NI 43-101 Technical Report

Lalor Mine

Snow Lake, Manitoba, Canada

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HUDBAY

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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, expectations, production, cost, capital and exploration expenditure guidance, including the anticipated capital and operating cost savings and anticipated production at our mines and processing facilities, events that may affect Hudbay’s operations and development projects, the anticipated timing, cost and benefits of developing the Lalor growth projects, anticipated mine plans, anticipated metals prices and the anticipated sensitivity of our financial performance to metals prices, the potential to increase throughput at the Stall mill and to refurbish the New Britannia mill and utilize it to process ore from the Lalor mine, 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 business and acquisition strategies. 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 success of mining, processing, exploration and development activities;
- 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 ability to complete project targets on time and on budget and other events that may affect our ability to develop our projects;
- the timing and receipt of various regulatory and governmental approvals;
- the availability of personnel for our exploration, development and operational projects and ongoing employee and union relations;
- maintaining good relations with the communities in which we operate, including First Nations communities surrounding our 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;
- the timing and possible outcome of pending litigation and no significant unanticipated litigation;
- certain tax matters, including, but not limited to current tax laws and regulations; 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), uncertainties related to the development and operation of our projects, dependence on key personnel and employee and union relations, risks related to the cost, schedule and economics of the capital projects intended to increase processing capacity for Lalor ore, 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, compliance with government and environmental regulations, including permitting requirements and anti-bribery legislation, 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 and our management’s discussion and analysis of Hudbay for the year ended December 31, 2016.

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.

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1 SUMMARY

1.1 Summary

This Technical Report (the Technical Report) has been prepared for Hudbay Minerals Inc. (Hudbay) to support the public disclosure of Mineral Resources and Mineral Reserves at the Lalor Mine and to provide an updated mine plan that contemplates 4,500 tpd of base metal, gold and copper-gold zone ore to Stall concentrator.

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, which also consists of an ore concentrator and zinc plant, and the Reed mine, which is located approximately 120 kilometres (km) by road southeast of Flin Flon. Operations near Snow Lake include the Lalor underground mine.

The Lalor mine consists of 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.

As of the date of this Technical Report, the Lalor mine is operating at approximately 3,000 to 3,500 metric tonnes per day (tpd) and is ramping-up production to 4,500 tpd by 2018. The production ramp-up is supported by the underground ore handling circuit capable of 4,500 tpd, transitioning to more bulk mining methods (65% of reserves) with additional mining fronts and design changes to improve mining efficiencies, developing ore passes and transfer raises to reduce truck haulage cycle times from the upper potions of the mine and commissioning of a paste backfill plant in the first quarter of 2018.

Autonomous operation of a Load Haul Dump loader underground is currently being trialed from the surface by tele-remote monitoring with changes to standard designs to allow isolation of autonomous areas and buffer storage for in-between shift mucking.

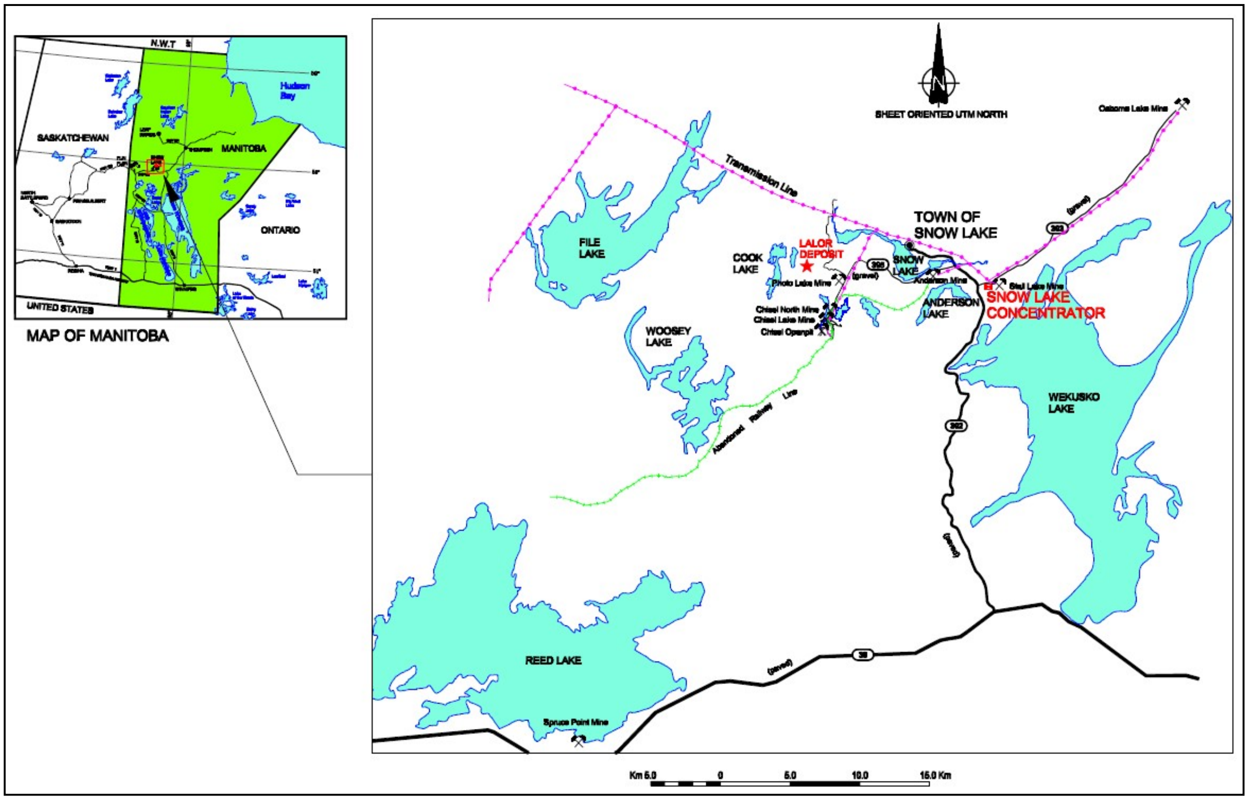
The increase in production to 4,500 tpd at Lalor is complemented by the Stall concentrator expansion to 4,500 tpd, which is currently underway and is expected to be commissioned in the third quarter of 2018.

The Qualifed Person (the "QP") who supervised the preparation of this Technical Report is Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

1.2 Property Description and Location

The Lalor mine is located approximately 208 km by road east of Flin Flon and 16 km by road west of Snow Lake in the Province of Manitoba at 54°52'N latitude, 100°08'W longitude and 303 metres above sea level (m ASL). Access to Lalor mine is from Provincial Road (PR) #395, a gravel road off PR #392, which joins the town of Snow Lake and PR #39 (Figure 1-1). From PR #395, there is an all-weather permanent road into the mine site.

FIGURE 1-1: LALOR MINE LOCATION



Hudbay owns 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 the purpose of working the lands and mining and producing minerals from the Lalor mine. Surface tenure, currently necessary to accommodate buildings and/or structures, required for the efficient and economical performance of the mining operations has been applied for and approved.

1.3 Geological Setting and Mineralization

The Lalor deposit is interpreted as a volcanic massive sulphide (VMS) deposit that precipitated at or near the seafloor in association with contemporaneous volcanism, forming a stratabound accumulation of sulphide minerals. The depositional environment for the mineralization at Lalor is similar to that of 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 hydrothermal alteration pipe.

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' to 'mature arc' to 'arc-rift'. The 'mature arc' Chisel sequence that hosts the zinc rich Chisel, Ghost, Chisel North, and Lalor deposits typically contains thin and discontinuous volcaniclastic deposits and intermediate to felsic flow-dome complexes.

The Chisel sequence is lithologically diverse and displays rapid lateral facies variations and abundant volcaniclastic rocks. Mafic and felsic flows both exhibit evolved geochemical characteristics consistent with one of, or a combination of, the following: within-plate enrichment, derivation from a more fertile mantle source, lower average extents of melting at greater depths, and contamination from older crustal fragments. These rocks have undergone metamorphism at the lower to middle almandine-amphibolite facies.

Rock units in the hanging wall of the Lalor deposit typically reflect this diversity and variation in rock types that include mafic and felsic volcanic and volocaniclastic units, mafic wacke, fragmental units of various grain sizes, and crystal tuff units.

1.3.1 Base Metal Mineralization

Lalor base metal mineralization begins at approximately 600 metres (m) from surface and extends to a depth of approximately 1,100 m. The mineralization trends about 320° to 340° azimuth and dips between 30° and 45° to the north. It has a lateral extent of about 900 m in the north-south direction and 700 m in the east-west direction.

Sulphide mineralization is 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.

Seven distinct stacked zinc rich mineralized zones have been interpreted within the Lalor deposit based on the zinc equivalency of 4.1% over a minimum three metre interval. The top two lenses of the stacked base metal zones (coded as Zone 10 and 11) have higher grade zinc and iron content. The footwall lenses coded as Zones 20, 30, 31, 32 and 40 have moderate to high zinc grades hosted in near solid sulphides containing higher grade gold and locally appreciable amounts of copper.

Overall, Zones 10 and 20 have the largest extent and volume of mineralization. Zone 10 extends approximately 400 m in the east-west and 550 m in the north-south direction and Zone 20, 250 m in the east-west and 700m in the north-south direction.

1.3.2 Gold Mineralization

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.

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.

Seven lense groups have been interpreted within the deposit area and are present between 750 m to 1,480 m below surface. Their general shape is similar to the base metals. However, the current interpretation suggests the deeper copper-gold lense tends to have a much more linear trend to the north than the rest of the zones. The gold mineralization associated with each zone was interpreted into three-dimensional wireframes based on a 2.4 grams per metric tonne (g/t) gold equivalent over a minimum 3 metre (m) interval.

1.4 Exploration

Exploration drilling since Hudbay's NI 43-101 technical report on the Lalor mine dated March 29, 2012 has focused on delineation of the inferred resource, confirming the continuity of the mineralization down plunge and testing for new mineralization peripheral to the known deposit. Surface drilling since the previous disclosure is limited to four drill holes while the focus was on underground exploration, definition, and delineation drilling, which has continued to expand the resource.

1.4.1 Underground Exploration Development

Since 2014 one exploration drift and one exploration ramp were developed at Lalor for a total of 1,891 m. The development was undertaken to establish underground platforms to conduct exploration drilling on targets that could not be drilled from existing mine infrastructure. Prudent care was taken in the placement and size of both the exploration ramp and drift to assure the selected locations can accommodate future mining equipment and related infrastructure.

1.4.2 Underground Exploration Drilling

In 2015, thirty-one drill holes were completed for a total of 10,395 m focusing on the copper-gold, Zone 27. Exploration drilling continued on gold Zone 25 from March to July of 2016 for a total of sixty nine drill holes and 16,098 m. The purpose of the exploration programs was to upgrade inferred resources, specifically focused on identifying areas of enriched gold and copper-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.

1.4.3 Borehole Electromagnetic (EM) Surveys

Time-domain borehole EM surveys with three-dimensional probes are routinely conducted on surface and underground drill holes. The survey results identify any off-hole conductors that were missed, indicate direction to the target, as well as the dimensions and the attitude of the conductor. The surveys can also detect possible conductors which may lie past the end of the hole allowing decisions to extend holes to be made.

1.4.4 Surface EM Surveys

Two time-domain surface EM surveys, for a total of approximately 35 line km, were completed north-northeast and south-southwest of the Lalor mine. Neither survey has identified any new significant targets of interest in the general Lalor mine area.

1.4.5 Airborne EM Survey

During the summer of 2014 an airborne EM survey was conducted to test the capabilities of the HeliSAM system for a total of 97.5 line km. This was performed by GAP Geophysics, based in Perth Australia, using a ground based transmitting loop and airborne total field magnetic sensor. The testing was aimed to identify the Lalor mine at a depth beyond the capabilities of conventional airborne EM systems. The test was successful and has led to further surveys of this type elsewhere in the mining camp.

1.5 Drilling

The Lalor mine was discovered by Hudbay drilling a surface exploration hole testing a electromagnetic geophysical anomaly in March 2007, which intersected appreciable widths of zinc-rich massive sulphides in hole DUB168. Surface drilling continued through July 2012.

A limited surface exploration drill program was conducted from August to October 2015 to explore for potential down plunge extensions of copper-gold Zone 27 and to test near mine geophysical conductors that could not be drilled from underground workings. As of January 1, 2017 a total of 203,037 m of surface drilling was completed at Lalor and Table 1-1 provides a summary by year.

TABLE 1-1: SUMMARY SURFACE DIAMOND DRILL HOLES WITH ASSAY RESULTS AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2007	Parent	Hudbay	2	BQ	2,342	Major Drilling Ltd.
	Parent	Hudbay	26	NQ	29,600	Major Drilling Ltd.
2008	Parent	Hudbay	41	NQ	45,454	Major Drilling Ltd.
	Wedge	Hudbay	32	NQ/AQ	12,112	Major Drilling Ltd.
2009	Parent	Hudbay	29	NQ	35,390	Major Drilling Ltd.
	Wedge	Hudbay	47	NQ/AQ	22,884	Major Drilling Ltd.
2010	Parent	Hudbay	13	NQ	17,438	Major Drilling Ltd.
	Wedge	Hudbay	17	NQ/AQ	11,576	Major Drilling Ltd.
2011	Parent	Hudbay	10	NQ	15,458	Major Drilling Ltd.
	Wedge	Hudbay	5	NQ/AQ	3,139	Major Drilling Ltd.
2012	Parent	Hudbay	3	NQ	4,688	Major Drilling Ltd.
2015	Parent	Hudbay	2	NQ	2,956	Rodren Drilling Ltd.
Total			**222**		**203,037**	

1.6 Sample Preparation, Analyses and Security

Since the start of exploration at Lalor the following different laboratories and sample shipment/preparation procedures have been in use:

- Discovery to November 1, 2009, Lalor samples were prepared and analyzed at Hudbay laboratory in Flin Flon, Manitoba. As part of Hudbay Quality Assurance and Quality Control (QAQC) procedures, pulp duplicates were sent to ACME Analytical Laboratories Ltd. (ACME) in Vancouver, BC.
- November 1, 2009 to March 12, 2012, Lalor samples were received, crushed and pulverized at Hudbay laboratory and pulps shipped to ACME for analysis. Pulp duplicates were analyzed at Hudbay laboratory as part of QAQC procedures.
- March 13, 2012 to May 21, 2014, all samples were prepared and analyzed at Hudbay laboratory. As part of Hudbay QAQC procedures, pulp duplicates were sent to ACME.
- May 22, 2014 to present, parts of the sample stream were and are shipped to ACME, (re-named to Bureau Veritas after January 1, 2015). The remainder of sample stream is shipped to the Hudbay laboratory.
- A set of 303 drill core pulps from samples originating from within known resource envelopes were submitted for check assaying at a SGS Laboratory in Burnaby, BC as part of the QAQC program for the 2017 resource estimate.

1.6.1 Sample Preparation

All samples arriving at the Hudbay analytical laboratory are checked against the geologist's sample submission sheets. Laboratory analytical work sheets are generated for the analysis areas. Any wet samples are dried at 105°C as per industry standard. The core samples are crushed to (-)10 mesh then split to approximately 250 g and pulverized with 90% passing (-)150 mesh before being deposited into labelled bags. Crusher and pulverizer checks are conducted daily to ensure there is no excessive wear on the crusher plates and pulverizer pots.

All samples arriving at ACME (Bureau Veritas) are checked against chain of custody information on sample submittal form and prepped according to codes WGHT and PR80-250. The sample preparation includes weighing of sample, crushing 1 kg to minimum 80% passing 2 mm. A 250 g split crushed to minimum 85% passing 75 µm.

1.6.2 Analyses

Samples sent to the Hudbay laboratory were analyzed for the following elements: gold, silver, copper, zinc, lead, iron, arsenic and nickel. Base metal and silver assaying was completed by aqua regia digestion and read by a simultaneous Inductively Coupled Plasma (ICP) unit. The gold analysis was completed on each sample by atomic absorption spectrometry (AAS) after fire assay lead collection. All samples with gold values (AAS) > 10 g/t were re-assayed using a gravimetric finish.

Two different assay methods are used for samples shipped to Bureau Veritas: AQ270 and AQ370. AQ370 was the only method used on samples submitted from May 2014 to the second quarter of 2016 after which the AQ270 method was applied to selected holes. After the fourth quarter of 2016 all samples submitted to Bureau Veritas were assayed using the AQ270 method. All samples using method AQ270 and AQ370 were run for gold using method FA430. The following elements were run for over range as necessary: gold, copper, zinc and lead using methods FA530, GC820, GC816, MA404 (assays returning lead values above 20% using MA404 were also run using method GC817) respectively.

Samples shipped to Bureau Veritas from November 1, 2009 to March 12, 2012, were run using the legacy codes (Group 7AR) and (Group 601) with over range samples being run with gravimetric finish (Group 612). The sample preparation for these legacy codes is essentially similar to those listed for the current AQ270 and AQ370 codes used after May 22, 2014.

For the multi-element methods AQ270 and AQ370, aliquots of 1.000 ± 0.002 g are weighed into 100 mL volumetric flasks. Bureau Veritas QAQC protocol requires one pulp duplicate to monitor analytical precision, a blank, and an aliquot of in-house reference material to monitor accuracy in each batch of 36 samples. 30 mL of Aqua Regia, a 1:1:1 mixture of ACS grade concentrated HCl, concentrated HNO_3 and de-mineralised H_2O, is added to each sample. Samples are digested for one hour in a hot water bath (> 95°C). After cooling for 3 hours, solutions are made up to volume (100 mL) with dilute (5%) HCl. Very high-grade samples may require a 1 g to 250 mL or 0.25 g to

250 mL sample/solution ratio for accurate determination. Bureau Veritas QAQC protocol requires simultaneous digestion of a reagent blank inserted in each batch.

For both AQ270 and AQ370 sample solutions are aspirated into an ICP emission spectrograph (ES) to determine 24 elements. For method AQ270 the solution is also run through an ICP mass spectrometer (MS) to provide values for an additional 10 elements bring the total number of elements to 34. Raw and final data from the ICP-ES/ICP-MS undergoes a final verification by a British Columbia Certified Assayer who then signs the Analytical Report before it is released to the client.

For the gold analysis FA430, 30 g charges are weighed into fire assay crucibles. The sample aliquot is custom blended with fire assay fluxes, PbO litharge and a silver inquart. Firing the charge at 1050°C liberates Au, Ag ± PGEs that report to the molten Pb-metal phase. After cooling the lead button is recovered, placed in a cupel, and fired at 950°C to render an Ag ± Au ± PGEs dore bead. The bead is weighed and parted (i.e. leached in 1 mL of hot HNO_3) to dissolve silver leaving a gold sponge. Adding 10 mL of HCl dissolves the Au ± PGE sponge. Solutions are analysed for gold on an ICP emission spectrometer. Gold in excess of 10 g/t forms a large sponge that can be weighed (gravimetric finish, method FA530).

1.6.3 Security

Security measures taken to ensure the validity and integrity of the samples collected include:

- Chain of custody of drill core from the drill site to the core logging area
- All facilities used for core logging and sampling located on a secure mine site
- Core sampling is undertaken by Hudbay geologists
- Sample splitting and shipping conducted by technicians under the supervision of Hudbay geologists
- Chain of custody for core cutting through to delivery of samples to laboratories
- Well documented and implemented receiving and processing procedures at the Hudbay and Bureau Veritas laboratories
- The Hudbay laboratory samples results are stored on a secure mainframe based Laboratory Information Management System (LIMS)
- The diamond drill hole database is stored on the secure Hudbay network, using the acQuire database management system with strict access rights

1.7 Data Validation

Drill core is logged, sample intervals selected and either whole core or split core with remaining half core kept for reference is submitted to the laboratory for assaying. As part of Hudbay's QAQC program, the following QAQC samples are inserted into the sample stream for every 100 samples:

1. Two blanks
2. Five duplicates
3. Five base metal standards, each of differing grade thresholds
4. Two gold standards of differing grade

It is concluded that the analytical accuracy and reproducibility of copper, zinc, and silver as indicated by the QAQC samples submitted at the Hudbay laboratory is appropriate for resource estimation. High grade gold standards are being under assayed at the Hudbay laboratory. This under assaying in turn affects the high-grade Lalor samples and will likely led to an underestimation of gold in the resource estimate, since the proportion of samples assayed at the Hudbay laboratory is approximately 80% of the total samples assayed between 2012 and 2016.

The accuracy and reproducibility of copper, zinc, silver, and gold assays, as indicated by the QAQC samples submitted at Bureau Veritas laboratories, is of good quality for resource estimation.

1.8 Mineral Processing and Metallurgical Testing

The Stall concentrator began processing ore from Lalor in August 2012, initially producing only a zinc concentrate and by October 2012 a copper concentrate. As Lalor increased ore production the mill underwent an initial expansion and by the summer of 2014 was capable of processing at 2,800 tpd throughput rate. Modifications and improvements at the Stall concentrator over the last few years has increased the milling performance to an average rate of 3,000 tpd in 2016.

The properties of the Lalor base metal ore 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, with the exception of a few higher grade copper years and 90 to 95% range for zinc for the remaining Life of Mine (LOM). The yearly LOM metal recoveries, shown in Table 1-2, 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. Optimal operating days are considered to be steady run and an appropriate control of parameters.

TABLE 1-2: EXPECTED LOM RECOVERIES AT STALL CONCENTRATOR

	Metal Recoveries				Concentrate Grade	
Year	**Au (%)**	**Ag (%)**	**Cu (%)**	**Zn (%)**	**Zn (%)**	**Cu (%)**
2017	59.6	51.9	83.9	93.6	51.0	21.0
2018	53.5	46.7	83.9	91.8	51.0	21.0
2019	55.8	48.2	83.1	91.7	51.0	21.0
2020	57.9	57.5	88.2	90.0	51.0	21.0
2021	61.9	66.1	89.1	90.5	51.0	21.0
2022	62.3	66.9	88.6	91.8	51.0	21.0
2023	60.1	60.8	88.2	91.8	51.0	21.0
2024	55.3	47.6	86.1	93.5	51.0	21.0
2025	59.7	52.2	87.7	91.7	51.0	21.0
2026	52.6	38.8	84.7	92.0	51.0	21.0
2027	55.8	39.5	81.7	90.6	51.0	21.0

Although it is the author’s opinion that actual plant performance overrides previous metallurgical testing it is appropriate to summarize the relevant testing completed on Lalor mineralization prior to processing ore at Stall concentrator.

The primary objectives of the test programs in 2009 and 2011 were to develop an appropriate flowsheet for either the design of a new concentrator or modifications to the existing Stall concentrator, and to determine expected concentrate grades and metal recoveries.

Mineralogical analysis showed that chalcopyrite in Lalor ore is mostly coarse grained and liberated at grind sizes of approximately 100 microns. However, 15 to 20% of the chalcopyrite remains locked, primarily with sphalerite, at sizes below 20 microns. Copper is present almost exclusively as chalcopyrite with minor bornite. Zinc is present mainly as sphalerite, with minor amounts of gahnite. The sphalerite is coarse-grained and liberated at a grind size of 250 microns. Lead is present as fine-grained galena and would require a grind size of 70 microns for liberation. However, there is insufficient galena in the ore to warrant a primary grind this fine.

Zinc recoveries achieved in the plant are generally higher than the laboratory recoveries while zinc concentrate grades in the plant have been lower than the laboratory results. This is a preferred operating option due to Hudbay’s short concentrate haul to their Flin Flon metallurgical site.

1.9 Mineral Resource Estimates

The mineral resources for Lalor are estimated either as base metal lenses or gold zones and classified as Measured, Indicated and Inferred resources, inclusive of mineral reserves, as of September 30, 2016. The Qualified Person for the mineral resource estimate is Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

The resource is based on integrated geological and assay interpretation of information recorded from diamond drill core logging and assaying and underground mapping and is comprised of the following steps: exploratory data analysis, high-grade capping, high yield grade restrictions, and estimation and interpolation parameters consistent with industry standards. A total of 420,310 m in 1,707 holes have been drilled at Lalor deposit.

Mineral resources were classified in accordance to CIM Definition Standards on Mineral Resources and Mineral Reserves, into measured, indicated and inferred, depending upon the confidence level of the resource based on experience at Lalor mine, with similar deposits and spatial continuity of the mineralization.

The base metal resources were categorized as measured when a distance to an underground development drift is generally less than 10 m, indicated when the closest distance is less than or equal to 50 m to a composite, and the remainder of the interpolated blocks within the interpreted lenses are classified as inferred.

The gold zone resources were classified using the relative difference between the kriged grade and the composite grade. The Resource Classification Index (RCI) uses the ordinary kriging combined variance, block model grade and a calibration factor based on the distance of the composite, number of composites, number of quadrants, and the number of drill holes are used in the formula. An RCI value that corresponds to the 50th percentile was used as a threshold for indicated resource, except in Zone 27 where a RCI of 70th percentile was used. Measured blocks were classified based on the approximate distance of 10 m to an underground development drift. All remaining blocks with minimum criteria of one drill hole to interpolate the grades were classified as inferred.

The Lalor block model was validated to ensure appropriate honouring of the input data by the following methods:

- Visual inspection of the ordinary kriging (OK) block model grades in plan and section views in comparison to composites grade
- Metal removed via grade capping and high yield restriction methodology
- Comparison between the interpolation methods of nearest neighbour and inverse distance squared weighted to confirm the absence of global bias in the OK grade model
- Swath plot comparisons of the estimation methods to investigate local bias
- Review of block model ordinary kriging quality control parameters
- Comparison of grade tonnage curves and statistics by estimation method
- Third party review of the block model and estimation process

The base metal mineral resources, inclusive of reserves by category with a zinc equivalency cut-off of 4.1% zinc, is shown in Table 1-3 and the gold mineral resources, inclusive of reserves by category with a gold equivalency cut-off of 2.4 g/t gold, is shown in Table 1-4, as of September 30, 2016.

TABLE 1-3: BASE METAL MINERAL RESOURCE, INCLUSIVE OF MINERAL RESERVES BY CATEGORY AND MINERALIZED ZONE WITH A CUT-OFF OF 4.1% ZN EQ, AS OF SEPTEMBER 30, 2016 (1)(2)(3)(4)(5)(6)(7)(8)(9)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Measured	5,126,000	8.34	2.46	0.86	31.34
Indicated	8,842,000	6.67	2.00	0.59	30.44
Measured + Indicated	13,967,000	7.28	2.17	0.69	30.77
Inferred	545,300	8.15	1.45	0.32	22.28

Notes:

1. Domains were modelled in 3D to separate mineralized zones from surrounding waste rock. The domains were based on core logging, grade, structural and geochemical data.
2. Raw drill hole assays were composited to 1.25 m lengths, honouring lithology boundaries.
3. Capping of high gold and silver grades was considered necessary and was completed on assays prior to compositing.
4. High yield restriction of base metal high grade and density was completed for each domain after compositing.
5. Block grades for zinc, gold, silver, copper, lead, iron, arsenic and density were estimated from the composites using ordinary kriging interpolation into 5 m x 5 m x 5 m blocks coded by domain.
6. Density values are from a multi elements regression formula based on 65,792 measurements
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Metal prices of $US 1.19/lb zinc, $US 1,300/oz gold, $US 2.67/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to calculate a zinc equivalence (Zn Eq) cut-off of 4.1%, where Zn Eq = Zn% + (1.98 x Cu%) + (1.11 x Au g/t) + (0.01 x Ag g/t) – (0.01 x Pb%). The Zn Eq considers the ratio of milling recovery, payability and value of metals after application of downstream processing costs. The Zn Eq cut-off of 4.1% covers administration overhead, mining removal, milling and general and administration costs.
9. Totals may not add up correctly due to rounding.

TABLE 1-4: GOLD MINERAL RESOURCE, INCLUSIVE OF MINERAL RESERVES BY CATEGORY AND MINERALIZED ZONE WITH A CUT-OFF OF 2.4 G/T AU EQ, AS OF SEPTEMBER 30, 2016 (1)(2)(3)(4)(5)(6)(7)(8)(9)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Measured	332,000	0.44	6.34	0.40	33.03
Indicated	4,108,000	0.50	6.23	0.89	34.80
Measured + Indicated	4,440,000	0.50	6.24	0.86	34.67
Inferred	4,124,000	0.31	5.01	0.90	27.61

Notes:

1. Domains were modelled in 3D to separate mineralized zones from surrounding waste rock. The domains were based on core logging, grade, structural and geochemical data.
2. Raw drill hole assays were composited to 1.25 m lengths, honouring lithology boundaries.
3. Capping of high gold and silver grades was considered necessary and was completed on assays prior to compositing.
4. High yield restriction of base metal high grade and density was completed for each domain after compositing.
5. Block grades for zinc, gold, silver, copper, lead, iron, arsenic and density were estimated from the composites using ordinary kriging interpolation into 5 m x 5 m x 5 m blocks coded by domain.
6. Density values are from a multi elements regression formula based on 65,792 measurements collected by Hudbay.
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Metal prices of $US 1,300/oz gold, $US 2.67/lb copper and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to calculate a gold equivalence (Au Eq) cut-off of 2.4 g/t Au Eq, where Au Eq = Au g/t + (1.34 x Cu %) + (0.01 x Ag g/t). The Au Eq considers the ratio of milling recovery, payability and value of metals after application of downstream processing costs. Au Eq cut-off of 2.4 g/t covers administration overhead, mining removal, milling and general and administration costs.
9. Totals may not add up correctly due to rounding.

1.9.1 Mine Reconciliation of Block Model

A mine reconciliation of the block model was carried out on the mined out areas. The process involved selecting all mined out blocks and comparing to the actual metal balance reported as ore received at the Stall concentrator. Mined out areas from the block model do not include dilution or pillars left behind after mining extraction. Table 1-5 list the mined out areas by lense from the block model and Table 1-6 is the ore reported by year at the Stall concentrator since Lalor commenced production in August 2012 until September 2016.

TABLE 1-5: MINED OUT AREAS FROM THE BLOCK MODEL

Lense	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
10	1,560,246	9.63	1.94	0.71	19.39
11	55,355	14.46	0.14	0.30	17.22
20	336,600	8.80	2.19	0.82	30.71
21	99,594	0.65	7.70	0.65	30.85
23	24,210	1.50	5.37	0.61	39.89
24	25,849	0.89	6.44	0.42	23.24
25	56,481	0.64	8.47	0.40	31.93
30	9,098	4.25	1.82	0.26	35.67
31	64,845	4.73	0.95	0.20	15.14
32	183,404	7.98	5.51	1.60	54.22
Total	**2,415,684**	**8.60**	**2.65**	**0.75**	**24.52**
		Zn (tonnes)	**Au (ounces)**	**Cu (tonnes)**	**Ag (ounces)**
In-Situ Metal		207,641	205,689	18,196	1,904,304

TABLE 1-6: ORE RECEIVED BY YEAR AT THE STALL CONCENTRATOR

Year	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
2012	72,294	11.83	1.68	0.63	19.30
2013	400,589	9.44	1.20	0.84	19.41
2014	551,883	8.52	2.29	0.88	23.83
2015	934,278	8.18	2.53	0.71	21.39
2016 Sep YTD	814,207	6.88	2.30	0.64	21.62
Stockpile as of Sep 2016	11,114	5.38	2.14	0.41	21.42
Total	2,784,365	8.13	2.20	0.74	21.60
		Zn (tonnes)	**Au (ounces)**	**Cu (tonnes)**	**Ag (ounces)**
Metal		226,438	196,908	20,525	1,933,617
	Tonnes	**Zn**	**Au**	**Cu**	**Ag**
Variance to Block Model	115%	109%	96%	113%	102%

The block model compared very well to the ore reported at the Stall concentrator. The mine reconciliation concludes a 15% mining dilution and a metal variance reported at the Stall concentrator of 109% for zinc, 96% for gold, 113% for copper and 102% for silver. A mine reconciliation of 5 to 10% variance is well within industry standard. The precious metals reconciled very well, while there might be some conservatism of the zinc and copper grade estimates in the block model. The conservatism of the zinc and copper grade is believed to be linked to the high yield radius parameter. A previous model selected a 40 m high yield radius compared to the current 20 m radius used for this block model.

1.10 Mineral Reserve Estimates

The mineral reserves were estimated based on a LOM plan prepared; using Deswik mine design software that generated mining inventory based on stope geometry parameters and mine development sequences. Appropriate dilution and recovery factors were applied based on cut and fill and longhole open stoping mining methods with a combination of paste and unconsolidated waste backfill material. The Qualified Person for the mineral reserve estimate is Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

The shallow dipping nature of the deposit and stacking of lenses results in multiple lenses being grouped together for mining purposes in the stope optimizer routines of Deswik so that they can be extracted as a single mining unit, based on stope mining parameters by mining method as shown in Table 1-7. Parameters most sensitive to Lalor mine are the minimum and maximum dip angles, which affects the dilution and recovery amounts of the optimized mining shape. The stope optimizer in Deswik generated an economic shape that honoured the geometric parameters.

TABLE 1-7: STOPING PARAMETERS BY MINING METHOD

Stope Shape Parameters	Unit	Longhole	Cut and Fill
Length Minimum Maximum	Metres	23 10	150 20
Width Minimum Maximum	Metres	3.5 50	5 50
Waste Pillar Width	Metres	5	-
Stope Height Minimum Maximum	Metres	10 20	5 5
Stope Dip Minimum	Degrees	35	75
Hanging Wall Dip Minimum Maximum	Degrees	20 90	70 90
Footwall Dip Minimum Maximum	Degrees	50 90	70 90
Dilution Hanging Wall Fixed Footwall Fixed	Metres	0.5 0.5	0.5 0.5

The space between the lenses is treated as internal dilution and external dilution is set at a fixed distance of 0.5 m 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. Dilution, set at zero grade and a bulk density of 2.8 t/m^3, is based on the full mining shape with internal and external dilution. Average dilution of the mineral resources that are in the LOM production plan is 18.9%.

Mining recovery is defined as the ratio of mineral resource tonnes delivered to the concentrator to the in-situ mineral resource tonnes. Some of the mineral resources are not recovered due to mining design, inefficiencies in mining and inefficiencies in mucking. Average recovery of the mineral resources that are in the LOM production plan is 81.1%

Diluted and recovered mineral resources exceeding a Net Smelter Return (NSR) cut-off of $88/t for longhole open stoping and $111/t for cut and fill mining method are included in the reserves. NSR's are based on metal grades from the stope optimizer and block model, long-term metal prices, concentrator recoveries, smelter treatment, refining and payabilities and a Hudbay Manitoba Business Unit administration cost.

Metal prices of $US 1.07/lb zinc (includes premium), $US 1,260/oz gold, $US 3.00/lb copper, and $US 18.00/oz silver with an CAD / US foreign exchange of 1.10 was used to estimate mineral reserves.

The orebody is polymetallic with economically significant metals being zinc, gold, copper and silver. There are two different ore type, both of which are assumed to be treated using conventional flotation at Hudbay Stall concentrator:

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

Metallurgical performance at Stall concentrator indicates that the base metal and gold rich ores can be blended and two concentrates will be produced, a zinc concentrate that will be shipped to the Hudbay Flin Flon metallurgical complex for production of refined zinc, and a gold enriched copper concentrate that will be shipped to third party smelters.

The Lalor mine mineral reserves as of January 1, 2017 are summarized in Table 1-8. The mine plan was prepared using measured and indicated mineral resources from the block model. Inferred resources were assumed as waste.

TABLE 1-8: SUMMARY OF MINERAL RESERVES AS OF JANUARY 1, 2017

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Proven	4,383,000	6.76	2.37	0.76	27.33
Probable	9,849,000	4.39	2.72	0.65	26.12
Proven + Probable	14,232,000	5.12	2.61	0.69	26.50

Notes:

1. CIM definitions were followed for mineral reserve

2. Mineral reserves are estimated at an NSR cut-off of $88/t for longhole open stoping mining method and $111/t for cut and fill mining method
3. Metal prices of $US 1.07/lb zinc (includes premium), $US 1,260/oz gold, $US 3.00/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.10 were used to estimate mineral reserves.
4. Bulk density of the resource is reported in the block model is from a multi elements regression formula based on 65,792 measurements. Stope geometry shapes include waste dilution based on a bulk density of 2.8 t/m^3.
5. Totals may not add up correctly due to rounding.

The conversion of resources to reserves is based on the LOM plan and NSR cut-offs that primarily focussed on capturing base metal resources for processing at the Stall concentrator. The secondary focus was to capture gold zone resources when in contact with or close proximity to base metal resources. In areas where a large separation existed between base metal and gold lenses, mining blocks were evaluated for economic stope mining shapes. When a non-economic shape was generated in a first pass, a second pass was evaluated for only base metal lenses and if an economic shape was generated the gold zone portion was removed. However, due to this larger separation, majority of these isolated gold lenses could have been evaluated independently of the base metal lenses and could potentially provide feed to a gold processing facility. Below approximately the 950 m level no attempt was made to generate an economic stope mining shape for gold zones 25 and 26 as the separation distance became too large. The author's opinion is that these resources are potentially better suited for a gold processing facility and should be re-evaluated when Hudbay has a better understanding of their New Britannia gold mill and Birch Tailings Impoundment Area in Snow Lake.

Of the current 14,232,000 tonnes of mineral reserves, approximately 80% is converted from base metal resources and approximately 20% is converted from gold zone resources. Of the total reserves approximately 3.8% is represented by the indicated resources of copper-gold zone 27, which is inclusive of the 20% from the gold zone, noted above. Although the indicated resource of copper-gold zone 27 as shown in Table 14-51, is converted to reserves and is planned to be processed at the Stall base metal concentrator, it has the potential to be milled at the New Britannia gold mill if the refurbishment plan and the installation of a copper pre-float facility proves to be economically viable.

The author recommends that base metal indicated resources, exclusive of reserves, as shown in Table 1-9 remain as indicated resources until such a time that detailed mine planning is completed. Furthermore, the author recommends that gold zone indicated resources, exclusive of reserves, as shown in Table 1-10 still have reasonable prospects of economic extraction at either Stall base metal concentrator or a gold processing facility.

TABLE 1-9: BASE METAL INDICATED RESOURCE, EXCLUSIVE OF RESERVES, WITH A CUT-OFF OF 4.1% ZN EQ, AS OF SEPTEMBER 30, 2016 (1)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Indicated	2,100,000	5.34	1.69	0.49	28.10

Notes:

1. Refer to the Notes for Table 1-3 of this Technical Report for more information

TABLE 1-10: GOLD INDICATED RESOURCE, EXCLUSIVE OF RESERVES, WITH A CUT-OFF OF 2.4 G/T AU EQ, AS OF SEPTEMBER 30, 2016 (1)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Indicated	1,750,000	0.40	5.18	0.34	30.61

Notes:

1. Refer to the Notes for Table 1-4 of this Technical Report for more information

1.11 Mining Methods

The Lalor mine is a multi-lens flat lying orebody with ramp access from surface and shaft access to the 955 m level. Internal ramps located in the footwall of the orebody provide access between mining levels. Stopes are accessed by cross cuts from the major mining levels.

The mining method process includes underground lateral advance (development rounds), production mining, backfilling and transporting ore to surface. Geotechnical information, orebody geometry interpreted from diamond drill core and recent experience mining within the deposit were the major considerations for selection of mining methods.

Mining methods that are currently in use or planned in the immediate future include: mechanized cut and fill, post pillar cut and fill, drift and fill and longhole open stoping (transverse and longitudinal retreat).

Ore is mucked using Load Haul and Dump (LHD) loaders which are operated remotely in inaccessible areas. The ore is then loaded into underground haul trucks or ore passes and transported to the ore handling system at the production shaft for hoisting to surface.

A paste backfill plant will be constructed on site, planned for the first quarter of 2018. Paste backfill will be used in higher grade areas to increase recovery and accelerate the mining cycle. Lower grade areas will be filled with waste rock from waste development. No waste is planned to be hoisted.

Ore delivered to the production shaft is sized to less than 0.55 m at one of the two rockbreaker/grizzly arrangements and hoisted from the mine by two 16 tonne capacity bottom dump skips in balance. Ore is truck hauled to a primary crusher at the Chisel North mine site, crushed to less than 0.15 m, and then trucked to the Stall concentrator for further processing.

1.11.1 Stope Mining

Two main mining methods are used at Lalor mine, cut and fill and longhole open stoping. Cut and fill methods include: mechanized cut and fill, post pillar cut and fill and drift and fill. Longhole open stoping methods include: transverse, longitudinal retreat and uppers retreat. Each mining area is evaluated to determine the most economic stoping method. In general, where the dip exceeds 35° and the orebody is of sufficient thickness, longhole open stoping is used and lateral based cut and fill mining methods are used in flatter areas.

Approximately 65% of the mineral reserves are mined using the longhole open stoping methods and 35% are mined with cut and fill methods.

1.11.2 Support Systems

Except when using cut and fill mining methods, all other drifts have arched backs for optimized shape and safety.

Ground support is broken down into primary support and secondary support.

Primary support refers to reinforcement of the rockmass immediately following excavation (first pass) to ensure safe working conditions before taking the next round. Primary support is typically undertaken with resin grouted rebar and #6 gauge galvanized welded wire mesh. Standard drift, <7 m wide: 2.2 m long #6 (for jackleg/stopper installations) or #7 (for bolter installations) resin rebar on a 1.2 m x 1.2 m square pattern. Rebar and screen are extended down the walls to within 1.8 m of the sill. Generally, this can be applied to a drift span up to 7.0 m if no major geological structures are encountered. Special design is needed for drift spans larger than 7.0 m or when major geological structure is present. Intersections, 7 m to 10.8 m wide: ground support uses the same support as for standard drifts except in the back where 3.6 m long #7 resin rebar on a 1.2 m x 1.2 m square pattern is installed.

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. Secondary support is installed at a later stage (second pass) and typically is a batch process. Examples of secondary support are rebar, cable bolts, strandlok bolts, inflatable rock anchors, split sets and shotcrete.

Secondary ground support is installed when excavation spans are larger than 10.8 m, major unfavorable ground conditions or rock structures are present, and/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, single cable bolts on a 1.8 m x 1.8 m pattern for long term excavations, or high strength inflatable rock anchors for temporary or short term excavations. The minimum bolt length should be equal to one-third of the final drift span.

1.11.3 Backfill

All stopes at Lalor mine are backfilled to maintain long term stability and to provide a floor to work from for subsequent mining. Unconsolidated waste rock fill 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 consolidated backfill. Consolidated backfill currently consists of cemented waste rock backfill and is planned to be primarily as paste backfill after commissioning of the paste plant in the first quarter of 2018. Where economically feasible consolidated backfill is used by adding cement to waste rock using a spray bar and placing it in stopes with LHDs or when paste is available using the underground distribution system to transport paste (via gravity) directly to the stope. Consolidated backfill is required to maintain long term stability and allow future recovery of sill pillars.

The majority of consolidated backfill will be paste. 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.

1.11.4 Ore Handling

Ore is mucked by LHD, loaded into underground haul trucks and hauled to one of the two ore passes that feed the shaft. Ore is dumped onto a grizzly at 910 m level for sizing to less than 0.55 m by a rockbreaker and grizzly. A 40 m raise and bin below the grizzly provides approximately 1,200 tonnes of coarse ore storage. A chute at the bottom of the raise at 955 m level feeds ore to a conveyor that loads a measuring flask with approximately 14 tonnes of ore. Ore is then skipped to surface by two 16-tonne capacity Bottom Dump skips in balance. The ore enters the head frame 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, and then trucked to the Stall concentrator to process. Opportunities to increase ore handling capacity and installation of additional shaft ore passes are currently being reviewed. However, based on an internal review the current ore handling system has the capacity to move 4,500 tpd.

1.11.5 Mining Operations

Typical development crew equipment consists of a two-boom electric hydraulic jumbo, and a mechanical bolter sized to excavate all lateral development (typical sizes include: 5.0 m x 5.0 m, 6.0 m x 5.0 m and 7.0 m x 5.0 m). The crew also uses LHDs, scissor lifts and backhoes for face preparation and extending services.

Current production is approximately 3,000 to 3,500 tonnes per day. At steady state by 2018, Lalor mine will produce 4,500 tonnes per day and be approximately 35% development based and 65% longhole. Development based production methods will produce approximately 4.2 ore rounds (1,600 tonnes) per day. Cut and fill mining areas are assumed to be in the ore producing portion of the mining cycle 75% of the time and in the backfill portion of the mining cycle or otherwise unavailable for mining 25% of the time. Longhole mining based production will produce approximately 2,900 tonnes per day.

1.11.6 Mine Equipment

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 tonnes per day is shown in Table 1-11.

TABLE 1-11: MINE EQUIPMENT

Description	Fleet
Underground Trucks 65 tonne	4
Underground Trucks 42 tonne	4
LHD 8yd	5
LHD 10yd	5
Two Boom Jumbo	4
Bolters (includes require for cable bolting)	8
Longhole Drills	3
Powder Trucks	3
Scissor Lift Trucks	8
Grader	1
Boom truck	2
Shotcrete Sprayer	1
Trans-mixers	2
Personnel Carriers Toyota	26
Miscellaneous Underground (Minecats, forklifts, etc.)	19
Miscellaneous Surface (Loader, forklift, pickups, etc.)	22
Total Mobile Equipment.	**117**
Ventilation Fans – Surface fans (250 HP – 2500 HP)	6
Ventilation Fans – U/G fans (50 HP – 400 HP)	54
U/G Submersible Pumps 100 HP	7
U/G Submersible Pumps 60 HP	1
U/G Submersible Pumps 50 HP	1
U/G Submersible Pumps 40 HP	13
U/G Submersible Pumps 34 HP	1
U/G Submersible Pumps 20 HP	7
U/G Submersible Pumps <20 HP	5
Portable Refuge Stations	3
Shotcrete Machine - Wet Mix	1
Grout Pump c/w Mixer	3
Portable Welder	1
Total Stationary Equipment	**103**

An allowance for replacement equipment has been included in the mine plan. As part of the mobile equipment fleet management plan, major mobile equipment will be replaced at approximately 15,000 operating hours.

1.11.7 Production Schedule

The LOM production schedule, shown in Table 1-12, is currently set to ramp-up to 4,500 tonnes tpd by 2018 and continues at that rate to the end of 2021 when a ramp-down begins to 3,000 tpd in 2026. Deswik software was used to assist with the LOM planning and generate the basis for the production schedule. The geologic block model was imported to the software, where a stope optimizer algorithm was applied to create economical mining shapes. These mining shapes were then linked to the development drifts and sequenced individually by their respective locations and geometrical limits. Mine resources and rates, realized through historical data, were applied and levelled through activity priority labels. From this output, adjustments were made to further balance resources and scheduling to create an improved plan.

TABLE 1-12: LOM PRODUCTION SCHEDULE

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2017	1,278,282	1.67	22.68	0.59	7.52
2018	1,616,285	2.13	24.37	0.52	5.71
2019	1,620,000	1.86	21.43	0.48	5.62
2020	1,603,652	2.79	28.43	0.79	4.61
2021	1,620,000	2.86	26.39	0.92	4.83
2022	1,473,657	3.16	26.72	0.95	5.72
2023	1,267,267	3.21	29.87	0.89	5.72
2024	1,212,738	3.14	28.35	0.60	4.49
2025	1,212,739	2.89	27.35	0.66	3.41
2026	1,022,918	2.78	32.83	0.49	3.55
2027	304,098	1.83	23.93	0.37	2.68
Total	**14,231,636**	**2.61**	**26.50**	**0.69**	**5.12**

Lalor mine will produce a total of 1,312,563 tonnes of zinc concentrate and 404,864 tonnes of copper concentrate in milling the ore from the LOM production plan, as shown in Table 1-13. The LOM contained metal in concentrate is shown in Table 1-14.

TABLE 1-13: LOM CONCENTRATE PRODUCTION BY YEAR

	Zinc Concentrate		Copper Concentrate			
Year	Tonnes	Zn (%)	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)
2017	176,396	51.0	30,158	42.2	499.1	21.0
2018	166,124	51.0	33,298	55.3	552.6	21.0
2019	163,716	51.0	30,863	54.5	541.8	21.0
2020	130,580	51.0	53,179	48.7	492.7	21.0

2021	138,843	51.0	63,025	45.4	448.7	21.0
2022	151,843	51.0	58,903	49.2	446.9	21.0
2023	130,488	51.0	47,567	51.5	483.5	21.0
2024	99,888	51.0	29,786	70.7	549.8	21.0
2025	74,437	51.0	33,629	62.3	514.7	21.0
2026	65,591	51.0	20,064	74.6	649.6	21.0
2027	14,658	51.0	4,394	70.8	653.4	21.0
Total	**1,312,563**	**51.0**	**404,864**	**53.4**	**502.8**	**21.0**

TABLE 1-14: LOM CONTAINED METAL IN CONCENTRATE

Year	Zn (tonnes)	Cu (tonnes)	Au (oz)	Ag (oz)
2017	89,962	6,333	40,917	483,928
2018	84,723	6,993	59,202	591,589
2019	83,495	6,481	54,079	537,611
2020	66,596	11,168	83,265	842,391
2021	70,810	13,235	91,994	909,201
2022	77,440	12,370	93,174	846,328
2023	66,549	9,989	78,760	739,424
2024	50,943	6,255	67,705	526,512
2025	37,963	7,062	67,359	556,492
2026	33,451	4,213	48,122	419,039
2027	7,476	923	10,002	92,306
Total	**669,408**	**85,022**	**694,578**	**6,544,821**

1.11.8 Mine Ventilation

The Chisel North mine ventilation system in sequence with the Lalor mine Downcast Raise, provide 400,000 cubic feet per minute (cfm) down the Lalor mine Access Ramp, with 150,000 cfm exhausting to surface via the Chisel North mine Ramp. An additional 555,000 cfm is downcast via the Lalor mine Production Shaft for a total of 955,000 cfm exhausting up the Main Exhaust Shaft. In the summer total volume of air increases slightly. Three heaters heat mine air in the winter: the 36M BTU Chisel North Mine Heater, the 30M BTU Lalor Mine Ramp Heater and an 80M BTU heater at the production shaft.

With the increase in mining rate from 3,000 to 4,500 tpd several new areas are being brought into production. As the footprint of the mine expands, the ventilation system will also require expansion to allow fresh air to be delivered to active mining areas.

1.11.9 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 feeds power to the Hudbay owned Main distribution substation consisting of two (2) 115-25 kV 24 MVA transformers. Substation is completely equipped with an E-house complete with 4 GE Powervac circuit breakers and Tie breaker. Lalor mine underground mine electrical distribution to the mine workings consist 13.8 kV that is further stepped down to 600 V.

1.11.10 Workforce

Lalor mine is operated on a continuous cycle. The majority of operations and maintenance personnel work 11.5 hour shifts on a 5-5-4 day cycle or a 7-7 day cycle. Operations support, technical and administrative personnel work 8 hour day shifts, 40 hours per week. The mine is operated under Collective Bargaining Agreements between Hudbay management and local unions.

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 manpower shortages. Personnel will vary year to year. Steady state personnel requirements are shown in Table 1-15.

TABLE 1-15: MINE OPERATIONS WORKFORCE

Discipline	Personnel
Direct Operations	180
Supervision and Administration	47
Health and Safety	4
Mine Maintenance	74
Mine Technical	34
Total Lalor Mine	**339**

1.11.11 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 operations including, but not limited to:

a) Positive Attitude Safety System (PASS) safety program

b) Health monitoring programs (hearing and lung)

c) Dust monitoring

d) Ongoing water and environmental monitoring

e) Personal Protective Equipment (ie. reflective outerwear, eye protection, hearing protection, respirators)

f) Task analysis and job procedures

1.11.12 Mining Method Opportunities

Lalor mine is considering different opportunities to improve mining efficiencies:

- Autonomous operation of LHDs are currently being trialed from surface by tele-remote with changes to standard designs to allow isolation of autonomous areas and buffer storage (transfer raises) for in between shift mucking
- A main ore pass from 755 m level to 910 m level is planned for 2017 to reduce trucking time from the upper levels of the mine
- Alternative truck loading systems are being investigated as an alternative to LHD loading
- Stoping block design changes are being considered to allow box hole primary mucking and circle route loading of trucks

1.12 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 will be expanded from the current 3,000 tpd rate to 4,500 tpd and the mill will operate 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 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 will be utilized to produce a cemented paste backfill for use underground at Lalor. Tailings not required for paste backfill will continue to be pumped to the existing Anderson TIA.

Run of mine ore as large as 0.55 m in one dimension is withdrawn from the head frame ore bin at Lalor and is transported to a crushing plant located at the Chisel North mine. The crushing plant reduces the ore to a range of 10 to 15 cm and is transported to the Stall concentrator coarse ore bins or to a stockpile at the mill. A stockpile of 36,000 tonnes, equivalent to 8 days production at the future expansion rate, is required to blend high-grade zinc to ensure a more consistent zinc feed.

The Stall concentrator consists of the following areas: stockpiles, crushing and screening, grinding circuits, copper and zinc flotation process, and concentrate dewatering. Flotation tailings currently pumped to the Anderson TIA will be pumped to the future paste plant at Lalor or to the Anderson TIA depending on the demand for paste. Water for the ore-grade mineral extraction process utilizes two sources: fresh and reclaimed water. Approximately 25% of the water usage is withdrawn from the fresh water of Snow Lake while the remaining approximately 75% of the water is reclaimed from Anderson TIA.

Engineering for the Stall concentrator expansion is currently underway and construction is planned to commence in the third quarter of 2017 with commissioning in the third quarter of 2018. This schedule is offset to the Lalor production ramp-up by about 6 months. Lalor is planned to be at a nominal mining rate of 4,500 tpd in the first quarter of 2018. The additional ore tonnage from Lalor is planned for either stockpiling or will be transported to the Flin Flon concentrator for processing.

1.13 Project Infrastructure

This section addresses the infrastructure facilities that support the current operation at the Lalor mine and process facilities with discussion on projects, expansions and improvements. The current infrastructure facilities include Lalor (underground, onsite and offsite), power, Stall concentrator, Anderson TIA. Projects, expansions and improvements are related to the Stall concentrator, paste plant, Anderson TIA and underground ore handling system.

1.13.1 Lalor Infrastructure

Lalor mine is designed to hoist 6,000 tpd combined ore and waste. Primary access to the mine is a concrete lined 6.9 m diameter production shaft with a secondary ramp access from surface through the Chisel North mine. Ore is hoisted to surface and trucked to the Chisel North site where it is crushed then hauled to the Stall concentrator for processing into two concentrates zinc and copper.

General area infrastructure includes provincial roads and 115 kV Manitoba Hydro grid power to within four km of Lalor, and Manitoba Telecom 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 914 m gravel airstrip to provide emergency medical evacuation.

Lalor is located 3.5 km from the Hudbay Chisel North mine. Chisel North infrastructure includes a mined out open pit used for waste rock disposal, fresh (process) water sources, pumps and waterlines, 4160 V and 550 V power, mine discharge water lines, a 2,500 gallons per minute (gpm) water treatment plant with retention areas, plus mine buildings including offices and a change house.

The permitted Hudbay Anderson TIA, located approximately 12km from Lalor is used for tailings disposal.

Underground Infrastructure

- Main Production shaft 6.9 m diameter concrete lined with five compartments
- Three main shaft stations at 835 m, 910 m and 955 m levels

- Lateral development consists of 6 m x 5 m ramps and level development
- Secondary egress consists of a ramp access from surface to the 810 m level where it joins the rest of the Lalor ramp system. Total distance is approximately 6.0 km.
- Power Distribution consists of 25 kV power down the shaft, 7.5 MVA 25 kV to 13.8 kV transformer to the 910 mL shaft station and primary distribution throughout the mine is 13.8 kV with transformers to 600 volt for local distribution
- Compressed air and process water are piped throughout the mine from surface
- Underground wireless radio communication throughout the mine is provided by a Leakey Feeder system
- Fiber-optic backbone for data and video
- Ore handling system consists of two rock breakers and bins (1,400 tonnes each) on 910 m level feeding chutes and conveyor system on 955 m level supplying the ore to the skips
- Mobile Maintenance shop is located in the Chisel North underground workings
- Discharge system consists of a series of drain holes and sumps with submersible pumps that feed the top of the two settling cones on 910 m level. Water from the 955 m level is pumped to surface by one 1,250 hp pump.

Onsite Infrastructure

- Office/change house complex with dry space for 341 personnel
- Hoisthouse consisting of the electrical distribution for the site, hoist and communication control room, production hoist, service hoist and compressors
- Headframe consisting of the utility hoist, bin house, external bunker, twin 250 hp downcast fans and mine heater
- Two 30,000 US gallon propane tanks
- Main pump station includes holding tanks and PAL water system and pumps for discharge, potable, process and fire water
- Bio Disk Sewage treatment plant for up to 38 m^3/day
- Fuel tanks and pumps for diesel and gas.
- Two backup generators one for the utility hoist and one for the main pump station
- Temporary offices for health safety, training and mine rescue, temporary change house for contractors on site and temporary trailer for onsite contractor.
- Warehouse/shop
- Vent shaft and two exhaust fans

Offsite Infrastructure

- 198 person camp in the Town of Snow Lake to house out of town personnel
- 3.5 km gravel access road connecting Provincial Road 395 to the mine site
- Two 24MVA - 115 kV to 25 kV power substations located on the Chisel North site
- The Chisel North complex is used for the diamond drilling core processing facility, shop to maintain the surface equipment fleet and offices for the project group
- Crushing of the Lalor ore is done at the Chisel North site with a maximum total stock pile capacity of 15,000 tonnes
- Booster pump station at Chisel North with holding tanks and pumps for process and discharge water. Equipped with a backup generator (350 kW).
- Two downcast fans, mine heaters with 30,000 US gallon propane tank each
- Discharge water from the Lalor site is pumped 3.5 km to the booster pump station at the Chisel North site where it is pumped the remaining 3.5 km to the settling ponds by the Chisel Pit
- A High Density Sludge Process Acidic Water Treatment Plant that can treat up to 2,500 gpm prior to being discharged to the environment

Ore Handling Improvements

Based on a review of Lalor's underground ore handling circuit by Stantec in early 2017 Hudbay is planning capital improvements in 2017. These improvements will ensure Lalor is able to maintain a steady 4,500 tpd of production through the ore circuit. These improvements pertain to maintenance repair and replacement of liners in the ore circuit and to reduce potential hang-ups in the system.

1.13.2 Stall Concentrator

Hudbay operates the Stall concentrator approximately 16 km from Lalor. The mill is currently operating seven days per week at 3,000 tpd, processing ore from the Lalor mine. The mill has two circuits, with design capacities of 909 tpd and 2,182 tpd. The concentrator has two flotation circuits producing a zinc concentrate and a copper concentrate. The tailings associated with the Lalor mine are deposited in the Anderson TIA.

Produced copper concentrate is currently hauled by 40 ton trucks to Flin Flon, where the concentrate is loaded onto gondola rail cars for market. Produced zinc concentrate is hauled by 40 ton trucks to Flin Flon and is processed at Hudbay's zinc processing facilities.

1.13.3 Stall Concentrator Expansion

The Stall concentrator expansion project to 4,500 tpd, slated for commissioning in third quarter of 2018, addresses the following areas: crushing, copper and zinc flotation, thickening, dewatering, electrical power distribution, compressed air systems and water distribution systems that will enable the concentrator to operate at 4,500 tpd.

1.13.4 Anderson TIA

Anderson TIA is located in the Snow Lake area between the Stall concentrator and Lalor mine. The purpose of the Anderson TIA is environmental management (storage) of mine tailings produced in the Stall concentrator. 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 construction of this expansion is anticipated to be complete prior to 2019.

1.13.5 Paste Plant

The Lalor paste plant project was approved in February 2017 and is critical for the sustainability of the mine production plan. The paste plant will be located northeast of the existing headframe complex and delivery capacity of the paste is 165 tph solids (tails) or 93 m^3/hr paste. 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 not to hoist waste from underground the combined paste/waste backfilling capacity is approximately 6,000 tpd. The paste plant will be capable of varying the binder content in the paste to provide flexibility in the strength gain of the paste where higher and early strength may be required depending on mining method.

Tails that are currently pumped from the Stall concentrator to the Anderson TIA will be diverted to the Anderson booster pump station. The tailings will be directed into the Anderson TIA when not required for the paste plant. Two pipelines will be installed between the Anderson booster pump station and the paste plant located at Lalor mine site, approximately a 13 km distance.

Paste will be delivered underground via one of two (nominal 8 inch diameter) cased boreholes from the surface to the 780 m level of the Lalor mine. The boreholes were drilled and cased in 2016. A network of underground lateral piping and level to level boreholes will transfer the paste from the base of the discharge hopper to the required underground locations. The majority of the underground distribution system will utilize existing drifts or planned future development.

1.14 Market Studies and Contracts

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. 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.

Lalor will produce a zinc concentrate and a copper concentrate with gold and silver credits. Zinc concentrates are trucked to Hudbay's operations in Flin Flon where they are processed into refined zinc and sold to customers in North America. The key long-term assumptions for the sale of Lalor's zinc metal and zinc concentrate are summarized in Table 1-16. This Technical Report assumes zinc concentrate will be processed at the Flin Flon zinc plant from 2017-2021 and after that time Lalor's zinc concentrate will be sold to third party refineries.

TABLE 1-16: KEY LONG-TERM ZINC METAL AND ZINC CONCENTRATE ASSUMPTIONS

	Units	LT Total / Average
Zinc Concentrate Grade	%	51%
Moisture Content of Zinc Concentrate	% H_2O	9%
Zinc Concentrate Base Treatment Charge	US$/ tonne concentrate	$200
Zinc Concentrate Metal Price Basis	US$ / tonne Zinc metal	$2,204.6
Zinc Concentrate Escalator	%	6%
Zinc Concentrate De-escalator	%	3%
Zinc Concentrate Payability	%	85%
Zinc Concentrate Minimum Deduction	%	8%
Zinc Concentrate Freight Cost	C$/wmt	$118
Freight Allowance/Capture	US$/ wmt concentrate	$40
Zinc Metal Premium	US$/lb	$0.07
Zinc Metal Distribution Cost	US$/lb	$0.055

The copper concentrate produced at Lalor is sent to copper smelters in North America by rail. The key assumptions for the sale of Lalor's copper concentrate are summarized in Table 1-17 below:

TABLE 1-17: KEY LONG-TERM COPPER CONCENTRATE ASSUMPTIONS

	Units	LT Total / Average
Copper Grade in Copper Concentrate	% Cu	21%
Moisture Content of Copper Concentrate	% H_2O	9%
Copper Concentrate Base Treatment Charge	US$ / dry tonne con	$80
Copper Refining Charge	US $ / lb Cu	$0.08
Silver Refining Charge	US $ / oz Ag	$0.50
Gold Refining Charge	US$ / oz Au	$5.00
Copper Concentrate Freight Cost	C$ / wet tonne con	$213
Copper Payability	%	96.5%
Copper Minimum Deduction	%	1%
Gold Payability	%	96%
Silver Payability	%	90%

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.

1.15 Environmental Studies, Permitting and Social or Community Impact

Commencing in 2007, AECOM carried out the environmental baseline investigations needed to conclude an environmental impact assessment for the Lalor project, including all necessary terrestrial and aquatic field studies. This baseline work was utilized in the Lalor Paste Plant Notice of Alteration (NoA) which was submitted to Manitoba Sustainable Development in the fourth quarter of 2016 and was approved in January 2017. AECOM also conducted baseline work and studies which are summarized in the Anderson TIA expansion NoA submitted in the third quarter of 2016 to Manitoba Sustainable Development for approval.

Due to the extensive work completed by AECOM and other existing studies completed as part of Environmental Effects Monitoring (EEM) programs at the various operations in the Snow Lake area, it is contemplated that no additional baseline studies are necessary for potential future improvement projects. There is no present indication that future approvals will not be obtained to meet potential future construction schedules.

There are no known environmental concerns which could adversely affect Hudbay's ability to mine ore from Lalor mine. Because of its location in close proximity to the existing facilities in the Snow Lake area, Lalor was able to utilize existing infrastructure, services, and previously disturbed land that is 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 NoA for the expansion of Anderson TIA was prepared by AECOM utilizing geotechnical tailings dam designs from Hudbay's Engineer of Record; BGC Engineering Inc. As detailed in the Environmental Assessment of the Proposed TIA Expansion submitted as a NoA, the entire volume of tailings from Lalor LOM was to be stored in Anderson TIA (AECOM, 2016). This conceptual design did not discount the volume of tails that could be used for paste backfill at Lalor.

The existing Lalor mine Environment Act Licence (EAL) was obtained in the first quarter of 2014 and covers all facilities on the Lalor site, including sewage and mine wastewater treatment facilities and the pipelines which carry freshwater into the site and remove treated wastewater from it. The main permits required for the Lalor operation are presently valid licenses and permits for the Lalor mine, Stall concentrator, Anderson TIA, and New Britannia site. Applications for the Anderson TIA expansion NoA have been submitted for approval. Other upgrades and augmentation plans may require the submission of a NoA to an existing licensed operation but no new tailings impoundment area will be required. No federal permits are anticipated.

Presently the New Britannia site inclusive of the Birch Lake Tailings Management Facility (BLTMF), in Snow Lake, although currently in care and maintenance, has a current EAL and the seasonal discharge from BLTMF is regulated under the Metal Mining Effluent Regulations (MMER). Hudbay is currently in the process of applying for a new water withdrawal licence for this site which is anticipated to be obtained before potential operational needs. Potential future use of the New Britannia site will require the submission of a NoA in order to process material from the Lalor mine.

Prior to commercial production of ore at Lalor mine a mineral lease was applied for and obtained from Manitoba Mines Branch. The mineral lease grants the holder the exclusive right to mine minerals within the lease area.

In specific areas associated with proposed pipeline routes and future improvements to the existing Anderson TIA, surface leases will be required. Activities are currently underway to apply for and obtain the required surface leases. There is no indication that theses leases cannot be obtained in the time lines of the expansion project.

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 Ecodistrict and surrounding area. Snow Lake has a population of approximately 840 according to the 2006 data from Statistics Canada, with the majority of these residents employed at or supported by the mines located throughout the area. Many other Snow Lake residents are employed in the industries and services that support the region's mining operations.

Hudbay and AECOM have carried out public consultation, including meetings to inform local communities about the progress of development of the Lalor mine and expansion of Anderson TIA and environmental effects of these projects. The projects will continue to provide jobs for both Flin Flon and the Town of Snow Lake during construction of upgrades and continued operation of the mine. The additional feed from the mine will also help ensure the continued employment of Hudbay employees in the Flin Flon and Snow Lake areas. Since the economies of both communities are based on mining, opposition to the projects is seen as unlikely.

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, other current operations, and potential future projects.

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 Manitoba Mines and Minerals Act requires a closure plan and financial assurance for any advanced exploration or mining project. Manitoba accepted 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 Lalor mine, Manitoba approved the Closure Plan for the Lalor AEP and accepted financial assurance in the amount of $1.5 million. NoA applications for the paste plant, expansion of the Anderson TIA, and potential upgrades to the New Britannia site also will require the submission of updated closure plans and financial assurance.

1.16 Capital and Operating Cost

Capital and operating costs are estimated in constant 2017 Canadian dollars.

1.16.1 Capital Costs

The total development capital required to increase throughput at Lalor to the targeted 4,500 tpd is estimated to be C$117 million, as shown in Table 1-18, which includes approximately an 18% contingency. This capital is expected to be spent during 2017 and 2018.

TABLE 1-18: DEVELOPMENT CAPITAL COST SUMMARY

Development Capital	000 C$
Paste Backfill	67,786
Ore Handling Underground	3,250
Stall Mill Upgrades	45,870
Total Development Capital	**116,906**

The development capital costs were estimated internally by Hudbay with input from Golder Associates, Stantec Inc. and Boge & Boge Ltd.

The LOM sustaining capital costs are estimated to be C$220 million. The breakdown of the sustaining capital over the next 5 years and for the LOM is shown in Table 1-19.

TABLE 1-19: SUSTAINING CAPITAL COST SUMMARY

Sustaining Capital (000 C$)	2017	2018	2019	2020	2021		2017-LOM
Mine Capital and Development	17,927	18,381	13,442	13,849	10,635		86,771
Normal Capital	3,000	10,000	3,000	3,000	3,000		28,000
Replacement Equipment	11,629	14,739	13,390	11,004	9,776		91,518
Major Installations	3,640	5,924	1,461	982	835		13,803
Total Sustaining Capital	**36,196**	**49,044**	**31,293**	**28,835**	**24,246**		**220,092**

Normal capital includes C$7 million related to the expansion of the Anderson tailings facility and C$21 million related to the Stall concentrator. This sustaining capital estimate has made no consideration for the availability of used equipment from the 777 and Reed mines. When these mines eventually close, some equipment may be available for use at Lalor and may reduce the sustaining capital estimate above. No contingency has been included in the sustaining capital estimate.

Reclamation costs, salvage value and severance costs have not been considered in this report.

1.16.2 Operating Costs

Operating costs were developed by Hudbay based on a bottom-up approach and utilizing budget quotes from local suppliers, Manitoba operations experience, labor costs within the region and actual costs at Lalor.

The mine plus mill unit operating costs are estimated to be C$99.83/tonne mined over the LOM. The addition of cemented rock fill and paste backfill has increased the mine unit costs, but maximizes recovery of the mineral resource and results in lower capitalized costs than prior years due to less underground development. Table 1-20 summarizes the mine plus mill unit operating costs over the next 5 years and for the LOM.

TABLE 1-20: UNIT OPERATING COST SUMMARY

(C$/tonne mined)	2017	2018	2019	2020	2021		2017-LOM
Mine Development	26.53	15.07	14.87	13.64	13.98		17.12
Ore Extraction	20.28	27.19	33.60	35.59	34.75		30.50
Ore Removal	29.87	30.24	28.22	28.20	28.47		30.70
Total Mine Operating Costs	**76.67**	**72.49**	**76.68**	**77.43**	**77.20**		**78.32**
Mill Operating Costs[1]	**22.02**	**20.14**	**20.12**	**20.19**	**20.12**		**21.51**
Total Mine + Mill Operating Costs	**98.70**	**92.63**	**96.80**	**97.63**	**97.32**		**99.83**

[1] Milling costs include concentrate haulage to Flin Flon

The total C1 cash costs and sustaining cash costs (net of by-product credits) per pound of zinc over the LOM and over the next 5 years are shown in Table 1-21. C1 cash costs include on-site and off-site costs. Sustaining cash costs include C1 costs plus sustaining capital.

TABLE 1-21: CASH COSTS (NET OF BY-PRODUCT CREDITS)

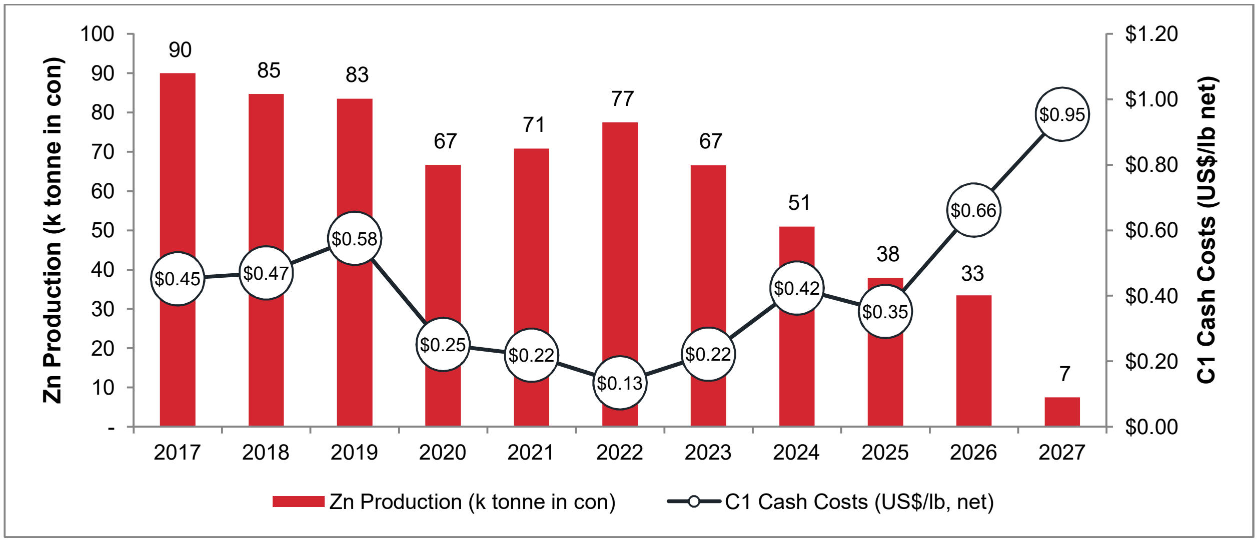
Cash Costs (Net of By-Product Credits[1])	Units	Next 5 Years	LOM
C1 Cash Costs	US$ / lb Zn in con	$0.41	$0.37
C1 Cash Costs + Sustaining Capex	US$ / lb Zn in con	$0.57	$0.50

[1] By-product credits are calculated using the following assumptions: copper price per pound - US$2.60 in 2017, US$2.75 in 2018, US$3.00 in 2019 to 2020 and long-term; gold price per ounce - US$1,300 in 2017 to 2020 and US$1,260 long-term; silver price per ounce - US$18.00 in 2017 to 2020 and long-term; CAD/USD exchange rate - 1.35 in 2017, 1.25 in 2018, 1.20 in 2019, 1.15 in 2020 and 1.10 long-term.

Lalor's annual zinc production (contained zinc in concentrate) and C1 cash costs (net of by-products) are shown below in Figure 1-2. Over the first 5 years, annual production is expected to average 79 thousand tonnes of zinc at an average C1 cash cost of US$0.41/lb. Over the 10.5 year

LOM, annual production is expected to average 64 thousand tonnes of zinc at an average C1 cash cost of US$0.37/lb. Lower C1 cash costs from years 2020 to 2023 are a result of mining the copper-gold (Zone 27).

FIGURE 1-2: LALOR ANNUAL ZINC PRODUCTION AND C1 CASH COSTS



1.17 Economic Analysis

Hudbay is a producing issuer and has excluded information required by Item 22 of Form 43-101F1 as the updated mine plan does not represent a material increase of Hudbay's current production.

1.18 Other Relevant Data and Information

1.18.1 Gold Bulk Sample Program

In the fourth quarter of 2016, Hudbay developed 37 drift rounds at Lalor to assess the continuity and variability of non-contact gold mineralization within discrete areas of zones 21 and 25. Approximately 10,000 tonnes of bulk sample material was mined and hauled to surface. The material was primary crushed and processed through a sample tower to collect a representative subsample of each development round. The integrity of the material was maintained at all times through a rigorous chain of custody process. The mined material, stored on surface, is available for milling pending the potential economic viability of refurbishing the New Britannia gold mill in Snow Lake by Hudbay.

The preliminary results indicate that the gold grades from the bulk sample program are as expected with minor variations when compared to those modeled based on diamond drill data. The bulk sample program has increased the confidence and the understating of the gold zones and gold mineralization at Lalor.

Following full assessment of the 2016 bulk sample data it is intended to collect additional subsamples in 2017 from other areas of gold zones 21 and 25 as well as from other previously untested gold zones for confirmation purposes.

1.18.2 Taxes and Royalties

Applicable Tax Rates

The Lalor mine is not directly taxable as Hudbay pays provincial and federal taxes on a legal entity basis. The combined federal and provincial tax rates are assumed to be approximately 27% for the LOM and Hudbay has approximately C$750 million in tax pools that can be used to offset future income taxes for federal and provincial purposes. Hudbay's mining operations in Manitoba are also subject to the Manitoba Mining Tax. The Manitoba Mining Tax is not applied to a new mining project until the original capital expenditures are recovered.

Royalties

There are no royalties applicable to Lalor.

1.19 Conclusions and Recommendations

1.19.1 Conclusions

The Lalor mine operation has been mining ore since August 2012. Since then the mine has operated uninterrupted and been in a continuous production ramp-up cycle, achieving the highest annual tonnage of approximately 1.1 million tonnes in 2016, with complementary throughput at the Stall concentrator. The production ramp-up is planned to continue in 2017 to reach a steady state of 4,500 tpd by the first quarter of 2018.

The production increase of 50%, compared to current production, is supported by an underground ore handling circuit capable of 4,500 tpd, transitioning to more bulk mining methods (65% of reserves) with additional mining fronts and design changes to improve mining efficiencies, developing ore passes and transfer raises to reduce truck haulage cycle times from the upper potions of the mine and commissioning of a paste plant backfill plant in the first quarter of 2018. Autonomous operation of a Load Haul Dump loader underground is currently being trialed from surface by tele-remote monitoring with changes to standard designs to allow isolation of autonomous areas and buffer storage for in between shift mucking.

The increase in production to 4,500 tpd at Lalor is complemented by the Stall concentrator expansion to 4,500 tpd, which is currently underway and is expected to be commissioned in the third quarter of 2018.

The mineral resources, as of September 30, 2016, are estimated as base metal lenses or gold zones based on geological and mineralization properties. The Hudbay validation process and third party review confirmed the resource block model is interpolated using industry accepted modelling techniques and classified in accordance with the 2014 CIM Definition Standards – For Mineral Resources and Mineral Reserves.

