simultaneously provided infrastructure such as power, water, air, and ore/waste passes are available.

16.8 Production Sequencing

16.8.1 Level Sequencing

JDS selected mining sublevel spacings of 25 m for the LH mining at McIlvenna Bay. This is not an aggressive spacing, nor too conservative, and should provide for accurate drilling using top-hammer rigs. Sublevels were designed such that a footwall drive is off-set 15 m from the mineralized zone, with crosscuts entering the zone for each stope on 20 m centres. The 15 m footwall off-set from the zone provides geotechnical stability to the footwall drift, and prevents potential drift failures due to over-blasting or ground failures in the stopes. Crosscut lengths were assumed based on other projects JDS has worked with and must be validated through geotechnical engineering.

Stope sequencing was designed in a primary/secondary fashion. Primary stopes are mined first, backfilled with cemented paste fill, and then the secondary's are mined and backfilled. Details on the paste plant and binder contents are located in Section 16.12.1. Sublevels were grouped into mining blocks containing a minimum of four levels each. Several mine blocks will be mined above insitu mineable resources and will require a cemented sill pillar as to not sterilize the resource below the mine block. Sill pillars were strategically placed on shaft access levels to minimize development required to start a mining block. Figure 16.9 below outlines a typical 125 m tall mining block.

Figure 16.9: Typical Long Hole Mining Block

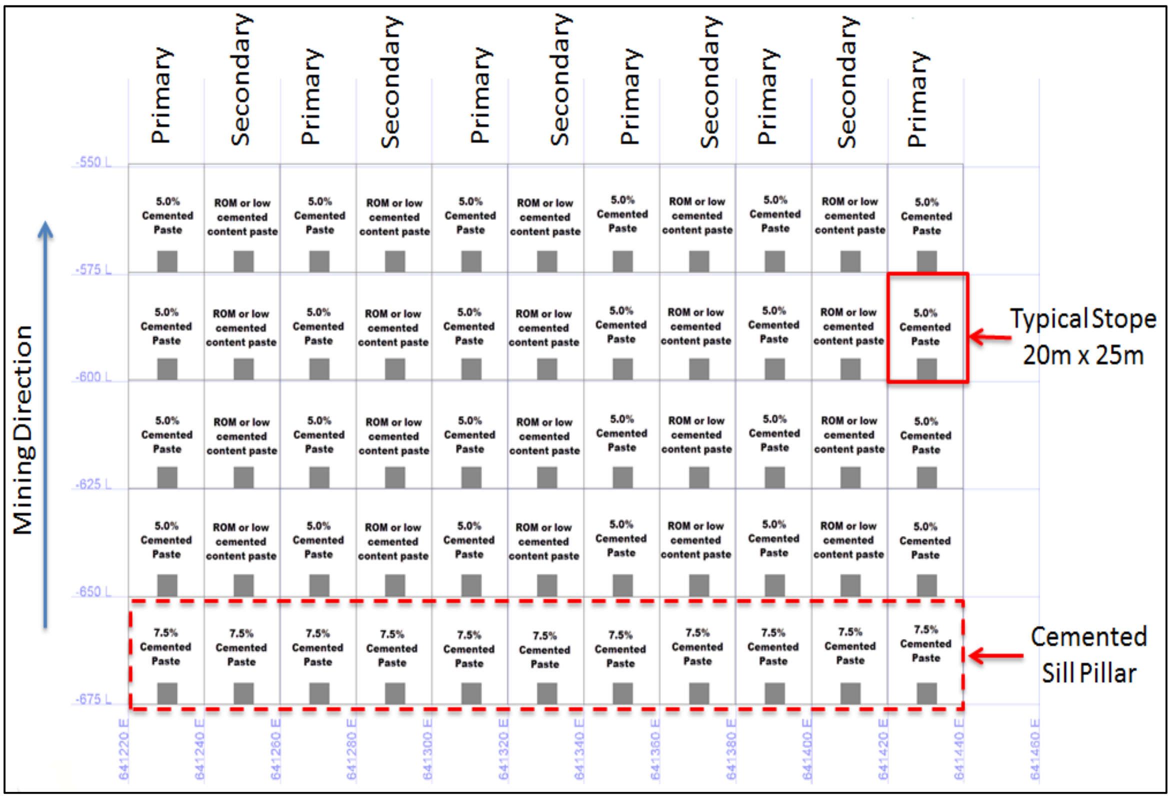
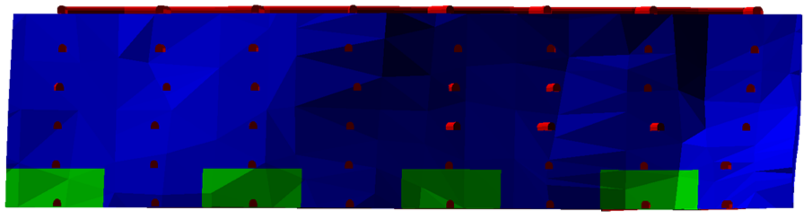


Figure 16.10 outlines the proposed stope sequence for each mine block. It will take approximately one year to fully develop one mine block, as each level will require footwall drives, crosscuts into the mineralized zone, ramp access, and typical level infrastructure such as vent drifts, sumps, and ore/waste passes.

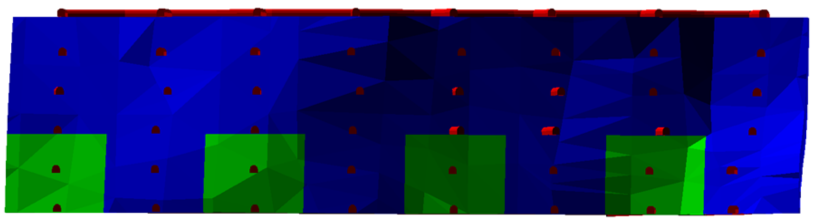
It is important to note that when mine blocks approach a sill pillar, tight backfilling of primary stopes will be important to ensure a stable back for secondary stope recovery. Stopes sitting beneath a sill pillar will not have top and bottom access for drilling or backfilling, and will be mined using blind upholes.

Tight backfilling may be achieved by drilling pilot holes from the crosscuts above the stope through which paste fill is pumped. It will be important to have multiple mining areas open when a mine block approaches the final lift, as production rate losses will be seen in the drilling and backfilling cycle in top lifts.

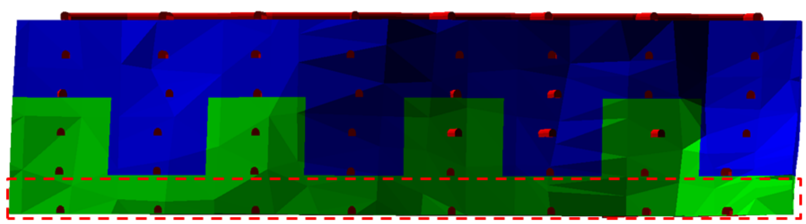
Figure 16.10: Long Hole Stope Sequencing



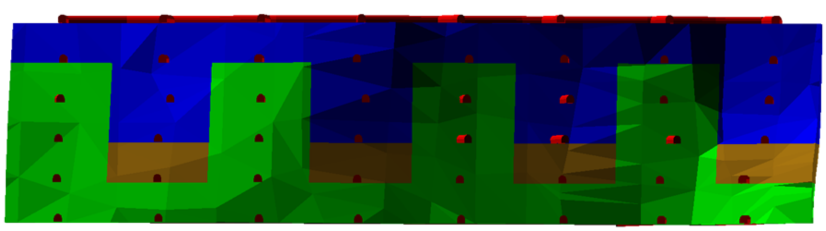
Mine and backfill Primaries with cemented paste fill suitable for sill pillar



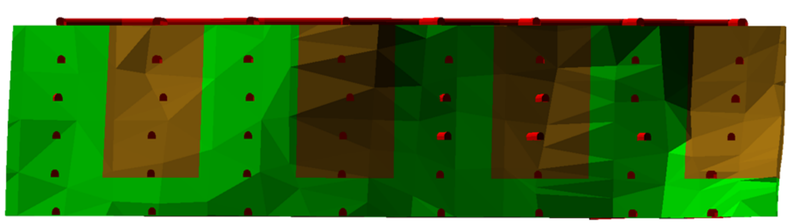
Mine and backfill Primary stopes on second level.



Mine and backfill primary stopes on third level, and secondary stopes on first level.



Continue mining primary stopes. Mine and backfill secondary with ROM waste or cemented fill



Source: JDS 2014

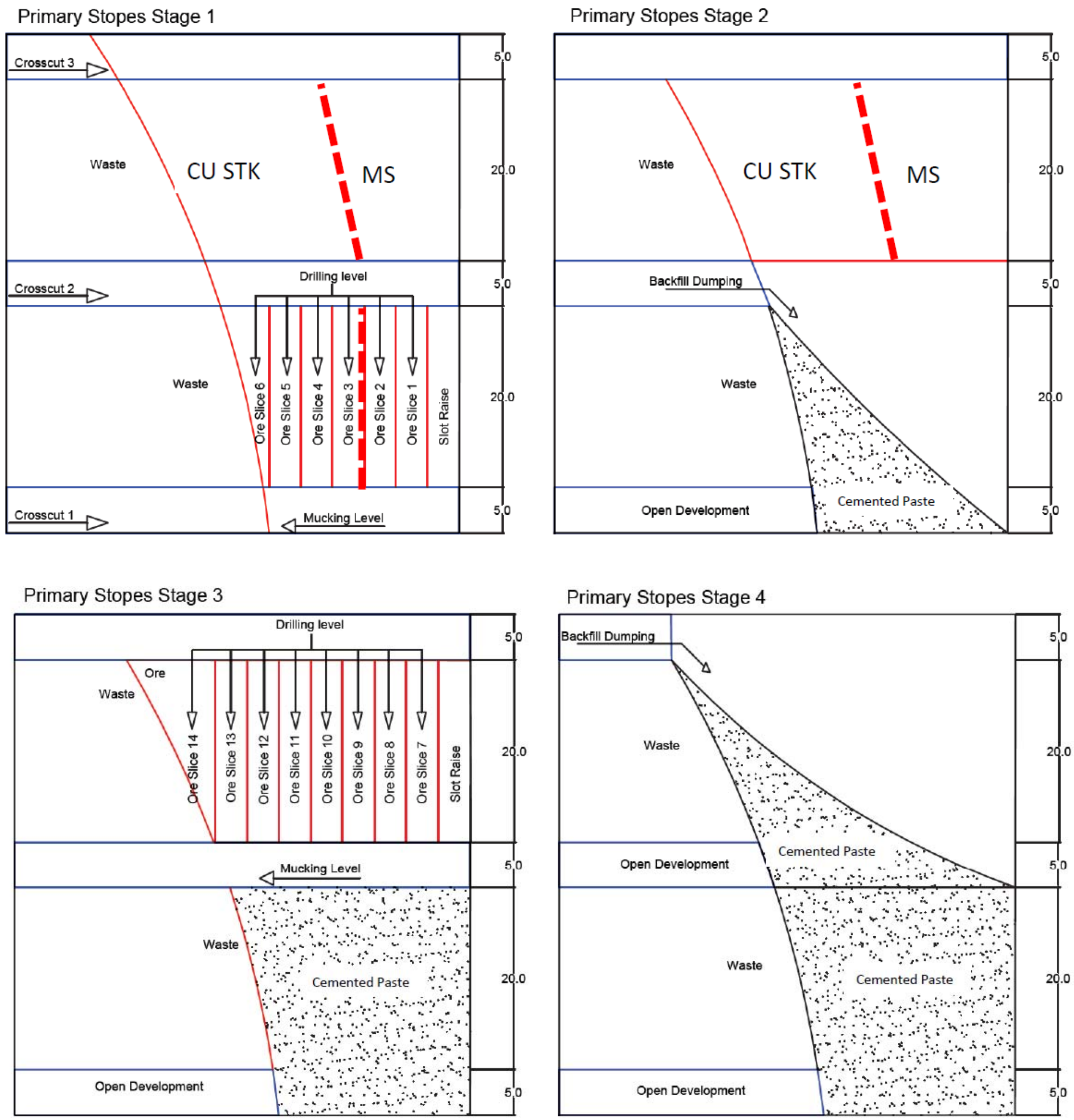
When mining under paste fill (e.g., mining up to a cemented paste sill pillar) the undercut paste spans will not exceed 7 m.

16.8.2 Mineral Zone Sequencing

It is anticipated that production will need to be scheduled such that the L2MS, CSZ, and UW-MS zones are mined separately. Level development will access the zones on the footwall, crosscutting the CSZ first, then the UW-MS and L2MS. The hanging wall should be taken first to avoid potential slabbing of the hanging wall material into the footwall muck pile. JDS recommends drilling production fans from the hanging wall to the footwall, loading and blasting only those intersecting the hanging wall mineralized zone. Once this material has been mucked out, the remaining blast holes are loaded and blasted into the sub level.

Grade control will be extremely important in drilling off two parallel mineralized zones separately. It is not known what metallurgical performances are lost by blending different mineralized materials, but it's anticipated that production over-break (blasting beyond the designed stopes) from one zone to another may have adverse effects on recovery. A metallurgical test program of blending CSZ, UW-MS, and L2MS material should be carried out to assess potential recoveries of taking the entire hanging wall to footwall as one stope, and to determine potential recovery losses from production over-break between mineralized zones. For this study, JDS has assumed clean separation of mineral zones. The parallel zone mining sequence is shown in Figure 16.11.

Figure 16.11: Stoping Parallel Zones



Source: JDS 2014

16.9 Mine Ventilation

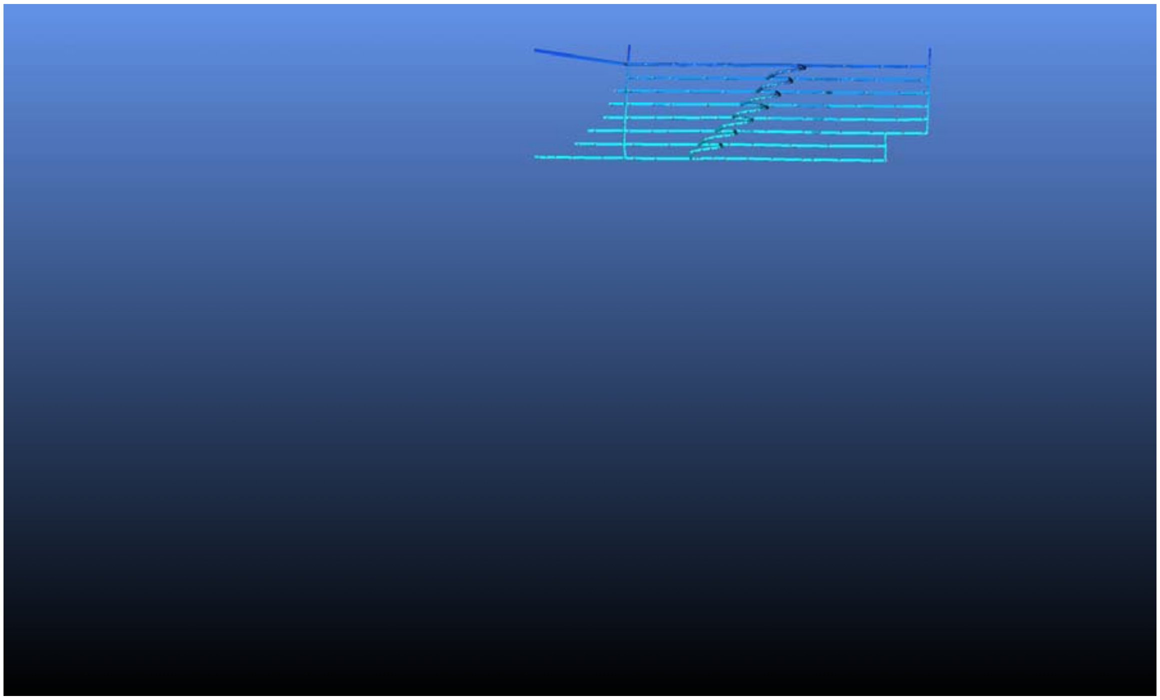
Ventilation simulations were performed using Ventsim™ Visual 3 software (Ventsim). Mine plan design lines were exported from Maptek Vulcan™ software and imported to Ventsim, where ventilation factors such as heading size and friction factors were applied to create a 3D volumetric network.

The ventilation network was designed in three major mine stages.

1. Stage one: Pre shaft;
2. Stage two: Initial shaft sinking; and
3. Stage three: Shaft extension.

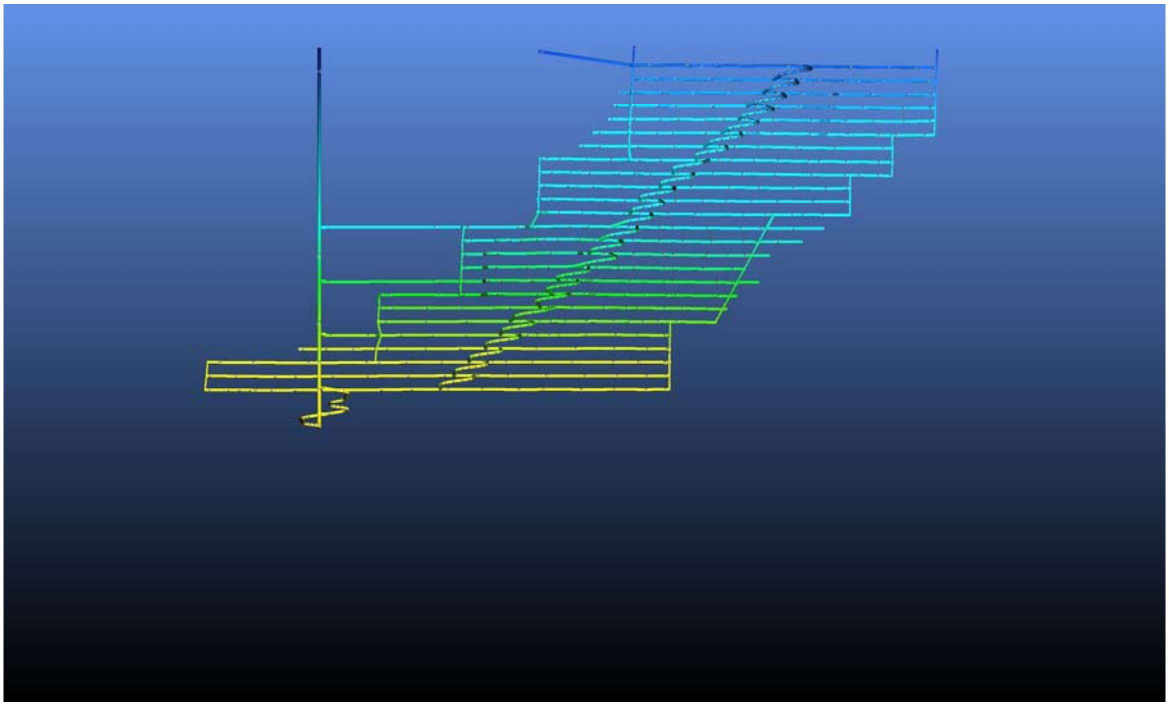
Figure 16.12, Figure 16.13, and Figure 16.14 below outline the three ventilation network stages.

Figure 16.12: Ventilation Network Stage 1



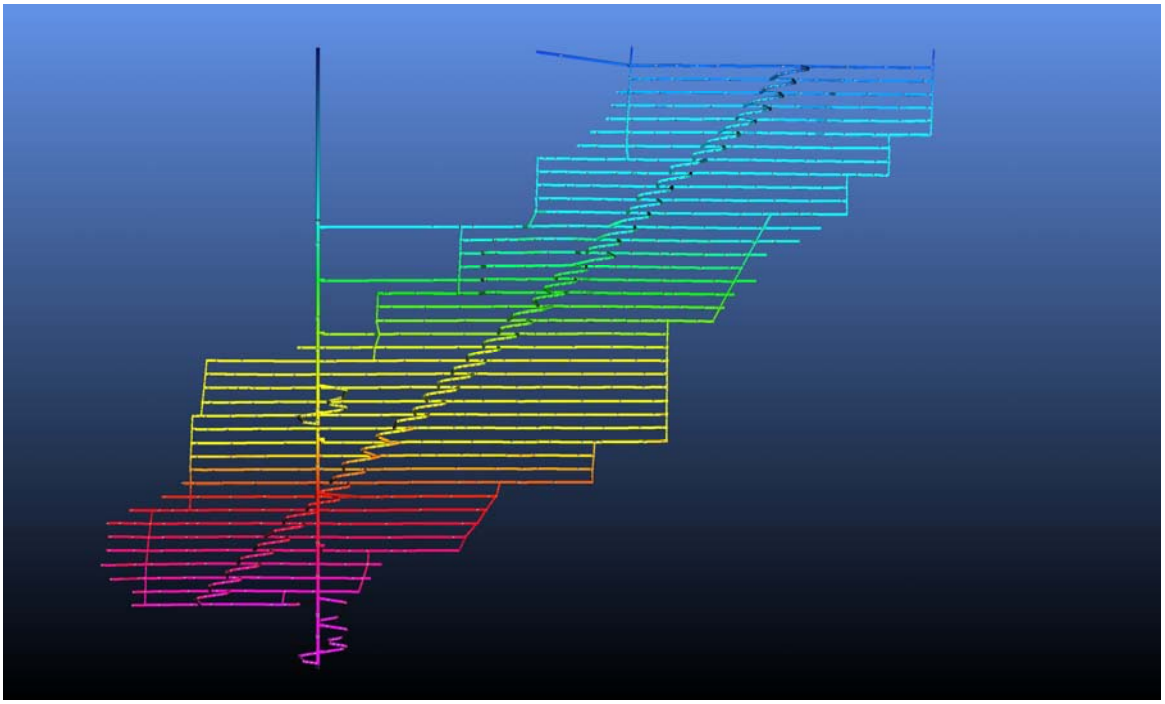
Source: JDS 2014

Figure 16.13: Ventilation Network Stage 2



Source: JDS 2014

Figure 16.14: Ventilation Network Stage 3



Source: JDS 2014

Ventilation requirements were based on the mine plan equipment operating horsepower, which is shown below in Table 16.11.

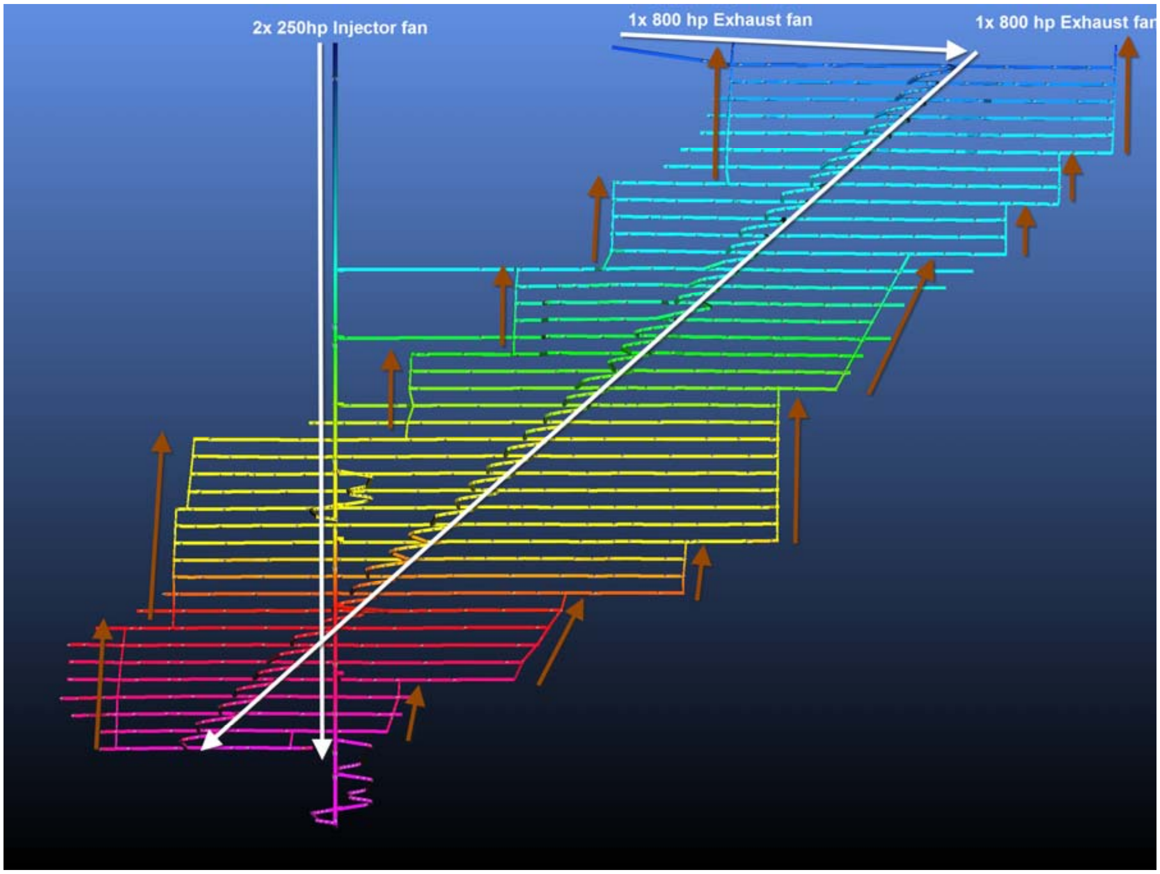
Table 16.11 Ventilation Requirements

Description	Hp	Quantity	Load Factor	Loaded Hp	Cfm Per Hp	Cfm Required
Jumbo 2 Boom	83	2	25%	21	100	4,150
Jumbo 1 Boom	241	1	25%	60	101	6,085
LH Drill Large	149	3	25%	37	102	11,399
LH Drill Narrow	99	1	25%	25	103	2,549
Bolter	148	2	40%	59	104	12,314
50t Truck	691	5	60%	415	105	217,665
3.7 cu.m LHD	165	2	75%	124	106	26,235
7.0 cu.m LHD	268	6	75%	201	107	129,042
Scissor Lifts	147	3	40%	59	108	19,051
ANFO Loader	147	2	40%	59	109	12,818
Boom Truck	147	1	40%	59	110	6,468
Mechanic Vehicles	125	2	40%	50	112	11,200
Sub Total						458,976
Contingency @ 10%						45,898
Grand Total					**cfm**	**504,874**
					m^3/s	**238.27**

Source: JDS 2014

The ventilation networks were designed such that the main ramp and shaft remain in fresh air at all times. In order to achieve this, the life of mine (LOM) ventilation network requires two 250 hp, 78.75” injector fans blowing fresh air into the shaft. These fans will operate in parallel to supply 134 m^3/s of fresh air. In addition, two exhaust raises will skirt the extents of footwall drives on each level to pull dirty air from the production levels to surface. Each exhaust raise will be equipped with one 800 hp, 72” fan to pull 120 m^3/s each. The exhaust fans will pull more air than the shaft provides, which will create suction through the ramp, drawing approximately 100 m^3/s of fresh air from surface to the closest open production level. A schematic of the LOM mine ventilation network and fan locations is shown below in Figure 16.15.

Figure 16.15 Mine Ventilation Schematic



Source: JDS 2014

Each level will be equipped with regulators on the exhaust raises and careful attention will need to be paid to prevent unnecessary recirculation of air.

In order to de-risk the operation, it may be beneficial to select two smaller hp fans to run in parallel on the exhaust raises, rather than one large fan. This way a spare exhaust fan can be purchased and installed on each raise, and two operate while leaving one spare. This would also allow for fans to undergo regular maintenance while not disrupting airflow quantities underground.

It is estimated that these four fans will require approximately 1.5 megawatts to operate.

Secondary ventilation requirements have been estimated based on the use of 30 hp development fans. Ten fans have been budgeted for use underground, in areas that the main fan circuits cannot reach, such as footwall drive development, ramp development, and longitudinal production.

16.10 Mine Air Heating

With average temperatures as low as -28°C in winter, McIlvenna Bay will require mine air heating. Given access to line power, it is recommended to utilize electric mine air heaters. Although more expensive to purchase, electric mine air heaters are more efficient and cost much less to operate than an equivalent propane fired heater. At McIlvenna Bay, it is estimated that electric heating can save up to $3 million per year in heating costs.

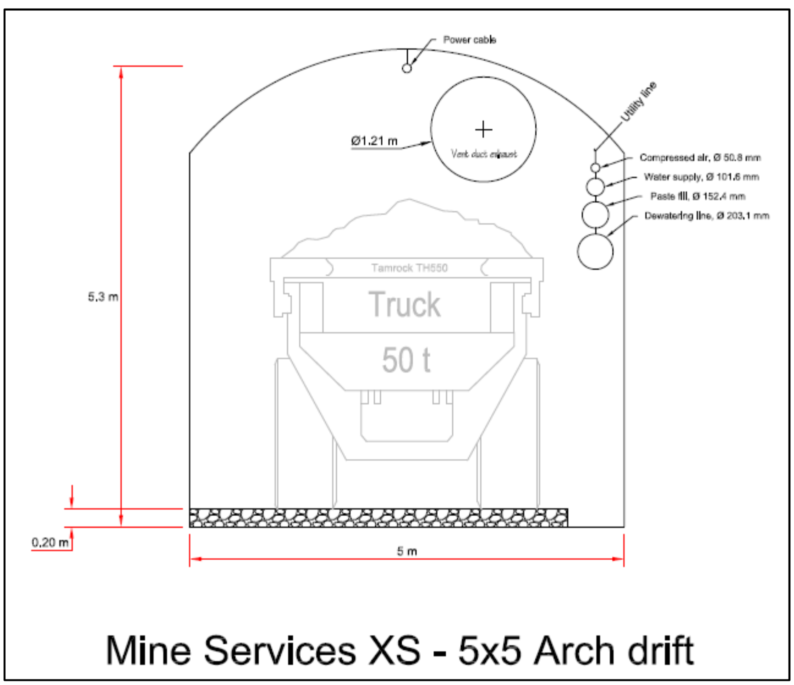
Two 4,160 volt mine air heaters will be installed on site, one 4,800kw unit at the shaft, and one 3,725kw unit just inside the ramp entrance. Together, these heaters will maintain a minimum of +2°C underground year round.

16.11 Underground Mine Services

16.11.1 Ramp

The ramp will be designed at 5.0 m x 5.3 m to accommodate fully loaded 50 t haul trucks and 1.2 m round vent ducting. Footwall drifts will also be driven at 5.0 m x 5.3 m to allow haul trucks access to the stope crosscuts. Crosscuts will be driven flat back style 4.0 m x 4.0 m to accommodate remote LHD entry. The Figure 16.16 depicts a typical ramp cross section.

