NI 43-101 Technical Report,

Lalor And Snow Lake Operations, Manitoba, Canada

March 28, 2019, effective as of January 1, 2019

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Prepared under the supervision of:

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CAUTIONARY NOTE REGARDING FORWARD-LOOKING INFORMATION

This Technical Report contains "forward-looking statements" and "forward-looking information" (collectively, "forward-looking information") within the meaning of applicable Canadian and United States securities legislation. All information contained in this Technical Report, other than statements of current and historical fact, is forward-looking information. Often, but not always, forward-looking information can be identified by the use of words such as “plans”, “expects”, “budget”, “guidance”, “scheduled”, “estimates”, “forecasts”, “strategy”, “target”, “intends”, “objective”, “goal”, “understands”, “anticipates” and “believes” (and variations of these or similar words) and statements that certain actions, events or results “may”, “could”, “would”, “should”, “might” “occur” or “be achieved” or “will be taken” (and variations of these or similar expressions). All of the forward-looking information in this Technical Report is qualified by this cautionary note.

Forward-looking information includes, but is not limited to, our objectives, strategies, intentions and expectations, production, cost, capital and exploration expenditure guidance, including the estimated economics of the Snow Lake operations mine, future financial and operating performance and prospects, anticipated production at our mine and processing facilities and events that may affect Hudbay’s operations, anticipated improvements to metallurgical recoveries, anticipated cash flows from operations and related liquidity requirements, the anticipated effect of external factors on revenue, such as commodity prices, estimation of mineral reserves and resources, mine life projections, reclamation costs, economic outlook, government regulation of mining operations, and expectations regarding community relations. Forward-looking information is not, and cannot be, a guarantee of future results or events. Forward-looking information is based on, among other things, opinions, assumptions, estimates and analyses that, while considered reasonable by us at the date the forward-looking information is provided, inherently are subject to significant risks, uncertainties, contingencies and other factors that may cause actual results and events to be materially different from those expressed or implied by the forward-looking information.

The material factors or assumptions that we identified and were applied by us in drawing conclusions or making forecasts or projections set out in the forward-looking information include, but are not limited to:

- The schedule for the refurbishment of the New Britannia mill and the success of our Lalor strategy;
- the success of mining, processing, exploration and development activities;
- the scheduled maintenance and availability of our processing facilities;
- the accuracy of geological, mining and metallurgical estimates;
- anticipated metals prices and the costs of production;
- the supply and demand for metals we produce;
- the supply and availability of concentrate for our processing facilities;
- the supply and availability of third party processing facilities for our concentrate;
- the supply and availability of all forms of energy and fuels at reasonable prices;
- the availability of transportation services at reasonable prices;
- no significant unanticipated operational or technical difficulties;
- the execution of our business and growth strategies, including the success of our strategic investments and initiatives;
- the availability of additional financing, if needed;
- the availability of personnel for our exploration, development and operational projects and ongoing employee relations;
- maintaining good relations with the communities surrounding Lalor mine;
- no significant unanticipated challenges with stakeholders at our various projects;

- no significant unanticipated events or changes relating to regulatory, environmental, health and safety matters;
- no contests over title to our properties, including as a result of rights or claimed rights of aboriginal peoples;
- no significant unanticipated litigation; and
- no significant and continuing adverse changes in general economic conditions or conditions in the financial markets (including commodity prices and foreign exchange rates).

The risks, uncertainties, contingencies and other factors that may cause actual results to differ materially from those expressed or implied by the forward-looking information may include, but are not limited to, risks generally associated with the mining industry, such as economic factors (including future commodity prices, currency fluctuations, energy prices and general cost escalation), risks related to the new Lalor mine plan including the schedule for the refurbishment of the New Britannia mill and the ability to convert inferred mineral resource estimates to higher confidence categories, uncertainties related to the development and operation of our projects, dependence on key personnel and employee and union relations, risks related to political or social unrest or change, risks in respect of aboriginal and community relations, rights and title claims, operational risks and hazards, including unanticipated environmental, industrial and geological events and developments and the inability to insure against all risks, failure of plant, equipment, processes, transportation and other infrastructure to operate as anticipated, depletion of Hudbay's reserves, volatile financial markets that may affect our ability to obtain additional financing on acceptable terms, the failure to obtain required approvals or clearances from government authorities on a timely basis, uncertainties related to the geology, continuity, grade and estimates of mineral reserves and resources, and the potential for variations in grade and recovery rates, uncertain costs of reclamation activities, Hudbay's ability to comply with its pension and other post-retirement obligations, our ability to abide by the covenants in our debt instruments and other material contracts, tax refunds, hedging transactions, as well as the risks discussed under the heading "Risk Factors" in our most recent Annual Information Form.

Should one or more risk, uncertainty, contingency or other factor materialize, or should any factor or assumption prove incorrect, actual results could vary materially from those expressed or implied in the forward-looking information. Accordingly, you should not place undue reliance on forward-looking information. We do not assume any obligation to update or revise any forward-looking information after the date of this Technical Report or to explain any material difference between subsequent actual events and any forward-looking information, except as required by applicable law.

Hudbay uses certain non-IFRS financial performance measures in its financial reports and in this 43-101 Technical Report, including net debt, cash cost, sustaining and all-in sustaining cash cost per pound of copper produced, and cash cost and sustaining cash cost per pound of zinc produced. These measures do not have a meaning prescribed by IFRS and are therefore unlikely to be comparable to similar measures presented by other issuers. These measures should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS and are not necessarily indicative of operating profit or cash flow from operations as determined under IFRS. Other companies may calculate these measures differently. For a description and reconciliation of each of these measures, please see the Non-IFRS Financial Performance Measures section on pages 45 to 56 of our management's discussion and analysis for the year ended December 31, 2018, a copy of which has been filed on SEDAR at www.sedar.com and EDGAR at www.sec.gov.

This 43-101 Technical Report contains references to both United States dollars and Canadian dollars. All dollar amounts referenced, unless otherwise indicated, are expressed in constant Canadian dollars.

SIGNATURE PAGE

This Technical Report titled "NI 43-101 Technical Report, Lalor and Snow Lake Operations, Manitoba, Canada", dated March 28th, 2019 and effective as of January 1st, 2019, was prepared under the supervision and signed by the following author:

Dated this 28th day of March 2019.

/s/ Olivier Tavchandjian
Signature of Qualified Person

Olivier Tavchandjian, P. Geo.
Vice President, Exploration and Geology
Hudbay Minerals Inc.

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1. SUMMARY

Hudbay is a Canadian integrated mining company with assets in North and South America. Hudbay operates multiple properties in the Province of Manitoba. Current operations near Snow Lake include the Lalor underground mine as well as an ore concentrator, a tailings impoundment area and other ancillary facilities that support the operation.

This Technical Report has been prepared to support the public disclosure of Mineral Resources and Mineral Reserves at the Lalor Mine and nearby Snow Lake satellite deposits and to provide an updated mine plan that contemplates 4,500 tpd of base metal, gold and copper-gold ore feed to the Stall, Flin Flon and New Britannia concentrators. Refurbishing the New Britannia mill is expected to significantly increase gold production from Lalor and enable new gold and copper-gold exploration opportunities in the Snow Lake region by having an operating processing facility with substantially higher gold and copper recoveries.

The Snow Lake satellite deposits that host mineral resources include the Wim, Pen II and New Britannia properties. These satellite deposits are not currently part of the mine plan and may provide an opportunity to increase Hudbay's mineral reserves in the area and extend the operating lives of the Stall and New Britannia processing facilities (see the Recommendations section of this Technical Report for further details).

1.1 QUALIFIED PERSON (QP)

This Technical Report has been prepared in accordance with National Instrument Form 43-101F1. The qualified person ("QP") who supervised the preparation of this Technical Report is Olivier Tavchandjian, P. Geo., Hudbay's Vice-President, Exploration and Geology.

1.2 PROPERTY LOCATION, LOCAL RESOURCES AND INFRASTRUCTURES

All the properties described in this technical report for which mineral resource and mineral reserve estimates are reported and their associated infrastructures are located within 20 km of the town of Snow Lake, approximately 200 km by road east of the city of Flin Flon where the nearest full service commercial airport is located. Hudbay owns a 100% interest in all the properties.

All these properties are accessible by road from the town of Snow Lake. The Snow Lake area has a typical mid-continental climate, with short summers and long, cold winters. Climate generally has only a minor effect on local exploration and mining activities. The area is surrounded by water bodies. Topography shows gentle relief that rarely exceeds 10 m.

The town of Snow Lake is a full-service community with available housing, hospital, police, fire department, potable water system, restaurants and stores. To house non-local employees during their work rotations, the company provides a camp located in town which services Hudbay employees and contractors for the mine and mill operations. Other infrastructure in the area includes provincial roads, a 115 kV Manitoba Hydro power grid within four kilometres of Lalor and Manitoba Telecom land line and cellular phone service.

Hudbay operates two concentrators in the Snow Lake / Flin Flon area: the Stall mill located approximately 16 km from Lalor and the Flin Flon mill located 200 km to the west. During the 2019-2021 period, Hudbay plans to truck approximately 1,000 tpd of ore from Lalor to the Flin Flon mill where the ore from Lalor will be processed separately from the ore produced from the 777 mine. The tailings from the Flin Flon mill are deposited at the Flin Flon tailings facility or used at the paste plant.

Hudbay also operates a zinc metallurgical plant in Flin Flon with a capacity of 115,000 tpa of refined zinc. However, Hudbay intends to cease operations and put the zinc plant and the Flin Flon mill on care and maintenance at the end of 2021. After the Zinc plant closure, the concentrates produced from the Stall and Flin Flon mill will be sold to market.

The New Britannia gold mill is located approximately 16 km east of the Lalor mine and it is currently under care and maintenance. Hudbay plans to complete the refurbishment of the New Britannia mill by the end of 2021, adding a new copper flotation circuit. The throughput capacity of the plant will be 1,500 tpd.

1.3 HISTORY

The Snow Lake area has a long exploration and mining history. Exploration in the Lalor-Chisel area has been occurring since the 1950s and the Chisel Basin area has hosted four past producing mines. This basin is also the host of the Lalor deposit. Lalor commenced initial ore production from the ventilation shaft in August 2012, only five years after its initial discovery hole and achieved commercial production from the main shaft in the third quarter of 2014. Since 2012, the mine has produced 5.6 million tonnes.

Gold was first discovered in 1914 approximately 20 km to the southeast of Snow Lake and in 1917, the Moose Horn-Ballast claims produced the first gold in Manitoba. First mine construction at the New Britannia site started in 1945 and in March 1949, the mine was opened as the Nor-Acme mine. Production continued until 1958. 4.9 Mt were mined at an average grade of 4.4 g/t and Nor-Acme mill recovered approximately 610,000 Oz of gold during this production period. TVX and High River formed a joint venture to reopen the mine and TVX became the operator. Full production from the main shaft was achieved in August 1996. Through various transactions, Kinross became the operator of the New Britannia mine-mill complex. Production ceased at the end of September 2004 and the mill was put on care and maintenance in 2005 due to a low gold price environment after producing 1.6 million ounces of gold.

1.4 GEOLOGY AND MINERALIZATION

The properties of interest for this Technical Report are all located within the Trans-Hudson Orogen of the Flin Flon Greenstone Belt. The volcanic assemblages consist of mafic to felsic volcanic rocks with intercalated volcanogenic sedimentary rocks. The Snow Lake arc assemblage that hosts the producing and past-producing mines in the Snow Lake area presents a temporal evolution from ‘primitive arc’ (Anderson sequence) to ‘mature arc’ (Chisel sequence).

The volcanogenic massive sulphide (VMS) deposits located near the town of Snow Lake have been subdivided into two different groups: Cu-Zn-rich (Cu-Zn, Cu-Zn-Au) and Zn-Cu-rich (Zn-Pb-Cu-Ag) types. The Cu-Zn-rich deposits mainly occur in the Anderson sequence and the Zn-Cu-rich deposits occur in the Chisel sequence. The location of VMS deposits is often controlled by synvolcanic faults and fissures, which permit a focused discharge of hydrothermal fluids. A typical VMS deposit will include the massive mineralization located proximal to the active hydrothermal vent, footwall stockwork mineralization, and distal products, which are typically thin but extensive. Massive sulphide deposits consist of usually more than 40% sulfides, dominated by pyrite, pyrrhotite, chalcopyrite, sphalerite, and galena. These deposits have stringer or feeder zones beneath the massive zone that consist of crosscutting veins and veinlets of sulfides in a matrix of pervasively altered host rock.

Mineralization of the lode-gold vein-type deposits are hosted in the Amisk group mafic and felsics volcanic rocks which are structurally controlled and associated with shear zones, faults, fold hinges and axial planes that host simple to complex vein systems. The mineralization is associated with lithological contacts of contrasting properties in the sequence of interlayered volcanic and volcaniclastic rocks. Alteration in the mineralized zones consists of quartz-carbonate-mica, with arsenopyrite as the primary sulphide accompanying the gold.

1.5 EXPLORATION

Since the last NI 43-101 Technical Report published in 2017, exploration drilling at Lalor has focused on adding and converting inferred mineral resource estimates with a strong emphasis on confirming the continuity of the gold mineralization. This effort will continue in 2019. In parallel, surface drilling has identified a new Cu-Au feeder type zone named Lens 17 which is deemed to be an analog to the now well-defined Lens 27 and to constitute the feeder zone of Lens 10. Exploration drilling of Lens 17 will continue in 2019 from underground platforms to establish an inferred mineral resource estimate by the end of 2019. Hudbay is also undertaking an exploration program to find additional massive sulfide lenses at depths below current drilling coverage, i.e. from 1,000 m to 1,500 m from surface. Exploration drilling will also follow-up on some initial successes obtained between Lalor and the former producing Chisel mine.

In 2018, Hudbay re-initiated a systematic exploration program for lode gold deposits in the Snow Lake area and completed an Induced Polarization (IP) survey and an unmanned aerial vehicle (UAV) magnetic survey over the northern part of the Snow Lake gold property encompassing the Birch, No. 3 Zone and the Boundary deposit areas. Further south, two additional properties: The Tern Lake and Purple Sandy Beach prospects were also covered by the survey. The IP results were deemed successful and have been used to define drill targets that will be tested in 2019.

1.6 DRILLING, SAMPLING, ASSAYING, QAQC AND VALIDATION

At Lalor, over 3,000 drill holes totaling more than 500,000 m were included in the Lalor database to support the mineral resource estimate while drilling supporting the Pen II and Wim mineral resource estimates totals respectively 2,000 m and 43,000 m. For the New Britannia resource estimates, over 500,000 m of drilling completed after 1995 were used. Drilling at all properties is a combination of NQ and BQ diamond drill holes, surveyed with either Reflex downhole tools or Gyro for deeper/longer holes.

At Lalor, Pen II and Wim, the core was photographed before samples were split and bagged for shipment. Completed sample bags were closed and secured at the neck using two zip ties. All saws and sampling buckets are rinsed with water after cutting each sample to prevent cross-contamination. All data from sample books including QAQC samples were entered into Hudbay's acQuire drill hole database, or in a Gemcom database in the case of Wim. Before dispatch to the laboratories, QAQC samples were inserted into the sample stream. Hudbay's practice involves insertion of the following every 100 samples: 2 blanks, 5 duplicates, 5 base metal standards and 2 gold standards.

Sample preparation has been conducted at three different laboratories over time. Since September 2016, nearly all samples are prepared and assayed at Bureau Veritas in Vancouver. All drill core samples analyzed since the last technical report (NI 43-101 published on March 30, 2017) have been sent for analysis at Bureau Veritas while the SGS laboratory in Vancouver was used as the umpire laboratory for quality control purposes.

The most common method of analysis for base metals and other elements was aqua regia digestions coupled with ICP-ES/MS (method AQ270 and AQ370). Assays returning "over the range" values were reassayed with higher reporting methods (GC820 for copper, GC816 for zinc, GC817 for lead and MA404 for copper and zinc when values were above 20%). All samples were analyzed for gold using a fire assay method coupled with atomic absorption (FA430). A gravimetric finish (FA530) was used when samples return gold values above 10 PPM.

Density measurements were initially collected using a non-wax-sealed immersion technique to measure the weight of each sample in air and in water. Since February 2018, the density measurements are performed with a gas pycnometer (method SPG04) at Bureau Veritas in Vancouver. Independent checks have been conducted to confirm that there is no material difference between the two methods.

Results from the QA/QC program for standards, blanks, duplicates and external checks show that the program has been working effectively for the Lalor, Pen II and Wim properties, meeting industry standards and the data used provides a representative and unbiased basis for resource modeling purposes.

The sampling methodology, analyses and security measures used by the previous owners at New Britannia have been documented in the Technical Report produced by Genivar for Alexis Resources in 2011 and available on SEDAR. Most of the drill cores and chips assays from 1995 to 2003 from the New Britannia mine were completed at the on-site mill laboratory using a fire assay/atomic absorption finish (FA/AA) method. Standard, blank and duplicate assay samples were added to each batch of 21 samples for drill core and to each batch of 24 samples for chip samples. The sampling and analytical procedures conformed to the industry standards at the time, and these were adequate to ensure a representative determination for the type of gold mineralization identified on the property.

It is the QP's opinion that the data is adequate and acceptable for use in the estimation of mineral resources at all the properties covered by this Technical Report.

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

The Stall concentrator is an operating plant running at steady state and, as a result, several of the initial metallurgical test results and assumptions have been revised to reflect the operating experience and performance of the plant over the past six years of operation in processing the ore produced from the Lalor mine. The Stall concentrator is producing a copper concentrate grade of 21% copper at 83 to 85% recovery and a zinc concentrate grade of 51% zinc at 90 to 95% recovery. Gold and silver are recovered in the copper concentrate as co-products. Since the metallurgical blend from Lalor is not expected to materially change over the life of mine, it is appropriate to assume that historical performance at Stall is expected to continue in the future. Should the Pen II Zone be developed, it is expected that ore from Pen II will be milled at the Stall Concentrator and blended with ore from the Lalor Mine. For the current mineral resource estimates at Wim, it has been assumed that production will be processed at the Stall mill. This is a conservative assumption as the proposed flowsheet for the refurbished New Britannia mill would better suit the Wim mineralization and likely allow a significant increase in gold recovery from the current assumptions.

In 2017 and 2018, the Flin Flon Concentrator ran a total of ten plant trials for the Lalor ore. The plant trial was generally quite smooth and successful from ore crushing to feed transition, grinding, flotation and filtration. Stable gold, copper and zinc recoveries were obtained after the optimization of operational parameters. The copper concentrate produced from the Lalor ore has a grade of approximately 18% Cu at 85 to 90% recovery and a zinc concentrate grade of approximately 53% Zn at 84 to 92% recovery.

To establish the future performance of the New Britannia mill, a laboratory program was conducted at SGS in 2015-2016. The objective of the program was to develop a flowsheet, design criteria, and metallurgical forecast information for these ore types. 20 composite samples from Lens 25 and 9 composite samples from Lens 27 were used to produce a "Blend" composite. The blend ratio was 67% from Zone 25 and 33% from Zone 27. All composites were submitted for mineralogical analysis by QEMSCAN to identify minerals and their liberation as well as for Bond rod and Bond ball mill work tests, gravity concentration test, rougher and cleaner flotation tests, cyanidation tests, and rolling bottle leach tests. In addition, CIP modelling was completed to predict the gold extraction performance of a CIP or CIL circuit. Cyanide destruction tests using the SO2/Air process were also carried out following standard SGS procedure of completing batch tests first to confirm applicability and to optimize retention times and reagent requirements. The results showed that the SO2/Air process is appropriate for use at New Britannia. Finally, A total of six locked cycle tests were completed to confirm the flowsheet and to generate tailings for the cyanidation testwork. Results of the locked cycle tests which covered a wide range of copper head grades from 0.22% Cu to 3.69% Cu form the basis for the recovery models. Detailed chemical analyses were completed on eight of the copper concentrates produced in the test program. Based on these analyses, no penalties are expected with the possible exception of lead, which at times may slightly exceed the penalty limit of 3% lead+zinc.

It is anticipated that any mineral resource estimates from the New Britannia property would be processed at the New Britannia mill, by-passing the copper flotation circuit and achieving similar gold recoveries as per historical performance of the mine.

1.8 MINERAL RESOURCE ESTIMATES

For the Lalor and Pen II properties, the construction of the 3D resource model and the estimation of mineral resources were performed by Hudbay personnel following Hudbay procedures in compliance with best industry standards and the CIM guidelines, using Leapfrog® version 4.2.3 and MineSight® version 15, two industry standard commercial geological and mining software.

The construction of the mineralized envelopes was solely based on the type of mineralization intersected. To this effect, assay intervals with high Au grade but no zinc or copper were excluded from base metal lenses and vice versa. Sample coding was checked to ensure proper tagging of the solids to actual drill hole locations. The mineral envelopes were used as hard boundaries in all cases for grade interpolation purposes.

Density values were generated from robust multi-regression formulas where no actual measurement was taken. Assay intervals were regularized by compositing drill hole data within each interpreted mineralized envelope. Although most of the drill holes were assayed in 1 m intervals, a composite length of 2.5 m was selected as more appropriate to conduct interpolation into the 5mx5mx5m block size selected by the mine engineering team as the optimum selective mining unit (SMU).

Exploratory data analysis (EDA) includes basic statistical evaluation of the composites for zinc (Zn), gold (Au), copper (Cu), silver (Ag) and density (SG). The EDA was conducted separately for each mineralized envelope and aimed to identify sub-domains that did not support the use of hard contacts for grade interpolation purposes but that justified being used for grade capping, block model validation, smoothing assessment and correction and resource classification.

The deciles[1] analysis (Parish) method was used to define high-grade outliers and to confirm the need for grade capping. This analysis was conducted on the 2.5 m composites for each sub-domain independently. Down-hole and directional variograms for Zn, Au, Cu, Ag, and SG were created for each individual mineral envelope. A linear combination of a nugget and two nested spherical models were adjusted in all cases. Once generated, a systematic visual check was conducted to ensure that the search ellipsoid would be correctly oriented with respect to the geometry of the mineral envelopes.

Where a block was intersected by two lenses, the mineral envelopes were used to assign the percentage of the block that belongs to each lens. Both nearest neighbour (NN) and ordinary kriging (OK) grade interpolations were completed, using a strict composite and block matching code by lens and three passes with increasing requirements.

The grade estimation process was validated for each envelope to ensure appropriate honouring of the input data and subsequent unbiased resource reporting and use of the model for reserve estimation. The validation steps included systematic visual investigations, checks for global bias and over-smoothing. Where the model was found to be over-smoothed, an indirect log-normal correction was applied to the kriged estimates.

The regression slope values obtained from the kriging of zinc (in the case of base metals lenses) or gold (in the case of gold zones) grade estimates was used as the primary criteria for resource classification. The block by block coding assignation was then smoothed to remove isolated blocks of one category within another. In assigning the final categorization, careful consideration was also given to the proportion of mineralization grading above the anticipated cut-off grade. In areas where the blocks grading above the anticipated cut-off grade would represent a small portion of the total volume of the Lens, the classification was downgraded.

For the present estimates, Hudbay has implemented a stringent approach to establish the potential for economic extraction of its resource reporting for underground deposits. With this approach, the potential for economic extraction of the mineral resource estimates at Lalor and Pen II are reported within the constraint of a 'stope optimization envelope'. This excludes small isolated individual blocks above the economic cut-off criteria from the resource estimate and includes some 'geological dilution' that would need to be included in the economic envelope to maintain minimum spatial continuity requirements to define mineable shapes.

The parameters used as input to define the stope optimization envelope cover all the relevant technical and economic constraints including minimum stope and waste pillar dimensions and a NSR value calculation for each block based on anticipated metal recoveries, long-term metal price forecast and

[1] I.S. Parrish (1997) Mining Engineering Journal, Geologist's Gordon Knot: to cut or not to cut (pp. 45-49)

operating and capital costs based on the 2019 Lalor mine and Stall concentrator budgets. At Lalor, two NSR values were calculated for each block to assess and compare the value of the blocks going to the Stall or Flin Flon mill (no material difference between the two) or going to the new Britannia mill. The mineral resource estimates are reported to ensure that each potential stope would cover all its associated operating mining and milling costs.

The resulting mineral resource estimates are presented in Table 1-1 and Table 1-2. There are no measured or indicated mineral resource estimates in addition (exclusive) to those that were converted to mineral reserve estimates. All measured and indicated mineral resource estimates were either converted to mineral reserve estimates or deemed non-economic after applying the resource to reserve appropriate conversion factors.

TABLE 1-1: LALOR MINERAL RESOURCE ESTIMATES
(Exclusive of Mineral Reserve Estimates) (1), (2), (3), (4)

Mineral Resource Inferred Category	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Base Metal Lens	1,385,000	4.5	43.6	0.70	2.30
Gold Lens	4,516,000	4.4	20.4	1.08	0.35
Total Inferred	**5,901,000**	**4.4**	**25.9**	**0.99**	**0.81**

Notes:
Totals may not add up correctly due to rounding.
1. Mineral resources are estimated as of January 1, 2019.
2. Mineral resources are estimated at a minimum NSR cut-off of $96.19 per tonne.
3. Mineral resources do not include mining dilution or recovery factors.
4. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

TABLE 1-2: PEN II MINERAL RESOURCE ESTIMATES
(1), (2), (3), (4)

Categories	Tonnes	Zn Grade (%)	Au Grade (g/t)	Cu Grade (%)	Ag Grade (g/t)
Indicated	500,000	8.89	0.35	0.49	6.81
Inferred	100,000	9.81	0.30	0.37	6.85

Notes:
1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.
4. Pen II mineral resources are estimated at a minimum NSR cut-off of $65 per tonne and assume that the Pen II mineral resources would be amenable to processing at the Stall mill.

In May 2015, Golder Associates (“Golder”) prepared a resource model for the Wim property using Datamine® version 2.1.1547.0, an industry standard commercial geological and mining software. The procedures used to create and validate the resource block model are compliant with best industry standards and the CIM guidelines. More details can be found in the “Technical Report on the Wim Copper-Gold Deposit, Snow Lake Manitoba Canada” prepared for Alexandria Minerals Corporation (Alexandria) dated May 2015 (Golder 2015) available on SEDAR.

A smooth and continuous 3D mineralized envelope was built using a copper equivalency cut-off grade to guide the construction of the base metal mineralized envelope while including material below cut-off where necessary to maintain the continuity of the envelope. Scatter plots were used to identify assay outliers for Au, Ag, Cu and Zn values for grade capping. Missing density values were generated using a linear regression formula from either the Fe grade or from the Cu grade. A composite length of 0.9 m, representing 77% of the population was selected to conduct interpolation into the blocks. Compositing was weighted by density.

A block size of 15 m east-west (strike), 3 m north-south (thickness), and 5 m elevation was selected. Grade for Cu, Zn, Au, Ag and density (SG) was estimated using an inverse distance interpolant (IDW^2) in the unfolded space and weighting by density. A nearest neighbor (NN) interpolation was also performed to provide declustered sample grade for block model validation. The search ellipsoids were defined from pairwise variograms modeled by a linear combination of a nugget and two nested spherical structures.

Resource classification was assigned based on the search parameters used for grade interpolation. The block by block coding assignation was then smoothed to remove isolated blocks of indicated within areas of mostly inferred category and vice versa.

To properly account for the polymetallic nature in the assessment of the economic potential of the Wim deposit, a Copper Equivalent (CuEq%) value was calculated. An open pit optimization was conducted on the resource block model using Whittle 4.5.2. A 20-metre zone of material below the bottom of the Whittle open pit shell was classified as crown pillar and excluded from the Mineral Resource. In a final step, the Datamine's Mineable Shape Optimizer (MSO) module was then applied to the Wim resource block model below the 20m crown pillar to identify contiguous volumes of resource above the underground using a minimum target stope head grade of 1. 3% CuEq. The resulting mineral resource estimates, as of January 1, 2019, are presented in Table 1-3 and, at Hudbay's request, have been revalidated by Golder. There are no mineral reserve estimates reported for the Wim deposit as the required engineering and metallurgical test work has not been completed yet

TABLE 1-3: WIM MINERAL RESOURCE ESTIMATES

(1), (2), (3), (4)

		CuEq% Cut-off	Tonnes	Cu%	Zn%	Au g/t	Ag g/t
Indicated	Open Pit	0.6	276,000	1.08	0.10	1.25	6.81
	Undergroung	1.3	3,623,000	1.76	0.28	1.59	6.67
	Total Indicated		**3,899,000**	**1.71**	**0.27**	**1.57**	**6.68**

		CuEq% Cut-off	Tonnes	Cu%	Zn%	Au g/t	Ag g/t
Inferred	Open Pit	0.6	63,000	0.95	0.09	1.05	6.4
	Undergroung East	1.3	604,000	1.12	0.44	1.69	4.7
	Undergroung West	1.3	64,000	0.26	0.01	3.03	2.5
	Total Inferred		**731,000**	**1.03**	**0.37**	**1.75**	**4.65**

1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.
4. Mineral resources reported based on a 1.3 CuEq% cut-off for the underground portion, and a 0.6% cut-off for the open pit portion, assuming processing recoveries of 90% for copper and zinc and 70% for gold and silver, and using long-term prices of $US3.00 per pound copper, $US1,200 per ounce gold, $US1.00 per pound zinc and $US15.00 per ounce silver. A 20 m crown pillar below the open pit bottom is excluded from resources.

New Britannia Mine

For the former New Britannia, mine and its satellite gold deposits, the historical resource estimate performed by Kinross and by Alexis Minerals followed a conventional and industry standard approach and have been independently validated in 2018 by WSP Engineering ("WSP"). A block model of 15 ft. x 15 ft. x 15 ft. was created for the main mine upper and lower zones while a 10 ft. x 10 ft. x 10 ft. block model was created for the No.3 Zone. The cut-off grades for the resource have been estimated over a 6-ft. minimum true width with a variable cut-off by zone as summarized in Table 1-4. The variation in the cut-off grade is related to new mining versus remnant mining. The New Britannia mill will have been refurbished when the mineral resource estimates would potentially be mined.

Given that WSP had to rely on historical documentation for some of the technical information supporting the estimation of the mineral resource estimates, the tonnes and grades previously estimated by Kinross and Alexis Minerals as measured and indicated resources were downgraded to an inferred category. Mineral resources that are not mineral reserves do not have demonstrated economic viability

TABLE 1-4: UPDATED GOLD RESOURCE ESTIMATE FOR THE NEW BRITANNIA MINE AND IT'S SATELLITE DEPOSITS
(1), (2), (3),(4)

Zone	Location	Cut-off (Au g/t)	Tonnes	Grade
Lower Dick NW	1780 -	3.3	7,000	4.76
Main Mine 1780 +	1780 +	3.5	1,142,000	4.55
Lower Dick NW	780 -	3.3	213,000	4.55
Lower Dick EW	1780 -	2	823,000	4.59
Lower Ruttan	1780 -	2	568,000	4.28
Main	3 Zone	2	744,000	6.37
Footwall	3 Zone	2	364,000	4.66
Birch	Birch	3.3	569,000	4.42
Total Inferred			4,430,000	4.82

1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.
4. Effective date of resource estimate is January 1, 2019

1.9 MINERAL RESERVE ESTIMATES

No mineral reserve estimates are reported for the Pen II, Wim and New Britannia Mine deposits.

At Lalor, the following steps were followed in developing the reserve estimates

- Calculate two payable (NSR) values for each individual block in the resource model depending on whether processing would occur at the Stall concentrator or at the New Britannia concentrator, using long-term metal prices, concentrator recoveries, metal payability and downstream smelter treatment and refining costs assumptions.
- Design stopes in the Deswik Stope Optimizer, considering depleted mineral resources, existing workings, resource categories and mine and mill operations costs. Dilution and recovery are estimated and applied at this step. Stopes are designed for both the Stall concentrator option and the New Britannia concentrator option.
- Considering grades, value and location in the mine, assign stopes to either Stall or New Britannia concentrator.
- Establish stope economics using a secondary NSR calculation where, along with mine and mill operations costs, mine capital, waste development and offsite administration costs are applied to each stope.
- Assign whether stopes can be upgraded to mineral reserves based on resource classification.
- Design ore development required for mining the reserves. Deplete development from the stopes. Interrogate grades of designed development for inclusion in mineral reserves. Sequence and schedule development and stope production for input to a financial Life of Mine (LOM) study to support mineral reserve economics.

The above methodology takes into consideration the different ore types and the milling options for the future production from Lalor mine and considers the three distinct ore types found at the mine.

The mineral reserve estimates exclude the mined out mineral resources, non-recoverable pillars (rib, post and sill) within mined out areas, mineral resources that are sterilized or not recoverable due to previous mining and stopes based on inferred mineral resource estimates.

A reconciliation between tonnes and grade predicted from the resource model and actual mine production credited by the mills between 2012 and 2018 shows a close comparison on all parameters. The reported tonnes and grade from the resource model are globally within 5% of actual production and on the conservative side, i.e. marginally under-estimate the quantity of metal recovered by the mill.

The author considers that the mineral reserves as classified and reported comply with all disclosure in accordance with requirements and CIM definitions. The author is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

The Lalor mine mineral reserves as of January 1, 2019 are summarized in Table 1-5.

TABLE 1-5: SUMMARY OF MINERAL RESERVES
(as of January 1, 2019) (1), (2), (3), (4)

	Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Base Metal	Proven	5,137,000	7.13	2.37	0.76	26.31
	Probable	5,552,000	4.19	3.52	0.44	27.39
Gold	Proven	58,000	2.65	5.46	0.80	39.09
	Probable	2,928,000	0.31	6.74	1.09	23.08
Proven + Probable		**13,675,000**	**4.46**	**3.78**	**0.70**	**26.11**

Notes:
1. Totals may not add up correctly due to rounding.
2. Mineral reserves are estimated as of January 1, 2019.
3. Mineral reserves are estimated at an NSR cut-off of $96.19/t for waste filled mining areas and a minimum of $104.58 per tonne for paste filled mining areas.
4. Metal prices of $US 1.17/lb zinc (includes premium), $US 1,260/oz gold, $US 3.10/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to estimate mineral reserves.

1.10 MINING METHODS AND PRODUCTION SCHEDULE

Geology, geotechnical information, orebody geometry, productivity and mining experience are the key drivers to select the best suited mining methods at Lalor including considerations for lateral development, production, backfilling and ore transportation used at Lalor mine.

The hanging wall and footwall rocks are in general of good quality allowing the use of mechanized drilling and blasting techniques. The mineralized lenses dip at an average of 30°, but locally varying from 10 to 55°. Mining methods currently in use include: cut and fill and longhole open stope. Paste backfill is used to increase recovery and accelerate the mining cycle. Low grade areas will be filled with rock from waste development.

Ore is mucked using Load Haul and Dump (LHD) loaders which are operated remotely in inaccessible areas. Ore is loaded into underground haul trucks at ore passes and transported to the ore handling system at the production shaft for hoisting to surface. Ore delivered to the production shaft is sized to less than 0.55 m by one of two rock breakers. Ore is hoisted from the mine by two 16 tonne capacity bottom dump skips in balance. On surface, ore is truck hauled to a primary crusher at the Chisel North mine site, crushed to less than 0.15 m, and will then be trucked to either the Stall, Flin Flon or New Britannia concentrator for processing.

Mine development includes sub-horizontal tunneling and a ramp system to provide access between levels.

Main levels are developed parallel to and in the footwall of the ore zones. Where possible, main levels are located to provide access to multiple ore zones and are connected to haul ramps to allow mechanized equipment to travel from level to level. Main ventilation and ore pass raises are developed using a raisebore and/or Alimak climber. Drain, paste backfill and electrical cable holes are drilled using longhole or raisebore drills and are reamed to designed diameter.

Stope mining currently in use at Lalor include: cut and fill and longhole open stope. Paste backfill is used to increase recovery and accelerate the mining cycle. Production is expanding to a steady state of 4,500 tpd in the first quarter of 2019. Lalor mine production will be approximately 20% jumbo and 80% longhole.

The Life of Mine (LOM) production schedule is shown in Table 1-6. The Deswik software was used to assist with the LOM sequencing and scheduling to generate the production schedule. Historic mining rates and systems, equipment or crew's capabilities were applied and levelled. From this output, adjustments were made to further balance the capabilities to create the final plan. Concentrates produced from Lalor and contained metal in concentrates are also shown in Table 1-6.

TABLE 1-6: LOM PRODUCTION SCHEDULE

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2019	1,588,810	2.41	22.96	0.63	5.43
2020	1,481,216	2.12	28.57	0.62	6.37
2021	1,559,418	2.75	28.23	0.64	6.18
2022	1,607,100	3.71	25.75	0.68	4.28
2023	1,631,103	3.68	24.00	0.72	4.65
2024	1,643,607	3.56	24.10	0.99	4.23
2025	1,440,842	5.28	24.89	0.81	2.54
2026	941,495	5.41	23.96	0.45	2.31
2027	952,003	5.59	28.66	0.51	3.09
2028	830,169	5.61	34.21	0.79	3.69
Total	**13,675,761**	**3.78**	**26.11**	**0.70**	**4.46**

	Zinc Concentrate		Copper Concentrate				Dore		
Year	Tonnes	Zn (%)	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Oz	% Au	% Ag
2019	155,533	50.8	41,698	51.7	485.8	20.3	0	-	-
2020	172,547	50.9	37,418	48.2	575.2	20.5	0	-	-
2021	175,154	50.8	41,626	56.2	542.2	20.3	0	-	-
2022	123,323	51	40,975	83.0	523.8	23.4	130,829	25.4%	69.6%
2023	137,137	51	43,139	81.1	490.5	23.9	91,003	26.9%	68.1%
2024	125,699	51	57,336	60.6	373.4	25.6	107,628	24.8%	70.2%
2025	63,350	51	42,285	102.0	494.1	24.8	116,201	28.5%	66.5%
2026	36,031	51	16,640	171.0	797.7	22.2	77,967	29.1%	65.9%
2027	50,859	51	18,982	156.2	785.3	22.1	96,944	24.8%	70.2%
2028	44,977	51	25,140	108.1	558.8	23.5	86,035	23.7%	71.3%
Total	**1,084,610**	**50.9**	**365,239**	**80.8**	**524.3**	**22.8**	**706,607**	**26.1%**	**68.9%**

Year	Zn (tonnes)	Cu (tonnes)	Au (oz)	Ag (oz)
2019	79,045	8,444	69,329	651,279
2020	87,794	7,664	57,968	692,018
2021	89,046	8,455	75,194	725,579
2022	62,895	9,575	142,529	781,195
2023	69,940	10,294	136,924	742,301
2024	64,106	14,672	138,475	763,879
2025	32,309	10,478	171,831	749,057
2026	18,376	3,689	114,181	478,153
2027	25,938	4,204	119,379	547,354
2028	22,938	5,914	107,709	513,064
Total	**552,387**	**83,389**	**1,133,519**	**6,643,879**

1.11 RECOVERY METHODS

The Stall concentrator complex is located approximately 16 km east of the Lalor Mine. Conventional crushing, grinding and flotation operations are used to process the ore. The nominal throughput rate is 3,500 tpd and the mill operates 24 hours per day, 365 days per year, with scheduled downtime for maintenance as required. The concentrator produces a copper concentrate with gold and silver credits and a zinc concentrate. Both concentrates are shipped by truck to Flin Flon. From there the copper concentrate is loaded onto rail cars and shipped to third party smelters and the zinc concentrate processed in the Zinc plant. Tailings from the flotation circuit are utilized to produce a cemented paste backfill for use underground. Tailings not required for paste backfill will continue to be pumped to the existing Anderson TIA.

The Flin Flon concentrator is located approximately 198 km from Snow Lake, Manitoba. Conventional crushing, grinding and flotation operations are used to process the excess of the Lalor ore that could not be processed at the Stall concentrator due to excess inventory. The nominal throughput rate is 7,200 tpd and the mill operates 24 hours per day, 365 days per year, with scheduled downtime for maintenance as required. The concentrator produces a copper concentrate with gold and silver credits and a zinc concentrate. The copper concentrate is loaded onto rail cars and shipped to third party smelters. Tailings from the flotation circuit are utilized to produce a cemented paste backfill for use underground in the Flin Flon mining operations. Tailings not required for paste backfill are pumped to an existing tailings pond.

The New Britannia mill is located approximately 16 km north of the Lalor mine. Conventional crushing, grinding, flotation, cyanide leach and recovery methods will be used to process the ore. The nominal throughput rate will be 1,500 tpd and the plant will operate 24 hours per day for 362 days per year, with an availability of 92%. The concentrator will produce concentrate and doré gold bars. The copper concentrate will be shipped by truck to Flin Flon and will then be delivered to third party smelters. It is anticipated that, once New Britannia is operational, the copper concentrate produced at the Stall mill will be dewatered at New Britannia. The doré bars will be shipped offsite using conventional third-party transportation.

Zinc concentrate from Stall is trucked to Hudbay's zinc plant in Flin Flon approximately 198 km from Snow Lake, Manitoba which produces pure zinc metal at its refinery. This refinery was refitted with the world's first two-stage pressure leach operation for recovering zinc from zinc sulphide concentrates in 1993. No roasters are used in this zinc extraction process and no sulfur dioxide (SO_2) gas is produced.

1.12 MARKET STUDIES AND CONTRACTS

Lalor produces the majority of its zinc concentrate and copper concentrate with gold and silver credits from its Stall mill. The refurbished New Britannia mill will process ore from the Lalor gold zone to produce a copper concentrate and gold doré bar.

Currently, the Lalor zinc concentrate is delivered by truck to Hudbay's operations in Flin Flon and processed at the Flin Flon zinc plant into refined zinc metal and sold to customers in North America. The economic assumptions presume the same into 2021. During or beyond 2021, it is anticipated that Lalor's zinc concentrate will be sold to third party North American refineries.

The Lalor copper concentrate produced at the Stall and Flin Flon mills is sold directly to a copper smelter in North America under a long-term agreement. The Lalor copper concentrate is delivered by rail. Lalor copper concentrate expected to be produced from the refurbished New Britannia mill has not been committed. It is anticipated that all sales contracts for the sale of copper concentrate produced from the refurbished New Britannia mill will be at standard industry terms.

The refurbished New Britannia mill will produce a gold doré commencing in 2022. It is forecast to contain approximately 69% silver, 26% gold and 5% other elements. The doré bar will be delivered and sold to refineries using conventional third-party transportation at standard industry terms.

Engineering, supply and construction contracts are initiated, managed and administrated by Hudbay's Manitoba Business Unit. Hudbay has a marketing division that is responsible for administering the company's marketing and sales of concentrates and metals. As well, Hudbay conducts ongoing research on metal prices and sales terms as part of normal business and long-range planning process to achieve market terms. Contract terms used in the Lalor financial evaluation are based on this research.

1.13 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The permits required for the current Lalor operation, including the Lalor mine, Stall concentrator and Anderson tailings facility have all been issued and remain valid.

At this time, there are no known environmental concerns which could adversely affect Hudbay's ability to operate the Lalor mine. Since the mine site is nearby existing facilities in the Snow Lake area, the Lalor mine was able to utilize infrastructure, services, and previously disturbed land associated with permitted, pre-existing and current mining operations in the Snow Lake area. The Lalor mine and associated projects are designed to minimize the potential impact on the surrounding environment by keeping the footprint of the operations as small as possible and by using existing licensed facilities for the withdrawal of water and disposal of wastes.

The re-commissioning of the New Britannia concentrator will involve the placement of the Lalor gold ore tailings in the fully licensed Anderson tailings facility or the Lalor Paste fill plant via a pipeline that will connect to the Stall concentrator. Environmental permits have been submitted for the refurbishment of the New Britannia mill and associated infrastructure.

The New Britannia site includes the Birch Lake Tailings Management Facility (BLTMF). The BLTMF is currently on care and maintenance since the closure of the mine in 2005. Hudbay is currently in the process of applying for a new water withdrawal licence for this site. At this stage, and given Hudbay's history in the area, there is no indication that the approvals will not be obtained within the project schedule.

Hudbay engaged AECOM in 2018 to carry out baseline studies for the Pen II property. These studies will comply with the regulatory requirements and will be part of the permitting submission process should the Pen II Zone project be advanced.

The Wim deposit is at a scoping study level and drilling is currently underway to assess the metallurgical characteristics of the mineralized material and to bring the engineering to a pre-feasibility level. If the project demonstrates economic viability, Hudbay will initiate a baseline study and the appropriate permit application(s) will be sent to the relevant government department(s).

Based on Hudbay's long-term (more than 50 years) mining experience in the Snow Lake region, and baseline studies to date, there is no known First Nation or Aboriginal hunting, fishing, trapping or other traditional use of the land in the zone of potential influence for the Lalor mine and associated facilities. Post closure, all water quality and earthen structures will be monitored and inspected in order to ensure the sites' conditions meet the applicable regulatory requirements.

1.14 CAPITAL AND OPERATING COSTS

The capital expenditures required to execute the life of mine plan at Lalor cover $125M in growth projects during the 2020-2021 period represented by the New Britannia mill refurbishment and the construction of a pipeline corridor to support the planned increase in gold production, and $394 m in sustaining capital over the life of the mine which includes the capitalized mine development, the replacement/acquisition of mining equipment and finally other capitalized expenditures related to milling and environmental activities. Sustaining capital costs for the new mine plan aim at preserving mineralization with higher precious metals content for extraction following commissioning of the New Britannia mill. As a result, capitalized development costs are relatively higher during the first three years as priority is given to maximizing base metals output. Overall, this approach results in a superior economic value for the Snow Lake Operations.

The operating cost estimates are based on the 2019 budget figures and were developed by Hudbay using a bottom-up approach and quotes from local suppliers, Manitoba operations experience, labor costs within the region and actual 2018 costs. The operating costs include all the relevant mining, milling and refining activities as well as the attributable general and administrative costs to the Lalor production. Labour costs constitute the largest component of the mining and milling operating costs. Labour costs include both employee and contractor costs arising from mining, milling and maintenance activities. Until the closure of the zinc plant in 2021, the Lalor zinc concentrate will be refined into finished zinc in order to maximize the realizable value from zinc produced. An operating cost has been allocated to the Snow Lake Operations for the refining of the zinc concentrates produced from the Lalor mine's production. Post 2021, the Lalor zinc concentrate will be sold directly to market. General and administration costs are comprised of an allocation of shared services, administration, and other office costs to the Snow Lake Operations based on relative usage versus the Flin Flon operations.

Table 1-7 illustrates that Lalor's significant by-product credits will reduce its cash operating costs and sustaining cash costs both on a zinc and gold basis. During the first five years of operation with New

Britannia (2022 to 2026), Lalor is estimated to produce approximately 140,000 ounces of gold annually at a sustaining cash cost, net of by-product credits, of $US 450/oz. This positions Lalor to be one of the lowest cost gold mines in Canada.

TABLE 1-7: CASH AND SUSTAINING UNIT CASH COSTS

CASH & SUSTAINING COST[1]		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Zinc Basis												
Contained zinc	M lbs	174	194	196	139	154	141	71	41	57	51	1,218
Cash cost	US$/lb	0.61	0.73	0.55	(0.05)	0.03	(0.18)	(1.21)	(0.87)	(0.68)	(1.08)	0.09
Sustaining cash cost	US$/lb	1.04	1.14	0.78	0.26	0.22	(0.04)	(0.95)	(0.77)	(0.67)	(1.07)	0.35
Gold Basis												
Contained gold	'000s oz	69	58	75	143	137	138	172	114	119	108	1,134
Cash cost	US$/oz	(672)	(308)	35	268	211	85	333	581	448	278	198
Sustaining cash cost	US$/oz	400	1,051	616	571	416	229	442	618	456	279	473

[1]Cash cost and sustaining cash cost are presented net of by-product credits

1.15 INTERPRETATION AND CONCLUSION

Lalor Mine has been in production continuously since mid-2012 and has operated without major interruptions. As of the end of 2018, the mine has produced 5.6 million tonnes. After a continuous production ramp-up cycle, the mine has reached its steady state optimal ore production of around 4,500 tpd.

The strategy for the 2018 update of the mine plan at Lalor was to mine base metals of higher grade zinc production from 2019 to the end of 2021 while at the same time advancing the development required to start mining the gold and copper gold zones. Once the New Britannia mill becomes fully operational in Q1 of 2022, the mine plan will target a production of 1,000 to 1,500 tonnes per day of gold and copper gold ore, supplemented by 2,400 to 3,400 tonnes per day of base metal ore until 2025, followed by lower production rates until depletion of the mineral reserves.

All the relevant capital and operating expenditures required to execute the life of mine plan at Lalor have been accounted for in the demonstration of economic viability of the mineral reserve estimates including growth and sustaining capital costs and operating costs for the mining, milling and refining activities as well as the attributable general and administrative costs.

The author considers that the mineral resources and reserves as classified and reported comply with all disclosure in accordance with requirements and CIM Definitions. The author is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

Since the last technical report published in March 2017, Hudbay has consolidated land nearby its Lalor mine which has resulted in the acquisition of Wim deposit. Combined with the former New Britannia mine and its satellite deposits, and with the recent discovery of new zinc and gold mineralization zones between Lalor and the former Chisel mine, Hudbay is in a strong position to increase its resources and reserves in the short term.

1.16 RECOMMENDATIONS

The author recommends continuing to advance engineering studies on Lalor and its satellite deposits in an attempt to continue to increase the tonnage of the mineral reserve estimates and the estimated operating life of the Snow Lake processing facilities at or near full capacity.

At Lalor, the author recommends pursuing in-mine exploration to continue to incrementally add both base metal and gold rich inferred resources where lenses are still open in their along strike or down dip directions. In addition, the drilling programs aimed at confirming the discovery of a new Cu-Au lens and to test the fold repeat of the favorable Lalor horizon below the 1,050m level should be accelerated. In addition, the in-mine drill program should continue to convert enough inferred mineral resource estimates to an indicated category in order to offset annual mining depletion. It is also recommended to test the applicability of conditional simulations to assess if the confidence in the recoverable tonnes and grade in some of the inferred resource areas could be elevated.

At the Wim and Pen II deposits, additional drilling is recommended in 2019 to collect representative core samples to conduct additional metallurgical testing. At the former New Britannia, mine and its satellite gold deposits (boundary, No. 3 Zone), the resource models should be re-constructed to support a new pre-feasibility study on the remaining mineral resource estimates.

The regional surface exploration program should continue to define the extent and quality of the new mineral deposit discovered between Lalor and the past producing Chisel mine and also around the past producing Snow Lake gold properties.

From a metallurgical perspective, it is recommended to conduct additional testing for gravity concentration. The collector dosage and flotation time should be modified at the New Britannia mill in order to account for the high copper grade from Lens 27 and recovery models should be developed based on rougher and cleaner flotation kinetics using the Aminpro methodology.

All recommendations can be funded from the 2019 Hudbay exploration budget of $17 M for Manitoba.

2. INTRODUCTION & TERMS OF REFERENCE

This Technical Report has been prepared for Hudbay to support the public disclosure of Mineral Resources and Mineral Reserves at the Lalor Mine and nearby Snow Lake satellite deposits and to provide an updated mine plan that contemplates 4,500 tpd of base metal, gold and copper-gold ore feed to the Stall, Flin Flon and New Britannia concentrators.

Hudbay is a Canadian integrated mining company with assets in North and South America, principally focused on the discovery, production and marketing of base and precious metals. Hudbay's objective is to maximize shareholder value through efficient operations, organic growth and accretive acquisitions, while maintaining its financial strength.

Hudbay operates multiple properties in the Province of Manitoba. Operations near Flin Flon include the 777 Mine, an ore concentrator and a zinc plant with all required tailings disposal facilities. Current operations near Snow Lake include the Lalor underground mine as well as an ore concentrator, a tailings impoundment area and other ancillary facilities that support the operation. The property is located approximately 16 km by road west of the town of Snow Lake, Manitoba. Hudbay owns a 100% interest in the Lalor property through one Mineral Lease and eight Order in Council Leases to the south of the mine. As of the issue date of this Technical Report, the Lalor mine is operating at a processing rate of approximately 4,500 tpd.

Refurbishing the New Britannia mill is expected to significantly increase gold production from Lalor and enable new gold and copper-gold exploration opportunities in the Snow Lake region by having an operating processing facility with substantially higher gold and copper recoveries. New Britannia was placed on care and maintenance in 2005 by its previous owner after producing 1.6 million ounces of gold. Refurbished, it will create additional value for Hudbay in the Flin Flon and Snow Lake regions as Hudbay pursues low-risk brownfield development opportunities.

The Wim deposit was acquired by Hudbay in the third quarter of 2018 for approximately C$0.5 million. Wim is a copper-gold deposit that starts from surface and is located approximately 15 km by road from the New Britannia mill. Hudbay is developing a mine plan and conducting metallurgical testing on the Wim deposit with the objective to upgrade the measured and indicated resources to mineral reserves. Wim has the potential for an underground ramp and could feed the New Britannia mill after the richest portions of the Lalor reserves and resources have been depleted.

New Britannia is a former producing gold mine that produced approximately 600,000 oz between 1949 and 1958 and an additional 800,000 oz between 1995 and 2005. Significant mineral resources remain accessible at New Britannia as well as in the nearby Birch and No. 3 Zone with some investments in existing mining infrastructure. WSP was engaged in 2018 to audit and restate the historical resource estimates previously reported for these deposits. Hudbay plans to initiate technical studies in the second half of 2019 to determine the technical and economic viability of the existing mineral resources and the potential to process this material at the New Britannia mill.

Pen II is a low tonnage and high-grade zinc deposit that starts from surface and is located approximately 6 km by road from the Stall mill. Pen II could constitute a supplemental source of feed for the Stall mill. In 2019, Hudbay will continue metallurgical testing, infill drilling on the inferred resource estimates and technical studies in an attempt to confirm the technical and economic viability of the mineral resource estimates.

2.1 QUALIFIED PERSON (QP)

This Technical Report has been prepared in accordance with National Instrument Form NI 43-101F1. The QP who supervised the preparation of this Technical Report is Olivier Tavchandjian, P. Geo., Hudbay's Vice-President, Exploration and Geology.

Mr. Tavchandjian is not independent of Hudbay, and this is not an independent technical report. Nevertheless, Hudbay is a "producing issuer" as defined in NI 43-101. As such, this technical report is not required to be prepared by or under the supervision of an independent QP.

Mr. Tavchandjian has been directly involved on a regular basis with the exploration, geology, resource-modelling, mine planning as well as with the estimation of operating and capital costs for the Lalor mine and the Pen II deposit. Mr. Tavchandjian has visited the mine on a monthly basis since September 2017 as well as the core storage facilities and the relevant internal and external laboratories and has directly overseen the mineral resource and reserve estimation process.

Greg Greenough and Paul Palmer, P. Geos for Golder have prepared the Wim mineral resource estimates and Todd McCracken, P. Geo for WSP has prepared the mineral resource estimates for the New Britannia mine site. Mr. Tavchandjian has reviewed the work conducted by Golder and WSP and is satisfied that the mineral resource estimates for the Wim and New Britannia properties were estimated in compliance with NI 43-101.

2.2 SOURCES OF INFORMATION

Sources of information for geological and mineral resources include: core drilling and sampling data, underground development and mapping, and assay and geochemistry analysis.

Sources of information for the mineral reserves include: the mineral resource block model, actual production and cost data, budget projections, life of mine inventory based on stope geometry parameters and mine development sequence.

Sources of information for the metallurgy, processing and economic analysis include: the actual operating data for the current processing facilities and a pre-feasibility study (AECOM, 2018) for the New Britannia mill and its associated infrastructures.

2.3 UNIT ABBREVIATIONS

Units of measurement in this report conform to the SI (metric) system unless otherwise noted. Table 2-1 above, lists the notable unit abbreviations utilized in this report.

TABLE 2-1: UNIT ABBREVIATIONS

Abbreviation	Unit	Abbreviation	Unit
$C or C$	Canadian dollars	m	Metre
$US or US$	US dollars	M	million
%	Percent	m ASL	metres above sea level
°C	degree Celsius	m^2	squared metre
µm	micrometre or micron	m^3	cubic metre
BTU	British thermal unit	m^3/hr	cubic metre per hour
CFM or cfm	Cubic feet per minute	mL	metre level
dmt	Dry metric tonnes	mm	millimetre
ft	foot	ml	millilitres
g	gram	MVA	Mega volt amp
g/t	gram per metric tonne	MW	Megawatt
Ga	billion years	nT	nanotesla
gpm	gallon per minute	oz	Troy ounces
Ha	hectare	ppm	parts per million
HP or hp	Horsepower	psi	Pounds per square inch
hr	hour	t	metric tonne
kg	kilogram	tpd	metric tonnes per day
km	kilometre	US GPM	United States gallon per minute
km/hr	kilometre per hour	US$ or $US	United States dollar
kV	kilovolt	V	Volt
kW	kilowatt	wmt	Wet metric tonne
L/min	litres per minute	Zn Eq	Percent
lb	pound (unit of weight)		

2.4 ACRONYMS & ABBREVIATIONS

Abbreviations of company names and other notable terms used in the report are as shown in Table 2-2.

TABLE 2-2: ACRONYMS AND ABBREVIATIONS

Abbreviation	Term	Abbreviation	Term	Abbreviation	Term
3D	Three-Dimensional	EEM	Environmental Effects Monitoring	NSS	Near solid sulphide
AAS	Atomic Absorption Spectrometry	EM	Electromagnetic	OIC	Order In Council
AEP	Advanced Exploration Project	ES	Emission spectrograph	OES	Optical Emission Spectrometry
Ag	Silver	Fe	Iron	OK	Ordinary Kriging
ANFO	Ammonium nitrate	FOB	Fine ore bin	OLS	Ordinary least square
As	Arsenic	GPSS	Global Positioning Satellite System	OREAS	Ore Research and Exploration
ASL	Above Sea Level	HCl	Hydrogen chloride	P. Eng.	Professional Engineer
Au	Gold	HNO_3	Nitric acid	Pb	Lead
Au Eq	Gold Equivalency	Hudbay	Collectively all Hudbay Minerals Inc. subsidiaries and business groups	PR	Provincial Road
AV	average	ICP	Inductively Coupled Plasma	QAQC	Quality Assurance and Quality Control
BLTMF	Birch Lake Tailings Management Facility	LCT	Lock cycle test	QP	Qualified Person
BQ	BQ drill core size 36.4mm	IDW	Inverse Distance Squared Weighted	Reflex	Reflex E-Z Shot
BV	Bureau Veritas	LHD	Load Haul and Dump	ROM	Run of mine
CBV	Certified Best Value	LIMS	Information management system	R^2	Coefficient of determination
CIM	Canadian Institute of Mining, Metallurgy and Petroleum	LOD	Limit of detection	RTK	Real Time Kinematic
CIP/CIL	Carbon-in-pulp / Carbon-in-leach	LOM	Life of Mine	SG	Specific Gravity
CRF	Cemented waste rock backfill	MBU	Manitoba Business Unit	SS	Solid sulphide
CRM	Certified Reference Materials	MCCN	Mathias Colomb First Nation	TIA	Tailings Impoundment Area
Cu	Copper	MIBC	Methyl isobutyl carbinol	RE	Relative Error
CV	Coefficient of Variation	ML	Mineral Lease	TMI	Total Magnetic Intensity
DH	Drill hole	MS	Mass Spectrometry	UCS	Unconfined compressive strength
DE	Data entry	NoA	Notice of Alteration	URF	Unconsolidated waste rock backfill
DGPS	Differential Global Positioning System	NI	National Instrument	UTM	Universal Transverse Mercator
EAL	Environmental Act Licence	NN	Nearest Neighbour	VMS	Volcanogenic Massive Sulphide
EDA	Exploratory data analysis	NSR	Net smelter return	Zn	Zinc
EDM	Electronic distance measurement	NQ	NQ drill core size 47.6mm	Zn Eq	Zinc Equivalency

2.5 OTHER CONTRIBUTORS

The responsible QP for all information is Olivier Tavchandjian. Table 2-3 presents the other contributors and participants for this report.

TABLE 2-3 CONTIBUTORS FOR THIS REPORT

Section	Description	Participants
1	Summary	Olivier Tavchandjian
2	Introduction	Olivier Tavchandjian
3	Reliance on Other Experts	Olivier Tavchandjian
4	Property Description and Location	Kenda Kozar
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Neil Richardson
6	History	Neil Richardson
7	Geological Setting and Mineralization	Keith Harrison, John Townend, Neil Richardson
8	Deposit Type	Keith Harrison, John Townend, Neil Richardson
9	Exploration	Keith Harrison, John Townend, Neil Richardson
10	Drilling	Keith Harrison, John Townend, Neil Richardson
11	Sample Preparation Analyses and Security	Marc-Andre Brulotte (Hudbay), Greg Greenough (Golder), Paul Palmer (Golder) and Todd McCracken (WSP)
12	Data Verification	Marc-Andre Brulotte (Hudbay), Greg Greenough (Golder), Paul Palmer (Golder) and Todd McCracken (WSP)
13	Mineral Processing and Metallurgical Testing	Jarid Medina, Karl Hoover
14	Mineral Resource Estimates	Marc-Andre Brulotte (Hudbay), Vicki Morrison (Hudbay), Greg Greenough (Golder), Paul Palmer (Golder) and Todd McCracken (WSP)
15	Mineral Reserve Estimates	Doug Salahub, Robert Carter
16	Mining Methods	Doug Salahub, Robert Carter
17	Recovery Methods	Jarid Medina, Karl Hoover
18	Project Infrastructure	Jarid Medina, Robert Carter
19	Market Studies and Contracts	Jon Douglas
20	Environmental Studies, Permitting, and Social or Community Impact	Jay Cooper
21	Capital and Operating Costs	Doug Salahub, Robert Carter
22	Economic Analysis	Doug Salahub, Robert Carter
23	Adjacent Properties	Neil Richardson
24	Other Relevant Data and Information	*NA*
25	Interpretation and Conclusions	Olivier Tavchandjian
26	Recommendations	Olivier Tavchandjian
27	References	Marc-Andre Brulotte

3. RELIANCE ON OTHER EXPERTS

Standard professional procedures were followed when preparing the contents of this Technical Report. Data used in this report has been verified where possible and the QP has no reason to believe that the data was not collected in a professional manner and no information has been withheld that would affect the conclusions made herein.