A mine reconciliation of the mined out areas compared to the ore reported at the concentrator was very close on the precious metals and a slight conservatism of the zinc and copper grades might be evident. This conservatism of the base metals is likely due to over constraining the high grade samples to 20 m as part of the high yield restriction step.

The mineral resources stated with a metal equivalency cut-offs provide for economic extraction of reserves from stated resources.

The mineral reserves, as of January 1, 2017, are based on a LOM plan that generated a mining inventory based on stope geometry parameters with appropriate dilution and recovery factors The conversion of resources to reserves is based on the LOM plan and NSR cut-offs that primarily focussed on capturing base metal resources for processing at the Stall concentrator. The secondary focus was to capture gold zone resources when in contact with or close proximity to base metal resources. In areas where a large separation existed between base metal and gold lenses, mining blocks were evaluated for economic stope mining shapes. When a non-economic shape was generated in a first pass, a second pass was evaluated for only base metal lenses and if an economic shape was generated the gold zone portion was removed. However, due to this larger separation, majority of these isolated gold lenses could have been evaluated independently of the base metal lenses and could potentially provide feed to a gold processing facility. Below approximately the 950 m level no attempt was made to generate an economic stope mining shape for gold zones 25 and 26 as the separation distance became too large. The author's opinion is that these resources are potentially better suited for a gold processing facility and should be re-evaluated when Hudbay has a better understanding of their New Britannia gold mill and Birch Tailings Impoundment Area in Snow Lake.

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 production and compilation of this technical report was supported by the capable and professional management and staff at Hudbay. The supervision, revision and approval of the assembly of this Technical Report is by the QP Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

1.19.2 Recommendations

Hudbay recommends the following:

- Investigate the impact of under assaying of high grade standards at Hudbay's Flin Flon laboratory and whether this in turn affects the high grade Lalor samples submitted and has potentially led to an underestimations of gold in the resource estimate, since the proportion of samples assayed at the laboratory was approximately 80% of the total samples assayed between 2012 and 2016.
- Investigate the high yield restriction parameters of the high grade base metal samples, and consider whether the restriction distance is suitable or were they over constrained, based on the conservatism noted in the mine reconciliation for zinc and copper.
- Pursue the option of a temporary paste backfill plant to utilize the boreholes from surface prior to commissioning of the permanent plant in the first quarter of 2018. This option provides assurance to achieve the ramp-up in production and is another source of backfill rather than relying on waste development.
- Due to the approximate 6 month timing offset of the production ramp-up at Lalor to 4,500 tpd and the Stall concentrator expansion to 4,500 tpd, Hudbay should pursue transporting of ore from Lalor to their Flin Flon concentrator for earlier processing.
- Finalize the evaluation of the gold bulk sample program conducted in the fourth quarter of 2016 and since Hudbay owns a sample tower consider collecting additional subsamples from other areas of gold zones 21 and 25 as well as from other previously untested gold zones for confirmation purposes.
- Hudbay owns the New Britannia mill, a gold leach plant on care and maintenance, in Snow Lake. Hudbay should continue to assess the feasibility of processing a portion of the material mined from the gold zone and copper-gold zone at Lalor at the New Britannia mill at a rate of 1,500 tpd starting in 2019. When combined with the processing capacity of the Stall concentrator, this would enable an aggregate throughput rate of up to 6,000 tpd and utilize the full capacity of the Lalor mine shaft.

2 INTRODUCTION AND 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 to provide an updated mine plan that contemplates 4,500 tpd of base metal, gold and copper-gold zone ore to Stall concentrator.

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, which also consists of an ore concentrator and zinc plant, and the Reed mine, which is located approximately 120 km by road southeast of Flin Flon. Operations near Snow Lake include the Lalor underground mine.

The Lalor mine consists of 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 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 3,000 to 3,500 tpd and is ramping up production to 4,500 tpd by 2018. The production ramp-up is supported by the underground ore handling circuit capable of 4,500 tpd, further transitioning to more bulk longhole open stoping mining methods with additional mining fronts and design changes to improve mining efficiencies, developing ore passes and transfer raises to reduce truck haulage cycle times from the upper potions of the mine and commissioning of a paste backfill plant in the first quarter of 2018. Autonomous operation of Load Haul Dump loaders underground are currently being trialed from surface by tele-remote monitoring with changes to standard designs to allow isolation of autonomous areas and buffer storage for in between shift mucking.

The increase in production to 4,500 tpd at Lalor is complemented by the Stall concentrator expansion to 4,500 tpd, which is currently underway and is expected to be complete in the second quarter of 2018.

This Technical Report represents an update of information pertaining to the Lalor mine previously disclosed in a Pre-Feasibility Study technical report dated March 2012. At the time of the previous technical report, Lalor was a development project, but the property has since commenced production starting in August 2012.

Major works completed at Lalor include the construction of a concrete-lined 6.9 m diameter production shaft capable of 6,000 tpd (which is the primary access), the development of a secondary ramp access from the surface through the Chisel North mine shaft, and the establishment of two shaft stations at 835 mL and 910 mL. The facilities at the Lalor mine have mined and processed more than 3 million tonnes of ore as of December 31, 2016.

2.1 Qualified Person (QP) and Site Visit

This Technical Report has been prepared in accordance with National Instrument Form NI 43-101 F1. The QP who supervised the preparation of this Technical Report is Robert Carter, P. Eng., Lalor Mine Manager at Hudbay's Manitoba Business Unit.

Robert Carter 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. Carter is directly involved with the Lalor mine on a permanent basis because of his role as mine manager and was involved in the early exploration, project evaluation and pre-feasibility study. Mr. Carter visits and inspects the Lalor operation on a routine basis and has overseen the mineral resource and reserve estimation process. Mr. Carter has acted as the Qualified Person for the overall Technical Report. Prior to the publication of this Technical Report, Mr. Carter's last site visit was on March 29, 2017.

2.2 Sources of Information

Geology and mineral resources sources of information include: core drilling and sampling data, underground development and mapping, assay and geochemistry analysis.

Mineral reserve sources of information are 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.

Metallurgy, processing and economic sources of information are the actual operating data since production and concentrating commenced in 2012 and operating budget estimates.

Multiple participants have worked on this Technical Report. Discussions were held with personnel from Hudbay Manitoba Business Unit (MBU) and Hudbay Toronto:

- Tony Scheres, Manager of Technical Services and Business Development, Lalor mine
- Sarah Bernauer, Chief Geologist, Lalor mine
- Jennifer Pakula, Chief Engineer, Lalor mine
- Jason Lanteigne, Mine Engineer, Lalor mine
- Doug Salahub, Mines Analyst, MBU
- Johan Krebs, Geologist, Lalor mine

- Jay Cooper, Superintendent Environment, MBU
- Jarid Medina, Stall Mill Superintendent, Stall concentrator
- Karl Hoover, Process Manager, Snow Lake Projects
- Marc-Andre Brulotte, Manager Project Evaluation, Toronto
- Mark Gupta, Manager Corporate Development, Toronto
- Juan Carlos, Ordóñez Calderón, Exploration Geochemist, Toronto
- Matthew Holden, Senior Geophysicists, Toronto

Table 2-1 lists the participants by section.

TABLE 2-1: CONTRIBUTORS AND RESPONSIBLE PARTIES FOR THIS REPORT

Section	Description	Participants	Responsible QP
1	Summary	Robert Carter	Robert Carter
2	Introduction	Robert Carter	Robert Carter
3	Reliance on Other Experts	Robert Carter	Robert Carter
4	Property Description and Location	Johan Krebs, Sarah Bernauer	Robert Carter
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Johan Krebs	Robert Carter
6	History	Johan Krebs	Robert Carter
7	Geological Setting and Mineralization	Johan Krebs	Robert Carter
8	Deposit Type	Johan Krebs	Robert Carter
9	Exploration	Matthew Holden	Robert Carter
10	Drilling	Johan Krebs	Robert Carter
11	Sample Preparation Analyses and Security	Johan Krebs, Sarah Krebs	Robert Carter
12	Data Verification	Juan Carlos, Ordóñez Calderón	Robert Carter
13	Mineral Processing and Metallurgical Testing	Jarid Medina, Karl Hoover	Robert Carter
14	Mineral Resource Estimates	Marc-Andre Brulotte, Robert Carter	Robert Carter
15	Mineral Reserve Estimates	Jason Lanteigne, Robert Carter	Robert Carter
16	Mining Methods	Jennifer Pakula, Jason Lanteigne	Robert Carter
17	Recovery Methods	Jarid Medina	Robert Carter
18	Project Infrastructure	Tony Scheres, Jarid Medina, Jay Cooper, Robert Carter	Robert Carter
19	Market Studies and Contracts	Mark Gupta	Robert Carter

Section	Description	Participants	Responsible QP
20	Environmental Studies, Permitting, and Social or Community Impact	Jay Cooper	Robert Carter
21	Capital and Operating Costs	Mark Gupta, Doug Salahub	Robert Carter
22	Economic Analysis	-	Robert Carter
23	Adjacent Properties	-	Robert Carter
24	Other Relevant Data and Information	Robert Carter	Robert Carter
25	Interpretation and Conclusions	Robert Carter	Robert Carter
26	Recommendations	Robert Carter	Robert Carter
27	References	Robert Carter	Robert Carter

2.3 Unit Abbreviations

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

TABLE 2-2: UNIT ABBREVIATIONS

Abbreviation	Term	Abbreviation	Term
$C or C$	Canadian 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
g	gram	ml	millilitres
g/t	gram per metric tonne	MVA	Mega volt amp
Ga	billion years	MW	Megawatt
gpm	gallon per minute	nT	nanotesla
Ha	hectare	oz	Troy ounces
HP or hp	Horsepower	ppm	parts per million
hr	hour	psi	Pounds per square inch
kg	kilogram	QP	Qualified Person
km	kilometre	t	metric tonne
km/hr	kilometre per hour	tpd	metric tonnes per day
kV	kilovolt	US GPM	United States gallon per minute
kW	kilowatt	US$ or $US	United States dollar
L/min	litres per minute	V	Volt
lb	pound (unit of weight)	wmt	Wet metric tonne
m	Metre	Zn Eq	Zinc equivalent

2.4 Acronyms and Abbreviations

Abbreviations of company names and other notable terms used in the report are as shown in Table 2-3.

TABLE 2-3: ACRONYMS AND ABBREVIATIONS

Abbreviation	Term
3D	Three-Dimensional
AAS	Atomic Absorption Spectrometry
ACME	ACME Analytical Laboratories Ltd.
acQuire	Drillhole Database Management Program
ADIST	Average distance of composites
AEP	Advanced Exploration Project
Ag	Silver
ANFO	Ammonium nitrate
As	Arsenic
ASL	Above Sea Level
Au	Gold
Au Eq	Gold Equivalency
AV	average
BLTMF	Birch Lake Tailings Management Facility
BQ	BQ drill core size 36.4mm
BV	Bureau Veritas
CBV	Certified Best Value
CDIST	Closest distance of a composite
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP/CIL	Carbon-in-pulp / Carbon-in-leach
CI	Confidence interval
CRF	Cemented waste rock backfill
CRM	Certified Reference Materials
Cu	Copper
CV	Coefficient of Variation
DH	Drill hole
DE	Data entry
DGPS	Differential Global Positioning System
EAL	Environmental Act Licence
EDA	Exploratory data analysis
EDM	Electronic distance measurement
EEM	Environmental Effects Monitoring
EM	Electromagnetic
ES	Emission spectrograph
Fe	Iron
FOB	Fine ore bin
GPSS	Global Positioning Satellite System
HCl	Hydrogen chloride
HNO_3	Nitric acid
Hudbay	Collectively all Hudbay Minerals Inc. subsidiaries and business groups
ICP	Inductively Coupled Plasma
LCT	Lock cycle test
IDW	Inverse Distance Squared Weighted
KSTD	Standard deviation of kriging
LHD	Load Haul and Dump
LIMS	Information management system
LOD	Limit of detection
LOM	Life of Mine
MBU	Manitoba Business Unit
MCCN	Mathias Colomb First Nation
MDIST	Maximum distance of a composite
MIBC	Methyl isobutyl carbinol
ML	Mineral Lease

Abbreviation	Term
MS	Mass Spectrometry
NCOMP	Number of composites
NoA	Notice of Alteration
NI	National Instrument
NN	Nearest Neighbour
NSR	Net smelter return
NQ	NQ drill core size 47.6mm
NSS	Near solid sulphide
OIC	Order In Council
OES	Optical Emission Spectrometry
OK	Ordinary Kriging
OLS	Ordinary least square
OREAS	Ore Research and Exploration
P. Eng.	Professional Engineer
Pb	Lead
PR	Provincial Road
QAQC	Quality Assurance and Quality Control
QP	Qualified Person
RCI	Resource classification index

Abbreviation	Term
RMA	Reduced-to-Major-Axis
Reflex	Reflex E-Z Shot
ROM	Run of mine
RSLOP	Regression slope
R squared or R^2	Coefficient of determination
RTK	Real Time Kinematic
SG	Specific Gravity
SS	Solid sulphide
TIA	Tailings Impoundment Area
RE	Relative Error
TMI	Total Magnetic Intensity
UCS	Unconfined compressive strength
URF	Unconsolidated waste rock backfill
UTM	Universal Transverse Mercator
VMS	Volcanogenic Massive Sulphide
Zn	Zinc
Zn Eq	Zinc Equivalency

3 RELIANCE ON OTHER EXPERTS

Standard professional procedures were followed in preparing the contents of this Technical Report. Data used in this report has been verified where possible and the author 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,
- Assumptions, conditions, and qualifications as set forth in this Technical Report

For the purpose of the report, the author has relied on title and property ownership information provided by Hudbay Manitoba Business Unit, Land Manager, Janelle Toffan on March 13, 2017.

4 PROPERTY DESCRIPTION AND LOCATION

The Lalor mine is located approximately 208 km by road east of Flin Flon and 16 km by road west of Snow Lake in the Province of Manitoba at 54°52'N latitude, 100°08'W longitude and 303 m ASL (Figure 4-1).

Hudbay owns 100% interest of the Lalor property through one (1) Mineral Lease (ML) and eight (8) Order in Council (OIC) Leases to the south (Figure 4-2).

4.1 Land Tenure

Mineral and Order in Council Leases are issued and administered by the Province of Manitoba Mines Branch. Annual payments of $10.50/ha, with a $193 minimum, for producing leases is due for each lease over the 21-year term.

ML334:

- ML334 was issued in 2012 and replaces mineral claims CB5361, CB10605, CB10606, CB10607 and CB10608 (Figure 4-2).
- Covers an area of approximately 796 hectares and encompasses the majority of the Lalor deposit.
- The required amount of expenditures, on approved work, within the ML334 area shall be no less than $1,250/ha during the 21-year term if a term renewal will be required. No later than 60 days after each of the 5th, 10th, 15th and 21st anniversaries of the issuance of the Mineral Lease, the Mineral Lessee shall submit a report to the Director of Mines setting out all work carried out on the mineral lease area for the applicable period.

OIC LEASES (M5778, M5779, M5780, M5781, M7278, M7279, M7280 and M7281):

- Covers an area of approximately 152 hectares. The southerly up-plunge extension of the mineralization lies within these OIC Leases.
- In addition to the annual payments mentioned above, an annual tax of $10.00/OIC is due by December 31st of each year.
- There are no work commitments due on OIC Leases.

Mineral and Order in Council Lease status for the Lalor property is shown in Table 4-1.

TABLE 4-1: MINERAL AND OIC LEASE PROPERTIES

Lease No.	Lease Name	Holder	Hectares	Annual Fees (excludes annual $10/OIC tax)	Anniversary Date	Termination Date
M5778	OX NO. 153	Hudbay	15.90	193.00	Apr 08, 2017	Apr 08, 2023
M5779	OX NO. 154	Hudbay	17.99	193.00	Apr 08, 2017	Apr 08, 2023
M5780	OX NO. 155	Hudbay	18.25	199.50	Apr 08, 2017	Apr 08, 2023
M5781	OX NO. 156	Hudbay	20.20	220.50	Apr 08, 2017	Apr 08, 2023
M7278	OX NO. 143	Hudbay	21.70	231.00	Sep 06, 2017	Sep 06, 2023
M7279	OX NO. 144	Hudbay	20.55	220.50	Sep 06, 2017	Sep 06, 2023
M7280	OX NO. 145	Hudbay	21.60	231.00	Sep 06, 2017	Sep 06, 2023
M7281	OX NO. 146	Hudbay	14.84	193.00	Sep 06, 2017	Sep 06, 2023
ML334		Hudbay	795.55	8,358.00	Mar 29, 2018	Mar 29, 2033
Totals			***946.58***	***$ 10,039.50***		

FIGURE 4-1: LOCATION MAP OF HUDBAY MINES

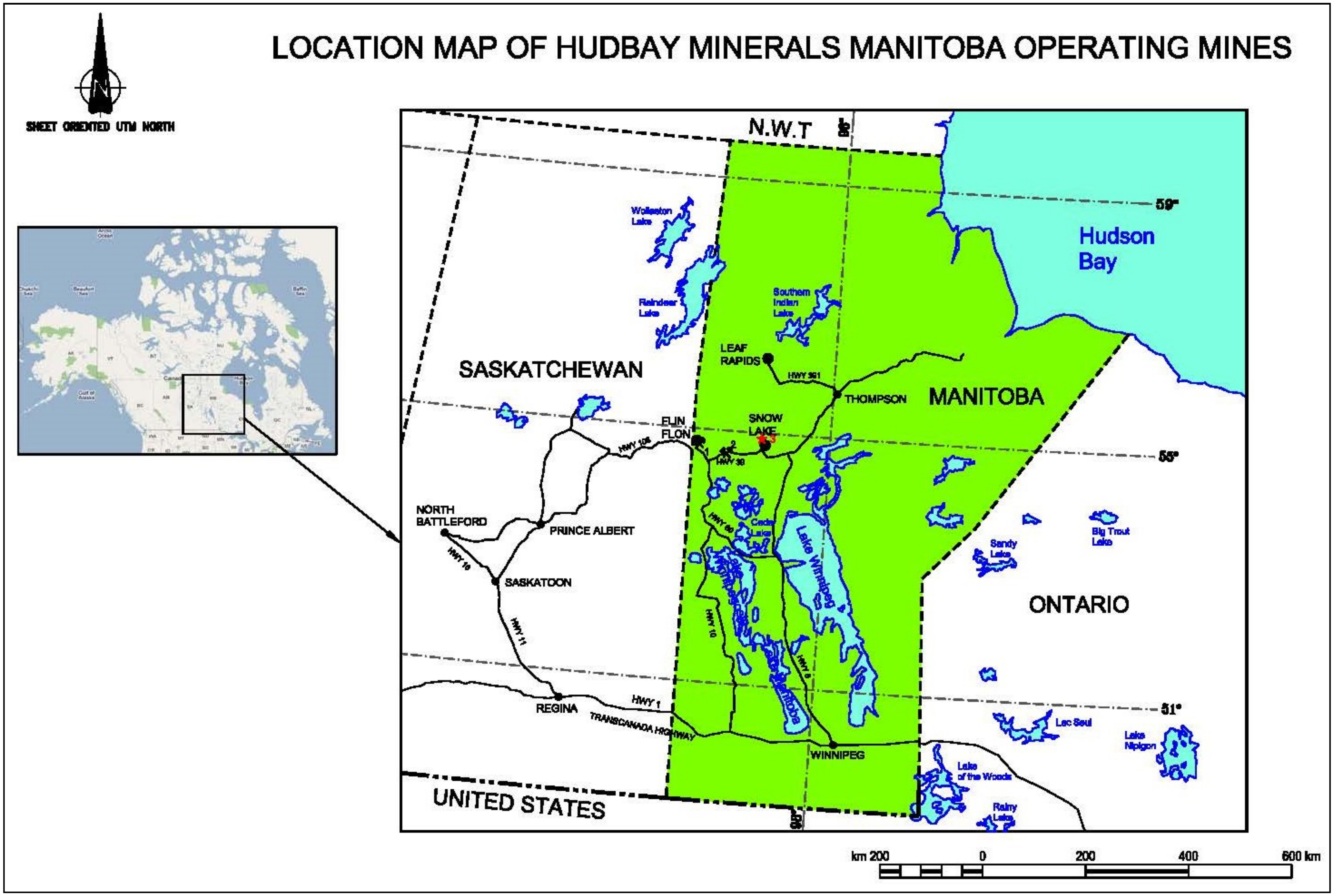
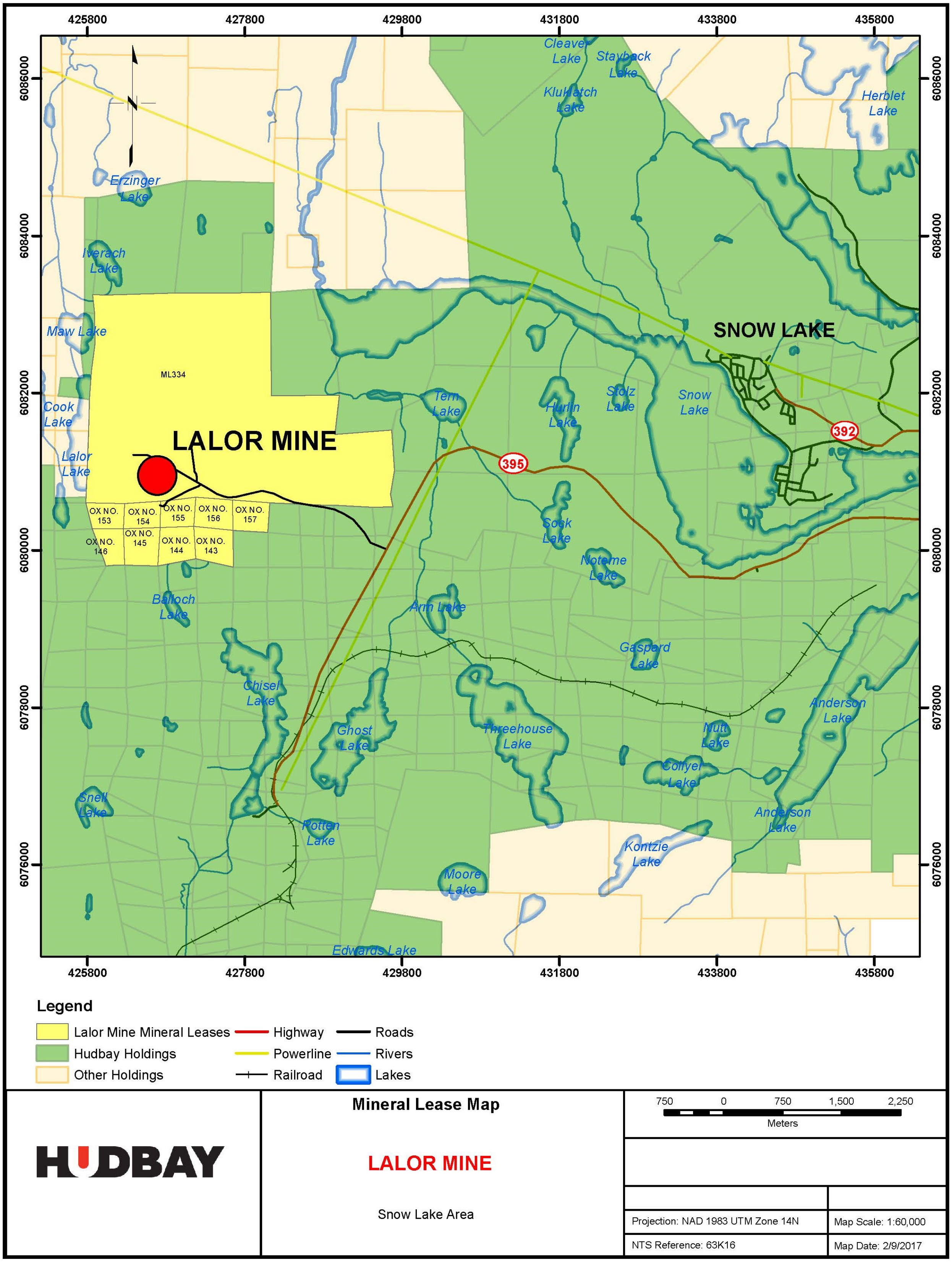


FIGURE 4-2: MINERAL CLAIMS AND LEASE MAP



4.2 Land Use Permitting

4.2.1 General Permits

General Permits are issued and administered by the Province of Manitoba Crown Lands and Property Agency. 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.

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

GP59093:

- **Specific Use:** 4.0 km x 5 m wide all weather access road (to accommodate a 4 km 25 kV transmission line, a 4 km discharge line and a 4 km fresh water line), a 200 m x 200 m parking lot and an additional access road in PT. NW 9-68-18W (0.25 km x 30 m).
- The annual fee for the All Weather Road is set at $100 plus one additional dollar for every kilometre of road. The annual fee for the Commercial Lot (parking) for one acre or less is set at $210 plus an additional $10 for each additional acre or portion of an acre.
- Annual Permit Renewal Fee is set at $10/permit.

GP63483:

- **Specific Use:** Mine Site
- The annual fee for Commercial Lot (mine site) is set at $2/acre or portion of an acre.
- Annual Permit Renewal Fee is set at $10/permit.

4.2.2 Quarry Lease

Quarry Leases are issued and administered by the Province of Manitoba Mines Branch.

One Quarry Lease is held by Hudbay and is required to carry on mining activities at Lalor:

- QL-1928 was issued November 29, 2007 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 2017 and provided this lease is still required it can be renewed for another 10-year term.
- The annual fee for a Quarry Lease is set at $27/ha, or fraction thereof, plus royalty and rehabilitation levies on extractions as prescribed by regulations.

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

TABLE 4-2: GENERAL PERMITS AND QUARRY LEASE

Permit / Quarry Number	Holder	Hectares	Issue Dates	Annual Fees (Excludes GST)	Anniversary Date
GP59093	Hudbay	4.05	Dec 31, 2007	415.00	Dec 31, 2017
GP63483	Hudbay	159.37	Jun 10, 2010	798.00	Dec 31, 2017
QL-1928	Hudbay	11.00	Nov 26, 2007	297.00	Nov 26, 2017
Total		***174.42***		***$ 1,510.00***	

Hudbay holds the exclusive right to the minerals, other than quarry minerals, and the mineral access rights required for the purpose of working the lands, mining, and producing minerals from the Lalor mine. Surface tenure, currently necessary to accommodate buildings and/or structures, required for the efficient and economical performance of the mining operations has been applied for and approved.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Lalor deposit is located approximately 208 km by road east of Flin Flon and 16 km by road west of the community of Snow Lake, Manitoba. Access to the deposit is from Provincial Road (PR) #395, a gravel road off PR #392, which joins the town of Snow Lake and PR #39 (Figure 5-1). From PR #395 there is an all weather permanent road into the mine site.

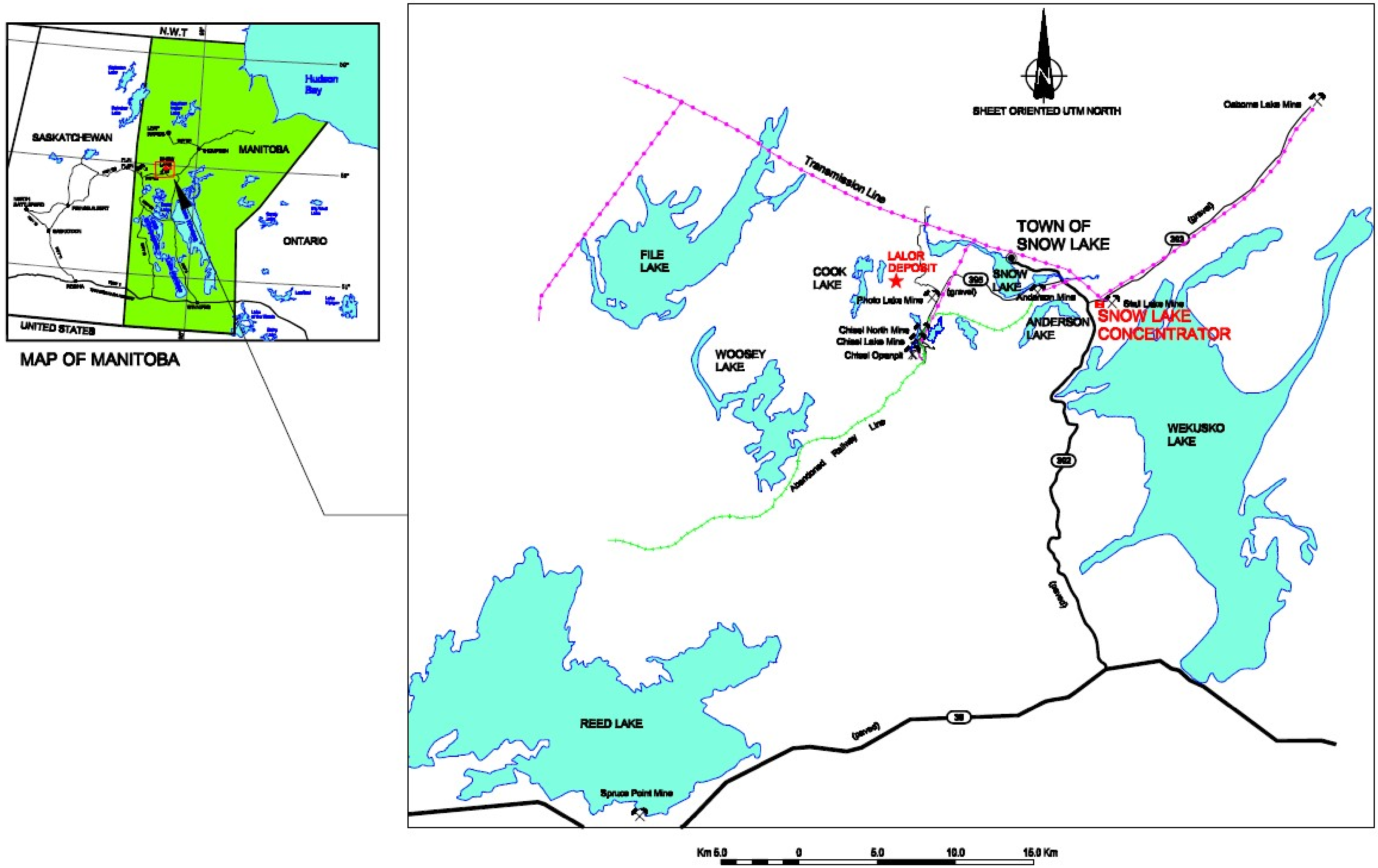
5.2 Climate

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 the Lalor deposit. 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, 35% as snow. Since 1960, extreme monthly precipitations have been zero to a high of 18.11 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, located approximately 16 km from Lalor. 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, 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, ambulance, 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+) are Flin Flon (208 km), The Pas (219 km) and Thompson (260 km), all accessible by paved highway. There is a 1,100 m by 20 m unserviced gravel municipal airstrip located approximately 30 km from Lalor 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 191 km from Lalor. The nearest international airport is located in Winnipeg, approximately 700 km from Snow Lake.

There is no rail in the immediate area of Lalor or Snow Lake. The nearest rail access is at Wekusko siding, approximately 65 km southeast of Lalor. 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.

5.4 Infrastructure

Hudbay operates a zinc metallurgical plant in Flin Flon, Manitoba, approximately 215 km from Lalor. Present capacity is 115,000 tpa refined zinc.

Hudbay operates an ore concentrator approximately 16 km from Lalor. The mill is currently operating seven days per week at 3,000 to 3,500 tonnes per day, processing ore from the Lalor mine. The mill has two circuits, with design capacities of 909 tpd and 2,182 tpd.

The concentrator has two flotation circuits producing a zinc concentrate and a copper concentrate. The tailings are deposited in the Anderson Tailings Impoundment Area. Concentrates are hauled by truck to Hudbay metallurgical facilities in Flin Flon.

General area infrastructure includes provincial roads and 115 kV Manitoba Hydro grid power to within four kilometres of Lalor, and Manitoba Telecom 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 914 m gravel airstrip to provide emergency medical evacuation.

Lalor is located 3.5 km from the Hudbay Chisel North mine. Chisel North infrastructure includes 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, plus mine buildings including offices and a change house. These facilities are used for geology 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, additional change house space, etc. The permitted Hudbay Anderson TIA, located approximately 12 km from Lalor, is used for tailings disposal.

5.5 Physiography

The deposit is located in the Boreal Shield Ecozone, the largest ecozone in Canada, extending as a broad inverted arch from northern Saskatchewan east to Newfoundland. The area of Lalor and surrounding water bodies (Snow, File, Woosey, Anderson and Wekusko lakes) are 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.

The property area is approximately 300 m ASL, with depressional lowlands, and has gentle relief that rarely exceeds 10 m, consisting of ridged to hummocky sloping rocks.

6 HISTORY

The Snow Lake area has a long exploration and mining history. The Lalor deposit was discovered in 2007.

6.1 Exploration in the Chisel Basin Area

Exploration in the Chisel Basin has been active since 1955. The Chisel Basin area has hosted three producing mines; namely, Chisel Lake, Chisel Open Pit and Chisel North. All three mines have 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 a band of conductive mineralization between 781.74 m and 826.87 m (45.13 m). Assay results include 0.30% Cu and 7.62% Zn over the 45.13 m including 0.19% Cu and 17.26% Zn over 16.45 m. Drilling at Lalor has been continuous since the discovery of mineralization on the property.

6.2 Historical Mining in the Snow Lake Area

The Snow Lake area has had an active mining history for more than 50 years. Hudbay has played an integral part in this history since the late 1950s by operating nine mines in the area including Photo Lake, Rod, Chisel Lake Chisel North and Chisel Open Pit, Stall Lake, Osborne Lake, Spruce Point, Ghost Lake, and Anderson Lake.

The Stall concentrator was commissioned in 1979 and operated continuously until shutdown in early 1993 as a result of the depletion of the Chisel Open Pit and Stall Lake mines. The concentrator was reopened in 1994 to process ore from the Photo Lake Mine and continued to process ore from the Chisel North mine until February 2009. With the reopening of Chisel North in 2010, the concentrator reopened and has remained open and is today the processing facility for all ore produced at Lalor.

6.3 Lalor Mine Production

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 2016.

TABLE 6-1: LALOR MINE ACTUAL PRODUCTION

Lalor Mine Actual Production						
	Units	**2016**	**2015**	**2014**	**2013**	**2012**
Ore Mined	tonnes	1,086,362	934,277	551,883	400,590	72,293
Gold	g/t	2.24	2.53	2.29	1.21	1.67
Silver	g/t	21.63	21.38	23.83	19.39	19.29
Copper	%	0.62	0.71	0.88	0.84	0.63
Zinc	%	7.01	8.18	8.52	9.44	11.83

Notes: Lalor ore production in 2014 includes partial production from the ventilation shaft, which began production in August 2012. Lalor ore production in each of 2012 and 2013 was from the ventilation shaft.

7 GEOLOGICAL SETTING AND MINERALIZATION

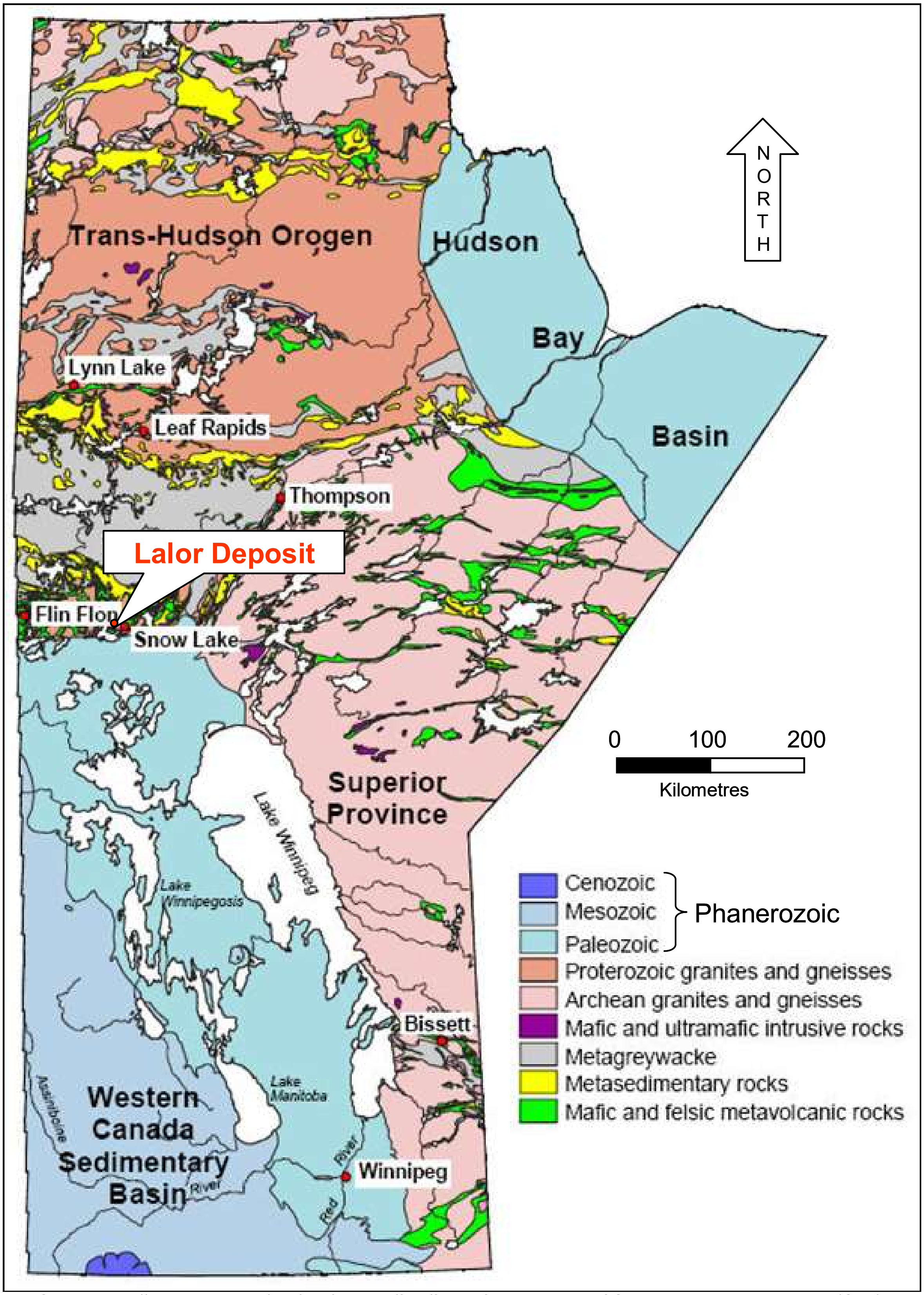
7.1 Regional Geology

The Lalor property lies in the eastern (Snow Lake) portion of the Paleoproterozoic Flin Flon Greenstone Belt (Figure 7-1) and is 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 ago, or giga-annums) tectonostratigraphic assemblages including juvenile arc, back-arc, ocean-floor and ocean-island and evolved volcanic arc assemblages that were amalgamated to form an accretionary collage (named the Amisk Collage) prior to the emplacement of voluminous intermediate to granitoid plutons and generally subsequent deformation (Syme et al., 1998). The volcanic assemblages consist of mafic to felsic volcanic rocks with intercalated volcanogenic sedimentary rocks. The younger plutons and coeval successor arc volcanics, volcaniclastic, and sedimentary successor basin rocks include the older, largely marine turbidites of the Burntwood Group and the terrestrial metasedimentary sequences of the Missi Group.

The Flin Flon belt is in fault and / or gradational contact with the Kisseynew Domain metasedimentary gneisses to the north and is unconformably overlain by the Phanerozoic cover of sandstone and dolostones to the south (Figure 7-2). Regional metamorphism at 1.82 to 1.81 Ga formed mineral assemblages in the Flin Flon belt that range from prehnite-pumpellyite to middle amphibolite facies in the east and upper amphibolite facies in the north and west (David and Machado, 1996; Froese and Moore, 1980; Syme et al., 1998).

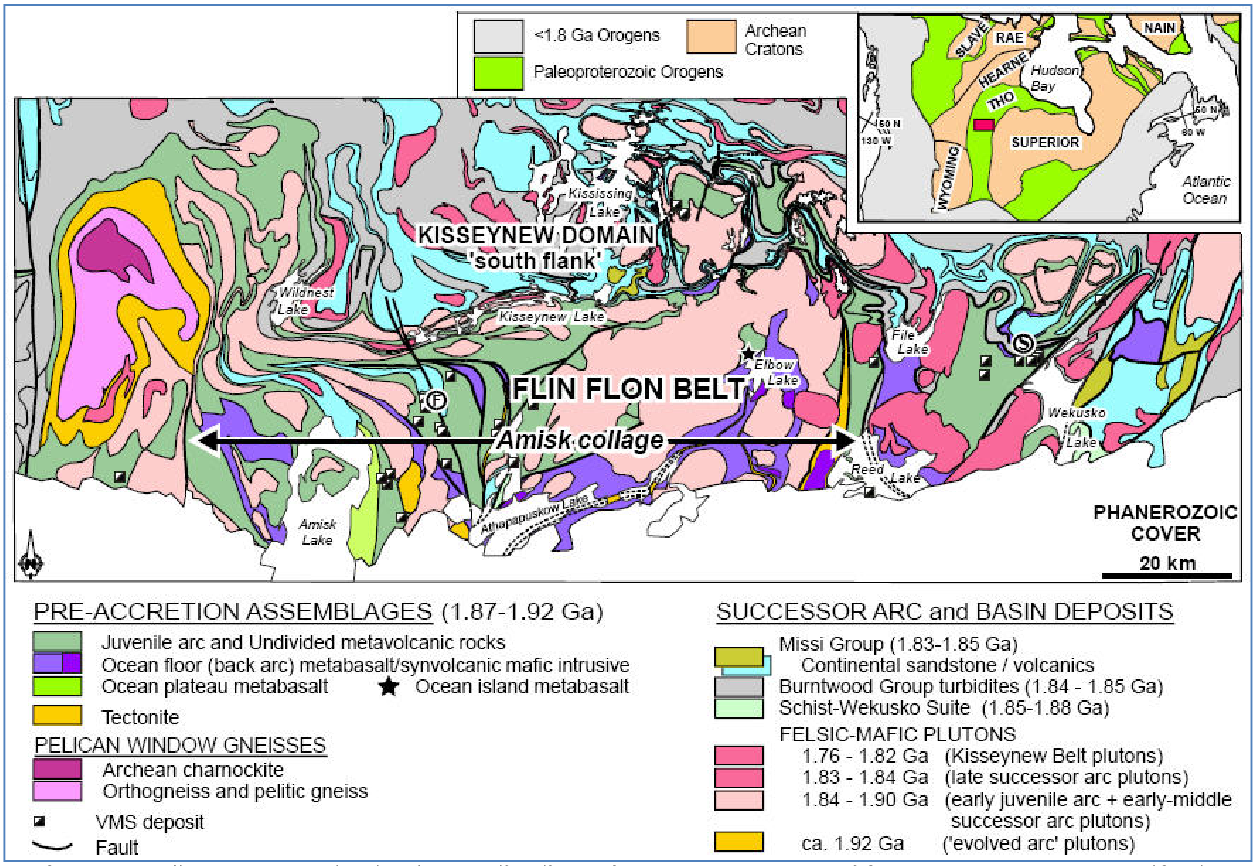
The Snow Lake portion of the Flin Flon belt is dominated by fold-thrust style tectonics that is atypical of western and central portions of the belt. It is a south-verging, northeast dipping imbricate that was thrust over the previously amalgamated collage of oceanic and arc rocks to the west (Bailes and Galley, 1999). The thrust package of the Snow Lake area has been modified by 1.82 to 1.81 Ga regional metamorphism of lower to middle almandine-amphibolite facies mineral assemblages (David and Machado, 1996; Froese and Moore, 1980).

FIGURE 7-1: GEOLOGY OF MANITOBA



Source: http://www.gov.mb.ca/iem/geo/exp-sup/files/fig1.pdf, Department of Growth, Enterprise and Trade (GET)

FIGURE 7-2: GEOLOGY OF THE FLIN FLON GREENSTONE BELT, MANITOBA



Source: http://www.gov.mb.ca/iem/geo/exp-sup/files/fig6.pdf. Manitoba Department of Growth, Enterprise and Trade (GET)

Intrusions in the belt are divided into pre-, syn- and post tectonic varieties where the pre-tectonic group includes intrusions that are coeval with the volcanic rocks, as well as those that crosscut volcanic and Missi supracrustal rocks. Numerous mafic to ultramafic dykes intrude the volcanic rocks.

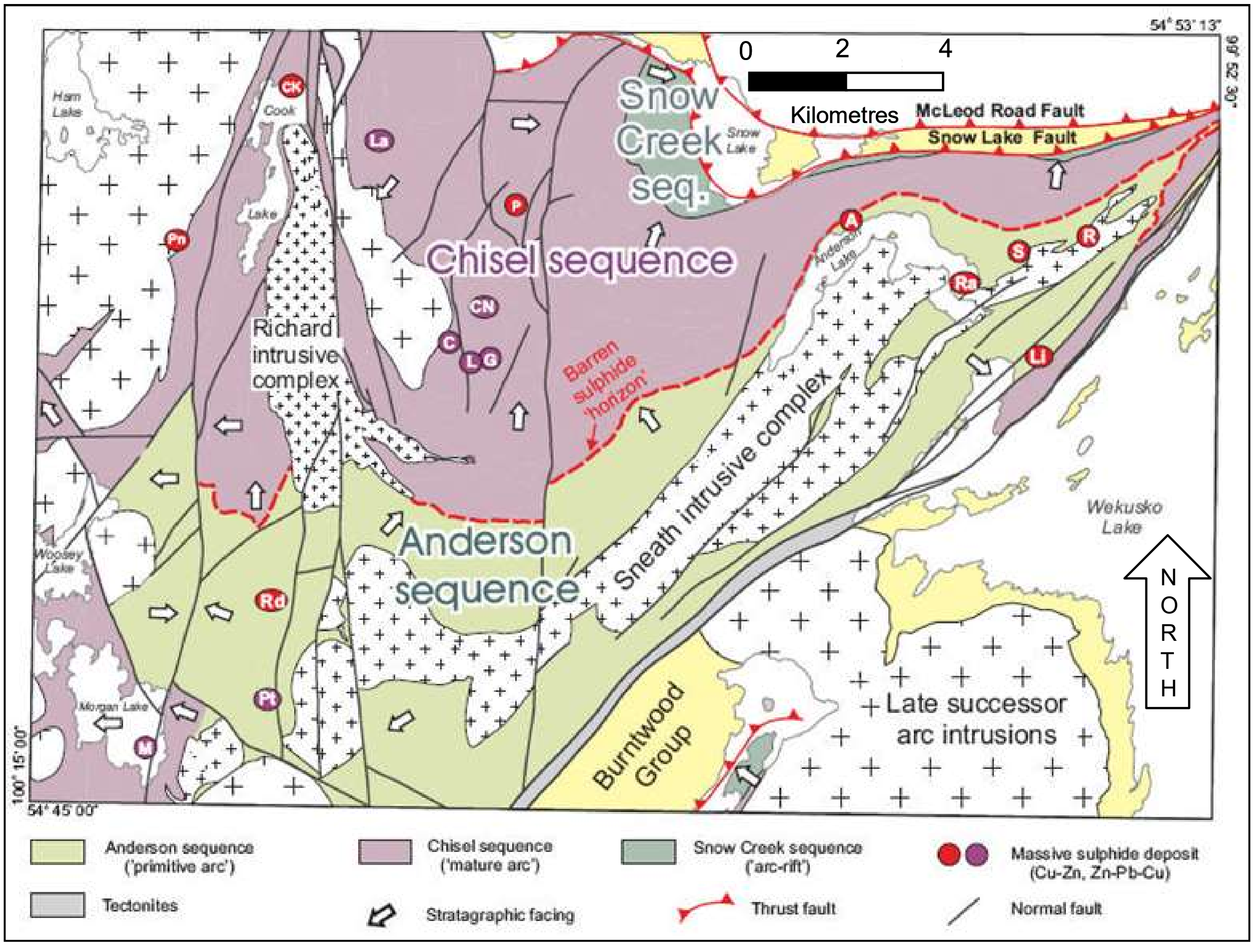
7.2 Property Geology

The Snow Lake arc assemblage (Figures 7-3 and 7-4) that hosts the producing and past-producing mines in the Snow Lake area is a 20km wide by 6km 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 'mature arc' Chisel sequence that hosts the zinc rich Chisel, Ghost, Chisel North, and Lalor deposits typically contains thin and discontinuous volcaniclastic deposits and intermediate to felsic flow-dome complexes.

The Chisel sequence is lithologically diverse and displays rapid lateral facies variations and abundant volcaniclastic rocks. Mafic and felsic flows both exhibit evolved geochemical characteristics (relative to the unevolved underlying Anderson sequence) consistent with one of, or a combination of, the following: within-plate enrichment, derivation from a more fertile mantle source, lower average extents of melting at greater depths, and contamination from older crustal fragments. These rocks have undergone metamorphism at the lower to middle almandine-amphibolite facies.

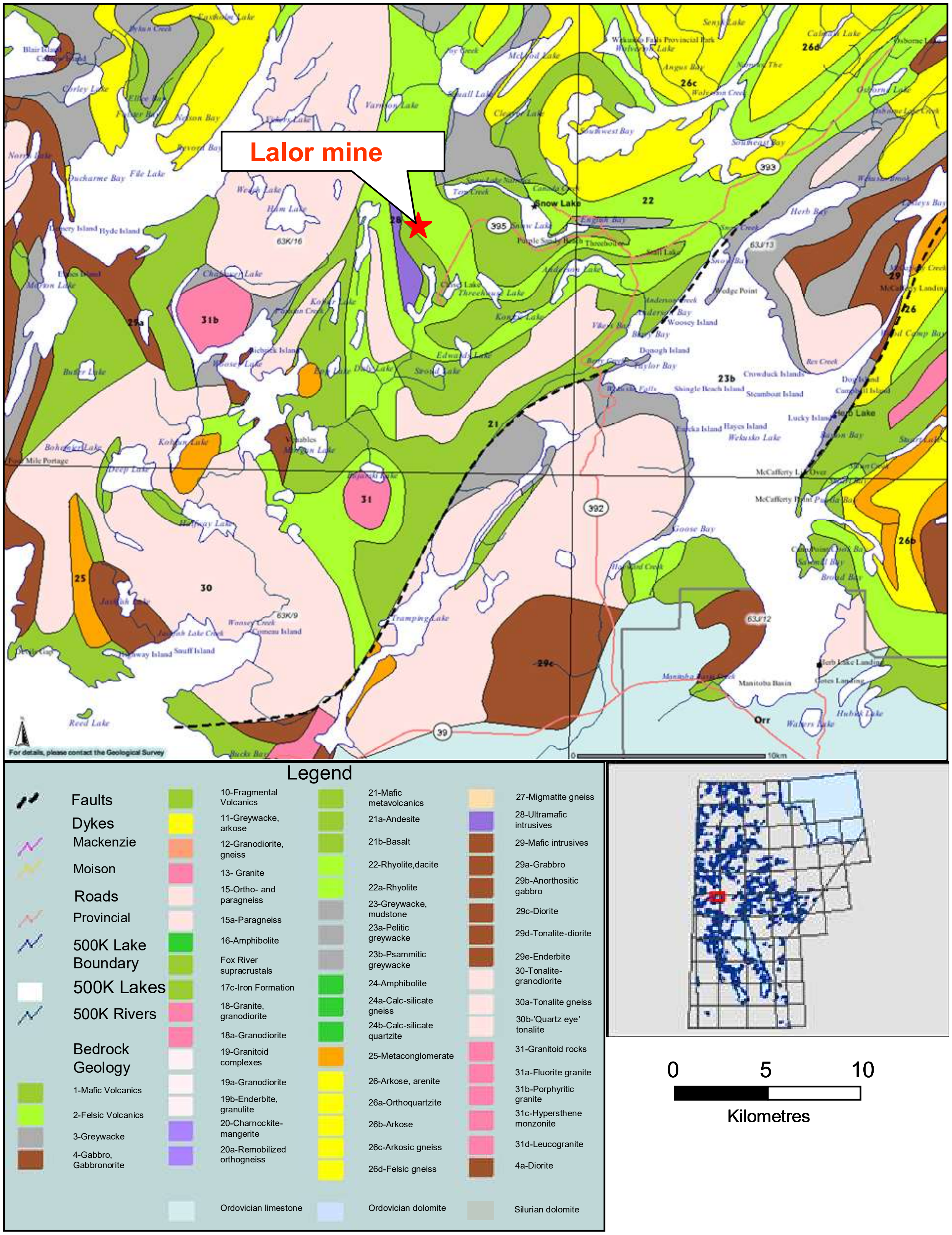
Rock units in the hanging wall of the Lalor deposit typically reflect this diversity and variation in rock types that include mafic and felsic volcanic and volocaniclastic units, mafic wacke, fragmental units of various grain sizes, and crystal tuff units.

FIGURE 7-3: VOLCANIC STRATIGRAPHY OF THE SNOW LAKE AREA



Source: Bailes and Galley, 2007

FIGURE 7-4: GEOLOGY OF THE SNOW LAKE AREA



Source: http://web33.gov.mb.ca/mapgallery/mgg-gmm.html [January 31, 2013]. Manitoba Department of Growth, Enterprise and Trade (GET)

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 most common dyke intrusion throughout these rocks is a fine grained feldspar-phyric gabbro to diorite. The Chisel Lake pluton, a late 1.8 km by 9.8 km layered ultramafic intrusion (Bailes and Galley, 2007), truncates the main lens of the Chisel Lake massive sulphide deposit but is not seen in any of the Lalor drill core.

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 volcanogenic massive sulphide (VMS) deposits and are similar to that at the other massive sulphide deposits in the Chisel Lake area.

7.3 Base Metal Mineralization

The Lalor VMS deposit is flat lying, with zinc mineralization beginning at approximately 600 m from surface and extending to a depth of approximately 1,100 m. The mineralization trends about 320° to 340° azimuth and dips between 30° and 45° to the north. It has a lateral extent of about 900 m in the north-south direction and 700 m in the east-west direction.

Sulphide mineralization is 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.

Seven distinct stacked zinc rich mineralized zones have been interpreted within the Lalor deposit based on the zinc equivalency of 4.1% over a minimum three metre interval (Table 7-1).

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

Overall, Zones 10 and 20 have the largest extent and volume of mineralization. Zone 10 extends approximately 400 m in the east-west and 550 m in the north-south direction and Zone 20, 250 m in the east-west and 700 m in the north-south direction.

TABLE 7-1: SUMMARY OF ZINC RICH INTERPRETED WIREFRAMES

Zone	Area (m^2)	Volume (m^3)	Average Thickness of Mineralization (m)	Number of Drillholes	Assayed Length of Drill Core (m)	Volume (m^3)/Number of Drillholes
10	198,253	1,898,251	9.6	557	7,923.07	3,408
11	45,079	127,763	2.8	117	472.61	1,092
20	140,481	1,435,583	10.2	491	6,141.25	2,924
30	36,694	317,443	8.7	87	801.26	3,649
31	42,656	305,428	7.2	119	979.49	2,567
32	79,474	585,311	7.4	327	2,698.28	1,790
40	74,218	693,170	9.3	102	1,223.09	6,796
		4,669,779			**20,239.05**	

7.4 Gold Mineralization

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.

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).

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. To date no definitive structural controls of the gold mineralization has been interpreted. 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).

Seven lense groups have been interpreted within the deposit area and are present between 750 m to 1,480 m below the surface (Table 7-2). Their general shape is similar to the base metals. However, the current interpretation suggests the deeper copper-gold lense tends to have a much more linear trend to the north than the rest of the zones. The gold mineralization associated with each zone was interpreted into three-dimensional wireframes based on a 2.4 g/t gold equivalent or 1.9% copper equivalent over minimum 3 metre interval.

A structural study with external consultant Jean-Francois Ravenelle, from SRK Consulting (Canada) Inc. is currently ongoing. As of March 2017, four, one week site visits have been completed. Purposes of the study are to:

- Define potential structural controls on higher grade sections within the gold lenses of 21 and 25
- Investigate the suspected presence of a mine scale folded geometry

TABLE 7-2: SUMMARY OF GOLD INTERPRETED WIREFRAMES

Zone	Area (m^2)	Volume (m^3)	Average Thickness of Mineralization (m)	Number of Drillholes	Assayed Length of Drill Core (m)	Volume (m^3)/Number of Drillholes
21	122,358	668,776	5.5	370	2,882.30	1,808
23	46,137	225,844	4.9	177	1,152.51	1,276
24	83,483	423,505	5.1	269	1,589.02	1,574
25	247,164	1,470,650	6.0	454	4,393.73	3,239
26	60,374	504,483	8.4	55	784.56	9,172
27	72,125	462,584	6.4	36	511.31	12,850
28	26,528	247,365	9.3	13	105.81	19,028
		3,755,842			**11,419.24**	

8 DEPOSIT TYPE

Lalor is interpreted as a VMS deposit that precipitated at or near the seafloor in association with contemporaneous volcanism, forming a stratabound accumulation of sulphide minerals. VMS deposits typically form during periods of rifting along volcanic arcs, fore arcs, and in extensional back arc basins. Rifting causes extension and thinning of the crust, providing the high heat source required to generate and sustain a high-temperature hydrothermal system (Franklin et al., 2005).

The location of VMS deposits is often controlled by synvolcanic faults and fissures, which permit a focused discharge of hydrothermal fluids. A typical 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).

The depositional environment for the mineralization at Lalor is similar to that of 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 hydrothermal alteration pipe.

9 EXPLORATION

Exploration in the Lalor deposit area was previously conducted by Hudbay through the main exploration office located in Flin Flon, Manitoba. More recently exploration is managed and conducted by the Lalor mine geology department with core logging and storage facilities, located at the Chisel North mine site, adjacent to the 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 and 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 two km by two km was used to maximize the EM field strength. The survey results were interpreted using three-dimensional computer modeling software. The model indicated a highly conductive, shallow-dipping zone at a vertical depth of 800 m. The Lalor drilling began in March 2007 to test the geophysical anomaly, and diamond drill hole DUB168 intersected conductive sulphides at a depth of approximately 780 m. Drilling is ongoing and has been continuous since the discovery hole.

Exploration drilling, since the disclosed March 29, 2012 NI 43-101 Technical Report, has focused on delineation of the inferred resource, confirming the continuity of the mineralization down plunge and testing for new mineralization peripheral to the known deposit. Surface drilling since the previous disclosure is limited to four drill holes while the focus was on underground exploration, definition, and delineation drilling, which has continued to expand the resource.

The first of the four surface drill holes, DUB288, was completed 500 m east-northeast of the Lalor mine in April of 2012 as a follow-up to copper and gold mineralization intersected in drill holes DUB283 and DUB283W2 that were completed in the fall of 2011. The results suggest further investigation in that area is warranted.

In mid-2015, two surface drill holes (DUB289 and DUB290) were collared approximately 750 m east of the deposit testing a conductive horizon identified by earlier BHEM surveys. A barren sulphide horizon was intersected at a depth of 930 m in DUB290. No further investigation was warranted.

Hole DUB291, 1,709 m in length, was completed in the summer of 2015 and tested the down plunge potential of the deposit. Several zones of mineralization were intersected suggesting future exploration in this area is warranted.

9.1 Underground Exploration

9.1.1 Exploration Development

Since 2014, one exploration drift and one exploration ramp were developed at Lalor for a total of 1,891 m. The development was undertaken to establish underground platforms to conduct exploration drilling on targets that could not be drilled from existing mine infrastructure. Prudent care was taken in the placement and size of both the exploration ramp and drift to assure the selected locations can accommodate future mining equipment and related infrastructure. Table 9-1 summarizes the underground exploration related development and intended future uses of the ramp and drift.

TABLE 9-1: UNDERGROUND EXPLORATION DEVELOPMENT

Location	Year	Meters	Exploration Target (Zone)	Definition Target (Zone)	Possible Future Use
865 Exploration Drift	2014	98	21, 25, 26, 28	20, 30, 40	Ventilation Drift
865 Exploration Drift	2015	396			
865 Exploration Drift	2016	282			
1025 Exploration Ramp	2014	637	27, 21, 25	20	Haulage and Ventilation
1025 Exploration Ramp	2015	478			

9.1.2 Exploration Drilling

In 2015, thirty-one drill holes were completed for a total of 10,395 m focusing on the copper-gold, Zone 27. Exploration drilling continued on Zone 25 from March to July of 2016 for a total of sixty-nine drill holes and 16,098 m. The purpose of the exploration programs was to upgrade inferred resources, specifically focused on identifying areas of enriched gold and copper-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.

9.2 Borehole Electromagnetic (EM) Surveys

Time-domain borehole EM surveys with three dimensional probes are routinely conducted on surface and underground drill holes. The survey results identify any off-hole conductors that were missed, indicate direction to the target, as well as the dimensions and the attitude of the conductor. The surveys can also detect possible conductors which may lie past the end of the hole allowing decisions to extend holes to be made.

9.3 Surface Electromagnetic (EM) Surveys

Two time-domain surface EM surveys, for a total of approximately 35 line km, were completed north-northeast and south-southwest of the Lalor mine. Neither survey has identified any new significant targets of interest in the general Lalor mine area.

9.4 Airborne Electromagnetic (EM) Survey

During the summer of 2014, an airborne EM survey was conducted to test the capabilities of the HeliSAM system for a total of 97.5 line km. Performed by GAP Geophysics, based in Perth Australia, using a ground based transmitting loop and airborne total field magnetic sensor. The testing was aimed to identify the Lalor mine at a depth beyond the capabilities of conventional airborne EM systems. The test was successful and has lead to further surveys of this type elsewhere in the mining camp.

10 DRILLING

The Lalor mine was discovered by drilling a surface exploration hole testing an electromagnetic geophysical anomaly in March 2007, which intersected appreciable widths of zinc-rich massive sulphides in hole DUB168. Surface drilling continued to July 2012. A limited surface exploration drill program was conducted from August to October 2015 to explore for potential down plunge extensions of 27 lens and to test near mine geophysical conductors that could not be drilled from underground workings. As of January 1, 2017, a total of 203,037 m of surface drilling was completed at Lalor and Table 10-1 provides a summary by year.

TABLE 10-1: SUMMARY SURFACE DIAMOND DRILL HOLES WITH ASSAY RESULTS AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2007	Parent	Hudbay	2	BQ	2,342	Major Drilling Ltd.
	Parent	Hudbay	26	NQ	29,600	Major Drilling Ltd.
2008	Parent	Hudbay	41	NQ	45,454	Major Drilling Ltd.
	Wedge	Hudbay	32	NQ/AQ	12,112	Major Drilling Ltd.
2009	Parent	Hudbay	29	NQ	35,390	Major Drilling Ltd.
	Wedge	Hudbay	47	NQ/AQ	22,884	Major Drilling Ltd.
2010	Parent	Hudbay	13	NQ	17,438	Major Drilling Ltd.
	Wedge	Hudbay	17	NQ/AQ	11,576	Major Drilling Ltd.
2011	Parent	Hudbay	10	NQ	15,458	Major Drilling Ltd.
	Wedge	Hudbay	5	NQ/AQ	3,139	Major Drilling Ltd.
2012	Parent	Hudbay	3	NQ	4,688	Major Drilling Ltd.
2015	Parent	Hudbay	2	NQ	2,956	Rodren Drilling Ltd.
Total			**222**		**203,037**	

Underground drilling began at Lalor with hole LP0001 in January 2012 and drilling has been continuous to date. Holes are drilled at all dips and azimuths needed to provide adequate coverage of the orebody for interpretation and mining purposes. 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 is divided into five different categories based on the primary planned purpose of the hole. The hole categories are as follows:

LD Prefix

Lalor definition holes with LD prefix 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. Drill costs are allocated to the yearly operational budget.

TABLE 10-2: SUMMARY OF LD UNDERGROUND DIAMOND DRILL HOLES AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2012	LD	Hudbay	48	BQ/NQ	5,264/2,194	Major Drilling Ltd.
2013	LD	Hudbay	142	BQ/NQ	4,660/13,873	Major Drilling Ltd.
2014	LD	Hudbay	315	BQ/NQ	9,995/22,377	Major Drilling Ltd.
2015	LD	Hudbay	258	BQ/NQ	14,144/13,038	Major Drilling Ltd.
2016	LD	Hudbay	174	BQ/NQ	10,643/3,411	Major Drilling Ltd.
Total			**937**		**99,599**	

LE Prefix

Lalor engineering holes with LE prefix are drilled for mine infrastructure purposes such as drain holes, holes for electrical cables and service holes for break through rounds. Drill costs are allocated to the yearly operational engineering budget.

TABLE 10-3: SUMMARY OF LE UNDERGROUND DIAMOND DRILL HOLES AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2012	LE	Hudbay	2	NQ	81	Major Drilling Ltd.
2015	LE	Hudbay	1	NQ	62	Major Drilling Ltd.
2016	LE	Hudbay	6	NQ	239	Major Drilling Ltd.
Total			**9**		**382**	

LP Prefix

Lalor project holes with LP prefix were holes drilled before the projects team handover to the operations team in 2012 and prior to mine production at Lalor. Project holes are similar to LD prefix holes noted above and costs of these holes were allocated to the project group budget in 2012.

TABLE 10-4: SUMMARY OF LP UNDERGROUND DIAMOND DRILL HOLES AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2012	LP	Hudbay	32	NQ	5,551	Major Drilling Ltd.
Total			32		5,551	

LQ Prefix

Lalor delineation holes with LQ prefix are drilled from within the mineralization. Holes are typically drilled to establish exact ore contacts for detailed mine planning and stope design purposes. Typical hole spacing is 10 to 15 m and costs are allocated to the yearly operational budget.

TABLE 10-5: SUMMARY OF LQ UNDERGROUND DIAMOND DRILL HOLES AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2012	LQ	Hudbay	2	BQ	61	Major Drilling Ltd.
2013	LQ	Hudbay	159	BQ	4,195	Major Drilling Ltd.
2014	LQ	Hudbay	80	BQ	2,692	Major Drilling Ltd.
2015	LQ	Hudbay	231	BQ	9,445	Major Drilling Ltd.
2016	LQ	Hudbay	74	BQ	2,113	Major Drilling Ltd.
Total			546		18,506	

LX Prefix

Lalor exploration holes with LX prefix are drilled for targets outside known lenses or in areas of inferred resources and low drill density. Costs are allocated to the yearly capital budget.

TABLE 10-6: SUMMARY OF LX UNDERGROUND DIAMOND DRILL HOLES AS OF JANUARY 1, 2017

Year	Hole Type	Operator	Number of Holes	Core Size	Length (m)	Drilling Company
2015	LX	Hudbay	31	NQ	10,395	Major Drilling Ltd.
2016	LX	Hudbay	69	BQ/NQ	16,098	Major Drilling Ltd.
Total			100		26,493	

10.1 Surveying of Property Grid and Drill Hole Collars

All 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 Lalor Geology. 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.

10.2 Downhole Surveying

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.

The Reflex instruments measure the azimuth relative to the earth's magnetic field and records magnetic azimuth, dip of the hole and Total Magnetic Intensity (TMI) in nanotesla (nT). Magnetic interference is likely to occur in areas where the TMI deviates significantly from the regional value. Regional TMI for the mine is 58,000 nT to 60,000 nT. The Reflex instrument is calibrated to notify user when the TMI deviates more than 1,000 nT from the programmed regional TMI value. All down hole survey results are validated by a geologist before being entered into the acQuire drill hole database.

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.

11 SAMPLING PREPARATION, ANALYSES, AND SECURITY

11.1 Laboratory/Laboratories Used

Since the start of exploration at Lalor the following different laboratories and sample shipment/prep procedures have been in use:

- Discovery to November 1, 2009, Lalor samples were prepared and analyzed at Hudbay laboratory in Flin Flon, Manitoba. As part of Hudbay QAQC procedures, pulp duplicates were sent to ACME Analytical Laboratories Ltd. (ACME) in Vancouver, BC.
- November 1, 2009 to March 12, 2012, Lalor samples were received, crushed and pulverized at Hudbay laboratory and pulps shipped to ACME for analysis. Pulp duplicates were analyzed at Hudbay laboratory as part of the QAQC procedure.
- March 13, 2012 to May 21, 2014, all samples were prepared and analyzed at Hudbay laboratory. As part of Hudbay QAQC procedures, pulp duplicates were sent to ACME.
- May 22, 2014 to present, parts of the sample stream were and are shipped to ACME, (re-named to Bureau Veritas after January 1, 2015). The remainder of the sample stream is shipped to Hudbay laboratory.
- A set of 303 drill core pulps from samples originating from within known resource envelopes were submitted for check assaying at a SGS Laboratory in Burnaby, BC as part of the QAQC program for the 2017 resource estimate.

Bureau Veritas and SGS are certified independent laboratories, while the certified laboratory in Flin Flon is owned by Hudbay.

11.2 Sample Collection

Drill core is logged, sample intervals selected and marked clearly on the core with the following information written directly on the core using a red grease pen:

1) Sample from: start depth of sample interval measured from collar of drill hole
2) Alphanumeric unique sample number consisting of:
    a) A letter prefix of either **M** or **N**.
        i) **M** prefix: Whole core is sampled and submitted to lab for assaying, rejects are discarded and pulps retained for reference
        ii) **N** prefix: pulps and rejects are to be stored for reference (bulk of holes are split with half core samples kept for reference)
    b) A unique six digit number

3) Sample to: end depth of sample interval measured from collar of drill hole

Visual metal mineral estimates are recorded for each individual sample and noted under sample number in sample book. The current practice for every 100 samples the following QAQC samples are inserted into the sample stream:

1) Two blanks
2) Five duplicates
3) Five base metal standards, each of differing grade thresholds
4) Two gold standards of differing grade

Once logging is complete all data from sample book including QAQC samples are entered into Hudbay's acQuire drill hole database by hole number. Before samples are split/bagged for shipment the core is photographed. The photographing of the core is the last step of the logging process as to assure that a full photographic record of exact locations of all contacts, sample locations and numbers is captured. A standard setup as well as a single camera type with standardized settings is use to ensure photographs of consistent quality. Each photo covers a maximum of five boxes and includes in the frame a placard with the following information in clear legible writing:

1) Hole ID
2) Project name
3) Name and number of any mineralized zones as well as start and end of zone given as distances in meter down hole
4) Current date
5) Box number for first and last box in photograph
6) Depth range of hole displayed in photograph measured in meters from collar with 2 significant digits

Once photographs have been captured they are saved on the company server and linked to acQuire drill hole database for network access. This procedure applies to all drill core at Lalor mine including drill core to be sampled by destruction. If intervals of a drill hole scheduled to be sampled by destruction are deemed not to be sampled by logging geologist the core is kept until assays of submitted samples from drill hole have been received and reviewed by the geologist. If the geologist deems that no further sampling is needed core is discarded.

11.3 Sample Preparation

11.3.1 Hudbay

All samples arriving at the Hudbay analytical laboratory are checked against the geologist's sample submission sheets. Laboratory analytical work sheets are generated for the analysis areas. Any wet samples are dried at 105°C as per industry standard. The core samples are crushed to (-)10 mesh then split to approximately 250g and pulverized with 90% passing (-)150 mesh before being deposited into labelled bags. Crusher and pulverizer checks are conducted daily to ensure there is no excessive wear on the crusher plates and pulverizer pots.

11.3.2 ACME/Bureau Veritas

All samples arriving at ACME (Bureau Veritas) are checked against chain information on sample submittal form and prepped according to codes WGHT and PR80-250. The sample preparation includes weighing of sample, crushing 1kg to minimum 80% passing 2 mm. A 250 g split crushed to minimum 85% passing 75 µm.

11.3.3 SGS

All samples arriving at SGS Canada Inc. laboratory are checked against chain information on sample submittal form upon arrival. No sample preparation was conducted at SGS as the laboratory was only used for check assays done on pulps previously assayed at either Hudbay or Bureau Veritas laboratories.

11.4 Assay Methodology

11.4.1 Hudbay

Samples sent to the Hudbay laboratory were analyzed for the following elements: gold, silver, copper, zinc, lead, iron, arsenic and nickel. Base metal and silver assaying was completed by aqua regia digestion and read by a simultaneous ICP unit. The gold analysis was completed on each sample by atomic absorption spectrometry (AAS) after fire assay lead collection. All samples with gold values (AAS) > 10 g/t were re-assayed using a gravimetric finish. Detection limits of the ICP and AAS are shown in Table 11-1.

TABLE 11-1: HUDBAY LABORATORY DETECTION LIMITS

Element	Detection Limit
Ag	0.439 g/t
As	0.002%
Au	0.034 g/t
Cu	0.003%
Fe	0.007%
Ni	0.001%
Pb	0.002%
Zn	0.010%

All analytical balances are certified annually by a third party. Check weights are used daily to verify calibration of balances. All metal standards used to make the calibration standards for the AAS and ICP are certified and traceable. Each is received with a certificate of analysis. Both the AAS and ICP are serviced twice per year by the instrument manufacturer's qualified service representative to ensure that the instruments meet original design specifications.

The Flin Flon assay laboratory has been participating in CANMET PTP/MAL round robin testing since 2000. PTP/MAL is a requirement for laboratories that are ISO 17025 certified. The laboratory has also been participating since 2002 in round robin testing conducted by GEOSTATS of Australia.

Fine sample pulps are kept in secure storage at the laboratory after analysis. Pulps are only released after all data is validated.

11.4.2 Acme/Bureau Veritas

Two different assay methods are used for samples shipped to Bureau Veritas: AQ270 and AQ370. AQ370 was the only method used on samples submitted from May 2014 to the second quarter of 2016 after which the AQ270 method was applied to selected holes. After the fourth quarter of 2016 all samples submitted to Bureau Veritas were assayed using the AQ270 method. All samples using method AQ270 and AQ370 were run for gold using method FA430. The following elements were run for over range as necessary: gold, copper, zinc and lead using methods FA530, GC820, GC816, MA404 (assays returning lead values above 20% using MA404 were also run using method GC817) respectively. Detection limits for over range methods, AQ270 and AQ370 are listed in Table 11-2 to Table 11-4.

Samples from selected holes were also submitted for determination of specific gravity using SPG02 method (volume determination by submersion followed by drying).

Samples shipped to Bureau Veritas from November 1, 2009 to March 12, 2012, were run using the legacy codes (Group 7AR) and (Group 601) with over range samples being run with gravimetric finish (Group 612). The sample preparation for these legacy codes is essentially similar to those listed for the current AQ270 and AQ370 codes used after May 22, 2014.

For the multi-element methods AQ270 and AQ370, aliquots of 1.000 ± 0.002 g are weighed into 100 mL volumetric flasks. Bureau Veritas QAQC protocol requires one pulp duplicate to monitor analytical precision, a blank, and an aliquot of in-house reference material to monitor accuracy in each batch of 36 samples. 30 mL of Aqua Regia, a 1:1:1 mixture of ACS grade concentrated HCl, concentrated HNO_3 and de-mineralised H_2O, is added to each sample. Samples are digested for one hour in a hot water bath (> 95°C). After cooling for 3 hours, solutions are made up to volume (100 mL) with dilute (5%) HCl. Very high-grade samples may require a 1 g to 250 mL or 0.25 g to 250 mL sample/solution ratio for accurate determination. Bureau Veritas QAQC protocol requires simultaneous digestion of a reagent blank inserted in each batch.

For both AQ270 and AQ370 sample solutions are aspirated into an ICP emission spectrograph (ES) to determine 24 elements. For method AQ270 the solution is also run through an ICP mass spectrometer (MS) to provide values for an additional 10 elements bring the total number of elements to 34. Raw and final data from the ICP-ES/ICP-MS undergoes a final verification by a

British Columbia Certified Assayer who then signs the Analytical Report before it is released to the client. The 24 element assay method AQ370 has detection limits displayed in Table 11-2 and the 34 element assay method AQ270 has detection limits displayed in Table 11-3.

For the gold analysis FA430, 30 g charges are weighed into fire assay crucibles. The sample aliquot is custom blended with fire assay fluxes, PbO litharge and a silver inquart. Firing the charge at 1050°C liberates Au, Ag ± PGEs that report to the molten Pb-metal phase. After cooling the lead button is recovered, placed in a cupel, and fired at 950°C to render an Ag ± Au ± PGEs dore bead. The bead is weighed and parted (i.e. leached in 1 mL of hot HNO_3) to dissolve silver leaving a gold sponge. Adding 10 mL of HCl dissolves the Au ± PGE sponge. Solutions are analysed for gold on an ICP emission spectrometer. Gold in excess of 10 g/t forms a large sponge that can be weighed (gravimetric finish, method FA530).

As part of Bureau Veritas QAQC protocol, a sample-prep blank is inserted as the first sample and carried through all stages of preparation to analysis as well as a pulp duplicate to monitor analytical precision. Two reagent blanks are inserted in each batch to measure background, and aliquots of Certified Reference Materials are used to monitor accuracy of the obtained gold assays. Raw and final data undergo a final verification by a British Columbia Certified Assayer who signs the Analytical Report before it is released to the client. Bureau Veritas is currently registered with ISO 9001 accreditation.