Figure 16.16: Typical Ramp Cross Section



Source: JDS 2014

16.11.2 Shaft

The shaft was designed to suit an operation of 5,000 tpd. Reference projects that were used to size the shaft at McIlvenna Bay are shown in Table 16.12.

Table 16.12: Shaft Design References

Location	Name	Shaft Size (m^2)	Shaft Depth (m)
Saskatchewan	Millenium	33.18	1,450
Ontario	Young Davidson	33.18	755

Source: JDS 2014

It is estimated that McIlvenna Bay will require a 6.2 m diameter shaft with a 1,500 kW motor to hoist 30 t skips at 250 tph. The shaft will be capable of delivering 6,000 t to the surface per day, covering 5,000 tpd of mineralized material plus 1,000 tpd of waste.

The deposit dips at 60-70^o and also plunges at 45^o, which makes sinking a shaft close to the deposit on all levels difficult. A minimum 100 m was established between the resource and the shaft for geotechnical considerations, but at depth this distance increases upwards to 400 m as the mineral zones dips away from the shaft.

Shaft sinking is very expensive, and it is recommended that the shaft be constructed in two stages. The initial sinking will provide the mine with enough potentially mineable resources to sustain six or more year's production, allowing a delay of capital spending. The initial shaft will be sunk 735 m below surface, deep enough that a shaft extension of 430 m can take place without disrupting normal operations. Total shaft length will be 1,165 m. The downfall of extending an existing shaft is the need to build two sets of shaft bottom infrastructure, such as conveyors, crushing chambers, and load out systems. However, the capital savings in delaying the full shaft sinking is beneficial for project pay-back timing.

The proposed shaft will include:

- Hoist and accessories;
- Headframe and hoistroom;
- Shaft sinking;
- Shaft loading pockets; and
- Underground Crusher and installation.

16.11.3 Mineralized Material & Waste Passes

Shaft levels are planned at 100 m spacing. Each shaft level will be equipped with a dumping station with sloped grizzly to feed mineralized material and waste passes to the underground crusher station at shaft bottom. Intermediate passes will also be installed on every sub-level to feed material to the nearest shaft level for re-handle to the main passes.

16.11.4 Crusher

A jaw crusher will be located on surface during shaft construction, and then at shaft bottom once the shaft is complete. It will be necessary to lease a mobile crusher while the underground crusher is installed.

16.12 Pastefill Distribution System

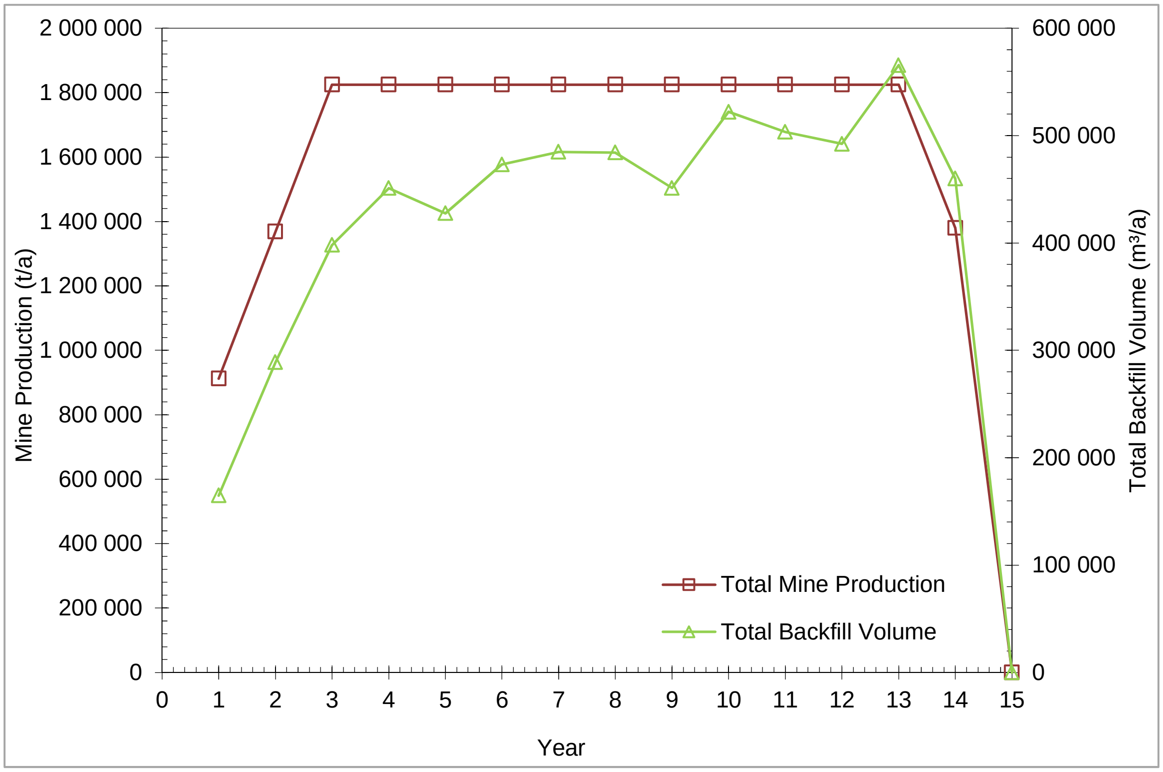
The following section is taken from Patterson & Cook's paste backfill plant and underground distribution report (JDM-32-0128 Report Section).

Paterson & Cooke (P&C) completed a preliminary economic assessment (PEA) level paste backfill plant and underground distribution for the Project (JDM-32-0128 R03). The proposed pastefill distribution system transports the paste from the surface backfill plant to the underground stopes through a pipeline system. The design of the backfill plant was conducted to meet the future mine production rate of 5000 tpd and provide a backfill with a minimum unconfined compressive strength of 1.0 MPa after 28 days. P&C have based the hydraulic design on the criteria developed in report JDM-32-0128 R01. The paste rheology and strength characteristics ware benchmarked against P&C historical internal database. A summary of the paste distribution system follows below.

16.12.1 System Capacity

The sizing of the backfill plant and underground reticulation system was based on the latest production and tailings summary provided by JDS. The total mine production and associated required backfill volume over the life of mine is presented in Figure 16.17. The required backfill volume is based on an average mineralized material density of 2.97 t/m³ and a void replacement factor which ramps up from 53.5% in Year 1 to 98.6% in Year 14.

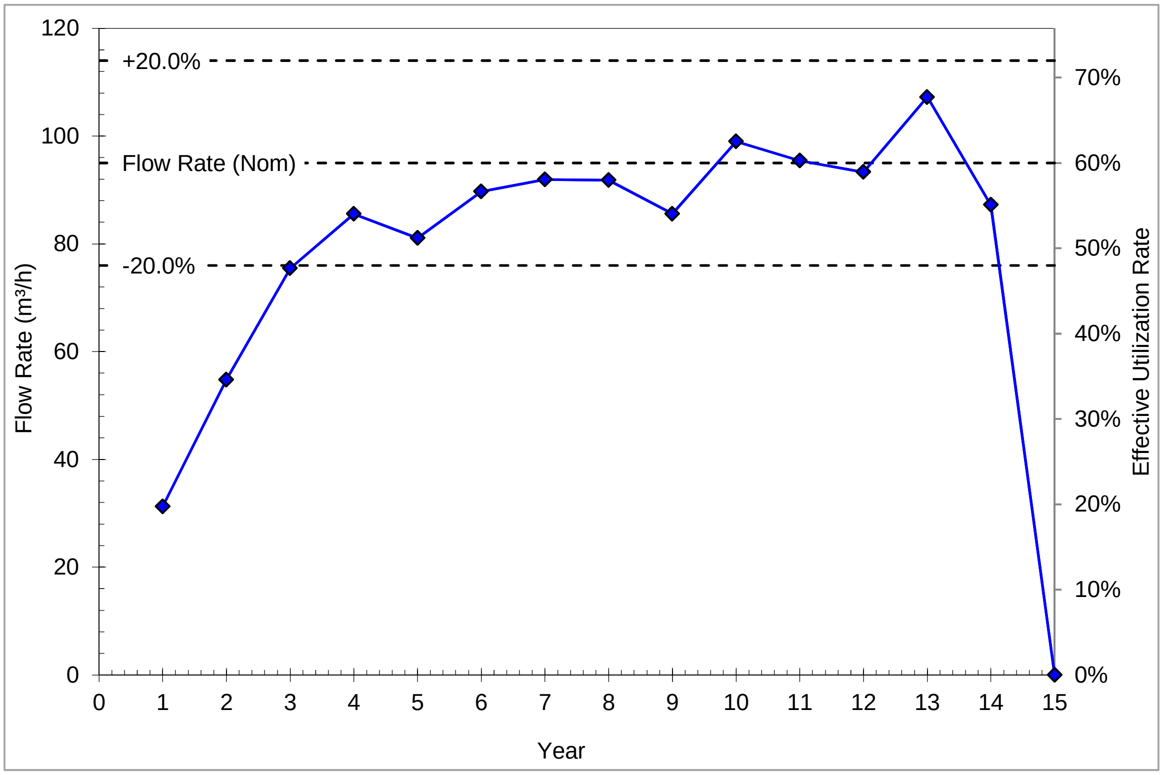
Figure 16.17: Total Mine Production and Required Backfill Volume



Source: Patterson & Cooke 2014

The system capacity is based on operating one pipeline system at a time. The utilization rate of a backfill plant is generally lower than a typical mine mill since there is a considerable amount of time lost to start-up and shutdown as well as waiting for stopes to be ready, pipe movements etc. Assuming an average backfill utilization rate of 60%, the nominal design flow rate per pipeline is 95 m³/h (Figure 16.18). The backfill plant will operate at a maximum utilization rate of 68% during Year 13. During the first couple of years of operation while the backfill plant ramps up to full production, the backfill plant will operate at lower utilization rates rather than operate continuously at reduced throughput.

Figure 16.18: Stope Total Production and Required Pastefill Volume



Source: Patterson & Cooke 2014

16.12.2 Distribution System Design

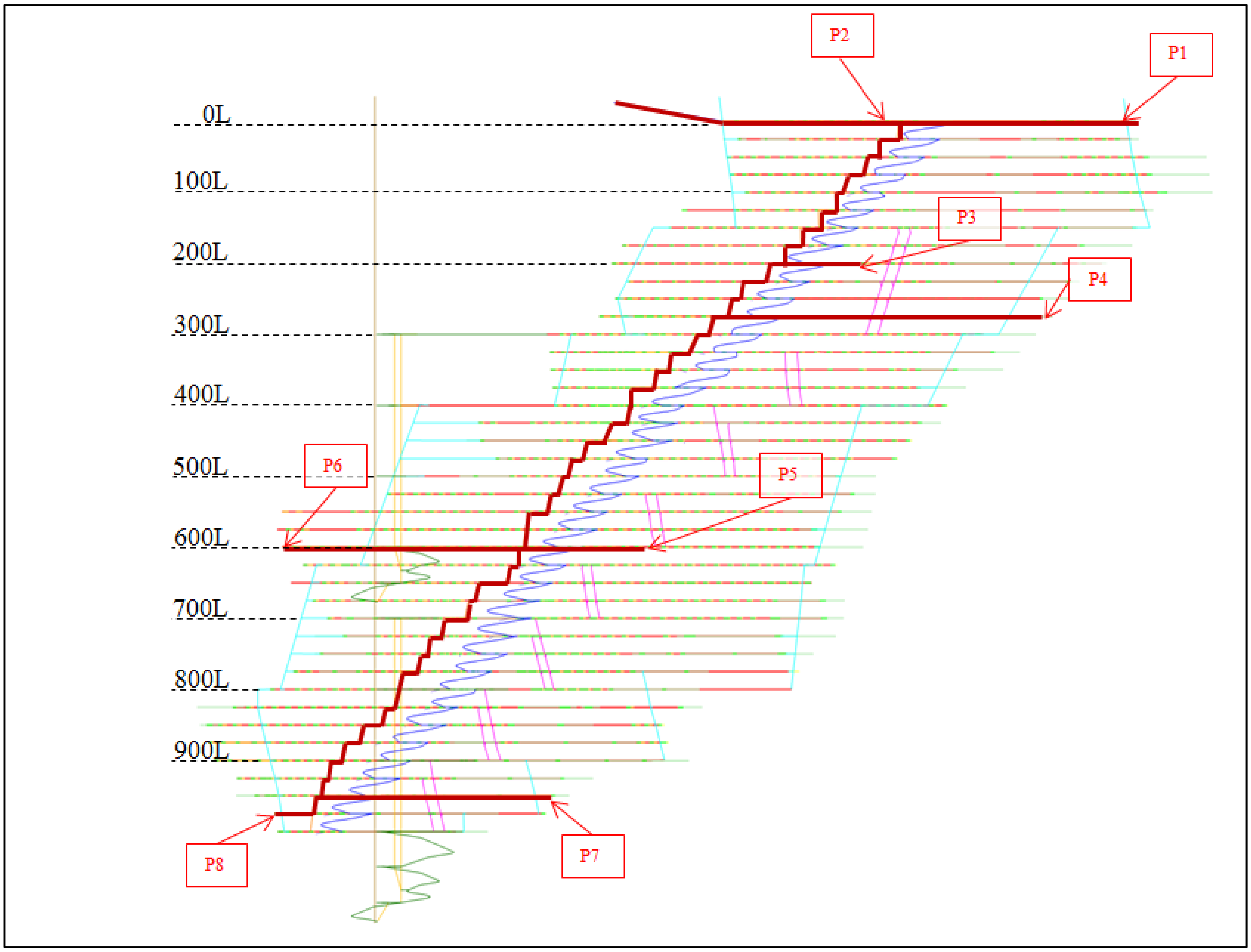
Due to the shallow location and length of the mineral deposit, PD pumps are required to pump the backfill material underground to the stopes. The backfill plant will be situated at the process plant next to the final tailings thickener to allow easy access to the final tailings and plant infrastructure. The backfill plant will use the decline (ramp) to access the underground workings. A site plan indicating the approximate location of the backfill plant and overland pipeline to the decline is provided in drawing JDM-32-0128-00-L01, found in the Appendix C.

The pipe routing for the underground distribution system was developed with consideration given to site conditions and hydraulic modelling. A schematic diagram of the underground workings is presented in Figure 16.19 with pressure model points indicated by “P”. Access for the backfill pipelines to the underground working is via a decline (ramp) to 300l. Two dedicated inter-level boreholes (duty and standby) are required from 0l to 975l.

The boreholes from 0L to 400L will be cased with carbon steel pipes lined with a ceramic epoxy liner. The boreholes from 400L to 975L will be cased with unlined carbon steel pipes. All drift will incorporate unlined steel pipes. HDPE pipe will be used for short distances within the cross-cuts and stopes as it is flexible and generally quicker to install.

The selection of the pipeline size is primarily based on ensuring that during normal operation, the friction losses are minimized. In addition, a minimum velocity is specified to reduce long pastefill transfer times. Based on the design flow rate, a 200 NB (8 inch) is recommended for the system. The flow velocities in the 200 NB Sch 80 steel mainline pipes are between 0.7 and 1.1 mps.

Figure 16.19: Schematic Diagram of Underground Distribution System



Source: JDS 2014

All above and underground steel slurry pipes are based on ASME B31.10 dimensions while all HDPE pipes are based on ASTM F714. The pressure rating for each pipe type are calculated according to ASME B31.3, including allowances for coupling grooving, wear and mill tolerance.

Hydraulic modeling showed that this system will provide paste to the stopes at a nominal yield stress of 235 Pa with a range of 189 to 275 Pa. This equates to a paste mass concentration of 78.3%m (ranging from 77.8 to 78.7%m).

Piping specified for this distribution system is A106 Gr. B. The schedule of the pipe varies with the pressure rating of the area. The overland pipeline from the backfill plant to the ramp and unlined borehole casing are Schedule 120 while the rest of the ceramic lined pipes and drift piping are of Schedule 80. Victaulic couplings have been used as the connection method.

Cemented backfill will be pumped by a hydraulic PD pump from the backfill plan to the decline shaft (ramp) to access the underground workings. Based on the hydraulic analysis, a maximum operating pressure of 6 MPA is expected for the hydraulic piston pump. Allowing for potential process upsets, a 7.5 MPa pump is recommended.

16.12.3 Backfill Plant

The process design on surface is strongly influenced by the requirements of the underground (both mining and reticulation) and the material properties of the feed. It is expected that the properties of the disc filtered tailings will remain fairly constant and the use of a continuous mixing process is therefore included in the design. Minor changes to the tailings properties, binder content, water content etc. will be controlled by specific sampling and monitoring measures included in the design to ensure a consistent backfill is produced.

The Process Flow Diagram (PFD) for the backfill plant is presented on P&C Drawing JDM-32-0128-00-F01 attached to the Appendix B. The backfill plant will be fed with tailings pumped from the final tailings tank located at the process plant. Initial analysis based on P&C's historical database suggests that the minimum mass concentration must be 65% m; however, this must be confirmed with test work. The backfill plant will require a constant feed of thickened tailings with a set mass concentration. The higher the mass concentration, the more benefit in reducing the overall binder requirements.

The thickened tailings will reports to a filter feed tank in the backfill plant area at a nominal rate of 130 m^3/h (i.e. 148 dry t/h). The filter feed tank is sized to provide buffer capacity for the control of the system and to allow for time to start up and shut down the distribution system. The tank is sized to provide a retention time of four hours.

From the filter feed tank thickened tailings is fed to two disc filters (one duty / one standby) which are sized to operate at a filtration rate of 550 kg/m^2/h. Based on historical data, it has been assumed the filter will produce a final filter cake product with a moisture content of 21%. The filtered tailings are continuously fed to a continuous mixer.

Dry binder is screw fed from two storage silos (one duty / one standby) and discharged into the mixer. A binder content of approximately 5% will be added depending on the underground distribution point and recipe requirements.

Trim water is fed to the mixer to achieve a design mass concentration between 77% and 78.5%m depending on the backfill deposition point. The bypassing of some thickened tailings directly to the mixer is a refinement that could be added to the process depending on the future test results.

The mixer is a continuous twin shaft mixer fitted with 2 x 55 kW electric motors. Filter cake, binder and trim water will be mixed to create a homogeneous backfill blend. The mixer has an inner volume of 5.5 m^3 and a mixing capacity of 120 m^3ph (peak) with 165 second retention time. The mixing action is performed by mixing arms and paddles which are hydrodynamically designed to reduce wear and promote optimum mixing. The mixer includes a high impact washing system.

The backfill from the mixer overflows into a paste hopper and is gravity fed into the suction side of one of the two piston positive displacement pumps. The piston pumps (one duty and one standby) pump backfill on a continuous basis down the surface decline for deposition underground at the nominal design flow rate of 95 m^3ph.

16.13 Mine Equipment

Underground equipment requirements were calculated based on equipment hours needed to meet production and development rates in the mine plan. Haulage cycles during ramp production years accounted for distances from footwall drifts, up the ramp, and to a surface stockpile. During shaft operations, haulage cycles included haulage from the footwall drifts and to the nearest material pass station.

Mucking and hauling cycles are based on a fixed distance between stopes and a truck, or re-muck bay and a truck. Non critical auxiliary equipment quantities were estimated based on the size of operation, and/or factored from other equipment requirements. Table 16.13 below shows equipment requirements during mine construction, year one during ramp production, and year five during full scale mining.

Table 16.13: Mobile Equipment Fleet

Underground Equipment	Year -1	Year 1	Year 5
Jumbo 2 Boom	1	3	4
Jumbo 1 Boom	0	1	1
LH Drill Large	0	2	3
LH Drill Narrow	0	1	1
Bolter	1	2	2
50t Truck	1	5	7
3.7 cu.m LHD	1	1	1
7.0 cu.m LHD	1	5	8
Scissor Lifts	3	3	4
ANFO Loader	1	2	3
Boom Truck	1	1	1
Fuel/Lube Truck	1	1	1
Mechanic Vehicles	2	2	2
Personnel Carriers (Kabota)	4	6	8
Shotcrete Machine	1	1	1
Grader	1	1	1
Fuel Sat	0	0	1

Source: JDS 2014

16.14 Mine Personnel

McIlvenna Bay is located approximately 100 km by road from the towns of Creighton, Saskatchewan andFlin Flon, Manitoba. It is anticipated that the workforce will bus to site daily from Creighton/Flin Flon. A total mining workforce of 211 will be employed at McIlvenna Bay. Underground mine labour is shown in Table 16.14.

Table 16.14: Underground Mine Labour

Mining Operations	Rotation	Total	Day Shift	Night Shift	Offsite
Mine Superintendent	4&4	1	1		
Mine Captain	4&4	2	1	1	
Mine Supervisor/Shift Boss	4&4	8	2	2	4
Production Drill Operator	4&4	10	3	2	5
Jumbo Operator	4&4	10	3	2	5
Ground Support/Bolter/Shotcrete	4&4	4	1	1	2
Development Service	4&4	8	2	2	4
Blaster	4&4	10	3	2	5
LHD Operator	4&4	28	7	7	14
Truck Driver	4&4	28	7	7	14
Utility Vehicle Operator/Nipper	4&4	10	3	2	5
Shaft Skip tenders/Shaftmen	4&4	8	2	2	4
Hoistmen	4&4	4	1	1	2
Mining Operations - Total		**131**	**36**	**31**	**64**
Paste Backfill Plant					
Paste Backfill Plant Operators	2&1	8	2	2	4
Backfill	4&4	8	2	2	4
Paste Backfill - Total		**16**	**4**	**4**	**8**
Mining Maintenance					
Mine Maintenance Superintendent	5&2				
Mine Maintenance Supervisor/Shift Boss	4&4	4	1	1	2
Mechanical & Electrical General Foreman	4&4	2	1	1	
Maintenance Planner	4&4	2	1	1	
HD Mechanic/Welder, Mobile	4&4	16	4	4	8
Electrician	4&4	8	2	2	4
Apprentice	4&4	8	2	2	4
Dry/Lapman/Bitman	4&4	4	1	1	2
Mine Maintenance - Total		**44**	**12**	**12**	**20**
Mining Technical Services					
Senior Mine Engineer & Planner	4&4	2	1	1	
Mine Ventilation/Project Engineer	4&4	2	1	1	
Geotechnical Engineer	4&4	2	1	1	
Sr. Mine Technician	4&4	2	1	1	
Surveyor/Mine Technician	4&4	4	2	2	
Chief Geologist	5&2	1	1		
Production Geologist	4&4	2	1	1	
Chief Mining Engineer	5&2	1	1		
Geotechnical Technician/Sampler	4&4	4	2	2	
Technical Services - Total		**20**	**11**	**9**	**0**
Grand Total UG Mining		**211**	**63**	**56**	**92**

Source: JDS 2014

16.15 Mine Production Plan

The production schedule begins with ramp supplied material. It is estimated that in the first year of production the average mining rate will not exceed 2,500 tpd, due to mine level development and ramp congestion. In Year 2, the shaft is commissioned and material crushing and extraction from the mine is done through the shaft infrastructure. Production increases in Year 2 from 2,500 tpd to 3,750 tpd. In Year 3 and beyond, production runs at full capacity at 5,000 tpd until Year 14 when the resource is depleted.

Table 16.15 outlines the mine plan production summary highlights.

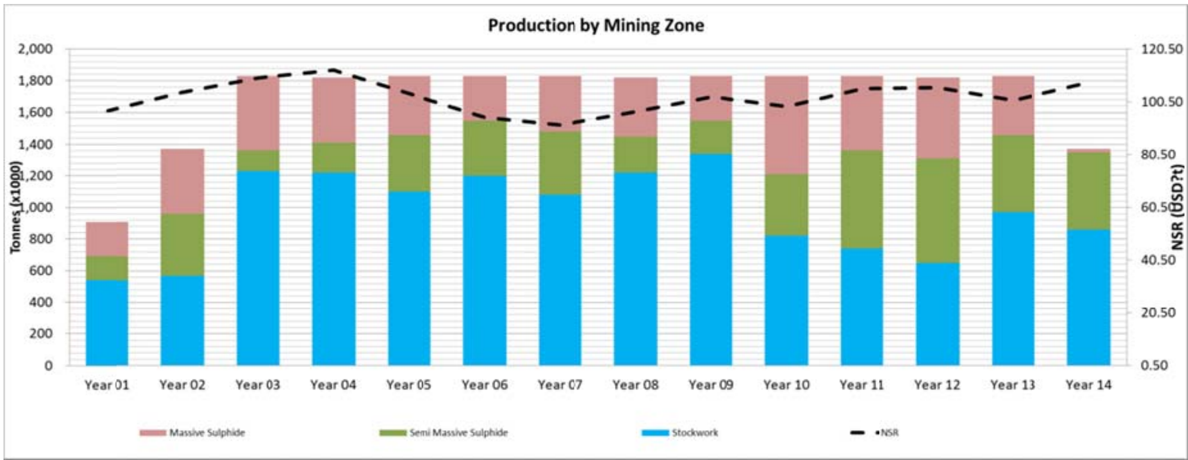
Table 16.15: Mine Production Summary

Parameter	Unit	Total	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09	Year 10	Year 11	Year 12	Year 13	Year 14
Production	Mt	**23.7**	0.91	1.37	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.83	1.38
Rate	ktpd	**4.6**	2.5	3.8	5	5	5	5	5	5	5	5	5	5	5	3.8
Au	g/t	**0.4**	0.3	0.4	0.5	0.5	0.4	0.3	0.4	0.5	0.4	0.4	0.4	0.3	0.4	0.4
Ag	g/t	**14.8**	14	18	15	15	17	12	14	12	11	19	19	15	13	13
Cu	%	**1.2**	1	1	1.2	1.3	1.2	1.2	1.1	1.2	1.3	1	1.1	1	1.2	1.5
Pb	%	**0.2**	0.1	0.2	0.2	0.2	0.2	0.1	0.2	0.1	0.1	0.2	0.2	0.1	0.1	0.1
Zn	%	**2.4**	2.7	3.3	2.2	2.1	2.1	1.7	1.9	1.8	1.4	3.3	3.4	3.8	2.2	1.3
NSR	CDN/t	**106**	99	108	113	116	107	97	94	100	105	103	110	110	104	111

Source: JDS 2014

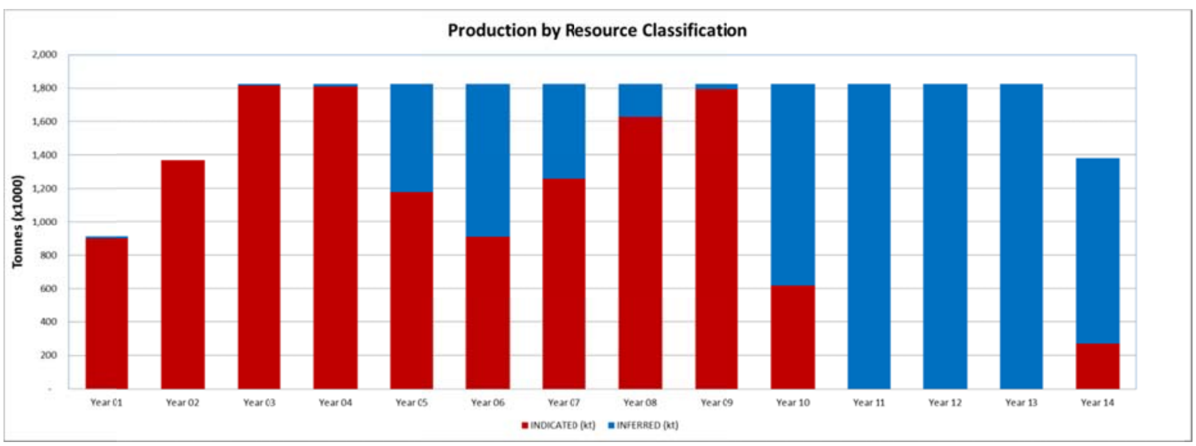
The mine plan has been scheduled to prioritize high grade levels where possible. A benefit of shaft access mining is the ability to target high grade spots within the resource faster than ramp which is limited to the area it uncovered annually. Figure 16.20 below provides the production schedule through all three mineralization zones and Figure 16.21 shows annual mining by resource class. Table 16.14 details annual waste development quantities.

Figure 16.20: Production Schedule by Zone



Source: JDS 2014

Figure 16.21 Production Schedule by Resource Classification



Source: JDS 2014

16.15.1 Underground Waste Production

Table 16.16: Waste Development Schedule

Development			Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Main Infrastructure	km	**39.4**	3.5	4.4	3.3	4.5	3.0	3.0	2.9	2.3	2.5	3.4	1.7	2.3	2.2	0.4	0.0
Alimak Raise	km	**4.3**	0.2	0.6	0.2	0.6	0.4	0.2	0.3	0.3	0.5	0.6	0.1	0.1	0.1	0.0	0.0
Production Ore	km	**27.0**	0.0	1.7	1.5	2.4	2.4	4.3	2.9	1.3	1.3	0.8	2.4	1.4	2.2	2.0	0.4
Production Waste Drifting	km	**19.5**	0.0	1.3	1.0	1.7	2.1	3.5	1.1	1.1	1.0	0.6	1.3	1.0	1.7	1.6	0.3
Secondary Infrastructure	km	**3.7**	0.1	0.4	0.3	0.5	0.3	0.1	0.4	0.3	0.2	0.2	0.1	0.3	0.3	0.0	0.0
Auxiliary Development	km	**0.4**	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
Raisebore	km	**1.5**	0.0	0.0	0.7	0.0	0.0	0.0	0.0	0.0	0.0	0.8	0.0	0.0	0.0	0.0	0.0
Shaft	km	**1.5**	0.3	0.6	0.1	0.0	0.0	0.0	0.0	0.3	0.1	0.0	0.0	0.0	0.0	0.0	0.0
Total		**97.4**	**4.2**	**9.1**	**7.2**	**9.7**	**8.3**	**11.2**	**7.7**	**5.7**	**5.6**	**6.4**	**5.7**	**5.2**	**6.5**	**4.1**	**0.7**
Tonnages																	
Waste Development	m^3	**1,546**	107.3	172.9	128.5	161.8	122.6	140.0	106.2	97.6	98.1	122.4	70.0	83.8	92.1	37.8	4.7
	tonnes	**4,174**	289.7	466.8	346.9	437.0	331.0	378.0	286.8	263.5	264.7	330.6	189.0	226.4	248.6	102.0	12.6
	tpd	**890**	794	1,279	950	1,197	907	1,036	786	722	725	906	518	620	681	280	35
Ore Development	m^3	**433**	0.0	27.3	23.8	38.0	39.1	69.1	46.6	21.5	20.1	12.9	38.5	23.0	34.9	31.8	6.2
	tonnes	**1,280**	0.0	80.9	70.3	112.4	115.7	204.4	138.1	63.7	59.5	38.1	113.8	68.0	103.2	94.1	18.2
	tpd	**310**	0	222	193	308	317	560	378	175	163	104	312	186	283	258	50

Source: JDS 2014

17 RECOVERY METHODS

17.1 Introduction

The metallurgical processing selected for the different types of mineralization, CSZ, L2MS) and UW-MS, were designed to produce copper concentrate, zinc concentrate or a bulk concentrate as final products depending on the type of mineralization fed to the plant.

