**NI 43-101 Technical Report**

**Feasibility Study**

**Updated Mineral Resource, Mineral Reserve and Financial Estimates**

# Rosemont Project

## Pima County, Arizona, USA

**Issue and Effective Date: March 30, 2017**


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**Revision 3**

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

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

- the 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, governmental and joint venture partner approvals;
- the availability of personnel for our exploration, development and operational projects and ongoing employee and union relations;
- the ability to secure required land rights to develop the Pampacancha deposit;
- maintaining good relations with the communities in which we operate, including the communities surrounding our Rosemont project;
- 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 (including risks associated with the permitting, development and economics of the Rosemont project and related legal challenges), dependence on key personnel and employee and union relations, risks related to political or social unrest or change, risks in respect of aboriginal and community relations, rights and title claims, operational risks and hazards, including unanticipated environmental, industrial and geological events and developments and the inability to insure against all risks, failure of plant, equipment, processes, transportation and other infrastructure to operate as anticipated, 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

The information that follows provides an executive summary of important information contained in this Technical Report.

## 1.1 Introduction

The author has prepared this Technical Report for Hudbay Minerals Inc. (“Hudbay”) with respect to its Rosemont Project (the “Project”), located in Arizona (the “Property”), issued and effective as of March 30, 2017. The purpose of this Report is to present Hudbay’s estimate of the mineral reserves and mineral resources for the Project based on the current mine plan, the current state of metallurgical testing, operating cost and capital cost estimates.

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 completed the acquisition of the Project on September 23, 2014 through its acquisition of all issued and outstanding common shares of Augusta Resource Corporation (“Augusta”) pursuant to the take-over bid, which expired July 29, 2014.

Hudbay owns a 92.05% interest in the 132 patented claims and 1,064 unpatented claims that comprise the Project, all of which are duly registered in the name of Hudbay’s wholly-owned subsidiary, Rosemont Copper Company[1]; Rosemont Copper Company also has the required surface rights to develop the Project. This Technical Report represents the first technical report filed by Hudbay since its acquisition of Augusta and also updates and supersedes Augusta’s Updated Feasibility Study dated August 28, 2012, prepared by M3 Engineering and Technology Corporation.

This Technical Report provides current estimates of the mineral reserves and mineral resources at the Project and describes the latest resource model, mine plan and the current state of the permitting process, metallurgical testing, operating cost and capital cost estimates. The information presented in this Technical Report relating to the Rosemont deposit, including the estimates of mineral reserves and resources therein, is the result of “feasibility study” level work conducted partly by external contractors and partly internally by Hudbay’s personnel under the overall supervision of, Cashel Meagher, the Qualified Person (the “QP”).

[1] Hudbay’s ownership in the Project is subject to an earn-in agreement and joint venture agreement dated September 16, 2010 between Rosemont Copper Company and United Copper & Moly LLC, pursuant to which UCM has earned a 7.95% interest in the project and may earn up to a 20% joint venture interest.

This Technical Report conforms with the 2014 CIM Definition Standards – for Mineral Resources and Mineral Reserves and the requirements in Form 43-101F1 of National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

The QP and Principal Author who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo., Senior Vice President and Chief Operating Officer for Hudbay. Mr. Meagher last visited the property on April 21, 2016 and numerous times prior to this date. The personal site inspections were conducted as part of the mineral resource estimation and technical report process, to become familiar with conditions on the Property and the Project, to observe the geology and mineralization and verify the work completed on the Property. Mr. Meagher has reviewed and approved the 3D block model and determination of mineral resources and mineral reserves of the Project.

As Hudbay is a "producing issuer", as defined in NI 43-101, this Technical Report is not required to be prepared by or under the supervision of an independent QP.

## 1.2 Property Description and Location

The Project is located within the historic Helvetia-Rosemont Mining District on the eastern flanks of the Santa Rita Mountain Range, approximately 30 miles southeast of Tucson in Pima County, Arizona. The property consists of a comprehensive land package that includes patented and unpatented mining claims, fee land and grazing leases that cover most of the old mining district. The lands are under a combination of private ownership by Rosemont Copper and Federal ownership. The lands occur within Townships 18 and 19 South, Ranges 15 and 16 East, Gila & Salt River Meridian. The Project's geographical coordinates are approximately 31º 50'N and 110º 45'W.

Hudbay's ownership in the Project is subject to an earn-in agreement and joint venture agreement dated September 16, 2010 between Rosemont Copper Company and United Copper & Moly LLC (''UCM''), pursuant to which UCM has earned a 7.95% interest in the Project and may earn up to a 20% joint venture interest.

Hudbay has all of the surface and mineral rights required to conduct the open pit mining operation, processing and concentrating facilities, storage of tailings, and disposal of waste rock as documented in this Technical Report. The core of the Project mineral resource is contained within the 132 patented mining claims that in total encompass an area of approximately 2,000 acres. Surrounding the patented claims is a contiguous package of 1,064 unpatented mining claims with an aggregate area of more than 16,000 acres.

There is a 3% Net Smelter Return ("NSR") royalty on all 132 patented claims, 603 of the unpatented claims, and one parcel of fee owned associated land. Pursuant to a precious metals stream agreement with Silver Wheaton Corp. ("Silver Wheaton") entered into on February 11, 2010, as amended and restated on February 15, 2011, Hudbay will receive deposit payments of $230 million against delivery of 100% of the payable gold and silver from the Project. The deposit will be payable upon the satisfaction of certain conditions precedent, including the receipt of permits for the Project and the commencement of construction. In addition to deposit payments, as gold and silver is delivered to Silver Wheaton, Hudbay will receive cash payments equal to lesser of (i) the market price and (ii) $450 per ounce (for gold) and $3.90 per ounce (for silver), subject to one percent annual escalation after three years. Approximately 50% of the copper concentrate has been contracted under existing commitments that are on benchmark-based terms.

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

Existing graded dirt roads connect the Project site with State Route 83, which provides easy access to the Project for the communities of Tucson and Benson to the north, and to Sierra Vista, Sonoita, Patagonia and Nogales to the south. The city of Tucson, Arizona provides the nearest major railroad and air transport services to support the Project.

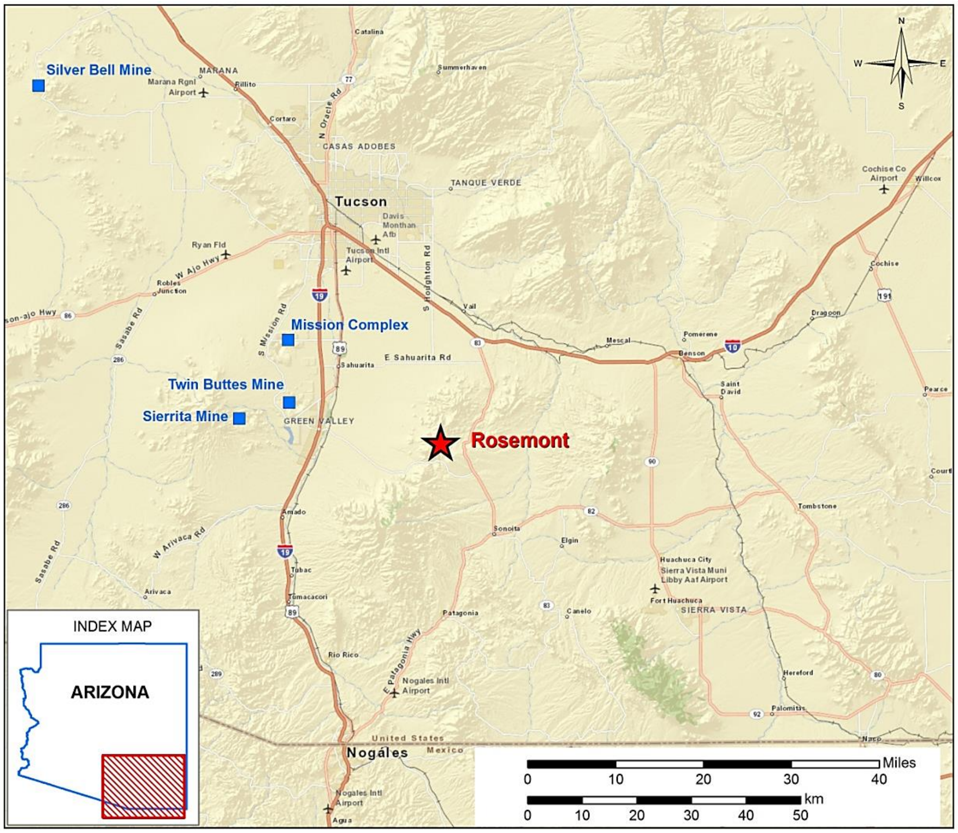
The Project site is located immediately adjacent and west of Arizona State Route 83, approximately eleven miles south of Interstate 10 (I-10). This system of state and interstate highways allows convenient access to the site for all major truck deliveries. The majority of the labour and supplies for construction and operations come from the surrounding areas in Pima, Maricopa, Cochise, and Santa Cruz counties.

The southern Arizona climate is typical of a semi-arid continental desert with hot summers and temperate winters. However, higher elevations in the Project area (4,550 to 5,350 feet AMSL) result in a milder climate than at lower elevations across the region. Summer daily high temperatures are above 90°F with significant cooling at night. Winter in the Project area is typically drier with mild daytime temperatures and overnight temperatures that are typically above freezing.

The average annual precipitation in the Project area is estimated between 16 and 18 inches with more than half of the annual precipitation occurring during the monsoon season from July through September. Rainfall has minor effects on a mining operation, which is considered to be 365 days per year.

The Project is located within the northern portion of the Santa Rita Mountains that form the western edge of the Mexican Highland section of the Basin and Range Physiographic Province characterized by high mountain ranges adjacent to alluvial filled basins. Vegetation in the Project area reflects the climate with the lower slopes of the Santa Rita Mountains dominated by mesquite and grasses while the higher elevations, receiving greater rainfall, support an open cover of oak, pine, juniper and cypress trees.

**FIGURE 1-1: ROSEMONT COPPER PROJECT PROPERTY LOCATION**




## 1.4 History

The first recorded mining activity in the Helvetia-Rosemont mining district occurred in 1875 and the mining district was officially established in 1878. Production from mines on both sides of the Santa Rita ridgeline supported the construction and operation of two smelters. Copper production from the district ceased in 1951 after production of about 227,300 tons of ore.

By the late 1950s, the Banner Mining Company ("Banner") had acquired most of the claims in the area and had drilled the discovery hole into the Rosemont deposit. In 1963, Anaconda Mining Co. acquired options to lease the Banner holdings. Over the next ten years, they carried out an extensive drilling program on both sides of the ridgeline.

In 1973, the Anaconda Mining Co. and Amax Inc. formed a 50/50 partnership to form the Anamax Mining Co. ("Anamax") and in 1985, Anamax ceased operations and liquidated their assets.

ASARCO Inc. ("Asarco") purchased the patented and unpatented mining claims from Anamax's real estate interests in August 1988 and renewed exploration and engineering studies. Asarco expanded the core of the mineral deposit in 1995 by patenting 347 acres, the last of the available claims for the orebody. In 1999, Grupo Mexico acquired the Helvetia-Rosemont property through a merger with Asarco and in 2004 Grupo Mexico sold the property to a Tucson real estate developer.

In April 2005, Augusta purchased the Property from Triangle Ventures LLC and initiated a series of extensive drill programs on the property. A Technical Report issued by Augusta in 2012 estimated mineral reserves of 667.2 million tons at an average grade of 0.44% copper, 0.015% molybdenum and 0.12 ounces per ton of silver based on $4.90 per ton net smelter return cut-off using metal prices of 2.50/lb. copper, $15.00/lb. molybdenum and $20.00/oz. silver.

*Note that Hudbay has treated Augusta's publicly disclosed estimated mineral reserves and resources as a historical estimate under NI 43-101 and not as current mineral reserves and resources, as a qualified person has not performed sufficient work for Hudbay to classify the 2012 estimate for the Project's mineral reserves or resources as current mineral reserves or mineral resources.*

Following its acquisition of Augusta, Hudbay acquired all of the issued and outstanding common shares of Augusta pursuant to a take-over bid, which expired July 29, 2014, and a subsequent acquisition transaction, which closed on September 23, 2014. Hudbay's ownership in the Project is subject to an earn-in agreement with UCM, pursuant to which UCM has earned a 7.95% interest in the Project and may earn up to a 20% interest. A joint venture agreement between Hudbay's subsidiary, Rosemont Copper Company, and UCM governs the parties' respective rights and obligations with respect to the Project.

Hudbay completed a 43-hole, 92,909 feet (28,319 m) drill program from September to December 2014 and a 46-hole, 75,164 feet (22,910 m) drill program from August to November 2015 in further efforts to better understand the geological setting and mineralization of the deposit and to collect additional metallurgical and geotechnical information.

## 1.5 Geological Setting and Mineralization

The Laramide belt is a major porphyry province that extends for approximately 621 miles (1,000 km) from Arizona to Sinaloa, Mexico. It hosts a number of world-class deposits including the Rosemont deposit. The northern block of the Santa Rita Mountains, where the Rosemont deposit lies, is dominated by Precambrian granite, with some dismembered slices of Paleozoic and Mesozoic sediments on the eastern and northern sides.

Paleozoic sedimentary carbonate units are the predominant host rocks for the copper mineralization. Structurally overlying these predominantly carbonate units at Rosemont are Mesozoic clastic units, including conglomerates, sandstones, and siltstones. These clastic upper sequences have andesitic flows and host mineralization. Quartz monzonite and quartz latite sill-shaped porphyries intruded both sequences and are associated with the porphyry/skarn mineralization.

Post-mineral features partially delimit the defined resource, dividing the deposit into major structural blocks with contrasting intensities and types of mineralization. The north-trending, steeply-dipping Backbone Fault juxtaposes marginally mineralized Precambrian granodiorite and Lower Paleozoic quartzite and limestone to the west against a block of younger, well-mineralized Paleozoic limestone units to the east.

The Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn that formed in the Paleozoic rocks as a result of the intrusion of quartz latite to quartz monzonite porphyry intrusions. Bornite-chalcopyrite-molybdenite mineralization occurs as veinlets and disseminations in the skarn.

Three mineralization domains (oxide, mixed and sulfide) were defined based on the soluble to total copper ratio (ASCu/TCu) collected in the Augusta (2005 to 2012) and Hudbay (2014 and 2015) drilling programs. The oxidation and mixed mineralization occurs mainly above a low angle fault defining the contact between the Palozoic and Mesozoic rocks as chrysocolla, copper carbonates and supergene chalcocite.

## 1.6 Deposit Types

As mentioned above, the Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn, genetically, it is a style of porphyry copper deposit, although intrusive rocks are volumetrically minor within the resource area. The skarns are formed as the result of thermal and metasomatic alteration of Paleozoic carbonate and to a lesser extent Mesozoic clastic rocks. Near surface weathering has resulted in the oxidation of the sulfides in the overlying Mesozoic units however, oxidation also occurs in the underlying Paleozoic carbonates.

## 1.7 Exploration

Prospecting began in the Rosemont and Helvetia Mining Districts in the mid-1800s and by 1875 copper production was first recorded, which continued sporadically until 1951. By the late 1950s, exploration drilling had discovered the Rosemont deposit. A succession of major mining companies subsequently conducted exploratory drilling of the Rosemont deposit and the nearby Broadtop Butte, Peach Elgin and Copper World mineralized areas.

Augusta acquired the Rosemont property in 2005 and performed infill drilling of the Rosemont deposit along with exploration geophysical surveys. A Titan 24 induced polarization/resistivity (DCIP) survey over the Rosemont deposit, performed in 2011, discovered significant chargeability anomalies, which were partially tested. These anomalies appear to define mineralization and certain unmineralized lithologic units. A regional scale airborne magnetics survey was also completed in 2008.

Two infill drilling campaigns were completed by Hudbay in and beneath the Rosemont deposit in the fall of both 2014 and 2015. In addition to chemical assaying, magnetic susceptibility and conductivity measurements were taken. A single test-line of DCIP data was collected over the Rosemont deposit using the DIAS Geophysical in April 2015 for comparison to the previously completed Titan 24 survey.

Hudbay analyzed all samples of the 2014 and 2015 drilling programs with ICP multi-element geochemistry. This new geochemical data set was used to classify rocks according to chemical indexes in a ternary diagram defined by siliciclatic, limestone and dolomitic vertices. The lithogeochemical groups honour the deposit stratigraphy and geochemical attributes and proved to be a useful tool for geological modeling and vectoring.

A mapping and geochemical sampling program was completed in the latter half of 2015 on the Rosemont property to reassess the interpretation of the regional geology and deposit setting. This was followed by a structural interpretation using both surface and drill core measurements to aid in the geotechnical evaluation of the Project.

## 1.8 Drilling

Extensive drilling has been conducted at the Rosemont deposit by several successive property owners. The most recent drilling was done by Hudbay, with prior drilling campaigns completed by Banner, Anaconda Mining Co., Anamax and Asarco and Augusta. Table 1-1 summarizes the drill holes used to estimate the current mineral resource estimate, with regional exploration holes excluded. The drillholes are approximately 200 feet apart over the core of the deposit.

**TABLE 1-1: ROSEMONT DEPOSIT DRILLING SUMMARY**

| Company | Time Period | Drill Holes | |
|---|---|---|---|
| | | Number | Feet |
| Banner Mining | 1950s to 1963 | 3 | 4,300 |
| Anaconda Mining | 1963 to 1973 | 113 | 136,838 |
| Anamax | 1973 to 1986 | 52 | 54,350 |
| ASARCO | 1988 to 2004 | 11 | 14,695 |
| Augusta | 2005 to 2012 | 87 | 132,525 |
| Hudbay | 2014 to 2015 | 90 | 168,286 |
| **Total** | | **355** | **510,780** |

The recent Hudbay drilling went deeper by approximately 300 feet on average than the Augusta drilling and almost twice as deep as the Anaconda and Anamax drilling program. This most recent drilling has helped to confirm the size and quality of the deposit estimated by previous owners and to also establish its continuation at depth resulting in an improved definition of the optimum open pit design.

## 1.9 Sample Preparation, Analyses, and Security

During the Hudbay 2014 and 2015 drill programs, the samples were transported to the Inspectorate America Corporation (“Inspectorate”) preparation facility at Sparks, Nevada, USA. Once the samples were pulverized, a 150 g subsample pulp was collected and air-freighted to Bureau Veritas Commodities Canada Ltd., in Vancouver, Canada, for analysis. A total of 18,361 drill core samples in 2014 and 14,868 samples in 2015 were analyzed for copper, molybdenum and silver, through a multi-element (45 elements) determination by Inductively Coupled Plasma Mass Spectrometry after 4-acid digestion. A total of 1,677 samples in 89 drill holes were collected for specific gravity determinations by a standard water displacement method at the Inspectorate preparation facility.

As part of Hudbay's quality control and quality assurance ("QA/QC") program, QA/QC samples were systematically introduced in the sample stream to assess adequate sub-sampling procedures, potential cross-contamination, precision, and accuracy. A total of 1,000 representative pulp samples (5.4%) from 2014 drilling and 742 representative pulp samples (5.0%) from 2015 drilling were selected and re-analyzed at the SGS Canada Inc., laboratory in Vancouver.

The core samples from the Augusta drilling programs from 2005 to 2012 were transported to Skyline Assayers and Laboratories (Skyline), in Tucson, Arizona, USA for preparation and analysis. In total, 21,197 samples were analyzed for total copper and 16,619 samples for molybdenum. Total copper and molybdenum were dissolved using a hot 3-acid digestion at 482°F and subsequently analyzed by AAS and ICP-OES, respectively. The lower detection limits for molybdenum are high relative to the average molybdenum grade of the Rosemont deposit. Silver was determined in 15,334 samples, which were digested using an aqua regia leach in 0.25 g subsample pulp and analyzed by AAS. A total of 391 drill core samples across the Rosemont deposit were measured for specific gravity at Skyline.

Augusta conducted its own internal QA/QC program to independently evaluate the quality of the assays reported by Skyline. Standards and blanks were systematically inserted in the sample stream. Duplicates were not periodically inserted.

Prior to Hudbay and Augusta, significant diamond drilling, drill core sampling, and assaying programs were executed by several property owners. Records are not available that detail the sampling and security protocols used by these property owners. There are no available QA/QC records for sample preparation and assaying methodologies for Banner, Anaconda, and Anamax. Copper, molybdenum, silver, and soluble copper were analyzed by Anaconda and Anamax at their in-house laboratories. Silver was regularly analyzed by Anamax, but not commonly assayed by Banner and Anaconda. Asarco assayed drill core samples for total copper, molybdenum, and acid soluble copper ("ASCu") at Skyline laboratory.

## 1.10 Data Verification

Hudbay built an entirely new drill hole database from all pre-Hudbay drilling and assaying information. Orix Geoscience Inc. was employed to digitally enter collar, downhole surveys and assay information from scanned drill logs and assay certificates for all holes drilled prior to ownership of the property by Augusta.

The infill drilling conducted by Hudbay and Augusta together with re-assaying of historical holes have closely replicated previous drilling campaign results confirming that the historical data can be used with a sufficient level of confidence for resource and reserve estimation. A bias was identified in the historical molybdenum assays and the data was corrected.

The author’s opinion is that the data verification is adequate for the purposes used in the Technical Report.

## 1.11 Mineral Processing and Metallurgical Testing

The earliest reported testwork on Rosemont ores comprising preliminary grinding and flotation tests was completed by Anamax in 1974. This early work was followed by a larger testwork campaign by Augusta in 2006 and 2007 to support the preparation of a feasibility study and technical report. Further testwork was then completed by Augusta between 2008 and 2012 to support engineering design and updates to the original technical report.

Historical metallurgical testwork programs were undertaken by Mountain State R&D International (MSRDI), SGS and G&T Metallurgical Services, with dewatering and rheology testing undertaken by Pocock, Outotec and FLSmidth. In 2014, Hudbay engaged XPS Consulting & Testwork Services (XPS) to undertake mineral characterization and metallurgical testwork. Base Met Laboratory (“BML”) was engaged in late 2015 to provide confirmation testwork of the XPS testwork and additional process optimization.

The testwork investigated key geo-metallurgical variables such as copper oxide content, swelling clays, magnesium clays and ore hardness. The copper oxide content, as measured by the acid soluble procedure, is a good indicator of the recoverable copper content of the ore. Clay content varies considerably in type and quantity throughout the oxide, transition and sulfide mineralization. Ore hardness varies from soft to very hard; testing results, together with geomet proxy modelling, were utilized to calculate hardness in the resource model.

Production period composites, together with the geo-metallurgical samples, underwent flotation testing for process engineering design as well as a recovery estimator for mine planning and the financial model.

Through the course of all the mineral processing and metallurgical testing, no deleterious elements were found to have a negative impact on plant performance or on the marketable value of the copper and molybdenum concentrates to be produced at the Project.

Based on the body of testwork that exists, including both the historical testwork, and the testing programs completed by Hudbay since the acquisition of the Project, forecasts of recovery, concentrate grade and quality, as well as characteristics of the resultant tailing product have been developed. The following summarizes long range mine plan (“LOM”) average recoveries expected.

| Concentrate | Average LOM recoveries |
|---|---|
| Copper (Cu) | 80.4% |
| Molybdenum (Mo) | 53.4% |

| Concentrate | Average LOM recoveries |
|---|---|
| Silver (Ag) | 74.4% |
| Gold (Au) | 65.1% |

## 1.12 Mineral Resource Estimate

Hudbay prepared a 3D block model of the Rosemont deposit. The 3D block model and determination of the mineral resources were reviewed and approved by Cashel Meagher, P.Geo., Senior Vice President and Chief Operating Officer for Hudbay and QP of this Technical Report.

### 1.12.1 Wireframe Models and Mineralization

The Rosemont deposit trends approximately along an azimuth of N020° with a general dip of 50° to the east. The Backbone Fault forms the footwall contact along the entire length of the deposit. Geologically, Rosemont is a skarn deposit. The deposit is continuous along a strike length of 4,000 feet in a north-south direction, 3,000 feet in an east-west direction and goes to a maximum vertical depth of approximately 2,500 feet.

Three sets of structures were recognized, a north-northeast trending set, an east-west trending set and a gently east dipping set. The structures locally offset mineralization but some also appear to control mineralization, especially the oxidation. Wireframes were constructed for each lithological unit, oxidation level and fault structure.

### 1.12.2 Exploratory Data Analysis

A statistical analysis (basic statistics, histograms, box plots, contact plots, regressions) for total copper, acid-soluble copper, molybdenum, silver and sample length was performed on all the assays and composites to ensure mineralized domains were understood and that bias was not introduced during the data preparation stage.