The information, conclusions, opinions, and estimates contained herein are based on:

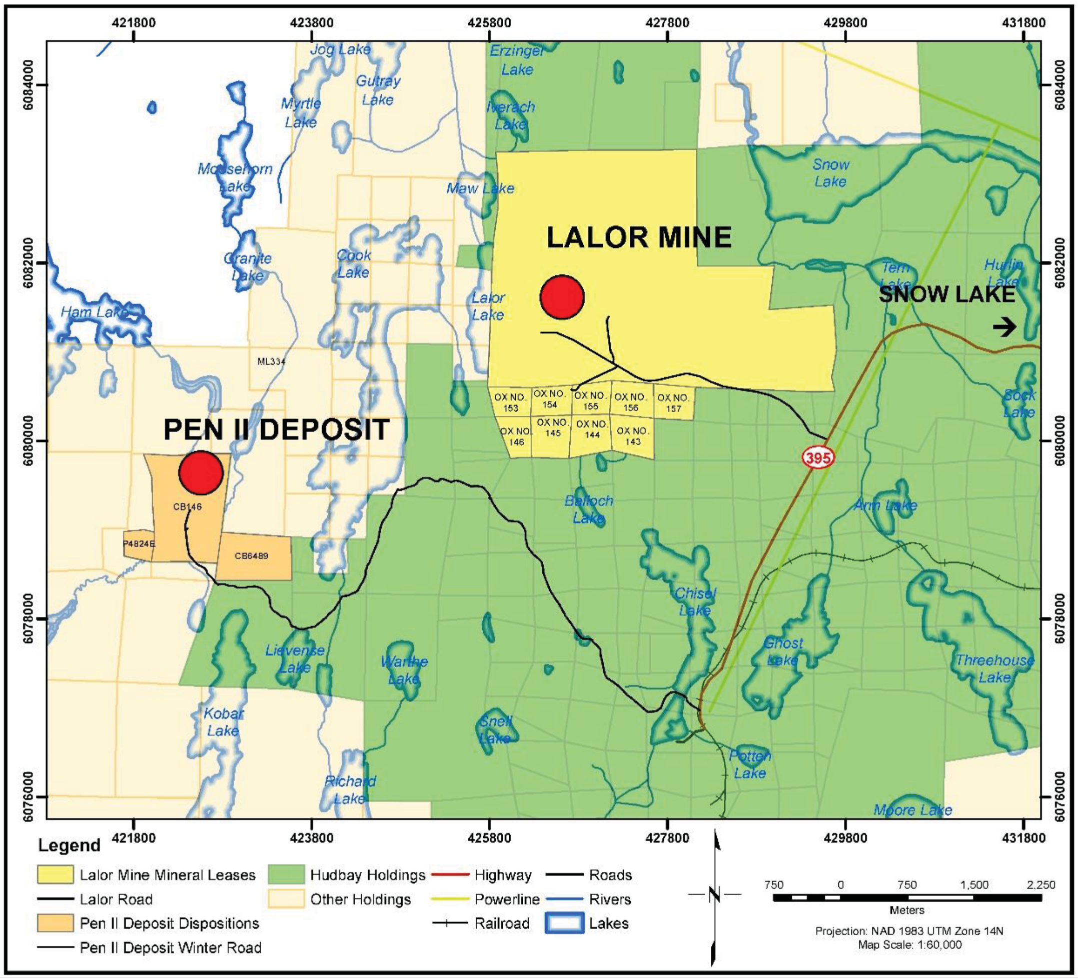
- Information available to Hudbay at the time of preparation of this technical report, and
- Assumptions, conditions, and qualifications as set forth in this technical report.

4. PROPERTY DESCRIPTION & LOCATION

4.1 LALOR MINE AND PEN II DEPOSITS

The Lalor mine is located 16 km by road west of the town of Snow Lake at 54°52’24” north latitude, 100°09’01” west longitude and 303 mASL on National Topographic System (NTS) map 63K/16 (Figure 4-1).

FIGURE 4-1: LOCATION OF THE LALOR AND PEN II DEPOSITS MINERAL LEASES AND MINERAL CLAIMS



Hudbay owns a 100% interest in the Lalor property through one Mineral Lease and eight Order in Council Leases to the south of the property.

Hudbay holds the exclusive right to the minerals, other than quarry minerals, and the mineral access rights required for working the lands, mining and producing minerals from the Lalor mine. The surface tenure required for the mining operations have all been approved.

The Pen II deposit is located 12 km west of the town of Snow Lake at 54°51’28” north latitude and 100°12’20” west longitude and 296 mASL on NTS map 63K/16.

Hudbay holds the mineral rights to claim CB146 which hosts the Pen II deposit. Other contiguous claims are held by Hudbay Minerals in the area and they comprise of one fraction and one regular claim for a total of 196 Ha (Figure 4-1 and Table 4-1)

TABLE 4-1: LIST OF BOTH MINERAL LEASES AND OIC LEASES

Project	Name	Number	Holder	Expiry Date	Hectares
Lalor Mine	OX NO. 155	M5780	Hudbay Minerals Inc.	4/8/2023	18.25
Lalor Mine	OX NO. 157	M5729	Hudbay Minerals Inc.	12/9/2022	20.52
Lalor Mine	OX NO. 153	M5778	Hudbay Minerals Inc.	4/8/2023	15.90
Lalor Mine		ML334	Hudbay Minerals Inc.	3/29/2033	195.50
Lalor Mine	OX NO. 143	M7278	Hudbay Minerals Inc.	9/6/2023	21.70
Lalor Mine	OX NO. 156	M5781	Hudbay Minerals Inc.	4/8/2023	20.20
Lalor Mine	OX NO. 146	M7281	Hudbay Minerals Inc.	9/6/2023	14.84
Lalor Mine	OX NO. 145	M7280	Hudbay Minerals Inc.	9/6/2023	21.60
Lalor Mine	OX NO. 144	M7279	Hudbay Minerals Inc.	9/6/2023	20.55
Lalor Mine	OX NO. 154	M5779	Hudbay Minerals Inc.	4/8/2023	17.99
Pen II Deposit		CB146	Hudbay Minerals Inc.	2/9/2025	126.00
Pen II Deposit	PEN 4824 FR.	P4824E	Hudbay Minerals Inc.	6/13/2025	9.00
Pen II Deposit		CB6489	Hudbay Minerals Inc.	7/16/2024	62.00

Land Use Permitting

General Permits are issued and administered by the Province of Manitoba Real Estate Services Division, Manitoba Department of Finance. Provided all terms and conditions of the General Permit are met, including payment of annual fees, the permit is automatically renewed for a 1-year term on an annual basis. Quarry Leases are issued and administered by the Province of Manitoba Mines Branch.

Two General Permits (GP59093 and GP63483) and one Quarry Lease are held by Hudbay and are required to carry on mining activities at Lalor. In detail, these are:

GP59093: This General Permit covers an area of 4.05Ha (4.0 km x 5m) and includes an all-weather access road, accommodates a 4 km 25 kV transmission line, a 4 km discharge line, a 4 km fresh water line, a 200m x 200m parking lot and an additional access road.

GP63483: The General Permit is related to the mine site.

QL-1928: QL-1928 was renewed November 26, 2017 for a 10-year term and provides the holder with the exclusive right to explore for, develop, and produce clay, gravel, rock or stone. QL-1928 will expire November 2027 and provided this lease is still required it can be renewed for another 10 year term.

General Permit and Quarry Lease status for the Lalor property is shown in Table 4-2 .

TABLE 4-2: GENERAL PERMITS AND QUARRY LEASE FOR LALOR

Permit/Quarry Number	Holder	Issue Dates	Hectares	Annual Fees (ExcludesGST)
GP59093	Hudbay	31-Dec-07	4.05	$415
GP63483	Hudbay	10-Jun-10	159.37	$798
QL-1928	Hudbay	26-Nov-17	11.00	$297
Total			***174.42***	***$1,510***

4.2 NEW BRITANNIA MINE AND OTHER SNOW LAKE GOLD PROPERTIES

Since 2015, Hudbay owns 100% interest in the Snow Lake Gold Properties through two Mineral Leases and 57 Mineral Claims. The New Britannia Mine-Mill Complex is located on the north side of the Town of Snow Lake at 54°53'10” north latitude and 100°01'24” west longitude and 293 mASL on NTS map 63K/16 (Figure 4-2)

The two mining leases have been legally surveyed and remain in good standing providing that the annual taxes and lease fees are paid. The lease fees are CAN$ 12.00/Ha each year and are subject to a 1.38% NSR, based on production, payable to W. Bruce Dunlop.

The following summarizes the sequence of corporate transactions and mergers leading to the current ownership and consolidation of the property:

Prior to December 22, 2006, the mining leases and the Hudbay Option claims were held by 1126774 Ontario Limited (1126774 Ontario) which was held by Kinross and High River Gold Ltd. in a joint venture, with each party holding 50%.

On December 22, 2006, 1126774 Ontario was combined with Pegasus Mines Ltd. (Pegasus), with the merged entity retaining the name Pegasus.

Pegasus was wholly-owned by the New Britannia Mine Joint Venture (NBM-JV) which consisted of Piper Capital Inc. (Piper) owning 60%, and Garson Resources Ltd. (Garson Resources) owning 40%. Prior to the establishment of the NBM-JV, Garson Resources Ltd. held the Squall Lake claims.

On June 28, 2007, Piper and Garson Resources merged to form Garson Gold Corp.

On September 19, 2007, Pegasus changed its name to New Britannia Mines Ltd.

The terms of the agreement with Kinross have been described in the previous Micon Technical Report (Lewis, 2006).

At the time of the original transaction with Kinross, the claims were optioned to Kinross by Hudbay. Exercise of the option required a one-time cash payment of CAN$ 400,000 and a royalty consisting of 1.5% of the gross proceeds from the sale or disposition of all metals mined and removed from the optioned claims. However, Hudbay has retained the rights to the base metals on the mineral claims and should a base metal deposit be discovered and exploited on the claims, then Garson would retain a 1.5% royalty.

On December 31, 2007, the CAN$ 400,000 option payment was made and, subject to the 1.5% royalty, the Hudbay claims are now 100%-owned by NBM Ltd. Prior to conducting any production mining, the claims would have to be either incorporated into the existing mineral lease ML-61 or incorporated into a new mining lease. Micon indicated that due to the pre-existing royalty on ML-61, the claims should be incorporated into a new mining lease (Lewis, 2009). This would create a boundary which could be surveyed underground so that no conflicts between the two separate royalty holders would exist.

The Squall Lake group of claims north of Snow Lake comprises 14 contiguous, unpatented mining claims with an area of 899 Ha and is 100% owned by Hudbay, subject to various underlying royalties. The property contains a nets profits royalty (NPI) interest payable on commencement of commercial production of 4% to a private individual, an additional 6% NPI payable to American Barrick (now Barrick), and a 30% NPI payable to Barrick up to a maximum of $550,000. The property also contains a production royalty of $0.10 per ton payable to a private individual on products milled from 6 of the claims. No mineral resource estimates have been estimated in this report for the Squall Lake group of claims.

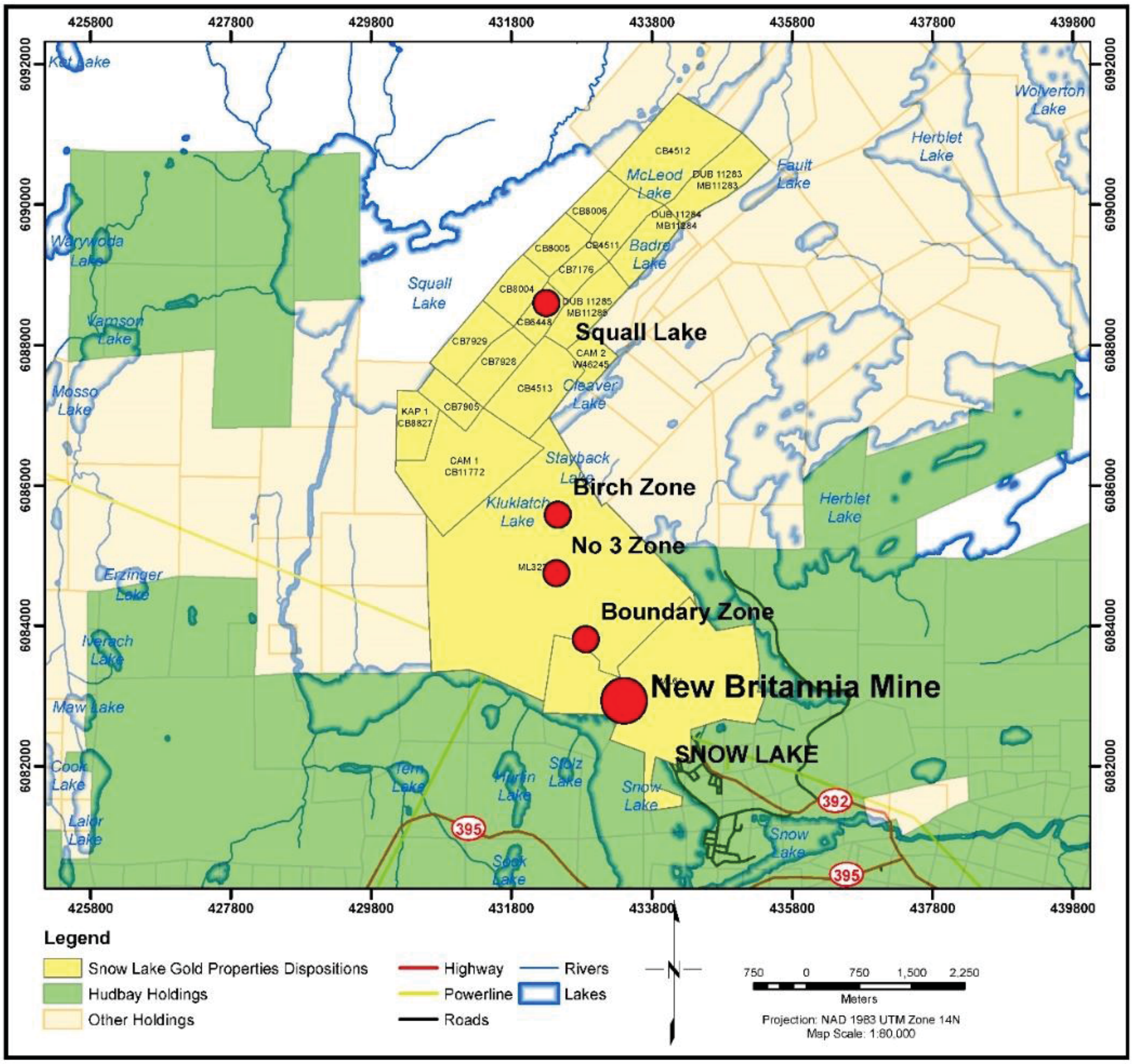
The mineral claims require annual assessment work in the amount of CAN$ 25.00/Ha to keep them in good standing beyond their current expiry dates. In Manitoba, when the claim holder exceeds the minimum annual assessment expenditures for the mineral claims, surplus expenditures can be banked and utilized in future years to meet the annual assessment requirements.

Mineral lease ML-61 (NBM) and mineral lease ML-323 are located on the NTS map sheet 63K/16NE. The mineral claims currently held by Hudbay are located on NTS map sheets 63K/16NE, 63J/13NW, and 63K-16SE. The Squall Lake claims are located on NTS map sheet 63K/16NE (Figure 4-2and Table 4-3).

TABLE 4-3: MINERAL LEASES AND MINERAL CLAIMS FOR THE SNOW LAKE GOLD AND SQUALL LAKE PROPERTIES

Project	Name	Number	Holder	Expiry	Hectares
Snow Lake Gold		ML61	Hudbay Minerals Inc.	4/1/2034	419
Snow Lake Gold		ML323	Hudbay Minerals Inc.	3/28/2037	828
Squall Area	DUB 11283	MB11283	Hudbay Minerals Inc.	10/13/2019	75
Squall Area	DUB 11284	MB11284	Hudbay Minerals Inc.	10/13/2019	53
Squall Area	DUB 11285	MB11285	Hudbay Minerals Inc.	10/13/2019	63
Squall Area	KAP 1	CB8827	Hudbay Minerals Inc.	11/23/2019	48
Squall Area		CB7905	Hudbay Minerals Inc.	3/11/2022	45
Squall Area	CAM 1	CB11772	Hudbay Minerals Inc.	3/16/2022	195
Squall Area		CB7928	Hudbay Minerals Inc.	5/13/2022	42
Squall Area	CAM 2	W46245	Hudbay Minerals Inc.	3/16/2026	17
Squall Area		CB7929	Hudbay Minerals Inc.	7/8/2026	43
Squall Area		CB8004	Hudbay Minerals Inc.	7/8/2026	43
Squall Area		CB8005	Hudbay Minerals Inc.	7/8/2026	43
Squall Area		CB8006	Hudbay Minerals Inc.	7/8/2026	43
Squall Area		CB7176	Hudbay Minerals Inc.	9/5/2026	33
Squall Area		CB6448	Hudbay Minerals Inc.	9/5/2026	33
Squall Area		CB4513	Hudbay Minerals Inc.	9/5/2026	119
Squall Area		CB4512	Hudbay Minerals Inc.	9/5/2026	152
Squall Area		CB4511	Hudbay Minerals Inc.	9/5/2026	43

FIGURE 4-2: LOCATION OF THE SNOW LAKE AND SQUALL LAKE MINERAL LEASES AND MINERAL CLAIMS



4.3 WIM DEPOSIT

The Wim deposit is located 16 km north of the Town of Snow Lake at 55°1’34” north latitude and 100°2’50” west longitude and 316 mASL on NTS map 63N/01(Figure 4-3).

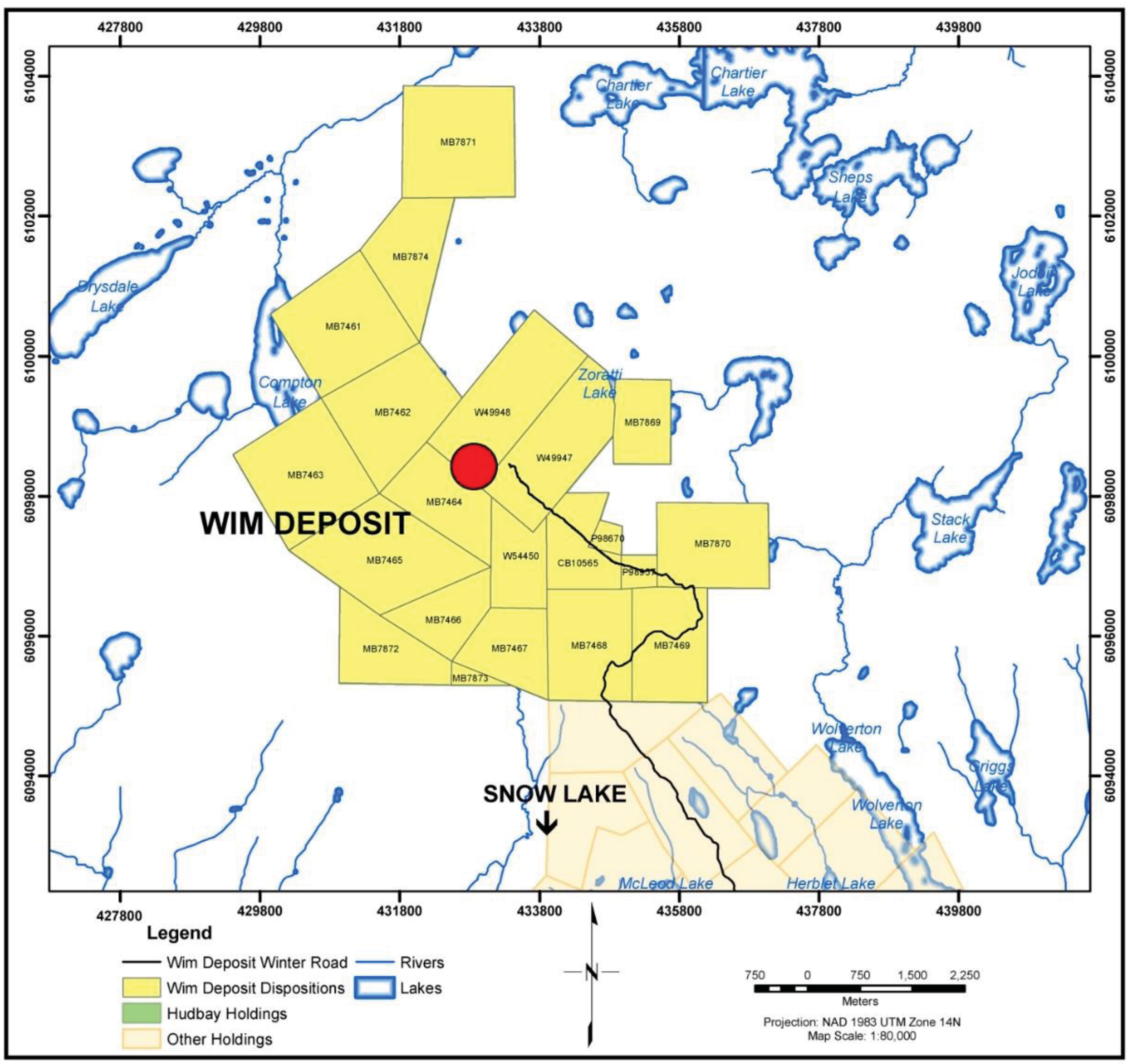
The Wim deposit is 100% owned by Hudbay. On October 31, 2018 it was announced that Alexandria had entered into a purchase agreement with Hudbay to sell the Hudvam and Wim properties. Previously, on December 22, 2014, Alexandria signed an Arrangement Agreement with Murgor Resources Inc. (Murgor) to acquire all Murgor’s shares and assets (the acquisition was completed on March 9, 2015).

Hudbay holds the mineral rights to 21 claims for a total 3,149 Ha. The Wim deposit is hosted by claims W49947 and W49948 (Table 4-4).

TABLE 4-4: MINERAL RIGHTS FOR THE WIM DEPOSIT

Project	Name	Number	Holder	Expiry Date	Hectares
Lalor Mine	OX NO. 155	M5780	Hudbay Minerals Inc.	4/8/2023	18.25
Lalor Mine	OX NO. 157	M5729	Hudbay Minerals Inc.	12/9/2022	20.52
Lalor Mine	OX NO. 153	M5778	Hudbay Minerals Inc.	4/8/2023	15.90
Lalor Mine		ML334	Hudbay Minerals Inc.	3/29/2033	195.50
Lalor Mine	OX NO. 143	M7278	Hudbay Minerals Inc.	9/6/2023	21.70
Lalor Mine	OX NO. 156	M5781	Hudbay Minerals Inc.	4/8/2023	20.20
Lalor Mine	OX NO. 146	M7281	Hudbay Minerals Inc.	9/6/2023	14.84
Lalor Mine	OX NO. 145	M7280	Hudbay Minerals Inc.	9/6/2023	21.60
Lalor Mine	OX NO. 144	M7279	Hudbay Minerals Inc.	9/6/2023	20.55
Lalor Mine	OX NO. 154	M5779	Hudbay Minerals Inc.	4/8/2023	17.99
Pen II Deposit		CB146	Hudbay Minerals Inc.	2/9/2025	126.00
Pen II Deposit	PEN 4824 FR.	P4824E	Hudbay Minerals Inc.	6/13/2025	9.00
Pen II Deposit		CB6489	Hudbay Minerals Inc.	7/16/2024	62.00

FIGURE 4-3: LOCATION OF THE WIM MINERAL CLAIMS



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

5.1 ACCESSIBILITY

The Lalor mine, its nearby satellite deposits and all the infrastructures associated with operations at Snow Lake are located approximately 200 km by road east of the city of Flin Flon and are within or close to the town of Snow Lake, Manitoba. Access to the four mineral deposits is described individually below (Figure 5-1)

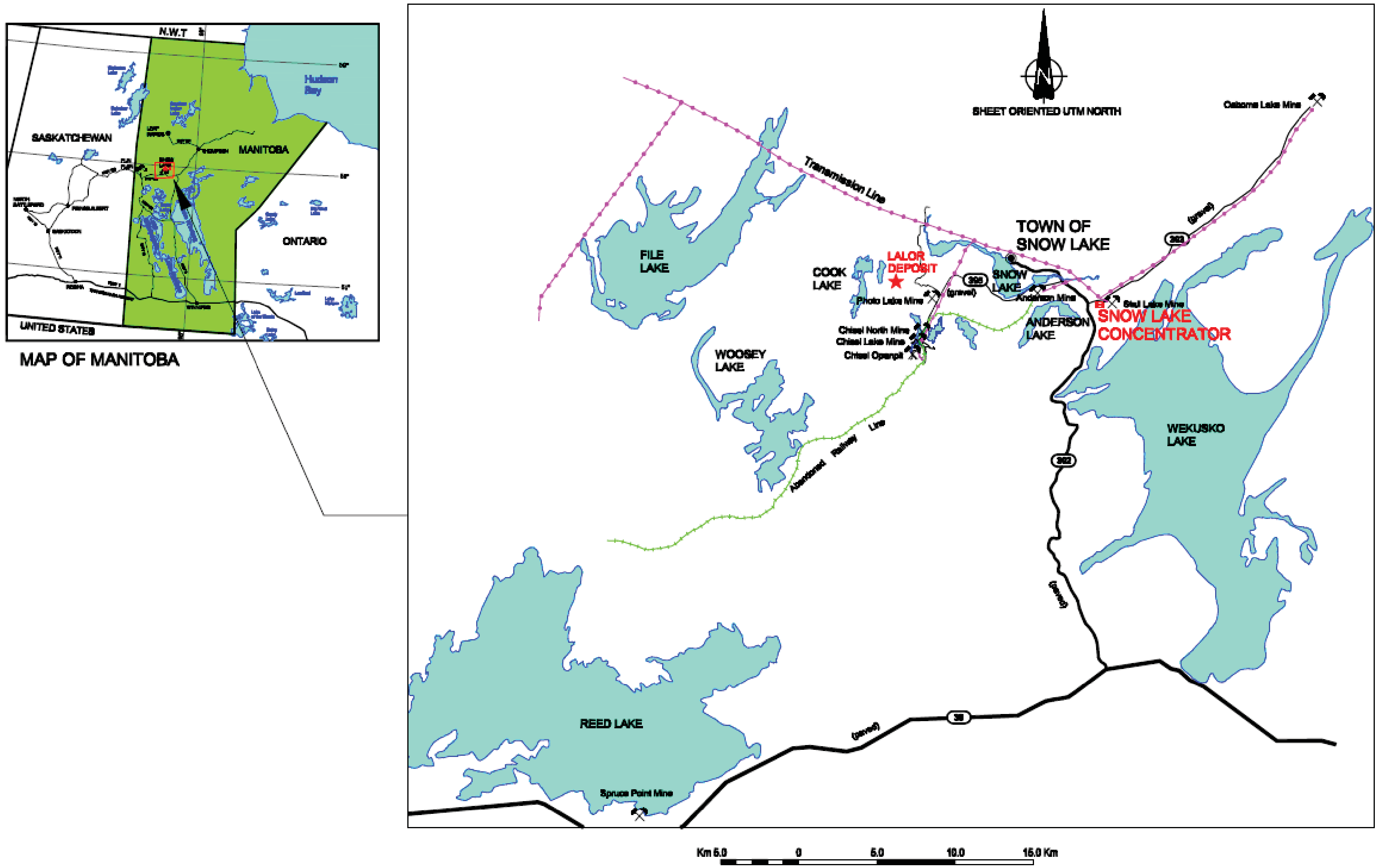
Access to Lalor mine is from Provincial Road (PR) #395 and a gravel road off PR #392 which joins the Town of Snow Lake and PR #39. From PR #395, there is an all-weather permanent road into the mine site.

- Access to the Pen II site is from Provincial Roads #392 and #395, located past the old Chisel North mine site.
- The New Britannia Mine-Mill Complex is located on the north side of the Town of Snow Lake and accessed from paved Cedar Avenue.
- Access to the Wim site is from the Town of Snow Lake, travelling north from the New Britannia mine site and over a tailings dam road along the Manitoba Hydro Transmission line.

5.2 CLIMATE

The Snow Lake area has a typical mid-continental climate, with short summers and long, cold winters. Climate generally has only a minor effect on local exploration and mining activities. The nearest Environment Canada weather station is located near Baker’s Narrows at the Flin Flon airport, approximately 16 km southeast of Flin Flon and approximately 100 km west of Snow Lake. The average annual temperature at the Baker’s Narrows weather station is 0.1°C. The average summer temperature is approximately 17°C, and the average winter temperature is -14°C. The lowest monthly average temperature occurs in January at -21.1°C and the highest monthly average temperature is in July at 18.3°C. Freeze-up of small bays and lakes occurs in mid-November, with breakup occurring in mid-May. There is an average of 115 frost-free days. On average, 45.7 cm of precipitation falls annually, of which, 35% is snow. Since 1960, extreme monthly precipitations have been zero to a high of 18.1 cm, with a maximum daily precipitation of 7.82 cm. Average monthly winds for the area range from 10km/hr to 13km/hr, with 40% of the winds originating from the northwest, northeast or north.

FIGURE 5-1: SNOW LAKE REGIONAL MAP



5.3 LOCAL RESOURCES

The nearest community is the town of Snow Lake, Manitoba. The community of 899 (Statistics Canada, 2016 census) has 498 private dwellings. There are two cottage subdivisions located on Wekusko Lake along PR #392, as well as residences at Herb Lake Landing, located approximately 40 km south of the town. There are also a small number of seasonal remote cottages located near lakes throughout the area.

Snow Lake community services include a health facility staffed with two doctors, an ambulance and a fire truck, a grocery store, two hotels/motels, three service stations, a kindergarten to grade 12 school, a hockey arena, a five-sheet curling rink and a nine-hole golf course.

The nearest larger centres (5,000+ residents) are Flin Flon (208 km), The Pas (219 km) and Thompson (260 km), all accessible by paved highway. There is a 1,100m by 20m un-serviced gravel municipal airstrip located approximately 16 km from Snow Lake along PR #393. A small craft charter service operates out of the community of Snow Lake, where small planes and helicopters can be chartered. Rental vehicles are available at the Flin Flon airport. The nearest full service commercial airport is located at Baker's Narrows near Flin Flon, approximately 185 km from Snow Lake. The nearest international airport is located in Winnipeg, approximately 700 km from Snow Lake.

There is no rail in the immediate area of Snow Lake. The nearest rail access is at Wekusko siding, approximately 65 km southeast of Snow Lake. Wekusko is accessible by an all-weather road. A gravel rail bed (ties and rail removed) connects the Stall concentrator to Chisel Lake mine and continues to a rail line at Optic Lake siding, approximately 65 km west of Chisel Lake. Optic Lake is not road accessible.

To house non-local employees during their work rotations, the company provides a camp located in the Town of Snow Lake which services Hudbay employees and contractors for the mine and mill operations. The camp began operations in January 2011 and has an area of 13,124 m^2. As of February 2019, the camp has seven dorms and three bunkers for a total of 260 rooms. There are also three dedicated trailers for non-dormitory purposes: a 640 m^2 trailer provides a dining room and kitchen for camp employees and the two remaining trailers are a recreation room and a gymnasium. Utilities are provided by the town of Snow Lake, including potable water, sewage and electricity. Heating is electrical and propane. The camp includes parking spots for employees.

5.4 EXISTING INFRASTRUCTURE

Hudbay operates a zinc metallurgical plant in Flin Flon, Manitoba, approximately 200 km from Snow Lake with a capacity of 115,000 tpa of refined zinc. However, Hudbay intends to cease operations and put the zinc plant on care and maintenance at the end of 2021.

Hudbay also operates two ore concentrators: the Stall mill located approximately 16 km from Lalor and the Flin Flon mill located in Flin Flon 200 km to the west. The mills are currently operating seven days per week, producing approximately 3,500 tpd at the Stall mill. During 2019-2021, plans include trucking 1,000 tpd of ore from Lalor to the Flin Flon mill until this facility shuts down at the same time as the zinc plant. The Stall mill has two circuits with design capacities of 909 tpd and 2,182 tpd. At the Flin Flon mill, the ore from Lalor is processed separately as it is free of the encumbrances that affect the other ore processed in Flin Flon from the 777 mine, i.e. ore subject to the Silver Wheaton precious metal streaming agreement.

Both concentrators have two flotation circuits producing a zinc concentrate and a copper concentrate. The zinc concentrate produced at Stall is hauled by truck to Hudbay metallurgical facilities in Flin Flon. The copper concentrates are sold to market. The tailings from the Stall mill are deposited in the Anderson Tailings Impoundment Area or used at the paste plant. The permitted Hudbay Anderson TIA, located approximately 13 km from Lalor, is used for tailings disposal. The tailings from the Flin Flon mill are deposited at the Flin Flon tailings facility or used at the paste plant.

The Lalor paste plant was commissioned in the third quarter of 2018, is located northeast of the existing head frame complex, and is capable of delivering 165 tph solids (tails) or 93 m^3/hr of paste. The paste plant is capable of varying the binder content in the paste to provide flexibility in the strength gained from the paste where higher and early strength may be required depending on mining method.

Tails are currently pumped from the Stall concentrator to the Anderson TIA and diverted, as needed, to the Anderson booster station. Two pipelines are installed between the Anderson booster pump station and the paste plant located at Lalor mine site, which are situated approximately 13 km apart.

General area infrastructure includes provincial roads, 115 kV power to within four kilometres of Lalor provided by Manitoba Hydro and telephone land line and cellular phone service provided by Manitoba Telecom.

The Lalor mine is located 3.5 km from the Hudbay Chisel North mine. Chisel North infrastructures include a mined out open pit used for waste rock disposal, fresh (process) water sources, pumps and waterlines, 4160V and 550V power, mine discharge water lines, a 2,500 gpm water treatment plant with retention areas, and mine buildings including offices and a change house. These facilities are used for geological core processing, surface mobile equipment shop, project offices, and the crushing of Lalor ore. The Chisel site is also the location of two electrical transformers 115 kV to 25 kV that feed Lalor mine.

The infrastructure on site at Lalor includes, the main office change house, headframe, hoistroom, down cast fans, exhaust fans, main pump station, potable water treatment plant, sewage treatment plant and several other smaller buildings for purposes such as health and safety and additional change house space.

The New Britannia mill is located approximately 16 km north of the Lalor mine and it is currently under care and maintenance. The mill infrastructure consists mainly of three buildings and houses the crusher and mill/gold circuits. There are two existing crusher buildings; primary and secondary, each approximately of 108m^2 and 136m^2 respectively. The primary building houses the jaw crusher and the secondary building houses the cone crusher and screen deck, each building with associated conveyors. The existing mill building is approximately 1,939m^2 and contains the grinding and gold circuits, elution, EW and refining. The outside equipment includes thickener and leach tanks.

5.5 PHYSIOGRAPHY

The Snow Lake mining camp is located in the Boreal Shield Ecozone, the largest ecozone in Canada, extending as a broad inverted arch from northern Saskatchewan east to Newfoundland. This area is surrounded by water bodies including the Snow, File, Woosey, Anderson and Wekusko Lakes, all located in the Churchill River Upland Ecoregion in the Wekusko Ecodistrict.

The dominant soils are well to excessively drained dystic brunisols that have developed on shallow, sandy and stony veneers of water-worked glacial till overlying bedrock. Significant areas consist of peat-filled depressions with very poorly drained typic and terric fibrisolic and mesisolic organic soils overlying loamy to clayey glaciolacustrine sediments.

Topography shows gentle relief that rarely exceeds 10 m at approximately 300 mASL, with depressional lowlands.

6. HISTORY

6.1 LALOR & THE CHISEL BASIN AREA

Exploration in the Lalor-Chisel area began in the 1950s. The Chisel Basin area hosted four previously producing mines; Chisel Lake, Chisel Open Pit, Photo Lake and Chisel North. All four mines had very similar lithological and mineralogical features. This basin is also the host of the Lalor deposit.

In early 2007, drill hole DUB168 was drilled almost vertically to test a 2003 surveyed Crone Geophysics deep penetrating pulse electromagnetic anomaly and intersected 45 m of mineralization starting at a downhole depth of 782 m. Drilling at Lalor has been continuous since the discovery of mineralization on the property and as of the date of this report, totals in excess of 500,000 drilled metres.

Lalor commenced initial ore production from the ventilation shaft in August 2012 and achieved commercial production from the main shaft in the third quarter of 2014. Table 6-1 summarizes the actual ore production by year at Lalor from 2012 to 2018.

TABLE 6-1: LALOR MINE PRODUCTION

	Tonnes	%Zn	%Cu	g/t Au	g/t Ag
2012	72,800	11.83	0.63	1.67	19.3
2013	400,600	9.44	0.84	1.21	19.4
2014	551,900	8.52	0.88	2.29	23.8
2015	934,300	8.18	0.71	2.53	21.4
2016	1,086,400	7.01	0.62	2.24	21.6
2017	1,293,400	7.73	0.68	1.93	23.2
2018	1,260,200	6.25	0.74	2.19	25.4
Total	**5,599,600**	**7.59**	**0.72**	**2.13**	**22.8**

6.2 NEW BRITANNIA AND SNOW LAKE DEPOSITS

Gold was first discovered in 1914 approximately 20 km to the southeast of Snow Lake on the eastern shores of Wekusko Lake, also known as Herb Lake. In 1917, the Moose Horn-Ballast claims produced the first gold in Manitoba. First significant industrial gold production came from the Laguna mine which was in production intermittently until 1939 and produced 58,962 oz of gold.

Mine construction at the New Britannia site first started in 1945. In March 1949, the mine was opened as Nor-Acme mine. Production continued until 1958. 4.9Mt were mined at an average grade of 4.4g/t and the the Nor-Acme mill recovered approximately 610,000 oz of gold during this production period.

Between 1958 and the early 1990s, various companies worked on the property with the intention to bring the mine back into production, however, these attempts were unsuccessful.

TVX and High River formed a joint venture in 1994 to reopen the mine. TVX became the operator of the mine in late 1994 and ore from the secondary mining areas (No. 3 Zone and Birch pit) located on mineral lease ML-323 was used to bring the mine and mill into production prior to the completion of the rehabilitation and construction of the main shaft.

The New Britannia mill poured the first gold bullion in November 1995 which was obtained from the gold produced at No. 3 Zone. When production commenced in 1995, the name of the Nor-Acme Mine was changed to the New Britannia Mine. Production ceased at No. 3 Zone in May 1996 and production from the Birch pit was limited to the spring and summer of 1996. Full production from the main shaft was achieved in August 1996. Through various transactions, Kinross became the operator of the New Britannia mine-mill complex. Production mining ceased at the end of September 2004 and the mill was put on care and maintenance in 2005 due to a low gold price environment after producing 1.6 million oz of gold.

6.3 WIM DEPOSIT

Initial exploration on the property was comprised of airborne electromagnetic (EM) and radiometric surveys carried out by HBED in 1961, which identified several geophysical conductors. An extensive program of staking, line cutting, ground geophysical surveying and diamond drilling over high priority anomalies immediately followed.

VMS mineralization was discovered on the Wim deposit in January 1962 by diamond drilling of a ground EM anomaly delineated during the ground follow-up of an airborne EM survey. Intense diamond drilling during 1962 to 1968, followed by sporadic diamond drilling until 1991, outlined a Cu rich sulphide lens at the Wim deposit.

6.4 PEN II

The first surface geophysics completed at Pen II was a Turam survey done in 1973. Early drilling in 1970’s focused on this geophysical Turam survey, without significant results. Two airborne surveys were subsequently completed in 1984 and 1999 and several holes were drilled and surveyed by Hudbay in the 1980’s and then by Callinan in 1999-2000. Drilling in the mid 1980's focused on a geophysical anomaly near the claim boundary of Granges Inc., which led to the discovery of the Pen II Zone.

7. GEOLOGICAL SETTING & MINERALIZATION

7.1 REGIONAL GEOLOGY

The properties of interest for this Technical Report are all within the Paleoproterozoic Flin Flon Greenstone Belt (Figure 7-1) and are overlain by a thin veneer of Pleistocene glacial/fluvial sediments. Located within the Trans-Hudson Orogen, the Flin Flon Greenstone Belt consists of a variety of distinct 1.92 to 1.87 Ga (billion years) tectonostratigraphic assemblages. These include juvenile arc, back-arc, ocean-floor, ocean-island and evolved volcanic arc assemblages that were amalgamated to form the Amisk accretionary collage prior to the emplacement of intermediate to granitoid plutons and subsequent deformation (Syme et al., 1998). The volcanic assemblages consist of mafic to felsic volcanic rocks with intercalated volcanogenic sedimentary rocks.

At the eastern end of the Flin Flon Greenstone Belt, the Snow Lake area includes a lithologically and structurally diverse sequence of deformed and metamorphosed volcanic, sedimentary and intrusive rocks. This sequence is dominated by 1.84–1.81 Ga fold-thrust–style tectonics and by a southwest-verging allochthon of volcanic rocks, the Snow Lake arc assemblage.

The Snow Lake arc assemblage that hosts the producing and past-producing mines in the Snow Lake area is a 20 km wide by 6 km thick section that records a temporal evolution in geodynamic setting from 'primitive arc' (Anderson sequence to the south) to 'mature arc' (Chisel sequence) to 'arc-rift' (Snow Creek sequence to the northeast, Bailes and Galley, 2007). The Anderson and Chisel sequences represent the geological units of interest (Figure 7-2).

The Anderson sequence is a bimodal primitive arc sequence dominated by basaltic andesite, which contains at least three rhyolite flow centres: Daly, Sneath and Anderson-Stall. The mafic and felsic volcanic rocks of the Anderson sequence are intruded by the Sneath Lake subvolcanic intrusive complex (Figure 7-2). The Anderson-Stall and Daly rhyolites occur directly above the Sneath Lake intrusion near its two ends.

The transition from the Anderson sequence to the Chisel sequence ranges from subaqueous mafic flows to a succession of units composed mostly of heterolithic and volcaniclastic detritus. Chisel volcanic rocks are typically thinner, discontinuous, display rapid lateral facies variations and include a large volume of volcaniclastic deposits suggesting that there was considerable topographic relief as a result of intra-arc extension, rifting and caldera development. The Chisel sequence has been divided into lower and upper subdivisions separated by a thrust fault (Bailes et al., 2009).

The Chisel sequence has been intruded by a variety of intrusive rocks from thin crosscutting dikes (<1 m) to large plutons. Most of the intrusions are synvolcanic in nature, while the others are extrusive equivalents.

FIGURE 7-1: GEOLOGY OF MANITOBA

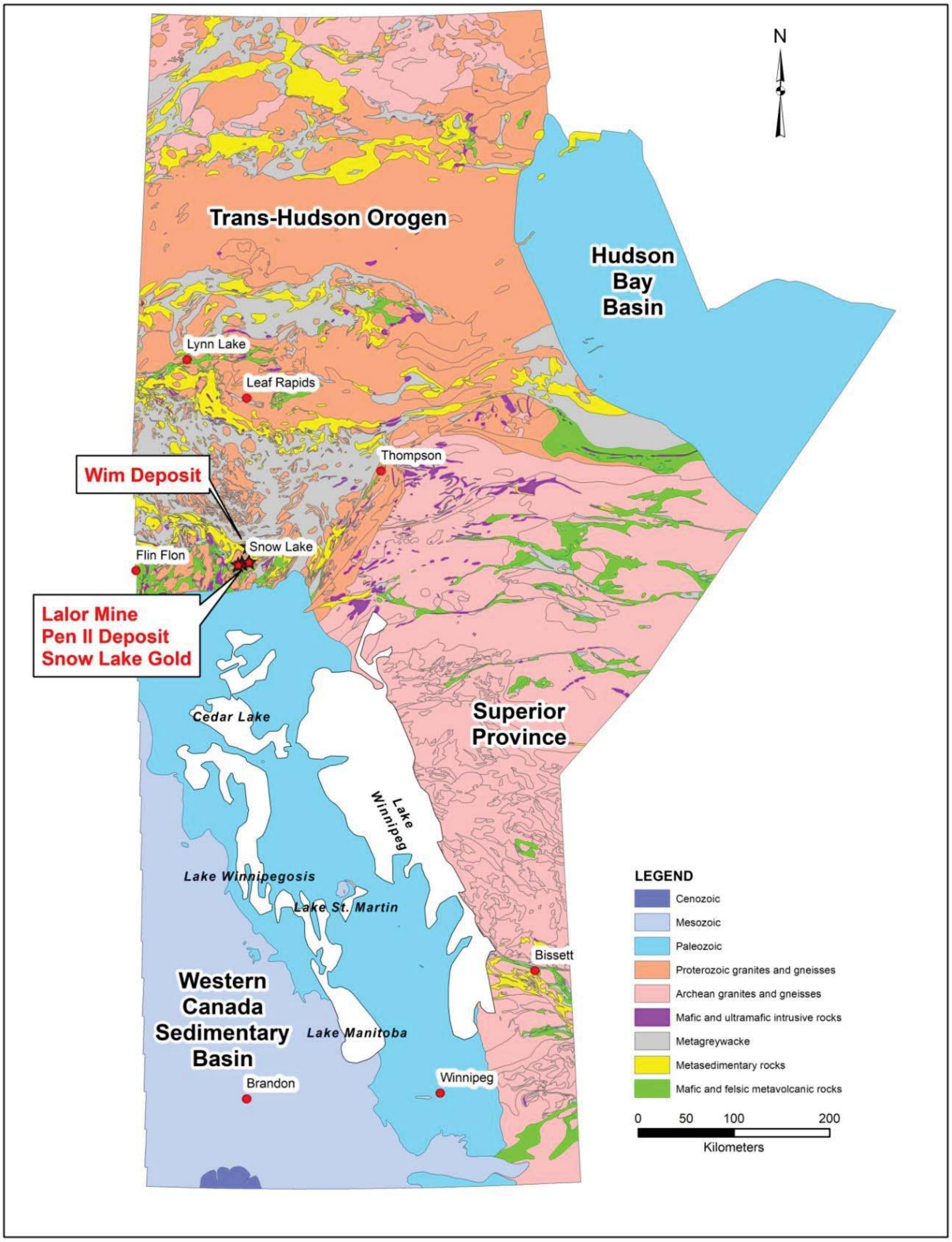
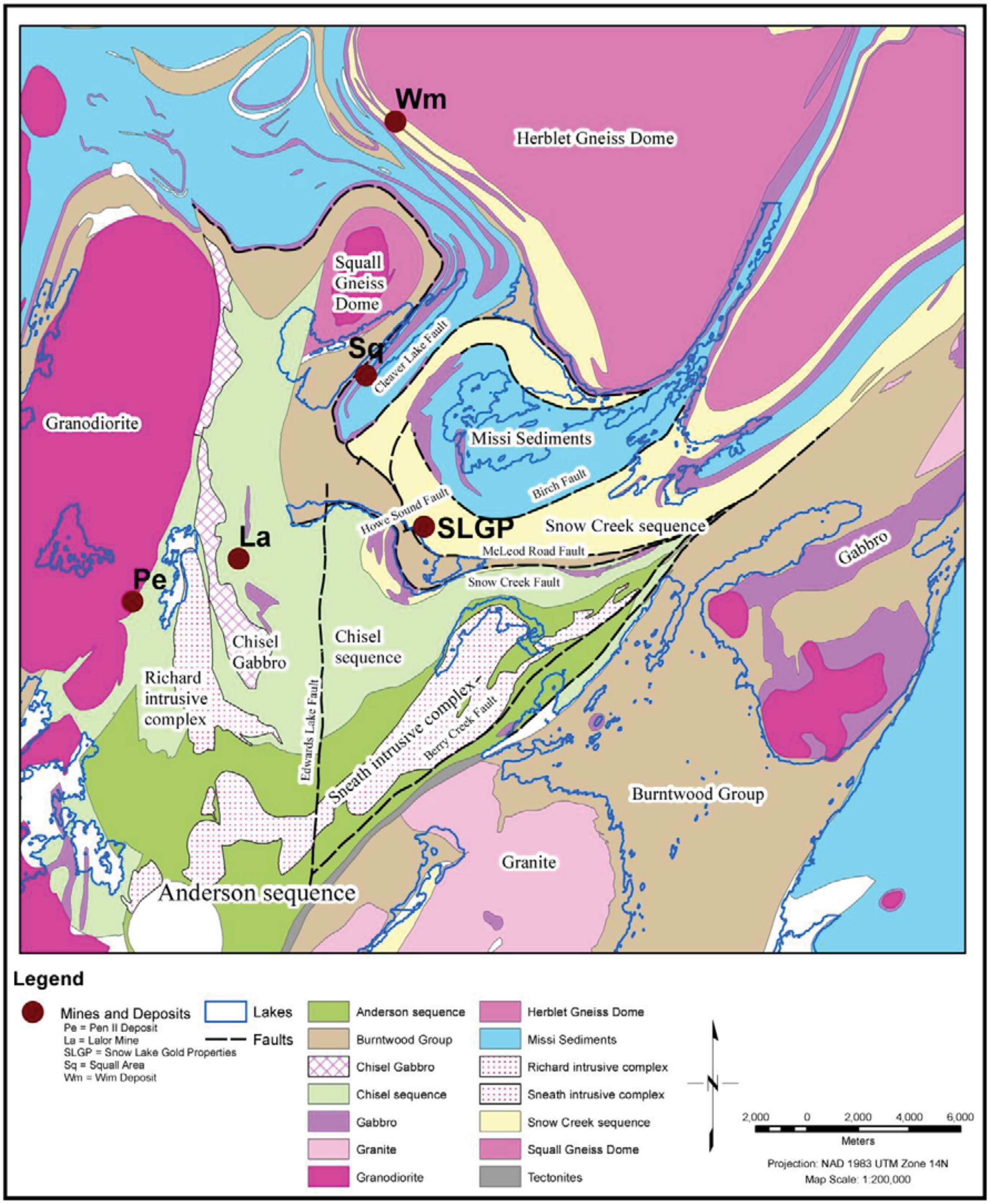


FIGURE 7-2: SNOW LAKE AREA

(After Bailes and Galley, 2011)



VMS Deposits

The volcanogenic massive sulphide (VMS) deposits located near the town of Snow Lake have been subdivided into two different groups: Cu-Zn-rich (Cu-Zn, Cu-Zn-Au) and Zn-Cu-rich (Zn-Pb-Cu-Ag) types. The Cu-Zn-rich deposits mainly occur in the Anderson sequence and the Zn-Cu-rich deposits occur in the Chisel sequence.

Cu-Zn-rich deposits and occurrences in the Anderson sequence are confined almost entirely to within the Anderson-Stall and Daly rhyolite centres. All the developed deposits are within the larger Anderson-Stall rhyolite. The Sneath Lake synvolcanic tonalite in the Anderson Lake area shows a steady increase in both intensity of alteration of supracrustal rocks and of the Cu/Zn ratio (Cu/Cu+Zn) contained in the VMS deposits as the distance to the pluton decreases. The variation of alteration and metal Cu/Zn ratio is interpreted to be the result from higher temperature for pluton-proximal portions of the regional synvolcanic hydrothermal system (Bailes and Galley, 1996).

The Photo Lake Cu-Zn-Au deposit occurs with the upper Chisel sequence and is located within a thick unit of rhyolite (Bailes and Simms, 1994). The Photo Lake deposit is thought to occur in the hanging wall of the structural panel to the Chisel-Lalor VMS system and is part of a separate and unrelated VMS forming event (Bailes et al., 2009).

There are six Zn-rich VMS deposits located within the mature arc Chisel sequence: Lalor, Chisel, Chisel North, Pen II and Ghost. The six deposits generally overlie the Powderhouse dacite and local discrete rhyolite flow complexes. They have been subjected to polyphase deformation.

The Wim deposit, located north of Snow Lake, consists of one large copper-gold (zinc and silver) VMS lens occurring in a highly prospective early Proterozoic island-arc assemblage of felsic and mafic volcanic rocks that stretches for an exposed length of 250 km east-west and 75 km north-south, on the western flank of the Herblet gneiss dome (Figure 7-2).

Snow Lake Gold Deposits

The Amisk mafic and felsic volcanic hosted deposits lie to the South of the Missi group hosted deposits of Squall Lake. The McLeod Road Thrust (MRT) is one of the main structural features of the area. The deposits are bound to the west by the MRT which is responsible for the emplacement of ca. 1.9-1.89 Ga volcanic and volcaniclastic rocks of the Snow Lake arc assemblage (Bailes and Galley, 2007) on top of the ca. 1.85 Ga Burntwood group metasedimentary rocks (Figure 7-2).

The southern part of the property comprises a moderately north-dipping and dominantly north facing bimodal sequence of coarsely re-crystallized and deformed mafic and felsic volcanic and volcaniclastic, epiclastic rocks and gabbro. The Birch Lake Thrust fault to the east was emplaced ca. 1845 Ma and thrust the Missi Group metasediments above the volcanic and volcaniclastic rocks.

The southern volcanic sequence is intruded by pre and syn ultramafic and mafic intrusions, and syn to post-tectonic granitoids. The metamorphic assemblages are of lower to mid-amphibolite facies. Imbrication of the sequence bound by the MRT and BLT occurred during south-directed transport and resulted in the formation of a strong regional northeast plunging stretching lineation, a regional foliation and the Nor-Acme anticline. These structures were subsequently folded around the northeast-trending Threehouse synform. (Rubingh et al. 2013)

Mineralization of the lode-gold vein-type deposits hosted in the Amisk group mafic and felsics volcanic rocks are structurally controlled and associated with shear zones, faults, fold hinges and axial planes that host simple to complex vein systems. The mineralization is associated with lithological contacts of contrasting properties in the sequence of interlayered volcanic and volcaniclastic rocks. Alteration in the mineralized zones consists of quartz-carbonate-mica, with arsenopyrite as the primary sulphide accompanying the gold.

The Squall Lake area (4 km northwest of the Town of Snow Lake) is comprised of several mineralized zones that are hosted in Missi group rocks which are the product of fluvial and alluvial sedimentation. In

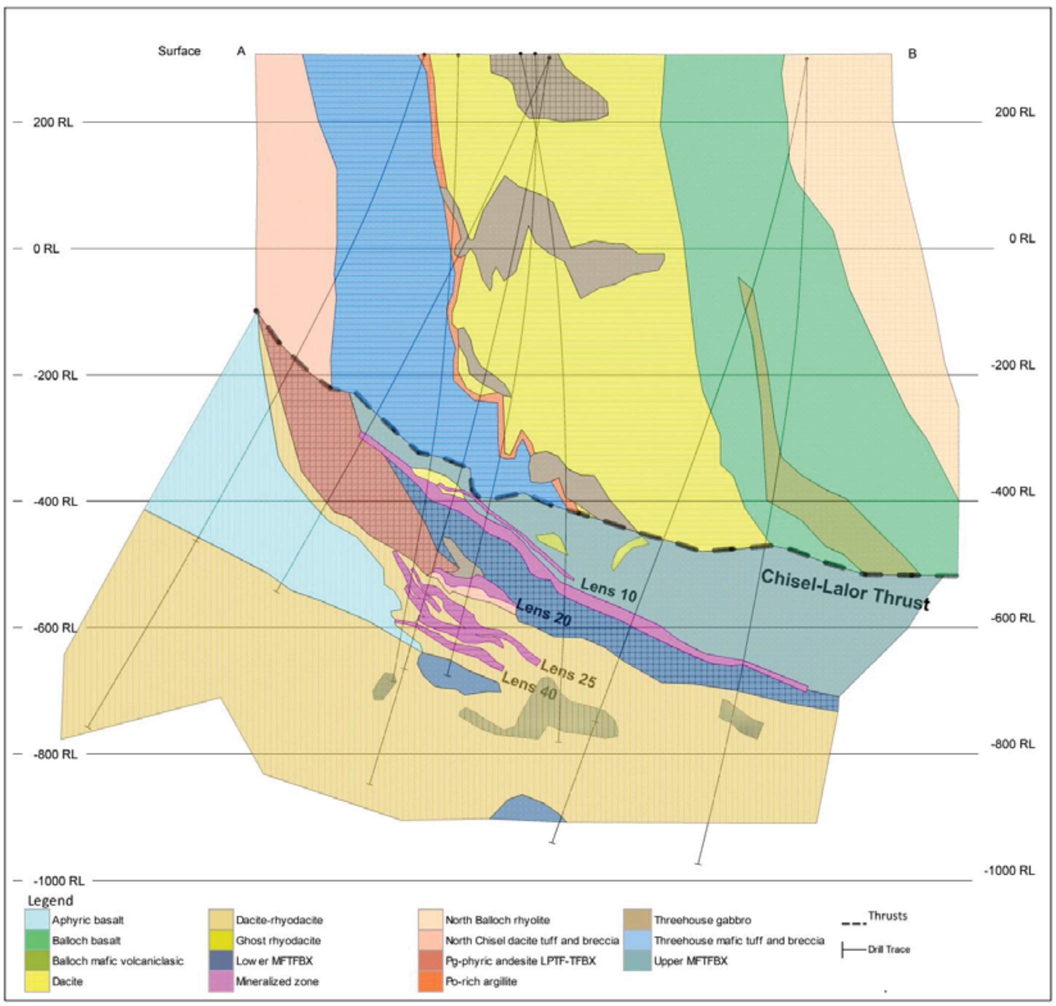
Snow Lake, the Missi Sequence is composed of cross bedded arenites and arkose. The Missi group is commonly intruded by discontinuous gabbroic sills and dykes that are medium to coarse grained. The Missi group arenites were thrusted over the garnet staurolite schist of the Burntwood group (Beilhartz, 2006). The lowermost sill, a 20 - 40 metre thick amphibolitic unit, is spatially associated with the known gold mineralization of the Squall lake area. This epigenetic gold mineralization is believed to be associated with structures generated by over-thrusting related to the McLeod Road Thrust Fault.

7.2 PROPERTY GEOLOGY

Lalor

The Lalor deposit is similar to other massive sulphide bodies in the Chisel sequence (Chisel Lake, Ghost Lake, Chisel North, and Photo Lake), and lies along the same stratigraphic horizon as the Chisel Lake and Chisel North deposits. It is interpreted that the top of the zone is near a decollement contact with the overturned hanging wall rocks, the Chisel-Lalor Thrust (Figure 7-3). The most common dyke intrusion throughout these rocks is a fine-grained feldspar-phyric gabbro to diorite.

FIGURE 7-3: LALOR MINE CROSS SECTION WITH DRILL HOLE TRACES



The extensive hydrothermal alteration and metamorphic recrystallization of the footwall rocks has produced some exotic aluminous mineral assemblages. These assemblages include chlorite and sericite dominant schists and cordierite+anthophyllite gneisses. Other minerals indicative of hydrothermal alteration that occur extensively throughout these rock assemblages include quartz, feldspar, kyanite, biotite, garnet, staurolite, hornblende, and carbonate. Clinopyroxene, gahnite and anhydrite also occur locally. These assemblages are typical of metamorphosed footwall hydrothermal alteration commonly associated with VMS deposits and are similar to that at the other massive sulphide deposits in the Chisel Lake area.