The 5,000 tpd process plant flowsheet design follows conventional crushing, a semi-autogenous mill with a pebble crushing circuit, a ball mill grinding circuit using cyclones for classification followed by a talc pre-flotation step to remove detrimental talc prior to copper/zinc/bulk flotation. Conventional sequential flotation for the recovery of copper, zinc and bulk concentrates are utilized in this flowsheet. Rougher and scavenger flotation cells are used for zinc, while the copper and bulk circuits have only rougher cells. Regrinding prior to cleaning is required for all ore types. Each of the three types of mineralization requires three stages of cleaning following regrind to produce final concentrates grades.

Each concentrate produced will have a dedicated concentrate thickener and concentrate filter press. Concentrates will be shipped out via bulk trucks. Thickener overflow from each concentrate thickener will be recycled back into its own flotation circuit to prevent cross reagent contamination.

Tailings from talc pre-float and the zinc flotation circuit will be will be sent to a common paste backfill system followed by a tailings thickening. The thickener underflow will be sent for disposal in the tailings management facility.

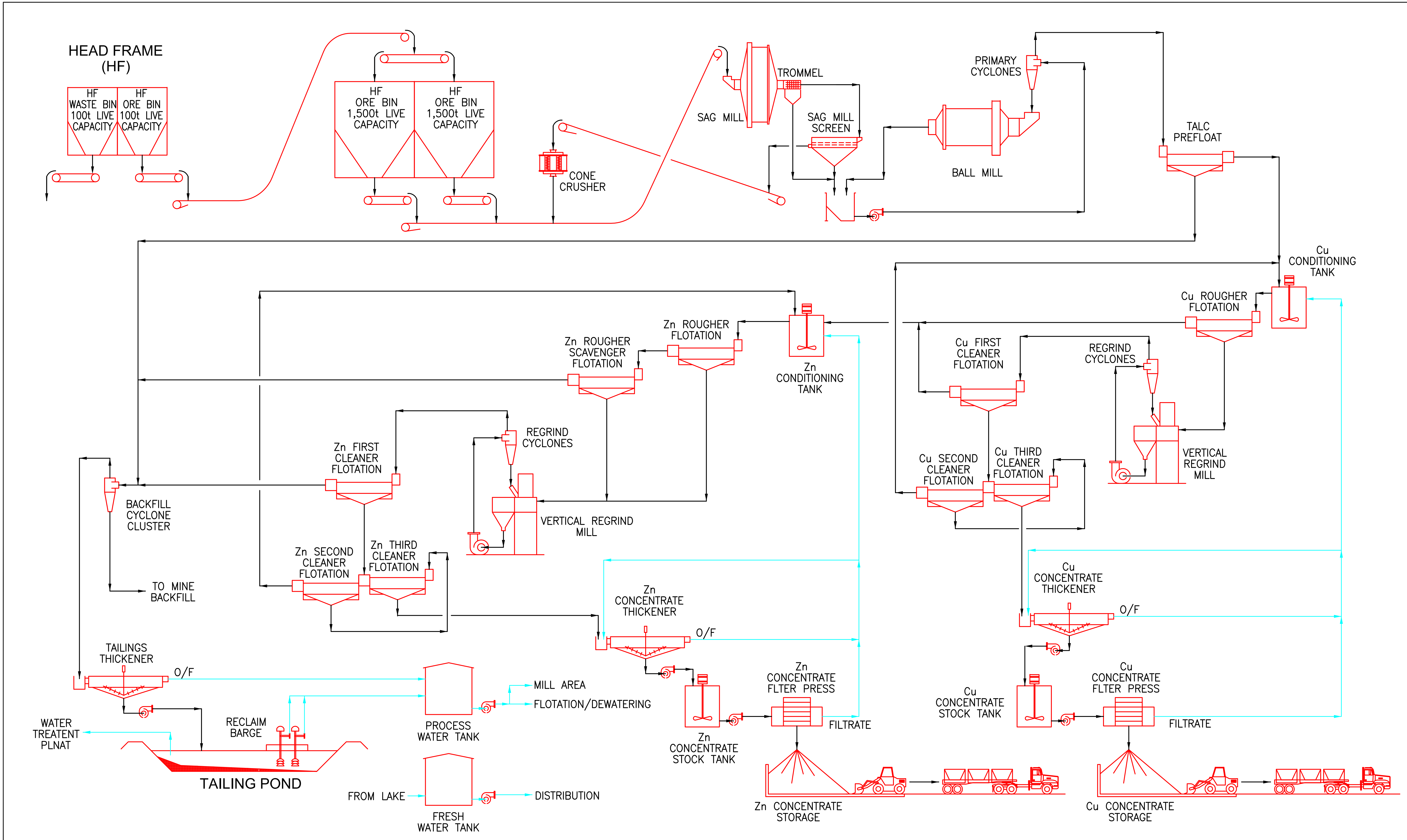
Process water recycled from the tailings thickener overflow and recovered water from the tailings management facility will supply the required process water for the plant. Fresh water from the lake will be used for gland service and reagent preparation.

The 5,000 tpd process plant will consist of the following unit operations and facilities:

- Crushed material (from underground) bins;
- SAG mill incorporating a pebble crushing circuit;
- Ball mill grinding circuit and cyclones for classification;
- Talc pre-flotation;
- Copper/Bulk conditioning tank followed by rougher flotation;
- Copper vertical regrind mill and cyclones for classification;
- Copper three stage cleaner flotation circuit;
- Copper concentrate thickener;
- Copper concentrate filter press;
- Copper concentrate bulk storage and handling;
- Zinc conditioning tank followed by rougher and scavenger flotation;

- Zinc vertical regrind mill and cyclones for classification;
- Zinc three stage cleaner flotation circuit;
- Zinc concentrate thickener;
- Zinc concentrate filter press;
- Zinc concentrate bulk storage and handling;
- Bulk vertical regrind mill and cyclones for classification;
- Bulk three stage cleaner flotation circuit;
- Bulk concentrate thickener;
- Bulk concentrate filter press;
- Bulk concentrate bulk storage and handling;
- Backfill cyclones;
- Tailings thickening;
- Tailings deposition;
- Process water reclamation;
- Reagent preparation facilities; and
- Utilities.

Simplified flowsheets are shown in Figures 17.1 and 17.2 for the processing of the different zone types.



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DRAWN BY: B. Wong	Dec 25 13
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Foran Mining Corp.

McLLVENNA BAY PROJECT
CSZ AND UW/MS ORE
CONCEPTUAL FLOWSHEET

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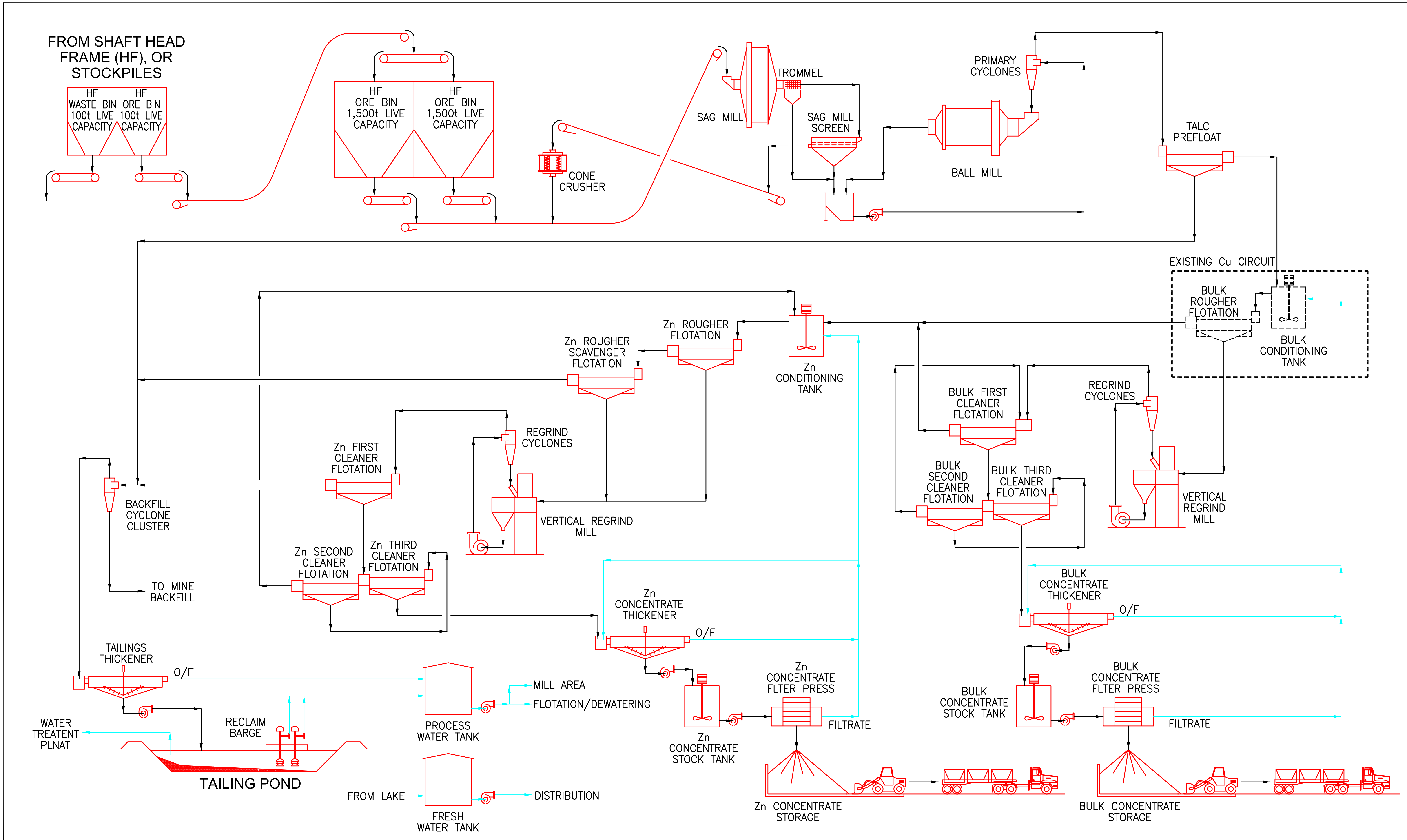
		100-PF-100	



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										REVISIONS		

SECTION:	
SCALE: None	
DESIGN. BY J.D.S.	Dec 25 13
DRAWN BY: B. Wong	Dec 25 13
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Foran Mining Corp.

McLLVENNA BAY PROJECT
MS ORE
CONCEPTURAL FLOWSHEET

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
		100-PF-101	

17.2 Plant Design

The concentrator has been designed to treat copper and zinc bearing material at a design rate of 5,000 tpd (1,825,000 tpa).

The process design parameters for the process plant were based on the results of metallurgical testwork results obtained by ALS Kamloops.

The grinding mills were sized based on the Bond work index data obtained from the testwork.

The flotation circuits were sized based on the times, grades and mass pulls as determined during the laboratory tests and industry practice. Typical plant design parameters have been used in the design of the thickening, and filtration circuits.

17.3 Operating Schedule and Availability

The processing plant will be designed to operate on the basis of two 12-hour shifts per day, for 365 days per year.

The reclaim, grinding, flotation, thickening and filtration utilization will be based a 92% availability. These utilizations will allow for sufficient downtime for scheduled and unscheduled maintenance of the process plant equipment.

18 PROJECT INFRASTRUCTURE

18.1 General

The project envisions construction of the following key infrastructure items:

- Upgraded 18 km long all-season access road from Saskatchewan Provincial Highway #106 to the project site;
- Approximately 95 km of new 138 kV transmission line from Creighton, SK to the project site;
- 25 kVA onsite sub-station fed from the SaskPower grid;
- Temporary construction camp;
- Process plant;
- Truck shop and warehouse;
- Bulk explosives storage and magazines;
- Bulk fuel storage and delivery system;
- Water treatment system; and
- Firewater tank and system.

18.2 General Site Arrangement

The overall site arrangement is shown in Figure 18.1.

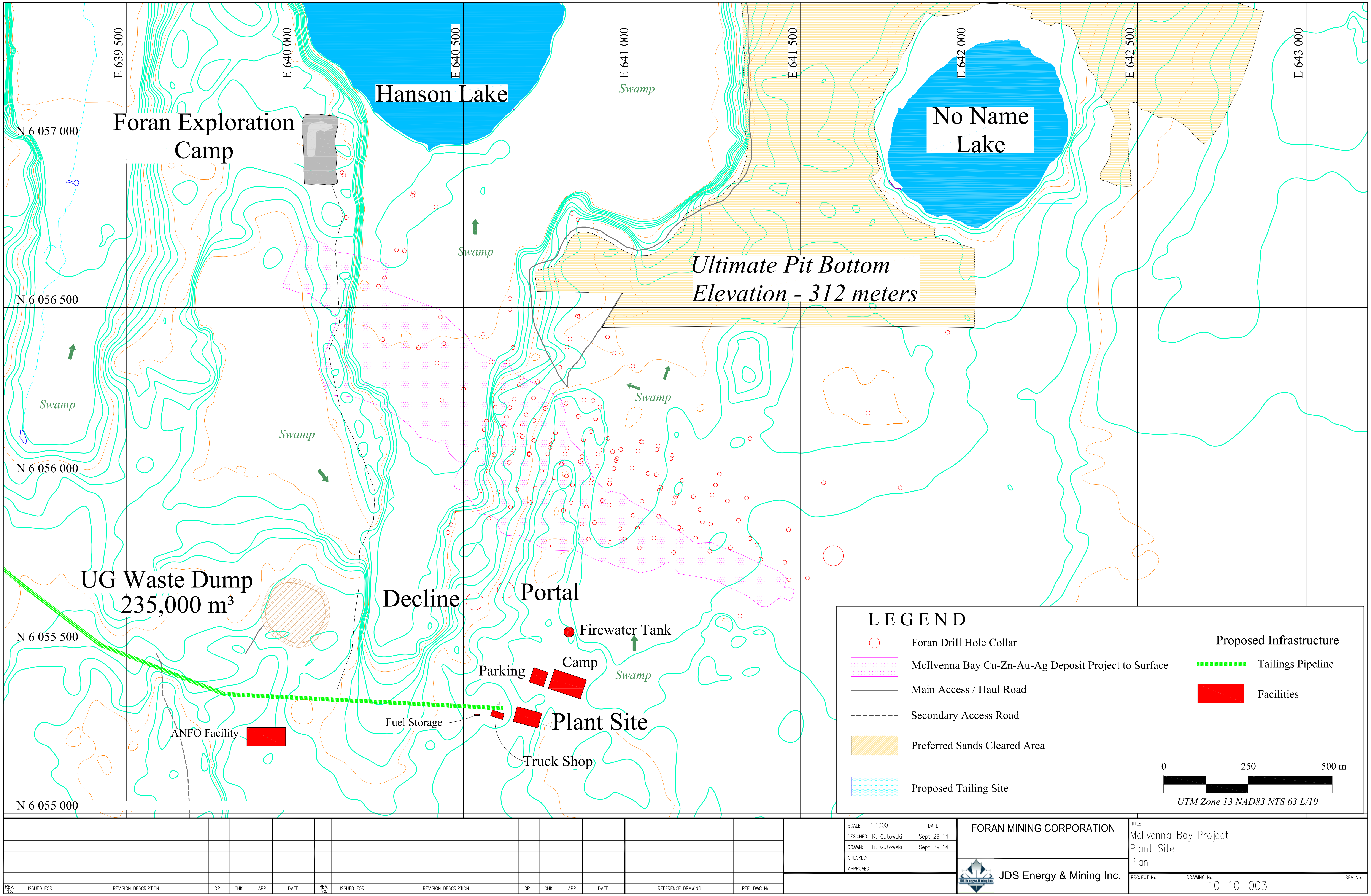
The site has been configured for optimum construction access and operational efficiency. Primary builds are located to allow easy access from the existing mine road. The process plant has been located 500 meters away from the mine portal to minimize material transfer distance. The new construction camp and parking is located within close proximity to the process plant and truck shop.

18.3 Site Access Road

The existing 18km mine access road from Saskatchewan Highway #106 to the mine site will receive upgrades to accommodate increased traffic. The road will require new gravel, grading and compaction. It will be suitable for transportation of mining equipment, fuel trucks, mobilization of construction equipment and ongoing operational requirements.

18.4 Light Vehicle Roads

Approximately 8 km of new light vehicle access roads will be constructed. These roads will provide access to the tailings facility, tailings pipeline, mine portal and plant site facilities. Single-lane light vehicle roads will be 6 m wide including berms (where required) and ditches.


Hanson Lake
Foran Exploration Camp
No Name Lake
Ultimate Pit Bottom Elevation - 312 meters
Swamp
Swamp
Swamp
Swamp
Swamp
Swamp
E 639 500
E 640 000
E 640 500
E 641 000
E 641 500
E 642 000
E 642 500
E 643 000
N 6 057 000
N 6 056 500
N 6 056 000
N 6 055 500
N 6 055 000
UG Waste Dump 235,000 m³
Decline
Portal
Firewater Tank
Parking
Camp
Plant Site
Fuel Storage
Truck Shop
ANFO Facility
LEGEND
Foran Drill Hole Collar
McIlvenna Bay Cu-Zn-Au-Ag Deposit Project to Surface
Main Access / Haul Road
Secondary Access Road
Preferred Sands Cleared Area
Proposed Tailing Site
Proposed Infrastructure
Tailings Pipeline
Facilities
0
250
500 m
UTM Zone 13 NAD83 NTS 63 L/10
REV. No.
ISSUED FOR
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DESIGNED: R. Gutowski
Sept 29 14
DRAWN: R. Gutowski
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APPROVED:
FORAN MINING CORPORATION
JDS Energy & Mining Inc.
TITLE
McIlvenna Bay Project
Plant Site
Plan
PROJECT No.
DRAWING No.
10-10-003
REV No.

18.5 Power Supply

Total average operating power, including the process plant, mine facilities and site infrastructure is 25MW. Electricity will be supplied by overhead line from Creighton SK.

18.5.1 Transmission Line

In order to meet the increased power demand a new 95 km transmission line will be constructed bringing reliable 138 kV power to site. This line is classified as an E84 rate code under SaskPower.

18.5.2 Sub-Station and Distribution

The sub-station will be located as close as possible to the process plant. Primary feed voltage will be transformed down to 4160V and delivered to adjacent e-houses via buried teck cables. From there power will be connected to all required facilities.

18.6 Camp

A temporary 200 person construction camp will be mobilized for the construction phase of the project. The camp will have 200 single occupancy rooms in a “common bathroom” arrangement. The complex will also incorporate 35 beds from the existing exploration camp that will be relocated to the construction camp complex. The construction camp will also include a kitchen, dining complex, offices, recreation room, laundry and gym facilities. Once in operations the construction camp will be removed leaving the offices, sewage treatment plant, potable water system and existing 35 bed exploration camp onsite.

18.7 Process Plant

The process plant is located adjacent to the truck shop and 500 m away from the mine portal. It contains the milling, flotation, regrind, concentrate thickening, concentrate thickening, filter presses and concentrate storage. The building will be a 45 m x 75 m pre-engineered steel structure.

18.8 Truck Shop and Warehouse

The truck shop and warehouse will be contained in one common pre-engineered building 35m x 20m building. The truck shop will house 4 separate bays: general maintenance, wash, lube & oil and miscellaneous storage bays. Each bay will have a dedicated 14’ x 14’ roll up to accommodate all vehicle sizes. Tire changing and large vehicle assembly will take place outdoors and utilize rough terrain mobile equipment.

18.9 Communications / IT

The camp and offices will include a wired and wireless computer network and satellite phone system.

A hand-held radio system will be used for voice-communication between personnel in the field.

18.10 First Aid / Emergency Services

A qualified nurse or first-aid attendant will be provided on-site. The first aid room will be located in the warehouse. The ambulance and fire truck will be parked at the ready in the truck shop.

Buildings will be equipped with smoke, carbon monoxide and heat detectors, overhead sprinklers, hydrants / hoses and appropriate chemical fire extinguishers.

18.11 Bulk Explosives Storage and Magazines

Explosives will be stored at a secured and monitored site located approximately 1 km from populated, high traffic areas. The final location of the explosives storage site will be determined as part of future pre-feasibility or feasibility studies. Boosters and detonators will be stored in barricaded magazines and separated according to Natural Resource Canada guidelines.

18.11.1 Powder Magazine

High explosive boosters and packaged explosives will be stored in a 24 tonne powder magazine.

18.11.2 Detonator Magazines

Detonators (caps) will be stored in a 3.6 tonne cap magazine.

18.12 Bulk Fuel Storage and Delivery

Diesel fuel will be stored in a 60,000L dual wall fuel tank located near the truck shop. The tank will be located within a lined compound that meets Saskatchewan and Environment Canada regulations for containment in the event of a spill.

A skid mounted, transportable delivery system capable of pumping 50 GPM will be used to fuel all site vehicles. Diesel will be delivered to mine mobile equipment by the fuel & lube truck. Light vehicles will fuel up at the fuel storage area.

18.13 Water Treatment

Any surplus water generated by the mining operation will be treated and tested prior to being discharged back into the environment. Potential discharge points will be assessed in the next level of study.

18.14 Firewater Tank and System

The firewater tank will be dual purpose serving as a freshwater and firewater storage tank. Internal risers on all non-firewater suction lines will ensure a minimum volume of 400,000 litres. This capacity will allow for approximately 2 hours of firefighting capability.

The buried firewater network will be pressurized by two pumps (one electric, one diesel stand-by). This network will be connected to all buildings requiring fire protection.

18.15 Potable Water

A potable water treatment system is included in the construction camp facility and will remain onsite during mine operations.

18.16 Sewage Treatment

A sewage water treatment system is included in the construction camp facility and will remain onsite during mine operations. The sewage treatment system will be sized to handle approximately 200 personnel on-site.

Contaminated water from the heavy equipment wash bay will pass through an oil water separator. Oily sludge will be stored in a transfer tank and back-hauled off-site for disposal. The remaining water will be stored in a pond and recycled to the wash bay.

18.17 Freight

Freight will be delivered to site on the all-season access road and offloaded at the warehouse or other designated area.

18.18 Personnel Transportation

During operations personnel will be transported between Flin Flon, MB and the mine site by charter buses. Multiple buses will operate on different schedules to accommodate varying work schedules.

18.19 Tailings

The tailings generated from milling activities during the life of mine will be used as backfill for underground mining operations, and stored on-site in a Tailings Storage Facility (TSF).

This section presents a mine plan and design criteria, conceptual designs, construction quantity and operating and capital cost estimates for tailings storage over the life of mine. A scoping level cost estimate (+/- 50% accuracy) is provided for the TSF. The cost estimate is based on the construction quantities completed by Golder (2013). The TSF considers storage of the quantity of tailings presented in the 2014 mine plan (JDS 2014).

18.19.1 Mine Plan Quantities

The mine plan presented in JDS (2014) indicates that total of 23.7 Mt tailings will be produced over a 14 year life of mine. The present estimate by JDS (2014) assumes some 11.7 Mt will be stored in a TSF, and the remainder of the tailings used as paste backfill underground.

18.19.2 Design Criteria

The conceptual level dam designs were based on the parameters listed in Table 18.1

Table 18.1: Design Criteria for Preliminary Mine Waste Management

Items	Value	Source/Comment
Base map	Digital elevation model based on 1:50,000 scale maps with 2 m contours	Foran 2013b, pers. comm.
Lake bathymetry	Selected lakes on 0.5 m contour, where available	TAEM 1990 (attached in Cameco 1990)
Climate and Hydrology		
Annual precipitation	500 mm	Cameco 1990
Annual evaporation	690 mm	Cameco 1990
Annual runoff (50% probability)	95,000 m^3/km^2	Cameco 1990
Mine Plan		
Mine life	14 years	JDS 2014
Ore	23.7 Mt	JDS 2014
Tailings		
Life-of-mine production	23.7 Mt	JDS 2014
Surface storage	11.7 Mt	JDS 2014
Underground paste backfill storage	12.0 Mt	JDS 2014.
Volume	7.8 M-m^3 for sub-aerial slurry 10.6 M-m^3 for sub-aqueous slurry	Calculated, excluding 50% underground backfilling
Specific gravity	3.3	G&T 2012
Void ratio (settled)	1.2 for sub-aerial slurry 2.0 for sub-aqueous slurry	Assumed, referenced G&T 2012 tailings gradation
Dry density (settled)	1.5 t/m^3 for sub-aerial slurry 1.1 t/m^3 for sub-aqueous slurry	Calculated
Tailings surface slope	0.5% for sub-aerial storage 1% to 2% for sub-aqueous storage	Included in freeboard
Dam		
Downstream slope	2H:1V	Assumed based on previous project experience
Upstream slope	2H:1V	Assumed based on previous project experience
Crest width	10 m	Assumed based on previous project experience
Raise method	Downstream	Assumed based on previous project experience
Pond volume	100,000 m^3	Assumed based on previous project experience
Freeboard requirement	5 m	Assumed based on previous project experience
Starter dam	3H:1V rockfill dam shells and 10 m wide clay core	Assumed based on previous project experience
Construction material	Earthfill	Development of a borrow area in or adjacent to the impoundment area is proposed
	Rockfill	Quarrying of local dolomite and sandstone is proposed; underground PAG waste rock is not suitable for dam construction

Geochemistry		
Tailings	Acid generating	Cameco 1990
	Metal leaching	No metal leaching test has been completed
Waste rock	Acid generating	Cameco 1990
Exclusion Zones		
Potential mineralization zone		Cameco 1990

18.19.3 Tailings Storage Facility

In 2013, Golder completed a Tailings Management Study for McIlvenna Bay which identified 12 suitable locations for tailings storage with in a 10 km radius of the deposit. The suitable locations were assessed and ranked based on the simplified “Guidelines for the Assessment of Alternatives for Mine Waste Disposal”, as published by Environment Canada in September 2011 (Environment Canada 2011) Based on the results of this assessment Option 4 was selected as the preferred TSF. The TSF is an in-lake option, requiring a small dam across the outlet of Guyader Lake. The TSF takes advantage of the storage volume of the lake basin and therefore had the lowest value of dam construction quantities, and presumably the lowest cost.

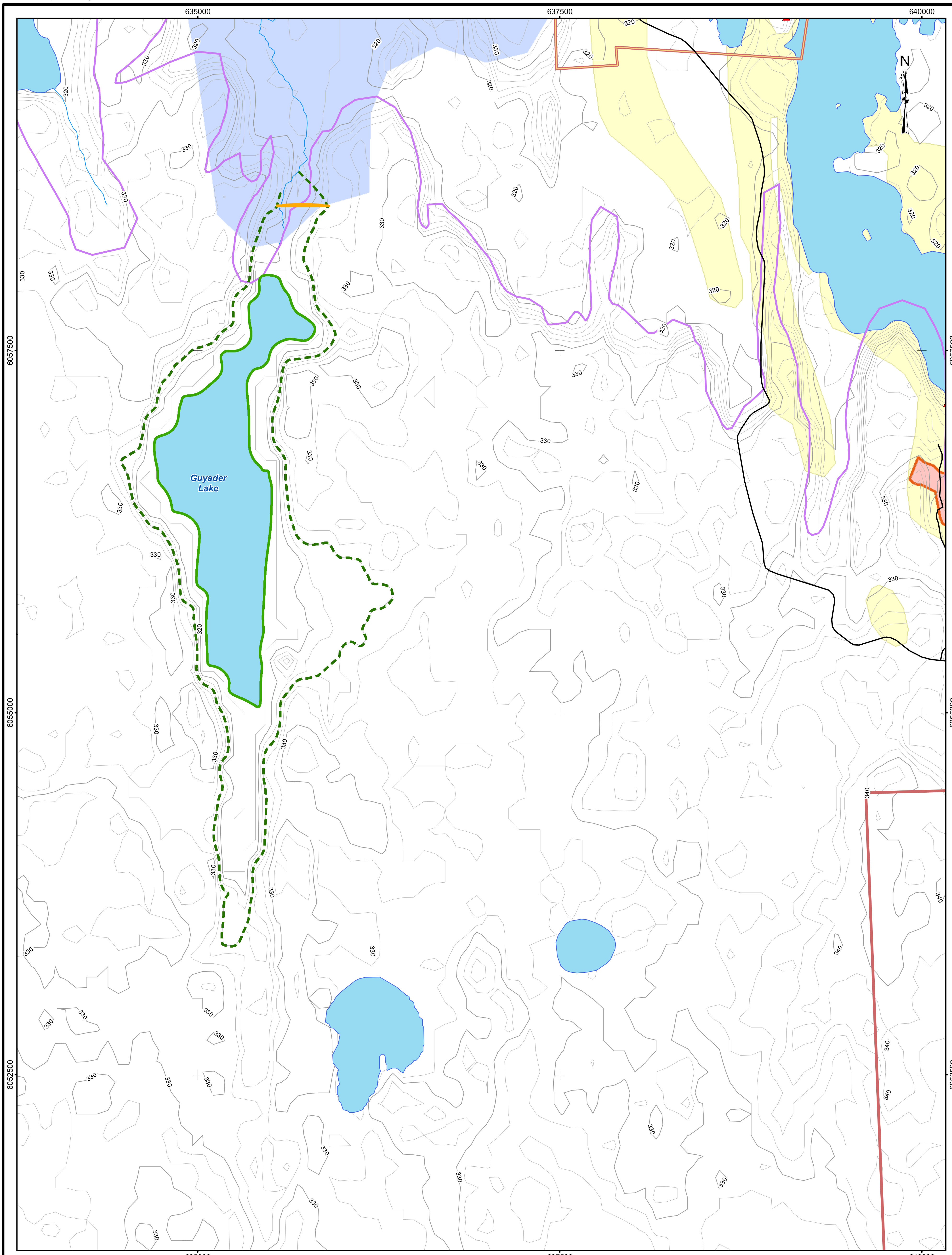
Most of the area required for the Option 4 impoundment is located to the south of the geological boundary and occupies Guyader Lake. The Option 4 dam alignment is illustrated in Figure 18.2.