### 1.12.3 Variography

Experimental variograms were calculated for total copper, acid-soluble copper, molybdenum and silver from the 25-foot capped composites. Directional and down-the-hole correlograms were fitted. The down-the-hole models were used to select the nugget used in subsequent modelling of directional correlograms. The total copper variograms show very low to moderate nugget effects with ranges of correlation generally varying between 340 to 2,000 ft, with the majority of the variability occurring within the first 200 to 300 feet in all directions.

### 1.12.4 Estimation and Interpolation Methods

The block model consists of regular blocks (50 feet along strike x 50 feet across strike x 50 feet vertically). The block size was chosen such that geological contacts are reasonably well reflected and to support a large-scale open pit mining scenario.

The interpolation plan was completed on the uncapped and capped composites via ordinary kriging ("OK") interpolation method using three passes with increasing search distances.

The first interpolation pass was restricted to a minimum of nine composites, a maximum of 12 composites (with a maximum of three composites per hole) and quadrant declustering. The second interpolation pass was restricted to a minimum of six composites, a maximum of 12 composites (with a maximum of three composites per hole) and quadrant declustering. Finally, the third interpolation pass was restricted to a minimum of four composites, a maximum of 12 composites (with a maximum of three composites per hole) without quadrant declustering.

### 1.12.5 Block Model Validation

The Rosemont block model was validated to ensure appropriate honouring of the input data and to verify the absence of bias by the following methods:

- Visual inspection of the OK block model grades in plan and section views in comparison to composites grades
- Assessment of the quantity of metal removed via the grade capping methodology
- Comparison between the different interpolation methods, including nearest neighbour, inverse distance squared and OK
- Swath plot comparisons of the estimation methods
- Review of block model OK quality control parameters
- Review of grade tonnage curves and statistics for each estimation method
- Third party and internal peer review

### 1.12.6 Classification of Mineral Resource

The resource category classification relies on the relative difference between the kriged grade and the composites grades, the number of composite used, the closest and farthest distance between the composites used and the centre of the blocks.

A smoothing algorithm was applied to remove isolated blocks of measured category blocks within areas of mostly indicated category or isolated indicated blocks within areas of mostly measured category blocks. Proportions of measured and indicated category blocks were not changed significantly by this process.

### 1.12.7 Reasonable Prospects of Economic Extraction

The component of the mineralization within the block model that meets the requirements for reasonable prospects of economic extraction was based on the application of a Lerchs-Grossman ("LG") cone pit algorithm. The mineral resources are therefore contained within a computer-generated open pit geometry.

The following assumptions were applied to the determination of the mineral resources:

- Economic benefit was applied to measured, indicated and inferred classified material within the resource cone.
- The LG optimization was conducted on a NSR value that reflects for each block in the resource model, the copper (“Cu”), molybdenum (“Mo”) and silver (“Ag”) grades, mill recoveries, contained metal in concentrate, deductions and payable metal values, metal prices, freight costs, smelting and refining charges and royalty charges.
- The pit shell selected to report mineral resources was based on a revenue factor of 1.0 (break-even value) using the following metal prices: $3.15/lb. copper, $11.00/lb. molybdenum, and $18.00/oz. silver.
- A constant 45-degree pit slope was used for the resource estimate.
- No haulage increment or bench discounting was applied to the resource estimate.

### 1.12.8 Mineral Resource Statement inclusive of the Mineral Reserve

Mineral resources for the Rosemont deposit were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves by application of a NSR calculation that reflects the combined benefit of producing copper, molybdenum and silver in addition to mine operating, processing and off-site costs. The cut-off used for resource reporting is based on a reasonable estimate of the investment required to construct and sustain a viable operating complex.

The mineral resources, classified as Measured, Indicated and Inferred and prior to any conversion to mineral reserves, inclusive of the portion of the mineral resources that was converted to mineral reserves, are summarized in Table 1-2.

*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 1-2: RESOURCE BY CATEGORY, MINERALIZED ZONE AND NSR CUT-OFF (1)(2)(3)(4)(5)(6)(7)(8)(9)(10)

| Measured | TONS | NSR Cut Off | CuEq (%) | Cu (%) | Mo (%) | Ag (opt) | *TONNES* | *Ag (g/t)* |
|---|---|---|---|---|---|---|---|---|
| Oxide | 111,800,000 | > = $5.70 | 0.38 | 0.38 | | | *101,400,000* | |
| Mix | 18,800,000 | > = $5.70 | 0.45 | 0.40 | 0.009 | 0.069 | *17,100,000* | *2.37* |
| Hypogene | 566,800,000 | > = $5.70 | 0.53 | 0.45 | 0.013 | 0.140 | *514,200,000* | *4.80* |
| **Summary** | **697,400,000** | | **0.51** | **0.44** | **0.011** | **0.116** | ***632,700,000*** | ***3.96*** |
| | | | | | | | | |
| **Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** | ***TONNES*** | ***Ag (g/t)*** |
| Oxide | 38,600,000 | > = $5.70 | 0.26 | 0.26 | | | *35,000,000* | |
| Mix | 7,000,000 | > = $5.70 | 0.40 | 0.36 | 0.007 | 0.055 | *6,400,000* | *1.88* |
| Hypogene | 521,500,000 | > = $5.70 | 0.32 | 0.26 | 0.011 | 0.081 | *473,100,000* | *2.79* |
| **Summary** | **567,100,000** | | **0.32** | **0.26** | **0.010** | **0.076** | ***514,500,000*** | ***2.59*** |
| | | | | | | | | |
| **Measured + Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** | ***TONNES*** | ***Ag (g/t)*** |
| Oxide | 150,400,000 | > = $5.70 | 0.35 | 0.35 | | | *136,400,000* | |
| Mix | 25,800,000 | > = $5.70 | 0.43 | 0.39 | 0.008 | 0.065 | *23,400,000* | *2.23* |
| Hypogene | 1,088,400,000 | > = $5.70 | 0.43 | 0.36 | 0.012 | 0.112 | *987,400,000* | *3.84* |
| **Summary** | **1,264,600,000** | | **0.42** | **0.36** | **0.011** | **0.098** | ***1,147,200,000*** | ***3.35*** |
| | | | | | | | | |
| **Inferred** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** | ***TONNES*** | ***Ag (g/t)*** |
| Oxide | 6,400,000 | > = $5.70 | 0.31 | 0.31 | | | *5,800,000* | |
| Mix | 1,600,000 | > = $5.70 | 0.46 | 0.44 | 0.004 | 0.024 | *1,500,000* | *0.84* |
| Hypogene | 74,100,000 | > = $5.70 | 0.35 | 0.30 | 0.011 | 0.050 | *67,200,000* | *1.70* |
| **Summary** | **82,100,000** | | **0.35** | **0.30** | **0.010** | **0.045** | ***74,500,000*** | ***1.55*** |

Notes:

1. The above mineral resources include mineral reserves.
2. Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were based on core logging, structural and geochemical data.
3. Raw drill hole assays were composited to 25-foot lengths broken at lithology boundaries.
4. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
5. Block grades for copper, molybdenum and silver were estimated from the composites using OK interpolation into 50 ft x 50 ft x 50 ft blocks coded by domain.
6. Tonnage factors were interpolated by lithology and mineralized zone. Tonnage factors are based on 2,066 measurements collected by Hudbay and previous operators.
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Mineral resources are constrained within a computer generated pit using the LG algorithm. Metal prices of $3.15/lb copper, $11.00/lb molybdenum and $18.00/troy oz silver. Metallurgical recoveries of 85% copper, 60% molybdenum and 75% silver were applied to sulfide material. Metallurgical recoveries of 40% copper, 30% molybdenum and 40% silver were applied to mixed material. A metallurgical recovery of 65% for copper was applied to oxide material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, and silver combined. Cut-off grades were set in terms of NSR based on current estimates of process recoveries, total process and G&A operating costs of $5.70/ton for oxide, mixed and sulfide material.
9. The oxide resource will be processed in the mill via flotation
10. Totals may not add up correctly due to rounding.

The reporting of the mineral resource by NSR within the LG pit shell reflects the combined benefit of producing copper, molybdenum and silver as per the following equations based on mineralized type, in addition to mine operating and processing costs:

| | | |
|---|---|---|
| NSR formula details | Copper Contribution | (Price of Copper -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) * unit conversion factor (% to ton) |
| | Molybdenum Contribution | (Price of Molybdenum -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) * unit conversion factor + (% to ton) |
| | Silver Contribution | (Price of Silver -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) |
| Sulfide: | Copper Contribution | ($3.15-$0.4307) * 0.85 * 0.96 * 0.97 * 20 + |
| | Molybdenum Contribution | ($11.00-$1.50) * 0.60 * 1 * 0.97 * 20 + |
| | Silver Contribution | ($18.00-$0.50) * 0.75* 0.90 * 0.97 |
| Mixed: | Copper Contribution | ($3.15-$0.4307) * 0.40 * 0.96 * 0.97 * 20 + |
| | Molybdenum Contribution | ($11.00-$1.50) * 0.30 * 1 * 0.97 * 20 + |
| | Silver Contribution | ($18.00-$0.50) * 0.40 * 0.90 * 0.97 |
| Oxide: | Copper Contribution | ($3.15-$0) * 0.65 * 1 * 0.97 * 20 |
| | Molybdenum Contribution | None |
| | Silver Contribution | None |

The copper equivalency is calculated, using metal contributions, for each block using the following formula:

CuEq = Copper + (Contribution of Molybdenum) + (Contribution of Silver)

Since molybdenum and silver are not considered in oxide material, the copper equivalency value equals the copper value.

### 1.12.9 Comparison with the 2012 Resource Estimate

A review and comparison of 2017 Hudbay mineral resource and 2012 Augusta Resource mineral resource was completed. The results of Measured, Indicated and Inferred are summarized in Table 1-3 and Table 1-4.

**TABLE 1-3: MEASURED AND INDICATED, COMPARISON TO 2012 AUGUSTA ESTIMATE**

| Mineralized Zone | Hudbay Resource 2017 Model | | | | | Augusta Resource 2012 Model | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NSR ($/ton) | Tons | Cu (%) | Mo (%) | Ag (opt) | CuEq Cut-Off (%) | Tons | Cu (%) | Mo (%) | Ag (opt) |
| Oxide | > = $5.70 | 150,400,000 | 0.35 | | | >=0.10 | 63,400,000 | 0.17 | | |
| Mixed | > = $5.70 | 25,800,000 | 0.39 | 0.008 | 0.07 | >=0.30 | 49,900,000 | 0.53 | 0.007 | 0.05 |
| Sulfides | > = $5.70 | 1,088,400,000 | 0.36 | 0.012 | 0.11 | >=0.15 | 869,300,000 | 0.40 | 0.014 | 0.11 |

**TABLE 1-4: INFERRED, COMPARISON TO 2012 AUGUSTA ESTIMATE**

| Mineralized Zone | Hudbay Resource 2017 Model | | | | | Augusta Resource 2012 Model | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NSR ($/ton) | Tons | Cu (%) | Mo (%) | Ag (opt) | CuEq Cut-Off (%) | Tons | Cu (%) | Mo (%) | Ag (opt) |
| Oxide | > = $5.70 | 6,400,000 | 0.31 | | | >=0.10 | 1,100,000 | 0.15 | | |
| Mixed | > = $5.70 | 1,600,000 | 0.44 | 0.004 | 0.02 | >=0.30 | 10,100,000 | 0.39 | 0.006 | 0.02 |
| Sulfides | > = $5.70 | 74,100,000 | 0.30 | 0.011 | 0.05 | >=0.15 | 128,400,000 | 0.40 | 0.013 | 0.10 |

The 2017 measured and indicated resource estimates constitute a 29% increase in tonnage with copper grades 8% lower to those estimated in 2012 by Augusta. Also, the molybdenum grade is 17% lower than reported in 2012 for the sulfide mineralized material while the 2016 oxide tonnage and grade have more than doubled. These differences result mainly from the reinterpretation of the oxide blanket surface. Molybdenum grades are also lower as a result of factoring historical molybdenum assays. The reduction of tonnage in the Inferred category is related to the infill drilling completed by Hudbay in 2014 and 2015 which resulted in the reclassification of the 2012 inferred and indicated resources to indicated and measured categories in 2017.

## 1.13 Mineral Reserves Estimate

The Mineral Reserves estimate for the Project are based in a LOM which uses the block model compiled under Section 14, Mineral Resource Estimates, with an economic value calculation per block (NSR in $/ton) and mining, processing, and engineering detail parameters. The Mineral Reserves estimate for the Project has been prepared by Hudbay senior mine engineer experts under the supervision of the QP. The mineral reserve economics are described in Section 22, Economic Analysis, of this Technical Report.

This Mineral Reserves estimate has been determined and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in 2014. NI 43-101 defines a Mineral Reserve as "the economically mineable part of measured and indicated mineral resources".

Mine design and reserves estimation for the Rosemont pit use the NSR block model, which consists of an NSR value calculation for each block in the block model, taking into account grade mill recoveries (Cu, Mo and Ag), contained metal in concentrate, deductions and payable metal values, metal prices, freight costs, smelting and refining charges and royalty charges. These parameters were applied to the block model to form the basis of the reserve estimate.

The LG analyses were conducted for the purpose of reporting reserves. The selected pit shell corresponds to a revenue factor of 0.8 (Pit shell 30), which represents metal prices 20% lower than the base case (revenue factor of 1.0 - pit 40). It was selected as the basis for the ultimate pit design, and is approximately 10% smaller than the base economic pit. This pit has better economic indicators in comparison with other pits in terms of free discounted cash flow and total revenue, stripping ratio and capital costs. All LG analyses were restricted to prevent the pit shells from crossing the topographic ridge immediately west of the deposit. This was done due to a permit commitment.

The selected LG pit shell 30 is shown in plan view in Figure 1-2 and in cross-section in Figure 1-3.

**FIGURE 1-2: PLAN VIEW CONTOURS OF SELECTED LERCHS-GROSSMAN PIT SHELL (PIT SHELL 30)**

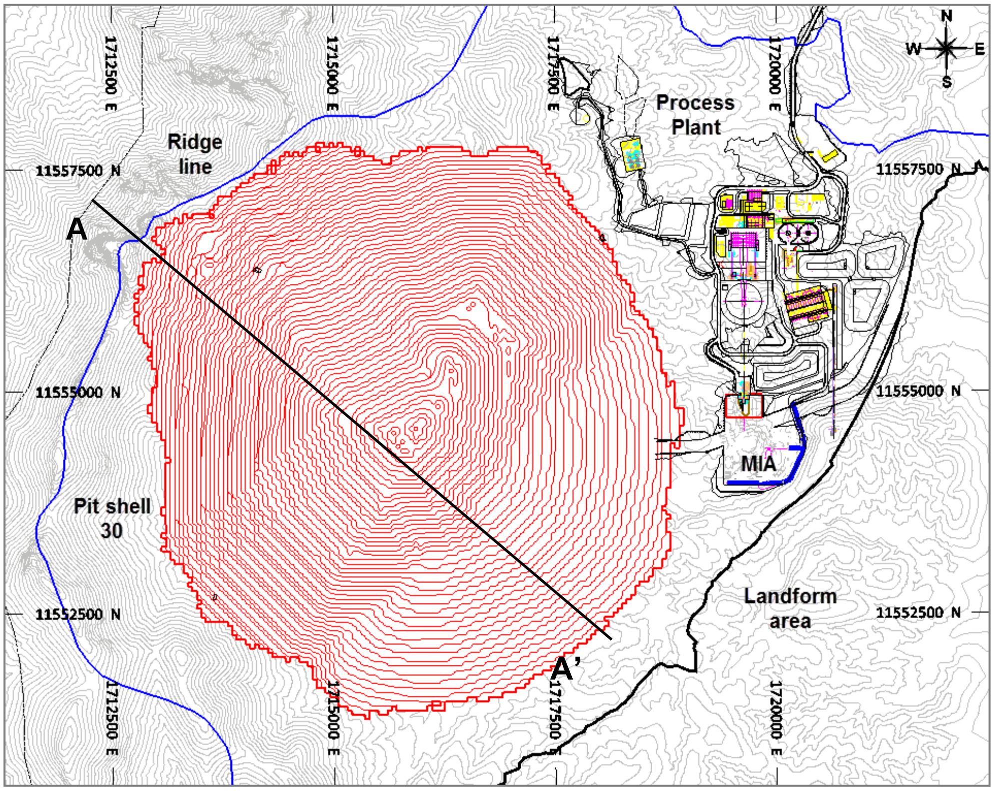
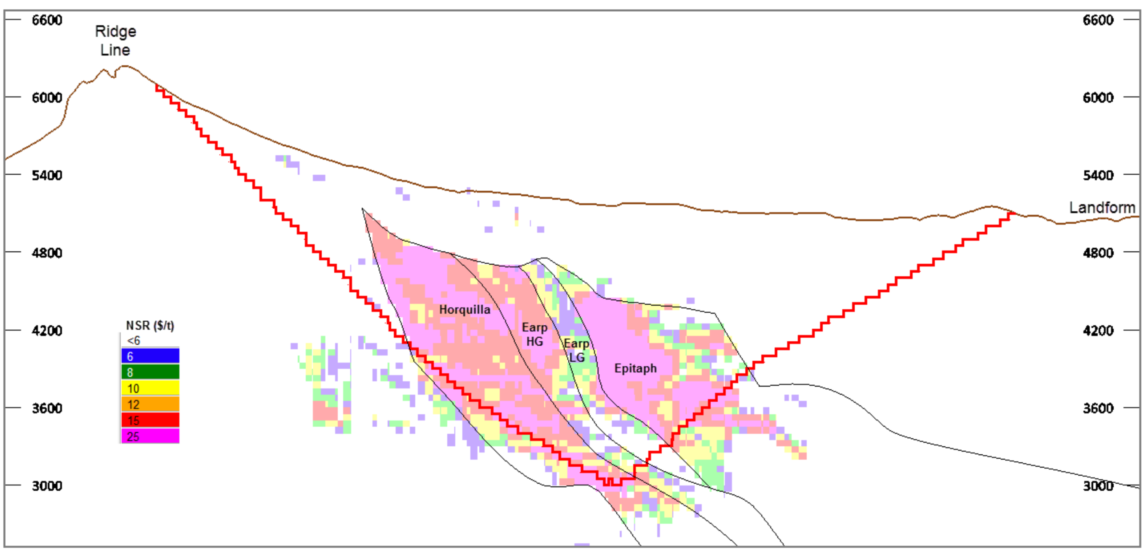



**FIGURE 1-3: AA' SECTION VIEW OF SELECTED LERCHS-GROSSMAN PIT SHELL (PIT SHELL 30)**




The Rosemont mineral reserve estimate is based on measured and indicated resources. Therefore, the potential exists for Inferred Mineral Resources within the ultimate pit to be included and reported as waste, as they currently do not meet the economic and mining requirements to be categorized as

Mineral Reserves. It cannot be assumed that all or any part of Inferred Mineral Resources will ever be upgraded to a higher category.

The selective mining unit ("SMU") dimension in the resource block model is 50x50x50 ft. The interpolated metal grade is averaged for the entire block. When the Project commences operations, ore feed will be delineated by implementing a detailed blasthole sampling program. Drill blast patterns will be smaller, 30 ft to 30 ft, than the resource block dimensions, thereby providing better definition than from the resource model. This new definition will be provided by a new block model built by assays from blasthole projects, dynamic or short-range block model, which is a common practice in Hudbay operations.

### 1.13.1 Mineral Resources and Mineral Reserves Statement

Proven and probable mineral reserves within the designed final pit total are 592 million tons grading 0.45% Cu, 0.012% Mo and 0.13 oz Ag/ton. There are 1.25 billion tons of waste material (including pre-stripping material), resulting in a stripping ratio of 2.1:1 (tons of waste per ton of ore). Total material in the pit is 1.84 billion tons. Contained metal in Proven and Probable mineral reserves is estimated at 5.30 billion pounds of copper, 142 million pounds of molybdenum and 79 million ounces of silver. Proven and Probable mineral reserves for the Rosemont deposit are summarized in Table 1-5.

**TABLE 1-5: PROVEN AND PROBABLE MINERAL RESERVES IN ROSEMONT FINAL PIT**

| | Short Tons | TCu % | SCu % | ASCu % | Mo % | Ag opt | NSR $/ton | CuEq % | Tonnes | Ag (g/t) |
|---|---|---|---|---|---|---|---|---|---|---|
| **Proven** | 469,708,117 | 0.48 | 0.43 | 0.05 | 0.012 | 0.14 | 22.11 | 0.56 | 426,112,017 | 4.96 |
| **Probable** | 122,324,813 | 0.31 | 0.28 | 0.03 | 0.010 | 0.09 | 14.66 | 0.38 | 110,971,199 | 3.09 |
| **Total** | **592,032,930** | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.57** | **0.53** | **537,083,216** | **4.58** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.

Economics of the Mineral Reserves were demonstrated by the mine plan's financial analysis, documented in Section 22 of this Technical Report, which confirmed a 15.6 percent after-tax internal rate of return, based on a copper price of $3.00/lb, silver price of $18.00/oz and molybdenum price of $11.00/lb.

Table 1-6 presents the mineral resource estimates exclusive of the Mineral Reserve estimate, i.e. the mineral resources located inside the resource pit shell but outside of the reserve pit design. The mineral reserve estimate represents the portion of the mineral resource estimates with potential for economic extraction after the current mineral reserves estimate has been mined and processed.

**TABLE 1-6: ROSEMONT MINERAL EXCLUSIVE RESOURCE ESTIMATES (1)(2)(3)(4)(5)(6)(7)(8)(9)**

| | Short Tons | NSR Cut-off | CuEq % | TCu % | Mo (%) | Ag opt | *Tonnes* | *Ag (g/t)* |
|---|---|---|---|---|---|---|---|---|
| **Measured** | | | | | | | | |
| Oxide | 54,000,000 | > = $5.70 | 0.41 | 0.41 | | | *49,000,000* | |
| Mixed | 5,000,000 | > = $5.70 | 0.45 | 0.41 | 0.01 | 0.047 | *4,500,000* | *1.63* |
| Hypogene | 118,700,000 | > = $5.70 | 0.44 | 0.36 | 0.01 | 0.117 | *107,700,000* | *4.01* |
| **Summary** | **177,700,000** | | **0.43** | **0.38** | **0.01** | **0.079** | ***161,200,000*** | ***2.72*** |
| **Indicated** | | | | | | | | |
| Oxide | 18,600,000 | > = $5.70 | 0.27 | 0.27 | | | *16,900,000* | |
| Mixed | 2,600,000 | > = $5.70 | 0.36 | 0.34 | 0.01 | 0.037 | *2,400,000* | *1.27* |
| Hypogene | 392,000,000 | > = $5.70 | 0.31 | 0.25 | 0.01 | 0.080 | *355,600,000* | *2.73* |
| **Summary** | **413,200,000** | | **0.31** | **0.25** | **0.01** | **0.076** | ***374,900,000*** | ***2.60*** |
| **Measured + Indicated** | | | | | | | | |
| Oxide | 72,700,000 | > = $5.70 | 0.38 | 0.38 | | | *66,000,000* | |
| Mixed | 7,600,000 | > = $5.70 | 0.42 | 0.38 | 0.01 | 0.044 | *6,900,000* | *1.50* |
| Hypogene | 510,700,000 | > = $5.70 | 0.34 | 0.27 | 0.01 | 0.088 | *463,300,000* | *3.03* |
| **Summary** | **591,000,000** | | **0.35** | **0.29** | **0.01** | **0.077** | ***536,200,000*** | ***2.64*** |
| **Inferred** | | | | | | | | |
| Oxide | 3,500,000 | > = $5.70 | 0.33 | 0.33 | | | *3,200,000* | |
| Mixed | 1,300,000 | > = $5.70 | 0.47 | 0.45 | 0.00 | 0.019 | *1,200,000* | *0.66* |
| Hypogene | 63,900,000 | > = $5.70 | 0.35 | 0.29 | 0.01 | 0.049 | *58,000,000* | *1.69* |
| **Summary** | **68,700,000** | | **0.35** | **0.30** | **0.01** | **0.046** | ***62,400,000*** | ***1.58*** |

Notes:

1. Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were based on core logging, structural and geochemical data.
2. Raw drill hole assays were composited to 25-foot lengths broken at lithology boundaries.
3. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
4. Block grades for copper, molybdenum and silver were estimated from the composites using OK interpolation into 50 ft x 50 ft x 50 ft blocks coded by domain.
5. Tonnage factors were interpolated by lithology and mineralized zone. Tonnage factors are based on 2,066 measurements collected by Hudbay and previous operators.
6. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
7. Mineral resources are constrained within a computer generated pit using the LG algorithm. Metal prices of $3.15/lb copper, $11.00/lb molybdenum and $18.00/troy oz silver. Metallurgical recoveries of 85% copper, 60% molybdenum and 75% silver were applied to sulfide material. Metallurgical recoveries of 40% copper, 30% molybdenum and 40% silver were applied to mixed material. A metallurgical recovery of 65% for copper was applied to oxide material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, and silver combined. Cut-off grades were set in terms of NSR based on current estimates of process recoveries, total process and G&A operating costs of $5.70/ton for oxide, mixed and sulfide material.
8. The oxide resource will be processed in the mill via flotation.
9. Totals may not add up correctly due to rounding.