Fine sample pulps are kept in secure storage at the laboratory after analysis. Pulps are only released after all data is validated.

TABLE 11-2: BUREAU VERITAS ELEMENTAL DETECTION LIMITS AQ370

Element	Detection Limit	Upper Limit
g	2 g/t	300 g/t
Al	0.01%	
As	0.01%	10%
Bi	0.01%	
Ca	0.01%	
Cd	0.001%	
Co	0.001%	
Cr	0.001%	
Cu	0.001%	10%
Fe	0.01%	
Hg	0.001%	
K	0.01%	
Mg	0.01%	
Mn	0.01%	
Mo	0.001%	20%
Na	0.01%	

Element	Detection Limit	Upper Limit
Ni	0.001%	
P	0.001%	
Pb	0.01%	4%
S	0.05%	
Sb	0.001%	
Sr	0.001%	
W	0.001%	
Zn	0.01%	20%

TABLE 11-3: BUREAU VERITAS ELEMENTAL DETECTION LIMITS AQ270

Element	Detection Limit	Upper Limit
Ag	0.5ppm	300ppm
Al	0.01%	
As	5ppm	100000ppm
Ba	5ppm	
Bi	0.5ppm	
Ca	0.01%	
Cd	0.5ppm	
Co	0.5ppm	
Cr	0.5ppm	
Cu	0.5ppm	100000ppm
Fe	0.01%	
Ga	5ppm	
Hg	0.05%	
K	0.01%	
La	0.5ppm	
Mg	0.01%	
Mn	5ppm	
Mo	0.5ppm	200000ppm
Na	0.01%	
Ni	0.5ppm	
P	0.00%	
Pb	0.5ppm	40000ppm
S	0.05%	
Sb	0.5ppm	
Sc	0.5ppm	
Se	2ppm	500ppm
Sr	5ppm	
Th	0.5ppm	

Element	Detection Limit	Upper Limit
Ti	0.00%	
Tl	0.5ppm	
U	0.5ppm	
V	10ppm	
W	0.5ppm	
Zn	5ppm	200000ppm

TABLE 11-4: BUREAU VERITAS ELEMENTAL DETECTION LIMITS AND RANGE FOR OVER RANGE CODES

Element	Over Range Codes	Technique	Detection Limit	Upper Limit
Au	FA530	Fire Assay/Gravemetric Finish	0.9ppm	1,000,000 ppm
Ag	FA530	Fire Assay/Gravemetric Finish	50ppm	1,000,000 ppm
Cu	GC820	Titration	0.01%	100%
Zn	GC816	Titration	0.01%	100%
Pb	GC817	Titration	0.01%	100%
Fe	GC818	Titration	0.01%	100%

TABLE 11-5: BUREAU VERITAS ELEMENTAL DETECTION LIMITS FOR LEGACY CODES (GROUP 7AR) AND (GROUP 601)

Element	Detection Limit
Ag	2.000 g/t
Al	0.010 %
As	0.010%
Au	0.010 g/t
Bi	0.010 %
Ca	0.010 %
Cd	0.001 %
Co	0.001 %
Cr	0.001 %
Cu	0.001 %
Fe	0.010 %
Hg	0.001 %
K	0.010 %
Mg	0.010 %
Mn	0.010 %
Mo	0.001 %
Na	0.010 %
Ni	0.001 %
P	0.001 %

Element	Detection Limit
Pb	0.010 %
Sb	0.001 %
Sr	0.001 %
W	0.001 %
Zn	0.010 %

11.5 Assay Certificates

Assay certificates since the discovery of Lalor have had two sources:

1) Hudbay laboratory in Flin Flon, Manitoba
2) ACME laboratory (renamed to Bureau Veritas as of January 1, 2015) in Vancouver, British Columbia

Assay certificates are received from both laboratories in digital form via e-mail. Data for import arrives as CSV files. Each CSV file is accompanied by a PDF certificate covering the sample number listed in the CSV file. The files are sent directly from the laboratory and independently to Lalor Senior Geologist and Hudbay database manager. Digital copies are archived with write privileges given only to the Hudbay database manager.

11.6 Security

Security measures taken to ensure the validity and integrity of the samples collected include:

- Chain of custody of drill core from the drill site to the core logging area
- All facilities used for core logging and sampling located on a secure mine site
- Core sampling is undertaken by Hudbay geologists
- Sample splitting and shipping conducted by technicians under the supervision of Hudbay geologists
- Chain of custody for core cutting through to delivery of samples to laboratories
- Well documented and implemented receiving and processing procedures at the Hudbay and Bureau Veritas laboratories
- The Hudbay Laboratory samples results are stored on a secure mainframe based Laboratory Information Management System (LIMS)
- The diamond drill hole database is stored on the secure Hudbay network, using the acQuire database management system with strict access rights

The author believes that there are no factors that could have materially impacted the accuracy and reliability of the sample preparation, security, and analytical procedures and that those used are appropriate and adequate for VMS type mineralization.

12 DATA VERIFICATION

Data verification procedures and results since the previous technical report are included in this section based on information collected and reviewed from 2012 to 2016. For data verification procedures and results prior to 2012, refer to previous technical report dated March 29th, 2012.

12.1 Bureau Veritas Assay Methods and QAQC

Over the past five years, 2012 to 2016, a total of 23,822 drill core samples were analyzed at Bureau Veritas laboratories. Copper, zinc, and silver were digested in aqua regia and analyzed by inductively coupled plasma optical emission spectrometry (ICP-OES) and more recently in 2016 by inductively coupled plasma mass spectrometry (ICP-MS) (Table 12-1). Samples with copper and zinc over the upper limit of detection (ULD) were analyzed by titration, whereas those samples with silver values over the ULD were analyzed by fire assay and gravimetric finish (Table 12-1).

TABLE 12-1: BUREAU VERITAS ASSAYS SPECIFICATIONS

Element	Unit	Lower Detection Limit	Upper Detection Limit	Digestion	Instrumental Finish	Method Code	Legacy Method Code
Cu_1	%	0.001	10	Aqua Regia	ICP-OES	AQ370	7AR
Cu_2	%	0.00005	10	Aqua Regia	ICP-OES and ICP-MS	AQ270	7AX
Cu overlimits	%	0.01	100		Cu Titration	GC820	G820
Zn_1	%	0.01	20	Aqua Regia	ICP-OES	AQ370	7AR2
Zn_2	%	0.0005	20	Aqua Regia	ICP-OES and ICP-MS	AQ270	7AX
Zn overlimits	%	0.01	100		Zn Titration	GC816	
Ag_1	ppm	2	300	Aqua Regia	ICP-OES	AQ370	7AR1
Ag_2	ppm	0.5	300	Aqua Regia	ICP-OES and ICP-MS	AQ270	
Ag overlimits	ppm	50		Fire Assay	Gravimetric	FA530	G603-G612
Au	ppm	0.01 - 0.005	10	Fire Assay	AAS	FA430	G601
Au overlimits	ppm	0.17 - 0.9	16600	Fire Assay	Gravimetric	FA530	G6 Grav

Gold was determined by lead-collection fire assay fusion, for total sample decomposition, followed by atomic absorption spectroscopy (AAS) instrumental analysis (Table 11-1). Fire assays were performed on 15 to 30g subsample pulps to circumvent problems due to potential nugget effect.

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 mostly by Ore Research and Exploration (OREAS). However, a few 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.

12.1.1 Blanks

Certified OREAS blanks were inserted into the sample stream commonly one every fifty samples to monitor potential cross-contamination (Table 12-2). Between 2012 and 2016, 841 blanks were analyzed representing 3.5% of the samples submitted to Bureau Veritas.

TABLE 12-2: OREAS CERTIFIED BLANKS

Element	Cu	Zn	Ag	Au
Unit	%	%	ppm	ppm
F6[1]	0.0035	0.0054	<0.1	0.009
F7[2]	0.005	0.0107	0.068*	<0.001
Certified Method	Aqua regia[1] - 4 acids[2]	Aqua regia[1] - 4 acids[2]	Aqua regia[1] - 4 acids[2]	Fire assay

*Indicative value

Blanks F6 and F7 are coarse blanks, <7mm chips, prepared exclusively for Hudbay by Ore Research and Exploration (Table 11-2). Both blanks have the same matrix consisting of fresh alkali olivine basalt. They are packaged in 100 g laminated foil pouches.

Gold assays in both blanks were determined by fire assay (Table 12-2). Copper, zinc, and silver were determined following an aqua regia digestion for blank F6 and a 4 acid digestion for blank F7. The 4 acid digestion is a stronger acid attack than the aqua regia digestion used for assaying material from the Lalor mine. However, to assess contamination both blanks are appropriate given their low concentration of base and precious metals. Silver is not certified in these blanks but their indicative values are a good guide to track potential contamination.

Blank failure thresholds due to potential contamination issues are set to values that exceed the certified best value (CBV) plus three standard deviations. A summary of the blank performance is shown on Table 12-3. The potential impact of contamination on resource estimation is assessed by (1) quantifying the amount of contamination relative to the certified blanks, and (2) the percent rate of failed blanks. An assay program is considered successful when the blank failure rate is <10%, and the average grades of contaminated blanks are not economic.

A total of 320 F6 blanks and 521 F7 blanks were systematically inserted along with the drill core samples analyzed at Bureau Veritas (Table 11-3). Contamination with copper and zinc was insignificant with an average contamination of 25 to 30 ppm for copper and 230 to 330 ppm for zinc. The blank failure rate for these metals is <6% and there are no cases of contamination at economic grade levels.

TABLE 12-3: SUMMARY OF BLANK PERFORMANCE

Coarse Blank F6					
Analyte	No. Blanks	Failed Blanks	Contamination Rate	Maximum Contamination	Average Contamination
Cu	320	17	5.3%	100 ppm	25 ppm
Zn	320	9	2.8%	440 ppm	230 ppm
Ag	320	0	0.0%	0	0
Au	320	12	4%	7 ppb	3 ppb

Coarse Blank F7					
Analyte	No. Blanks	Failed Blanks	Contamination Rate	Maximum Contamination	Average Contamination
Cu	521	16	3.1%	60 ppm	30 ppm
Zn	521	24	4.6%	0.25%	330 ppm
Ag	521	3	0.6%	1.9	1 ppm
Au	521	31	6.0%	85 ppb	13 ppb

Blank F6 shows no contamination with silver, whereas blank F7 indicates an average contamination with silver of 1ppm. However, the failure rate for silver on blank F7 is <1% which renders the contamination an isolated case and not significant (Table 12-3).

The blank failure rate for gold is 4 to 6%. The contamination recorded by the blanks is on average 3 to 13ppb which is below grades of economic interest. Blank F7 carries contamination of up to 85ppb gold. The contamination is not economically significant, but it is recommended to visit the laboratory and review the sample preparation facilities since cleaner laboratory conditions are achievable and desired for gold assays (Table 12-3).

The performance of blanks indicates that there are no significant problems with contamination at the Bureau Veritas laboratories, samples were handled with care, and the assay results are free of contamination and adequate for the resource estimation.

12.1.2 Standards

From 2012 to 2016 a total of 1,601 OREAS and Rocklabs CRMs were analyzed at Bureau Veritas representing 6.7% of the sample stream. Table 12-4 presents a list of all CRMs used by Hudbay for quality control over the past five years.

TABLE 12-4: CERTIFIED REFERENCE MATERIALS

Element	Cu	Zn	Ag	Au
Unit	%	%	ppm	ppm
OREAS A5	0.0945	0.506	1.91	0.141
OREAS B5	1.33	0.125	2.86	0.476
OREAS C5	3.37	5.28	21.1	2.493
OREAS D5	9.42	2.46	90.0	7.344
OREAS E5	0.393	23.70	19.7	0.780
OREAS A6	0.055	0.039	0.535	0.121
OREAS B6	0.872	0.702	2.94	0.689
OREAS C6	2.05	2.41	5.18	0.987
OREAS D6	4.20	3.31	25.17	3.04
OREAS E6	0.245	18.14	15.23	0.316
Rocklabs SN60	-	-	-	8.595
Rocklabs SN75	-	-	-	8.671
Rocklabs SP59	-	-	-	18.12
Rocklabs SP73	-	-	-	18.17
Certified Method	Aqua Regia	Aqua Regia	Aqua Regia	Fire assay

OREAS standards are a series of matrix-matched polymetallic CRMs prepared exclusively for Hudbay (Table 12-4). The geological materials were sourced from Hudbay Manitoba mine ore bodies. The mines are located in northern Manitoba within the Flin Flon Greenstone Belt in the Canadian Shield. These ore bodies are typical of polymetallic copper, zinc, silver, and gold rich volcanogenic massive sulphide (VMS) deposits. The gangue materials comprise felsic and mafic volcanic rocks; commonly the host rocks of economic mineralization. These CRMs are packaged into 50g laminated foil pouches sealed under dry nitrogen. The OREAS series covers a wide range of low-, medium-, and high-grade metal values adequate for the QAQC program.

Rocklabs standards are a series of commercially available high-grade gold CRMs prepared from pulverized feldspar minerals, basaltic rocks, and barren pyrites (Table 12-4). These materials are blended with pulverized gold-bearing minerals screened to avoid nugget effect.

More than 50 samples were analyzed per CRM which provide sufficient information to set acceptance criteria relative to the average (AV) and standard deviation (SD) of the actual assay values of the CRMs. However, the acceptance criteria is re-set to the CBV and SD recommended by the CRM in those cases where (1) the absolute bias is >10%, relative to the certified best value (CBV) provided by the CRM, and (2) the laboratory determinations have unexpected larger variance than the recommended SD of the CRM. Table 12-5 summarizes the performance of the CRM assays at Bureau Veritas laboratories.

TABLE 12-5: SUMMARY OF CRM PERFORMANCE AT BUREAU VERITAS

Cu (%)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (%)	ACME Average	CRM CV	Lab CV	Relative Bias
OREAS A5	83	2	2.4%	0.0945	0.0915	3%	2%	-3.1%
OREAS B5	74	1	1.4%	1.33	1.30	3%	2%	-1.9%
OREAS C5	56	0	0%	3.37	3.29	6%	2%	-2.3%
OREAS D5	58	0	0%	9.42	9.22	4%	4%	-2.1%
OREAS E5	59	1	1.7%	0.393	0.389	8%	2%	-1.1%
OREAS A6	191	1	0.5%	0.055	0.053	4%	3%	-3.9%
OREAS B6	190	0	0.0%	0.872	0.870	2%	2%	-0.3%
OREAS C6	185	2	1.1%	2.05	2.03	3%	2%	-1.0%
OREAS D6	184	2	1.1%	4.20	4.12	4%	2%	-2.0%
OREAS E6	183	2	1.1%	0.245	0.247	5%	2%	0.7%
Zn (%)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	ACME Average	CRM CV	Lab CV	Relative Bias
OREAS A5	83	1	1.2%	0.506	0.542	5%	3%	7.0%
OREAS B5	74	0	0%	0.125	0.132	7%	3%	5.7%
OREAS C5	56	0	0%	5.28	5.35	11%	3%	1.2%
OREAS D5	58	1	2%	2.46	2.55	7%	3%	3.8%
OREAS E5	59	1	1.7%	23.70	25.10	4%	2%	5.9%
OREAS A6	191	0	0%	0.039	0.037	5%	14%	-6.0%
OREAS B6	190	1	0.5%	0.702	0.703	3%	3%	0.2%
OREAS C6	185	1	0.5%	2.41	2.43	3%	3%	1.0%
OREAS D6	184	0	0.0%	3.31	3.37	4%	2%	1.8%
OREAS E6	183	1	0.5%	18.14	18.07	4%	2%	-0.4%
Au (ppm)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	ACME Average	CRM CV	Lab CV	Relative Bias
OREAS A5	83	2	2.4%	0.141	0.132	7%	9%	-6.4%
OREAS B5	74	1	1.4%	0.476	0.460	3%	6%	-3.3%
OREAS C5	56	2	4%	2.493	2.488	2%	4%	-0.2%
OREAS D5	58	2	3%	7.344	7.367	4%	8%	0.3%
OREAS E5	59	2	3.4%	0.780	0.758	3%	6%	-2.8%
OREAS A6	190	3	1.6%	0.121	0.126	4%	4%	3.9%
OREAS B6	190	4	2.1%	0.706	0.720	2%	4%	1.9%
OREAS C6	185	5	2.7%	0.987	0.988	2%	3%	0.1%
OREAS D6	184	1	0.5%	3.04	3.040	4%	4%	0.0%
OREAS E6	183	3	1.6%	0.316	0.325	4%	4%	2.8%
Rocklabs SN75	175	2	1.1%	8.671	8.626	2%	2%	-0.5%
Rocklabs SP73	163	3	1.8%	18.17	18.00	2%	2%	-1.0%
Ag (ppm)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	ACME Average	CRM CV	Lab CV	Relative Bias
OREAS B5	73	4	5.5%	2.86	2.95	15%	16%	3.0%
OREAS C5	56	1	2%	21.1	22.45	11%	5%	6.4%
OREAS D5	58	0	0%	90.0	98.21	13%	3%	9.1%
OREAS E5	59	1	1.7%	19.7	20.80	10%	5%	5.6%
OREAS B6	182	2	1.1%	2.94	2.92	18%	16%	-0.6%
OREAS C6	185	4	2.2%	5.18	5.12	10%	11%	-1.2%
OREAS D6	184	4	2.2%	25.17	26.32	6%	5%	4.5%
OREAS E6	183	2	1.1%	15.23	15.66	6%	7%	2.8%

The performance gates were set such that CRM assayed values within AV±2SD and isolated values between AV±2SD and AV±3SD were accepted. In contrast, two consecutive assayed values between AV±2SD and AV±3SD and all values outside the AV±3SD were rejected.

To evaluate the accuracy of the assays using the CRMs, the relative bias was calculated after excluding the outlier values located outside the AV±3SD:

Bias (%) = 100*[(AVeo/CBV)-1]

AVeo represents the average of the actual assay values after excluding outliers. The analytical bias was assessed according to the following ranges: good between 0 and ±5%, reasonable between ±5% and ±10%, and unacceptable for values ±10%.

A successful assaying program aims at having good to reasonable bias and good to reasonable reproducibility of the CRM assays. The reproducibility can be assessed by the failure rate of the CRM assays considered to be good at <5%, reasonable between 5 and 10%, and unacceptable at >10%. In addition, comparison of the coefficient of variation (CV) of the CRM assays with the reported CV in the CRM certificate is a good guide to assess the variability of the assays at a given laboratory.

The analytical bias is good for copper and good to reasonable for zinc. The CRM failure rate for both base metals is <3%. The laboratory coefficient of variation (CV) for copper and zinc is in most cases better than or comparable to the variation documented by the CRM certificates.

The bias for gold and silver is good to reasonable. In addition, the CRM failure rate is <4% for gold and <6% for silver. The laboratory coefficient of variation (CV) for gold and silver is comparable to the variation reported in the CRM certificates.

It is concluded that the accuracy and reproducibility of copper, zinc, silver, and gold assays, as indicated by the CRMs assayed at Bureau Veritas laboratories, is of good quality for resource estimation.

12.1.3 Duplicates

The following analysis is based on coarse duplicates analyzed during 2015 and 2016. There is no record of coarse duplicates between 2012 and 2014. Coarse duplicates, approximately one in every twenty samples, were submitted to Bureau Veritas laboratory in order to monitor sub-sampling precision. After crushing to 10 mesh (2mm), a coarse duplicate sub-sample was riffle split and pulverized to ≥85% passing through 200 mesh (75µm). The duplicate sample was analyzed immediately after its paired sample. Quarter-core twin sample duplicates and pulp duplicates were not analyzed in the sample stream.

Coarse duplicates were evaluated using the hyperbolic method developed by AMEC (Simón, 2004). Minimum and maximum element concentrations of the sample pairs are plotted in the y and x axis, respectively. In the Minimum-Maximum diagrams all samples plot along and above the y = x line and the failure boundary is given by the hyperbolic equation $y^2=m^2x^2+b^2$.

The coarse duplicates were evaluated using a failure boundary that asymptotically approaches the line with slope m corresponding to a 20% absolute relative error (RE), and an intercept b representing the practical detection limit, set at 5 times the lower limit of detection (LOD) of the laboratory. The RE, expressed in percentage, is calculated as the absolute value of the pair difference divided by the pair average. An acceptable level of sub-sampling variance is achieved when the failure rate does not exceed 10% of the total pairs (Table 12-6).

TABLE 12-6: SUMMARY OF COARSE DUPLICATE PERFORMANCE

Analyte	Duplicate Samples	Duplicate Failures	Total Rate of Failures	Practical Detection Limit	Accepted Absolute Relative Error (RE)
Cu	1035	17	1.6%	50 ppm	20%
Zn	677	7	1.0%	500 ppm	20%
Ag	436	30	6.9%	2.5 ppm	20%
Au	962	53	5.5%	0.05 ppm	20%

Table 12-6 and Figure 12-1 to Figure 12-4 show the results of the coarse duplicates submitted to Bureau Veritas. The duplicate pairs display failure rates ranging between 1 and 7% for copper, zinc, silver, and gold when evaluated by the hyperbolic method for an absolute relative error of 20%. The low failure rates indicate that a good level of sub-sampling variance was achieved by Bureau Veritas laboratories for copper, zinc, silver, and gold assays. Therefore, it is concluded that the sub-sampling procedures were adequate for all metals used in the resource model.

FIGURE 12-1: COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

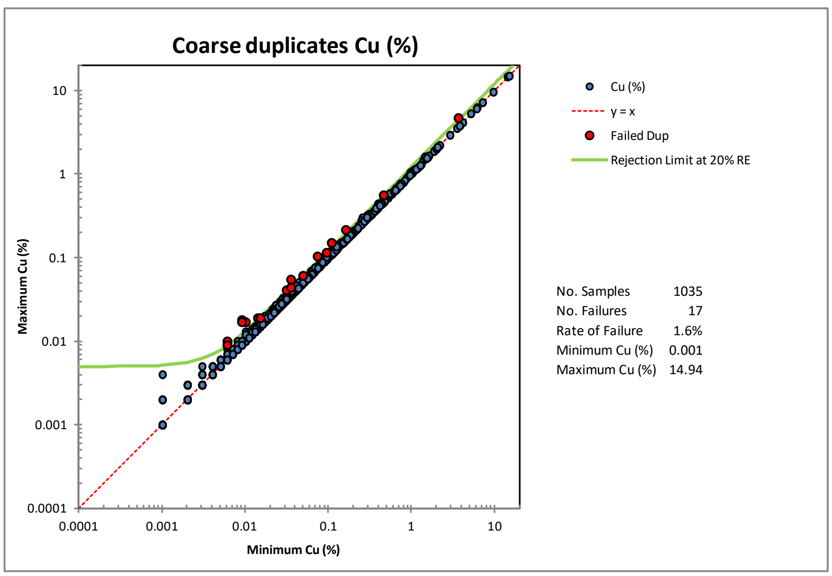


FIGURE 12-2: ZINC COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

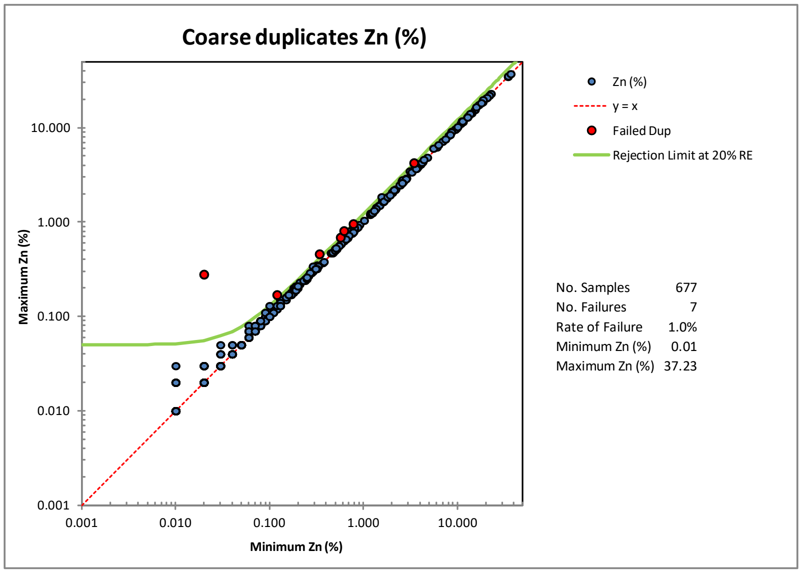


FIGURE 12-3: SILVER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

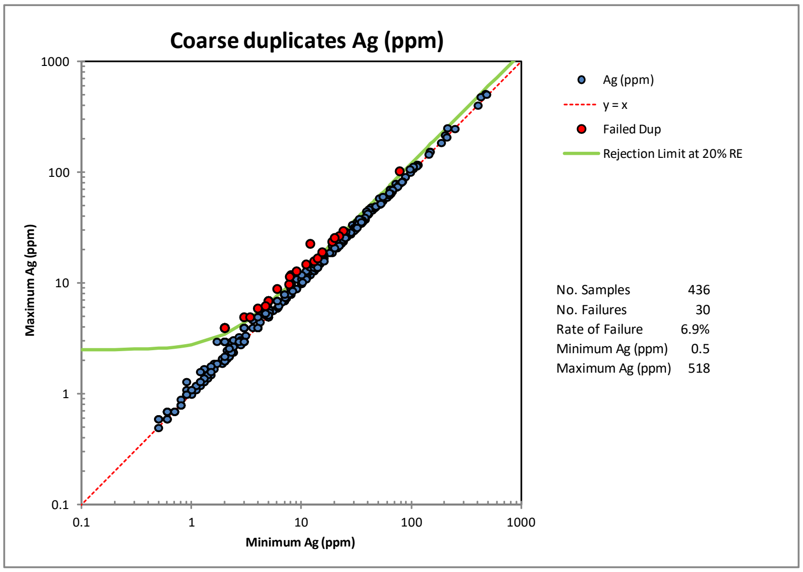
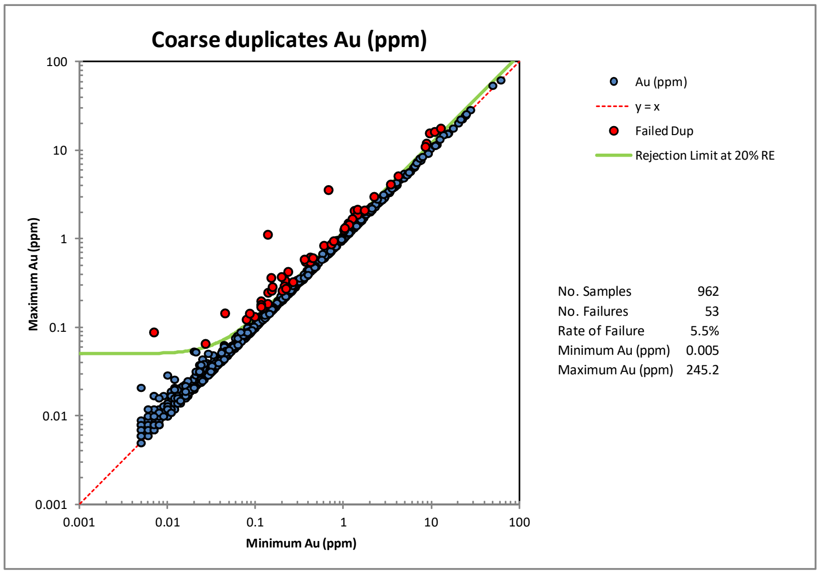


FIGURE 12-4: GOLD COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT



12.2 Hudbay Laboratory Methods and QAQC

From 2012 to 2016 a total of 104,024 drill core samples were analyzed at the Hudbay laboratory in Flin Flon, Manitoba. Copper, zinc, and silver were digested in aqua regia and analyzed by ICP-OES (Table 12-7). Gold was determined by lead-collection fire assay fusion, for total sample decomposition, followed by atomic absorption spectroscopy (AAS) analysis. Fire assays were performed on 15 to 30 g subsample pulps to avoid problems due to potential nuggetty gold.

TABLE 12-7: HUDBAY LABORATORY ASSAYS SPECIFICATIONS

Element	Unit	Lower Detection Limit	Upper Detection Limit	Digestion	Instrumental Finish	Method Code
Cu	%	0.01	16.5	Aqua Regia	ICPA-OES	LAI-006
Cu overlimits	%	0.05	80	Aqua Regia	ICPA-OES	LAI-006
Zn	%	0.01	33	Aqua Regia	ICPA-OES	LAI-006
Zn overlimits	%	0.05	100	Aqua Regia	ICPA-OES	LAI-006
Ag	ppm	0.446	500	Aqua Regia	ICPA-OES	LAI-006
Ag overlimits	ppm	2.23	2500	Aqua Regia	ICPA-OES	LAI-006
Au	ppm	0.103	6.857	Fire Assay	AAS	LAI-044
Au overlimits	ppm	6.857		Litharge FA	Gravimetric	LAI-115

Quality control samples were systematically introduced in the sample stream to assess sub-sampling procedures, potential cross-contamination, precision, and accuracy. Hudbay sampling program commonly consists of 5% certified reference materials (CRM), 2% certified blanks, and 5% coarse duplicates. Blanks and CRMs are mostly from OREAS and a few high gold grade standards from Rocklabs. All QAQC samples were analyzed following the same analytical procedures as those used for drill core samples.

12.2.1 Blanks

Certified OREAS blanks F6 and F7 were inserted into the sample stream approximately one every fifty samples to monitor potential cross-contamination (Table 12-2). A total of 2,278 blanks were inserted, representing 2.2% of the total number of samples analyzed at the Hudbay laboratory. The performance of the certified blanks was assessed using the same protocols explained in detail on Section 12.1.1.

A total of 218 F6 blanks and 2,060 F7 blanks were systematically inserted along with the drill core samples analyzed at the Hudbay laboratory in Flin Flon. A summary of the blank performance is shown on Table 12-8.

TABLE 12-8: SUMMARY OF BLANK PERFORMANCE

Coarse Blank F6					
Analyte	No. Blanks	Failed Blanks	Contamination Rate	Maximum Contamination	Average Contamination
Cu	218	5	2.3%	160 ppm	80 ppm
Zn	218	67	30.7%	1040 pm	170 ppm
Ag	218	7	3.2%	0.929 ppm	0.51 ppm
Au	218	18	8.3%	0.71 ppm	0.12 ppm

Coarse Blank F7					
Analyte	No. Blanks	Failed Blanks	Contamination Rate	Maximum Contamination	Average Contamination
Cu	2060	36	1.7%	1340 ppm	130 ppm
Zn	2060	240	11.7%	2680 ppm	195 ppm
Ag	2060	12	0.6%	1.82 ppm	0.72 ppm
Au	2060	76	3.7%	2.53 ppm	0.23 ppm

Copper displays a low blank failure rate of <3%. The failure rate for zinc is high with 12 to 31% failed blanks. However, the grades of contaminated blanks are low with an average of 80 to 130 ppm for copper and 170 to 195 ppm for zinc. There are a few cases of contamination with moderate grades in the range of 0.1 to 0.13% copper, and 0.1 to 0.27% Zn.

The blank failure rate for silver is <5%. On average, silver contamination is in the range 0.5 to 0.7 ppm. A few blanks are contaminated with 0.9 to 1.8 ppm silver.

The blank failure rate for gold is 3 to 8% which is acceptable. The contamination with gold is high, relative to the blanks, with average grades ranging from 0.5 to 0.7 ppm. A few cases of contamination with up to 1.8 ppm gold are documented.

It is recommended to audit the laboratory and request a cleaner assaying protocol to (1) reduce the contamination with copper and zinc to levels below 0.1%, and (2) to reduce the average contamination with gold and silver to levels below 0.1 ppm. These cleaner laboratory conditions are achievable and desired for base and precious metals.

Overall the performance of blanks indicates no significant issues with contamination at the Hudbay laboratory in Flin Flon. The contamination rates are generally low for copper, silver and gold, and high for zinc but at grade levels that are not economic. Therefore the results are acceptable for the resource estimation.

12.2.2 Standards

From 2012 to 2016 a total of 5,570 OREAS and Rocklabs CRMs were analyzed representing 5.4% of the total samples submitted to the Hudbay laboratory. Table 12-4 presents a list of all CRMs used by Hudbay for quality control over the past five years.

OREAS standards are a series of matrix-matched polymetallic CRMs prepared exclusively for Hudbay and sourced from the Flin Flon mine ore bodies. Rocklabs standards are a series of commercially available high-grade gold standards.

Section 12.1.2 explains the methodology for estimation of bias and assessment of the CRM performance. In summary, a successful assaying program aims at having a bias better than ±10%, and good reproducibility of the CRM assays given by a rate of failed CRMs <10%. Table 11-9 summarizes the performance of the CRMs at the Hudbay laboratory in Flin Flon.

The analytical bias is good for copper, and good to reasonable for zinc (Table 12-9). The CRM failure rate is <8% for copper and <5% for zinc. The laboratory coefficient of variation for copper and zinc is better than or comparable to the variation indicated by the CRM certificates (Table 12-9). These results suggest that the copper and zinc assays at the Hudbay laboratory have good accuracy and reproducibility for resource estimation.

The analytical bias for silver is good to reasonable for all standards except OREAS A6 which has a bias of 20%. However, the certified SD of OREAS A6 indicates a coefficient of variation of 20% for this standard, and the average silver value at Hudbay for OREAS A6 is within 2SD of the certified value. In addition, the silver grade of this standard (0.535 ppm) is close to the lower limit of detection for silver (0.4 ppm) at the Hudbay laboratory (Table 12-7 and Table 12-9). The failure rates for CRMs assayed for silver, including OREAS A6, are <4%. The coefficients of variation of the silver determinations at the laboratory are better than or comparable to the variation indicated in the CRM certificate. It is concluded that the accuracy and reproducibility of the silver assays at the Hudbay laboratory is of good quality for resource estimation.

Several problems were documented by the analysis of CRMs assayed for gold at the Hudbay laboratory in Flin Flon.

First, the relative bias ranges from good to reasonable, better than ±7%, for all OREAS CRMs (Table 12-9). However, negative biases of up to -20% are estimated for the high gold grade Rocklab standards. Rocklab standard SN75 was also analyzed at Bureau Veritas where the estimated bias was -0.5%, versus a bias of -10.8% at the Hudbay laboratory. Therefore, the large negative bias in the high-grade gold standards needs to be investigated with the laboratory in Flin Flon.

Second, the variability of gold indicated by the CRMs assayed at the Hudbay laboratory is relatively large. For instance, the CRM gold values determined by Hudbay have coefficients of variation (CV) ranging from 8 to 24%, which is high relative to the indicated CVs in the CRM certificates ranging from 2 to 7%, and the CRMs assayed for gold at Bureau Veritas with CVs in the range of 2 to 9%.

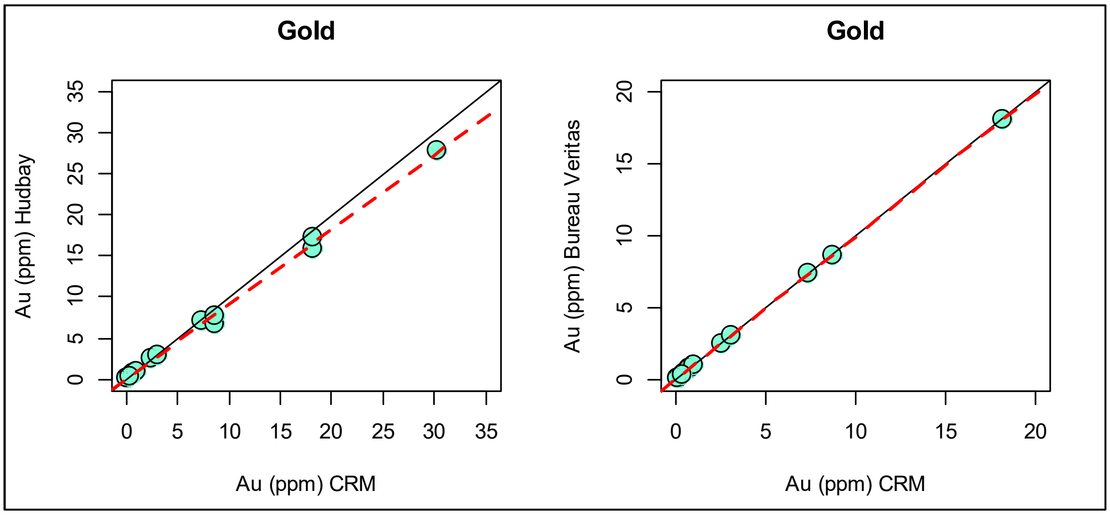
The variability in the low gold grade (<1 ppm gold) OREAS CRMs can be explained by the relatively high lower limits of detection (0.1 ppm LOD) for gold at the Hudbay laboratory. Most of these low grade standards are within 10 times the LOD and therefore large variance is expected. However, the laboratory LOD does not satisfactorily explain the large variability of gold recorded by the assays of high gold grade Rocklab CRMs with CVs ranging from 10 to 17%. These variations are larger than the expected variation based on the CRMs certificates. The large variation in the determination of gold at the Hudbay laboratory results in a large number of CRM failures exceeding acceptable thresholds. This problem with the high-grade gold standards needs to be addressed with the laboratory.

To further investigate the impact of the poor performance of gold CRMs at the laboratory in Flin Flon, an ordinary least square regression of the average CRM values obtained at the laboratory was fit onto the recommended best values reported in the CRM certificates. For comparison, the same regression analysis was conducted for the average CRM assays from Bureau Veritas. The results are summarized on Figure 12-5 and Table 12-10.

TABLE 12-9: SUMMARY OF CRM PERFORMANCE AT HUDBAY LABORATORY

Cu (%)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (%)	HBMS Average	CRM CV	Lab CV	Relative Bias
OREAS A5	78	6	7.7%	0.0945	0.0908	3%	3%	-3.9%
OREAS B5	78	0	0%	1.33	1.31	3%	2%	-1.9%
OREAS C5	76	1	1.3%	3.37	3.28	6%	2%	-2.7%
OREAS D5	76	1	1.3%	9.42	9.47	4%	2%	0.5%
OREAS E5	76	0	0%	0.393	0.384	8%	2%	-2.2%
OREAS A6	761	2	0.3%	0.055	0.053	4%	9%	-4.3%
OREAS B6	747	6	0.8%	0.872	0.855	2%	2%	-2.0%
OREAS C6	748	7	0.9%	2.05	2.01	3%	2%	-2.1%
OREAS D6	747	4	0.5%	4.20	4.00	4%	2%	-4.8%
OREAS E6	736	5	0.7%	0.245	0.239	5%	3%	-2.3%
Zn (%)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	HBMS Average	CRM CV	Lab CV	Relative Bias
OREAS A5	78	0	0%	0.506	0.496	5%	3%	-2.1%
OREAS B5	78	2	3%	0.125	0.130	7%	8%	3.7%
OREAS C5	76	0	0%	5.28	5.09	11%	2%	-3.6%
OREAS D5	76	1	1%	2.46	2.28	7%	3%	-7.2%
OREAS E5	76	1	1.3%	23.70	23.77	4%	2%	0.3%
OREAS A6	761	17	2%	0.039	0.040	5%	10%	3.2%
OREAS B6	747	2	0.3%	0.702	0.665	3%	2%	-5.3%
OREAS C6	748	3	0.4%	2.41	2.30	3%	2%	-4.5%
OREAS D6	747	4	0.5%	3.31	3.10	4%	2%	-6.3%
OREAS E6	736	6	0.8%	18.14	18.06	4%	2%	-0.4%
Au (ppm)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	HBMS Average	CRM CV	Lab CV	Relative Bias
OREAS A5	72	39	54.2%	0.141	0.140	7%	24%	-0.7%
OREAS B5	79	58	73.4%	0.476	0.460	3%	20%	-3.4%
OREAS C5	76	1	1%	2.493	2.524	2%	5%	1.2%
OREAS D5	76	1	1%	7.344	7.107	4%	8%	-3.2%
OREAS E5	76	1	1.3%	0.780	0.813	3%	8%	4.3%
OREAS A6	569	569	100%	0.121	0.128	4%	23%	6.2%
OREAS B6	738	487	66.0%	0.706	0.676	2%	15%	-4.2%
OREAS C6	742	371	50.0%	0.987	1.007	2%	13%	2.0%
OREAS D6	741	24	3.2%	3.04	2.98	4%	8%	-2.0%
OREAS E6	731	428	58.5%	0.316	0.328	4%	13%	3.9%
Rocklabs SN60	181	170	93.9%	8.595	6.804	3%	13%	-20.8%
Rocklabs SN75	497	261	52.5%	8.671	7.734	2%	17%	-10.8%
Rocklabs SP59	187	127	67.9%	18.12	15.891	2%	15%	-12.3%
Rocklabs SP73	490	134	27.3%	18.17	17.28	2%	10%	-4.9%
Rocklabs SQ48	92	50	54.3%	30.25	27.84	2%	10%	-8.0%
Ag (ppm)								
CRM	No. of Samples	No. of Failures	Failure Rate (%)	CRM Value (ppm)	HBMS Average	CRM CV	Lab CV	Relative Bias
OREAS A5	78	0	0%	1.91	2.03	7%	15%	6.4%
OREAS B5	72	2	2.8%	2.86	2.98	15%	8%	4.2%
OREAS C5	76	0	0%	21.1	21.33	11%	6.1%	1.1%
OREAS D5	76	2	3%	90.0	88.61	13%	6.0%	-1.5%
OREAS E5	76	1	1.3%	19.7	20.37	10%	6.6%	3.4%
OREAS A6	634	4	0.6%	0.535	0.642	20%	21.2%	20.1%
OREAS B6	747	4	0.5%	2.94	2.81	18%	6.6%	-4.3%
OREAS C6	748	4	0.5%	5.18	5.02	10%	4.6%	-3.1%
OREAS D6	747	3	0.4%	25.17	23.72	6%	3.2%	-5.7%
OREAS E6	736	7	1.0%	15.23	15.03	6%	3.4%	-1.3%

FIGURE 12-5: XY COMPARISON OF THE AVERAGE GOLD VALUES DETERMINED BY HUDBAY AND BUREAU VERITAS LABORATORIES ON CRMS VERSUS THE CRM RECOMMENDED BEST VALUE*



*Ordinary least square regression in red, and the y=x line in black

The 95% confidence interval (CI) of the regression intercepts for both Hudbay and Bureau Veritas laboratories contain the zero value indicating that the intercepts are not significant and do not carry a major weight in the analysis of bias (Table 12-10).

TABLE 12-10: SUMMARY OF REGRESSION PARAMETERS

Analyte	Laboratory	Method	Intercept	Slope	95% CI-Intercept		95% CI-Slope		Bias
Au	HBMS	OLS	0.016	0.91	-0.285	0.317	0.89	0.94	-9%
Au	Bureau Veritas	OLS	0.011	0.99	-0.012	0.034	0.99	1.00	-1%

The average bias was calculated from the regression analysis as Bias (%) = OLSS-1; in which OLSS is the slope of the ordinary least square regression (OLS). Evidently, the Hudbay laboratory carries an average negative bias for gold of -9%, which may range between -11% and -6% (95% CI). In contrast, Bureau Veritas laboratory carries a very small negative bias of -1%, ranging from 0 to -1%. In summary, the laboratory in Flin Flon may underestimate the gold grades by -9% ±2.5 in the range of 0.1 to 30 ppm gold (Table 12-10, Figure 12-5).

It is concluded that the analytical accuracy and reproducibility of copper, zinc, and silver as indicated by the CRM analysis, at the Hudbay laboratory is appropriate for resource estimation. Gold grades are being under assayed and this issue needs to be discussed with the laboratory. This under assaying of gold standards will likely lead to an underestimation of gold in the resource estimate, since the proportion of samples assayed at the Hudbay laboratory is approximately 80% of the total samples assayed between 2012 and 2016.

12.2.3 Duplicates

The following analysis is based on coarse duplicates analyzed during 2015 and 2016. There is no record of coarse duplicates between 2012 and 2014. Coarse duplicates, approximately one in every twenty samples, were submitted to the Hudbay laboratory in Flin Flon to monitor sub-sampling precision. After crushing to 10 mesh (2 mm), a coarse duplicate sub-sample were split and pulverized to ≥95% passing through 150 mesh (105 µm). The laboratory in Flin Flon uses riffle splitting and rotator splitting. However, rotator splitting is a relatively new feature in the laboratory and most of the samples discussed here were riffle split. The duplicate sample was analyzed immediately after its paired sample. Quarter-core twin sample duplicates and pulp duplicates were not analyzed in the sample stream.

Coarse duplicates were evaluated using the hyperbolic method developed by AMEC (Simón, 2004) and described in detail in Section 12.1.3. The coarse duplicates submitted at Hudbay laboratory were also evaluated using a failure boundary of 20% absolute relative error (RE). An acceptable level of sub-sampling variance is achieved when the failure rate does not exceed 10% of all sample pairs. Table 12-11 summarizes the results of duplicate analysis at the Hudbay laboratory and minimum and maximum plots are presented on Figures 12-6 to 12-9.

TABLE 12-11: SUMMARY OF COARSE DUPLICATE PERFORMANCE

Analyte	Duplicate Samples	Duplicate Failures	Total Rate of Failures	Practical Detection Limit	Accepted Absolute Relative Error (RE)
Cu	2208	16	0.7%	500 ppm	20%
Zn	2124	37	1.7%	500 ppm	20%
Ag	2173	76	3.5%	2 ppm	20%
Au	1456	162	11.1%	0.5 ppm	20%

The duplicate pairs display failure rates of <5% for copper, zinc, and silver indicating that the sub-sampling procedures employed by the Hudbay laboratory are of good quality for base metals and silver (Table 12-11, Figures 12-6 to 12-8).

Gold duplicate analysis display a failure rate of 11% when evaluated at 20% RE and a practical detection limit of 0.5ppm corresponding to five times the lower limit of detection (Table 12-11, Figure 12-9). When the gold duplicates are evaluated at a practical detection limit of 1 ppm, 10 times the LOD, and 20% RE, the failure rate is 7% which is acceptable. It is concluded that an adequate sub-sampling variance for gold at the laboratory in Flin Flon is reached only for gold grades above 1ppm.

FIGURE 12-6: COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

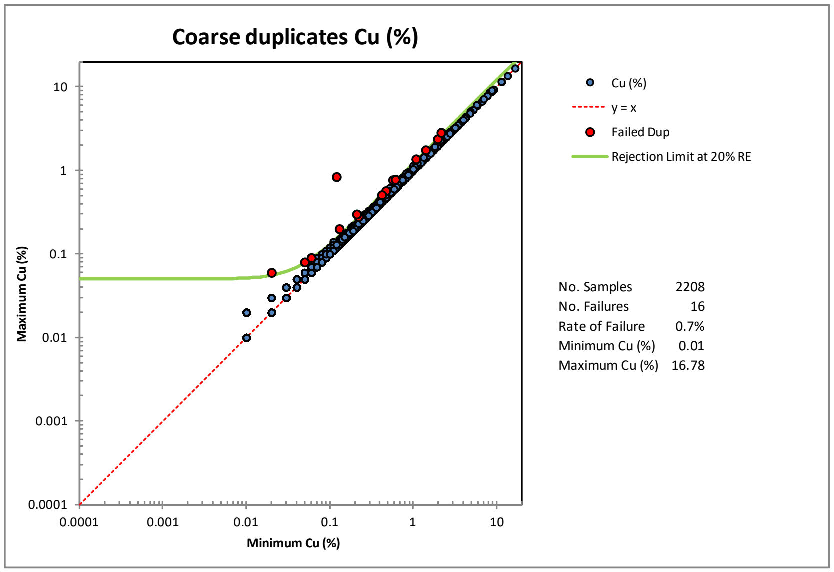


FIGURE 12-7: ZINC COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

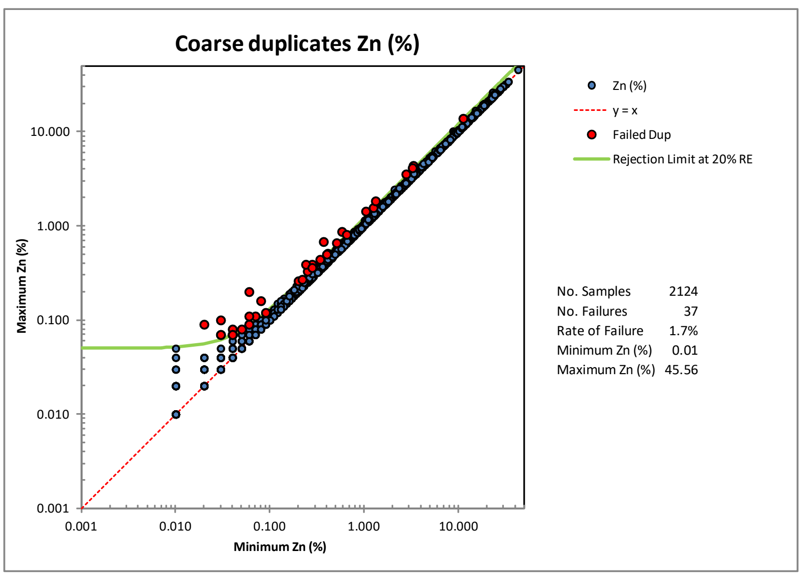


FIGURE 12-8: SILVER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

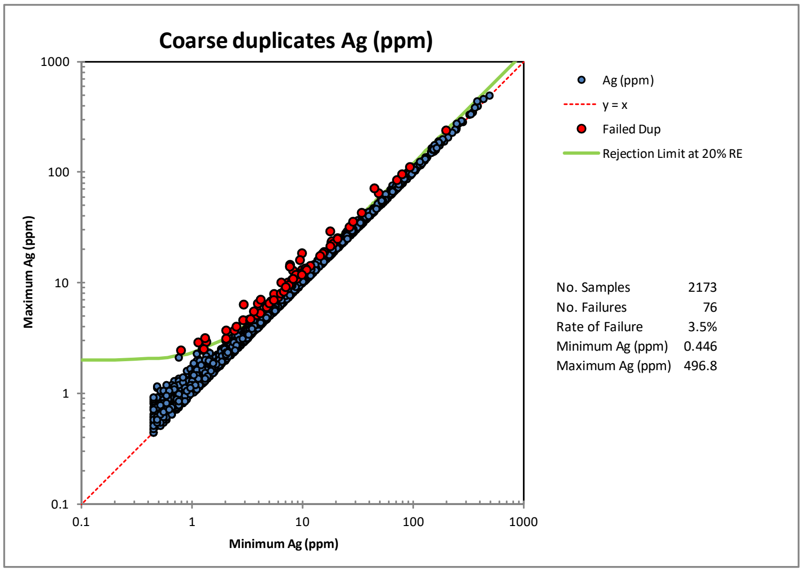
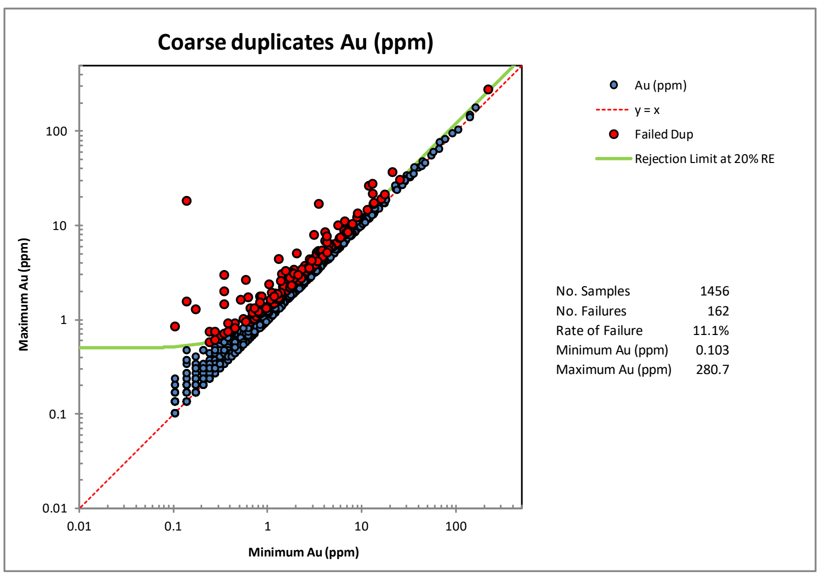


FIGURE 12-9: GOLD COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT



12.3 Check Assaying

The check assays of duplicate pulp samples evaluated here only represent sample batches submitted to Bureau Veritas and Hudbay between 2015 and 2016. Check assays are not available for samples submitted before these dates.

A total of 304 representative pulp samples (1.5%) were selected and re-analyzed at SGS Canada Inc. (SGS) laboratory in Vancouver to assess the accuracy of assay results reported by Bureau Veritas (BV) and the Hudbay laboratory in Flin Flon relative to the umpire laboratory SGS. Only samples with ≥0.5ppm gold were submitted for re-analysis at the secondary laboratory. Copper, zinc, and silver were digested in aqua regia and analyzed by ICP-OES. Gold was fire assayed and analyzed by AAS. These methods are comparable to those used by Bureau Veritas and the Hudbay laboratory.

CRMs, certified blanks, and pulp duplicates were not inserted along with the check samples, which leaves the secondary laboratory, SGS, untested for its quality.

A Reduced-to-Major-Axis regression (RMA) was used to evaluate the check samples. Assays from the primary laboratory were regressed onto the umpire laboratory. The RMA regression calculates an unbiased fit for values that are independent from each other. The results of the regression analysis are presented on Table 12-12 and Figures 12-10 and 12-11.

TABLE 12-12: SUMMARY OF RMA REGRESSION ANALYSIS

Analyte	Laboratory	Method	R.square	Intercept	Slope	95% CI-Intercept		95% CI-Slope		Bias
Cu	Bureau Veritas	RMA	0.999	0.000	1.003	-0.015	0.015	0.995	1.011	0.3%
Zn	Bureau Veritas	RMA	0.997	0.005	0.992	-0.003	0.013	0.979	1.006	-0.8%
Ag	Bureau Veritas	RMA	0.981	0.992	0.997	0.066	1.887	0.962	1.032	-0.3%
Au	Bureau Veritas	RMA	0.992	0.188	0.990	-0.009	0.380	0.970	1.010	-1.0%
Cu	HBMS	RMA	0.998	-0.018	1.044	-0.025	-0.011	1.038	1.050	4.4%
Zn	HBMS	RMA	0.998	0.013	0.975	0.008	0.017	0.970	0.981	-2.5%
Ag	HBMS	RMA	0.989	-1.305	1.031	-1.962	-0.659	1.015	1.046	3.1%
Au	HBMS	RMA	0.989	-0.462	1.036	-0.586	-0.340	1.022	1.051	3.6%

The coefficient of determination (R squared, R^2) is used to assess the variance explained by the linear relationship between the pairs. Samples analysed at Hudbay and Bureau Veritas laboratories show a very good fit (R^2 > 0.98), relative to SGS, for copper, zinc, silver, and gold.

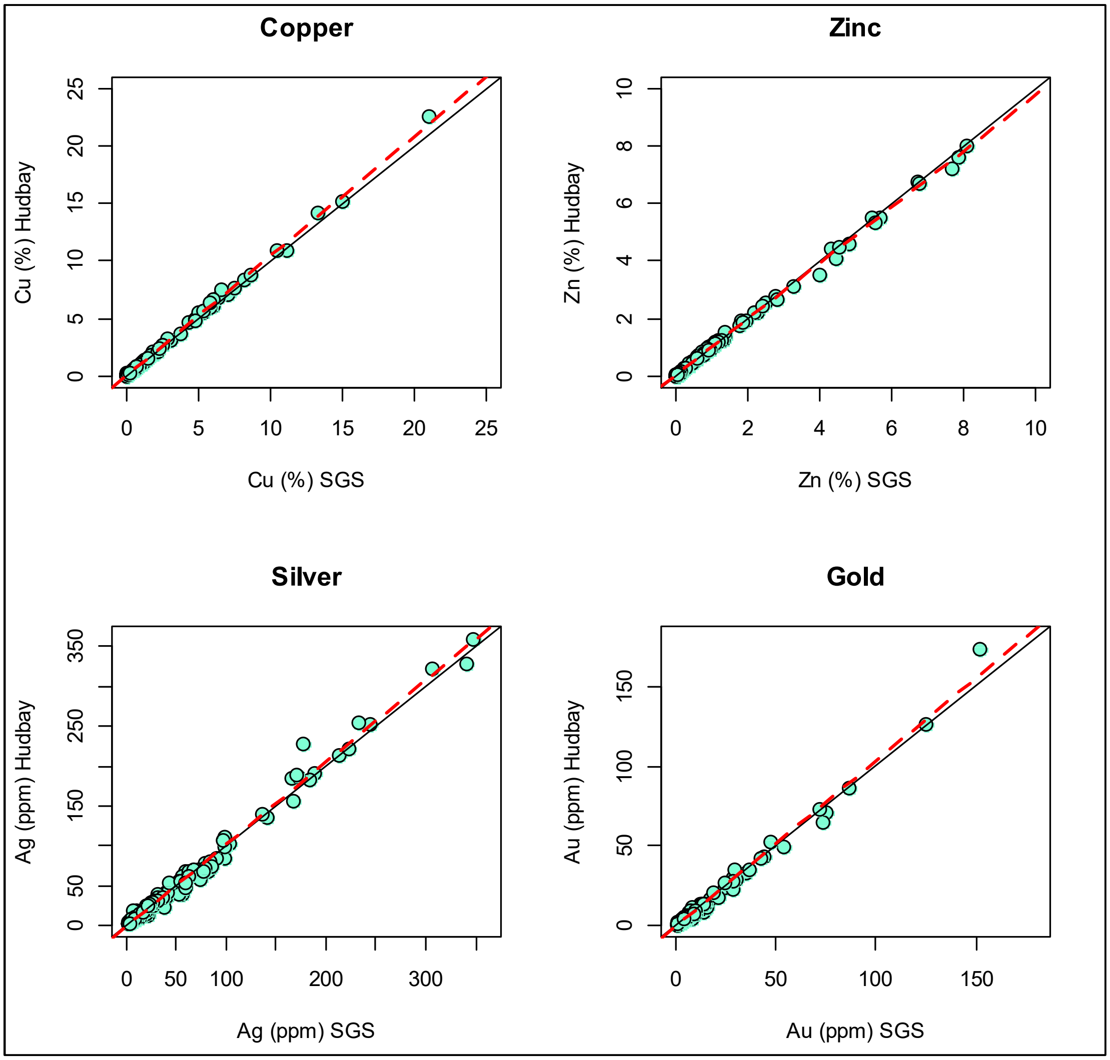
For Bureau Veritas-SGS pairs and Hudbay-SGS pairs, the 95% confidence interval of the regression intercepts include zero or are within five times the lower limit of detection indicating that the effect of the intercept on the analysis of bias is not significant.

The bias, expressed as a percent, is calculated as Bias (%) = RMAS-1; in which RMAS is the slope of the RMA regression. The slope of the RMA regression for the pairs analyzed at Bureau Veritas and SGS ranges between 0.990 and 1.003, and between 0.975 and 1.044 for those analyzed at the Hudbay laboratory and SGS. The estimated bias, relative to SGS, for all base and precious metals is < ±2% for Bureau Veritas and < ±5% for Hudbay laboratory.

The RMA regression results are illustrated on Figures 12-10 and 12-11, where the black line represents the y = x function and the red dashed line represents the RMA regression line.

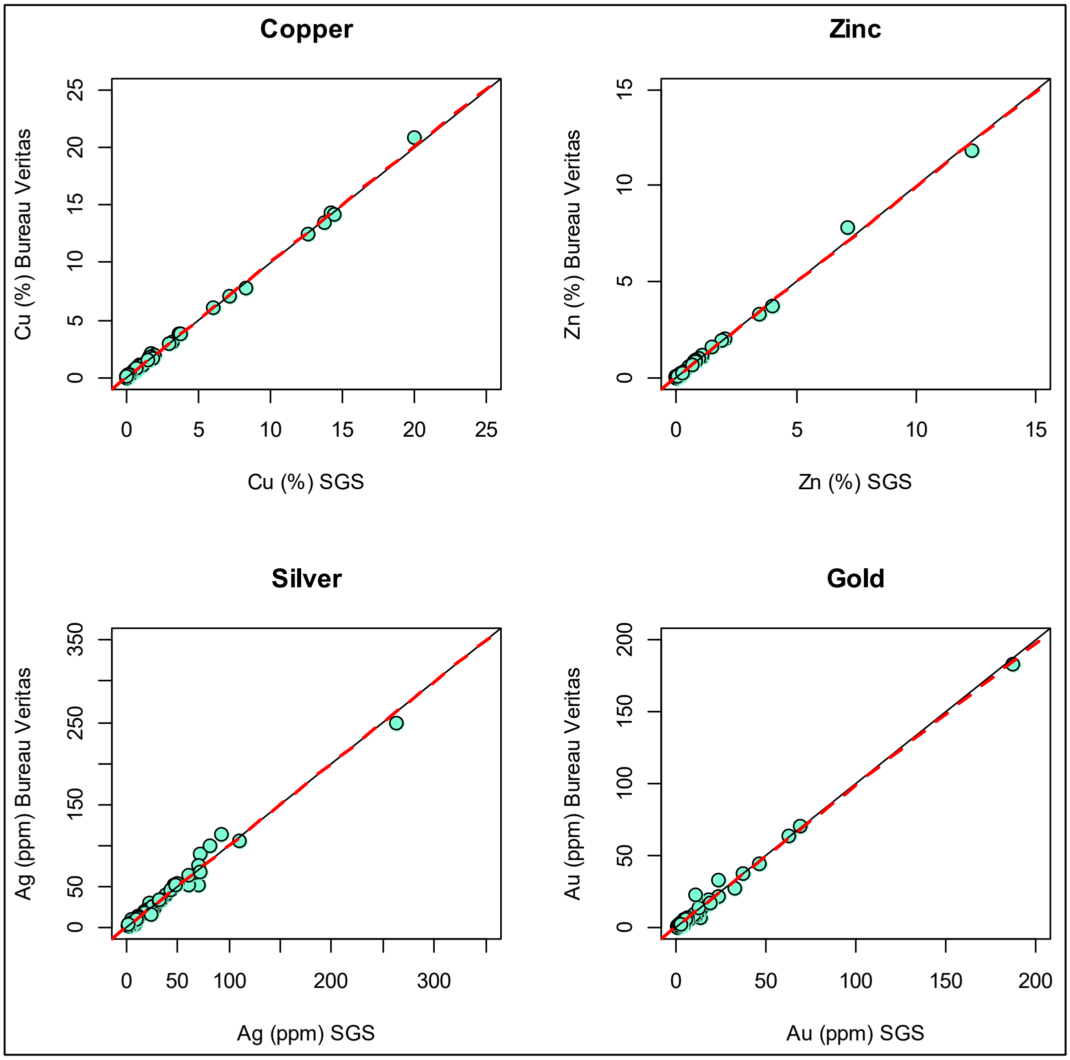
The overall bias estimated by the RMA regression analysis, regression intercepts, and r-squared indicates that the accuracy achieved by Bureau Veritas and Hudbay for copper, zinc, silver, and gold, during 2015 and 2016, is of good quality for resource estimation.

FIGURE 12-10: XY PLOTS OF CHECK ASSAY DATA COMPARING PRIMARY LABORATORY HUDBAY TO SECONDARY LABORATORY SGS*



* The RMA Regression Line in Red, and the Y=X Function in Black.

FIGURE 12-11: XY PLOTS OF CHECK ASSAY DATA COMPARING PRIMARY LABORATORY BUREAU VERITAS TO SECONDARY LABORATORY SGS



* The RMA Regression Line in Red, and the Y=X Function in Black.

12.4 Site Visit

Robert Carter, P.Eng., Lalor Mine Manager at Hudbay Manitoba Business Unit is a regular employee at Lalor and in his role continually conducts personal site inspections to become familiar with conditions in the mine, to observe the geology and mineralization and verify work completed. He last visited the mine on March 29, 2017.

12.5 Core Review

Robert Carter, P.Eng., reviewed the geological data and verifies drill core mineralization during his personal site inspections.

12.6 Drilling Database

Drill hole data (header, down hole surveys, geological intervals, sample intervals, QAQC samples, and geotechnical details) is entered into the local acQuire database through various data entry (DE) objects. The core logger enters geological data into the acQuire database via a customized core logging DE object. The DE object is designed to follow the work flow of the data entry process and applies built in business rules and pick lists to ensure all data is entered consistently. This is essentially a preliminary validation check before data is committed to the database. Therefore all data such as: reported lengths, geology intervals, and sample intervals are automatically validated on entry to prevent overlapping intervals, duplicate sample numbers, spelling errors, etc. Specific gravity (SG) results, if measured in the core shack, are stored within Excel spreadsheets and emailed to the database manager. The SG values are uploaded in acQuire using a customized interface tool.

Once the core logging is complete the drill hole data is reviewed and approved by the senior mine geologist or a designate. This includes signing approvals that are built in various validation/verification objects to ensure all data (collar details, surveys, geology, sample details, etc) is checked thoroughly and is complete. The Hudbay database manager also does routine checks and reports any inconsistencies/discrepancies to the attention of the senior geologist or the core loggers. Once the senior geologist is satisfied with the drill hole input data, diamond drill reports are generated within acQuire and stored in the local directory in PDF format.

On receipt of the analytical results from the Hudbay Flin Flon laboratory and Bureau Veritas laboratory, the assay files (certificates, csv files, etc) are inspected prior to uploading into the acQuire database. QC results are examined and any discrepancies are flagged and reported either back to the laboratory or to the database manager. The Hudbay database manager also compares the analytical results to the logged visual estimates for copper, zinc and iron. Discrepancies are brought to the attention of the senior geologist and re-assaying may be requested if the significance of the interval is warranted. Once the preliminary checks are complete, the analytical results (including laboratory measured SG values) for assay intervals are uploaded directly into the acQuire database using the sample number as the unique identifier. The database manager informs the senior geologist after the assays have been uploaded in the database. The senior geologist then completes the assay verification and approval process.

QC and duplicate assay results from both laboratories are kept in the acQuire database as unique identified variables. This information is readily available and can be easily charted externally or directly within the database system.

Drill core from the initial surface drilling program is stored at the Hudbay Hangar site near Flin Flon and a designated core storage location near the Stall concentrator. Drill core from the underground drilling programs is stored at the Lalor mine core storage area near the Chisel North mine site. Fine pulp rejects are kept in perpetuity in lidded plastic pails at the Hudbay Hangar and at the Lalor core storage area in sea cans.

12.7 Mineral Resource Database Management

All information used in the estimation of the mineral resources was extracted directly from the Hudbay acQuire database management system managed by the database manager. The MineSight software package used for the 3D modeling and grade interpolation downloads information directly from this database.

In the author's opinion, the drill hole and assay database is acceptable for resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The Stall concentrator, located 16km from Lalor mine was commissioned in 1979. Since then, it has processed ore from many of the Snow Lake area mines. Prior to processing ore from Lalor mine, Stall concentrator processed ore from the Chisel North mine from 2000 to 2013 with similar mineralogy to that of Lalor. The processed ore from Chisel North mine produced only a zinc concentrate, while Lalor ore with increased copper and precious metal content warranted the refurbishment of the defunct copper circuit and an overall throughput increase as the Lalor mine ramped up production.

The Stall concentrator began processing ore from Lalor in August 2012, initially producing only a zinc concentrate and by October 2012 a copper concentrate. As Lalor increased ore production the mill underwent an initial expansion and by the summer of 2014 was capable of processing at 2,800 tpd throughput rate.

It is the author's opinion that actual plant metallurgical performance overrides previous metallurgical testing results, since the metallurgical blend from Lalor is not expected to materially change over the life of mine. It is appropriate to assume that previous actual performance recoveries are expected in the future.

13.2 Plant Metallurgical Performance

The Stall concentrator metallurgical performance milling of Lalor ore from October 2012 to December 2016 is shown in Table 13-1 and actual concentrate produced is shown in Table 13-2.

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

		Head Assays				Metal Recoveries			
Year	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Zn (%)**	**Au (%)**	**Ag (%)**	**Cu (%)**	**Zn (%)**
2012	59,787	1.33	15.31	0.60	9.23	47.6	55.5	68.0	94.0
2013	422,287	1.51	29.38	0.60	8.91	59.6	56.5	77.7	94.9
2014	526,015	2.31	24.05	0.89	8.49	53.3	50.8	71.4	93.7
2015	928,501	2.53	21.28	0.71	8.21	55.8	54.8	84.5	90.8
2016	1,089,530	2.25	21.67	0.63	7.03	57.4	56.4	82.1	92.8

TABLE 13-2: HISTORICAL PLANT CONCENTRATE PRODUCED

	Zinc Concentrate		Copper Concentrate			
Year	**Tonnes**	**Zn (%)**	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**
2012	10,260	56.4	1,450	28.0	383.3	18.5
2013	67,346	51.5	9,784	39.2	386.9	20.2
2014	81,946	51.1	16,518	39.2	388.4	20.1
2015	133,808	51.8	26,848	48.7	403.1	20.7
2016	138,056	51.5	27,299	51.6	488.2	20.7

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 metal recoveries, shown in Table 13-3 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. Optimal operating days are considered to be steady run and appropriate control of parameters.

TABLE 13-3: EXPECTED LOM RECOVERIES AT STALL CONCENTRATOR

	Metal Recoveries				Concentrate Grade	
Year	Au (%)	Ag (%)	Cu (%)	Zn (%)	Zn (%)	Cu (%)
2017	59.6	51.9	83.9	93.6	51.0	21.0
2018	53.5	46.7	83.9	91.8	51.0	21.0
2019	55.8	48.2	83.1	91.7	51.0	21.0
2020	57.9	57.5	88.2	90.0	51.0	21.0
2021	61.9	66.1	89.1	90.5	51.0	21.0
2022	62.3	66.9	88.6	91.8	51.0	21.0
2023	60.1	60.8	88.2	91.8	51.0	21.0
2024	55.3	47.6	86.1	93.5	51.0	21.0
2025	59.7	52.2	87.7	91.7	51.0	21.0
2026	52.6	38.8	84.7	92.0	51.0	21.0
2027	55.8	39.5	81.7	90.6	51.0	21.0

13.3 Metallurgical Testing

Although it is the author’s opinion that actual plant performance overrides previous metallurgical testing it is appropriate to summarize the relevant testing completed on Lalor mineralization prior to processing ore at Stall concentrator.

The following reports were used in the preparation of this summary:

1. A Report on the Recovery of Copper, Zinc, Gold and Silver from Lalor Samples
   SGS Vancouver Metallurgy
   July 20, 2009

2. An Investigation into the Recovery of Copper, Lead, Zinc and Gold from Lalor Samples – Phase II
   SGS Vancouver Metallurgy
   February 25, 2011

3. Pre-Feasibility Study Technical Report on the Lalor Deposit, Snow Lake, Manitoba, Canada.
   Effective Date: March 29, 2012

The primary objectives of the test programs were to develop an appropriate flowsheet for either the design of a new concentrator or modifications to the existing Stall concentrator, and to determine expected concentrate grades and metal recoveries.

All aspects of the test program including work on gold zone material were covered in the previous technical report, dated March 29th, 2012. 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. This summary only includes laboratory test results on base metal ores from Zone 10 and Zone 20, focussing on the locked cycle flotation tests done on the variability composites from these two zones.

These ore types have been processed at the Stall Concentrator for several years now, and plant recoveries from 2014 through 2016 will be compared to the earlier locked cycle flotation results.

Mineralogical analysis showed that chalcopyrite in Lalor ore is mostly coarse grained and liberated at grind sizes of approximately 100 microns. However, 15 to 20% of the chalcopyrite remains locked, primarily with sphalerite, at sizes below 20 microns. Copper is present almost exclusively as chalcopyrite with minor bornite. Zinc is present mainly as sphalerite, with minor amounts of gahnite. The sphalerite is coarse grained and liberated at a grind size of 250 microns. Lead is present as fine grained galena and would require a grind size of 70 microns for liberation. However there is insufficient galena in the ore to warrant a primary grind this fine.

Ore hardness was measured with Bond Work Index tests in the first phase of work. The average rod mill work index was 6.8 kwh/t on the base metals and 10.6 kwh/t on the contact gold material. The ball mill work indexes averaged 10.5 kwh/t on the base metals and 13.2 kwh/t on the contact gold material.

In the second phase of work at SGS, four variability composites were prepared from Zone 10 and Zone 20 ores as shown in Table 13-4.

TABLE 13-4: MAKE-UP OF VARIABILITY COMPOSITES

	Composite 1 **High grade Zn** **High grade Cu** **High grade Au**	**Composite 2** **High grade Zn** **Low grade Cu** **Low grade Au**	**Composite 3** **Low grade Zn** **High grade Cu** **High grade Au**	**Composite 4** **Low grade Zn** **Low grade Cu** **Low grade Au**
Percent by Zone				
10	35	75	40	34
20	65	25	60	66
Head Assays, g/t, %				
Au	2.98	0.29	4.29	1.14
Ag	21.10	12.30	21.30	16.70
Cu	0.95	0.22	0.96	0.42
Zn	11.00	10.30	5.23	4.42
Pb	0.23	0.23	0.21	0.26
Fe	22.90	23.20	17.10	16.30
S	24.60	22.80	16.60	14.70

The conditions for the locked cycle tests on the variability composites are shown in Table 13-5, and the results are shown in Table 13-6.

TABLE 13-5: LOCKED CYCLE TEST CONDITIONS – VARIABILITY COMPOSITES

Process	Conditions	Composite 1	Composite 2	Composite 3	Composite 4
Primary Grind	P_{80} Size, microns Water pH $NaCN/ZnSO_4$, g/t	79 Tap Water 8.6 20/60	80 Tap Water 8.6 20/60	82 Tap Water 8.6 20/60	80 Tap Water 8.6 20/60
Cu-Pb Bulk Roughing	3418A, g/t pH	40 9.5 - 9.6	40 9.5 - 9.6	40 9.5 - 9.6	40 9.5 - 9.6
Cu-Pb Bullk Conc Regrind	P_{80} Size, microns	30	30	30	30
Cu-Pb Bulk Cleaning	3418A, g/t pH No. of stages	15 10.5 3	15 10.5 3	15 10.5 3	15 10.5 3
Cu-Pb Separation	NaCN, g/t 3418A, g/t	300 4	400 4	400 4	400 4
Pb Cleaning	3418A, g/t pH No. of stages	4.5 10.5 4	4.5 10.5 4	4.5 10.5 4	4.5 10.5 4
Zn Roughing	$CuSO_4$, g/t_t Xanthate (SIPX), g/t pH	550 55 11.5	500 50 11.5	210 30 11.5	180 25 11.5
Zn Cleaning and 1st Cleaner Scavenging	Xanthate (SIPX), g/t pH No. of stages	6 11.5 3	2 11.5 3	2 11.5 3	3 11.5 3

TABLE 13-6: LOCKED CYCLE TEST RESULTS – VARIABILITY COMPOSITES

		Assays, % or g/t				Distribution, %			
Product		Composite 1	Composite 2	Composite 3	Composite 4	Composite 1	Composite 2	Composite 3	Composite 4
Feed	Wt					100	100	100	100
	Cu	0.95	0.25	0.90	0.44	100	100	100	100
	Pb	0.22	0.23	0.19	0.27	100	100	100	100
	Zn	10.76	10.30	5.27	4.17	100	100	100	100
	Au	2.87	0.20	3.40	0.93	100	100	100	100
	Ag	22.1	11.3	22.6	15.7	100	100	100	100
Lead Concentrate	Wt					1.27	1.00	1.17	1.51
	Cu	0.71	0.21	0.54	0.36	0.95	0.84	0.71	1.25
	Pb	14.68	20.36	12.86	15.22	84.72	87.07	76.19	85.89
	Zn	0.98	0.68	0.72	0.54	0.12	0.07	0.16	0.20
	Au	24.07	0.93	32.75	2.04	10.66	4.56	11.25	3.30
	Ag	145.0	196.0	154.0	151.0	8.4	17.3	8.0	14.6
Copper Concentrate	Wt					3.06	0.72	3.05	1.38
	Cu	27.40	24.30	27.60	27.20	88.52	70.27	93.32	86.49
	Pb	0.44	1.24	0.66	1.01	6.09	3.81	10.46	5.21
	Zn	6.52	7.71	3.32	3.64	1.85	0.54	1.92	1.21
	Au	58.74	12.43	62.60	36.79	62.66	43.66	56.04	54.60
	Ag	409.0	447.0	434.0	567.0	56.7	28.4	58.6	50.1
Bulk Concentrate	Wt					4.33	1.72	4.21	2.89
	Cu	19.57	10.28	20.10	13.20	89.47	71.11	94.03	87.74
	Pb	4.62	12.37	4.04	8.42	90.81	90.88	86.65	91.10
	Zn	4.89	3.62	2.60	2.02	1.97	0.61	2.08	1.41
	Au	48.57	5.74	54.32	18.66	73.32	48.22	67.29	57.90
	Ag	331.6	300.9	356.4	349.9	65.0	45.7	66.6	64.7
Zinc Concentrate	Wt					16.17	15.75	7.92	6.71
	Cu	0.22	0.19	0.26	0.27	3.70	12.25	2.29	4.16
	Pb	0.02	0.02	0.03	0.04	1.22	1.35	1.14	1.09
	Zn	60.55	60.97	57.42	57.95	90.99	92.36	86.38	93.24
	Au	0.23	0.06	0.78	0.32	1.28	4.36	1.82	2.30
	Ag	13.5	15.5	17.9	16.3	9.9	21.6	6.3	7.0
Zinc Cleaner Scavenger Tails	Wt					5.46	11.25	4.22	3.53
	Cu	0.15	0.36	0.34	0.45	2.13	6.60	1.67	2.78
	Pb	0.04	0.08	0.10	0.17	1.24	1.68	1.65	1.32
	Zn	3.23	4.16	3.45	3.26	3.14	3.53	3.33	2.92
	Au	0.15	3.70	1.41	5.31	2.85	8.02	4.59	5.34
	Ag	7.5	26.2	22.3	28.4	4.2	7.4	4.9	5.0
Zinc Rougher Tails	Wt					74.05	71.28	83.65	86.86
	Cu	0.08	0.03	0.02	0.03	4.69	10.03	2.02	5.32
	Pb	0.02	0.02	0.02	0.02	6.73	6.09	8.56	6.49
	Zn	0.57	0.38	0.52	0.12	3.90	2.61	8.21	2.43
	Au	0.88	0.11	1.07	0.37	22.75	39.40	26.31	34.46
	Ag	6.20	4.00	6.00	4.20	20.82	25.33	22.24	23.30

A copper/lead separation stage was in the laboratory flowsheet, but will not be in the plant flowsheet.