The bathymetric data provided by TAEM (1990) attached in Cameco (1990) indicates that the lake is about 5 m deep and holds approximately 6.4 M-m3 of water. At this early stage, no field investigation has been conducted to find out the geological and hydrogeological conditions of the lake. The lake is assumed to hold approximately 6.4 M-m3 water. This would need to be confirmed in future work.

According to regional survey data provided by Foran in January 2013 (Foran 2013b pers. comm.), the elevation of the upland plateau or platform surrounding Guyader Lake is approximately 330 m (El. 330 m), while the elevation of the lake surface is approximately 315 m. Assuming El. 315 m is the surface elevation according to the TAEM (1990) data (elevation was not included in the TAEM bathymetry data), the bottom of Guyader Lake is approximately El. 310 m and the elevation difference from the lake bottom to the surrounding platform would be some 20 m.

In developing a conceptual design and cost estimate for this study, it is assumed that the TSF would be lined starting at an elevation approximately 1 m below the existing lake water surface (El. 315 M) and extend to El. 314 m. This requires pumping approximately 2.4 M-m3 of water from the lake in Year 1 for the construction of the low permeability liner. These assumptions will need to be checked by future field investigation.

N:\Bur-Graphics\Projects\2011\1426\11-1426-0006\GIS\Mapping\MXD\General\5000\Figure-3_Dam_Align_Option_4.mxd



LEGEND

- PROPOSED TSF SITE DAM CENTRE ALIGNMENT
- DIVERSION DITCH
- ROAD
- MAJOR CONTOUR (10 METRE)
- MINOR CONTOUR (2 METRE)
- TSF POND ALIGNMENT (EL. 317 METRES)
- WATERCOURSE
- WATERBODY
- GLACIOLACUSTRINE
- PRECAMBRIAN DOMAIN BOUNDARY
- FORAN MINING CORPORATION MINERAL CLAIM
- THIRD PARTY MINERAL CLAIM
- MCILVENNA BAY DEPOSIT (Figure 10.1, RPA 2011)
- BASE METAL OCCURENCE/DEPOSIT (Figure 7.3, CAMECO 1990)

DRAFT

REFERENCE

MINERAL CLAIMS AND GEOLOGY FEATURES OBTAINED FROM THE PROVINCE OF SASKATCHEWAN. ROAD FEATURES OBTAINED FROM GEOBASE. WATER FEATURES OBTAINED FROM GEOGRATIS. CONTOURS OBTAINED FROM THE CLIENT.
DATUM: NAD83 PROJECTION: UTM ZONE 13

PROJECT
FORAN MINING CORPORATION
MCILVENNA BAY PROJECT
SASKATCHEWAN, CANADA

TITLE
OPTION 4

Golder Associates

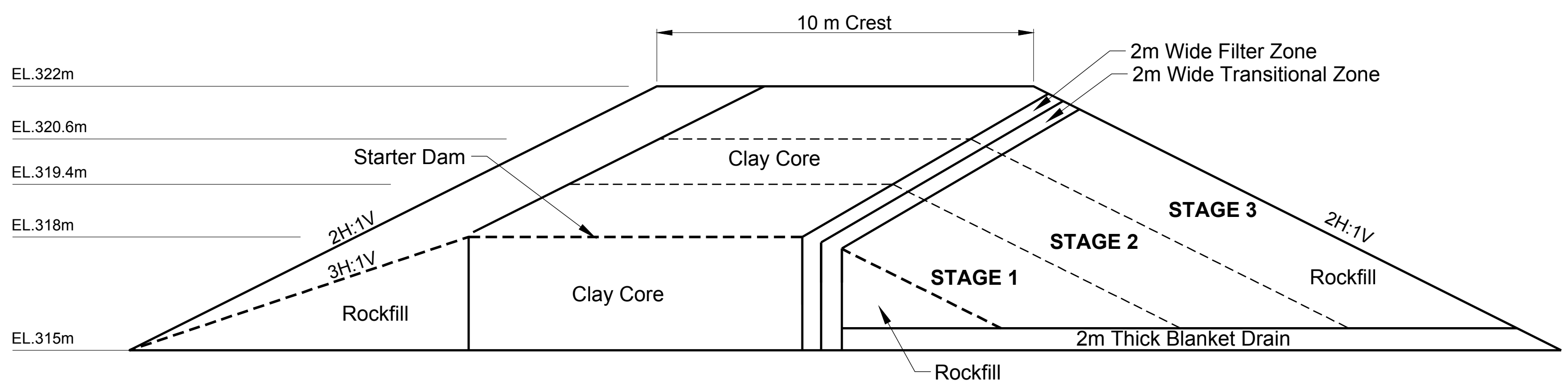
PROJECT	11-1426-0006		FILE No.	
DESIGN	WD	26 Mar. 2013	SCALE AS SHOWN	REV. 0
GIS	JP	2 May. 2013		
CHECK			FIGURE: 3	
REVIEW				

18.19.4 Diversion Ditches

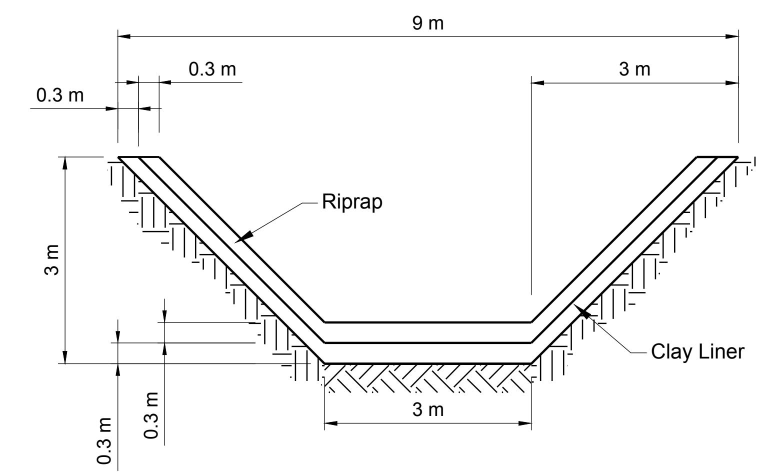
Cameco reported that the sub-catchment area of TSF Option 4, Guyader Lake, ranges from approximately 14 km2 to 27 km2 (Cameco 1990). It is assumed a diversion ditch that is up to 3 m deep and 9 m wide at top with 1 Horizontal to 1 Vertical (1H:1V) side slopes will be constructed in Year 1 to divert the run-on from reaching the TSF. The ditch would be lined with 0.3 m thick clay layer to reduce seepage loss and overlain by 0.3 m thick riprap to minimize potential erosion during peak/design events. It is expected that ditch geometry would be optimized in the next phase of design.

Some information, such as the sub-catchment area, seepage loss in the TSF options, stratigraphy in the impoundment area, and dam foundation conditions, is unknown at this stage, so the design of other appurtenant structures (such as the spillway and seepage collection / pump back systems) was not considered at this stage.

See Figure 18.3 for the conceptual ditch design.



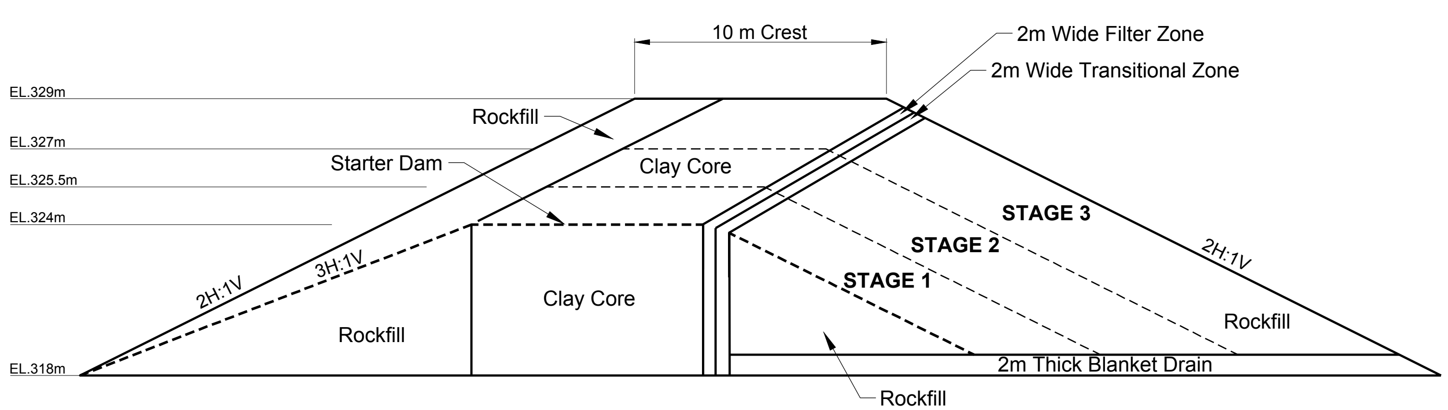
TYPICAL DAM SECTION OF OPTION 4



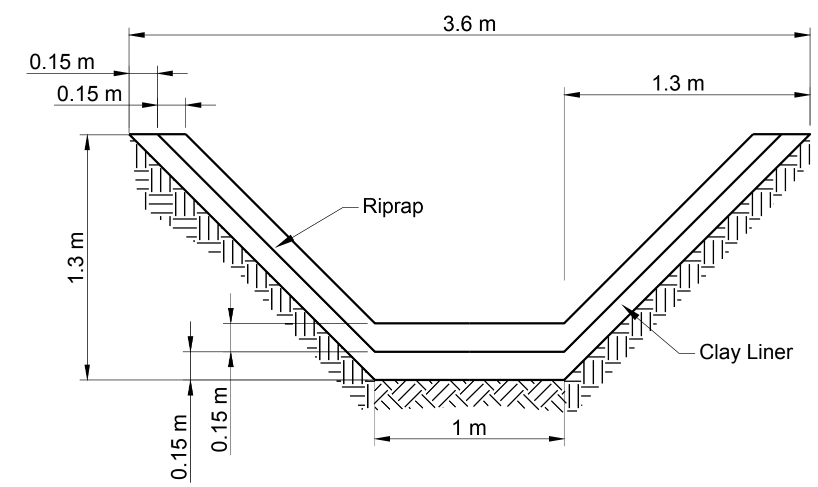
NOT TO SCALE

DIVERSION DITCH PROFILE OF OPTION 4

Riprap Zone: 0.3 m true and nominal thickness
Clay Zone: 0.3 m true and nominal thickness



TYPICAL DAM SECTION OF OPTION 11



NOT TO SCALE

DIVERSION DITCH PROFILE OF OPTION 11

Riprap Zone: 0.15 m true and nominal thickness
Clay Zone: 0.15 m true and nominal thickness

NOTES

1. Clay Core: 10 m true and nominal thickness
2. Filter Zone: 2 m true and nominal thickness
3. Transitional Zone: 2m true and nominal thickness
4. Assume low permeable dam foundation to the north of the Geological Boundary. Grouting the dam foundation is not required.
5. Toe drain is not considered for the conceptual design.

PROJECT
FORAN MINING CORPORATION
MCILVENNA BAY PROJECT
SASKATCHEWAN, CANADA

TITLE
TYPICAL DAM AND DITCH SECTIONS

Golder Associates

PROJECT No.	11-1426-0006-5000		FILE No. 1114260006-5000-3000-01
DESIGN	WD	02MAY13	SCALE NTS
CADD	TS	02MAY13	
CHECK	DYH	03MAY13	**FIGURE 4**
REVIEW			

18.19.5 Dam Design

Conceptual designs of the TSF dam for Option 4 is shown in Figure 18.3. Both of the conceptual dam designs assume rockfill dams with low permeability clay core. A 2H:1V downstream and upstream slope is suitable for the rockfill dams used in this study, while a 3H:1V slope was used for the general design as the dam zoning was unknown.

Clifton (1999) reported 0.5 M-m3 of clayey soils available around the project area. The surficial geological map obtained from the Government of Saskatchewan shows a large area of lacustrine deposit at the northwest of the project area (see Figure 18.1). While no field investigation has been conducted for the Project at this stage, it is assumed for this study that enough low permeability soils can be found for impervious dam core, liner, and soil cover construction.

Although waste rock from the operation will not be produced because an open pit is not considered economical for the Project, rockfill may be still accessible for the dam construction by developing a quarry locally. Initial planning anticipates the quarry would be developed inside the impoundment, if practical.

There is a large difference in particle size between earthfill and rockfill, so a filter zone and a transition zone between the earthfill and rockfill are designed to minimize the potential for the fine particles to pipe into the voids in the coarse rockfill zones. Further, as sand can be easily quarried from local unconsolidated sandstone, a drainage blanket, instead of finger drains, is considered in this design for the downstream portion of the dam.

Grouting the dam foundation is not considered at this stage for any of the options because no investigation has been completed to date. The locations of the TSF options are shown in Figure 18.1.

The preliminary evaluation anticipates the TSF to store:

- 10.6 M-m3 tailings by sub-aqueous deposition in Option 4, the dam will be approximately 7.5 m high dam with crest elevation of 322.6 m and 5 m freeboard.

18.19.6 Consideration of Tailings Storage facility Closure

The McIlvenna Bay tailings are potentially acid generating. The tailings stored in the TSF must be covered using a water cover or engineered soil cover for closure to minimize the potential acid generation.

This study has assumed that a 2 m deep water cover can be maintained for Option 4 for closure.

18.19.7 Construction Quantities

Dam construction quantities for Options 4 is based on the JDS (2014) mine plan, the conceptual designs presented in Golder (2013). Construction items are described in Table 18.2, and the construction quantities are summarised in Table 18.3.

Table 18.2: Dam and Impoundment Construction Items

Item		Unit	Comment
Pumping lake water		Mm^3	For clay liner construction around Guyader Lake in Year 1, not for water recovery
Clearing and grubbing		ha	Assumes logging and brush removal to stockpiles within 100 m
Topsoil removal		m^3	Assumes average of 2 m thickness based on general description of surficial geology (Cameco 1990), to be confirmed by future field investigation; topsoil to be stored in TSF facility for reclamation
Dam foundation preparation		ha	Assumes preparation limited to proof rolling of foundation after topsoil removal
Rockfill		m^3	Assumes run-of-mine material from an existing or new open pit or a new quarry, including haul, placement, and compaction
Earthfill	Clay dam core	m^3	Clay borrow from approximately 2 km directly northwest of the TSF centre, based on general description of surficial geology (Cameco 1990), to be confirmed by future field investigation
	Clay liner on lake slope	m^3	For Option 4 to avoid acid seepage loss; assumed to be 0.5 m thick
	Clay liner on diversion ditch	m^3	Channel bed is clay lined, then covered with riprap; assume to be 0.15 m to 0.3 m thick
	Engineered soil cover	m^3	For Option 11 closure to cap the tailing and waste rock from air and water ingress and acid rock drainage; assumed to be 1 m thick
Coarse filter		m^3	Processed material for grain size; likely screened only
Fine filter		m^3	Likely using unconsolidated sandstone or similar, or crushed to meet specifications
Excavation in soil		m^3	Assumes soil is rippable and is used as fill with no haul
Riprap		m^3	Rockfill obtained from site excavation; assumed to be 0.15 m to 0.3 m thick

Source: Golder 2014

Table 18.3: Construction Quantities

Option 4							
Item	**Unit**	**Starter Dam Quantity [(a)]**	**Stage 1 Quantity [(b)]**	**Stage 2 Quantity [(b)]**	**Stage 3 Quantity [(b)]**	**Closure Quantity**	**Total Quantity**
Pumping Lake Water	m^3	2,400,000	-	-	-	-	2,400,000
Clearing and Grubbing	ha	12.4	-	40.1	20.3	-	72.8
Topsoil Removal	m^3	199,500	7,200	15,600	11,300	-	233,600
Dam Foundation Preparation	ha	0.3	0.4	0.8	0.6	-	2.10
Rockfill (quarry)	m^3	2,500	2,900	8,800	6,100	-	20,300
Earthfill (clayey soils) [(c)]	m^3	41,000	4,500	203,000	104,900	-	353,400
Fine Filter	m^3	2,000	8,100	16,900	12,500	-	39,500
Coarse Filter	m^3	300	900	1,300	1,200	-	3,700
Ditch Excavation (soil)	m^3	55,200	-	-	-	-	55,200
Rip-rap	m^3	33,500	-	-	-	-	33,500

Note:
a) The construction quantities for Starter Dam include the starter dams, diversion ditches, and liner of Option 4.
b) The construction quantities of Stage 1 are the sum of those of Year 2 to Year 5; Stage 2 is of Year 6 to Year 10; and Stage 3 is of Year 11 to Year 14.
c) Earthfill comprises material for the dam core and diversion ditch clay liner.
Source: Golder 2014

18.19.8 Cost Estimates

Cost estimate for construction of Options 4 was were completed, with an accuracy of +/- 50%, based on the following assumptions:

- The dam sections are based on the designs presented in Golder (2013) including a zoned earthfill dam with a clay core as the low permeability element;
- The option is built in stages to minimise the capital cost for construction;
- The starter dam, which comprises the starter embankment and the ditches, is completed prior to mill start up and has been sized to store the first year's deposition. The subsequent stages comprise downstream raises of the dam;
- Clearing and grubbing assumes logging and brush removal to stockpiles within 100 m;
- Soil removal assumes an average of 2 m thickness based on general description of surficial geology. This should be confirmed by future field investigations;
- Dam foundation preparation includes proof rolling after soil removal;
- Rockfill is sourced from quarry. The unit rate for rockfill includes drill, blast, load, haul, placement, and compaction;
- A clay borrow source is within approximately 2 km of the TSF. This assumption should be confirmed by future field investigation;
- Coarse filter material is sourced from screened run-of-mine material or a quarry. The unit rate includes haul, placement, and compaction;
- Fine filter material is sourced from sandstone crushed to meet specifications. The unit rate includes haul, placement, and compaction;
- Ditches are clay lined and then covered with rip-rap. The clay liner and rip-rap are 0.3 m thick for Option 4. It is assumed that rip-rap can be sourced on site;
- Access roads, tailings delivery and water reclaim systems are not included;
- Operating costs for power, equipment and labour required for the daily operation of the TSF are included in an assumed unit rate for tailings placement;
- The construction rates used are assumed based on Golder's project experience;
- Option 4 includes an additional 0.5 m thick clay liner above the lake level, and requires that an estimated 2,400,000 m^3 of water be pumped from the lake prior to construction. The rate used for pumping assumes that there is existing line power available; and

The lower bound cost estimates are summarized in Table 18.4. The operating cost for the tailings facility is expected to range between $3 to $10 / tonne. This quantity excludes tailings placed as underground backfill.

Table 18.4: Lower Bound Cost Estimates

Item	Unit Rate ($)	Starter Dam Cost	Stage 1 Cost	Stage 2 Cost	Stage 3 Cost	Closure Cost	Total Cost
Pumping Lake Water	0.04/m^3	96,000	-	-	-	-	96,000
Clearing and Grubbing	2,625/ha	32,550	-	105,262	53,288	-	191,100
Topsoil Removal	4/m^3	798,000	28,800	62,400	45,200	-	934,400
Dam Foundation Preparation	37,500/ha	11,250	15,000	30,000	22,500	-	78,750
Rockfill (quarry)	20/m^3	50,000	58,000	176,000	122,000	-	406,000
Earthfill (clayey soils)	10/m^3	410,000	45,000	2,030,000	1,049,000	-	3,534,000
Fine Filter	25/m^3	50,000	202,500	422,500	312,500	-	987,500
Coarse Filter	25/m^3	7,500	22,500	32,500	30,000	-	92,500
Ditch Excavation (soil)	4/m^3	220,800	-	-	-	-	220,800
Rip-rap	25/m^3	837,500	-	-	-	-	837,500
TOTAL COST		2,513,600	371,800	2,858,662	1,634,488	-	7,378,550

Notes: All rates and costs are in Canadian Dollars.
Source: Golder 2014

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

At this time, no formal offtake agreements for concentrate sales exist for the McIlvenna Bay project. Preliminary market studies were completed by an independent consultant who has provided Foran with indicative terms of the market conditions with respect to the concentrates to be produced. The study and indicative terms have been reviewed and/or modified where applicable and found to be acceptable by QP Mike Makarenko.

Smelter terms were identified for the copper, zinc and bulk concentrates and are considered to be in line with current market conditions and have been considered in the economic analysis.

Concentrate transportation will be conducted using trucks from the mine site to Flin Flon. The PEA envisages that both copper concentrates and the Bulk concentrate will be shipped offshore via Vancouver. The zinc concentrates will be transported to Trail, BC. No contractual arrangements exist at this time for concentrate trucking, port usage, shipping, and smelting or refining.

Table 19.1 through Table 19.5 outline the concentrate transportation costs and smelter terms used in the economic analysis.

Table 19.1: Copper Concentrate No. 1 Smelter Terms

Copper Concentrate No. 1	Unit	Value
Cu Recovery	%	94.4
Zn Recovery	%	33.8
Au Recovery	%	84.6
Ag Recovery	%	76.9
Concentrate Grade		
Cu	%	29.2
Zn	%	1.1
Au	g/t	6.38
Ag	g/t	126
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	100
Min. Cu deduction	% Cu/tonne	1
Zn Payable	%	0
Au Payable	%	95
Min. Au deduction	g/t conc	0
Ag Payable	%	90
Min. Ag deduction	g/t conc	0
Treatment & Refining Costs		
Cu TC	US$/dmt conc	75
Cu RC	US$/payable lb	0.08
Au RC	US$/payable oz	15
Ag RC	US$/payable oz	1
Calculated Penalties	US$/dmt	3.5
Transport Costs	US$/dmt	198.48

Table 19.2: Copper Concentrate No. 2 Smelter Terms

Copper Concentrate No. 2	Unit	Value
Cu Recovery	%	83.4
Zn Recovery	%	9.6
Pb Recovery	%	43.4
Au Recovery	%	59.7
Ag Recovery	%	50.3
Concentrate Grade		
Cu	%	24.2
Zn	%	6.4
Pb	%	1.3
Au	g/t	6.50
Ag	g/t	216.00
Moisture Content	%	8
Smelter Payables		
Cu Payable	%	100.0
Min. Cu deduction	% Cu/tonne	1.1
Zn Payable	%	0.0
Au Payable	%	95.0
Min. Au deduction	g/t conc	0.0
Ag Payable	%	90.0
Min. Ag deduction	g/t conc	0.0
Treatment & Refining Costs		
Cu TC	US$/dmt conc	75.00
Cu RC	US$/payable lb	0.08
Au RC	US$/payable oz	15.00
Ag RC	US$/payable oz	1.00
Calculated Penalties	US$/dmt	9.12
Transport Costs	US$/dmt	198.48

Source: JDS 2014

Table 19.3: Zinc Concentrate No. 1 Smelter Terms

Zinc Concentrate No. 1	Unit	Value
Zn Recovery	%	85.4
Au Recovery	%	14.6
Ag Recovery	%	27.3
Concentrate Grade		
Zn	%	55.0
Au	%	0.29
Ag	g/t	38.00
Moisture Content	%	8.0
Smelter Payables		
Zn Payable	%	85.0
Min. Zn deduction	% Zn/tonne	0.0
Au Payable	%	80.0
Min. Au deduction	g/t conc	1.0
Ag Payable	%	70.0
Min. Ag deduction	g/t conc	93.3
Treatment & Refining Costs		
Zn TC	US$/dmt conc	215.00
Zn RC	US$/payable lb	0.00
Au RC	US$/payable oz	0.00
Ag RC	US$/payable oz	0.00
Calculated Penalties	US$/dmt	2.50
Transport Costs	US$/dmt	97.27

Source: JDS 2014

Table 19.4: Zinc Concentrate No. 2 Smelter Terms

Zinc Concentrate No. 2	Unit	Value
Cu Recovery	%	0.0
Zn Recovery	%	76.3
Pb Recovery	%	0.0
Au Recovery	%	6.9
Ag Recovery	%	13.6
Concentrate Grade		
Cu	%	0.0
Zn	%	54.3
Au	%	0.81
Ag	g/t	63.00
Moisture Content	%	8.0
Smelter Payables		
Zn Payable	%	85.0
Min. Zn deduction	% Zn/tonne	0.0
Au Payable	%	80.0
Min. Au deduction	g/t conc	1.0
Ag Payable	%	70.0
Min. Ag deduction	g/t conc	93.3
Treatment & Refining Costs		
Zn TC	US$/dmt conc	215.00
Zn RC	US$/payable lb	0.00
Au RC	US$/payable oz	0.00
Ag RC	US$/payable oz	0.00
Calculated Penalties	US$/dmt	2.50
Transport Costs	US$/dmt	97.27

Source: JDS 2014

Table 19.5: Bulk Concentrate Smelter Terms

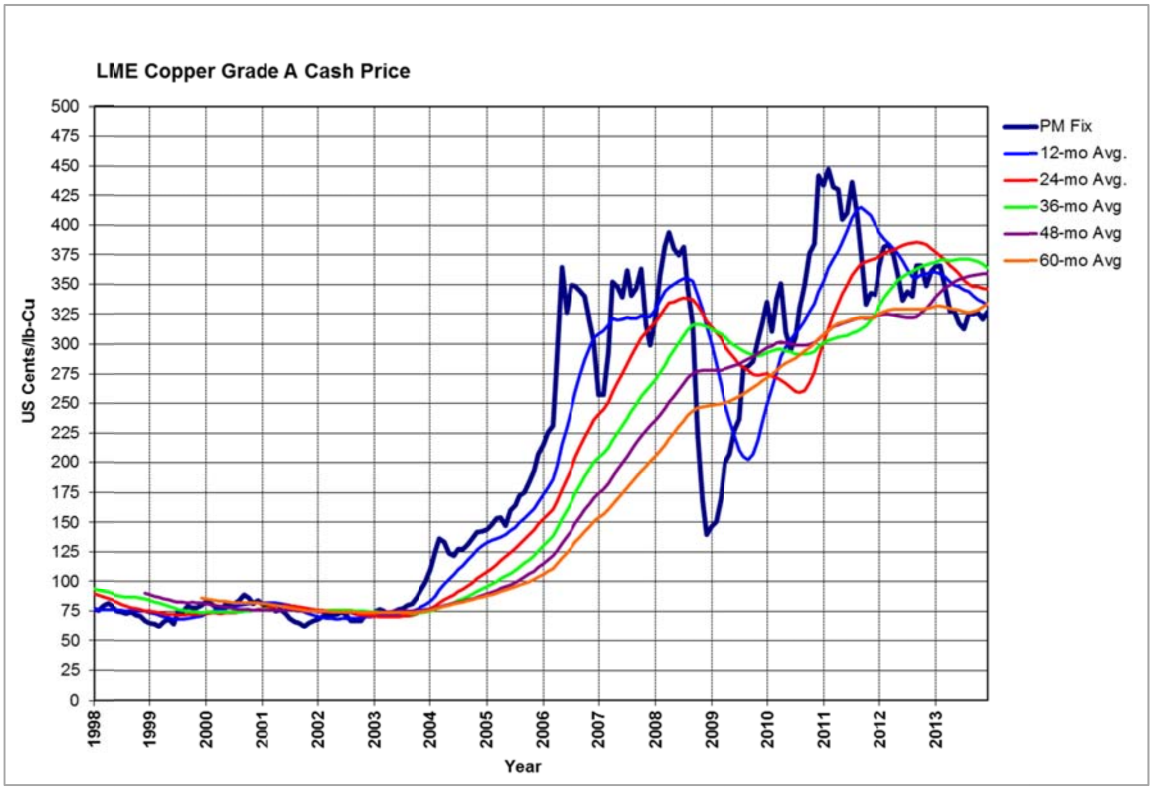
Bulk Concentrate	**Unit**	**Value**
Cu Recovery	%	56.0
Zn Recovery	%	2.1
Au Recovery	%	59.0
Ag Recovery	%	42.0
Concentrate Grade		
Cu	%	12.0
Zn	%	9.2
Pb	%	14.0
Au	g/t	5.27
Ag	g/t	332.00
Moisture Content	%	8.0
Smelter Payables		
Cu Payable	%	25.0
Min. Cu deduction	% Cu/tonne	2.0
Zn Payable	%	100.0
Min. Zn deduction	% Zn/tonne	5.0
Pb Payable	%	100.0
Min. Pb deduction	% Pb/tonne	3.0
Au Payable	%	90.0
Min. Au deduction	g/t conc	1.0
Ag Payable	%	90.0
Min. Ag deduction	g/t conc	93.3
Treatment & Refining Costs		
Bulk Conc TC	US$/dmt conc	325
Zn RC	US$/payable lb	0.00
Au RC	US$/payable oz	0.00
Ag RC	US$/payable oz	0.00
Calculated Penalties	US$/dmt	35.20
Transport Costs	US$/dmt	198.48

Source: JDS 2014

19.2 Metal Prices

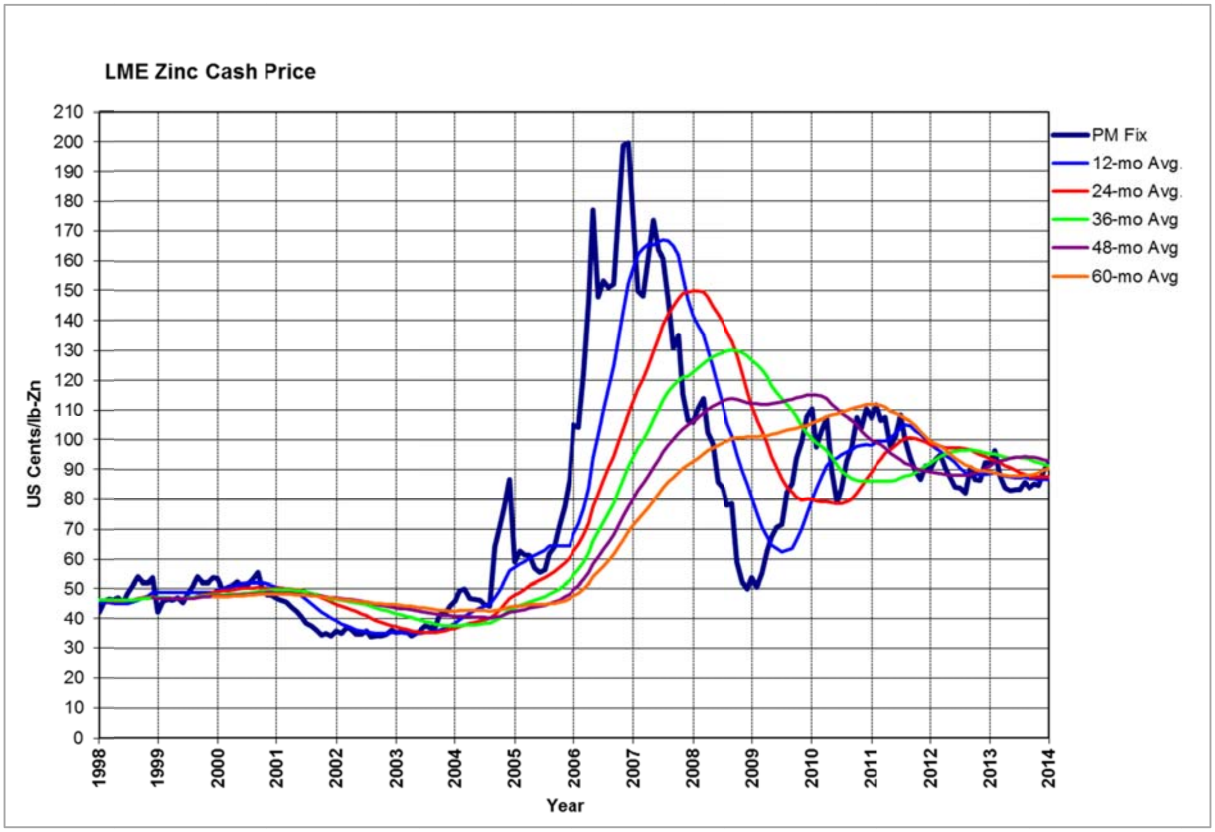
The base and precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong) on an almost continuous basis. Historical metal prices are shown in Figure 19.1 through x and demonstrate the change in metal prices from 1998 to 2014.