### 1.13.2 Comparison with the 2012 Mineral Reserves

A review and comparison of the 2017 Hudbay mineral resource and 2012 Augusta mineral reserves was completed. The results in Table 1-7 of proven and probable reserves show that Hudbay reports a tonnage 11% lower; with copper grades 2% higher, molybdenum grades 17% lower and silver grades 11% higher compared to those estimated in 2012.

**TABLE 1-7: PROVEN AND PROBABLE, COMPARISON TO 2012 AUGUSTA RESERVE ESTIMATE**

| Category | Hudbay Reserves 2017 | | | | Augusta Reserves 2012 Model | | | |
|---|---|---|---|---|---|---|---|---|
| | Tons | Cu (%) | Mo (%) | Ag (opt) | Tons | TCu (%) | Mo (%) | Ag (opt) |
| Proven | 469,708,117 | 0.48 | 0.012 | 0.14 | 308,075,000 | 0.46 | 0.015 | 0.12 |
| Probable | 122,324,813 | 0.31 | 0.010 | 0.09 | 359,131,000 | 0.42 | 0.014 | 0.12 |
| **TOTAL** | **592,032,930** | 0.45 | 0.012 | 0.13 | **667,206,000** | 0.44 | 0.014 | 0.12 |

The changes between the 2012 and 2017 mineral reserve estimates can be mostly attributed to a revision of the mining, processing and general & administration cost assumptions resulting in a marginally higher cut-off in 2017.

## 1.14 Mining Methods

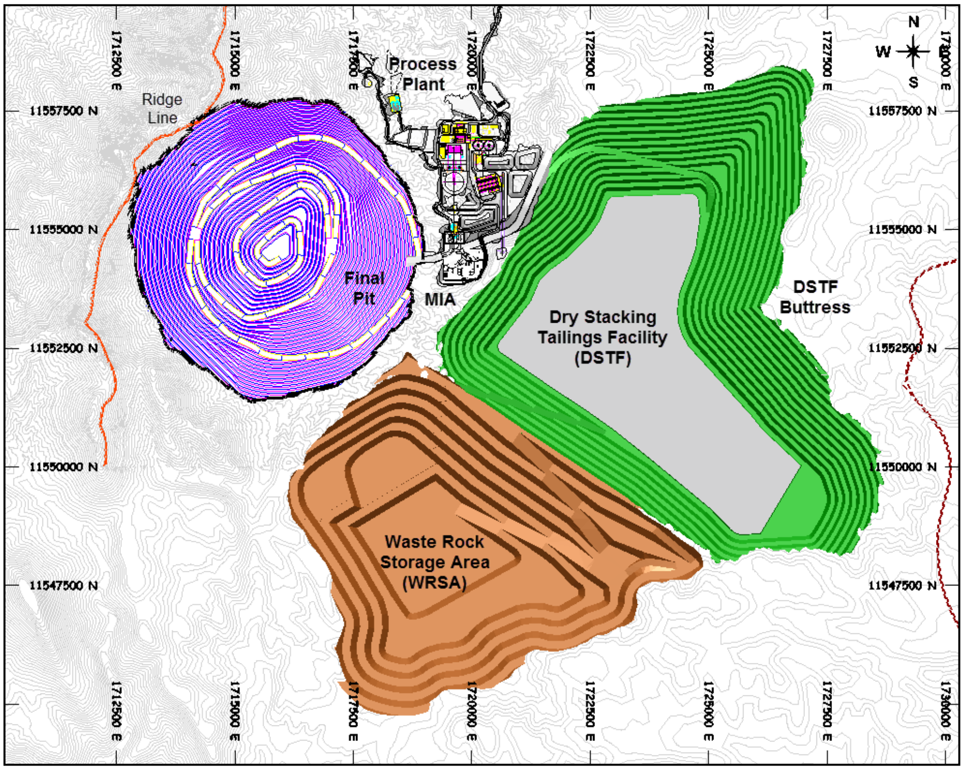
The Rosemont deposit is a high-tonnage, skarn-hosted, porphyry-intruded, copper-molybdenum deposit located in close proximity to the surface. The Project will be a traditional open pit shovel/truck operation. It consists of open pit mining and flotation of sulfide minerals to produce commercial grade concentrates of copper and molybdenum. Payable silver and gold will report to the copper concentrate.

The proposed pit operations will be conducted from 50-foot-high benches using large-scale mine equipment, including: 10-5/8-inch-diameter rotary blast hole drills, 60 $yd^3$ class electric mining shovels, 46 $yd^3$ class hydraulic shovels, 25 $yd^3$ front-end loaders, and 260-ton off-highway haul trucks.

The Rosemont final pit will measure approximately 6,000 feet east to west, 6,000 feet north to south, and will have a total depth of approximately 2,900 feet down to 3,100 feet (AMSL). There is one primary waste rock storage area ("WRSA"), which is located 1,200 feet southeast of the Rosemont final pit. The processing facility is located approximately 1,000 feet east of the final pit, while the dry stacking tailings facility ("DSTF") is located 1,500 feet southeast of the Rosemont pit. The final pit and facilities can be seen in Figure 1-4.

The mine production plan contains 592 million tons of ore and approximately 1.25 billion tons of waste, yielding a life of mine waste to ore stripping ratio of 2.1 to 1 (including pre-stripping material). The mine has a 19-year life, with ore to be delivered to the processing plant at a throughput ramping up to 90,000 tons per day (tpd).

**FIGURE 1-4: ROSEMONT MINE PLAN SITE LAYOUT**




### 1.14.1 Mine Phases

The mine phases and ultimate pit for the Project are designed for large-scale mining equipment (specifically, 60 $yd^3$ class electric shovels and 260-ton haulage trucks) and is derived from the selected LG pit shells described in the previous section. The design process included smoothing pit walls, eliminating or rounding significant noses and notches that may affect slope stability, and providing access to working faces by developing internal ramps (including a dual ramp for the final pit).

For the pit design, the targeted minimum mining width is 320 ft. and honored the wall slope design provided by Call and Nicholas, Inc. ("CNI") and Hudbay. Table 1-8 lists the configuration of the recommended pit slope configuration for each sector.

**TABLE 1-8: ROSEMONT SLOPE GUIDANCE**

| Geotechnical Sector | Bench Height, ft. | Bench Face Angle° | Inter-Ramp Slope Angle° | Catch Bench, ft. | Overall Slope Angle° |
|---|---|---|---|---|---|
| **1** | 100 | 70 | 50 | 48 | 42 |
| **2** | 100 | 65 | 46 | 50 | 40 |
| **3** | 100 | 65 | 48 | 44 | 45 |
| **4** | 100 | 65 | 48 | 44 | 45 |
| **5** | 50 | 65 | 46 | 25 | 43 |
| **6** | 50 | 65 | 44 | 29 | 41 |
| **7** | 50 | 55 | 33 | 42 | 31 |
| **8** | 50 | 55 | 33 | 42 | 31 |

Total ore reserves in the final pit are estimated to be 592 million tons. Approximately 55 million tons of medium and low grade oxide, mixed and sulfide ore will be stockpiled. This material will be reclaimed and processed during operations.

Final configuration of mine phases in plan view is presented in Figure 1-5 and in cross section in Figure 1-6. Mineral reserves for the Rosemont deposit by mine phase are summarized in Table 1-9.

**FIGURE 1-5: PLAN VIEW OF ROSEMONT MINE PHASES**

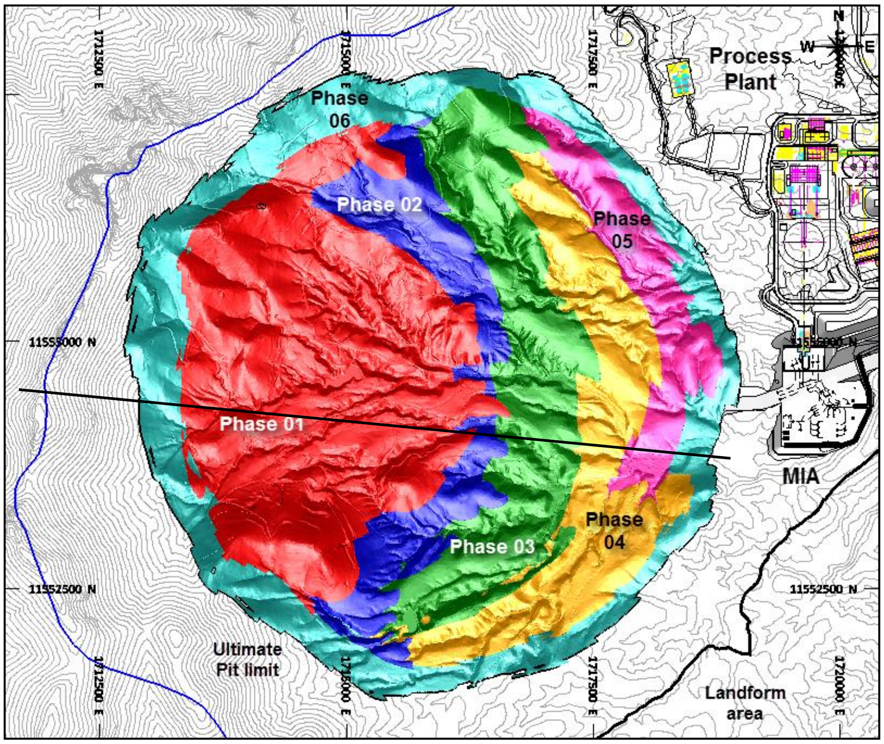



**FIGURE 1-6: AA' SECTION VIEW OF ROSEMONT MINE PHASES**

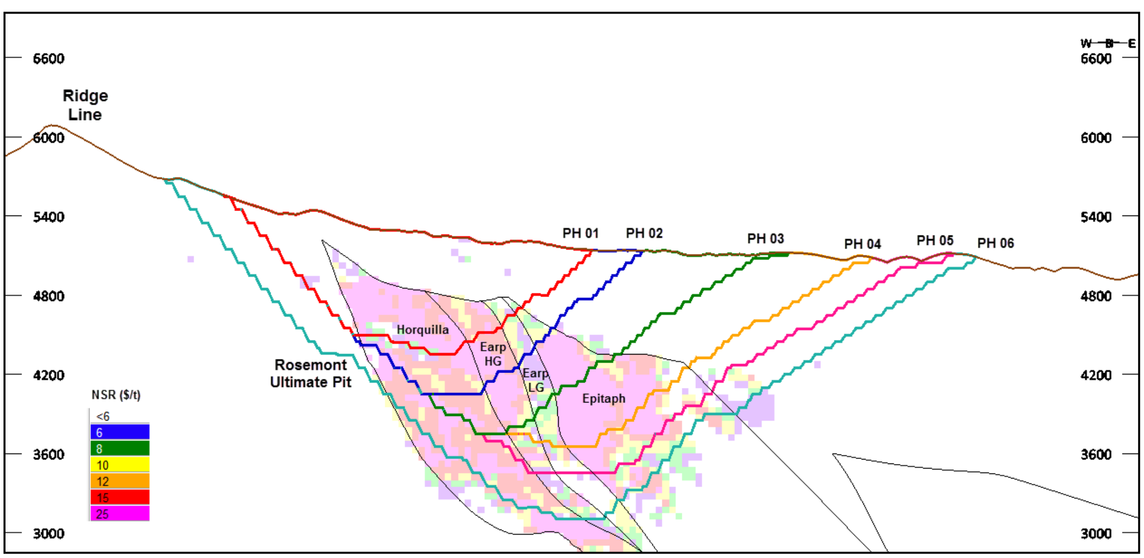


**TABLE 1-9: ROSEMONT MINE PHASES MINERAL RESERVES**

| | Ore M Tons | TCu % | SCu % | ASCu % | Mo % | Ag opt | NSR $/ton | CuEq % | Waste M Tons | Total M Tons | S.R. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| PH01 | 84.8 | 0.49 | 0.43 | 0.06 | 0.011 | 0.16 | 21.80 | 0.57 | 190.3 | 275.1 | 2.24 |
| PH02 | 88.3 | 0.43 | 0.38 | 0.05 | 0.010 | 0.15 | 19.77 | 0.51 | 115.6 | 203.9 | 1.31 |
| PH03 | 74.8 | 0.50 | 0.45 | 0.04 | 0.012 | 0.15 | 23.18 | 0.58 | 177.9 | 252.7 | 2.38 |
| PH04 | 63.5 | 0.53 | 0.50 | 0.03 | 0.014 | 0.13 | 25.26 | 0.62 | 182.5 | 246.0 | 2.87 |
| PH05 | 59.4 | 0.47 | 0.44 | 0.03 | 0.014 | 0.12 | 22.65 | 0.56 | 150.3 | 209.8 | 2.53 |
| PH06 | 221.2 | 0.39 | 0.34 | 0.05 | 0.012 | 0.12 | 17.64 | 0.46 | 431.9 | 653.1 | 1.95 |
| **Total** | **592.0** | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.57** | **0.53** | **1,248.6** | **1,840.6** | **2.11** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.

### 1.14.2 Mine Schedule and Production Plan

The operating and scheduling criteria used to develop the mining sequence plans are summarized in Table 1-10 below.

**TABLE 1-10: MINE PRODUCTION SCHEDULE CRITERIA**

| Parameter | Value |
|---|---|
| Annual Ore Production Base Rate<br>Daily Ore Production Base Rate | 32,850,000 tons<br>90,000 tons |
| Operating Hours per Shift<br>Operating Shifts per Day<br>Operating Days per Week<br>Scheduled Operating Days per Year | 12<br>2<br>7<br>365 |
| Number of Mine Crews | 4 |

Pit operations and mine maintenance will be scheduled around the clock. Allowances for down time and weather delays have been included in the mine equipment and manpower estimations.

A mill ramp up period for concentrator start-up has been considered. Provisions are included to reach a full and steady production (throughput) by the end of the sixth month of operation. This assumption is based on the actual ramp-up achieved by Hudbay in 2016 at the Constancia Project in Peru.

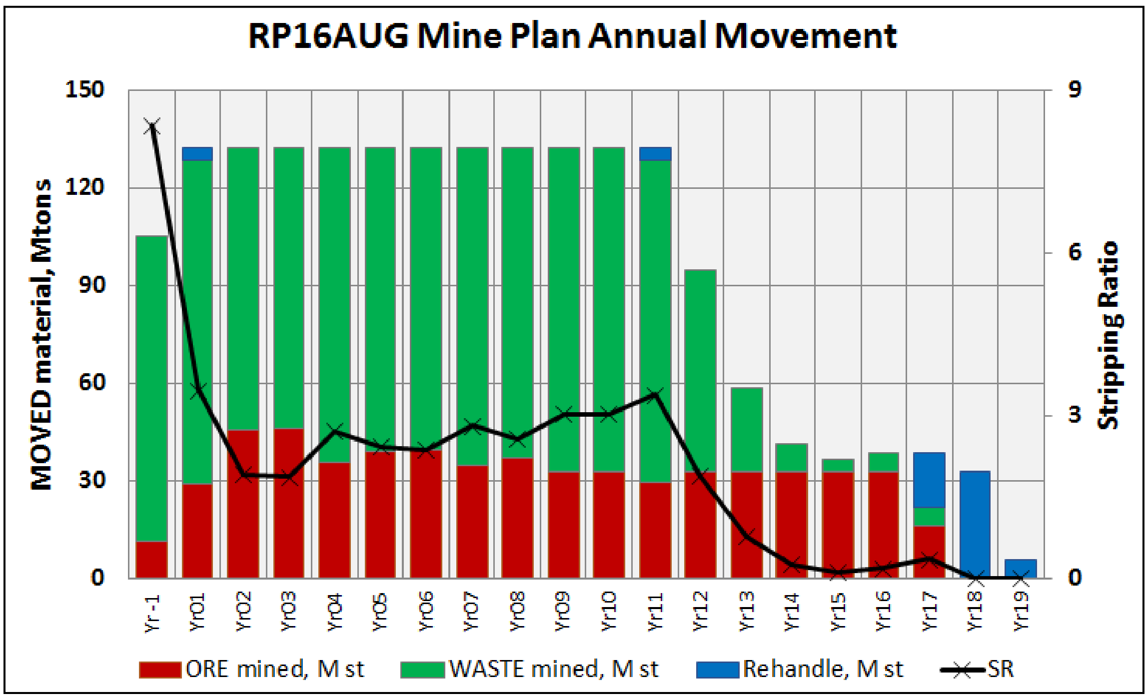
An elevated cut-off grade strategy has been implemented to bring forward a slightly higher-grade ore from the pit to the early part of the ore production schedule. Delivering higher-grade ore to the mill in the early years will improve the net present value and internal rate of return of the Project. NSR values were calculated for each block in the resource model to represent the net Cu, Mo, and Ag metal values. The pit reserves were estimated at a cut-off with an NSR value of $6.00/ton. This is the minimum value of mineralized material that will cover the processing and G&A costs and is therefore reserved for mill feed. Priority plant feed will consist of high grade material (NSR above $12.00/ton). The medium and low grade material (NSR between $6.00/ton and $12.00/ton) will be fed as needed to make up any immediate ore short-fall, but the bulk of this material will be stockpiled.

The stripping analysis determined a minimum preproduction stripping requirement of approximately 94 million tons of waste. Approximately 11 million tons of ore will also be mined and stockpiled during this period.

A mine life of approximately 19 years is projected by this development plan. Peak mining rates of 367,000 tpd of total material will be realized in year 1 through year 11. Average mining rates during years 12-14 will be 180,000 tpd of total material, and will then be reduced to an average of 105,000 tpd from years 15 – 17 as the strip ratio drops.

The estimated mine production schedule, in terms of annual movement of material, is summarized in Figure 1-7.

**FIGURE 1-7: ROSEMONT MINE SCHEDULE, MATERIAL MOVEMENT**




### 1.14.3 Waste Rock Storage Area (WRSA)

Overburden and other waste rock encountered during the course of mining will be placed into a WRSA located to the south and southeast of the planned open pit and within the permitted landform area (i.e., combined WRSA and DSTF). The design criteria for the WRSA and associated haul roads are summarized in Table 1-11 below. The general mine site layout is shown in Figure 1-4.

**TABLE 1-11: WASTE ROCK FACILITY DESIGN CRITERIA**

| Description | Criteria |
|---|---|
| Angle of Repose | 37° |
| Average Tonnage Factor (with swell) | 16.02 $ft^3$/ton |
| Overall Slope Angle | 3.5H:1V |
| Total Height, ft | 600 |
| Lift, ft | 100 |
| Haul Road, ft | 120 |
| Max Elevation, ft (AMSL) | 5,700 |

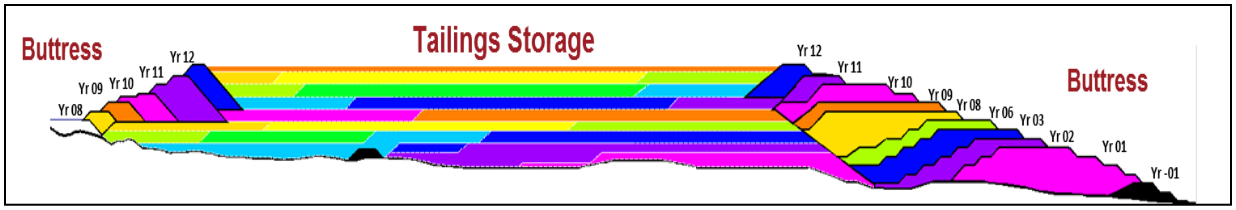
### 1.14.4 Dry StackTailings Facility (DSTF) Buttress

The DSTF is north of the WRSA area and east-northeast of the pit. The DSTF is the repository where processed ore tailings will be placed behind large containment buttresses constructed from mine waste rock. The design criteria for the DSTF and associated haul roads are summarized in Table 1-12 below. The general mine site layout is shown in Figure 1-4 and a N-S cross section view of the DSTF buttress by year is shown in Figure 1-8 below.

**TABLE 1-12: DSTF BUTTRESS ROCK STORAGE DESIGN CRITERIA**

| Description | Criteria |
|---|---|
| Angle of Repose | 37° |
| Average Tonnage Factor (with swell) | 16.02 $ft^3$/ton |
| Overall Slope Angle | 3.5H:1V |
| Total Height, ft | 700 |
| Haul Road, ft | 120 |
| Max Elevation, ft (AMSL) | 5,490 |

**FIGURE 1-8: DRY STACK TAILINGS FACILITY NS SECTION VIEW, LOM BUTTRESS BY YEAR**




### 1.14.5 Mine Equipment

The proposed pit operations will be conducted from 50-foot-high benches using large-scale mine equipment, including: 10-5/8-inch-diameter rotary blast hole drills, 60 $yd^3$ class electric mining shovels, 46 $yd^3$ class hydraulic shovels, 25 $yd^3$ front-end loaders, and 260 ton off-highway haul trucks.

The mine will operate two shifts per day, 12 hours per shift for 365 days a year. No significant weather delays are expected and the mine will not be shut down for holidays. Crew work schedule will consist of a standard four crew rotation.

A summary of fleet requirements by time period for major mine equipment is shown in Table 16-12. This represents the equipment necessary to perform the following mine tasks:

- Mine Site clearing and topsoil salvage and stockpiling.
- Construction of the main haul roads.
- Production drilling.
- Loading and hauling of sulfide ore to the primary crusher (located on the east side of the pit), and waste rock to the WRSA and DSTF buttresses.
- Maintain mine haulage and access roads.
- Maintain WRSA, DSTF and berms, and allow re-grading of slopes and final surfaces for concurrent reclamation.
- Control dust.

### 1.14.6 Mine Manpower Requirements

Mine supervision, technical staff, mine maintenance, workshop personnel and equipment operator requirements over the life of the mine is based on the mine plan. During the Pre-Production period, direct (workshop and operators) and indirect (staff, supervision and technicians) requirements are 337, building up to a peak of 459 in year 7.

Mine staff manpower employees and salaries were developed for Mine Administration, Mine Geology, Mine Operations, and Mine Maintenance. Salaries were a composite of information provided by Hudbay which was calibrated against local mine salaries. Salary information includes wages, burden and bonus for staff employees.

## 1.15 Recovery Methods

The Rosemont process plant is a conventional copper-molybdenum concentrator and its process design is typical of concentrators treating low sulfur copper porphyry-skarn style ores. The process involves crushing, grinding, flotation, concentrate dewatering, molybdenum separation and tailings dewatering.

The process plant design is based on a combination of metallurgical testwork, Project production plan, and in-house information, and is modelled after the Constancia Copper Project design with changes made where necessary to address differences. With minor modifications, the process plant is designed to treat an average of 90,000 tons/d (32.8 million tons/y).

## 1.16 Project Infrastructure

The Project Infrastructure consists of access and plant roads, electric power supply and distribution, water supply and distribution, voice and data communication, and DSTF, and other ancillary facilities.

### 1.16.1 Access and Plant Roads

Access and plant roads consist of an access road into the plant from State Highway 83, in-plant roads, haul roads and a perimeter road around the toe of the WRSA and DSTF. The plant and access roads are shown in Figure 18-1.

### 1.16.2 Power Supply and Distribution

An agreement between Tucson Electric Power (“TEP”), Trico Electric, and Hudbay will be realize to provide the electrical power supply, estimated to be approximately 183 MVA, for the Project. A proposed switchyard (Toro Switchyard) will tap into the existing TEP 138 kV transmission line that extends from the South Substation to the Green Valley Substation. A 13.2-mile-long proposed 138 kV transmission line originates at the Toro Switchyard and terminates on private property to the Rosemont substation as shown in Figure 18-2.

### 1.16.3 Water Supply and Distribution

The fresh water requirement for the Rosemont facilities is approximately 6,000 acre-feet per year. The water supply source identified for the Project is groundwater from the Santa Cruz basin, which lies west of the Project and the Santa Rita Mountains.

There are 4 pump stations located strategically to pump the necessary water to the storage tank located at the mine site. Water from the storage tank will be provided for the following systems:

- potable water system,
- fresh water system,
- process water system, and
- fire water system.

### 1.16.4 Tailings Management

The Rosemont tailings dry stack is designed as a low hazard facility with fully drained waste rock placed as buttressing material. The slope stability analyses performed on the outer slope indicate the dry tailings stack operations can be constructed with stable 3H:1V inter-bench slopes and an overall stable slope of approximately 3.5H:1V. The design was developed based on hydrological and geotechnical studies that included review of regional climate data, drilling and testing programs, and laboratory characterization of subsurface and tailings samples.

An initial starter buttress around the tailings facility will be constructed with waste rock. Concurrent tailings and waste rock placement in the buttress will occur throughout the life of the tailings facility.