The copper concentrate plant product is equivalent to the bulk concentrate in Table 13-5.

Comparisons between laboratory locked cycle test (LCT) results and monthly average plant operating results are shown in Figure 13-1 to **Error! Reference source not found.** and discussed below.

Recoveries for copper and gold (Figure 13-1) in the plant are generally lower than the laboratory recoveries. This is probably due to the combined effects of the following factors:

- The ore blends in plant feeds are more variable and include small amounts of ore from zones that were not represented in the variability composites tested in the laboratory
- The plant grind is coarser than the laboratory grind (approximately 100 micron vs. 80 micron). The grind design criterion for the proposed concentrator upgrade project is 100 micron.
- The plant has been able to improve upon the concentrate grades that were achieved in the laboratory (Figure 13-2). Concentrate grades and recoveries are inversely related.

FIGURE 13-1: COPPER AND GOLD RECOVERY TO CONCENTRATE, LCT AND PLANT DATA

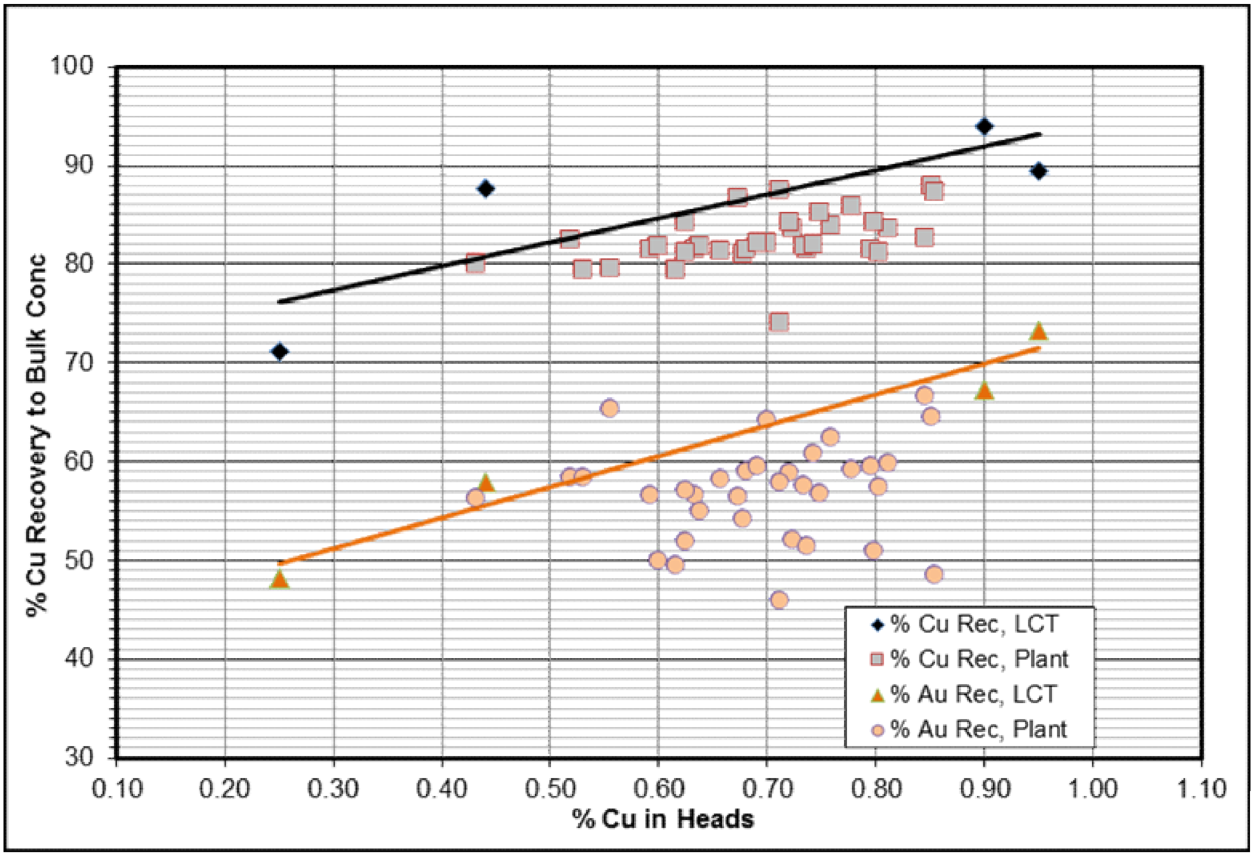
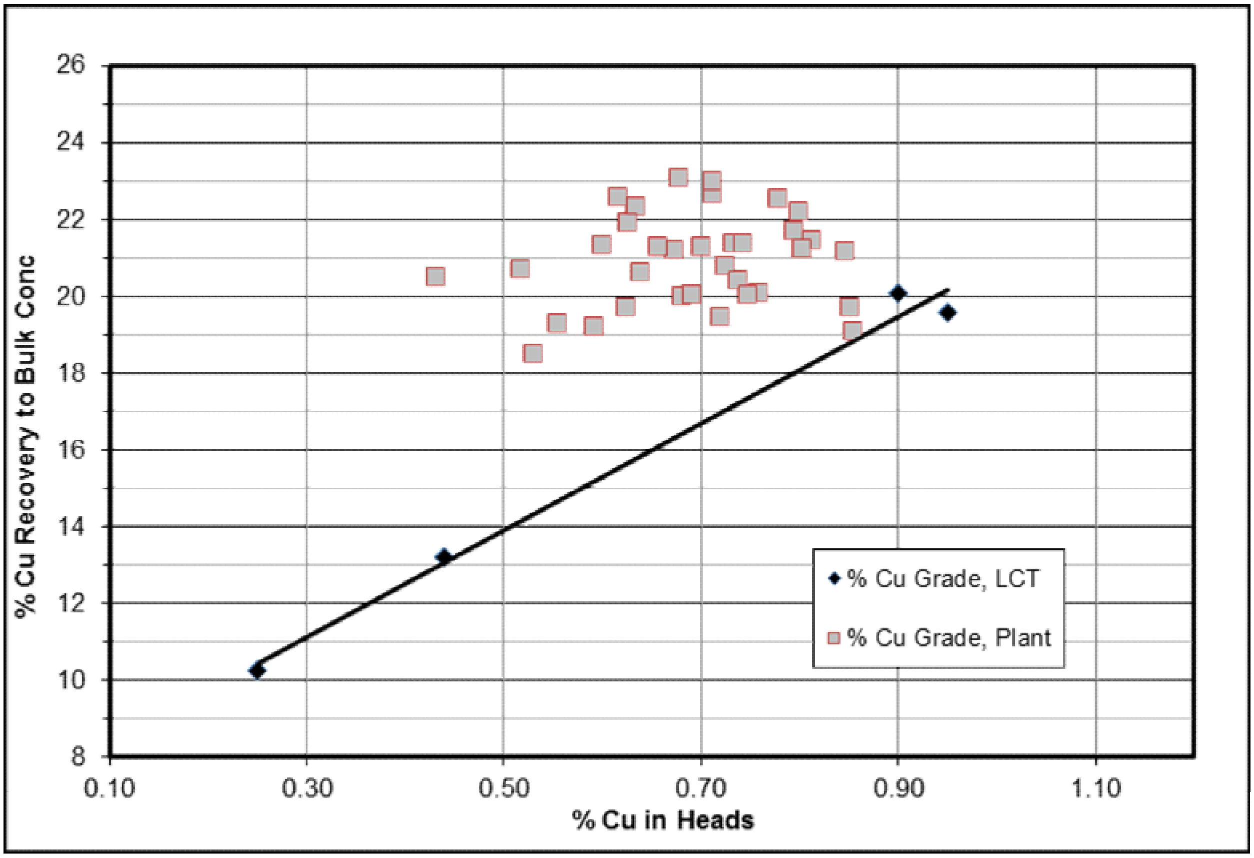


FIGURE 13-2: COPPER CONCENTRATE GRADES, LCT AND PLANT DATA



Average gold and copper head grades in the plant feed have been a little higher than the laboratory head grades at 0.71% Cu and 2.36 g/t Au, compared to 0.64% Cu and 1.85 g/t Au in the laboratory head samples.

Zinc recoveries achieved in the plant are generally higher than the laboratory recoveries (Figure 13 - 3) while zinc concentrate grades in the plant have been lower than the laboratory results (Figure 13 – 4). This is a preferred operating option due to Hudbay's short concentrate haul to their Flin Flon metallurgical site.

FIGURE 13-3: ZINC RECOVERY TO CONCENTRATE, LCT AND PLANT DATA

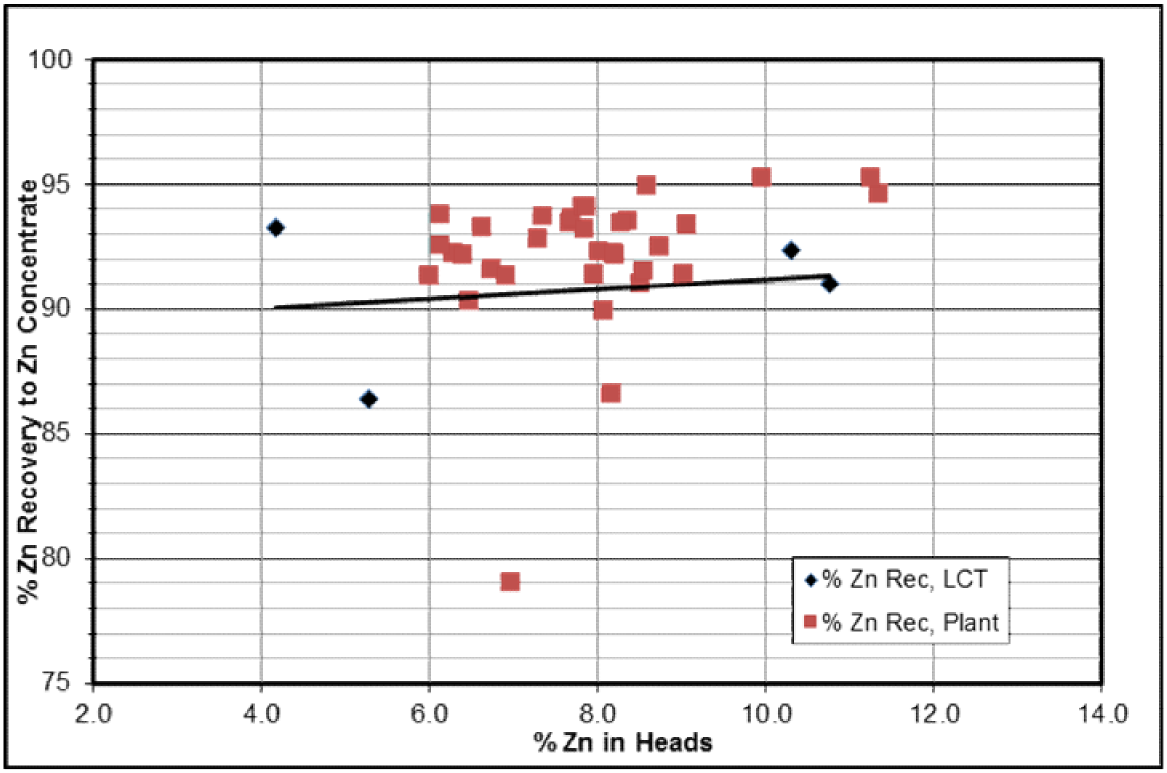
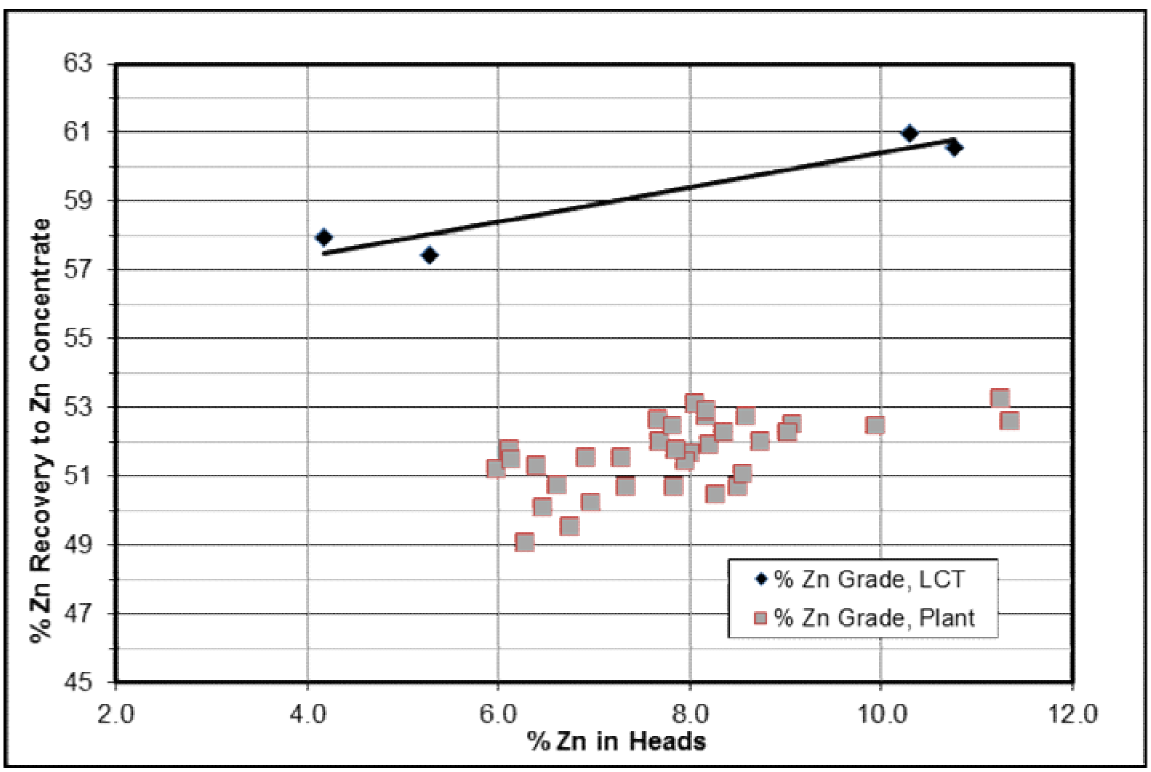


FIGURE 13-4: ZINC CONCENTRATE GRADES, LCT AND PLANT DATA



Zinc head grades in the plant feed have been slightly higher than the laboratory head grades at 7.76% Zn compared to 7.64% Zn in the laboratory head samples.

Metallurgical forecasts for this report will be based on recent plant data.

Penalty charges of the copper concentrate contract are as follows:

Zinc and Lead (Combined)

- US $3.00 per dmt for every 1% above 3% up to 8%
- US $5.00 per dmt for every 1% above 8% up to 12%
- US $8.00 per dmt for every 1% above 12%

Mercury

- US $2.00 per dmt for every 10ppm above 15ppm

Moisture

- US $2.00/wmt for every 1% above 10%

In 2016, approximate penalty charges of the deleterious elements of the copper concentrate in $US per tonne were 23.20, 2.21, and 15.06 for zinc and lead, mercury, and moisture respectively. The penalty charges do not have a significant effect on the potential economic extraction.

14 MINERAL RESOURCE ESTIMATES

Hudbay prepared an update of the Lalor mine 3D block model using MineSight® version 11.60, industry standard commercial software that specializes in geologic modelling and mine planning. The 3D block model and determination of the updated mineral resources at the Lalor mine were performed by Hudbay personnel following Hudbay procedures. The work was reviewed and approved by Robert Carter, P.Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit and Qualified Person of this Technical Report.

14.1 Key Assumptions of Model

As shown in Table 14-1, there are 1,707 assayed drill holes totalling approximately 420,310 m within the Lalor database used to support the mineral resource estimate.

TABLE 14-1: DRILLING DATA BY YEAR

Year	Drill Holes	Meters 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	199	25,849
Total	**1,707**	**420,310**

Note: the most recent drill hole in the database was drilled in July 2016

The drill hole database was exported from acQuire®, validated and provided in Microsoft Excel® format with a cut-off date for mineral resource estimate purposes of September 30, 2016. The files were imported as collar, downhole survey and assay data into MineSight.

14.2 Wireframe Models and Mineralization

The Lalor mineralized envelopes trend along an azimuth of approximately 320° with a general dip of approximately 37° to the north east. The volcanogenic massive sulphide deposit is hosted in a Paleoproterozoic bimodal volcanic sequence. Seven zinc-rich massive sulphide lenses (i.e. 10, 11, 20, 30, 31, 32 and 40) and seven gold-rich disseminated to semi-massive zones (i.e. 21, 23, 24, 25, 26, 27 and 28) have now been identified. The base metal lenses and gold rich zones are stratigraphically stacked and are mostly aligned to the main regional deformation S2 foliation, averaging N320°/37°. The Lalor mine is continuous along a strike length of approximately 1,600 m in north-south direction, approximately 700 m in an east-west direction and with a vertical extent of approximately 830 m.

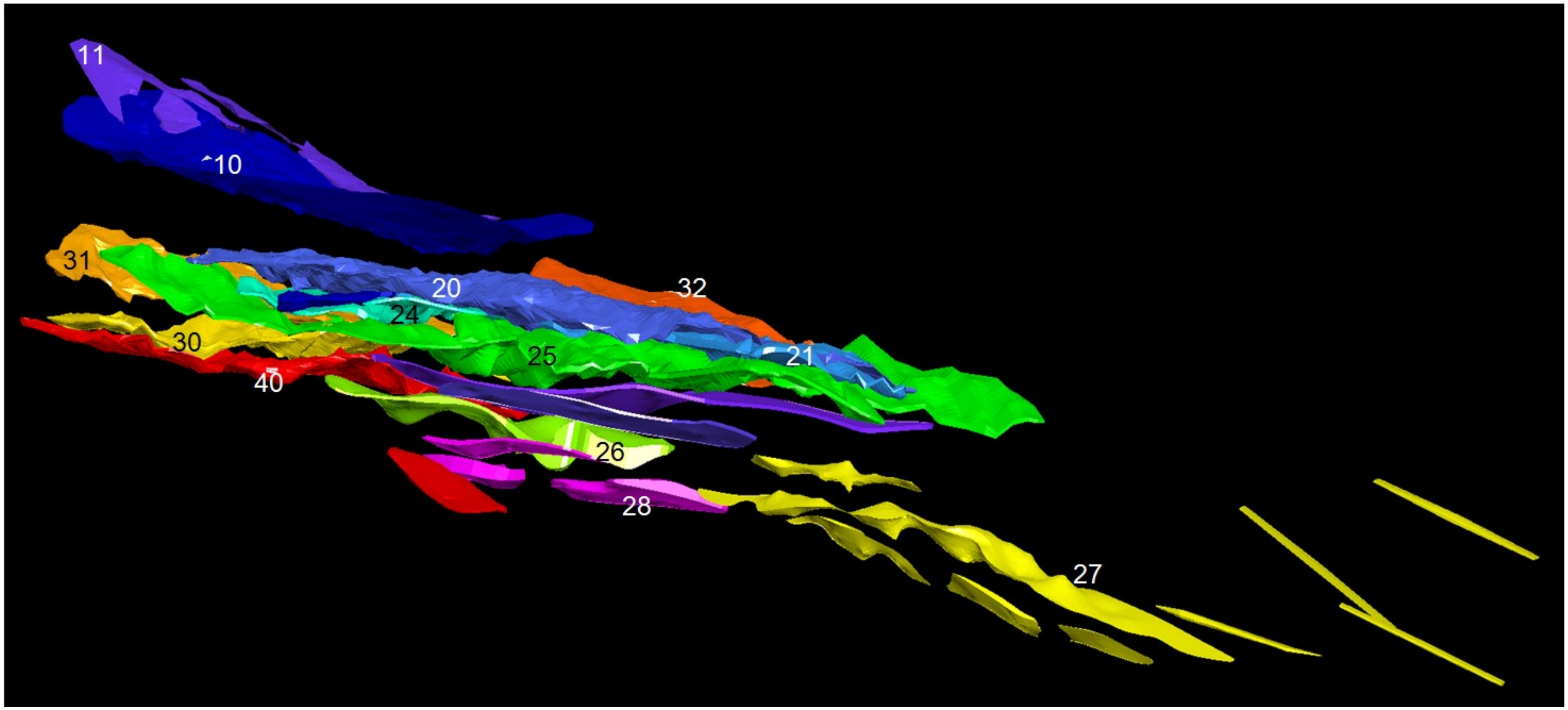
Four sets of structures were recognized by Hudbay and SRK geologists (Ravenelle, 2016), the Manitoba Geological Survey and the Geological Survey of Canada: the bedding S0 with an average

N317°/36°, the dominant foliation S2 averaging N318°/37°, the late foliation S3 averaging N012°/81° and the fold F3 with a fold axis of 26° → 012°.

The base metal grade shells were built using MineSight, from 2D cross sections linked to create solids and verified in plan, using an approximate 4.1% zinc equivalency cut-off. The gold-rich grade shell were built with Leapfrog® version 4.0 and were constrained with the logging, lithogeological data and an approximate 2.4 g/t gold equivalency or an approximate 1.9% copper equivalency. The gold-rich mineralized envelopes interpretation was stretched to follow the geological continuity of the zones. Wireframes of the mineralized envelopes are shown in Figure 14-1 (refer to Table 14-2 for the grade shells legend).

The term “Zone”, “Lense” and “Mineralized Envelope” is used interchangeably throughout this section and the report in general. Although Zone 27 is referred to as a gold-rich zone it contains appreciable amounts of copper and is amenable to processing in either a base metal or gold concentrator.

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



Note: Lense 23 is located under lense 32 and is hidden in this view.

TABLE 14-2: LEGEND OF INTERPRETED WIREFRAMES

Lense	Mineralization	Code
10	Base Metals	10
11	Base Metals	11
20	Base Metals	20
21	Gold and Copper	21
23	Gold	23
24	Gold	1 & 24
25	Gold	2 & 25
26	Gold	3 & 26
27	Copper and Gold	4, 5 & 27
28	Gold	6 & 28
30	Base Metals	30
31	Base Metals	31
32	Base Metals	32
40	Base Metals	40

14.3 Density Assignation

Density values generated from two multi elements regression formulas based on 65,792 measurements collected by Hudbay and measured at Flin Flon laboratory, ACME/Bureau Veritas laboratory and at Hudbay logging facility, using a non-wax-sealed immersion technique to measure the weight of each sample in air and in water.

Two regression formulas were used to predict the density value of intervals without a measured density:

Regression formula in the base metal lenses

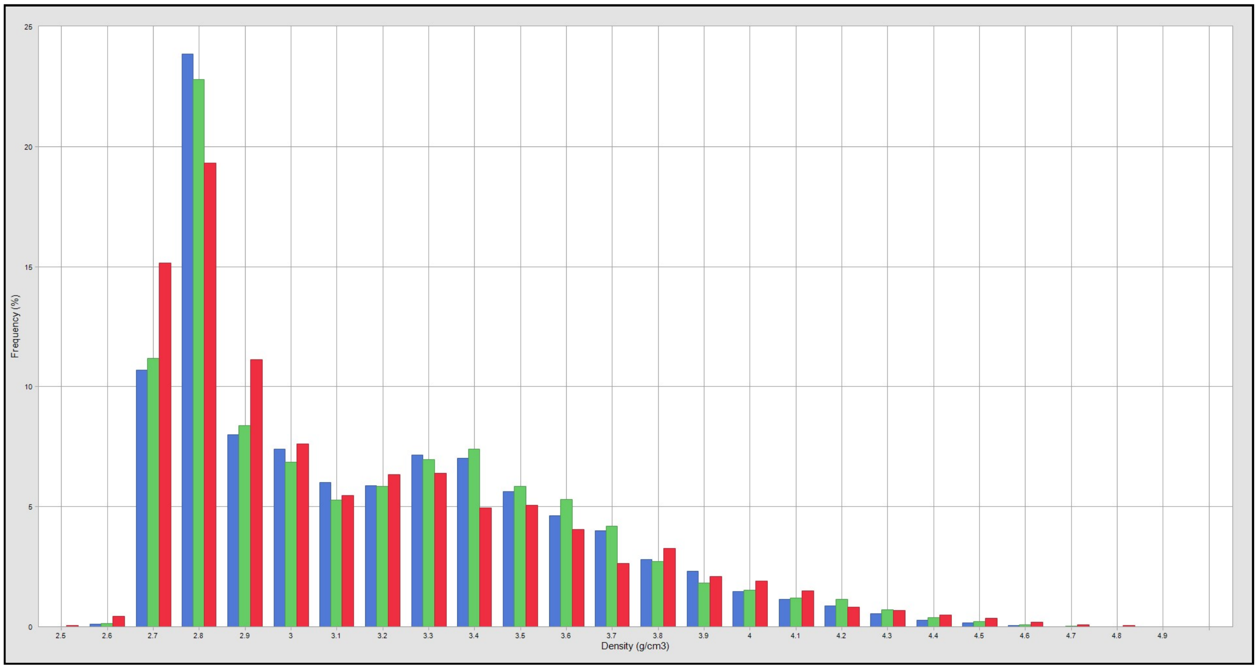
$$SG = Zn * 0.0194 + Pb * 0.0554 - Cu * 0.0001 + As * 0.0349 + Fe * 0.0494 + 2.7$$

Regression formula in the gold zones

$$SG = Zn * 0.0194 + Pb * 0.0554 + Cu * 0.0170 + As * 0.0494 + Fe * 0.0299 + 2.7$$

The hybrid density (i.e. measured values when available, predicted when missing) was interpolated to each mineralized envelop via ordinary kriging. The interpolation results were validated against a nearest neighbour model (NN) and an inverse distance weighting (IDW) models which show similar distribution (Figure 14-2).

FIGURE 14-2: OK, IDW AND NN SPECIFIC GRAVITY DISTRIBUTION



Note: OK SG in blue, IDW SG in green and NN SG in red.

14.4 Exploratory Data Analysis

Exploratory data analysis (EDA) includes basic statistical evaluation of the assays and composites for zinc (Zn), gold (Au), silver (Ag), copper (Cu), lead (Pb), iron (Fe), arsenic (As), density (SG), and samples length.

14.4.1 Assays

The Table 14-3 presents the number of samples collected and total length analyzed.

TABLE 14-3: SAMPLES AND LENGTH ANALYZED

	Number of Samples	Total Length in Metre
Zinc (%)	194,276	186,033
Copper (%)	194,276	186,033
Gold (g/t)	193,977	182,986
Silver (g/t)	194,276	186,033
Lead (%)	194,275	186,032
Iron (%)	194,276	186,033
Arsenic (%)	194,276	186,033
Density (g/cm3)	65,792	63,636

Box Plots

Box plots of the basic statistics for zinc, gold, copper and silver for each mineralized envelope are displayed in Figure 14-3 to Figure 14-6. These box plots confirm the occurrence of two mineralization systems at Lalor: higher grade zinc mineralization (zones 10, 11, 20, 30, 31, 32 and 40) and gold rich mineralization (zones 21, 23, 24, 25, 26, 27 and 28).

FIGURE 14-3: BOX PLOT OF ZINC (%) BY LENSE

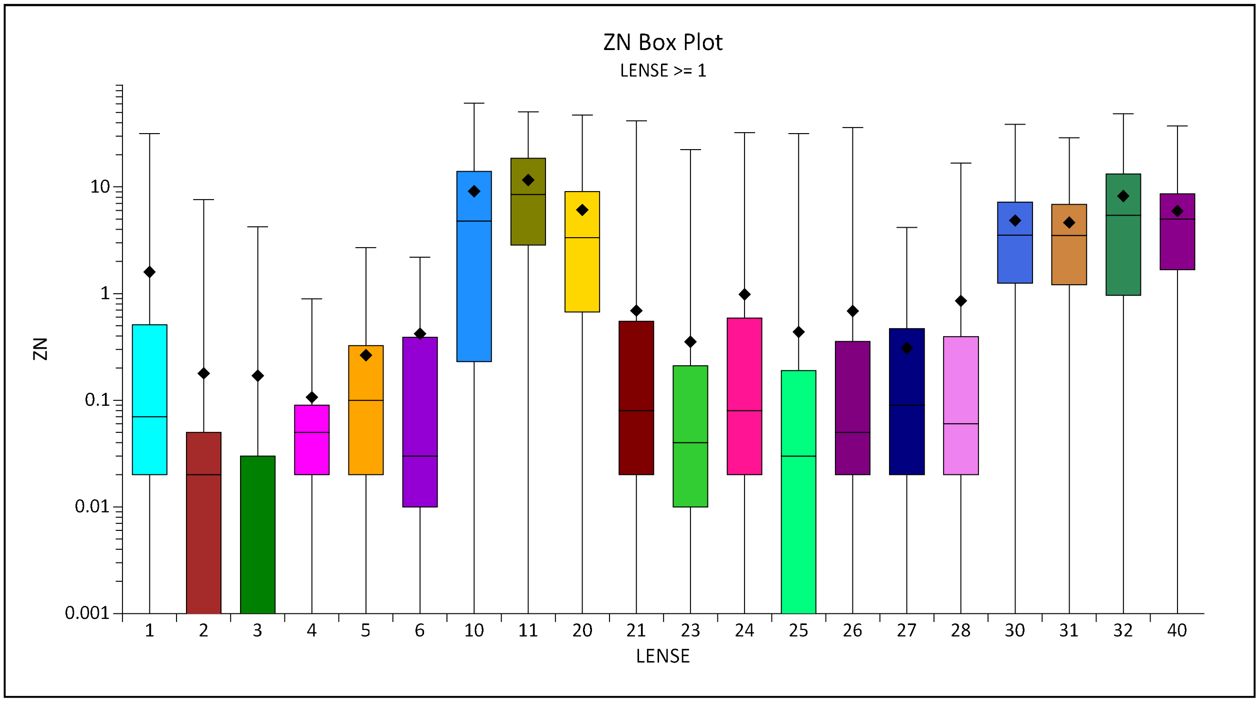


FIGURE 14-4: BOX PLOT OF GOLD (G/T) BY LENSE

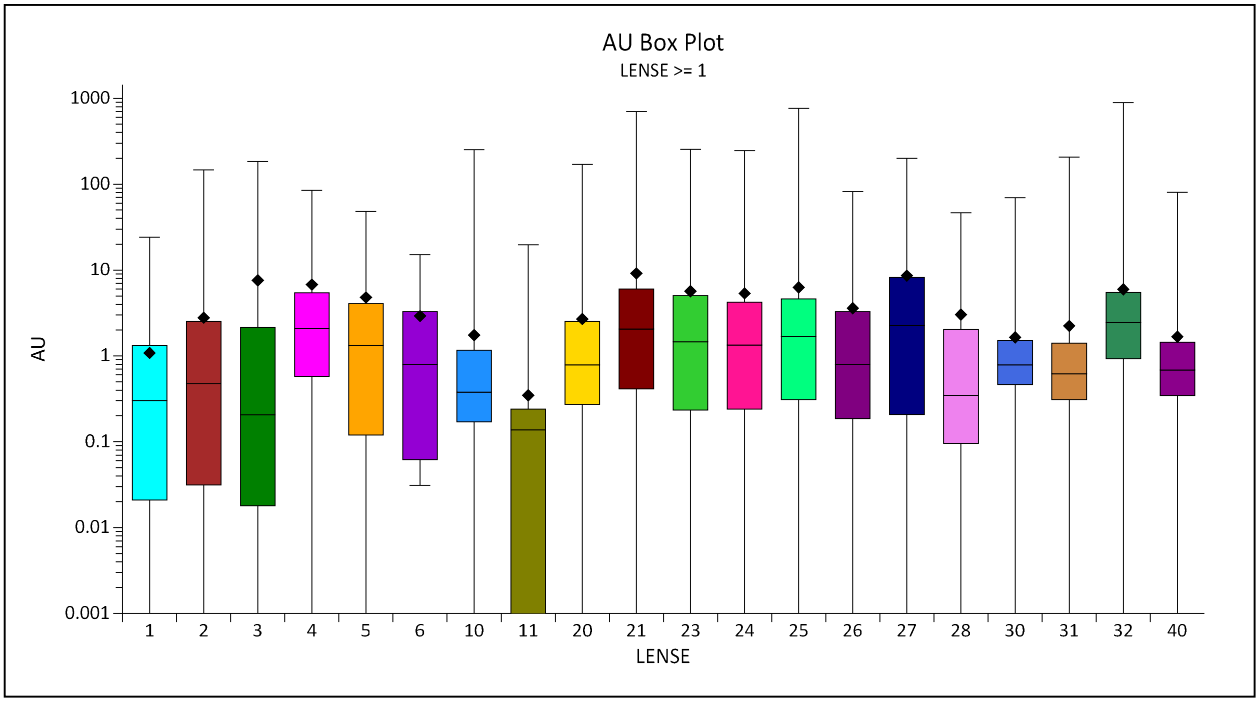


FIGURE 14-5: BOX PLOT OF COPPER (%) BY LENSE

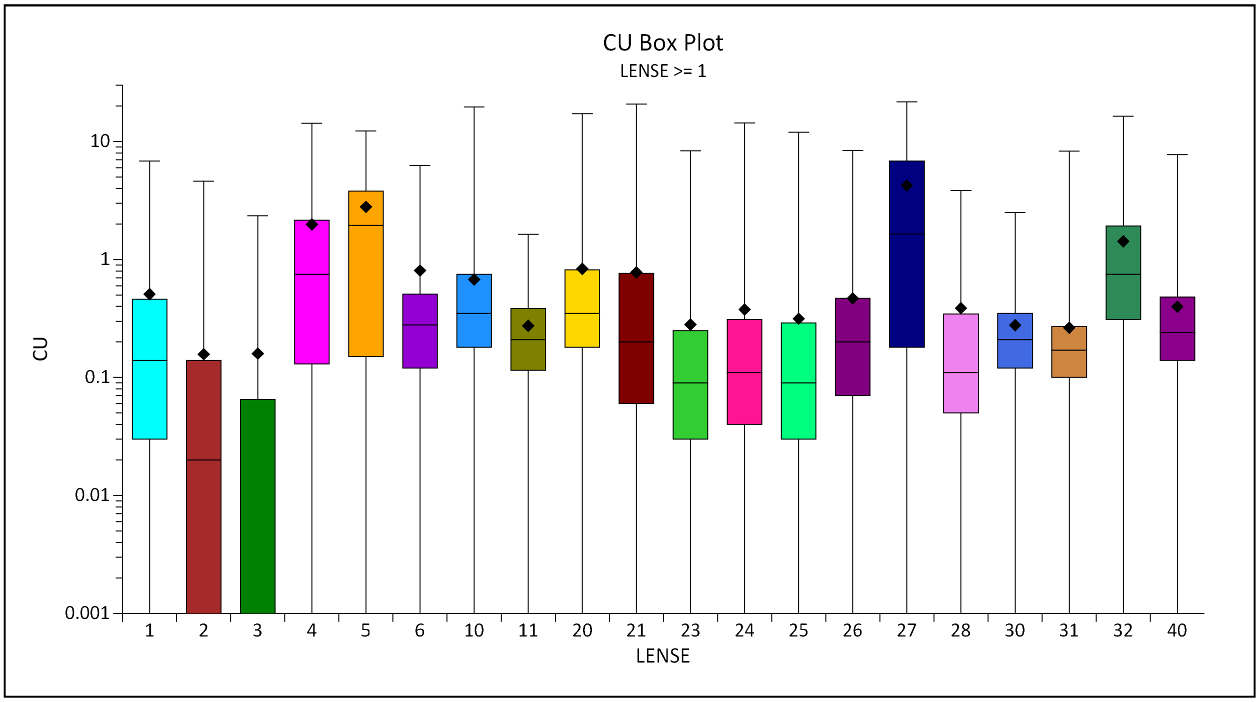
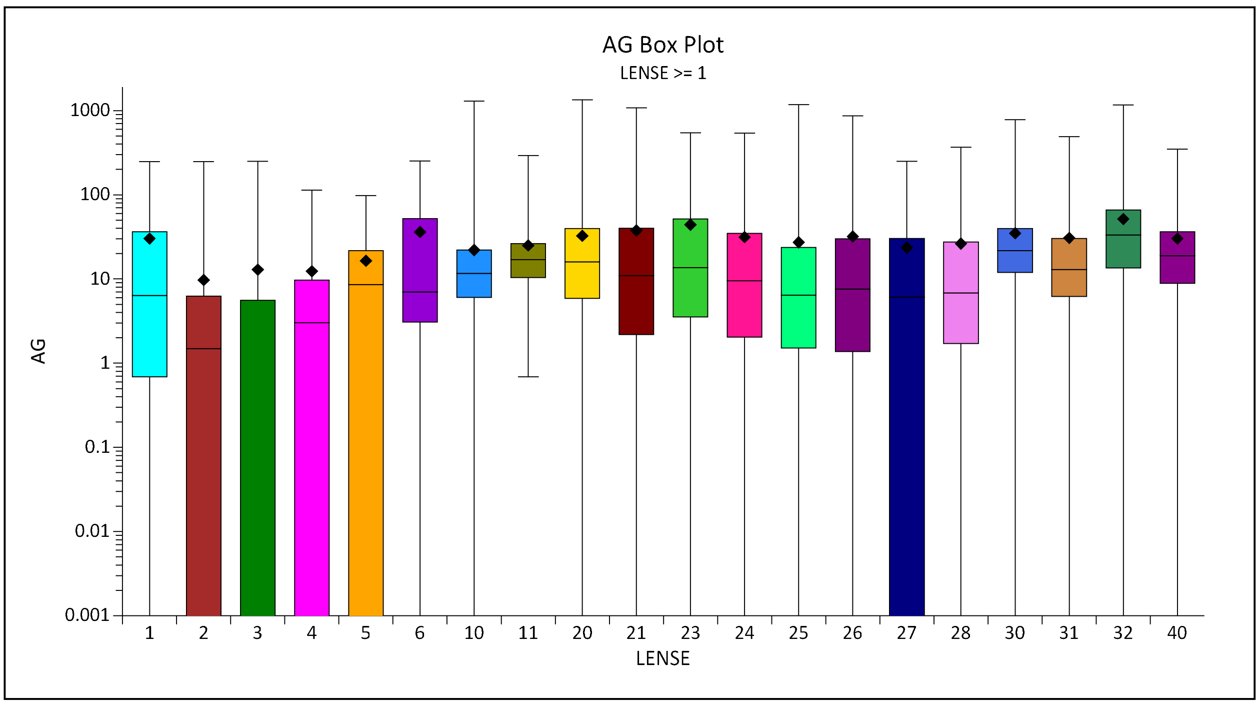


FIGURE 14-6: BOX PLOT OF SILVER (G/T) BY LENSE



Assay Statistics

The EDA of assay statistics of zinc, gold, copper, silver, lead, iron, arsenic and density are summarized in Table 14-4 to Table 14-11.

TABLE 14-4: ZINC ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
	1	24 Raft	177	0	31.60	1.59	0.07	25.32	5.03	3.16	0	0.01	0.02	0.07	0.51	2.42	10.36	19.28	29.59
	2	25 (Raft)	253	0	7.61	0.18	0.02	0.52	0.72	3.97	0	0	0	0.02	0.06	0.22	0.79	1.99	3.05
	3	26 (Raft)	30	0	4.22	0.20	0.01	0.61	0.78	3.95	0	0	0	0.01	0.03	0.08	0.72	1.89	3.29
	4	27 (upper raft)	93	0	0.89	0.11	0.05	0.03	0.18	1.71	0	0.02	0.02	0.05	0.09	0.28	0.52	0.77	0.81
	5	27 (lower raft)	94	0	2.69	0.27	0.11	0.17	0.41	1.53	0	0.02	0.02	0.11	0.33	0.69	0.95	1.25	1.66
	6	28 Raft	17	0	2.19	0.42	0.03	0.50	0.71	1.69	0	0	0.01	0.03	0.39	1.48	2.02	2.11	2.16
	10	10	9065	0	61.02	9.12	4.81	128.90	11.35	1.24	0.04	0.1	0.24	4.81	13.98	24.55	31.98	43.44	49.20
	11	11	583	0	50.63	11.61	8.52	113.50	10.65	0.92	0.21	1.4	2.85	8.52	18.6	25.85	30.45	38.15	43.23
	20	20	7185	0	47.12	6.09	3.35	53.03	7.28	1.20	0.07	0.39	0.68	3.35	9.03	15.94	20.87	26.37	32.17
ZN	21	21	3237	0	41.74	0.69	0.08	3.34	1.83	2.63	0	0.01	0.02	0.08	0.55	1.87	3.30	5.53	8.58
	23	23	1143	0	22.45	0.35	0.04	1.23	1.11	3.14	0	0.01	0.01	0.04	0.21	0.92	1.76	2.81	4.14
	24	24	1654	0	32.34	0.99	0.09	8.79	2.97	3.00	0	0.01	0.02	0.09	0.59	2.37	4.50	8.78	16.11
	25	25	4640	0	31.53	0.44	0.03	2.45	1.57	3.54	0	0	0.01	0.03	0.19	1.07	2.15	3.72	6.92
	26	26	938	0	35.94	0.69	0.05	6.51	2.55	3.68	0	0.02	0.02	0.05	0.38	1.49	3.01	4.61	9.81
	27	27	621	0	4.17	0.31	0.09	0.21	0.46	1.48	0	0.01	0.02	0.09	0.47	0.92	1.12	1.29	1.64
	28	28	125	0	16.70	0.87	0.06	6.09	2.47	2.85	0	0.01	0.02	0.06	0.41	2.58	4.26	6.64	14.63
	30	30	975	0.01	38.74	4.85	3.54	20.82	4.56	0.94	0.35	0.86	1.26	3.54	7.23	11.36	13.32	15.67	19.34
	31	31	1181	0	28.92	4.64	3.51	19.74	4.44	0.96	0.22	0.86	1.21	3.51	6.87	10.26	13.79	16.70	20.46
	32	32	3154	0	48.44	8.22	5.42	72.77	8.53	1.04	0.12	0.56	0.97	5.42	13.25	21.49	25.42	28.93	32.13
	40	40	1467	0	37.23	6.01	5.07	28.50	5.34	0.89	0.28	1.15	1.82	5.07	8.7	12.75	15.18	18.60	24.23

TABLE 14-5: GOLD ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
	1	24 Raft	177	0	24.21	1.08	0.30	5.17	2.27	2.10	0	0.00	0.02	0.30	1.32	2.94	3.88	4.58	8.23
	2	25 (Raft)	253	0	147.29	2.84	0.53	104.60	10.23	3.60	0	0.03	0.05	0.53	2.59	6.47	10.95	16.01	27.37
	3	26 (Raft)	30	0	183.30	8.84	0.36	1113.00	33.36	3.77	0.02	0.04	0.06	0.36	4.23	13.21	18.98	64.33	135.71
	4	27 (upper raft)	93	0	84.75	6.76	2.08	213.10	14.60	2.16	0.02	0.33	0.58	2.08	5.42	15.37	25.97	60.96	77.94
	5	27 (lower raft)	94	0	48.20	4.85	1.33	79.53	8.92	1.84	0.00	0.04	0.14	1.33	4.14	13.25	20.75	36.26	40.87
	6	28 Raft	17	0.031	15.05	2.92	0.80	17.86	4.23	1.45	0.03	0.06	0.06	0.80	3.26	8.13	10.80	12.93	14.20
	10	10	9061	0	251.59	1.75	0.38	39.03	6.25	3.57	0	0.14	0.17	0.38	1.17	3.91	7.10	12.41	22.18
	11	11	583	0	19.68	0.35	0.14	1.14	1.07	3.08	0	0	0	0.14	0.24	0.58	1.36	2.79	4.24
	20	20	7185	0	170.15	2.70	0.79	39.52	6.29	2.33	0.12	0.21	0.27	0.79	2.54	6.27	11.18	17.80	28.59
AU	21	21	3237	0	698.60	9.14	2.06	1030.00	32.09	3.51	0.08	0.27	0.41	2.06	6.03	19.36	36.04	57.36	124.38
	23	23	1143	0	254.40	5.68	1.47	237.40	15.41	2.71	0	0.17	0.24	1.47	5.06	11.75	23.38	40.03	67.04
	24	24	1654	0	245.20	5.36	1.34	230.60	15.18	2.83	0	0.17	0.24	1.34	4.22	11.91	23.93	40.60	67.99
	25	25	4654	0	766.27	6.34	1.71	478.20	21.87	3.45	0.05	0.21	0.32	1.71	4.66	13.36	25.57	45.38	73.71
	26	26	940	0	81.94	3.62	0.82	70.13	8.37	2.32	0.03	0.14	0.21	0.82	3.29	8.99	15.60	23.65	48.63
	27	27	621	0	200.37	8.60	2.25	389.60	19.74	2.29	0	0.10	0.21	2.25	8.16	21.81	32.30	60.68	111.44
	28	28	125	0	46.50	3.07	0.35	53.30	7.30	2.38	0.04	0.06	0.11	0.35	2.05	8.53	16.30	23.34	39.64
	30	30	975	0	69.81	1.65	0.79	16.69	4.09	2.48	0.28	0.41	0.46	0.79	1.51	2.82	4.71	8.01	16.34
	31	31	1181	0	206.95	2.23	0.62	82.42	9.08	4.07	0.19	0.27	0.31	0.62	1.41	3.46	6.99	14.61	32.11
	32	32	3151	0	892.15	5.97	2.44	407.90	20.20	3.38	0.34	0.69	0.93	2.44	5.52	12.41	20.64	33.57	55.13
	40	40	1472	0	80.37	1.68	0.69	19.42	4.41	2.62	0.19	0.28	0.34	0.69	1.44	3.29	5.94	8.66	16.38

TABLE 14-6: COPPER ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
	1	24 Raft	177	0	6.85	0.51	0.14	0.92	0.96	1.89	0	0.02	0.03	0.14	0.46	1.46	1.92	3.49	4.68
	2	25 (Raft)	253	0	4.61	0.16	0.02	0.18	0.43	2.67	0	0	0	0.02	0.15	0.40	0.71	0.86	2.14
	3	26 (Raft)	30	0	2.35	0.19	0.01	0.27	0.52	2.77	0	0	0	0.01	0.10	0.24	1.19	1.84	2.14
	4	27 (upper raft)	93	0	14.29	1.99	0.75	9.75	3.12	1.57	0.02	0.1	0.13	0.75	2.15	5.26	9.36	11.63	14.24
	5	27 (lower raft)	94	0	12.29	2.82	1.98	9.42	3.07	1.09	0.05	0.11	0.17	1.98	3.84	7.19	9.16	11.16	11.86
	6	28 Raft	17	0	6.28	0.81	0.28	2.37	1.54	1.91	0	0.05	0.12	0.28	0.51	1.98	2.90	4.59	5.60
	10	10	9065	0	19.71	0.68	0.36	1.21	1.10	1.62	0.08	0.15	0.18	0.36	0.75	1.42	2.25	3.56	5.65
	11	11	583	0	1.64	0.27	0.21	0.05	0.23	0.83	0.06	0.09	0.12	0.21	0.39	0.57	0.69	0.81	1.09
	20	20	7185	0	17.29	0.83	0.35	1.80	1.34	1.61	0.09	0.16	0.18	0.35	0.82	2.12	3.40	4.97	7.00
CU	21	21	3237	0	20.84	0.78	0.20	2.56	1.60	2.06	0.02	0.04	0.06	0.20	0.77	2.11	3.50	5.31	7.92
	23	23	1143	0	8.37	0.28	0.09	0.38	0.62	2.19	0	0.02	0.03	0.09	0.25	0.64	1.23	1.95	3.13
	24	24	1654	0	14.35	0.38	0.11	1.00	1.00	2.65	0	0.03	0.04	0.11	0.32	0.80	1.48	2.65	5.00
	25	25	4640	0	12.01	0.32	0.09	0.56	0.75	2.34	0.01	0.02	0.03	0.09	0.30	0.74	1.37	2.08	3.59
	26	26	938	0	8.40	0.47	0.20	0.80	0.90	1.90	0.02	0.05	0.07	0.20	0.47	1.08	1.89	2.84	5.07
	27	27	621	0	21.75	4.24	1.65	27.93	5.28	1.25	0.02	0.11	0.18	1.65	6.84	13.20	15.36	17.33	18.86
	28	28	125	0	3.86	0.39	0.13	0.43	0.66	1.67	0.02	0.05	0.05	0.13	0.36	1.09	1.45	2.39	3.26
	30	30	975	0.01	2.50	0.28	0.21	0.07	0.26	0.92	0.08	0.1	0.12	0.21	0.35	0.56	0.73	0.91	1.27
	31	31	1181	0	8.33	0.26	0.17	0.21	0.46	1.75	0.06	0.09	0.1	0.17	0.27	0.45	0.76	1.23	1.92
	32	32	3154	0	16.44	1.43	0.75	3.27	1.81	1.27	0.13	0.25	0.32	0.75	1.92	3.53	4.84	6.22	8.57
	40	40	1467	0	7.77	0.40	0.25	0.25	0.50	1.24	0.09	0.13	0.15	0.25	0.48	0.84	1.20	1.55	2.25

TABLE 14-7: SILVER ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
AG	1	24 Raft	177	0	248.06	30.17	6.38	2503.00	50.03	1.66	0	0.48	0.686	6.38	36.31	89.17	158.65	189.31	205.08
	2	25 (Raft)	253	0	248.23	9.92	1.75	676.50	26.01	2.62	0	0	0	1.75	6.50	24.98	37.97	82.81	131.57
	3	26 (Raft)	30	0	250.00	15.04	0.25	2300.00	47.96	3.19	0	0	0	0.25	8.18	13.22	70.81	135.45	204.18
	4	27 (upper raft)	93	0	114.00	12.35	3.02	532.60	23.08	1.87	0	0	0	3.02	9.67	35.80	69.20	84.37	100.20
	5	27 (lower raft)	94	0	98.00	16.63	9.12	521.60	22.84	1.37	0	0	0	9.12	21.84	47.66	70.75	85.50	96.14
	6	28 Raft	17	0	251.31	36.21	7.00	4215.00	64.92	1.79	0.69	1.99	3.09	7.00	52.11	91.89	145.99	198.65	230.25
	10	10	9065	0	1297.03	22.10	11.66	1867.00	43.21	1.96	3.43	5.18	6.07	11.66	22.18	44.19	70.97	116.47	180.43
	11	11	583	0.686	291.77	25.03	16.94	983.10	31.35	1.25	5.86	9.15	10.42	16.94	26.35	46.44	72.46	114.76	148.52
	20	20	7185	0	2090.00	32.88	15.98	3059.00	55.31	1.68	3.00	4.77	5.93	15.98	39.77	79.95	116.34	161.75	237.79
	21	21	3237	0	2065.00	38.41	11.00	6817.00	82.57	2.15	0	1.54	2.23	11.00	40.08	100.14	167.04	258.01	389.62
	23	23	1143	0	547.00	44.22	13.82	5873.00	76.63	1.73	0.82	2.41	3.55	13.82	51.74	115.66	195.07	301.76	406.57
	24	24	1654	0	538.97	31.52	9.60	3408.00	58.38	1.85	0.51	1.37	2.06	9.60	34.96	80.51	143.61	217.51	279.23
	25	25	4640	0	1178.00	27.61	6.72	3954.00	62.89	2.28	0	1.06	1.65	6.72	24.28	68.72	129.01	202.85	321.68
	26	26	938	0	870.00	32.28	7.89	4963.00	70.45	2.18	0	0.69	1.54	7.89	30.41	88.30	144.18	205.34	365.73
	27	27	621	0	249.33	23.60	6.07	1308.00	36.16	1.53	0	0	0	6.07	30.14	78.00	101.59	122.73	147.05
	28	28	125	0	367.54	26.77	6.86	2885.00	53.71	2.01	0	1.37	1.90	6.86	28.00	73.57	122.65	187.54	275.52
	30	30	975	1.749	781.03	35.01	21.81	2867.00	53.54	1.53	6.22	10.21	12.00	21.81	39.67	67.69	96.89	140.47	202.20
	31	31	1181	0	489.26	30.73	12.89	2893.00	53.79	1.75	3.70	5.42	6.21	12.89	30.31	73.89	110.00	174.70	296.71
	32	32	3154	0	1164.00	51.49	33.33	4004.00	63.28	1.23	6.00	10.77	13.58	33.33	66.05	110.27	157.28	218.42	299.62
	40	40	1467	0	349.03	30.47	19.13	1259.00	35.48	1.16	4.89	7.49	9.05	19.13	37.05	71.98	94.80	124.08	173.69

TABLE 14-8: LEAD ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
PB	1	24 Raft	177	0	4.92	0.35	0.02	0.69	0.83	2.40	0	0	0	0.02	0.15	1	2.23	2.95	3.99
	2	25 (Raft)	253	0	2.37	0.06	0	0.08	0.28	4.33	0	0	0	0	0.02	0.07	0.21	0.75	1.56
	3	26 (Raft)	30	0	0.44	0.03	0	0.01	0.10	3.27	0	0	0	0	0	0.02	0.22	0.35	0.41
	4	27 (upper raft)	93	0	0.04	0	0	0	0.01	2.47	0	0	0	0	0	0.01	0.02	0.02	0.04
	5	27 (lower raft)	94	0	0.19	0.01	0	0	0.02	2.83	0	0	0	0	0.01	0.02	0.02	0.04	0.05
	6	28 Raft	17	0	0.17	0.03	0.02	0	0.04	1.48	0	0	0	0.02	0.03	0.06	0.12	0.15	0.16
	10	10	9065	0	23.60	0.26	0.03	0.82	0.91	3.54	0	0	0	0.03	0.13	0.62	1.24	2.11	3.60
	11	11	583	0	13.42	0.22	0.02	0.57	0.75	3.36	0	0	0	0.02	0.13	0.63	1.08	1.57	2.02
	20	20	7185	0	37.81	0.44	0.07	1.33	1.15	2.65	0	0.01	0.02	0.07	0.37	1.18	2.07	3.06	4.68
	21	21	3237	0	20.20	0.17	0.02	0.56	0.75	4.54	0	0	0	0.02	0.09	0.33	0.64	1.15	2.45
	23	23	1143	0	18.30	0.38	0.07	1.12	1.06	2.77	0	0	0.01	0.07	0.32	0.92	1.52	3.01	5.02
	24	24	1654	0	9.70	0.30	0.04	0.53	0.73	2.39	0	0	0.01	0.04	0.23	0.88	1.58	2.35	3.61
	25	25	4640	0	24.41	0.18	0.02	0.48	0.69	3.78	0	0	0	0.02	0.09	0.41	0.96	1.59	2.49
	26	26	938	0	19.09	0.36	0.01	1.94	1.39	3.84	0	0	0	0.01	0.07	0.77	1.99	3.70	6.82
	27	27	621	0	1.53	0.01	0	0	0.07	5.68	0	0	0	0	0.01	0.02	0.03	0.07	0.12
	28	28	125	0	3.28	0.16	0	0.27	0.52	3.33	0	0	0	0	0.06	0.33	0.58	2.00	3.03
	30	30	975	0	13.20	0.57	0.27	0.92	0.96	1.67	0.02	0.04	0.06	0.27	0.69	1.31	2.14	3.01	4.50
	31	31	1181	0	11.70	0.44	0.12	0.76	0.87	2.00	0	0.02	0.03	0.12	0.48	1.15	1.93	2.75	4.37
	32	32	3154	0	25.15	0.44	0.10	1.18	1.09	2.45	0	0.01	0.02	0.10	0.46	1.13	1.74	2.51	5.46
	40	40	1467	0	8.35	0.65	0.27	0.95	0.97	1.51	0.01	0.03	0.04	0.27	0.84	1.86	2.49	3.27	4.43

TABLE 14-9: IRON ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
FE	1	24 Raft	177	1.22	29.94	7.24	5.47	33.74	5.81	0.80	2.15	2.85	3.19	5.47	9.22	14.63	18.68	25.20	29.25
	2	25 (Raft)	253	0.44	27.54	3.99	3.20	7.48	2.73	0.69	1.65	2.18	2.41	3.20	4.94	6.91	8.44	9.71	11.38
	3	26 (Raft)	30	1.19	9.73	3.64	3.43	3.18	1.78	0.49	1.84	2.32	2.35	3.43	4.61	5.68	5.82	6.97	8.63
	4	27 (upper raft)	93	1.39	19.93	6.50	5.85	11.71	3.42	0.53	3.15	4.39	4.79	5.85	7.32	9.39	13.97	16.71	18.71
	5	27 (lower raft)	94	1.60	31.13	6.53	5.35	26.09	5.11	0.78	2.10	2.53	3.06	5.35	7.32	13.74	17.96	19.09	22.07
	6	28 Raft	17	1.02	22.42	3.72	2.54	24.31	4.93	1.33	1.41	1.56	1.56	2.54	3.70	4.07	7.82	15.12	19.50
	10	10	9065	0.30	56.46	21.36	20.07	108.20	10.40	0.49	8.20	11.50	13.00	20.07	29.80	36.20	38.92	41.10	43.47
	11	11	583	0.78	46.23	24.78	26.20	97.57	9.88	0.40	10.22	15.07	17.00	26.20	32.39	37.04	39.19	40.37	42.70
	20	20	7185	0.27	46.17	14.60	12.28	100.80	10.04	0.69	3.32	4.96	5.81	12.28	22.20	29.67	33.14	35.80	37.96
	21	21	3237	0.29	38.05	4.73	3.37	16.93	4.12	0.87	1.70	2.13	2.29	3.37	5.65	9.28	13.21	16.61	21.58
	23	23	1143	0.21	34.03	4.24	3.34	11.14	3.34	0.79	1.29	1.76	2.01	3.34	5.56	8.11	10.10	12.37	16.19
	24	24	1654	0.17	39.80	4.97	3.24	26.67	5.16	1.04	1.40	1.82	2.01	3.24	5.59	10.46	15.89	21.45	26.48
	25	25	4640	0.24	40.06	4.24	3.46	11.09	3.33	0.79	1.43	1.88	2.07	3.46	5.40	7.58	9.90	12.51	17.41
	26	26	938	0.39	40.10	6.01	4.51	28.56	5.34	0.89	1.73	2.46	2.78	4.51	6.95	12.16	16.61	20.68	27.69
	27	27	621	1.39	32.70	8.18	5.10	48.66	6.98	0.85	2.41	2.81	3.01	5.10	10.83	19.50	22.65	27.34	30.37
	28	28	125	0.79	36.26	5.41	3.84	23.79	4.88	0.90	1.61	1.99	2.24	3.84	6.91	11.32	14.59	17.34	19.65
	30	30	975	1.49	35.89	15.18	14.49	57.77	7.60	0.50	5.52	7.79	8.97	14.49	20.42	25.90	29.18	30.58	33.66
	31	31	1181	0.90	43.87	17.85	17.02	99.86	9.99	0.56	5.00	7.72	9.09	17.02	25.99	31.88	34.47	36.46	38.65
	32	32	3154	0	44.30	18.27	19.24	126.30	11.24	0.62	3.52	5.69	7.07	19.24	27.74	32.97	35.69	37.86	39.91
	40	40	1467	0.32	43.10	17.18	17.68	82.35	9.07	0.53	4.80	7.40	8.91	17.68	24.09	28.91	31.19	33.86	36.98

TABLE 14-10: ARSENIC ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
	1	24 Raft	177	0	0.04	0	0	0	0.01	2.66	0	0	0	0	0	0.01	0.02	0.03	0.03
	2	25 (Raft)	253	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	3	26 (Raft)	30	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	4	27 (upper raft)	93	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	5	27 (lower raft)	94	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	6	28 Raft	17	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	10	10	9065	0	5.20	0.04	0.03	0.01	0.10	2.49	0	0	0	0.03	0.05	0.07	0.10	0.13	0.23
	11	11	583	0	0.25	0.03	0.03	0	0.03	0.98	0	0	0	0.03	0.05	0.07	0.09	0.10	0.11
	20	20	7185	0	2.25	0.05	0.02	0.01	0.12	2.42	0	0	0	0.02	0.05	0.10	0.20	0.34	0.57
AS	21	21	3237	0	0.46	0	0	0	0.02	7.30	0	0	0	0	0	0	0	0.03	0.06
	23	23	1143	0	2.12	0.01	0	0	0.07	9.33	0	0	0	0	0	0.01	0.03	0.06	0.10
	24	24	1654	0	1.94	0.01	0	0	0.06	7.95	0	0	0	0	0	0.01	0.03	0.05	0.12
	25	25	4640	0	0.68	0	0	0	0.02	10.33	0	0	0	0	0	0	0	0.01	0.05
	26	26	938	0	0.21	0	0	0	0.01	6.92	0	0	0	0	0	0	0	0.02	0.04
	27	27	621	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	28	28	125	0	0	0	0	0	0	NaN	0	0	0	0	0	0	0	0	0
	30	30	975	0	0.41	0.01	0	0	0.02	1.91	0	0	0	0	0.02	0.03	0.04	0.06	0.07
	31	31	1181	0	0.54	0.02	0.01	0	0.03	1.66	0	0	0	0.01	0.03	0.04	0.05	0.06	0.09
	32	32	3154	0	0.53	0.03	0.02	0	0.04	1.41	0	0	0	0.02	0.04	0.06	0.08	0.12	0.20
	40	40	1467	0	0.30	0.02	0.01	0	0.02	1.42	0	0	0	0.01	0.03	0.04	0.05	0.06	0.08

TABLE 14-11: DENSITY ASSAY STATISTICS BY LENSE

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	10%	20%	25%	50%	75%	90%	95%	97.50%	99%
	1	24 Raft	177	2.63	3.92	2.97	2.88	0.07	0.27	0.09	2.77	2.79	2.80	2.88	3.01	3.33	3.65	3.80	3.87
	2	25 (Raft)	197	2.70	3.21	2.84	2.80	0.01	0.10	0.04	2.74	2.75	2.76	2.80	2.89	2.98	3.04	3.09	3.11
	3	26 (Raft)	27	2.73	3.14	2.83	2.81	0.01	0.09	0.03	2.75	2.75	2.77	2.81	2.87	2.94	2.96	3.02	3.09
	4	27 (upper raft)	83	2.65	3.50	3.00	2.98	0.02	0.14	0.05	2.90	2.91	2.92	2.98	3.03	3.10	3.40	3.42	3.47
	5	27 (lower raft)	94	2.74	3.81	2.97	2.90	0.04	0.19	0.07	2.78	2.81	2.82	2.90	3.03	3.23	3.29	3.46	3.51
	6	28 Raft	4	2.74	2.83	2.79	2.80	0	0.04	0.02	2.75	2.76	2.76	2.80	2.82	2.83	2.83	2.83	2.83
	10	10	8879	2.45	5.30	3.65	3.52	0.35	0.59	0.16	2.93	3.06	3.12	3.52	4.16	4.50	4.64	4.74	4.87
	11	11	556	2.69	5.80	3.83	3.91	0.28	0.53	0.14	3.05	3.30	3.41	3.91	4.24	4.43	4.59	4.71	4.83
	20	20	7170	2.26	5.10	3.32	3.18	0.23	0.48	0.14	2.81	2.88	2.92	3.18	3.69	4.06	4.21	4.32	4.45
SG	21	21	3205	2.23	4.80	2.84	2.79	0.03	0.16	0.06	2.73	2.74	2.75	2.79	2.87	3.03	3.15	3.29	3.49
	23	23	1100	2.45	4.25	2.84	2.81	0.02	0.13	0.05	2.73	2.75	2.76	2.81	2.88	2.98	3.04	3.14	3.30
	24	24	1605	2.04	4.40	2.87	2.80	0.05	0.21	0.07	2.73	2.75	2.76	2.80	2.89	3.08	3.31	3.53	3.73
	25	25	4400	2.07	4.45	2.83	2.80	0.02	0.14	0.05	2.73	2.74	2.75	2.80	2.88	2.98	3.06	3.17	3.31
	26	26	664	2.22	4.38	2.95	2.86	0.06	0.25	0.09	2.75	2.77	2.79	2.86	3.05	3.24	3.40	3.57	4.11
	27	27	511	2.70	4.02	3.09	3.04	0.06	0.25	0.08	2.82	2.88	2.92	3.04	3.20	3.45	3.60	3.71	3.92
	28	28	49	2.69	3.59	2.84	2.80	0.02	0.14	0.05	2.74	2.75	2.75	2.80	2.88	2.94	3.01	3.05	3.33
	30	30	967	2.67	4.57	3.33	3.29	0.12	0.34	0.10	2.93	3.03	3.09	3.29	3.55	3.81	3.98	4.10	4.20
	31	31	1181	2.50	4.80	3.42	3.33	0.21	0.45	0.13	2.89	3.00	3.05	3.33	3.72	4.10	4.29	4.40	4.50
	32	32	3142	2.28	5.28	3.53	3.55	0.36	0.60	0.17	2.80	2.89	2.93	3.55	4.06	4.33	4.45	4.56	4.69
	40	40	1455	2.26	4.59	3.46	3.47	0.16	0.40	0.11	2.93	3.08	3.13	3.47	3.79	4.00	4.11	4.18	4.30

The high coefficient of variation (CV) values of copper and gold in the gold zones suggest that the gold and copper grades are variable and that a method of modeling using additional constraints may be better suited to these zones.

Contact Plots

Given the fact that for the most part, the different mineralized envelopes are separated by barren volcanic rocks, contact plots were not analyzed and a strict code matching system between the composites and block model has been used for the grade interpolation.

Grade Capping and High Yield Restriction

Since most of the mineralized envelopes show a high skewness in the statistical distribution of gold and silver grade, length weighted, log-scaled probability plots, deciles analysis and disintegration analysis of the assays were used to define high-grade outliers for gold and silver within each of the separately evaluated domains. These high-grade outliers can lead to overestimation of average grades unless some means of moderating the effect of the high-grade samples is applied. A common method for accomplishing this is by capping high assays at some predetermined level prior to grade estimation. In reviewing the statistics by zone it was decided to cap high grade gold and silver for all zones. Capping was completed on the assays prior to compositing. As an example, the log probability plot and the histograms of the gold distribution in zone 25 are displayed below in Figure 14-7 and Figure 14-8.

FIGURE 14-7: LOG PROBABILITY PLOT OF AU (G/T) IN ZONE 25

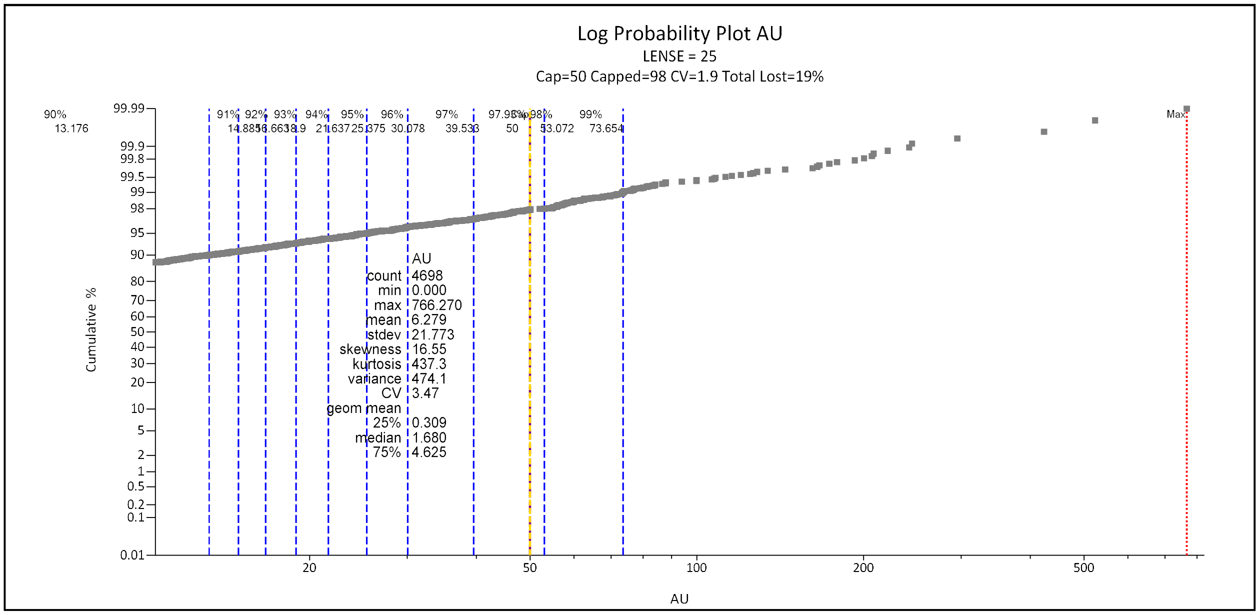
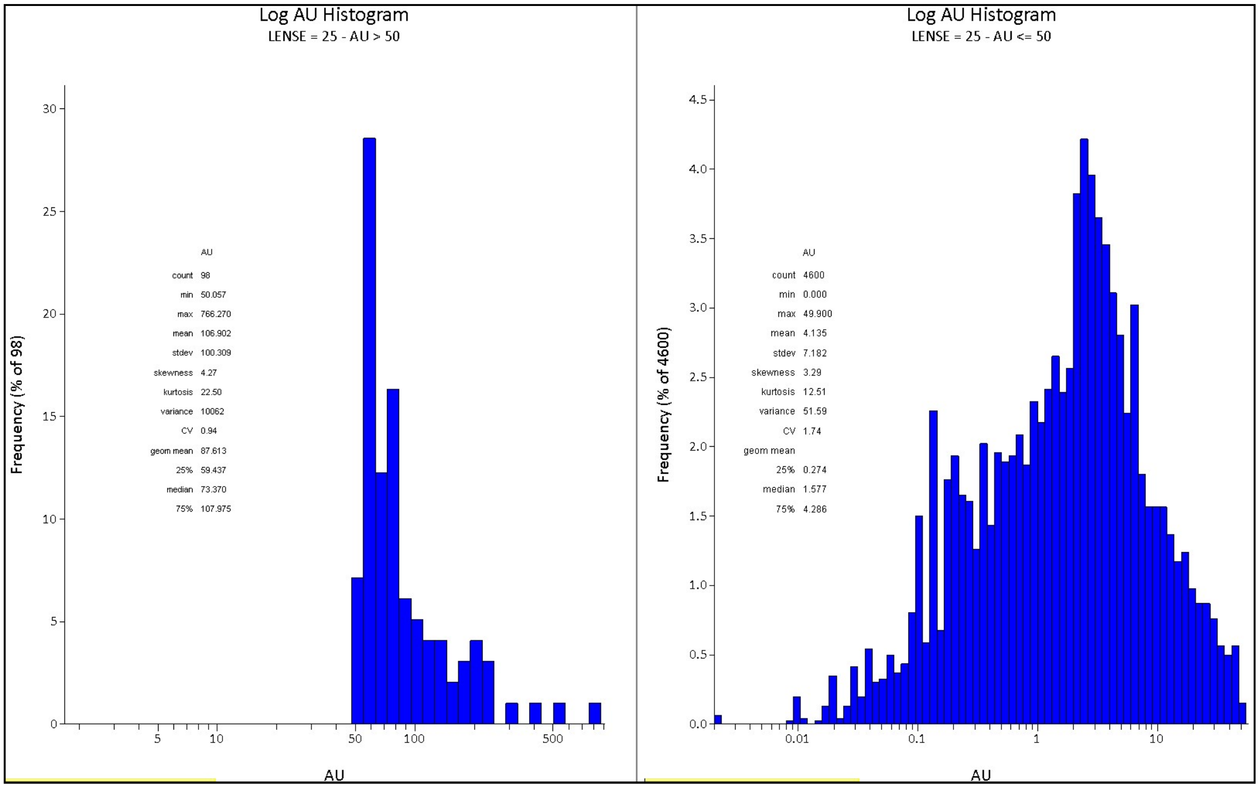


FIGURE 14-8: HISTOGRAMS AND BASIC STATISTICS OF THE POPULATION ABOVE AND BELOW CAPPING IN ZONE 25 FOR AU (G/T)



In reviewing statistics of zinc, copper, lead, iron and density the composite data showed some high grade values were discontinuous from the remainder of the data set and there was justification of restricting these higher grade values by limiting their search distance. A high yield restriction of 20 m (i.e. four blocks) was applied at the interpolation stage based on the 95th percentile of the 1.25 m composites values instead of capping. The capping and high yield thresholds are shown below in Table 14-12 and Table 14-13 respectively.

TABLE 14-12: CAPPING THRESHOLDS BY LENSE OF GOLD AND SILVER (G/T)

Assays Capping Values						
Domain	Lense	Total number of samples	AU Cap	Au samples capped	AG Cap	Ag samples capped
1	24 Raft	177	4.4	7	200	3
2	25 (Raft)	258	13.0	11	80	8
3	26 (Raft)	35	19.0	2	n/a	0
4	27 (upper raft)	93	71.0	2	80	4
5	27 (lower raft)	95	38.5	2	70	6
6	28 Raft	17	*n/a*	*0*	*n/a*	0
10	10	9,036	22.0	93	180	91
11	11	583	4.3	6	120	15
20	20	7,192	27.9	75	320	30
21	21	3,240	64.0	71	350	42
23	23	1,146	54.1	20	350	21
24	24	1,659	40.0	42	230	32
25	25	4,698	50.0	98	275	65
26	26	947	36.0	36	260	19
27	27	621	71.0	13	130	12
28	28	127	19.0	6	130	6
30	30	977	16.0	10	195	15
31	31	1,181	32.0	12	280	16
32	32	3,157	55.0	32	320	25
40	40	1,483	16.0	16	180	14

TABLE 14-13: HIGH-YIELD RESTRICTION THRESHOLDS BY LENSE

High Yield Restriction Calculated from 95th Percentile from 1.25m composites							
Domain	Lense	Zinc	Copper	Lead	Iron	Arsenic	Density
1	24 Raft	7.83	1.90	2.00	16.65	0.02	3.42
2	25 (Raft)	0.40	0.60	0.17	7.93	n/a	3.01
3	26 (Raft)	0.90	1.01	0.19	5.93	n/a	2.96
4	27 (upper raft)	0.48	6.13	0.02	10.89	n/a	3.12
5	27 (lower raft)	0.99	7.51	0.02	15.33	n/a	3.30
6	28 Raft	1.24	1.36	0.13	5.00	n/a	2.81
10	10	29.81	2.11	1.19	37.66	0.10	4.60
11	11	28.28	0.64	0.90	37.67	0.08	4.56
20	20	18.84	3.24	1.94	32.12	0.20	4.16
21	21	2.71	2.68	0.61	10.42	0.01	3.07
23	23	1.56	1.10	1.46	9.22	0.03	3.02
24	24	3.79	1.28	1.42	13.90	0.03	3.19
25	25	1.69	1.05	0.86	8.78	n/a	3.02
26	26	2.68	1.39	1.82	13.95	0.01	3.32
27	27	1.02	14.55	0.03	21.20	n/a	3.52
28	28	3.37	1.14	0.91	10.12	n/a	2.90
30	30	12.54	0.64	1.79	27.41	0.04	3.90
31	31	11.97	0.70	1.71	33.18	0.05	4.20
32	32	23.63	4.50	1.55	34.54	0.08	4.41
40	40	14.07	1.04	2.24	30.76	0.05	4.06

Table 14-14 presents the overall gold and silver metal removed by capping of the assays. It is not uncommon to see large differences between the original assay and capped value as in zone 21, 24, 25 and 28. These gold zones could benefit from a more constraining modeling method which could generate a statistical distribution closer to stationarity, and could present a smaller impact on the overall metal loss.