Figure 19.1: Historical Copper Price



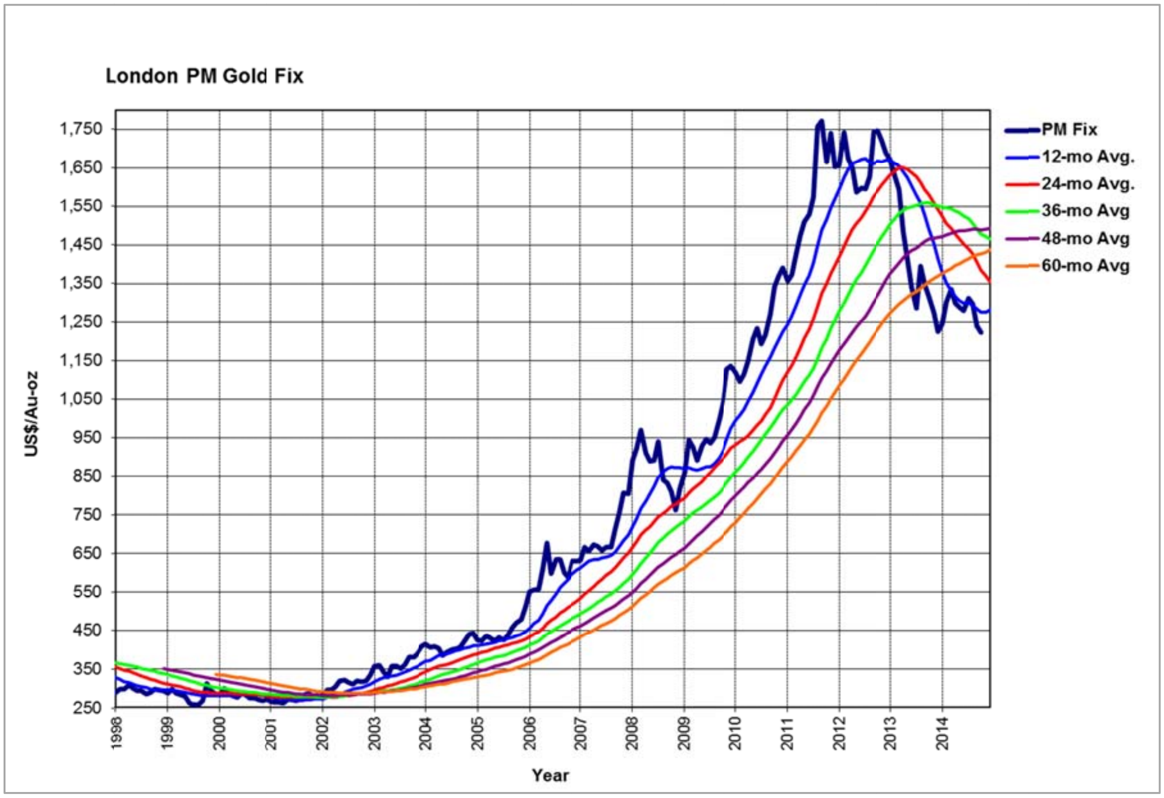
Source: LME 2014

Figure 19.2: Historical Zinc Price



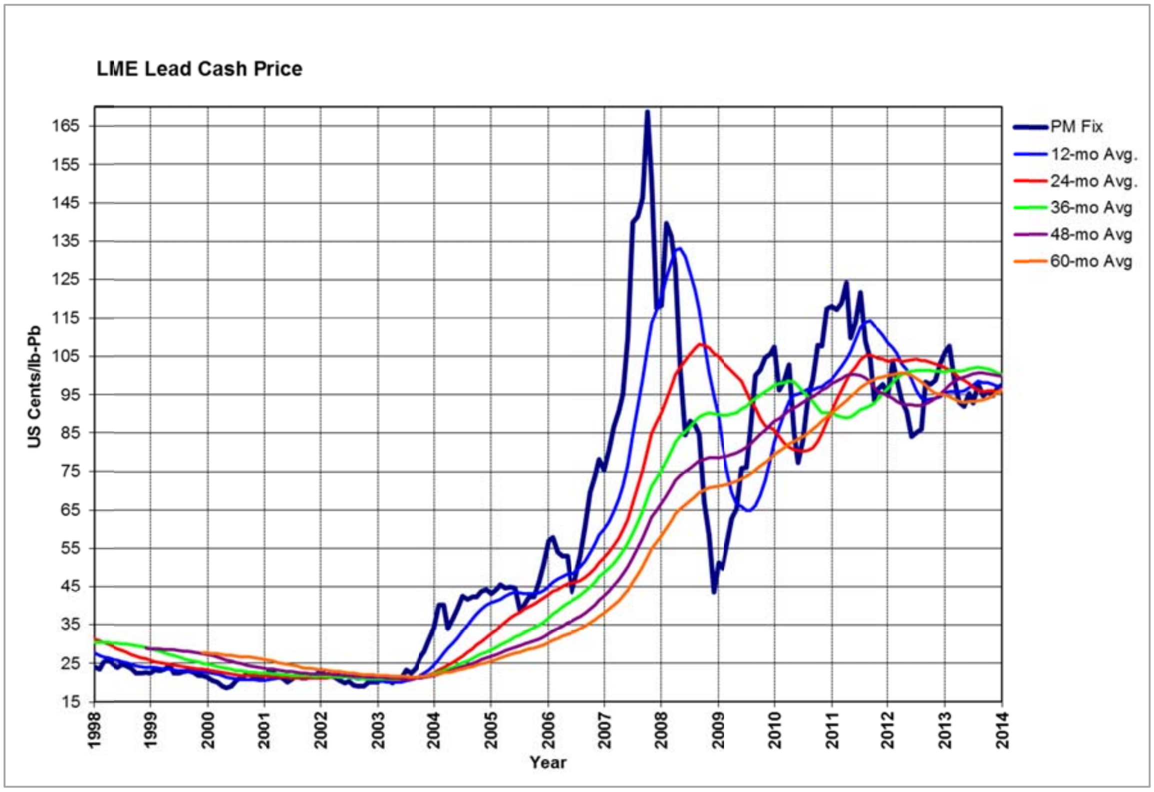
Source: LME 2014

Figure 19.3: Historical Gold price



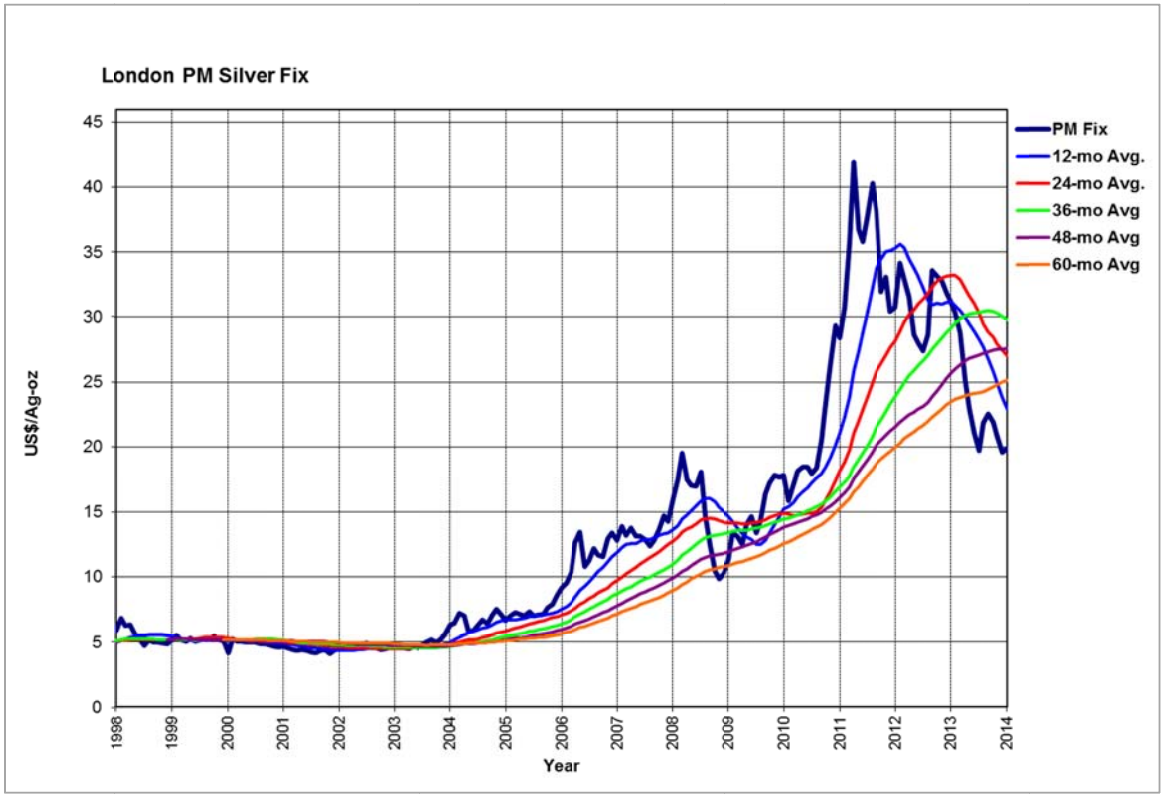
Source: LME 2014

Figure 19.4: Historical Lead Price



Source: LME 2014

Figure 19.5: Historical Silver Price



Source: LME 2014

The metal prices used in the Base Case economic analysis were the spot metal prices and exchange rate as at October 15, 2014. Table 19.6 summarizes the spot metal prices and exchange rate as at October 15, 2014.

Table 19.6: Metal Prices and Exchange Rate used in the Economic Analysis

Commodity	Unit	Base Case Spot as at October 15, 2014
Copper Price	US$/lb	3.08
Lead Price	US$/lb	0.93
Zinc Price	US$/lb	1.06
Gold Price	US$/oz	1,238
Silver Price	US$/oz	17.00
Exchange Rate	US$:C$	0.89

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Related to the Project

McIlvenna Bay involves the development and operation of an underground mine and mill facility and installation of associated surface amenities including a truck shop, warehouse, a power line and distribution facility, fuel storage and distribution, a fire water system, water treatment plant, mineral waste storage (tailings and waste rock), temporary camp accommodations, administration buildings, and other miscellaneous infrastructure. The Project and deposit area are accessible via an 18 km long all-weather gravel road which connects to Highway 106 (the Hanson Lake Road). It is expected this road will need upgrading for heavy machinery delivery and concentrate trucking; however, road condition is still to be determined.

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

The Namew Lake Upland landscape area of the Mid-Boreal Lowland Ecoregion is characterized by a gently undulating to nearly level landscape, featuring deciduous and coniferous forests with numerous wetlands (Acton et al. 1998). Vegetation is generally influenced by landscape and soil types. Peatlands, which comprise approximately one third of the ecoregion, typically consist of tamarack and black spruce interspersed with wet meadows (Acton et al. 1998). The Flin Flon Plain landscape area of the Churchill River Upland Ecoregion lies in eastern Saskatchewan's southernmost stretch of Precambrian Shield. Bedrock predominates in this area, with thin deposits of sandy glacial till or glaciolacustrine silt and clay (Acton et al. 1998). Vegetation of the Flin Flon Plain landscape is characterised by mixed wood forests. Black spruce is the most common tree species and is largely found in poorly drained peaty areas along with tamarack; however, black spruce is not as abundant as it is in other landscape areas of the boreal shield (Acton et al. 1998).

Extensive mining and exploration activities associated with other metal and 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 Corporation (Cameco) and Esso Minerals Canada. 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.

The site of the past-producing Hanson Lake Mine, operated by Western Nuclear Mines Ltd., 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, 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.

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

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 is located in the immediate vicinity of the Project, approximately 3.6 km from McIlvenna Bay. This mine was formerly operated by Winn Bay Sand Limited Partnership. Another silica sands project in the area operated by Hanson Lake Sands Company Ltd. is in the exploration phase, but not yet operational.

20.2 Environmental and Heritage Studies Results

Comprehensive environmental baseline studies for McIlvenna Bay were completed by Canada North Environmental Services (CanNorth) in 2012. The baseline program was designed to prepare the Project for future licensing and regulatory requirements, and included collection of a full suite of environmental data including climate and meteorology; noise; surface water hydrology; water and sediment quality; plankton, benthic invertebrate, and fish communities; fish habitat; fish chemistry; fish spawning; ecosite classification; vegetation communities; wildlife communities; species at risk; and heritage resources (CanNorth 2013).

20.2.1 Aquatic Resources

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

At least 15 species of fish are known to be present in McIlvenna Bay ASA, including lake whitefish, northern pike, walleye, white sucker, and yellow perch; however, none of these species are considered to be of conservation concern. Unnamed Pond is the only waterbody in the Project ASA which does not contain fish. Aquatic habitat mapping indicated a variety of habitat types are present in McIlvenna Bay ASA, with suitable habitat for fish spawning, rearing, feeding, and overwintering provided by most waterbodies. Evidence of spawning (i.e., eggs) by northern pike and yellow perch was abundant throughout most of the ASA, and the Bad Carrot River was found to be an important spawning migration route/area for white sucker, walleye, northern pike, and yellow perch.

20.2.2 Terrestrial Resources

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

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

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

20.2.3 Heritage Resources

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

20.3 Environmental Issues

20.3.1 Tailings Disposal

A preliminary study on mine waste management for the Project was conducted by Golder Associates (Golder) in 2012 and 2013. The study was divided into three phases. Phase 1 identified 10 potential TSF site options; two additional sites were identified following this stage. Phase 2 assessed the 12 options using the Guidelines for the Assessment of Alternatives for Mine Waste Disposal (EC 2011a). The results of the Phase 2 study indicated that Option 4, an in-lake option utilizing the basin of Guyader Lake, stood out among all of the options.

20.3.1.1 Option 4

Guyader Lake is a medium-sized headwater lake known to contain several species of fish, including lake whitefish, northern pike, spottail shiner, walleye, white sucker, and yellow perch. A project which includes a proposal to use a natural waterbody frequented by fish for mine waste disposal requires an amendment to the Metal Mining Effluent Regulations (MMER), which is a federal legislative action and also triggers a requirement for a federal environmental assessment (EA) under the Canadian Environmental Assessment Act (CEAA), where applicable.

20.3.2 Species at Risk

Additional mitigation and/or management consideration for species of provincial and federal conservation concern may be required for the Project.

As noted previously, a number of vegetation species considered rare in the province of Saskatchewan were identified in the Project LSA and RSA. The MOE recommends a 50 m setback distance for high level disturbance (e.g. road building, blasting) for all plants rated S1 to S3 (MOE 2014), therefore, mitigation for any rare plants within the construction footprint may be required.

A number of species listed under the federal Species at Risk Act (SARA) were observed during baseline surveys in the Project LSA and RSA. Of particular note, woodland caribou (boreal population; Suggi-Amisk-Kississing management unit) occur in and near the Project LSA and RSA. The boreal population of woodland caribou (including woodland caribou in Saskatchewan) is listed on SARA Schedule 1 as threatened (SARPR 2014). A federal recovery strategy for the boreal population of woodland caribou has been recently proposed by EC (EC 2011b). The long-term goal of the strategy is to achieve or maintain self-sustainability in as many of the local populations as possible, and to stabilize the remaining populations. It should be noted that the project area has seen significant previous development,does not represent undisturbed baseline conditions and the linear disturbance from the access road has already been in existence for a number of years.

Proponents have an obligation to notify the competent minister or ministers of a project if the project is likely to affect a listed wildlife species or its critical habitat under SARA. Additionally, adverse effects of the project on a listed wildlife species and its critical habitat must be identified, and, if the project is carried out, those effects must be mitigated and monitored.

These obligations are in addition to the requirements set out in CEAA for an assessment of the environmental effects of the project, including in particular any change it may cause to a listed wildlife species, its critical habitat or the residences of individuals of that species as those terms are defined in SARA.

A number of provincially and federally listed avian species at risk were also observed within the Project LSA and RSA during baseline field surveys, thus, it is possible that disturbances to breeding migratory birds may occur during construction or operation of the Project via habitat destruction and/or disruption of breeding and/or nesting activities. Under the Migratory Birds Convention Act (MBCA) (GC 1994), destruction of birds, nests, and eggs is prohibited. In the case of sensitive, rare, and at risk species, additional guidelines may apply (MOE 2014; EC 2009).

20.4 Operating and Post Closure Requirements and Plans

Plans to address requirements for environmental mitigation and monitoring during all stages of McIlvenna Bay will be developed based in part on the results of the EA process. These plans may include (but are not limited to) the following:

- Environmental Protection Plan;
- Environmental Contingency Plan;
- Emergency Spill Response Plan;
- Waste Management Plan;
- Emergency Response Plan;
- Tailings Management Plan; and
- Rehabilitation and Closure Plan.

Additional plans will be developed in response to conditions of release of the EA process, as well as to address compliance monitoring requirements pursuant to applicable legislation. Monitoring may be required during construction and operation of the Project depending on the conditions of release issued by governments. Monitoring and follow-up is the responsibility of the proponent to demonstrate that the project is carried out according to the regulatory conditions and authorizations issued, to determine the accuracy of the environmental effects predicted in the EIS, and to evaluate the effectiveness of the mitigation measures.

Proponents must comply with the terms and conditions set forth in any Approval to Operate Pollutant Control Facilities issued by the MOE pursuant to the Environmental Management and Protection Act (EMPA), 2002 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 MOE 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 MOE 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 MOE. 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 MOE 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 McIlvenna Bay pursuant to the federal Fisheries Act and the MMER. This will apply to any deposit of mine tailings and other waste matter produced during mining operations into natural fish bearing waters.

Prior to decommissioning of McIlvenna Bay mine, mill, and ancillary facilities, a number of approvals must be obtained from the MOE, including:

- Approval to Decommission Pollutant Control Facilities;
- Release from Decommissioning and Reclamation; and,
- 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 Required Permits and Status

The EA and permitting framework for metal mining in Canada is well established. Following a successful EA, the Project would undergo a licensing and permitting phase to allow operations to proceed. The Project is then regulated through all phases (construction, operation, closure, and post-closure) by both federal and provincial departments and agencies.

20.5.1 Environmental Assessment Process

McIlvenna Bay will be subject to EA in accordance with provincial and federal requirements. The federal and provincial EA processes, guided respectively by CEAA and *The Environmental Assessment Act*, respectively, are coordinated where possible utilizing established protocols and milestones for projects with joint federal and provincial jurisdiction and provide a mechanism for reviewing major projects to assess potential impacts. This approach aligns with the 'one project, one assessment' model for the proponent and the public while not affecting the independent decision-making of the two levels of governments or hindering any statutory or process requirements by either legislation.

20.5.1.1 Provincial Environmental Assessment Process

The *Environmental Assessment Act* requires that a proponent receives the approval of the Minister of Environment before proceeding with a development that is likely to have significant environmental implications. Since the Project is considered a development under the Act, Foran will be required to submit a technical proposal to the EA Branch for review or screening. It is assumed that the EA Branch will determine that the Project is a development under the Act and therefore Foran will be required to prepare a terms of reference (TOR) which will identify the key impacts to be studied. The initial environmental impact statement (EIS) submission will then be circulated by the EA Branch for a technical review by experts (including those from other provincial ministries) and, where required, to federal government reviewers.

The *Environmental Assessment Act* requires the Minister to give notice that an EIA is being conducted. Additionally, where the Minister's decision on a development leads to actions that have the potential to adversely impact Treaty and Aboriginal rights and the pursuit of traditional uses, the province has a duty to consult with First Nations and Métis communities in advance of the decision.

Following the completion of the technical review, the EA Branch advises the proponent and the Minister notifies the public that the EIS will be made available for review and comment by the public. Following the end of the period provided for public review and comment, the Minister will decide whether to approve or deny the project as proposed. If approved, the Minister may also require the proponent to implement additional environmental protection measures as a condition of an approval. Once ministerial approval is received, the proponent has 'cleared' the EA process and may proceed to obtain any other provincial approvals.

20.5.1.2 Federal Environmental Assessment Process

Designated Projects require a screening under CEAA to determine whether an EA is required. Under CEAA, an EA focuses on potential adverse environmental effects that are within federal jurisdiction including fish and fish habitat, other aquatic species, migratory birds, federal lands, effects that cross provincial or international boundaries, effects that impact on Aboriginal peoples such as their use of lands and resources for traditional purposes, and environmental changes that are directly linked or necessarily incidental to any federal decisions about a project. The federal decision-making and coordinating authority under CEAA is the Canadian Environmental Assessment Agency (CEA Agency). Other federal departments may also provide specialized knowledge or expert advice through both the federal and provincial EA processes.

A federal EA may be required for McIlvenna Bay if it is developed as it is currently proposed. McIlvenna bay would trigger under CEAA 2012, specifically Section 16(a) of the Regulations Designating Physical Activities, because in its current form it involves the construction, operation, decommissioning, and abandonment of a metal mine, other than a gold mine, with an ore production capacity greater than 3,000 tpd. The current projected production rate for the Project is 4,600 tpd. Additionally, a project which includes a proposal to use a natural, fish-frequented waterbody for the disposal of mine waste triggers a federal EA. In the case that Guyader Lake is selected as the preferred option for tailings storage, Foran will be required to prepare an assessment of alternatives for mine waste disposal for consideration, a fish habitat offset plan for consideration as part of the EA, and also to participate in public and aboriginal consultations on the EA, including on possible amendments to the MMER.

Since the Project would likely be considered a Designated Project under CEAA 2012, Foran may be required to submit a project description to the CEA Agency for screening. The agency will then screen the project to determine if a federal EA is required. If a federal assessment is required, the minister then determines what type of EA the project will require. There are two types of EAs conducted under CEAA: an environmental assessment by responsible authority (standard EA) and an environmental assessment by a review panel. Both types of assessments can be conducted by the federal government alone or in conjunction with another jurisdiction. The responsible authority in the case of base and precious metal mining is the CEA Agency.

20.5.2 Environmental Permitting

McIlvenna Bay will require a number of approvals, permits, and authorizations during all stages of the Project following release from the provincial and federal EA processes in accordance with various standards outlined in legislation, regulations, and guidelines. Foran will also be required to comply with any other terms and conditions issued by regulatory agencies associated with release from the EA process. A preliminary list of permits, approvals, and authorizations that may be required for the Project is presented in Table 20.1, subject to confirmation with the responsible agencies. Permits and authorizations may also be required from other jurisdictions, such as municipalities, if any are affected.

Table 20.1: Potential Permits, Approvals, and Authorizations Anticipated to be Required

Permit, Approval, or Authorization	Issuing Agency
Provincial	
Release from Environmental Assessment Process	Saskatchewan Environmental Assessment Branch
Approval to Construct and Operate Waterworks	Water Security Agency
Water Rights License	Water Security Agency
Approval to Construct and Operate Drainage Works	Water Security Agency
Approval to Construct and Operate Sewage Works	Water Security Agency
Aquatic Habitat Protection Permit	Water Security Agency
Temporary Work Camp Site Permit	MOE
Forest Product Permit	MOE
Miscellaneous Use Permit	MOE
Construction Permit	MOE
Approval to Operate Pollutant Control Facilities	MOE
Approval to Construct and Operate an Industrial Effluent Works	MOE
Approval to Construct and Operate a Storage Facility (Hazardous Materials and Waste Dangerous Goods)	MOE, Industrial Branch
Federal	
Release from EA Process	CEA Agency
Fisheries Act Authorization	Fisheries and Oceans Canada
Species at Risk Permit	Environment Canada
Designation of a Tailings Impoundment Area	EC
Aquatic Environmental Effects Monitoring Program	EC
License to Store, Manufacture, or Handle Explosives	Natural Resources Canada

Source: Can North 2014

These permits, approvals, and authorizations will be required at various stages throughout the life of the Project. Applications for initial construction activities should be submitted in advance of release from the provincial and federal EA processes with the expectation that approvals can be granted shortly after EA release to allow activities to begin. Close communication with the regulators is recommended to understand the likely timeframes for their review and approval processes. In all cases, applications will be made in a timeframe to accommodate regulatory review and allow the construction, operation, closure, and decommissioning activities to proceed in a timely manner.

20.6 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.

20.7 Social and Community

McIlvenna Bay is located near Hanson Lake in east-central Saskatchewan, approximately 375 km northeast of Saskatoon, Saskatchewan. The closest communities include Creighton, Saskatchewan and Flin Flon, Manitoba, which are located approximately 65 km west-southwest of the Project. Creighton and Flin Flon have a combined population of approximately 7,100 residents, with 5,600 living in Flin Flon and the remainder in Creighton (Statistics Canada 2012a, 2012b). The economy of the area is primarily based on copper and zinc mining, while tourism and forestry are also of some importance.

McIlvenna Bay lies within the area traditionally occupied by the Peter Ballantyne Cree Nation (PBCN), which is made up of approximately 9,000 members living on more than 36 reserves and/or settlements. The PBCN's traditional territory encompasses roughly 52,000 km^2, from the Saskatchewan/Manitoba border west to the west end of Trade Lake, north to Reindeer Lake, and south to Sturgeon Landing (ASKI 2012). The Project is located approximately 55 kilometers southwest of the settlement of Deschambault Lake and approximately 100 kilometers northeast of the community of Denare Beach. Approximately 1,500 PBCN members reside in these communities (ASKI 2012).

The isolated nature of these communities creates special circumstances for PBCN members working to strengthen their local economies and personal economic well-being. Although rich in natural resources, this sparsely populated region is challenged by infrastructure, education levels, and average income when compared to the rest of the province (ASKI 2012).

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

While most acknowledged that the mining sector does provide the potential for employment and to create spin-off opportunities such as service business in catering, janitorial, trucking, security, grocery and retail supplies, such development must be tempered against the continued reliance of PBCN members on the waters, lands and forests relied on for sustenance, livelihood and spiritual support (ASKI 2012). As the Project proceeds, Foran will continue to engage the traditional users of the Project area in order to receive input on potential ways and means to minimize, to the extent possible, negative impacts on the traditional use of the lands in the vicinity of McIlvenna Bay site.

Foran will undertake public consultation with First Nations groups and area stakeholders as part of the EA process required for approval of McIlvenna Bay. These groups must be adequately informed about the financial, social and environmental impacts, and opportunities associated with the Project. All First Nations groups and stakeholders in the area will be made aware of the Project, be informed of and invited to attend Open House meetings, and have an opportunity to provide feedback to Foran regarding McIlvenna Bay.

20.8 Mine Closure

A conceptual plan for decommissioning and reclamation of the Project site is required as part of the environmental assessment of a mining development in Saskatchewan. As such, Foran will prepare a conceptual decommissioning and reclamation plan for inclusion in the environmental assessment, including details related to the predicted impacts of the project on the surrounding ecosystems; a description of how the impacts will be mitigated and what the residual impacts, if any, will be; a general overview on how the site will be decommissioned (i.e. buildings removed; pits filled in, etc.); and the final decommissioning objective, which will in part be based on the residual impacts of the project.

Following release from the environmental assessment process, an operational decommissioning plan is required pursuant to Section 12 of *The Mineral Industry Environmental Protection Regulations*, 1996. The plan must include the provision of a financial security or assurance and, as such, will be reviewed at least every five years or sooner when there are significant changes to the mining project such as expansion of the operation or when the minister believes the financial assurance is underfunded.

The operational plan should include:

- Details regarding the proposed end use of the decommissioned site;
- The predicted timelines for reclamation of the site;
- A discussion of alternative procedures that may be used for decommissioning the various site facilities (i.e., camp and office buildings, pit mine(s), underground mines and waste rock stockpiles, etc.);
- Identification of the preferred procedures for decommissioning the site facilities;
- The time frame and sequence of decommissioning activities;
- Environmental mitigation and reclamation measures (e.g., contouring of waste rock piles, covering of wastes and re-vegetation);

- An estimate of post-decommissioning contaminant loadings and residual impacts to the local drainage system and to groundwater;
- A proposed program for monitoring during the decommissioning and transition (post-decommissioning) phases;
- Proposed contingency measures if initial plans are not successful; and,
- An estimate of the cost to undertake the decommissioning and reclamation plan and the cost of monitoring the site after decommissioning and reclamation has been completed.

As per Section 18 of *The Mineral Industry Environmental Protection Regulations*, 1996, proponents are required to give sixty days' notice in writing prior to the initiation of an approved plan to permanently close pollutant control facilities, mines, or mills. If a sufficient period of transition phase monitoring demonstrates that the site has achieved an appropriate level of environmental and physical stability in accordance with the decommissioning and reclamation plan, the operator may make a written application for a Release from Decommissioning and Reclamation.

Based in part on this section of the regulations, the application for Release from Decommissioning and Reclamation should contain, at a minimum:

- A summary of the decommissioning and reclamation activities that have been completed by the operator;
- A description of the performance of the site during the transition (decommissioning and post decommissioning) monitoring phase;
- Predictions that are based on the documented performance of the site during the post decommissioning phase monitoring, of any potential ongoing expenditures the province may be expected to accrue in order to adequately maintain and monitor the site if it assumes custodial responsibility for the property;
- A list and assessment of remaining environmental liabilities; and,
- An estimate of the potential costs to the province to address such liabilities should it assume custodial responsibility.