### 1.16.5 Communication

The proposed approach is to integrate data networking and telecommunication systems into a common infrastructure to meet the requirements for accounting, purchasing, maintenance, and general office business as well as specialized requirements for control systems. Mobile radios will also be used by the mine and plant operation personnel for daily control and communications while outside the offices.

A security system has been incorporated into the plant network. Using a dedicated video server and monitors, I/P cameras utilizing power over ethernet connections will be plugged into dedicated switches.

## 1.17 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. Rosemont copper concentrates are expected to be a clean, high grade concentrate containing small gold and silver by-product credits which will be suitable as a feedstock for smelters globally. Approximately 50% of the copper concentrate production has been contracted under long term sales contracts.

Table 1-13 below summarizes the key assumptions for the sale of Rosemont's copper concentrate.

**TABLE 1-13: COPPER CONCENTRATE**

| | Units | LOM Total / Average |
|---|---|---|
| Copper Concentrate Base Treatment Charge | $ / dry short ton con | $73 |
| Copper Refining Charge | $ / lb Cu | $0.08 |
| Silver Refining Charge | $ / oz Ag | $0.50 |
| Copper Concentrate Transport & Freight | $ / wet short ton con | $127 |
| LOM Copper Grade in Copper Concentrate | % Total Cu | 34.3% |
| Moisture Content of Copper Concentrate | % $H_2O$ | 8.0% |

No deleterious elements are expected to be produced in quantities which would result in material selling penalties.

Pursuant to a precious metals stream agreement with Silver Wheaton entered into on February 11, 2010, as amended and restated on February 15, 2011, Hudbay will receive deposit payments of $230 million against delivery of100% of the payable gold and silver from the Project . The deposit will be payable upon the satisfaction of certain conditions precedent, including the receipt of permits for the Project and the commencement of construction. In addition to deposit payments, as gold and silver is delivered to Silver Wheaton, Hudbay will receive cash payments equal to lesser of (i) the market price and (ii) $450 per ounce (for gold) and $3.90 per ounce (for silver), subject to one percent annual escalation after three years.

Rosemont is expected to produce a marketable 45% molybdenum concentrate. Table 1-14 below summarizes the key assumptions for the sale of Rosemont's molybdenum concentrate.

**TABLE 1-14: MOLYBDENUM CONCENTRATE**

| | Units | LOM Total / Average |
|---|---|---|
| Molybdenum Concentrate Base Treatment Charge | $ / lb Mo | $1.50 |
| Molybdenum Concentrate Transport & Freight | $ / wet short ton con | $124 |
| LOM Molybdenum Grade in Molybdenum Concentrate | % Mo | 45.0% |
| Moisture Content of Molybdenum Concentrate | % $H_2O$ | 8.0% |

## 1.18 Environmental Studies, Permitting and Social or Community Impact

Permitting status for the Project is well advanced and has continued to progress since July 2007. The final approvals required include the Final Record of Decision (“ROD”) from the U.S. Forest Service (“USFS”) and the 404 Permit from the U.S. Army Corps of Engineers (“USACE”). These final Federal permits are currently in the review process.

Since 2013 when the Final Environmental Impact Statement (“EIS”) and Draft ROD were issued, the USFS has finalized two Supplemental Information Reports (“SIRs”) and a Supplemental Biological Assessment (“SBA”)and completed a consultation with the U.S. Fish and Wildlife Services (“USFWS”) culminating in an Amended Final Biological and Conference Opinion (“BO”) in April 2016. The SIRs determined that nothing disclosed to date would indicate that the information in the SIR falls outside the information disclosed in the EIS. The BO determined that none of the endangered species were jeopardized by the Project.

The USFS is expected to issue its ROD once the USACE is clear on the decision it will make for the Project. This will allow the USFS to include additional analysis into their record and review it against the disclosed impacts in the EIS if the USACE determines it is necessary to make adjustments to their portion of the Project, mitigation, or evaluations. Once the ROD is issued, Hudbay will submit the Mine Plan of Operations (“MPO”) to the USFS for their approval. This approval is expected to take up to six months, and once the MPO is approved, site access is granted.

The USACE is evaluating the overall project record and a mitigation package that provides mitigation for impacts to the ephemeral channels on the Project site. This mitigation incorporates the restoration of a floodplain that was impacted by agriculture; mitigation for two sites impacted by grazing, poor roadway maintenance, and other activities; as well as preservation of sites near to the Project site. Once the USACE evaluation is complete, a decision will be made by the USACE on permit issuance, terms and conditions and appropriate financial assurance will be negotiated.

At this time, the State of Arizona Permits and Approvals dealing with the environment have been issued for the Project, and all permits remain in force and are current. The Project continues to comply with permit terms and conditions. No additional environmental permits are necessary to begin construction of the facilities, and only minor environmental permits (e.g., septic system permits, water system approvals and registration) will be needed during construction.

The Project refinements included in this document were specifically designed and evaluated to fall within the envelope included in the EIS review and as such are not expected to cause concern by the agency. State of Arizona permits that were issued based on early designs will be amended to include designs included in the EIS. Such amendments are customary in the state of Arizona.

## 1.19 Capital and Operating Cost

Initial project capital costs are estimated to be $1,921 million including 15% contingency on all items. The LOM sustaining capital costs are estimated to be $387 million excluding capitalized stripping and $1,168 million including capitalized stripping. The capital cost estimate is considered to be a Class 3 estimate as defined by AACE Recommended Practice 47R-11 for the mining and mineral process industry.

The average LOM operating costs (mining, milling and G&A) are estimated to be $9.24/ton milled (before deducting capitalized stripping) and $7.92/ton milled (after deducting capitalized stripping). Refer to Section 21 for greater capital and operating cost detail.

Over the first 10 years, C1 cash costs (net of by-product credits at stream prices) are estimated to average $1.40 per pound of copper before deducting capitalized stripping and $1.14 per pound of copper after deducting capitalized stripping. LOM C1 cash costs are estimated to be $1.47 per pound of copper before deducting capitalized stripping and $1.29 per pound of copper after deducting capitalized stripping. Including royalties and sustaining capital, sustaining cash costs are estimated to be $1.59 per pound of copper over the first 10 years and average $1.65 over the LOM.

## 1.20 Economic Analysis

The economic viability of the Project has been evaluated using the metal prices outlined in Table 1-15. The metal prices used in the economic analysis are based on a blend of consensus metal price forecasts from over 30 well-known financial institutions and Wood Mackenzie.

**TABLE 1-15: METAL PRICE ASSUMPTIONS**

| Metal | Units | Price |
|---|---|---|
| Spot Copper | $/lb | $3.00 |
| Spot Molybdenum | $/lb | $11.00 |
| Spot Silver | $/oz | $18.00 |
| Streamed Silver[1] | $/oz | $3.90 |

1. Subject to a 1% annual inflation adjustment

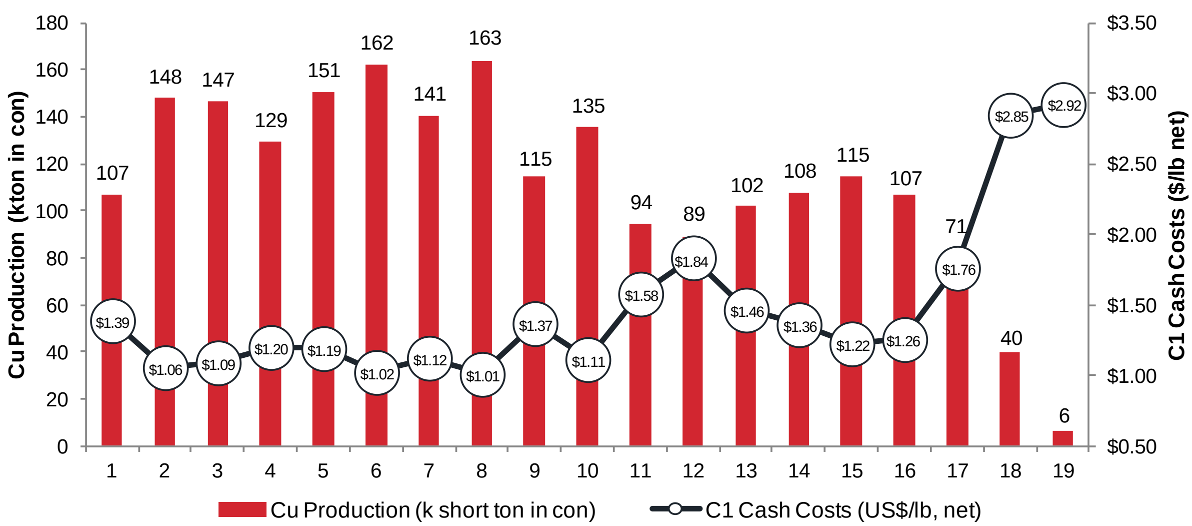
The terms of the existing precious metals streaming agreement with Silver Wheaton were included in the analysis. Silver Wheaton will make upfront cash payments totalling $230 million to fund initial development capital in exchange for 100% of the silver and gold production from Rosemont. Silver Wheaton will make ongoing payments of $3.90 per ounce of silver and $450 per ounce of gold subject to a 1% inflation adjustment starting on the third anniversary of production.

Although gold is not part of the current reserve estimate, metallurgical testing has demonstrated economic concentrations of gold in copper concentrate as outlined in Section 13. Over the LOM, approximately 309 thousand ounces of gold are expected to be recovered in copper concentrate (although the financial impact has not been included).

At the effective realized prices including the impact of the stream, the revenue breakdown at Rosemont is approximately 92% copper, 6% molybdenum, and 2% silver.

Rosemont's annual copper production (contained copper in concentrate) and C1 cash costs (net of by-products at stream prices after deducting capitalized stripping) are shown below in Figure 1-9. Over the first 10 years, annual production is expected to average 140 thousand tons of copper at an average C1 cash cost of $1.14/lb. Over the 19-year LOM, annual production is expected to average 112 thousand tons of copper at an average C1 cash cost of $1.29/lb.

**FIGURE 1-9: ROSEMONT ANNUAL COPPER PRODUCTION AND C1 CASH COSTS**




Rosemont (on a 100% basis) has an unlevered after-tax NPV8% of $769 million and a 15.5% after-tax IRR using a copper price of $3.00/lb as summarized in Table 1-16. The Project NPV and IRR are calculated using end of period quarterly discounting in the quarter immediately before development capital is spent.

**TABLE 1-16: LIFE OF MINE FINANCIAL METRICS (100% PROJECT BASIS)**

| Metric | Units | LOM Total |
|---|---|---|
| Gross Revenue (Stream Prices) | $M | $13,377 |
| TCRCs | $M | ($1,837) |
| On-Site Operating Costs (After Deducting Capitalized Stripping) | $M | ($4,691) |
| Royalties | $M | ($368) |
| **Operating Margin** | **$M** | **$6,480** |
| Development Capital | $M | ($1,921) |
| Stream Upfront Payment | $M | $230 |
| Sustaining Capital (excludes capitalized stripping) | $M | ($387) |
| Capitalized Stripping | $M | ($781) |
| **Pre-Tax Cash Flow** | **$M** | **$3,622** |
| Cash Income Taxes | $M | ($718) |
| **After-Tax Free Cash Flow** | **$M** | **$2,903** |
| After-Tax NPV8% | $M | $769 |
| After-Tax NPV10% | $M | $496 |
| After-Tax IRR | % | 15.5% |
| After-Tax Payback Period | Years | 5.2 |

The NPV8% (100% Project basis) was sensitized based on percentage changes in various input assumptions above or below the base case. Each input assumption change was assumed to occur independently from changes in other inputs. The sensitivity analysis is summarized in Figure 1-10. The Project is most sensitive to the copper price, followed by initial capital costs, on-site operating costs, and the molybdenum price.

**FIGURE 1-10: NPV8% SENSITIVITY (100% BASIS)**

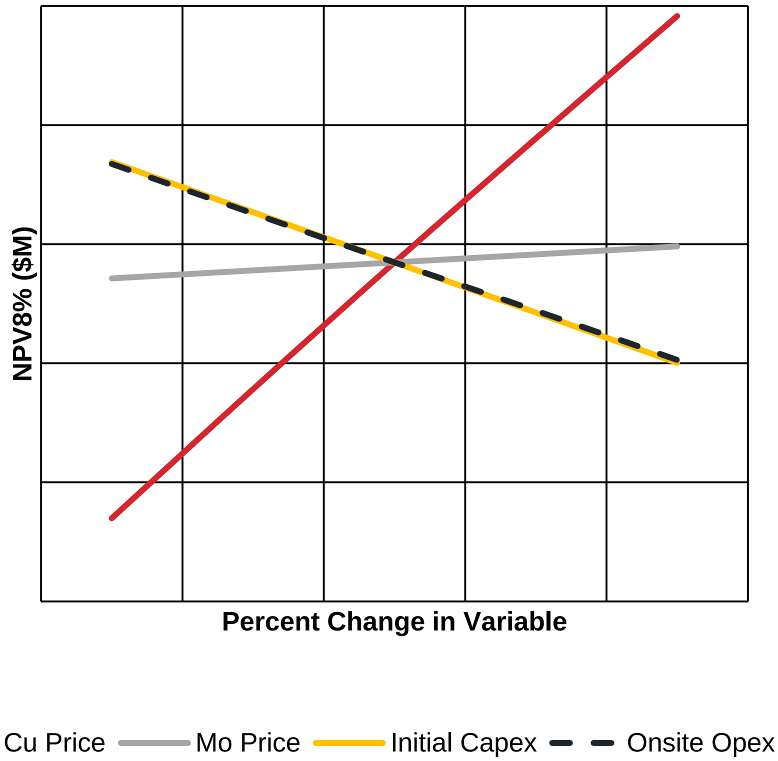



Table 1-17 below reports the after-tax NPV8%, NPV10%, IRR and Payback of the Project at various flat copper prices assuming all other inputs remain constant.

**TABLE 1-17: AFTER-TAX NPV8%, NPV10%, IRR AND PAYBACK SENSITIVITY AT VARIOUS FLAT COPPER PRICES (100% BASIS)**

| | Flat Copper Price ($/lb) | | | | |
|---|---|---|---|---|---|
| | $2.50 | $2.75 | **$3.00** | $3.25 | $3.50 |
| After-Tax NPV8% ($M) | $45 | $412 | **$769** | $1,115 | $1,448 |
| After-Tax NPV10% ($M) | ($122) | $192 | **$496** | $792 | $1,076 |
| After-Tax IRR (%) | 8.5% | 12.2% | **15.5%** | 18.5% | 21.2% |
| After-Tax Payback (years) | 6.9 | 5.9 | **5.2** | 4.4 | 4.3 |

The existing Joint Venture Agreement requires cash payments from UCM totaling $106 million to the Joint Venture (“JV”) in order for UCM to complete its earn-in for 20% ownership of the Project. The payments will be made on an installment basis to fund the initial development capital and payments will commence once certain milestones are achieved. The NPV attributable to Hudbay is improved beyond 80% of the standalone project NPV due to the JV payments, and the IRR attributable to Hudbay is improved beyond the standalone project IRR as a result of the reduced time period between development capital spending and positive project cash flow. Table 1-18 shows the adjusted key financial metrics attributable to Hudbay.

**TABLE 1-18: KEY FINANCIAL METRICS ATTRIBUTABLE TO HUDBAY**

| Metric | Units | LOM Total |
|---|---|---|
| Development Capital (100% Basis) | $M | $1,921 |
| Stream Upfront Payment | $M | ($230) |
| Joint Venture Earn-in Payment | $M | ($106) |
| JV Share of Remaining Capital (20%) | $M | ($317) |
| JV Loan Repayment to Hudbay[1] | $M | ($20) |
| **Hudbay's Share of Development Capital** | **$M** | **$1,248** |
| After-Tax NPV8% to Hudbay | $M | $719 |
| After-Tax NPV10% to Hudbay | $M | $499 |
| After-Tax IRR to Hudbay | % | 17.7% |
| After-Tax Payback Period to Hudbay | Years | 4.9 |

1. Hudbay is funding the JV's share of project expenditures until receipt of material permits and approximately $20M in principal and accrued interest is due to Hudbay

## 1.21 Adjacent Properties

There is no material information concerning mineral properties immediately adjacent to the Project.

## 1.22 Other Relevant Data and Information

The author is not aware of any other information that would impact the reported estimate of mineral resources for the Project.

A draft feasibility study was completed for the Project which included information on the basis of design, infrastructure, design strategies, Project Execution Strategy, risks assessments and recommendations. The EPCM team has also completed a draft construction execution plan.

The Project has undergone various risk assessments and workshops during the years and continues to hold quarterly risk assessment workshops.

## 1.23 Conclusions

The purpose of this Technical Report is to present Hudbay's estimate of the mineral reserves and mineral resources for the Project based on the current mine plan, the current state of metallurgical testing, operating cost and capital cost estimates. The results of "feasibility study" level work conducted partly by external contractors and partly internally by Hudbay, completion of the drill program and bench-marking against other mines including Hudbay's Constancia mine has resulted in the following fundamental conclusions:

- The Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn formed on a chemical/siliciclastic sedimentary sequence after the intrusion of Laramide quartz monzonite porphyry intrusions.

- A new geological model was built based on chemostratigraphy and lithogeochemistry from 33,000 samples covering the full footage of the 2014 and 2015 Hudbay drilling programs. Geochemical based geological model reduces uncertainties in the formational, lithological and alteration logging. The updated geological model incorporated a revised structural framework based on a surface mapping and downhole structural review.
- The resource economic parameters utilized are slightly different than those supporting the mineral reserve statement. The cost and price inputs for the mineral resource economic parameters are considered an approximation and were used to test the economic viability of the resource. Although these costs and prices differ from the ones used for the mineral reserves, it is the opinion of the QP that changing the resource parameters would not materially change the output of the reserve.
- A proven and probable reserve of 592 million tons has been identified and its economic viability demonstrated. An additional 591 million tons of measured and indicated resources and 69 million tons of inferred resources have been identified and outlined as having further potential for economic extraction once the mineral reserves have been extracted.
- Mining equipment performance and maintenance requirements and costs are benchmarked from Constancia's actual operating information and are robust.
- Metallurgical testwork has confirmed that the Rosemont ores respond well to proven and widely used sulfide mineral processing techniques.
- The tailing properties have been sufficiently characterized as well as the dewatering performance of vendor equipment over the life of the operation to satisfy the estimated number, type and size of tailing filters for this Project. To be conservative, expansion space has been allocated for additional filtering equipment, to the extent that it may be necessary.
- The Rosemont Process Plant design has been modelled after the operating Constancia processing plant's flow sheet and is sufficiently robust to routinely achieve key production targets such as throughput, recovery, and concentrate grade as stated in the production schedule based on the metallurgical testing conducted.
- Flexibility exists within the mine plan to optimise plant recovery and performance through the management of feed types including clays, oxides and hardness.
- The Project is one of many large projects scheduled to be constructed. The author believes that schedule slip will be the principal pressure on cost should the Project experience construction delays. In the opinion of the author, the contingency allotted to construction capital cost should mitigate most cost risk. At the time of publication of this Technical Report, committed and spent dollars would raise this contingency to approximately 15% of the required project capital as estimated in the draft Detailed Feasibility Study (dDFS).

- Related to the capital costs, M3 Engineering and Amec Foster Wheeler (acting under a joint venture agreement) and Ausenco each completed a value engineering phase and independently produced capital estimates for the Project which were within 5% of each other. Since then, Ausenco has also completed a feasibility quality capital estimate, benchmarked their findings with Constancia (and other similar projects), and engaged third party construction contractors and various consulting firms to provide additional input into the estimate. Hudbay has also completed an independent third party review of the feasibility study estimate.
- The net present value of the project is most sensitive to the price of copper. The resulting project NPV8% ($769 million) and IRR (15.5%) utilizing the current Hudbay long term view on metal prices, TCRC's and other economic assumptions, in the opinion of the author, support the declaration of mineral reserves as outlined in the CIM guidelines.
- The project execution plan is modelled after the Constancia delivery method. Many key personnel who developed the Constancia mine are members of the Project who will be involved in the development of the Project and therefore considered a robust project execution plan.

The Project is uniquely located in a copper mining jurisdiction that has sustained economic copper production for close to 140 years. Since it is located approximately 30 miles from Tucson it is expected to have a significant impact on employment and economic gain for the region. The proposed mining, processing, and logistics plan provides a step forward in innovation and sustainability. The dry stack tailings deposition proposed would be among the largest in size and address industry and stakeholder concerns regarding the use of water and the stability of tailings impoundment facilities. The proposed design and operating practice that will be applied in respect of the Project is expected to set a new standard by which other large mining projects are judged with respect to their impact on stakeholders, the ecology and the environment.

In recognition of the scarcity of world economic copper reserves in an environment of ever increasing consumption of the metal, Hudbay has carefully considered the ecological, environmental, and ethical extraction methods to be applied to the Project in an effort to set it apart from others in the world. The Project is located in a first world leading nation, where extraction and production is governed by laws with due process and human rights fundamental to the consumer.

This Technical Report also concludes that the estimated mineral reserves and mineral resources for the Project conform to the requirements of 2014 CIM Definition Standards – for Mineral Resources and Mineral Reserves and requirements in Form 43-101F1 of NI 43-101, Standards of Disclosure.

## 1.24 Recommendations

The Author recommends the following:

- The author recommends that Hudbay further investigate the cause(s) of the differences in average molybdenum grade of the historical assays. Hudbay should also evaluate the application of non-linear interpolation or wireframing methods in the minor geological units.

- A drill hole twinning program of the pre-Augusta drilling to bolster the confidence of the re-assaying campaign as conducted by Augusta.
- Further investigate change-of-support correction and alternative approaches to resource classification taking into account the high production rate. This should be performed to ensure that the resource classification properly reflects the reduced risk when a large volume is mined and delivered to the mill on a quarterly and annual basis.
- It is recommended by the author that 5% of the samples should be sent for check assay in future drill hole campaigns.
- Future drilling campaigns should consider the by-product credit contribution of gold noted contained in copper concentrate produced through testing and increase the confidence in geological continuity such that it can be included in the reserve statement.
- Metallurgical test-work has confirmed that marketable copper concentrates can be produced. Mine and mill production sequencing and planning will require care to manage clay content that can adversely affect flotation and tailings filtration.
- A geometallurgical program is recommended as a component of the operating plan to further refine, monitor and optimise the mine to mill performance. It is concluded that fluorine can be readily rejected from copper concentrate, and further study is recommended to develop understanding of ore conditions and indicators that trigger elevated fluorine content in concentrate.
- Ongoing optimization of pit slope designs should be conducted during operation and should be based on observed conditions accounting for more detailed mapping of local alteration, jointing and faulting. These observations and compilation can then provide the basis for revised slope geometry and pushback (mining phase) configurations that have the potential to increase mineral reserves and reduce overall stripping requirements.

# 2 INTRODUCTION AND TERMS OF REFERENCE

The Principal Author has prepared this Feasibility-Level Technical Report for Hudbay Minerals Inc. ("Hudbay") on the Project, located approximately 30 miles (50 km) southeast of Tucson, in Pima County, Arizona. This Technical Report conforms with the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and requirements in Form 43-101F1 of NI 43-101, Standards of Disclosure for Mineral Projects.

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 acquired all of the issued and outstanding common shares of Augusta Resource Corporation ("Augusta") pursuant to take-over bid, which expired July 29, 2014, and a subsequent acquisition transaction, which closed on September 23, 2014.

This Technical Report describes the latest resource model and mine plan and the current state of metallurgical testing, operating cost and capital cost estimates. The information presented in this Technical Report is the result of "feasibility study" level work conducted partly by external contractors and partly by internal Hudbay personnel under the overall supervision of the Qualified Person ("QP").

The QP who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo, Senior Vice President & Chief Operating Officer for Hudbay. Mr. Meagher last visited the property on April 21, 2016 and numerous times prior to this date. The personal site inspections were conducted as part of the 2014-2015 diamond drilling program to become familiar with conditions on the property, to observe the geology and mineralization and to verify the work completed on the Property. Mr. Meagher has also reviewed and conducted sufficient confirmatory work to act as QP for the reporting of the mineral resource and mineral reserve estimates for the Project.

## 2.1 Information Sources

Information used to support this Technical Report was based on current primary-source data when available, previous technical reports on the property and from the reports and documents listed in Section 27, References. Notable information reviewed and relied upon by the QP was as follows:

- **3D Block Model** – Hudbay prepared a 3D block model of the Rosemont deposit. The 3D block model and determination of the mineral resources at the Rosemont deposit were performed by internal Hudbay employees, following Hudbay procedures and were reviewed and approved by Cashel Meagher, Chief Operating Officer for Hudbay and Qualified Person of this Technical Report.

- **Lerchs-Grossman Analyses** – A Lerchs-Grossman ("LG") cone algorithm was applied by Hudbay to the block model to establish the component of the deposit that has a "reasonable prospect of economic extraction".