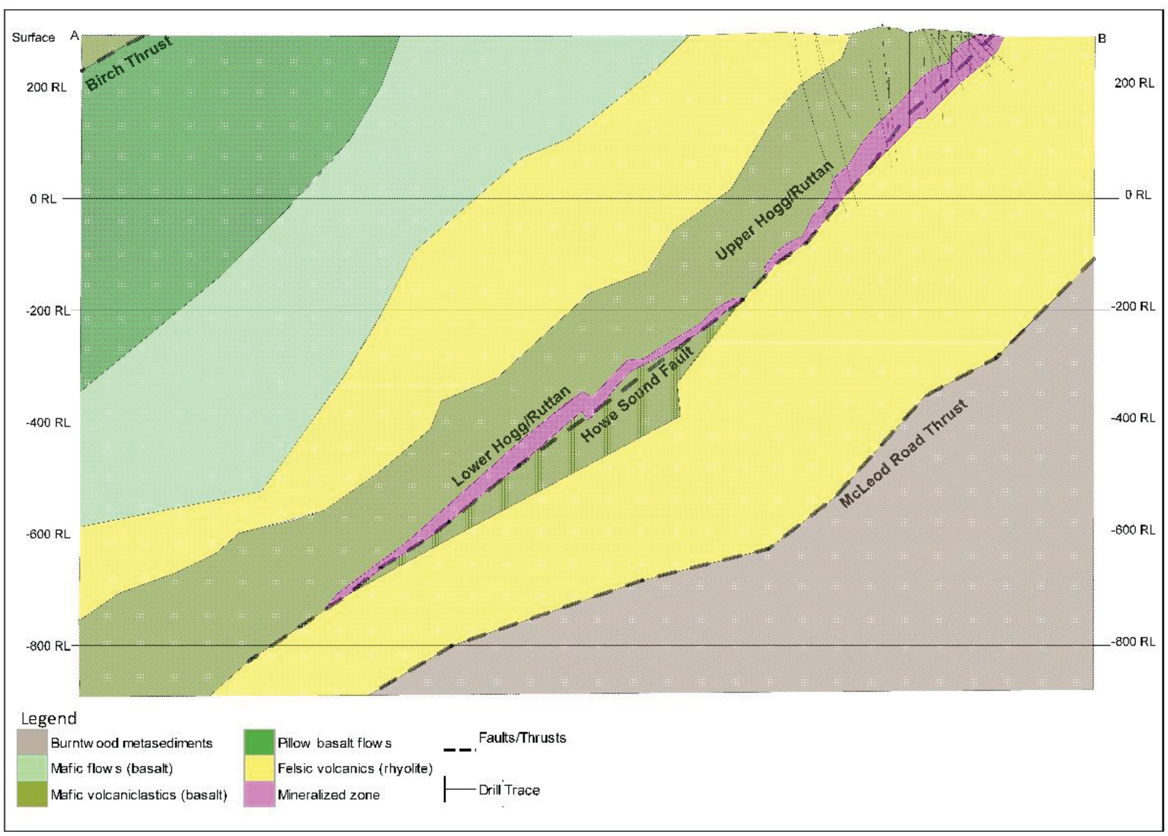
Snow Lake Gold Properties

The Snow Lake gold properties consists of the New Britannia Mine, No. 3 Zone, Birch Zone, Boundary Zone and Squall Lake deposits including the K1, K5, K7, K10, K11, Margaret (North and South), Margaret Extension, South, F1, Moon Gertie Zones, Bay and North Zones.

The New Britannia mine is a classical lode gold, quartz-carbonate vein type deposit. The mineralization is associated with shear zones, faults, and simple to complex vein systems. The auriferous zones (Toots, Dick, Ruttan and Hogg) lie along a curvilinear shear zone named the Howe Sound fault also referred to as the Nor-Acme fault (Figure 7-4). The main mineralized zones consist of sheared or "mylonitic" quartz-carbonate-mica alteration with arsenopyrite mineralization accompanying the gold emplaced in a simple intercalated sequence of altered felsic and mafic volcanics and pyroclastics of the Amisk Group. The host rocks are altered and include varying proportions of quartz and carbonate.

The gold bearing horizons in the No. 3 Zone are fault-hosted vein occurrences, consisting of a main shear vein and numerous extensional ladder veins. Gold mineralization is located within a 50 m wide fault zone at the folded contact between the coarse pyroxene mafic volcaniclastic rocks and the plagioclase-phyric pillowed basalt flows.

FIGURE 7-4: NEW BRITANNIA REPRESENTATIVE CROSS SECTION WITH DRILL HOLE TRACES



The Birch Zone is interpreted as hosted in a silicified shear zone and several parallel and sub parallel shears cut the mafic volcanic, mafic volcaniclastic and metasedimentary rocks (including some graphitic argillite) which were intruded by gabbroic dykes. Alteration and mineralization at the Birch Zone is similar to the No. 3 Zone.

The Boundary Zone is located 0.5 km south of the No. 3 Zone and approximately 1 km north of the New Britannia Mine, and in the hanging wall of the McLeod Road Thrust fault. The lode-type gold deposit is located along the Nor-Acme anticline at the contact between mafic and felsic volcanics.

The Squall Lake area is host to several mineralized zones in Missi group rocks. Epigenetic gold mineralization on the property is believed to be associated with structures generated by over-thrusting related to the McLeod Road Thrust Fault (Beilhartz, 2006). Two economically significant gold mineralized sets of structures exist on the property. The first type of structure was generated by an initial shearing event at the contact of differing lithologies due to contrasting competencies. This type of structure accounts for the Lower Silicified Zone at the Margaret Extension and Margaret South and occur as a series of sub-parallel en echelon mineralized lenses striking 050° to 070°. The second type of mineralized structure is a transcurrent type of shearing which developed large scale, sub-vertical shears across the property. (Beilhartz, D. 2006)

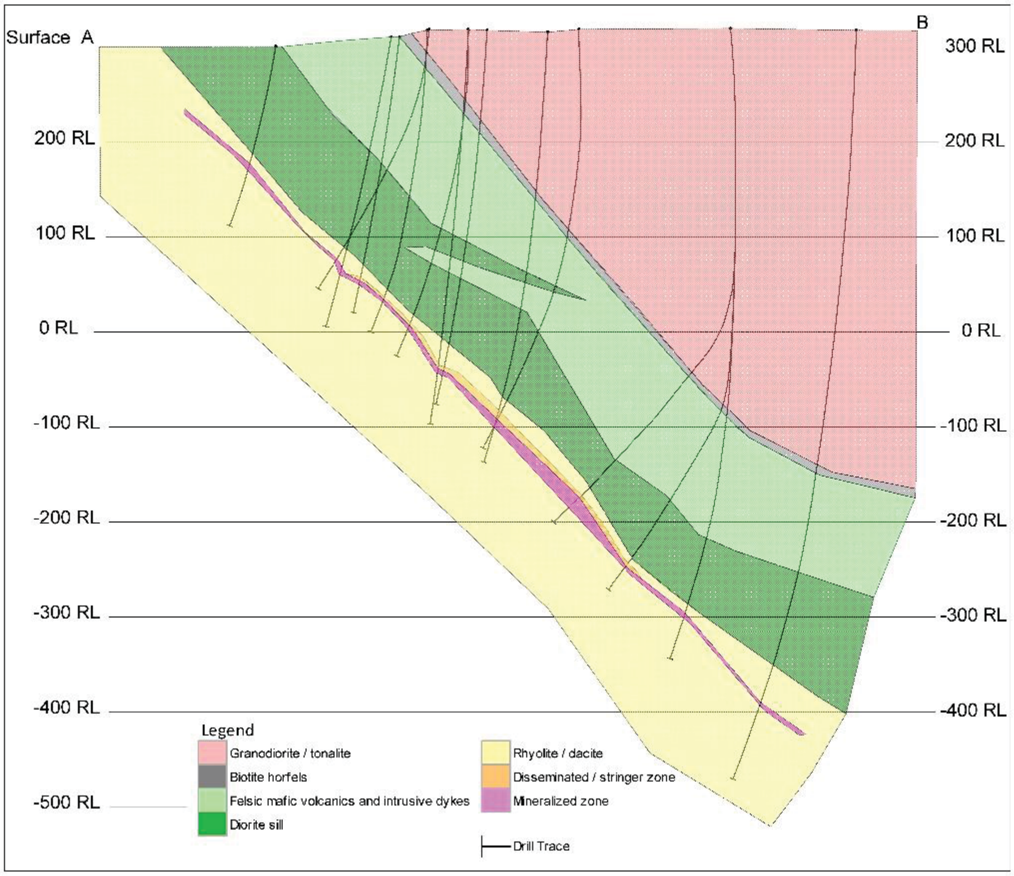
Pen II Deposits

The Pen II deposit is located in the Snow Lake arc assemblage (Figure 7-2). The volcanic stratigraphy that hosts the Pen II deposit is on the eastern margin of the younger Ham Lake intrusion. The stratigraphy strikes north - northeast and dips moderately to steeply west. The deposit has extensive granitic to gabbro dykes throughout the hanging wall and numerous felsic to mafic volcanic flows in the footwall of the deposit. The mineralization is generally hosted within felsic volcanic or volcaniclastics or at a contact between felsic and mafic volcanics. All the rocks have undergone lower to middle amphibolite facies metamorphism and primary volcanic features have been obliterated.

Wim Deposit

The Wim deposit is hosted within a northwest-striking, northeast-dipping package of mafic and felsic metavolcanic gneisses along the southern rim of the Herblet Lake Gneiss Dome Complex (Figure 7-5).

FIGURE 7-5: WIM DEPOSIT CROSS SECTION WITH DRILL HOLE TRACES



The meta-volcanic gneisses are interpreted to be derived from the early Proterozoic subaqueous and subaerial volcanics of the Amisk Group that is locally overlain by marine turbidites of the Burntwood Group and terrestrial sediments of the Missi Group, and intruded by various dykes. Tectonic foliations in the vicinity of the Wim deposit trend northwest and dip moderately to steeply towards the northeast.

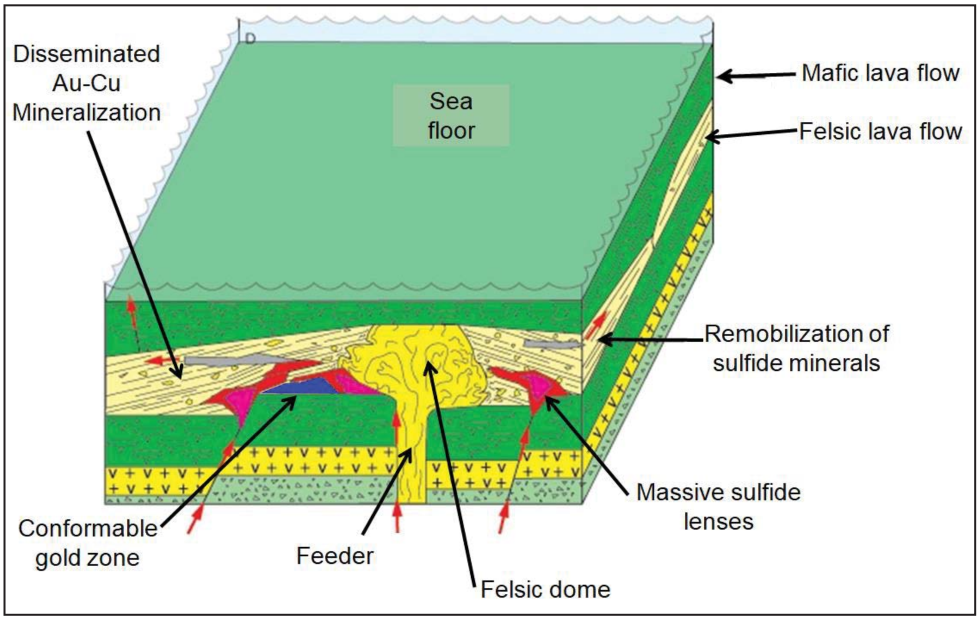
Several alteration types associated with the sulphide mineralization have been documented in outcrop and drill core. Other alteration types encountered in outcrop and drill core include medium to coarse-grained, cordierite-anthophyllite, sieve textured garnets with chalcopyrite inclusions, muscovite-sillimanite and garnet zones. There is a general coarsening of grain size with proximity to mineralization due to the metamorphic recrystallization of hydrothermally altered rock and the growth of strongly porphyroblastic minerals such as garnet, anthophyllite, cordierite and sillimanite. altered rhyolites occur to the northwest and southeast of the deposit along the geophysical trend that coincides with the Wim deposit mineralization.

8. DEPOSIT TYPES

8.1 VMS DEPOSITS

The deposits of interest in this Technical Report are classified as VMS (volcanogenic massive sulphides). VMS deposits are an important source of copper, zinc, lead, gold and silver (Cu, Zn, Pb. Au and Ag) and form at or near the seafloor where circulating hydrothermal fluids driven by magmatic heat that have quenched through mixing with bottom waters or porewaters in near-seafloor lithologies (Figure 8-1).

FIGURE 8-1: CONCEPTUAL MODEL OF A VOLCANOGENIC MASSIVE SULFIDE (VMS) DEPOSIT
(Modified After S.Lavoie, 2003)



Massive sulfide lenses vary widely in shape and size and may be podlike or sheetlike. They are generally stratiform and may occur as multiple lenses from small pods of less than a ton (which are commonly scattered through prospective terrains) to giant accumulations like the Rio Tinto mine (Spain), Kholodrina (Russia), Windy Craggy (Canada), Brunswick No. 12 (Canada) or Ducktown (United States). Volcanogenic massive sulfide deposits are actively forming in extensional settings on the seafloor, especially mid-ocean ridges, island arcs, and back-arc spreading basins.

The location of VMS deposits is often controlled by synvolcanic faults and fissures, which permit a focused discharge of hydrothermal fluids. As illustrated on Figure 8-1, a typical VMS deposit will include the massive mineralization located proximal to the active hydrothermal vent, footwall stockwork mineralization, and distal products, which are typically thin but extensive. Footwall, and less commonly, hanging wall semiconformable alteration zones are produced by high temperature water-rock interactions (Franklin et al., 2005).

Massive sulphide ore in VMS deposits consists of usually more than 40 percent sulfides, dominated by pyrite, pyrrhotite, chalcopyrite, sphalerite, and galena. Non-sulfide gangue typically consists of quartz, barite, anhydrite, iron (Fe) oxides, chlorite, sericite, talc, and their metamorphosed equivalents. Sulphide ore composition may vary from Pb-Zn- to Cu-Zn-, or Pb-Cu-Zn-dominated, and some deposits are zoned vertically and laterally. VMS deposits have stringer or feeder zones beneath the massive zone that consist of crosscutting veins and veinlets of sulfides in a matrix of pervasively altered host rock.

Lalor Mine

The depositional environment for the mineralization at Lalor is similar to present and past producing base metal deposits in felsic to mafic volcanic and volcaniclastic rocks in the Snow Lake mining camp. The deposit appears to have an extensive associated regional hydrothermal alteration.

The mineralized envelopes at the Lalor mine are shallow dipping (Figure 7-3), with zinc mineralization defined to date beginning at approximately 600 m from surface and extending to a depth of approximately 1,100 m. The mineralization trends about N320° to N340° azimuth and dips between 30° and 45° to the north. It has a lateral extent of about 1,400 m in the north-south direction and 800 m in the east-west direction.

Sulphide mineralization is dominated by pyrite and sphalerite. In the near solid (semi-massive) to solid (massive) sulphide sections, pyrite occurs as fine to coarse grained crystals ranging one to six millimetres and averages two to three millimetres in size. Sphalerite occurs interstitial to the pyrite. A crude bedding or lamination is locally discernable between these two sulphide minerals. Near solid coarse grained sphalerite zones occur locally as bands or boudins that strongly suggest that remobilization took place during metamorphism.

Disseminated blebs and stringers of pyrrhotite and chalcopyrite occur locally within the massive sulphides, adjacent to and generally in the footwall of the massive sulphides. The hydrothermally altered rocks in the footwall commonly contain some very low concentrations of sulphide minerals. Some sections of massive pyrrhotite occur, but these tend to give way to pyrite-sphalerite-dominant zones.

The top two lenses of the stacked base metal zones (referred to as Zone 10 and 11) have higher grade zinc and iron content. Lenses located lower in the stratigraphy and coded as Zones 20, 30, 31, 32, 40 and 42 have moderate to high zinc grades hosted in near solid sulphides containing higher grade gold and locally appreciable amounts of copper.

The footwall gold mineralization is typical of any VMS footwall feeder zone with copper-rich, disseminated and vein style mineralization overlain by a massive, zinc-rich lens. The fact that the footwall zone is strongly enriched in gold suggests a copper-gold association which is comparable to other gold enriched VMS camps and deposits (Mercier-Langevin, 2009). Some of the footwall zones tend to be associated with silicification and the presence of gahnite. These zones are often characterized by copper-gold association and are currently interpreted as being associated with higher temperature fluids below a zone of lower temperature base-metal accumulations.

Gold and silver enriched zones occur near the margins of the zinc rich sulphide lenses and as lenses in local silicified alteration. Remobilization is illustrated in some of the gold-rich zones by late veining that is more or less restricted to the massive lenses.

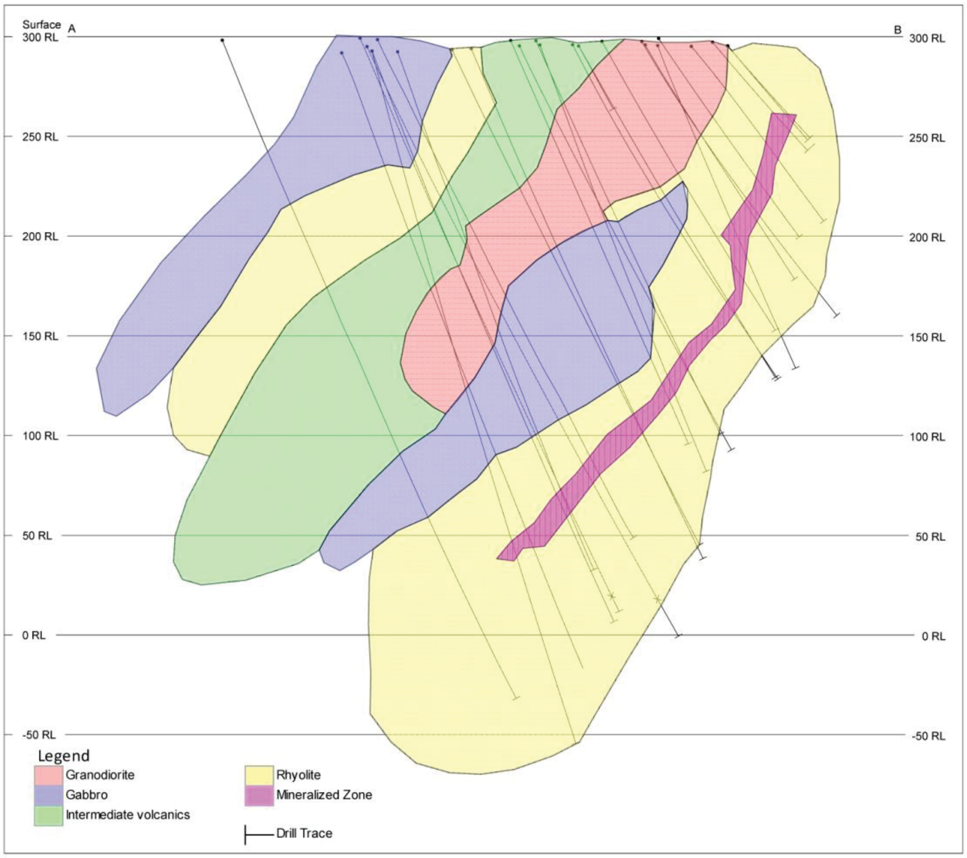
Some of the footwall zones tend to be associated with silicification and the presence of gahnite. These zones are often characterized by copper-gold association and are currently interpreted as being associated with higher temperature fluids below a zone of lower temperature base-metal accumulations.

General observations of the known gold zones indicate areas which are coarse-grained and porphyroblastic in nature are gold poor, while fine grain siliceous (± veins ± sulphide traces) and strained looking stratigraphy tend to be gold rich. However, the intensity and style of alteration can vary strongly over short distances and may suggest that the alteration was forming discordant stockwork like zones that are now strongly transposed in the main foliation (Mercier-Langevin, 2009).

Pen II Deposit

The Pen II deposit comprises a stratabound, semi-massive to massive sulphide lens with an adjacent stringer/disseminated sulphide zone (Figure 8-2).

FIGURE 8-2: TYPICAL CROSS-SECTION THROUGH THE PEN II DEPOSIT



Mineralization is characterized by disseminated to massive, recrystallized and medium to coarse-grained sphalerite, pyrite, pyrrhotite and minor chalcopyrite. Based on the drilling completed by Hudbay and Callinan Mines Ltd. the mineralization extends from surface to 500 m below surface. The current strike length of the deposit is 400 m with an average thickness of 4 m. The deposit is conformable to stratigraphy, trends to the northeast at a N40º azimuth, a 45-65º dip towards the northwest.

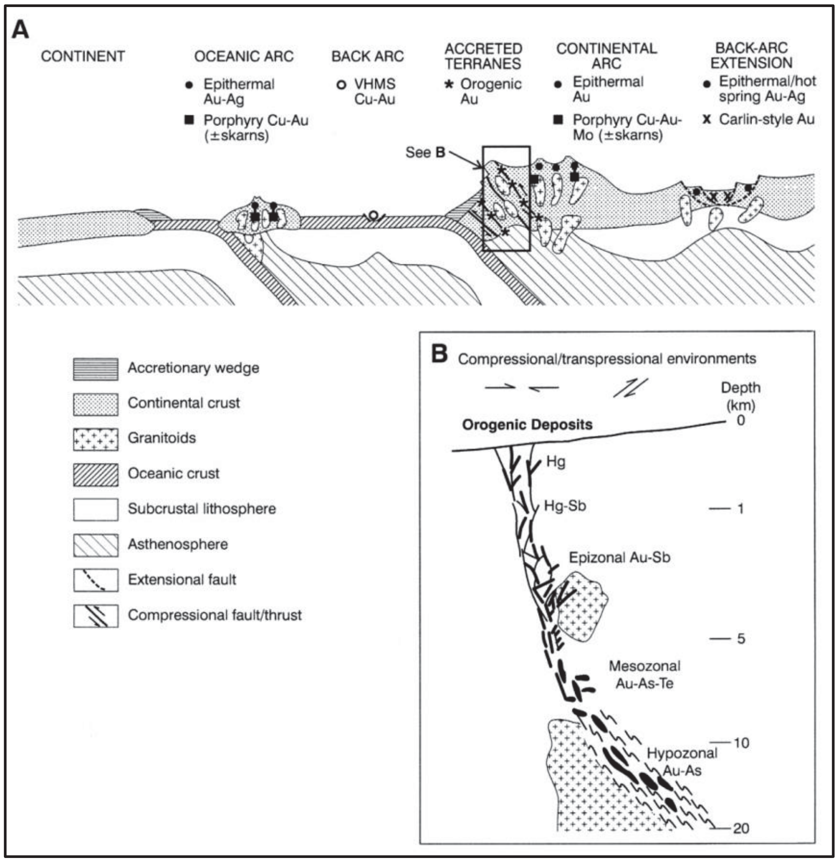
Wim Deposit

The Wim deposit comprises a stratabound, semi-massive to massive sulphide lens with an adjacent stringer/disseminated sulphide zone (Figure 7-5). Mineralization is characterized by disseminated to massive, recrystallized and medium to coarse grained pyrite, pyrrhotite, chalcopyrite and minor sphalerite. Based on the drilling completed by Murgor, the VMS mineralization extends from surface to 720 m below surface. The current strike length of the deposit is 725 m with an average thickness of 10 m. The Wim deposit is conformable to stratigraphy, trends to the northwest at a N310º azimuth, a 40-45º dip towards the northeast and a plunge of 40º to the north. In addition, disseminated and stringer sulphides occur in the stratigraphic footwall of the Wim sulphide lens and may represent the feeder zone of the stratabound sulphide lens.

8.2 SNOW LAKE GOLD PROPERTIES

The Snow Lake Gold Properties (NBM, No. 3 Zone, Boundary Zone, Birch Zone and Squall Lake Area) are considered to belong to the quartz-carbonate vein gold subtype of orogenic lode gold deposits. This subtype of gold deposits consists of simple to complex quartz carbonate vein systems associated with brittle-ductile rock behaviour, corresponding to intermediate depths within the crust, and compressive tectonic settings (Figure 8-3). Deposits of this type have been commonly referred to as mesothermal gold quartz vein deposits, but they encompass both mesothermal and hypothermal classes as initially defined by Lindgren (1933).

FIGURE 8-3: SCHEMATIC DIAGRAM SHOWING THE SETTING AND NATURE OF OROGENIC LODE-GOLD DEPOSITS



Note: The plate tectonic environments of formation of orogenic lode-gold deposits and other gold-rich deposit styles, adapted from Goldfarb et al. (2005) and Groves et al. (2000). (B) Depth profile of orogenic lode-gold deposits, adapted from Groves et al. (1998)

At the regional scale, the quartz-carbonate vein gold deposits occur in two contrasting geological environments: deformed clastic sedimentary terranes and deformed volcano-plutonic terranes containing diverse volcanic assemblages of island-arc and oceanic affinities. Despite lithological and structural differences, these two types of environment share the following characteristics: greenschist to locally lower amphibolite metamorphic facies, brittle-ductile nature of deformation, and geological structures recording compressional to transpressional tectonic settings. Quartz-carbonate vein gold deposits in these environments tend to occur in clusters, or districts. Both types of environments are present in several districts, in which they are separated by major fault zones. However, in such cases auriferous quartz-carbonate veins preferentially occur in the volcano-plutonic domains.

At the New Britannia mine the mineralization is associated with shear zones, faults, and simple to complex vein systems. The auriferous zones (Toots, Dick, Ruttan and Hogg from west to east respectively) within the mine lie along a curvilinear shear (fault) zone named the Howe Sound fault also referred to as the Nor-Acme fault (Figure 7-2). One or more “slip planes” accompanied by a variety of altered mylonitic zones mark the location of the fault. The mylonitic zones (quartz-carbonate-mica) are predominantly less than 0.3 m thick but can thicken locally to between 3 m to 6 m. The gold bearing quartz-carbonate rocks are almost always situated next to or astride the fault. The main mineralized zone consists of quartz-carbonate alteration zones with arsenopyrite as the main sulphide accompanying the gold, emplaced in a simple intercalated sequence of altered felsic and mafic volcanics and pyroclastics. The contacts of the mineralized zones are often gradational; however, there have been sharp shear and fault contacts noted within some parts of the deposit¸ and a biotite alteration halo occurs within a few feet of the mineralized zone.

Mineralization in No. 3 Zone, Boundary and Birch Zones is hosted within sheared biotite-quartz and carbonate altered volcanic and volcaniclastic rocks that host quartz-albite-iron carbonate veins with fine to coarse gold, and is associated with acicular arsenopyrite within the veins, wallrocks and wallrock fragments (Galley et al., 1991). Observations of the mineralization from underground at the No. 3 Zone noted that the quartz veins varied in true width from 0-1.5m and that the mineralization was not confined to the veins and can occur in up to 2 m from the main vein system.

Ten zones of gold-arsenopyrite mineralization have been identified in the Squall Lake area through previous exploration programs from 1943 to 1987. These zones occur within a 40-60 m stratigraphic interval above the staurolite schist unit and are aligned over a 10 km strike length along the northwest flank of the McLeod Lake Synform. These zones have been named K1, K5, K7, K10, K11, Margaret (North and South), Margaret Extension, South, F1 and Moon Gertie Zones. Two other similar zones to the South are the Bay and North Zones. The gold-arsenopyrite mineralization is directly associated with tabular quartz veins that are rimmed by fine-grained arsenopyrite, sub-parallel to the enclosing stratigraphy, or as tightly folded with a near vertical attitude. (Beilhartz, 2006).

8.3 OTHER DEPOSITS

Far Resources Ltd. is exploring for lithium on the Zoro Lithium property that is located 20 kilometres east of the Town of Snow Lake. The lithium mineralization is associated with several pegmatite dykes that strike north to northwest and dip vertically (NI 43-101 Technical Report on Zoro Lithium Project). Rockcliff Metal Corp. is exploring for base metals and graphite on their Phenex property, which is adjacent to Pen II.

Wolfden Resources Corporation has a 100% ownership of the Rice Island Ni-Cu-Co deposit. The property is located 5 kilometres from Hudbay's Snow Lake Stall concentrator. Mineralization occurs at the basal contact of a northeast-striking, steeply plunging gabbroic intrusion and underlying sedimentary rocks. Ni-Cu-Co mineralization was initially explored by Inco Ltd in the 1950's. Recent drilling completed by Wolfden has confirmed the grade and nature of Ni-Cu-Co mineralization and was successful in discovering an underlying feeder zone containing high-grade nickel sulphide mineralization.

9. EXPLORATION

It is the QP's opinion, that sampling to date is both representative and adequate to develop a sound interpretation of the geology and the grade distribution for all the deposits covered by this Technical Report.

9.1 LALOR MINE

In 2003, a Crone Geophysics high power time-domain electromagnetic (EM) system experimental survey was conducted over the deepest portion (approximately 600 m vertical depth) of the Chisel North Mine. The survey was designed and interpreted by Hudbay and was conducted by Koop Geotechnical Services Inc. The survey provided conclusive evidence that the system could detect conductive bodies at depths greater than 500 m. It was decided to extend the survey coverage further down-dip and down-plunge of the known mineralized lenses. A double-wired transmitter loop measuring 2 km by 2 km was used to maximize the EM field strength. The survey results were interpreted using three-dimensional (3D) computer modeling. The model indicated a highly conductive, shallow-dipping zone at a vertical depth of 800 m at Lalor.

Since the last NI 43-101 Technical Report published in 2017, exploration drilling at Lalor has both focused on adding and converting inferred mineral resource estimates with a strong emphasis on confirming the continuity of the gold mineralization. Due to the low angle and stacking nature of the mineralization at Lalor, holes were extended beyond the gold target depths to explore the on-strike and plunge potential of known base metal lenses, which led to increases in mineral resource inventory. Approximately 30,000 m of exploration underground drilling has been conducted per year at Lalor since 2017.

Surface drilling since the previous disclosure has identified a new Cu-Au feeder type zone named Lens 17 which is deemed to be an analog to the now well-defined Lens 27 and to constitute the feeder zone of Lens 10. So far, Lens 17 is defined from 7 holes as illustrated on Figure 9-1 and described in Table 9-1 and Table 9-2. Hudbay has identified a suitable platform to continue infill drilling in Lens 17 from underground in 2019. The objective is to be in a position of reporting inferred mineral resource estimates for Lens 17 by the end of 2019. In addition, surface drilling also aims at testing the up-dip continuation of Lens 10. In parallel, Hudbay is undertaking an exploration program to find additional massive sulfide lenses at depths below current drilling coverage, i.e. from 1,000 m to 1,500 m depth. The first step will be to drill 3 deep holes from underground platform and conduct borehole EM surveys to test for the presence of significant sulfide occurrences in a possible repeat of the lower limb of the fold of the Lalor sequence (Figure 9-3). This program also has the potential to help identify additional Cu-Au feeder zones.

TABLE 9-1: SURFACE DRILLING RESULTS IN LENS 17

Hole ID	From (m)	To (m)	Intercept (m)	Depth (m)	Estimated true width (m)[1]	Cu (%)[2]	Au (g/t)[2]
193W01	1,041.2	1,046.5	5.4	1,028	4.1	1.1	2.8
267W01	1,120.8	1,127.2	6.3	1,098	4.5	2.7	11.3
273	1,211.8	1,215.8	4.0	1,202	2.9	1.9	1.2
283	1,242.7	1,249.0	6.3	1,240	4.2	7.8	5.9
283W02	1,270.8	1,276.3	5.5	1,263	4.1	7.8	2.5
296	1,227.5	1,233.0	5.5	1,184	4.2	5.2	5.6
296W01	1,220.5	1,228.3	7.8	1,175	6.1	3.7	5.4

1 based on the apparent orientation of the mineralization

2 uncut grades

TABLE 9-2: INTERCEPTS LOCATIONS

Hole ID	From			To			Azimuth at intercept	Dip at intercept	Core Size
	Easting	Northinig	Elevation	Easting	Northinig	Elevation			
193W01	427,051	6,081,272	4,273	427,051	6,081,270	4,268	185	-76	NQ
267W01	427,185	6,081,266	4,204	427,183	6,081,266	4,197	242	-79	NQ
273	427,163	6,081,570	4,101	427,162	6,081,570	4,098	206	-79	NQ
283	427,223	6,081,530	4,064	427,222	6,081,530	4,057	248	-83	NQ
283W02	427,263	6,081,461	4,040	427,263	6,081,460	4,035	186	-77	NQ
296	427,251	6,081,311	4,121	427,251	6,081,310	4,115	154	-76	NQ
296W01	427,243	6,081,301	4,130	427,244	6,081,299	4,123	163	-73	NQ

FIGURE 9-1: SURFACE DRILLING OF LENS 17

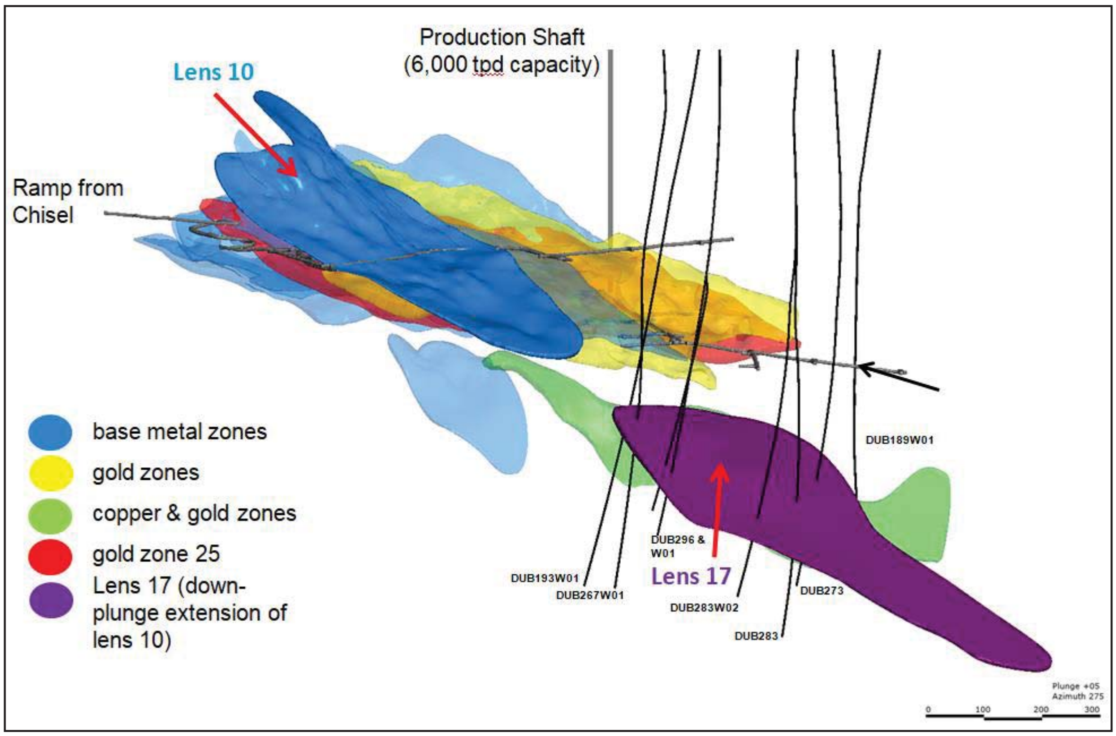


FIGURE 9-2: PROPOSED UNDERGROUND EXPLORATION PROGRAMS FOR LENS 17

(3D Solid of Lens 17 is Only Conceptual at This Stage for Exploration Purposes)

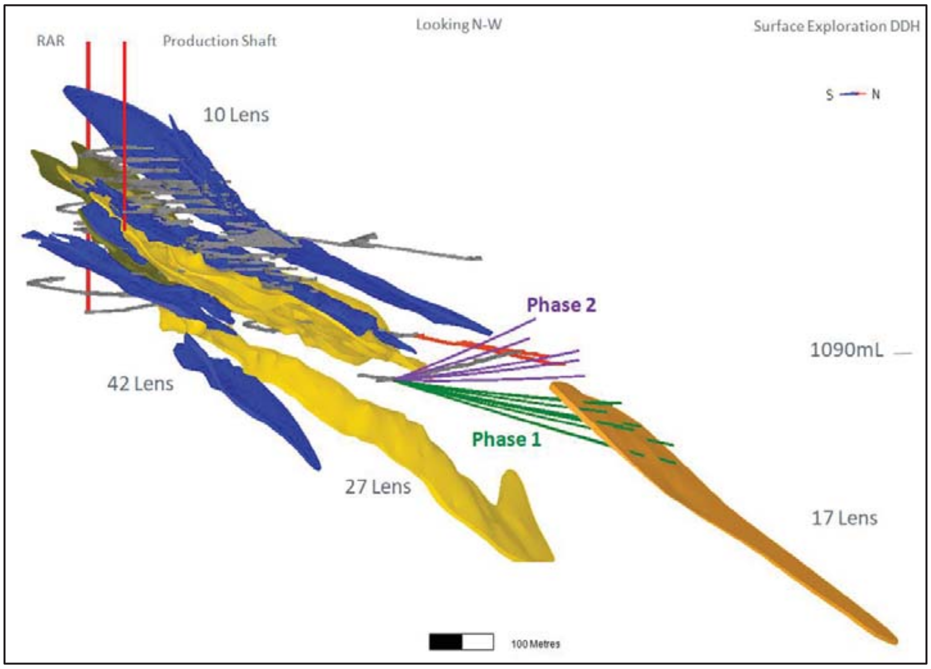
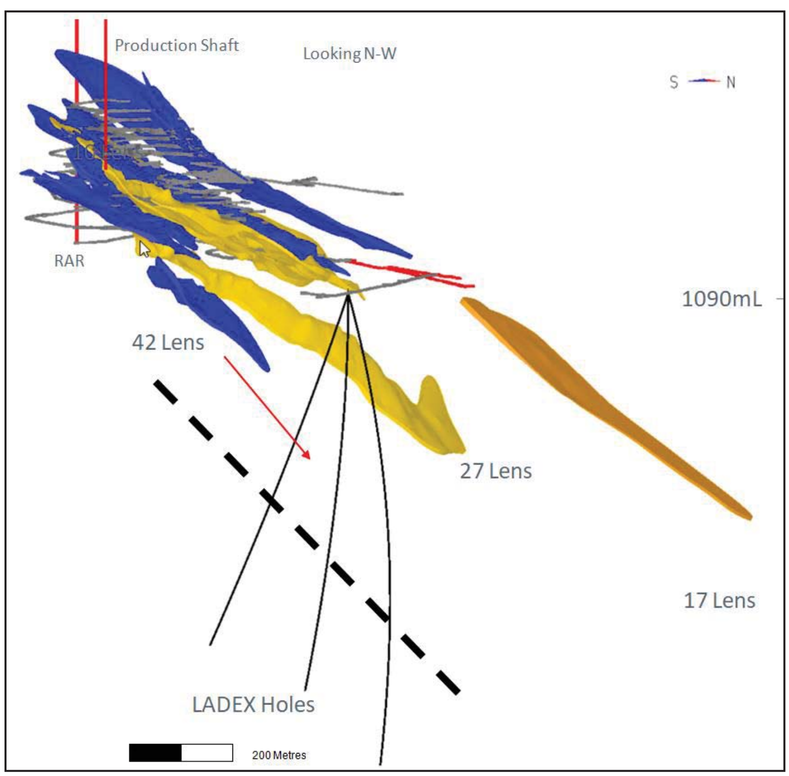


FIGURE 9-3: PROPOSED UNDERGROUND DEEP EXPLORATION PROGRAM



9.2 SNOW LAKE GOLD PROPERTIES

Exploration programs and exploration history on the Snow Lake Gold Properties (SLGP) are described in the previously disclosed NI 43-101 prepared for Alexis Minerals Corp. by Genivar, in 2010. A brief outline of the most recent exploration work on these properties is provided below.

In 2011, Bailes Geoscience was contracted by Alexis Minerals Corp. to perform a geochemical study of the lithological units in this area (McLeod Road-Birch Lake Thrust (MB)) sequence. The evaluation was conducted over 649 high-quality ICP-MS litho-geochemical samples collected from 35 exploration drill holes. The highlights of the results were:

- The McLeod Road-Birch Lake volcanic rocks are a thrust repetition of the Snow Lake arc assemblage to the south.
- Gabbro intrusions are geochemically identical to specific supracrustal units and therefore likely to be synvolcanic feeders for the rocks.
- Rhyolite flows in the MB are all very similar in geochemistry and indistinguishable from one another. They display arc tholeiite type FII rhyolite geochemistry.
- Alteration within the MB is very subtle compared to rocks associated with VMS. No gold-associated major element alteration trends were identified.

In 2018, Hudbay re-initiated a systematic exploration program for lode gold deposits in the Snow Lake area and completed an Induced Polarization (IP) survey and an unmanned aerial vehicle (UAV) magnetic survey over the northern part of the (SLGP) encompassing the Birch, No. 3 Zone and the Boundary deposit areas. Further south, two additional properties: The Tern Lake and Purple Sandy Beach prospects were also covered by the survey. The IP results were deemed successful and have been used to define drill targets that will be tested in 2019.

9.3 PEN II DEPOSIT

In 2018, 23 holes were drilled, totaling 6,374 m to upgrade the majority of the mineral resource estimates at the Pen II deposit to an indicated category. During this program, two new small high-grade satellite zones were discovered within the top 400 m. In 2019, the drilling at Pen II will focus on confirming the continuity of the mineralization in these two satellite zones, define the exact extent of the mineralization towards surface, and to collect geotechnical data and samples for metallurgical sampling.

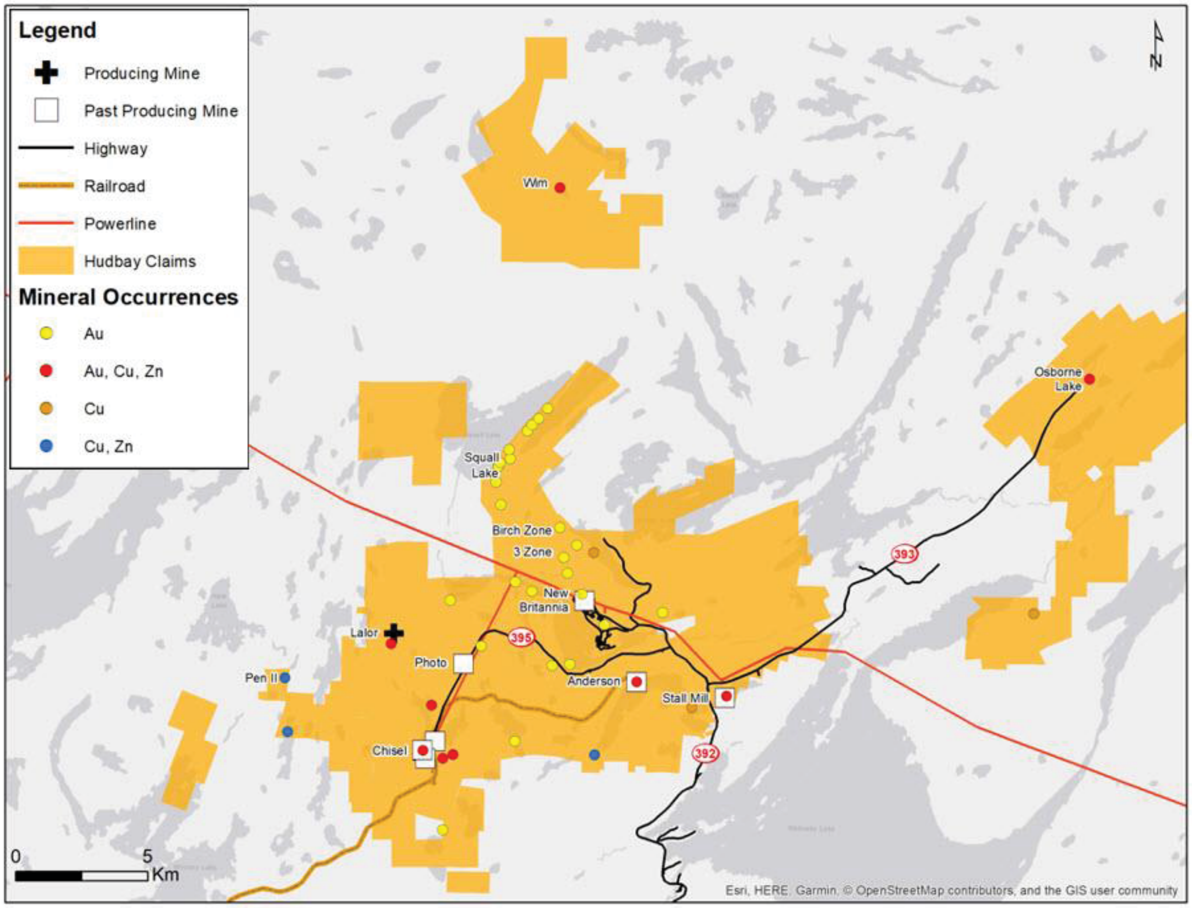
9.4 WIM DEPOSIT

Exploration programs and exploration history at Wim is described in the previously disclosed NI 43-101 prepared for Alexandria resources by Golder, in 2015. Since acquiring the property at the end of 2018, Hudbay has initiated a drilling program to collect geotechnical and metallurgical samples. The main objectives are to test the flowsheet of the future New Britannia mill and to confirm that the mineralization at Wim could attain processing recoveries in copper, silver and gold similar to the ones estimated for the Lalor copper-gold zones.

9.5 REGION EXPLORATION

Hudbay has an extensive land package in the Snow Lake belt of 17,517 Ha that provides significant additional upside for further gold and base metals exploration (Figure 9-4).

FIGURE 9-4 MINERAL OCCURRENCES IN THE SNOW LAKE CAMP



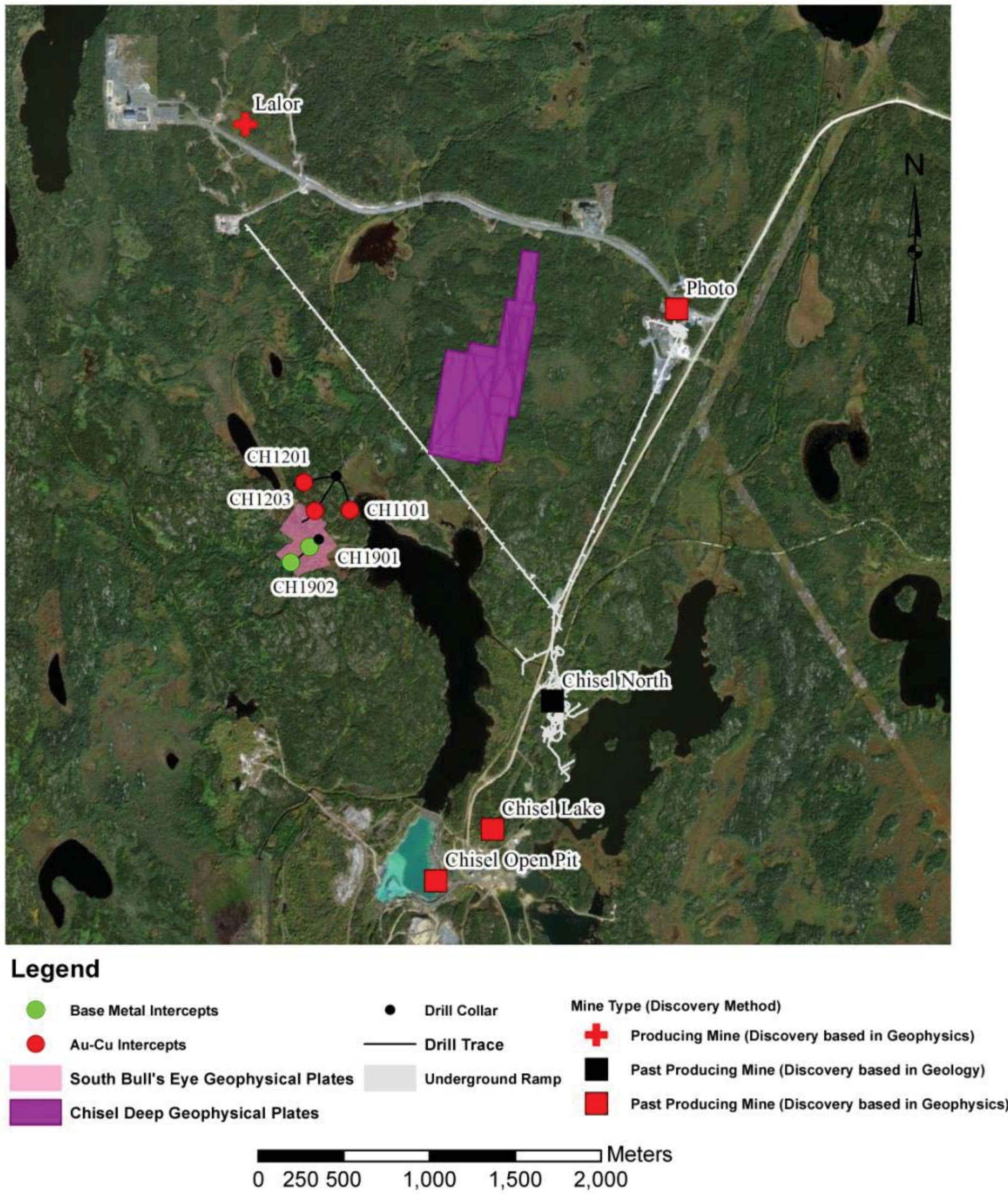
In 2018, Hudbay spent C$14 million on major airborne and ground geophysical surveys as well as on surface exploration drilling in the Flin Flon and Snow Lake areas. This work was instrumental in identifying several base metal and gold targets that are to be tested in 2019 with a comparable exploration budget.

An area of focus is the Lalor-Chisel-Photo area where Hudbay has discovered a new deposit, the 1901 Zone (Figure 9-5), by following a favorable up-dip plane with discrete copper-gold rich feeder intersections from drilling completed in 2011 and 2012 (see News Release on February 26, 2019 "Hudbay Announces Discovery of a New Deposit with High-Grade Zinc and High-Grade Gold Intersections Between the Chisel North and Lalor Mines").

This deposit is located between the old Chisel North mine and the producing Lalor mine at a down hole depth between 540.7-576.3m. The property is 100% owned by Hudbay and free of any encumbrances. The drill hole intersections occurred less than 1,000m from the existing active ramp between Chisel North and Lalor.

Drilling will continue in 2019 in the immediate vicinity of this initial discovery in order to further confirm the continuity of the mineralization and its lateral extent while testing a recently reinterpreted electromagnetic conductive plate west of the two recent intersections. Depending on the success of future drilling, the company expects to develop an exploration drift off the ramp from Chisel North to the Lalor mine in order to drill this new discovery from an underground platform.

FIGURE 9-5: LOCATION MAP OF THE 1901 ZONE DISCOVERY



10. DRILLING

10.1 LALOR MINE

Surface Drilling

As of January 1, 2019, a total of 305,487 m of surface drilling was completed at Lalor. Table 10-1 provides a summary by year.

TABLE 10-1: SURFACE DRILLING PROGRAMS BY YEAR (in Metres Drilled)

Year	Hole Type	Drilling Company	Core Size	Number of Holes	Length (m)
2007	Surface drilling	Major Drilling Ltd.	NQ/BQ	33	37,431
2008	Surface drilling	Major Drilling Ltd.	NQ/AQ	75	78,470
2009	Surface drilling	Major Drilling Ltd.	NQ/AQ	78	90,676
2010	Surface drilling	Major Drilling Ltd.	NQ/AQ	31	41,680
2011	Surface drilling	Major Drilling Ltd./ Boart Longyear Inc.	NQ/AQ	22	28,891
2012	Surface drilling	Major Drilling Ltd.	NQ	3	4,722
2015	Surface drilling	Major Drilling Ltd.	NQ	2	2,956
2017	Surface drilling	Rodren Drilling Ltd.	NQ	4	4,869
2018	Surface drilling	Rodren Drilling Ltd.	NQ	13	13,979
2019	Surface drilling	Rodren Drilling Ltd.	NQ	2	1,813
Total				**263**	**305,487**

All surface holes have been surveyed using reflex downhole tools (Ez-Shot or Ez-Trac) systems at a spacing of 30 metres down the hole. The survey instruments measure magnetic azimuth, dip, temperature, earth's magnetic field and roll face of tool for wedging operations. This data is then copied onto reflex survey instruments by the drillers and provide to the geologist daily. Readings are then inputted into the drilling database and a correction is applied to the magnetic azimuth to calculate true azimuth based on yearly diurnal drift of the magnetic north pole. All down hole survey results are validated by a geologist before being entered into the drill hole database.

Underground Drilling

Underground drill holes are drilled at all dips and azimuths needed to provide adequate coverage of the orebody for interpretation and mining purposes (Table 10-2). Holes with dips steeper than +70° are preferably avoided due to poor ergonomics and the increased risk for the drill crews.

Underground drilling at Lalor can be divided into four different categories based on the primary planned purpose of the hole:

- Exploration holes for targets outside known lenses or in areas of low drill density.
- Definition holes are drilled into known lenses to upgrade inferred resources to a higher category and to identify mineralization contacts for preliminary mine design purposes. Typical hole spacing is about 15 to 20 m.
- Delineation holes within indicated resources establish exact ore contacts for detailed mine planning and stope design purposes. Typical hole spacing is 10 to 15 m.
- Engineering holes are drilled for mine infrastructure purposes such as drain holes, holes for electrical cables and service holes for break through rounds.

TABLE 10-2: UNDERGROUND DRILLING PROGRAMS BY YEAR
(in metres drilled)

Year	Drilling Company	Core Size	Number of Holes	Length (m)
2012	Major Drilling Ltd.	NQ/BQ	50	7,519
2013	Major Drilling Ltd.	NQ/BQ	301	22,728
2014	Major Drilling Ltd.	NQ/BQ	396	35,669
2015	Major Drilling Ltd.	NQ/BQ	519	46,418
2016	Major Drilling Ltd.	NQ/BQ	336	34,070
2017	Major Drilling Ltd.	NQ/BQ	610	44,809
2018	Major Drilling Ltd.	NQ/BQ	744	73,691
2019	Major Drilling Ltd.	NQ/BQ	114	11,392
Total			**3,070**	**276,295**

All underground surveying at Lalor mine is conducted in Universal Transverse Mercator projection using NAD83 datum Zone 14. The mine survey was tied into the Canadian Spatial Reference System grid through the use of the Global Navigation Satellite System (GNSS) and an initial reference point at surface. Using the initial reference point, a reference azimuth was established by the use of a gyro compass. Based on the initial GNSS point and gyro azimuth a traverse was conducted to carry the survey down the ramp to the 810 m level. Once the main underground development workings were completed, another traverse was done to bring the survey down to 955 m level. All surveys into the individual levels are based on the control points created by the initial survey located at 810 m level and 955 m level. All surveying is done using electronic theodolites using the resection/free station technique.

Diamond drill lines are marked up according to layouts issued by the Mine's Geology Department. An electronic version of the layout is created for viewing on the electronic theodolite. In the field, the electronic theodolite is set up using resection or free station technique. Reflectorless Electronic Distance Measurement (EDM) is used to locate the planned drill azimuth line location according to the electronic drill hole layout. Collar location and a drill azimuth back sight is spray painted on the walls of the drift to function as front and back sights for diamond drillers to line up the drill. An anchor is installed on the painted lines to provide a permanent reference line. Once drilling is completed the collar survey is recorded using the resection setup method. Collar location is surveyed using reflectorless EDM surveying and collar location stored as single point data.

Downhole surveys were completed using a Reflex EZ-Shot®, EZ-A® or EZ-Trac® (Reflex) instrument. Surveys were completed at regular intervals of 30 m down the hole.

A down hole gyro survey was conducted at the Lalor Mine in June and July of 2016 to check the accuracy of the magnetic Reflex readings. The survey was conducted by company technicians using a Reflex TN14 gyro compass and a MEMS down hole gyro probe. Thirty-eight underground holes were surveyed by gyro method for a total of 11,358 m. No significant discrepancies were identified between azimuth values measured by magnetic Reflex survey methods and azimuth values measured by the gyro survey methods.

Snow Lake Gold Properties

Table 10-3 lists all the surface and underground drilling on the Snow Lake Gold Properties since the 1940s.

TABLE 10-3: SUMMARY OF SURFACE AND UNDERGROUND DIAMOND DRILL HOLES

(as of December 31,2018)

Year	Area	Hole Type	Operator	Number of Holes	Total Meters Drilled
1945-1946	Squall	Surface	Squall Lake Gold Mines Ltd.	363	15,250
1950	Squall	Surface	Wekusko Consolidated Ltd.	8	503
1971-1972	Squall	Surface	Stall lake Mines Ltd.	6	367
1979	Squall	Surface	Corporate Oil and Gas	NA	NA
1980	Squall	Surface	W.B. Kobar	1	315
1980-1981	Squall	Surface	Camflo Mines Ltd.	35	3,223
1984	Squall	Surface	Barrick Resources Corp. (formerly Camflo Mines Ltd.)	114	8,559
1987	Squall	Surface	Zenco Resources	30	3,272
1990	Squall	Surface	Graham Gold Mining Corp.	11	1,067
2003	Squall	Surface	Coniagas/Trienergy/ MBMI	19	1,448
Total				**587**	**34,004**

1948-1956	NB	Surface & underground	Howe Sound Exploration Co. Ltd. & Britannia Mining & Smelting Co. Ltd.	1,629	92,766
1988-1989	NB	Underground	High River Gold Mines Ltd.	88	9,809
1988-1989	NB	Surface & underground	Inco Gold Co.	86	17,741
1994-2002	NB	Surface & underground	TVX Gold Inc. & High River Gold Mines	7,107	494,750
2003-2004	NB	Surface & underground	Kinross Gold Corp. & High River Gold	1,127	67,886
2007-2008	NB	Surface	Garson Gold Corp.	10	3,618
2009	NB	Surface	Garson Gold Corp.	3	913
2010-2011	NB	Surface	Alexis Minerals Corporation	146	50,872
Total				**10,196**	**738,355**

Alexis Minerals Corp. was the last operator to conduct drilling on the property. The 2010-2011 Alexis drilling campaigns focused on collecting information in areas with the potential to provide mineral resources to the mining plan and additional surface targets in the Snow Lake Gold area. the drill programs concentrated on areas around the Boundary, Birch, Mine East and near the New Britannia Mine.

This most recent drilling completed by Garson Gold Corp. and Alexis Minerals Corp was NQ diameter core utilizing a drilling company based in Alida, Saskatchewan called Dig-It Exploration. The location of all the drill holes were surveyed and tied into the established grid on the property. Down hole surveys were completed using Fordia Flexit Smart-tool System. The first reading was taken approximately 10 m down the hole with subsequent readings being taken every 50 to 60 metres down the hole.

Orientated core has been performed on some of the holes drilled at No. 3 Zone and at Boundary using the Ball Mark Orientation System. The orientated core was logged for structural measurements on veins and rock fabric.

Pen II

Historical drilling at the Pen II deposit included 15 holes for a total of 2,029.26 metres, for which downhole surveys were completed sporadically using acid tests and tropari instrument. Since 2017, 9,531 metres of additional drilling have been collected with downhole survey utilizing reflex (Ez-Shot and Ez-Trac) systems. Readings were then inputted into the drilling database and a correction is applied to the magnetic azimuth to calculate true azimuth based on yearly diurnal drift of the magnetic north pole. All down hole survey results are validated by a geologist before being entered into the drill hole database. Hole collars were picked up utilizing a Differential Global Positioning System (DGPS) supplied by Hudbay's mine technical service group.

Wim

In total, 136 diamond drill holes and 43,285.36 m have been completed on the Wim deposit from 1962 to 2008. All drilling prior to Murgor's exploration campaigns on the Wim deposit was completed by HBED. HBED completed six drilling campaigns between 1962 and 1991, which included 75 diamond drill holes (AX, EX and BQ core sizes) for a total of 25,018.90 m of drilling of which 17 were wedge drill holes. In 2007 and 2008, Murgor completed a total of 61 NQ-sized diamond drill holes for a total of 18,266 m of drilling on the Wim deposit. Half core samples from the historical drilling programs (pre-2007) are stored at the Stall Lake Mill Complex located approximately 5 km from Snow Lake. All half core samples from Murgor's 2007 and 2008 drilling campaigns are permanently stored on the Wim project site or at the Manitoba government core storage at the past producing Centennial mine.

11. SAMPLE PREPARATION, ANALYSES & SECURITY

Sample preparation, analysis, and security procedures applied to Lalor and Pen II were reviewed by Olivier Tavchandjian,P.Geo. the Qualified Person of this Technical Report. The sampling methodology, analyses and security measures have been documented in great detail in past published technical reports by Hudbay for its Manitoba operations and will only be summarized in this document. Generally, the same procedures were followed by the exploration team of Murgor Resources at Wim, who was under the supervision of a former chief geologist at Hudbay. Unless specified in the report, all the laboratories used for samples preparation, analysis and security are independent of Hudbay. The procedures followed at the New Britannia mine and other Snow Lake gold properties are summarized at the end of this section and are described in greater details in the Technical Report produced by Genivar for Alexis Resources in 2011 and available on SEDAR.

11.1 LALOR

At the Lalor mine, drill core is logged directly into the acQuire drill hole database using dedicated laptop computers. Lithologies, alteration, structures, mineralization core quality and recovery are described in detail throughout the core length. Sample intervals are clearly marked on the core.

The core is photographed before samples are split and bagged for shipment. Photographing the core is the last step of the logging process to assure exact locations of all contacts, sample locations and numbers are captured. A standard setup as well as a single camera type with standardized settings is used to ensure photographs are of consistent quality. Once photographs have been captured, they are saved on the company server and linked to the acQuire drill hole database for easy network access.

Prior to splitting core, the database manager prints a sample list for each drill hole that includes the sample identification number, hole name, sample type and the start and end footage of each sample. The core is split along the cut line drawn by the geologist. In gouge and rubble intervals, an aluminium sampling scoop is used to separate the gouge into two halves in the core boxes. Completed sample bags are closed using the bag drawstrings and secured at the neck using two zip ties. All saws and sampling buckets are rinsed with water after cutting each sample to prevent cross-contamination.