TABLE 14-14: PRECIOUS METAL REMOVED BY CAPPING

Base metal zones	10	11	20	30	31	32	40
Au g/t	1.75	0.35	2.69	1.64	2.23	5.96	1.67
Capped Au g/t	1.52	0.31	2.49	1.46	1.86	5.28	1.47
Au Removed	-15%	-11%	-8%	-13%	-20%	-13%	-14%
Ag g/t	22.05	25.03	32.56	34.94	30.73	51.44	30.14
Capped Ag g/t	20.57	23.65	32.10	32.56	29.52	50.32	29.59
Ag Removed	-7%	-6%	-1%	-7%	-4%	-2%	-2%

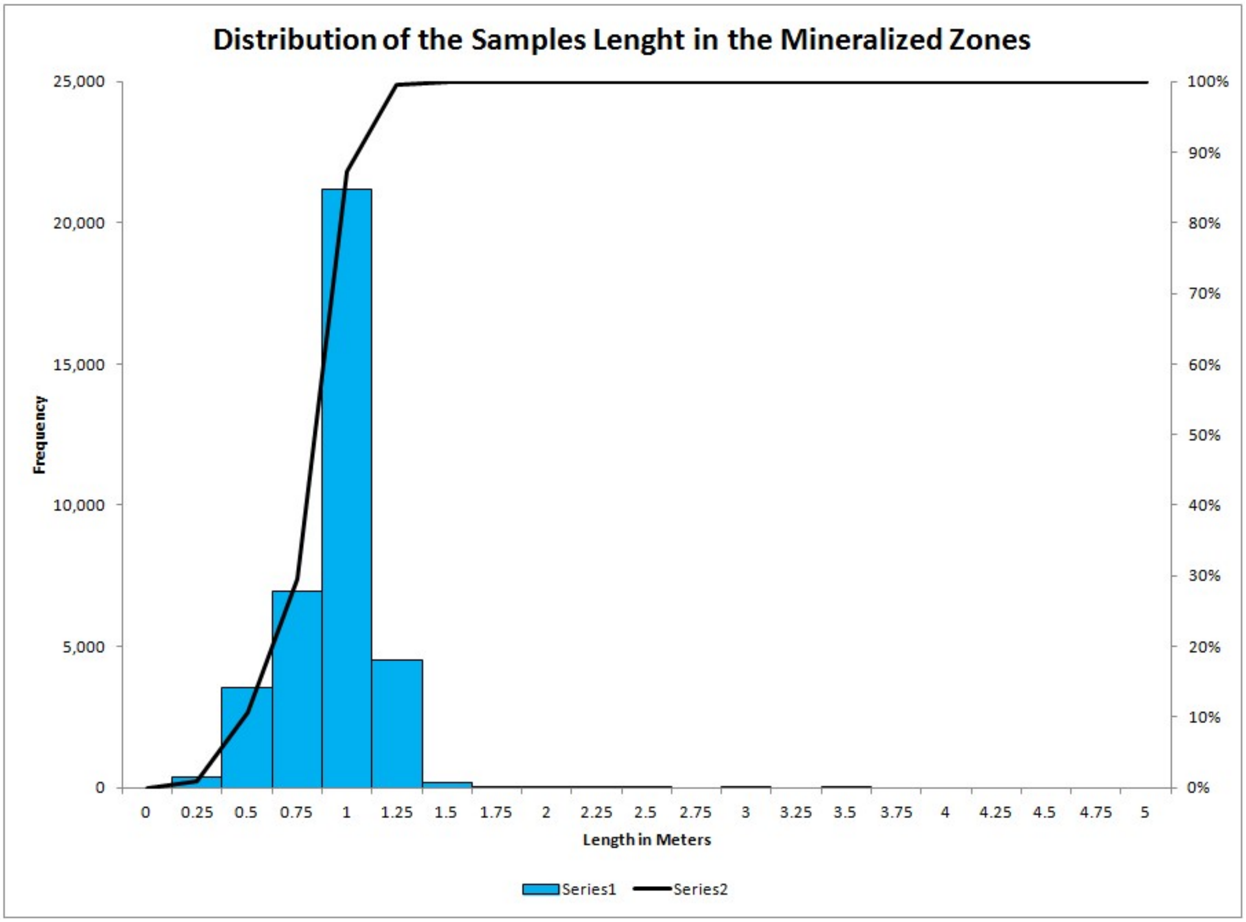
Gold zones	21	23	24	25	26	27	28
Au g/t	9.13	5.67	5.35	6.28	3.59	8.60	3.02
Capped Au g/t	6.86	4.96	4.38	5.09	3.27	7.61	2.47
Au Removed	-33%	-14%	-22%	-23%	-10%	-13%	-22%
Ag g/t	37.74	44.10	31.42	27.27	31.97	23.60	26.35
Capped Ag g/t	36.13	42.74	29.62	25.43	29.03	22.92	21.87
Ag Removed	-4%	-3%	-6%	-7%	-10%	-3%	-20%

Gold zones (raft)	1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Au g/t	1.08	2.79	7.58	6.76	4.80	2.92
Capped Au g/t	0.91	1.98	2.88	6.55	4.68	2.92
Au Removed	-19%	-41%	-163%	-3%	-3%	0%
Ag g/t	30.17	9.73	12.90	12.35	16.46	36.21
Capped Ag g/t	29.81	8.29	12.90	11.70	15.62	36.21
Ag Removed	-1%	-17%	0%	-6%	-5%	0%

14.4.2 Composites

In order to normalize the weight of influence of each sample, assay intervals were regularized by compositing drill hole data using the interpreted mineralized envelopes to break composites. A 1.25 m length was selected as the composites length, which represents approximately 99% of all the samples intervals. Figure 14-9 presents the distribution of the assay intervals.

FIGURE 14-9: SAMPLE LENGTH IN MINERALIZED ZONES



The 1.25 m intervals (+/- 0.62 m of threshold) were composited using "honour geology" from the coded drill hole file. The compositing was weighted both on length and density, and the weighting factor stored in the composite file (WFACT).

For bias assessment purposes, assay intervals were also composited into 5 m lengths (+/- 2.5 m of threshold) using the same methodology. The 5 m composites were used to estimate a nearest neighbour model (NN). EDA of the 1.25 m and 5 m composite statistics for Zn, Au Cu and Ag are shown in Table 14-15 to Table 14-22.

TABLE 14-15: LENGTH WEIGHTED 1.25 M COMPOSITE STATISTICS, ZINC (%)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
ZN%	1	24 Raft	131	0	25.92	1.17	0.09	12.44	3.53	3.01	7.83
	2	25 (Raft)	215	0	7.19	0.15	0.02	0.42	0.65	4.21	0.40
	3	26 (Raft)	31	0	4.22	0.20	0	0.65	0.81	3.98	0.90
	4	27 (upper raft)	66	0	0.74	0.10	0.05	0.02	0.15	1.55	0.48
	5	27 (lower raft)	65	0	1.37	0.25	0.14	0.10	0.32	1.26	0.99
	6	28 Raft	12	0	1.59	0.37	0.10	0.27	0.52	1.40	1.24
	10	10	6342	0	58.72	9.04	5.67	107.20	10.35	1.15	29.81
	11	11	374	0.01	48.36	11.72	9.92	85.20	9.23	0.79	28.28
	20	20	4927	0	46.54	6.00	3.95	43.11	6.57	1.09	18.84
	21	21	2306	0	27.30	0.59	0.12	1.71	1.31	2.22	2.71
	23	23	921	0	11.78	0.31	0.05	0.66	0.81	2.58	1.56
	24	24	1273	0	30.35	0.83	0.11	5.04	2.24	2.69	3.79
	25	25	3504	0	13.57	0.34	0.04	0.85	0.92	2.72	1.69
	26	26	629	0	33.42	0.60	0.07	4.03	2.01	3.34	2.68
	27	27	412	0	3.16	0.29	0.11	0.17	0.41	1.40	1.02
	28	28	84	0	12.73	0.71	0.08	3.07	1.75	2.48	3.37
	30	30	636	0.03	21.21	4.81	3.74	14.80	3.85	0.80	12.54
	31	31	787	0	18.56	4.60	3.70	13.76	3.71	0.81	11.97
	32	32	2156	0	45.98	8.16	5.65	61.37	7.83	0.96	23.63
	40	40	980	0	28.81	5.98	5.32	20.99	4.58	0.77	14.07

TABLE 14-16: LENGTH WEIGHTED UNCAPPED AND CAPPED 1.25 M COMPOSITE STATISTICS, GOLD (G/T)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AU	1	24 Raft	131	0	17.64	1.03	0.45	3.81	1.95	1.90	3.62
	2	25 (Raft)	215	0	110.96	2.70	0.57	74.75	8.65	3.21	8.92
	3	26 (Raft)	31	0	146.03	7.18	0.19	690.40	26.28	3.66	18.95
	4	27 (upper raft)	66	0	72.83	5.79	2.61	121.50	11.02	1.90	21.02
	5	27 (lower raft)	65	0	37.77	4.96	1.82	66.08	8.13	1.64	23.30
	6	28 Raft	12	0.04	6.76	2.08	1.75	4.04	2.01	0.97	5.28
	10	10	6342	0	163.46	1.71	0.43	25.63	5.06	2.97	7.16
	11	11	374	0	10.72	0.35	0.15	0.76	0.87	2.49	1.46
	20	20	4927	0	79.75	2.70	1.00	27.12	5.21	1.93	10.51
	21	21	2306	0	333.86	7.78	2.55	360.80	19.00	2.44	30.39
	23	23	921	0	213.36	5.40	1.98	165.60	12.87	2.39	20.49
	24	24	1273	0	169.86	5.03	1.85	132.80	11.52	2.29	22.46
	25	25	3504	0	301.93	5.55	2.06	205.00	14.32	2.58	21.92
	26	26	629	0	77.68	3.35	1.28	47.23	6.87	2.05	11.14
	27	27	412	0	137.93	8.00	2.58	221.70	14.89	1.86	31.26
	28	28	84	0	28.45	2.34	0.58	20.41	4.52	1.93	9.50
	30	30	636	0.06	31.91	1.58	0.88	8.10	2.85	1.81	4.57
	31	31	787	0.04	85.28	2.24	0.71	46.01	6.78	3.03	7.00
	32	32	2156	0	356.52	5.81	2.97	146.30	12.10	2.08	19.53
	40	40	980	0	48.91	1.63	0.78	10.62	3.26	2.00	5.86

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AUCAP	1	24 Raft	131	0	4.40	0.86	0.45	1.17	1.08	1.25	3.55
	2	25 (Raft)	215	0	13.00	1.94	0.57	9.33	3.06	1.57	8.92
	3	26 (Raft)	31	0	19.00	3.03	0.19	33.43	5.78	1.91	18.15
	4	27 (upper raft)	66	0	63.94	5.66	2.61	104.30	10.21	1.80	21.02
	5	27 (lower raft)	65	0	33.45	4.82	1.82	58.34	7.64	1.59	23.29
	6	28 Raft	12	0.04	6.76	2.08	1.75	4.04	2.01	0.97	5.28
	10	10	6342	0	22.00	1.50	0.43	7.76	2.79	1.86	7.05
	11	11	374	0	4.23	0.31	0.15	0.34	0.58	1.88	1.37
	20	20	4927	0	27.90	2.50	1.00	15.16	3.89	1.56	10.35
	21	21	2306	0	64.00	6.20	2.55	93.70	9.68	1.56	27.15
	23	23	921	0	54.10	4.77	1.98	66.89	8.18	1.71	19.92
	24	24	1273	0	40.00	4.18	1.85	45.23	6.73	1.61	18.95
	25	25	3504	0	50.00	4.60	2.06	55.86	7.47	1.63	19.87
	26	26	629	0	36.00	3.04	1.28	25.09	5.01	1.65	10.94
	27	27	412	0	68.86	7.12	2.58	115.60	10.75	1.51	29.29
	28	28	84	0	16.59	1.91	0.58	8.65	2.94	1.54	8.11
	30	30	636	0.06	13.94	1.43	0.88	3.45	1.86	1.30	4.46
	31	31	787	0.04	32.00	1.83	0.71	14.32	3.78	2.07	7.00
	32	32	2156	0	55.00	5.30	2.97	50.36	7.10	1.34	19.26
	40	40	980	0	14.43	1.46	0.78	4.02	2.01	1.37	5.29

TABLE 14-17: LENGTH WEIGHTED 1.25 M COMPOSITE STATISTICS, COPPER (%)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
CU%	1	24 Raft	131	0	5.58	0.51	0.16	0.76	0.87	1.71	1.90
	2	25 (Raft)	215	0	3.61	0.14	0.02	0.13	0.36	2.50	0.60
	3	26 (Raft)	31	0	2.35	0.16	0	0.27	0.52	3.18	1.01
	4	27 (upper raft)	66	0	12.37	1.70	0.92	5.27	2.30	1.35	6.13
	5	27 (lower raft)	65	0	9.83	2.71	2.05	6.30	2.51	0.93	7.51
	6	28 Raft	12	0	1.68	0.46	0.29	0.23	0.48	1.03	1.36
	10	10	6342	0	13.38	0.66	0.39	0.81	0.90	1.36	2.11
	11	11	374	0	1.04	0.27	0.22	0.03	0.19	0.70	0.64
	20	20	4927	0	12.57	0.81	0.39	1.34	1.16	1.43	3.24
	21	21	2306	0	12.57	0.66	0.26	1.23	1.11	1.68	2.68
	23	23	921	0	4.99	0.25	0.09	0.21	0.46	1.84	1.10
	24	24	1273	0	7.95	0.32	0.13	0.43	0.65	2.02	1.28
	25	25	3504	0	7.56	0.27	0.10	0.25	0.50	1.89	1.05
	26	26	629	0	7.25	0.42	0.22	0.43	0.66	1.57	1.39
	27	27	412	0	19.36	4.01	1.86	23.23	4.82	1.20	14.55
	28	28	84	0	1.80	0.30	0.12	0.15	0.39	1.32	1.14
	30	30	636	0.02	2.17	0.27	0.22	0.04	0.21	0.76	0.64
	31	31	787	0	3.95	0.26	0.18	0.12	0.34	1.33	0.70
	32	32	2156	0	13.91	1.42	0.82	2.73	1.65	1.16	4.50
	40	40	980	0	4.28	0.39	0.27	0.14	0.38	0.97	1.04

TABLE 14-18: LENGTH WEIGHTED UNCAPPED AND CAPPED 1.25 M COMPOSITE STATISTICS, SILVER (G/T)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AG	1	24 Raft	131	0	187.37	26.95	6.35	1754.00	41.88	1.55	133.32
	2	25 (Raft)	215	0	189.29	8.95	1.51	502.70	22.42	2.50	46.37
	3	26 (Raft)	31	0	215.50	12.11	0	1699.00	41.22	3.40	52.55
	4	27 (upper raft)	66	0	93.39	10.70	4.10	309.70	17.60	1.64	44.84
	5	27 (lower raft)	65	0	81.32	16.02	10.28	364.80	19.10	1.19	56.24
	6	28 Raft	12	0.39	96.35	24.14	8.04	1019.00	31.92	1.32	79.20
	10	10	6342	0	1106.82	21.45	12.63	1382.00	37.18	1.73	66.07
	11	11	374	1.09	210.34	23.87	17.81	623.90	24.98	1.05	63.24
	20	20	4927	0	477.48	32.05	18.71	1612.00	40.15	1.25	104.98
	21	21	2306	0	547.54	32.34	14.52	2523.00	50.23	1.55	131.93
	23	23	921	0	501.26	42.81	17.44	4646.00	68.16	1.59	162.99
	24	24	1273	0	474.17	29.55	11.98	2162.00	46.49	1.57	121.17
	25	25	3504	0	568.11	23.97	7.53	2241.00	47.34	1.97	108.96
	26	26	629	0	540.68	31.09	9.64	3810.00	61.72	1.99	134.12
	27	27	412	0	186.10	22.18	8.00	1045.00	32.32	1.46	92.02
	28	28	84	0	328.60	21.27	8.49	1672.00	40.90	1.92	79.42
	30	30	636	2.51	643.03	33.67	23.04	1941.00	44.06	1.31	84.32
	31	31	787	1.18	478.10	30.40	15.42	2252.00	47.45	1.56	101.57
	32	32	2156	0	470.29	50.90	36.06	2820.00	53.11	1.04	141.79
	40	40	980	0	291.00	29.95	21.09	894.50	29.91	1.00	86.00

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AGCAP	1	24 Raft	131	0	187.37	26.76	6.35	1719.00	41.46	1.55	132.06
	2	25 (Raft)	215	0	80.00	7.69	1.51	237.80	15.42	2.01	45.79
	3	26 (Raft)	31	0	215.50	12.11	0	1699.00	41.22	3.40	52.55
	4	27 (upper raft)	66	0	70.61	10.22	4.10	253.10	15.91	1.56	44.84
	5	27 (lower raft)	65	0	70.00	15.36	10.28	303.50	17.42	1.13	56.19
	6	28 Raft	12	0.39	96.35	24.14	8.04	1019.00	31.92	1.32	79.20
	10	10	6342	0	180.00	20.07	12.63	594.40	24.38	1.22	65.18
	11	11	374	1.09	120.00	22.61	17.81	372.50	19.30	0.85	62.28
	20	20	4927	0	320.00	31.70	18.71	1457.00	38.16	1.20	104.47
	21	21	2306	0	350.00	31.45	14.52	2125.00	46.09	1.47	126.24
	23	23	921	0	350.00	41.43	17.44	3770.00	61.40	1.48	161.86
	24	24	1273	0	230.00	28.09	11.98	1632.00	40.40	1.44	118.92
	25	25	3504	0	275.00	22.53	7.53	1585.00	39.82	1.77	105.93
	26	26	629	0	260.00	28.01	9.64	2163.00	46.50	1.66	131.69
	27	27	412	0	130.00	21.66	8.00	921.90	30.36	1.40	92.02
	28	28	84	0	130.00	17.78	8.49	549.20	23.44	1.32	62.49
	30	30	636	2.51	181.36	31.39	23.04	742.70	27.25	0.87	83.99
	31	31	787	1.18	280.00	29.27	15.42	1642.00	40.53	1.38	101.57
	32	32	2156	0	320.00	50.01	36.06	2374.00	48.72	0.97	140.39
	40	40	980	0	180.00	29.39	21.09	741.30	27.23	0.93	85.63

TABLE 14-19: LENGTH WEIGHTED 5 M COMPOSITE STATISTICS, ZINC (%)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
ZN%	1	24 Raft	37	0	13.32	1.11	0.15	6.71	2.59	2.33	5.77
	2	25 (Raft)	69	0	3.37	0.15	0.02	0.22	0.47	3.10	0.87
	3	26 (Raft)	15	0	2.70	0.20	0.01	0.48	0.69	3.55	0.88
	4	27 (upper raft)	15	0.01	0.38	0.10	0.07	0.01	0.10	0.99	0.29
	5	27 (lower raft)	17	0.02	1.01	0.26	0.20	0.05	0.23	0.89	0.63
	6	28 Raft	6	0.01	0.68	0.23	0.02	0.11	0.33	1.45	0.67
	10	10	1625	0	54.86	9.10	7.11	78.43	8.86	0.97	25.29
	11	11	128	0.01	39.86	11.40	9.97	58.44	7.64	0.67	23.63
	20	20	1250	0	41.62	6.14	4.77	29.84	5.46	0.89	16.55
	21	21	630	0	11.84	0.60	0.24	1.02	1.01	1.68	2.18
	23	23	262	0	5.40	0.30	0.07	0.32	0.57	1.90	1.36
	24	24	388	0	22.50	0.83	0.22	3.36	1.83	2.21	3.43
	25	25	934	0	5.25	0.35	0.06	0.46	0.68	1.91	1.74
	26	26	163	0	19.13	0.62	0.13	2.97	1.72	2.78	2.42
	27	27	108	0	2.24	0.30	0.20	0.14	0.37	1.23	0.83
	28	28	24	0	5.29	0.67	0.14	1.50	1.22	1.82	2.64
	30	30	169	0.14	13.69	4.89	4.30	9.50	3.08	0.63	11.00
	31	31	210	0.29	14.50	4.73	4.36	7.51	2.74	0.58	9.81
	32	32	585	0	27.68	8.21	6.91	41.95	6.48	0.79	20.12
	40	40	247	0	19.65	6.07	5.79	11.79	3.43	0.57	12.11

TABLE 14-20: LENGTH WEIGHTED UNCAPPED AND CAPPED 5 M COMPOSITE STATISTICS, GOLD (G/T)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AU	1	24 Raft	37	0	9.53	1.28	0.84	4.24	2.06	1.61	3.89
	2	25 (Raft)	69	0	32.42	2.47	0.66	25.44	5.04	2.04	11.13
	3	26 (Raft)	15	0	93.22	9.32	1.05	565.70	23.78	2.55	41.41
	4	27 (upper raft)	15	0.97	19.15	5.81	4.20	21.95	4.69	0.81	13.49
	5	27 (lower raft)	17	0.24	19.07	5.02	3.62	23.48	4.85	0.97	12.55
	6	28 Raft	6	0.40	3.26	2.12	2.70	1.63	1.28	0.60	3.21
	10	10	1625	0	82.57	1.71	0.50	14.26	3.78	2.21	6.90
	11	11	128	0	3.12	0.29	0.15	0.22	0.47	1.61	1.00
	20	20	1250	0	42.27	2.65	1.31	14.14	3.76	1.42	9.43
	21	21	630	0	120.67	7.60	3.92	151.40	12.30	1.62	26.09
	23	23	262	0	114.27	5.50	2.87	99.64	9.98	1.82	17.08
	24	24	388	0	125.28	5.34	2.75	94.10	9.70	1.82	25.09
	25	25	934	0	92.84	5.59	2.96	74.71	8.64	1.55	19.88
	26	26	163	0	37.15	3.53	1.84	29.32	5.42	1.53	10.47
	27	27	108	0	54.73	8.04	4.48	104.60	10.23	1.27	30.83
	28	28	24	0	10.90	2.46	0.99	11.70	3.42	1.39	10.61
	30	30	169	0.14	9.31	1.54	1.08	2.33	1.53	0.99	4.84
	31	31	210	0.11	40.14	2.03	0.89	18.46	4.30	2.11	6.72
	32	32	585	0	93.26	5.58	3.64	50.26	7.09	1.27	16.86
	40	40	247	0	20.80	1.69	0.96	5.92	2.43	1.44	4.53

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AUCAP	1	24 Raft	37	0	4.40	0.96	0.72	0.92	0.96	1.00	2.58
	2	25 (Raft)	69	0	10.74	1.81	0.66	6.68	2.58	1.43	7.07
	3	26 (Raft)	15	0	19.00	4.13	1.05	36.60	6.05	1.47	16.64
	4	27 (upper raft)	15	0.97	16.86	5.66	4.20	17.93	4.24	0.75	12.80
	5	27 (lower raft)	17	0.24	17.02	4.88	3.62	20.06	4.48	0.92	12.15
	6	28 Raft	6	0.40	3.26	2.12	2.70	1.63	1.28	0.60	3.21
	10	10	1625	0	15.76	1.49	0.50	4.95	2.23	1.49	6.33
	11	11	128	0	2.73	0.27	0.15	0.14	0.37	1.39	0.90
	20	20	1250	0	20.49	2.45	1.31	8.49	2.91	1.19	8.63
	21	21	630	0	37.83	6.07	3.91	39.43	6.28	1.03	19.46
	23	23	262	0	47.74	4.81	2.87	38.99	6.24	1.30	15.13
	24	24	388	0	40.00	4.42	2.75	32.90	5.74	1.30	15.41
	25	25	934	0	35.78	4.66	2.96	25.44	5.04	1.08	15.54
	26	26	163	0	24.93	3.17	1.84	15.23	3.90	1.23	9.86
	27	27	108	0	35.86	7.09	4.48	59.92	7.74	1.09	23.57
	28	28	24	0	7.36	1.99	0.99	5.52	2.35	1.18	6.84
	30	30	169	0.14	6.70	1.42	1.08	1.40	1.18	0.83	4.04
	31	31	210	0.11	18.31	1.69	0.89	5.92	2.43	1.44	6.72
	32	32	585	0	33.26	5.11	3.64	24.38	4.94	0.97	15.22
	40	40	247	0	10.74	1.48	0.96	2.19	1.48	1.00	4.22

TABLE 14-21: LENGTH WEIGHTED 5 M COMPOSITE STATISTICS, COPPER (%)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
CU%	1	24 Raft	37	0	4.53	0.63	0.30	0.82	0.91	1.43	2.15
	2	25 (Raft)	69	0	1.64	0.14	0.02	0.08	0.29	2.10	0.62
	3	26 (Raft)	15	0	2.01	0.18	0.02	0.26	0.51	2.91	0.76
	4	27 (upper raft)	15	0.03	3.84	1.75	1.75	1.10	1.05	0.60	3.26
	5	27 (lower raft)	17	0.16	7.73	2.73	2.59	2.42	1.56	0.57	4.67
	6	28 Raft	6	0.17	0.73	0.39	0.29	0.05	0.23	0.58	0.70
	10	10	1625	0	6.17	0.66	0.44	0.48	0.69	1.05	2.01
	11	11	128	0	0.66	0.26	0.25	0.02	0.15	0.58	0.54
	20	20	1250	0	6.66	0.83	0.44	0.99	0.99	1.20	3.06
	21	21	630	0	5.79	0.64	0.37	0.60	0.77	1.21	1.96
	23	23	262	0	2.81	0.23	0.11	0.13	0.36	1.54	0.93
	24	24	388	0	4.28	0.33	0.16	0.26	0.51	1.55	1.08
	25	25	934	0	6.20	0.28	0.16	0.18	0.42	1.52	0.88
	26	26	163	0	3.70	0.42	0.28	0.28	0.53	1.26	1.34
	27	27	108	0	15.90	4.06	3.11	14.48	3.81	0.94	11.11
	28	28	24	0	1.30	0.31	0.22	0.11	0.33	1.06	0.90
	30	30	169	0.05	1.57	0.28	0.23	0.03	0.18	0.65	0.60
	31	31	210	0.02	2.47	0.26	0.19	0.07	0.26	1.00	0.69
	32	32	585	0	10.11	1.40	0.93	1.97	1.40	1.00	3.81
	40	40	247	0	2.01	0.39	0.29	0.09	0.30	0.77	0.99

TABLE 14-22: LENGTH WEIGHTED UNCAPPED AND CAPPED 5 M COMPOSITE STATISTICS, SILVER (G/T)

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AG	1	24 Raft	37	0.06	128.73	27.02	14.09	1061.00	32.57	1.21	88.59
	2	25 (Raft)	69	0	92.60	8.58	1.72	317.40	17.82	2.08	50.48
	3	26 (Raft)	15	0	166.62	14.37	2.30	1792.00	42.33	2.95	59.15
	4	27 (upper raft)	15	1.02	28.55	10.89	6.99	75.36	8.68	0.80	26.76
	5	27 (lower raft)	17	0.60	57.57	16.28	14.62	157.90	12.57	0.77	33.33
	6	28 Raft	6	4.92	48.58	15.08	7.03	290.50	17.04	1.13	40.79
	10	10	1625	0	544.10	21.41	14.75	761.60	27.60	1.29	60.71
	11	11	128	2.73	97.59	22.29	18.95	236.30	15.37	0.69	51.83
	20	20	1250	0	223.72	32.02	23.12	926.80	30.44	0.95	89.38
	21	21	630	0	243.88	32.10	21.50	1171.00	34.22	1.07	95.57
	23	23	262	0	389.89	43.68	24.62	2847.00	53.36	1.22	139.00
	24	24	388	0	227.04	30.52	17.82	1311.00	36.20	1.19	109.51
	25	25	934	0	271.68	24.65	12.24	1277.00	35.73	1.45	97.17
	26	26	163	0	295.07	32.43	13.46	2373.00	48.71	1.50	123.90
	27	27	108	0	109.47	22.45	13.03	628.30	25.07	1.12	71.08
	28	28	24	0	143.54	22.83	14.18	991.60	31.49	1.38	62.30
	30	30	169	4.32	193.34	33.68	27.02	714.70	26.73	0.79	75.55
	31	31	210	2.15	246.51	29.74	18.59	1288.00	35.89	1.21	100.41
	32	32	585	0	293.54	49.90	40.22	1704.00	41.28	0.83	120.60
	40	40	247	0	150.34	30.03	24.60	486.80	22.06	0.73	76.25

	Domain	Lense	Count	Min	Max	Mean	Median	Variance	StDev	CV	95%
AGCAP	1	24 Raft	37	0.06	128.73	26.88	14.09	1055.00	32.47	1.21	88.59
	2	25 (Raft)	69	0	65.41	7.36	1.72	184.60	13.59	1.85	40.01
	3	26 (Raft)	15	0	166.62	14.37	2.30	1792.00	42.33	2.95	59.15
	4	27 (upper raft)	15	1.02	26.00	10.49	6.99	63.32	7.96	0.76	23.91
	5	27 (lower raft)	17	0.60	52.51	15.62	14.62	127.40	11.29	0.72	29.78
	6	28 Raft	6	4.92	48.58	15.08	7.03	290.50	17.04	1.13	40.79
	10	10	1625	0	173.92	20.03	14.74	354.40	18.83	0.94	54.71
	11	11	128	2.73	82.76	21.42	18.86	173.60	13.18	0.61	42.06
	20	20	1250	0	200.77	31.68	23.12	864.30	29.40	0.93	86.75
	21	21	630	0	225.35	31.22	21.50	1022.00	31.96	1.02	91.70
	23	23	262	0	350.00	42.29	24.62	2399.00	48.98	1.16	135.03
	24	24	388	0	192.61	29.24	17.82	1118.00	33.43	1.14	99.64
	25	25	934	0	203.74	23.18	12.24	973.30	31.20	1.35	84.13
	26	26	163	0	217.95	29.40	13.46	1573.00	39.67	1.35	110.12
	27	27	108	0	106.58	21.95	13.03	568.10	23.84	1.09	71.08
	28	28	24	0	70.14	18.82	14.18	382.30	19.55	1.04	58.22
	30	30	169	4.32	101.19	31.64	27.02	383.70	19.59	0.62	71.17
	31	31	210	2.15	171.12	28.74	18.59	984.10	31.37	1.09	93.49
	32	32	585	0	264.31	49.11	40.22	1516.00	38.93	0.79	116.89
	40	40	247	0	150.34	29.49	24.47	450.10	21.22	0.72	74.03

The length weighted mean grades of both 1.25 m and 5 m length composites are similar to those of the assays; therefore providing confidence that the compositing process is working as intended. The low to moderate CV values of zinc within the based metal zones suggest that no further domaining is warranted and that a linear interpolation method can be used. As for the gold zones, the relatively high CV values suggest that the gold and copper grades are variable and that a linear interpolation method may produce skewed results. Applying non-linear interpolation methods and/or revisions of the wire framing criteria should be further investigated for these zones in the future update of the resource model.

14.4.3 Variography

Down-hole and directional correlograms for Zn, Au, Cu, Ag, Pb, Fe, As and SG based on each individual grade shells were created using SAGE® software. Due to limited number of composite pairs in the raft portions of lenses 24, 25, 26, 27 and 28, the analysis was conducted on the primary lenses only. Lense 28 did not compute valid variograms and IDW was the interpolation method used.

The Zn variograms show low nugget effects in the base metal lenses with values lower or equal to 5% of the total sill. The ranges of correlation generally vary between 60 and 200 m in the major axis, 37 to 80 m in the semi-major axis and 12 to 54 m in the minor axis. As for gold, the nugget effect varies from low to high, up to 66% of the total sill. The ranges of correlation generally vary between 40 and 90 m in the major axis, 19 to 40 m in the semi-major axis and 7 to 25 m in the minor axis.

As an example, the downhole variogram of Zn in lense 32 and gold in zone 25 are shown in Figure 14-10 and Figure 14-11 respectively. A nugget and a nested model were fitted to the experimental correlograms. Correlogram model parameters for Zn, Cu, Au and Ag are shown in Table 14-23 and Table 14-24, both for the base metal lenses and the gold zones.

FIGURE 14-10: DOWNHOLE VARIOGRAM ZINC, LENSE 32

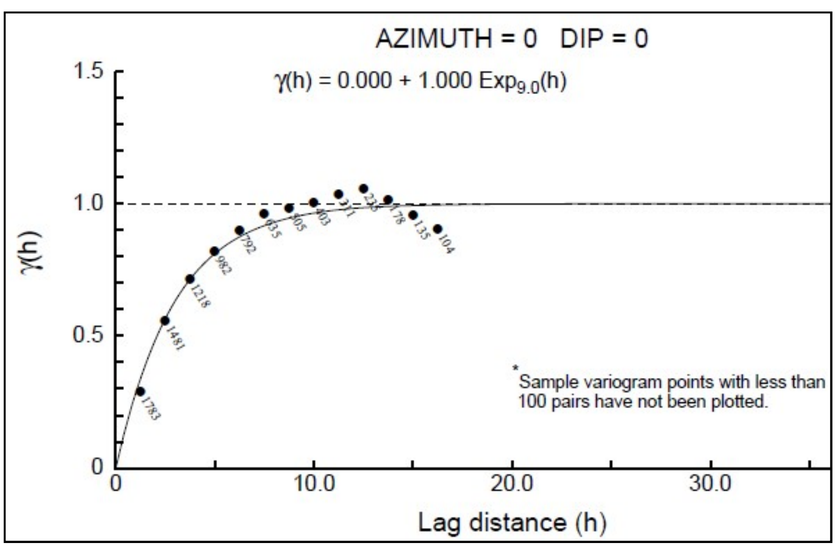


FIGURE 14-11: DOWNHOLE VARIOGRAM GOLD, LENSE 25

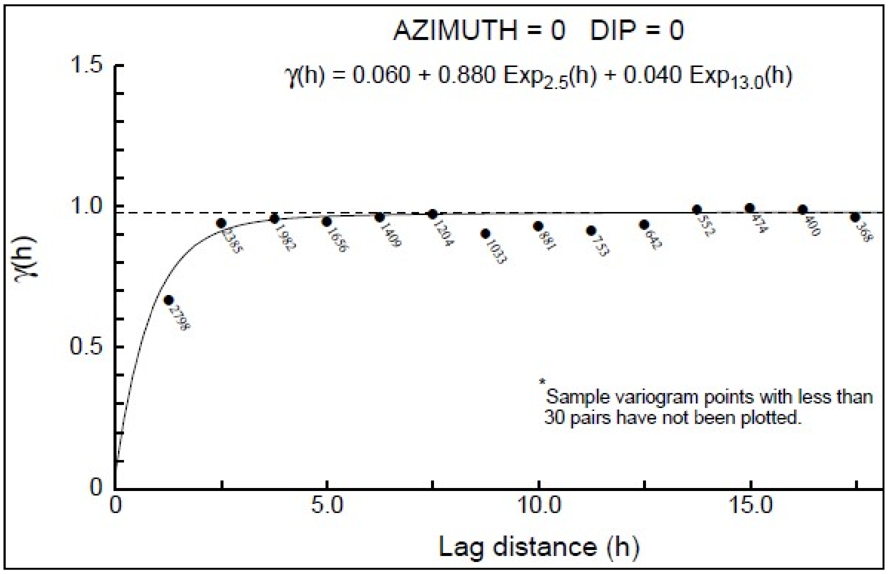


TABLE 14-23: VARIOGRAM MODELS AND ROTATION ANGLES IN BASE METAL LENSES

				First Structure							Nested Structure						
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Zinc	10	EXP	0.050	0.733	46.4	8.4	32.0	45	-42	82	0.217	80.0	54.2	200.0	114	7	-45
	11	EXP	0	0.591	18.9	40.0	1.9	-43	14	119	0.409	62.6	14.4	37.0	50	-50	0
	20	EXP	0	0.543	6.0	43.4	4.1	24	67	-21	0.457	38.3	44.3	150.0	-38	73	54
	30	EXP	0.050	0.809	36.5	20.5	4.8	-73	25	-14	0.141	95.1	90.0	30.0	55	-25	0
	31	EXP	0.001	0.879	6.4	5.1	20.0	27	61	21	0.120	131.0	40.0	26.0	50	-55	0
	32	EXP	0	0.747	8.2	15.8	7.4	66	48	58	0.253	20.4	72.4	150.0	-98	-36	30
	40	EXP	0.100	0.846	17.2	13.4	7.4	-14	19	34	0.054	100.0	49.3	12.7	50	-20	0
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Copper	10	EXP	0.100	0.765	16.7	6.0	22.2	-41	0	51	0.135	150.0	17.8	200.0	82	-19	-64
	11	EXP	0.051	0.640	36.2	19.4	1.8	-18	-39	-49	0.309	149.7	43.0	27.2	50	-50	0
	20	EXP	0	0.508	12.9	19.5	2.8	121	-6	42	0.492	37.0	96.3	150.0	-11	62	39
	30	EXP	0	0.642	27.5	23.1	3.2	42	13	-3	0.358	90.0	47.3	20.0	55	-25	0
	31	EXP	0	0.624	64.2	5.6	20.3	21	-16	28	0.376	101.0	54.0	26.0	50	-55	0
	32	EXP	0.006	0.687	20.0	5.1	17.6	30	-18	64	0.307	100.0	81.7	60.0	55	-40	0
	40	EXP	0.100	0.505	10.0	5.3	20.0	-9	8	63	0.395	54.5	46.4	30.0	50	-20	0
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Gold	10	EXP	0.100	0.756	18.3	5.7	6.5	162	12	-12	0.144	29.3	100.0	150.0	21	55	-31
	11	EXP	0.403	0.553	58.4	10.0	2.6	0	-50	-59	0.043	58.4	54.8	15.0	50	-50	0
	20	EXP	0.200	0.615	6.9	18.2	4.8	147	102	65	0.185	31.7	47.9	100.0	-3	67	32
	30	EXP	0.500	0.448	10.0	20.0	4.7	22	-42	15	0.052	60.0	60.0	20.0	55	-25	0
	31	EXP	0.390	0.462	60.0	2.8	19.6	0	6	6	0.148	100.0	56.0	38.0	50	-55	0
	32	EXP	0.300	0.607	1.6	9.2	13.5	-79	63	84	0.093	163.3	66.7	44.6	55	-40	0
	40	EXP	0.560	0.296	20.0	3.2	5.6	-13	61	27	0.144	109.7	49.2	20.0	50	-20	0
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Silver	10	EXP	0	0.970	8.4	2.9	10.9	40	8	-13	0.030	150.0	10.9	150.0	-49	-14	64
	11	EXP	0.365	0.453	28.9	12.1	3.8	87	31	-70	0.181	50.0	4.6	150.0	-58	91	58
	20	EXP	0.100	0.708	8.5	7.1	23.4	152	92	18	0.192	25.6	50.1	150.0	5	63	-120
	30	EXP	0.363	0.566	20.0	15.0	5.5	-97	-1	-28	0.071	90.0	42.3	20.0	55	-25	0
	31	EXP	0.212	0.607	30.0	4.3	10.0	-8	30	15	0.181	114.9	80.4	20.0	50	-55	0
	32	EXP	0.120	0.646	31.1	9.3	4.3	7	-2	55	0.234	100.0	64.6	48.8	55	-40	0
	40	EXP	0.012	0.811	14.8	22.6	3.5	-12	33	18	0.177	69.5	64.6	30.0	50	-20	0

Note: Ranges are in metres and search ellipse orientations are given using GSLIB-MS rotation convention.

TABLE 14-24: VARIOGRAM MODELS AND ROTATION ANGLES IN GOLD ZONES

				First Structure							Nested Structure						
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Zinc	21	SPH	0.100	0.600	5.0	6.0	8.0	-41	26	63	0.300	70.0	30.0	90.0	-41	26	63
Zinc	23	EXP	0	0.653	37.5	2.2	26.5	-32	5	22	0.347	86.7	70.9	20.0	50	-45	0
Zinc	24	EXP	0.012	0.725	20.0	12.7	2.3	3	0	-1	0.263	70.0	40.0	15.0	80	-25	0
Zinc	25	SPH	0.360	0.500	0.1	6.0	9.0	15	-39	37	9.000	75.0	50.0	120.0	-39	37	9
Zinc	26	EXP	0.001	0.576	10.0	16.9	3.0	-49	4	13	0.423	58.0	38.0	15.0	61	-27	0
Zinc	27	EXP	0.050	0.500	10.0	6.0	15.0	-1	36	-1	0.450	55.0	25.0	80.0	-1	36	-1
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Copper	21	SPH	0.240	0.500	5.0	10.0	30.0	-12	-16	39	0.260	60.0	30.0	90.0	-12	-16	39
Copper	23	EXP	0.150	0.391	30.0	21.3	3.4	-58	8	67	0.459	80.0	59.7	23.9	50	-45	0
Copper	24	EXP	0.343	0.466	8.5	2.8	23.4	-56	-9	24	0.191	77.4	40.0	15.0	80	-25	0
Copper	25	SPH	0.360	0.500	17.0	11.0	11.0	2	6	90	0.140	60.0	30.0	120.0	2	6	90
Copper	26	EXP	0.010	0.671	29.0	36.3	10.1	97	44	34	0.319	50.0	30.0	15.0	61	-27	0
Copper	27	EXP	0.050	0.500	6.0	4.0	12.0	-4	28	-2	0.450	40.0	15.0	60.0	-4	28	-2
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Gold	21	SPH	0.380	0.420	8.0	3.0	10.0	-86	-27	6	0.200	40.0	25.0	60.0	-86	-27	6
Gold	23	EXP	0.700	0.263	30.6	6.4	10.4	8	-16	27	0.039	40.0	19.4	7.0	50	-45	0
Gold	24	EXP	0.500	0.355	25.6	1.9	4.6	-27	19	-15	0.145	70.6	40.0	20.0	80	-25	0
Gold	25	SPH	0.060	0.770	10.0	10.0	8.0	-68	40	-32	0.170	70.0	30.0	15.0	-68	40	-32
Gold	26	EXP	0.660	0.039	26.0	16.0	4.0	-22	-36	-10	0.301	55.0	32.1	20.0	61	-27	0
Gold	27	SPH	0.050	0.750	5.0	5.0	10.0	-4	18	5	0.200	40.0	20.0	90.0	-4	18	5
	LENSE CODE	Variogram Model	Nugget	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3	Sill	Major axis	Semi Major axis	Minor axis	ROT1	ROT2	ROT3
Silver	21	SPH	0.470	0.430	10.0	15.0	30.0	-9	-16	9	0.100	30.0	50.0	120.0	-9	-16	9
Silver	23	EXP	0.600	0.247	24.7	4.0	30.0	65	-3	3	0.153	36.2	18.7	10.0	50	-45	0
Silver	24	EXP	0.003	0.789	8.2	1.6	5.1	-72	-12	7	0.208	90.0	40.0	20.0	80	-25	0
Silver	25	SPH	0.230	0.510	8.0	6.0	7.0	-20	-27	86	0.260	50.0	25.0	80.0	-20	-27	86
Silver	26	EXP	0.466	0.255	20.0	10.0	4.4	-33	-35	57	0.279	80.0	40.0	20.0	61	-27	0
Silver	27	SPH	0.050	0.500	5.0	10.0	8.0	91	-3	56	0.450	20.0	50.0	30.0	91	-3	56

Note: Ranges are in metres and search ellipse orientations are given using GSLIB-MS rotation convention.

14.5 Estimation and Interpolation Methods

The mineralized envelopes were used to code blocks in the model using a multi ore percent method, where ORE1 and ORE%1 represent the majority block, ORE2 and ORE%2 represent the partial block and ORE3 and ORE%3 represent the minority block. Grade interpolation used a strict composite and block matching system based on the mineralized envelopes codes. For example, in the case of Lense 10, only composites coded as Lense 10 were used to interpolate blocks grades.

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 based on the resource wireframe widths and to match the smallest mining unit at Lalor mine. The ordinary kriging (OK) grade interpolation was completed on the uncapped (or unrestricted) and capped (or restricted) with composites of 1.25 m in length, using three passes with increasing requirements. The composite selection parameters for grade estimation in each domain (minimum, maximum, maximum number of composites per hole) were selected to minimize bias.

The first interpolation pass uses 100% of the variogram ranges and is restricted to a minimum of four composites, a maximum of 16 composites (with a maximum of four composites per hole), without quadrant declustering. The second interpolation pass uses 75% of the variogram ranges and is restricted to a minimum of six composites, a maximum of 16 composites (with a maximum of four composites per hole) and uses quadrant declustering. Finally, the third interpolation pass uses 50% of the variogram ranges and is restricted to a minimum of nine composites, a maximum of 16 composites (with a maximum of four composites per hole) and uses quadrant declustering. At the end of the interpolation, more than 95% of the blocks within the mineralized envelopes had been interpolated. In order to estimate the remaining un-interpolated blocks, a “fill pass” was added at the process, using a minimum of two composites to interpolate the blocks. The “fill pass” used ellipses sizes equivalent to 150% to 225% of the variograms ranges.

The interpolations use both the length and density of the composites (WFACT) to weight the grade of the nearest neighbour (NN using 5 m composites), the inverse distance (IDW) using and the ordinary kriging (OK) interpolations. The NN and IDW interpolations were used to monitor the quality of the OK interpolation.

14.6 Block Model Validation

The Lalor block model was validated to ensure appropriate honouring of the input data by the following methods:

- Visual inspection of the OK block model grades in plan and section views in comparison to composites grade
- Metal removed via grade capping and high yield restriction methodology

- Comparison between the interpolation methods of NN and IDW to confirm the absence of global bias in the OK grade model
- Swath plot comparisons of the estimation methods to investigate local bias
- Review of block model ordinary kriging quality control parameters
- Comparison of grade tonnage curves and statistics by estimation method
- Third party review of the block model and estimation process

14.6.1 Visual Inspection

Visual inspection of block grade versus composited data was conducted in section view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model. As an example, two long sections (looking west) are presented. Figure 14-12 presents the zinc grade in Lense 10 while Figure 14-13 presents the gold grade in Lense 25.

FIGURE 14-12: EW CROSS SECTION N630 SHOWING OK MODEL AND COMPOSITES ZINC GRADE OF BASE METAL LENSE 10

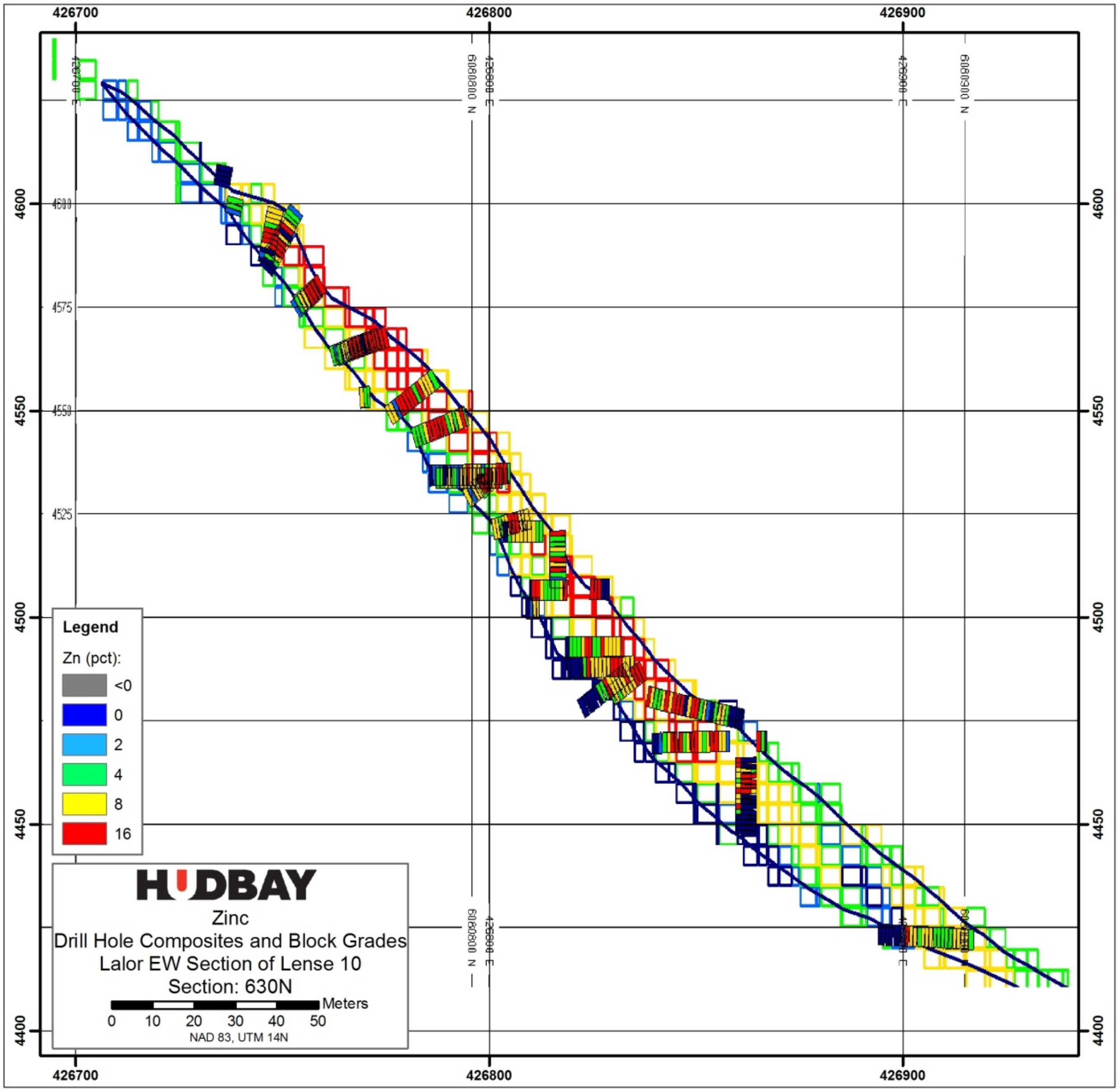
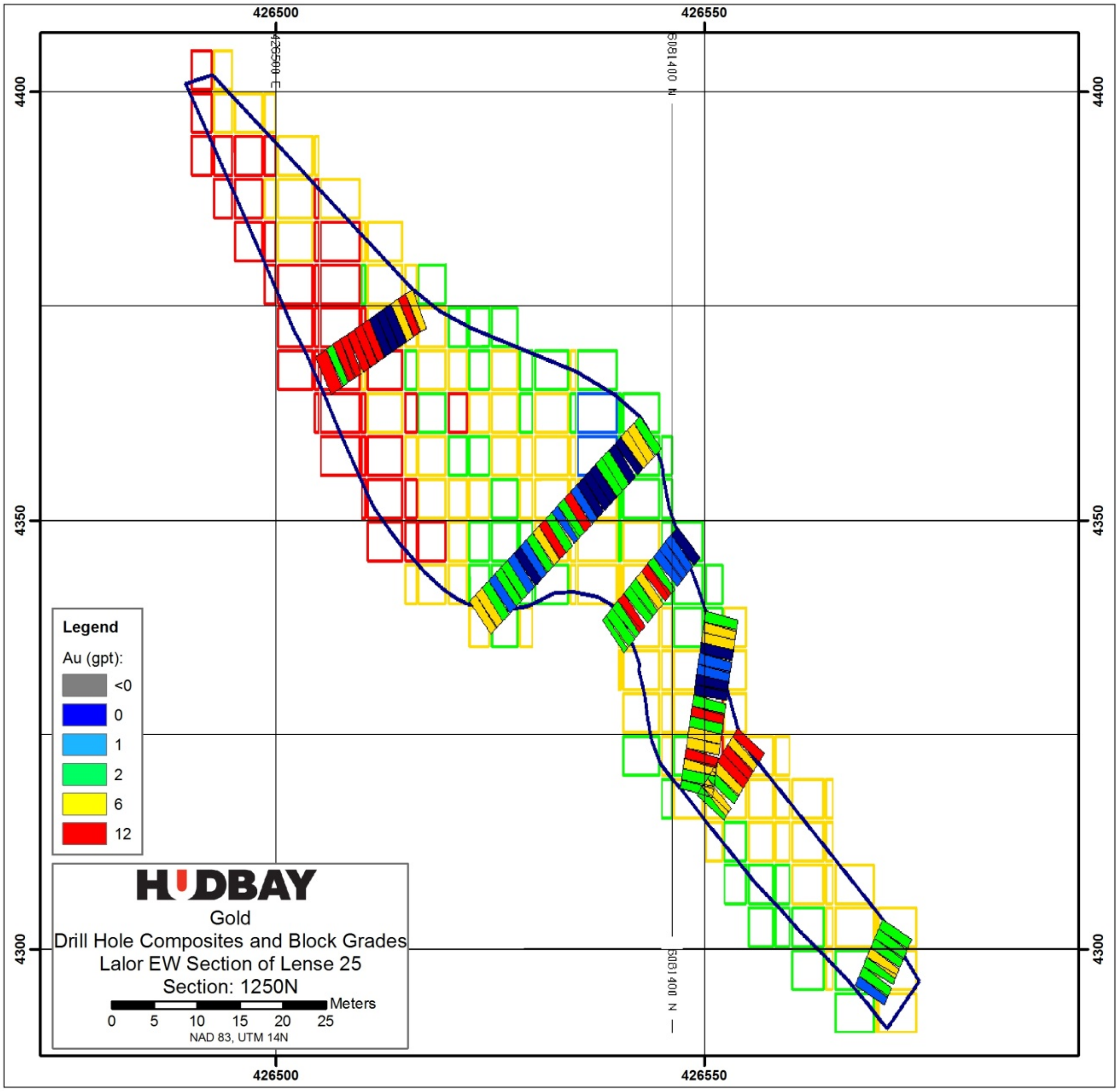


FIGURE 14-13: EW CROSS SECTION 1250N SHOWING OK MODEL AND COMPOSITES GOLD GRADE IN GOLD ZONE 25



14.6.2 Metal Removed by Capping

The impact of capping was evaluated by estimating uncapped and capped grade and the unrestricted and restricted grade in the block model. Generally the amounts of metal removed by capping from the different interpolation methods are consistent with the difference of the capped and uncapped assays.

The percentages of metal removed by capping from the NN, IDW and OK models are shown in Table 14-25 to Table 14-28. The amount of capping for gold in the raft part of zone 26 is high. The limited drilling (35 samples only) is the likely source of difference between the un-capped and capped values. This zone is considered inferred resource and should be drilled in the future.

The amount of zinc metal loss in the base metal lenses is lower than 5% for all the lenses except for lense 31 (-9%) and lense 40 (-13%). It is believed that the larger drill spacing in parts of these two lenses is the main source of difference between the un-restricted and restricted values.

TABLE 14-25: NN, IDW AND OK MODEL, ZINC REMOVED BY RESTRICTION IN BASE METAL LENSES

Base metal zones	10	11	20	30	31	32	40
Zn NN	7.71	11.00	6.70	5.08	4.84	8.56	6.34
Restricted Zn NN	7.66	10.89	6.70	5.02	4.76	8.56	6.09
Zn removed NN	-1%	-1%	0%	-1%	-2%	0%	-4%
Zn IDW	8.44	11.51	7.10	5.44	5.02	9.25	6.66
Restricted Zn IDW	8.36	10.99	7.09	5.15	4.60	9.25	5.98
Zn removed IDW	-1%	-5%	0%	-6%	-9%	0%	-11%
Zn OK	7.68	10.80	6.67	5.06	4.90	8.32	6.43
Restricted Zn OK	7.60	10.31	6.66	4.85	4.48	8.32	5.67
Zn removed OK	-1%	-5%	0%	-4%	-9%	0%	-13%

TABLE 14-26: NN, IDW AND OK MODEL, GOLD REMOVED BY CAPPING IN BASE METAL LENSES AND GOLD ZONES

Base metal zones	10	11	20	30	31	32	40
Au NN	2.09	0.54	2.42	1.47	1.79	6.38	1.71
Restricted Au NN	1.84	0.51	2.19	1.40	1.57	5.42	1.48
Au removed NN	-14%	-6%	-11%	-5%	-14%	-18%	-16%
Au IDW	1.83	0.55	2.31	1.48	1.73	5.95	1.57
Restricted Au IDW	1.61	0.51	2.10	1.39	1.49	5.21	1.41
Au removed IDW	-14%	-8%	-10%	-6%	-16%	-14%	-11%
Au OK	1.96	0.49	2.36	1.55	1.76	6.01	1.59
Restricted Au OK	1.68	0.45	2.15	1.45	1.51	5.27	1.42
Au removed OK	-17%	-9%	-10%	-7%	-17%	-14%	-12%

Gold zones	21	23	24	25	26	27	28
Au NN	7.21	5.20	5.49	5.86	3.69	7.29	2.83
Restricted Au NN	5.82	4.71	4.63	4.96	3.35	6.59	2.08
Au removed NN	-24%	-10%	-19%	-18%	-10%	-11%	-36%
Au IDW	7.17	5.31	5.31	5.95	3.54	7.19	2.52
Restricted Au IDW	5.87	4.79	4.48	5.00	3.20	6.62	1.92
Au removed IDW	-22%	-11%	-19%	-19%	-11%	-9%	-31%
Au OK	6.96	5.47	5.19	5.96	3.52	6.90	1.92
Restricted Au OK	5.75	4.89	4.40	5.11	3.16	6.39	1.60
Au removed OK	-21%	-12%	-18%	-17%	-11%	-8%	-20%

Gold zones (raft)	1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Au NN	1.27	2.26	17.47	5.90	5.06	2.08
Restricted Au NN	0.96	1.88	5.43	5.73	4.91	2.08
Au removed NN	-32%	-20%	-222%	-3%	-3%	0%
Au IDW	1.04	2.31	19.60	7.23	5.81	1.95
Restricted Au IDW	0.86	1.88	5.16	7.00	5.62	1.95
Au removed IDW	-21%	-23%	-280%	-3%	-3%	0%
Au OK	1.13	2.58	11.85	7.59	4.88	1.97
Restricted Au OK	0.88	1.94	4.04	7.32	4.76	1.97
Au removed OK	-28%	-33%	-193%	-4%	-3%	0%

TABLE 14-27: NN, IDW AND OK MODEL, COPPER REMOVED BY RESTRICTION IN BASE METAL LENSES AND GOLD ZONES

Base metal zones	10	11	20	30	31	32	40
Cu NN	0.67	0.23	0.79	0.26	0.23	1.46	0.39
Restricted Cu NN	0.61	0.23	0.79	0.25	0.23	1.44	0.39
Cu removed NN	-10%	0%	0%	-4%	0%	-1%	0%
Cu IDW	0.65	0.23	0.80	0.26	0.23	1.54	0.39
Restricted Cu IDW	0.59	0.22	0.80	0.24	0.22	1.47	0.37
Cu removed IDW	-10%	-5%	0%	-8%	-5%	-5%	-5%
Cu OK	0.65	0.24	0.79	0.26	0.23	1.42	0.39
Restricted Cu OK	0.58	0.22	0.79	0.25	0.22	1.35	0.37
Cu removed OK	-12%	-9%	0%	-4%	-5%	-5%	-5%

Gold zones	21	23	24	25	26	27	28
Cu NN	0.59	0.20	0.33	0.30	0.42	4.01	0.38
Restricted Cu NN	0.58	0.19	0.30	0.29	0.41	3.72	0.30
Cu removed NN	-2%	-5%	-10%	-3%	-2%	-8%	-27%
Cu IDW	0.62	0.21	0.33	0.31	0.40	4.14	0.35
Restricted Cu IDW	0.59	0.18	0.27	0.28	0.37	3.78	0.27
Cu removed IDW	-5%	-17%	-22%	-11%	-8%	-10%	-30%
Cu OK	0.61	0.22	0.32	0.30	0.40	4.02	0.31
Restricted Cu OK	0.57	0.18	0.26	0.27	0.37	3.56	0.28
Cu removed OK	-7%	-22%	-23%	-11%	-8%	-13%	-11%

Gold zones (raft)	1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Cu NN	0.69	0.11	0.14	1.79	2.71	0.46
Restricted Cu NN	0.60	0.10	0.09	1.79	2.69	0.46
Cu removed NN	-15%	-10%	-56%	0%	-1%	0%
Cu IDW	0.56	0.12	0.12	1.97	2.81	0.46
Restricted Cu IDW	0.51	0.10	0.08	1.66	2.56	0.37
Cu removed IDW	-10%	-20%	-50%	-19%	-10%	-24%
Cu OK	0.62	0.11	0.16	2.13	2.73	0.39
Restricted Cu OK	0.52	0.09	0.08	1.72	2.57	0.34
Cu removed OK	-19%	-22%	-100%	-24%	-6%	-15%

TABLE 14-28: NN, IDW AND OK MODEL, SILVER REMOVED BY CAPPING IN BASE METAL LENSES AND GOLD ZONES

Base metal zones	10	11	20	30	31	32	40
Ag NN	25.69	22.95	30.33	32.84	29.49	54.32	28.67
Restricted Ag NN	23.54	22.14	30.03	30.85	28.72	52.64	28.29
Ag removed NN	-9%	-4%	-1%	-6%	-3%	-3%	-1%
Ag IDW	23.44	23.01	29.67	33.04	29.49	54.04	28.39
Restricted Ag IDW	21.91	22.24	29.40	31.02	28.71	52.52	28.03
Ag removed IDW	-7%	-3%	-1%	-7%	-3%	-3%	-1%
Ag OK	24.01	24.12	30.47	32.98	30.16	53.00	28.68
Restricted Ag OK	22.30	23.20	30.14	30.92	29.34	51.55	28.27
Ag removed OK	-8%	-4%	-1%	-7%	-3%	-3%	-1%

Gold zones	21	23	24	25	26	27	28
Ag NN	32.72	41.75	32.65	27.96	40.08	21.68	22.59
Restricted Ag NN	31.92	40.62	31.19	26.36	36.38	20.83	19.73
Ag removed NN	-3%	-3%	-5%	-6%	-10%	-4%	-14%
Ag IDW	33.62	42.53	31.73	28.31	37.18	22.36	21.02
Restricted Ag IDW	32.87	41.19	30.12	26.59	33.64	21.47	18.46
Ag removed IDW	-2%	-3%	-5%	-6%	-11%	-4%	-14%
Ag OK	33.50	42.34	31.18	28.63	36.37	21.56	17.97
Restricted Ag OK	32.72	41.03	30.00	27.09	33.16	20.70	16.01
Ag removed OK	-2%	-3%	-4%	-6%	-10%	-4%	-12%

Gold zones (raft)	1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Ag NN	21.81	8.32	29.98	11.68	14.85	21.88
Restricted Ag NN	21.73	7.10	29.98	11.00	14.50	21.88
Ag removed NN	0%	-17%	0%	-6%	-2%	0%
Ag IDW	21.26	9.00	32.28	13.15	16.03	21.25
Restricted Ag IDW	21.13	7.43	32.28	12.24	15.64	21.25
Ag removed IDW	-1%	-21%	0%	-7%	-2%	0%
Ag OK	22.05	9.11	21.37	13.32	15.26	21.02
Restricted Ag OK	21.94	7.40	21.37	12.18	14.90	21.02
Ag removed OK	-1%	-23%	0%	-9%	-2%	0%

14.6.3 Global Bias Checks

A comparison between the interpolation methods estimates was completed on all the blocks within the lenses. Differences between the NN, IDW and OK grades are acceptable in most lenses aside from the raft were only limited information is available. The differences are summarized in Table 14-29 to Table 14-32.

TABLE 14-29: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR ZINC

		10	11	20	30	31	32	40
Base Metal Lenses	NN	7.66	10.89	6.70	5.02	4.76	8.56	6.09
	IDW	8.36	10.99	7.09	5.15	4.60	9.25	5.98
	OK	7.60	10.31	6.66	4.85	4.48	8.32	5.67
	NN & OK %	-1%	-5%	-1%	-3%	-6%	-3%	-7%
	IDW & OK %	-9%	-6%	-6%	-6%	-3%	-10%	-5%

		21	23	24	25	26	27	28
Gold Zones	NN	0.63	0.28	0.79	0.35	0.56	0.24	0.55
	IDW	0.59	0.26	0.72	0.33	0.50	0.24	0.52
	OK	0.55	0.24	0.68	0.33	0.46	0.23	0.47
	NN & OK %	-13%	-14%	-14%	-6%	-18%	-4%	-15%
	IDW & OK %	-7%	-8%	-6%	0%	-8%	-4%	-10%

		1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Gold Zones (raft)	NN	0.93	0.12	0.08	0.10	0.20	0.33
	IDW	0.91	0.10	0.08	0.10	0.20	0.26
	OK	0.83	0.10	0.05	0.09	0.19	0.27
	NN & OK %	-11%	-17%	-38%	-10%	-5%	*-18%*
	IDW & OK %	-9%	0%	-38%	-10%	-5%	*4%*

TABLE 14-30: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR GOLD

		10	11	20	30	31	32	40
Base Metal Lenses	NN	1.84	0.51	2.19	1.40	1.57	5.42	1.48
	IDW	1.61	0.51	2.10	1.39	1.49	5.21	1.41
	OK	1.68	0.45	2.15	1.45	1.51	5.27	1.42
	NN & OK %	-9%	-12%	-2%	4%	-4%	-3%	-4%
	IDW & OK %	4%	-12%	2%	4%	1%	1%	1%

		21	23	24	25	26	27	28
Gold Zones	NN	5.82	4.71	4.63	4.96	3.35	6.59	2.08
	IDW	5.87	4.79	4.48	5.00	3.20	6.62	1.92
	OK	5.75	4.89	4.40	5.11	3.16	6.39	1.60
	NN & OK %	-1%	4%	-5%	3%	-6%	-3%	-23%
	IDW & OK %	-2%	2%	-2%	2%	-1%	-3%	-17%

		1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Gold Zones (raft)	NN	0.96	1.88	5.43	5.73	4.91	2.08
	IDW	0.86	1.88	5.16	7.00	5.62	1.95
	OK	0.88	1.94	4.04	7.32	4.76	1.97
	NN & OK %	-8%	3%	-26%	28%	-3%	*-5%*
	IDW & OK %	2%	3%	-22%	5%	-15%	*1%*

TABLE 14-31: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR COPPER

		10	11	20	30	31	32	40
Base Metal Lenses	NN	0.61	0.23	0.79	0.25	0.23	1.44	0.39
	IDW	0.59	0.22	0.80	0.24	0.22	1.47	0.37
	OK	0.58	0.22	0.79	0.25	0.22	1.35	0.37
	NN & OK %	-5%	-4%	0%	0%	-4%	-6%	-5%
	IDW & OK %	-2%	0%	-1%	4%	0%	-8%	0%

		21	23	24	25	26	27	28
Gold Zones	NN	0.58	0.19	0.30	0.29	0.41	3.72	0.30
	IDW	0.59	0.18	0.27	0.28	0.37	3.78	0.27
	OK	0.57	0.18	0.26	0.27	0.37	3.56	0.28
	NN & OK %	-2%	-5%	-13%	-7%	-10%	-4%	-7%
	IDW & OK %	-3%	0%	-4%	-4%	0%	-6%	4%

		1 (raft of	2 (raft of	3 (raft of	4 (upper	5 (lower	6 (raft of
Gold Zones (raft)	NN	0.60	0.10	0.09	1.79	2.69	0.46
	IDW	0.51	0.10	0.08	1.66	2.56	0.37
	OK	0.52	0.09	0.08	1.72	2.57	0.34
	NN & OK %	-13%	-10%	-11%	-4%	-4%	-26%
	IDW & OK %	2%	-10%	0%	4%	0%	-8%

TABLE 14-32: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS SILVER

		10	11	20	30	31	32	40
Base Metal Lenses	NN	23.54	22.14	30.03	30.85	28.72	52.64	28.29
	IDW	21.91	22.24	29.40	31.02	28.71	52.52	28.03
	OK	22.30	23.20	30.14	30.92	29.34	51.55	28.27
	NN & OK %	-5%	5%	0%	0%	2%	-2%	0%
	IDW & OK %	2%	4%	3%	0%	2%	-2%	1%

		21	23	24	25	26	27	28
Gold Zones	NN	31.92	40.62	31.19	26.36	36.38	20.83	19.73
	IDW	32.87	41.19	30.12	26.59	33.64	21.47	18.46
	OK	32.72	41.03	30.00	27.09	33.16	20.70	16.01
	NN & OK %	3%	1%	-4%	3%	-9%	-1%	-19%
	IDW & OK %	0%	0%	0%	2%	-1%	-4%	-13%

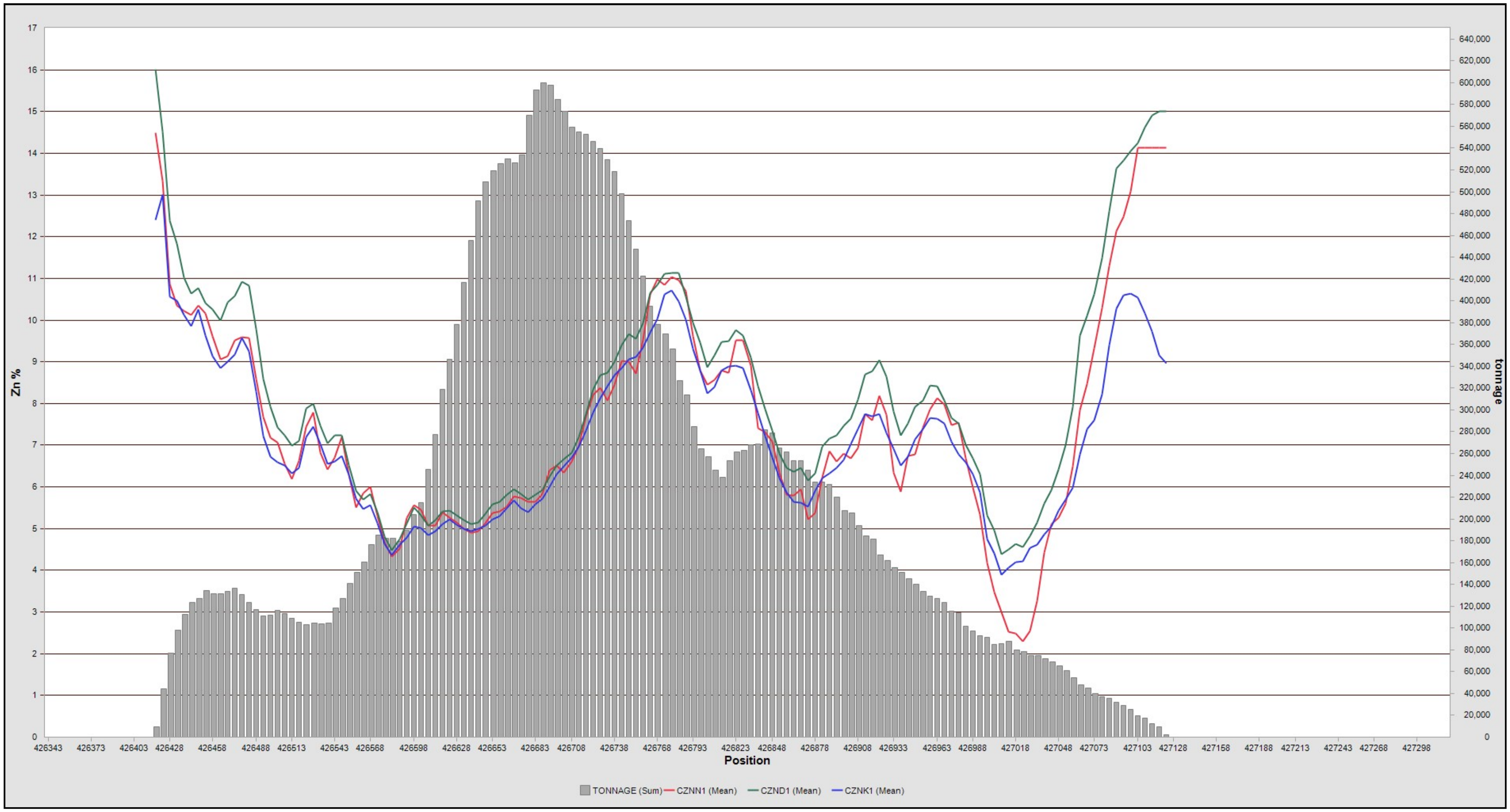
		1 (raft of 24)	2 (raft of 25)	3 (raft of 26)	4 (upper raft of 27)	5 (lower raft of 27)	6 (raft of 28)
Gold Zones (raft)	NN	21.73	7.10	29.98	11.00	14.50	21.88
	IDW	21.13	7.43	32.28	12.24	15.64	21.25
	OK	21.94	7.40	21.37	12.18	14.90	21.02
	NN & OK %	1%	4%	-29%	11%	3%	-4%
	IDW & OK %	4%	0%	-34%	0%	-5%	-1%

14.6.4 Local Bias Checks

A local bias check was performed by plotting the average zinc, gold, copper and silver of the NN, IDW and OK models in swaths plots oriented along the model easting.

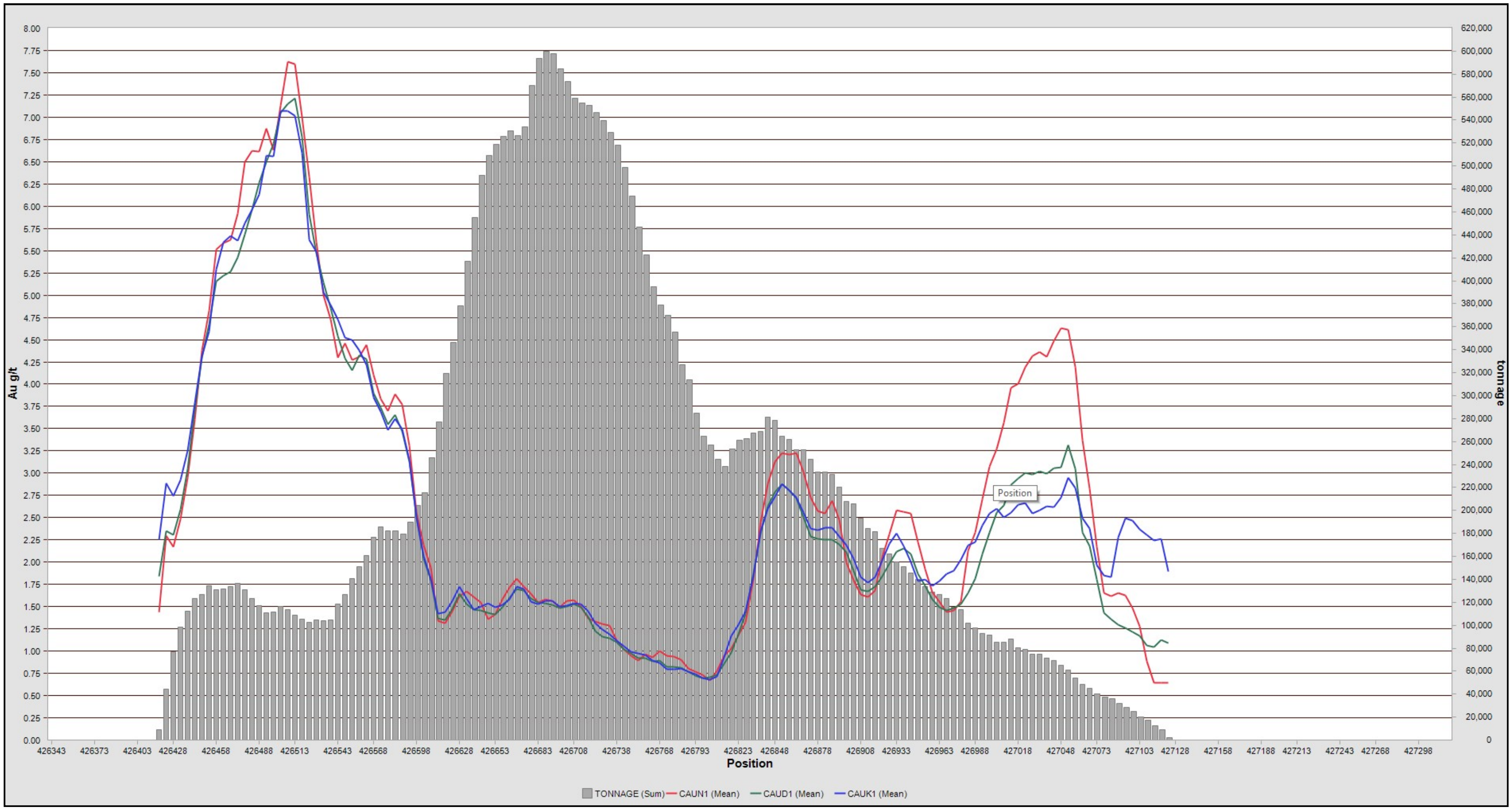
In reviewing the swath plots, only minor discrepancies were found between the different grade models. In areas where there is extrapolation beyond the drill holes, the swath plots indicate less agreement for all variables. The swath plots are shown below in Figure 14-14 to Figure 14-20.