Upon receiving the application, the MOE will initiate a detailed review of the application. That review will include opportunities for public input on any conditions that may be applied before the Release from Decommissioning and Reclamation is issued and the type of institutional controls that will be applied to the site. Only after these steps are completed to the satisfaction of the Minister of Environment will a Release from Decommissioning and Reclamation be issued to the operator and the custodial responsibility for the property is transferred from the operator to the provincial institutional control management framework.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

A summary of the capital costs is shown in Table 21.1.

Table 21.1: Capital Cost Summary

Capital Costs	Pre-Production $M	Sustaining/ Closure $M	Total $M
Site Development	0.9	0.0	0.9
Mining	72.9	119.6	192.5
Primary Crushing & Coarse	5.8	0.0	5.8
Concentrator	53.8	0.0	53.8
Tailings & Waste Rock Management	3.1	4.9	7.9
On-Site Infrastructure	18.3	0.0	18.3
Off-Site Infrastructure	14.9	0.0	14.9
Project Indirects	18.8	0.0	18.8
Engineering & EPCM	15.8	0.0	15.8
Owner's Costs	3.0	0.0	3.0
Closure	0.0	10.0	10.0
Salvage	0.0	-9.3	-9.3
Subtotal	**207.3**	**125.2**	**332.5**
Contingency (20%)	41.5	25.0	66.5
Total Capital Costs	**248.8**	**150.3**	**399.1**

21.1.1 Mine Capital

Mine capital is summarized below in Table 21.2. Mine capital includes all underground work and capital purchases required to prepare the mine for production.

Table 21.2: Mine Capital

Mine Capital Costs	Pre-Production Total ($M)
Contractor Raise Development	0.9
Contractor Shaft Development	39.9
Infill Drilling	0.6
Preproduction Lateral Development	17.3
Mobile Equipment	10.3
Stationary Equipment	3.9
Total Mine Capital Costs	**72.9**

Source: JDS 2014

21.1.2 Mill Capital

The mill capital cost estimate was prepared using stochastic estimating methods, in-house data base and previous project experience. The estimate is derived from engineers and estimators who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

Mill capital is summarized in Table 21-3. Mill capital includes all capital purchases required to build the process plant.

Table 21.3: Summary of Mill Capital Costs

Mill Capital Costs	Pre-Production Total ($M)
Primary Crushing	2.9
Coarse Ore Bins & Reclaim	3.0
Total Primary Crushing & Coarse Material Stockpile	**5.8**
Concentrator	
Concentrator Building	20.0
Grinding & Classification	11.4
Flotation	12.4
Concentrate Dewatering	3.0
Concentrate Drying & Loadout	2.8
Tailings & Paste	3.0
Reagents	2.0
Total Concentrator	**54.6**
Total Mill Capital Costs	**60.4**

Source: Samuel 2014

21.1.3 Site Infrastructure

Infrastructure capital costs are summarized in Table 21-4.

Table 21.4: Summary of Site Infrastructure Capital Costs

Site Infrastructure Capital Costs	$M
On-Site Infrastructure	
Accommodation & Administration Facilities	6.5
Truck Shop & Warehouse	2.5
Power Distribution	1.0
Fuel Storage And Distribution	0.6
Fire Water System	1.0
Water Treatment	2.5
Plant Mobile Fleet	2.2
Surface Mobile Equipment	0.5
Miscellaneous Infrastructure	0.3
Total On-Site Infrastructure	**17.0**
Off-Site Infrastructure	
Access Road Improvements	1.9
Power Line	19.0
Total Off-Site Infrastructure	**20.9**
Total On-Site & Off-Site Infrastructure	**37.9**

Source: JDS 2014

21.2 Operating Costs

21.2.1 Mine Operating Costs

Mine operating costs were built using a combination of first principals engineering and scaling from other feasibility level studies JDS has prepared for Canadian underground operations.

Mine operating costs are summarized below in Table 21.5. Costs are subdivided into operating categories.

Table 21.5: Mine Operating Costs

Mine OPEX by Activity	CA$
Development	6.58
Production	12.76
Hoisting	0.38
Infill Drilling	0.08
Backfill	6.98
Mine General	3.86
Mine Maintenance	2.89
Grand Total by Activity	**33.54**

21.2.2 Mill Operating Costs

The mill operating cost estimate was prepared using stochastic estimating methods, in-house data base and previous project experience. The estimate is derived from engineers and estimators who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in this study.

A breakdown of the cost by material type is shown below in Table 21.6.

Table 21.6: Summary of Process Operating Costs

Process Operating Costs	Unit	Copper Stockwork	Upper West	Massive Sulphide
Labour	$/t	3.32	3.32	3.32
Supplies	$/t	5.97	7.13	7.00
Power	$/t	3.63	3.63	3.63
Total	$/t	12.91	14.08	13.94

21.2.3 General and Administration Costs

General and administrative costs are summarized in Table 21.7.

Table 21.7: Summary of General & Administrative Costs

G&A Operating Costs	Unit	Value
Labour	$/t	2.11
Equipment	$/t	0.18
Expenses	$/t	1.34
Transportation	$/t	0.47
Total	$/t	4.10

22 ECONOMIC ANALYSIS

An engineering economic model was developed to estimate annual cash flows and sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variation in metal prices, grades, recoveries, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules and forecast of resulting cash flows as part of this study. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis, to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ materially from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this project and are summarized in Section 21 of this report and are presented in 2014 dollars. The economic analysis has been run with no inflation (constant dollar basis).

22.1 Assumptions

The metal price and exchange rate assumptions used in the economic analysis are outlined in Table 19-6. All costs and economic results are reported in Canadian dollars ($), unless otherwise noted, while metal prices are reported in US dollars (US$).

Other economic factors used in the economic analysis include the following:

- Discount rate of 7% (sensitivities using other discount rates have been calculated);
- Closure costs of $10M and salvage value of $9.3M were utilized;
- Working capital of $9M (equivalent to 3 months of first year operating costs);
- Nominal 2014 dollars;
- No inflation;
- Numbers are presented on 100% ownership and do not include management fees or financing costs; and
- Exclusion of all pre-development sunk costs (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, etc.).

Mine revenue is derived from the sale of concentrates into the international marketplace. No contractual arrangements for concentrate smelting or refining exist at this time, however, preliminary market studies on the potential concentrate sales were completed to provide indicative market smelter terms.

Table 22.1 outlines the LOM plan tonnage and grade estimates used in the economic analysis.

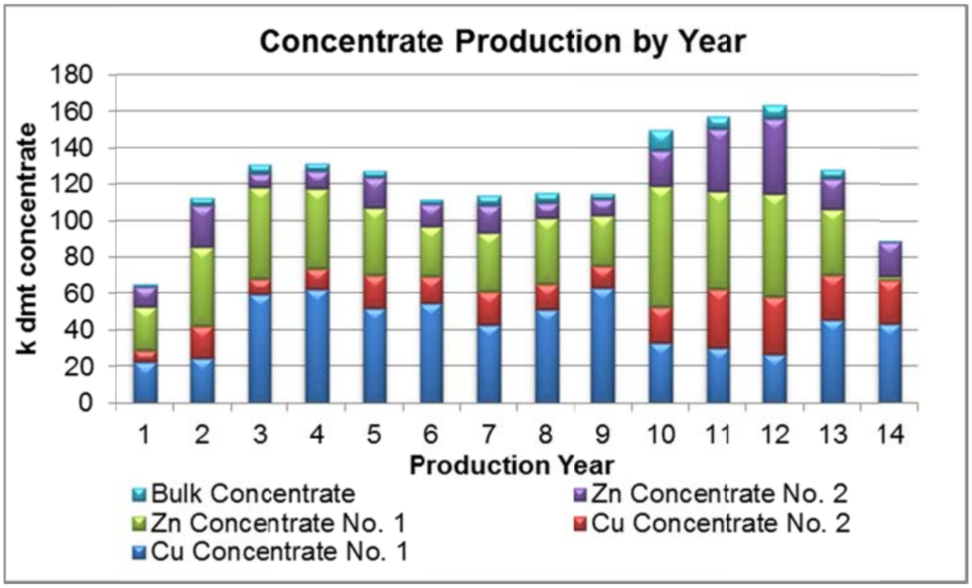
The NSR Parameters used in the economic analysis are outlined in Section 19.

Table 22.1: Life of Mine Plan Summary

Parameter	Unit	Value
Mine Life	Years	13.7
Total Mined	**M tonnes**	**23.7**
Throughput Rate	tpd	4,761
Average Head Grade		
Cu	%	1.17%
Zn	%	2.36%
Pb	%	0.15%
Au	g/t	0.42
Ag	g/t	14.82
Metal Production		
Cu Concentrate No. 1 (CSZ)	dmt	611,377
	dmtpa	44,759
Cu Concentrate No. 2 (UW)	dmt	253,809
	dmtpa	18,581
Zn Concentrate No. 1 (MS)	dmt	533,476
	dmtpa	39,056
Zn Concentrate No. 2 (UW)	dmt	244,582
	dmtpa	17,906
Bulk Concentrate (MS)	dmt	65,173
	dmtpa	4,771
Cu Payable	M lbs	513.7
	M lbs /yr	37.6
Pb Payable	M lbs	15.8
	M lbs /yr	1.2
Zn Payable	M lbs	804.7
	M lbs /yr	58.9
Au Payable	k oz	218.0
	k oz/yr	16.0
Ag Payable	k oz	5,437
	k oz/yr	398.0

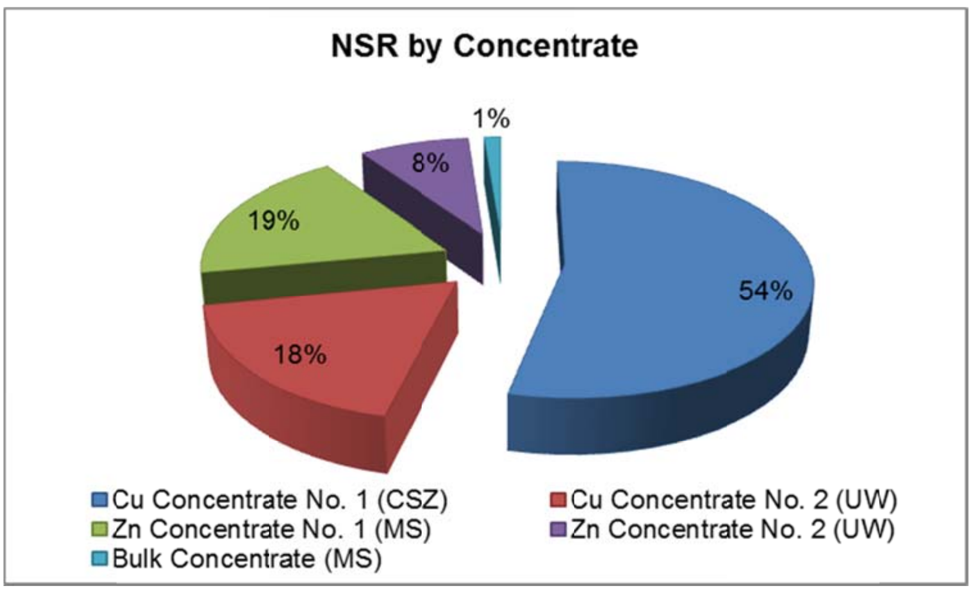
Source: JDS 2014

Figure 22.1: Life of Mine Concentrate Production by Year



Source: JDS 2014

Figure 22.2: NSR by Concentrate



Source: JDS 2014

22.2 Summary of Capital Costs

Table 22.2 outlines the capital costs use in the economic analysis. Detailed information can be found in Section 21 of this report.

Table 22.2: Summary of Capital Costs

Capital Costs	Pre-Production $M	Sustaining/ Closure $M	Total $M
Site Development	0.9	0.0	0.9
Mining	72.9	119.6	192.5
Prim. Crushing & Coarse Ore	5.8	0.0	5.8
Concentrator	53.8	0.0	53.8
Tailings & Waste Rock Management	3.1	4.9	7.9
On-Site Infrastructure	18.3	0.0	18.3
Off-Site Infrastructure	14.9	0.0	14.9
Project Indirects	18.8	0.0	18.8
Engineering & EPCM	15.8	0.0	15.8
Owner's Costs	3.0	0.0	3.0
Closure	0.0	10.0	10.0
Salvage	0.0	-9.3	-9.3
Subtotal	**207.3**	**125.2**	**332.5**
Contingency (20%)	41.5	25.0	66.5
Total Capital Costs	**248.8**	**150.3**	**399.1**

Source: JDS 2014

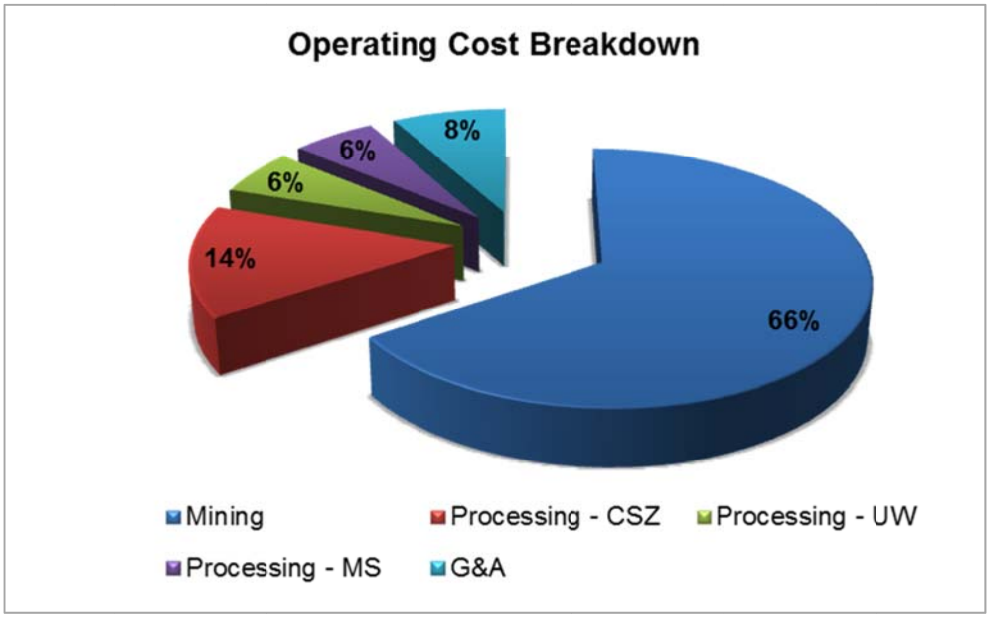
22.3 Summary of Operating Costs

Total operating cost over the life of mine amount to $1,211.3M. This translates to an average cost of $51.03/tonne processed over the life of mine. These costs are outlined in Table 22.3. Figure 22.3 shows the distribution of operating costs over the life of mine.

Table 22.3: Breakdown of Operating Costs

Operating Costs	$/t milled	LOM C$M
Mining	33.54	796.2
Processing - CSZ	12.91	174.9
Processing - UW	14.08	71.2
Processing - MS	13.94	71.6
G&A	4.10	97.4
Total	**51.03**	**1,211.3**

Figure 22.3: Distribution of Operating Costs



Source: JDS 2014

22.4 Taxes

The project has been evaluated on an after-tax basis in order to reflect a more indicative, but still approximate, value of the project. Saskatchewan Federal and Provincial Income Tax rates were applied to the project. A detailed tax analysis was completed by independent consultants for the purpose of the After-Tax valuation of the project.

Specific assumptions and methodology in the analysis includes the following:

22.4.1 Saskatchewan Mineral Tax

Saskatchewan mineral tax rate is equal to 10% of taxable income. Saskatchewan allows a 10-year mineral tax holiday which has also been considered in the tax analysis. Total life of mine Saskatchewan mineral taxes amount to $40.9M.

22.4.2 Federal & Provincial Corporate Income Tax

Federal tax rate of 15.0% and a combined BC (11.0%) and Saskatchewan (11.5%) rate were used to determine a blended 26.5% rate which was used to calculate income taxes.

22.4.3 Mineral Property Tax Pools

Canadian Exploration Expense (CEE) and Canadian Development Expense (CDE) tax pools were used with appropriate opening balances to calculate income taxes.

22.4.4 Federal Investment Tax Credits

Appropriate opening balances were used to calculate the Federal Investment Tax Credits for the project with respect to the pre-production capital costs of the project.

22.4.5 Capital Cost Allowance (CCA)

Capital cost specific CCA rates were applied to and used to calculate the appropriate amount of CCA the Company can claim during the life of the project.

The tax analysis completed amount to a life of mine taxes payable of $248.4M. The after-tax values are determined solely for project valuation purposes.

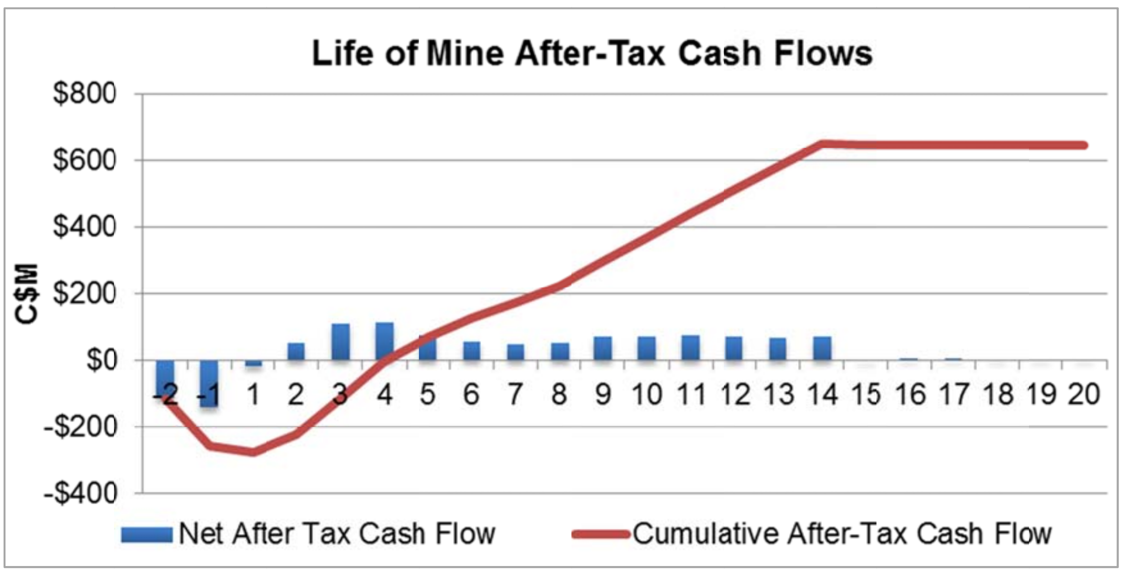
22.5 Economic Results

The project is economically viable with an after-tax internal rate of return (IRR) of 18.9% and a net present value at 7% ($NPV_{7\%}$) of 262.6M using the October 15, 2014 spot metal prices and Canadian/US Dollar foreign exchange rate.

Table 22.4: Summary of Base Case Economic Results

Category	Unit	Value
LOM Pre-Tax Free Cash Flow	$M	894.6
Average Annual Pre-Tax Free Cash Flow	$M	65.5
LOM Taxes	$M	248.4
LOM After-Tax Free Cash Flow	$M	646.2
Average Annual After-Tax Free Cash Flow	$M	47.3
Discount Rate	%	7.0
Pre-Tax NPV	$M	381.7
Pre-Tax IRR	%	21.9
Pre-Tax Payback	Years	4.1
After-Tax NPV	$M	262.6
After-Tax IRR	%	18.9
After-Tax Payback	Years	4.1

Figure 22.4: Life of Mine Cash Flows



22.6 Sensitivity Analysis

A sensitivity analysis was performed to test project value drives on project Net Present Value using a 7% discount rate. The results of this analysis are demonstrated in Table 22.5 and Figure 22.5. The project proved to be most sensitive to changes in foreign exchange rates and metal prices, and least sensitive to capital costs.

A sensitivity analysis of the pre-tax and after-tax results was performed using various discount rates. The results of this analysis are demonstrated in Table 22.6.

Table 22.5: After-Tax $NPV_{7\%}$ Sensitivity Test Results

Factor	-10%	100%	10%
Metal Price	129.4	262.6	394.5
Head Grade	156.3	262.6	368
OPEX	313.7	262.6	211.3
CAPEX	297.5	262.6	227.7
F/X Rate	378.7	262.6	166.9

Figure 22.5: After-Tax $NPV_{7\%}$ Sensitivity Graph

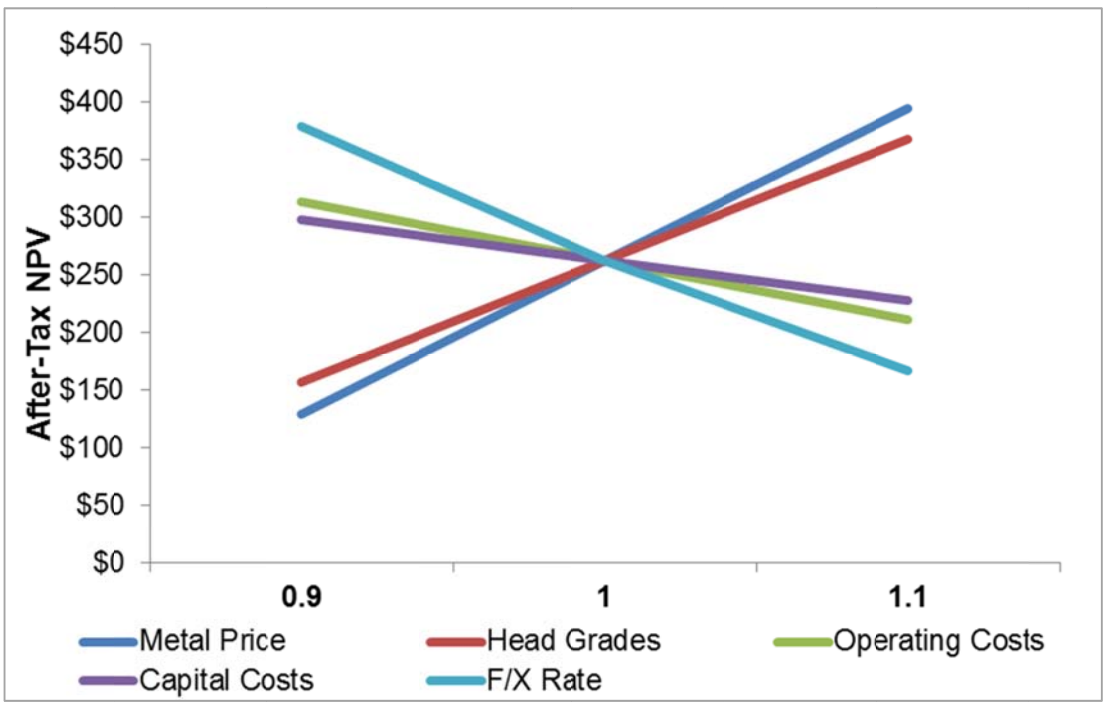


Table 22.6: Discount Rate Sensitivity Test Results

Discount Rate	Pre-Tax NPV $M	After-Tax NPV $M
0%	894.6	646.2
5%	490.2	344.4
7%	381.7	262.6
8%	335.5	227.5
10%	256.2	167
12%	191.2	117.2

23 ADJACENT PROPERTIES

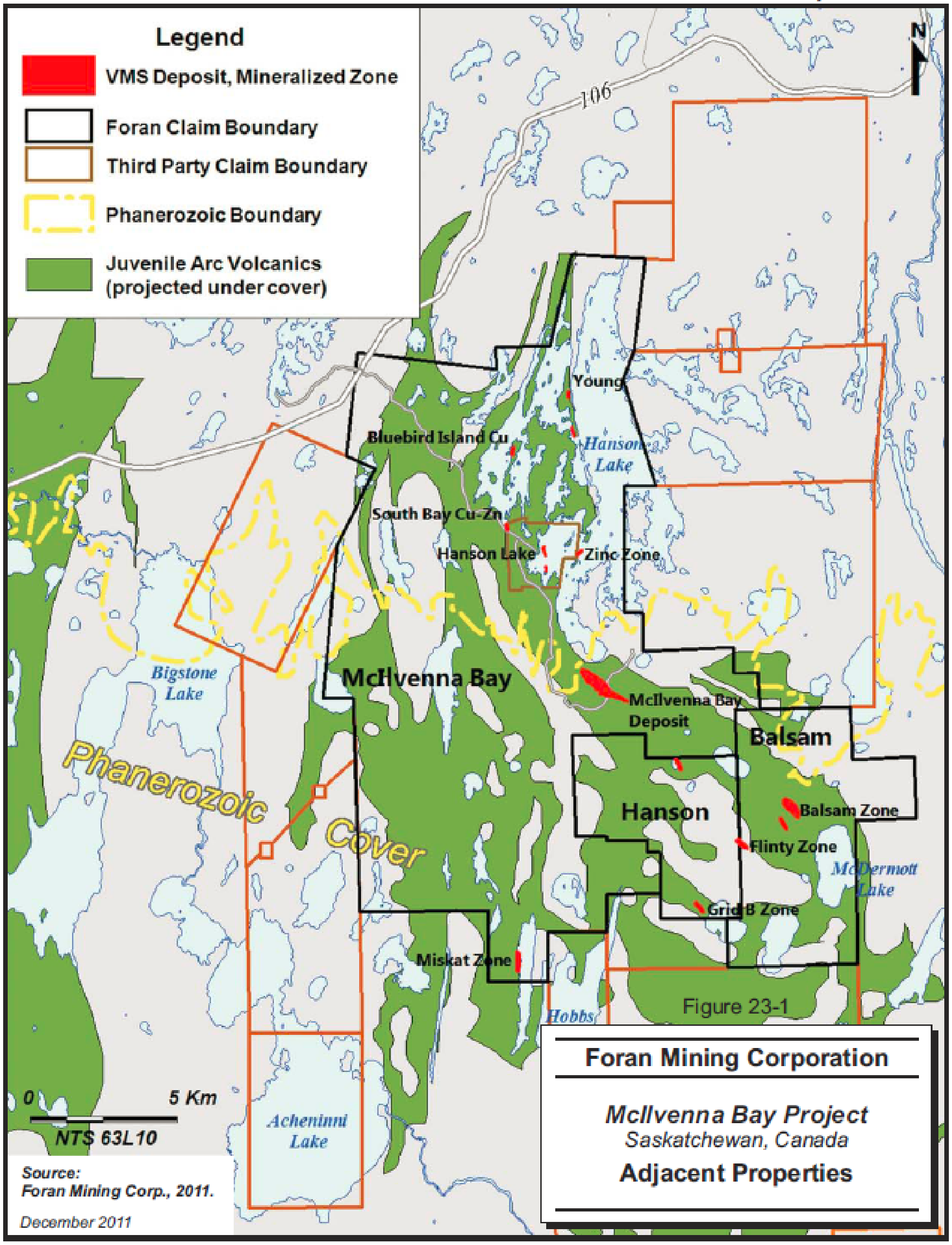
There are no producing metal mines adjacent to the McIlvenna Bay property. However, Preferred has active silica sand quarry leases immediately northeast of McIlvenna Bay that, in part, overlies Foran's mineral dispositions (acquired originally in 1986). Preferred and its predecessors have held the quarry leases since at least 1998 and have acquired additional leases up to 2006. In order to access the sand, Preferred blasts up to 25 m of dolomite cap rock, allowing them to access three to five metres of silica sand, which it mines, washes, and sorts into various grain sizes. The sand is marketed throughout western Canada and the US where it is used for hydraulic fracturing ("fracing").

Other VMS-style prospects are known to exist on Foran's claims and on adjacent ground (see Figure 23.1). The more significant of these include the Balsam/Thunder Zone, located southeast of McIlvenna Bay, the Miskat Zone, which is located in the southernmost extremity of the property and the historic Bigstone deposit located on an adjacent property 25km to the west.

The past producing Hanson Lake Mine is also located approximately 5 km to the northwest of McIlvenna 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.

McIlvenna Bay remains the most important deposit in the district.

Figure 23.1: Adjacent Properties



Source: RPA 2014

24 OTHER RELEVANT DATA AND INFORMATION

At the current time, Preferred has a silica sand quarry within one kilometre of the areas where Foran is drilling, which does not impact on Foran exploration activities. Foran retains a Miscellaneous Use Permit (MUP 602369) for the southern 8.9 km of the Project access road with the Saskatchewan Ministry of Environment, for which annual fees are paid for by Foran and reimbursed by Preferred. Preferred maintains the road as an active haul route for its operations. Foran currently uses the road for mineral exploration access to McIlvenna Bay and its exploration camp site.

Several quarry leases were obtained by Preferred (through a precursor company) and approved by the Ministry of Environment for mining in April 2009. These leases are located in proximity to and partially overlie a portion of the east-central part of the McIlvenna Bay deposit. In this area, the upper edge of the deposit is at a depth of 100 m or more below surface as it plunges off to the northwest. There is a possibility that future quarrying in some portions of those leases could interfere with ongoing exploration at McIlvenna Bay by restricting surface access for drill stations, depending on the timing of quarry development and ongoing exploration drilling by Foran. Detailed engineering and mine planning work will need to be completed for the McIlvenna Bay deposit in order to determine if ongoing quarrying operations by Hanson Lake could have an impact on or possibly indirectly interfere with future mining at McIlvenna Bay.

Foran management has met with Preferred management to discuss the possible conflict that could arise in the future due to the overlying mineral and quarrying claims. Both sides have agreed that regular communication of their current and planned activities will help prevent any possible short-term conflicts from arising. Foran has communicated with various departments within the Saskatchewan Ministry of Energy and Resources and Ministry of Environment that regulate permitting and the granting of licences. The government departments have been apprised that the conflicting overlying mineral and quarry claims will require a resolution, so that Foran can proceed with future proposed development at McIlvenna Bay unimpeded, and is working with both parties towards a resolution of the potential conflicts.

25 INTERPRETATION AND CONCLUSIONS

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the positive economic outcome shown for McIlvenna Bay. Standard industry practices, equipment and design methods were used in the PEA.

McIlvenna Bay contains a substantial base metal resource that can be mined by underground methods and recovered with conventional processing.

Using the assumptions contained in this report, the project is economic and should proceed to the pre-feasibility or feasibility stage.

To date, the QPs are not aware of any fatal flaws for the McIlvenna project.

25.1 Risks

As with most mining projects there are many risks that could affect the economic viability of the project. Many of these risks are based on lack of detailed knowledge and can be managed as more sampling, testing, design, and engineering are conducted at the next study stages. Table 25.1 identifies what are currently deemed to be the most significant internal project risks, potential impacts, and possible mitigation approaches.

The most significant potential risks associated with the project are uncontrolled dilution, uncontrolled groundwater inflow in the underground mine, lower metal recoveries than those projected, operating and capital cost escalation, permitting and environmental compliance, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and metal price. These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning and pro-active management.