Once logging is completed, all data from sample books including QAQC samples are entered into Hudbay's acQuire drill hole database by hole number. Before dispatch, QAQC samples are inserted into the sample stream. Hudbay's practice involves insertion of the following every 100 samples:

1) Two blanks

2) Five duplicates

3) Five base metal standards

4) Two gold standards

Sample Preparation

Sample preparation for Lalor has been conducted at three different laboratories over time. Since September 2016, nearly all samples are prepared and assayed at Bureau Veritas. All drill core samples analyzed since the last technical report (NI 43-101 published on March 30, 2017) have been sent for analysis at Bureau Veritas. SGS was used as the umpire laboratory for quality control purposes, and in rare instances, the Flin Flon laboratory was also used for external checks. Table 11-1 presents the number of samples prepared and analyzed by year.

TABLE 11-1: LALOR SAMPLES PREPARATION AND ANALYSIS BY YEAR

Year	Number of Samples Dispatched	Flin Flon Laboratory	Acme / Bureau Veritas Laboratory
2007	8,601	8,563	38
2008	16,004	15,586	418
2009	26,753	14,737	12,016
2010	13,756	0	13,756
2011	8,684	0	8,684
2012	11,310	10,398	912
2013	24,207	24,207	0
2014	28,393	28,393	0
2015	39,019	29,735	9,284
2016	47,621	23,394	24,227
2017	52,472	3,549	48,923
2018	55,539	2,242	53,297
Total	**332,359**	**160,804**	**171,555**

ACME/Bureau Veritas

All samples arriving at ACME (Bureau Veritas) were checked against the custody chain form and prepped according to the lab protocols. The sample preparation includes weighing the sample, crushing 1 kg to minimum 80% passing 2 mm and a 250 g split crushed to minimum 85% passing 75 µm.

SGS

All samples arriving at SGS are checked against the custody chain form. No sample preparation was conducted at SGS since the laboratory was only used to check assays done on pulps.

Specific Gravity

A total of 74,299 density measurements collected by Hudbay were either measured at the Flin Flon laboratory, ACME/Bureau Veritas laboratory or at Hudbay's logging facility, using a non-wax-sealed immersion technique to measure the weight of each sample in air and in water. Since February 2018, the density measurements are performed with a gas pycnometer (method SPG04) at Bureau Veritas in Vancouver. Independent checks have been conducted to confirm that there is no material difference between the two methods. Overall, there were 12,912 density measurement made in all the mineralized envelopes. Table 11-2 presents the number of density measurements by year.

TABLE 11-2: DENSITY MEASUREMENTS AT LALOR BY YEAR

Year	Number of Specific Density Measurements	Flin Flon Laboratory	Acme / Bureau Veritas Laboratory	Hudbay Logging Facility
2007	4,350	0	0	4,350
2008	7,118	0	0	7,118
2009	1,965	0	0	1,965
2010	0	0	0	0
2011	0	0	0	0
2012	6,337	0	0	6,337
2013	7,548	7,180	0	368
2014	19,578	16,590	2,988	0
2015	10,767	6,485	4,282	0
2016	7,373	1,640	5,733	0
2017	2,881	0	2,881	0
2018	6,382	0	6,382	0
Total	**74,299**	**31,895**	**22,266**	**20,138**

Sample Analysis

As stated previously, all drill core samples analyzed since the last technical report (NI 43-101 published on March 30, 2017) have been sent to Bureau Veritas for analysis.

ACME/Bureau Veritas

Base metals and other elements were analyzed with aqua regia digestions coupled with ICP-ES/MS (method AQ270 and AQ370). Assays returning “over the range” values were reassayed with higher reporting methods (GC820 for copper, GC816 for zinc, GC817 for lead and MA404 for copper and zinc when values were above 20%).

All samples were analyzed for gold using a fire assay method coupled with atomic absorption (FA430). A gravimetric finish (FA530) was used when samples return gold values above 10 PPM.

Bureau Veritas is currently registered with ISO 9001 and ISO/IEC 17025 accreditations. Fine sample pulps are kept in secure storage at the laboratory after analysis. Pulps are only released after all data is validated.

SGS

Base metals and other elements were analyzed with aqua regia digestions coupled with ICP-ES (method ICP14B). Assays returning “over the range” values were reassayed with higher reporting methods (ICP13B when values were above 1% for zinc, copper and lead).

All samples were analyzed for gold using a fire assay method coupled with atomic absorption (FAA313). A gravimetric finish (FAG303) was used when samples return gold values above 10 PPM.

SGS is currently registered with ISO 9001 and ISO/IEC 17025 accreditations. Fine sample pulps are kept in secure storage at the laboratory after analysis. Pulps are only released after all data is validated.

Quality Control and Quality Assurance

Data verification procedures and results since the previous technical report are included in this section based on information collected and reviewed from 2016 to 2018. During this period, all samples were assayed at the Acme/Bureau Veritas laboratory. For data verification procedures and results prior to 2016, refer to previous technical report dated March 29, 2017.

As part of Hudbay QAQC program, QAQC samples were systematically introduced in the sample stream to assess sub-sampling procedures, potential cross-contamination, precision and accuracy. Hudbay commonly includes 5% certified reference materials (CRM), 2% certified blanks, and 5% coarse duplicates. Blanks and CRMs were prepared by Ore Research and Exploration (OREAS) and Analytical Solutions Ltd. (ASL). High-grade gold standards are from Rocklabs. All QAQC samples were analyzed following the same analytical procedures as those used for the drill core samples.

Standards

From 2016 to 2018 a total of 5,363 OREAS and Rocklabs CRMs were analyzed at Bureau Veritas representing 4.9% of the sample stream. Table 11-3 presents a list of all CRMs used by Hudbay for quality control over the past two years.

TABLE 11-3: CRM

Name	Origine	CRM type	Number of insertion
A6	OREAS	low grade	773
B6	OREAS	low grade	770
C6	OREAS	medium Cu grade	763
D6	OREAS	high Cu grade	764
E6	OREAS	high Cu grade	766
SN75	Rocklabs	high Cu grade	672
SN91	Rocklabs	high Cu grade	97
SP73	Rocklabs	high Cu grade	757
		Total	5,362

Table 11-4 presents the certified value of the CRMs along with the passing rates per element for each standard. A failure rate that is less than 5% is considered good, while a failure rate between 5% and 10% is considered reasonable. A failure rate that is greater than 10% would trigger additional investigations.

TABLE 11-4: CERTIFIED VALUES AND PASSING RATES

Certified Value	A6	B6	C6	D6	E6	SN75	SN91	SP73
Gold (g/t)	0.121	0.706	0.987	3.040	0.316	8.671	8.679	18.17
Zinc (%)	0.035	0.643	2.410	3.310	18.14	*n/a*	*n/a*	*n/a*
Copper (%)	0.055	0.881	2.050	4.200	0.245	*n/a*	*n/a*	*n/a*
Silver (g/t)	0.535	2.960	5.180	25.170	15.23	*n/a*	*n/a*	*n/a*

Analytical Bias	A6	B6	C6	D6	E6	SN75	SN91	SP73
Gold	94.6%	86.6%	89.3%	96.9%	91.4%	92.7%	97.9%	91.5%
Zinc	97.4%	13.8%	94.2%	99.9%	99.6%	*n/a*	*n/a*	*n/a*
Copper	94.7%	99.1%	97.4%	99.2%	100.0%	*n/a*	*n/a*	*n/a*
Silver	60.9%	88.6%	95.0%	95.8%	96.5%	*n/a*	*n/a*	*n/a*

In most cases, the analytical bias is considered adequate for resource estimate purposes. High levels of failures occurred mostly for the B6 CRM. The relevance of this standard is questionable as its composition, reflecting a medium grade for both gold, zinc and copper, does not match any mineralization type at Lalor. In any case, when failure of a CRM occurred, part of the assays batch including the standard was re-assayed until all the QAQC results cleared the built-in checks in the acQuire database.

Blanks

Between 2016 to 2018, a total of 2,149 certified OREAS and ASL blanks (about 2% of the samples stream), were inserted into the sample stream to monitor potential cross-contamination or sample swaps at Bureau Veritas. Table 11-5 presents a list of the blank CRMs used by Hudbay for quality control over the past two years.

TABLE 11-5: BLANK CRM

Name	Origine	CRM type	Number of insertion
F7	OREAS	blank	1,182
F8	ASL	blank	966
		Total	2,149

Table 11-6 presents the certified value of the blank CRMs along with the failure per element for each standard. A failure rate that is less than three times the detection limit (3xDL) 90% of the time is considered acceptable.

TABLE 11-6: BLANK CERTIFIED VALUES AND PASSING RATES

		Performance Gates		Results				
F7	Certified Value	DL	3DL	< 3DL	> 3DL	> 5DL	> 10DL	% of < 3DL
Au PPM	0.0025	0.005	0.015	1,148	35	16	9	95%
Zn %	0.0107	0.01	0.03	1,171	12	8	1	98%
Cu %	0.005	0.001	0.003	41	1,142	105	3	3%
Ag PPM	0.068	2	6	1,183	0	0	0	100%

		Performance Gates		Results				
F8	Certified Value	DL	3DL	< 3DL	> 3DL	> 5DL	> 10DL	% of < 3DL
Au PPM	0.005	0.005	0.015	951	15	11	4	97%
Zn %	0.001	0.01	0.03	961	5	1	1	99%
Cu %	0.0005	0.001	0.003	950	16	8	1	97%
Ag PPM	1	2	6	966	0	0	0	100%

In all cases aside from copper in the blank F7, the cross-contamination or sample swaps are considered adequate for resource estimate purposes. The failure of copper in F7 is linked to its certified value being equal to the 3xDL criteria. In this case, 99.7% of the results came back within five times the detection limit, which is a more appropriate performance gate, given its certified value. In any case, when failure of a blank CRM occurred, part of the assays batch, including the blank, was re-assayed until all the QAQC results cleared the database acQuire built-in check checks.

Coarse Duplicates

From 2016 to 2018 a total of 5,385 coarse duplicates were analyzed at Bureau Veritas representing 4.9% of the sample stream. Basic statistics, scatter plots, quantitle-quantile plots, probability plots and absolute difference plots were used to assess the precision of the laboratory. Based on the paired analysis results, the laboratory appeared to be precise. Table 11-7 presents the basic statistics results between the original and the duplicates.

TABLE 11-7: STATISTICS OF ORIGINAL AND DUPLICATES RESULTS

	Originals Au	Duplicate Au	Distribution	Originals Au	Duplicate Au	Units
Population	5,385		25.0%	0.02	0.02	PPM
Minimum	0.00	0.00	50.0%	0.03	0.03	PPM
Maximum	374.20	388.30	75.0%	0.04	0.04	PPM
Mean	1.37	1.37	80.0%	0.08	0.08	PPM
Std Dev	9.21	9.54	90.0%	0.15	0.15	PPM
CV	6.73	6.95	97.5%	0.29	0.30	PPM
Correlation	0.994		99.9%	0.43	0.42	PPM
	Originals Ag	**Duplicate Ag**	**Distribution**	**Originals Ag**	**Duplicate Ag**	**Units**
Population	5,385		25.0%	0.70	0.70	PPM
Minimum	0.25	0.25	50.0%	1.00	1.00	PPM
Maximum	1271.00	1287.00	75.0%	1.00	1.00	PPM
Mean	10.65	10.68	80.0%	1.30	1.30	PPM
Std Dev	36.75	37.07	90.0%	2.60	2.50	PPM
CV	3.45	3.47	97.5%	4.60	4.60	PPM
Correlation	0.998		99.9%	6.30	6.50	PPM
	Originals Cu	**Duplicate Cu**	**Distribution**	**Originals Cu**	**Duplicate Cu**	**Units**
Population	5,385		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	12.01	11.69	75.0%	0.02	0.02	%
Mean	0.15	0.15	80.0%	0.02	0.02	%
Std Dev	0.51	0.51	90.0%	0.03	0.03	%
CV	3.43	3.42	97.5%	0.07	0.07	%
Correlation	0.999		99.9%	0.10	0.10	%
	Originals Zn	**Duplicate Zn**	**Distribution**	**Originals Zn**	**Duplicate Zn**	**Units**
Population	5,385		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	46.26	45.91	75.0%	0.01	0.01	%
Mean	0.72	0.72	80.0%	0.01	0.01	%
Std Dev	2.82	2.83	90.0%	0.02	0.02	%
CV	3.91	3.90	97.5%	0.04	0.04	%
Correlation	1.000		99.9%	0.07	0.07	%

Umpire Laboratory

From 2016 to 2018 a total of 3,272 pulps rejects were sent to and analyzed at SGS to validate the reproducibility of Bureau Veritas results. This external check, represents 3% of the sample stream. Basic statistics, scatter plots, quantitle-quantile plots, probability plots and absolute difference plots were used to assess the precision of the laboratory. Based on the paired analysis results, Bureau Veritas appeared to be reproducible. Table 11-8 presents the basic statistics results between the original results (BV) and the duplicate results (SGS).

Additional check assays conducted at the Flin Flon laboratory were found to have a negative bias compared to Bureau Veritas of approximately 3.5% on gold assays.

TABLE 11-8: STATISTICS OF ORIGINAL (BV) AND DUPLICATES (SGS) RESULTS

	Originals Au	Duplicate Au	Distribution	Originals Au	Duplicate Au	Units
Population	3,272		25.0%	0.02	0.02	PPM
Minimum	0.00	0.00	50.0%	0.03	0.03	PPM
Maximum	374.20	385.06	75.0%	0.05	0.05	PPM
Mean	1.43	1.43	80.0%	0.09	0.09	PPM
Std Dev	9.31	9.65	90.0%	0.17	0.17	PPM
CV	6.52	6.75	97.5%	0.33	0.33	PPM
Correlation	0.998		99.9%	0.48	0.48	PPM
	Originals Ag	**Duplicate Ag**	**Distribution**	**Originals Ag**	**Duplicate Ag**	**Units**
Population	3,272		25.0%	1.00	1.00	PPM
Minimum	0.25	1.00	50.0%	1.00	1.00	PPM
Maximum	1271.00	1322.00	75.0%	1.00	1.00	PPM
Mean	10.43	10.74	80.0%	1.60	1.00	PPM
Std Dev	36.62	37.87	90.0%	3.00	3.00	PPM
CV	3.51	3.53	97.5%	5.00	5.00	PPM
Correlation	0.994		99.9%	6.83	7.00	PPM
	Originals Cu	**Duplicate Cu**	**Distribution**	**Originals Cu**	**Duplicate Cu**	**Units**
Population	3,272		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	12.01	11.40	75.0%	0.02	0.02	%
Mean	0.16	0.16	80.0%	0.02	0.02	%
Std Dev	0.55	0.56	90.0%	0.04	0.04	%
CV	3.44	3.42	97.5%	0.08	0.08	%
Correlation	0.999		99.9%	0.11	0.11	%
	Originals Zn	**Duplicate Zn**	**Distribution**	**Originals Zn**	**Duplicate Zn**	**Units**
Population	3,272		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	46.26	47.10	75.0%	0.01	0.01	%
Mean	0.63	0.63	80.0%	0.01	0.01	%
Std Dev	2.62	2.61	90.0%	0.02	0.02	%
CV	4.14	4.13	97.5%	0.04	0.04	%
Correlation	0.998		99.9%	0.06	0.06	%

11.2 PEN II

Sample Preparation

The samples preparation methods performed on the Pen II samples are the same as the ones used for the Lalor mine samples. Table 11-9 presents the number of samples prepared and analyzed by year.

TABLE 11-9: PEN II SAMPLES PREPARATION AND ANALYSIS BY YEAR

Year	Number of Samples Dispatched	Flin Flon Laboratory	Acme / Bureau Veritas Laboratory
1971	37	37	
1983	15	15	
1985	28	28	
1986	519	519	
1987	255	225	
1988	116	116	
1990	54	54	
1991	63	63	
1999	75		75
2000	33		33
2007	127	127	
2017	674		674
2018	868		868
Total	**2,864**	**1,184**	**1,650**

Specific Gravity

1,306 density measurements were collected by Hudbay in 2017 and 2018. These measurements were performed at Bureau Veritas laboratory in Vancouver using a gas pycnometer (method SPG04). Table 11-10 presents the number of density measurements by year.

TABLE 11-10: DENSITY MEASUREMENTS AT PEN II BY YEAR

Year	Number of Specific Density Measurements	Acme / Bureau Veritas Laboratory
2017	575	575
2018	731	731
Total	**1,306**	**1,306**

Sample Analysis

The laboratory analysis for the Pen II deposit was also conducted at the Bureau Veritas laboratory in Vancouver, using the same methods as those performed on the Lalor samples with SGS laboratory conducting external checks.

Quality Assurance and Quality Control

Data verification procedures and results are based on information collected and reviewed from the 2017 and 2018 drilling campaign. During this period, all samples were assayed at the Acme/Bureau Veritas laboratory.

As part of Hudbay's QAQC program, QAQC samples were systematically introduced into the sample stream to assess sub-sampling procedures, potential cross-contamination, precision and accuracy. Hudbay commonly includes 5% certified reference materials (CRM), 2% certified blanks, and 5% coarse duplicates in the sample stream. Blanks and CRMs were prepared by Ore Research and Exploration (OREAS) and Analytical Solutions Ltd. (ASL). High-grade gold standards are from Rocklabs. All QAQC samples were analyzed following the same analytical procedures as those used for the drill core samples.

Standards

In 2017 to 2018 a total of 77 OREAS were analyzed at Bureau Veritas, representing 4.9% of the sample stream. Table 11-11 presents a list of all CRMs used by Hudbay for quality control over the past two years and Table 11-12 presents the certified value of the CRMs and the passing rates per element for each standard.

TABLE 11-11: CRM

Name	Origine	CRM type	Number of insertion
A6	OREAS	low grade	3
B6	OREAS	low grade	20
C6	OREAS	medium Cu grade	19
D6	OREAS	high Cu grade	14
E6	OREAS	high Cu grade	21
		Total	77

TABLE 11-12: CERTIFIED VALUES AND PASSING RATES

Certified Value	A6	B6	C6	D6	E6
Gold (g/t)	0.121	0.706	0.987	3.040	0.316
Zinc (%)	0.035	0.643	2.410	3.310	18.14
Copper (%)	0.055	0.881	2.050	4.200	0.245
Silver (g/t)	0.535	2.960	5.180	25.170	15.23

Analytical Bias	A6	B6	C6	D6	E6
Gold	100%	90%	94.7%	100%	100%
Zinc	100%	100%	89.5%	92.9%	100%
Copper	100%	80%	84.2%	100%	95.2%
Silver	100%	100%	94.7%	71.4%	66.7%

In most cases, the analytical bias on zinc is considered adequate for resource estimate purposes. High failure rates mostly impact copper and silver for standard values which are rarely found at Pen II. In any case, when failure of a CRM occurred, part of the assays batch including the standard was re-assayed until all the QAQC results cleared the built-in checks in the acquire database.

Blanks

In 2017 to 2018, a total of 82 certified OREAS and ASL blanks, or 5.3% of the samples stream, were inserted into the sample stream to monitor potential cross-contamination or sample swaps at Bureau Veritas. Table 11-13 presents a list of the blank CRMs used by Hudbay for quality control over the past two years.

TABLE 11-13: BLANK CERTIFIED VALUES AND PASSING RATES

		Performance Gates		Results				
F7	Certified Value	DL	3DL	< 3DL	> 3DL	> 5DL	> 10DL	% of < 3DL
Au PPM	0.0025	0.005	0.015	33	0	0	0	100%
Zn %	0.0107	0.01	0.03	33	0	0	0	100%
Cu %	0.005	0.001	0.003	0	33	0	0	0%
Ag PPM	0.068	2	6	33	0	0	0	100%

		Performance Gates		Results				
F8	Certified Value	DL	3DL	< 3DL	> 3DL	> 5DL	> 10DL	% of < 3DL
Au PPM	0.005	0.005	0.015	49	0	0	0	100%
Zn %	0.001	0.01	0.03	49	0	0	0	100%
Cu %	0.0005	0.001	0.003	49	0	0	0	100%
Ag PPM	1	2	6	49	0	0	0	100%

In all cases aside from copper in the blank F7, the cross-contamination or sample swaps are considered adequate for resource estimate purposes. The failure of copper in F7 is linked to its certified value been equal to the 3xDL criteria. In this case, 100% of the results came back within five times the detection limit, which is a more appropriate performance gate given its certified value. In any case, when failure of a blank CRM occurred, part of the assays batch including the blank was re-assayed until all the QAQC results cleared the acQuire database built-in check checks.

Coarse Duplicates

In 2017 and 2018 a total of 43 coarse duplicates were analyzed at Bureau Veritas representing 2.7% of the sample stream. Basic statistics, scatter plots, quantile-quantile plots, probability plots and absolute difference plots were used to assess the precision of the laboratory. Based on the paired analysis results, the laboratory appeared to be precise. Table 11-14 presents the basic statistics results between the original and the duplicates.

TABLE 11-14: STATISTICS OF ORIGINAL AND DUPLICATES RESULTS

	Originals Au	Duplicate Au	Distribution	Originals Au	Duplicate Au	Units
Population	43		25.0%	0.00	0.00	PPM
Minimum	0.00	0.00	50.0%	0.00	0.00	PPM
Maximum	1.10	1.27	75.0%	0.01	0.01	PPM
Mean	0.07	0.07	80.0%	0.01	0.01	PPM
Std Dev	0.19	0.20	90.0%	0.01	0.02	PPM
CV	2.50	2.76	97.5%	0.03	0.04	PPM
Correlation	0.973		99.9%	0.07	0.06	PPM

	Originals Ag	Duplicate Ag	Distribution	Originals Ag	Duplicate Ag	Units
Population	43		25.0%	0.23	2.00	PPM
Minimum	0.05	0.05	50.0%	0.24	0.25	PPM
Maximum	135.00	110.00	75.0%	0.34	0.40	PPM
Mean	6.28	5.74	80.0%	0.48	0.46	PPM
Std Dev	21.03	17.47	90.0%	0.63	0.84	PPM
CV	3.35	3.05	97.5%	1.29	1.61	PPM
Correlation	0.989		99.9%	2.37	2.48	PPM

	Originals Cu	Duplicate Cu	Distribution	Originals Cu	Duplicate Cu	Units
Population	43		25.0%	0.00	0.00	%
Minimum	0.00	0.00	50.0%	0.00	0.00	%
Maximum	0.58	0.82	75.0%	0.01	0.00	%
Mean	0.06	0.06	80.0%	0.01	0.01	%
Std Dev	0.13	0.15	90.0%	0.01	0.02	%
CV	2.26	2.39	97.5%	0.03	0.03	%
Correlation	0.980		99.9%	0.04	0.05	%

	Originals Zn	Duplicate Zn	Distribution	Originals Zn	Duplicate Zn	Units
Population	43		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	40.56	41.28	75.0%	0.01	0.01	%
Mean	1.16	1.15	80.0%	0.01	0.01	%
Std Dev	6.20	6.30	90.0%	0.02	0.02	%
CV	5.33	5.46	97.5%	0.04	0.03	%
Correlation	1.00		99.9%	0.05	0.04	%

Umpire Laboratory

In 2017 and 2018 a total of 67 pulps rejects were sent to and analyzed at SGS to validate the reproducibility of Bureau Veritas results. This external check represents 4.3% of the sample stream. Basic statistics, scatter plots, quantile-quantile plots, probability plots and absolute difference plots were used to assess the precision of the laboratory. Based on the paired analysis results, the Bureau Veritas results appeared to be reproducible. Table 11-15 presents the basic statistics results between the original results (BV) and the duplicate results (SGS).

TABLE 11-15: STATISTICS OF ORIGINAL (BV) AND DUPLICATES (SGS) RESULTS

	Originals Au	Duplicate Au	Distribution	Originals Au	Duplicate Au	Units
Population	67		25.0%	0.01	0.00	PPM
Minimum	0.00	0.00	50.0%	0.01	0.01	PPM
Maximum	1.10	1.06	75.0%	0.01	0.01	PPM
Mean	0.07	0.07	80.0%	0.01	0.01	PPM
Std Dev	0.16	0.15	90.0%	0.01	0.02	PPM
CV	2.31	2.34	97.5%	0.04	0.04	PPM
Correlation	0.996		99.9%	0.06	0.06	PPM
	Originals Ag	**Duplicate Ag**	**Distribution**	**Originals Ag**	**Duplicate Ag**	**Units**
Population	67		25.0%	0.18	0.17	PPM
Minimum	0.04	0.02	50.0%	0.21	0.20	PPM
Maximum	135.00	100.00	75.0%	0.28	0.27	PPM
Mean	5.10	4.51	80.0%	0.45	0.42	PPM
Std Dev	17.08	13.13	90.0%	0.55	0.51	PPM
CV	3.35	2.91	97.5%	1.36	1.25	PPM
Correlation	0.994		99.9%	2.92	2.87	PPM
	Originals Cu	**Duplicate Cu**	**Distribution**	**Originals Cu**	**Duplicate Cu**	**Units**
Population	67		25.0%	0.00	0.00	%
Minimum	0.00	0.00	50.0%	0.00	0.00	%
Maximum	0.90	0.96	75.0%	0.01	0.01	%
Mean	0.07	0.08	80.0%	0.01	0.01	%
Std Dev	0.16	0.17	90.0%	0.01	0.01	%
CV	2.23	2.29	97.5%	0.03	0.03	%
Correlation	0.995		99.9%	0.04	0.04	%
	Originals Zn	**Duplicate Zn**	**Distribution**	**Originals Zn**	**Duplicate Zn**	**Units**
Population	67		25.0%	0.01	0.01	%
Minimum	0.00	0.00	50.0%	0.01	0.01	%
Maximum	40.56	30.00	75.0%	0.01	0.01	%
Mean	2.06	1.79	80.0%	0.01	0.01	%
Std Dev	7.47	6.13	90.0%	0.02	0.02	%
CV	3.63	3.42	97.5%	0.03	0.03	%
Correlation	0.992		99.9%	0.05	0.05	%

11.3 WIM

Hudbay has not performed any work on the Wim property since 1991. The sampling methodology, analyses and security measures used by the previous owners (i.e. Murgor Resources and Alexandria Minerals) have been documented in Alexandria Minerals' 2015 technical report and will only be summarised in this document.

Sample Preparation

The samples preparation methods performed on the Wim samples are the same as the ones used for the Lalor mine samples. The logs were entered in a Gemcom Access® database.

Specific Gravity

572 density measurements were collected by Murgor Resources in 2007 and 2008. These measurements were performed at TSL Laboratories in Saskatoon, using a non-wax-sealed immersion technique to measure the weight of each sample in air and in water.

Sample Analysis

"All drill cores assayed from the 2007 and 2008 drilling programs were completed by TSL Laboratories, using multi-acid digestion and atomic absorption spectrometry for the base metals and fire assay, atomic absorption and gravimetric finish for gold. All samples selected for analysis were prepared and analyzed for Cu, Zn, Au, Ag, Fe, As and Pb.". TSL Laboratories is currently registered with ISO/IEC 17025 accreditation

Quality Assurance and Quality Control

Data verification procedures and results are based on information collected and reviewed from the 2007 and 2008 drilling campaign. During this period, all samples were assayed at the TSL laboratory in Saskatoon while whole rock geochemistry pulp samples were sent to ALS Laboratory in Vancouver. As part of Murgor's QAQC program, QAQC samples were systematically introduced in the sample stream to assess sub-sampling procedures, potential cross-contamination, precision, and accuracy, typically using a 10% insertion rate (i.e. one CRM, blank or duplicate every 10 samples). All QAQC samples were analyzed following the same analytical procedures as those used for the drill core samples at Wim.

The results of the review done by Golder indicated that Murgor's QAQC standard sampling program was working effectively, and the program met industry standards.

11.4 SNOW LAKE GOLD PROPERTIES

The sampling methodology, analyses and security measures used by the previous owners (i.e. Kinross and Alexis Resources) have been documented in the Technical Report produced by Genivar for Alexis Resources in 2011, available on SEDAR and will only be summarized in this document.

Sample Preparation

Details of the sampling methods conducted by earlier operators of the New Britannia mine prior to 1995 have been lost. Between 1995 and 2003, the New Britannia sampling method consisted in core being logged on paper at the mine site by geologists. The written logs were later transferred into a Gemcom Access database. Technicians would lay out the core, split and samples to the core according to the geologists' instructions. Chip samples were also collected during the mining operation by the geologist either taken from the back of the drift if the muck pile had not been removed or taken 1.40 m to 1.52 m above the floor of the drift if the blast muck has been removed. The geologist was responsible for bringing the samples to the preparation area of the mill laboratory and recording the number of each sample into a record book.

Sample Analysis

Most of the drill cores and chips assays from 1995 to 2003 from the New Britannia mine were completed at the on-site mill laboratory using a fire assay/atomic absorption finish (FA/AA) method.

At the other Snow Lake properties, sample analysis, including determination of gold and arsenic on all samples and specific gravity and multi-element geochemistry on selected samples, was completed at TSL Laboratories Inc. (TSL) in Saskatoon, SK. TSL was established in 1981 and certified as ISO/IEC 17025. At TSL, the samples are dried and crushed and a 250 g riffle split of the crushed sample is pulverized to 95% passing 150 mesh (106 microns). Gold concentration is determined on all samples from a 30-g split of the pulverized sample by fire assay with atomic absorption finish. All samples with a gold concentration greater than 1,000 ppb (1 gram/tonne) were re-assayed using fire assay with a gravimetric finish. Specific gravity and ICP-MS multi-element analyses were completed on selected samples as requested by Garson project geologists.

Quality Assurance and Quality Control

Standard, blank and duplicate assay samples were added to each batch of 21 samples for drill core and to each batch of 24 samples for chip samples.

The sampling methods and approaches and concluded that the sampling and analytical procedures conformed to the industry standards at the time, and these were adequate to ensure a representative determination for the type of gold mineralization identified on the property. The QAQC data documented in the Genivar Technical Report (2011) is not available for the QP to review. The QP is confident that the audits done by Micon in 2005 and 2009 are sufficient for the purpose of reporting inferred mineral resource estimates for the Snow Lake gold properties.

12. DATA VERIFICATION

12.1 LALOR MINE AND PEN II ZONE

Site Visits

As mentioned in section two of this report, monthly site visits were conducted by the Qualified Person since September 2017. These visits included the core logging facilities and the underground mine to review the geology and mineralization of the different mineralized zones. Visits of the exploration and the mine technical services offices to meet with the geologists were also part of the process. The Flin Flon Hudbay's laboratory and the Bureau Veritas laboratory in Vancouver were also inspected by the Qualified Person.

Database Management

As described in the previous sections of this report (10 and 11), drilling, logging, sampling data and assays results are captured and stored in an acQuire database. The acQuire platform provides robust data security and long-term data storage solutions. The assay database is administered by the database manager with working copies kept on the local drive of a secure computer and backups placed on a secure location on a Hudbay server. Any requests for edits to the database are made to the database manager who updates all the copies. All .pdf copies of the assay certificates and logs are available on the Hudbay's internal SharePoint website with restricted access.

Database Review

The drill holes samples used for the mineral Resource estimate included Zn, Cu, Au, Ag, Fe, Pb, As, Ni, and bulk density (SG). The drill database was directly imported into MineSight. Under the direct supervision of the Qualified Person, reviews of the drill holes to look for obvious errors were performed along with random checks of the acQuire database to ensure honouring of the original data such as the collar location, the downhole survey data and the assays results. Minor errors in the downhole surveys data of a few holes were detected and immediately corrected. It is the QP's opinion that the data is adequate and acceptable for use in the estimation of mineral resources.

12.2 WIM DEPOSIT

Hudbay has not performed any work on the Wim property since 1991. The data verification used by the previous owners (i.e. Murgor Resources and Alexandria Minerals) are summarized in this section but more details can be found in Alexandria Minerals 2015 technical report available on SEDAR.

Site Visits

Paul Palmer, P.Geo. P. Eng., of Golder who prepared the mineral resource estimates for Wim, visited the property on February 7, 2008. At the time, two drill rigs were active. During the visit, Golder confirmed that the Murgor core log information matched the information that was observed in the drill core recovered. Sulphide mineralization (chalcopyrite, sphalerite and pyrite and pyrrhotite) was observed in each of the drill holes that were in quantities consistent with the logging and the general mineralization for the Wim deposit.

Database Management

Drilling, logging, sampling data and assays results are captured and stored in a Gemcom Access database.

Database Review

As part of the data verification checked by Golder, a review of the historical log, historical drill hole database and historical assay certificates was completed. The entire assay database was checked by Golder and corrections made after an initial 5% check of the database showed an unacceptable error rate.

Hand written assay certificates were entered into an electronic database by Murgor and compared with an electronic version of drill logs known as Diamond Drill Record. Direct comparison of samples from each electronic file uncovered typographical errors corresponding to sample numbers, location of samples and assay values. Obvious errors detected were corrected by Golder after referring to photocopies of the original data. Discussions with Murgor solved the remaining errors.

In addition to this comparison, there were three occurrences of samples whose “From” was not equal to the previous “To” field. These were discussed with Murgor and an agreed upon correction made. This check also revealed additional samples reported at the end of two holes in the Diamond Drill Records which overlapped with existing samples in the hole. These were eliminated from the database as they were considered as a type of check assay.

Observations during the February 2008 site visit by Golder and data validation procedures completed for the 2008 mineral resource update indicate that industry standard practices were followed for the QAQC program and Murgor’s 2008 Gemcom Access database is acceptable to be used in a resource evaluation

12.3 SNOW LAKE GOLD PROPERTIES

The data verification used by the previous owners are summarized in this section, but more details can be found in the report from Technical Report produced by Genivar for Alexis Resources in 2011 and available on SEDAR.

Site Visits

Todd McCracken, P. Geo, for WSP, conducted a site visit in November 2018. During the site visit, the following validation tasks were completed:

- Transfer of all available digital records to WSP;
- Review of the state of geological/mineralization knowledge;
- Review of the drilling quality assurance/control (QAQC) procedures;
- Review of survey techniques for collar positions and down-hole surveys;
- Review of the core handling and sampling procedures;
- Review of the SG/bulk density determination procedures;
- Review of database integrity and storage procedures;
- Inspection of the core logging and core storage facilities.

Database Review

WSP reviewed the process completed by Micon (*Lewis, 2009*) to verify completeness and reliability of electronic database files; Micon found no errors. Consequently, it can be reasonably assumed that the database is relatively free of errors.

Based on the technical review of data collection, quality control procedures, and inspection of previous studies, the QP is satisfied that both the integrity and quality of the mineral resource database met the industry norms and formed a good basis upon which the inferred mineral resource estimates can be reported.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 STALL CONCENTRATOR

The Stall concentrator is an operating plant running at steady state and as a result, several of the initial metallurgical testing and assumptions have been revised to reflect the operating experience and performance of the plant over the past six years of operation in processing the ore produced from the Lalor mine. Since the metallurgical blend from Lalor is not expected to materially change over the life of mine, it is appropriate to assume that historical performance at Stall is expected to continue in the future.

The Stall concentrator metallurgical performance milling of Lalor ore from October 2012 to December 2018 is shown in Table 13-1.

TABLE 13-1: STALL CONCENTRATOR HISTORICAL PLANT HEAD ASSAY AND METAL RECOVERY

Year	Head Assays				Metal Recoveries			
	Au (g/t)	Ag (g/t)	Cu%	Zn%	Au%	Ag%	Cu%	Zn%
2012*	0.75	26.78	0.33	8.91	46.5	55.1	68.5	96.4
2013	1.38	14.47	0.6	8.66	59.6	56.5	77.7	94.9
2014	2.27	23.85	0.88	8.48	61.3	57.7	81.13	92.31
2015	2.53	21.29	0.71	8.21	55.8	54.75	84.47	90.81
2016	2.25	21.68	0.63	7.03	57.39	56.42	82.17	92.82
2017	1.9	22.67	0.65	7.79	55.92	56.45	81.77	92.34
2018	2.13	25.26	0.71	6.39	57.51	59	84.91	92.85

*Stall Concentrator began to produce Zn and Cu concentrate from Lalor ore since Oct 2012.

Currently, the Stall concentrator is producing a copper concentrate grade of 21% copper at 83 to 85% recovery and a zinc concentrate grade of 51% zinc at 90 to 95% recovery. Gold and silver are recovered to the copper concentrate as co-products. Lalor ore has about 0.2 to 0.3% lead head grade and since the mill process is not configured to separate the lead from copper, lead reports to the copper concentrate. Lead grade in copper concentrate ranges from 5 to 10% and is dependent on the lead head grade and copper/lead ratio. The smelter charges a penalty for lead in copper concentrate and no economic value is received from lead.

The copper concentrate produced at Stall concentrator contains 3 to 6% zinc primarily due to the liberation. According to previous mineralogy studies, there is a certain amount of sphalerite inevitably locked in chalcopyrite in ultrafine size fractions.

The Lalor ore base metal properties are not expected to vary significantly from the previous four years of milling and it is appropriate to assume that the metal recoveries will remain in the 80 to 85% range for copper and 90 to 95% range for zinc for the remaining LOM.

The yearly LOM planned metal recoveries, shown in Table 13-2 were calculated using Hudbay's in-house metallurgical model that considers the relationship of metal grade versus recovery from historical data at optimal operating days.

TABLE 13-2: LOM STALL CONCENTATOR METAL RECOVERY AND CONCENTRATE GRADE FORECAST

Year	Ore Grade Forecasts				Metal Recovery Forecasts			
	Au (g/t)	Ag (g/t)	Cu%	Zn%	Au%	Ag%	Cu%	Zn%
2019	2.41	22.96	0.63	5.43	54.9	55.94	84.02	92.75
2020	2.12	28.57	0.62	6.37	56.54	50.46	83.37	93.94
2021	2.75	28.23	0.64	6.18	52.74	50.77	84.33	93.57
2022	2.35	26.04	0.58	5.92	50.58	51.41	82.99	93.68
2023	2.78	24.48	0.51	5.94	54.77	53.78	81.55	94.09
2024	2.5	24.66	0.46	5.93	53.48	52.07	79.99	94.46
2025	4.7	28.35	0.51	3.98	51.97	59.97	84.36	90.8
2026	4.69	26.15	0.53	3.4	51.62	61.9	85.19	88.37
2027	4.52	28.15	0.56	4.28	51.15	55.81	85.19	90.86
2028	4.66	37.57	0.72	4.82	51.19	44.81	87.52	92.97

Additional information about the relevant testing completed on Lalor mineralization prior to processing ore at the Stall concentrator and including work on Gold Zone material were covered in the previous technical report, dated March 29, 2017. The current LOM plan does not specifically target mining of Gold Zone material unless it is in contact with base metals, which is consistent with previous mining practices at Lalor since 2012. Hudbay intends to preferentially process gold rich ore at the New Britannia mill that is planned to be refurbished and re-commissioned by the end of 2021.

13.2 PEN II PROCESSING PFS TESTWORK

It is anticipated that Pen II Zone ore will be milled at the Stall Concentrator as a blend with ore from the Lalor Mine. The Lalor Mine has been in production for five years, and the metallurgical characteristics are well understood. The expected blend ratio is 85% Lalor and 15% Pen II Zone, based on the planned mining rate for Pen II relative to the Lalor mining rate.

The primary objective of the test program was to determine the recoveries and concentrate grades that can be expected from milling a blend of Pen II and Lalor ore. Given that Pen II ore is the minor component in the blend, and that the mineralogy is simple, no significant changes will be required to the flowsheet which is currently in place at the concentrator.

The metallurgical program for Pen II Zone was completed at Blue Coast Research. The program included mineralogical analysis, Bond work index tests, batch rougher and cleaner flotation tests, batch rougher tests on variability composites, locked cycle tests and analysis of concentrates for deleterious elements.

Ore Samples and Composite Preparation

Hudbay geology personnel provided guidance on selection of the most appropriate intersections to use for the test program. Waste dilution in the amount of 15% of the ore weight was incorporated into the composites. Lalor mill feed samples were crushed and blended into one composite. Head assays for the drill core composites and Lalor mill feed are shown in Table 13-3.

A total of nine composites were prepared from the Pen II zone drill core composites. An overall composite was prepared for the bulk of the initial work, and for blending with Lalor mill feed in a ratio of 15% Pen II to 85% Lalor. Variability composites based on spatial distribution within the deposit were prepared to determine the response of these to flotation conditions developed for the overall composite.

TABLE 13-3: DRILL CORE AND LALOR MILL FEED COMPOSITE HEAD ASSAYS

Analyte Units	Cu %	Pb %	Zn %	Fe %	Ag g/t	As %	S %	Au g/t	MgO %	MgO %
PEN 211 Comp	0.78	0.01	8.09	32.09	5.35	0.05	30.30	0.12	0.46	0.61
PEN 215 Comp	0.37	0.03	4.68	14.35	5.40	0.02	12.00	0.10	0.82	0.98
PEN 216 Comp	0.86	0.04	6.66	19.74	13.00	0.15	18.43	0.30	0.82	1.10
PEN 220 Comp	0.17	0.04	16.75	18.88	7.50	0.01	23.32	0.10	0.53	0.65
PEN 221 Comp	0.67	0.02	19.41	18.88	5.80	0.00	23.58	0.08	0.81	1.22
PEN 225 Comp	0.30	0.02	5.08	12.25	7.20	0.02	11.09	0.13	0.47	0.63
PEN 213 Comp	0.34	0.02	4.85	14.76	7.20	0.05	12.51	0.16	0.88	1.10
PEN 223 Comp	0.27	0.02	11.10	16.68	5.70	0.09	18.13	0.22	0.38	0.55
PEN 224 Comp	0.33	0.10	6.18	15.50	19.50	0.10	14.41	0.73	0.74	0.87
PEN 227 Comp	0.33	0.00	7.18	24.82	2.80	0.00	20.42	0.07	0.70	0.86
PEN 228/9 Comp	0.51	0.01	8.68	22.98	4.90	0.02	21.28	0.14	0.40	0.48
Lalor Mill Feed Comp	0.84	0.17	5.70	16.18	19.40	0.04	18.25	1.18	2.73	6.94

Minerology

Copper occurs almost exclusively as chalcopyrite. Chalcopyrite grain sizes are quite fine with P_{80} sizes in the 10 micron range for the early work and 8 micron for the current test samples. Chalcopyrite liberation was reasonably good, averaging 67% liberated/free. Association with sphalerite averages 16% and varies from 1% to 40%. A copper concentrate re-grind will be required for selectivity, as is the case for Lalor ores currently treated at Stall.

Sphalerite is somewhat coarser than the chalcopyrite with P_{80} sizes in the 15 to 20 micron range in the early samples and 12 micron in the current samples. Sphalerite liberation was above 90%. Gahnite was identified in three samples. On average, 3.5% of the zinc occurred as gahnite in the drill core samples. This is not unusual for ores in this area. Gahnite is non-recoverable and is refractory to acid digestion, so it is not reported in the chemical analyses.

Grindability

Standard Bond ball mill work index tests were completed on samples of Lalor mill feed and the Pen II Zone overall composite with a closing screen size of 150 micron. The two ore types were similar in hardness, the Pen II BWi was 13.9 kwh/t, and the Lalor BWi was 13.4 kwh/t.

Batch Flotation Testwork

A series of scoping tests were run on samples of Lalor mill feed, Comp 9, and blends of Lalor and Pen II to establish reagent addition rates, flotation times and mass pull rates. Starting points were based on plant practice and previous Lalor flotation testing at SGS in 2014. MIBC was used as frother, Aerophine 3418A as copper collector, Aero 7279 as zinc collector, and lime for pH control. Pen II Zone copper recovery appeared to be reaching an upper limit near 80%, compared to 90% for Lalor mill feed. This is likely due to the disparity in copper head grades between the ore types.

Zinc rougher tests were completed on tailings from eight of the batch copper rougher tests. Zinc floated readily from all samples. Zinc tailings grades were consistently low, regardless of head grade. Zinc metallurgy in the plant will probably not be affected by inclusion of Pen II Zone ore in the blend.

Four cleaner flotation tests were completed to establish flotation times for the locked cycle tests.

Variability Tests

Batch rougher flotation tests were completed on six spatial composites. Metal grades and recoveries were directly related to head grades as would be expected. All rougher flotation results are summarized in Table 13-4.

TABLE 13-4: BATCH ROUGHER FLOTATION RESULTS

Test	Ore	Head Grades, %		Cu Rougher Flotation							Zn Rougher Flotation		
				Aeration	Float		Assays, %		Recovery, %			Assay, %	Rec, %
		Cu	Zn	min	min	Wt %	Cu	Zn	Cu	Zn	Wt %	Zn	Zn
Lalor Mill Feed, Overall Composites and Lalor/Pen Blends													
F-1	Lalor MF	0.88	5.68	0	5	20.6	4.1	7.2	95.4	26.2			
F-2	Lalor MF	0.85	5.65	0	3	16.0	6.0	6.3	80.8	17.8	12.8	35.7	80.8
F-3	Pen 10_20	0.45	8.57	0	5	6.1	6.9	7.1	74.5	5.0			
F-4	Pen 10_20	0.42	8.95	5	4	4.3	7.1	8.6	73.3	4.1	30.8	27.5	94.7
F-5	Lalor MF	0.84	5.74	0	6	17.8	4.4	8.8	92.9	27.4	11.1	36.6	70.9
F-6	Lalor MF	0.84	5.70	0	6	18.3	4.2	7.3	92.1	23.3	11.8	36.1	74.6
F-7	Pen 10_20	0.48	8.93	5	6	7.2	5.2	11.5	78.8	9.3	31.0	25.8	89.6
F-8	Pen 10_20	0.46	8.57	5	6	7.3	5.0	10.5	79.0	9.0	30.6	25.2	89.9
F-10	Lalor_PL10	0.79	6.17	4	6	17.0	4.2	8.8	91.2	24.2	13.6	33.4	73.8
F-11	Lalor_PL20	0.77	6.08	1	6	16.0	4.4	9.7	91.2	25.4	14.7	30.0	72.6
F-12	Pen 10_20	0.46	8.59	2	7	6.9	5.5	9.6	83.2	7.7			
Pen Variability Composites													
F-16	Pen Comp 4	0.23	16.30	2	4	2.4	5.4	14.8	55.8	2.2	40.7	39.0	97.4
F-17	Pen Comp 6	0.54	6.22	2	5	7.7	5.5	5.7	78.5	7.1	19.2	28.8	89.0
F-18	Pen Comp 3	0.46	7.92	2	6	4.1	8.1	12.8	72.6	6.6	22.5	32.5	92.4
F-20	Pen Comp 5	0.79	7.03	2	8	9.2	7.3	8.2	85.1	10.7	24.8	24.9	87.7
F-21	Pen Comp 7	0.32	8.74	2	4	5.1	4.4	9.5	70.0	5.6	32.3	25.1	92.9
F-22	Pen Comp 8	0.50	9.17	2	7	7.2	5.1	11.5	72.6	9.0	29.7	27.5	88.9

Locked Cycle Tests

Two locked cycle tests (LCT) were completed on blends of Lalor mill feed and Pen II composites. Results are summarized in Table 13-5. The first test (LCT-1) produced results in line with previous locked cycle testing done at SGS on Lalor ore. Given Hudbay's operating experience with this material, differences between the LCT results and plant performance are likely to results in LCT showing lower concentrate grades, higher zinc losses to copper concentrate and higher copper recoveries.

TABLE 13-5: LCT-1, PROJECTED METALLURGY BASED ON CYCLES 4-6

Product	Weight		Assays						% Distribution					
	g	%	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	MgO (%	S (%)	Cu	Pb	Zn	Au	MgO	S
Cu Clearner 3 Concentrate	225.0	3.74	18.28	2.98	8.76	19.06	2.84	2.84	88.66	74.47	5.32	59.97	2.19	2.19
Zn Clearner 2 Concentrate	672.9	11.18	0.23	0.04	49.66	0.42	0.08	0.08	3.30	2.73	90.11	3.94	0.19	0.19
Zn Clearner 1 Tail	303.8	5.05	0.33	0.11	2.94	1.35	1.82	1.82	2.16	3.88	2.41	5.71	1.89	1.89
Rougher Tail	4,816.1	80.03	0.06	0.04	0.17	0.45	5.80	5.80	5.88	18.93	2.16	30.38	95.72	95.72
Calculated Feed	6,017.8	100.00	0.77	0.15	6.16	1.19	4.85	4.85	100.00	100.00	100.00	100.00	100.00	100.00

The second LCT did not reach an acceptable level of stability and produced low concentrate grades compared to LCT-1.

Projected Metallurgy

Ultimately, the historical plant recovery models for Lalor provide more realistic estimates of expected plant metallurgy with Pen II Zone and the Lalor recovery model for zinc will be used while the copper and gold recovery models will be adjusted downward to account for the lower head grades in Pen II Zone assuming a blend of the two ore types in a ratio of 85% Lalor, 15% Pen II. Table 13-6 presents the projected metallurgy of Pen II and Lalor ore.

TABLE 13-6: PROJECTED METALLURGY FOR PEN II ZONE AND LALOR

		Assays, g/t or Percent					Recovery, Percent				
	Wt%	Au	Ag	Cu	Zn	Pb	Au	Ag	Cu	Zn	Pb
Pen Zone											
Heads	100	0.29	7.8	0.40	7.81	0.10					
Cu Concentrate	1.6	9.3	192.0	19.3	6.4	3.7	51.4	39.7	77.8	1.3	59.0
Cu Tailings	98.4	0.14	4.8	0.09	7.8	0.04	48.6	60.3	22.2	98.7	41.0
Zn Concentrate	14.0	0.4	13.9	0.4	51.7	0.2	17.0	24.9	13.1	92.8	23.3
Final Tailings	84.4	0.11	3.3	0.04	0.5	0.02	31.6	35.4	9.1	5.9	17.7
Lalor LOM											
Heads	100	2.78	26.4	0.64	4.32	0.26					
Cu Concentrate	2.6	61.6	593.3	20.5	3.62	5.8	57.1	57.9	82.6	2.2	57.2
Cu Tailings	97.4	1.22	11.4	0.11	4.3	0.11	42.9	42.1	17.4	97.8	42.8
Zn Concentrate	8.1	4.5	51.3	0.8	49.0	0.8	13.0	15.7	10.6	91.6	24.1
Final Tailings	89.3	0.93	7.8	0.05	0.3	0.05	29.8	26.5	6.9	6.2	18.8
Blend, Lalor+Pen Zone (85/15)											
Heads	100	2.41	23.64	0.60	4.84	0.24					
Cu Concentrate	2.4	53.7	533.1	20.3	4.0	5.4	56.3	55.1	81.9	2.0	57.4
Cu Tailings	97.6	1.06	10.4	0.11	4.9	0.10	43.7	44.9	18.1	98.0	42.6
Zn Concentrate	9.0	3.9	45.7	0.8	49.4	0.7	13.6	17.0	11.0	91.8	24.0
Final Tailings	88.6	0.81	7.1	0.05	0.34	0.0	30.1	27.8	7.2	6.2	18.6

Minor Elements and Concentrate Penalties

The final concentrates from LCT-1 were submitted for minor element scans to identify any deleterious elements. Other than lead and zinc, no penalties for minor elements are expected.

Based on existing smelter contracts, penalties in the range of $22 to $28 per tonne of copper concentrate may apply due to above limit content of lead, zinc or moisture.

Further Work

The initial program scope was complete at the time of report writing. There will be some further work using Aminpro methodology to develop recovery models based on rougher and cleaner flotation kinetics.

13.3 NEW BRITANNIA MILL HISTORICAL PERFORMANCE AND PFS TESTWORK

A laboratory program was conducted at SGS in 2015-2016 on samples from Zone 25 and Zone 27 from Lalor mine. The objective of the program was to develop a flowsheet, design criteria, and metallurgical forecast information for these ore types. Hudbay geology provided guidance on the selection of core samples for the program. Melis Engineering provided technical guidance for the test program, and BBA completed an audit of the program after its completion.

The program included mineralogy, gravity concentration, comminution, flotation, leaching, carbon-in-pulp (CIP) modeling, cyanide destruction and environmental characterization of tailings. The objectives of the flotation and leaching testwork were to determine:

1. If saleable copper concentrates could be produced.
2. If removal of copper by flotation prior to the leach was necessary, and whether it would improve the performance of the leach circuit (copper minerals are known to be problematic in cyanide leach circuits).

Metallurgical Samples and Composite Preparation

In Zone 25, 20 composite samples and a waste sample were prepared from eleven drill holes. These were used to produce an overall diluted composite and four variability composites representing a range of copper values. The Zone 25 overall diluted composite assayed 5.11 t/t Au, 40.3 g/t Ag, and 0.22% Cu.

In Zone 27, nine composite samples and a waste sample were prepared from seven drill holes. These were used to produce an overall diluted composite and three zonal composites. The Zone 27 overall diluted composite assayed 8.41 g/t Au, 20.5 g/t Ag, and 4.28% Cu.

The overall diluted composites from the two zones were used to produce a “blend” composite. The blend ratio was 67% from Zone 25 and 33% from Zone 27.

Minerology

Sub-samples of the Zone 25 and Zone 27 overall composites were submitted for mineralogical analysis by QEMSCAN to identify minerals and their liberation.

The Zone 25 composite was composed mainly of quartz, amphibole/pyroxene and feldspars. Sulphides accounted for less than 5% of the mineral mass. 81.6% of the chalcopyrite was free and liberated.

The Zone 27 composite was composed mainly of quartz, amphibole/pyroxene, chalcopyrite and garnet. Sulphides accounted for more than 20% of the mineral mass. 96.6% of the chalcopyrite was free and liberated.

Comminution Testwork

Five composites were submitted for Bond rod mill work index tests, and seven of the composites were submitted for Bond ball mill work index tests. Results are summarized in Table 13-7. The rod mill results correspond to the soft to medium-soft range of hardness in the SGS data base. The ball mill results correspond to the medium to very hard range of hardness. However, it was recognized that the high hardness values were likely due to the presence of difficult to screen silicates.

TABLE 13-7: BOND WORK INDEX TEST SUMMARY

	Work Index, kwh/t	
Composites	**Rod Mill at 14 mesh**	**Ball Mill at 150 mesh**
Composite A	13.3	21
Composite B	11.4	17.7
Composite C		15.4
Composite D		16.1
Zone 25 Gangue	11.8	17.1
Zone 27 Main central	9.3	14.5
Zone 27 Gangue	10.2	18

Gravity Concentration

A single gravity concentration test was completed on a sample of Zone 27 overall composite to determine the amenability of this sample to gold and silver recovery by gravity separation. 18% of the gold and 5.4% of the silver were recovered. These are below the threshold values for consideration of a gravity circuit, but any future work on this ore should include gravity testing.

Flotation

Rougher flotation conditions were optimized first on the blend composite. Test variables included grind size, pH and collector type. Based on previous test programs with this ore it was known that a depressant would be necessary to control floatable gangue minerals, and Depramin 347 CMC was used for this purpose. Collector Aero 3418A provided the best copper and gold recoveries and was used for all subsequent tests. Batch rougher flotation results are summarized in Table 13-8.

Following the batch rougher tests, a total of 20 cleaner tests were completed on the remaining composites to confirm that saleable copper concentrates could be produced (Table 13-9).

The initial tests on Zone 25 composites as well as the blend composites indicated very high lead grades in the copper concentrates, which would be expected given the high ratio of lead to copper in these composites. It was found that a combination of Depramin addition to the cleaner stages, and a higher pH (pH>11.0) successfully depressed the lead in the cleaners. This strategy was carried forward into the locked cycle tests.

Excellent recoveries and concentrate grades were obtained for all composites with the exception of Zone 27 Main North. This composite had a high copper head grade of 6.79% Cu. Both the collector dosage and flotation time used in the test were probably too low for this grade. It's unlikely that the plant would see head grades this high, but nonetheless there will be work done in the future to optimize conditions for high grade ore types.