FIGURE 14-14: ZINC SWATH PLOT IN BASE METAL LENSES (BY EASTING)



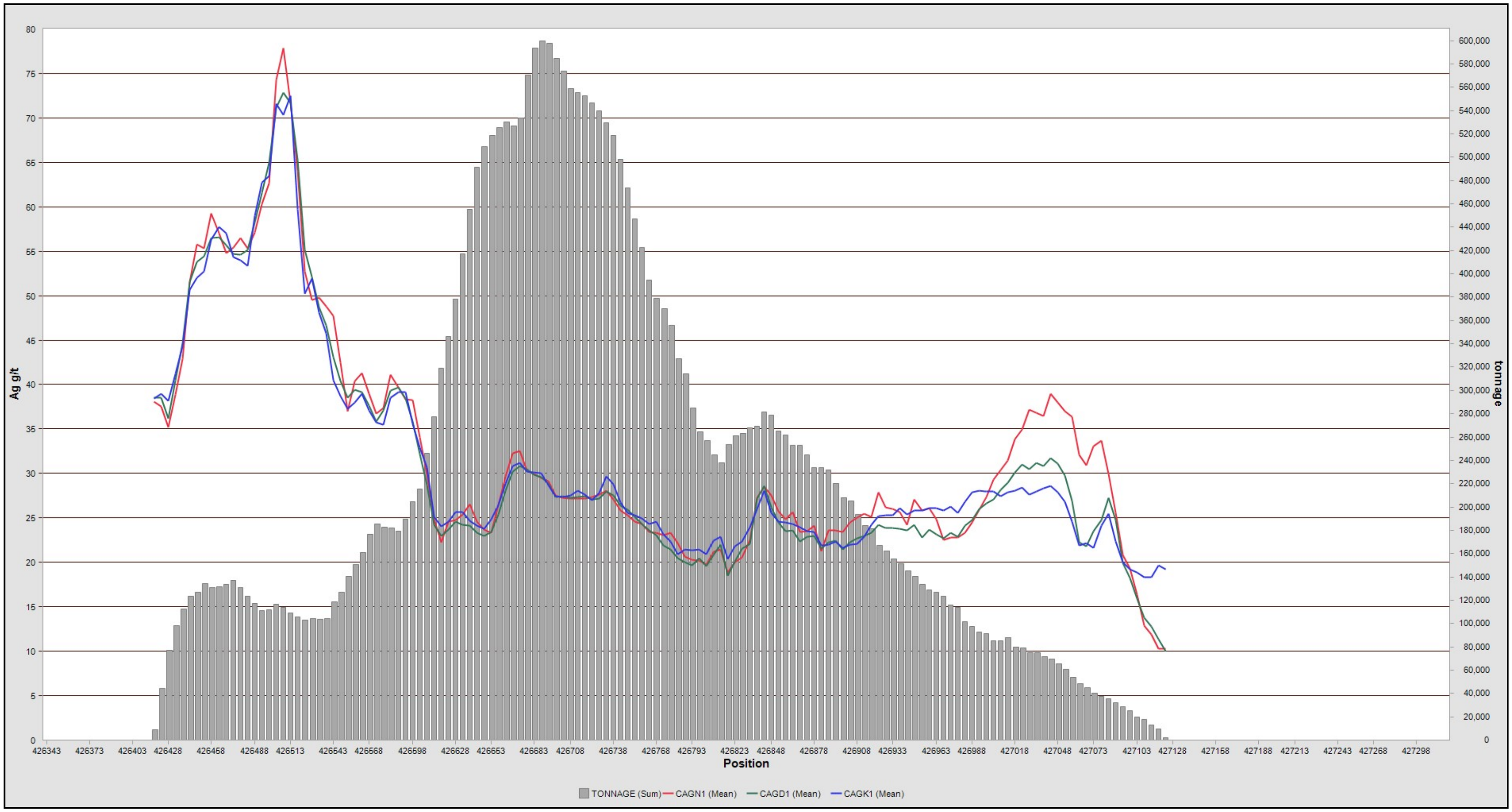
Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-15: GOLD SWATH PLOT IN BASE METAL LENSES (BY EASTING)



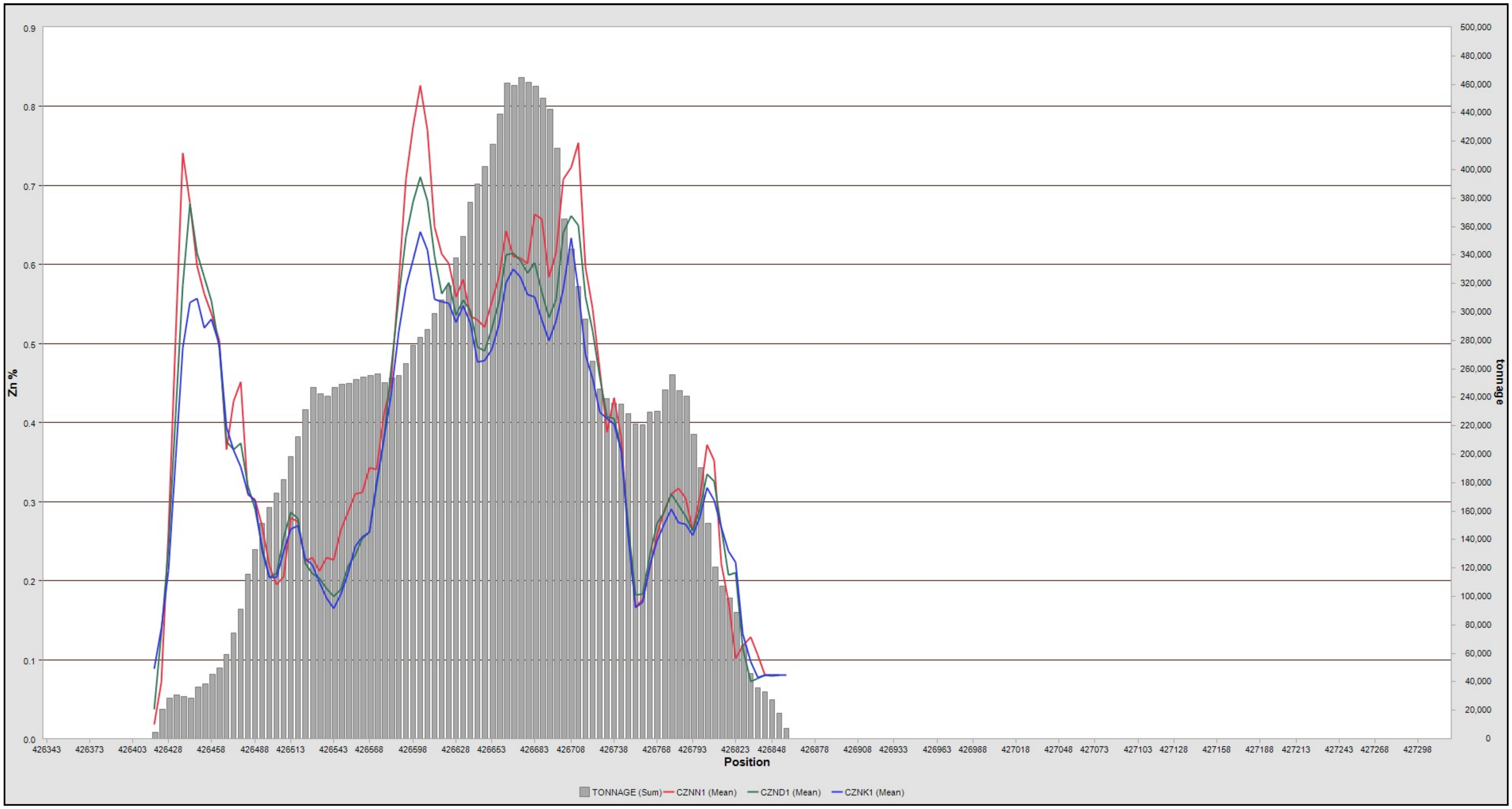
Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-16: SILVER SWATH PLOT IN BASE METAL LENSES (BY EASTING)



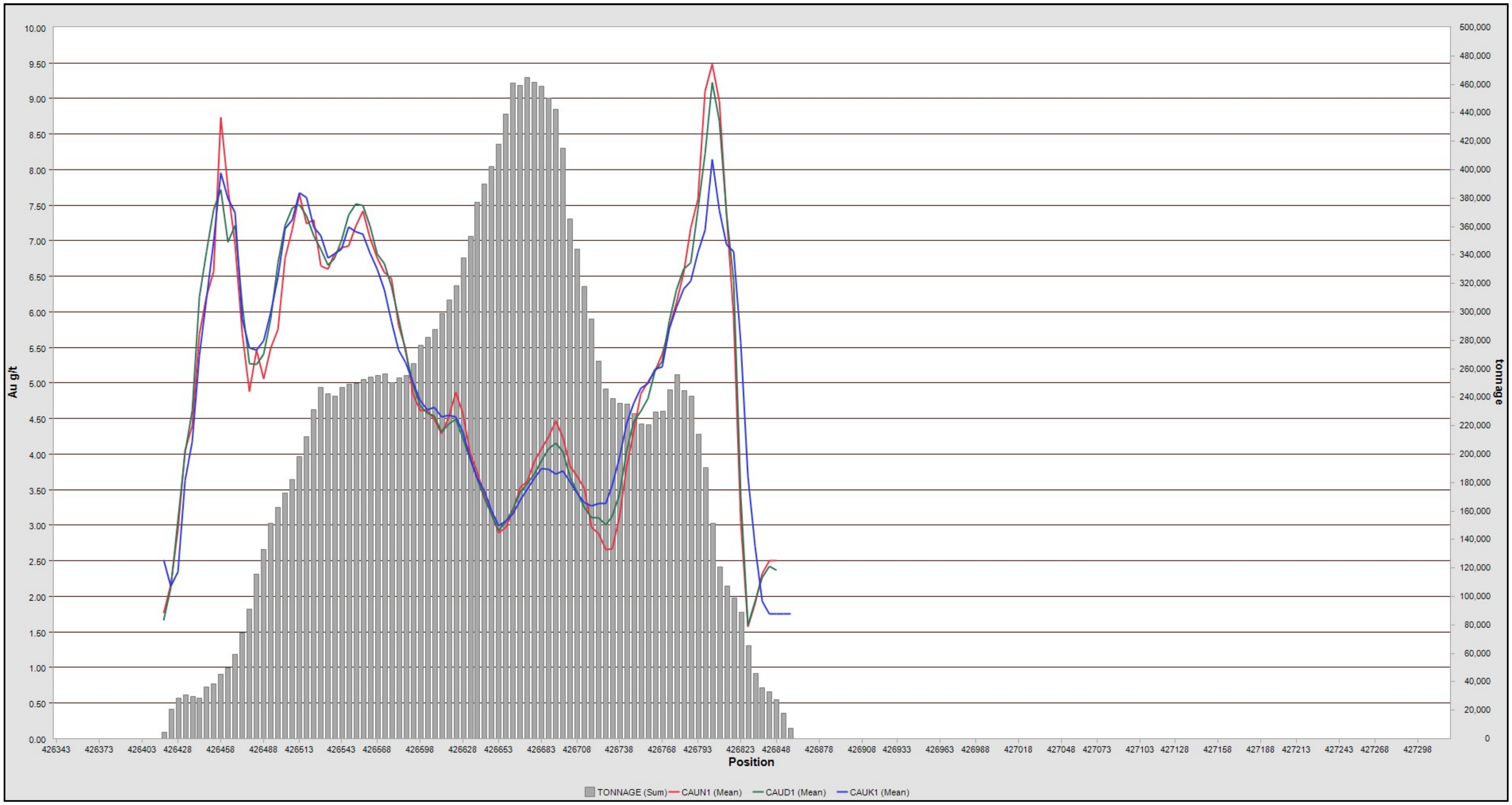
Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-17: ZINC SWATH PLOT IN GOLD ZONES (BY EASTING)



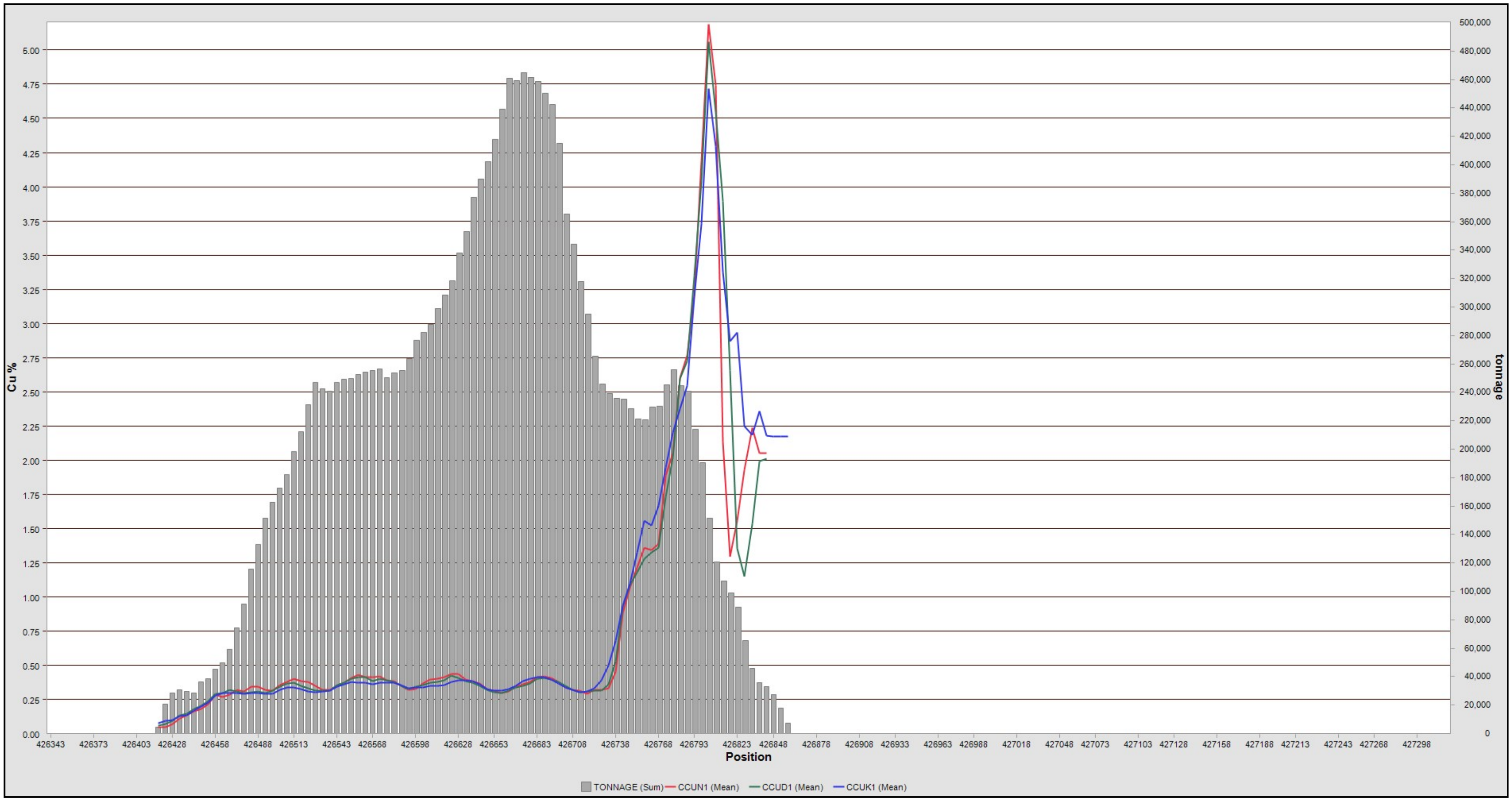
Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-18: GOLD SWATH PLOT IN GOLD ZONES (BY EASTING)



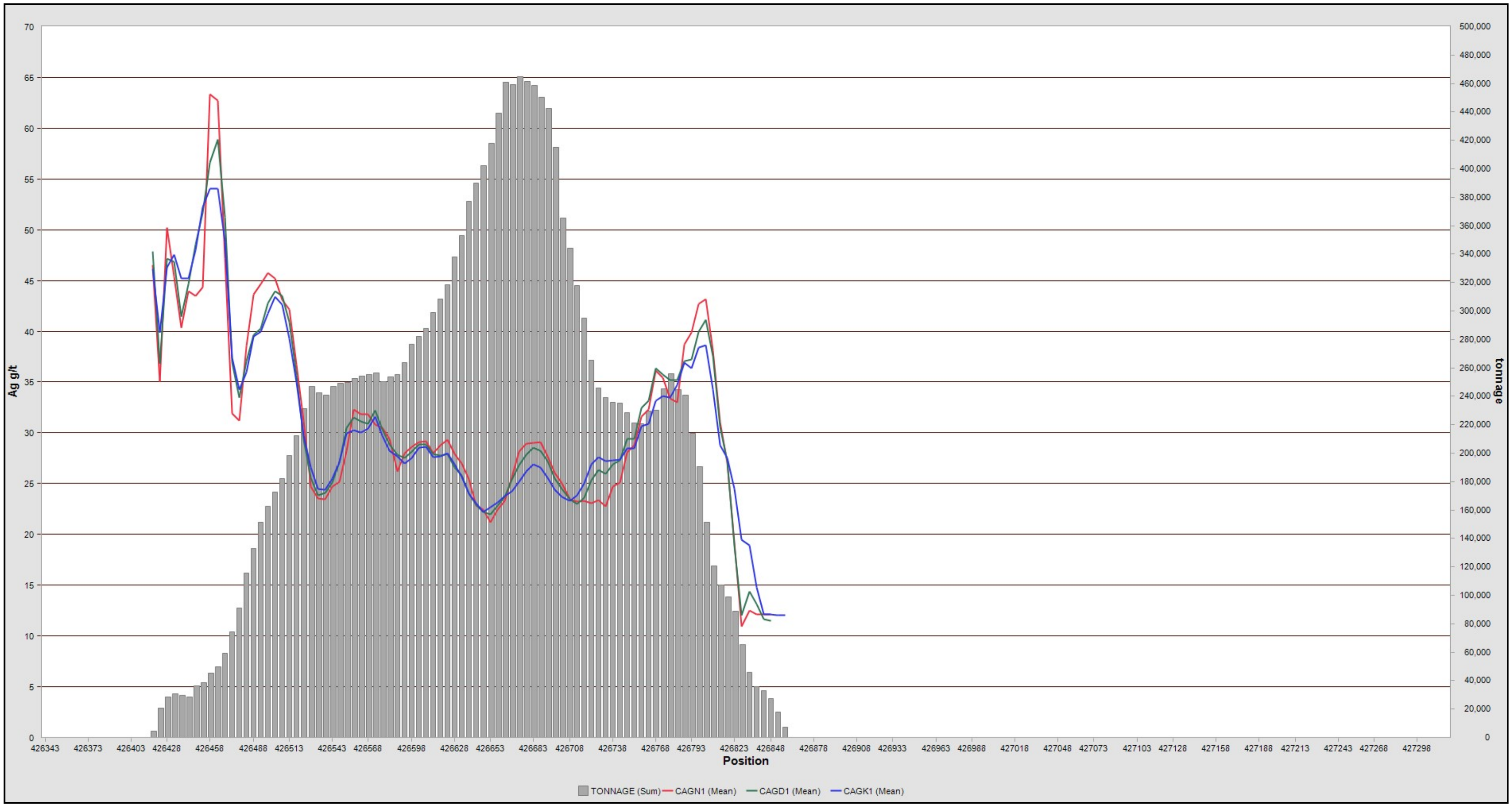
Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-19: COPPER SWATH PLOT IN GOLD ZONES (BY EASTING)



Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

FIGURE 14-20: SILVER SWATH PLOT IN GOLD ZONES (BY EASTING)



Note: Line charts show the grades and histogram shows the tonnes. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.

14.6.5 Block Model Quality Control

The closest distance of a composite (CDIST), the maximum distance of a composite (MDIST), the average distance of composites (ADIST), the number of composites (NCOMP) and the number of holes (NHOLE) used for the OK interpolation of zinc, gold, silver and copper were recorded in the block model.

The standard deviation of the kriging (KSTD) and the regression slope (RSLOP) were also recorded in the block model. Table 14-33, to Table 14-36 present the quality control parameters recorded in the block model from the OK interpolation.

TABLE 14-33: QUALITY CONTROL STATISTICS OF THE ZINC INTERPOLATION WITHIN THE BASE METAL LENSES

LENSE	CDIST	MDIST	ADIST	NCOMP	NHOLE	KSTD	RSLOP
10	12	27	20	15	5	0.50	0.86
11	31	70	53	12	5	0.58	0.80
20	5	13	9	15	5	0.56	0.89
30	13	39	26	14	5	0.52	0.90
31	21	59	40	14	5	0.69	0.72
32	2	6	4	16	6	0.58	0.87
40	24	60	42	12	4	0.77	0.57

TABLE 14-34: QUALITY CONTROL STATISTICS OF THE GOLD INTERPOLATION WITHIN THE BASE METAL LENSES

LENSE	CDIST	MDIST	ADIST	NCOMP	NHOLE	KSTD	RSLOP
10	5	11	8	14	5	0.66	0.67
11	25	56	43	12	5	0.55	0.54
20	5	12	8	15	5	0.59	0.76
30	13	32	22	12	4	0.56	0.56
31	15	42	28	15	5	0.49	0.76
32	16	43	29	16	6	0.55	0.58
40	25	59	42	12	4	0.54	0.51

Aside from lenses 31 and 40, the average standard deviation and the regression slope of the kriging are indicating a low variability of the zinc OK interpolation which is to be expected in a VMS deposit. As mentioned previously, it is believed that the overall drill spacing in lenses 31 and 40 is the source of the higher standard deviation and kriging regression slope.

TABLE 14-35: QUALITY CONTROL STATISTICS OF THE GOLD INTERPOLATION WITHIN THE GOLD ZONES

LENSE	CDIST	MDIST	ADIST	NCOMP	NHOLE	KSTD	RSLOP
21	11	24	17	12	4	0.56	0.44
23	15	31	24	10	4	0.53	0.78
24	14	34	25	14	5	0.48	0.62
25	22	46	34	11	4	0.79	0.33
26	18	37	28	10	3	0.43	0.69
27	26	48	38	9	3	0.82	0.22
28	30	48	38	8	2	0.81	0.62

TABLE 14-36: QUALITY CONTROL STATISTICS OF THE COPPER INTERPOLATION WITHIN THE GOLD ZONES

LENSE	CDIST	MDIST	ADIST	NCOMP	NHOLE	KSTD	RSLOP
21	10	26	18	14	5	0.60	0.65
23	12	32	22	15	6	0.49	0.89
24	15	39	28	13	5	0.53	0.73
25	11	28	19	14	5	0.66	0.47
26	17	34	26	10	3	0.65	0.81
27	29	57	44	9	3	0.80	0.58
28	29	51	40	8	3	0.79	0.65

The average standard deviation and the regression slope of the kriging are indicating a relative high variability of the gold and copper OK interpolation. As mentioned previously, the interpretation of the gold mineralized envelopes could be reviewed to improve the stationarity.

14.6.6 Grade-Tonnage Statistics

Table 14-37 to Table 14-40 present the grade-tonnage statistics of zinc, gold and copper for each interpolation method at different cut-offs. The grade-tonnage curve for zinc, gold and copper are shown in Figure 14-21 to Figure 14-24 as a way to present the overall assessment of the resources.

TABLE 14-37: ZINC GRADE-TONNAGE STATISTICS IN BASE METAL LENSES

	NN Model			IDW Model			OK Model			NN vs IDW Difference			NN vs OK Difference			IDW vs OK Difference		
Capped Zn Znt-Off	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV	Percent of Tons Above Znt-Off	Mean Zn Grade	STDEV
0	100.00	7.09	6.26	100.00	7.50	5.25	100.00	6.97	4.91	0%	6%	-16%	0%	-2%	-21%	0%	-7%	-6%
1	87.62	8.05	6.10	96.06	7.79	5.15	95.64	7.27	4.81	10%	-3%	-16%	9%	-10%	-21%	0%	-7%	-6%
2	80.43	8.64	6.03	91.32	8.11	5.07	90.73	7.58	4.75	14%	-6%	-16%	13%	-12%	-21%	-1%	-7%	-6%
3	72.90	9.27	5.98	84.20	8.59	5.00	82.77	8.06	4.69	16%	-7%	-16%	14%	-13%	-22%	-2%	-6%	-6%
4	65.16	9.96	5.96	74.97	9.21	4.95	72.08	8.74	4.66	15%	-8%	-17%	11%	-12%	-22%	-4%	-5%	-6%
5	54.60	11.02	5.96	63.75	10.04	4.93	60.13	9.58	4.67	17%	-9%	-17%	10%	-13%	-22%	-6%	-5%	-5%
6	46.15	12.02	5.95	52.09	11.06	4.90	48.23	10.59	4.69	13%	-8%	-18%	5%	-12%	-21%	-7%	-4%	-4%
7	39.34	12.98	5.94	42.76	12.06	4.86	38.38	11.65	4.70	9%	-7%	-18%	-2%	-10%	-21%	-10%	-3%	-3%
8	32.92	14.06	5.92	35.44	13.00	4.83	30.96	12.65	4.71	8%	-8%	-19%	-6%	-10%	-20%	-13%	-3%	-2%
9	28.64	14.90	5.91	29.19	13.97	4.79	24.65	13.71	4.72	2%	-6%	-19%	-14%	-8%	-20%	-16%	-2%	-1%
10	24.32	15.85	5.93	24.12	14.92	4.76	19.87	14.74	4.72	-1%	-6%	-20%	-18%	-7%	-20%	-18%	-1%	-1%

TABLE 14-38: GOLD GRADE-TONNAGE STATISTICS IN BASE METAL LENSES

	NN Model			IDW Model			OK Model			NN vs IDW Difference			NN vs OK Difference			IDW vs OK Difference		
Capped Au Aut-Off	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV
0	100.00	2.15	3.05	100.00	2.01	2.33	100.00	2.05	2.20	0%	-7%	-23%	0%	-5%	-28%	0%	2%	-6%
1	50.48	3.84	3.54	56.45	3.18	2.54	59.90	3.09	2.31	12%	-17%	-28%	19%	-19%	-35%	6%	-3%	-9%
2	30.28	5.44	3.80	32.92	4.43	2.69	35.66	4.22	2.41	9%	-19%	-29%	18%	-22%	-37%	8%	-5%	-11%
3	21.15	6.72	3.91	21.02	5.55	2.80	22.01	5.33	2.47	-1%	-17%	-28%	4%	-21%	-37%	5%	-4%	-12%
4	15.62	7.86	3.96	13.97	6.60	2.91	14.32	6.32	2.56	-11%	-16%	-26%	-8%	-20%	-35%	3%	-4%	-12%
5	12.00	8.88	3.98	9.23	7.70	3.04	9.00	7.43	2.66	-23%	-13%	-24%	-25%	-16%	-33%	-2%	-4%	-13%
6	8.99	10.06	3.94	6.17	8.82	3.17	5.79	8.52	2.76	-31%	-12%	-20%	-36%	-15%	-30%	-6%	-3%	-13%
7	7.41	10.83	3.93	4.20	9.92	3.30	3.70	9.68	2.84	-43%	-8%	-16%	-50%	-11%	-28%	-12%	-2%	-14%
8	5.85	11.72	3.97	2.78	11.18	3.42	2.36	10.94	2.86	-52%	-5%	-14%	-60%	-7%	-28%	-15%	-2%	-16%
9	4.49	12.69	4.06	1.93	12.39	3.47	1.64	12.03	2.81	-57%	-2%	-15%	-63%	-5%	-31%	-15%	-3%	-19%
10	2.86	14.46	4.16	1.41	13.47	3.49	1.19	13.00	2.73	-51%	-7%	-16%	-58%	-10%	-34%	-16%	-3%	-22%

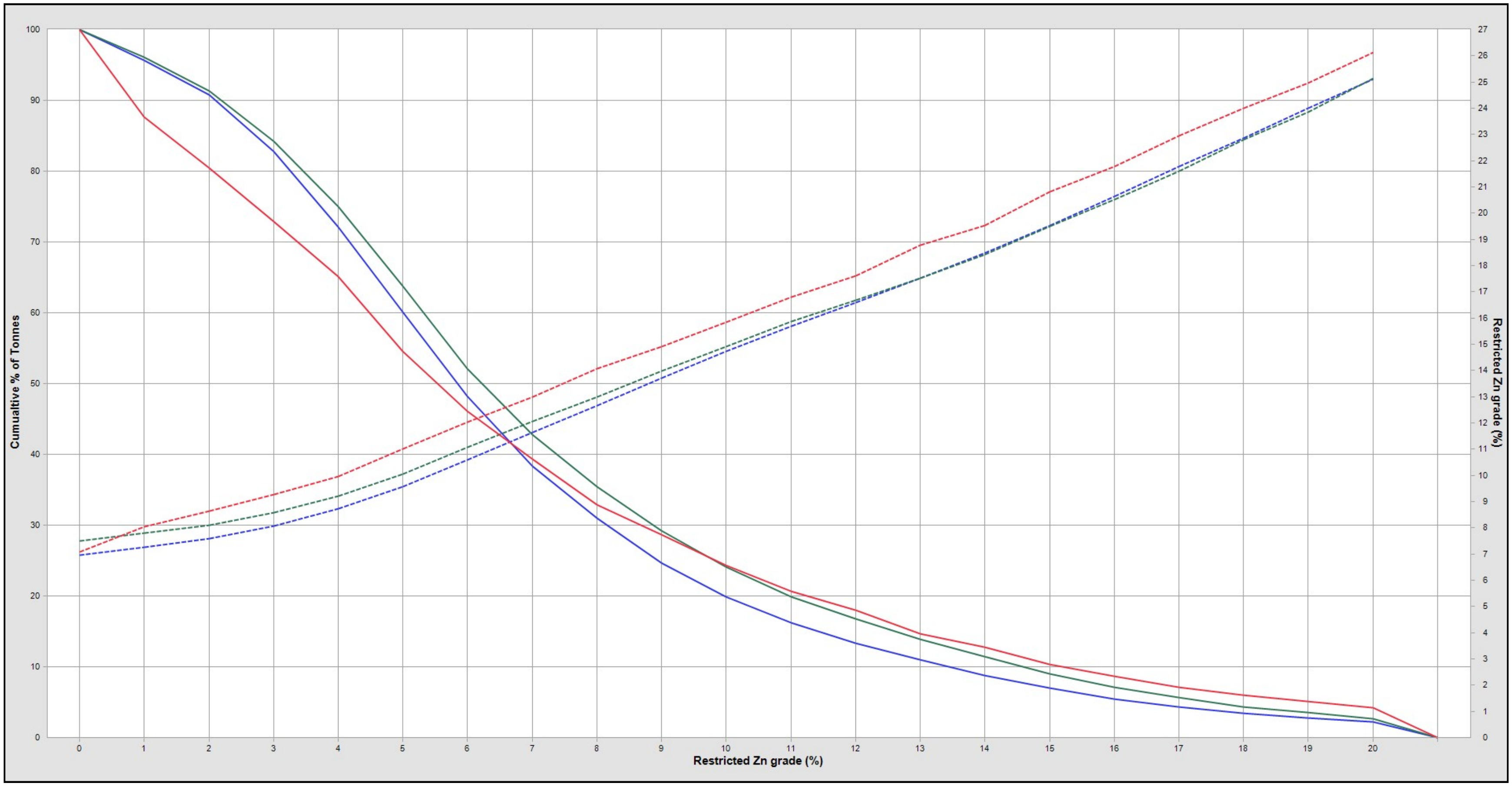
TABLE 14-39: GOLD GRADE-TONNAGE STATISTICS IN GOLD ZONES

	NN Model			IDW Model			OK Model			NN vs IDW Difference			NN vs OK Difference			IDW vs OK Difference		
Capped Au Aut-Off	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV	Percent of Tons Above Aut-Off	Mean Au Grade	STDEV
0	100.00	4.94	5.53	100.00	4.94	4.31	100.00	4.87	3.76	0%	0%	-22%	0%	-2%	-32%	0%	-1%	-13%
1	81.66	5.96	5.64	90.81	5.38	4.28	92.78	5.20	3.70	11%	-10%	-24%	14%	-13%	-34%	2%	-3%	-14%
2	64.87	7.10	5.80	75.58	6.16	4.29	78.22	5.88	3.65	17%	-13%	-26%	21%	-17%	-37%	3%	-5%	-15%
3	50.97	8.36	5.95	58.72	7.21	4.33	61.26	6.81	3.60	15%	-14%	-27%	20%	-19%	-40%	4%	-5%	-17%
4	40.62	9.60	6.07	45.48	8.30	4.35	47.36	7.79	3.53	12%	-14%	-28%	17%	-19%	-42%	4%	-6%	-19%
5	32.85	10.81	6.16	35.47	9.37	4.35	36.73	8.75	3.46	8%	-13%	-29%	12%	-19%	-44%	4%	-7%	-21%
6	26.84	11.99	6.23	28.01	10.41	4.35	28.92	9.63	3.39	4%	-13%	-30%	8%	-20%	-46%	3%	-7%	-22%
7	21.16	13.46	6.24	21.97	11.49	4.32	22.30	10.57	3.33	4%	-15%	-31%	5%	-22%	-47%	2%	-8%	-23%
8	18.52	14.32	6.21	17.72	12.45	4.28	17.23	11.47	3.27	-4%	-13%	-31%	-7%	-20%	-47%	-3%	-8%	-23%
9	15.40	15.49	6.18	14.31	13.40	4.24	13.33	12.36	3.22	-7%	-14%	-31%	-13%	-20%	-48%	-7%	-8%	-24%
10	13.27	16.45	6.14	11.62	14.31	4.21	10.16	13.25	3.20	-12%	-13%	-31%	-23%	-19%	-48%	-13%	-7%	-24%

TABLE 14-40: COPPER GRADE-TONNAGE STATISTICS IN GOLD ZONES

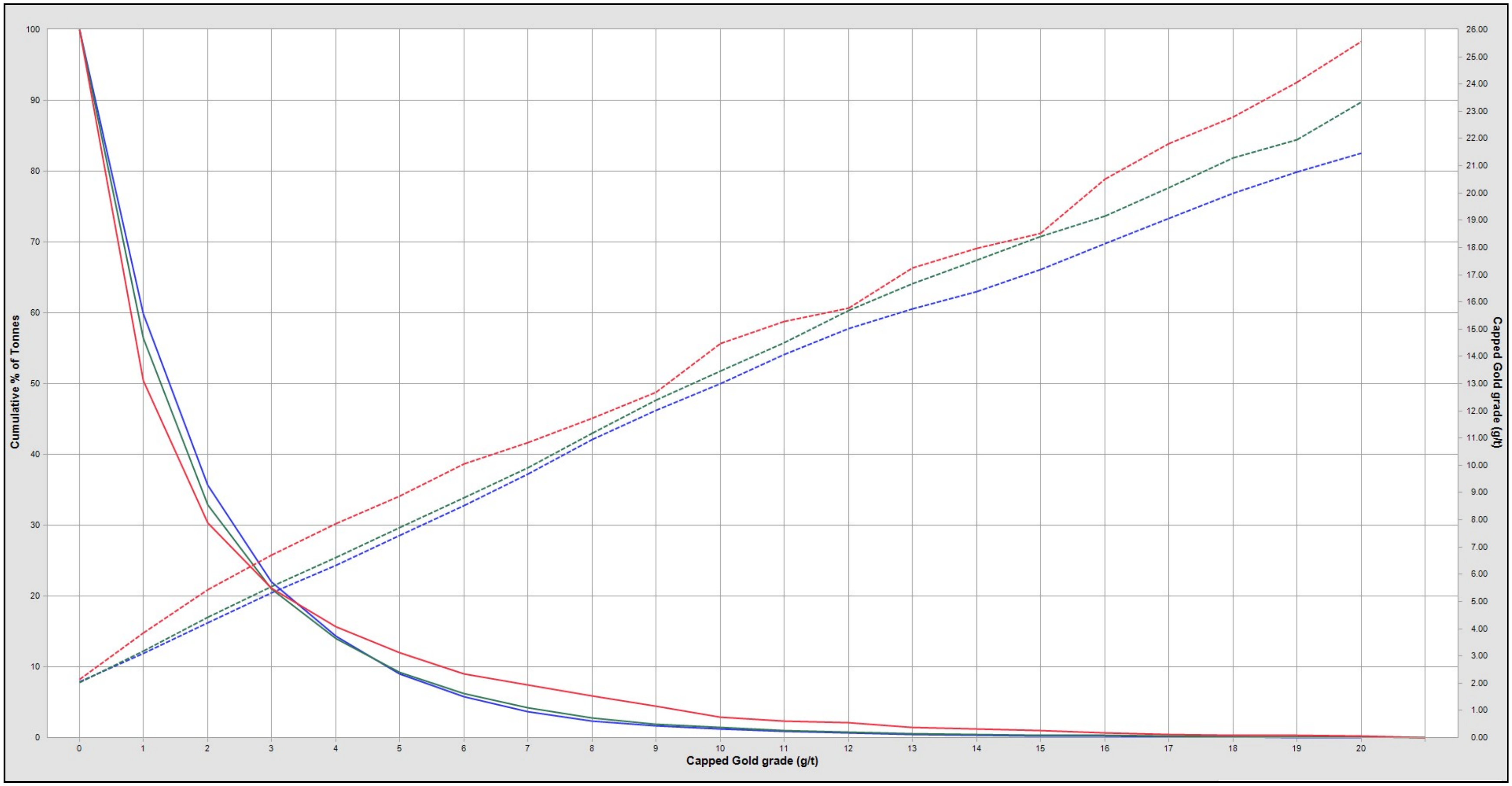
	NN Model			IDW Model			OK Model			NN vs IDW Difference			NN vs OK Difference			IDW vs OK Difference		
Capped Cu Cut-Off	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV	Percent of Tons Above Cut-Off	Mean Cu Grade	STDEV
0	100.00	0.66	1.46	100.00	0.66	1.38	100.00	0.69	1.26	0%	0%	-6%	0%	5%	-14%	0%	5%	-8%
1	12.52	3.42	2.83	12.31	3.31	2.65	13.57	3.20	2.05	-2%	-3%	-6%	8%	-6%	-28%	10%	-3%	-23%
2	7.04	5.01	2.90	6.39	5.09	2.62	8.93	4.16	1.91	-9%	2%	-9%	27%	-17%	-34%	40%	-18%	-27%
3	4.88	6.16	2.79	5.06	5.79	2.51	5.96	5.03	1.77	4%	-6%	-10%	22%	-18%	-37%	18%	-13%	-30%
4	3.93	6.80	2.75	3.88	6.49	2.48	4.01	5.77	1.71	-1%	-5%	-10%	2%	-15%	-38%	3%	-11%	-31%
5	2.67	7.87	2.74	2.41	7.74	2.39	2.49	6.54	1.77	-10%	-2%	-13%	-7%	-17%	-35%	3%	-15%	-26%
6	1.80	8.99	2.69	1.74	8.61	2.27	1.16	7.80	1.91	-3%	-4%	-15%	-36%	-13%	-29%	-33%	-9%	-16%
7	1.65	9.22	2.70	1.28	9.38	2.17	0.58	9.21	1.81	-22%	2%	-20%	-65%	0%	-33%	-55%	-2%	-17%
8	1.11	10.20	2.80	0.86	10.34	2.07	0.38	10.11	1.58	-23%	1%	-26%	-66%	-1%	-44%	-56%	-2%	-24%
9	0.41	13.23	2.58	0.54	11.45	1.87	0.28	10.75	1.38	32%	-14%	-28%	-32%	-19%	-46%	-48%	-6%	-26%
10	0.34	13.98	2.07	0.41	12.06	1.71	0.18	11.47	1.20	21%	-14%	-17%	-47%	-18%	-42%	-56%	-5%	-30%

FIGURE 14-21: GRADE TONNAGE OF RESTRICTED ZINC IN THE BASE METAL LENSES



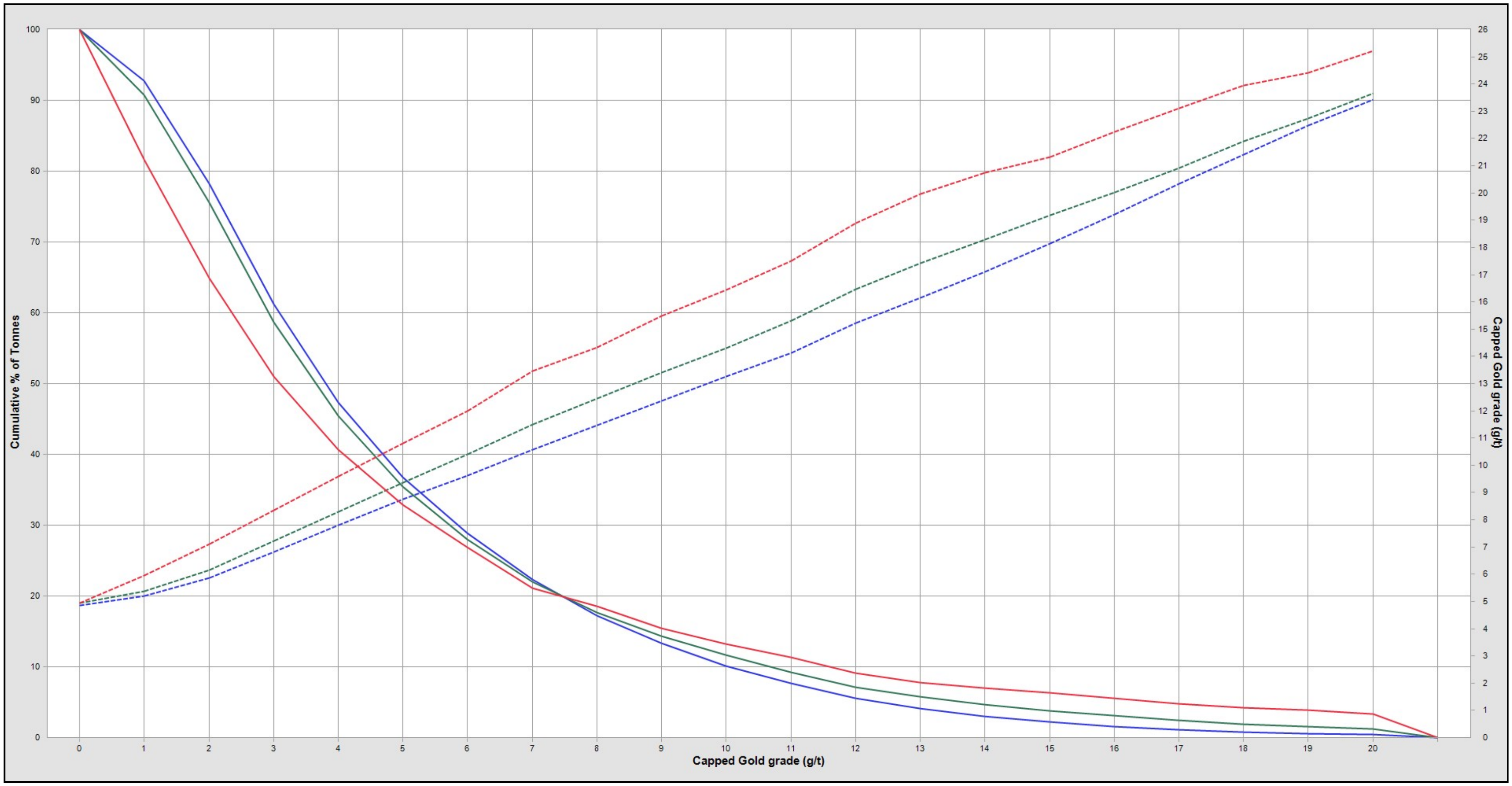
Note: Solid lines represent tonnes, dashed lines represent grades, green represents IDW model, red represents NN model and blue represents OK model

FIGURE 14-22: GRADE TONNAGE OF CAPPED GOLD IN THE BASE METAL LENSES



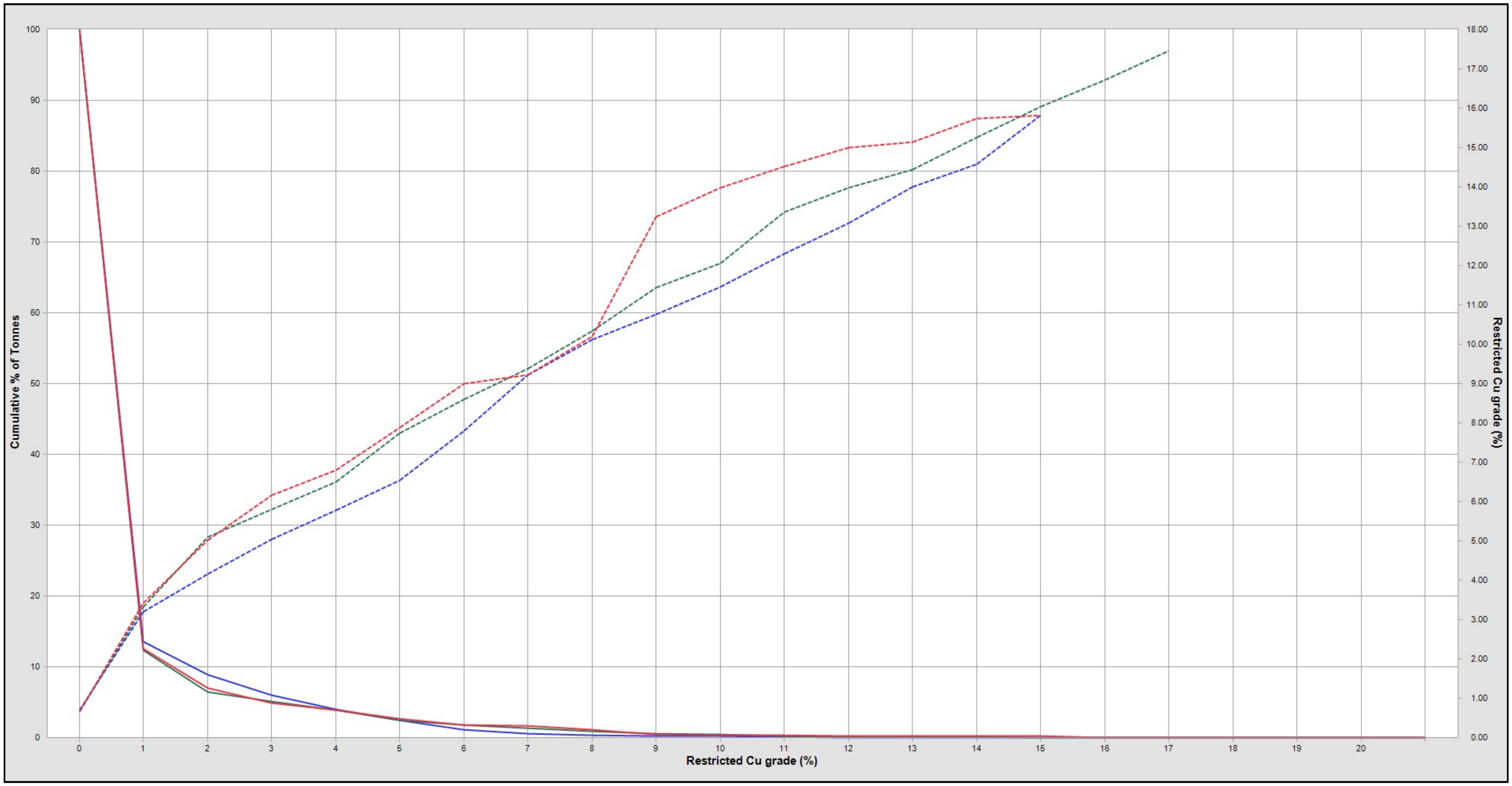
Note: Solid lines represent tonnes, dashed lines represent grades, green represents IDW model, red represents NN model and blue represents OK model

FIGURE 14-23: GRADE TONNAGE OF CAPPED GOLD IN THE GOLD ZONES



Note: Solid lines represent tonnes, dashed lines represent grades, green represents IDW model, red represents NN model and blue represents OK model

FIGURE 14-24: GRADE TONNAGE OF RESTRICTED COPPER IN THE GOLD ZONES



Note: Solid lines represent tonnes, dashed lines represent grades, green represents IDW model, red represents NN model and blue represents OK model

14.7 Classification of Mineral Resource in Base Metal Lenses

Mineral resources have been classified according to the 2014 CIM Definition Standards on Mineral Resources and Mineral Reserves (CIM definitions), as incorporated in NI 43-101. Resource blocks are classified as Measured, Indicated or Inferred, depending upon the confidence level of the resource based on experience at Lalor mine, with similar deposits and the spatial continuity of the mineralization.

The resource category classification relies on the following method:

Measured

- Distance to an underground development is generally less than 10 m

Indicated

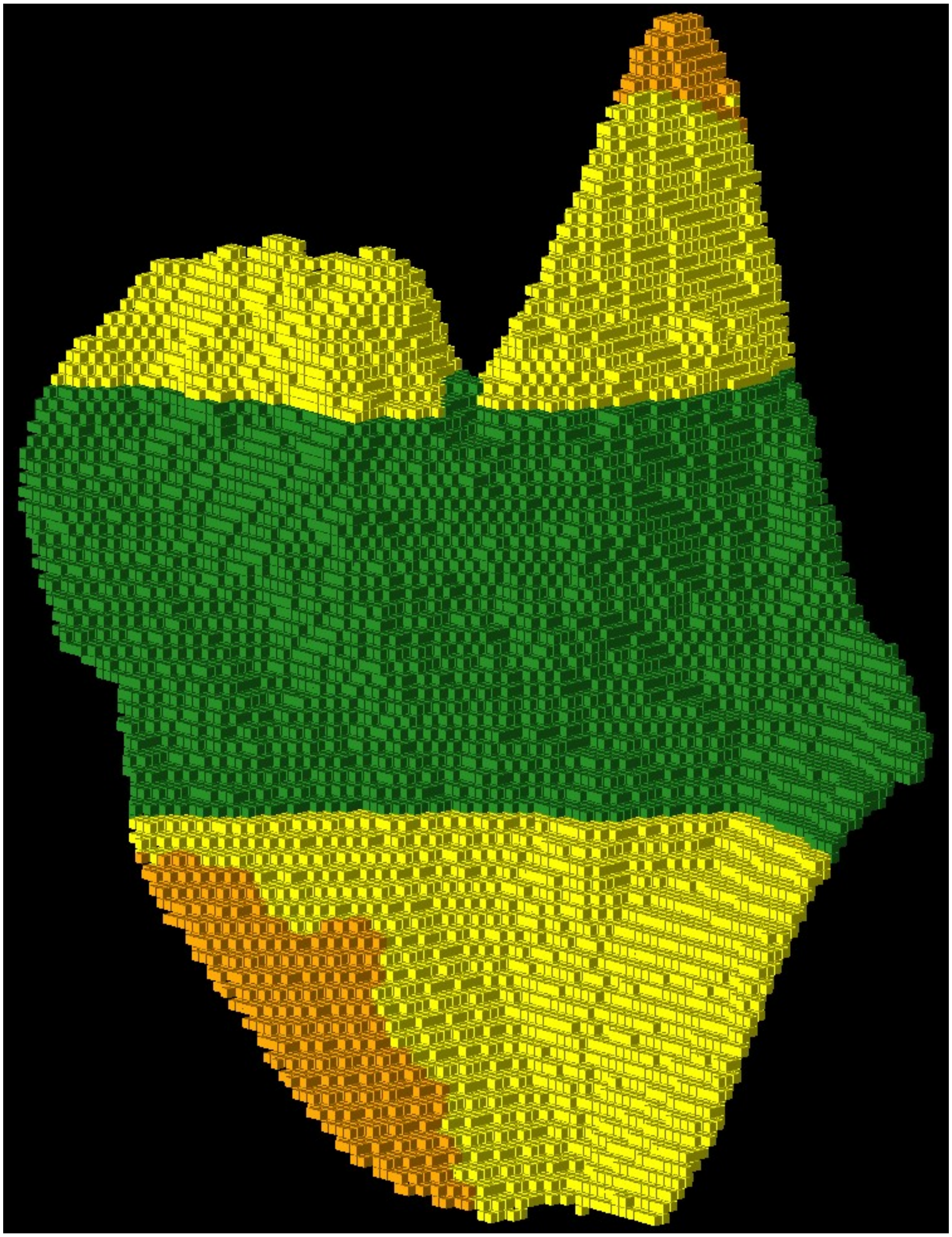
- Distance to closest composite is less than or equal to 50 m
- Blocks interpolated from the last two interpolation pass (i.e. the most restrictive)
- Blocks estimated from at least two drill holes

Inferred

- The remainder of the interpolated blocks within the interpreted lenses are classified as Inferred Resources

A smoothing algorithm was applied to remove isolated blocks of measured within areas of mostly indicated category or isolated indicated blocks within areas of mostly measured category blocks and solids were created for all the different base metal lenses resource categories. Proportions of measured and indicated category blocks were not changed significantly by this process. Figure 14 25 presents a 3D view of the resource categories of lense 10.

FIGURE 14-25: 3D VIEW (LOOKING SW) DISPLAYING THE RESOURCE CLASSIFICATION IN LENSE 10



Note: One block equals 5m by 5m by 5m. Measured = green blocks, Indicated = yellow blocks and Inferred = orange blocks.

14.8 Classification of Mineral Resource in Gold Zones

The resource category classification relies on the relative difference between the kriged grade and the composites grades. The Resource Classification Index (RCI) uses the following formula[1]:

$$RCI = \sqrt{\left(\frac{Ordinary\ Kriging\ combined\ variance}{block\ grade}\right)} * C$$

Where C is a calibration factor based the distance of the composites, the number of composites, number of quadrants and number of drill holes using the following formula:

$$C = \exp\frac{closest\ distance}{maximum\ distance} \Big/ \left(\exp\frac{composites\ used}{maximum\ possibility} * \exp\frac{quadrant\ used}{4} * \exp\frac{drill\ hole\ used}{maxim\quad possibility}\right)$$

The RCI value corresponding to the 50th (0.084) percentiles of the distribution of blocks with gold grade above 1 g/t contained within the interpreted gold zones was determined and used as a threshold for the indicated resource category in all the gold zones except for zone 27 (i.e. a copper-gold zone), where the RCI values corresponding to the 70th (0.131) percentiles was used as a threshold for the indicated resource category. Table 14-41 presents the RCI deciles values along with their corresponding average distance, closest distance and maximum distance of the composites to the blocks centre.

TABLE 14-41: RCI VALUES WITH AVERAGE ADIST, CDIST AND MDIST

		Averages		
Percentiles	Au RCI values	ADIST	CDIST	MDIST
10th	0.027	19	8	28
20th	0.040	20	10	30
30th	0.053	22	11	32
40th	0.067	23	12	34
50th	0.084	25	13	35
60th	0.104	26	14	36
70th	0.131	27	15	37
80th	0.175	28	16	38
90th	0.284	29	17	39

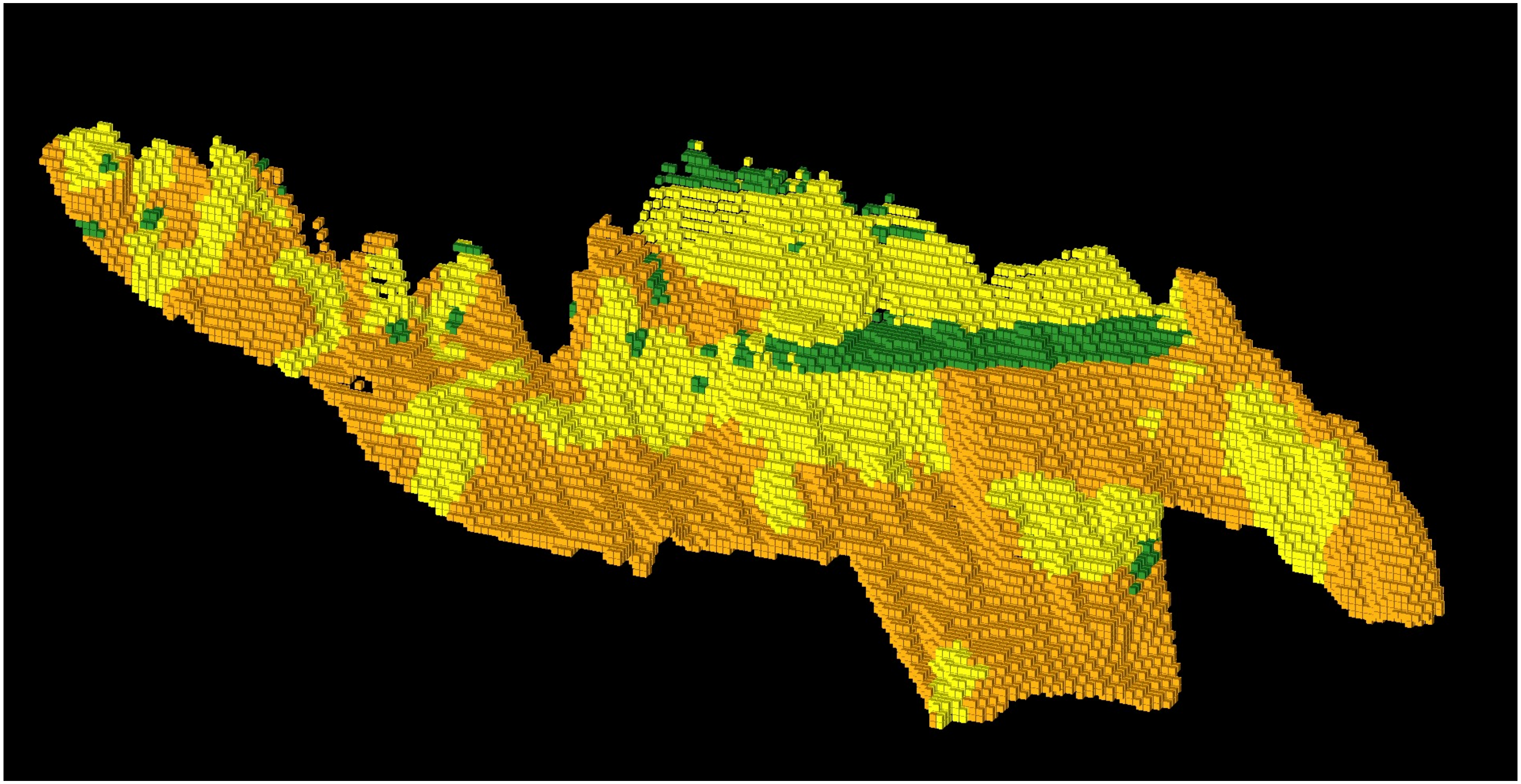
Under this classification system, in order for a block to be considered as measured, the blocks must be within approximately 10 m of an underground development. To be considered as indicated, a block must have a RCI value lower than 0.084 (or lower 0.131 in zone 27) and a CDIST of less than or equal to 25 m. All remaining blocks were classified as inferred resources with minimum criteria of one drill hole to interpolate the grades within one of the three interpolation passes.

A smoothing algorithm was applied to remove isolated blocks of measured within areas of mostly indicated category or isolated indicated blocks within areas of mostly measured category blocks and

[1] *Arik, A. 2002," Resource Classification Index", MineSight in the Foreground.*

solids were created for all the different gold zones resource categories. Proportions of measured and indicated category blocks were not changed significantly by this process. Figure 14 26 presents a 3D view of the resource categories of zone 25.

FIGURE 14-26: 3D VIEW (LOOKING SW) DISPLAYING THE RESOURCE CLASSIFICATION IN ZONE 25



Note: One block equals 5m by 5m by 5m. Measured = green blocks, Indicated = yellow blocks and Inferred = orange blocks.

14.9 Third Party Review

Hudbay requested that T. Maunula & Associates Consulting Inc., an independent consultant, perform a validation of gold zones and selected base metals lenses. The following minor issues were highlighted by the third party validation:

- Entry error noted and corrected in OK interpolation parameter.
- It was noted that some variograms ranges (example in lense 10 and 40) could be increased. The nugget effect was quite low for some of the variograms which seems inconsistent with the skewed data population and high CV, the next round of modelling should inspect this further.
- Swath plots demonstrated good correlation between composites and block models except in areas of lower data density or smaller resource volumes
- Based on the visual inspection, the block model grades appeared to honour the data well. The interpolated block model grades exhibit satisfactory consistency with the drill hole composites. However, it was noted that high coefficients of variation (CV) are present. This may cause a bias using linear estimation techniques. Compositing and capping reduced the CV to an acceptable level for most lenses.
- 0.24% of blocks were estimated with one drill hole and classified as Measured or Indicated Resource

The third party recommends the following:

- Review interpolation methodology to reduce number of passes. This could be evaluated as part of the change of support analysis. The interpolation parameters for minimum and maximum number of composites, use of octants, etc. and their impact could be evaluated against the Herco adjusted NN model.
- Assess the calculation of the density from the interpolated grades rather than interpolating density.
- Use reconciliation information from mined areas to validate the capping levels and density calculations.

Based on the review, the third party has concluded that the Lalor block model has been interpolated using industry accepted modeling techniques using MineSight desktop software. This included geologic input, appropriate block model cell sizes, grade capping, assay compositing, and reasonable interpolation parameters. The results have been verified by visual review and statistical comparisons between the estimated block grades and the composites used to interpolate. The OK model has been selected as the best representation of the grade distribution based on the current geological understanding and zone interpretation. The OK model has been validated with alternate estimation methods: NN and IDW. No biases have been identified in the model. Mineral resources were classified in accordance with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 "Standards of Disclosure for Mineral Projects."

14.10 Reasonable Prospects of Economic Extraction

The component of the mineralization within the block model that meets the requirements for reasonable prospects of economic extraction is based on the 2017 Lalor mine and Stall concentrator budgets. The following table presents the metal equivalency formulas used in the 2017 block model:

TABLE 14-42: LALOR METAL EQUIVALENCIES

Mineralization Type	Metal Equivalence	Formula
Base Metal Lenses	Zn Eq	Zn Eq = Zn% + (1.98 x Cu%) + (1.11 x Au g/t) + (0.01 x Ag g/t) – (0.01 x Pb%)
Gold Zones	Au Eq	Au Eq = Au g/t + (1.34 x Cu%) + (0.01 x Ag g/t)

Table 14-43 to Table 14-49 present the economic parameters and recoveries used to determine the metal equivalency formulas specified above. Metal equivalence considers the ratio of recovery, payability and value of metals after application of downstream processing costs.

TABLE 14-43: METAL PRICES – 2017 MINERAL RESOURCE ($US)

	Unit	Value
Gold	$US/oz	1,300
Silver	$US/oz	18.00
Copper	$US/lb	2.67
Zinc[1]	$US/lb	1.19
FX Rate	CAD / USD	1.25

[1]Net zinc price includes premium and distribution costs based on processing and refining at Hudbay's Flin Flon Zinc Plant

The cost and price inputs are considered approximation and were used to test the economic viability of the resource. The cost and price inputs may differ from the mineral reserve.

TABLE 14-44: METAL RECOVERIES

	Base Metal Concentrator	Gold Leach Concentrator
Gold to Copper Concentrate	58.0%	
Silver to Copper Concentrate	55.0%	
Copper to Copper Concentrate	85.0%	
Copper Concentrate Grade	21.0%	
Gold to Dore		92.0%
Silver to Dore		53.1%
Zinc to Zinc Concentrate	92.5%	

Zinc Concentrate Grade	51.0%	

TABLE 14-45: MANITOBA BUSINESS UNIT – GENERAL MANAGEMENT, ADMINISTRATION AND UNALLOCATED SERVICES COSTS

Administration Overhead	Unit	Value
Base Metal	$C/tonne mined	16.00
Gold Zone	per dollar of revenue	0.08

TABLE 14-46: COPPER AND ZINC CONCENTRATE TERMS

	Unit
Payables	
Gold in Copper Concentrate	96% of content
Silver in Copper Concentrate	90% of content
Copper in Copper Concentrate	Minimum deduction 1 unit
Zinc in Zinc Concentrate	97.50%
Treatment	
Copper Concentrate	$US 90.00/dmt
Zinc Concentrate (includes refining)	$US 320.00/dmt
Refining	
Gold	$US 5.00/oz
Silver	$US 0.50/oz
Copper	$US 0.09/lb
Freight	
Copper Concentrate (third party smelter)	$US 160.00/dmt
Zinc Concentrate	Freigth to Flin Flon included with milling

TABLE 14-47: GOLD DORÉ TERMS

	Unit
Payable	
Gold in Dore	98.5% of content
Silver in Dore	99% of content
Treatment	
Dore	$US 0.40/oz
Refining	
Gold	$US 5.00/oz
Silver	$US 0.50/oz
Freight	
Dore	$US 0.50/oz

TABLE 14-48: MINING

	Unit	Value
Ore Removal	$C/tonne mined	20.00
General & Administration	$C/tonne mined	33.00

TABLE 14-49: MILLING

	Unit	Value
Base Metal (includes concentrate freight to Flin Flon)	$C/tonne milled	23.00
Gold Concentrator	$C/tonne milled	40.00

14.11 Mineral Resource Statement

Mineral resources for the Lalor mine were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves[2].

The mineral resources, classified as Measured, Indicated and Inferred, inclusive of mineral reserves are summarized in Table 14-50 and Table 14-51 as of September 30, 2016. The Qualified Person for the mineral resource estimate is Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

[2] *Ontario Securities Commission web site (http://www.osc.gov.on.ca/en/15019.htm)*

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Due to the uncertainty that may be associated with Inferred mineral resources it cannot be assumed that all or any part of Inferred resources will be upgraded to an Indicated or Measured resource.

TABLE 14-50: BASE METAL MINERAL RESOURCE, INCLUSIVE OF MINERAL RESERVES BY CATEGORY AND MINERALIZED ZONE WITH A CUT-OFF OF 4.1% ZN EQ, AS OF SEPTEMBER 30, 2016 (1)(2)(3)(4)(5)(6)(7)(8)(9)

Category	Lense	Tonnes	Zinc	Gold	Copper	Silver
Measured	10	1,792,000	10.67	1.11	0.62	19.27
	11	107,000	9.98	0.20	0.29	20.26
	20	1,868,000	6.61	2.36	0.83	32.08
	30	25,000	4.86	1.78	0.22	32.95
	31	227,000	4.56	1.32	0.20	25.27
	32	1,107,000	8.17	5.26	1.50	51.90
Subtotal		**5,126,000**	**8.34**	**2.46**	**0.86**	**31.34**
Indicated	10	2,225,000	6.34	2.03	0.59	26.91
	11	112,000	11.87	0.31	0.23	24.10
	20	2,083,000	8.00	1.88	0.84	25.03
	30	968,000	5.24	1.46	0.24	32.25
	31	646,000	4.50	1.61	0.22	31.40
	32	687,000	9.61	5.44	1.42	54.69
	40	2,121,000	5.79	1.41	0.38	30.80
Subtotal		**8,842,000**	**6.67**	**2.00**	**0.59**	**30.44**
Measured + Indicated	10	4,016,000	8.27	1.62	0.60	23.50
	11	219,000	10.95	0.26	0.26	22.23
	20	3,951,000	7.34	2.11	0.84	28.36
	30	993,000	5.23	1.47	0.24	32.27
	31	873,000	4.51	1.54	0.21	29.81
	32	1,794,000	8.72	5.33	1.47	52.97
	40	2,121,000	5.79	1.41	0.38	30.80
Subtotal		**13,967,000**	**7.28**	**2.17**	**0.69**	**30.77**

Category	Lense	Tonnes	Zinc	Gold	Copper	Silver
Inferred	10	178,000	8.69	1.84	0.36	18.97
	11	131,000	9.22	0.92	0.17	36.69
	20	11,000	5.86	2.38	0.92	27.25
	30	11,000	4.44	1.71	0.28	23.51
	31	27,000	6.07	0.65	0.18	16.14
	32	300	1.88	8.01	2.05	97.33
	40	187,000	7.54	1.49	0.38	15.75
Subtotal		**545,300**	**8.15**	**1.45**	**0.32**	**22.28**

Notes:

1. Domains were modelled in 3D to separate mineralized zones from surrounding waste rock. The domains were based on core logging, grade, structural and geochemical data.
2. Raw drill hole assays were composited to 1.25 metre lengths, honouring lithology boundaries.
3. Capping of high gold and silver grades was considered necessary and was completed on assays prior to compositing.
4. High yield restriction of base metal high grade and density was completed for each domain after compositing.
5. Block grades for zinc, gold, silver, copper, lead, iron, arsenic and density were estimated from the composites using ordinary kriging interpolation into 5 m x 5 m x 5 m blocks coded by domain.
6. Density values are from a multi elements regression formula based on 65,792 measurements.
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Metal prices of $US 1.19/lb zinc, $US 1,300/oz gold, $US 2.67/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to calculate a zinc equivalence (Zn Eq) cut-off of 4.1%, where Zn Eq = Zn% + (1.98 x Cu%)

+ (1.11 x Au g/t) + (0.01 x Ag g/t) – (0.01 x Pb%). The Zn Eq considers the ratio of milling recovery, payability and value of metals after application of downstream processing costs. The Zn Eq cut-off of 4.1% covers administration overhead, mining removal, milling and general and administration costs.

9. Totals may not add up correctly due to rounding.

TABLE 14-51: GOLD MINERAL RESOURCE, INCLUSIVE OF MINERAL RESERVES, BY CATEGORY AND MINERALIZED ZONE WITH A CUT-OFF OF 2.4 G/T AU EQ, AS OF SEPTEMBER 30, 2016 (1)(2)(3)(4)(5)(6)(7)(8)(9)

Category	Zones	Tonnes	Zinc	Gold	Copper	Silver
Measured	21	116,000	0.53	7.10	0.65	32.87
	23	72,000	0.28	5.90	0.25	45.24
	24 Zones	52,000	0.64	4.71	0.25	23.95
	25 Zones	92,000	0.34	6.65	0.28	28.81
Subtotal		**332,000**	**0.44**	**6.34**	**0.40**	**33.03**
Indicated	21	920,000	0.53	6.58	0.70	32.11
	23	367,000	0.31	5.69	0.21	49.87
	24 Zones	716,000	0.79	5.55	0.33	36.98
	25 Zones	1,362,000	0.44	5.88	0.33	29.02
	26 Zones	288,000	0.53	5.26	0.43	55.33
	27 Zones	455,000	0.34	8.71	4.72	28.97
Subtotal		**4,108,000**	**0.50**	**6.23**	**0.89**	**34.80**
Measured + Indicated	21	1,036,000	0.53	6.63	0.70	32.20
	23	439,000	0.31	5.72	0.22	49.11
	24 Zones	768,000	0.78	5.49	0.32	36.09
	25 Zones	1,454,000	0.43	5.93	0.32	29.01
	26 Zones	288,000	0.53	5.26	0.43	55.33
	27 Zones	455,000	0.34	8.71	4.72	28.97
Subtotal		**4,440,000**	**0.50**	**6.24**	**0.86**	**34.67**

Inferred	21	524,000	0.60	4.90	0.49	38.68
	24 Zones	31,000	0.73	1.69	0.95	25.04
	25 Zones	1,727,000	0.25	5.35	0.25	27.38
	26 Zones	674,000	0.33	4.06	0.38	35.99
	27 Zones	896,000	0.19	5.98	2.93	16.85
	28 Zones	272,000	0.34	2.78	0.38	22.79
Subtotal		**4,124,000**	**0.31**	**5.02**	**0.90**	**27.61**

Notes:

1. Domains were modelled in 3D to separate mineralized zones from surrounding waste rock. The domains were based on core logging, grade, structural and geochemical data.
2. Raw drill hole assays were composited to 1.25 metre lengths, honouring lithology boundaries.
3. Capping of high gold and silver grades was considered necessary and was completed on assays prior to compositing.
4. High yield restriction of base metal high grade and density was completed for each domain after compositing.
5. Block grades for zinc, gold, silver, copper, lead, iron, arsenic and density were estimated from the composites using ordinary kriging interpolation into 5 m x 5 m x 5 m blocks coded by domain.
6. Density values are from a multi elements regression formula based on 65,792 measurements collected by Hudbay.
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Metal prices of $US 1,300/oz gold, $US 2.67/lb copper and $US 18.00/oz silver with a CAD / US foreign exchange of 1.25 were used to calculate a gold equivalence (Au Eq) cut-off of 2.4 g/t Au Eq, where Au Eq = Au g/t + (1.34 x Cu %) + (0.01 x Ag g/t). The Au Eq considers the ratio of milling recovery, payability and value of metals after application of downstream processing costs. Au Eq cut-off of 2.4 g/t covers administration overhead, mining removal, milling and general and administration costs.
9. Totals may not add up correctly due to rounding.

14.12 Mine Reconciliation of Block Model

A mine reconciliation of the block model was carried out on the mined out areas. The process involved selecting all mined out blocks and comparing to the actual metal balance reported as ore received at the Stall concentrator. Mined out areas from the block model do not include dilution or pillars left behind after mining extraction. Table 14-52 list the mined out areas by lense from the block model and Table 14-53 is the ore reported by year at the Stall concentrator since Lalor commenced production in August 2012 until September 2016.

TABLE 14-52: MINED-OUT AREAS FROM THE BLOCK MODEL

Lense	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
10	1,560,246	9.63	1.94	0.71	19.39
11	55,355	14.46	0.14	0.30	17.22
20	336,600	8.80	2.19	0.82	30.71
21	99,594	0.65	7.70	0.65	30.85
23	24,210	1.50	5.37	0.61	39.89
24	25,849	0.89	6.44	0.42	23.24
25	56,481	0.64	8.47	0.40	31.93
30	9,098	4.25	1.82	0.26	35.67
31	64,845	4.73	0.95	0.20	15.14
32	183,404	7.98	5.51	1.60	54.22
Total	**2,415,684**	**8.60**	**2.65**	**0.75**	**24.52**
		Zn (tonnes)	**Au (ounces)**	**Cu (tonnes)**	**Ag (ounces)**
In-Situ Metal		207,641	205,689	18,196	1,904,304

TABLE 14-53: ORE RECEIVED BY YEAR AT THE STALL CONCENTRATOR

Year	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
2012	72,294	11.83	1.68	0.63	19.30
2013	400,589	9.44	1.20	0.84	19.41
2014	551,883	8.52	2.29	0.88	23.83
2015	934,278	8.18	2.53	0.71	21.39
2016 Sep YTD	814,207	6.88	2.30	0.64	21.62
Stockpile as of Sep 2016	11,114	5.38	2.14	0.41	21.42
Total	2,784,365	8.13	2.20	0.74	21.60
		Zn (tonnes)	**Au (ounces)**	**Cu (tonnes)**	**Ag (ounces)**
Metal		226,438	196,908	20,525	1,933,617
	Tonnes	**Zn**	**Au**	**Cu**	**Ag**
Variance to Block Model	115%	109%	96%	113%	102%

The block model compared very well to the ore reported at the Stall concentrator. The mine reconciliation concludes a 15% mining dilution and a metal variance reported at the Stall concentrator of 109% for zinc, 96% for gold, 113% for copper and 102% for silver. A mine reconciliation of 5 to 10% variance is well within industry standard. The precious metals reconciled very well, while there might be some conservatism of the zinc and copper grade estimates in the block model. The conservatism of the zinc and copper grade is believed to be linked to the high yield radius parameter. A previous model selected a 40 m high yield radius compared to the current 20 m radius used for this block model.

14.13 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the mineral resource estimate includes:

- Medium and long-term commodity price assumptions
- Operating cost assumptions
- Metal recovery assumptions used and changes to the metallurgical recovery assumptions as a result of new metallurgical information
- Changes to the tonnage and grade estimates may vary as a result of more drilling and new assay information
- Assumptions as to the ability to maintain mining claims and surface rights, access to the site, obtain environmental and other regulatory permits and obtain social license to operate

14.14 Conclusions

The mineral resource estimation in the base metal lenses is well-constrained by three-dimensional wireframes representing realistic volumes of mineralization. Exploratory data analysis conducted on assays and composites shows that the wireframes are suitable domains for mineral resource estimation. As for the gold mineralized zones, which are displaying high coefficient of variation, they could benefit from a modelling method with additional constraints.

As a result of validation steps conducted on the mineral resource block model the following was concluded:

- Visual inspection of block grade versus composited data shows a good reproduction of the data by the model.
- Checks for global bias in the grade estimates of the block model show differences within acceptable levels between the NN, IDW and the OK models. The variation are less than 7% for zinc and 4% for gold in the base metal lenses, while the variation of gold is less than 6% (aside from Zone 28) in the gold zones.
- Checks for local bias (swath plots) indicate good agreement between the NN, IDW and OK for all variables.
- A mine reconciliation of the mined out areas compared to the ore reported at the concentrator was very close on the precious metals and a slight conservatism of the zinc and copper grades might be evident.

- Applying linear interpolation method to the gold zones method may not produce highly accurate results given the high coefficient of variation of gold.

The impact of grade capping was evaluated by estimating uncapped and capped grade models. Generally, the amounts of metal removed by capping in the models are consistent with the amounts calculated during the grade capping study on the assays.

Mineral resources are constrained and reported using economic and technical criteria such that the mineral resource has reasonable prospects of economic extraction.

The estimated mineral resources for the Lalor mine conform to the requirements of 2014 CIM Definition Standards – for Mineral Resources and Mineral Reserves and requirements in Form 43-101F1 of National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

14.15 Recommendations

The author recommends that the following points be further investigated in order to increase the geological knowledge and confidence.