External risks are, to a certain extent, beyond the control of the project proponents and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. External risks are things such as the political situation in the project region, metal prices, exchange rates and government legislation. These external risks are generally applicable to all mining projects. Negative variance to these items from the assumptions made in the economic model would reduce the profitability of the mine and the mineral resource estimates.

Table 25.1 Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Batch Material Handling and Processing	The handling and batch processing of three different mineral zones may be difficult and may negatively affect metal recovery.	Detailed planning, stope sequencing and material handling. Stockpiling of individual mineral zones for batch processing.
Dilution	Higher than expected dilution has a severe impact on project economics. The mine must ensure accurate drilling and blasting practices are maintained to minimize dilution, minimize secondary breaking and optimize extraction. The ability to segregate higher grade material, early in the mine life, is critical to project economics.	A well planned and executed grade control plan is necessary immediately upon commencement of mining.
Resource Modelling	All mineral resource estimates carry some risk and are one of the most common issues with project success. 43% of the resources in the mine plan are Inferred.	Infill drilling may be recommended in order to provide a greater level of confidence in the resource.
Metallurgical Recoveries	Changes to metallurgical assumptions could lead to reduced metal recovery, increased processing costs, and/or changes to the processing circuit design. If LOM metal recovery is lower than assumed, the project economics would be negatively impacted.	Additional sampling and test work could be conducted if applicable.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of project success. If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineable resource would reduce yielding fewer mineable tonnes.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
Permit Acquisition	The ability to secure all of the permits to build and operate the project is of paramount importance. Failure to secure the necessary permits could stop or delay the project.	The development of close relationships with the local communities and government along with a thorough Environmental and Social Impact Assessment and a project design that gives appropriate consideration to the environment and local people is required. Maintain direct control with a clear solution.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Development Schedule	The project development could be delayed for a number of reasons and could impact project economics. A change in schedule would alter the project economics.	If an aggressive schedule is to be followed, PFS or FS field work should begin as soon as possible.
Overall Mine Stability	Mining with backfill may not be sustainable. The current design calls for all mined voids to be filled with paste backfill.	Overall geotechnical stability of the mine needs to be assessed.
Ability to Attract Experienced Professionals	The ability to attract and retain competent, experienced professionals is a key success factor for the project. High turnover or the lack of appropriate technical and management staff at the project could result in difficulties meeting project goals.	The early search for professionals as well as competitive salaries and benefits identify, attract and retain critical people.

Source: JDS 2014

25.2 Opportunities

There are significant opportunities that could improve the economics, timing, and/or permitting potential of the project. The major opportunities that have been identified at this time are summarized in Table 25-2, excluding those typical to all mining projects, such as changes in metal prices, exchange rates, and etcetera. Further information and assessments are needed before these opportunities should be included in the project economics.

Table 25.2 Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Expansion of the Mine	The mineral resource has not been fully delineated and there is an opportunity to expand the mineable resource.	Increased mine life.
Increased Production	Increased production may be possible in high TVPM levels. There is an opportunity for the mine to produce more tonnes for short durations on the high tonnage levels of the mine.	Reduced unit operating costs and increased revenue.
Optimize Mine Plan	Optimize the mine plan and stope sequence.	Decrease ramp-up duration and potentially higher grades earlier in the mine life.

Contract Mining	Contract mining instead of owner mining.	Reduce Capex
Backfill Cement Content	Paste backfill testing may reduce the cement content assumptions.	Reduce mining costs
Concentrate Smelting	Copper and bulk concentrates are currently assumed to be shipped overseas. There may be potential to source north American smelter capacity to reduce concentrate transport costs. It may be possible to obtain better treatment and/or refining terms from smelters through formal negotiations in the future	Reduced transportation and concentrate shipping costs. Reduced concentrate treatment and refining costs
Satellite Deposits	There exist several historic deposits and numerous both new and historic prospects in close proximity to McIlvenna Bay which could provide additional mill feed for the development	Additional mill feed (especially at higher grade) could improve the project economics by speeding up project payback and/or extending the mine life

26 RECOMMENDATIONS

It is recommended that the project proceed to the feasibility study stage in line with Foran's desire to advance the project. It is also recommended that environmental and permitting continue as needed to support Foran's project development plans.

Prior to and/or concurrent with the studies and work programs designed to advance the McIlvenna Bay deposit towards Feasibility level studies, work should also be completed to validate and define several of the known additional sources of mineral resources which lie in close proximity of the deposit. These potential resources consist of an historic deposit and an advanced exploration target, namely the historic Bigstone deposit and Thunder Zone prospect, which occur in proximity of McIlvenna Bay and may provide additional higher grade mill feed to the project early in the project life to reduce the payback period. Additional resources defined in these areas (assuming the completion of successful exploration programs) could substantially enhance the economics of development of McIlvenna Bay by adding additional higher grade mill feed early in the mine cycle and at the same time extend the life of any such operations.

An exploration program which encompasses diamond drilling and ancillary studies designed to produce 43-101 compliant resources and determine the suitability of the two target areas for mining is recommended. It is envisioned that exploration programs would be conducted in two phases. The phase I program should consist of 4,000-5,000m of diamond drilling on the two target areas to establish that the required thresholds of potentially economic resources are present. It is estimated that the cost of the phase I program would be approximately $1.2-1.5M. Contingent on positive results from the initial program, a phase II program would be recommended consisting of additional diamond drilling of a density sufficient to support indicated resource classifications and initial metallurgical studies to determine potential recoveries, etc.

Assuming the successful completion of these proposed exploration programs, the additional satellite deposits should then be included in a revised PEA for the project and/or the deposits included in future Feasibility studies.

It is estimated that a feasibility study and supporting field work would cost approximately $13.0 million. A breakdown of the key components of the next study phase is as follows in Table 26.1.

Table 26.1 Cost Estimate to Advance the Project to Feasibility Stage

Component	Estimated Cost (M$)	Comment
Resource Drilling & Updated Resource	6.1	Conversion of inferred resources to indicated within and immediately adjacent to the proposed mine. Drilling will include holes for combined resource, geotech and hydrogeology purposes.
Metallurgical Testing	0.5	Variability test work including expanded grinding testwork, evaluation of blending of mineralization types and testwork for ancillary processes (thickening and filtering)
Condemnation Drilling	0.4	Drilling under infrastructure and TMF to ensure no sterilization of resources
Geochemistry	1	ABA accounting tests and humidity cell testing to determine acid generating potential of all rock units and mitigation plans
Geotechnical/ Hydrology/Hydrogeology	0.5	Mine and surface facilities geotechnical investigations (logging, test pitting, sampling, lab tests, etc.)
Engineering & Paste Backfill Testing	4	FS-level mine, infrastructure, paste backfill & process design, cost estimation, scheduling & economic analysis
Environment	0.5	Other investigations including, water quality, fisheries, wildlife, weather, traditional land use & archaeology
Total	**13.0**	Excludes corporate overheads and future permitting activities

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APPENDIX A:
QP CERTIFICATES

PARTNERS IN ACHIEVING MAXIMUM RESOURCE DEVELOPMENT VALUE

JDS Energy & Mining Inc.

CERTIFICATE OF AUTHOR

I, Michael E. Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of November 12, 2014, prepared for Foran Mining Corporation;

2. I am currently employed as a Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 860 – 625 Howe Street, Vancouver British Columbia, V6C 2T6;

3. I am a graduate of the University of Alberta with a BSc. In Mining Engineering, 1988. I have practiced my profession continuously since 1988;

4. I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over seven years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

6. 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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have visited the McIlvenna Bay Project site July 23, 2014;

8. I am responsible for Sections1 (except 1.3-1.6, 1.9), 2, 3, 15, 16 (except (16.2 and 16.12), 18 (Except 18.19), 20, 21 (except 21.1.2 and 21.2.2), 22, 24-27;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of the date of this certificate, 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;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: November 12, 2014
Signing Date: December 22, 2014

"Original Signed & Stamped"

Michael E. Makarenko, P. Eng.

Golder Associates

December 22, 2014 Reference No. 11-1426-0006-00?-L-Rev0-??

CERTIFICATE OF AUTHOR

I, Darren Kennard, P.Eng. do hereby certify that:

1) This certificate applies to the Technical Report entitled "Preliminary Economic Assessment, Technical Report, McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of November 12th, 2014, prepared for the Foran Mining Corporation.;

2) I am currently employed as a Senior Mining Geotechnical Engineer – Associate, with Golder Associates Ltd. with an office at Suite 200 - 2920 Virtual Way, Vancouver, BC, V5M 0C4.

3) I am a graduate of the University of Saskatchewan with a Bachelors Degree in Geological Engineering in 1992 and a Master's of Science Degree in 1998. I have practiced my profession continuously since 1995;

4) I am a Registered Professional Mining Engineer in British Columbia (#25639);

5) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and in provision of geotechnical consulting services to underground mines in similar geotechnical settings to Silvertip in: BC; Nevada; and Alaska over a period of 12 years, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

6) I was responsible for preparing Section 16.2.

7) I have not visited the McIlvenna Bay project site.

8) Prior involvement in the project included geotechnical consulting work (Golder) for Foran Mining Corporation on the McIlvenna Bay Project since late 2011.

9) As of the date of this certificate, 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 compliance with NI 43-101 and Form 43-101F1.

Signing Date: December 22, 2014

GOLDER ASSOCIATES LTD.

Signature and Stamp

Darren Kennard

o:\active_2011\1426\11-1426-0006 foran mining corp\draft pea - dec 10 2014\consent forms and experience\qp form foran mcilvenna darren kennard golder 17dec_14.docx

Golder Associates Ltd.
500 - 4260 Still Creek Drive, Burnaby, British Columbia, Canada V5C 6C6
Tel: +1 (604) 296 4200 Fax: +1 (604) 298 5253 www.golder.com

Golder Associates

December 22, 2014

Reference No. 1114260006-039-L-Rev0-1000

CERTIFICATE OF AUTHOR

I, John Hull, M.Sc., P.Eng. do hereby certify that:

1) This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of November 12, 2014, prepared for Foran Mining Corporation.
2) I am currently employed as a Senior Project Manager with Golder Associates with an office at Suite 200 – 2920 Virtual Way, Vancouver British Columbia, V5M 0C4.
3) I am a graduate of the Queen's University with a BSc and MSc., in Civil Engineering, 1972 and 1973, I have practiced my profession continuously since 1974.
4) I have been an independent consultant for over 15 years and have performed tailings facility designs, tailings closure planning, cost estimation, technical due diligence reviews and report writing for tailings facilities for mining projects.
5) I am a Registered Professional Mining Engineer in British Columbia (#9835) and the Northwest Territories (#L1045).
6) 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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
7) I have not visited the McIlvenna Bay Project site.
8) I am responsible for Section 18.19.
9) I have had no prior involvement with the property that is the subject of this Technical Report.
10) As of the date of this certificate, 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.
11) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signing Date: December 22, 2014

GOLDER ASSOCIATES LTD.

ORIGINAL SIGNED AND SEALED

John Hull, M.Sc., P.Eng.
Principal, Mining Division

JAH/kp

o:\final\2011\1426\11-1426-0006 foran\1114260006-039-l-rev0-1000\1114260006-039-l-rev0-1000-qp form foran mcilvenna john hull golder 22dec_14.docx

CERTIFICATE OF QUALIFIED PERSON

I, David W. Rennie, P.Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada”, with an effective date of November 12, 2014, prepared for Foran Mining Corporation.

2. I am a Principal Geologist with Roscoe Postle Associates Inc. My office address is Suite 388, 1130 West Pender Street, Vancouver, British Columbia, Canada V6E 4A4.

3. I am a graduate of the University of British Columbia in 1979 with a Bachelor of Applied Science degree in Geological Engineering. I have worked as a geological engineer for a total of 35 years since my graduation.

4. My relevant experience for the purpose of the Technical Report is:
    - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements
    - Consultant Geologist to a number of major international mining companies providing expertise in conventional and geostatistical resource estimation for properties in North and South Americas, and Africa
    - Chief Geologist and Chief Engineer at a gold-silver mine in southern British Columbia
    - Exploration geologist in charge of exploration work and claim staking with two mining companies in British Columbia

5. I am registered as a Professional Engineer in the Provinces of British Columbia (Reg. #13572) and Nova Scotia (Lic. #L4989).

6. 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. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

7. I visited the McIlvenna Bay Project site on September 22-23, 2011.

8. I am responsible for Sections 4 to 12, inclusive, and Section 14; and contributed to Sections 1, 25, and 26 of the Technical Report.

9. I previously prepared an independent Technical Report on the property that is the subject of the Technical Report.

10. As of the date of this certificate, to the best of my knowledge, information, and belief, the Sections for which I am responsible in the Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: November 12, 2014
Signing Date: December 22, 2014

"Original Signed & Stamped"

David W. Rennie, P. Eng.

KWM CONSULTING INC.
KEN MAJOR

12576 206St
Maple Ridge, BC, V2X 3M2]
T (604) 465-8349
F N/A
E kwmajor@shaw.ca

CERTIFICATE of AUTHOR

I, Kenneth W Major, P.Eng., do certify that:

1. I am a Mineral Processing Engineer with KWM Consulting Inc. ("KWM") with an office at 12576 - 206St Maple Ridge, British Columbia
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of November 12, 2014, prepared for Foran Mining Corporation.
3. I graduated with the degree of Bachelor of Engineering (Metallurgical) from McGill University in Montreal, Quebec in 1976. I have practiced my profession since 1976. I have worked in mineral processing operations in technical and management positions and also in engineering and design as a process engineer, project manager and minerals processing consultant.
4. I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia (#13149). As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.
5. I have not visited the McIlvenna Bay Project site.
6. I am responsible for the preparation of: Section 1.5 and Section 13 of the Technical Report.
7. I am independent of Foran Mining Corporation in accordance with the application of Section 1.5 of National Instrument 43-101.
8. I have not had prior involvement with Foran Mining and the McIlvenna Bay Project that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 12, 2014
Signing Date: December 22, 2014

"Kenneth W Major"

Original signed

Kenneth W Major, P.Eng.

Paterson & Cooke Canada Inc. +1 (705) 222-1720
1351-C Kelly Lake Road, Unit 2, Sudbury, Ontario P3E 5P5, Canada
Sudbury@PatersonCooke.com, www.PatersonCooke.com

CERTIFICATE OF AUTHOR

I, Leslie D.C. Correia, Pr. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada", with an effective date of November 12, 2014, prepared for Foran Mining Corporation;

2. I am currently employed as a Process Engineer with Paterson & Cooke Canada Inc. with business address at 1351-C Kelly Lake Road, Unit #2, Sudbury, Ontario, P3E 5P5;

3. I am a graduate of the University of Stellenbosch (Bachelors of Engineering (Chemical), 2005. I am a member in good standing of the Engineering Council of South Africa (ECSA), License #20130236. My relevant experience is 7 years as an independent consultant. I have been responsible for hydraulic, process and mechanical design of slurry pump and pipeline systems, backfill plant and reticulation system design, capital and operation cost estimates and project management of mining projects worldwide;

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 am independent of Foran Mining Corporation as defined by Section 1.5 of the NI 43-101;

6. I did not complete a personal inspection of the McIlvenna Bay Project;

7. I am responsible for Sections 16.12 of the Technical Report;

8. I have had no prior involvement with the property that is the subject of the Technical Report;

9. I have read NI 43-101, and the sections of the Technical Report that I am responsible for has been prepared in compliance with NI 43-101.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: November 12, 2014
Signing Date: December 22, 2014

"Original Signed & Stamped"

Leslie D.C. Correia, Pr. Eng.

8450 East Crescent Parkway, Suite 200
Greenwood Village, CO 80111

Phone: 303.714.4840
FAX: 303.714.4800

CERTIFICATE OF AUTHOR

I, Matthew R. Bender, P. E., do hereby certify that:

1. This certificate applies to the Technical Report entitled “Preliminary Economic Assessment Technical Report, McIlvenna Bay Project, Saskatchewan, Canada”, with an effective date of November 12, 2014, prepared for Foran Mining Corporation;

2. I am currently employed as Director of Metallurgy with Samuel Engineering, Inc. with an office at 8450 E. Crescent Pkwy, Suite 200, Greenwood Village, CO 80111, USA;

3. I am a graduate of the Colorado School of Mines with a BSc. in Metallurgical Engineering, 1987. I have practiced my profession continuously since 1987;

4. I have worked in technical, operations and management positions at mines in the United States. I have been an independent consultant for over 15 years and have performed metallurgical testwork programs, process design, process engineering, start-ups, plant audits, process technology and equipment sales, cost estimation, technical due diligence reviews and report writing for mining projects worldwide;

5. I am a Registered Professional Metallurgical Engineer in the states of Nevada, USA (#11594) and Colorado, USA (#31471);

6. 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. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have not visited the McIlvenna Bay Project site;

8. I am responsible for Sections 1.9, 17, 21.1.2 and 21.2.2;

9. I have had no prior involvement with the property that is the subject of this Technical Report;

10. As of the date of this certificate, 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;

11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Effective Date: November 12, 2014
Signing Date: December 22, 2014

“Original Signed & Stamped”

Matt R. Bender, P. E.

APPENDIX B:
PROCESS FLOW DIAGRAM
(JDSM-32-0128-00)

STREAM NUMBER	1	2	3	4	5	6	7	8	9	10
DESCRIPTION	BACKFILL PLANT FEED	FILTER FEED PUMPS	FILTER CAKE DISCHARGE	BINDER	TRIM WATER	PASTE TO U/G	FILTRATE WATER	POTABLE WATER SUPPLY	CLARIFER OVERFLOW	PROCESS TANK OVERFLOW
UTILIZATION RATE (%)	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%
SOLIDS (t/h)	148	148	148	7.79	-	156	-	-	-	-
WATER (m^3/h)	79.8	79.8	39.4	0.00	3.33	42.7	40.4	13.0	40.4	37.1
TOTAL MIXTURE (m^3/h)	130	130	89.2	2.48	3.33	95.0	40.4	13.0	40.4	37.1
MIXTURE DENSITY (t/m^3)	1.756	1.756	2.099	3.138	0.998	2.088	0.998	0.998	0.998	0.998
VOLUME CONCENTRATION (%)	38.4%	38.4%	55.8%	100%	-	55.0%	-	-	-	-
MASS CONCENTRATION (%)	65.0%	65.0%	79.0%	100%	-	78.5%	-	-	-	-
SOLIDS DENSITY (t/m^3)	2.970	2.970	2.970	3.138	0.998	2.978	0.998	0.998	0.998	0.998
COMMENT										

REVISIONS					
REV	DATE	DESCRIPTION	DESG	DRWN	CHKD
A	23-SEP-14	ISSUED TO CLIENT	L.C.	T.S.	R.B.

REFERENCE DRAWINGS	
DRG No.	DESCRIPTION

PATERSON & COOKE

1351-C KELLY LAKE ROAD, UNIT #2
SUDBURY, ONT, CANADA
TEL: 705-222-1720, FAX: 705-222-1719
WWW.PATERSONCOOKE.COM

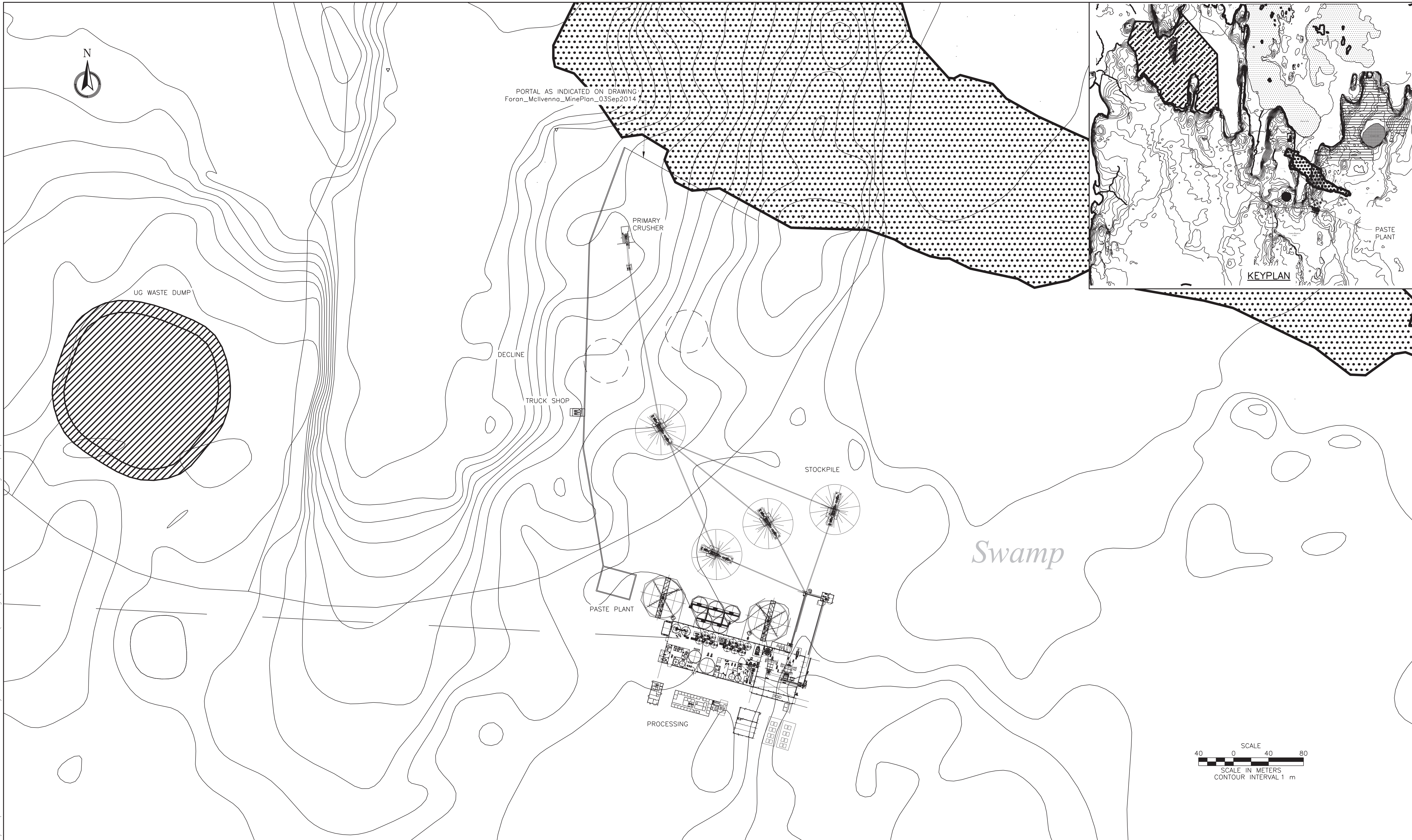
FORAN MINING CORPORATION

MCILVENNA BAY
BACKFILL PASTE PLANT – PEA

PASTE PLANT
PROCESS FLOW DIAGRAM

SCALE	N.T.S.	PROJECT NUMBER	JDM-32-0128	SIZE	D
DESIGNED	L.C.	09-SEP-14	DRAWING NUMBER	00-F01	REV A
DRAWN	T.S.	10-SEP-14			

APPENDIX C:
SITE PLAN
(JDM-32-0128-00)



REVISIONS					
REV	DATE	DESCRIPTION	DESG	DRWN	CHKD
A	23-SEP-14	ISSUED TO CLIENT	L.C.	T.S.	R.B.

REFERENCE DRAWINGS	
DRG No.	DESCRIPTION

PATERSON & COOKE

1351-C KELLY LAKE ROAD, UNIT #2
SUDBURY, ONT, CANADA
TEL: 705-222-1720, FAX: 705-222-1719
WWW.PATERSONCOOKE.COM

FORAN MINING CORPORATION

MCILVENNA BAY
BACKFILL PASTE PLANT – PEA

SITE PLANT WITH PROPOSED
PASTE PLANT LOCATION

SCALE		PROJECT NUMBER	SIZE
2 mm=1 m		JDM-32-0128	D
DESIGNED	L.C. 09-SEP-14	DRAWING NUMBER	REV
DRAWN	T.S. 10-SEP-14	00-L01	A