Additional sources of information that the QP relied upon are described in Section 3 of this Technical Report.

## 2.2 Unit Abbreviations

The units of measure in this report are a combination of US standard units and metric units. Unless stated otherwise, all dollar amounts ("$") are in United States dollars. Unit abbreviations used in this report are noted below:

**TABLE 2-1: UNIT ABBREVIATIONS**

| Abbreviation | Description |
|---|---|
| $ | United States dollar |
| °C | degree Celsius |
| °F | degree Fahrenheit |
| % | percent |
| μm | microns |
| cm | centimetres |
| ft | feet |
| $ft^2$ | Square feet |
| g | gram |
| g/mt | grams per (metric) tonne |
| HP | horsepower |
| km | Kilometre |
| kV | Kilovolt |
| kW | kilowatt |
| kWh | Kilowatt-hour |
| $m^2$ | Square meter |
| $m^3$ | Cubic meter |
| mm | Millimetres |
| tonne | metric tonne |
| Mt | Million (short) tons |
| ppb | parts per billion |
| ppm | parts per million |
| t, ton | short ton |
| w/w | Weight per weight |

## 2.3 Name Abbreviations

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

**TABLE 2-2: NAME ABBREVIATIONS**

| Abbreviation | Description |
|---|---|
| 3D | Three-Dimensional |
| AAS | Atomic Absorption Spectrometry |
| ADWR | Arizona Department of Water Resources |
| Ag | Silver |
| AMSL | Above mean seal level |
| ASCu | Acid soluble copper |
| Augusta | Augusta Resource Corporation |
| AV | Average |
| BADCT | Best Available Demonstrate Control Technology |
| BQ | BQ drill core size 1.43 inches or 36.4mm |
| Bureau Veritas | Bureau Veritas Commodities Canada Ltd. |
| CBV | Certified best value |
| CEC | Cation Exchange Capacity |
| CIM | Canadian Institute of Mining, Metallurgy and Petroleum |
| Cu | Copper |
| Cu-Mo | Copper-molybdenum |
| CRM | Certified reference materials |
| CV | Coefficient of Variance |
| DGM | Discrete Gaussian Model |
| DSTF | Dry Stack Tailings Facility or Tailings Management Facility (TMF) |
| EDX | Energy Dispersive X-ray |
| EGL | Equivalent Grinding Length |
| EIS | Environmental Impact Statement |
| EPMA | Electron Probe Micro-analysis |
| FEL | Front End Loader |
| FileMaker | FileMaker Inc. |
| GMD | Gearless Motor Drive |
| GT | Grade-tonnage |
| $H_2SO_4$ | Sulfuric acid |
| HCT | High Compression Thickeners |
| HQ | HQ drill core size 2.50 inches or 63.5 mm diameter |
| Hudbay | Collectively all Hudbay Minerals Inc. subsidiaries and business groups |
| ICP | Inductively Coupled Plasma |
| ICP-MS | Inductively Coupled Plasma Mass Spectrometry |
| ICP-OES | Inductively Coupled Plasma Optical Emission Spectroscopy |
| ID2 | Inverse Distance Squared |
| Inspectorate | Inspectorate America Corporation |
| LAF | Low Angle Fault |
| LG | Lerchs-Grossman |
| MPO | Mine Plan of Operations |
| Mo | Molybdenum |
| NaHS | Sodium Hydrosulfide |
| NEPA | National Environmental Policy Act |
| NI | National Instrument |
| NN | Nearest Neighbour |
| NQ | HQ drill core size 1.875 inches or 47.6 mm diameter |
| NSR | Net Smelter Return |
| OK | Ordinary Kriging |
| OSA | On-Stream Analyser |
| OREAS | Ore Research and Exploration |
| PQ | PQ drill core size 3.3 inch or 83 mm diameter |
| QA/QC | Quality Assurance and Quality Control |
| QUEMSCAN | Quantitative Evaluation of Minerals by Scanning electron microscopy |
| $R^2$ | Coefficient of Determination |
| RE | Absolute relative error |
| RMA | Reduced-to-Major-Axis regression |
| ROD | Record of Decision |
| RQD | Rock Quality Designation |
| RSE | Relative standard error of the kriged estimate |
| RSD | Relative standard deviations |
| SAG | Semi-Autogenous Grinding |
| SABC | SAG and Ball Mill and Crushing Comminution Circuit |
| SBA | Supplemental Biological Assessment |
| SCu | Copper in sulfides |
| SD | Standard deviation |
| SEM | Scanning Electron Microscopy |
| SFR | Staged Flotation Reactor |
| SG | Specific Gravity |
| SGS | SGS Canada Inc. |
| SIR | Supplemental Information Report |
| Skyline | Skyline Assayers & Laboratories |
| SMU | Selective mining unit |
| SRM | Standard reference materials |

| Abbreviation | Description |
|---|---|
| TCu | Total copper |
| TEP | Tucson Electric Power Company |
| TIA | Tucson International Airport |
| TRICO | TRICO Electric Cooperative Inc. |
| UCM | United Copper & Moly LLC |
| USFS | U.S. Forest Service |
| USFWS | U.S. Fish and Wildlife Service |
| USACE | U.S. Army Corps of Engineers |
| XPS | XPS Consulting & Testwork Services |
| XRD | X-Ray Diffraction |

# 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.

Hudbay has retained a number of contractors/consultants to prepare technical and cost information to support this Technical Report. All the information used from this work in the current Technical Report has been duly verified and validated by the author.

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

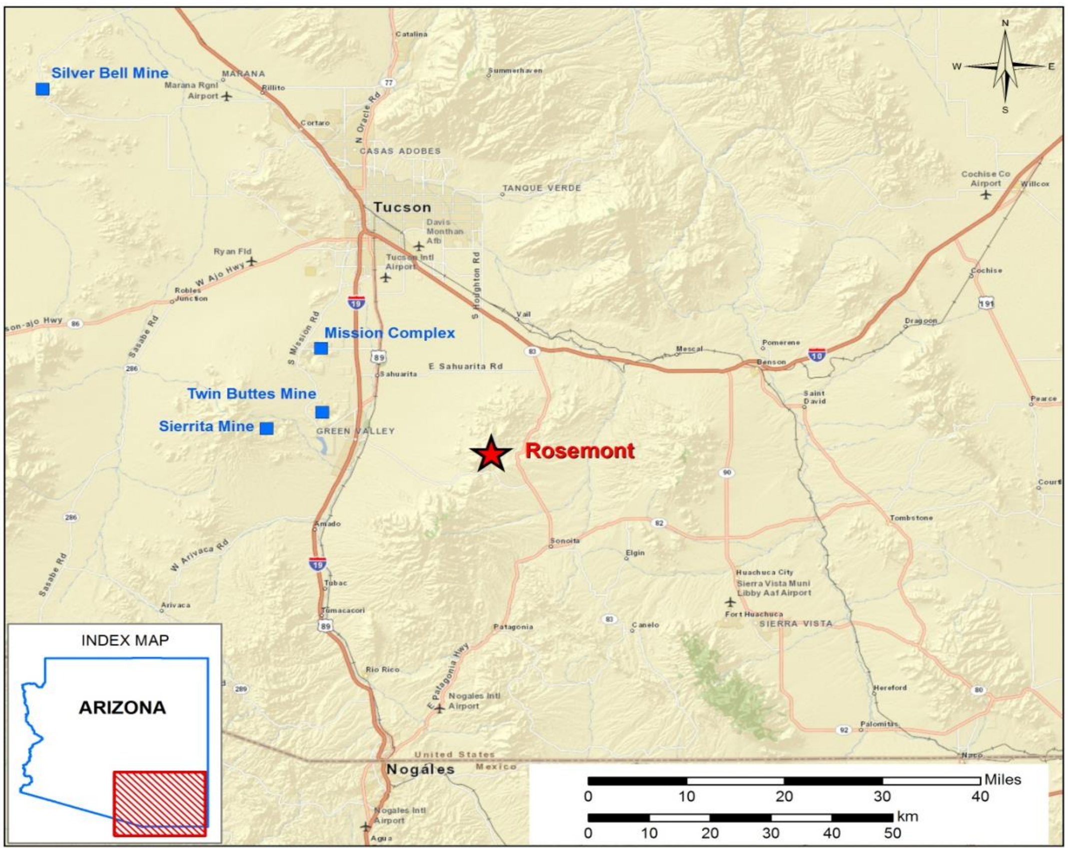
- Information available to Hudbay at the time of preparation of this Technical Report,
- Assumptions, conditions, and qualifications as set forth in this Technical Report, and
- For purposes of this Technical Report, the author has relied on title and property ownership information based on select records of the U.S. Bureau of Land Management and the Assessor's Office in Pima County, Arizona.
- The author has also relied on tax information provided by Hudbay's tax department, Social and Environmental information as provided by appropriate Arizona based personnel from Hudbay, marketing information from Hudbay's marketing group and forward price forecasts from Hudbay's treasury department.

# 4 PROPERTY DESCRIPTION AND LOCATION

## 4.1 Location

The Project is located within the historic Helvetia-Rosemont Mining District that dates back to the 1800's. The deposit lies on the eastern flanks of the Santa Rita Mountain range approximately 30 miles (50 km) southeast of Tucson, in Pima County, Arizona off of State Route 83 (see Figure 4-1). The core land position includes patented and unpatented mining claims, fee land and grazing leases that cover most of the old Mining District. The lands are under a combination of private ownership by Rosemont and Federal ownership. The lands occur within Townships 18 and 19 South, Ranges 15 and 16 East, Gila & Salt River Meridian. The Project geographical coordinates are approximately 31º 50'N and 110º 45'W.

**FIGURE 4-1: PROPERTY LOCATION OF ROSEMONT PROJECT**




## 4.2 Land Tenure

Hudbay acquired all of the issued and outstanding common shares of Augusta Resource Corporation pursuant to take-over bid, which expired July 29, 2014, and a subsequent acquisition transaction, which closed on September 23, 2014. Hudbay's ownership in the Project is subject to an earn-in agreement with United Copper & Moly LLC (''UCM''), pursuant to which UCM has earned a

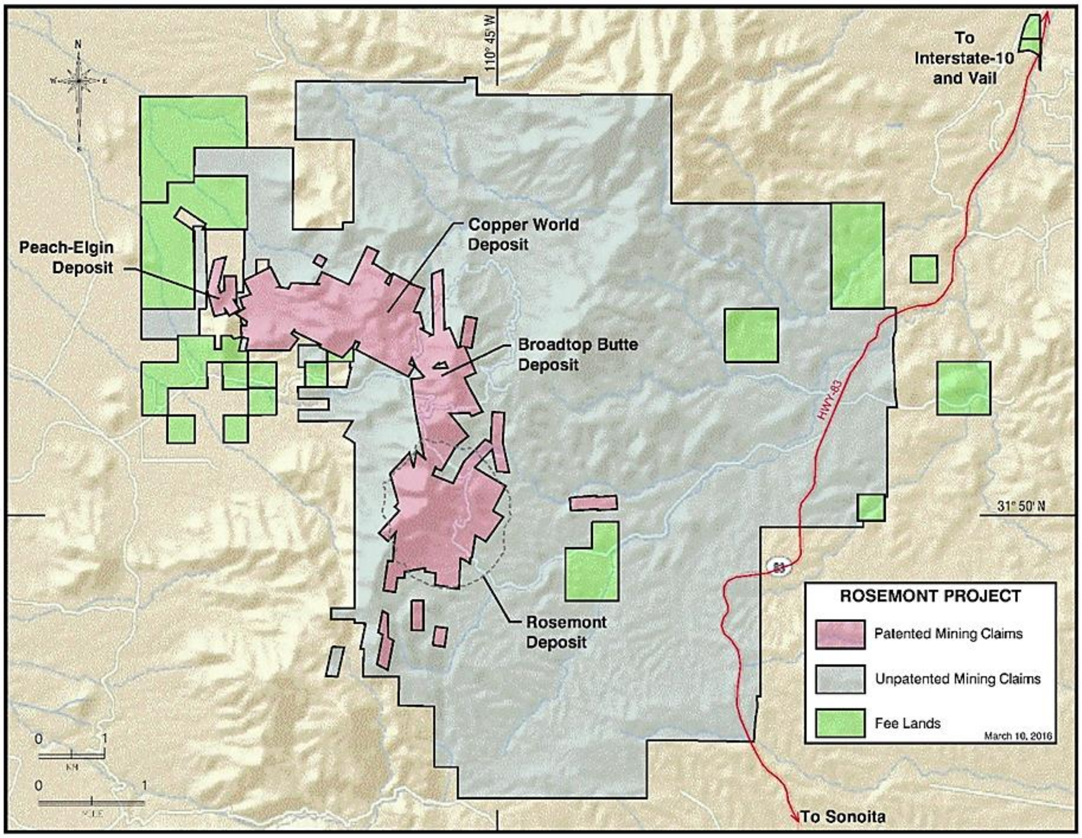
7.95% interest in the Project and may earn up to a 20% interest subject to cash payments from UCM totaling $106 million to the Joint Venture. A joint venture agreement between Hudbay's subsidiary, Rosemont Copper Company, and UCM governs the parties' respective rights and obligations with respect to the Project.

Hudbay continues to maintain the property in good standing which consists of a combination of fee land, patented and unpatented lode, mill site mining claims, and rights-of-way from the Arizona State Land Department. Taken together, the land position is sufficient to allow access to an open pit mining operation, processing and concentrating facilities, storage of tailings, disposal of waste rock and a utility corridor to bring water and power to site. The Federal lands covered by unpatented mining claims are accessible under the provisions of the Mining Law of 1872, subject to approval from the U.S. Forest Service ("USFS") after the completion of an Environmental Impact Statement ("EIS") as per the National Environmental Policy Act ("NEPA") process.

The core of the Project mineral resource is contained within the 132 patented mining claims that in total encompass an area of approximately 2,000 acres (809 hectares) as shown in Figure 4-2. Surrounding the patented claims is a contiguous package of 1,064 unpatented mining claims with an aggregate area of more than 16,000 acres (6,475 hectares). Associated with the mining claims are 38 parcels of fee (private) land consisting of approximately 2,300 acres (931 hectares) (the Associated Fee Lands). The area covered by the patented claims, unpatented claims and Associated Fee Lands totals approximately 20,300 acres (8,215 hectares). A listing of the patented claims, unpatented claims and Associated Fee Lands is provided in Appendix A1-1 & A1- 2.

Rosemont has also acquired 62 parcels of fee (private) land and one parcel of leased land that are more distal from the Project area that are planned for: (1) various infrastructure purposes including, well fields, pump stations, utilities and ranch operation; and (2) for environmental mitigation and conservation purposes (together, the Distal Fee Lands). The Distal Fee Lands constitute an additional approximately 3,700 acres (1,497 hectares).

**FIGURE 4-2: ROSEMONT PROPERTY OWNERSHIP**




The patented mining claims are considered to be private lands that provide the owner with both surface and mineral rights. The patented mining claim block, including the core of the mineral resource, is monumented in the field by surveyed brass caps on short pipes cemented into the ground. The fee lands are located by legal description recorded at the Pima County Recorder's Office. The patented claims and Associated Fee Lands are subject to annual property taxes amounting to a total of approximately $8,800.

Mineral Rights on USFS and Bureau of Land Management ("BLM") lands have been reserved to Rosemont Copper Company, via the unpatented claims that surround the patented claims. Wooden posts and stone cairns mark the unpatented claim corners, end lines and discovery monuments, all of which have been surveyed. The unpatented claims are maintained through the payment of annual maintenance fees of $155.00 per claim, for a total of approximately $165,000 per year, payable to the BLM.

The rights-of-way over State Land are all non-exclusive but grant Rosemont the rights to construct certain utility infrastructure connecting the well field and power supply to the site. Two of these rights-of-way have a term of ten years while the other four have a term of fifty years. These rights-of-way across State Land are not shown in Figure 4-2, but generally run northwest from the project site towards the Town of Sahuarita.

There is a 3% NSR royalty on all 132 patented claims, 603 of the unpatented claims, and one parcel of the Associated Fee Lands with an area of approximately 180 acres. In the original royalty deeds, a 1.5% NSR is reserved to each of (1) Dennis Lauderbach et. ux. and (2) Pioneer Trust Company of Arizona, as Trustee under Trust No. 11778. These royalties have since been assigned and Rosemont is in the process of verifying current ownership.

A precious metals stream agreement with Silver Wheaton Corp. for 100% of payable gold and silver from the Project was entered into on February 11, 2010. Under the agreement, Hudbay will receive payments equal to lesser of either the market price or $450 per ounce for gold and $3.90 per ounce for silver, subject to 1% annual escalation after three years.

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

## 5.1 Accessibility

The Project is easily accessible to the communities of Tucson and Benson to the north and Sierra Vista, Sonoita, Patagonia and Nogales to the south by way of State Route 83.

Existing graded dirt roads connect the property with State Route 83 which include Forest Service ("FS") roads 4062 into the Hidden Valley complex, FS4064 into Rosemont Camp and FS231 transecting the property north to south. FS4051 and FS4059 provide good access into and around the Project area.

The city of Tucson, Arizona provides the nearest major railroad and air transport services to the Project and is approximately 30 miles southeast of Tucson in Pima County.

## 5.2 Climate

The southern Arizona climate is typical of a semi-arid continental desert with hot summers and temperate winters. The Project area is at the north end of the Santa Rita Mountain Range at elevations between 4,550 and 5,350 feet (1,387 and 1,631 meters) above mean sea level ("AMSL"). The higher elevation in the Project area results in a milder climate than at the lower elevations across the region.

Summer daily high temperatures are above 90°F (32°C) with significant cooling at night. Winter in the Project area is typically drier with mild daytime temperatures and overnight temperatures that are typically above freezing. Winter can have occasional low intensity rainstorm and light snowfall patterns that can last for several days.

The average annual precipitation in the Project area is estimated between 16 and 18 inches (41 and 46 cm) based on historical data from eight meteorological stations within a 30 mile (50 km) radius of the Project area. More than half of the annual precipitation occurs during the monsoon season from July through September. The monsoon season is characterized by afternoon thunderstorms that are typically of short duration, but with high-intensity rainfall that has minor effects on a mining operation, which is considered to be 365 days per year. The lowest precipitation months are April through June.

A meteorological station was installed at the property in April, 2006. The station is located near the center of the deposit at an elevation of 5,350 feet (1,631 m) AMSL. The station monitors site-specific weather data including temperature, precipitation, wind speed, and wind direction. Pan evaporation was added to this station in mid-2008. This station was decommissioned and a new station was located near the core shed on private property in 2015. Data from the weather station is automatically recorded and downloaded monthly by site personnel.

## 5.3 Local Resources

The largest city near the Project area is Tucson with a population of over 520,000 based on the 2014 census. Tucson is also the county seat for Pima County with a population of approximately one million, which encompasses the Tucson Metropolitan Area.

Arizona produces 65% of the copper in the USA[2] and Tucson is a mining industry hub in the state with nine operating copper mines within a 125 mile (200 km) radius. The cultural and educational facilities provided in the Tucson Metropolitan Area attract experienced technical staff into the area. Tucson is home to a well-established base of contractors and service providers to the mining industry.

## 5.4 Infrastructure

The Project site is located immediately adjacent to and west of Arizona State Route 83 (South Sonoita Highway), approximately 11 miles (18 km) south of Interstate 10 ("I-10"). This system of state and interstate highways allows convenient access to the site for all major truck deliveries. The majority of the labour and supplies for construction and operations can come from the surrounding areas in Pima, Cochise and Santa Cruz Counties.

The Union Pacific mainline east-west railroad route passes through Tucson, Arizona and generally follows I-10. The Port of Tucson has rail access from the Union Pacific mainline consisting of a two mile (3.2 km) siding complimented by an additional 3,000 foot (914 m) siding.

The Tucson International Airport ("TIA") is located approximately 30 miles (50 km) from the Project site and in close proximity to Interstate highways I-10 and I-19. TIA provides international air passenger and air freight services to businesses in the area with seven airlines currently providing nonstop service to 15 destinations with connections worldwide.

The power supply to the Project falls within the Tucson Electric Power Company ("TEP") and TRICO Electric Cooperative Inc. ("TRICO") service territories. Presently there exists a 13.8 kV TEP distribution line that routes through the Project site and power from this distribution line was used to service the related activities for the 2014 and 2015 drill program.

Geographically, the area east of the deposit that includes the majority of the mineral resource is in the TEP service territory, while the area west of the deposit falls within the TRICO service territory. Since most of the estimated electrical load for a mining and process operations would be located in the TEP service territory, TEP will be the electrical utility service provider for the entire facility. A joint venture business arrangement is expected to be established between TEP and TRICO to compensate both service providers in accordance with the Arizona Corporation Commission review and approval.

[2] *Arizona Mining Association economic impact 2014 (azmining.com)*

The most viable source of water supply for the Project is from groundwater in the upper Santa Cruz basin aquifer. The Project currently holds a permit granted by the Arizona Department of Water Resources ("ADWR") in 2009 to pump 6,000 acre-feet of water annually for 20 years that meet mining and processing operations requirements. In addition, there are bedrock and/or shallow alluvium aquifers on or near the Project area that supplied water for the 2014 and 2015 drill program; however, they are considered to be insufficient as a primary source of water supply for a mining operation.

The Project consists of sufficient area of land to the east of the deposit, which is suitable for mining and processing operations, waste rock storage area ("WRSA") and dry stack tailings facility ("DSTF") for a deposit of this size.

## 5.5 Physiography

The Project is located within the northern portion of the Santa Rita Mountains that form the western edge of the Mexican Highland section of the Basin and Range Physiographic Province of the southwest United States (Wardrop, 2005). The Basin and Range physiographic province is characterized by high mountain ranges adjacent to alluvial filled basins. The property occupies flat to mountainous topography in the northeastern and northwestern flanks of the Santa Rita Mountains.

Vegetation in the Project area reflects the climate with the lower slopes of the Santa Rita Mountains dominated by mesquite and grasses. The higher elevations, receiving greater rainfall, support an open cover of oak, pine, juniper and cypress trees.

# 6 HISTORY

The early history and production from the Property has been described in Anzalone (1995), M3 (2012) and Briggs (2014) from which the following summarization is taken. Hudbay considers the mineral reserve and resource estimates referred to in this chapter (including the estimates prepared by Augusta) to be historical in nature since no work was done by a qualified person to verify such estimates and such estimates should not be relied upon.

## 6.1 Helvetia-Rosemont Mining District (1875 – 1973)

The first recorded mining activity in the Helvetia-Rosemont mining district occurred in 1875. The Helvetia-Rosemont mining district was officially established in 1878. Production from mines on both sides of the Santa Rita ridgeline supported the construction and operation of the Columbia Smelter in Helvetia and the Rosemont Smelter in Old Rosemont. Copper production from the district ceased in 1951 after production of about 227,300 tons of ore containing 17,290,000 pounds of copper, 1,097,980 pounds of zinc and 180,760 ounces of silver.

By the late 1950s, the Banner Mining Company (Banner) had acquired most of the claims in the area and had drilled the discovery hole into the Rosemont deposit. In 1963, the Anaconda Mining Co. acquired options to lease the Banner holdings and over the next ten years they carried out an extensive drilling program on both sides of the mountain for a total of 136,838 feet (41,708 m) from 113 drill holes. The exploration program demonstrated that a large scale porphyry/skarn existed at Rosemont. Regional exploration included targets at Broadtop Butte and Peach-Elgin prospects. In 1964, Anaconda produced a historical resource estimate for the Peach-Elgin deposit located in the Helvetia District. Based on assays from 67 churn and diamond drill holes, the estimate identified 14 million tons of sulfide material averaging 0.78% copper and 10 million tons of oxide material averaging 0.72% copper.

## 6.2 Anamax Mining Company (1973 - 1985)

In 1973, Anaconda Mining Co. and Amax Inc. formed a 50/50 partnership to form the Anamax Mining Co. In 1977, following years of drilling and evaluation, the Anamax joint venture commissioned the mining consulting firm of Pincock, Allen & Holt, Inc. to estimate a resource for the Rosemont deposit. Their historical resource estimate of about 445 million tons of sulfide mineralization averaged 0.54% copper using a cut-off grade of 0.20% copper. In addition to the sulfide material, 69 million tons of oxide mineralization averaging 0.45% copper was estimated. Subsequent engineering designed a pit based on 40,000 tons/day production rate for a mine life of 20 years.