- The host rocks are heavily altered, metamorphosed and deformed, making the protoliths and the relevant alteration assemblage often hard to identify. Therefore, full geochemistry analytical package should be used to assay the core. This would enable the utilization of geochemical proxies and would allow the identification of the host rocks and alterations, hence increasing the geological knowledge and confidence.
- Lithological, alteration and structural interpretations should be added to the current grade shell model to increase the geological knowledge and confidence.
- As suggested by the statistical analysis and the validation process, the gold zones could benefit from a modeling method that would use additional constraints instead of relying on the interpreted geological continuity.
- The gold raft zones should be the focus of exploration to increase the geological knowledge and confidence. For instance, the rafts of zones 24 and 25 have developments that could be use for mapping purposes.
- In the event that a revision of the wire framing method does not improve the statistics and stationarity of the gold zones, applying non-linear interpolation methods should be investigated in order to ensure that the grade interpolation produce accurate results in the gold zones.
- Review interpolation methodology to reduce number of passes. The interpolation parameters for minimum and maximum number of composites, use of octants, etc. And their impact could be evaluated against the Herco adjusted NN model.

- Assess the calculation of the density from the interpolated grades rather than interpolating SG.

It is also recommended that Hudbay perform a check on grade smoothing using a global change-of-support correction. This should be performed to ensure that the grade smoothing is acceptable around the cut-off grades of interest.

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Lalor mine mineral reserves as of January 1, 2017 are summarized in Table 15-1. The mine plan was prepared using measured and indicated mineral resources from the block model. Inferred resources were assumed as waste. The mineral reserves were estimated based on a Life of Mine (LOM) plan prepared; using Deswik mine design software that generated mining inventory based on stope geometry parameters and mine development sequences. Appropriate dilution and recovery factors were applied based on cut and fill and longhole open stoping mining methods with a combination of paste and unconsolidated waste backfill material. The Qualified Person for the mineral reserve estimate is Robert Carter, P. Eng., Lalor Mine Manager, Hudbay Manitoba Business Unit.

TABLE 15-1: SUMMARY OF MINERAL RESERVES AS OF JANUARY 1, 2017

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Proven	4,383,000	6.76	2.37	0.76	27.33
Probable	9,849,000	4.39	2.72	0.65	26.12
Proven + Probable	14,232,000	5.12	2.61	0.69	26.50

Notes:

1. CIM definitions were followed for mineral reserve
2. Mineral reserves are estimated at an NSR cut-off of $88/t for longhole open stoping mining method and $111/t for cut and fill mining method
3. Metal prices of $US 1.07/lb zinc (includes premium), $US 1,260/oz gold, $US 3.00/lb copper, and $US 18.00/oz silver with a CAD / US foreign exchange of 1.10 were used to estimate mineral reserves.
4. Bulk density of the resource is reported in the block model is from a multi elements regression formula based on 65,792 measurements. Stope geometry shapes include waste dilution based on a bulk density of 2.8t/m3.
5. Totals may not add up correctly due to rounding.

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.

15.2 Dilution and Recovery

There are two sources of dilution: internal (planned) dilution and external (unplanned) dilution. Dilution amounts are included in the conversion of mineral resource to mineral reserves. The shallow dipping nature of the deposit and stacking of lenses results in multiple lenses being grouped together for mining purposes in the stope optimizer routines of Deswik so that they can be extracted as a single mining unit, based on stope mining parameters by mining method as shown in Table 15-2.

TABLE 15-2: STOPING PARAMETERS BY MINING METHOD

Stope Shape Parameters	Unit	Longhole	Cut and Fill
Length			
Minimum	Metres	23	150
Maximum		10	20
Width			
Minimum	Metres	3.5	5
Maximum		50	50
Waste Pillar Width	Metres	5	-
Stope Height			
Minimum	Metres	10	5
Maximum		20	5
Stope Dip			
Minimum	Degrees	35	75
Hanging Wall Dip			
Minimum	Degrees	20	70
Maximum		90	90
Footwall Dip			
Minimum	Degrees	50	70
Maximum		90	90
Dilution			
Hanging Wall Fixed	Metres	0.5	0.5
Footwall Fixed		0.5	0.5

Parameters most sensitive to Lalor mine are the minimum and maximum dip angles, which affects the dilution and recovery amounts of the optimized mining shape. The stope optimizer in Deswik generated an economic shape that honoured the geometric parameters.

The space between the lenses is treated as internal dilution and external dilution is set at a fixed distance of 0.5 m 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. Dilution, set at zero grade and a bulk density of 2.8 t/m^3, is based on the full mining shape with internal and external dilution. Average dilution factors by lense grouping are shown in Table 15-3.

TABLE 15-3: AVERAGE DILUTION FACTORS BY LENSE GROUPING

Lense Grouping	Diluted Tonnes	Average Dilution (%)
10, 11	4,278,000	19.8
20, 21, 24 ,25	6,277,000	19.7
23, 32	2,222,000	15.6
26, 28, 30, 40	2,639,000	14.8
27	626,000	29.4
31	618,000	11.3
Development	896,000	27.1
Total	**17,556,000**	**18.9**

Mining recovery is defined as the ratio of mineral resource tonnes delivered to the concentrator to the in-situ mineral resource tonnes. Mining recovery factors used for each mining method and by lense are shown in Table 15-4. Average recovery factors by lense grouping are shown in Table 15-5. Some of the mineral resources are not recovered due to:

- Mining design. This includes rib, post and sill pillars that are not recovered to maintain rock stability.
- Inefficiencies in mining. This includes small blocks of ore along ore/waste contacts and underbreak.
- Inefficiencies in mucking. This includes losses of broken rock in longhole stopes mucked by remote control LHD and broken rock that is mixed with waste backfill and is not mucked.

TABLE 15-4: RECOVERY FACTORS BY MINING METHOD AND LENSE

Mining Method	Lense	Criteria	Backfill	Recovery (%)
PPCF and CF	All	Standard 7 x 7m pillars	Waste	75
		Reduced pillar sizes	Paste	80
		>$250/t NSR ore	Paste	85
Drift and Fill	27	>$300/t NSR ore	Paste	90
Longhole	10 and 30	5m rib pillars	Waste	70
	10 and 30	No rib pillars	Paste	85
	20, 27, 32 and 10 above 755m level	4m rib pillars	Waste	72
	20, 27, 32 and 10 above 755m level	No rib pillars	Paste	90

TABLE 15-5: AVERAGE RECOVERY FACTORS BY LENSE GROUPING

Lense Grouping	Recovered Tonnes	Average Recovery (%)
10, 11	3,310,000	77.4
20, 21, 24 ,25	5,196,000	82.8
23, 32	1,941,000	87.3
26, 28, 30, 40	1,908,000	72.3
27	548,000	87.6
31	433,000	70.0
Development	896,000	100.0
Total	**14,232,000**	**81.1**

15.3 Conversion of Mineral Resources to Mineral Reserves

The mine plan was prepared using measured and indicated mineral resources from the block model. Inferred resources were assumed as waste. Appropriate dilution and recovery factors were applied based on cut and fill or longhole open stoping mining methods and backfill or unconsolidated waste backfill. Upon determination of diluted and recovered mineral resources, stope economic criteria were applied.

Diluted and recovered mineral resources exceeding a Net Smelter Return (NSR) cut-off of $88/t for longhole open stoping and $111/t for cut and fill mining method are included in the mineral reserves. NSR's are based on metal grades from the stope optimizer and block model, long-term metal prices, concentrator recoveries, smelter treatment, refining and payabilities and a Hudbay Manitoba Business Unit administration cost.

15.3.1 Metal Prices

Metal prices of $US 1.07/lb zinc (includes premium), $US 1,260/oz gold, $US 3.00/lb copper, and $US 18.00/oz silver with an CAD / US foreign exchange of 1.10 was used to estimate mineral reserves.

15.3.1 Metallurgy

The orebody is polymetallic with economically significant metals being zinc, gold, copper and silver. There are two different ore types, both of which are assumed to be treated using conventional flotation at the Hudbay Stall concentrator:

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

Metallurgical performance at Stall concentrator indicates that the base metal and gold rich ores can be blended and metallurgical assumptions are shown in Table 15-6. Two concentrates will be produced, a zinc concentrate that will be shipped to the Hudbay Flin Flon metallurgical complex for production of refined zinc, and a gold enriched copper concentrate that will be shipped to third party smelters.

TABLE 15-6: METALLURGICAL ASSUMPTIONS

	Base Metal Concentrator
Gold to Copper Concentrate	57.6%
Silver to Copper Concentrate	50.9%
Copper to Copper Concentrate	82.8%
Copper Concentrate Grade	21.0%
Zinc to Zinc Concentrate	94.2%
Zinc Concentrate Grade	51.0%

15.3.1 Payability, Treatment Charges, Refining Charges

The basis of the NSR cut-off is further determined by applying the copper and zinc concentrate terms as shown in Table 15-7.

TABLE 15-7: COPPER AND ZINC CONCENTRATE TERMS

	Unit
Treatment	
Copper Concentrate	$US 85.00/dmt
Zinc Concentrate (includes refining)	$C 404.97/dmt
Freight	
Copper Concentrate (third party smelter)	$C 215.00/wmt
Zinc Concentrate (Hudbay smelter)	Included with milling
Payables	
Gold in Copper Concentrate	96% of content
Silver in Copper Concentrate	90% of content
Copper in Copper Concentrate	Minimum deduction 1 unit
Zinc in Zinc Concentrate	97.5%
Refining	
Gold	$US 5.00/oz
Silver	$US 0.50/oz
Copper	$US 0.085/lb
Zinc Marketing	
Distribution	$US 0.06/lb
Premium	$US 0.07/lb
Administration	
Copper Concentrate	$C 106.92/dmt

Zinc Concentrate	$C 139.95/dmt

The NSR cut-offs of $88/t for longhole open stoping and $111/t for cut and fill mining method are sufficient to cover all expenses related to mining and milling the mineral reserves as shown in Table 15-1.

The mineral reserves are supported by a LOM plan prepared using Deswik mine design software that generated mining inventory based on stope geometry parameters, and mine development and production sequences. This is further detailed in the LOM production plan as described in Section 16 of this report.

The author considers the dilution and loss factors to be appropriate for the mining methods selected and the stope geometry shape and applicable stoping 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 plan and NSR cut-offs that primarily focussed on capturing base metal resources for processing at the Stall concentrator. The secondary focus was to capture gold zone resources when in contact with or close proximity to base metal resources. In areas where a large separation existed between base metal and gold lenses, mining blocks were evaluated for economic stope mining shapes. When a non-economic shape was generated in a first pass, a second pass was evaluated for only base metal lenses and if an economic shape was generated the gold zone portion was removed. However, due to this larger separation, majority of these isolated gold lenses could have been evaluated independently of the base metal lenses and could potentially provide feed to a gold processing facility. Below approximately the 950 m level no attempt was made to generate an economic stope mining shape for gold zones 25 and 26 as the separation distance became too large. The author’s opinion is that these resources are potentially better suited for a gold processing facility and should be re-evaluated when Hudbay has a better understanding of their New Britannia gold mill and Birch Tailings Impoundment Area in Snow Lake.

Of the current 14,232,000 tonnes of mineral reserves, approximately 80% is converted from base metal resources and approximately 20% is converted from gold zone resources. Of the total reserves approximately 3.8% is represented by the indicated resources of copper-gold zone 27, which is inclusive of the 20% from the gold zone, noted above. Although the indicated resource of copper-gold zone 27 as shown in Table 14-51, is converted to reserves and is planned to be processed at the Stall base metal concentrator, it has the potential to be milled at the New Britannia gold mill if the refurbishment plan and the installation of a copper pre-float facility proves to be economically viable.

The author recommends that base metal indicated resources, exclusive of reserves, as shown in Table 15-8 remain as indicated resources until such a time that detailed mine planning is completed. Furthermore, the author recommends that gold zone indicated resources, exclusive of reserves, as shown in Table 15-9 still have reasonable prospects of economic extraction at either Stall base metal concentrator or a gold processing facility.

TABLE 15-8 BASE METAL INDICATED RESOURCE, EXCLUSIVE OF RESERVES, WITH A CUT-OFF OF 4.1% ZN EQ, AS OF SEPTEMBER 30, 2016 (1)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Indicated	2,100,000	5.34	1.69	0.49	28.10

Notes:

1. Refer to the Notes for Table 14-50 of this Technical Report for more information

TABLE 15-9 GOLD INDICATED RESOURCE, EXCLUSIVE OF RESERVES, WITH A CUT-OFF OF 2.4 G/T AU EQ, AS OF SEPTEMBER 30, 2016 (1)

Category	Tonnes	Zn (%)	Au (g/t)	Cu (%)	Ag (g/t)
Indicated	1,750,000	0.40	5.18	0.34	30.61

Notes:

1. Refer to the Notes for Table 14-51 of this Technical Report for more information

16 MINING METHODS

16.1 Introduction

The mining method process includes underground lateral advance (development rounds), production mining, backfilling and transporting ore to surface. Geotechnical information, orebody geometry interpreted from diamond drill core and recent experience mining within the deposit were the major considerations for selection of mining methods.

A geotechnical report, along with stoping recommendations was prepared in 2010. The ore zones, hanging wall and footwall rock is of good quality allowing the use of mechanized drilling and blasting techniques. The orebody dips at an average of -30°, with localized dips as flat as -10° and as steep as -55°. Mining methods that are currently in use or planned in the immediate future include: mechanized cut and fill, post pillar cut and fill, drift and fill and longhole open stoping (transverse and longitudinal retreat).

Ore is mucked using Load Haul and Dump (LHD) loaders which are operated remotely in inaccessible areas. The ore is then loaded into underground haul trucks or ore passes and transported to the ore handling system at the production shaft for hoisting to surface.

A paste backfill plant will be constructed on site, planned for the first quarter of 2018. Paste backfill will be used in high grade areas to increase recovery and accelerate the mining cycle. Lower grade areas will be filled with waste rock from waste development. No waste is planned to be hoisted.

Ore delivered to the production shaft is sized to less than 0.55 m at one of the two rock breaker/grizzly arrangements and hoisted from the mine by two 16 tonne capacity bottom dump skips in balance. Ore is truck hauled to a primary crusher at the Chisel North mine site, crushed to less than 0.15 m, and then trucked to the Stall concentrator for further processing.

16.2 Lateral Development

Lateral advance is made in 4 m long segments (rounds), with typical dimensions of 6 m wide by 5 m high. Lateral drilling is completed with two boom electric hydraulic jumbo drills, each round requires approximately 80 holes. Rounds mined in low sulphide areas or waste is blasted using ANFO, while rounds mined in high sulphide areas (ore or waste) are blasted using an emulsion with a sulphide blast inhibitor. Ore and waste are mucked using LHDs. Following mucking, standard ground support, consisting of resin grouted rebar and welded wire mesh to within 1.8 m of the sill, is installed. Mine services, including compressed air, process water and discharge water pipes, paste backfill pipeline, power cables, leaky feeder communications antenna and ventilation duct are installed in main levels and stope entrances.

Generally, main levels are developed parallel to and in the footwall of the ore zones. To optimize development, in some areas of the mine, main levels are located to provide access to multiple ore zones. As levels are developed, stope entrance crosscuts are stubbed off and used as temporary remucks. Main levels are connected by a haul ramp to allow mechanized equipment to travel from level to level.

Stope access crosscuts to cut and fill stopes are driven at -15% to allow multiple cuts from a single crosscut. Cut and fill stope entrances are located approximately every 150 m along strike. For longhole stopes, transverse mining areas have drawpoint crosscuts developed in waste from the footwall and longitudinal mining areas have primary access points and then sill along the orebody with stopes being retreated to the access point.

16.3 Vertical Development

Main ventilation raises and ore pass raises are developed by a mining contractor using a raisebore and/or Alimak climber and hand held drills. Ground support and ladder ways, if required, are installed to Hudbay standards.

Longhole slot raises, transfer ore passes and auxiliary ventilation raises are limited to approximately 35 m long and are developed using longhole conventional drop raise techniques.

Drain, backfill and electrical cable holes are drilled using longhole or raisebore drills, and reamed to designed diameter.

16.4 Stope Mining

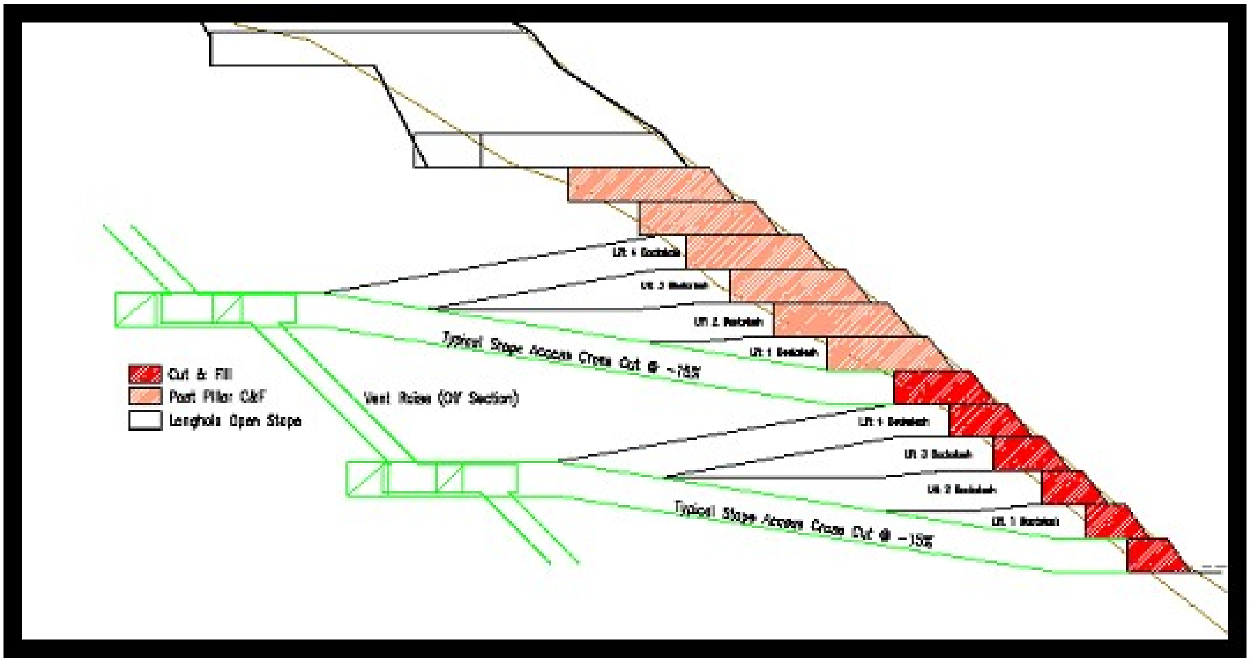
Two main mining methods are used at Lalor mine, cut and fill and longhole open stoping. Cut and fill methods include: mechanized cut and fill, post pillar cut and fill and drift and fill. Longhole open stoping methods include: transverse, longitudinal retreat and uppers retreat. Each mining area is evaluated to determine the most economic stoping method. In general where the dip exceeds 35° and the orebody is of sufficient thickness, longhole open stoping is used and lateral based cut and fill mining methods are used in flatter areas.

Approximately 65% of the mineral reserves are mined using the longhole open stoping methods and 35% are mined with cut and fill methods.

16.4.1 Cut and Fill Mining

Where the ore is flatter than 35°, single pass overhand mechanized cut and fill mining is generally the mining method chosen. The ore is accessed from a footwall drift by a crosscut developed at approximately -15%, a typical cross section is shown in Figure 16-1. Ore is mined 5 m high up to the hanging wall and footwall contacts. When ore mining is complete, ore remaining on the sill is mucked, pipe and ventilation duct is stripped, backfill is placed and the entrance crosscut is back slashed to provide access to the next cut.

FIGURE 16-1: TYPICAL CUT AND FILL MINING CROSS SECTION

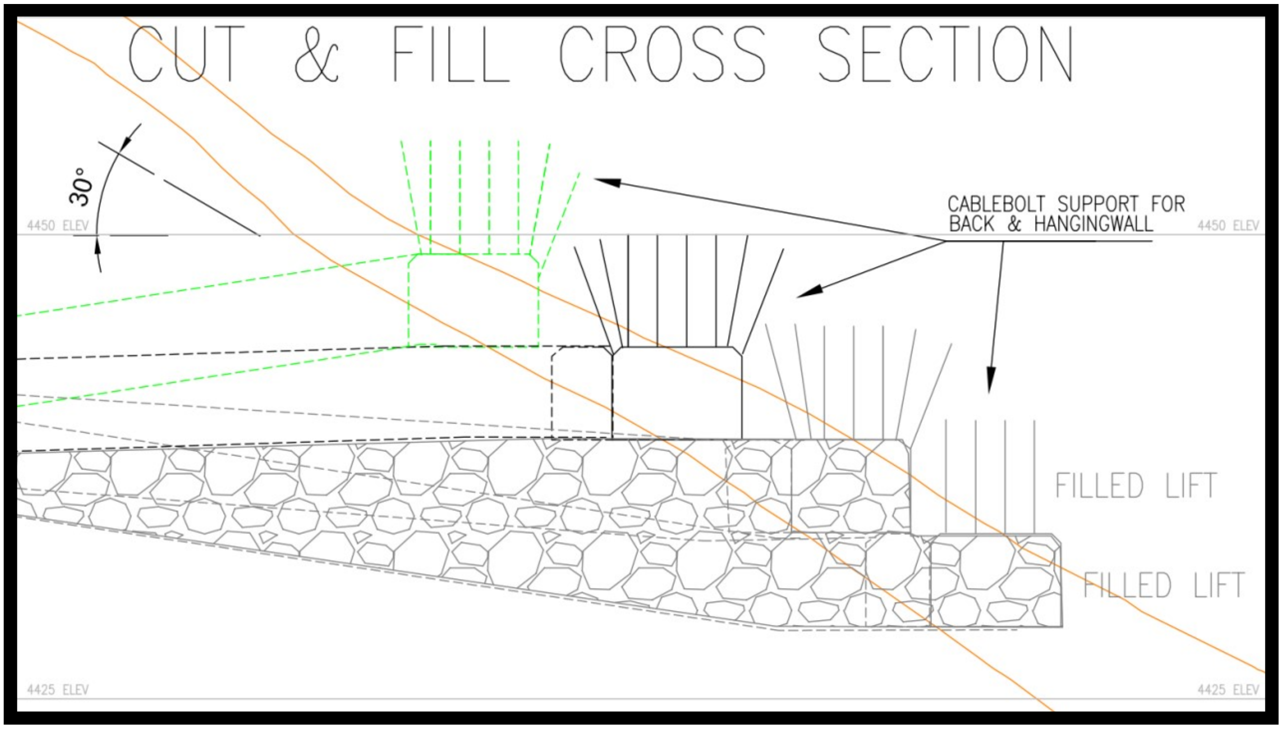


16.4.1.1 Mechanized Cut and Fill Mining

Where mining is planned to be less than 14 m wide and economics do not warrant consolidated fill, single pass overhand mechanized cut and fill mining is generally the cut and fill mining method chosen. Unconsolidated backfill is placed tight to the back and the entrance crosscut is then back slashed to provide access to the next cut.

Ground control used in mechanized cut and fill mining is 3.6 m resin rebar in the back for sections <10.8 m wide, 2.2 m resin rebar in the walls to within 1.8 m of the sill and welded wire mesh. In areas of excessive width, cement grouted cablebolts are installed to provide additional support and the area is mined in two passes or slashed and mucked/filled remotely. See Figure 16-2 for a typical cross section.

FIGURE 16-2: TYPICAL MECHANIZED CUT AND FILL MINING CROSS SECTION



16.4.1.2 Post Pillar Cut and Fill Mining

Where mining is planned to be greater than 14 m wide and economics do not warrant consolidated fill, overhand post pillar cut and fill mining is generally the mining method chosen. Post pillars provide ground support to allow selective mining of wide stopes. Backfill is placed to within 1.8 m of the back and the entrance crosscut will be back slashed to provide access to the next cut.

Design for drifts and crosscuts in post pillar stopes is 7 m wide x 5 m high, with 7 m x 7 m post pillars.

Ground control used in post pillar cut and fill mining is 2.2 m resin rebar in the back outside of intersections, 3.6 m resin rebar in the back in intersections, 2.2 m resin rebar in the walls and pillars to within 1.8 m of the sill and welded wire mesh.

Post pillar mining is sequenced to retreat mining towards the access to reduce risk of encountering poor ground conditions. Typical mining sequences are shown for two different mining areas in Figure 16-3 as a section view and Figure 16-4 as a plan view.

FIGURE 16-3: TYPICAL POST PILLAR CUT AND FILL MINING CROSS SECTION

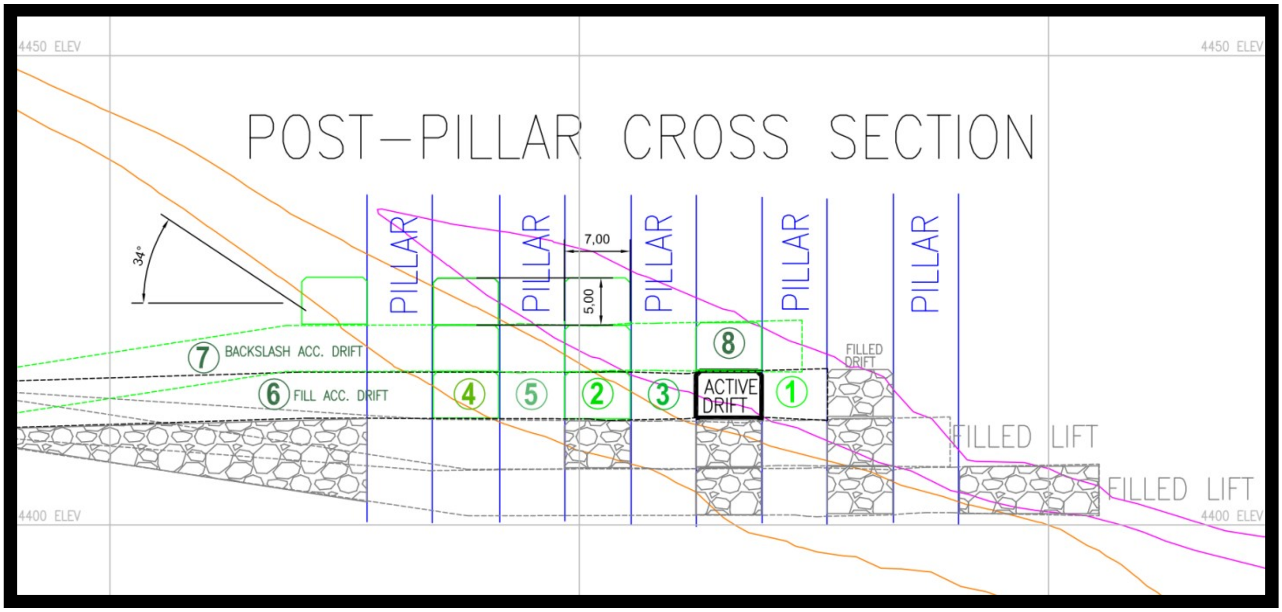
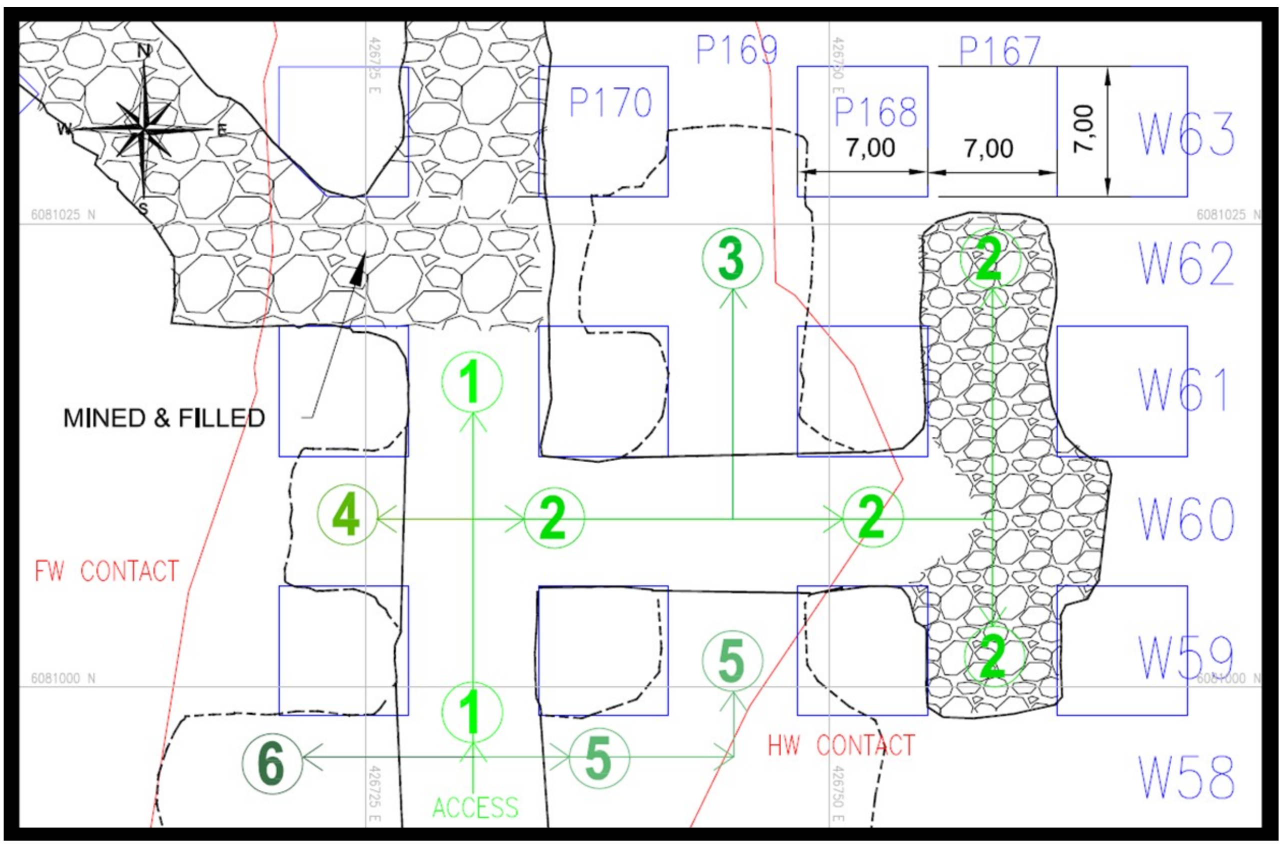


FIGURE 16-4: TYPICAL POST PILLAR CUT AND FILL MINING PLAN VIEW



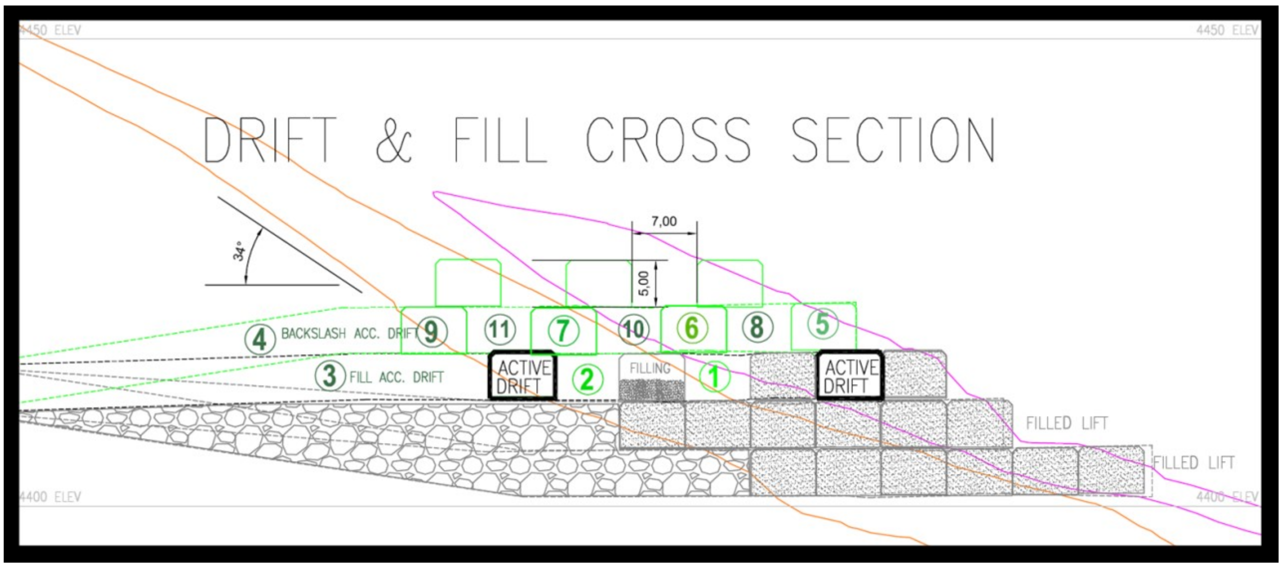
16.4.1.3 Drift and Fill Mining

Where cut and fill mining is planned and economics do warrant consolidated fill, drift and fill mining is generally the mining method chosen. Filling the initial ore drifts with consolidated fill allows successive mining of ore drifts adjacent to the filled areas without leaving pillars allowing complete recovery of the resource. Figure 16-5 shows a two lift sequence of drift and fill mining.

Design for drifts in drift and fill stopes is planned 7 m wide x 5 m high.

Ground control used in drift and fill mining is 2.2 m resin rebar in the back and walls to within 1.8m of the sill and welded wire mesh.

FIGURE 16-5: TYPICAL DRIFT AND FILL MINING CROSS SECTION



16.4.2 Longhole Open Stope Mining

Where the footwall of the ore is steeper than 35°, longhole open stope mining is generally the mining method chosen. The advantages of longhole mining over cut and fill methods include:

- Longhole open stoping is a non-entry method. This reduces our employee's exposure to risk on a per tonne basis.
- Longhole open stoping is a bulk mining method. The amount of ground support and labour per tonne is reduced lowering the overall cost per tonne compared to cut and fill mining methods.

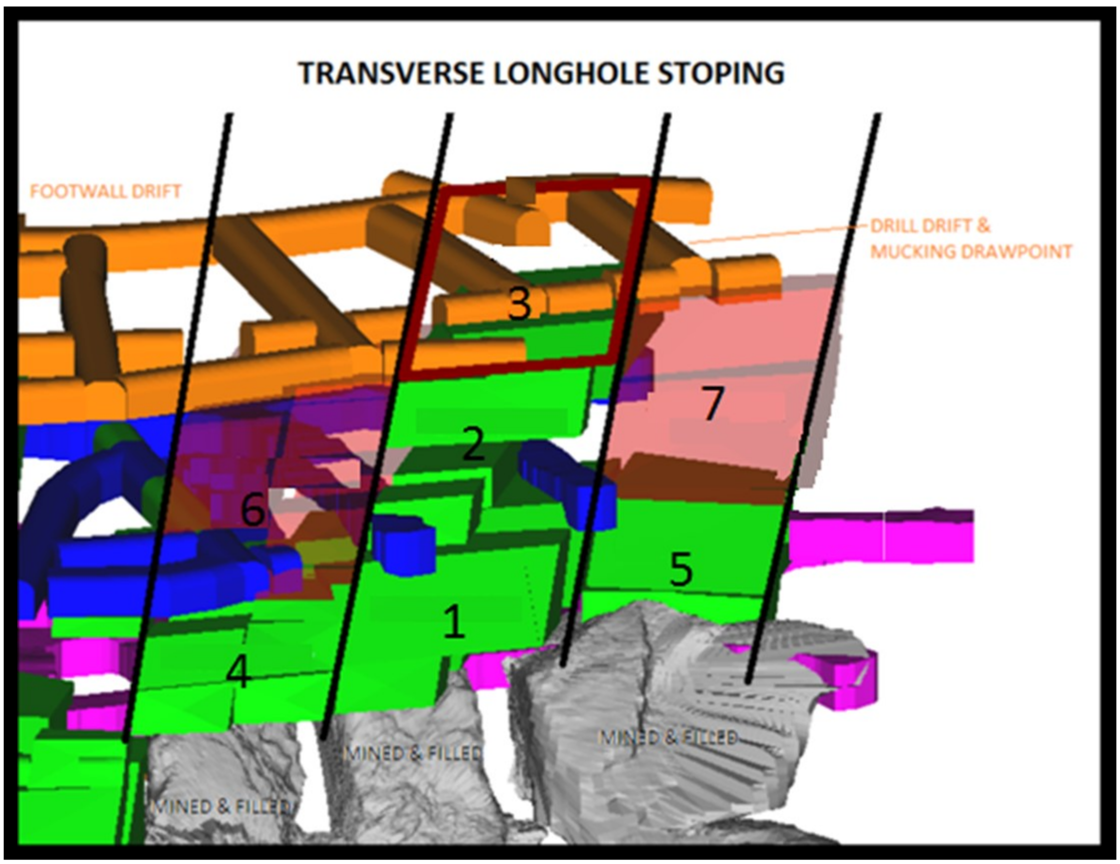
Longhole stoping areas are evaluated to determine the optimum sill to sill vertical interval. In the shallower dipping areas of the ore this interval is typically 15 m, and in steeper dipping areas the interval is planned up to 25 m. Optimizing the sublevel interval allows minimal dilution and optimal recovery. Stope widths are also evaluated by area typically being less than 25 m and more than 15 m wide. Hole diameter currently used is 7.6 cm, but may be adjusted based on ore fragmentation and cost.

Depending on economics and availability, consolidated or unconsolidated fill will be used. When using unconsolidated fill a nominal 5 m pillar is established between the stopes to contain the fill.

16.4.2.1 Transverse Longhole Open Stope Mining

The 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 16-6. Drilling is down the hole with a top hammer longhole drill. Stope sequence is reviewed for each area and typically progresses center out, bottom up. In future areas a primary-secondary sequence may be feasible.

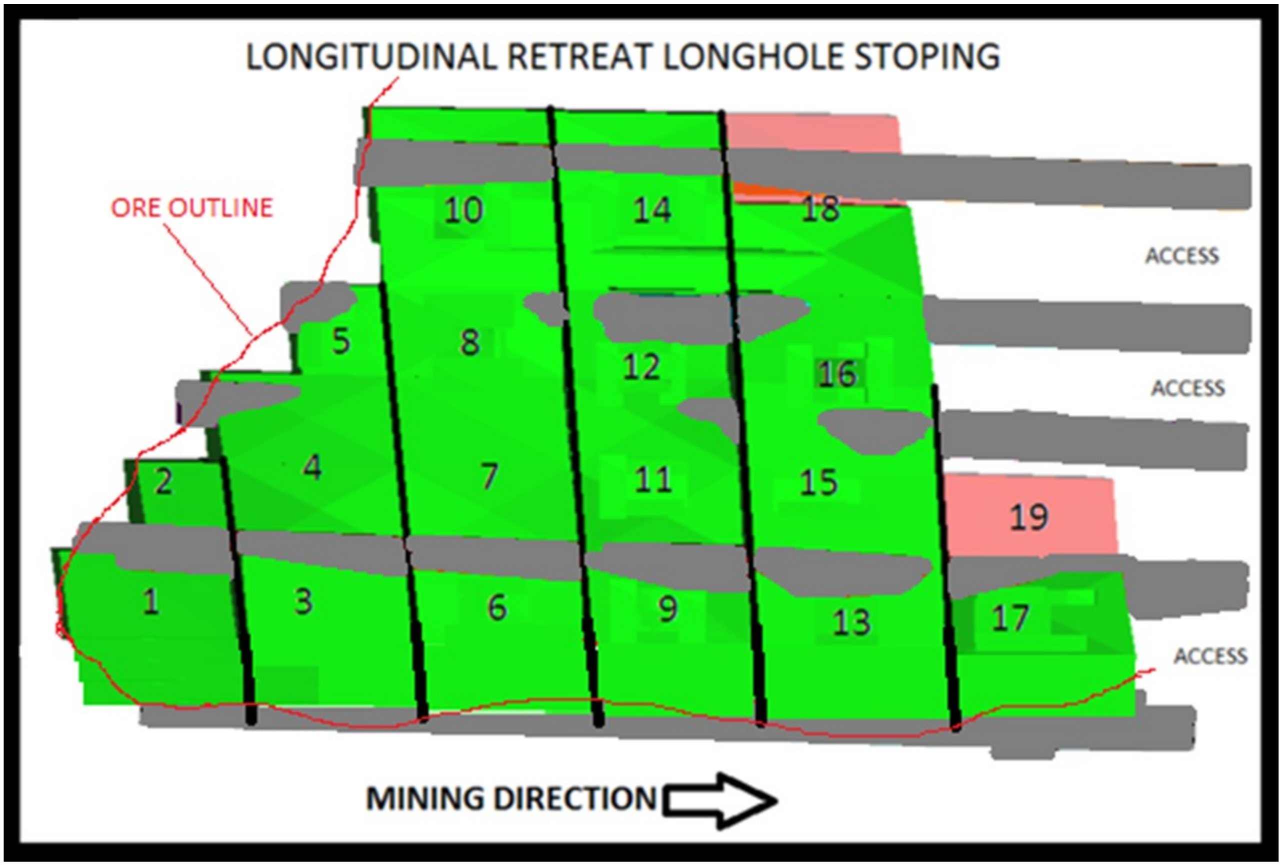
FIGURE 16-6: TYPICAL ISOMETRIC VIEW – TRANSVERSE LONGHOLE OPEN STOPING



16.4.2.2 Longitudinal Retreat Longhole Open Stope Mining

The ore is undercut at the top and bottom of the block with sill drifts off the main access drift, providing access for drilling and mucking. Drilling is down the hole with a top hammer longhole drill. Stope sequence is reviewed for each area typically bottom up and retreating along the length of the sill to an access point at the end of the lense or from both ends of the sill toward a central access point, see Figure 16-7.

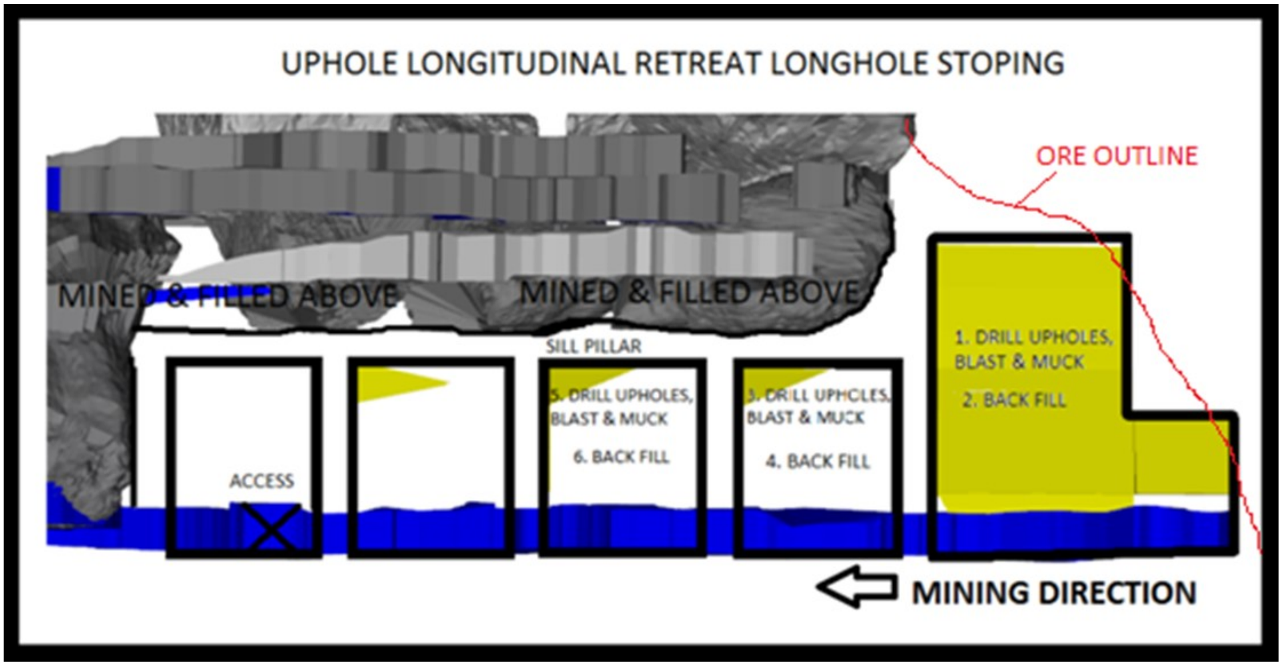
FIGURE 16-7: TYPICAL LONG SECTION – LONGITUDINAL RETREAT LONGHOLE OPEN STOPING



16.4.2.3 Uppers Retreat Longhole Open Stope Mining

The ore is undercut at the bottom of the block with sill drifts off the main access drift, providing access for drilling and mucking. Drilling is up the hole with a top hammer longhole drill. Stope sequence is reviewed for each area typically top down and retreating along the length of the sill to an access point at the end of the lens or from both ends of the sill toward a central access point. This method is generally used to recover sill pillars and requires consolidated fill, see Figure 16-8.

FIGURE 16-8: TYPICAL LONG SECTION – UPPERS RETREAT LONGHOLE OPEN STOPING



16.5 Backfill

All stopes at Lalor mine are backfilled to maintain long term stability and to provide a floor to work from for subsequent mining. Backfill is either:

a) Unconsolidated waste rock backfill (URF)

b) Consolidated backfill

   a. Cemented waste rock backfill (CRF)

   b. 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 consolidated backfill.

Consolidated backfill currently consists of cemented waste rock backfill and is planned to be primarily as paste backfill after commissioning of the paste plant in the first quarter of 2018. Where economically feasible consolidated backfill is used by adding cement to waste rock using a spray bar and placing it in stopes with LHDs or when paste is available using the underground distribution system to transport paste (via gravity) directly to the stope. Consolidated backfill is required to maintain long term stability and allow future recovery of sill pillars.

The majority of consolidated backfill will be paste. 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. Hudbay has experience with the design and operation of a paste backfill system, currently in use at the Flin Flon concentrator and 777 Mine. Paste backfill has advantages over unconsolidated fill such as slurried mill tailings or loose waste rock as follows:

a) Confines pillars in post pillar cut and fill stopes to increase pillar stability.

b) Flows to the hanging wall to seal off previous cut and fill cuts, improving ventilation control and limiting the potential for hanging wall failure. By comparison, unconsolidated waste rock backfill typically rills to approximately 50°.

c) Cures to a solid product. This allows mining up to the paste backfill walls in adjacent longhole stopes, and creates a good mucking floor in all stopes. This also eliminates the possibility of the build up of hydraulic head in stopes and potential flows of re-liquefied unconsolidated tails.

16.6 Ore Handling

Ore is mucked by LHD, loaded into underground haul trucks and hauled to one of the two ore passes that feed the shaft. Ore is dumped onto a grizzly at 910 m level for sizing to less than 0.55 m by a rockbreaker and grizzly. A 40 m raise and bin below the grizzly provides approximately 1,200 tonnes of coarse ore storage. A chute at the bottom of the raise at 955 m level feeds ore to a conveyor that loads a measuring flask with approximately 14 tonnes of ore. Ore is then 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, and then trucked to the Stall concentrator to process. Opportunities to increase ore handling capacity and installation of additional shaft ore passes are currently being reviewed. However, based on an internal review the current ore handling system has the capacity to move 4,500 tpd.

16.7 Surface Infrastructure

The Lalor mine surface infrastructure includes the following:

a) 3.5 km mine site access corridor, which includes the mine haul road, 25 kV overhead power line and covered/heat traced process water and mine discharge water pipelines.

b) Onsite services distribution. This includes pole and buried electrical services and switchgear and buried freshwater and mine discharge water pipelines.

c) Hoist house, including: hoist foundations, mine service hoist, and production hoist.

d) Production shaft collar and foundations, including ventilation plenum.

e) Ore bunker, ore bin and head frame.

f) Warehouse/shop complex.

g) Propane Storage.

h) Haulage truck route.

i) Mine water handling systems, including onsite pump house, mine discharge water tanks and pumps and freshwater tanks and pumps. Offsite booster pump station for freshwater and mine discharge water.

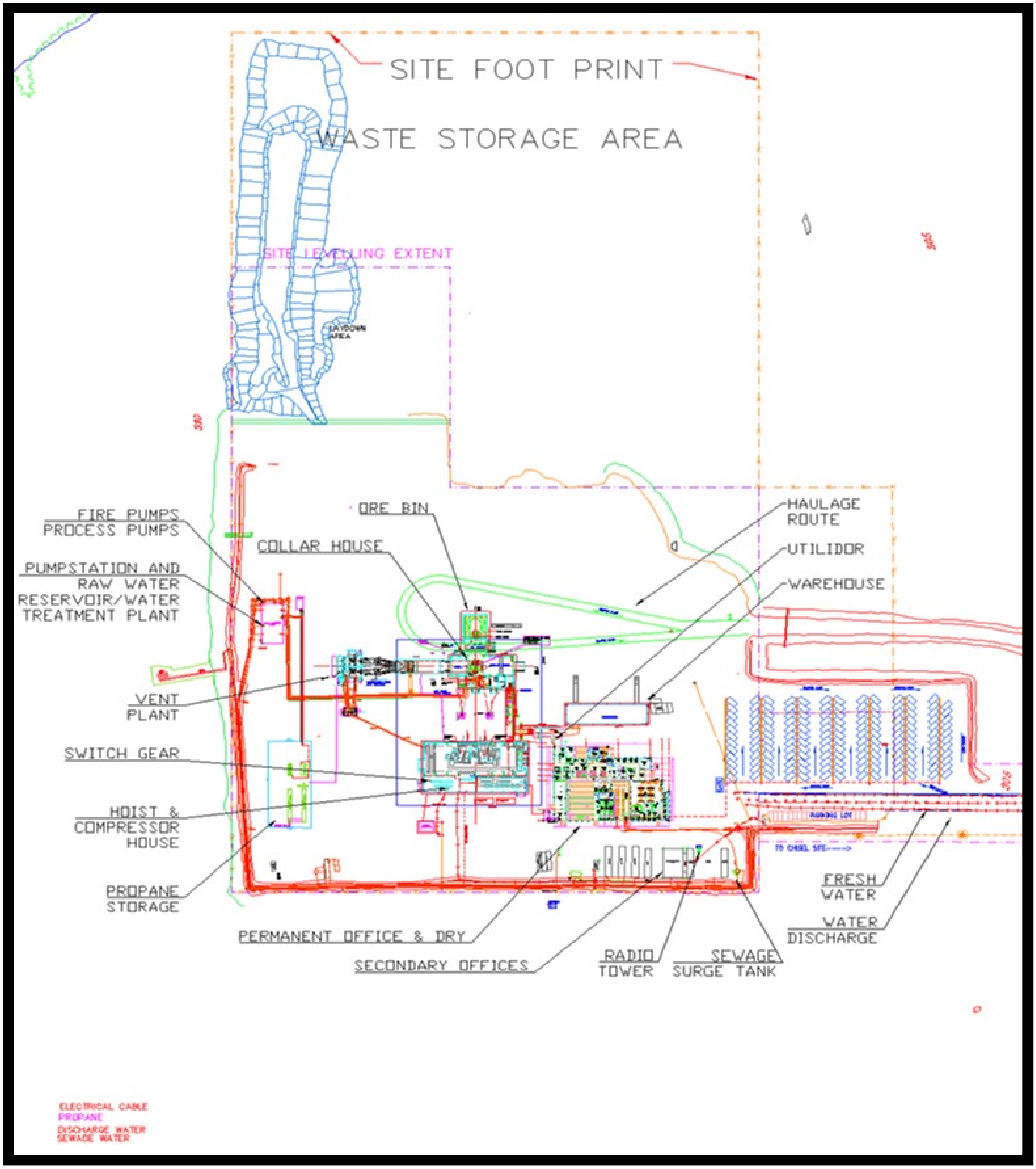
j) Offsite permanent substation with two (2) 115-25 kV 24 MVA transformers.

k) 341 person change house complex that also houses staff offices.

l) Permanent ventilation installations, including offsite exhaust fan and onsite mine downcast air heater and fans.

Additional information regarding mine infrastructure is included in Section 18. Refer to Figure 16-9 for the Lalor mine site general arrangement drawing.

FIGURE 16-9: SITE GENERAL ARRANGEMENT



16.8 Geotechnical Design

Initial geotechnical design was completed by Stantec Engineering in 2009 and 2010. The orebody is flat lying beginning at approximately 600m from surface and extending to a depth of approximately 1,100 m. It trends between 320° to 340° azimuth and dips between 30° and 45° to the north with a lateral extent of about 900 m north to south and 700 m east to west. A number of distinct stacked mineralized zones have been interpreted. The geotechnical design was based on a preliminary lithology of the Lalor mine orebody as shown in Table 16-1.

TABLE 16-1: LATERAL JUMBO DEVELOPMENT

Domain/Zone	Rock/Mineralization Type
Hanging wall Rock	Basaltic wacke, crystal wacke/fragmental polymodal fragmenta. Cordierite + anthophyllite, mafic tuffs
Zone 10	Solid to near solid sulphides and minor disseminated sulphides
Footwall to Zone 10	Sericite + kyanite + pyrite schist, staurolite + garnet + biotite gneiss
Hangingwall Zone 20	Altered dacite, amphibolitized mafic volcanic, calc. Silicate gneiss, diorite
Zone 20	Near to solid sulphides
Footwall Zone 20	Chlorite schist, altered dacite, chlorite sericitic talc schist, quartz-biotite-amphibolite gneiss
Hanging wall Zone 30	Mineralized chlorite schist, quartz chlorite schist
Zone 30	Disseminated to near/solid sulphides
Footwall Rock	Rhyolite fragmental, quartz biotite schist and chlorite schist

The comments below refer to both short term and long term requirements for Lalor mine geotechnical data acquisition, including training requirements.

16.8.1 Geotechnical Requirements

Geotechnical Logging

a) To ensure that rock mass quality is logged and assessed to industry standard, Lalor has established diamond drill hole (DH) core logging by trained technical personnel.

b) Lalor core logging format includes geology descriptions and geotechnical data collection (SG, UCS, E, ν, RQD, Q', Jn, Jr, Ja) and is located in the same summary sheet for ease of data analysis and geotechnical modeling.

Geology and Structures

a) Geotechnical data from all the know zones is continuously collected following standard procedures. This is done by the Lalor geology department and data is shared with ground control.

b) All geological core logging is examined to identify and characterize any major structural features, including major fault and/or shear zones that may intersect zones of the mineralization.

c) Representative samples from various geotechnical domains, including hanging wall, various ore zones and footwall, are collected to test rock properties, including Unconfined Compressive Strength (UCS), Young Modulus and Poisson Ratio.

d) A geotechnical database has been established from drill core logging data and geotechnical core sample laboratory tests.

e) Footwall lithology is interpreted to identify the location and extent of the weak zones where future internal ramp and other major infrastructure may be proposed.

f) The interpretation of the various lithological units in and around various zones of the orebody, i.e. the major geotechnical domains, and the major structural interpretation, i.e. major faults and shear zones, is generated/mapped in geology and mine planning applications for mine design and ground control use.

Rock Mass Classification Plots and Rock Properties Determination

a) Rock mass classification data (RQD) is generated in 3D space, similar to the ore grade distribution from all diamond drill holes.

b) 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.

16.8.2 Rock Mechanics Numerical Modelling

Numerical modeling is being conducted to help understand mining ground conditions and assist ground control decision making using available numerical modeling software on site (e.g. Examine 2D, ExamineTab). With mining progressing and stress interactions becoming more complicated, 3D numerical modeling may be considered.

16.8.3 Rock Mechanics Instrumentation

Ground monitoring instruments are currently being installed for ground movement monitoring. More advanced ground control instrumentation will be considered in the future.

16.9 Support Systems

Except when using cut and fill mining methods, all other drifts have arched backs for optimized shape and safety.

Ground support is broken down into primary support and secondary support.

Primary support refers to reinforcement of the rockmass immediately following excavation (first pass) to ensure safe working conditions before taking the next round. Primary support is typically undertaken with resin grouted rebar and #6 gauge galvanized welded wire mesh.

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. Secondary support is installed at a later stage (second pass) and typically is a batch process. Examples of secondary support are rebar, cable bolts, strandlok bolts, inflatable rock anchors, split sets and shotcrete.

Lalor mine ground support standards are summarized in the following sections.

16.9.1 Primary Support

Standard drift, <7 m wide: 2.2 m long #6 (for jackleg/stopper installations) or #7 (for bolter installations) resin rebar on a 1.2 m x 1.2 m square pattern. Rebar and screen are extended down the walls to within 1.8 m of the sill. Screen overlaps at least 2 squares during installation. FS-33 or FS-39 Friction Stabilizers may be used to pin screen to back and walls but not as primary ground support. Generally, this can be applied to a drift span up to 7.0 m if no major geological structures are encountered. Special design is needed for drift spans larger than 7.0 m or when major geological structure is present.

Intersections, 7 m to 10.8 m wide: ground support uses the same support as for standard drifts except in the back where 3.6 m long #7 resin rebar on a 1.2 m x 1.2 m square pattern is installed.

16.9.2 Secondary Support

Secondary ground support is installed when excavation spans are larger than 10.8 m, major unfavorable ground conditions or rock structures are present, and/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, single cable bolts on a 1.8 m x 1.8 m pattern for long term excavations, or high strength inflatable rock anchors for temporary or short term excavations. The minimum bolt length should be equal to one-third of the final drift span.

16.9.3 Developed in Two Passes – Mechanized Cut and Fill or Infrastructure Headings

Headings wider than 7.0 m are developed using two methods, single pass and double pass. Ground support will depend upon the development method used to excavate the opening.

First pass development ground support is as described in Sections 16.9.1 and 16.9.2 above.

On completion of the first pass and before starting the second pass, secondary ground support is installed. The type of secondary ground support can be longer resin rebar, cable bolts or shotcrete. It should be noted that cablebolts and shotcrete must be allowed to cure for at least 24 hours before any blasting within 30 m. The second pass is supported similarly to the first pass or the area is designated for remote operations only.

16.9.4 Sill Pillar Support

Sill pillars between mining blocks are occasionally formed. Depending on ore geometry, strength of the mineralization and depth of mining, poor ground conditions are anticipated in the last three cuts (15 m) or last longhole stope of each block, which will then be underneath excavated stopes. Drifts driven under excavated stopes require cablebolting when the sill pillar thickness is less than twice the width of the drift. The minimum lengths of cablebolts are to be equal to the width of the drift. Some shotcreting may also be required if conditions warrant.

All stopes will be backfilled. The fill type and mechanism of placement is important (refer to the following):

a) Where paste fill is chosen, filling tight will require the use of sacrificial pipes (pours in long and small drifts) with burst disks, or the use of blasting cord to cut the pipe to stage pour locations. Tight fill is defined as 70% fill contact with the back.

b) Where CRF is chosen, tight fill may be placed. However, the composition of the rock fill must be tightly controlled, as in many cases the mix can be too sloppy to tight fill with a steep rill.

c) Where URF is chosen, fill may be placed and pushed to within 0.5 m of the back or if the mining plan calls for mining directly above the area to be filled a larger void may be left.

As additional geotechnical data becomes available ground support designs are reviewed. In addition, calibrated two or three dimensional numerical modeling is undertaken to identify problematic areas. Control measures, including enhanced ground support, are installed to reduce the risk of future ground instability if needed.

16.10 Underground Development

16.10.1 Lateral Development

Drifts and ramps will be developed by dedicated mine development crews using the following equipment:

a) Drilling – Atlas Copco M2D Jumbos equipped with 4.88 m feeds and AC1838HD rockdrills. Development advance is nominally 4.0 m per round.

b) Bolting – 3 methods are used: 1) Maclean Bolters (MEM-928) equipped with a scissor deck and bolting boom, 2) Atlas Copco Boltec MC equipped with remote arm bolting boom and 3) Maclean Scissor Decks (MEM 977) as a platform for jacklegs and stopers. Bolters have rod adding systems to allow cablebolt and testhole drilling. The units are equipped with AC1638HD, AC50 Monobear or AC1435 rockdrills and screen handling features.

c) Mucking – LHDs, primarily Atlas Copco ST-14 and ST-1530.

d) Trucking – Atlas Copco haul trucks primarily MT-42 and MT-6020.

e) Development Loading – Rounds will be charged with explosives using a Maclean Explosives Loader (MEM-AC3) equipped to load both ANFO (~1360 kg capacity) and emulsion (~450 kg capacity).

f) Production Loading – Holes will be charged with emulsion (~1,800 kg capacity) using a Marcotte (M40) toe loader.

Lateral development required by year to mine the LOM reserves is shown in Table 16-2.

TABLE 16-2: LATERAL JUMBO DEVELOPMENT

Year	Sustaining Capital (m)	Mine Operating Waste (m)	Mine Operating Ore (m)	Total (m)
2017	2,918	2,471	4,526	9,915
2018	2,925	1,552	5,933	10,410
2019	1,785	2,120	6,506	10,411
2020	1,837	1,888	6,271	9,996
2021	1,369	1,830	6,672	9,871
2022	1,000	1,815	5,629	8,444
2023	77	2,303	4,788	7,168
2024		2,896	3,925	6,821
2025		1,571	3,816	5,387
2026		1,394	2,175	3,569
2027		71	414	485
Total	**11,911**	**19,911**	**50,655**	**82,477**

16.10.2 Vertical Development

Vertical development or raising is required at Lalor mine for the following reasons: 1) extension of the ventilation system, 2) secondary egress, 3) main orepass for the upper portion of the mine above the 910 m level rock breakers, 4) temporary transfer ore passes to optimize stope extraction and 5) slot raises for longhole stopes.

When a new raise is planned, three options are considered: raise boring, Alimak and drilled drop raising. Typically raises less than 35 m are designed as conventional drop raises and drilled with the production drill on site. 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 16-3.

TABLE 16-3: VERTICAL DEVELOPMENT

Year	Ventilation Raises (m)	Ore Handling Raises (m)	Total (m)
2017	465	438	903
2018	411	253	664

Year	Ventilation Raises (m)	Ore Handling Raises (m)	Total (m)
2019	654	178	832
2020	682	184	866
2021	548	137	685
2022	350	100	450
2023	344	48	392
2024	240		240
2025	80		80
2026			
2027			
Total	**3,774**	**1,338**	**5,112**

16.11 Diamond Drilling

Diamond drilling is completed by specialist contractor. Ore delineation holes are typically drilled from one zone to the other, and less than 200 m in length. Stope definition holes are drilled from undercut drifts through the contact to determine mucking sill elevations and grade planning (which could vary sequence in some cases). The diamond drill is equipped with universal rotation to allow test holes to be drilled in all azimuths and dips.

16.12 Drainage System

Lalor mine is a relatively dry underground mine with no significant hydrological concerns. The drainage for Lalor mine, Photo Lake mine and Chisel North mine is interconnected. Mine water reports to the water treatment plant at Chisel Lake where it is treated and released. The main collection areas feeding the water treatment plant are the 140 m level Pump Station at Photo Lake mine via the main ramp, surface portal and Photo Lake Pump House; the 840 m level Sump via the Lalor mine Ventilation Raise and Lalor mine Lift station; and the 955 m level main pumps via the Lalor mine Production Shaft and Lalor mine Lift Station. The Chisel North mine water is collected and that water is pumped to the 410 m level sump where it continues to the 955 m level main pumps at Lalor mine. All water within the mine is collected in intermediary collection sumps and proceeds to the main collection areas via drain lines, drain holes or drainage ditches.

16.13 Mining Operations

Typical development crew equipment consists of a two-boom electric hydraulic jumbo, and a mechanical bolter sized to excavate all lateral development (typical sizes include: 5.0 m x 5.0 m, 6.0 m x 5.0 m and 7.0 m x 5.0 m). The crew also uses LHDs, scissor lifts and backhoes for face preparation and extending services.

Production includes an ore handling system capable of removing 4,500 tpd from underground to surface.

Current production is approximately 3,000 to 3,500 tpd. At steady state by 2018, Lalor mine will produce 4,500 tonnes per day and be approximately 35% development based and 65% longhole. Development based production methods will produce approximately 4.2 ore rounds (1,600 tonnes) per day. Cut and fill mining areas are assumed to be in the ore producing portion of the mining cycle 75% of the time and in the backfill portion of the mining cycle or otherwise unavailable for mining 25% of the time. Longhole mining based production will produce approximately 2,900 tpd.

16.13.1 Mine Equipment

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 16-4.

TABLE 16-4: MINE EQUIPMENT

Description	Fleet
Underground Trucks 65 tonne	4
Underground Trucks 42 tonne	4
LHD 8yd	5
LHD 10yd	5
Two Boom Jumbo	4
Bolters (includes require for cable bolting)	8
Longhole Drills	3
Powder Trucks	3
Scissor Lift Trucks	8
Grader	1
Boom truck	2
Shotcrete Sprayer	1
Trans-mixers	2
Personnel Carriers Toyota	26
Miscellaneous Underground (Minecats, forklifts, etc.)	19
Miscellaneous Surface (Loader, forklift, pickups, etc.)	22
Total Mobile Equipment.	**117**
Ventilation Fans – Surface fans (250HP – 2500HP)	6
Ventilation Fans – U/G fans (50HP – 400HP)	54
U/G Submersible Pumps 100HP	7
U/G Submersible Pumps 60HP	1
U/G Submersible Pumps 50HP	1
U/G Submersible Pumps 40HP	13
U/G Submersible Pumps 34HP	1
U/G Submersible Pumps 20HP	7
U/G Submersible Pumps <20HP	5

Portable Refuge Stations	3
Shotcrete Machine - Wet Mix	1
Grout Pump c/w Mixer	3
Portable Welder	1
Total Stationary Equipment	**103**

An allowance for replacement equipment has been included in the mine plan. As part of the mobile equipment fleet management plan, major mobile equipment will be replaced at approximately 15,000 operating hours.

16.13.2 Production Schedules

The LOM production schedule, shown in Table 16-5, is currently set to ramp-up to 4,500 tonnes per day by 2018 and continues at that rate to the end of 2021 when a ramp-down begins to 3,000 tonnes per day in 2026. Deswik software was used to assist with the LOM planning and generate the basis for the production schedule. The geologic block model was imported to the software, where a stope optimizer algorithm was applied to create economical mining shapes. These mining shapes were then linked to the development drifts and sequenced individually by their respective locations and geometrical limits. Mine resources and rates, realized through historical data, were applied and levelled through activity priority labels. From this output, adjustments were made to further balance resources and scheduling to create an improved plan.

TABLE 16-5: LOM PRODUCTION SCHEDULE

Year	Tonnes	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)
2017	1,278,282	1.67	22.68	0.59	7.52
2018	1,616,285	2.13	24.37	0.52	5.71
2019	1,620,000	1.86	21.43	0.48	5.62
2020	1,603,652	2.79	28.43	0.79	4.61
2021	1,620,000	2.86	26.39	0.92	4.83
2022	1,473,657	3.16	26.72	0.95	5.72
2023	1,267,267	3.21	29.87	0.89	5.72
2024	1,212,738	3.14	28.35	0.60	4.49
2025	1,212,739	2.89	27.35	0.66	3.41
2026	1,022,918	2.78	32.83	0.49	3.55
2027	304,098	1.83	23.93	0.37	2.68
Total	**14,231,636**	**2.61**	**26.50**	**0.69**	**5.12**

Lalor mine will produce a total of 1,312,563 tonnes of zinc concentrate and 404,864 tonnes of copper concentrate in milling the ore from the LOM production plan, as shown in Table 16-6. The LOM contained metal in concentrate is shown in Table 16-6.

TABLE 16-6: LOM CONCENTRATE PRODUCTION BY YEAR

	Zinc Concentrate		Copper Concentrate			
Year	**Tonnes**	**Zn (%)**	**Tonnes**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**
2017	176,396	51.0	30,158	42.2	499.1	21.0
2018	166,124	51.0	33,298	55.3	552.6	21.0
2019	163,716	51.0	30,863	54.5	541.8	21.0
2020	130,580	51.0	53,179	48.7	492.7	21.0
2021	138,843	51.0	63,025	45.4	448.7	21.0
2022	151,843	51.0	58,903	49.2	446.9	21.0
2023	130,488	51.0	47,567	51.5	483.5	21.0
2024	99,888	51.0	29,786	70.7	549.8	21.0
2025	74,437	51.0	33,629	62.3	514.7	21.0
2026	65,591	51.0	20,064	74.6	649.6	21.0
2027	14,658	51.0	4,394	70.8	653.4	21.0
Total	**1,312,563**	**51.0**	**404,864**	**53.4**	**502.8**	**21.0**

TABLE 16-7: LOM CONTAINED METAL IN CONCENTRATE

Year	Zn (tonnes)	Cu (tonnes)	Au (oz)	Ag (oz)
2017	89,962	6,333	40,917	483,928
2018	84,723	6,993	59,202	591,589
2019	83,495	6,481	54,079	537,611
2020	66,596	11,168	83,265	842,391
2021	70,810	13,235	91,994	909,201
2022	77,440	12,370	93,174	846,328
2023	66,549	9,989	78,760	739,424
2024	50,943	6,255	67,705	526,512
2025	37,963	7,062	67,359	556,492
2026	33,451	4,213	48,122	419,039
2027	7,476	923	10,002	92,306
Total	**669,408**	**85,022**	**694,578**	**6,544,821**

16.13.3 Mine Ventilation

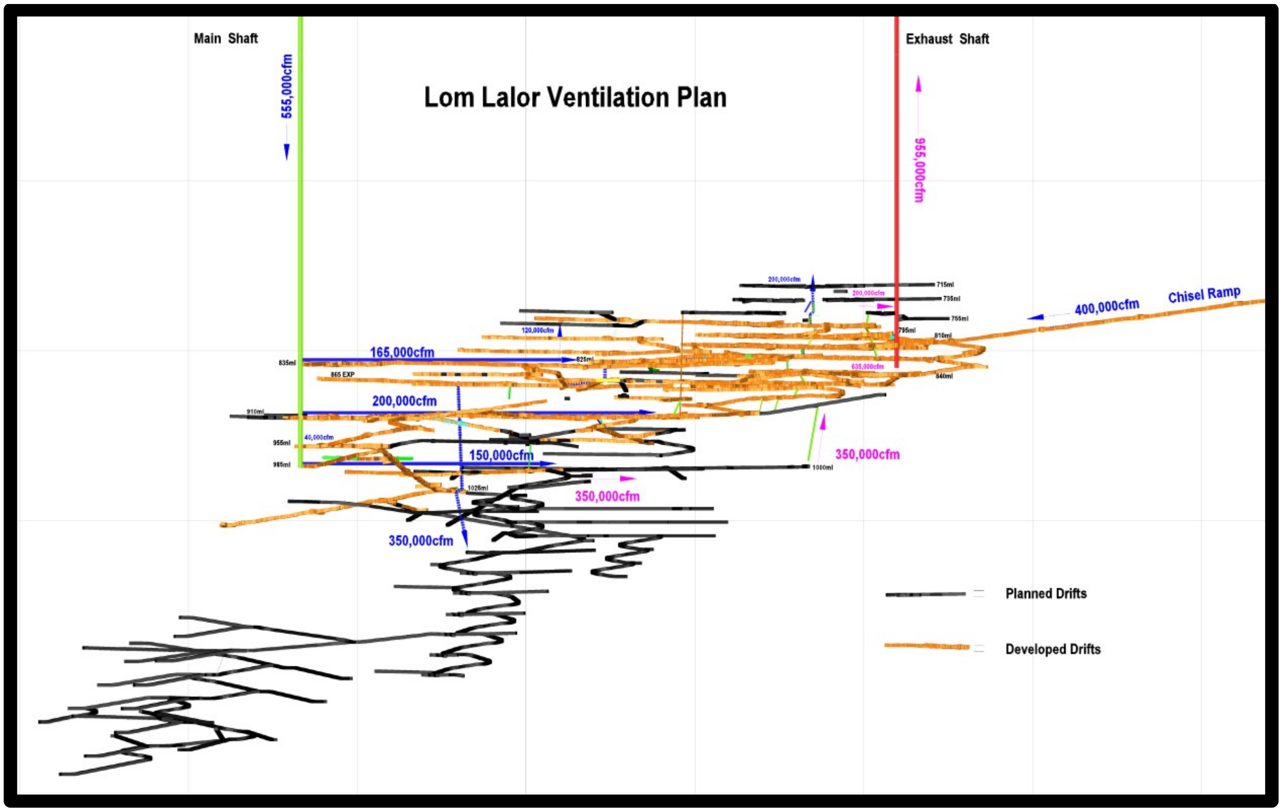
The Chisel North mine ventilation system in sequence with the Lalor mine Downcast Raise, provide 400,000 cfm down the Lalor mine Access Ramp, with 150,000 cfm exhausting to surface via the Chisel North mine Ramp. An additional 555,000 cfm is downcast via the Lalor mine Production Shaft for a total of 955,000 cfm exhausting up the Main Exhaust Shaft. In the summer total volume of air increases slightly. Three heaters heat mine air in the winter: the 36M BTU Chisel North Mine Heater, the 30M BTU Lalor Mine Ramp Heater and an 80M BTU heater at the production shaft.

To mine reserves below the 1025 m level there are plans for modifications to the original ventilation system to provide at least 200,000 cfm from the Chisel North mine Ramp System via existing levels and connect 1025 m level and the 865 m level North exploration drift with a 170 m (2.4 m x 2.4 m) Alimak raise. With approximately 150,000 cfm coming down the 1025 Ramp, the mine expects to have 350,000 cfm at its disposal to ventilate future mining horizons below 1025 m level over the LOM. Fresh air is distributed to different areas of the mine via a series of ventilation raises and cross cuts that are developed off the main ramp. Currently 200,000 cfm ventilates the upper section of 10 Lense, 120,000 cfm ventilates the mining of the upper sections of all other levels. Individual mining faces continue to be ventilated using 75 HP to 250 HP fans and 1.2 m ventilation duct. Several larger fans (3 – 300 HP fans) will be needed to provide air to the bottom of the mine as development continues.

With the increase in mining rate from 3,000 to 4,500 tpd several new areas are being brought into production. As the footprint of the mine expands, the ventilation system will also require expansion to allow fresh air to be delivered to active mining areas.

Refer to Figure 16-10 for a ventilation longsection of Lalor mine.

FIGURE 16-10: LALOR MINE VENTILATION PLAN



16.13.4 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 feeds power to the Hudbay owned main distribution substation consisting of two (2) 115-25 kV 24 MVA transformers. Substation is completely equipped with an E-house complete with 4 GE Powervac circuit breakers and Tie breaker. From there power is routed as follows:

1. **<u>Breaker 52-C1 – Provides power to the following areas</u>**

   a) The Chisel North mine site substation provides power to the:

      a. Surface ore crusher

      b. Photo Lake pump house

      c. Office buildings

      d. 4160V power to the upper part of the Chisel North mine ramp

   b) The Lalor mine waste water treatment plant area located 8 km from Lalor mine shaft. The area is equipped with:

a. 7.5 MVA Interstate 25-6.9 kV transformer that feeds the waste water treatment plant and Chisel Lake fresh water pump house.

b. 500 KVA 25-0.6 kV transformer for power line technician's workshop.

c) Chisel North mine downcast fan substation. This substation is equipped with

a. 5.5 MVA 25-4.16 kV transformer that provides power to;

i. 600 HP 4160V Alphair Model 10150 AMF550 downcast fan that supply 350,000 cfm heated mine air to the Chisel North mine underground workings.

ii. 4.16-0.6 kV Portable mine power centers in the lower part of the old Chisel North mine.

b. 5 MVA 25-13.8 kV transformer that provides power to the upper part of the Lalor mine.

d) Lalor mine ramp downcast fan site 750 KVA 25-0.6 kV Transformer for the Lalor mine ramp 400 HP Alphair 8400 VAX 3150 Jetstream downcast fan. Fan supply 240,000 cfm heated air to the upper part of the Lalor mine ramp.

2. <u>Breaker 52-L1 – Lalor CCT1 – Provides power to the following areas</u>

a) Lalor mine site pump house and water treatment plant 750 KVA 25-0.6 kV Transformer

b) Lalor mine exhaust fans 6 MVA 25-4.16 kV substation providing power to two (2) Howden 2500 HP exhaust fans. Fans are VFD controlled utilizing Rockwell Powerflex 7000 drives.

c) Hoist house electrical room Bus 1. Bus 1 further distributes power to the following areas:

a. Mine underground Feeder A feeds

I. 910 m level Shaft Station 750 KVA 25-0.6 kV transformer for

i. Rock breakers

ii. Settling cones

II. 955 m level 3 MVA 25-4.16 kV transformer for 2 X 1250 HP 10 Stage Mather and Platt dewatering pumps.

III. 835 m level 7.5 MVA 25-13.8 kV Underground distribution transformer. – Currently under construction.

b. Mill Feeder A (Future)

c. Davey Markham production hoist 3MVA 25-0.6 kV transformer for VFD drives.

d. Production Hoist 500 KVA 25-0.6 kV MCC.

e. Hoist house PDC 2.5 MVA 25-0.6 kV transformer. PDC provides to 3 X GA250 1477 cfm, 120 psi Atlas Copco Compressors and Surface Warehouse/Mechanical shop.

f. Davey Markham – Service hoist 750 KVA 25-0.6 kV MCC

g. Davey Markham – Auxiliary hoist

Note: Hoist House electrical room switchgear arrangement is equipped with a 1200A Tie breaker between Bus 1 and 2.