APPENDIX D:
CROSS SECTIONS

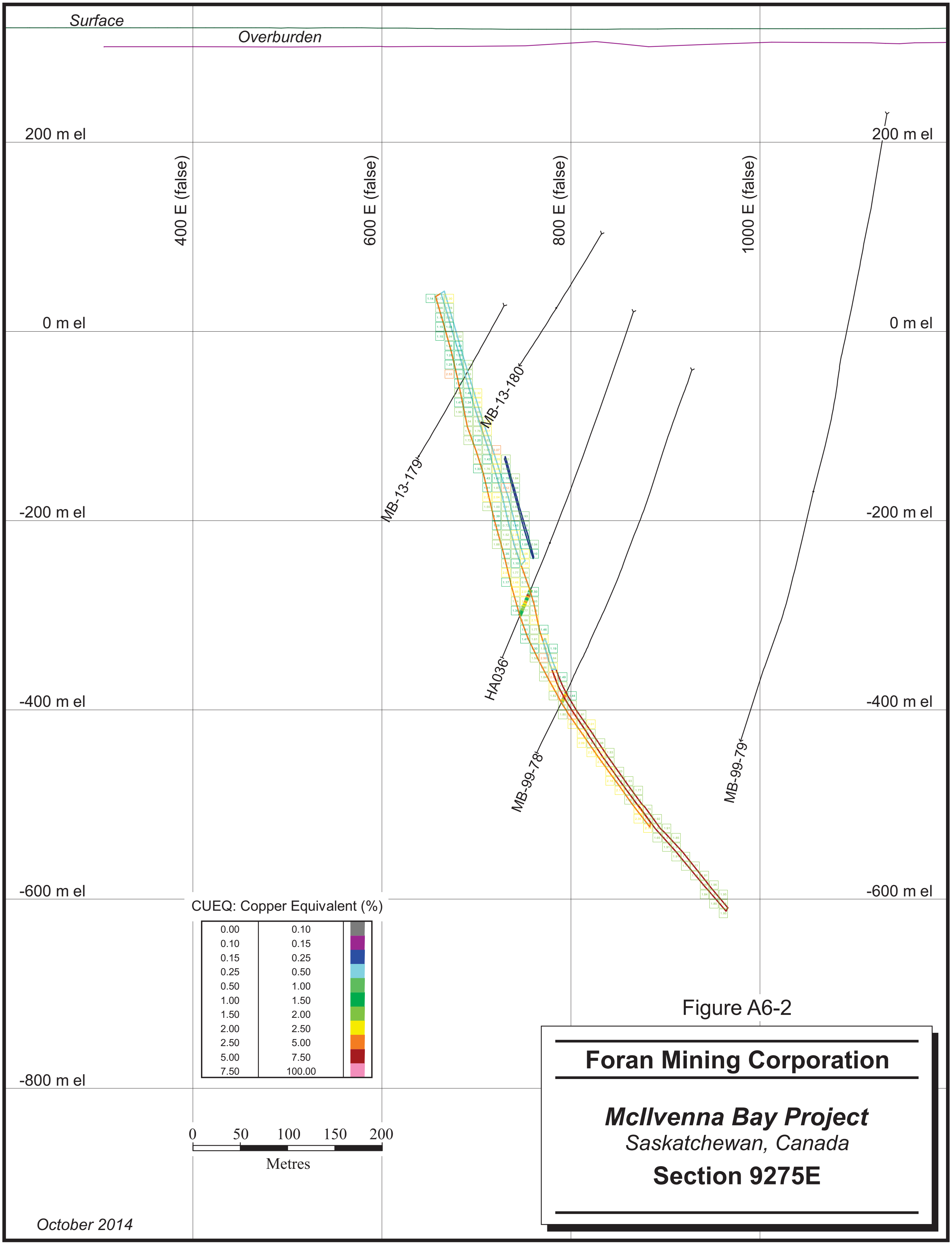


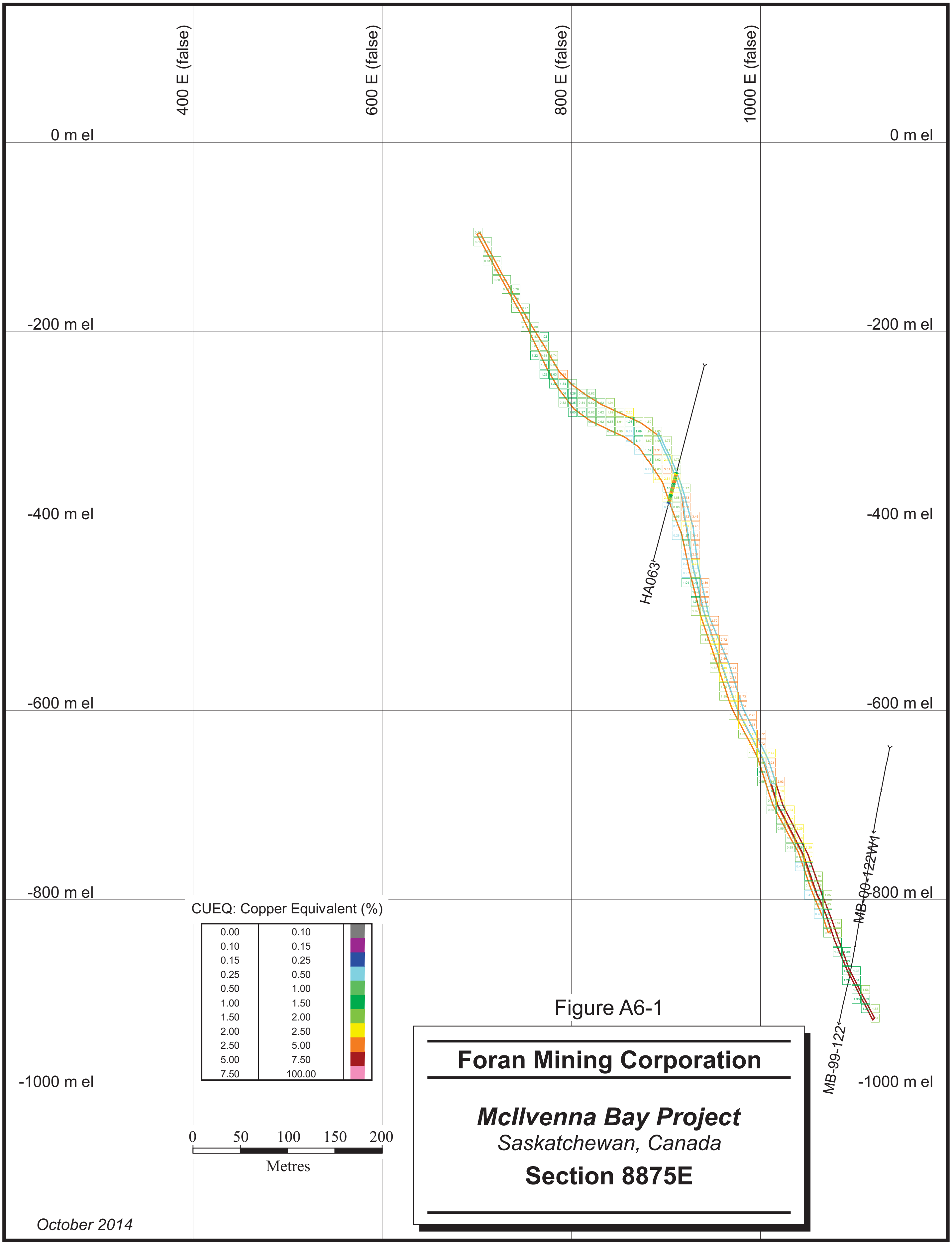
Figure A6-2

Foran Mining Corporation

McIlvenna Bay Project

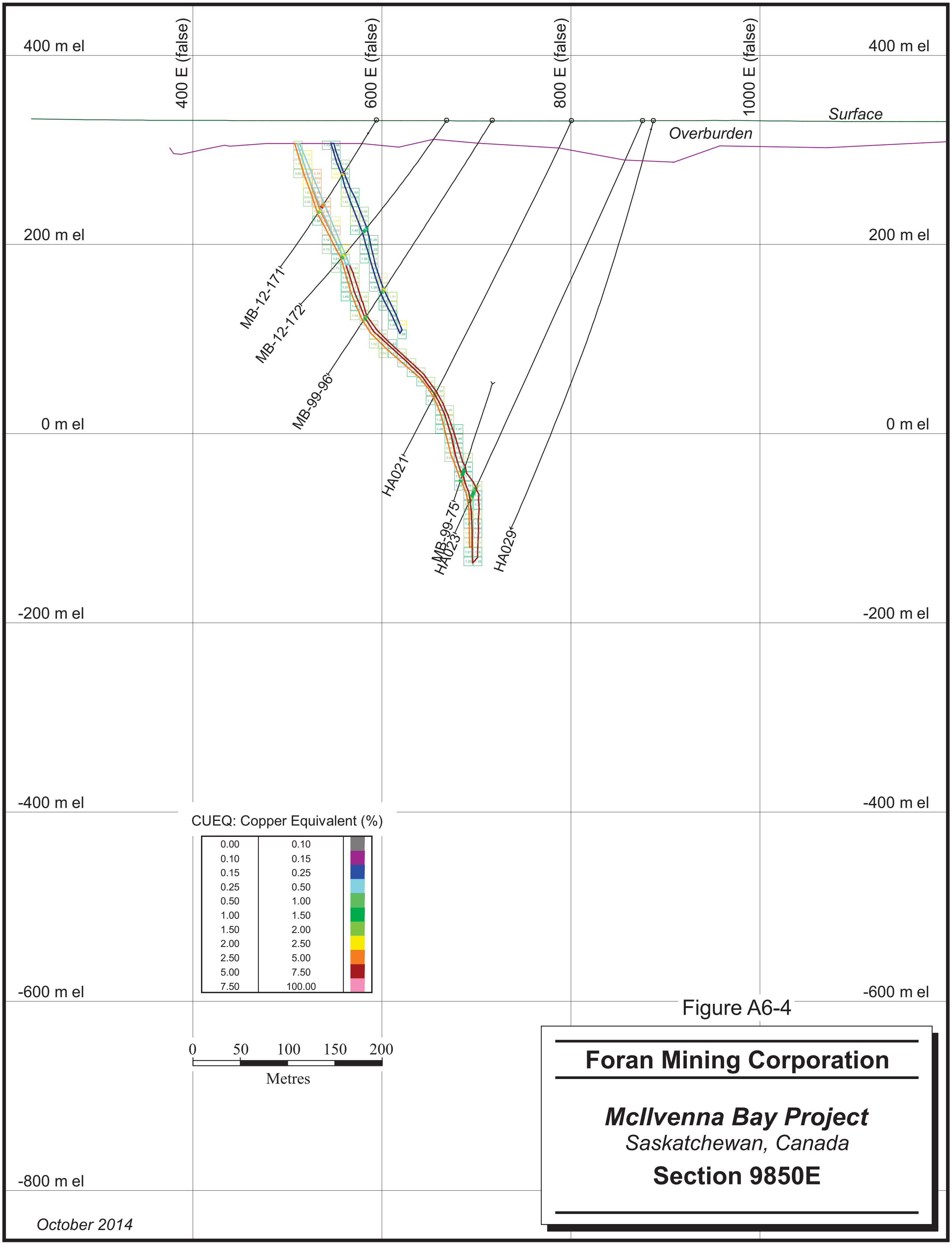
Saskatchewan, Canada

Section 9275E



400 E (false)
600 E (false)
800 E (false)
1000 E (false)
0 m el
-200 m el
-400 m el
-600 m el
-800 m el
-1000 m el
HA063
MB-00-122W1
MB-99-122
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 50 100 150 200
Metres
October 2014
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Section 8875E

Figure A6-1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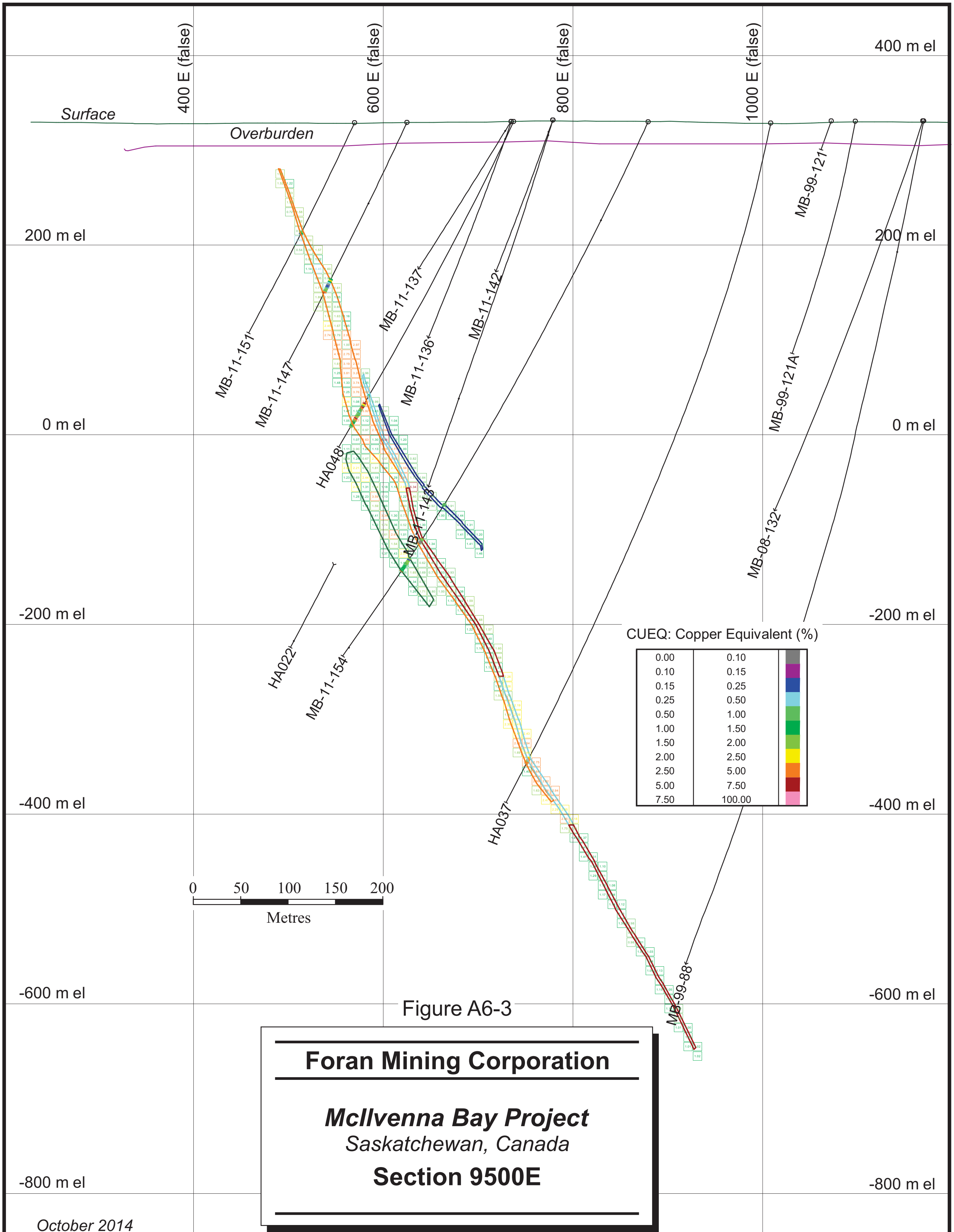
200 m el
0 m el
-200 m el
-400 m el
-600 m el
-800 m el
400 E (false)
600 E (false)
800 E (false)
1000 E (false)
Surface
Overburden
MB-12-171
MB-12-172
MB-99-96
HA021
MB-99-75
HA023
HA029
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 50 100 150 200
Metres
October 2014
Figure A6-4
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Section 9850E



0.00	0.10
0.10	0.15
0.15	0.25
0.25	0.50
0.50	1.00
1.00	1.50
1.50	2.00
2.00	2.50
2.50	5.00
5.00	7.50
7.50	100.00

Figure A6-3

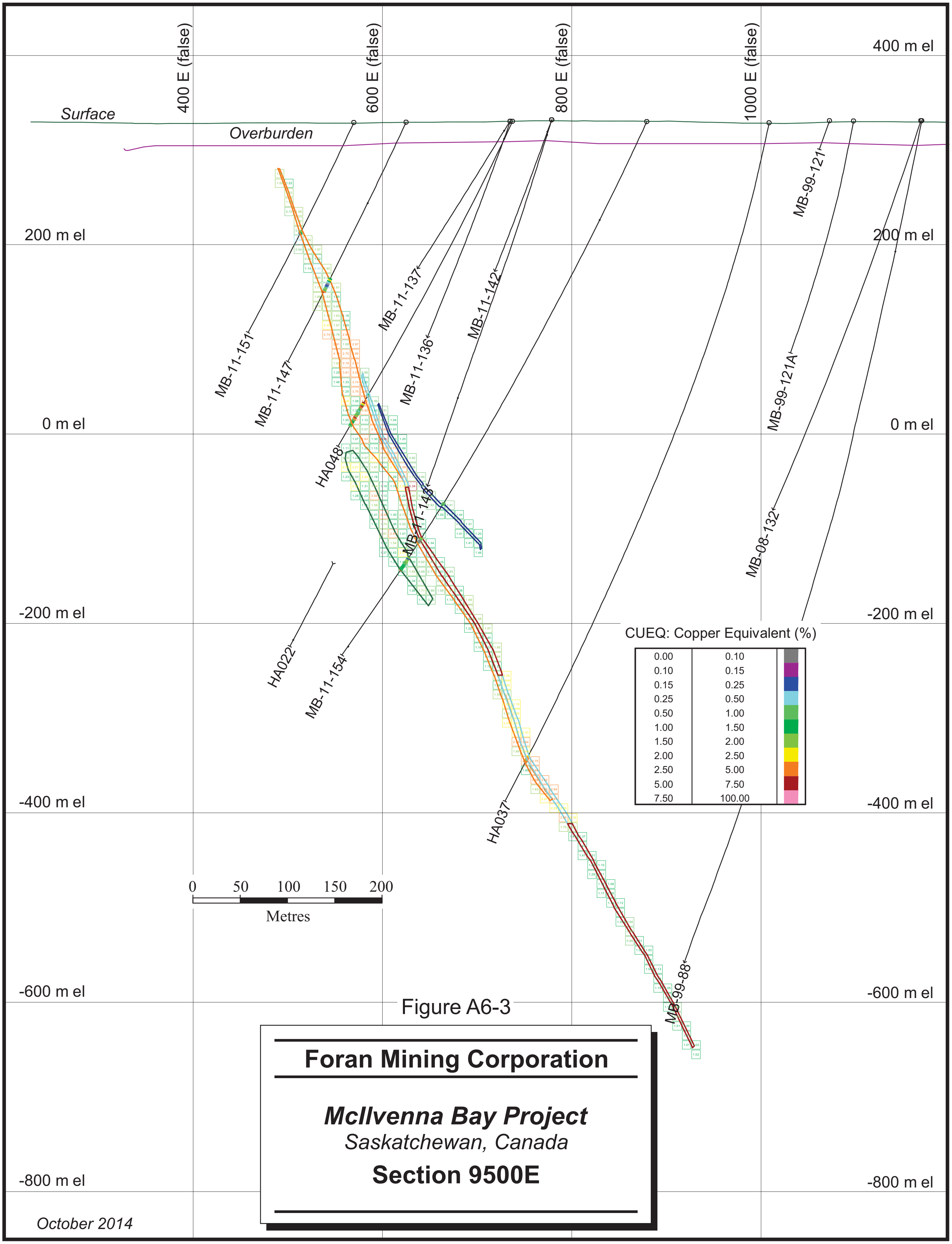
Foran Mining Corporation

McIlvenna Bay Project

Saskatchewan, Canada

Section 9500E

October 2014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600 E (false)
800 E (false)
1000 E (false)
400 m el
200 m el
0 m el
-200 m el
-400 m el
-600 m el
-800 m el
Surface
Overburden
MB-11-151
MB-11-147
MB-11-137
MB-11-136
MB-11-142
MB-11-143
MB-11-154
MB-99-121
MB-99-121A
MB-08-132
MB-99-88
HA048
HA022
HA037
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 50 100 150 200
Metres
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Section 9500E
October 2014
Figure A6-3

APPENDIX E:
LEVEL PLANS

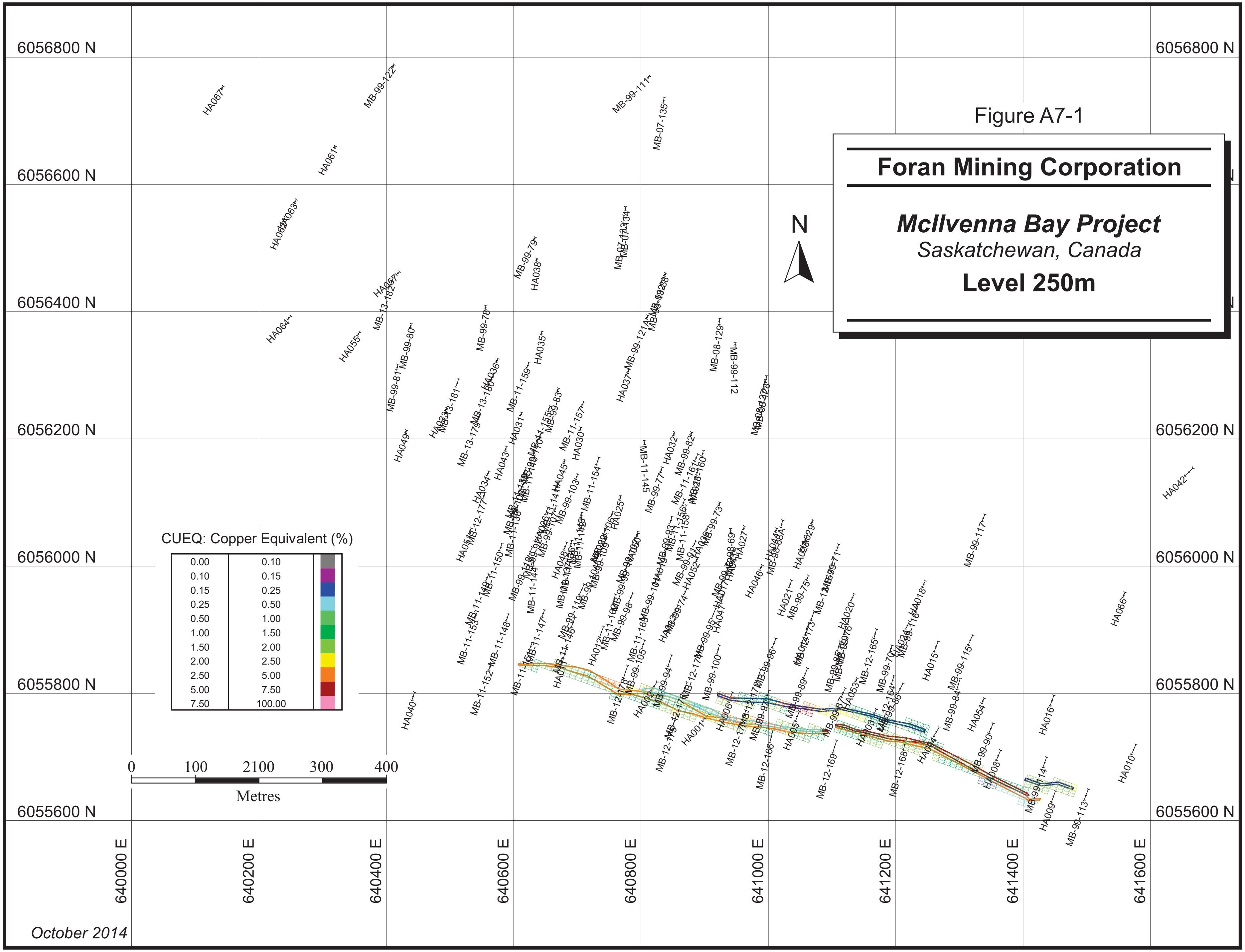
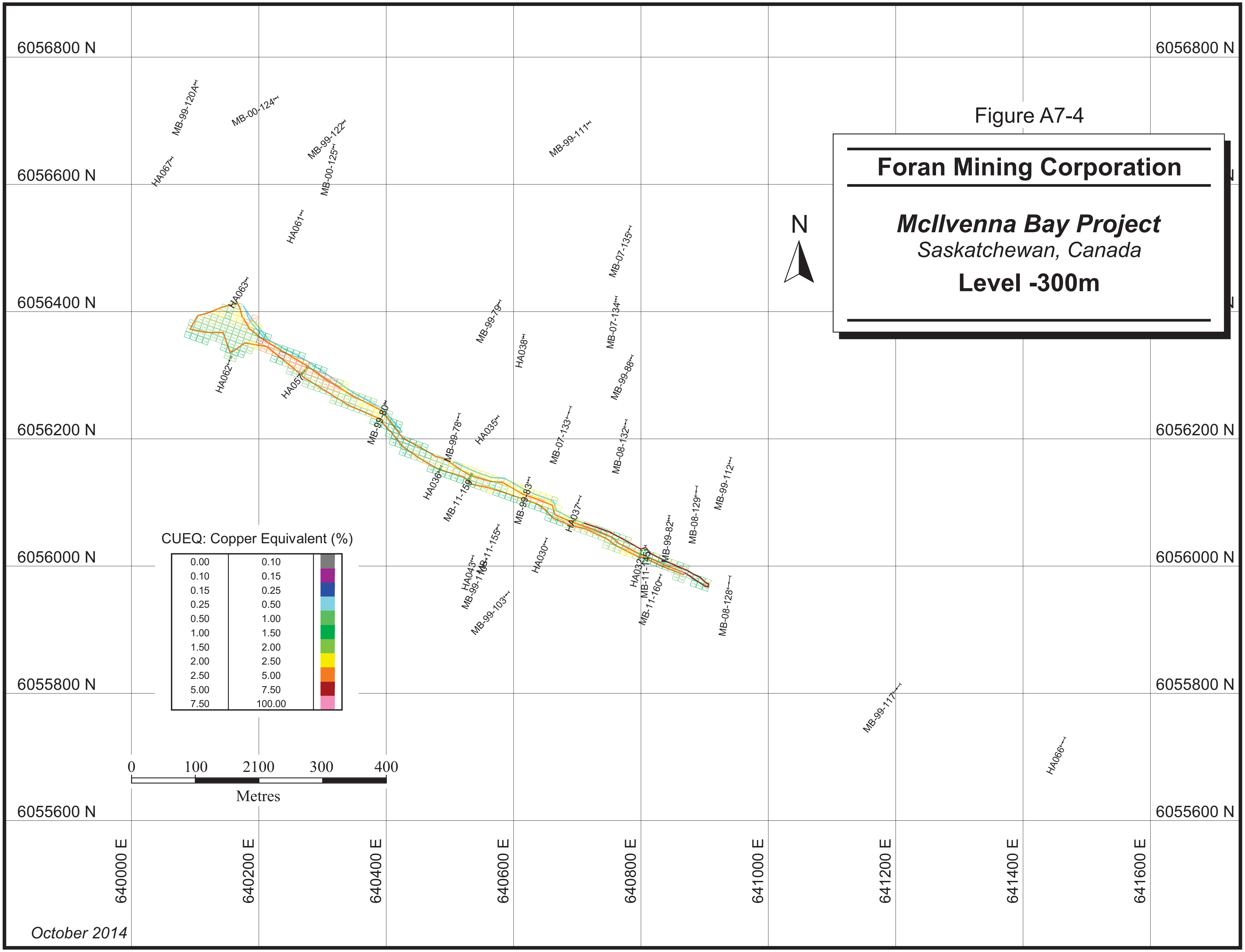


Figure A7-1
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Level 250m
N
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 100 200 300 400
Metres
6056800 N
6056600 N
6056400 N
6056200 N
6056000 N
6055800 N
6055600 N
640000 E
640200 E
640400 E
640600 E
640800 E
641000 E
641200 E
641400 E
641600 E
October 2014

Figure A7-4


Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Level -300m
N
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 100 2100 300 400
Metres
6056800 N
6056600 N
6056400 N
6056200 N
6056000 N
6055800 N
6055600 N
640000 E
640200 E
640400 E
640600 E
640800 E
641000 E
641200 E
641400 E
641600 E
MB-99-120A
MB-00-124
MB-99-122
MB-00-125
HA067
MB-99-111
HA061
MB-07-135
HA063
MB-99-79
HA038
MB-07-134
MB-99-88
HA062
HA057
MB-99-80
MB-99-78
HA035
MB-07-133
MB-08-132
MB-99-112
HA036
MB-11-159
MB-99-83
HA037
MB-08-129
MB-99-82
MB-11-155
HA030
HA043
HA032
MB-99-103
MB-11-160
MB-08-128
MB-99-117
HA066
October 2014

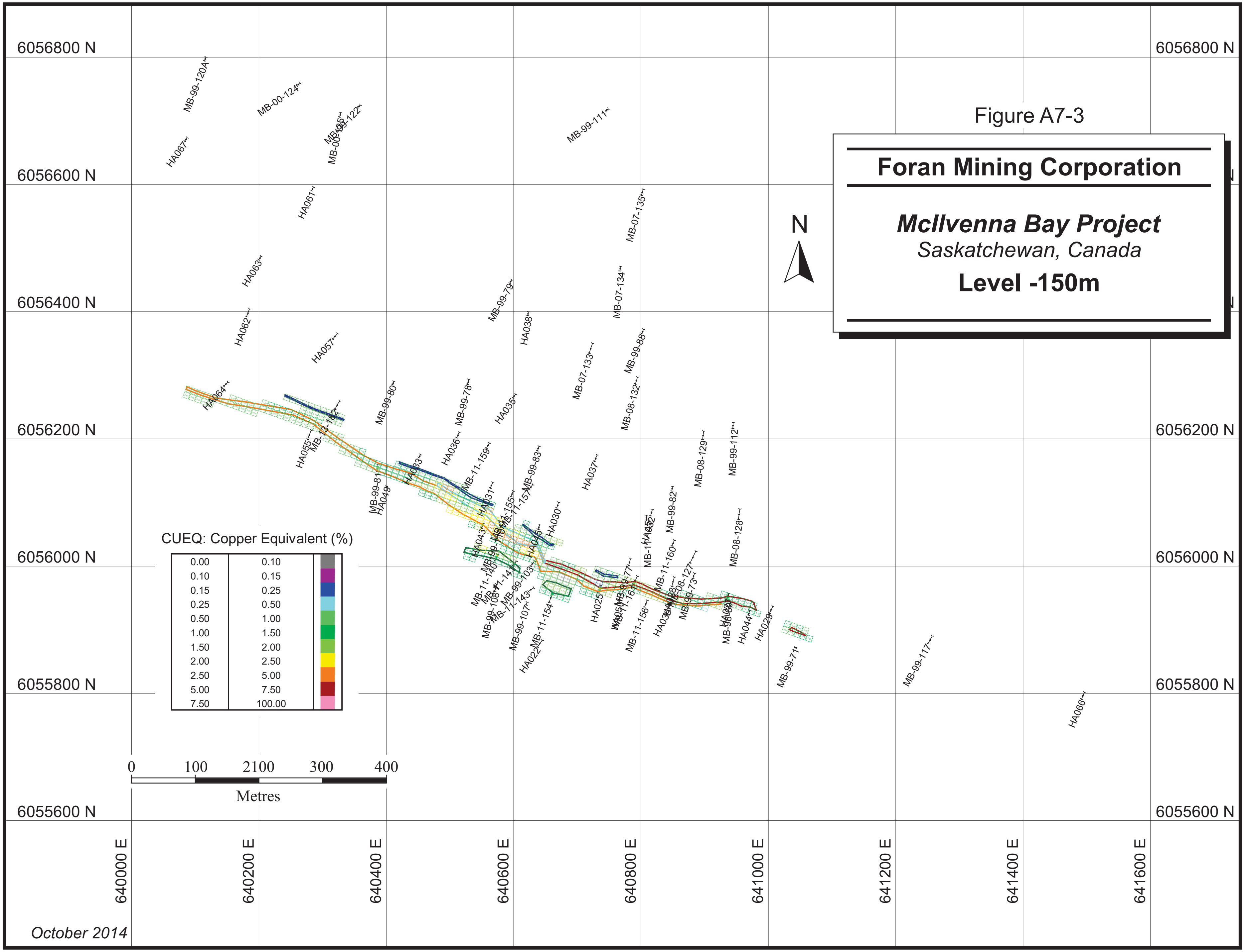
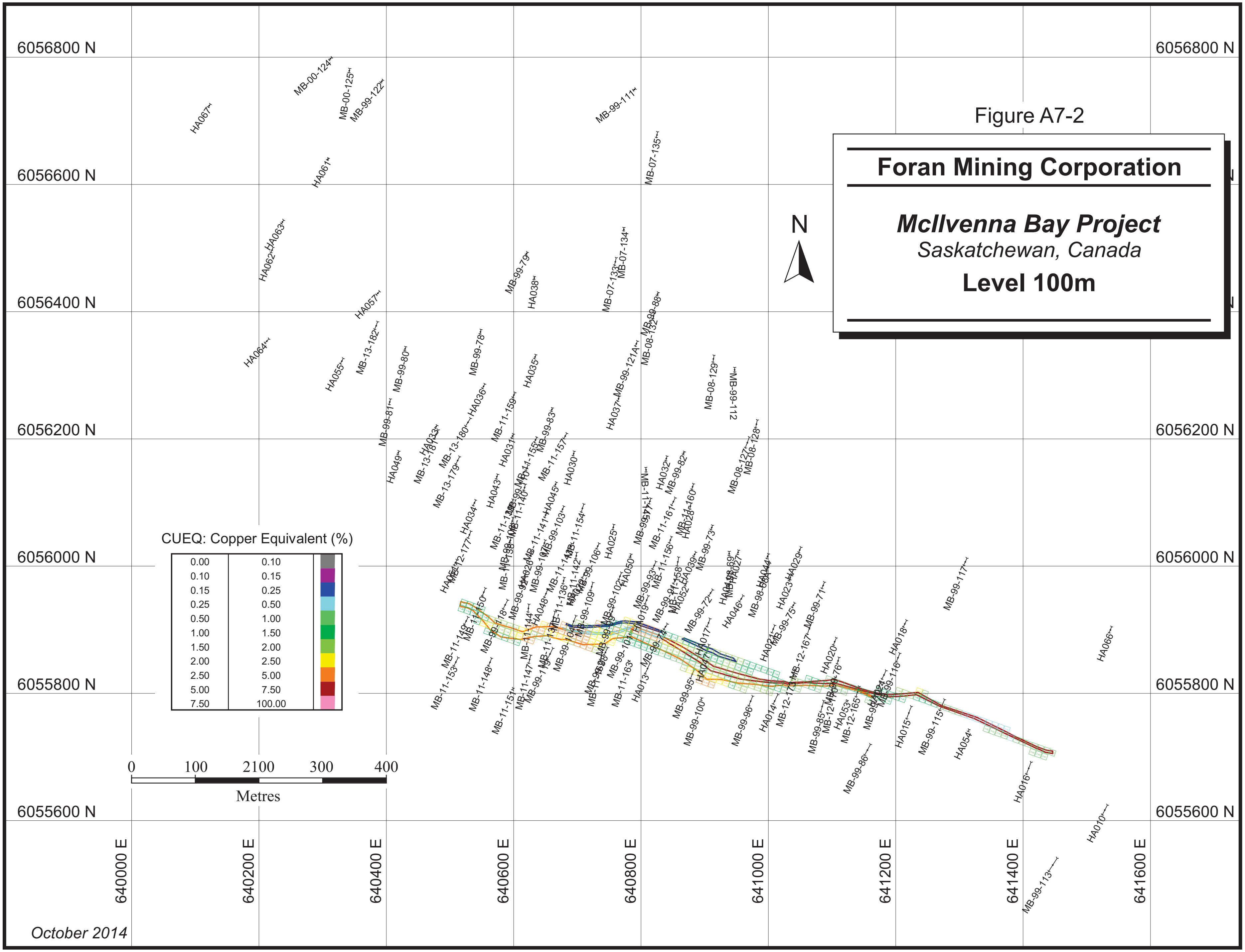


Figure A7-3
Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Level -150m
N
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 100 2100 300 400
Metres
6056800 N
6056600 N
6056400 N
6056200 N
6056000 N
6055800 N
6055600 N
640000 E
640200 E
640400 E
640600 E
640800 E
641000 E
641200 E
641400 E
641600 E
October 2014



Foran Mining Corporation
McIlvenna Bay Project
Saskatchewan, Canada
Level 100m
N
CUEQ: Copper Equivalent (%)
0.00 0.10
0.10 0.15
0.15 0.25
0.25 0.50
0.50 1.00
1.00 1.50
1.50 2.00
2.00 2.50
2.50 5.00
5.00 7.50
7.50 100.00
0 100 2100 300 400
Metres
6056800 N
6056600 N
6056400 N
6056200 N
6056000 N
6055800 N
6055600 N
640000 E
640200 E
640400 E
640600 E
640800 E
641000 E
641200 E
641400 E
641600 E
October 2014

Figure A7-2