In 1979, Anamax carried out a resource estimate for the Broadtop Butte deposit located about a mile north of the Rosemont deposit. Based on assays from 18 widely spaced diamond drill holes, a historical estimate identified 9 million tons averaging 0.77% copper and 0.037% molybdenum. In 1985, Anamax ceased operations and liquidated their assets. Today, most of the Anaconda/Anamax core is currently stored at Hidden Valley core storage facility at Project site. *Hudbay considers the estimate done by Anamax to be historical in nature since no work has been done by a Hudbay QP to verify the estimate and the estimate should not be relied upon.*

## 6.3 ASARCO, Inc. (1988 – 2004)

Asarco purchased the patented and unpatented mining claims in the Helvetia-Rosemont mining district from real estate interests in August 1988 and renewed exploration of the Peach-Elgin and initiated engineering studies on Rosemont. In 1995, Asarco succeeded in acquiring patents on 21 mining claims in the Rosemont area just prior to the moratorium placed on patented mining claims in 1996.

In 1999, Grupo Mexico acquired the Helvetia-Rosemont property through a merger with Asarco. During the 16 years of ownership by Asarco and Grupo Mexico, 11 diamond drill holes were completed for a total of 14,695 feet (4,479 m) at Rosemont. Asarco estimated historical reserves of 294,834,000 tons at 0.673% copper based on a mine production schedule with a strip ratio of 3.7:1. The Asarco drill core is currently stored at the Hudbay core storage facility on site. In 2004, Grupo Mexico sold the Rosemont property to a Tucson developer.

## 6.4 Augusta Resource Corporation (2005 – 2014)

In April 2005, Augusta purchased the property from Triangle Ventures LLC. Between mid-2005 and January 2007, Augusta drilled 55 diamond drill holes for a total of 96,129 feet (29,300 meters) in order to bring the resource estimate at Rosemont into compliance with NI 43-101 standards. The program was designed to better define the geology, distribution of copper mineralization as well as gather geotechnical data required to design a pit. In June 2006, Washington Group Int. completed a preliminary assessment and economic evaluation of the Project. Augusta submitted a mine plan of operations (“MPO”) to the USFS in July 2006. It was deemed incomplete and in 2007, Augusta resubmitted the MPO. Following a positive feasibility study conducted by M3 Engineering in August 2007 the Forest Service accepted a revised MPO in March 2008 marking the start of the formal EIS process mandated by the National Environmental Protection Act (“NEPA”) of 1969.

Over the next several years, Augusta continued to evaluate the mineral potential at Rosemont and refine the economics of developing this resource. A 20-hole diamond drilling program (17,522 feet or 5,341 meters) was conducted from December 2007 through July 2008. This was followed by a twelve-hole diamond drilling program (18,874 feet or 5,753 meters), which was completed in February 2012. A Technical Report issued by Augusta in 2012 estimated mineral reserves of 667.2 million tons at an average grade of 0.44% copper, 0.015% molybdenum and 0.12 ounces per ton of silver based on $4.90 per ton NSR cut-off using metal prices of $2.50/lb copper, $15.00/lb molybdenum and $20.00/oz silver. Augusta’s mineral resource estimate, shown in Table 6-1, is inclusive to their mineral reserves, stated above.

*Hudbay is treating Augusta's publicly disclosed estimated mineral reserves and resources as a historical estimate under NI 43-101 and not as current mineral reserves and resources, as a Qualified Person has not done sufficient work for Hudbay to classify Rosemont's mineral reserves or resources as current mineral reserves or mineral resources.*

**TABLE 6-1: HISTORICAL SULFIDE MINERAL RESOURCE (AUGUSTA 2012)**

| Category | Tons (millions) | Cu (%) | Mo (%) | Ag (oz/ton) |
|---|---|---|---|---|
| Measured | 334.619 | 0.440 | 0.015 | 0.124 |
| Indicated | 534.735 | 0.373 | 0.014 | 0.105 |
| Inferred | 128.488 | 0.397 | 0.013 | 0.104 |

## 6.5 Hudbay (2014 – Present)

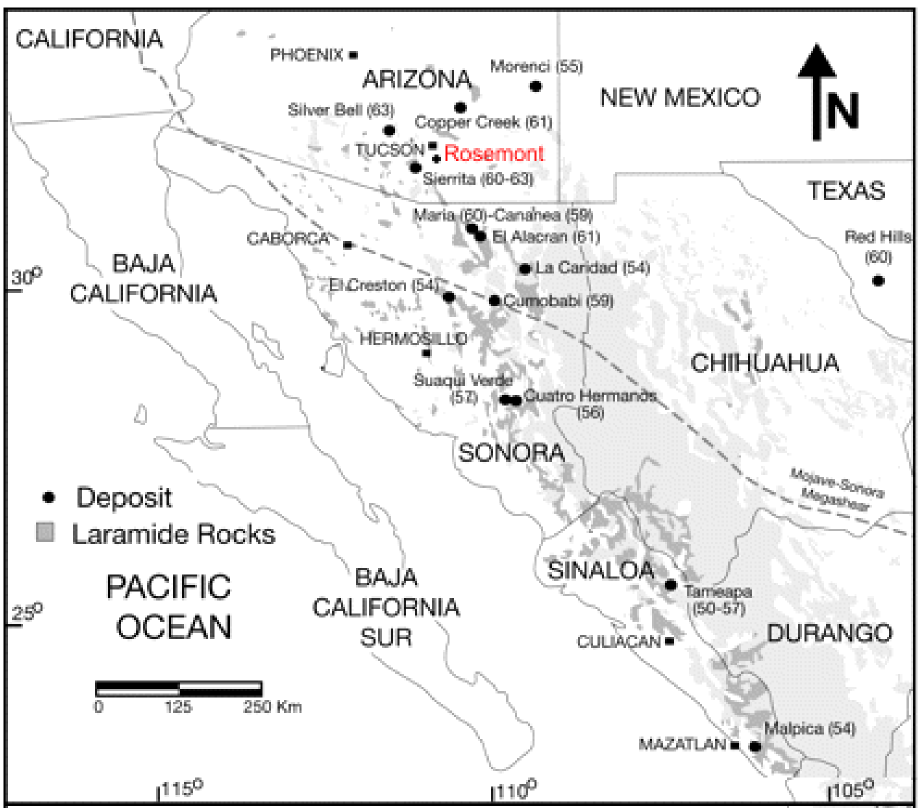
Hudbay completed a 43-hole, 92,909 feet (28,319 meters) drill program from September to December 2014 and a 46-hole, 75,164 feet (22,910 meters) drill program from August to November 2015. These drilling programs were completed in further efforts to gain a better understanding of the geological setting and mineralization of the deposit and to collect additional metallurgical and geotechnical information.

# 7 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 Tectonic and Metallogenic Setting

Mesozoic subduction and associated magmatism and tectonism in the southwestern United States and northern Mexico, generated extensive and relevant porphyry copper mineralization (Figure 7-1). Compressional tectonism during the Mesozoic and early Cenozoic Laramide Orogeny caused folding and thrusting, accompanied by extensive calc-alkaline magmatism (Barra et al., 2005). The Laramide belt is a major porphyry province that extends for approximately 600 miles (1,000 km) from Arizona to Sinaloa, Mexico, and includes the Rosemont deposit and hosts a number of other world class deposits (e.g. Morenci, Resolution, and Cananea).

**FIGURE 7-1: LARAMIDE BELT AND ASSOCIATED PORPHYRY COPPER MINERALIZATION (BARRA ET AL., 2005)**




Tertiary extensional tectonism followed the Laramide Orogeny, accompanied by voluminous felsic volcanism (Barra et al., 2005). Steeply-to shallowly-dipping normal faults became active during this time, including rotational listric faulting. At Rosemont, it appears that tertiary faulting has significantly segmented the original deposit, juxtaposing mineralized and unmineralized rocks. The extensional tectonics culminated in the large-scale block faulting that produced the present basin and range geomorphology that is typical throughout southern Arizona (Maher, 2008).

## 7.2 Regional Geology

The geology of the Santa Rita Mountains has recently been reviewed by Rasmussen et. al. (2012) identifying two main blocks (Figure 7-2). The northern block, where the Rosemont deposit lies, is dominated by Precambrian granite (brown on the map), with some slices of Paleozoic and Mesozoic sediments on the eastern and northern sides (blue and green on the map). This block includes small stocks and dikes of quartz monzonite or quartz latite porphyry that are related to porphyry copper and skarn mineralization.

## 7.3 District Geology

The Precambrian meta-sedimentary and intrusive rocks in the Rosemont area form the regional basement beneath a Paleozoic carbonate and siliciclastic sedimentary sequence (Figure 7-3 and Figure 7-4). Paleozoic sedimentary carbonate units are the predominant host rocks for the copper mineralization. Structurally overlying these predominantly carbonate units at Rosemont are Mesozoic clastic units, including conglomerates, sandstones, and siltstones. These clastic upper sequences have andesitic intercalations and also host mineralization. Quartz monzonite and quartz latite sill-shaped porphyries intruded both sequences and are associated with the porphyry/skarn mineralization. Tertiary conglomerates locally lay over the Mesozoic sedimentary units in late fault grabens. The Rosemont stratigraphy is summarized in Figure 7-4 and the geological configuration of the deposit is shown on a level section in Figure 7-5 and in a vertical section in Figure 7-6.

**FIGURE 7-2: SANTA RITA MOUNTAIN GEOLOGY (ADAPTED FROM DREWES ET AL., 2002)**

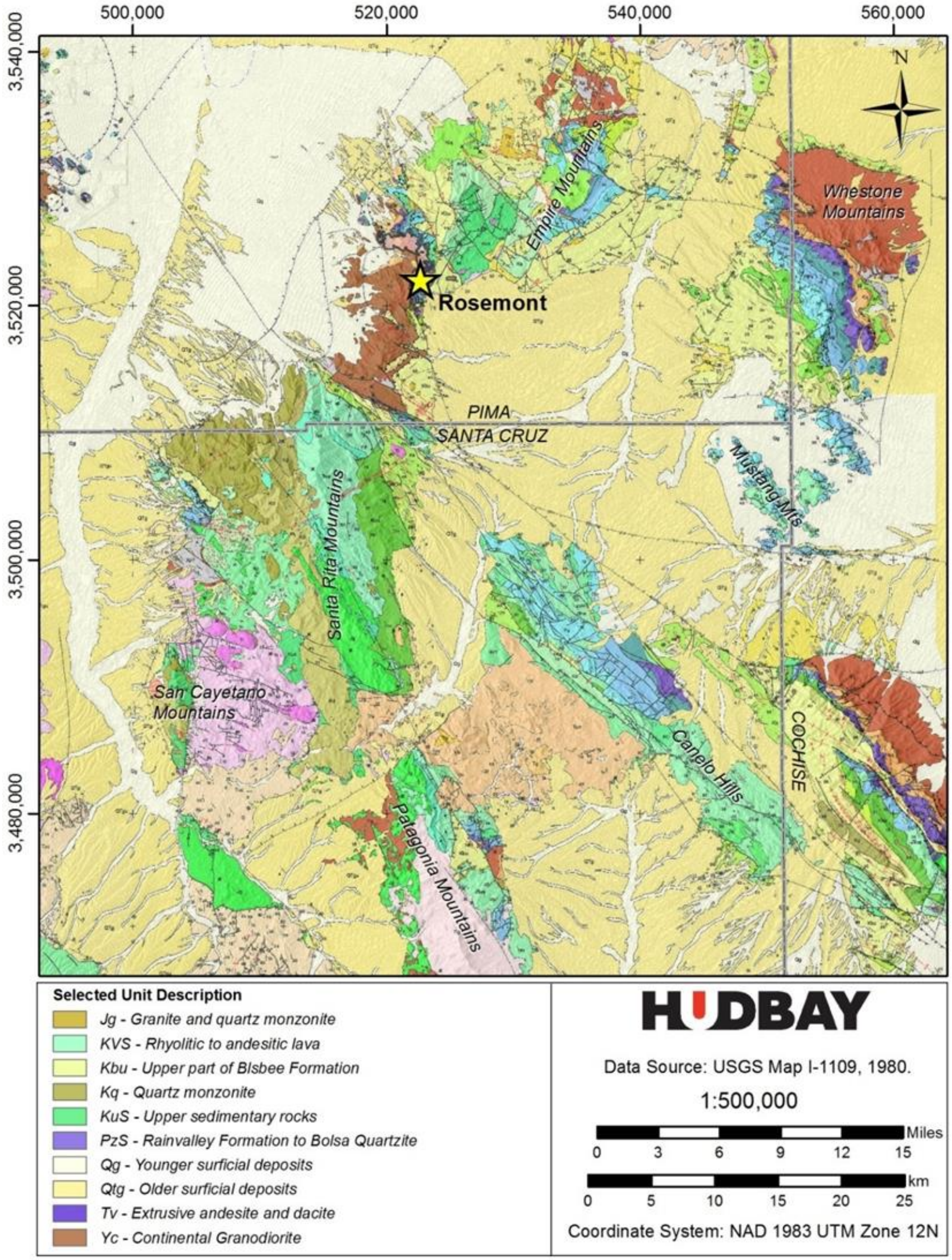
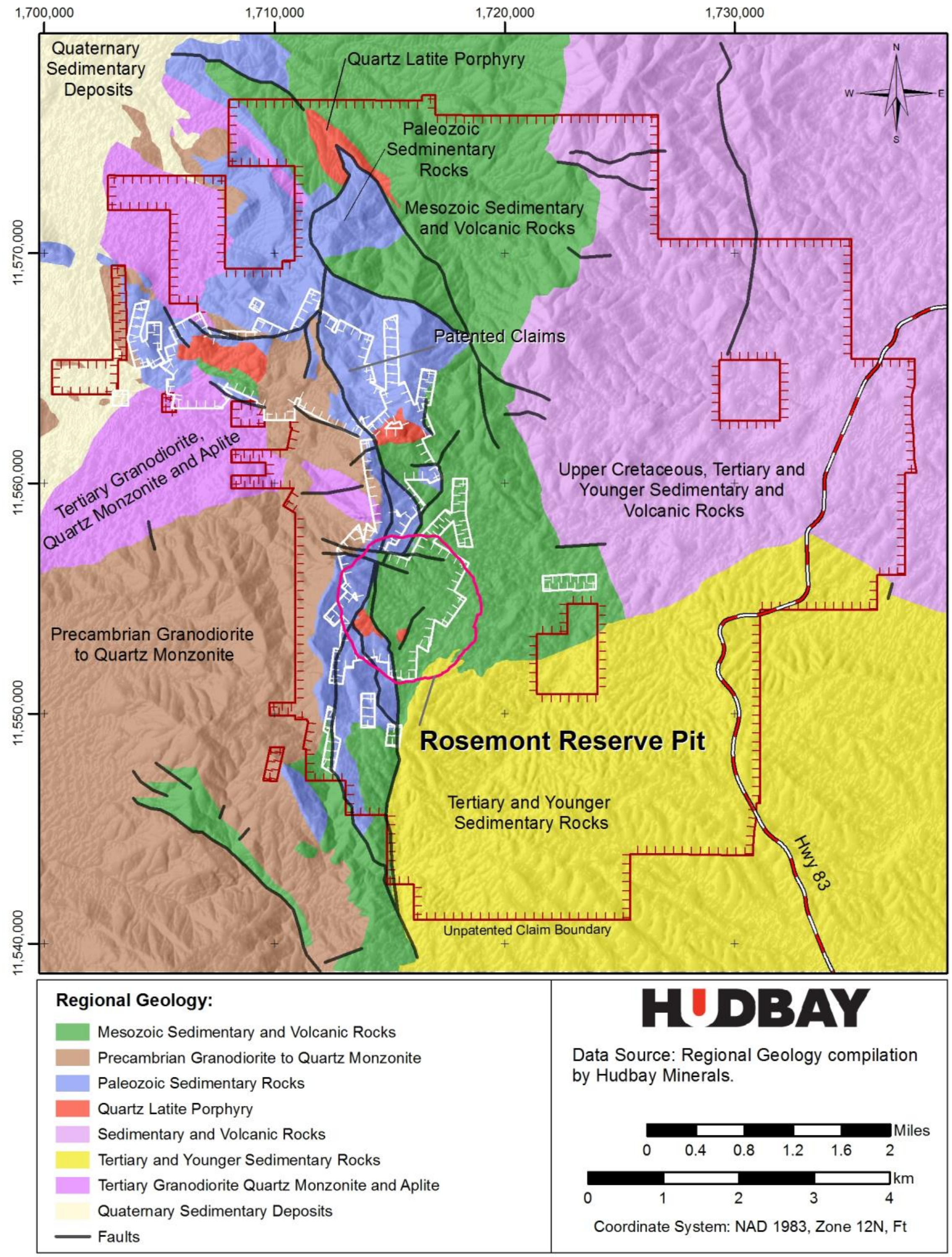


**FIGURE 7-3: ROSEMONT REGIONAL GEOLOGY**



# FIGURE 7-4: ROSEMONT STRATIGRAPHIC COLUMN

| Era | Period | Chemo-stratigraphy | Formation (lith code) | Thickness (ft) | Section | Lithology | Skarn/Alteration | Mineralization |
|---|---|---|---|---|---|---|---|---|
| Cenozoic | Tertiary | QMP | Quartz-Feldspar Porphyry (8) | | | Quartz monzonite porphyry to quartz latite porphyry. 55.7-56.3 my | Qtz-ksp; qtz-ser-pyr; minor epidote-chlorite | Qtz +/- pyr-cpy-bn-mo veining, commonly oxidized. Mo disseminated and vein controlled. |
| Mesozoic | Cretaceous | Andesite / Arkose | Willow Canyon Formation (10) | 2,200 | | Interbedded arkosic sandstone, siltstone and conglomerate. Internal andesite flow sequence locally present. | Arkose: wk; ksp, epidote, calcite. Rare qtz veining. Andesite: stronger quartz-chlorite-epidote. | Arkose: wk limonite and secondary Cu mins. Rare qtz-pyr-cpy-bn vns. Andesite: stronger qtz-pyr-cpy-bn veining, commonly oxidized. |
| Mesozoic | Cretaceous | Glance Group | Glance Conglomerate (11) | 0-1500 | | Limestone conglomerate. Locally underlain by Paleozoic(?) bedded limestone +/- quartzite ("Mystery Limestone"). Fault at base. | Moderately to strongly marblized. Local calc-silicate alteration. | Mineralized locally (rarely). |
| Paleozoic | Permian | Glance Group | Rainvalley Fm (18) | 0-300 | | Fossiliferous limestone, dolomite and quartz sandstone | Serpentine-magnetite. | Secondary Cu and minor sulfides in BT Buttes area |
| Paleozoic | Permian | Glance Group | Concha Limestone (14) | 400-575 | | Thick-bedded, cherty, fossiliferous limestone | Marblized. Wollastonite and garnet skarns locally. | Secondary Cu and minor sulfides in BT Buttes area |
| Paleozoic | Permian | Scherrer | Scherrer Fm (12) | 720 | | Quartzite, locally x-bedded; less dolomite and limestone. Basal siltstone. | Wk. chlorite in clastics. Garnet skarn after limestone. | Pyr, less common cpy and bn |
| Paleozoic | Permian | Epitaph Group | Epitaph Fm (2) | 1000 | | Limestone, dolomite and marl; less quartzite. Distinctive gypsum beds. Basal thick-bedded dolomite. | Strong chlorite-serpentine-magnetite; less garnet-diopside-chlorite skarn; hornfels | Bn-cpy +/- pyr vns and disseminated in skarn |
| Paleozoic | Permian | Epitaph Group | Colina Limestone (3) | 350 | | Dark, thick-bedded dolomitic limestone. Fossiliferous. Minor quartzite. | Marblized. Some serpentine-magnetite vns. Some garnet skarn. | Bn-cpy +/- pyr |
| Paleozoic | Permian / Pennsylvanian | Earp | Earp 2 / Earp 1 | 800 | | Siltstone, shale, sandstone, chert-pebble conglomerate and limestone. Earp 2: siliciclastic dolostone Earp 1: siliciclastic limestone | Hornfels and garnet-diopside-chlorite skarn. | Qtz-cal-chl and qtz vns with pyr-cpy-bn |
| Paleozoic | Pennsylvanian | Horquilla | Horquilla Limestone (5) | 1000 | | Thin- to thick-bedded limestone, siltstone, minor shale. Basal Black Prince Limestone | Skarn, hornfels, and marble. Garnet-pyroxene skarn with lessor chlorite and serpentinite. Local wollastonite. | Qtz and chl-serp-mag vns; bn-cpy-cc +/- pyr in vns and disseminated in skarn. Secondary Cu minerals near faults. Main mineralized formation. |
| Paleozoic | Miss. | Escab-rosa | Escabrosa Limestone (6) | 560 | | Thick-bedded to massive limestone, cherty. | Marble. Serpentine at faults. Garnet-diopside-magnetite at intrusive contact. | Commonly barren. Where altered, strong py-bn-cpy mineralization. |
| Paleozoic | Dev. | Martin | Martin Formation (7) | 400 | | Thin- to medium bedded domomite; less limestone, siltstone and sandstone. Faulted contacts. | Marble; minor chorite and serpentine | Weakly mineralized. |
| Paleozoic | Cambrian | Abrigo | Abrigo Formation (13) | 740-900 | | Thin-bedded limestone, siltstone and shale | Weakly marblized | Weakly mineralized. |
| Paleozoic | Cambrian | Bolsa | Bolsa Quartzite (15) | 460 | | Course-grained, thick-bedded quartzite | Weakly altered, unreactive. | Trace disseminated pyr and cpy |
| pC | pC | Grano-diorite | Continental Granodiorite (16) | | | Granodiorite porphyry | Weak alteration locally. | Weak mineralization locally |

*Notes: Stratigraphic thicknesses taken from H. Drewes, Professional Paper 748 (1972), and may exceed thicknesses found locally at the Rosemont Deposit. Skarn/Alteration and Mineralization from Daffron and others, Augusta Resource internal report, 2007. Chemostratigraphy units from Hudbay 2014-15 multi-element study.*