3. **Breaker 52-L2 – Lalor CCT2 – Provides power to the following areas:**

a) Hoist house electrical room Bus 2. Bus 2 further distributes power to the following areas;

a. Mine UG Feeder B feeds

I. 910 m level 7.5 MVA 25-13.8 kV mine underground distribution transformer.

II. 955 m level 750 KVA 26-0.6 kV Conveyor belt and loading pocket arrangement.

b. Head frame Complex 750 KVA 25-0.6 kV MCC

c. Mill Feeder B (Future)

d. Service hoist 2 MVA 25-0.6 kV transformer for VFD Drives.

e. 1.5 MVA 25-0.6 kV transformer for the 2 X Alphair 1015 AMF 5500 Intake fans that supply heated mine air down the Lalor mine shaft.

f. Lalor mine Office/Dry Unit 4 MVA 25-0.6 kV substation.

Lalor mine pump house and treatment plant is further backed up with a Cummins 350 kW generator that will supply enough power to keep Freshwater, waste water and fire pumps running in case of an emergency.

The booster pump station located 3.5 km from the Lalor mine shaft is also equipped with a Cummins 350 kW generator that will supply essential power to:

a) Process water pumps that will supply process water to Lalor mine pump house
b) Waste water pumps that will pump Lalor mine effluent water to the Chisel Pit for further waste water treatment.

The Lalor mine Auxiliary hoist is also backed up with a Cummins 1000 KVA generator that will supply sufficient power to the auxiliary hoist to move employees up the shaft in the event of an emergency / power failure.

All generators are equipped with automatic transfer switch that will in the event of a power failure transfer to generator power and vice versa.

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 up:

a) Auxiliary ventilation fans,
b) Auxiliary pumps, and
c) Mobile electric Jumbo drills and Bolters

Lalor mine underground mine workings currently consists of:

a) 20 X 1 MVA 13.8-0.6 kV mine power centers (aka. portable sleds) and,
b) 16 X 0.75 MVA 13.8-0.6 kV mine power centers

16.14 Workforce

Lalor mine is operated on a continuous cycle. The majority of operations and maintenance personnel work 11.5 hour shifts on a 5-5-4 day cycle or a 7-7 day cycle. Operations support, technical and administrative personnel work 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 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	180
Supervision and Administration	47
Health and Safety	4
Mine Maintenance	74
Mine Technical	34
Total Lalor mine	**339**

16.15 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 operations including, but not limited to:

a) Positive Attitude Safety System (PASS) safety program

b) Health monitoring programs (hearing and lung)

c) Dust monitoring

d) Ongoing water and environmental monitoring

e) Personal Protective Equipment (i.e. reflective outerwear, eye protection, hearing protection, respirators)

f) Task analysis and job procedures

16.15.1 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 and a backup oxygen generator, potable water, stretcher kit and first aid supplies, and supplies to seal off the bulkheads.

In new development where it is impractical to excavate a refuge station, portable refuge stations will be used.

16.15.2 Second Egress

Underground mines require a second means of egress. 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.

In the case that the production shaft is not usable, the second egress from the mine is the main ramp to surface at Chisel North mine.

16.16 MINING METHOD OPPORTUNITIES

The Lalor mine is considering different opportunities to improve mining efficiencies:

- Autonomous operation of LHDs are currently being trialed from surface by tele-remote with changes to standard designs to allow isolation of autonomous areas and buffer storage (transfer raises) for in between shift mucking
- A main ore pass from 755 m level to 910 m level is planned for 2017 to reduce trucking time from the upper levels of the mine
- Alternative truck loading systems are being investigated as an alternative to LHD loading, and
- Stoping block design changes are being considered to allow box hole primary mucking and circle route loading of trucks

17 RECOVERY METHODS

17.1 Introduction

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 will be expanded from the current 3,000 tpd rate to 4,500 tpd and the mill will operate 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 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 will be 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.

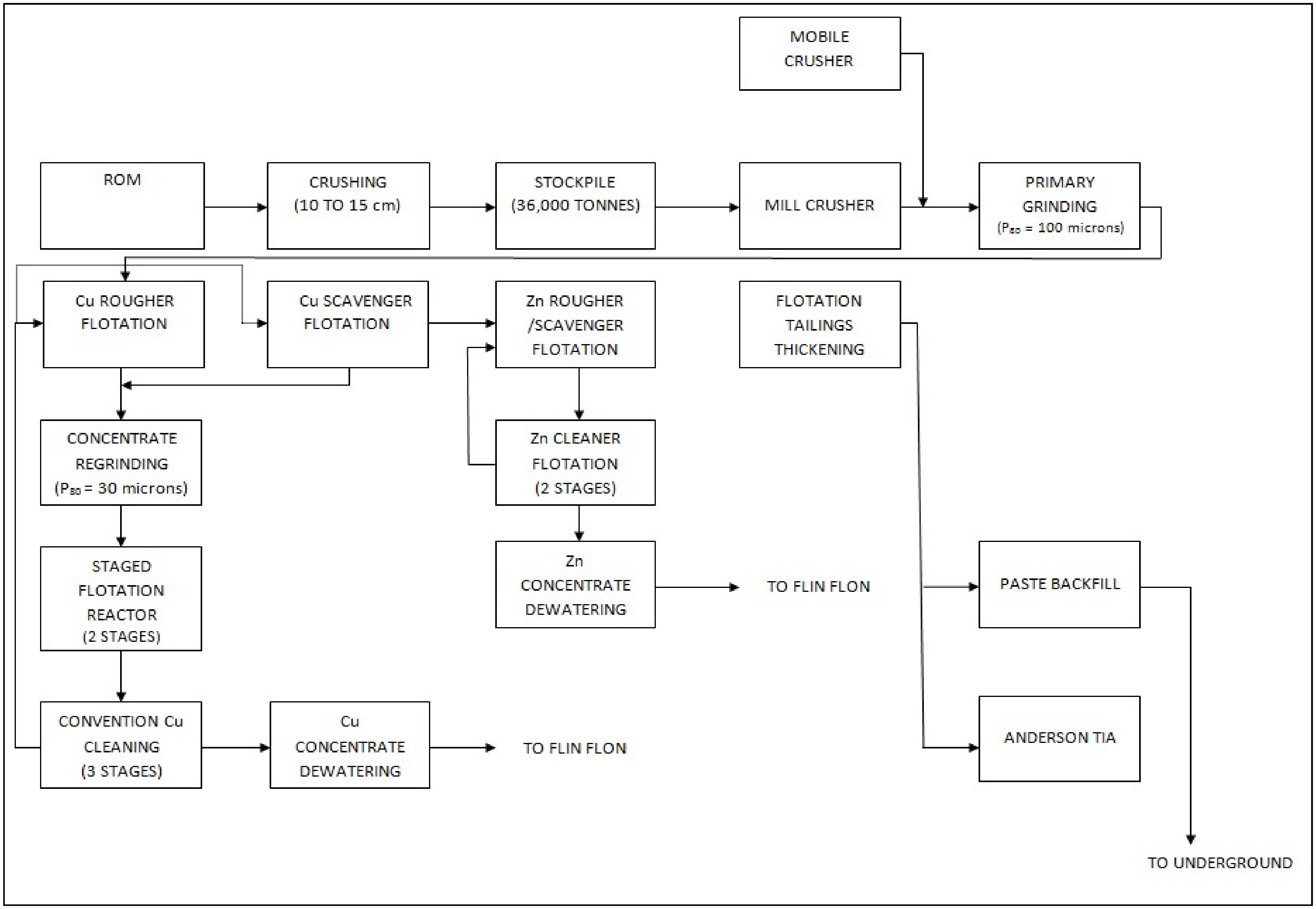
17.2 Stall Concentrator

A simplified block flow diagram for the planned Lalor concentrator is shown in Figure 17-1. Run of mine ore as large as 0.55 m in one dimension is withdrawn from the head frame ore bin by an apron feeder and transported to a crushing plant located at the Chisel North mine, which is ran by a third party contractor.

The contractor crushing plant reduces the ore to a range of 10 to 15 cm and transports the crushed ore to the Stall concentrator coarse ore bins using belly-dump trucks or to a stockpile located at the Stall concentrator using regular dump-trucks. From the stockpile, belly-dump trucks are loaded using a loader and directed to the coarse ore bin.

A stockpile of 36,000 tonnes, equivalent to 8 days production, is required to blend high-grade zinc to ensure a more consistent zinc feed of less than 13%. Ore is reclaimed from the bins via one of two 1220 mm x 1829 mm Syntron vibrating feeders which discharge onto a 1219 mm wide conveyor No 3. This conveyor discharges directly into a 30 x 48 Hewitt Robins jaw crusher 110 kW (150 HP) installed drive and the discharge is combined with the secondary Symons cone discharge to feed a 2440 mm x 6100 mm double deck vibrating screen which has a 39 mm top deck and 19 mm bottom deck.

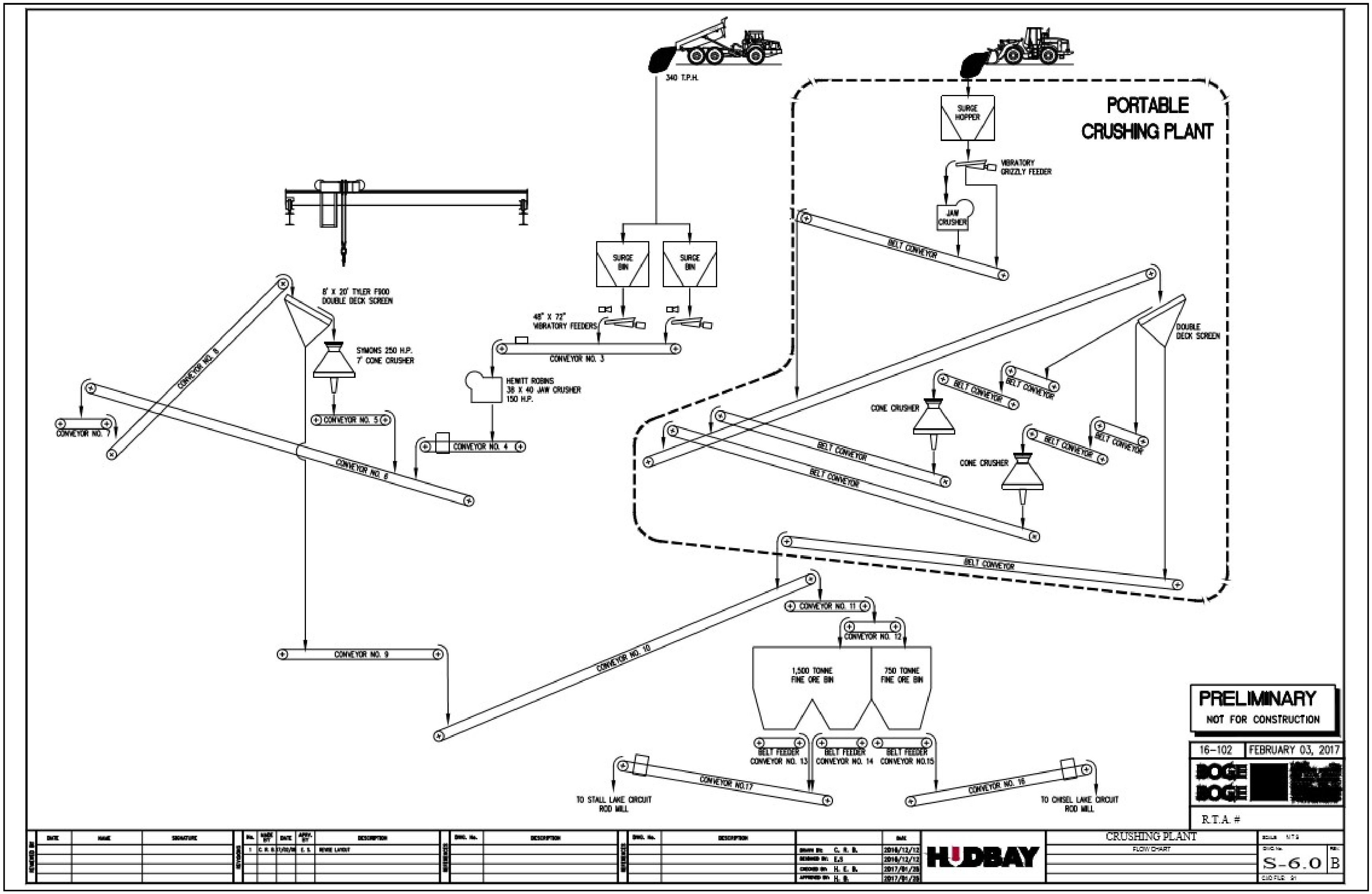
FIGURE 17-1: LALOR CONCENTRATOR SIMPLIFIED BLOCK FLOW DIAGRAM



The undersize from the bottom deck is the final product from the crusher circuit which is conveyed to the fine ore bins (FOB) while the oversize of the two decks is combined to feed a 186 kW (250 HP) 7 ft Symons cone crusher. The product out of the Symons at 19 mm is recirculated back to the screen deck in a closed loop.

For the expansion the crusher will retain all of the existing crushing and screening equipment as mentioned above, except for minor modifications (i.e screen openings, minor conveyor upgrades), as shown in Figure 17-2. If the crushing plant falls short of the target production of 4,500 tpd, a standby outdoor portable crusher will be used to make up the shortfall. This portable crusher will produce a particle size of 19 mm and envisions feeding conveyor No. 10 and eventually the FOB.

FIGURE 17-2: STALL CONCENTRATOR CRUSHING PROCESS FLOW DIAGRAM



From the FOB, the crushed ore is reclaimed via three 1370 mm wide variable speed belt feeders; two feed the larger Stall rod mill while the single one feeds the smaller Chisel rod mill. The Chisel grinding circuit will continue operating independent of the Stall circuit, providing the ability to continue operating the plant if one of the lines is down for maintenance. It also permits the use of the existing feed bins and feeders without modification.

17.2.1 Chisel Grinding Circuit

The ore is reclaimed from the FOB via a 1370 mm wide variable speed belt and discharges onto the rod mill feed conveyor which feeds into the 2.13 m x 3.05 m rod mill with 150 kW (200 HP) installed. The rod mill discharges into the cyclone feed pump box where it is combined with the ball mill discharge and with dilution water. The variable speed cyclone feed pump discharges into the cyclone pack fitted with two 500 mm cyclones, one operating and one spare. The cyclone underflow returns to the 3.20 m x 3.96 m ball mill with 600 kW (800 HP) installed. The cyclone overflow flows to the combined cyclone overflow pump which feeds the copper flotation feed trash screen. Flotation reagent 3418A is added to the rod mill and lime is added to the ball mill to control the pH of the feed to the flotation circuit.

17.2.2 Stall Grinding Circuit

The ore is reclaimed from the FOB via two 1370 mm wide variable speed belts and discharge onto the rod mill feed conveyor which feeds into the 3.20 m x 4.88 m rod mill with 600 kW (800 HP) installed. The rod mill discharges into the cyclone feed pump box where it is combined with the ball mill discharge and with dilution water. The variable speed cyclone feed pump discharges into the cyclone pack fitted with two 500 mm cyclones, one operating and one spare. The cyclone underflow returns to the 3.81 m x 5.49 m ball mill with 1,200 kW (1,600 HP) installed. The cyclone overflow flows to the two tertiary high frequency each with five decks 1.22 m x 1.52 m with a 150 micron screen cloth. The screen oversize is discharged into the 2.44 m x 3.66 m ball mill with 300 kW (400 HP) installed. Water is added to the screen oversize to optimize the grinding mill density. The screen undersize flows to the combined cyclone overflow pump which feeds the copper flotation feed trash screen. Flotation reagent 3418A is added to the rod mill and lime is added to the ball mills to control the pH of the feed to the flotation circuit.

17.2.3 Flotation Process

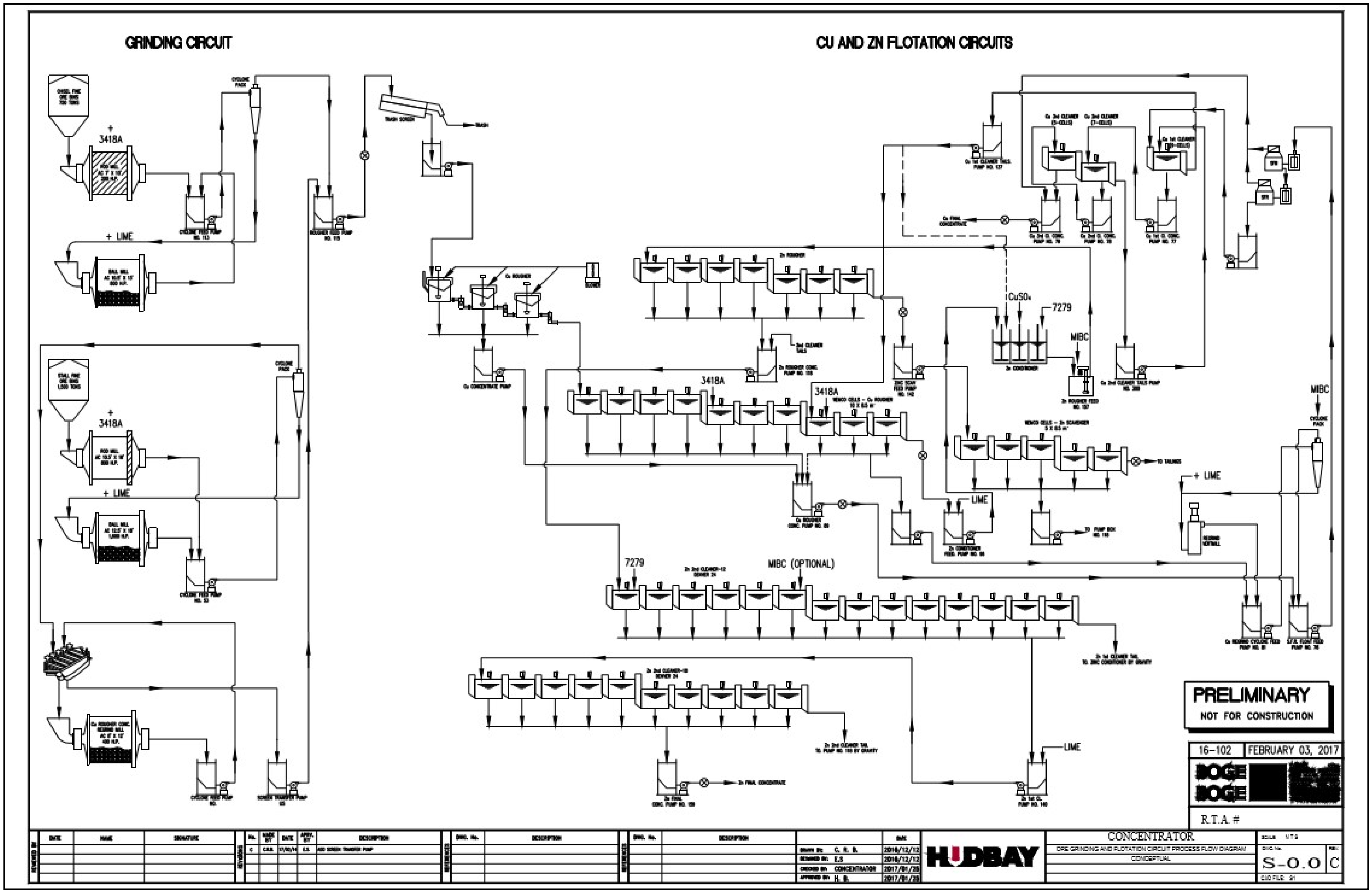
The combined primary grinding circuit cyclone overflow and screen undersize are pumped to a bank of new 20 m^3 copper rougher flotation cells, which is part of the concentrator expansion, where it is combined with the copper first cleaner tailings, as shown in Figure 17-3. The tailings from the third cell flows by gravity to the existing copper rougher flotation cells.

The existing copper roughers consist of ten Wemco 8.5 m^3 cells. The concentrate from the new 20 m^3 copper rougher flotation cells is combined with the concentrate from the first four of the existing Wemco rougher flotation cells and is then pumped to the feed of the Woodgrove SFR flotation cells.

The concentrate from the remaining copper scavenger flotation cells is pumped to the copper regrind mill circuit which consists of a VTM50 mill with 37 W (50 HP) installed. The mill is operated in closed circuit with a cyclopac with 250 mm cyclones. Methyl isobutyl carbinol (MIBC) frother will be added to stabilize the froth. The regrind cyclone overflow is pumped to the two new Woodgrove SFR flotation cells, which is part of the concentrator expansion. The concentrate from these cells, containing 50% of the final copper concentrate is discharged directly to the final copper concentrate pump box. The tailings of the SFR flotation cells are pumped to the existing copper first cleaners where they are combined with the copper second cleaner tailings. The existing first cleaners consist of nine Denver DR24 1.4 m^3 cells. The first cleaner tailings are returned to the feed end of the copper rougher scavenger 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 second cleaners consist of seven DR24 cells, with the concentrate going to the third cleaners. The third cleaner concentrate, produced from five DR24 flotation cells is combined with the SFR concentrate and pumped to the copper concentrate thickener.

The zinc flotation feed consists of the copper rougher scavenger flotation tailings and the zinc first cleaner tailings. It is first conditioned with copper sulphate, 7279 collector and MIBC frother and is then pumped to the seven existing Wemco 8.5 m^3 flotation cells zinc rougher flotation cells. The rougher tailings are pumped to the existing zinc scavengers consisting of five Wemco 8.5 m^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 combined with the zinc rougher concentrate and the zinc second cleaner tailings which are then pumped to the existing fourteen DR24 flotation cells. The first cleaner concentrate is pumped to the second cleaners which consist of the existing ten DR24 flotation cells. The second cleaner concentrate is the final zinc concentrate and is pumped to the zinc concentrate thickener.

FIGURE 17-3: STALL CONCENTRATOR FLOTATION PROCESS FLOW DIAGRAM



17.2.4 Concentrate Dewatering

Flocculated copper concentrate will be pumped to thickeners, two 3.4 m existing and one new one 4.6 m, as part of the concentrator expansion, as shown in Figure 17-4. Thickener overflow will be pumped to the copper thickener overflow pumpbox. Underflow, at a target density of 65% solids, will be pumped to an existing agitated stock tank.

Thickened copper concentrate will be further dewatered to approximately 9% moisture on a pressure filter, as part of the concentrator expansion. Filtrate will be recycled to the copper concentrate thickener to prevent the loss of fine solids and reuse the water. Filter cake will be conveyor fed and gravity dropped to the concentrate shed.

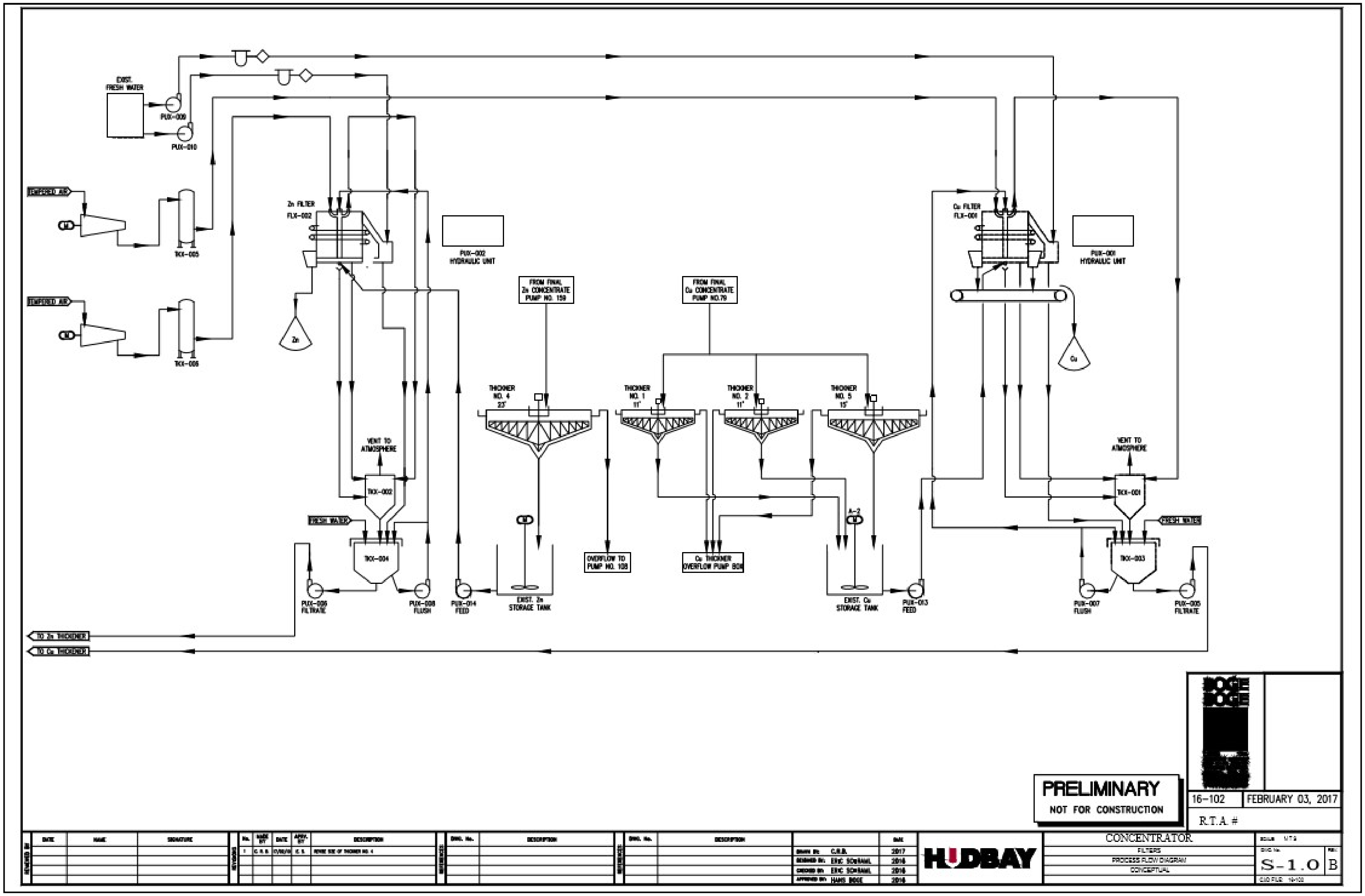
Flocculated zinc concentrate will be pumped to a 7.0 m new thickener, as part of the concentrator expansion. The overflow will be recycled to pump No. 108. The underflow, at a target density of 65% solids, will be pumped to an existing agitated stock tank.

Thickened zinc concentrate will be further dewatered to approximately 9% moisture using a pressure filter. Filtrate will be recycled to the zinc concentrate thickener to prevent the loss of fine solids and reuse the water. Filter cake will be gravity fed to the concentrate shed.

There will be no changes to the concentrate shed area. The shed building 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 trucks for transport to the Hudbay concentrate handling facilities in Flin Flon. Each truck is weighed on a truck scale located in Flin Flon.

FIGURE 17-4: STALL CONCENTRATOR DEWATERING PROCESS FLOW DIAGRAM



17.2.5 Tailings

Flotation tailings, from the tailings pump box, can be pumped to the paste plant or to the Anderson TIA, depending on the demand for paste. In the scenario of pumping to the paste plant, the tailings will be sent to a booster station pump box before reaching the paste plant thickener. In the scenario when paste is not required in the mine, the thickened tailings will be diverted at the splitter box to the tailings pump box and pumped to the Anderson TIA.

In terms of utilities, there will be modifications to the compressed air and electrical requirements that are related to the installation of new equipment. Regarding the electrical power distribution; secondary transformation capacity will be added, new power distribution centers will be established, and centralized process control points for new systems. In terms of mill compressed air systems, sufficient, dedicated compressed air capacity will be added to meet the demands of the filter presses.

17.2.6 Water

The Stall Concentrator ore-grade mineral extraction process utilizes two sources of water – fresh water and reclaim water (or recycled water).

- **Fresh Water**: approximately 25% of the water usage is fresh water withdrawn from Snow Lake. 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. The fresh water pipe-line run is approximately 7 km from Snow Lake to the Stall concentrator. It is fed into a storage tank located in the concentrator and then distributed to the process.
- **Reclaim Water**: approximately 75% of the water usage is reclaim water withdrawn from the Anderson TIA; also called recycled water, this water is deposited into the Anderson TIA as final process tailings (approximately 30% solids, 70% water). The water is then recycled from the TIA for re-use in the concentrator and utilized for process density control, launder sprays, hosing and other processes or activities that do not require high quality water.

The reclaim water pipe-line run is approximately 5km from the Anderson TIA to the Stall concentrator. It is fed into a storage tank located in the concentrator and then distributed to the process.

The water balance is summarized in Table 17-1.

TABLE 17-1: STALL MILL WATER BALANCE DATA FOR 2016

Month	Operating Hours	Down Time Hours	Reclaim (m^3)	Fresh (m^3)	Total Water Use (m^3)	% Reclaim	% Fresh Water
Jan	588.99	155.01	251236.38	83969.90	335206.28	74.95	25.05

Month	Operating Hours	Down Time Hours	Reclaim (m^3)	Fresh (m^3)	Total Water Use (m^3)	% Reclaim	% Fresh Water
Feb	604.11	91.89	233411.67	71326.09	304737.76	76.59	23.41
Mar	626.07	117.93	261091.93	83196.46	344288.38	75.84	24.16
Apr	682.75	37.25	256293.89	85732.02	342025.91	74.93	25.07
May	687.34	56.66	260402.83	85917.26	346320.09	75.19	24.81
Jun	544.58	175.42	203135.96	81736.84	284872.80	71.31	28.69
Jul	682.09	61.91	219229.19	84953.77	304182.96	72.07	27.93
Aug	628.7	115.3	218134.34	78646.05	296780.39	73.50	26.50
Sep	671.75	48.25	218331.58	78871.64	297203.22	73.46	26.54
Oct	673.74	70.26	216471.97	80180.81	296652.78	72.97	27.03
Nov	644.43	75.57	252413.37	66937.53	319350.90	79.04	20.96
Dec	582.16	161.84	204013.93	67165.48	271179.41	75.23	24.77
Total	**7616.71**	**1167.29**	**2794167.02**	**948633.86**	**3742800.88**	**74.65**	**25.35**

17.2.7 Operating Costs

The 2016 cost per tonne milled at Stall concentrator was $23.62 based on milling 1,089,530 tonnes (average 2,985 tpd). The expansion plan at Stall concentrator from the current tonnage of approximately 3,000 tpd to 4,500 tpd in 2018, increases the daily tonnage by 50% and a yearly expenditure increase of 27.5% is envisaged.

The expansion costs per tonne were further dissected into labour, power, operating supplies, maintenance supplies, outside services and G&A. The estimated percentage of the total cost per tonne by allocated area is shown in Table 17-2.

The methodology assumes there will be a cost increase corresponding with the expansion primarily in power, operating and maintenance supplies. However the higher tonnage will offset cost incrementally, resulting in a $20.09 cost per tonne based on 4,500 tpd.

TABLE 17-2: EXPANSION ESTIMATE COST PER TONNE

Allocation	2016 Unit Operating at 3,000 tpd		Tonnes per day increase (%)	Expansion Unit Operating at 4,500 tpd	
	$/tonne	% of Cost		$/tonne	% of Cost
Labour	$7.56	32.0%	50%	$5.54	27.6%
Power	$1.42	6.0%		$1.65	8.2%
Operating Supplies	$7.20	30.5%		$7.20	35.9%
Maintenance Supplies	$1.30	5.5%		$1.39	6.9%
Services	$4.72	20.0%		$3.31	16.5%
GandA	$1.42	6.0%		$0.99	4.9%
Total	$23.62	100.0%	50%	$20.09	100.0%

17.2.8 Expansion Schedule

In terms of schedule of the 4,500 tpd expansion project, it entails primarily engineering starting in the first quarter of 2017, procurement of long lead items in second quarter of 2017, construction phase in third quarter of 2017 and tie-ins/start-up and commissioning in second quarter of 2018.

18 PROJECT INFRASTRUCTURE

18.1 Lalor Infrastructure

Lalor mine is designed to hoist 6,000 tpd combined ore and waste. Primary access to the mine is a concrete-lined 6.9 m diameter production shaft with a secondary ramp access from the surface through the Chisel North mine. Ore is hoisted to the surface and trucked to the Chisel North site where it is crushed, then hauled to the Stall concentrator for processing into two concentrates (zinc and copper).

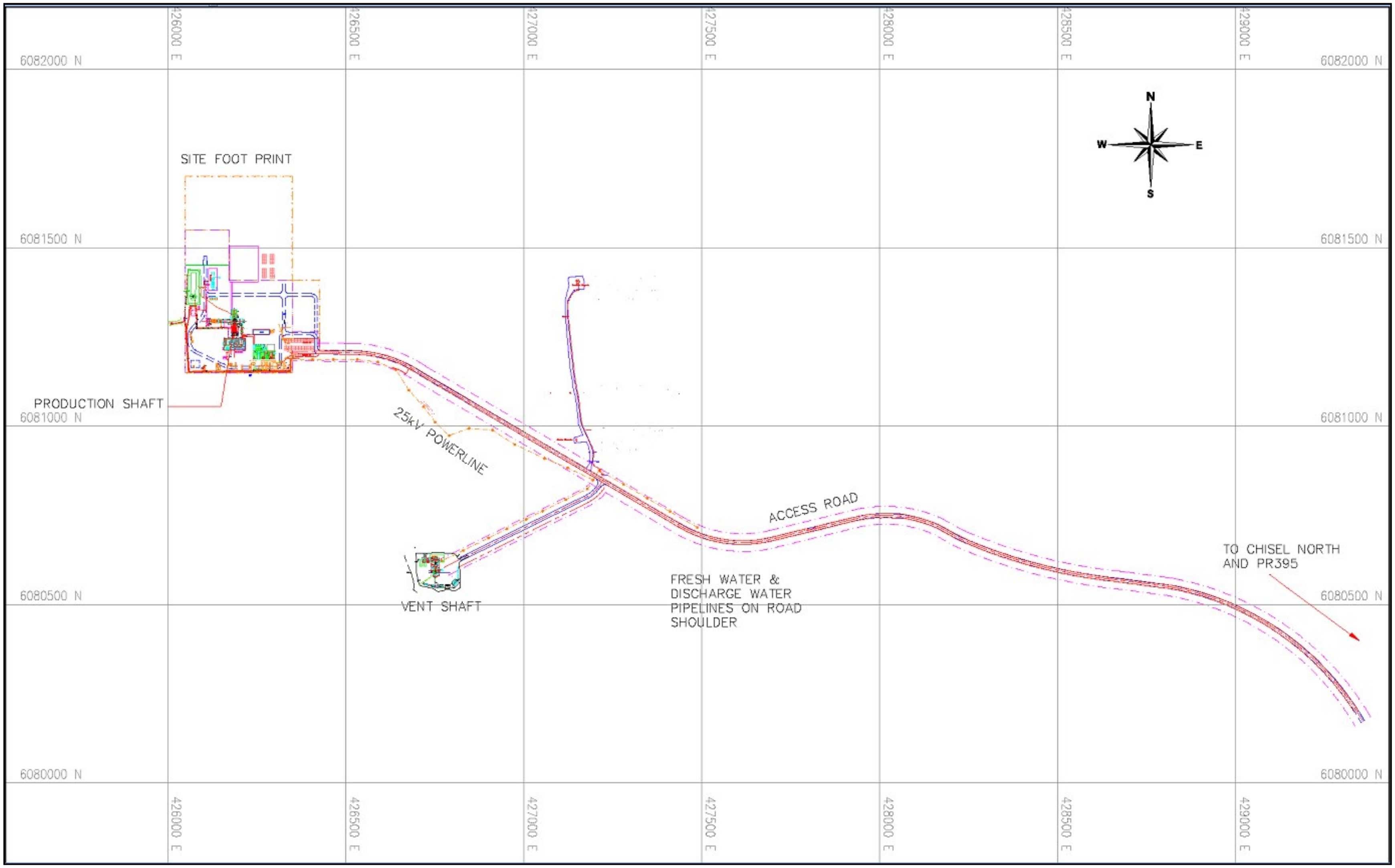
Lalor is 16 km by road from the Town of Snow Lake, Manitoba. General area infrastructure includes provincial roads and 115 kV Manitoba Hydro grid power to within four km of Lalor, and Manitoba Telecom land line and cellular phone service. The Town of Snow Lake is a full-service community with available housing, hospital, police, fire department, potable water system, restaurants and stores. The community is serviced by a 914 m gravel airstrip to provide emergency medical evacuation.

Lalor is located 3.5 km from the Hudbay Chisel North mine. Chisel North infrastructure includes 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, plus mine buildings including offices and a change house.

The permitted Hudbay Anderson TIA, located approximately 12 km from Lalor is used for tailings disposal.

A site drawing of the Lalor access road and services is shown in Figure 18-1.

FIGURE 18-1: SITE ACCESS ROAD AND SERVICES



As of March 2017, offsite infrastructure for the mine operation included:

- 198 person camp in the Town of Snow Lake to house out of town personnel
- 3.5 km gravel access road connecting Provincial Road 395 to the mine site. The road was constructed to Manitoba Class B Feeder road standard.
- Two 24 MVA - 115 kV to 25 kV power substations located on the Chisel North site. These substations provide power for surface and underground mining activities.
- Four km of 25 kV overhead power lines from Chisel North to Lalor
- Process water is pumped 4 km from Chisel Lake through to the booster pump station at Chisel North where it is then pumped the remaining 3.5 km to Lalor
- The Chisel North complex, which is used for the diamond drilling core processing facility, shop to maintain the surface equipment fleet and offices for the project group
- Crushing of the Lalor ore, which is done at the Chisel North site with a maximum total stock pile capacity of 15,000 tonnes
- Booster pump station at Chisel North with holding tanks and pumps for process and discharge water. Equipped with a backup generator (350 kW).
- Two downcast fans, mine heaters (each with a 30,000 US gallon propane tank); the Chisel North downcast (600 hp and 350,000 cfm) and the Lalor ramp downcast (400 hp and 250,000 cfm)
- Discharge water from the Lalor site is pumped 3.5 km to the booster pump station at the Chisel North site where it is pumped the remaining 3.5 km to the settling ponds by the Chisel Pit
- A high density sludge process acidic water treatment plant that can treat up to 2,500 gpm prior to being discharged to the environment

As of March 2017, onsite surface infrastructure includes:

- Office/change house complex with dry space for 341 personnel
- Hoisthouse containing:
  - 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 which also contains:
  - The utility hoist (Davy Markham, single drum, 314 kW)

  - Bin house (capacity 850 tonnes of ore)
  - External bunker (capacity 1,000 tonnes of ore)
  - Twin 250 hp downcast fans (313,000 cfm each) and mine heater

- Two 30,000 US gallon propane tanks
- Main pump station includes holding tanks (discharge water, process water and potable), PAL water system and pumps for discharge, potable, process and fire water
- Bio Disk Sewage treatment plant (is a natural biological process based on the principle of rotating biological contactor) for up to 38 m^3/day
- Fuel tanks and pumps for diesel and gas.
- Two backup generators one for the utility hoist (1,000 kW) and one for the main pump station (350 kW).
- Temporary offices for health safety, training and mine rescue.
- Temporary change house for contractors on site.
- Temporary trailer for onsite contractor.
- Warehouse/shop
- Vent shaft and two exhaust fans (2,500 hp and 575,000 cfm each)

As of March 2017, underground infrastructure includes:

- Main Production shaft 6.9 m diameter concrete lined with five compartments
  - Two - 16 tonne skips
  - One double deck cage (50 people per deck)
  - Counter weight
  - Utility hoist (6 person cage)
  - Three main shaft stations at 835 m, 910 m and 955 m levels
- Lateral development consists of 6 m x 5 m ramps and level development
- Secondary egress consists of a ramp access from surface to the 810 m level where it joins the rest of the Lalor ramp system. Total approximately distance of 6.0 km.
- Power Distribution consists of:
  - 25 kV power lines down the shaft
  - 7.5 MVA 25 kV to 13.8 kV transformer to the 910 mL shaft station
  - Primary distribution throughout the mine is 13.8 kV with transformers to 600 V for local distribution
- Compressed air and process water are piped throughout the mine from surface

- Underground wireless radio communication throughout the mine is provided by a Leakey Feeder system
- Fiber-optic backbone for data and video
- Ore handling system consists of two rock breakers and bins (1,400 tonnes each) on 910 mL feeding chutes and conveyor system on 955 mL supplying the ore to the skips.
- Mobile Maintenance shop is located in the Chisel North underground workings
- Discharge system consists of a series of drain holes and sumps with submersible pumps that feed the top of the two settling cones on 910 mL. The over flow from the cones goes to the main clean water sumps on 955 mL where the water is then pumped to surface by one 1,250 hp 10 stage Mather Platt (740 gpm). There is a second installed spare.

As of March 2017, the schedule for additional permanent infrastructure is:

- Paste plant is scheduled to start construction in 2017 with completion in early 2018
- Ore pass and chute from 910 mL to 755 mL to be operational in the second half of 2017
- A second 7.5 MVA 25 kV to 13.8 kV transformer is to be installed at 835 mL shaft station in 2017

18.2 Lalor Ore Handling Improvements

Based on a review of Lalor's ore handling circuit by Stantec in early 2017 Hudbay is planning capital improvements in 2017. These improvements will ensure Lalor is able to maintain a steady 4,500 tpd of production through the ore circuit.

Mine personnel have identified that maintenance, particularly repair and replacement of liners in the ore circuit is challenging and to reduce potential hang-ups in the system, as a result the following activities are envisaged:

- Construct muck deadbed in the skip dump in the headframe
- Modifications to transfer car at loading pocket to replace diverter liners with deadbeds
- Install tramp metal grapples at current 910 m level rockbreakers to remove the tramp metal before entering the ore circuit could help to reduce hang-ups, and
- Measuring flask chute angle shallowing to reduce the energy of the muck striking the wall of the skip and to further reducing maintenance

18.3 Paste Plant

The Lalor paste plant project was approved in February 2017 and is critical for the sustainability of the mine production plan. The paste plant will be located northeast of the existing headframe complex and delivery capacity of the paste is 165 tph solids (tails) or 93 m^3/hr paste. 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 not to hoist waste from underground the combined paste/waste backfilling capacity is approximately 6,000 tpd. The paste plant will be capable of varying the binder content in the paste to provide flexibility in the strength gain of the paste where higher and early strength may be required depending on mining method.

Tails that are currently pumped from the Stall concentrator to the Anderson TIA will be diverted to the Anderson booster pump station. Capacity of the pumping station will range from 110 to 130 tph to allow for some variation in the output of tailings from the concentrator. The tailings will be directed into the Anderson TIA when not required for the paste plant.

Two pipelines will be 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 will be 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 location.

Paste will be delivered underground via one of two – nominal 8 inch diameter, cased boreholes from surface to the 780 mL of the Lalor mine. Only one borehole is required during normal operation, with the second borehole 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 will transfer the paste from the base of the discharge hopper to the required underground locations. The 780 mL will be the main distribution level to direct the paste to other levels above and below. Underground development is required to extend an existing drift on the 780 mL to intersect the surface boreholes and short cross-cuts on several levels for level to level boreholes. The majority of the underground distribution system will utilize existing drifts or planned future development.

18.4 Stall Concentrator

Hudbay operates the Stall concentrator approximately 16 km from Lalor. The mill is currently operating seven days per week at 3,000 tpd, processing ore from the Lalor mine. The mill has two circuits, with design capacities of 909 tpd and 2,182 tpd.

The concentrator has two flotation circuits producing a zinc concentrate and a copper concentrate. The tailings associated with the Lalor mine are deposited in the Anderson TIA.

Produced copper concentrate is currently hauled by 40 ton trucks to Flin Flon, where the concentrate is loaded onto gondola rail cars for market. Produced zinc concentrate is hauled by 40 ton trucks to Flin Flon and is processed at the Flin Flon Zinc Plant.

As of March 2017, the current Stall concentrator technical specification includes:

- 2 - 350 tonnes coarse ore bins
- 1 Hewitt Robins model W, 30 inch by 48 inch, 150 hp direct V-belt driven jaw crusher
- 1 Symons Standard 7 ft cone, 250 hp direct V-belt driven, direct coupling
- 1 Tyroc double deck screen – 8 ft by 20 ft, 2 inch on top deck, ½ inch x 2½ inch on bottom deck
- 1 Rod mill – AC – 7 ft diameter by 10 ft, 200 hp induction motor
- 1 Ball mill – AC – 10.6 ft diameter by 13ft, 800 hp induction motor
- 1 Rod mill – AC – 10.5 ft diameter by 16ft, 800 hp induction motor
- 1 Ball mill – AC – 12.5 ft diameter by 18ft, 1,600 hp synchronous motor
- 17 - 8.5 m^3 Wemco cells
- 1 Regrind mill – AC – 8 ft diameter by 12 ft, 400 hp induction motor
- 22 – Denver cells

18.5 Stall Concentrator Expansion

Engineering work is currently underway at the Stall concentrator as part of the expansion to 4,500tpd with construction slated in third quarter of 2017 and commissioning in third quarter of 2018. The expansion project will address the following areas (Boge & Boge, 2017):

- Crushing – Increase the ore handling capacity for the crusher house conveyors, and change the screen deck media to allow the crushing system to operate to its maximum capacity to produce 4500tpd at a P80 of 16mm from a coarse ore feed F80 of 150mm.
- Grinding – Increase the ore handling capacity of fine ore bin reclaim conveyors and mill feed conveyors. Reconfigure existing mills into two parallel grinding circuits (a smaller two stage (Chisel) and larger three stage (Stall)), and introduce high frequency screening in closed circuit with the third stage Stall ball mill to ensure product sorting by size rather than density to produce a blended flotation feed P80 of 100 µm at 4500tpd.
- Copper Flotation – Expanded capacity at the copper roughers with 3 new 20m3 tank cells, and expanded capacity of copper cleaners with addition of staged flotation reactors. There will be some pump capacity modifications to address the increased volumes through the circuit. Introduction of a regrind mill to reduce the rougher scavenger tails to target a P80 of 50µm prior to the copper cleaners to improve metal recovery.
- Zinc Flotation – The zinc flotation capacities predicted by the mine planned and head grades are similar to the existing operating conditions and no flotation capacity upgrade

requirements were identified. Some pump capacity modifications are required to address extraneous process streams.

- Thickening – Introduction of a large (8m) zinc thickener to address the existing and future limitations on zinc concentrate production, and add a new copper thickener (or modify the two existing 3.4m thickeners with de-aeration tanks and froth rings) to extend the existing thickeners' capacity to meet the increased copper concentrate loads.
- Dewatering – Replace the existing disc thickeners with pressure filters to increase concentrate production capacity to match the new throughput. This will improve copper concentrate from 19% moisture to 9% moisture, and will improve zinc moisture from 14% to 8% moisture.
- Electrical Power Distribution – Add secondary transformation capacity, establish new power distribution centers, and centralize process control points for new systems.
- Mill Compressed Air Systems – Add sufficient, dedicated compressed air capacity to meet the significant demands of the filter presses.
- Mill water distribution systems – Increase the mill distribution pump capacity to satisfy the increase water demand.

18.6 Anderson TIA

Anderson TIA is located in the Snow Lake area between the Stall concentrator and Lalor mine. The purpose of the Anderson TIA is environmental management (storage) of mine tailings produced in the Stall concentrator which processes ore from the Lalor operation.

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 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 construction of this expansion is anticipated to be complete prior to 2019.

19 MARKET STUDIES AND CONTRACTS

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. 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.

Lalor will produce a zinc concentrate and a copper concentrate with gold and silver credits. Zinc concentrates are trucked to Hudbay's operations in Flin Flon where they are processed into refined zinc and sold to customers in North America. The key long-term assumptions for the sale of Lalor's zinc metal and zinc concentrate are summarized in Table 19-1. This report assumes zinc concentrate will be processed at the Flin Flon zinc plant from 2017-2021 and after that time Lalor's zinc concentrate will be sold to third party refineries.

TABLE 19-1: KEY LONG-TERM ZINC METAL AND ZINC CONCENTRATE ASSUMPTIONS

	Units	LT Total / Average
Zinc Concentrate Grade	%	51%
Moisture Content of Zinc Concentrate	% H_2O	9%
Zinc Concentrate Base Treatment Charge	US$/ tonne concentrate	$200
Zinc Concentrate Metal Price Basis	US$ / tonne Zinc metal	$2,204.6
Zinc Concentrate Escalator	%	6%
Zinc Concentrate De-escalator	%	3%
Zinc Concentrate Payability	%	85%
Zinc Concentrate Minimum Deduction	%	8%
Zinc Concentrate Freight Cost	C$/wmt	$118
Freight Allowance/Capture	US$/ wmt concentrate	$40
Zinc Metal Premium	US$/lb	$0.07
Zinc Metal Distribution Cost	US$/lb	$0.055

The copper concentrate produced at Lalor is sent to copper smelters in North America by rail. The key assumptions for the sale of Lalor's copper concentrate are summarized in Table 19-2 below:

TABLE 19-2: KEY LONG-TERM COPPER CONCENTRATE ASSUMPTIONS

	Units	LT Total / Average
Copper Grade in Copper Concentrate	% Cu	21%
Moisture Content of Copper Concentrate	% H_2O	9%
Copper Concentrate Base Treatment Charge	US$ / dry tonne con	$80
Copper Refining Charge	US $ / lb Cu	$0.08
Silver Refining Charge	US $ / oz Ag	$0.50
Gold Refining Charge	US$ / oz Au	$5.00
Copper Concentrate Freight Cost	C$ / wet tonne con	$213
Copper Payability	%	96.5%
Copper Minimum Deduction	%	1%
Gold Payability	%	96%
Silver Payability	%	90%

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.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Planning

Commencing in 2007, AECOM carried out the environmental baseline investigations needed to conclude an environmental impact assessment for the Lalor project, including all necessary terrestrial and aquatic field studies. Much of the early baseline work was summarized in AECOM's Lalor Advanced Exploration Project Plan ("Lalor AEP"), which was submitted to and approved by Manitoba Mines Branch and the Lalor Mine Environment Act Licence (EAL) application submitted to and approved by Manitoba Sustainable Development. Baseline work was utilized in the Lalor Paste Plant Notice of Alteration (NoA) which was submitted to the Manitoba Sustainable Development in Q4 2016 and was approved in January 2017. AECOM has also conducted baseline work and studies which are summarized in the Anderson TIA expansion NoA submitted in Q3 2016 to Manitoba Sustainable Development for approval.

Hudbay is currently reviewing improvement plans for existing operations in the Snow Lake area. For each project associated with previously permitted sites, Hudbay will submit a NoA to Manitoba Sustainable Development for review and approval.

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, it is contemplated that no additional baseline studies are necessary for potential future improvement projects. There is no present indication that future approvals will not be obtained to meet potential future construction schedules.

20.2 Waste, Tailings Disposal and Water Management

There are no known environmental concerns which could adversely affect Hudbay's ability to mine ore from Lalor mine. Because of its location in close proximity to the existing facilities in the Snow Lake area, Lalor was able to utilize existing infrastructure, services, and previously disturbed land that is 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 NoA for the expansion of Anderson TIA was prepared by AECOM utilizing geotechnical tailings dam designs from Hudbay's Engineer of Record; BGC Engineering Inc. As detailed in the Environmental Assessment of the Proposed TIA Expansion submitted as a NoA, the entire volume of tailings from Lalor LOM was to be stored in Anderson TIA (AECOM, 2016). This conceptual design did not discount the volume of tails that could be used for paste backfill at Lalor. In order to de-risk the construction of the proposed dams the project was proposed in 3 separate stages with the first to occur prior to 2019 based on available storage in the TIA.

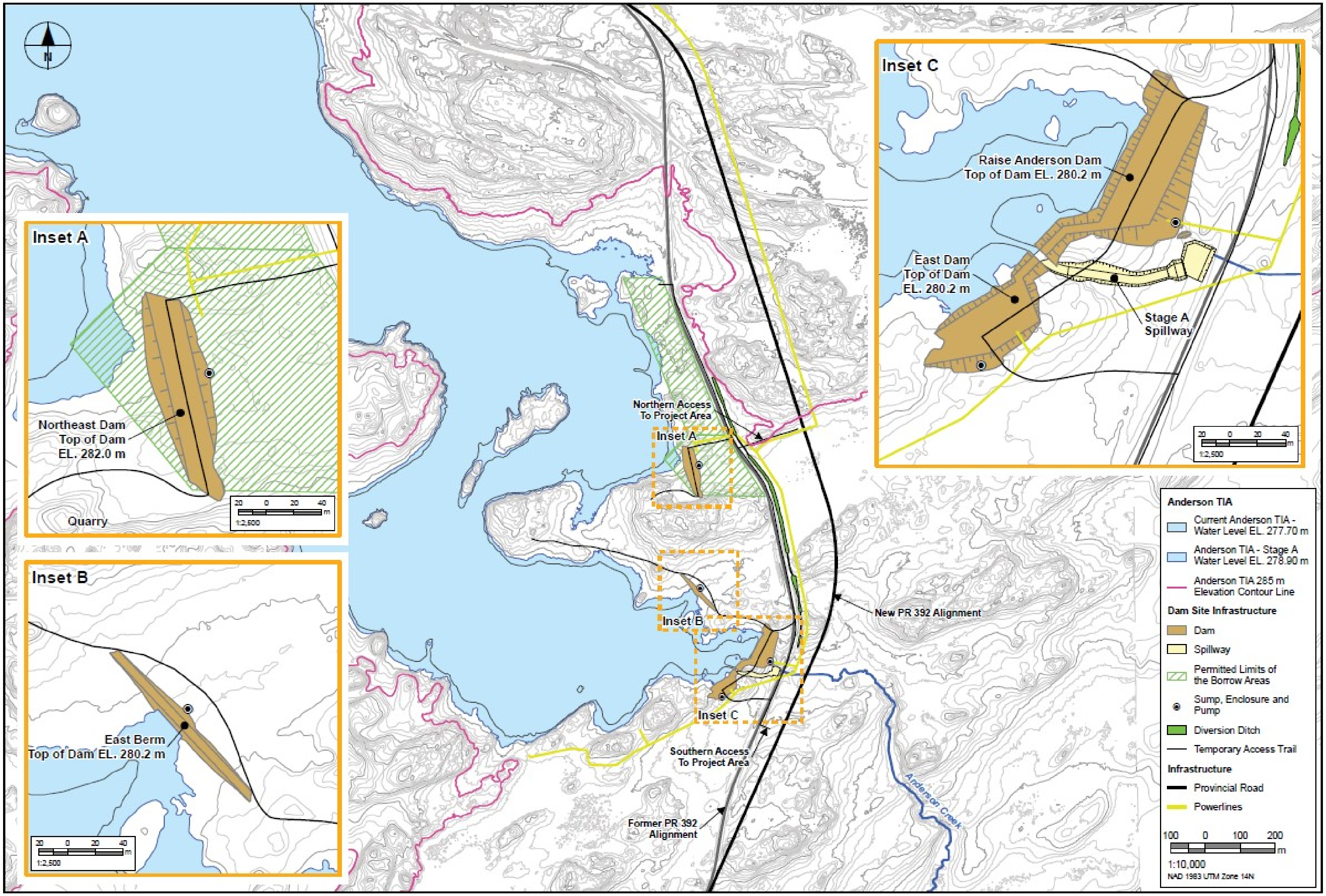
The initial stage of expansion (Stage A) was to provide sufficient storage for tailings production volumes such that future dam raises could be planned based on actual production rates. This would be possible as the actual tailings production could be measured against remaining volumes as ongoing bathymetry surveys are a requirement of the existing licence. Compared to the original design criteria, the predicted volume of tailings from Lalor operation is currently much less due to paste plant operation. This overall reduction in required tailings storage will result in adequate storage volumes with only Stage A being constructed. It is anticipated that Stage A will cost in the order of $7M in 2018. Stage A construction is detailed in Figure 20-1, which was submitted as part of the NoA.

20.3 Permitting Requirements

The existing Lalor mine EAL was obtained in the first quarter of 2014 and covers all facilities on the Lalor site, including sewage and mine wastewater treatment facilities and the pipelines which carry freshwater into the site and remove treated wastewater from it. The sources of freshwater and other facilities where treated wastewater are discharged to the environment are existing operating sites which are licensed provincially and regulated under the federal Metal Mines Effluent Regulation.

The main permits required for the Lalor operation are presently valid licenses and permits for the Lalor mine, Stall Concentrator, Anderson TIA, and New Britannia site. Applications for the Anderson TIA expansion NoA have been submitted for approval. Other upgrades and augmentation plans may require the submission of a NoA to an existing licensed operation but no new tailings impoundment area will be required. No federal permits are anticipated.

FIGURE 20-1: ANDERSON TIA STAGE 1A CONSTRUCTION PLAN (AECOM, 2016)



Presently the New Britannia site inclusive of the Birch Lake Tailings Management Facility (BLTMF) although currently in care and maintenance, has a current EAL and the seasonal discharge from BLTMF is regulated under the MMER. Hudbay is currently in the process of applying for a new water withdrawal licence for this site which is anticipated to be obtained before potential operational needs. Potential future use of the New Britannia site will require the submission of a NoA in order to process material from the Lalor mine.

The approval process and time requirements have been contemplated in regards to overall project milestones. There is no indication that the approvals will not be obtained within the project schedule.

20.4 Mineral Lease and Surface Lease

Prior to commercial production of ore at the Lalor mine, a mineral lease was applied for and obtained from the Manitoba Mines Branch. The mineral lease grants the holder the exclusive right to mine minerals within the lease area.

As the entire Paste Plant is on Hudbay held property and it utilizes existing infrastructure, there are no land tenure concerns or additional leases required. Approval for this project was received in January 2017.

In specific areas associated with proposed pipeline routes and future improvements to the existing Anderson TIA, surface leases will be required. Activities are currently underway to apply for and obtain the required surface leases. There is no indication that theses leases cannot be obtained in the time lines of the expansion project.

20.5 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 Ecodistrict and surrounding area. Snow Lake has a population of approximately 840 according to the 2006 data from Statistics Canada, with the majority of these residents employed at or supported by the mines located throughout the area. Many other Snow Lake residents are employed in the industries and services that support the region's mining operations.

Hudbay and AECOM have carried out public consultation, including meetings to inform local communities about the progress of development of the Lalor mine and expansion of Anderson TIA and environmental effects of these projects. Manitoba Sustainable Development has taken these meetings into account in the environmental licensing process.

The projects will continue to provide jobs for both Flin Flon and the Town of Snow Lake during construction of upgrades and continued operation of the mine. The additional feed from the mine will also help ensure the continued employment of Hudbay employees in the Flin Flon and Snow Lake areas. Since the economies of both communities are based on mining, opposition to the projects is seen as unlikely.

20.6 Aboriginal People and 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, other current operations, 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 First 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 continues attempts to initiate an information sharing process with MCCN and expects to be able to provide Manitoba regulators with all information necessary to support previous Crown consultation decisions made prior to approval of the Lalor mine EAL. No impact to current operations or delays in project schedule is anticipated.

20.7 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. Approximately 20 km south of the Lalor operation is Tramping Lake, which is the site 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.

20.8 Mine Closure Requirements and Plans

The Manitoba Mines and Minerals Act requires a closure plan and financial assurance for any advanced exploration or mining project. Manitoba accepted 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 and accepted financial assurance in the amount of $1.5 million. 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 submitted for approval in September 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.

NoA applications for the Lalor paste plant, expansion of the Anderson TIA, and upgrades to the New Britannia site also will require the submission of updated closure plans and financial assurance. It is expected that as a condition of Manitoba's approval, closure planning and financial assurance will be required within a year of final construction activities. Allowances for these applications are contemplated items as part of future years budgeting process.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

Capital and operating costs are estimated in constant 2017 Canadian dollars.

21.2 Capital Costs

The total development capital required to increase throughput at Lalor to the targeted 4,500 tpd is estimated to be C$117 million, as shown in Table 21-1, which includes approximately an 18% contingency. This capital is expected to be spent during 2017 and 2018.

TABLE 21-1: DEVELOPMENT CAPITAL COST SUMMARY

Development Capital	000 C$
Paste Backfill	67,786
Ore Handling Underground	3,250
Stall Mill Upgrades	45,870
Total Development Capital	**116,906**

The development capital costs were estimated internally by Hudbay with input from Golder Associates, Stantec Inc. and Boge & Boge Ltd.

The LOM sustaining capital costs are estimated to be C$220 million. The breakdown of the sustaining capital over the next 5 years and for the LOM is shown in Table 21-2.

TABLE 21-2: SUSTAINING CAPITAL COST SUMMARY

Sustaining Capital (000 C$)	2017	2018	2019	2020	2021		2017-LOM
Mine Capital and Development	17,927	18,381	13,442	13,849	10,635		86,771
Normal Capital	3,000	10,000	3,000	3,000	3,000		28,000
Replacement Equipment	11,629	14,739	13,390	11,004	9,776		91,518
Major Installations	3,640	5,924	1,461	982	835		13,803
Total Sustaining Capital	**36,196**	**49,044**	**31,293**	**28,835**	**24,246**		**220,092**

Normal capital includes C$7 million related to the expansion of the Anderson tailings facility and C$21 million related to the Stall concentrator. This sustaining capital estimate has made no consideration for the availability of used equipment from the 777 and Reed mines. When these mines eventually close, some equipment may be available for use at Lalor and may reduce the sustaining capital estimate above. No contingency has been included in the sustaining capital estimate.

Reclamation costs, salvage value and severance costs have not been considered in this report.

21.3 Operating Costs

Operating costs were developed by Hudbay based on a bottom-up approach and utilizing budget quotes from local suppliers, Manitoba operations experience, labor costs within the region and actual costs at Lalor.

The mine plus mill unit operating costs are estimated to be C$99.83/tonne mined over the LOM. The addition of cemented rock fill and paste backfill has increased the mine unit costs, but maximizes recovery of the mineral resource and results in lower capitalized costs than prior years due to less underground development. Table 21–3 summarizes the mine plus mill unit operating costs over the next 5 years and for the LOM.

TABLE 21-3: UNIT OPERATING COST SUMMARY

(C$/tonne mined)	2017	2018	2019	2020	2021		2017-LOM
Mine Development	26.53	15.07	14.87	13.64	13.98		17.12
Ore Extraction	20.28	27.19	33.60	35.59	34.75		30.50
Ore Removal	29.87	30.24	28.22	28.20	28.47		30.70
Total Mine Operating Costs	**76.67**	**72.49**	**76.68**	**77.43**	**77.20**		**78.32**
Mill Operating Costs[1]	**22.02**	**20.14**	**20.12**	**20.19**	**20.12**		**21.51**
Total Mine + Mill Operating Costs	**98.70**	**92.63**	**96.80**	**97.63**	**97.32**		**99.83**

[1] Milling costs include concentrate haulage to Flin Flon

The total C1 cash costs and sustaining cash costs (net of by-product credits) per pound of zinc over the LOM and over the next 5 years are shown in Table 21-4. C1 cash costs include on-site and off-site costs. Sustaining cash costs include C1 costs plus sustaining capital.

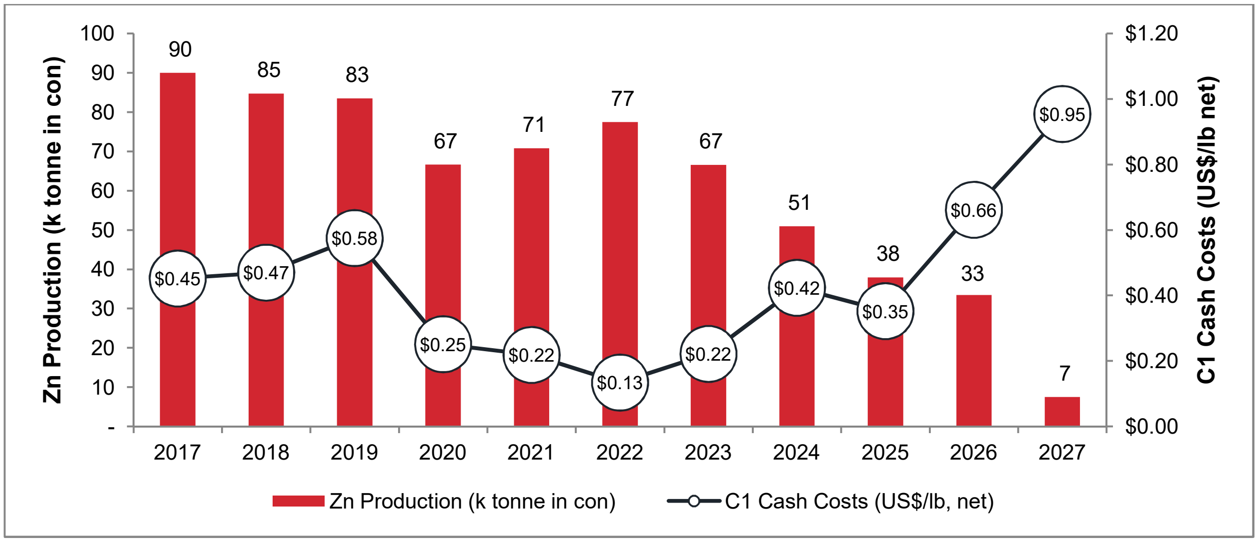
TABLE 21-4: CASH COSTS (NET OF BY-PRODUCT CREDITS)

Cash Costs (Net of By-Product Credits[1])	Units	Next 5 Years	LOM
C1 Cash Costs	US$ / lb Zn in con	$0.41	$0.37
C1 Cash Costs + Sustaining Capex	US$ / lb Zn in con	$0.57	$0.50

[1] By-product credits are calculated using the following assumptions: copper price per pound - US$2.60 in 2017, US$2.75 in 2018, US$3.00 in 2019 to 2020 and long-term; gold price per ounce - US$1,300 in 2017 to 2020 and US$1,260 long-term; silver price per ounce - US$18.00 in 2017 to 2020 and long-term; CAD/USD exchange rate - 1.35 in 2017, 1.25 in 2018, 1.20 in 2019, 1.15 in 2020 and 1.10 long-term.

Lalor's annual zinc production (contained zinc in concentrate) and C1 cash costs (net of by-products) are shown below in Figure 21-1. Over the first 5 years, annual production is expected to average 79 thousand tonnes of zinc at an average C1 cash cost of US$0.41/lb. Over the 10.5 year LOM, annual production is expected to average 64 thousand tonnes of zinc at an average C1 cash cost of US$0.37/lb. Lower C1 cash costs from years 2020 to 2023 are a result of mining the copper-gold (Zone 27).

FIGURE 21-1: LALOR ANNUAL ZINC PRODUCTION AND C1 CASH COSTS



22 ECONOMIC ANALYSIS

Hudbay is a producing issuer and has excluded information required by Item 22 of Form 43-101F1 as the updated mine plan does not represent a material increase of Hudbay's current production.

23 ADJACENT PROPERTIES

The author is not aware of any current relevant work on properties immediately adjacent to the Lalor deposit.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Gold Bulk Sample Program

In the fourth quarter of 2016, Hudbay developed 37 drift rounds at Lalor to assess the continuity and variability of non-contact gold mineralization within discrete areas of zones 21 and 25. Approximately 10,000 tonnes of bulk sample material was mined and hauled to surface. The material was primary crushed and processed through a sample tower to collect a representative subsample of each development round. The integrity of the material was maintained at all times through a rigorous chain of custody process. The mined material, stored on surface, is available for milling pending the potential economic viability of refurbishing the New Britannia gold mill in Snow Lake by Hudbay.

The run of mine (ROM) material for the bulk sampling program came from mining in three different areas of the northern parts of gold zones 21 and 25 on 975 m level, 995 m level and 1000 m level. The areas were selected as to provide maximum spatial distribution of the samples with minimal need for additional capital expenditures on underground development.

The bulk sample program was conducted to evaluate the mining potential of the gold zones and to increase the confidence in results obtained by modeling of the gold zones based on diamond drill hole data.

The grade of the mined material was derived by the collection of manageable sized subsamples representative of each individual round for assaying. The sub samples were collected by the use of a contractor crushing and a custom built sample tower.

To ensure the validity of the obtained results QAQC procedures exceeding best industry practices were in place. The QAQC procedures allowed for the ROM material to be traced back to individual LHD buckets from any particular round. Total round separations was assured by the use of a ROM surface bunker, and elaborate cleaning procedures for all equipment used between rounds.

Other parts of the project included:

- Testing feasibility and performance of various grade control procedures and techniques to be used in connection with the potential to mine future non-contact gold resources (diamond drilling down drifts, sampling while mucking and chip sampling)

- Structural studies to develop a better understanding of the controls of grade distribution within the gold zones

The preliminary results indicate that the gold grades from the bulk sample program are as expected with minor variations when compared to those modeled based on diamond drill data. The bulk sample program has increased the confidence and the understating of the gold zones and gold mineralization at Lalor.

Following full assessment of the 2016 bulk sample data it is intended to collect additional subsamples in 2017 from other areas of gold zones 21 and 25 as well as from other previously untested gold zones for confirmation purposes.

24.2 Taxes and Royalties

24.2.1 Applicable Tax Rates

The Lalor mine is not directly taxable as Hudbay pays provincial and federal taxes on a legal entity basis. The combined federal and provincial tax rates are assumed to be approximately 27% for the LOM and Hudbay has approximately C$750 million in tax pools that can be used to offset future income taxes for federal and provincial purposes. Hudbay's mining operations in Manitoba are also subject to the Manitoba Mining Tax. The Manitoba Mining Tax is not applied to a new mining project until the original capital expenditures are recovered.

24.2.2 Royalties

There are no royalties applicable to Lalor.

The author is not aware of any other information that would impact the reported estimate of mineral resources or estimated mineral reserves for the Lalor deposit.

25 INTERPRETATION AND CONCLUSIONS

The Lalor mine operation has been mining ore since August 2012. Since then the mine has operated uninterrupted and been in a continuous production ramp-up cycle, achieving the highest annual tonnage of approximately 1.1 million tonnes in 2016, with complementary throughput at the Stall concentrator. The production ramp-up is planned to continue in 2017 to reach a steady state of 4,500 tpd by the first quarter of 2018.

The production increase of 50%, compared to current production, is supported by an underground ore handling circuit capable of 4,500 tpd, transitioning to more bulk mining methods (65% of reserves) with additional mining fronts and design changes to improve mining efficiencies, developing ore passes and transfer raises to reduce truck haulage cycle times from the upper potions of the mine and commissioning of a paste plant backfill plant in the first quarter of 2018. Autonomous operation of a Load Haul Dump loader underground is currently being trialed from surface by tele-remote monitoring with changes to standard designs to allow isolation of autonomous areas and buffer storage for in between shift mucking.

The increase in production to 4,500 tpd at Lalor is complemented by the Stall concentrator expansion to 4,500 tpd, which is currently underway and is expected to be commissioned in the third quarter of 2018.

The mineral resources, as of September 30, 2016, are estimated as base metal lenses or gold zones based on geological and mineralization properties. The Hudbay validation process and third party review confirmed the resource block model is interpolated using industry accepted modelling techniques and classified in accordance with the 2014 CIM Definition Standards – For Mineral Resources and Mineral Reserves.

A mine reconciliation of the mined out areas compared to the ore reported at the concentrator was very close on the precious metals and a slight conservatism of the zinc and copper grades might be evident. This conservatism of the base metals is likely due to over constraining the high grade samples to 20 m as part of the high yield restriction step.

The mineral resources stated with a metal equivalency cut-offs provide for economic extraction of reserves from stated resources.

The mineral reserves, as of January 1, 2017, are based on a LOM plan that generated a mining inventory based on stope geometry parameters with appropriate dilution and recovery factors The conversion of resources to reserves is based on the LOM plan and NSR cut-offs that primarily focussed on capturing base metal resources for processing at the Stall concentrator. The secondary focus was to capture gold zone resources when in contact with or close proximity to base metal resources. In areas where a large separation existed between base metal and gold lenses, mining blocks were evaluated for economic stope mining shapes. When a non-economic shape was generated in a first pass, a second pass was evaluated for only base metal lenses and if an economic shape was generated the gold zone portion was removed. However, due to this larger separation, majority of these isolated gold lenses could have been evaluated independently of the base metal lenses and could potentially provide feed to a gold processing facility. Below approximately the 950 m level no attempt was made to generate an economic stope mining shape for gold zones 25 and 26 as the separation distance became too large. The author's opinion is that these resources are potentially better suited for a gold processing facility and should be re-evaluated when Hudbay has a better understanding of their New Britannia gold mill and Birch Tailings Impoundment Area in Snow Lake.

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 production and compilation of this technical report was supported by the capable and professional management and staff at Hudbay. The supervision, revision and approval of the assembly of this Technical Report is by the QP Robert Carter, P. Eng., Lalor Mine Manager at Hudbay Manitoba Business Unit.

26 RECOMMENDATIONS

It is recommended that the following actions be performed:

- Investigate the impact of under assaying of high grade standards at Hudbay's Flin Flon laboratory and whether this in turn affects the high grade Lalor samples submitted and has potentially led to an underestimations of gold in the resource estimate, since the proportion of samples assayed at the laboratory was approximately 80% of the total samples assayed between 2012 and 2016
- Investigate the high yield restriction parameters of the high grade base metal samples, and consider whether the restriction distance is suitable or were they over constrained, based on the conservatism noted in the mine reconciliation for zinc and copper
- Pursue the option of a temporary paste backfill plant to utilize the boreholes from surface prior to commissioning of the permanent plant in the first quarter of 2018. This option provides assurance to achieve the ramp-up in production and is another source of backfill rather than relying on waste development.
- Due to the approximate 6 month timing offset of the production ramp-up at Lalor to 4,500 tpd and the Stall concentrator expansion to 4,500 tpd, Hudbay should pursue transporting of ore from Lalor to their Flin Flon concentrator for earlier processing.
- Finalize the evaluation of the gold bulk sample program conducted in the fourth quarter of 2016 and since Hudbay owns a sample tower consider collecting additional subsamples from other areas of gold zones 21 and 25 as well as from other previously untested gold zones for confirmation purposes.
- Hudbay owns the New Britannia mill, a gold leach plant on care and maintenance, in Snow Lake. Hudbay should continue to assess the feasibility of processing a portion of the material mined from the gold zone and copper-gold zone at Lalor at the New Britannia mill at a rate of 1,500 tpd starting in 2019. When combined with the processing capacity of the Stall concentrator, this would enable an aggregate throughput rate of up to 6,000 tpd and utilize the full capacity of the Lalor mine shaft.

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28 SIGNATURE PAGE

This Technical Report titled "NI 43-101 Technical Report, Lalor Mine, Snow Lake, Manitoba, Canada", dated March 30, 2017 and effective as of March 30, 2017 was prepared under the supervision and signed by the following author:

Dated effective this 30th day of March, 2017.

(signed) Robert Carter

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Signature of Qualified Person

Robert Carter, P. Eng.
Lalor Mine Manager, Hudbay Manitoba Business Unit

29 CERTIFICATES OF QUALIFIED PERSONS

Robert Carter

CERTIFICATE OF QUALIFICATION

Re: Lalor Mine Technical Report, March 30, 2017

I, Robert Carter, P.Eng., of Burlington, Ontario, do hereby certify that:

1. I am currently employed as Lalor Mine Manager, Hudbay Manitoba Business Unit, with Hudbay Minerals Inc. (the "Issuer"), 25 York Street, Suite 800, Toronto, Ontario, Canada, M5J 2V5

2. I graduated from University of Manitoba with a Bachelor of Sciences in Geological Engineering in 1997.

3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of Manitoba, Registration #21836.

4. I am a member in good standing of the Association of Professional Engineers of Ontario, Registration #100089189.

5. I have practiced my profession continuously for over 19 years and have been involved in mineral exploration, mine site engineering and geology, mineral resource and mineral reserve evaluations, and mine operations for base metal deposits and operations in North and South America.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.

7. I have reviewed and approved and I am responsible for the preparation of this Technical Report titled "NI 43-101 Technical Report, Lalor Mine, Snow Lake, Manitoba, Canada", dated March 30, 2017 (the "Technical Report") and effective as of March 30, 2017.

8. I last visited the property on March 29, 2017. I am directly involved with Lalor mine on a permanent basis because of my role as mine manager and I personally inspect the operation on a routine basis.

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

10. I am not independent of the Issuer. Since I am an employee of the Issuer, a producing issuer, I fall under subsection 5.3 (3) of NI 43-101 where “a technical report required to be filed by a producing issuer is not required to be prepared by or under the supervision of an independent qualified person.”

11. I have been involved with the Lalor property, which is the subject of the Technical Report, continuously since discovery in 2007.

12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument and form.

13. I consent to the public filing of the Technical Report with any stock exchange, securities commission or other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 30th day of March, 2017.

Original signed by:

Robert Carter

Robert Carter, P. Eng.

Lalor Mine Manager, Hudbay Manitoba Business Unit