TABLE 13-8: BATCH ROUGHER FLOTATION RESULTS

Test #	Purpose	Product	Time (Min)	Wt (%)	Assays (g/t, %)							Distribution (%)						
					Au	Ag	Cu	Pb	Fe	Zn	S	Au	Ag	Cu	Pb	Fe	Zn	S
F1 Blend Comp	Baseline conditions	Cu Rougher 1	1	3.55	50.4	307	28.6	0.56	31.4	0.62	33.0	34.5	38.8	66.9	11.2	12.7	12.5	43.6
		Cu Rougher 1-2	2	5.30	55.2	321	26.0	1.70	30.4	0.81	32.0	56.3	60.6	90.7	50.7	18.4	24.3	63.1
		Cu Rougher 1-3	4	6.44	56.6	323	22.8	2.33	29.0	1.01	29.4	70.2	74.1	96.4	84.4	21.3	36.8	70.4
		Cu Rougher 1-4	8	8.26	49.4	279	18.1	1.97	26.3	1.03	24.6	78.6	82.0	98.2	91.7	24.8	48.1	75.8
		Tail		91.7	1.21	5.5	0.03	0.02	7.18	0.10	0.71	21.4	18.0	1.75	8.26	75.2	51.9	24.2
		Head (calc):		100.00	5.2	28.1	1.5	0.2	8.8	0.2	2.7							
F2 Blend Comp	As F1, substitute 3418A with Danafloat 262	Cu Rougher 1	1	1.72	124	350	26.9	0.16	32.0	0.48	33.9	35.9	21.4	29.2	1.52	6.15	4.40	21.3
		Cu Rougher 1-2	2	2.64	99.6	302	25.1	0.18	30.3	0.56	31.4	44.4	28.4	41.9	2.70	8.97	7.97	30.4
		Cu Rougher 1-3	4	4.29	77.3	254	22.9	0.22	28.9	0.73	28.8	56.0	38.9	62.2	5.16	13.9	16.8	45.2
		Cu Rougher 1-4	8	7.49	55.5	214	19.4	0.31	27.2	0.89	25.2	70.2	57.1	91.8	13.0	22.9	35.8	69.2
		Head (calc):		100.00	5.9	28.1	1.6	0.2	8.9	0.2	2.7							
F3 Blend Comp	As F1, replace 3418A with TNC-312	Cu Rougher 1	1	1.87	62.5	291	26.4	0.09	32.0	0.47	-	21.8	18.3	31.6	0.97	6.55	4.65	-
		Cu Rougher 1-2	2	2.80	63.1	266	25.9	0.10	31.0	0.53	-	33.0	25.1	46.7	1.54	9.54	7.94	-
		Cu Rougher 1-3	4	4.39	58.4	243	23.9	0.13	29.5	0.70	-	47.8	36.1	67.4	3.21	14.2	16.3	-
		Cu Rougher 1-4	8	7.01	48.8	217	20.4	0.19	27.4	0.83	-	63.8	51.4	91.6	7.54	21.0	31.0	-
		Head (calc):		100.00	5.4	29.6	1.6	0.2	9.1	0.2	-							
F4 Blend Comp	As F1, add Aero 208 to increase Au/Ag recovery	Cu Rougher 1	1	3.79	56.7	336	28.8	0.45	32.7	0.70	-	40.6	44.2	70.1	9.2	13.7	13.9	-
		Cu Rougher 1-2	2	5.27	60.7	354	26.7	1.92	32.0	0.88	-	60.6	64.9	90.5	54.7	18.6	24.4	-
		Cu Rougher 1-3	4	6.19	59.6	348	24.0	2.44	30.6	1.04	-	69.8	74.9	95.5	81.6	20.9	33.7	-
		Cu Rougher 1-4	8	8.38	48.6	283	18.1	1.99	26.3	0.96	-	77.1	82.5	97.7	90.1	24.4	42.2	-
		Head (calc):		100.00	5.3	28.8	1.6	0.2	9.1	0.2	-							
F5 Blend Comp	As F1, finer primary grind	Cu Rougher 1	1	3.40	81.5	449	27.1	0.66	32.6	0.73	-	52.9	52.5	60.1	12.5	12.5	13.8	-
		Cu Rougher 1-2	2	5.45	65.6	374	25.4	2.16	31.2	0.94	-	68.1	70.0	90.4	65.8	19.1	28.6	-
		Cu Rougher 1-3	4	6.98	58.2	336	21.2	2.21	28.9	1.07	-	77.4	80.7	96.2	86.2	22.7	41.4	-
		Cu Rougher 1-4	8	8.80	48.7	284	17.1	1.84	25.8	1.01	-	81.6	85.9	97.9	90.3	25.5	49.4	-
		Head (calc):		100.00	5.3	29.1	1.5	0.2	8.9	0.2	-							
F6 Blend Comp	As F5, high pH to reduce Au/Ag recovery	Cu Rougher 1	1	3.56	77.5	404	29.7	0.23	28.7	0.58	-	48.2	49.6	68.6	4.2	11.2	11.1	-
		Cu Rougher 1-2	2	4.68	72.1	371	28.1	0.37	28.7	0.77	-	59.0	59.9	85.6	8.9	14.7	19.5	-
		Cu Rougher 1-3	4	5.78	67.1	340	24.7	0.90	27.2	0.94	-	67.8	67.9	92.8	26.8	17.2	29.3	-
		Cu Rougher 1-4	8	7.69	56.7	293	19.4	1.49	24.6	1.10	-	76.3	77.7	96.6	59.0	20.7	45.4	-
		Head (calc):		100.00	5.7	29.0	1.5	0.2	9.1	0.2	-							
F7 Blend Comp	As F5, 3418A / TNC-312 blend	Cu Rougher 1	1	3.78	79.1	387	26.6	0.66	31.9	0.81	-	55.6	47.8	60.1	12.8	13.1	17.3	-
		Cu Rougher 1-2	2	5.22	67.5	346	25.8	1.23	31.4	0.96	-	65.5	59.0	80.6	32.8	17.8	28.3	-
		Cu Rougher 1-3	4	6.74	59.5	324	21.6	2.30	29.2	1.14	-	74.5	71.4	87.0	79.5	21.4	43.4	-
		Cu Rougher 1-4	8	9.45	46.4	256	15.8	1.78	25.3	1.04	-	81.6	79.3	89.2	86.1	26.1	55.5	-
		Head (calc):		100.00	5.4	30.6	1.7	0.2	9.2	0.2	-							
F17 Zone 27 Main Central	Increase collector / retention time to reduce tailings grade	Cu Rougher 1	1	14.9	46.3	104	28.9	0.04	32.1	0.71	-	57.6	45.2	71.1	35.5	32.1	25.0	-
		Cu Rougher 1-2	2	21.1	40.4	97.0	22.0	0.02	32.7	1.71	-	71.1	59.7	93.4	45.0	45.7	49.8	-
		Cu Rougher 1-3	6	24.5	40.0	101	8.77	0.03	28.5	2.42	-	81.6	72.5	98.4	50.9	52.2	69.3	-
		Cu Rougher 1-4	10	25.3	39.9	103	3.59	0.02	21.6	1.37	-	84.1	75.8	98.8	52.0	53.4	71.9	-
		Head (calc):		100.00	12.0	34.3	6.1	0.0	14.9	0.4	-							
F21 Zone 27 Main North	Increase collector / retention time to reduce tailings grade	Cu Rougher 1	1	24.8	33.5	88.1	25.4	0.09	32.8	0.7	-	76.8	65.5	90.4	63.7	45.6	49.9	-
		Cu Rougher 1-2	2	29.6	30.7	88.4	22.7	0.09	32.4	0.7	-	83.9	78.5	96.5	74.4	53.8	64.2	-
		Cu Rougher 1-3	6	32.4	28.9	87.1	21.1	0.08	31.8	0.8	-	86.6	84.7	98.4	79.1	57.7	74.5	-
		Cu Rougher 1-4	10	33.7	28.1	85.9	20.5	0.08	31.4	0.8	-	87.4	86.7	98.9	80.9	59.1	76.5	-
		Head (calc):		100.00	10.8	33.4	7.0	0.0	17.9	0.3	-							

TABLE 13-9: BATCH CLEANER FLOTATION RESULTS

Test		Calculated Head						Copper Cleaner Concentrate									Copper Rougher Concentrate								
		Assays (g/t, %)						Assays (g/t, %)					Distribution (%)				Assays (g/t, %)					Distribution (%)			
#	Feed	Au	Ag	Cu	Pb	Fe	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag	Cu	Pb	Au	Ag	Cu	Pb	Zn	Au	Ag	Cu	Pb
F8	Blend Comp	5.10	30.4	1.57	0.19	0.19	0.18	62.4	370	26.0	1.76	1.25	68.9	68.7	93.4	53.4	50.2	306	19.4	2.13	1.14	75.4	77.2	94.8	87.6
F20	Blend Comp	5.42	29.6	1.51	0.19	0.19	0.18	71.6	370	28.3	0.65	0.84	65.1	61.6	92.4	17.0	58.3	316	20.6	2.28	0.79	74.8	74.3	94.8	84.2
F22	Blend Comp	5.40	28.3	1.46	0.20	0.20	0.19	73.7	365	29.6	0.43	0.79	59.9	56.6	89.0	9.5	-	-	-	-	-	-	-	-	-
F23	Blend Comp	5.92	29.1	1.50	0.17	0.17	0.17	78.2	285	27.5	0.44	1.01	65.2	48.3	90.7	13.0	61.9	258	20.5	1.64	0.98	72.3	61.4	94.6	67.2
F25	75% Zone 25 / 25% Zone 27	5.94	35.0	1.16	0.24	0.24	0.33	86.5	516	27.6	0.69	0.88	60.2	57.0	91.1	11.5	67.3	429	18.9	2.47	1.12	70.4	71.2	93.8	61.4
F26	50% Zone 25 / 50% Zone 27	6.76	30.1	2.10	0.16	0.16	0.40	54.9	251	29.4	0.35	0.95	57.3	50.4	87.4	12.0	49.8	254	22.8	1.71	1.11	72.8	71.3	94.9	81.5
F27	25% Zone 25 / 75% Zone 27	7.59	25.3	3.04	0.09	0.09	0.46	50.8	161	30.3	0.18	0.97	65.3	53.3	90.7	17.9	47.9	170	25.6	0.68	1.10	76.9	70.4	95.7	82.6
F30	50% Blend / 50% Comp 1	3.80	22.5	1.07	0.19	0.19	3.79	27.9	153	11.0	0.46	4.79	67.1	51.9	87.2	17.2	14.4	86.9	5.23	0.78	6.06	76.0	64.7	90.9	63.7
F9	Zone 25 Overall Diluted	5.03	40.5	0.22	0.33	0.33	0.13	254	1992	14.8	14.3	1.23	69.0	67.3	90.1	59.3	89.6	718	4.77	5.91	0.67	77.8	77.6	92.8	78.3
F18	Zone 25 Overall Diluted	5.02	40.6	0.20	0.29	0.29	0.13	368	2631	21.2	5.27	1.01	61.1	54.0	87.9	15.3	132	1040	6.51	7.26	0.70	75.6	73.7	93.2	72.9
F11	Zone 25 Comp C	7.72	73.4	0.33	0.77	0.77	0.19	312	2767	16.6	12.1	1.40	66.6	62.1	83.5	25.9	134	1229	6.62	10.7	0.88	74.2	71.3	86.0	59.0
F19	Zone 25 Comp C	7.76	70.2	0.29	0.58	0.58	0.18	406	3736	21.4	4.21	1.22	58.8	59.9	82.3	8.2	106	995	4.30	8.50	0.72	83.3	86.1	89.4	89.0
F12	Zone 25 Comp D	7.54	29.2	0.65	0.15	0.15	0.47	184	633	23.3	2.74	2.04	61.8	55.0	91.3	45.3	79.0	287	8.80	1.61	1.36	73.1	68.7	95.0	73.3
F16	Zone 25 Comp D	7.47	26.7	0.63	0.16	0.16	0.47	195	498	24.9	1.21	0.89	57.2	40.8	85.8	16.3	99.1	299	10.4	1.96	2.17	72.8	61.4	90.2	66.3
F10	Zone 27 Overall Diluted	6.55	20.5	3.69	0.02	0.02	0.27	35.8	101	27.8	0.04	1.31	69.2	62.6	95.5	35.8	31.3	93.0	22.7	0.04	1.17	75.3	71.6	96.8	44.7
F24	Zone 27 Overall Diluted	7.18	22.0	4.44	-	-	-	-	-	-	-	-	-	-	-	-	27.6	84.3	26.1	-	-	63.5	63.5	97.4	-
F28	Zone 27 Overall Diluted	7.26	21.3	3.86	0.01	0.01	0.29	34.4	100	26.1	0.04	1.59	69.4	68.9	99.1	36.5	31.3	91.0	21.6	0.03	1.34	76.6	76.0	99.3	41.4
F29	Zone 27 Overall Diluted	6.40	19.4	3.61	0.01	0.01	0.26	35.2	101	27.4	0.03	1.19	70.2	66.4	96.8	29.1	29.6	86.3	22.3	0.03	1.02	72.9	70.1	97.4	33.8
F13	Zone 27 Main Central	10.7	35.0	6.35	0.02	0.02	0.45	37.4	102	30.7	0.03	1.27	66.5	55.4	92.0	34.4	35.6	109	27.7	0.04	1.26	72.4	67.4	94.9	51.2
F17	Zone 27 Main Central	12.0	34.3	6.07	0.02	0.02	0.42	-	-	-	-	-	-	-	-	-	40.0	101	24.4	0.03	1.20	81.6	72.5	98.4	50.9
F14	Zone 27 Upper	3.16	8.8	1.16	0.02	0.02	0.10	46.9	111	26.1	0.16	0.91	61.6	52.5	93.0	39.2	33.9	87.9	16.9	0.12	0.71	70.4	65.9	95.2	44.7
F15	Zone 27 Main North	8.72	26.0	6.79	0.03	0.03	0.40	47.2	99	29.5	0.06	1.01	91.9	64.9	73.7	38.3	40.3	87.7	25.4	0.08	0.96	95.1	69.4	76.9	56.6
F21	Zone 27 Main North	10.8	33.4	6.97	0.03	0.03	0.34	-	-	-	-	-	-	-	-	-	28.9	87.1	21.1	0.08	0.78	86.6	84.7	98.4	79.1

Cyanidation Testwork –Cyanidation Tests

Whole ore (i.e., without pre-flotation) were completed on the various composites. The purpose of these tests was to determine if the removal of copper was essential for operation of the cyanide leach circuit. Cyanide consumptions and gold recoveries were reasonable on the low copper grade composites from Zone 25, but the whole ore leach approach on composites containing Zone 27 material resulted in poor recovery, high cyanide consumption and unacceptably high Cu in solution levels (>400 ppm). This approach is no longer relevant as the production plan is to send all gold and copper ores to New Britannia.

Cyanidation Testwork – Flotation Tailings

A total of 27 rolling bottle leach tests were completed on samples of combined rougher and first cleaner tailings samples from both batch flotation tests and locked cycle tests. The baseline conditions used for these tests were:

- Pulp density = 40% solids
- pH 10.5 – 11.0
- NaCN concentration = 0.5 g/l
- Retention time = 48 hr
- Lead nitrate addition = 50 g/t
- Pre-Aeration time = 8 hours
- Leach time = 48 hr

Based on previous work pre-aeration had a positive impact on gold recovery and this step was included in all the tailings leach tests. Also, lower NaCN levels (0.25 g/l and 0.35 g/l) had been tested in the earlier work and were slightly less effective than 0.5 g/l, so were not tested again. Tailings cyanidation results are summarized in Table 13-10.

Deleterious Elements and Penalties

Detailed chemical analyses were completed on eight of the copper concentrates produced in the test program, as shown below. Based on these analyses, no penalties are expected with the possible exception of lead, which at times may slightly exceed the penalty limit of 3% lead+zinc as per current concentrate agreements.

TABLE 13-10: TAILINGS CYANIDATION RESULTS

Test No.	Sample	Gold					Silver				
		Extraction (%) Time (hours)		Residue	Assay (g/t) Head		Extraction (%) Time (hours)		Residue	Assay (g/t) Head	
		24	48		Calc	Direct	24	48		Calc	Direct
CN-13	F10-Ro Tail & Cl Tail	68.0	74.3	0.59	2.29	2.31	39.9	50.7	4.25	8.62	8.77
CN-14	F14-Ro Tail & Cl Tail	75.4	78.2	0.28	1.26	1.26	59.9	65.3	1.45	4.18	4.34
CN-15	F17-Ro Tail	68.9	72.3	0.75	2.71	2.55	52.4	64.2	3.25	9.09	11.1
CN-16	F21-Ro Tail	64.6	72.9	0.28	1.03	2.05	32.9	51.0	3.25	6.6	6.70
CN-17	LCT-1 Comb Tail	80.6	80.7	0.46	2.35	2.51	62.5	65.2	4.50	12.9	11.6
CN-18	LCT-1 Comb Tail	79.8	78.9	0.48	2.27	2.51	60.2	60.3	4.90	12.3	11.6
CN-19	LCT-1 Comb Tail	80.3	79.1	0.44*	2.24	2.51	n/d	n/d	n/d	n/d	11.6
CN-20	LCT-1 Comb Tail	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	n/d	11.6
CN-21	LCT-1 Comb Tail	n/d	81.5	0.46	2.49	2.51	n/d	n/d	n/d	n/d	11.6
CN-25	LCT-2 - Comb Tail	85.3	85.0	0.27	1.76	1.94	63.5	69.2	4.50	14.6	21.9
CN-26	LCT-3 - Comb Tail	86.8	83.9	0.39	2.43	2.29	53.7	56.4	11.8	27.1	26.8
CN-27	LCT-4 - Comb Tail	82.3	81.9	0.48	2.65	3.21	75.1	79.7	2.50	12.3	13.6
CN-28	LCT-5 - Comb Tail	*81.0*	*@ 48 h*	0.40	2.08	1.91	*64.4*	*@ 48 h*	4.40	12.4	11.9
CN-29	LCT-5 - Comb Tail	*80.2*	*@ 36 h*	0.39	1.95	1.91	*63.4*	*@ 36 h*	4.20	11.5	11.9
CN-30	LCT-5 - Comb Tail	*78.3*	*@ 24 h*	0.42	1.94	1.91	*62.5*	*@ 24 h*	4.30	11.5	11.9
CN-31	LCT-5 - Comb Tail	*76.1*	*@ 16 h*	0.47	1.95	1.91	*57.8*	*@ 16 h*	4.80	11.4	11.9
CN-32	LCT-5 - Comb Tail	*74.1*	*@ 8 h*	0.52	2.01	1.91	*56.0*	*@ 8 h*	4.90	11.1	11.9
CN-33	F-22 - Comb Tail	79.6	-	0.37	1.79	1.87	62.4	-	4.20	11.2	10.9
CN-34	F-22 - Comb Tail	79.9	-	0.36	1.76	1.87	63.2	-	4.00	10.9	10.9
CN-35	F-24 - Comb Tail	-	74.0	0.79	3.02	3.14	-	62.4	3.60	9.6	9.60
CN-36	F-25 - Comb Tail	-	85.6	0.35	2.43	2.38	-	69.0	4.90	15.8	16.2
CN-37	F-26 - Comb Tail	-	78.2	0.53	2.43	2.79	-	60.8	5.40	13.8	16.8
CN-38	F-27 - Comb Tail	-	72.2	0.75	2.70	2.94	-	43.3	7.40	13.1	15.3
CN-41	F-29 - Comb Tail	74.2	74.1	0.54	*2.07*	2.19	62.1	66.0	2.20	*5.81*	7.48
CN-42	F-29 - Comb Tail	71.9	72.9	0.60	*2.13*	2.19	59.9	63.3	2.40	*5.98*	7.48
CN-43	LCT-6 - Comb Tail	78.8	78.5	0.36	*1.67*	1.85	67.0	65.3	3.40	*10.3*	12.1
CN-44	LCT-6 - Comb Tail	78.4	78.3	0.38	*1.76*	1.85	66.4	64.6	3.60	*10.7*	12.1

Notes: * at 72 hours, n/d - not determined   *24 hr calculated heads*

CIP Modeling

CIP modeling was completed as per standard SGS procedures (SGS, 2015). The kinetic data obtained from these tests were fitted to equations developed by SGS, and used in proprietary CIP and CIL models to predict the gold extraction performance of a CIP or CIL circuit. The following plant configuration and operating parameters were recommended for a CIP plant treating flotation tailings:

- Number of adsorption stages: 4 or 5
- Slurry residence time per stage: 1 hour
- Carbon concentration: 20 to 25 g/l (2.2 to 2.7 t/tank)
- Carbon advance rate: ~1.5 t/day

Cyanide Destruction

Cyanide destruction tests using the SO_2/Air process were carried out following standard SGS procedure of completing batch tests first to confirm applicability and to optimize retention times and reagent requirements. The results showed that the SO_2/Air process is appropriate for use at New Britannia. Pulp from the blend leach test was reduced from 285 ppm CN_T to less than 1 ppm CN_T with 60 minutes retention time at pH 8.6. Reagent requirements were 4.7 g equivalent SO_2, 2.4 g hydrated lime, 0.04 g Cu (added as copper sulphate) and 1.02 g Zn (added as zinc sulphate) per gram CN_{WAD} in the feed.

Locked Cycle Test Results and Recovery Method

A total of six locked cycle tests were completed to confirm the flowsheet, and to generate tailings for the cyanidation testwork. Results of the locked cycle tests are summarized in Table 13-11. Also included in this table are two batch cleaner tests on Zone 27 composites (Test F-10 and F-14) for which locked cycle tests were not required as a single stage of cleaning was sufficient to obtain final concentrate grade. These tests covered a wide range of copper head grades from 0.22% Cu to 3.69% Cu. These flotation results form the basis for the recovery models (Table 13-12).

TABLE 13-11: LOCKED CYCLE TEST RESULTS

	Zone 25			Zone 27		Zone 25 & Zone 27 Blend		
Composite	Overall	Comp C	Comp D	Overall	Upper			
Float Test ID	LCT-2	LCT-3	LCT-4	F-10	F-14	LCT-1	LCT-5	LCT-6
Head Grades								
Au, g/t	5.03	7.81	8.44	6.55	3.16	5.68	4.91	5.36
Ag, g/t	44.90	75.50	30.40	20.50	8.76	28.70	27.90	30.00
Cu, %	0.22	0.31	0.74	3.69	1.16	1.58	1.36	1.58
Pb, %	0.27	0.70	0.10	0.02	0.02	0.18	0.18	0.16
Zn, %	0.13	0.16	0.42	0.27	0.10	0.17	0.16	0.17
Flotation								
Concentrate Grades								
Au, g/t	371	433	212	36	46.9	65.2	76.8	68.0
Ag, g/t	2,776	3,749	676	101	111	336	411	351
Cu, %	22.90	22.50	26.60	27.80	26.10	28.60	31.50	28.20
Pb, %	3.72	3.56	1.23	0.04	0.16	0.45	0.35	0.67
Zn, %	1.08	1.26	1.57	1.31	0.91	0.92	0.68	1.05
Recovery to Concentrate, %								
Wt, %	0.84	1.30	2.60	12.70	4.14	5.20	4.00	5.30
Au	61.70	70.00	65.60	69.20	61.60	59.80	62.60	67.50
Ag	51.70	62.70	58.20	62.60	52.50	61.00	59.00	62.30
Cu	89.00	90.30	94.40	95.50	93.00	94.50	92.90	95.20
Pb	11.50	6.40	31.80	35.80	39.20	12.90	7.90	22.20
Zn	7.10	9.60	9.80	62.00	38.70	27.60	16.80	32.20
Tailings Grades								
Wt, %	99.20	98.70	97.40	87.3	95.86	94.80	96.00	94.70
Au, g/t	1.94	2.37	2.98	2.31	1.00	2.41	1.91	1.84
Ag, g/t	21.90	28.50	13.10	8.80	3.20	11.80	11.90	12.00
Cu, %	0.02	0.031	0.043	0.190	0.060	0.09	0.10	0.08
Pb, %	0.24	0.660	0.070	0.010	0.010	0.17	0.17	0.13
Zn, %	0.12	0.150	0.390	0.100	0.050	0.13	0.14	0.12
Cyanide Leaching								
Extraction %, from tailings								
Au	86.0	83.5	84.0	74.6	72.0	80.5	77.0	80.0
Ag	79.5	58.6	80.9	51.7	53.0	60.2	62.1	70.8
Recovery %, from heads								
Au	32.9	25.0	28.9	23.0	21.9	32.4	28.8	26.0
Ag	38.4	21.8	34.0	19.4	18.6	23.4	25.4	26.8
Leach Residue Grades								
Au, g/t	0.27	0.39	0.48	0.59	0.28	0.47	0.44	0.37
Ag, g/t	4.50	11.80	2.50	4.25	1.50	4.70	4.52	3.50
Carbon Adsorption								
Recovery, %								
Au	99.5	99.5	99.5	99.5	99.5	99.5	99.5	99.5
Ag	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0
Final Tailings Grades								
Wt, %	99.20	98.70	97.40	87.30	95.86	94.80	96.00	94.70
Au, g/t	0.28	0.40	0.49	0.60	0.28	0.48	0.45	0.38
Ag, g/t	7.98	15.14	4.62	5.16	1.84	6.12	5.99	5.20
Cu, %	0.02	0.03	0.04	0.19	0.06	0.09	0.10	0.08
Pb, %	0.24	0.66	0.07	0.01	0.01	0.17	0.17	0.13
Zn, %	0.12	0.15	0.39	0.10	0.05	0.13	0.14	0.12
Overall Recovery, %								
Wt	0.80	1.30	2.60	12.70	4.14	5.20	4.00	5.30
Au	94.5	94.9	94.4	92.1	91.4	92.0	91.3	93.4
Ag	82.4	80.2	85.2	78.0	79.8	79.8	79.4	83.6
Cu	89.0	90.1	94.3	95.5	95.1	94.5	92.9	95.1
Pb	11.5	6.9	31.8	56.4	43.6	10.5	7.8	23.1
Zn	7.1	7.5	9.6	67.7	50.6	27.5	17.0	33.2

TABLE 13-12: RECOVERY MODEL AT RESERVE GRADES

Head Grades	
Gold (g/t)	6.72
Silver (g/t)	23.48
Copper (%)	1.17
Recovery to Copper Concentrate (%)	
Gold	63.2
Silver	55.0
Copper	93.5
Recovery to Doré (%)	
Gold	30.1
Silver	22.8
Overall Recovery (%)	
Gold	93.3
Silver	77.7
Copper	93.5
Assays (%)	
Gold in Doré	26.1
Silver in Doré	69
Copper in Copper Concentrate	27.6

13.4 FLIN FLON CONCENTRATOR

The Flin Flon concentrator is an operating plant running at steady state, with a capacity of 7,200 tpd that has treated the Lalor ore since 2017 in dedicated plant batches.

Historically, the Flin Flon Concentrator treated both Reed ore from the reed mine and 777 ore from the 777 mine from 2014 to 2016 at a blend mix of 7:3, 777/Reed, producing copper and zinc concentrates. However, upon closure of the Reed mine in July 2018, the Flin Flon concentrator now treats primarily 777 ore and Lalor ore in batches as scheduled.

The processing of 777 ore results in a copper concentrate with a grade of 23% copper at 90 to 93% recovery and a zinc concentrate with a grade of 50% zinc at 80 to 85% recovery. Lead head grade ranges from 0.1-0.2% and there is less than 1% lead in the copper concentrate.

In 2017 and 2018, the Flin Flon Concentrator ran a total of ten plant trials for the Lalor ore. The plant trial was generally quite smooth and successful from ore crushing, feed transition, grinding, flotation to filtration. Stable gold, copper and zinc recoveries were obtained after the optimization of operational parameters. The copper concentrate produced from the Lalor ore has a grade of about 18% Cu at 85 to 90% recovery and a zinc concentrate grade of about 53% Zn at 84 to 92% recovery (Table 13-13).

TABLE 13-13: FLIN FLON CONCENTRATOR PERFORMANCE FROM LALOR PLANT TRIALS

Year	Head Assays				Metal Recoveries			
	Au (g/t)	Ag (g/t)	Cu%	Zn%	Au%	Ag%	Cu%	Zn%
2017	2.12	26.03	0.92	6.94	58.44	55.32	85.01	87.63
2018	2.53	24.92	0.89	6.22	60.57	54.94	87.66	84.31

Since the metallurgical blend from Lalor is not expected to materially change over the life of mine, it is appropriate to assume similar metallurgical performance at the Flin Flon concentrator is expected to continue in the future. Similar to the Stall Concentrator, the process is not configured to separate the lead which reports to the copper concentrate. The smelter charges a penalty for lead in copper concentrate and there is no economic value received from lead.

The Flin Flon Concentrator is scheduled to cease operations in the fourth quarter of 2021.

14. MINERAL RESOURCE ESTIMATES

14.1 LALOR MINE

Hudbay prepared an update of the Lalor mine resource model using Leapfrog® version 4.2.3 and MineSight® version 15, two industry standard commercial geological and mining software. The construction of this 3D resource model and the estimation of mineral resources were performed by Hudbay personnel following Hudbay procedures in compliance with best industry standards and the CIM guidelines. The work was supervised and validated by Olivier Tavchandjian, P.Geo., Vice President, Exploration and Geology and Qualified Person for the purpose of the present Technical Report

Modeling Database

As shown in Table 14-1, 3,052 drill holes totaling approximately 527,610 m were included in the Lalor database to support the mineral resource estimate. Amongst these samples approximately 280,400 m were analyzed for Zn, Cu, Au and Ag and density was measured for 74, 299 samples.

TABLE 14-1: DRILLING DATA BY YEAR

Year	Drill Holes	Metres Drilled
2007	39	44,905
2008	69	73,377
2009	76	89,749
2010	28	38,656
2011	19	24,723
2012	82	17,651
2013	299	22,650
2014	381	33,771
2015	515	48,981
2016	325	26,328
2017	587	44,118
2018	632	62,701
Total	**3,052**	**527,610**

The drill hole database was directly imported in MineSight® from acQuire® with a cut-off date for mineral resource estimate purposes of November 15th, 2018. In parallel, the drill hole database was also imported in Leapfrog® for additional and independent comparison and validation. No significant differences in the projection of the drill hole traces were found between the two datasets. Additional validations conducted on the drill hole database are described in section 12.

Modeling of the Mineralized Envelopes

The Lalor mineralized envelopes trend along an azimuth of approximately N320° with a general dip of approximately 37° to the north east. Economic mineralization delineated so far at Lalor occurs in eight semi-massive to massive sulfide lenses typical of VMS deposits (lenses 10, 11, 20, 30, 31, 32, 40 and 42), three conformable gold-rich disseminated to stringer zones (lenses 21, 23 and 25) and as a copper-gold rich feeder (lens 27). The base metal and gold rich zones are stratigraphically stacked and are mostly aligned to the main regional deformation, averaging N320°/37°. The Lalor mine is continuous along a strike length of approximately 1,400 m in north-south direction, approximately 780 m in an east-west direction and with a vertical extent of approximately 760 m.

Geological units of sulphides horizons were created in 3D using core logging data and assay results. These broad geological envelopes were then refined to create base metal and gold rich mineralized envelopes. The distribution of gold mineralization follows two styles: the first one, which is synchronous to the base metal deposition, is directly associated and contained within the base metal mineralized envelopes, typically on their footwall. The second one, posterior to the base metal deposition, is contained outside the sulphides horizons and is typically sulphides poor with the exception of the Au-Cu rich mineralized envelope 27 which is an Au-Cu copper stringer zone.

The construction of the mineralized envelopes was solely based on the type of mineralization intersected. To this effect, assay intervals with high gold grade but no zinc or copper were excluded from base metal lenses and vice versa. Instead, smooth and continuous 3D solids were built using 4% zinc or 1% copper cut-off to guide the construction of the base metal mineralized envelopes while a 2.5 g/t cut-off was used to guide the construction of the gold rich mineralized envelopes.

In a final validation step, these envelopes were loaded into MineSight® in order to ensure proper tagging of the solids to actual drill hole locations. The mineral envelopes were used as hard boundaries in all cases for grade interpolation purposes so as to correctly prevent spreading of sulfide mineralization into the gold zones and vice-versa. Wireframes of the mineralized envelopes are shown in Figure 14-1.

Density

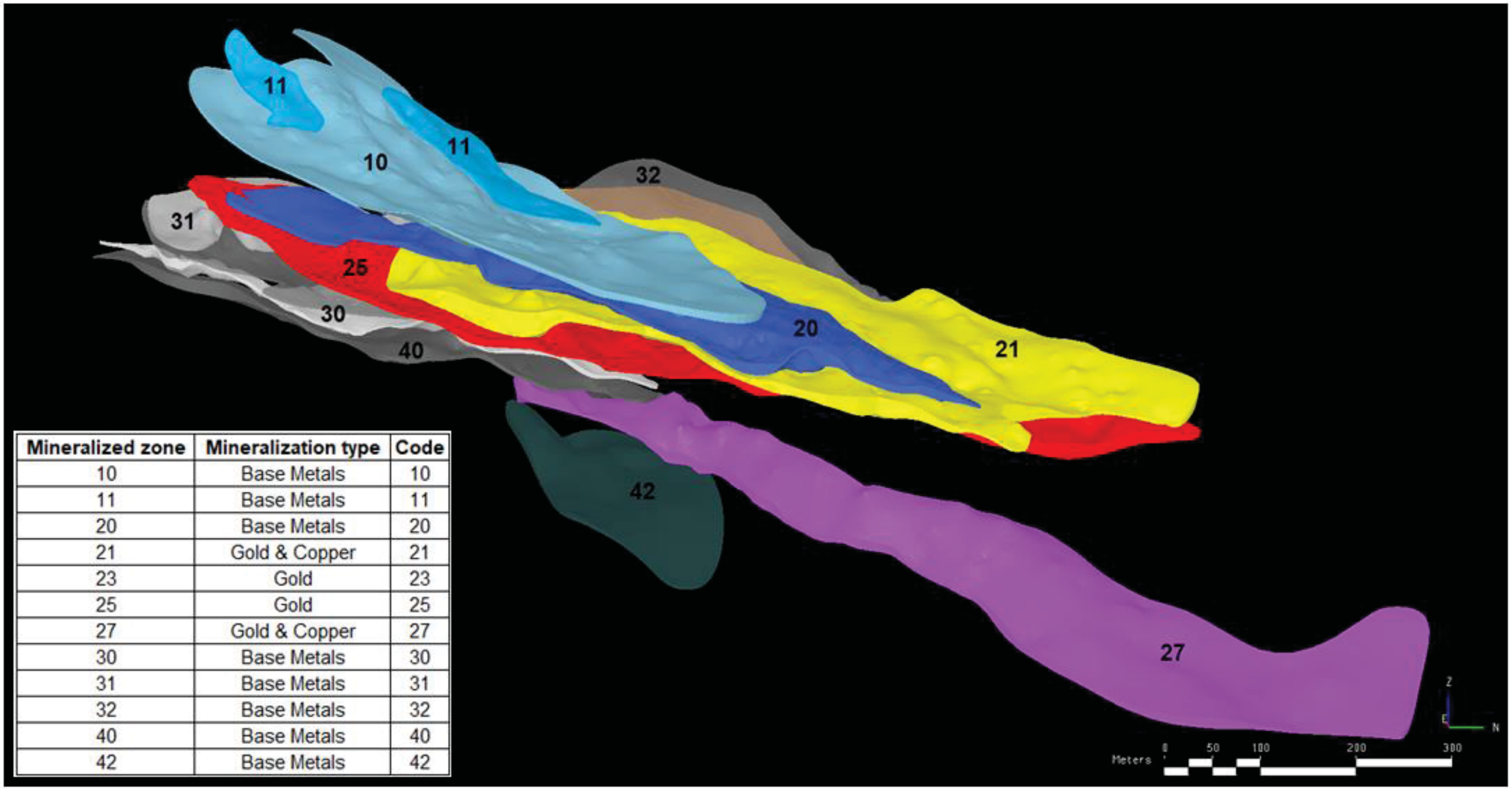
Density values generated from multi regression formulas are based on 12,912 measurements. Table 14-2 presents the regression formulas used to predict the density value where no actual measurement was taken:

TABLE 14-2: DENSITY REGRESSION FORMULA

Envelope	MX1	MX2	MX3	B
10	0.0154*Zn	0.0418*Fe		2.61509
11	0.0169*Zn	0.0394*Fe		2.68323
20		0.0416*Fe		2.72064
21		0.0342*Fe		2.69354
23	0.0327*Pb	0.0202*Fe	0.0762*Cu	2.78016
25		0.0354*Fe		2.71023
27	0.0006*AG	0.0232*Fe	0.0109*Cu	2.85108
30		0.0391*Fe		2.79713
31	0.0235*Zn	0.0326*Fe		2.73048
32		0.0430*Fe		2.70953
40		0.0354*Fe		2.86909
42	0.0114*Zn	0.0005*Fe	0.1422*Cu	2.91883

The predicted density values were validated through comparisons against the measured density.

FIGURE 14-1: 3D VIEW OF MINERALENVELOPE WIREFRAMES, LOOKING WEST



Mineralized zone	Mineralization type	Code
10	Base Metals	10
11	Base Metals	11
20	Base Metals	20
21	Gold & Copper	21
23	Gold	23
25	Gold	25
27	Gold & Copper	27
30	Base Metals	30
31	Base Metals	31
32	Base Metals	32
40	Base Metals	40
42	Base Metals	42

- Note: Lens 23 is located under lens 32 and is shown semi transparent in this view.

Compositing

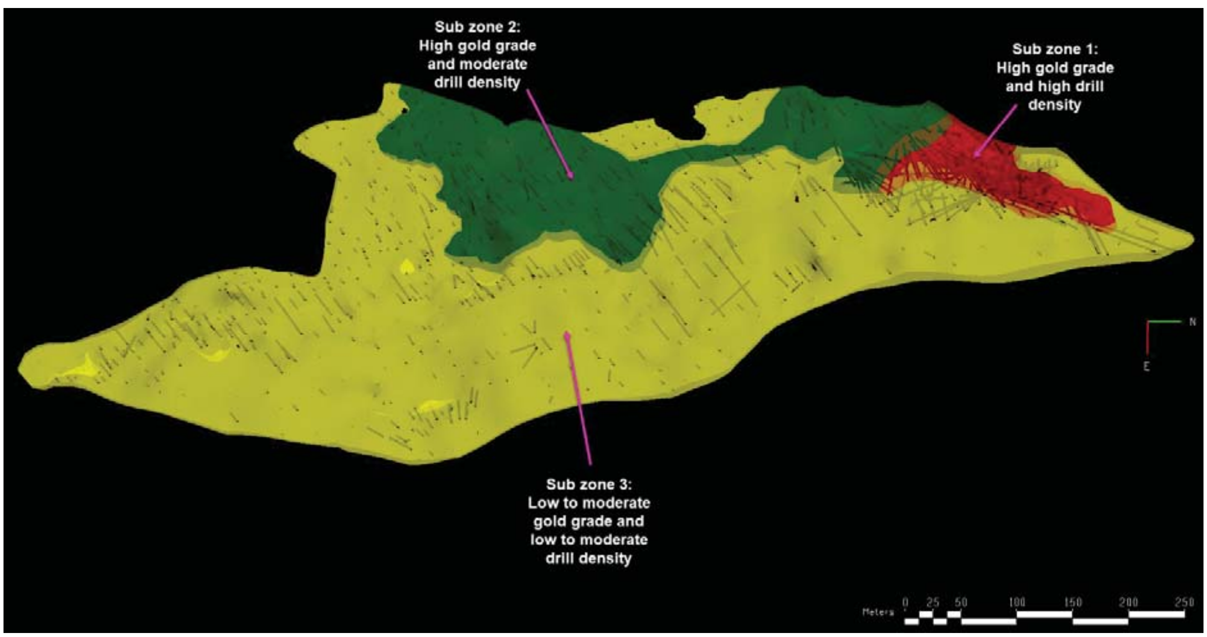
Assay intervals were regularized by compositing drill hole data within each interpreted mineralized envelope. Although most of the drill holes were assayed on 1m interval, a composite length of 2.5 m was selected as more appropriate to conduct interpolation into the 5mx5mx5m block size selected by the mine engineering team as the optimum selective mining unit (SMU).

Grade compositing was weighted by specific gravity as there is a significant positive correlation between some of the metal grade and density. The compositing process was validated by comparing total length, density and length weighted average grade for each metal of the 2.5m composites to the original assays.

Exploratory Data Analysis

Exploratory data analysis (EDA) includes basic statistical evaluation of the composites for zinc (Zn), gold (Au), copper (Cu), silver (Ag) and density (SG). The EDA was conducted separately for each mineralized envelope and also aimed to identify sub-domains that did not support the use of hard contacts for grade interpolation purposes but that justified being used for the purpose of block model validation, smoothing assessment and correction and resource classification. Capping analysis and smoothing assessment were also conducted separately in each sub-domain. An example of sub-domaining is illustrated for gold zone 25 on Figure 14-2 where the high-grade core of the lens is subdivided based on drilling coverage and also from its lower grade halo.

FIGURE 14-2: SUB-DOMAIN IN MINERALIZED ENVELOPE 25



Note: drill hole traces represented by the lines in grey

The 2.5 m composite statistics for Zn, Au Cu, Ag and calculated SG are summarized in Table 14-5 in the grade estimation validation section. As expected, zinc shows significantly lower CV and skewness values in the base metal lenses while the opposite is observed for gold.

Grade Capping

Because the composites of the four metals of economic potential show highly skewed histograms in most of the mineralized envelopes, the deciles[2] analysis (Parish) method was used to define high-grade outliers and to confirm the need for grade capping. This analysis was conducted on the 2.5m composites for each sub-domain independently.

This method considers capping when the last decile of the population contains more than 40% of the metal and that the last percent contains more than 10% of the metal. Capping values were selected to limit the weight of the high-grade outliers on the overall population. Figure 14-3 summarizes the capping applied to all the mineralized envelopes for gold and silver and the proportion of metal removed through this process. In all cases, it was found that no capping would be required for zinc or copper.

TABLE 14-3: CAPPING THRESHOLDS BY LENSE OF GOLD AND SILVER (G/T)

Mineralized zone	Total number of composites	Au capping value	Number of composites	Metal Removed	Ag capping	Number of composites capped	Metal Removed
10	4,603	45	4	8%	270	22	4%
11	359	n/a	0	0%	n/a	0	0%
20	3,079	25	13	4%	n/a	0	0%
21 HGO	4,105	110	4	2%	150	61	7%
21 LGO		50	11	9%			
23	542	65	6	3%	n/a	0	0%
25 HGO 1	5,522	120	11	6%	320	10	6%
25 HGO 2		70	4	3%	220	8	2%
25 LGO		60	9	3%	220	21	3%
27 sub zone 1	1,013	35	4	2%	n/a	0	0%
27 sub zone 2		55	11	3%	n/a	0	0%
30	532	n/a	0	0%	n/a	0	0%
31	904	30	5	7%	n/a	0	0%
32	2,481	65	10	5%	n/a	0	0%
40	686	n/a	0	0%	n/a	0	0%
42	153	n/a	0	0%	n/a	0	0%

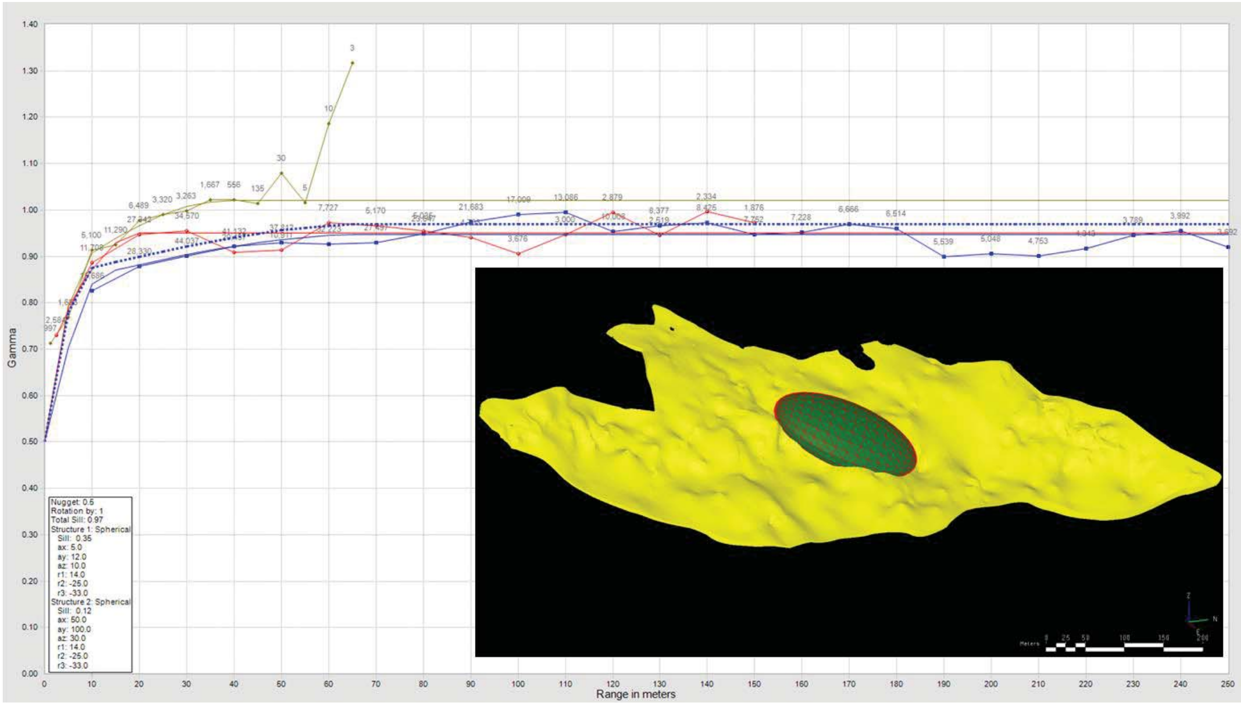
The results obtained from the Parish method were then compared to the natural breaks in the cumulative frequency distribution in the last decile. The metal at risk was also assessed in the population of outliers by comparing the difference in metal using the median and mean values of these outliers. An independent third-party review was conducted in November 2018 by Mr. Dominique Francois-Bongarçon (Agoratek) who concurred with the use of the Parrish as a valid approach for gold capping in the gold zones.

Variography

Down-hole and directional variograms for Zn, Au, Cu, Ag, and SG were created for each individual mineral envelope using the MineSight data analysis software. Pairwise relative variogram were preferred due to the skewed nature of metal distribution in most lenses with the exception of zinc in envelope 40 and silver in envelope 30 which used normal variograms, The major, semi-major and minor axis were built in order to fit the attitude of each mineral envelope. A linear combination of a nugget and two nested spherical models were adjusted in all cases. Once generated, a systematic visual check was conducted to ensure that the search ellipsoid would be correctly oriented with respect to the geometry of the mineral envelopes (Figure 14-3).

[2] I.S. Parrish (1997) Mining Engineering Journal, Geologist's Gordon Knot: to cut or not to cut (pp. 45-49)

FIGURE 14-3: 3D VARIOGRAM MODEL OF GOLD IN MINERALIZED ENVELOPE 25



Note: the insert presents the mineralized envelope in yellow and the matching search ellipse in green

Grade Estimation and Interpolation Methods

The block model consists of regular blocks (5 m along strike by 5 m across strike by 5 m vertically). The block dimensions were selected to match the smallest mining unit (SMU) at Lalor mine. Where a block was intersected by two lenses, the mineral envelopes were used to assign the percentage of the block that belong to each lens.

Both nearest neighbour (NN) and ordinary kriging (OK) grade interpolations were completed on the uncapped and capped grades, using a strict composite and block matching code by lens and three passes with increasing minimum information requirements (Table 14-4). The search passes were selected to ensure best local estimates recognizing that OK has a smoothing effect but making no attempt during interpolation to reduce this smoothing as it would negatively impact the quality of the local estimates. Over-smoothing is addressed through the post-processing of the model described in sub section "smoothing assessment".

Grade Estimation Validation

The grade estimation process was validated for each envelope to ensure appropriate honouring of the input data and subsequent unbiased resource reporting and use of the model for reserve estimation.

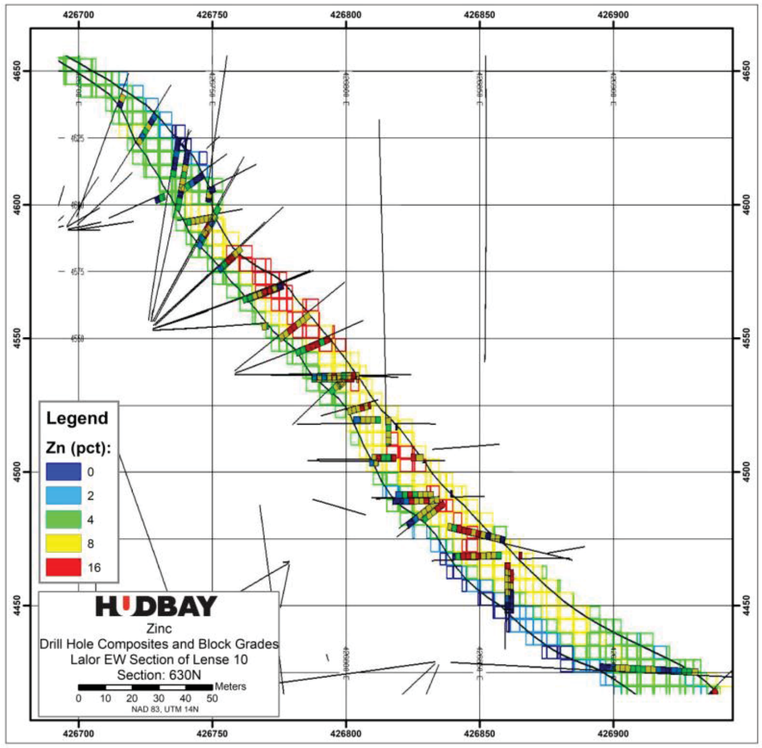
TABLE 14-4: SEARCH ELLIPSE PARAMETRES

	Min # of Comps	Max # of Comps	max # of Comps per hole	Declustering	max Comps per Quadrant	Search ellipse
Pass #1	1	32	6	*no*	-	150% of variogram ranges
Pass #2	16	32	6	yes	8	75% of variogram ranges
Pass #3	16	32	6	yes	8	50% of variogram ranges

Visual Inspection

Visual inspection of block grade versus composited data was systematically conducted in section and plan views. This check confirmed a good reproduction of the data by the model. An example is illustrated in Figure 14-4 for the zinc grade in zone 10.

FIGURE 14-4: VISUAL CHECK ON OK MODEL FOR ZN GRADE IN LENS 10



Global Bias Checks

This validation step consists in comparing the global average grade of each element (after capping for gold and silver) between the original composites, the kriged block estimates and the nearest neighbour estimates. This investigation was conducted not only by envelope but also for each individual sub-domain. The average grade obtained from the nearest neighbor interpolation is a useful benchmark but not a perfect one as it fails to incorporate the nugget effect measured by the variogram. A global check was performed to verify that the kriged mean block estimate was located between the mean of the composites and the mean of the nearest neighbour model. Differences between the 2.5m composites, the NN and OK grades are acceptable in all the sub-domains within each mineralized envelope (Table 14-5).

TABLE 14-5: COMPARATIVE ZINC AND GOLD STATISTICS BETWEEN COMPOSITES AND INTERPOLATION RESULTS

	Zone	Data Type	Count	Min	Max	Mean	Median	Variance	CV
Zinc (%)	10	Zinc in 2.5m composites	4,540	0.00	57.06	8.34	5.27	87.93	1.12
		Zinc in NN	12,902	0.00	57.06	7.42	3.92	93.01	1.30
		Zinc in OK	12,902	0.01	41.86	7.53	6.02	34.92	0.78
	11	Zinc in 2.5m composites	359	0.00	46.44	6.54	3.84	56.69	1.15
		Zinc in NN	3,653	0.00	46.44	6.48	4.03	51.27	1.1
		Zinc in OK	3,653	0.01	25.54	6.67	5.82	17.32	0.62
	20	Zinc in 2.5m composites	3,039	0.00	43.93	7.04	5.37	42.46	0.93
		Zinc in NN	22,266	0.00	43.93	6.66	4.84	41.06	0.96
		Zinc in OK	22,266	0.00	32.39	6.77	5.63	17.94	0.63
	30	Zinc in 2.5m composites	492	0.00	24.10	5.03	4.05	16.31	0.80
		Zinc in NN	6,732	0.00	24.10	4.86	3.87	18.55	0.89
		Zinc in OK	6,732	0.30	14.26	4.88	5.16	5.54	0.48
	31	Zinc in 2.5m composites	897	0.00	29.30	5.13	3.60	22.21	0.92
		Zinc in NN	5,250	0.00	29.30	5.23	3.68	24.11	0.94
		Zinc in OK	5,250	0.97	14.14	5.26	4.73	5.53	0.45
	32	Zinc in 2.5m composites	2,379	0.00	39.57	7.18	5.00	44.36	0.93
		Zinc in NN	19,223	0.00	39.57	6.31	4.36	40.07	1
		Zinc in OK	19,223	0.06	25.22	6.40	5.32	14.58	0.6
	40	Zinc in 2.5m composites	631	0.00	26.06	6.31	5.75	15.30	0.62
		Zinc in NN	10,633	0.00	26.06	6.22	5.45	15.05	0.62
		Zinc in OK	10,633	1.12	13.19	6.30	6.19	2.69	0.26
	42	Zinc in 2.5m composites	139	0.01	23.79	2.81	1.48	14.32	1.35
		Zinc in NN	6,245	0.01	23.79	3.52	2.09	18.03	1.21
		Zinc in OK	6,245	0.87	9.47	3.78	3.24	3.37	0.49

	Zone	Data Type	Count	Min	Max	Mean	Median	Variance	CV
Zinc (%)	21	Zinc in 2.5m composites	3,586	0	17.25	0.39	0.06	0.850	2.37
		Zinc in NN	33,477	0	17.25	0.30	0.04	0.605	2.59
		Zinc in OK	33,477	0	3.90	0.30	0.18	0.112	1.11
	23	Zinc in 2.5m composites	532	0	19.48	0.36	0.06	1.318	3.21
		Zinc in NN	4,600	0	19.48	0.28	0.04	0.806	3.26
		Zinc in OK	4,600	0	3.98	0.32	0.21	0.120	1.07
	25	Zinc in 2.5m composites	4,972	0	22.40	0.30	0.03	0.984	3.34
		Zinc in NN	30,069	0	18.54	0.34	0.04	1.126	3.14
		Zinc in OK	30,069	0	6.24	0.34	0.18	0.235	1.42
	27	Zinc in 2.5m composites	1,013	0	8.37	0.32	0.07	0.605	2.47
		Zinc in NN	19,672	0	8.37	0.23	0.06	0.304	2.45
		Zinc in OK	19,672	0.01	4.03	0.25	0.15	0.065	1.03

	Zone	Data Type	Count	Min	Max	Mean	Median	Variance	CV
Gold (PPM)	10	Capped gold in 2.5m composites	4,540	0.00	45.00	1.52	0.44	10.73	2.15
		Capped gold in NN	12,902	0.00	45.00	1.34	0.39	8.21	2.14
		Capped gold in OK	12,902	0.03	11.41	1.38	0.63	3.00	1.25
	11	Gold in 2.5m composites	359	0.00	33.39	0.81	0.17	4.78	2.72
		Gold in NN	3,653	0.00	33.39	0.70	0.18	2.98	2.48
		Gold in OK	3,653	0.05	9.24	0.74	0.48	0.56	1.02
	20	Capped gold in 2.5m composites	3,039	0.00	25.00	1.78	0.75	8.96	1.68
		Capped gold in NN	22,266	0.00	25.00	1.73	0.63	10.93	1.91
		Capped gold in OK	22,266	0.04	14.43	1.63	1.06	2.56	0.98
	30	Gold in 2.5m composites	492	0.00	21.46	1.14	0.67	4.12	1.78
		Gold in NN	6,732	0.00	21.46	1.04	0.62	3.74	1.87
		Gold in OK	6,732	0.06	7.76	1.14	0.98	0.57	0.66
	31	Capped gold in 2.5m composites	897	0.00	30.00	1.35	0.72	7.58	2.04
		Capped gold in NN	5,250	0.00	30.00	1.50	0.71	11.39	2.25
		Capped gold in OK	5,250	0.22	11.09	1.45	1.08	1.44	0.83
	32	Capped gold in 2.5m composites	2,379	0.00	65.00	3.74	1.54	44.18	1.78
		Capped gold in NN	19,223	0.00	65.00	3.29	1.25	38.09	1.88
		Capped gold in OK	19,223	0.23	35.01	3.42	2.42	10.42	0.94
	40	Gold in 2.5m composites	631	0.00	25.20	1.32	0.76	3.96	1.50
		Gold in NN	10,633	0.00	25.20	1.62	0.87	7.65	1.70
		Gold in OK	10,633	0.22	10.29	1.53	1.28	0.99	0.65
	42	Gold in 2.5m composites	139	0.00	5.70	0.69	0.39	0.83	1.33
		Gold in NN	6,245	0.00	5.70	0.95	0.48	1.61	1.33
		Gold in OK	6,245	0.23	2.16	0.79	0.73	0.12	0.44

	Zone	Data Type	Count	Min	Max	Mean	Median	Variance	CV
Gold (PPM)	21	Capped gold in 2.5m composites	3,586	0	110.00	4.81	1.50	108.868	2.17
		Capped gold in NN	33,477	0	110.00	4.66	1.45	101.023	2.16
		Capped gold in OK	33,477	0.011	39.00	4.19	3.12	12.110	0.83
	23	Capped gold in 2.5m composites	532	0	65.00	5.89	2.59	90.421	1.62
		Capped gold in NN	4,600	0	65.00	5.59	2.93	68.575	1.48
		Capped gold in OK	4,600	0.689	22.27	5.65	4.94	8.526	0.52
	25	Capped gold in 2.5m composites	4,972	0	120.00	5.12	1.67	129.118	2.22
		Capped gold in NN	30,069	0	120.00	4.54	2.05	86.118	2.04
		Capped gold in OK	30,069	0.143	48.54	4.25	3.21	10.963	0.78
	27	Capped gold in 2.5m composites	1,013	0	55.00	5.44	1.75	83.814	1.68
		Capped gold in NN	19,672	0	55.00	4.47	1.31	64.545	1.8
		Capped gold in OK	19,672	0.063	27.90	4.59	3.85	8.703	0.64

Smoothing Assessment

The larger number of composites used for grade estimation in the block model significantly improves the individual block grade estimates but, at the same time results in a much smoother model requiring a careful assessment and in many cases a post-processing of the OK estimates. The extent of grade 'over-smoothing' in the model was investigated separately by sub domains based on material differences in grade distribution and/or drilling density. The mean and variance of the kriged estimates were compared to the variance of the composites after declustering (obtained from a nearest neighbour interpolation). The expected true variance between SMUs can be calculated from the variogram models.

Smoothing Correction

An indirect log-normal correction was used to perform a change of support on the kriged models in order to obtain unbiased grade tonnage curves for the selected mining block size of 5mx5mx5m. This correction is only valid globally and provides poorer local estimates than the smoothed OK model but does not materially alter the global average grade within each zone and provides the correct grade-tonnage curve for the variogram models fitted on the drill hole data. The targeted variance was reached within very close limits in most cases, as illustrated in Table 14-6.

TABLE 14-6: SUMMARY OF THE APPLIED SMOOTHING CORRECTIONS

Mineralized Envelope	Sub Zone	NN model variance	OK model variance	Theoritical variance between 5m x 5m x5m blocks	Corrected model variance for 5m x 5m x5m blocks
10	1 Zn	81.95	9.80	43.50	31.18
	2 Zn	82.39	27.20	43.73	39.88
	3 Zn	23.62	5.39	12.54	11.30
11	global Zn	53.86	19.19	28.59	25.47
20	1 Zn	16.33	4.14	9.69	8.76
	2 Zn	50.69	19.39	30.06	28.90
	3 Zn	11.78	2.48	7.00	6.77
21	1 Au	33.78	4.35	9.83	8.53
	2 Au	270.05	14.10	78.37	82.80
23	1 Au	50.13	6.09	16.50	18.13
	2 Au	110.21	8.14	36.29	30.32
25	1 Au	329.73	29.19	86.99	72.17
	2 Au	200.18	15.79	52.91	45.64
	3 Au	30.72	2.71	8.11	8.41
27	1 Au	49.84	6.61	20.50	16.97
	2 Au	120.89	11.41	49.76	36.71
	3 Au	75.59	6.06	31.08	23.67
	4 Au	14.58	1.29	6.00	4.03
30	1 Zn	19.60	2.82	6.90	7.01
	2 Zn	5.30	1.36	1.86	1.83
31	1 Zn	30.50	5.31	10.24	9.42
	2 Zn	6.88	1.01	2.31	2.23
	3 Zn	6.19	0.90	2.08	1.95
32	1 Zn	8.13	1.99	3.98	3.76
	2 Zn	54.27	14.51	26.54	24.01
40	1 Zn	12.23	1.58	6.63	5.99
	2 Zn	16.00	2.91	8.66	7.74
	3 Zn	14.65	1.96	7.95	7.89
42	1 Zn	8.04	0.91	2.55	2.53
	2 Zn	25.51	2.98	8.10	6.88

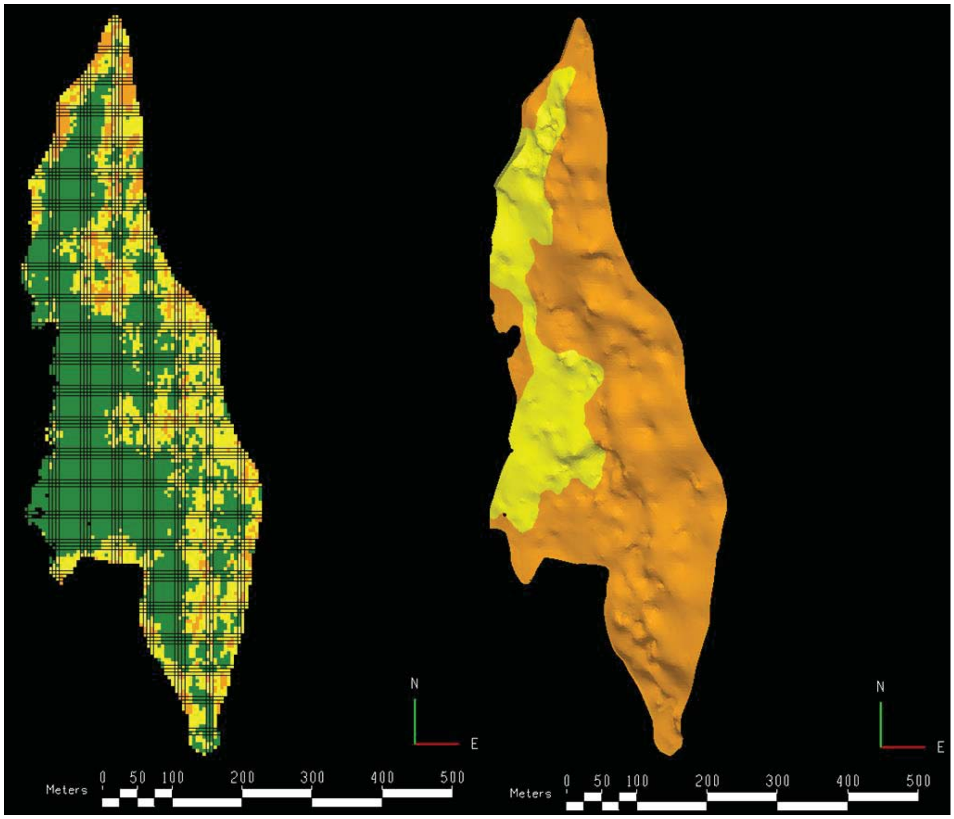
Classification of Mineral Resources

During the interpolation process, several control parameters were recorded for each block, e.g. number of samples, number of holes, the distance to the nearest sample and the average distance to all the samples used for the interpolation as well as the number of quadrants with samples, the kriging variance and the regression slope of kriging for each individual block estimate.

The regression slope values obtained from the kriging of zinc (in the case of base metals lenses) or gold (in the case of gold zones) grade estimates was used as the primary criteria for resource classification with 80% and 60% regression slope thresholds used respectively to separate measured from indicated and from inferred resources. From detailed reserves to mill reconciliations exercises conducted by Hudbay at its operating mines, including Lalor, this criteria was found to be a reliable first pass measure of quarterly and annual performance in tonnes and grade prediction.

The block by block coding assignation was then smoothed to remove isolated blocks of one category within another. Proportions of measured, indicated and inferred category blocks were not changed significantly through this process. In assigning the final categorization, careful consideration was also given to the proportion of mineralization grading above the anticipated cut-off grade. In areas where the blocks grading above the anticipated cut-off grade would represent a small portion of the total volume of the Lens, the classification was downgraded. Figure 14-5 illustrates the classification process followed for Lens 25.

FIGURE 14-5: RESOURCE CLASSIFICATION IN MINERALIZED ENVELOPE 25



Note: the insert on the left presents the block by block resource classification (Measured in green, indicated in yellow and Inferred on orange) based on the regression slope of gold in mineralized envelope 25 while the insert on the right present the final classification. Confidence was down graded based on the proportion of blocks above cut off.

Reasonable Prospects of Economies Extraction and Mineral Resource Estimates

In the past, the reasonable potential for economic extraction required to report mineral resource estimates at Lalor was established by applying an economic cut-off grade to individual 5mx5mx5m blocks using a metal equivalent calculation on zinc or on gold depending on the type of mineralization. This approach is commonly used in the mining industry for underground deposit but usually results in a low conversion factor from mineral resource to mineral reserve estimates when the spatial continuity of these 'economic' blocks' is poor.

For the present estimates, Hudbay has implemented a more stringent approach to resource reporting for underground deposits. With this approach, the potential for economic extraction of the mineral resource estimates at Lalor are reported within the constraint of a 'stope optimization envelope' process similar in concept to a Lerchs-Grossman pit shell commonly used in the mining industry for open pit deposits. This excludes from the resource estimate small isolated individual resource blocks that may meet an economic cut-off criteria on an individual basis but could not be aggregated into mineable shapes and include some 'geological dilution' representing small block grading below cut-off grade but that would need to be included in the economic envelope in order to maintain minimum spatial continuity requirements to define mineable shapes.