**FIGURE 7-5: ROSEMONT DEPOSIT GEOLOGIC – 4,000 FOOT LEVEL PLAN**

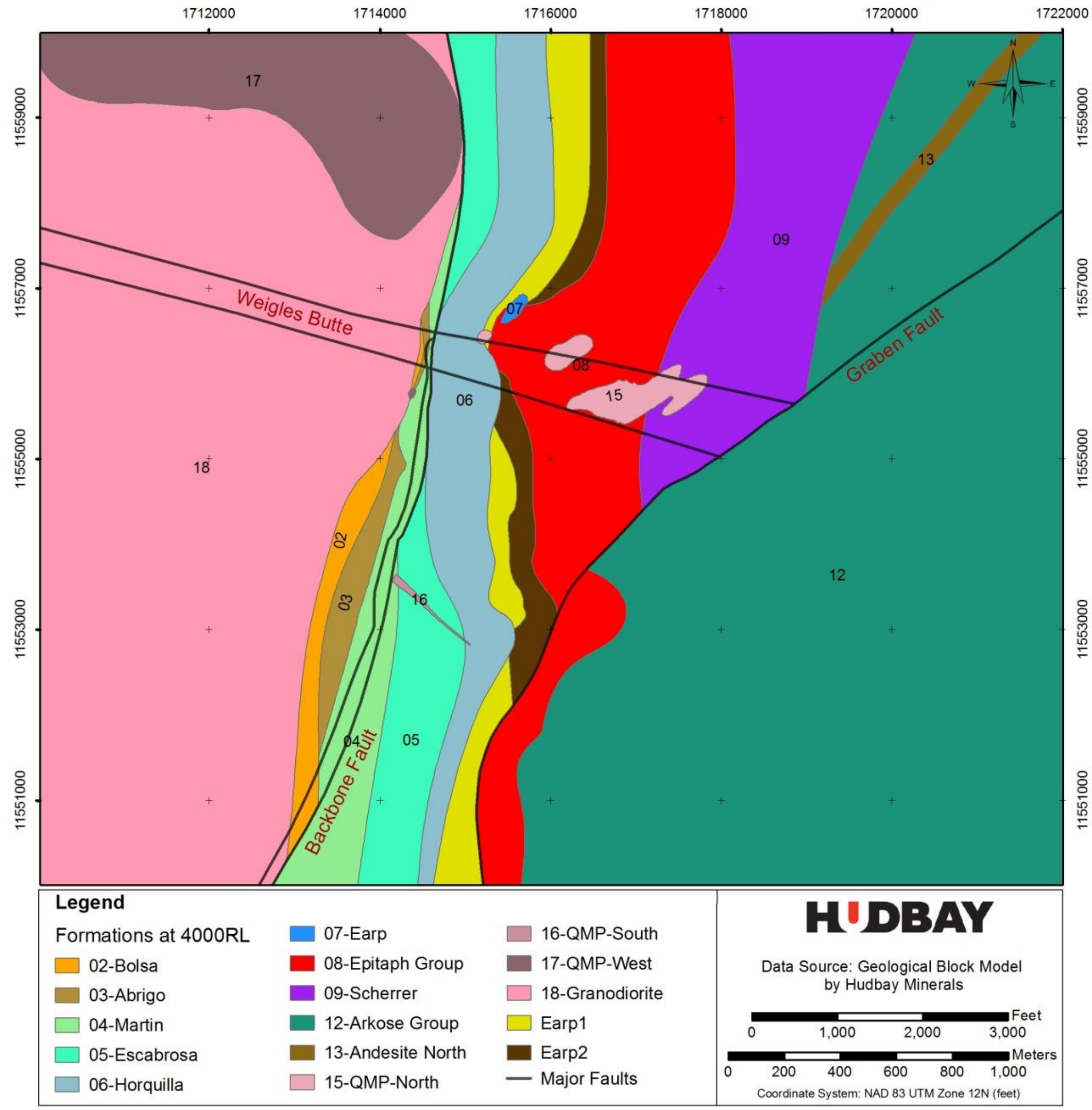
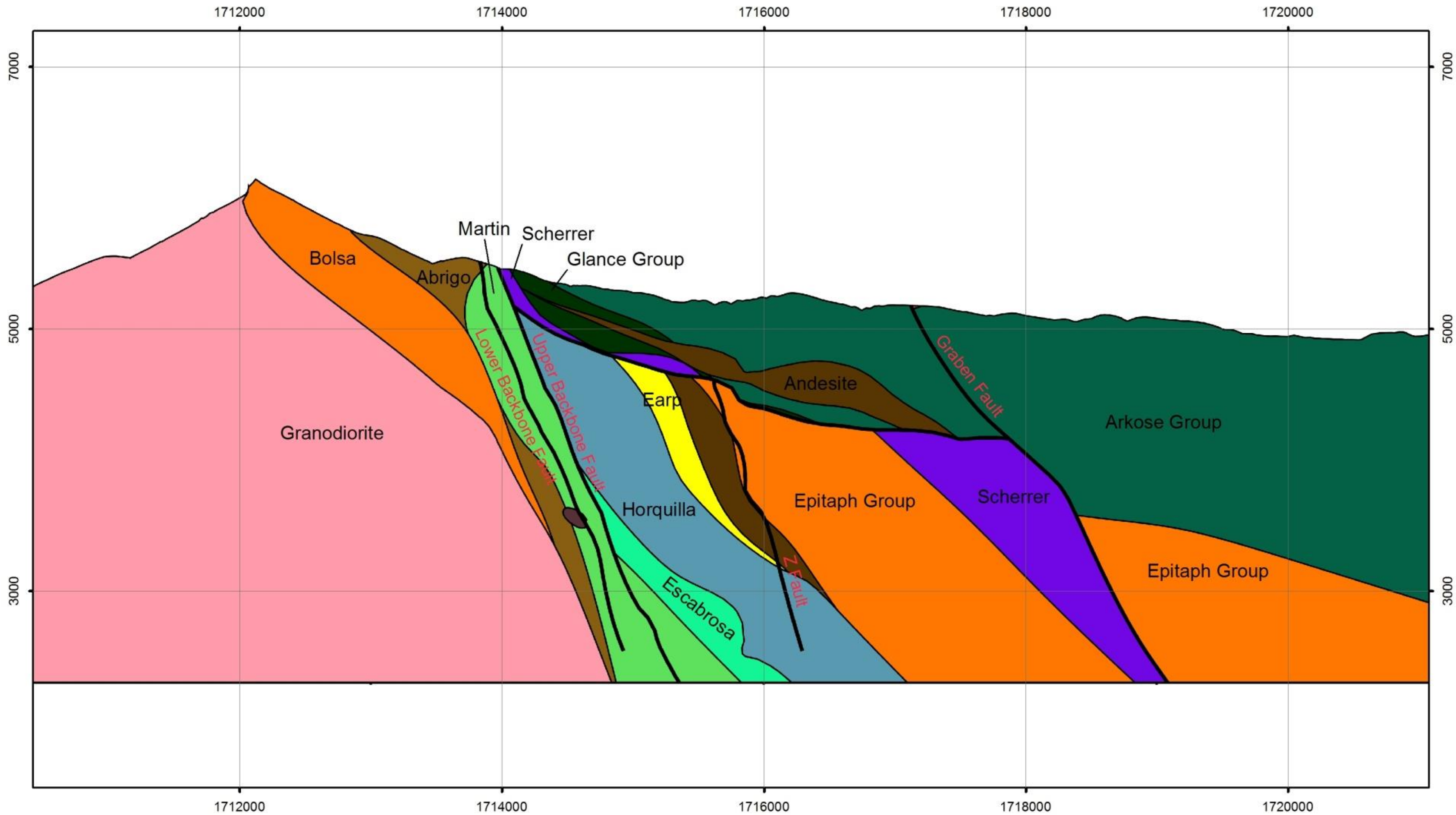


**FIGURE 7-6: ROSEMONT DEPOSIT GEOLOGIC – 11,555,050 VERTICAL SECTION**



## 7.4 Chemostratigraphy

Hudbay 2014 and 2015 drilling programs included complete inductively coupled plasma multi-element assays (4 acid digestion) for every sample. The new data set (> 33,000 samples) was used to classify the different stratigraphic units according to their geochemical affinities. The original formations were grouped into equivalent chemostratigraphic units that reflect chemical changes induced by mixing of siliciclastic, dolomitic, and calcareous sediments as well as a hydrothermal component. The chemostratigraphic groups honour both the deposit stratigraphy and geochemical attributes and ultimately reflect the mineralogy (Figure 7-7). The geological model built implicitly in Leapfrog is based mainly on the downhole chemostratigraphy of the holes drilled between 2014 and 2015 (90 holes). The density of the chemostratigraphy data is higher in the pit area, where the new model was exclusively based on chemostratigraphy. In zones that are distal to the recent drilling (e.g. Backbone footwall domain), lithology (logged) data was incorporated into the model.

**FIGURE 7-7: CHEMOSTRATIGRAPHY ROSEMONT DEPOSIT GEOLOGY**

| Formation | Chemostratigrapy | Lithogeochemical Description |
|---|---|---|
| QMP | QMP | The quartz-monzonite porphyries can be discriminated from other intrusive and volcanic rocks by their high silica, and low thorium and titanium content. |
| Willow Canyon | Andesite | Andesites are geochemically distinctive by their high titanium content. |
| | Arkose | The upper siliciclastic rocks are distinctive in their Ca, Mg, and trace element content relative to the lower chemical sedimentary units. The Arkose display a provenance signature indicating a larger contribution of mafic and intermediate sources. |
| Glance | Glance Group | Strong calcareous signature indicating pure limestones with scarse silicistic component. Dolomitization is localized along the upper and lower members, with stonger dolomitic signatures towards the Concha Formation. |
| Rainvalley | | |
| Concha | | |
| Scherrer | Scherrer | Dominated by siliclastic component, but also contains less abundant impure dolostones and limestones. The dolomitic component increases downwards into the Epitaph Group. |
| Epitaph | Epitaph Group | Pure dolostones and limestones with minor siliciclastic component. Relative to the limestone geochemical component, dolomitization is stronger in the upper parts of the group, and strong but less dominant towards the lower part of the Epitaph Group. |
| Colina | | |
| Earp | Earp 2 (Upper) | Siliciclastic dolostne dominated. |
| | Earp 1 (lower) | Siliciclastic limestone dominated. |
| Horquilla | Horquilla | Strong calcareous signature indicating pure limestones and interlayered mixed siliciclastic and chemical sediments. Poor dolomitization. |
| Escabrosa | Escabrosa | Strong calcareous signature indicating pure limestones with scarse silicistic component. Poor dolomitization. |
| Martin | Martin | Mainly dolostones and minor limestone. |
| Abrigo | Abrigo | Impure limestone characterized by mixed chemical sedimentary and siliciclastic sedimentary rocks indicated by geochemical signatures. |
| Bolsa | Bolsa | Characterized by higher silica and lower titanium content relative to siliciclastic and volcanic rocks. |
| Continental Granodiorite | Granodiorite | The Precambrian granodiorite is distinctive by its thorium enrichment relative to other intrusive, volcanic and siliciclastic rocks in the Rosemont Property. |

## 7.5 Structural Domains

The updated geological model incorporated a revised structural framework based on a surface and downhole structural review. The temporal and special relations between the main fault surfaces define 4 structural domains: Backbone Footwall, Lower Plate, Upper Plate and Graben Block (Figure 7-8).

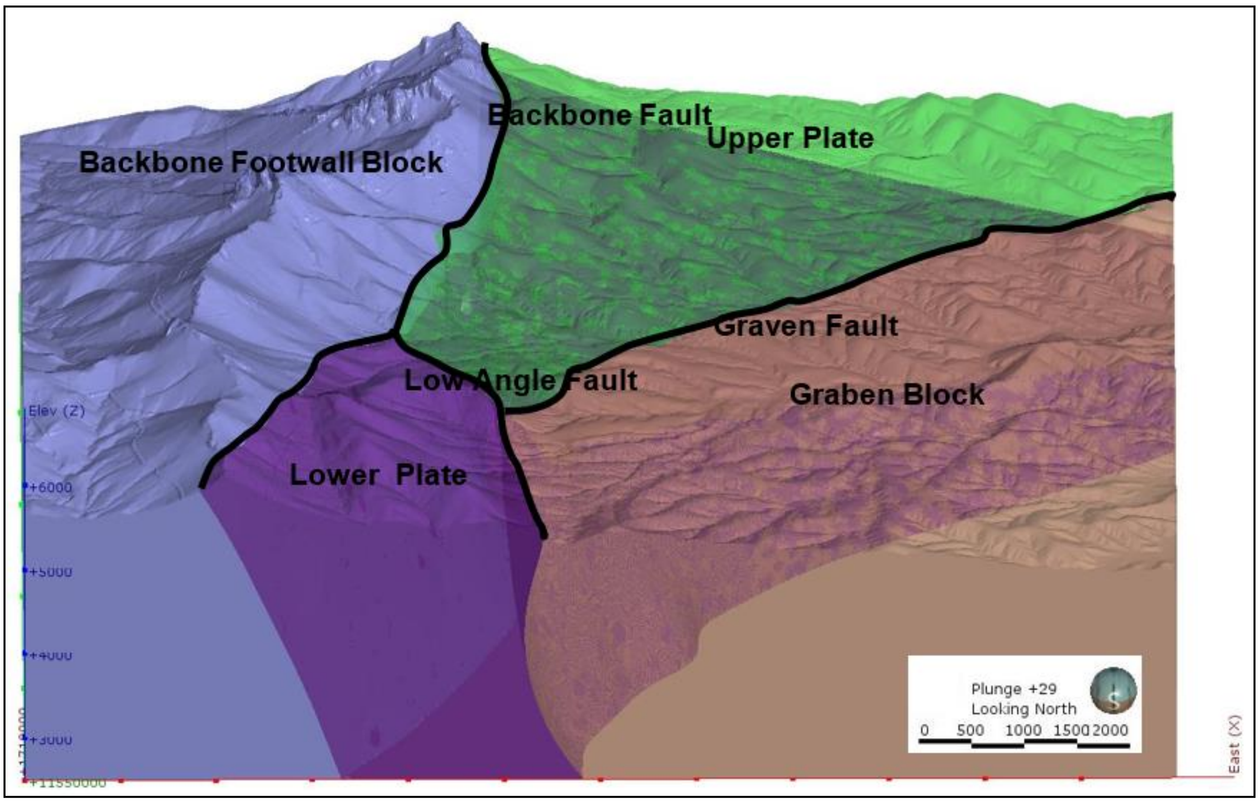
The north trending, steeply dipping Backbone Fault juxtaposes Precambrian granodiorite and Lower Paleozoic quartzite and limestone marginally mineralized to the west (Backbone Footwall block) against a block of an homoclinical sequence of younger mineralized metamorphosed sedimentary units to the east (Lower Plate). A series of subparallel, anastomosing, curviplanar faults that generally strike north and dip steeply within the Lower Plate define a zone along the Backbone Fault strike.

The Low Angle Faults are a series of shallowly east-dipping faults that are comprised of one major fault and a series of steep to shallow splay structures. The main Low Angle Fault forms the non-conformable contact between the Upper (siliciclastics and volcanics) and Lower Plate rocks.

The southeast-dipping Graben (60-65°) fault terminates mineralization continuity to the southeast and east. This fault postdates mineralization and has been interpreted as an extensional normal fault.

The east-west striking faults, including the Weigle Faults are a series of steeply-dipping, anastomosing structures that are oriented oblique to bedding and rock contacts. Locally, the most well-known fault is the Weigle Fault Zone, which displaces the Precambrian, Paleozoic, and Mesozoic rocks and generates a deepening of the oxide front.

**FIGURE 7-8: ROSEMONT DEPOSIT GEOLOGICAL MODEL STRUCTURAL DOMAINS 3D VIEW (LOOKING NORTH)**




## 7.6 Mineralization

Drilling to date at Rosemont has defined a mineral resource approximately 4,000 feet (1,200 meters) in diameter that extends to a depth of approximately 2,500 feet (750 meters) below the surface. The main fault systems partially delimit the defined resource, dividing the deposit into major structural blocks with contrasting intensities and types of mineralization. The north-trending, steeply dipping Backbone Fault juxtaposes marginally mineralized Precambrian granodiorite and Lower Paleozoic quartzite and limestone to the west (Back Bone Footwall Block) against a block of younger, well-mineralized Paleozoic limestone units to the east (Lower Plate).

Most of the copper sulfide resource is contained in the eastern hanging wall of the Backbone Fault. Structurally overlying the sulfide resource is a block of Mesozoic sedimentary and volcanic rocks (Upper Plate) that contains lower grade copper mineralization (predominantly as oxides). These two blocks are separated by the shallowly dipping Low Angle Fault ("LAF"). Other post-mineral features include a deep, gravel-filled Tertiary paleo valley on the south side of the deposit and a significant thickness of Cretaceous and Tertiary volcaniclastic material to the northeast of the deposit.

Sulfide mineralization on the east side of the Backbone Fault and below the LAF is hosted in an east-dipping package of Paleozoic-age sedimentary rocks that includes the Escabrosa Limestone, Horquilla Limestone, Earp Formation, Colina Limestone, and Epitaph Formation.

Relatively minor mineralization occurs in the other Paleozoic units. To the south, the mineralization in this block appears to weaken and eventually die out. To the north, mineralization appears to narrow but continues under cover amid complex faulting (Weigles Fault system). To the east, mineralization is covered by an increasingly thick block of Mesozoic sediments due to normal faulting (east block down) along the graben fault.

The Mesozoic rocks of the structural block above the LAF consist predominantly of arkosic siltstones, sandstones, and conglomerate. Subordinate andesite flows or sills within the Arkose range from a few tens of feet to several hundred feet thick. Also, structurally wedged into the Upper Plate block at the base of the Arkose is the Glance Conglomerate, a limestone-cobble conglomerate, and some occurrences of relatively fresh Paleozoic formations.

New QUEMSCAN® data from 107 composite samples (averaging 30 feet (9.1 meters) of core each) collected from Augusta and Hudbay drilled core provides a preliminary mineralogical characterization of the Rosemont deposit. In bulk terms, the total sulfide volume content of the non-oxidized mineralized skarn is less than 2.7%. Pyrite and chalcopyrite comprise approximately 25% and 35% of the total sulfides content, respectively; along with bornite (20%) and chalcocite (12%). The ratio of these main sulfide minerals is variable through the stratigraphy of the deposit owing to competing, over-printing pulses of mineralization and possible supergene effects. Mineralization in the Horquilla formation is richer in bornite and chalcocite (40% and 35% of total sulfides, respectively) and lower in pyrite and chalcopyrite (5% and 15% respectively) compared to the other mineralized units. Molybdenite is a minor phase but appears to be distributed throughout the skarn and in peripheral portions of the deposit. Gold and silver are present in small amounts across the deposit and are thought to be contained in the primary sulfide minerals. Chalcocite, covellite, native copper and a suite of other secondary copper oxide and carbonate minerals are found in fault and fracture zones in the skarn.

## 7.7 Mineralization Domains

Three mineralization domains (oxide, mixed and sulfide) were defined based on the soluble to total copper ratio ("ASCu/Tcu") collected in the Augusta (2005-2012) and Hudbay (2014 and 2015) drilling programs. The Augusta analytical protocol included soluble copper assays only in zones where copper oxides were observed in core. Hudbay's 2014 and 2015 programs included soluble copper assays for all samples regardless of the dominant logged mineralogy.

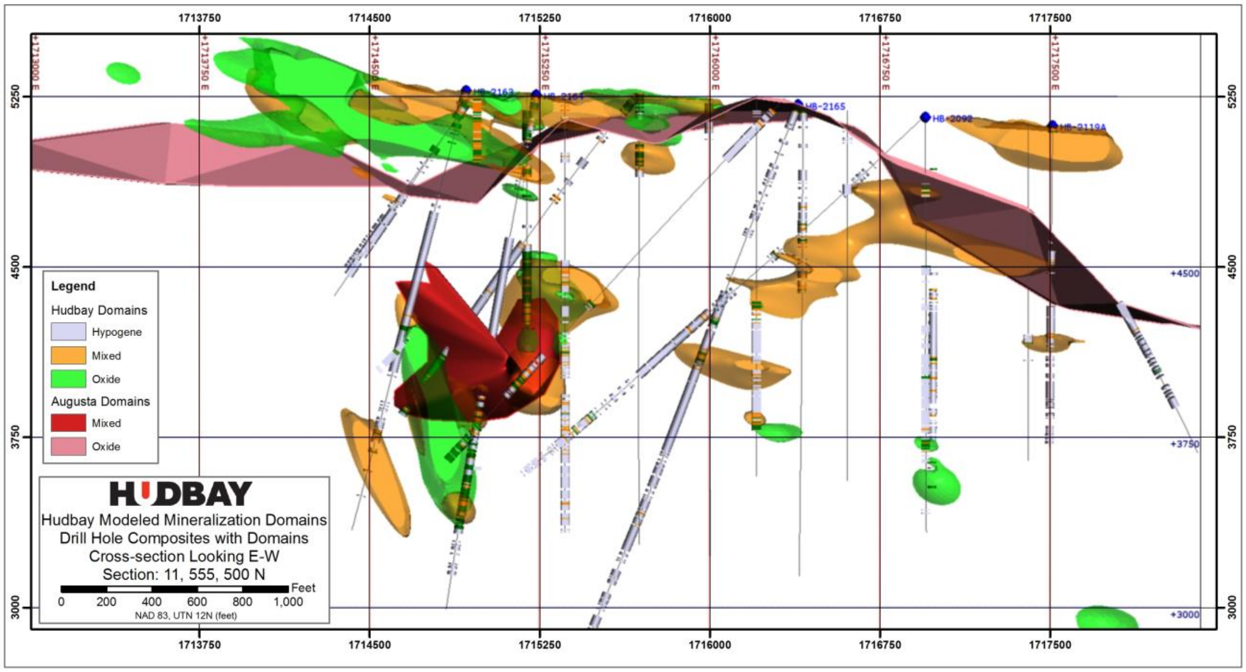
For the domains definition the ASCu/TCu ratio was modelled in Leapfrog using samples with TCu > 0.05%. Two ASCu/TCu ratio shells were interpolated (spheroidal indicator interpolant) for values of > 0.3 and > 0.5. The remaining part of the deposit constitutes the sulfide domain.

The oxidized zones including the 0.3 – 0.5 ASCu/TCu (Mixed) and the > 0.5 ASCu/TCu (Oxide) in part are bounded by a continuous blanket with a gentle east-dipping attitude. The blanket is defined by a sharp decrease in ASCu/TCu ratios that coincides with the LAF.

Other irregular oxidized zones are located in the hanging wall of the Backbone Fault with special development at the intersection with the Weigles Butte Fault, where a bulbous body of mixed mineralization projects deeply down-dip into the Horquilla Formation.

Some significant differences were noted between the current and previous domaining outcomes (Figure 7-9). Augusta domain modelling was built on down hole coding based on the soluble copper data selectively collected in some holes and visual examination of core. The new modelling is based exclusively on analytical data spread throughout the mineralized zones, including the hole core length unbiased acid soluble data collected by Hudbay. This new approach allows refining the shape and continuity of the domains within the upper and lower plate. In the upper plate the oxide blanket topography is better resolved and in the lower plate the continuity of the mixed and oxide zones associated with faults and fractures is better constrained.

**FIGURE 7-9: MINERALIZATION DOMAINS SECTION 11,555,500 N**




## 7.8 Alteration and Skarn Development

The Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn that formed in the Paleozoic rocks as a result of the intrusion of quartz latite to quartz monzonite porphyry intrusions. Bornite-chalcopyrite-molybdenite mineralization occurs as veinlets and disseminations in the skarn.

Garnet-diopside-wollastonite skarn, which formed in impure limestone, is the most important skarn type volumetrically. Diopside-serpentine skarn which formed in dolomitic rocks is less significant. Marble was developed in the most purest carbonate rocks, while the more siliceous, silty rocks were converted to hornfels. Both marble and hornfels are relatively poor hosts to mineralization. The main skarn minerals are accompanied by quartz, potassium feldspar, amphibole, magnetite, epidote, chlorite and clay minerals. Quartz latite to quartz monzonite intrusive rocks host strong quartz-sericite-pyrite alteration with minor mineralization. Where the mineralized package of Paleozoic rocks and quartz-latite intrusive outcrop on the western side of the deposit, near surface weathering and oxidation has produced disseminated and fracture-controlled copper oxide minerals.

The Mesozoic and lesser Paleozoic rocks above the LAF are propylitically altered to an assemblage including epidote, chlorite, calcite, and pyrite. Copper mineralization is irregularly developed. The rocks are commonly deeply weathered and limonitic. The original chalcopyrite is typically oxidized to chrysocolla, copper wad and copper carbonates. Supergene chalcocite is locally present.

## 7.9 Clay Proxies

Geochemical proxies were developed using mineralogy data paired to multi-element geochemistry. This allowed populating the whole length of the core drilled in 2014 and 2015 with relevant clay estimates, and modelling the distribution of both Mg and swelling clays within the deposit.

Magnesium-rich clays ("Mg-clays"), defined as the combined percentage of talc + serpentine, were analyzed by XRD and QEMSCAN as part of the metallurgical work. A total of 431 samples were analyzed by XRD and 107 samples were analyzed by QEMSCAN.

Swelling clays ("Sw-clays") were analyzed using cation exchange capacity ("CEC") for a total of 431 samples also for metallurgical work. Sw-clays were collectively determined as a group in which individual swelling clay types were not discriminated.

For Mg-clays, the QEMSCAN data was used as a training data set to fit a multi-linear regression (MLR) using the multi-element geochemical analysis as input variables. QEMSCAN was preferred, given that it has better detection limits than XRD mineralogy. The XRD data has approximately 83% of observations below the detection limit for Mg-clays and, accordingly, it is not suitable to fit the models. For Sw-clays, the CECF proxies data was used as training data set to fit a MLR model using the multi-element geochemical analysis as input variables.

Higher magnesium clay content is generally associated with a dolomitic protore (e.g. Epithaph) and higher swelling clays with the siliciclastic component of the chemical sedimentary rocks (e.g. Earp).

### 7.9.1 Ore Types

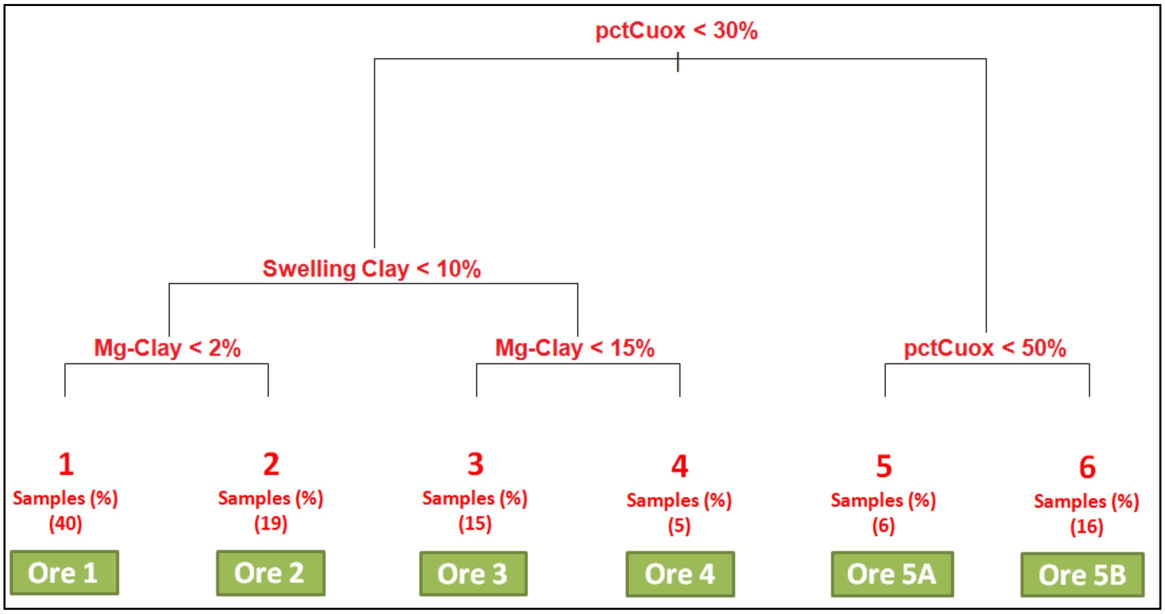
A 6 node classification and regression tree ("CART") was used to classify ore types using 107 measurements of total copper rougher recoveries ("RCu %") as the response variable. The bond work index ("BWI"), sag power index ("SPI"), Sw-clay, Mg-clay; talc + serpentine, and the ratio of soluble copper relative to total copper (pctCuox = Soluble Cu/TCu), a proxy for oxidation of ore minerals, were used as predictor variables in the CART model. The results of the regression tree indicate that the most important variables in hierarchical order are pctCuox, Sw-clay, and Mg-clay.