The parameters used as input to define the stope optimization envelope cover all the relevant technical and economic constraints including minimum stope and waste pillar dimensions and a NSR value calculation for each block based on anticipated metal recoveries, long-term metal price forecast and operating and capital costs based on the 2019 Lalor mine and Stall concentrator budgets. Two NSR values were calculated for each block in order to assess and compare the value of the blocks going to the Stall or

Flin Flon mill (no material difference between the two) or going to the new Britannia mill. The mineral resource estimates are reported to ensure that each potential stope would cover all its associated operating mining and milling costs. The technical and economic parameters used to estimate the NSR value and the cut-off considered to build the stope optimization envelopes are detailed in section 15.

The stope optimization process was ran on all resource categories combined and as a result, a small amount of one resource category could be included in a stope shape dominated by another resource category. This usually occurred on the fringes of the zones defined for resource classification. Resource classification is based on broad smoothing of interpolation parameters used in the construction of the resource model and visual inspections confirmed that minor modifications to the resource categories were warranted so that all stope shapes would be entirely assigned to one resource category.

The resulting mineral resource estimates, as of January 1, 2019, are presented in Table 14-7. There are no measured or indicated mineral resource estimates in addition (exclusive) to those that were converted to mineral reserve estimates. All measured and indicated mineral resource estimates were either converted to mineral reserve estimates, as documented in section 15, or deemed non-economic after applying the appropriate resource to reserve conversion factors.

TABLE 14-7: LALOR MINERAL RESOURCE ESTIMATES
(1), (2), (3), (4), (5)

	Mineral Resource (Inclusive of Mineral Reserve estimate)	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
Measured	Base Metal Lens	5,410,000	2.7	30.8	0.86	8.21
	Gold Lens	-	-	-	-	-
	Total Measured	**5,410,000**	**2.7**	**30.8**	**0.86**	**8.21**
Indicated	Base Metal Lens	4,130,000	2.6	33.0	0.54	7.30
	Gold Lens	4,290,000	8.1	31.5	1.01	0.39
	Total Indicated	**8,420,000**	**5.4**	**32.2**	**0.78**	**3.78**
Measured + Indicated	Base Metal Lens	9,540,000	2.7	31.8	0.72	7.82
	Gold Lens	4,290,000	8.1	31.5	1.01	0.39
	Total Measured + Indicated	**13,830,000**	**4.3**	**31.7**	**0.81**	**5.51**
	Mineral Resource (Exclusive of Mineral Reserve estimate)	***Tonnes***	***Au (g/t)***	***Ag (g/t)***	***Cu (%)***	***Zn (%)***
Inferred	*Base Metal Lens*	*1,385,000*	*4.5*	*43.6*	*0.70*	*2.30*
	Gold Lens	*4,516,000*	*4.4*	*20.4*	*1.08*	*0.35*
	Total Inferred	***5,901,000***	***4.4***	***25.9***	***0.99***	***0.81***

Note:
1. Totals may not add up correctly due to rounding.
2. Mineral resources are estimated as of January 1, 2019.
3. Mineral resources are estimated at a minimum NSR cut-off of C$96.19 per tonne.
4. Mineral resources do not include mining dilution or recovery factors.
5. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.2 PEN II ZONE

Modeling Database

As shown in Table 14-8, 95 drill holes totalling 19,563 m were included in the Pen II zone database to support the mineral resource estimate with a cut-off date of August 29th, 2018. The files were imported as collar, downhole survey and assay data into MineSight® and Leapfrog® for independent comparison and validation. No significant differences in the projection of the drill hole traces were found between the two datasets. Additional validations conducted on the drill hole database are described in section 12.

TABLE 14-8: DRILLING DATA BY YEAR

Year	Drill Holes	Metres Drilled
1964	4	107
1999	3	1,000
2000	2	754
1971	3	268
1983	1	74
1985	2	296
1987	10	2,461
1986	15	2,029
1988	3	922
1990	1	395
1991	1	473
2007	4	1,253
2017	17	3,120
2018	29	6,411
Total	**95**	**19,563**

Modeling of The Mineralized Envelope

The Pen II Zone mineralized envelope trends along an azimuth of approximately 40° with a general dip of approximately 40° to the north east. Economic mineralization delineated occurs in a semi-massive to massive sulfide lens typical of VMS deposits.

The drill hole database was imported in Leapfrog®. Geological units of sulphides horizons were created in 3D using core logging data and assay results. This broad envelope of sulfide occurrence was then refined to create a narrow smooth and continuous 3D solid including most of the zinc rich mineralization. In a final validation step, the envelope was loaded into MineSight® in order to ensure proper tagging of the solids to actual drill hole locations. The mineral envelope was used as hard boundaries for grade interpolation purposes so as to correctly prevent spreading of sulfide mineralization into barren zones and vice-versa.

Density Assignation

Density values were calculated using a linear regression formula from the zinc and iron content based on 166 measurements and supported by a coefficient of correlation of 97%.

$$SGPR = 2.6286 + (Zn\% * 0.0239) + \ (Fe\% * 0.035)$$

Exploratory Data Analysis

Exploratory data analysis (EDA) includes basic statistical evaluation of the assays and composites for zinc (Zn), gold (Au), copper (Cu), silver (Ag), density (SG), and samples length for the samples located inside the mineralized envelope.

Composites

The majority of the mineralized zone at Pen II has a width of less than 7 m. Due to the narrow nature of the mineralization, the grade interpolation process was simplified to a pseudo-2D approach. Assay intervals were regularized to a single full-length composite from hanging wall to footwall. This modeling approach is adopted to prevent any lateral selectivity in the mine design that is not deemed realistic at Pen II. Grade compositing was weighted by specific gravity as there is a significant positive correlation between some of the metal grade and density in the mineral envelope.

After compositing, the total length, the density and length weighted average grade for each metal of the composites was compared to those of the assays before compositing to ensure they were similar. The composite statistics for Zn, Au Cu and Ag are shown in the grade estimation validation section (Table 14-9).

Variography

Down-hole and directional variograms for Zn, Au, Cu, Ag, and SG were created using the MineSight data analysis software. Pairwise relative variogram were preferred due to the skewed nature of metal distribution. The major, semi-major and minor axis were built in order to fit the attitude of the mineral envelope. A linear combination of a nugget and at two nested spherical models were adjusted. Once generated, a systematic visual check was conducted to ensure that the search ellipsoid would be correctly oriented with respect to the geometry of the mineral envelope.

Grade Estimation and Interpolation Methods

Both nearest neighbour (NN) and ordinary kriging (OK) grade interpolations were completed, using three passes with increasing minimum information requirements. The composite selection parameters for grade estimation in each domain were selected to ensure best local estimates. The percentage of the blocks contained within the mineralized envelope is recorded in the model.

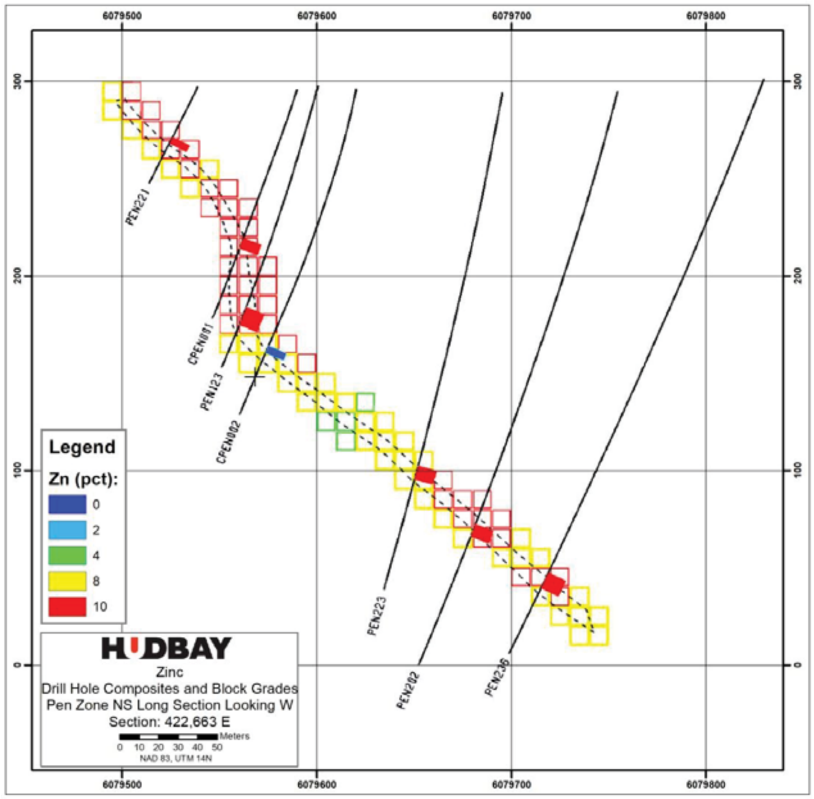
The first interpolation pass uses 150% of the variogram ranges and is restricted to a minimum of one composite and a maximum of eight composites. The second interpolation pass uses 75% of the variogram ranges and is restricted to a minimum of three composites and a maximum of eight composites. The third interpolation pass uses similar constraints to the second pass but with a reduced 50% of the variogram ranges

At Pen II, all the composites have a different length and as a result the interpolation was both length and density weighted. The NN interpolation is solely used to validate that there is no global bias.

Grade Estimation Validation

The grade estimation process was validated to ensure appropriate honouring of the input data and unbiased resource reporting and use of the model for reserve estimation. Visual inspection of block grade versus composited data was systematically conducted in section and plan views. This check confirmed a good reproduction of the data by the model. As an example, one long sections (looking west) is presented. Figure 14-5 presents the zinc grade in the mineralized envelope.

FIGURE 14-6: LONG SECTION SHOWING OK MODEL AND COMPSITES ZINC GRADE



Global Bias Checks

A valid global check was performed to verify that the kriged mean block estimate was located between the mean of the composites and the mean of the nearest neighbour model. Differences between the composites, the NN and OK grades are acceptable. The comparison of the mean and variance for each metal between the composites, the NN and OK models are summarized in Table 14-9.

TABLE 14-9: COMPARATIVE ZINC AND GOLD STATISTICS BETWEEN COMPOSITES AND INTERPOLATION RESULTS

	Data Type	Count	Min	Max	Mean	Median	Variance	CV
Zinc (%)	Zinc in composites	41	1.92	25.425	8.834	7.283	28.23	0.6
	Zinc in NN	2,945	1.92	25.43	8.38	6.6	25.52	0.6
	ZN%	2,945	3.79	18.15	8.63	8.55	6.86	0.3
Gold (ppm)	Gold in composites	41	0	3.307	0.259	0.123	0.28	2.05
	Gold in NN	2,945	0	3.31	0.381	0.14	0.58	1.99
	Gold in OK	2,945	0.03	2.12	0.346	0.17	0.14	1.09
Silver (ppm)	Silver in composites	41	0	33.014	6.726	5.453	36.8	0.9
	Silver in NN	2,945	0	33.01	7.338	6.21	40.51	0.87
	Silver in OK	2,945	1.93	25.77	7.312	6.71	8.72	0.4
Copper (%)	Copper in composites	41	0.072	0.826	0.451	0.417	0.04	0.47
	Copper in NN	2,945	0.07	0.83	0.42	0.36	0.04	0.47
	Copper in OK	2,945	0.15	0.68	0.44	0.45	0.02	0.29

Classification of Mineral Resources

As for Lalor, the regression slope values obtained from the kriging of zinc grade estimates was used as the primary criteria for resource classification. Blocks from the second and third interpolation pass combined with a 60% regression slope threshold were used to separate indicated from inferred resources. There are no measured resource estimates reported at Pen II. The block by block coding assignation was then smoothed to remove isolated blocks of indicated resources within areas of mostly inferred resources and vice versa. Figure 14-6presents a 3D view of the resource categories.

FIGURE 14-7: PLAN VIEW DISPLAYING THE RESOURCE CLASSIFICATION

Reasonable Prospects of Economics Extraction and Mineral Resource Estimates

The mineral resource estimates for Pen II, as of January 1, 2019, are reported within the constraint of a ‘stope optimization envelope’. The parameters used as input to define the stope optimization envelope cover all the relevant technical and economic constraints including realistic minimum stope and waste pillar dimensions. The NSR value calculation is based on the one used at Lalor for the base metal lenses assuming beneficiation at the Stall mill. The mineral resource estimates were reported to ensure that each potential stope would cover all its associated operating and capital costs. The resulting mineral resource estimates are presented in Table 14-10. There are no mineral reserve estimates reported for the Pen II as the required engineering work has not been completed yet.

TABLE 14-10: PEN II MINERAL RESOURCE ESTIMATES
(1), (2), (3), (4)

Categories	Tonnes	Zn Grade (%)	Au Grade (g/t)	Cu Grade (%)	Ag Grade (g/t)
Indicated	500,000	8.89	0.35	0.49	6.81
Inferred	100,000	9.81	0.30	0.37	6.85

1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.
4. Pen II mineral resources are estimated at a minimum NSR cut-off of C$65 per tonne and assume that the Pen II mineral resources would be amenable to processing at the Stall mill.

14.3 WIM DEPOSIT

In May 2015, Golder prepared a resource block model using Datamine® version 2.1.1547.0, an industry standard commercial geological and mining software. The construction of this 3D resource model and the estimation of mineral resources were performed by Greg Greenough, P.Geo., and reviewed by Paul Palmer, P.Geo. P.Eng., The procedures used to create and validate the resource block model are compliant with best industry standards and the CIM guidelines. More details can be found in the “technical Report on The Wim Copper-Gold Deposit, Snow Lake, Manitoba, Canada”, prepared for Alexandria Minerals Corporation (Alexandria), dated May 2015 (Greenough, Palmer) available on SEDAR.

Modeling Database

121 drill holes (2,035 samples) were used to support the mineral resource estimate. The drill holes were checked for overlaps, missing intervals and missing surveys.

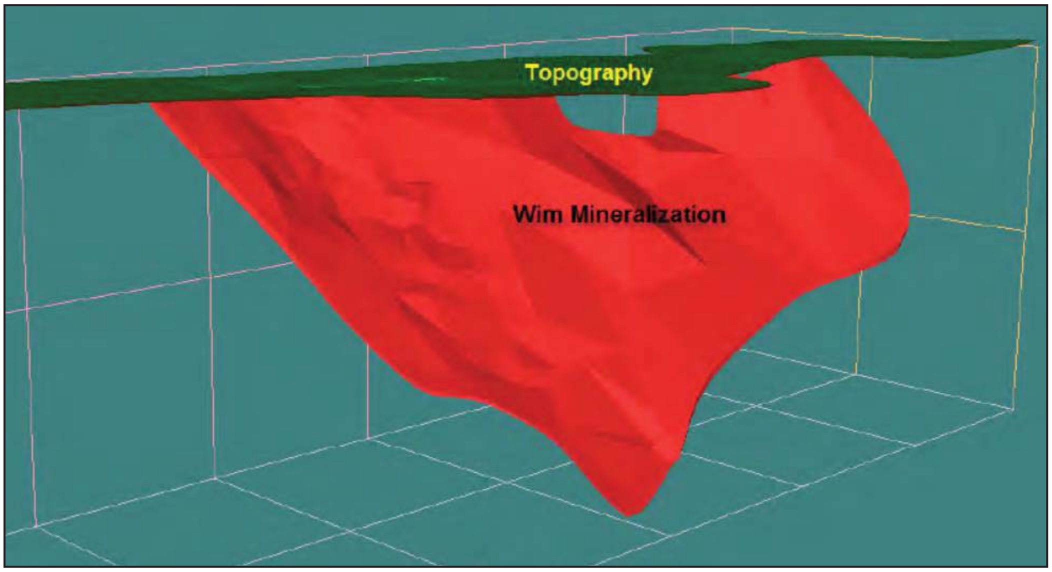
Modeling of the Mineralized Envelopes

The Wim mineralized envelope trends approximately to the North with a general dip of approximately 45° and has an average a true thickness of about 10 m. A smooth and continuous 3D solid was built in Datamine using a 0.5% copper equivalency cut-off to guide the construction of the base metal mineralized envelope while including material below cut-off where necessary to maintain the continuity of the envelope (Figure 14-8).

Density Assignation

Missing density values for 91 samples were generated using a linear regression formula between Fe grade and density based on 572 density measurements. For 1,337 intervals with missing density and Fe grade, a linear regression from the Cu grade was used instead. Finally, 35 samples without any analytical results were assigned a density value of 2.55 g/cm^3.

FIGURE 14-8: 3D VIEW WIM MINERALIZED ENVELOPE, LOOKING SOUTH WEST



Capping

Scatter plots were used to identify assay outliers for Au, Ag, Cu and Zn values for the purpose of grade capping. Prior to compositing, one Au sample that was above 30 g/t, containing 2% of the metal was capped at 30 g/t; and Ag for the same sample containing 1% of the metal was capped at 60 g/t. No samples were capped for Cu or Zn content.

Composites

A histogram and cumulative percentage plot of the sample lengths were used to determine an appropriate composite length for grade estimation of the Wim deposit. The range of sample length for the Wim deposit is between 0.1 m to greater than 5 m with a sample length mean of 0.63 m. A composite length of 0.9 m, representing 77% of the population was selected to conduct interpolation into the blocks. Compositing was weighted by density.

Variograms

Variogram contours in the plane of the deposit were calculated for Cu, Zn, Au and Ag. A direction of preferential continuity was identified at approximately a 40º angle in the unfolded plane of the deposit coinciding with the plunge of the eastern limits of the deposit, as well as an observed trend communicated by Murgor personnel. Down-hole and directional pairwise variograms for Zn, Au, Cu, Ag, and SG were modeled by a linear combination of a nugget and two nested spherical structures.

Grade Interpolation

A block size of 15 m east-west (strike), 3 m north-south (thickness), and 5 m elevation was selected to provide sufficient detail without causing improper smoothing during distance estimates. Grade for Cu, Zn, Au, Ag and density (SG) was estimated using an inverse distance squared interpolant ($IDW^{2)}$ in the unfolded space and weighting by density. A nearest neighbor (NN) interpolation was also performed to provide declustered sample grade for block model validation. Three passes were used to populate the blocks in the model. The first pass uses 100% of the variogram ranges, while the second pass uses 200% of the variogram ranges. Finally, the third pass uses 500% of the variogram ranges. In any case, the three interpolation passes are restricted to a minimum of eight composites and a maximum of 15 composites (maximum of four per hole).

Block Model Validation

The NN and IDW2 estimates, as well as the composites were compared to ensure that the differences between their mean estimated grade was well within acceptable limits. Higher mean grades for the composites reflect increased drill hole clustering in the higher grade east end of the Wim deposit.

Resource Classification

Blocks estimated from the first interpolation pass and using a minimum of 12 composites were classified as indicated. The remainders of the blocks were classified as inferred. The block by block coding assignation was then smoothed to remove isolated blocks of indicated within areas of mostly inferred category and vice versa. Proportions of indicated and inferred category blocks were not changed significantly by this process. The classification process is illustrated on Figure 14-9.

Potential for Economic Extraction and Mineral Resource Estimates

In order to properly account for the polymetalic nature in the assessment of the economic potential of the Wim deposit, a Copper Equivalent (CuEq%) value was calculated as illustrated in Table 14-11.

FIGURE 14-9: RESOURCE CLASSIFICATION

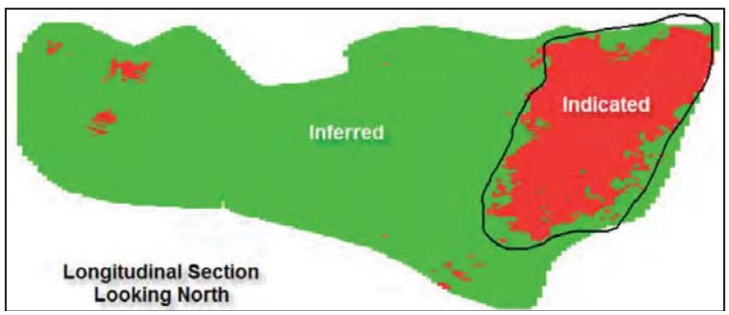


TABLE 14-11: COPPER EQUIVALENCY

Metal	Revenu factor formula	Revenu factor
Cu	1% * pounds per tonne * sell prince = 1/100 * 2204.627 * $3 =	$66.14 per tonne at 1% Cu
Zn	1% * pounds per tonne * sell prince = 1/100 * 2204.627 * $1 =	$22.05 per tonne at 1% Zn
Au	1% * grams per once * sell prince = 1/100 * 31.1035 * $1200 =	$38.58 per tonne at 1 g/t Au
Ag	1% * grams per once * sell prince = 1/100 * 31.1035 * $15 =	$0.48 per tonne at 1 g/t Au

CuEq =	Cu * 90% recovery	Copper = $3 per pound
	+ (Au * 70% revovery * Au revenu factor (38.58) / 66.14	Gold = $1,200 per once
	+ (Zn * 90% revovery * Zn revenu factor (22.05) / 66.14	Zinc = $1 per pound
	+ (Ag * 70% revovery * Ag revenu factor (0.48) / 66.14	Silver = $15 per once

An open pit optimization was conducted on the resource block model using Whittle 4.5.2 and assuming $5/t for mining, $1.50/t for transport, $25/t for processing, $15/t for G&A and metal prices of respectively $3/lb Cu, $1,200 Au/oz, $1/lb Zn and $15/oz Ag and metallurgical recovery factors of 90% Cu, 70% Au, 90% Zn and 70% Ag. An overall average slope angle of 45 degrees was used and a base case pit shell was generated using both indicated and inferred blocks in the model. The open pit mineral resource estimates are reported as individual blocks above a 0.6% CuEq cut-off grade inside the pit shell defined by a revenue factor of 1.

A 20 m zone of material below the bottom of the Whittle open pit shell was classified as crown pillar and excluded from the Mineral Resource. A more detailed geotechnical study will be required as the project advances to determine the minimum crown pillar thickness or potential methods of mining the crown pillar.

The Datamine's Mineable Shape Optimizer (MSO) module was then applied to the Wim resource block model below the 20m crown pillar in order to identify contiguous volumes of resource using a minimum target stope head grade of 1. 3% CuEq. This optimization was conducted on both indicated and inferred mineral resources.

The resulting mineral resource estimates, as of January 1, 2019, are presented in Table 14-12. There are no mineral reserve estimates reported for the Wim deposit as the required engineering and metallurgical test work has not been completed yet.

TABLE 14-12: WIM MINERAL RESOURCE ESTIMATES
(1), (2), (3), (4)

		CuEq% Cut-off	Tonnes	Cu%	Zn%	Au g/t	Ag g/t
Indicated	Open Pit	0.6	276,000	1.08	0.10	1.25	6.81
	Undergroung	1.3	3,623,000	1.76	0.28	1.59	6.67
	Total Indicated		**3,899,000**	**1.71**	**0.27**	**1.57**	**6.68**

		CuEq% Cut-off	Tonnes	Cu%	Zn%	Au g/t	Ag g/t
Inferred	Open Pit	0.6	63,000	0.95	0.09	1.05	6.4
	Undergroung East	1.3	604,000	1.12	0.44	1.69	4.7
	Undergroung West	1.3	64,000	0.26	0.01	3.03	2.5
	Total Inferred		**731,000**	**1.03**	**0.37**	**1.75**	**4.65**

1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.
4. Mineral resources reported based on a 1.3 CuEq% cut-off for the underground portion, and a 0.6% cut-off for the open pit portion, assuming processing recoveries of 90% for copper and zinc and 70% for gold and silver, and using long-term prices of $3.00 per pound copper, $1,200 per ounce gold, $1.00 per pound zinc and $15.00 per ounce silver. A 20m crown pillar below the open pit bottom is excluded from resources.

14.4 NEW BRITANNIA

Main Mine Upper and Lower

The mineral resource estimate performed by Kinross followed a conventional and industry standard approach. The approach and general methods of resource estimation are summarized as follows:

- Sectional interpretation of mineralization zones at a cutoff grade of 0.097 oz. per ton and a minimum true width of 7 ft.
- All assays greater than 0.50 oz. per ton were cut to 0.500 oz. per ton.
- The sectional interpretation of the mineralization was digitized and wireframed, using Gemcom Software.
- The assay data was composited into 3-ft. intervals and basic statistics were completed on the assay and composited assay data.
- Geostatistical analysis was completed and variogram models established.
- A block model with dimensions 15 ft. x 15 ft. x 15 ft. was used for grade estimation.
- Grade interpolation was completed using OK using a number of interpolation passes.
- Validation of the block model using graphical and statistical techniques.
- Tonnage was established using the percent model utility in Gemcom and tabulated by rock type using the percentage of the block within the mineralized wireframe.
- The blocks were composited, using Gemcom, into 50-ft. levels and 50-ft. sections for tabulation and reporting of tons and grade.
- Tabulation and reporting of the resources was completed at US$400, US$450, and US$500 gold prices corresponding to cutoff grades of 0.097 oz. per ton, 0.086 oz. per ton, and 0.077 oz. per ton.

The review of the reports and the two procedures verify that the Kinross resource estimate, (audited and reported in the Micon International Ltd. NI-43-101 technical report dated October 2006), is a consistent representation of the drill data.

No.3 Zone and Birch

The mineral resource estimates were performed by SRK for Alexis Mineral Resources following a conventional and industry standard approach.

The database contains 227 surface diamond drill holes totaling 64,471m for No. 3 Zone. A total of 11,426 samples were taken from the No.3 Zone. A portion of this zone was mined between 1995 and 1996 from a surface portal down to approximately 137m vertical. For the purposes of geological interpretation, as well removing the mined volume for the resource estimate, a mined-out volume was created in Gemcom using surveyed AutoCAD level plans.

The Main and Footwall Lenses are based on sectional interpretation of mineralized intercepts greater than 1.77g/t and 1.8m minimum true width. The Main Lens exhibits very good continuity down plunge with only the occasional hole below the cutoff grade parameters. The Footwall Lens exhibits good continuity in the upper part of the No.3 Zone where is appears parallel the Main Lens. The continuity becomes more sporadic down plunge on the Footwall Lens where the interpretation has been expanded to include foliated and altered, but weakly mineralized mafic volcanic rocks.

Samples were composited to 0.9m was determined to be an appropriate sample length as 95% of the samples in the Main Lens are 3 ft. or less.

SRK evaluated the composite data for both the Main Lens and the Footwall Lens zones using histograms and probability plots. SRK found clear breaks in the composite grade distributions at 35.8g/t for the Main Lens, and 13.1g/t for the Footwall Lens.The composite data was evaluated by the author and although a top grade of 14g/t may also be appropriate for the Footwall Lens, the author concurs with the choices of top cuts made by SRK.

Variography on the Main Lens and the Footwall Lens was completed by SRK using the capped composite values. SRK examined 1) traditional semi-variograms, 2) traditional correlograms, 3) normal scores semi-variograms. and 4) normal scores correlograms. SRK noted the limited quantity of composites for No.3 Zone variography, and the variograms do not illustrate well-defined structures.

A block model was created in Gemcom to model the Main Lens and the Footwall Lens. Block size is 10 ft. x 10 ft. x 10 ft. The block size is similar to the average width of the Main Lens. Given the lack of well-defined structure in the variography combined with the observation that the Main Zone is very planar feature with very good continuity and where most intersections are above the minimum grade and width modeling parameters, an ID^2 method was used for block grade estimation. The interpolation parameters are described in Table 14-13.

TABLE 14-13: INTERPOLATION PARAMETRES FOR BLOCK GRADE ESTIMATION USING ID^2 AT THE NO.3 ZONE

Main	Min # of Comps	Max # of Comps	max # of Comps per hole	Declustering	max Comps per Quadrant	Search ellipse
Pass #1	2	12	-	*yes*	3	40m x 24m x 1.5m
Pass #2	2	12	-	*yes*	3	80m x 48m x 3m
Pass #3	2	12	-	*yes*	3	120m x 72m x 4.5m

Footwall	Min # of Comps	Max # of Comps	max # of Comps per hole	Declustering	max Comps per Quadrant	Search ellipse
Pass #1	4	6	-	-	-	45m x 24m x 1.5m
Pass #2	2	6	-	-	-	90m x 24m x 3m
Pass #3	2	4	-	-	-	135m x 72m x 4.5m

The tonnage factor used in the previous No.3 Zone resource estimate (*Micon, 2006*) was 0.0893 tons per cubic foot. This number is based on the specific gravity determinations of 357 samples. The specific gravity data was evaluated and reported by Micon (20068) and is considered to be reliable for the tonnage factor.

The results of the modelling process was validated through:

- A visual review of the results;
- Block model and wireframe volume comparisons;
- Comparison with other in interpolation methods;
- Comparison of the mined out and not mined out parts of the model;
- Swath plots.

For comparison purposes, grade block models were generated using nearest neighbour and ordinary kriging methods. The ordinary kriging parameters used are reported by SRK. The nearest neighbour model is interpreted as the de-clustered representation of the underlying composite data, and similarity with the estimate is interpreted as a measure of the ID^2 estimate being globally unbiased. The ordinary kriging model has a 3.6% higher average grade; this is interpreted as a reasonable replication of the global average grade.

The cutoff grades for the resource have been estimated over a 6-ft. minimum true width and the parameters described in

Table 14-14. The variation in the mining cost is related to new mining versus remnant mining. The mill exists on the project, and the G&A is higher due to the camp costs. The formula to calculate the cutoff grade is:

$$costs/(gold\ price * recovery) * 31.103/exchange\ rate$$

TABLE 14-14: COST AND RECOVERY PRICE ASSUMPTION

Parameter	Units	Value
Estimated Mining Cost	CA$/tonne	43 – 115
Estimated Milling Cost	CA$/tonne	35
Estimated G&A	CA$/tonne	20
Total Operating Cost	CA$/tonne	98 - 170
Assumed Gold Price	US$/Tr.Oz	1,250
Assumed Exchange Rate	CA$/US$	1.3
Gold Recovery	%	93

Table 14-15 presents the resource estimate for the New Britannia property, as of January 1, 2019, using a variable cut off grade. Given its historical nature, the entire resource is classified as inferred. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

TABLE 14-15: UPDATWED GOLD RESOURCE ESTIMATE FOR THE SNOW LAKE PROPERTY
(1), (2), (3)

Zone	Location	Cut-off (Au g/t)	Tonnes	Grade
Lower Dick NW	1780 -	3.3	7,000	4.76
Main Mine 1780 +	1780 +	3.5	1,142,000	4.55
Lower Dick NW	780 -	3.3	213,000	4.55
Lower Dick EW	1780 -	2	823,000	4.59
Lower Ruttan	1780 -	2	568,000	4.28
Main	3 Zone	2	744,000	6.37
Footwall	3 Zone	2	364,000	4.66
Birch	Birch	3.3	569,000	4.42
Total Inferred			4,430,000	4.82

1. Totals may not add up correctly due to rounding.
2. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
3. Mineral resources in the above tables do not include mining dilution or recovery factors.

WSP reviewed all the available data, modeling procedures, block models and technical reports and more specifically, audited the geological model, mineralization wireframes, domain statistical validation, and visual trend slice validation. Available historical data and technical reports were also examined. Based on the independent checks conducted by WSP, the QP is satisfied that the procedures followed by Kinross, SRK and Alexis Minerals were adequate to support the declaration of inferred mineral resource estimates.

15. MINERAL RESERVE ESTIMATES

No mineral reserve estimates are reported for the Pen II, Wim and New Britannia Mine deposits and as a result, this section documents how the Lalor LOM plan and mineral reserve estimates were developed. The Lalor mine mineral reserves as of January 1, 2019 are summarized in Table 15-1.

TABLE 15-1: SUMMARY OF MINERAL RESERVE

(as of January 1, 2019) (1), (2), (3), (4)

	Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Base Metal	Proven	5,137,000	7.13	2.37	0.76	26.31
	Probable	5,552,000	4.19	3.52	0.44	27.39
Gold	Proven	58,000	2.65	5.46	0.80	39.09
	Probable	2,928,000	0.31	6.74	1.09	23.08
Proven + Probable		**13,675,000**	**4.46**	**3.78**	**0.70**	**26.11**

Notes:
1. Totals may not add up correctly due to rounding.
2. Mineral reserves are estimated as of January 1, 2019.
3. Mineral reserves are estimated at an NSR cut-off of $96.19/t for waste filled mining areas and a minimum of $104.58 per tonne for paste filled mining areas.
4. Metal prices of $US 1.17/lb zinc (includes premium), $US 1,260/oz gold, $US 3.10/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to estimate mineral reserves.

The Deswik mine design software was used to optimise the mine plan based on cut-off costs, stope geometry parameters, mineral resource shapes from the resource model described in section 14. Appropriate dilution and recovery factors were applied based on mining method with a combination of paste and unconsolidated waste backfill material.

The mineral reserves were sequenced and scheduled into a Life of Mine (LOM) schedule using Deswik interactive scheduling software and exported to spreadsheets for financial analysis.

The author considers that the mineral reserves as classified and reported comply with all disclosure in accordance with requirements and CIM definitions. The author is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

There are no legal, political, environmental or other risks that could materially affect the potential developments of the reserves in the report.

15.1 MINING STRATEGY AND RESERVE ESTIMATIONS PROCESS

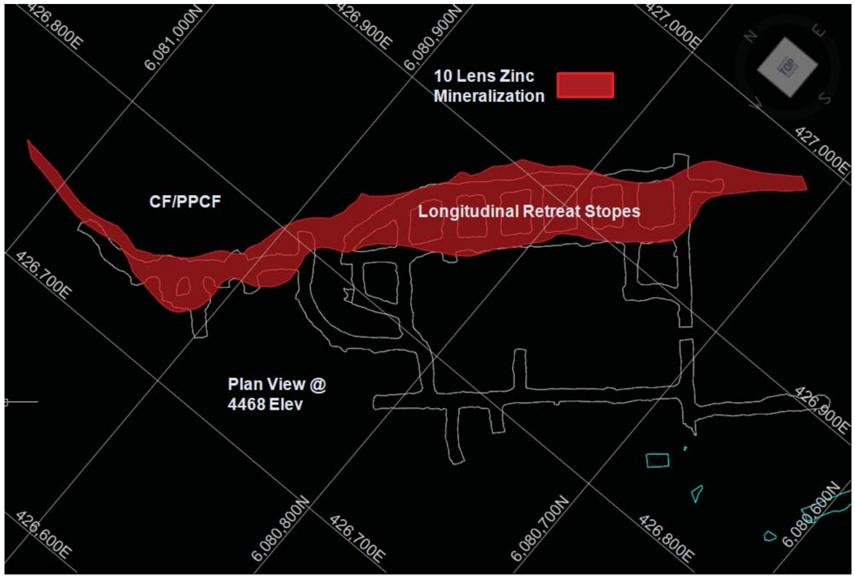
Mining Method Selection & Mining Strategy

Lalor Mine has been in production continuously since mid 2012 and has produced 5.6 million tonnes until the end of 2018. Mining methods are the major determinants of dilution, recovery, economics and production scheduling, which are used to determine mineral reserves. Production at Lalor is planned to be done using predominantly transverse longhole mining with minor longitudinal retreat and cut and fill mining.

Early in the mine life, the complexity of stacked zinc and gold zones within the deposit was realized. In the zinc rich 10 and 11 Lenses, where there is no lens stacking, longhole retreat, transverse longhole and post pillar cut and fill mining were used successfully. See Figure 15-1 for typical mining arrangement in 10 Lens.

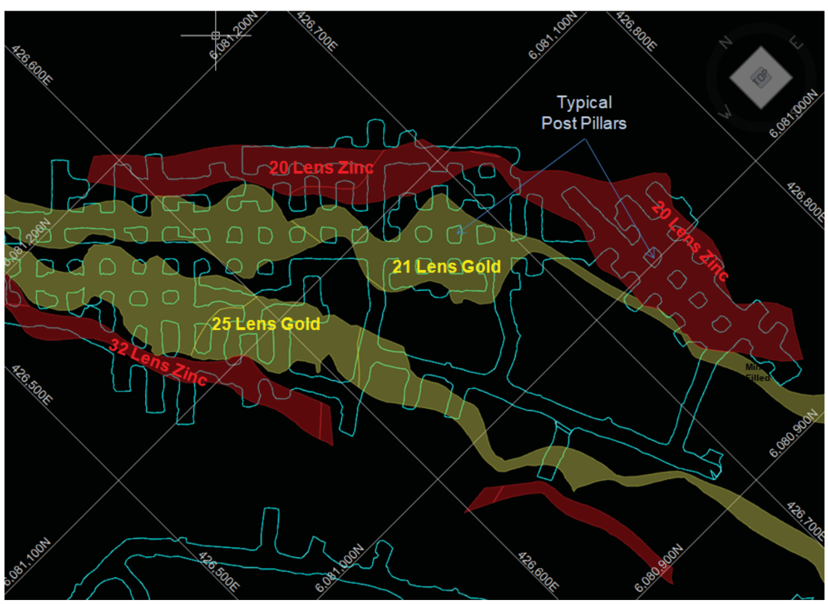
Selective post pillar cut and fill mining was initially used in 20, 21, 25 and 32 lenses as shown in Figure 15-2. Post pillar mining allowed geological evaluation of each round, leading to a better understanding of the ore. This helped to determine: 1) the possibility of mining the 20 and 32 lens zinc ores separately from 21 and 25 lens gold ores and 2) the ability to visually identify and segregate gold ores from surrounding waste. The result was that most zinc ore stopes can be designed and mined separately from the gold, but that the microscopic nature and distribution of the gold in gold zones required additional diamond drilling, sampling and assaying information than required for the zinc stopes to determine production grades.

FIGURE 15-1: 840 LEVEL 10 LENS PLAN VIEW



Updated geological interpretations and the decision to construct a paste backfill plant supported the decision to convert some areas of the mine similar to those in Figure 15-2 from post pillar cut and fill to longhole mining across all lenses. For narrow and steeply dipping stopes typical of 31 and 32 lens, longitudinal retreat mining was preferred, while for wide stopes crossing multiple lenses, typical of 20 and 21 lenses shown in Figure 15-2, transverse longhole mining using footwall drawpoints was preferred.

FIGURE 15-2: 910 LEVEL POST PILLAR C&F



Longitudinal retreat mining was tested in the North end of 32 Lens. Poor hanging wall conditions prompted a review of the geological structures and the contact of the gold lenses which tended to overbreak or slough in contact with the host waste rock. This sloughing resulted in stope failures and delays in the mine production schedule. In order to mitigate the effects of poor hanging wall conditions, future production from undeveloped areas of 32 Lens is planned using transverse longhole mining with paste backfill. Some risk of hanging wall failure into the gold lenses remains, but the effect on production is minimized by having multiple stopes available. This mining method along with paste backfill also allows the division of stopes into hanging wall and footwall blocks, further reducing the risks and effects of stope failure. This approach was also selected for the upper areas of Lenses 10 and 27 and for the lower part of Lens 27.

The flexibility provided by transverse longhole mining is expected to provide consistent production of 4,500 tonnes per day and will allow concurrent production from both zinc and gold stopes. In addition, commissioning of the paste plant in the third quarter of 2018 has significantly reduced the time to backfill a stope, provides tight competent fill up against the hanging wall and a noticeable improvement to ground conditions. In addition, paste backfill allows greater flexibility in mine design and sequencing options.

The strategy for this update of the mine plan at Lalor was to mine base metals of higher grade zinc production from 2019 to the end of 2021 while at the same time advancing the development required to start mining the gold and copper gold zones. Following the planned opening of the New Britannia mill in Q1 2022, the strategy changes to targeting a production of 1,000 to 1,500 tonnes per day of gold and copper gold ore, supplemented by 2,400 to 3,400 tonnes per day of base metal ore until 2025, followed by lower production rates until depletion of the mineral reserves.

Reserve Methodology

The following steps were followed in developing the reserve estimates:

- Calculate two payable (NSR) values for each individual block in the resource model depending on whether processing would occur at the Stall concentrator or at the New Britannia concentrator, using long-term metal prices, concentrator recoveries, metal payability and downstream smelter treatment and refining costs assumptions.
- Design stopes in the Deswik Stope Optimizer, considering depleted mineral resources, existing workings, resource categories and mine and mill operations costs. Dilution and recovery are estimated and applied at this step. Stopes are designed for both the Stall concentrator option and the New Britannia concentrator option
- Establish stope economics using a secondary NSR calculation where, along with mine and mill operations costs, mine capital, waste development and offsite administration costs are applied to each stope. Assign whether stopes can be upgraded to mineral reserves based on resource classification.
- Design ore development required for mining the reserves. Deplete development from the stopes. Interrogate grades of designed development for inclusion in mineral reserves.
- Sequence and schedule development and stope production for input to a financial Life of Mine (LOM) study to support mineral reserve economics.

The above methodology takes into consideration the three different ore types and the milling options for the future production from Lalor mine.

15.2 MINERAL RESOURCE NSR VALUE

The average block value by lens for flotation applicable to the Stall concentrator (NSR ZN) and by copper flotation with gold leach (NSR AU) applicable to the New Britannia concentrator are shown in Table 15-2. The NSR values shown are before applying mining or milling costs and are derived from geological block model grades that contain low grade material that did not convert to mineral reserves. The metal price assumptions were based on long-term forecast of $US 1.17/lb zinc (includes premium), $US 1,260/oz gold, $US 3.10/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25.

TABLE 15-2: IN-SITU MINERAL INVENTORY VALUE

Lens	Ore Type	NSR Zn (Stall)	NSR Au (New Brit)
10	Zinc	**$247.93/t**	$139.38/t
11	Zinc	**$161.91/t**	$48.80/t
20	Zinc	**$259.75/t**	$135.04/t
30	Zinc	**$141.30/t**	$73.53/t
31	Zinc	**$148.50/t**	$87.77/t
32	Zinc	**$358.01/t**	$284.81/t
40	Zinc	**$199.40/t**	$109.49/t
42	Zinc	**$103.91/t**	$58.07/t
21	Gold	$131.28/t	**$232.38/t**
23	Gold	$145.23/t	**$288.31/t**
25	Gold	$128.37/t	**$232.10/t**
27	Copper/Gold	$259.03/t	**$334.92/t**

Metallurgy

The orebody is polymetallic with economically significant metals being zinc, gold, copper and silver. There are three different mineralization types, which are assumed to be treated either using conventional copper and zinc flotation at the Stall concentrator or using copper flotation followed by gold and silver leaching at the New Britannia concentrator:

- Base metal zinc: Near solid to solid sulphide, with dominant pyrite and sphalerite with minor blebs and stringers of chalcopyrite and pyrrhotite.
- Gold rich: Silicified gold and silver enriched with stringers to disseminated chalcopyrite and sphalerite mineralization.
- Copper rich.

Metallurgical performance at Stall concentrator indicates that base metal and gold can be blended. Metallurgical assumptions are shown in Table 15-3.

Two concentrates will be produced at Stall concentrator; 1) a zinc concentrate that will be shipped to the Flin Flon metallurgical complex for production of refined zinc and 2) a copper concentrate that will be shipped to third party smelters. A small amount of zinc ore will be processed at the Flin Flon concentrator during the 2019-2021 period which has similar metallurgical performance to Stall.

The New Britannia gold concentrator will process gold and copper rich ores which are separated from zinc ore at the mine. Copper concentrate will be shipped to third party smelters while tails from the copper circuit will be processed in the CIL circuit to produce doré containing gold and silver. Gold mill metallurgical assumptions are shown in Table 15-3

TABLE 15-3: METALLURGICAL ASSUMPTIONS
(1)

	Base	Gold
Gold to Copper Concentrate	55.6%	63.3%
Silver to Copper Concentrate	56.3%	55.4%
Copper to Copper Concentrate	81.8%	94.1%
Copper Concentrate Grade	20.3%	27.8%
Zinc to Zinc Concentrate	92.2%	
Zinc Concentrate Grade	50.3%	
Gold to Doré in Gold Circuit [1]		82.3%
Silver to Doré in Gold Circuit		50.7%
Overall Gold Recovery	55.6%	93.5%
Overall Silver Recovery	56.3%	78.0%

1. Recovery of gold contained in copper circuit tails.

Payability, Treatment Charges, Refining Charges

Hudbay's long term copper and zinc payability assumptions, concentrate freight to third party smelters costs, concentrate treatment costs, metal refining charges, and marketing terms including premiums and metal distribution costs are applied.

15.3 DILUTION AND RECOVERY

Dilution and recovery factors are input in the stope optimizer by mining method assuming those currently experienced at Lalor Mine for transverse longhole open stope, longitudinal retreat longhole and post pillar cut and fill mining.

Dilution

Dilution is defined as the tonnage of waste added to the mineral resource tonnage. Dilution is included in the conversion of mineral resource to mineral reserves and can be both internal or external. Dilution is reported as the ratio of total waste divided by mineral resource.

Internal dilution is primarily incurred due to the shallow dipping nature of the deposit and stacking of lenses which can result in multiple lenses being grouped together for mining purposes. The space between the lenses that are mined in the course of stope mining is considered as internal dilution. In the Deswik stope optimizer routines multiple lenses may be required to be extracted as a single mining unit based on stope mining parameters as shown in Table 15-4.

Internal dilution included in mineral reserves can also include low grade development required to achieve footwall dip angles in longhole stopes and may include material mined between lenses where multiple lenses are grouped together. Current grade control practices will continue to be used to assign material as ore or waste on a round by round basis to attempt to minimize dilution.

TABLE 15-4: STOPING PARAMETERS BY MINING METHOD

Stope Shape Parameters	Unit	Longhole	Cut and Fill
Length along Strike			
Minimum	Metres	16	20
Maximum		23	150
Width FW to HW			
Minimum	Metres	3.5	5
Maximum		50	-
Waste Pillar Width	Metres	5	-
Stope Height			
Minimum	Metres	10	5
Maximum		20	5
Hanging Wall Dip			
Minimum	Degrees	30	90
Maximum		90	90
Footwall Dip			
Minimum	Degrees	50	90
Maximum		90	90
Dilution			
Hanging Wall Fixed	Metres	0.5	11%
Footwall Fixed		0.5	11%

External dilution is set at a fixed distance of 0.5m into the footwall and hanging wall after the stope geometry shape is finalized. Internal dilution and external dilution are included as part of the optimized mining shape. If dilution occurs within the block model, it is assigned block model grades. Otherwise, it is set at zero grades and a bulk density of 2.8 t/m^3. Dilution incurred in converting resources to reserves is summarized by mining method in Table 15-5.

TABLE 15-5: DILUTION INCURRED BY MINING METHOD

	Average Dilution (%)
Development	39.9
PPCF & CF	11
Longhole	24.3
Total	**23.8**

Mining Recovery

Mining recovery is defined as the ratio between mineral resource tonnes delivered to the concentrator and in-situ mineral resource tonnes estimated in the model. Mining recovery assumptions used by type of stoping and backfilling method in the Deswik Stope Optimizer are shown in Table 15-6. These assumptions were based on a combination of first principles and actual performance at the mine since the start of operations. Resultant recovery factors by mining method are shown in Table 15-7. Some of the mineral resources not recovered relate to: rib, post and sill pillars required to maintain rock stability, inefficiencies in mining represented by small blocks of ore along ore/waste contacts and underbreak and inefficiencies in mucking reflecting ore unable to be mucked from longhole stopes.

TABLE 15-6: RECOVERY FACTORS BY MINING METHOD

Mining	Criteria	Backfill	Recovery (%)
PPCF and	Standard 7 x 7m pillars	Waste	75
Longhole	Transverse & Retreat	Paste	89
	Transverse & Retreat	Waste	71

TABLE 15-7: AVERAGE RECOVERY FACTORS BY MINING METHOD

Method	Average Recovery (%)
Development	100
PPCF & CF	75.4
Longhole	88.2
Total	**87.8**

15.4 DESWIK STOPE OPTIMZER

Stopes were built in the Deswik stope optimizer considering measured, indicated and inferred mineral inventories from the block model. Stope construction considered the NSR block values and appropriate dilution and recovery factors based on resource geometry, mining and backfill methods.

To reduce the possibility of including low grade material in stopes, cut-off costs attempt to replicate the steady state cost of mine and mill operations, and include ore extraction, onsite Mine G&A and mill operating costs. This generates stopes that might be mined economically during the normal operation of the mine. Cut-off costs ($/tonne) used to construct stopes in the stope optimizer are shown in Table 15-8.

TABLE 15-8: CUT-OFF COSTS

Mining Backfill Mill	Longhole Paste Stall	Longhole Waste Stall	PPCF Waste Stall	Longhole Paste New Brit	PPCF Waste New Brit
Mining	$15.57	$15.57	$37.15	$15.57	$37.15
Ore	$15.70	$15.70	$15.70	$15.70	$15.70
Backfill	$10.02	$1.63	$1.63	$10.02	$1.63
General	$36.36	$36.36	$36.36	$36.36	$36.36
Subtotal	**$77.65**	**$69.26**	**$90.84**	**$77.65**	**$90.84**
Mill	$26.93	$26.93	$26.93	$34.38	$34.38
DSO	**$104.58**	**$96.19**	**$117.77**	**$112.03**	**$125.22**

Iterations were run considering mining methods and preferred milling location. At this stage, the multiple runs of the stope optimizer continued to consider measured, indicated and inferred resources. The sequence used to build the preliminary Deswik stope inventory is:

1. Attempt to build longhole stopes in the stope optimizer using payability from the flotation only milling process (NSR Stall) assuming longhole mining and Stall milling cutoff costs.
2. Where longhole stopes were unable to be built by the stope optimizer, or where excessive dilution was designed into the stope, attempt to design post pillar cut and fill stopes using the stope optimizer, using higher cut and fill mining costs which incurs less planned dilution and the Stall milling cutoff cost.
3. For gold lenses that are distinctly separated from zinc lenses, attempt to build longhole stopes using payability and costs from the New Britannia concentrator. This assumes higher gold payability and higher mill operating costs.
4. For gold lenses that are distinctly separated from zinc lenses and for which longhole stopes cannot be designed for, or for areas incurring high dilution, attempt to design post pillar cut and fill stopes using payability from New Britannia concentrator and using higher cut and fill mining costs, which incurs less dilution.

In the first pass, 15.3 million tonnes (diluted and recovered) of stopes were built assuming all material would go to Stall. In the second pass, the stope optimizer was run considering material would be sent to New Britannia and higher payable gold and silver for Lenses 21, 25 and 27. 7.9 million tonnes (diluted and recovered) of stopes were built on this run. A level by level analysis was done to remove stopes that were duplicated in the two runs. This removed 2.0 million tonnes (diluted and recovered) from the Stall option.

15.5 STOPE NET SMELTER RETURN CALCULATION

To determine if stopes could be included in the final mine plan and in the mineral reserve estimates, only those stopes supported by measured and indicated resource estimates were retained. Then, the tonnes and grades contained in the DSO stopes were subjected to a secondary financial test using 2019 budget unit costs. Each stope was assigned to a mill feed to determine stope revenues, and a mining method and backfill method to determine mining costs. Direct mining costs derived from mining and backfill method, onsite G&A costs, offsite administration costs, mine sustaining capital costs and stope specific waste development costs were applied on a stope by stope basis to determine final stope economics:

- Mining, milling and onsite G&A costs are based on the 2019 budget and are applied to each stope generated in the DSO as shown in Table 15-8
- Capital equipment costs were applied at $7.17/t to all stopes generated in the DSO.
- Capital waste development includes ramp and level development metres required to access new mining areas:

- Ramp and ancillary development (remucks, sumps, etc.) metres were estimated and applied to mining areas at 2019 budget unit costs.
- Haulage drift metres (footwall drifts) were estimated based on the stope arrangement and were applied as required to access additional stopes along strike at 2019 budget unit costs.
- Stope specific operating waste development includes longhole stope cross cuts and cut and fill entrance cross cuts. Estimated metres were applied on a stope by stope basis at 2019 budget unit costs.

Offsite administration costs were applied at 2019 budget unit costs and are based on copper and zinc concentrates produced from stopes.

15.6 ORE DEVELOPMENT

The final step to establishing mineral reserves was to create ore development shapes required to mine stopes. 3D development shapes were built and removed from the stopes using Boolean operations in Deswik. The development shapes were interrogated and assigned as mineral reserves as shown in Table 15-9.

TABLE 15-9: MEASURED AND INDICATED STOPE AND DEVELOPMENT RESERVE

NSR Option Stall and New Britannia Option	Tonnes (million)	Au	Ag	Cu	Zn
Stall Stopes	9.77	2.97	27.15	0.60	5.74
New Britannia Stopes	2.61	6.86	23.91	1.10	0.36
Total Stopes - Development	**12.38**	**3.79**	**26.47**	**0.71**	**4.60**
Stall Development	0.91	2.89	23.89	0.46	4.22
New Britannia Development	0.38	5.65	19.83	0.95	0.30
Total Ore Development for Stopes	**1.29**	**3.69**	**22.71**	**0.61**	**3.08**
Total Proven and Probable Reserve	**13.68**	**3.78**	**26.11**	**0.70**	**4.46**

15.7 RECONCILIATION OF RESERVES TO PRODUCTION

Reconciliation between reserve estimates and past production provide a useful insight into the reliability of the LOM plan to predict future production. Although no new stope design is available for the areas that have been already mined out at Lalor, the resource model with properly calibrated factors can be used as a proxy to mineral reserve estimates to conduct this reconciliation exercise.

Conversion factors were developed in areas where stopes have been designed to approximate mineral reserve estimates included in the present LOM plan from tonnes and grade reported as individual 5m x 5m x 5m blocks above a cut-off grade from the resource model. This exercise was restricted to the areas with proven reserves only for the base metal lenses as they are deemed to resemble most the portions of the mine that have been mined out to date.

Applying a recovery of 65% for metal lenses and 60% for gold metal lenses together with a dilution 15% on individual blocks above a cut-off of 3% Zn or 2g/t Au, depending on the type of lens, were found to provide a very close match between tonnes and grade reported above cut-off and mineral reserve estimates as illustrated in Table 15-10. The low recovery factors used for the base metal and gold lenses to match the reserve estimates illustrate the potential for low conversion factors when using a methodology based on block above cut off to report mineral resource estimates in an underground mine.

TABLE 15-10: ADJUSTMENT FACTORS FROM BLOCKS ABOVE CUT-OFF TO RESERVE ESTIMATES

Blocks above cut-off		Tonnes	%Zn	g/t Ag	%Cu	g/t Au	Recovery	Dilution	Tonnes	%Zn	g/t Ag	%Cu	g/t Au
Base Metal	Proven	7,126,761	9.12	27.60	0.81	2.13	65%	15%	5,327,254	7.76	23.46	0.69	1.81
Gold	21,23&25	2,835,704	0.44	39.04	0.45	9.47	60%	15%	1,956,636	0.38	33.18	0.38	8.05
							Total after adjustment		**7,283,890**	**5.77**	**26.07**	**0.61**	**3.49**

- The following cut-off were used:
  - Base metal: %Zn>3 or %Zn<3 and %Cu>1
  - Lens 21,23&25: g/t Au>2g/t

2019 Reserves		Tonnes	%Zn	g/t Ag	%Cu	g/t Au
Base Metal	Proven only	5,361,955	7.33	26.91	0.81	2.46
Gold	21&25 only	1,809,000	0.34	25.60	0.33	6.76
Total		**7,170,955**	**5.57**	**26.58**	**0.69**	**3.54**
	Calibration	2%	4%	-2%	-12%	-2%

In a second step, the adjustment factors calibrated from mineral reserve estimates were applied to the tonnes and grade reported as individual blocks above cut-off in the areas already depleted by mining production between 2012 and 2018 and reconciled to the tonnes and grade credited by the mills to Lalor mine. Recognising that the areas mined out to date in lenses 10 and 20 were slightly more continuous than what is left to be mined, the recovery factor applied to the base metal lenses in Table 15-10 was increased from 65% to 70%. The results of this reconciliation are summarised in Table 15-11 and show a close comparison on all parameters. The reported tonnes and grade from the resource model are globally within 5% of actual production and on the conservative side, i.e. marginally under-estimate the quantity of metal actually recovered by the mill. The performance of the reserve model against actual production credited from the mills will continue to be monitored on a continuous basis.

TABLE 15-11: RECONCILIATION BETWEEN MINE PRODUCTION AND THE RESOURCE MODEL

Blocks above cut-off in mined out		Tonnes	%Zn	g/t Ag	%Cu	g/t Au	Recovery	Dilution	Tonnes	%Zn	g/t Ag	%Cu	g/t Au
Base Metal	Proven	5,818,829	9.60	24.93	0.85	1.96	70%	15%	4,684,157	8.16	21.19	0.72	1.67
Base Metal	Probable	318,393	8.57	23.77	0.60	1.43	70%	15%	256,306	7.28	20.20	0.51	1.22
Gold	Probable	526,561	0.45	36.18	0.40	9.50	60%	15%	363,327	0.38	30.75	0.34	8.08
Gold	Inf	75,812	0.49	27.37	0.23	5.95	60%	15%	52,310	0.41	23.27	0.19	5.06
Total	2P + Inf	6,739,595	8.74	25.78	0.80	2.57	**Total after adjustment**		**5,356,101**	**7.52**	**21.81**	**0.68**	**2.12**
							Actual Production		**5,599,564**	**7.59**	**22.82**	**0.72**	**2.13**
							Reconciliation		**-4%**	**-1%**	**-4%**	**-5%**	**-1%**

- The following cut-off were used:
  - Base metal: %Zn>3 or %Zn<3 and %Cu>1
  - Lens 21,23&25: g/t Au>2g/t

The author considers the dilution and recovery factors to be appropriate for the mining methods selected and the stope geometry shape and applicable parameters of the Lalor ore body and the methodology of converting mineral resources to mineral reserves.

The conversion of resources to reserves is based on the LOM strategy and NSR cut-offs that primarily focused on capturing base metal resources with gold resources in contact with base metal resources for processing at the Stall concentrator. In areas where a large separation existed between base metal and gold lenses, gold lenses were evaluated independently of the base metal lenses to provide feed to a gold mill.

Of the 13,675,000 tonnes of mineral reserves, approximately 78% is converted from base metal resources and approximately 22% is converted from gold or copper gold zone resources.

16. MINING METHODS

Geology, geotechnical information, orebody geometry, productivity and mining experience are major drivers to select the best suited mining methods at Lalor including considerations for lateral development, production, backfilling and ore transportation used at Lalor mine.

The hanging wall and footwall rocks are generally of good quality, allowing the use of mechanized drilling and blasting techniques. The mineralized lenses dip at an average of 30°, but locally varying from 10 to 55°. Mining methods currently in use include: cut and fill and longhole open stope. Paste backfill is used to increase recovery and accelerate the mining cycle. Low grade areas will be filled with rock from waste development.

Ore is mucked using Load Haul and Dump (LHD) loaders which are operated remotely in inaccessible areas. Ore is loaded into underground haul trucks or ore passes and transported to the ore handling system at the production shaft for hoisting to surface. Ore delivered to the production shaft is sized to less than 0.55m by one of two rock breakers. Ore is hoisted from the mine by two 16 tonne capacity bottom dump skips in balance. On surface, ore is truck hauled to a primary crusher at the Chisel North mine site, crushed to less than 0.15m, and then trucked to either the Stall or New Britannia concentrator for processing.

16.1 LATERAL DEVELOPMENT

Lateral development is horizontal or near horizontal tunnelling, travelable by rubber tired mechanized equipment. Ramps are typically mined at up to +/- 15% grade to provide access between levels. Rounds are 4m long and typical dimensions are 6m wide by 5m high. Standard ground support, consists of resin grouted rebar and welded wire mesh across the back and to within 1.8m of the sill on the walls and face. Mine services, including compressed air, process water and discharge water pipelines, paste backfill pipeline, power cables, leaky feeder communications antenna and ventilation duct, are installed in main levels and stope entrances.

Main levels are developed parallel to and in the footwall of the ore zones. Where possible, main levels are located to provide access to multiple ore zones and are connected to haul ramps to allow mechanized equipment to travel from level to level. Cut and fill access crosscuts are driven at -15% to allow for multiple cuts. For transverse longhole mining, crosscuts are developed in waste from the footwall through the ore. Longitudinal retreat longhole stopes have a primary access crosscut and a drift is mined in ore along the orebody to provide drilling and mucking access.

16.2 VERTICAL DEVELOPMENT

Main ventilation raises, and ore pass raises are developed by a mining contractor using a raisebore and/or Alimak climber. Ground support and ladder ways are installed as required. Longhole slot raises, transfer ore passes, and auxiliary ventilation raises are limited to approximately 35m long and are developed using longhole conventional drop raise techniques. Drain, paste backfill and electrical cable holes are drilled using longhole or raisebore drills and are reamed to designed diameter.

16.3 STOPE MINING

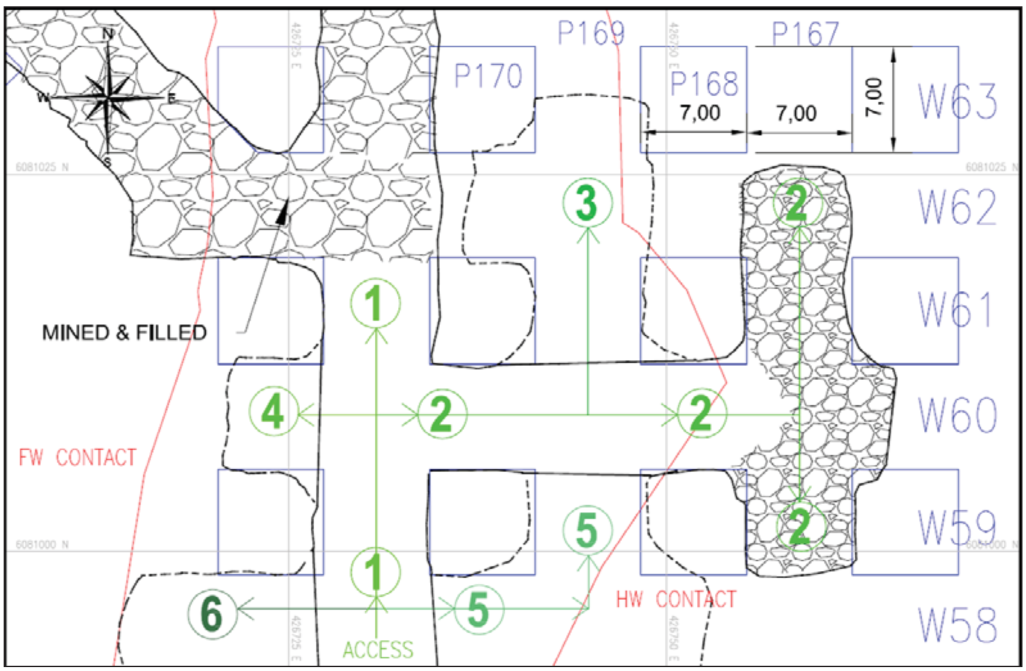
The two mining methods used at Lalor mine are variations of cut and fill mining or longhole open stope mining. In general, where the dip exceeds 35° and the orebody is of sufficient thickness, longhole open stope mining is preferred and represents 89% of total stope mining. Cut and fill mining methods are used in narrower, flatter areas. Stope mining accounts for 91% of all mining with development accounting for the other 9%.

Cut and Fill Mining

Single pass overhand mechanized cut and fill mining is the preferred method where the ore is flatter than 35° and horizontal widths are less than 10m. The ore is accessed from a footwall drift by a crosscut developed at approximately -15%. Ore is mined in 5m high horizontal cuts. Unconsolidated backfill is placed tight to the back and the entrance crosscut is then back slashed to provide access to the next cut.

Where ore widths are greater than 14m wide, mining is conducted by overhand post pillar cut and fill. Post pillars provide ground support to allow for selective mining of wide stopes. Backfill is placed to within 1.8m of the back and the entrance crosscut is back slashed to provide access to the next cut. Drifts and crosscuts in stopes are typically 7m wide x 5m high, with 7m x 7m post pillars. Mining retreats towards the access to reduce risks associated with poor ground conditions. A typical mining sequence is shown in Figure 16-1 as a plan view.

FIGURE 16-1: TYPICAL POST PILLAR CUT AND FILL MINING PLAN VIEW



Longhole Open Stope Mining

Benefits of longhole mining over cut and fill methods include reducing exposure to risk as a non-entry method and higher productivity/lower cost than cut and fill methods.

The minimum design height of a longhole stope is 17m. In shallower dipping areas of the ore the interval is typically 17 to 20m, and in steeper dipping areas the interval is 20 to 23m. Stope widths assume a minimum mining width of 5m. Maximum open stope length and width is determined considering the rock quality and hydraulic radius of the combined exposed hanging wall and back span of the stope. A 5m rib pillar is added when unconsolidated rock is used as backfill.

In transverse longhole open stope mining, ore is undercut at the top and bottom of the block from cross-cuts off the footwall drift, providing access for drilling and mucking, as shown in Figure 16-2. A primary-secondary sequence is applied when using paste backfill.

In longitudinal retreat mining, the stope is undercut in ore at the top and bottom of the block providing access for drilling and mucking. Stope sequence is typically bottom up and retreating along the length of the sill to an access cross cut at the end of the lens or from both ends of the sill toward a central access point, see Figure 16-3. Ore at the top of plunge lines or sill pillars will be recovered from longhole stopes below.

FIGURE 16-2: TYPICAL ISOMETRIC VIEW- TRANSVERSE LONGHOLE OPEN STOPING

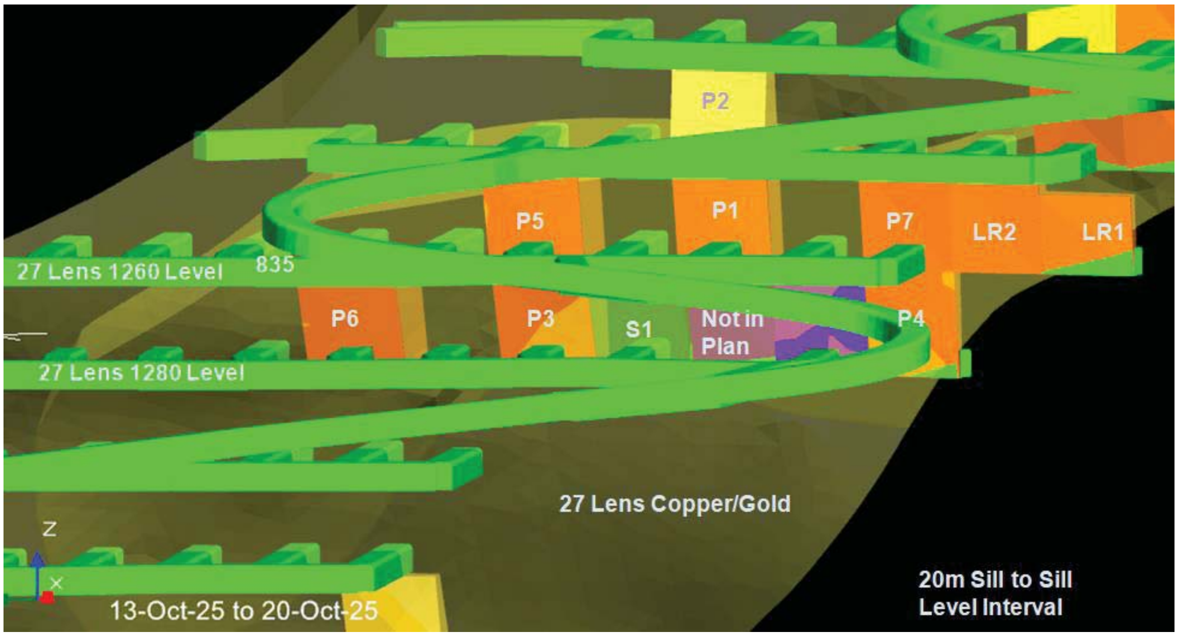
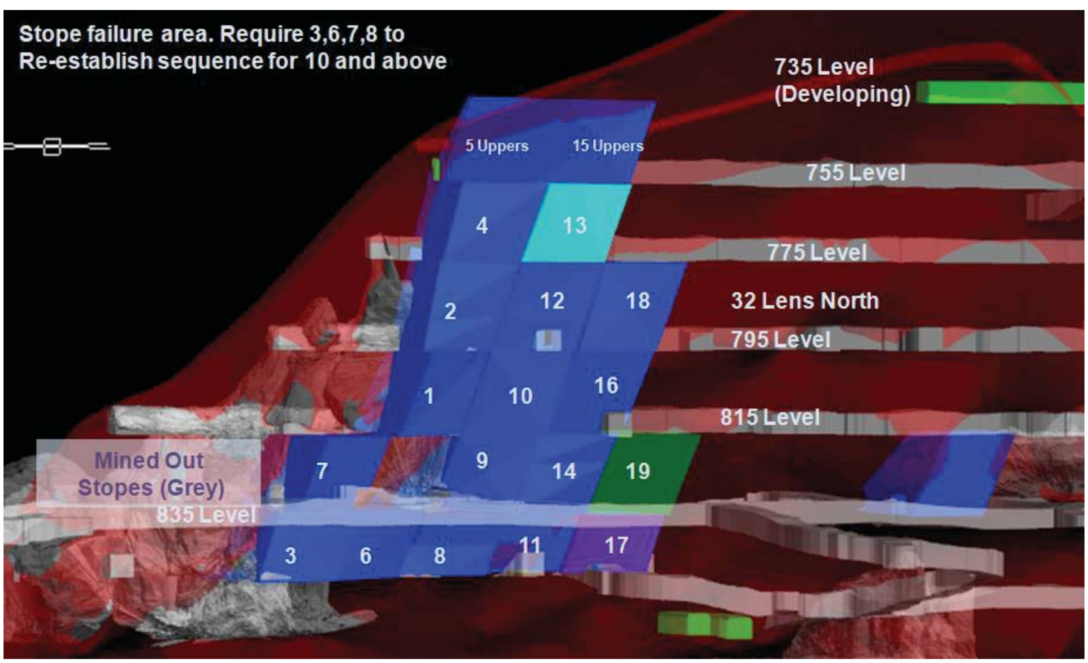


FIGURE 16-3: TYPICAL LONG SECTION - LONGITUDINAL RETREAT LH OPEN STOPING



Teleremote and Autonomous Mucking

Teleremote and autonomous mucking allows stope mucking to and orepass to continue through shift change and blast clearing times. Lalor Mine uses Epiroc's Autonomous Mucking System which includes an operator's station on surface allowing control of up to three LHD's concurrently. The surface operator drives the LHD remotely to record a route and then switches to autonomous mode. The LHD travels the recorded route without operator input. Safety devices are installed on the mucking level to restrict access by mine workers and equipment during the operation. Future plans are to isolate truck routes from the stoping areas allowing teleremote loading of trucks as the mine expands below shaft bottom.

16.4 BACKFILL

All stopes at Lalor mine except uppers require backfill to maintain long term stability and to provide a floor to work from for subsequent mining. Backfill is either unconsolidated waste rock backfill (URF) or consolidated paste backfill. URF is used in stopes where pillar or wall confinement is not required and the value of the adjacent pillars does not warrant the added expenditure of paste backfill. Paste backfill is an engineered product comprised of mill tailings and a binder (3-5% cement by weight), mixed with water to provide a thickened paste that is delivered by borehole and pipes to stopes. Paste backfill presents several advantages over unconsolidated fill such as slurried mill tailings or loose waste rock including increased stability, better ventilation controls, higher mining recovery and lower mining dilution and it also eliminates potential flows of re-liquefied unconsolidated tails.

16.5 ORE HANDLING

Ore is mucked by LHD, loaded to underground haul trucks and hauled to one of the two rock breaker/grizzlies on 910m level for sizing to less than 0.55m. A 40m bin below the grizzly provides approximately 1,200 tonnes of coarse ore storage. A chute at the bottom of the bin at 955m level feeds ore to a conveyor that loads a measuring flask with approximately 14 to 16 tonnes of ore. Ore is skipped to surface by two 16 tonne capacity bottom dump skips in balance. The ore enters the headframe chute from the skips and is deposited into the surface ore bin or to the exterior concrete bunker via gravity. From the surface bin or bunker, ore is truck hauled to a primary crusher at the Chisel North mine site, crushed to minus150mm, and then trucked to a concentrator to process.

16.6 GEOTECHNICAL DESIGN

A geotechnical database has been established from drill core logs and geotechnical core sample laboratory tests. Geotechnical data is continuously collected following standard procedures. This is done by the Lalor Geology department and data is shared with the Ground Control team. Adequate core samples are collected for all the ore zones to ensure we have sufficient geotechnical data from hanging wall, orebody and footwall to further test rock properties. Ground monitoring instruments (dCABLES) are used for ground movement monitoring and more advanced ground control instrumentation will be considered going forward.

Core logs are examined to identify and characterize any major structural features, including major faults and/or shear zones. Representative samples of hanging wall, ore zones and footwall rock are collected to test rock properties, including Unconfined Compressive Strength (UCS), Young Modulus and Poisson Ratio. Rock mass classification data (RQD) is generated in 3D space, similar to the ore grade distribution from all diamond drill holes.

Numerical modeling is being done to help understand ground conditions and to assist ground control decisions. Software used is Map 3D, Examine 2D, ExamineTab and phases 2. A Map 3D model simulates the mine wide stress situation and is used for localized stress and ground stability analysis.

Lalor Mine has a formal Ground Support Standard Policy in place to guide the mining ground support practice. The standard policy is updated on a regular basis to reflect the current mining environment and ground condition changes as mining advances into new areas. Third party ground control experts conduct bi-annual ground control reviews and audits to maintain high mine safety standards.

16.7 GROUND SUPPORT SYSTEMS

Except when using cut and fill mining methods, drifts have arched backs to increase stability and factor of safety. Ground support is classified as either primary support or secondary support.

Primary support refers to reinforcement of the rock mass immediately following excavation to ensure safe working conditions before re-entry. Secondary support is additional support applied after the installation of primary support to provide further support in large spans, long term infrastructure excavations and structurally controlled areas where wedge failures may be a concern.

Primary Support

The standard for Lalor is #6 or #7 2.3m (7’4”) resin rebar installed on a 1.2m x 1.2m (4’ x4’) square pattern with a same size rebar in the center of the pattern. Screen (#6 gauge welded wire mesh) and bolts are installed on the walls, back and face prior to re-entry. At drift intersections, 3.6m (12’) resin rebar in a 1.2m x 1.2m pattern with a center #7 7’4” rebar is used for primary back ground support.

Secondary Support

Secondary ground support is installed when excavation spans are larger than 10.8m, unfavorable ground conditions or rock structures are present, or after a site ground condition evaluation indicates it is required. Secondary ground support uses heavy duty longer bolts, such as cement grouted cable bolts. Typically, double cable bolts on a 1.8m x 1.8m pattern are used for long term excavations. High strength inflatable rock anchors may be used for temporary or short-term excavations. The minimum bolt length should be equal to one-third of the final drift span.

Developed in Two Phases- Mechanized C&F and Infrastructure

Headings wider than 7.0m are developed using two methods, single pass and double pass. First pass development ground support is as described in the sections entitled: Primary Support and Secondary Support above. On completion of the first pass and before starting the second pass, secondary ground support is installed. It should be noted that cablebolt grout and shotcrete must be allowed to cure for at least 24 hours before any blasting within 30m. The second pass is supported similarly to the first pass or the area is designated for remote operations only.

Sill Pillars

Poor ground conditions are to be expected in the last three cuts (15m) in post pillar or for the last lift of longhole stopes in each block. Drifts driven under excavated stopes require cable bolting when the sill pillar thickness is less than twice the width of the drift. The minimum length of the cable bolts is equal to the width of the drift. Shotcreting may also be required if conditions warrant.

16.8 UNDERGROUND DEVELOPMENT

Drifts and ramps will be developed by dedicated mine development crews. Lateral development required by year to mine the LOM mineral reserve estimates is shown in Table 6-1.

TABLE 16-1: LATERAL JUMBO DEVELOPMENT

Year	Sustaining Capital (m)	Mine Operating Waste (m)	Mine Operating Ore (m)	Total (m)
2019	2,584	1,986	5,457	10,027
2020	3,182	3,127	3,803	10,112
2021	2,701	3,320	3,600	9,621
2022	2,255	2,278	3,475	8,008
2023	1,338	2,016	3,441	6,795
2024	706	1,058	1,617	3,381
2025	565	674	680	1,920
2026	285	460	563	1,308
2027	0	317	217	534
Total	**13,616**	**15,236**	**22,853**	**51,706**

Vertical Development

RAISES

Vertical development or raises are planned at Lalor mine for extension of the ventilation system and secondary egress. Drop raises used as slots in longhole stopes are included in stope drilling and blasting. Raises less than 35m can be mined as drop raises. Longer raises are evaluated to choose between raise boring and Alimak considering the conditions, end use and cost of both methods. Vertical development required by year to mine the LOM reserves is shown in Table 16-2.

TABLE 16-2: VERTICAL DEVELOPMENT

Year	Vent Raises (m)	Ore Handling Raises (m)	Total (m)
2019	364	84	448
2020	636	0	636
2021	540	0	540
2022	451	0	451
2023	268	0	268
2024	141	0	141
2025	113	0	113
2026	57	0	57
2027	0	0	0
Total	**2,570**	**84**	**2,654**

PASTE BACKFILL, DRAIN AND SERVICE HOLES

Paste backfill is distributed to stopes via a series of steel cased vertical holes between levels and pipelines installed in footwall drifts. Paste backfill holes are drilled using a raisebore pilot bit. Service holes are holes drilled between levels for electrical cable distribution throughout the mine. An allowance of 240m per year of paste and service holes has been included in the mine plan.

16.9 DIAMOND DRILLING

Diamond drilling is done to define the extents of mineralization and is required for forecasting grades from stopes. The delineation holes are typically drilled from a drift located in the footwall or hanging wall. Stope definition holes are drilled from stope drawpoint stubs or undercut drifts for grade planning (which could vary sequence in some cases).

16.10 MINE DRAINAGE AND DEWATERING

Lalor mine is a relatively dry underground mine with no significant hydrological concerns. The drainage for Lalor mine and the past producing Photo Lake and Chisel North mines is interconnected. Mine water from all mine sources is pumped to the Chisel Open Pit, where it is pumped to a water treatment plant and then released to the environment. Main collection areas feeding the water treatment plant are: 140m level pump station at Photo Lake, a pumping station at Chisel North 540m level (referred to as the 990 sump) and satellite sumps at Lalor mine.

16.11 MINING OPERATIONS

Production is expanding to a steady state of 4,500 tpd in the first quarter of 2019.

Mining Fleet

Lalor mine is a ramp and shaft accessible mine with production and development done by rubber tired underground mining equipment. The mine equipment fleet required to achieve 4,500 tpd is shown in Table 16-3. Lalor mine production will be approximately 20% jumbo and 80% longhole. Additional contractor equipment not listed in Table 16-3 is planned to be on-site to complete the sustaining capital lateral development metre requirements in the LOM plan. An allowance for replacement of mine equipment has been included in the mine plan.

Mine Power

Grid electricity is supplied by Manitoba Hydro, the provincial power utility. Manitoba Hydro's 115 kV power line terminates at the Chisel North mine site, approximately 3.5 road km from the Lalor mine site. This supplies power to the Hudbay owned main distribution substation consisting of two (2) 115-25 kV 24 MVA transformers.

TABLE 16-3: MINE MOBILE EQUIPMENT

Description	Fleet
Underground Trucks 65 tonne	2
Underground Trucks 42 tonne	6
Underground Trucks 42 tonne Ejector	1
LHD 14 Tonne	5
LHD 18 Tonne	6
LHD – Service	3
Two Boom Jumbo	5
Roof Bolters (includes require for cable bolting)	8
Longhole Drills	4
Blockholer	2
Powder Truck	3
Scissor Lift	8
Boom Truck	3
Grader	2
Backhoe	4
Cable Reeler	1
Forklift	12
General Service Truck (Fuel, water, etc.)	7
Skid Steers	3
Front End Loader	3
Personnel Carriers Toyota	25
Surface Light Trucks	15
Total Mine Equipment	**128**

Production Schedule

The Life of Mine (LOM) production schedule is shown in Table 16-4 to Table 16-6. The Deswik software was used to assist with the LOM sequencing and scheduling to generate the production schedule. Historic mining rates and systems, equipment or crews capabilities were applied and levelled. From this output, adjustments were made to further balance the capabilities to create the final plan. Concentrates produced from Lalor and contained metal in concentrates are also shown in tables bellow.

TABLE 16-4: LOM PRODUCTION SCHEDULE

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2019	1,588,810	2.41	22.96	0.63	5.43
2020	1,481,216	2.12	28.57	0.62	6.37
2021	1,559,418	2.75	28.23	0.64	6.18
2022	1,607,100	3.71	25.75	0.68	4.28
2023	1,631,103	3.68	24.00	0.72	4.65
2024	1,643,607	3.56	24.10	0.99	4.23
2025	1,440,842	5.28	24.89	0.81	2.54
2026	941,495	5.41	23.96	0.45	2.31
2027	952,003	5.59	28.66	0.51	3.09
2028	830,169	5.61	34.21	0.79	3.69
Total	**13,675,761**	**3.78**	**26.11**	**0.70**	**4.46**

TABLE 16-5: LOM CONCENTRATE AND DORÉ SCHEDULE

	Zinc Concentrate		Copper Concentrate				Dore		
Year	Tonnes	Zn (%)	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Oz	% Au	% Ag
2019	155,533	50.8	41,698	51.7	485.8	20.3	0	-	-
2020	172,547	50.9	37,418	48.2	575.2	20.5	0	-	-
2021	175,154	50.8	41,626	56.2	542.2	20.3	0	-	-
2022	123,323	51	40,975	83.0	523.8	23.4	130,829	25.4%	69.6%
2023	137,137	51	43,139	81.1	490.5	23.9	91,003	26.9%	68.1%
2024	125,699	51	57,336	60.6	373.4	25.6	107,628	24.8%	70.2%
2025	63,350	51	42,285	102.0	494.1	24.8	116,201	28.5%	66.5%
2026	36,031	51	16,640	171.0	797.7	22.2	77,967	29.1%	65.9%
2027	50,859	51	18,982	156.2	785.3	22.1	96,944	24.8%	70.2%
2028	44,977	51	25,140	108.1	558.8	23.5	86,035	23.7%	71.3%
Total	**1,084,610**	**50.9**	**365,239**	**80.8**	**524.3**	**22.8**	**706,607**	**26.1%**	**68.9%**

TABLE 16-6: LOM METAL PRODUCTION SCHEDULE

Year	Zn (tonnes)	Cu (tonnes)	Au (oz)	Ag (oz)
2019	79,045	8,444	69,329	651,279
2020	87,794	7,664	57,968	692,018
2021	89,046	8,455	75,194	725,579
2022	62,895	9,575	142,529	781,195
2023	69,940	10,294	136,924	742,301
2024	64,106	14,672	138,475	763,879
2025	32,309	10,478	171,831	749,057
2026	18,376	3,689	114,181	478,153
2027	25,938	4,204	119,379	547,354
2028	22,938	5,914	107,709	513,064
Total	**552,387**	**83,389**	**1,133,519**	**6,643,879**

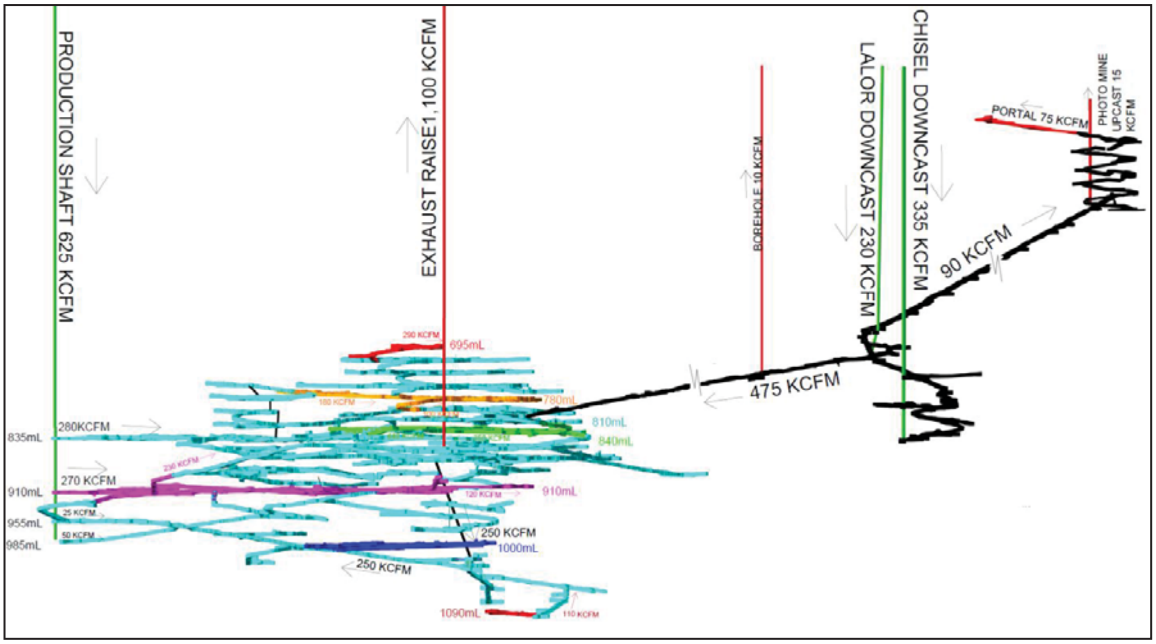
Mine Ventilation

Lalor mine is a push pull system primarily driven by pull. The exhaust shaft is equipped with two fans drawing 1.1 million cfm from the mine (Figure 16-4). Fresh air is supplied into the mine from the Chisel and Lalor downcast raises (0.6 million cfm). The Chisel and Lalor downcast raises are equipped with 600HP and 400HP fans respectively. An additional 0.6 million cfm is downcast via the Lalor production shaft, equipped with two 250HP fans. In the upper mine, ventilation raises and an internal ramp between 795mL to 755mL will provide ventilation to expanding stoping fronts. In the lower mine, an extension of the 810 to 1025mL raise is planned to connect 1025mL to 1090mL. An internal exhaust raise is planned from the

1000mL to the bottom of the exhaust shaft at 840mL. This will improve blast clearing times for the lower part of the mine. Individual mining faces are ventilated using 60 HP to 250 HP fans and 1.37 m ventilation duct. Three propane heaters heat air in the mine during the winter from Chisel, the access ramp and the production shaft.

Ventilation planning utilizes Ventsim modelling to create intermediate stages from the LOM Deswik schedule. The Ventsim model is continually updated as the mine continues to expand both upward and downward in multiple lenses.

FIGURE 16-4: CURRENT LALOR MINE VENTILATION LONG SECTION



16.12 WORKFORCE

The majority of operations, maintenance and technical personnel work 11.5 hour shifts on a 5-5-4 day cycle or a 7-7 day cycle or 8 hour day shifts, 40 hours per week. The mine is operated under Collective Bargaining Agreements between Hudbay management and local unions. The mine operations workforce is comprised of Hudbay hourly operations and maintenance personnel as well as salaried supervision, mine administration and technical staff, plus contractor personnel for specialized work and workforce shortages. Personnel count will vary year to year. Steady state personnel requirements are shown in Table 16-7 .

TABLE 16-7: MINE OPERATIONS WORKFORCE

Discipline	Personnel
Direct Operations	279
Supervision and Administration	64
Health and Safety	11
Mine Maintenance	131
Mine Technical	52
Total Lalor mine	**537**

16.13 MINE SAFETY AND HEALTH

All personnel are required to work under the applicable laws of the Province of Manitoba, Canada. All contractors working on site are required to have an approved health and safety program in place and have on site representation. Hudbay Plant Safety Rules and Regulations are used at Lalor mine including safety and health monitoring programs, dust, water and environmental monitoring, personal protective equipment and task analysis and job procedures completed by employees daily.

Refuge Stations

Refuge stations are required at Lalor mine as per mine regulations and Hudbay standards and are incorporated into the mine design. Hudbay's standard refuge station is excavated from rock and requires two ventilation bulkheads, compressed air. Depending on the size of the refuge and the people working in the area a backup oxygen generator may be installed. Potable water, stretcher kit first aid supplies and supplies to seal off the bulkheads are installed. In new development areas portable refuge stations replace excavated refuge station.

Personal Evacuation Infrastructures

The primary route in and out of the mine is the production shaft equipped with a service cage. The shaft is equipped with a small auxiliary hoist and six person cage. In case of power failure, the auxiliary hoist can be operated by an emergency diesel generator to evacuate personnel from the mine. If the production shaft is not usable, the second egress from the mine is the main ramp to surface at Chisel North mine.

17. RECOVERY METHODS

The Lalor run of mine ore, as large as 0.55m in one dimension, is withdrawn from the Lalor head frame ore bin by an apron feeder and transported to a crushing plant. This crushing plant is located at the Chisel North mine and is operated by a third party contractor. The product from the crushing plant is trucked to the Stall or Flin Flon concentrators for base metal feed. In the case of gold feed, the crushed product will be sent to the New Britannia mill when refurbishment is completed after 2021.

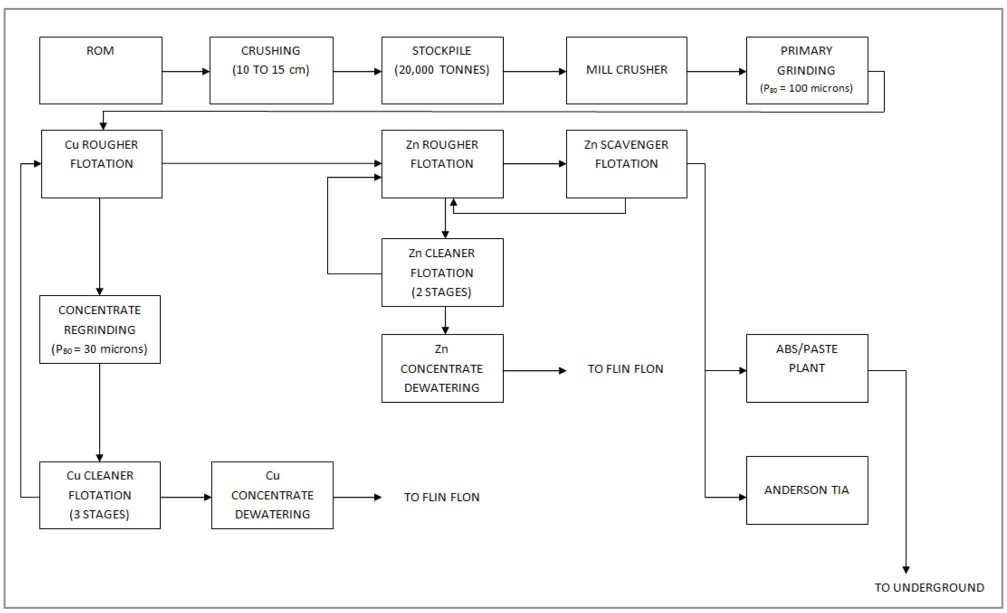
17.1 STALL CONCENTRATOR

General Layout and Facility Description

The Stall concentrator complex is located approximately 16 km east of the Lalor Mine. Conventional crushing, grinding and flotation operations are used to process the feed. The nominal throughput rate is 3,500 tpd. The mill operates 24 hours per day, 365 days per year, with scheduled downtime for maintenance as required.

The concentrator produces a copper concentrate with gold and silver credits and a zinc concentrate. Both concentrates are shipped by truck to Flin Flon. From there the copper concentrate is loaded onto rail cars and shipped to third party smelters. Tailings from the flotation circuit are utilized to produce a cemented paste backfill for use underground. Tailings not required for paste backfill will continue to be pumped to the existing Anderson TIA. A simplified block flow diagram is shown in Figure 17-1.

FIGURE 17-1: LALOR CONCENTRATOR SIMPLIFIED BLOCK FLOW DIAGRAM



Crushing and Grinding

The crushed material is received at the coarse ore bins or directed to a stockpile. The stockpile has a capacity of 20,000 tonnes, equivalent to approximately 6.5 days of production. This stockpile is used for blending in order to achieve a more consistent feed of less than 13% Zn and less than 1% Cu. Material is reclaimed from the bins via one of two vibrating feeders and discharged via a conveyor into a jaw crusher. The discharge is combined with the secondary Symons cone discharge to feed a double deck vibrating screen. The undersize from the bottom deck and the final product from the crusher circuit is conveyed to the fine ore bins (FOB) while the oversize of the two decks is combined to feed the Symons cone crusher and then recirculated to the screen deck in a closed loop.

From the FOB, the crushed feed is reclaimed and fed to each mill via two conveyors. One feeds the Stall circuit and the other feeds the Chisel circuit. The Chisel grinding circuit operates independently of the Stall circuit, enabling the plant to operate if one of the lines is down for maintenance.

The grinded product discharges onto each rod mill which in turn discharges into a dedicated cyclone feed pump box where it is combined with the ball mill discharge and with dilution water. Each cyclone underflow returns to their respective ball mill. The cyclone overflow from each circuit combines and discharges into the cyclone overflow pump box. The product reports to a copper conditioning tank through a trash screen. Water is added in the milling process to optimize the grinding density. Flotation reagent 3418A and lime are added to each rod and ball mill to control the pH of the feed of the flotation circuit.

Flotation

From the copper conditioning tank, the slurry is pumped to the copper rougher consisting of ten 8.5m^3 flotation cells, where it is combined with the copper first cleaner tailings. The concentrate from the copper rougher flotation cells is pumped to the copper regrind mill circuit which consists of a 2.43m x 3.65m (400 HP) mill. The mill is operated in closed circuit with a cyclopac with four cyclones. Methyl isobutyl carbinol (MIBC) frother is added to stabilize the froth. The regrind cyclone overflow is pumped to the copper first cleaners where it is combined with the copper second cleaner tailings.

The first cleaner tailings are returned to the copper rougher cells and the concentrate progresses to the second cleaners where it is combined with the third cleaner tailings. There is an option to direct the copper first cleaner tailings to the zinc rougher feed conditioner. The third cleaner concentrate, produced from five DR24 flotation cells is pumped to the copper concentrate thickener.

The zinc flotation feed consists of the copper rougher flotation tailings, the zinc first cleaner tailings and scavenger tailings. The zinc rougher feed is first conditioned in the zinc conditioner tank using copper sulphate, 7279 collector and MIBC frother and then pumped to seven 8.5m^3 zinc rougher flotation cells. The concentrate is then pumped to two successive stages of zinc cleaning. The second cleaner concentrate is the final zinc concentrate and is pumped to the zinc concentrate thickener.

The zinc rougher tailings are pumped to the existing zinc scavengers consisting of five 8.5m^3 flotation cells. The zinc scavenger tailings and other streams collect in a tailings box and are the final plant tailings. The zinc scavenger concentrate is returned to the zinc conditioner tank for further recovery.

Slurry samplers and an on-stream X-Ray analyzer in the plant allows for control of the entire flotation circuit and metallurgical accounting.

Concentrate, Water and Tailings Management

Flocculated copper concentrate is pumped to a 3.35m diameter thickener. Underflow, at a target density of 65% solids, is pumped to an existing agitated stock tank. Thickened copper concentrate is further dewatered to approximately 19% moisture on a disc filter. Filter cake is conveyor fed and gravity dropped to the concentrate shed. The flocculated zinc concentrate follows the same process as describe above but is further dewatered to approximately 11% moisture using two disc filters.

The shed is a fully enclosed building and contains partitions for separate areas for zinc and copper concentrate storage. A front end loader is used to separately load the filtered concentrates into dedicated trucks for transport to the Hudbay concentrate handling facilities in Flin Flon.

Plant tailings, which are composed of zinc scavenger tailings, certain sumps and flows from dewatering, are pumped out of the plant to the Anderson TIA for final disposal. The paste plant can divert the tails to the paste plant, depending on the demand for paste.

The Stall Concentrator process utilizes two sources of water:

- Fresh Water: (approximately 25% of water usage) is pumped from Snow Lake 7 km away and used in areas such as the worker's change-rooms/showers, pump gland water, flocculant mixing, On-Stream X-Ray Analyzer and other processes or equipment that requires good quality water with low levels of suspended solids and dissolved compounds.
- Reclaim Water (approximately 75% of water usage) is reclaimed from the Anderson TIA 5 km away and re-used in the concentrator for process density control, launder sprays, hosing and other processes or activities that do not require high quality water.

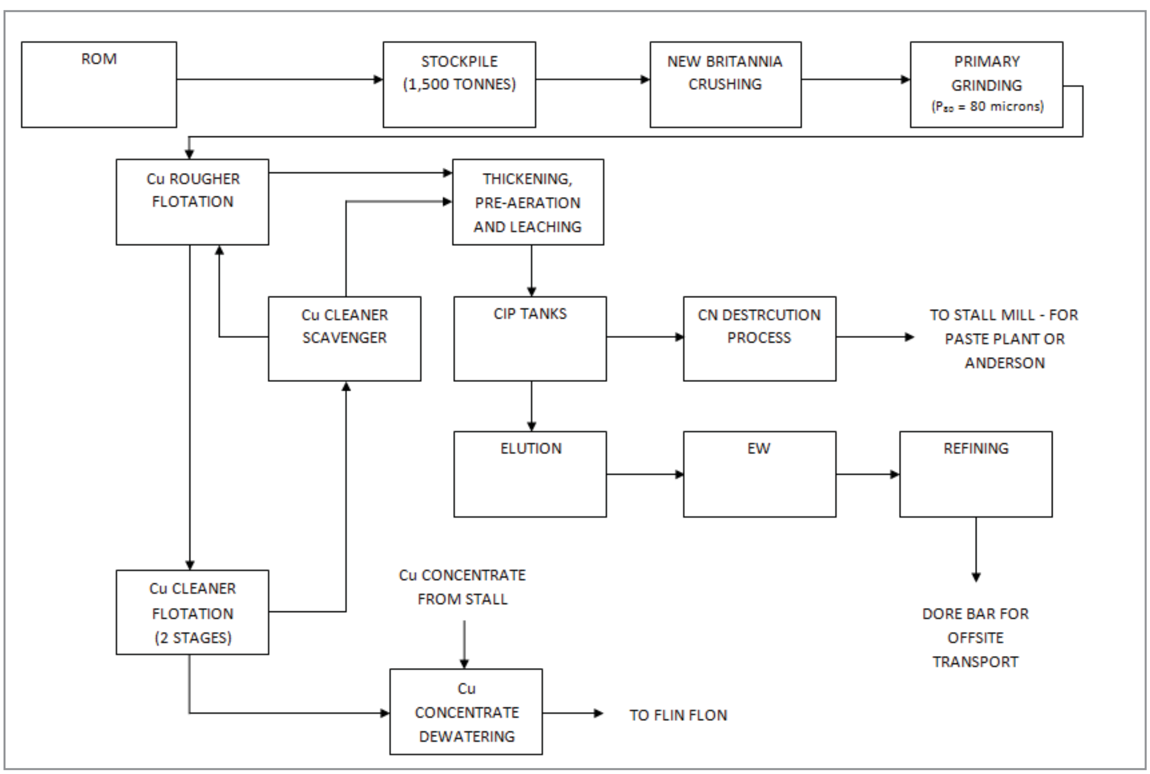
17.2 NEW BRITANNIA MILL

General Layout and Facility Description

The New Britannia mill is located approximately 16 km north of the Lalor mine. Conventional crushing, grinding, flotation and cyanide leach and recovery methods will be used to process the feed. The nominal throughput rate will be 1,500 tpd and the plant will operate 24 hours per day for 362 days per year, with an availability of 92%.

The concentrator will produce concentrate and doré gold bars. The copper concentrate will be shipped by truck to third party smelters. Once the New Britannia mill is fully refurbished and operational, the copper concentrate from the Stall mill will be processed at New Britannia. The doré bar will be shipped offsite using conventional third party transportation. A simplified flow diagram for the planned New Britannia mill is shown in Figure 17-2.

FIGURE 17-2: NEW BRITANNIA MILL SIMPLIFIED BLOCK FLOW DIAGRAM



Crushing and Grinding

Run of mine will be delivered to the New Britannia site using 40 tonne trucks. Once at destination, the material will be loaded into a hopper and conveyed to a jaw crusher set to 50 mm. The feed will then be screened using a double deck vibrating screen with a 19mm bottom deck. The oversize will be sent to a cone crusher while the undersize will be fed to three fine ore bins (FOB) totalling 1,800 tonnes.

The crushed material will be reclaimed from the FOB via conveyors into a rod mill and then discharged into a cyclone feed pump box where it will be combined with the ball mill discharge and dilution water. A variable speed cyclone feed pump will discharge into a cyclone pack. The cyclone underflow will return to the ball mill while the overflow will be directed to the flotation feed pump box and fed to a trash screen and then to the copper rougher conditioning tank.

Flotation and Dewatering of Leach Feed and Copper Concentrate

From the copper rougher conditioning tank, the feed will be sent to the copper rougher stage which will be composed of seven 10 m^3 tanks. The concentrate will go through two stages of cleaning. The concentrate from the second cleaner stage will mix with the copper concentrate from Stall and together the final copper concentrate will be pumped to an eight metre copper thickener. The concentrate will be thickened, filtered, stored in a concentrate shed. Finally, the copper concentrate will be loaded onto dedicated trucks and transported offsite.

The tails from the second cleaning stage will be pumped to the first cleaner stage and the tails from the first cleaner stage will be sent to the cleaner scavenger stage. The cleaner scavenger cells will attempt to do a final recovery on the stream and the concentrate will report to the rougher while the tails will report to the flotation tails pumps.

This cleaner scavenger tail stream will mix with the original rougher tailings. The combined stream, depleted of copper and rich in gold, will report to a 12m pre-leach thickener for thickening the feed and optimize reagent additions prior to cyanidation and eventual gold recovery. The underflow product of the thickener will be set at 50% solids.

Both thickeners overflows (from the copper concentrate and pre-leach thickeners) will be recovered back to the process as process water. Additionally, slurry samplers and an on-stream X-Ray analyzer have been included in the design of the plant for control of the flotation circuit and metallurgical accounting.

Leaching, CIP, Elution and Electrowinning

The thickened cleaner scavenger tails and rougher tails will be fed to one pre-aerating tank followed by three leach tanks in series with aeration for a total retention time of 56 hours at a minimum. Hydrated lime will be added to ensure the pH stays above 11.5 and sodium cyanide solution will be added to leach the gold. The slurry stream will enter six CIP leach tanks in series with 1.2 hr retention time in each. Carbon will be added to the sixth tank and will be transferred in counter-current, absorbing gold as a cyanide-Au complex as it progresses upstream in the tanks.

At the first tank, gold-loaded carbon will be transferred to a fine vibrating screen where it will separate the carbon via elution. The loaded carbon will be washed with a mild acid solution to eliminate impurities and then stripped using a hot caustic cyanide solution. This solution, referred to as pregnant solution, will proceed to electrowinning (EW). The sludge collected will then be treated with fluxes and chemicals to obtain a doré bar.

The stripped carbon, after elution, will retrain significant quantity of organic and inorganic contaminants (foulants) present in the cyanide slurry. Prior to returning to the process, the carbon will be regenerated by removing these contaminants.

The final tails will proceed to a cyanide destruction stage. Prior to leaving the tank, sodium metabisulfite will be added to ensure thorough mixing and conditioning. The cyanide destruction stage will provide 1 hr of retention time and the resulting product of the chemical reaction will have converted cyanide (CN-) to cyanate (CNO-). The less stable cyanate will hydrolyze to ammonium and carbonate ions respectively. In the chemical reactions for cyanide destruction, any base metal present will be liberated and precipitated as metal hydroxides.

The final tails from the New Britannia mill will be sent to the Stall Concentrator via a 6.8 km pipeline. This same pipeline corridor will house the copper concentrate pipe and the reclaim water (both with flow from Stall to New Britannia). Once the tailings line reaches the Stall Concentrator tails pumpbox, the paste plant will have full control to direct the flow to the paste plant or to the Anderson TIA, depending on the needs for paste.

The New Britannia flotation/gold leaching will utilize two sources of water:

Fresh Water (approximately 10% of water usage) will be pumped from Snow Lake. Used in areas such as the worker's change-rooms/showers, pump gland water, flocculant mixing, On-Stream X-Ray Analyzer, carbon elution, carbon regeneration and quenching and other processes or equipment that require good quality water with low levels of suspended solids and dissolved compounds.

Reclaim Water (approximately 90% of water usage) will be reclaimed from the Anderson TIA, pumped from the Stall mill 7 km away and used for density control, launder sprays, hosing and other processes or activities that do not require high quality water.

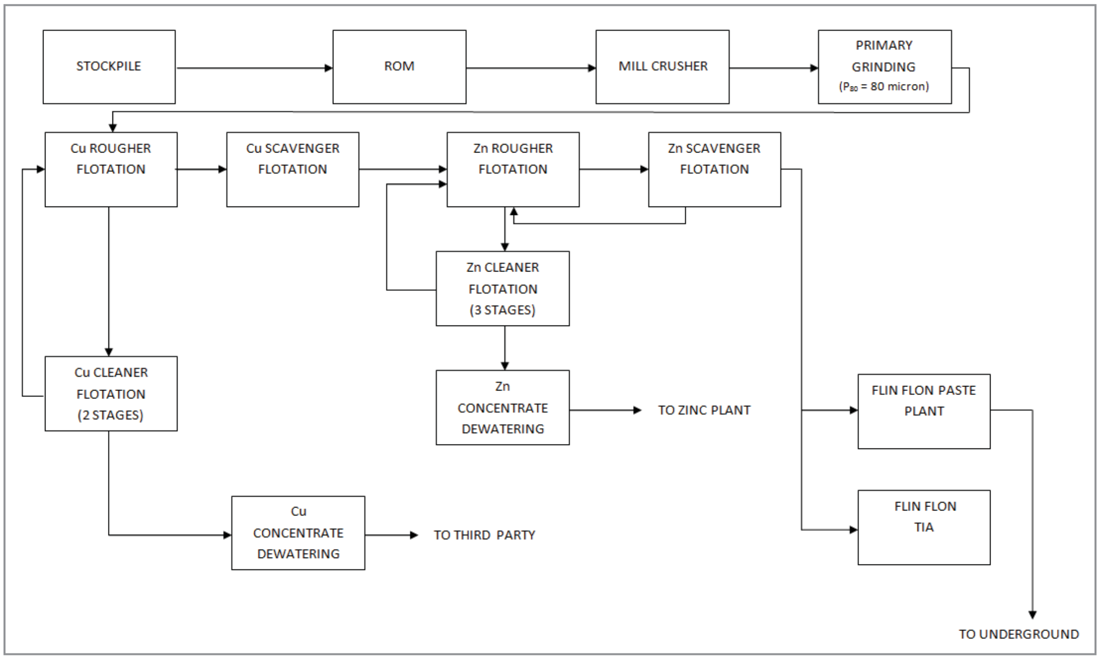
17.3 FLIN FLON CONCENTRATOR

General Layout and Process Description

The Flin Flon concentrator is located approximately 198 km from Snow Lake, Manitoba. Conventional crushing, grinding and flotation operations are used to process the excess of the Lalor run of mine that could not be processed at the Stall concentrator due to excess inventory. The nominal throughput rate is 7,200 tpd and the mill operates 24 hours per day, 365 days per year, with scheduled downtime for maintenance as required.

The concentrator produces a copper concentrate with gold and silver credits and a zinc concentrate. The copper concentrate is loaded onto rail cars and shipped to third party smelters. Tailings from the flotation circuit are utilized to produce a cemented paste backfill for use underground at the Flin Flon 777 mine. Tailings not required for paste backfill are pumped to an existing tailings pond. A simplified block flow diagram is shown in Figure 17-3.

FIGURE 17-3: FLIN FLON CONCENTRATOR PROCESS FLOW DIAGRAM



Run of mine is withdrawn from several coarse bins by apron feeders and transported to the crushing plant to reduce the size to 31mm. The material is then grounded to a particle size P80 of 80 micron via a rod and ball mill in closed circuit, coupled with a cyclone. The slurry is sent to the copper flotation circuit, composed of a rougher stage and two cleaner stages. The product of the second cleaner stage is sent to dewatering for thickening and filtration. The tailings of the copper rougher are scavenged in the copper scavenger stage and the concentrate returned to the rougher, while the tailings are sent to the zinc flotation circuit.

In the zinc flotation stage, the feed is sent to the rougher and the concentrate sent to three stages of cleaning. The product of the third cleaner stage is sent to dewatering for thickening and filtration. The tailings of the zinc rougher are scavenged in the zinc scavenger stage and the concentrate returned to the rougher, while the tailings are sent to either the Flin Flon paste plant or the Flin Flon tailings pond.

17.4 ZINC PLANT

General Layout and Process Description

The Zinc Plant produces pure zinc metal at its refinery in Flin Flon, Manitoba. This refinery was refitted with the world's first two-stage pressure leach operation for recovering zinc from zinc sulphide concentrates in 1993. No roasters are used in this zinc extraction process, therefore no sulfur dioxide (SO2) gas is produced. A simplified block flow diagram for the Zinc Plant is shown in Figure 17-4.

The zinc circuit consists of feed preparation, pressure leaching, four-step purification, flash cooling, electrowinning, and casting. Purification begins with neutralization of excess sulfuric acid (gypsum removal), followed by the removal of iron and two stages of purification in contact with zinc dust (copper-cadmium removal, then hot purification).

Leaching is a continuous two stage process using autoclave vessels operated at approximately 150 OC and 1100 kPa. The concentrate is mixed with oxygen, preheated acid, acidic solution from the second stage leach and excess solution from the ferric reduction leach circuit. The zinc concentrate is purposely not fully reacted in the first autoclave to allow the exothermic heat of reaction to provide most of the heat required by both autoclaves.

Extraction of zinc after the two stages of leaching is usually above 99%. Leach extraction of copper typically ranges from 80 to 90%. Each autoclave is 3.9m in diameter by 21.4m long, has five 150 HP agitators and four compartments. A spare autoclave is available to allow scheduled preventive maintenance.

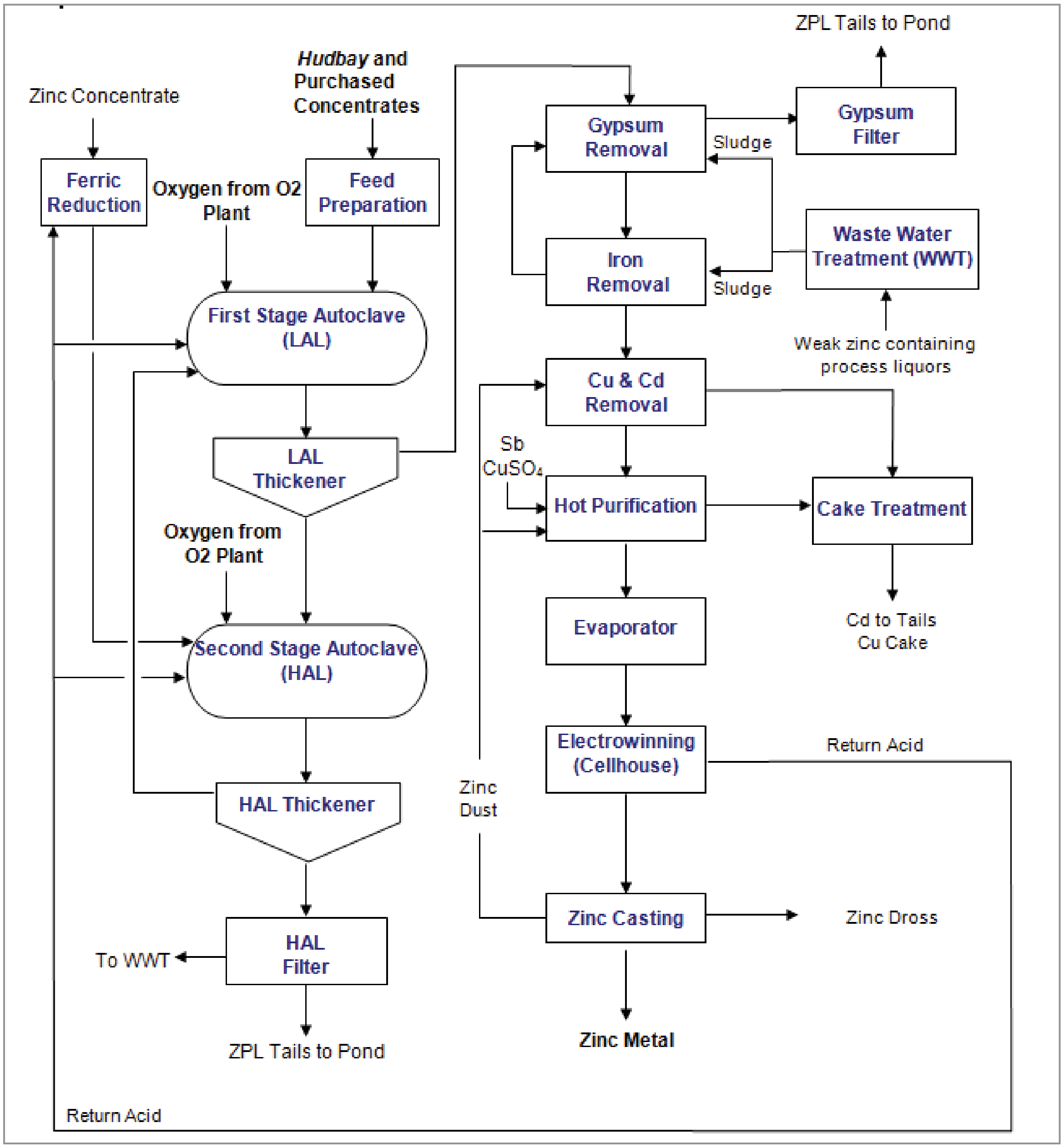
The excess acid from the overflow is neutralized with a zinc hydroxide sludge and the gypsum/iron residues are thickened and filtered. This residue is deposited along with the leach filter residue in the zinc plant tailing containment pond. The second step in the purification process precipitates the remaining iron and other impurities in the leach solution by neutralizing further with zinc hydroxide sludge while sparging oxygen into atmospheric stir tanks. The iron residues are thickened and are returned to the beginning of the gypsum removal step for recovery of valuable metals and disposal of the iron residue.

Copper and cadmium in solution are removed by contact with leaded zinc dust in a series of stir tanks and the solids are pumped to the cake treatment area for recovery of copper, cadmium and unreacted zinc. The zinc-cobalt solids are filtered from solution using filter presses, re-pulped and treated in the cake treatment area for recovery of zinc and copper. A four stage flash evaporator is used to cool the solution before it is sent for electrowinning in the cellhouse.

In the cellhouse, the zinc metal is electrochemically deposited from the electrolyte onto aluminum cathode plates, and after 48 hours, mechanically stripped to produce sheets of zinc.

The zinc strip is melted and cast for sale in one of three ingot shapes. These are: (i) continuous galvanizing grade, CGG, "long blocks", (ii) ASTM slabs and (iii) ASTM blocks "jumbo blocks". Zinc can be alloyed with aluminum or aluminum-cadmium. The final cast zinc is shipped by rail or by truck to customers.

FIGURE 17-4: ZINC PLANT SIMPLIFIED BLOCK FLOW DIAGRAM



18. PROJECT INFRASTRUCTURE

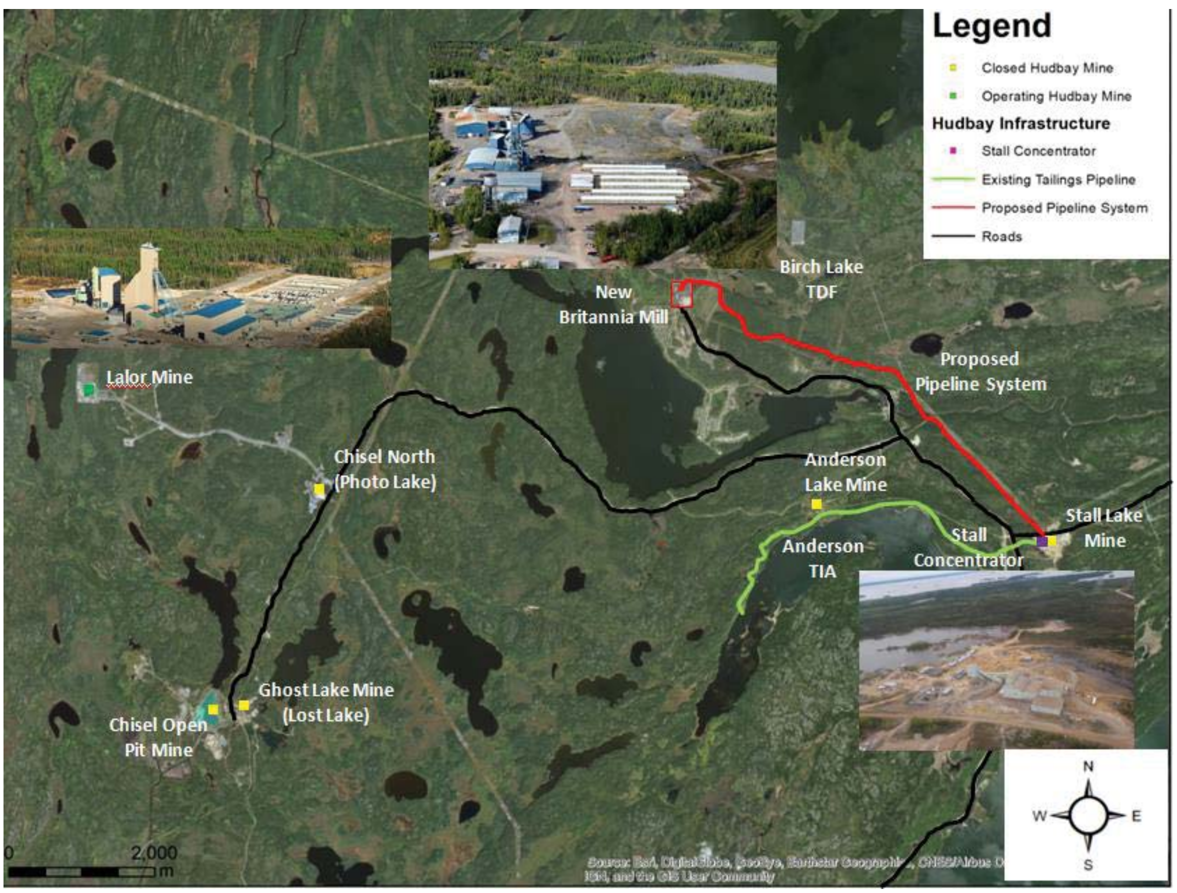
18.1 AREA SURFACE INFRASTRUCTURE

Lalor Mine is located 16 km by road from the Town of Snow Lake, Manitoba. General area infrastructure includes provincial roads, 115 kV Manitoba Hydro grid power within four km of Lalor and Manitoba Telecom (Bell) land line and cellular phone service.

The Town of Snow Lake is a full-service community with available housing, hospital, police, fire department, potable water system, restaurants and stores. The community is serviced by a 914m gravel airstrip to provide emergency medical evacuation. Hudbay owns a contractor operated 260 person camp in Snow Lake and the New Britannia mine, mill and office.

Figure 18-1 provides an overview of the various infrastructures and of the material flow from the mine to the tailings disposal facility in Snow Lake.

FIGURE 18-1: MAP OF SNOW LAKE MINING, PROCESSING AND TAILINGS FACILITIES



Chisel North Mine Site

The Chisel North mine site is located 3.5 km from Lalor Mine. Infrastructure includes:

- Building complex with offices, changehouse and shop,
- Lalor Mine electrical substation with two (2) 115-25 kV 24 MVA transformers,
- Water pumping station (booster pump station) with holding tanks and pumps, and
- Contractor owned and operated ore crusher. Stockpile capacity at Chisel North is 15,000 tonnes.

South of the Chisel North mine site, along PR 395, there are:

- Two mine fresh air raises equipped with downcast fans, mine heaters and each with a 30,000 US gallon propane tank.

Chisel Lake Mine Site

The Chisel Lake mine site is a decommissioned mine located approximately 7.5 km from Lalor. Infrastructure includes:

- Mined out open pit used for potentially acid generating (PAG) waste rock disposal and storing mine discharge water and water treatment plant sludge,
- High density sludge process acidic water treatment plant that can treat up to 2,500 gpm, and
- Fresh (process) water pumps for Lalor Mine.

Anderson Tailings Impoundment Area Site

The permitted Hudbay Anderson TIA is located approximately 12 km from Lalor. Infrastructure includes:

- Slurried tails pump station,
- 13 km of 14" buried pipeline from Anderson TIA to Lalor to deliver slurried tailings to the Lalor paste backfill plant, and
- 13 km of 10" buried pipeline to return water from the paste plant to Anderson TIA.

Mine Access Corridor

The mine access corridor is the roadway between the PR395 at the Chisel North mine site to Lalor mine site. Infrastructure in the corridor includes:

- Four km of 25 kV overhead power lines,
- Buried and heat traced process water and mine discharge pipelines, and
- Buried slurried tailings and return water pipelines.

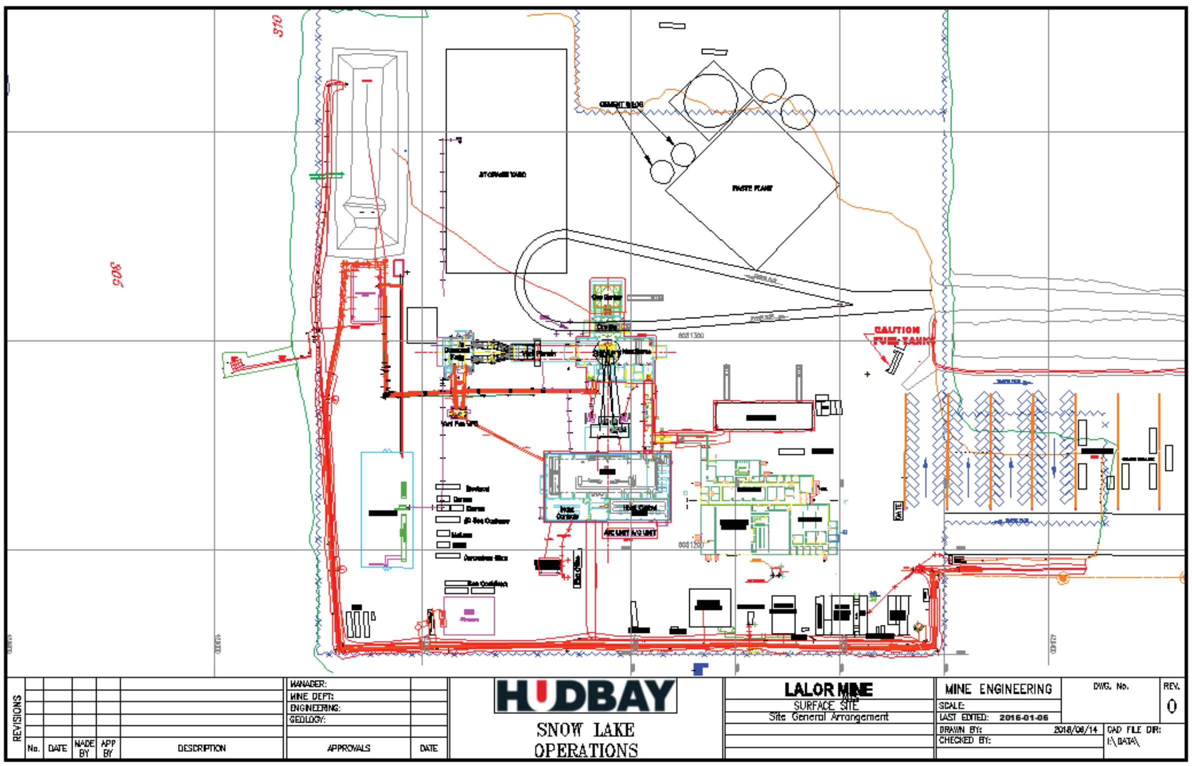
18.2 MINE SITE SURFACE INFRASTRUCTURE

A site drawing of the Lalor access road and services is shown in Figure 18-2. The Lalor mine surface infrastructure includes the following:

- Mine site services: This includes pole and buried electrical power lines, transformers and switchgear, and buried freshwater, mine discharge water and tailings pipelines.
- Hoist house: Contains electrical distribution for the site, hoist and communication control room, production hoist (Davy Markham, double drum, 4,828 kW), service hoist (Davy Markham, double drum, 2,414 kW), three GA 250 screw compressors (1,477 cfm @120 psi each).
- Headframe: Contains utility hoist (Davy Markham, single drum, 314 kW), bin house (approximate capacity 1,000 tonnes of ore), external bunker (capacity 1,200 tonnes of ore), two 250 hp downcast fans (313,000 cfm each) and mine air heater.

- Two exhaust fans (2,500 hp and 575,000 cfm each) located approximately 1.0 km from the production shaft.
- Main pump station: Includes holding tanks for discharge water, process water and potable water, PAL water system and pumps for discharge, potable, process and fire water.
- Paste backfill plant with tailings receiving and storage tanks, tailings thickener, tailings filters and cement storage silos.
- 341 person change house complex that also houses staff offices and 100 person and 50 person change houses adjacent to the main complex.
- Two 30,000 US gallon propane tanks.
- Office complex for health & safety, training and mine rescue.
- Fuel tanks and pumps for diesel and gas.
- Warehouse and shop.
- Multiple storage containers (sea containers) and contractors' office trailers.

FIGURE 18-2: LALOR MINE SITE GENERAL ARRANGEMENT



18.3 MINE UNDERGROUND INFRASTRUCTURE

Mining infrastructure includes excavations and equipment normally used below the shaft collar. It includes:

- Main Production shaft; 6.9m diameter concrete lined with five compartments equipped with:
  - Two - 16 tonne skips
  - One double deck cage (50 people per deck)
  - Counter weight
  - Utility conveyance (6 person cage)
  - Three main shaft stations at 835m, 910m and 955m levels

- Lateral development, consisting of 6m x 5m ramps and level development connecting all mine workings.
- Ramp from surface to 810m level. Secondary egress from the mine. Total distance is approximately 6.0 km.
- Power Distribution consisting of:
  - 2 x 25 kV power cables down the shaft
  - 7.5 MVA 25 kV to 13.8 kV transformers at the 835m level and 910m level shaft stations
  - Primary distribution throughout the mine is 13.8 kV with transformers to 600 V for local distribution
- Compressed air, process water and discharge water pipelines throughout the mine.
- Leaky Feeder underground wireless radio communication system.
- Fiber-optic backbone for data and video.
- Ore handling system consisting of two rock breakers and bins on 910m level feeding chutes and conveyor system on 955m level.
- Mobile Maintenance shop located at the Chisel North underground workings. Lalor maintenance shop located near the 910m level shaft station is under construction.
- Water Discharge system consisting of a series of drain holes and sumps with submersible pumps. Two settling cones on 910 m level. Clean water sumps on 955m level with 1,250 hp 10 stage Mather Platt pump (740 gpm).
- Internal ore passes to transfer ore from upper levels of the mine to 910m level.

Electrical Distribution and Equipment

Grid electricity is supplied by Manitoba Hydro, the provincial power utility. Manitoba Hydro's 115 kV power line terminates at the Chisel North Mine site, approximately 3.5 km by road from the Lalor mine site. This feeds power to the Hudbay owned main distribution substation consisting of two 115-25 kV 24 MVA transformers. The substation is completely equipped with an E-house complete with four GE Powervac circuit breakers and Tie breaker. From there, power is routed as follows:

1. Breaker 52-C1 provides power to:
   a) The Chisel North mine site
   b) The Chisel Lake mine site
   c) The Chisel North mine downcast fan substation
   d) Lalor mine ramp downcast fan site
2. Breaker 52-L1 – Lalor CCT1 provides power to the following areas:
   a) Lalor mine site pump house and water treatment plant
   b) Lalor mine exhaust fans
   c) Lalor paste plant
   d) Hoist house electrical room Bus 1 which distributes power to:
      - Mine underground Feeder A
      - VFD drives for production hoist
      - MCC for production hoist
      - Hoist house PDC (compressors)
      - MCC for Service hoist
      - Auxiliary hoist
3. Breaker 52-L2 – Lalor CCT2 provides power to the following areas:
      - Mine UG Feeder B which feeds
      - Headframe Complex
      - VFD Drives for Service hoist.
      - Intake fans
      - Lalor mine Office/Dry

Mine Underground Power Supply

Lalor mine underground mine electrical distribution to the mine workings consist 13.8 kV that is further stepped down to 600V. This power is used to power:

a) Auxiliary ventilation fans,
b) Auxiliary pumps,
c) Mobile electric jumbo drills and bolters, and
d) Underground fiber communication network.

Lalor mine underground mine power distribution equipment currently consists of:

a) 27 X 1 MVA 13.8-0.6 kV mine power centres (aka. portable sleds),
b) 16 X 0.75 MVA 13.8-0.6 kV mine power centres, and
c) 6 x 200A S&C load break switches.

The mine power centers are located in constructed underground substations.

Emergency Generators

Emergency generators are required to maintain water flows and to evacuate the mine in the case of power failure. Emergency generators are located at:

- Lalor mine pump house and treatment plant - Cummins 350 kW
- Booster pump station (Cummins 350 kW)
- Lalor auxiliary hoist (Cummins 1000 KVA)
- Paste backfill plant (Cummins 1000KVA)
- Anderson pump station (Cummins 1000 KVA)

All generators are equipped with automatic transfer switch that will in the event of a power failure transfer to generator power and vice versa.

18.4 PASTE PLANT AND ANDERSON PUMP STATION

Construction and commissioning of the paste plant and distribution system was completed in the third quarter of 2018. Paste is critical for the sustainability of the mine production plan. The paste plant is designed to fill voids left by mining of approximately 4,500 tpd. Taking into account waste generated from development in the LOM and the plan to not hoist waste from underground, the combined paste/waste backfilling capacity is approximately 6,000 tpd.

The paste plant is located northeast of the existing headframe complex at the Lalor mine site. Paste delivery capacity is 165 tph solids (tails) or 93 m^3/hr paste. The paste plant is capable of varying the binder content in the paste to provide flexibility in the cure rate of the paste where higher and early strength may be required depending on mining method.

Tails are pumped from the Stall concentrator to the Anderson TIA. Tails required for paste are diverted to the Anderson booster pump station. The capacity of the pumping station ranges from 110 to 130 tph to allow for variation in the output of tailings from the concentrator. Tailings are directed into the Anderson TIA when not required for the paste plant.

Two pipelines have been installed between the Anderson booster pump station and the paste plant located at Lalor mine site, approximately a 13 km distance. The tails slurry pipeline is a nominal 14 inch diameter and the return water pipeline is a nominal 10 inch diameter. The main route of this pipeline is on top of the existing abandoned rail bed (property owned by Hudbay), then along the west side of Provincial Road #395 and finally along the south side of the Lalor mine access road to the paste plant.

Paste is delivered from a discharge hopper to underground via one of two nominal 8 inch diameter, hardened steel cased boreholes from surface to the 780m level of the Lalor mine. Only one borehole is required during normal operation. The second borehole is available as a spare in the event of a plug or excessive wear on the primary hole. The boreholes were drilled and cased in 2016.

A network of underground lateral piping and level to level boreholes transfers the paste from the base of boreholes to the required underground locations. The 780m level is the main distribution level to direct the paste to other levels above and below. Underground development connects the paste fill distribution system to mine workings via existing and planned development into stopes.

Operational improvement projects have been implemented to enhance the reliability of the tailings delivery system on surface. This includes the establishment of additional water recirculation, improved environmental containment, flexibility for water input due to tailings level expansion at Anderson tails area, and winterizing improvements at the Lalor Paste Plant. While some of these improvements are already in place, remaining construction on these improvements is slated for completion in the second quarter of 2019.

The paste plant utilizes the Lalor mine compressors. Paste plant and Anderson booster station infrastructure includes:

- Four slurry booster pumps
- Thickener – 14m x 5.5m
- Two filter feed tanks – 14m x 14.5m
- Disc filter – 9.4m x 4.8m x 4.5m
- Vacuum pump
- Paste hopper
- Continuous mixer
- Bag house dust collector and dust collector blower
- Two cement silos – 4.1m x 15.2m
- Two screw conveyors
- Tow rotary valves and loss in Wt feeder
- One flocculent system
- Two MCC rooms – ABS and Paste Plant
- One crane
- Electric heating installations

18.5 STALL CONCENTRATOR AND ANDERSON TIA

Hudbay operates the Stall concentrator approximately 16 km from Lalor. The mill infrastructure consists mainly of two buildings. The crusher building houses the jaw and cone crushers, the screen deck and conveyors. The mill building hosts the grinding mills, flotation circuits, dewatering processes and concentrate shed areas. Three trailers are on site to provide office space. As of January 2019, the current Stall concentrator infrastructure includes:

- Two 350 tonnes coarse ore bins and 3 750 tonnes fine ore bins
- One Hewitt Robins 762mm x 1219.2mm jaw crusher
- One Symons Standard 7 ft cone crusher
- One Tyroc double deck vibrating screen – 2.4m by 6.1m screen frame, 50.8mm on top deck, 19.05mm bottom deck
- Two Rod mills – AC – (2.1m x 3.0m) and (3.2mx4.9m)
- Two Ball mills – AC – (3.2m x 4.0m) and (3.8m x 5.5m)
- One Regrind mill – AC – (2.4m x 3.7m)
- 17 - 8.5 m^3 Wemco cells
- 22 Denver cells
- Two thickeners – 3.4m and a thickener/clarifier – 2.1m

- Two Zn Filters – 2.7m and 1.8m x 6 discs agitdisc - and one Cu Filter – 1.8m x 7 discs
- Eight cranes
- One lime silo
- Two MCC Rooms
- Three compressors and 3 blowers
- Utility pipeline – crusher and mill
- Two transformers on site to 4.16 kV and 6 Power Distribution Centers (PDCs)
- Two fuel tanks and pumps for diesel and gas respectively, 2 main propane tanks and auxiliary propane tanks for office trailers
- One backup generator (1 MW).
- Tree x water pump stations including holding tanks, MCCs and generators
- Quonset and shed for part storage and muster point and 3 trailers for office space
- Dry space for approximately 90 personnel in total

The Anderson Tailings impoundment area (TIA) is located between the Stall concentrator and Lalor mine. The Anderson TIA has been in use since 1979, when a control dam was built at the east end of Anderson Lake across Anderson Creek. Seasonal discharge of water out of the Anderson TIA occurs during the open-water season, usually from May to October. Water quality at the final Anderson TIA discharge point has at all times been in compliance with applicable regulatory requirements. Tailings are deposited subaqueously into the TIA and no treatment, other than retention in the TIA, has ever been required.

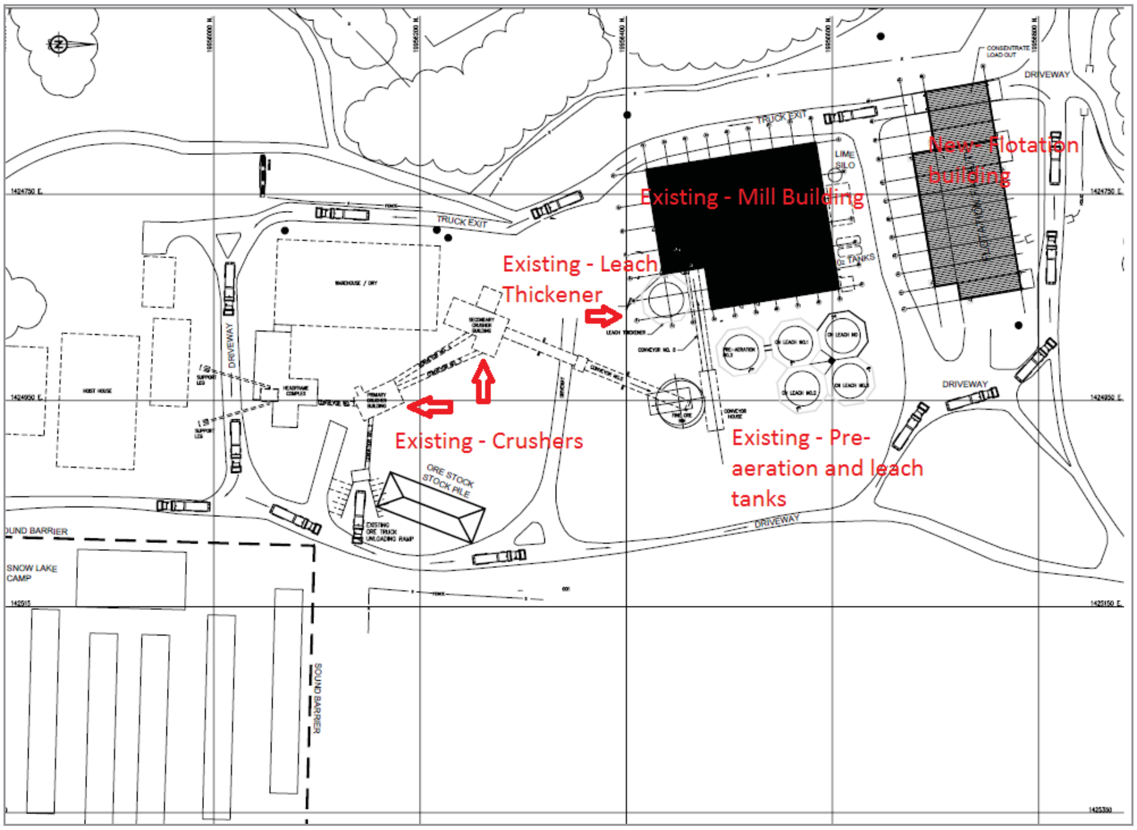
Hudbay has submitted a Notice of Alteration to Manitoba Sustainable Development to expand the TIA within the existing limits to accommodate the future tailings produced through the entire Lalor mine operations. The commissioning of this expansion is anticipated to be completed in the fall of 2019.

18.6 NEW BRITANNIA MILL

The New Britannia mill is located approximately 16 km north of the Lalor mine. As of January 2019, the refurbishment of the New Britannia mill is under a Pre-feasibility study (PFS). The following list is an overview of major equipment and infrastructure existing and future that will be available for the operation of the New Britannia mill. As part of the refurbishment, some structures and equipment will be new and others will be refurbished.

The mill infrastructure consists mainly of three buildings – crusher, mill/gold circuits and flotation/dewatering buildings. There are two existing crusher buildings; primary and secondary, approximately of 108m^2 and 136m^2 respectively. The primary building houses the jaw crusher and the secondary building houses the cone crusher and screen deck. Both buildings will also host conveyors and scrubber dust systems. The existing mill building is close to 2,000m^2 and contains the grinding and gold circuits – elution, EW and refining. A new flotation building will be added with an area of approximately 1,120m^2 to host the copper flotation circuit, copper concentrate thickener, filter press and reagent mixing systems.

FIGURE 18-3: GENERAL MAP OF NEW BRITANNIA MILL



The thickener for the leach feed along with the pre-aeration tank and leach tanks are located outside. The following list provides an inventory of the infrastructures (current and future) at the New Britannia mill:

- Reclaim hopper
- C100 Metso Jaw crusher and HP100 Metso Cone crusher
- Allis Chalmers double deck vibrating screen – 1.8m x 4.9m
- 1,800 tonnes fine ore bin
- Rod mill – AC – 3.5m x 4.6m
  Ball mill – AC – 4.3m x 6.1m
- Seven – copper rougher cells 10 m^3 OK-TK10
- Eight – copper cleaner cells 8 m^3 OK-8
- Thickener - Cu concentrate – 8 m diameter
- Thickener - leach feed – 12 m diameter
- Filter – Cu concentrate Larox
- Pre-aeration tank (694m^3) and 3 leach tanks (4163m^3)
- 6 CIP tanks
- Carbon strip vessel and carbon filter press
- Electrowinning cell
- Refinery filter press
- Refinery drying oven and induction furnace
- Lime silo
- Four cranes for crushers, mill and flotation buildings
- Three compressors and two blowers

- Utility pipeline – crusher and mill, mill and flotation building
- Pipeline corridor to/from Stall - tails, copper concentrate, reclaim water
- Four transformers
- Three MCCs – crushing, milling/Au and flotation areas
- Backup generator (750 kW)
- Fresh water pump station from Snow Lake
- Dry space for approximately 72 personnel in total

18.7 SNOW LAKE CAMP

The camp is located in the town of Snow Lake and services Hudbay employees and contractors for the mine and mill operations. The camp began operations in January 2011 and has an area of 13,124m^2. As of February 2019, the camp has seven dorms and three bunkers for a total of 260 rooms. There are also three dedicated trailers for non-dormitory purposes: a 640m2 trailer provides a dining room for employees and kitchen for camp employees and the two remaining trailers are a recreation room and a gym.

Utilities are provided by the Town of Snow Lake, including potable water, sewage and electricity. Heating is electrical and propane. The camp also provides parking spots for employees.

19. MARKET STUDIES & CONTRACTS

19.1 METAL MARKETING

Lalor produces the majority of its zinc concentrate and copper concentrate with gold and silver credits from the Stall Mill. During the 2019-2021 period, a portion of Lalor's production will be delivered to and processed in the Flin Flon Mill.

The refurbished New Britannia mill will process run of mine from the Lalor gold zone to produce a copper concentrate and gold doré bar.

Hudbay does not use intermediate agencies to sell its products from Lalor. The production from Lalor is free of encumbrances and from any streaming agreement and there are no existing contracts that would materially impact the planned revenues in the demonstration of economic viability of the mineral reserve estimates.

Zinc Concentrates

Lalor zinc concentrate is delivered by truck to Hudbay's operations in Flin Flon and processed at the Flin Flon zinc plant into refined zinc metal and sold to customers in North America. The economic analysis supporting the declaration of the mineral reserve is based on the same assumption until 2021. Beyond 2021, it is anticipated that Lalor's zinc concentrate will be sold to third party North American refineries. To date, no sales contracts have been established for the sale of Lalor zinc concentrate but it is anticipated that all sales contracts will be at standard industry terms for the quality as noted below. Table 19-1 presents the zinc concentrate composition.

TABLE 19-1: ZINC CONCENTRATE COMPOSITION

Metal	Unit	Average	Range
Zn	%	50.0	49.0 – 54.0
Fe	%	11.5	10.0 – 12.0
Au	g/t	1.6	1.2 – 2.5
S	%	33.0	30 - 35
Cd	%	0.1	0.10 – 0.14
Pb	%	0.3	0.2 – 0.7

Copper Concentrates

Lalor copper concentrate produced from its Stall Mill and the Flin Flon Mill is sold directly to a copper smelter in North America under a long term frame agreement. This agreement references the annual benchmark agreements between major concentrate producers and non-integrated smelters for the purpose of fixing the annual treatment and refining charge. The Lalor copper concentrate is delivered by rail.

Lalor copper concentrate expected to be produced from the refurbished New Britannia Mill have not been committed. It is anticipated that all sales contracts for the sale of copper concentrate produced from the refurbished New Britannia mill will be at standard industry terms for the quality as noted below.

Table 19-2 and Table 19-3 provide the expected composition of the copper concentrate to be produced from both mills

TABLE 19-2: COPPER CONCENTRATE COMPOSITION - STALL MILL

Metal	Unit	Average	Range
Cu	%	21.0	17.0 – 23.0
Ag	g/t	350.0	300 – 400
Au	g/t	35.0	35.0 – 40.0
Zn	%	5.5	2.0 – 8.0
Pb	%	5.5	0.5 - 10.0
As	%	0.03	0.01 - 0.05

TABLE 19-3: COPPER CONCENTRATE COMPOSITION - NEW BRITANNIA MILL

Metal	Unit	Average	Range
Cu	%	27.5	25.0 – 30.0
Ag	g/t	550.0	150 – 1,200
Au	g/t	175.0	50 – 400

Gold Doré

The refurbished New Britannia mill will produce a gold doré commencing in 2022. It is forecast to contain approximately 69% silver, 26% gold and 5% other elements. The doré bar will be delivered and sold to refineries using conventional third party transportation at standard industry terms.

19.2 CONTRACTS

Engineering, supply and construction contracts are initiated, managed and administrated by Hudbay’s Manitoba Business Unit. Hudbay follows a standard contracting-out process that specifies contractors’ requirements to be eligible to be considered for work. Contractor selection criteria include ability to complete the work within the required time, safety record and programs, price, and proposed alternatives. The Lalor contracts that are in place have rates and charges that are within industry norms.

Hudbay has a marketing division that is responsible for establishing and maintaining all marketing and sales administrations of concentrates and metals. As well, Hudbay conducts ongoing research of metal prices and sales terms as part of normal business and long range planning process to achieve market terms. Contract terms used in the Lalor financial evaluation are based on this research and the author has reviewed these results and they support the assumptions made in this technical report.

20. ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES AND PLANNING

Lalor Mine Stall Mill and Paste Plant

Prior to commencing advanced exploration work, environmental impact assessments including a baseline investigation were initiated in 2007 by AECOM, a consulting group that specializes in environmental service. This investigation included all the required terrestrial and aquatic field studies. The Lalor Advanced Exploration Project plan (AEP) was submitted in the first quarter of 2010 and approved in the second quarter of 2010. The Lalor mine Environment Act Licence (EAL) application was submitted during the second quarter of 2012 and was approved in the first quarter of 2014.

Baseline work and studies were also performed by AECOM for the Anderson Tailings Impoundment Area (TIA) expansion. The notice of alteration was submitted in the third quarter of 2016 and approved in the second quarter of 2018. Baseline work from other Lalor related projects was used for the Lalor paste plant Notice of Alteration (NoA) which was submitted in the fourth quarter of 2016 and approved in the first quarter of 2017. At this time, there are no known environmental issues which could materially impact the ability to extract mineral resources or mineral reserves.

New Britannia Mill

Ongoing baseline work and environmental studies are being conducted by AECOM as part of a (NoA) submission to restart the New Britannia concentrator and associated pipeline development that will link the New Britannia and the Stall concentrators.

Pen II Zone

Baseline studies were initiated in 2018 by Hudbay via AECOM. These studies will be used as part of the submittal process once the Pen II Zone project mine design is completed.

Due to the extensive work completed by AECOM and other existing studies completed as part of environmental effects monitoring programs at the various operations in the Snow Lake area, only limited additional baseline studies may be required for potential future modification of Hudbay’s current properties. There is no present indication that future approvals will not be obtained to meet potential future construction schedules.

20.2 WASTE AND TAILINGS DISPOSAL AND WATER MANAGEMENT

Lalor Mine, Stall Mill and Paste Plant

There are no known environmental concerns which could adversely affect Hudbay’s ability to operate Lalor mine. Since the mine site is near existing facilities in the Snow Lake area, the Lalor mine was able to utilize infrastructure, services, and previously disturbed land associated with permitted, pre-existing and current mining operations in the Snow Lake area.

Water management initiatives for the Lalor mine and associated projects are designed to minimize the potential impact on the surrounding environment by keeping the footprint of the operations as small as possible and by using existing licensed facilities for the withdrawal of water and disposal of waste. For instance, the NoA for the expansion of the Anderson TIA was prepared by AECOM and utilized the geotechnical tailings dam designs from Hudbay’s ongoing Engineer of Record (performed by BGC Engineering Inc.). The permitted design allows for the entire volume of tailings from Lalor LOM to be stored in Anderson TIA. This design does not discount the volume of tails that are and will be used in the future for paste backfill at the Lalor mine.

The initial stage of expansion will provide sufficient storage for tailings production volumes such that future dam raises will be planned based on actual production rates. The actual tailings production will be measured against remaining volumes based on ongoing bathymetry surveys, as per the existing licence requirements.

Compared to the original design criteria, the predicted volume of tailings from Lalor operation is currently less due to paste plant operation. It is anticipated that the initial stage of expansion will be completed in 2019. Future raises will be scheduled based on storage requirements, taking into account actual production values and paste backfill usage. The planned tailings facility raises are incorporated in the Lalor long range plan. All base metal and gold ore tailings are incorporated into the long term storage capacity of Anderson TIA. New Britannia Concentrator

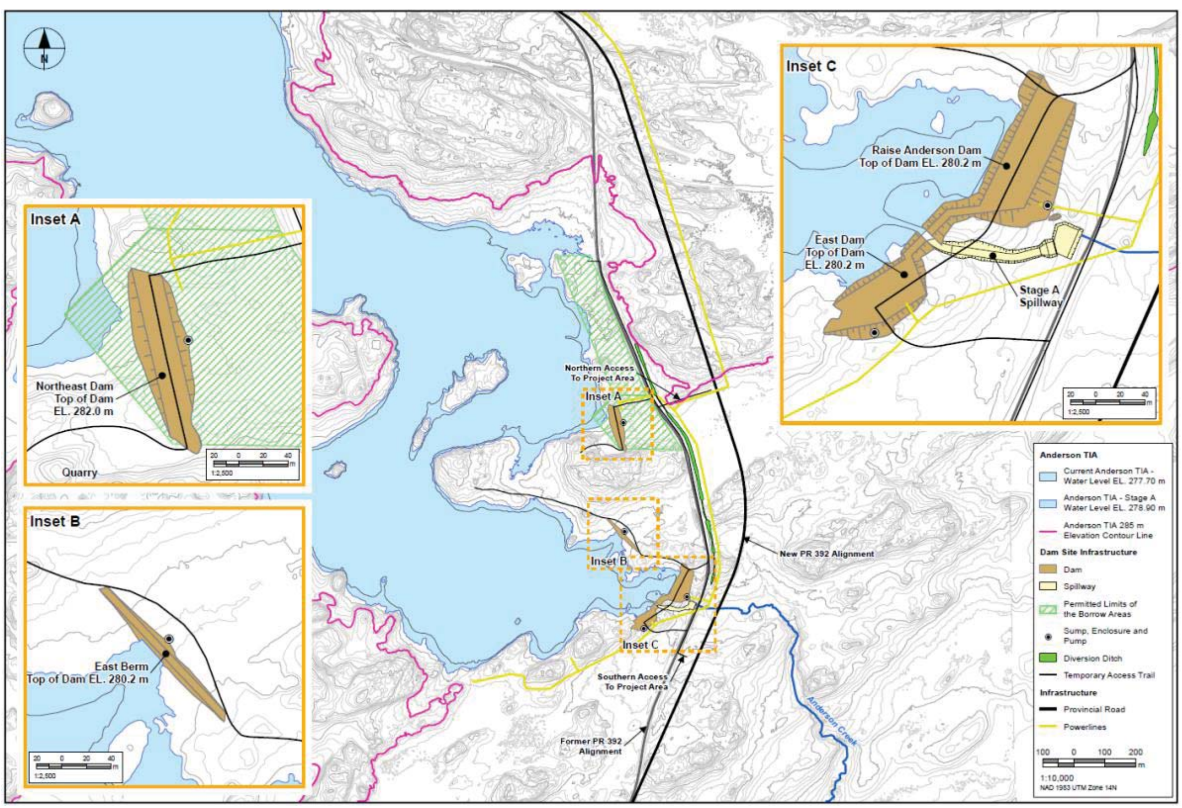
New Britannia Mill

The re-commissioning of the New Britannia mill will involve the placement of the Lalor gold ore tailings in the fully licensed Anderson TIA via a pipeline that will connect to the Stall mill. Trade-off studies and environmental assessments are ongoing to ensure that the use of Anderson TIA for all Lalor tailings is the most effective option. A NoA for the refurbishment and upgrade of the New Britannia concentrator was submitted in the first quarter of 2019

Pen II

Future production from the Pen II Zone would be processed in the Stall mill and the tailings would be stored at the Anderson TIA. Figure 20-1 displays the initial expansion of Anderson TIA.

FIGURE 20-1: ANDRESON TAILINGS IMPOUNDMENT



20.3 PERMITING REQUIREMENTS

Lalor Mine, Stall Mill and Past Plant

The existing Lalor mine EAL was obtained in the first quarter of 2014 and covers all facilities on the Lalor site. These include sewage, water treatment facilities and the pipelines which carry freshwater and wastewater. The Lalor mine was permitted such that it uses to the greatest extent possible, facilities that were already licensed. The sources of freshwater and discharge of wastewater at Hudbay's existing operating sites are licensed provincially. Discharge of effluent to the environment is regulated federally under the Metal and Diamond Mining Effluent Regulations (MDMER).

The permits required for the Lalor operation, including the Lalor mine, Stall concentrator and Anderson TIA are current and valid licenses are held. The Anderson TIA expansion NoA was submitted for approval in the third quarter of 2016 and were approved in the second quarter of 2018. No additional tailings impoundment area will be required. Federal permits are not required for this expansion.

Lalor paste plant NoA was submitted in the fourth quarter of 2016 and was approved in the first quarter of 2017.

New Britannia Mine Mill

The New Britannia site includes the Birch Lake Tailings Management Facility (BLTMF). The BLTMF is currently in care and maintenance since the closure of the mine in 2005. The seasonal discharge from BLTMF, which has an EAL is regulated under the MDMER. Hudbay is currently in the process of applying for a new water withdrawal permit for this site. In order to process material from the Lalor mine, a NoA was submitted for approval in the first quarter of 2019. At this stage, and given Hudbay's history in the area, there is no indication that the approvals will not be obtained within the project schedule.

Pen II Zone

Hudbay has initiated baseline studies through AECOM in 2018. At the moment, the Pen II Zone is at a prefeasibility study level. Once the full production and engineering details are available, applications will be sent to the appropriate regulatory body for approval.

Wim Deposit

The Wim deposit is at a scoping study level and drilling is currently underway to assess the metallurgical characteristics of the mineralized material and bring the project to a prefeasibility level. When the project demonstrates economic viability, Hudbay will initiate baseline studies and applications will be sent to the appropriate regulatory body for approval.

20.4 COMMUNITY SUPPORT

The main settlement in the region of the Lalor mine is the town of Snow Lake, which is an important mining and service centre for the surrounding area. Snow Lake had a population of 899 according to the 2016 Statistics Canada census. This number is not expected to have significantly changed. The majority of Snow Lake residents are directly employed or support the Lalor mine through contracting services.

Through the years, Hudbay and AECOM have carried out public consultation, including meetings to inform local communities on the progress and development of the Lalor mine, expansion of Anderson TIA and the environmental effects of these projects. Manitoba Sustainable Development has taken these meetings into account in the environmental licensing process. Since the economy of Snow Lake is based on mining, opposition to the projects is seen as unlikely.

20.5 FIRST NATIONS

Based on Hudbay's long-term (more than 50 years) mining experience in the Snow Lake region, there is no known current First Nation or Aboriginal hunting, fishing, trapping or other traditional use in the zone of potential influence for the Lalor mine and potential future projects. There is no First Nation Registered Trapline District or Reserve in the area that will be affected by the Lalor operation. Although development on the Mine Site involved a loss of vegetation and habitat for wildlife, the vegetation and habitat type is common throughout the region.

The Mathias Colomb Cree Nation ("MCCN"), located 125 km northwest of Snow Lake at the community of Pukatawagan, has asserted a right to be consulted in connection with the Lalor operation and expansion of Anderson TIA. Hudbay keeps MCCN informed on potential future projects and will provide Manitoba regulators with all information necessary to support Crown Consultation decisions. Regarding First Nations and Crown Consultation it is anticipated that there will be no impact to current operations or delays in project schedule.

20.6 HERITAGE RESOURCES

Operation of the Lalor mine and construction of potential future upgrades will not affect any known site of potential historical, archaeological or cultural significance. The only known site, Trampling Lake, is located approximately 20 km south of the Lalor operation. This site is of one of Manitoba's largest known concentrations of aboriginal pictographs. These paintings are thought to have been created 1,500 to 3,000 years ago by the Algonkian-speaking ancestors of the Cree and Ojibway First Nations. Activities associated with the Lalor operation will not have any impact on this historical site.

20.7 MINE CLOSURE

The Manitoba Mines and Minerals Act require a closure plan and financial assurance for any advanced exploration or mining project. Manitoba accepted Hudbay's Closure Plans prepared by SRK in 2005 and financial assurance to cover the cost of closure for all existing infrastructure that will continue to be used during operation of the Lalor mine. Existing facilities which support the Lalor mine include the Chisel North mine, which is connected by an underground ramp to Lalor, Stall Concentrator and Anderson TIA piping systems associated with milling and tailings deposition, the Chisel Open Pit and the Chisel North water treatment plant.

Prior to commercial production at the Lalor mine, Manitoba approved the Closure Plan for the Lalor AEP. The Lalor AEP Closure Plan was prepared in 2010 and approved as part of the AEP application process. As a requirement of the Lalor mine EAL, an updated closure plan was prepared by SRK and was submitted in the third quarter of 2014. The estimated cost of the closure and post-closure activities detailed in the updated Closure Plan is $1.73 million. It is anticipated that the site of the Lalor mine will be substantially returned to its natural state in about five to ten years post closure, after which no monitoring or other measures will be required.

The expansion of Anderson TIA, and upgrades to the New Britannia site also will require the submission of updated closure plans and financial assurance. Allowances for these applications have been included in the cash flow model supporting reserve declaration.

Post closure, all water quality and earthen structures will be monitored and inspected in order to ensure the sites' conditions meet the applicable regulatory requirements.

21. CAPITAL AND OPERATING COSTS

Capital and operating costs are estimated in constant 2018 Canadian dollars.

21.1 CAPITAL COSTS

The capital expenditures required to execute the life of mine plan at Lalor can be separated between the growth projects represented by the New Britannia mill refurbishment and the construction of a pipeline corridor to support the planned increase in gold production, and the sustaining capital which includes the capitalized mine development, the replacement/acquisition of mining equipment and finally other capitalised expenditures related to milling and environmental activities.

New Britannia Mill Refurbishment and Pipeline Corridor

Based on the detailed work completed in the last 12 months, Hudbay believes that the refurbishment of the New Britannia mill, including the addition of a copper flotation circuit, is the optimal processing solution for the future copper and gold rich mineral reserves to be mined at Lalor. The mill has been on care and maintenance since 2005 and was acquired by Hudbay in 2015.

To produce the planned copper concentrate and gold doré bars, the New Britannia mill will require refurbishment of existing equipment and facilities as well as the purchase and installation of new equipment. In addition, a pipeline corridor will connect New Britannia and the Stall mill to transport tailings, copper concentrate, and reclaim water. The New Britannia mill refurbishment and pipeline corridor capital estimates were engineered to a pre-feasibility level by AECOM and are expected to total $107M and $18M respectively including a 20% contingency.

For the mill (Table 21-1), the plan includes refurbishment costs for the existing building, crushers, pre-aeration and leach tanks, and leach thickener. In addition, a new flotation building will be constructed to house the flotation, thickening, dewatering, and concentrate shed components required to produce copper concentrate. New major equipment purchases will include jaw and cone crushers, screen deck, flotation circuit, thickener, filter press, acid wash vessel, reagent packages and lime silo, and electrical equipment such as transformers and an emergency generator. The total mill refurbishment capital cost includes $55M in direct costs, $14M in engineering and indirect costs, $15M in owner's costs, $17M in contingency and $6M in provincial taxes.

The pipeline corridor (Table 21-2) includes two individual pipes to facilitate movement of material between the New Britannia mill and the Stall mill (with adjacent Anderson tailings facility). An 8" pipe will carry tailings from New Britannia to Stall and a 6" pipe will carry copper concentrate from Stall to New Britannia. The copper concentrate produced at Stall will be piped to New Britannia for dewatering purposes. This arrangement facilitates environmental monitoring, improves paste plant efficiency, and provides for lower conc. moistures. The total pipeline corridor capital cost includes $11M in direct costs, $1M in engineering costs, $1M in owner's indirect costs, $1M in provincial sales taxes and $3M in contingency.

TABLE 21-1: NEW BRITANNIA MILL REFURBISHMENT CAPITAL COSTS

MILL REFURBISHMENT		LOM
Direct Costs		
Site General	C$M	21
Ore Handling	C$M	5
Grinding	C$M	3
Flotation and Dewatering	C$M	23
Leaching and Refining	C$M	4
Tailings (pipeline connect)	C$M	0
Subtotal	**C$M**	**55**
Engineering & Indirects	C$M	14
Owners Costs	C$M	15
Contingency (20%)	C$M	17
PST	C$M	6
Total - Mill Refurb	**C$M**	**107**

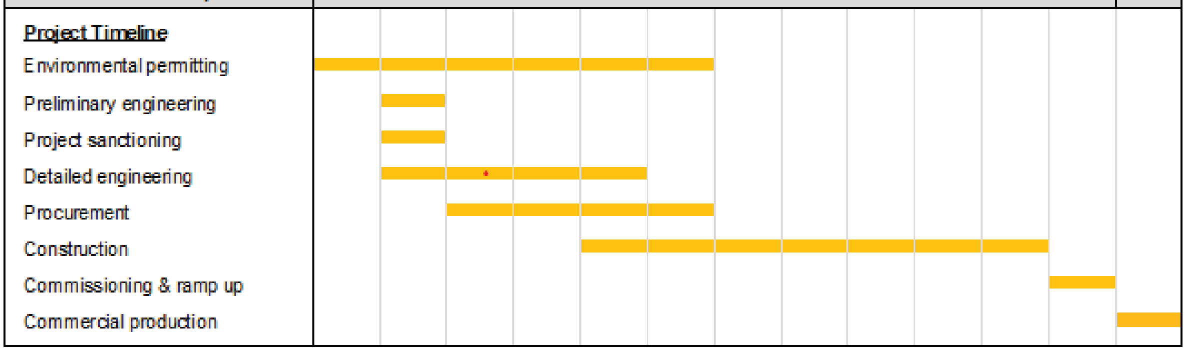
TABLE 21-2: PIPELINE CORRIDOR CAPITAL COSTS

PIPELINE CORRIDOR		LOM
Direct Costs		
Piping	C$M	3
Mechanical	C$M	0
Civil Structural	C$M	8
E&I	C$M	1
Subtotal	**C$M**	**11**
Engineering Costs	C$M	1
Owners Indirect Cost	C$M	1
Contingency (20%)	C$M	3
PST	C$M	1
Total - Pipeline Corridor	**C$M**	**18**

The development plan and its associated capital expenditure schedule are summarised on Table 21-3. Hudbay plans to complete detailed engineering and environmental permitting by Q1 2020 and Q2 2020 respectively. Construction activities will occur between Q2 2020 and Q3 2021, followed by plant commissioning and ramp-up in Q4 2021. The estimated capital expenditures and schedules for completion and plant ramp-up are deemed to be low risk since this project involves industry standard equipment and proven processing technology in a brownfield environment. Permitting activities have already started in 2018 and are proceeding in line with the proposed development plan.

TABLE 21-3: NEW BRITANNIA MILL REFURBISHMENT - PROJECT DEVELOPEMENT SCHEDULE

DEVELOPMENT CAPEX		Q1'19	Q2'19	Q3'19	Q4'19	Q1'20	Q2'20	Q3'20	Q4'20	Q1'21	Q2'21	Q3'21	Q4'21	LOM
New Britannia Mill														
Mill refurbishment	C$M	-	3	3	5	11	11	19	19	19	8	5	4	107
Pipeline corridor	C$M	-	-	-	-	3	3	3	3	3	3	-	-	18
Total - Snow Lake Ops	**C$M**	**-**	**3**	**3**	**5**	**14**	**14**	**22**	**22**	**22**	**11**	**5**	**4**	**124**



Sustaining Capital and Capitalized Development

The total sustaining capital cost (Table 21-4) of $394M is comprised primarily of capitalized development and mine equipment costs at the Lalor mine, totaling $246M and $102M respectively, in addition to milling and other sustaining capital costs of $46M. No contingency amount is included in the sustaining capital estimate.

Sustaining capital costs for the new mine plan are formulated assuming a focus on base metal ore extraction over the 2019 to 2021 time period, to preserve gold ore with higher precious metals content for extraction following commissioning of the New Britannia mill. As a result, capitalized development costs are relatively higher during the first three years as priority is given to maximizing mining base metal output. Overall, this approach results in a superior economic value for the Snow Lake Operations due to the significantly higher precious metals recoveries of the New Britannia mill.

The largest component of capitalized development is mechanical drift costs of $80M arising from approximately 14,000 metres of development required to access stopes over the life of mine. These costs are incurred primarily in the earlier years of the mine life. Similarly, various types of raises are required for venting, paste pass, ore pass, and other purposes with approximately 7,000 metres required over the life of mine at a cost of $21M.

TABLE 21-4: SUSTAINING AND CAPITALIZED DEVELOPMENT COSTS

SUSTAINING CAPEX		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Capitalized Development												
Mechanical drift	C$M	17	20	17	12	6	3	3	1	-	-	80
Bored & alimak raises	C$M	3	5	4	4	2	1	1	0	-	-	21
Electrical installations	C$M	4	5	4	3	2	1	1	0	-	-	21
Pastefill distribution	C$M	1	4	4	3	2	2	1	1	0	-	19
Main underground shop	C$M	14	5	-	-	-	-	-	-	-	-	19
General mine expense	C$M	18	16	13	11	7	5	4	2	0	-	75
Other	C$M	7	2	1	1	0	0	0	0	0	-	12
Subtotal	C$M	64	57	44	33	20	12	9	5	1	-	246
Mine Equipment												
Replacement equipment	C$M	7	7	8	8	2	5	7	-	-	-	43
Additional equipment	C$M	-	23	-	-	-	2	7	-	-	-	33
Major repairs	C$M	1	5	1	4	12	1	-	-	-	-	25
Subtotal	C$M	9	35	10	11	14	8	14	-	-	-	102
Subtotal - Lalor Mine	**C$M**	**73**	**92**	**54**	**45**	**35**	**20**	**23**	**5**	**1**	**-**	**348**
Other												
Stall mill equip. & building	C$M	4	3	1	1	1	1	0	0	0	0	11
Shared general plant	C$M	11	3	-	-	-	-	-	-	-	-	14
Environmental	C$M	8	-	-	8	-	4	-	-	-	-	21
Subtotal - Other	**C$M**	**23**	**6**	**1**	**9**	**1**	**5**	**0**	**0**	**0**	**0**	**46**
Total - Snow Lake Ops	**C$M**	**96**	**98**	**55**	**54**	**35**	**25**	**23**	**5**	**1**	**0**	**394**

Commencing in 2019, construction of the main underground shop is estimated to cost $19M with completion in early 2020. This shop will be used to service mine equipment underground and will reduce maintenance costs versus the currently used surface shop as vehicles will no longer need to be driven to the surface for repairs. General mine expense includes a variety of costs such as power, mine admin, heating, ventilation, level maintenance, engineering, geology, and Snow Lake area costs that are allocated between capitalized development and mining operating costs based on accounting rules. The amount applicable to capitalized development is $75M over the life of mine.

Replacement equipment comprises the largest component of sustaining capital requirements for mine equipment with a life of mine cost of $43M. Replacement costs are based on the scheduled useful life of individual equipment. Additional equipment costs of $33M are comprised primarily of $23M in mining equipment purchases scheduled for 2020 to replace contractor provided equipment with Hudbay owned equipment as a measure to reduce operating costs.

Stall mill sustaining capital costs of $11M over the life of mine includes scheduled replacement of key equipment and building upkeep required to maintain production at up to 3,500 tpd. Shared general plant costs of $14M in 2019 and 2020 represent the share of Flin Flon surface facilities upkeep costs allocated to Snow Lake Operations, primarily costs related to the zinc plant. Finally, environmental capital costs of $21M relate to the scheduled Anderson tailings facility raises and stabilization work expected to be completed over the life of mine to accommodate tailings from both the Stall and New Britannia mills as a result of regular operations.

21.2 OPERATING COSTS

Operating cost estimates are based on the 2019 budget figures and were developed by Hudbay using a bottom-up approach and quotes from local suppliers, Manitoba operations experience, labour costs within the region and actual 2018 costs. The operating cost estimates are presented below separately for mining, milling, refining and general; and administrative activities.

Mining Operating Costs

Table 21-5 presents the details of the Lalor mine's operating costs. Labour costs constitute the largest component of the mining operating costs with a total of $514M estimated for the life of mine. Labour costs include both employee and contractor costs arising from mining and maintenance activities. The staffing plan includes approximately 500 positions, comprised primarily of hourly paid employees.

TABLE 21-5: LALOR MINE'S OPERATING COCTS

OPEX - MINING		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Mining - Lalor	C$M	144	147	156	145	146	141	126	92	92	70	1,259
Per tonne milled	C$/t	90.9	99.4	99.8	90.0	89.3	85.6	87.6	98.2	96.2	84.6	92.0

General mine expense includes a variety of costs such as power, mine admin, heating, ventilation, level maintenance, engineering, geology, and Snow Lake area costs that are allocated between capitalized development and mining operating costs based on accounting rules. The amount applicable to mining operating costs is $372M over the life of mine.

In terms of mining activities, ore extraction represents the largest cost driver accounting for approximately 52% of mining operating costs over the life of mine, while ore removal and mine development activities account for 29% and 19% of total costs respectively.

Milling Operating Costs

Table 21-6 presents the details of the Stall, Flin Flon and New Britannia operating costs. As with mining, labour costs are the largest component of milling operating costs with a total of $177M in total between the three mills used to process Lalor ore. The staffing plan includes a total of approximately 80 positions at Stall mill and 50 at New Britannia mill, comprised primarily of hourly paid employees. Concentrates produced at Stall and New Britannia mills are transported to Flin Flon by truck where they are loaded onto railcars and shipped to off takers. The cost of the concentrate haulage to Flin Flon totals $32M or approximately $24/tonne during the 2019-2021 period.

TABLE 21-6: MILLING OPERATING COSTS

OPEX - MILLING		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Stall mill	C$M	35	36	33	28	29	27	21	17	18	16	260
Per tonne milled	C$/t	27.1	27.5	25.5	24.5	23.0	23.8	23.8	27.8	27.0	30.9	25.7
New Britannia mill	C$M	-	-	-	18	17	20	20	15	14	15	118
Per tonne milled	C$/t	-	-	-	38.1	44.2	39.9	35.9	45.0	49.0	46.4	41.6
Flin Flon mill	C$M	9	6	9	-	-	-	-	-	-	-	24
Per tonne milled	C$/t	30.2	33.3	34.8	-	-	-	-	-	-	-	32.6
Total - Snow Lake Ops	**C$M**	**44**	**42**	**42**	**46**	**46**	**47**	**41**	**32**	**32**	**31**	**402**
Per Tonne Milled	**C$/t**	**27.7**	**28.3**	**27.0**	**28.5**	**27.9**	**28.7**	**28.4**	**33.8**	**33.6**	**36.8**	**29.4**

[1] Allocation of Flin Flon mill costs applicable to the processing of Lalor ore prior to the closure of Flin Flon ops in 2021

Refining Costs

Until the closure of the zinc plant in Q2 2021, the Lalor zinc concentrate will be refined into finished zinc in order to maximize the realizable value from zinc produced. The operating costs presented in Table 21-7 summarize the costs allocated to the Snow Lake Operations for the refining of zinc concentrates produced from Lalor mine's production (either through the Stall or Flin Flon mills).

TABLE 21-7: REFINING OPERATING COSTS

OPEX - REFINING		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Total - Snow Lake Ops	C$M	87	89	42	-	-	-	-	-	-	-	218
Per lb zinc produced	C$/lb	0.52	0.52	0.47	-	-	-	-	-	-	-	0.51
Per tonne milled	C$/t	55.0	59.8	54.2	-	-	-	-	-	-	-	56.7

[1] Following closure of the zinc plant in 2021, Lalor zinc concentrates will be sold directly to the market

General and Administration Costs

General and administration costs (Table 21-8) are comprised of an allocation of shared services, administration, and other office costs to Snow Lake Operations based on relative usage versus Flin Flon Operations. Significant costs in the shared services line item include health & safety, environmental control, logistics, and analytical services. Significant costs in the administration line item include insurance and property taxes, information technology, human resources, and finance.

TABLE 21-8: GENERAL AND ADMINISTRATION COSTS

OPEX - G&A		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Total - Snow Lake Ops	C$M	31	31	30	33	34	36	32	17	19	18	280
Per Tonne Milled	C$/t	19.4	20.9	18.9	20.7	20.8	21.7	22.3	18.1	19.7	21.5	20.5

[1] Decommissioning, pension, and severance costs are considered as corporate legacy costs and therefore not included in the economic valuation of the business plan of Lalor

21.3 UNIT CASH AND SUSTAINING CASH COSTS

With the benefit of the New Britannia mill, the net revenue at Lalor will shift from primarily zinc during the 2019-2021 period to primarily gold in the latter portion of the mine life (post 2021), positioning Lalor as a gold mine with significant zinc, copper and silver by-products. The life of mine net revenue from Lalor is approximately 50% precious metals, 33% zinc and 17% copper. Once the New Britannia mill is operational in 2022, revenue from precious metals through the remaining life-of-mine is expected to be approximately 60% of total revenue. Significant zinc and copper revenue provides a diversified commodity exposure with a significant upside potential from the mineral resource estimates in the Snow Lake that have not yet been converted to mineral reserve estimates both at Lalor and at other Snow Lake gold properties.

Table 21-9 illustrates that Lalor's significant by-product credits will reduce its cash operating costs and sustaining cash costs both on a zinc and gold basis. During the first five years of operation with New Britannia (2022 to 2026), Lalor is estimated to produce approximately 140,000 ounces of gold annually at a sustaining cash cost, net of by-product credits, of $450/oz. This positions Lalor to be one of the lowest cost gold mines in Canada.

TABLE 21-9: CASH AND SUSTAINING UNIT CASH COSTS

CASH & SUSTAINING COST[1]		2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM
Zinc Basis												
Contained zinc	Mlbs	174	194	196	139	154	141	71	41	57	51	1,218
Cash cost	US$/lb	0.61	0.73	0.55	(0.05)	0.03	(0.18)	(1.21)	(0.87)	(0.68)	(1.08)	0.09
Sustaining cash cost	US$/lb	1.04	1.14	0.78	0.26	0.22	(0.04)	(0.95)	(0.77)	(0.67)	(1.07)	0.35
Gold Basis												
Contained gold	'000s oz	69	58	75	143	137	138	172	114	119	108	1,134
Cash cost	US$/oz	(672)	(308)	35	268	211	85	333	581	448	278	198
Sustaining cash cost	US$/oz	400	1,051	616	571	416	229	442	618	456	279	473

[1]Cash cost and sustaining cash cost are presented net of by-product credits

22. ECONOMIC ANALYSIS

Hudbay is a producing issuer and has excluded the information required by Item 22 of Form 43-101F1 as the updated mine plan does not represent a material increase in current production.

23. ADJACENT PROPERTIES

RockCliff owns the claims located immediately to the North of the PEN II deposit. RockCliff has not reported any mineral resource estimates and no significant exploration results on these claims that would have an impact on the mineral resource estimates reported by Hudbay in this report for the PEN II property.

There are no other relevant adjacent properties in relation to the other properties discussed in this technical report.

24. OTHER RELEVANT DATA & INFORMATION

There is no other relevant data or information that we are aware of that has not been presented in the other sections of this report which is necessary to make this Technical Report understandable and not misleading.

25. INTERPRETATIONS & CONCLUSIONS

Lalor was discovered in 2007 on 100% owned land using Hudbay's innovative exploration techniques and it is the largest VMS deposit found in the Snow Lake region to-date. In 2009, the company commenced construction of Lalor with initial ramp access from the Chisel North mine. Construction of the main production shaft was approved in 2010 and was completed on time and on budget, achieving commercial production in 2014, only seven years from initial discovery.

Hudbay acquired the New Britannia gold mill in 2015 for approximately $10 million as a potential long-term processing option for the Lalor gold and copper-gold zones. New Britannia was placed on care and maintenance in 2005 by its previous owner after producing 1.6 million ounces of gold. Since acquiring New Britannia, Hudbay has conducted significant technical work to assess the grade, tonnage, mineability and metallurgy of the gold and copper-gold zones at Lalor to support and de-risk the investment required to refurbish New Britannia and maximize the net present value of Lalor.

During the last two years, Hudbay has also dedicated considerable efforts to drilling, test mining in the gold-rich lenses and revising its geological interpretation and updating the resource models for all the mineralised lenses at Lalor. The updated estimate of mineral reserves at Lalor has increased in-situ contained gold by 65%, copper by 23%, zinc by 11% and silver by 15%, relative to the previous estimate of mineral reserves in Hudbay's 2018 AIF, adjusted for 2018 production depletion.

Based on the detailed work completed in the last 12 months, Hudbay believes that the refurbishment of the New Britannia mill, including the addition of a copper flotation circuit, is the optimal processing solution for Lalor, as it capitalizes on existing infrastructure, significantly grows gold production from a deposit that is unencumbered by any royalties or streams and offers further upside potential from nearby satellite deposits.

The New Britannia development plan contemplates completion of detailed engineering by February 2020, environmental permitting completion in April 2020 and construction activities occurring between June 2020 and August 2021, with plant commissioning and ramp-up occurring during the fourth quarter of 2021. The estimated capital expenditures and the schedules for completion and plant ramp-up are deemed to be low risk since this project involves industry standard equipment and proven processing technology in a brownfield environment. Permitting activities started in 2018 and are proceeding in line with the development plan.

The revised mine plan for the Lalor mineral reserves optimizes net present value by preserving gold-rich ore for processing at the New Britannia mill and zinc-rich ore for the Stall mill, which is expected to result in significantly higher gold and copper recoveries. Life-of-mine production of gold, copper, zinc, and silver has increased by 91%, 16%, 13% and 21%, respectively, compared to the 2017 Technical Report for the period starting January 1, 2019. This revised mine plan for Lalor supports a 10 year mine life, based solely on proven and probable reserves, and utilizes the existing mining capacity of 4,500 tonnes per day at Lalor for the first six years of the mine plan. The technical work completed supports 4,500 tonnes per day as the optimal mining rate to maximize net present value, although the Lalor production shaft has the potential to hoist at higher throughput rates.

Compared with the 2017 Technical Report life-of-mine plan, with the inclusion of the New Britannia mill, net revenue at Lalor has shifted from primarily zinc to primarily gold in the updated production profile, positioning Lalor as a primary gold mine with significant zinc, copper and silver by-products. Lalor's significant by-product credits reduce its cash operating costs and sustaining cash costs on both a zinc and gold basis. During the first five years of operation with New Britannia (2022 to 2026), Lalor is estimated to produce approximately 140,000 ounces of gold annually at a sustaining cash cost, net of by-product credits, of $450/oz, positioning Lalor as one of the lowest cost gold mines in Canada. In addition, significant zinc and copper revenue provides diversified commodity exposure.

Refurbishing New Britannia is expected to significantly increase gold production from Lalor and enable new gold and copper-gold exploration opportunities in the Snow Lake region by having an operating processing facility with substantially higher gold and copper recoveries. It also demonstrates the opportunity to create additional value through owning multiple processing facilities in the Flin Flon and Snow Lake regions.

As documented in this report, Hudbay has also implemented a more stringent approach to resource reporting for underground deposits. With this approach, the potential for economic extraction of the mineral resource estimates at Lalor, the Pen II and Wim deposits, are reported within the constraint of a 'stope optimization envelope' process similar in concept to a Lerchs-Grossman pit shell for an open pit deposit. This excludes from the resource estimate small individual resource blocks that may meet economic cut-off criteria on an individual basis but could not be aggregated into mineable shapes. It is anticipated that this approach will result in higher resource to reserve conversion ratios than those experienced in the past.

The updated resource models at Lalor includes 5.9 million tonnes of inferred mineral resources, which has the potential to extend the mine life beyond 10 years while feeding both the Stall and New Britannia mills. In addition, the mineral resources at Hudbay's satellite deposits in the Snow Lake region, including the copper-gold Wim deposit, the former gold producing New Britannia mine and the zinc-rich Pen II deposit could provide feed for the Stall and New Britannia processing facilities and further extend the mine life.

In 2018, Hudbay has continued to invest in significant exploration efforts in Manitoba including major airborne and ground geophysical surveys and surface exploration drilling in the Flin Flon and Snow Lake areas. This work has identified several base metal and gold targets to be tested in 2019. In parallel, in-mine exploration at Lalor continues with the intent to convert inferred resources to indicated mineral resources and add additional inferred mineral resources in base metal and gold-rich mineralization. Hudbay will continue to advance engineering studies on Lalor and its satellite deposits in an attempt to continue to increase the tonnage of the mineral reserve estimates and the estimated operating life of the Snow Lake processing facilities at or near full capacity.

The Hudbay validation process has confirmed that the resource models are constructed using industry accepted modelling techniques and classified in accordance with the 2014 CIM Definition Standards – For Mineral Resources and Mineral Reserves. The author considers that the mineral resources and reserves as classified and reported comply with all disclosure in accordance with requirements and CIM Definitions. The author is not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimate.

26. RECOMMENDATIONS

Significant value has been unlocked by leveraging Hudbay's exploration expertise, its existing portfolio of processing infrastructures and several inexpensive acquisitions over the past two years. These efforts have highlighted the potential to further increase the tonnage of mineral reserve estimates and the estimated operating life of the Snow Lake processing facilities at or near full capacity through continued exploration success, a disciplined and methodic approach to engineering studies while maintaining excellence in tailings management and environmental stewardship.

On the exploration front, the author recommends to pursue in-mine exploration to confirm the potential size and quality of the copper-gold rich Lens 17 so as to be able to report an inferred resource for this zone by the end of 2019. In-mine exploration should also be pursued to incrementally add both base metal and gold rich inferred resources where lenses are still open in their along strike or down dip directions. In addition, a deep drilling program of three holes has been proposed and should be pursued to test the fold repeat of the favorable Lalor horizon below the 1,050m level. The in-mine drill program should also continue to convert enough inferred mineral resource estimates to an indicated category so that they can be converted to mineral reserve estimates and replace the planned 2019 mining depletion.

Hudbay has developed a well focused surface exploration program aimed at confirming near surface exploration targets between Lalor and the past producing Chisel mine and also around the past producing Snow Lake gold properties. The author supports this program which has the potential to define additional inferred resources in the Snow Lake camp over the next one to two years.

At Lalor, in the gold and copper-gold rich zones, some of the mineral resource estimates have a drilling density sufficient to support an indicated category but were classified as inferred due to the small proportion of mineralization grading above cut-off. It is recommended to test the applicability of conditional simulations in these areas in order to assess if the confidence in the recoverable tonnes and grade could be strengthened depending on the mining method considered (C&F vs LH). Theoretical results obtained from conditional simulations could be readily tested through a mining test in Lens 25 or in Lens 21.

At the Wim deposit, additional drilling is recommended in 2019 to collect representative core samples to conduct additional metallurgical testing and to establish if the mineralization is amenable to high gold recovery through the New Britannia mill flowsheet. This work together with a geotechnical study and a base line environmental study could support a pre-feasibility and the declaration of mineral reserve estimates in 2019.

At the Pen II deposit, a pre-feasibility study is on-going to establish the potential economic and technical viability of the indicated mineral resource estimates. In parallel, a shallow drilling program should be completed to convert a small tonnage of high grade inferred resources, to collect additional material for metallurgical testing and to fully establish how far the mineralization extend towards the topographic surface and to establish the geotechnical properties of the potential crown pillar.

At the former New Britannia mine and its satellite gold deposits (boundary, 3 zone), the resource models should be re-constructed to support a new pre-feasibility study on the remaining mineral resource estimates. With a more thorough modeling approach based on best current industry practices, it is deemed that a portion of the current inferred mineral resource estimates could be upgraded to an indicated and category and are potentially amenable to a conversion to mineral reserve estimates. This work should be initiated in 2019.

From a metallurgical perspective, it is recommended to conduct testing for gravity concentration at the New Britannia mill from core samples from Lens 27 and the collector dosage and flotation time should be modified at the New Britannia mill in order to account for the high copper grade that will be fed from mining in this lens. Also, recovery models should be developed based on rougher and cleaner flotation kinetics using the Aminpro methodology.

The updated closure plans for the Anderson tailings facility and for the New Britannia site should be completed and submitted to the appropriate regulatory body for approval.

The 2019 Hudbay exploration budget of approximately $17 M for Manitoba will provide all the funding required to implement the recommendations. The exploration budget for both surface and underground programs is under direct authority of the QP.

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