The CART model suggests that the Rosemont deposit can be subdivided in at least 6 ore types (Figure 7-10). Ore types 1 and 2 are considered clean material. Ore types 3 and 4 are clay-rich material; in which ore type 4 is Mg-clay rich. Ore types 5 and 6 are highly oxidized ores including mixed material (Ore 5) and oxide (Ore 6) material (Figure 7-10).

Once the ore types were established, the geochemical data of these groups were used to create proxies for ore types of the entire deposit. Mineralogical data collected for metallurgical purposes was used to train geochemical data. Mg-clays were analyzed by QEMSCAN (107 samples). Sw-clays were analyzed using CEC for a total of 431 samples. Sw-clays were collectively determined as a group in which; individual Sw-clay types were not discriminated.

Mixed and oxide materials (Ore 5a and Ore 5b, respectively) were modeled using the pctCuox. Geochemical proxies for both Sw-Clays and Mg-clays were developed. For Mg-clays, the QEMSCAN data was used as a training data set to fit a multi-linear regression ("MLR") using the multi-element geochemical analysis as input variables. For Sw-clays, the CEC data was used as training data set to fit a MLR model using the multi-element geochemical analysis as input variables.

**FIGURE 7-10: ORE TYPES CART MODEL**



# 8 DEPOSIT TYPE

The Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn that formed in the Paleozoic rocks as a result of the intrusion of quartz latite to quartz monzonite porphyry intrusions. Genetically, skarns form part of the suite of deposit styles associated with porphyry copper centers, although intrusive rocks are volumetrically minor within the resource area. The skarns were formed as the result of thermal and metasomatic alteration of Paleozoic carbonate and to a lesser extent Mesozoic clastic rocks. Near surface weathering has resulted in the oxidation of the sulfides in the overlying Mesozoic units.

Mineralization is mostly in the form of primary (hypogene) copper, molybdenum and silver bearing sulfides, found in stockwork veinlets and disseminated in the altered host rock. Some oxidized copper mineralization is also present in the upper portion of the deposit. The oxidized mineralization is primarily hosted in Mesozoic rocks, but is also found in Paleozoic rocks on the west side of the deposit and deeper along some faults. The oxidized mineralization occurs as mixed copper oxide and copper carbonate minerals. Locally, enrichment of supergene chalcocite and associated secondary mineralization are found in and beneath the oxidized mineralization.

The Twin Buttes Mine, operated by Anaconda and later by Cyprus, was developed on a deposit with a number of geologic similarities, located approximately 20 miles (32 kilometers) to the west of Rosemont. The Twin Buttes mine was in production from 1969 to 1994. In addition, the Asarco Mission Mine, also located about 20 miles (32 kilometers) to the west of Rosemont, has some common geologic characteristics.

# 9 EXPLORATION

Prospecting began in the Rosemont and Helvetia Mining Districts in the mid-1800s and by 1875 copper production was first recorded, which continued sporadically until 1951. By the late 1950s, exploration drilling had discovered the Rosemont deposit. A succession of major mining companies subsequently conducted exploratory drilling of the Rosemont deposit and the nearby Broadtop Butte, Peach Elgin and Copper World mineralized areas.

Augusta acquired the Rosemont property in 2005 and performed infill drilling at the Rosemont deposit along with exploration geophysical surveys. A Titan 24 induced polarization/resistivity (“DCIP”) survey over the Rosemont deposit, performed in 2011, discovered significant chargeability anomalies which are partially-tested. These anomalies appear to define mineralization and also certain unmineralized lithologic units. A regional scale airborne magnetics survey was also completed in 2008.

Two infill drilling campaigns were completed by Hudbay in and beneath the Rosemont deposit in the fall of both 2014 and 2015. In addition to chemical assaying, magnetic susceptibility and conductivity measurements were taken using the Terraplus’ KT-10 & KT-20 instruments at approximately every 10-feet (3 meters) intervals of recovered core from the drilling program. The magnetic susceptibility data has been used from both drilling programs as a constraint for a 3D inversion of the deposit with an interpretation in progress. A single test-line of DCIP data was collected over the Rosemont deposit using the DIAS Geophysical (3D Survey/Mapping) in April 2015 for comparison to the previously completed Titan 24 survey.

Hudbay analyzed all samples of the 2014 and 2015 drilling programs with ICP multi-element geochemistry. This new geochemical data set was used to classify rocks according to chemical indexes in a ternary diagram defined by Siliciclatic, Limestone and Dolomitic vertices. The lithogeochemical groups honour the deposit stratigraphy and geochemical attributes and proved to be a useful tool for geological modeling and vectoring.

A mapping and geochemical sampling program was completed in the latter half of 2015 on the Rosemont property to reassess the interpretation of the regional geology and deposit setting. This was followed by a structural interpretation using both surface and drill core measurements to aid in the geotechnical evaluation of the Project.

# 10 DRILLING

Extensive drilling has been conducted at the Rosemont deposit by several successive property owners. The most recent drilling was by Hudbay, with prior drilling campaigns completed by Banner, Anaconda Mining Co., Anamax and Asarco and Augusta. Table 10-1 summarizes the drill holes used to estimate the current mineral resource estimate, with regional exploration holes excluded.

**TABLE 10-1: ROSEMONT DEPOSIT DRILLING SUMMARY**

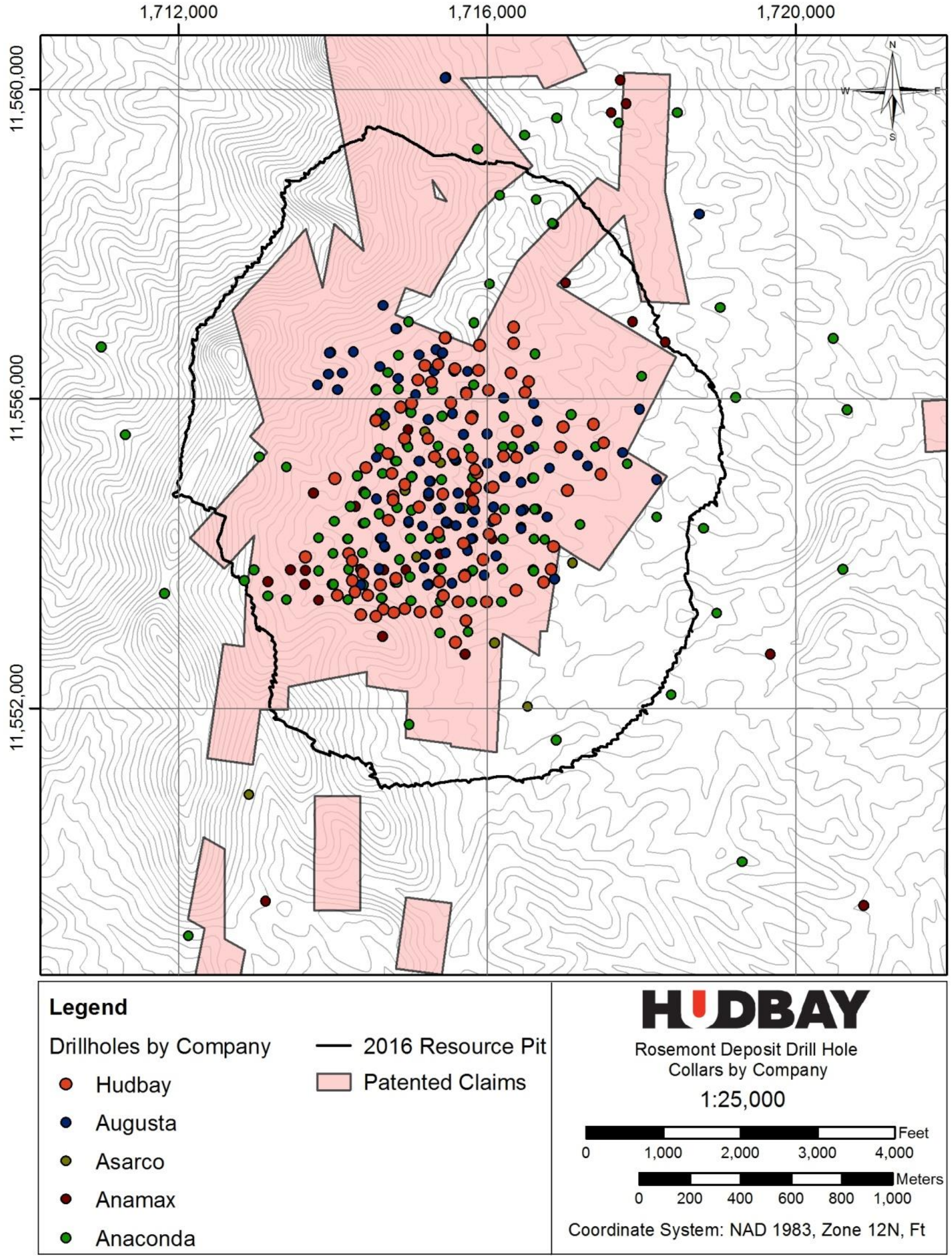
| | | Drill Holes | | |
|---|---|---|---|---|
| **Company** | **Time Period** | **Number** | **Feet** | **Meters** |
| Banner Mining | 1950s to 1963 | 3 | 4,300 | 1,311 |
| Anaconda Mining | 1963 to 1973 | 113 | 136,838 | 41,708 |
| Anamax | 1973 to 1986 | 52 | 54,350 | 16,566 |
| ASARCO | 1988 to 2004 | 11 | 14,695 | 4,479 |
| Augusta | 2005 to 2012 | 87 | 132,525 | 40,394 |
| Hudbay | 2014 to 2015 | 90 | 168,286 | 51,294 |
| **Total** | | **355** | **510,780** | **155,686** |

The drill holes in the database were all drilled using diamond drilling (coring) methods. In some cases, the top portion of the older holes were drilled using a rock bit to set the collar or by rotary drilling methods and then switching to core drilling before intercepting mineralization. A map showing the location of the drill holes by company is provided in Figure 10-1 along with an outline of the mineral resource pit shell limits for the Rosemont deposit. Exploration holes drilled using rotary or older “churn” drill holes were excluded from the resource database.

In all of the drilling campaigns, efforts were consistently made to obtain representative samples by drilling either H-size (2.5 inch or 63.5 mm diameter) or N-size (1.9 inch or 47.6 mm diameter) core.

Core recoveries within the ore zone for the Hudbay and Augusta drilling programs average 96% and core recoveries within the pit elsewhere average 89%, lending confidence that quality samples were obtained including for oxidised intervals. Generally, drill programs were on east-west grid lines spaced approximately 200 feet (61 meters) apart.

**FIGURE 10-1: ROSEMONT DEPOSIT DRILL HOLE LOCATIONS BY COMPANY**




The majority of the Anaconda Mining Co., Anamax and Asarco drill core is available and was systematically re-logged by Augusta personnel to be geologically consistent with their drilling from 2005 to 2012. In addition, with re-logging, they completed some re-sampling for geochemical analyses.

## 10.1 Banner Mining Company (1961 to 1963)

The first significant core drilling campaign on the Property was by the Banner, beginning in about 1961. Banner completed primarily shallow diamond drill holes, many of which were subsequently deepened by Anaconda Mining Co. Three drill holes included in the resource database were shallow holes initially collared by Banner and were significantly deepened during subsequent drilling programs conducted by Anaconda Mining Co.,. These holes have a combined length of 4,300 feet (1,311 meters).

## 10.2 The Anaconda Mining Co., (1963 to 1986)

Anaconda acquired Banner Rosemont Holdings around 1963 and conducted exploration at the Rosemont deposit and in adjacent mineralized areas. Between the years of 1963 and 1973, they completed 113 diamond drill holes at Rosemont for a total of 136,838 feet (41,708 meters). These holes were primarily drilled vertically. Down-hole and collar surveys completed by company surveyors were conducted during drilling or immediately following drill hole completion for selected holes. Anaconda drilled approximately 85% of the larger N-size core and 15% of the smaller B-size core (1.4 inch or 36.4 mm diameter). Overall core recovery was more than 85%.

Exploration subsequently transferred to Anamax Mining Co., (an Anaconda Mining Co., and Amax Inc., joint venture) around 1973, which continued extensive diamond drilling and analytical work until 1986. Anamax completed 52 core holes for a total of 54,350 feet (16,566 meters). These holes were almost exclusively drilled as angle holes inclined -45° to -55° to the west, approximately perpendicular to the east-dipping, Paleozoic, metasedimentary host rocks. Down-hole and collar surveys by company surveyors were conducted during drilling or immediately following drill hole completion for the majority of the holes. Anamax drilled approximately 80% N-sized core and 20% B-sized core, with an overall core holes recovery of more than 88%.

## 10.3 ASARCO Mining Co., (1988 to 2004)

Asarco acquired the Rosemont property in 1988 and conducted exploration until 2004, completing 11 vertical drill holes for a total of 14,695 feet (4,479 meters) in the deposit area (a 12th hole was drilled to the east of the deposit and is not part of the Project’s database). Data was available from eight of the Asarco core holes in the Rosemont deposit area and were incorporated into Hudbay’s resource estimate. Down-hole survey data, if taken, were not available for the Asarco holes. Drill hole collars were surveyed by company surveyors. The size of core collected by Asarco was predominantly N-sized. Core recovery information was not available but re-logging by Augusta personnel indicated it to be of similar quality to that of other drilling campaigns.

## 10.4 Augusta Resource (2005 to 2012)

Augusta optioned the Rosemont property in 2005 and conducted diamond drilling in several campaigns, from 2005 to 2012. In total, Augusta completed 87 core holes for a total of 132,525 feet

(40,394 meters). Of these, 60 holes were drilled for the purposes of delineating the deposit and providing infill information, while six were exploration holes outside of the planned pit area, but close enough to be a part of the resource database. The remaining 21 core holes support geotechnical (13) or metallurgical (8) studies. Augusta holes were usually collared by rock-bitted through overburden, and then drilled with larger HQ-sized core as deeply as possible and finished with NQ-sized core if a reduction in core size was required due to ground conditions.

Most of the holes were oriented vertically, although a few of the holes were inclined to intercept targets from reasonably accessible drill pad locations. All drill holes were down-hole surveyed using a Reflex EZ-Shot survey instrument which measures inclination/dip and azimuth direction, with measurements generally taken every 100 feet (30 meters) down the hole during 2008 and every 200 or 500 feet (61 or 152 meters) down the hole during 2005, 2006 and 2011 to 2012 drill campaigns. The initial drill hole collar locations were surveyed by Putt Surveying of Tucson, Arizona, while all later drilling locations were measured and certified by Darling Environmental & Surveying of Tucson, Arizona.

## 10.5 Hudbay (2014 to 2015)

Shortly after acquiring the Project, Hudbay initiated a 44 core hole drill program in September 2014 and completed 93,122 feet (28,384 meters) of diamond drilling by December 2014. The Phase I drill program was conducted entirely within the Rosemont resource, on patented claims and was designed to gain an initial understanding of the geological setting and mineralization, provide infill drilling density along with metallurgical, geochemical and geophysical data.

Diamond drilling was primarily HQ-sized core as deeply as possible and finished with NQ-sized core, if a reduction in core size was required due to ground conditions. If ground conditions warranted, drill holes were collared in larger PQ size (3.3 inch or 83 mm diameter) and reduced to HQ as ground conditions improved. Drilled length and respective recoveries were PQ 4,326 feet (1,319 meters) with 83.5% recovery, HQ 85,583 feet (26,086 meters) with 95.9% recovery, and NQ 3,213 feet (979 meters) with 92.8% recovery (statistics include HB-2119 that was abandoned due to poor ground conditions after drilling approximately 200 feet (60 meters).

Forty-three of the drill holes were orientated vertically, with one inclined in order to intercept a target area from an accessible drill pad location. Down hole surveying was conducted on 200 feet (61 meters) intervals with either a Multishot Reflex or a Surface Recording Gyro Survey instrument, both instruments measured inclination/dip and azimuth direction. Collar locations were surveyed and certified by Darling Environmental & Surveying of Tucson, Arizona

From August to November 2015, Hudbay completed a 46 core hole, 75,164 feet (22,910 meters) diamond drill program. The Phase II drill program was conducted entirely within the Rosemont resource, on patented claims and was designed to gain a further understanding of the geological setting and mineralization, provide infill drilling density along with metallurgical, geotechnical, geochemical and geophysical data.

Diamond drilling was primarily HQ-sized core as deeply as possible and finished with NQ-sized core, if a reduction in core size was required due to ground conditions. If ground conditions warranted, drill holes were collared in larger PQ size and reduced to HQ as ground conditions improved. Twenty-two of the drill holes were oriented vertically, with 24 inclined drill holes. Eight holes were inclined for drilling oriented core utilizing the Reflex ACT III instrument to gather geotechnical structural data, and 16 holes were inclined in order to intercept a target area from an accessible drill pad location. Down hole surveying was conducted on 200 feet (61 m) intervals with either a Multishot Reflex or a Surface Recording Gyro Survey instrument, both instruments measured inclination/dip and azimuth direction. Collar locations were surveyed and certified by Darling Environmental & Surveying of Tucson, Arizona.

# 11 SAMPLING PREPARATION, ANALYSES, AND SECURITY

## 11.1 Hudbay 2014

### 11.1.1 Core Logging

The drilling contractors thoroughly cleaned the drill core retrieved from the core tube before piecing all the segments together in the core boxes. Footage marker blocks were inserted in the core boxes after each run to indicate the relative down-hole depth. Core boxes were labelled with the hole name, box number and from - to footage measurements before securely closing the box with a tightly fitted lid. Core boxes were delivered to the core processing areas of either Rosemont Camp or Hidden Valley Ranch by the drilling contractors, and neatly stacked on top of pallets. Private 24-hour per day security guards administered by Securitas Inc., controlled site access and oversaw sample security at each camp and drill site.

Core boxes were loaded onto conveyor racks by the geotechnicians and geologists for logging. Prior to measuring the core recovery parameters and Rock Quality Data ("RQD"), visual checks were performed for incorrect placement and orientation of core fragments. Any discrepancies caused by mislabeled or misplaced footage tags were resolved by consulting the drilling contractors. The drill core was marked with cut lines designed to provide the most representative split.

Standard parameters for core recovery and RQD for each drill run were measured by either the trained geotechnicians or geologists and recorded on tablets. The RQD program was administrated and monitored by the consulting engineering firm CNI. All core logging was completed by experienced contract geologists. At the start of each drilling campaign, all geologists were provided with three days of training on the rock types, alterations, mineralization and structures found on the property.

All drill holes were logged using tablets networked to a FileMaker Inc. (FileMaker) database hosted on a laptop using a local hotspot network. Drill core was divided into sub intervals based on the rock types observed by the geologists. A local formation name was assigned to each interval if a positive identification was made. Each interval was further described for alteration, mineralization, and oxidation state of the primary sulfides. Any significant veins found were also logged along with identifiable structures.

### 11.1.2 Sample Selection

All core samples for assaying were assigned by the core logging geologist. The typical sample interval was about 5 feet (1.5 meters) while being mindful of lithological contacts. Geologists were responsible for filling out two of the three paper tags for each sample in the sampling book with the hole name and sample interval. Sample tag numbers along with the sampled intervals were also entered into the core logging database. For core samples, two of the three tags were stapled into the core box at the starting point of each sample, one to remain there, and the second to accompany the sampled split in the sample bag. Lines were drawn on the core using a permanent marker to indicate the beginning and end of each sample. For QA/QC samples consisting of duplicates (two coarse rejects and analyses to be generated from the same interval), two sets of sample tags were stapled into the core box, and a double line was drawn on the core. For other QA/QC samples (standards and blanks), a single sample tag was stapled into the core box indicating the QA/QC sample's relative position in the sequence and the sample type.

#### 11.1.3 Core Photographs

Completely logged core boxes with the sample intervals marked and sample tags inserted were passed on to geotechnicians who photographed all core boxes individually or in groups of two using a digital camera (Nikon Coolpix L830) mounted to a tripod in natural light. The hole name, box numbers and the from and to intervals were written on a white board, which was photographed with the core boxes along with a color patch and a scale for reference. All photos taken were loaded on to a laptop computer dedicated to core photos and reviewed by the on-site database manager or the lead geotechnician. Any photos deemed unacceptable were retaken by the geotechnicians. All core photos were renamed by the site database manager based on hole name, box numbers and intervals and uploaded to the Google Drive.

#### 11.1.4 Core Cutting

Prior to cutting core, the database manager printed a sample list for each drill hole that included the sample identification number, hole name, sample type and the start and end footage of each sample. This list was used by the geotechnicians to label sample bags. At the core cutting station, a bucket was lined with the correctly labelled sample bag and the corresponding core box was placed on to the work table next to the core saw. The core cutters separated one of the two sample tags stapled in the core box at the start of the sample and placed them in the sample bag. For coarse duplicate samples, the second QA/QC sample tag was also placed in the same bag. Core was cut along the cut line drawn by the geologist and the right half of the sample was placed in the sample bag and the left was placed back in the core box. In gouge and rubble intervals, an aluminium sampling scoop was used to separate the gouge into two halves in the core boxes and the right half was scooped into the sample bag. Completed sample bags were closed using the bag draw strings and secured at the neck using two zip ties. These bags were moved to a dry storage area away from the core cutting saws and stacked in orderly rows. All saws and sampling buckets were rinsed with water after cutting each sample to prevent cross contamination.

#### 11.1.5 Sample Dispatching

Samples were dispatched using the dispatching module in the core logging database. Samples were dispatched typically in batches of 100 samples from the same drill hole. A requisition form was automatically created that listed the range of sample numbers, job order number, requested analytical codes and any special instructions. A corresponding sample list for each requisition was also created. The requisition form and the sample list were emailed to the preparation laboratory prior to sample shipment. QA/QC samples including Blanks and Standards were prepared by the database manager prior to sample shipment. On the day of sample shipment, sample bags were cross-checked with the sample requisition list before packing them into large sacks placed on wooden pallets. These sacks were secured to the wooden pallets using shrink wrap and strap cables before being loaded on the truck.

### 11.1.6 Sample Preparation

Drill core samples were picked up at the core processing facilities and transported via UPS to Inspectorate America Corporation's ("Inspectorate") preparation facility at Sparks, Nevada, USA. Samples were weighed upon arrival, dried at 60°C, and crushed in jaw crushers to ≥70% passing through 10 mesh (2 mm). The entire crushed sample was homogenized, riffle split, and a 1,000 g subsample was pulverized to ≥85% passing through 200 mesh (75 µm) using Essa standard steel grinding bowls. Jaw crushers, preparation pans, and grinding bowls were cleaned by brush and compressed air between samples. Cleaning with a quartz wash was conducted between jobs and between highly mineralized samples.

Once samples were pulverized a 150 g subsample pulp was collected and air freighted to Bureau Veritas Commodities Canada Ltd., ("Bureau Veritas") in Vancouver, Canada, for analysis. The remaining 850 g master pulps and the coarse rejects were stored temporarily at the Inspectorate laboratory and then moved to a storage facility in Tucson.

Bureau Veritas is independent from Hudbay and has a quality system that is compliant with the International Standards Organization ("ISO") 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories.

The sample preparation, analysis, and security procedures are considered industry standard, adequate, and acceptable.

### 11.1.7 Bulk Density

A total of 922 samples in 43 drill holes were collected for specific gravity determinations at the Inspectorate preparation facility. The drill core samples, 4 to 6 inches (10 to 15 cm) long, were taken every 100 feet (30 meters) from half-split core.

At the laboratory, samples were dried at 105°C overnight and then allowed to cool to room temperature. The initial weight of the sample is determined using a top loading balance and recorded. Balances are calibrated using a 10 g, 50 g and 250 g calibration weight. The sample is immersed in a pan containing molten paraffin, then immediately removed from the molten paraffin and shaken a few times to remove excess wax while hardening. The wax coated sample is re-weighed using the top loader and the weight is recorded. The standard water displacement method is then used to calculate the specific gravity.

The method is industry standard and suitable for bulk density determinations.

### 11.1.8 Assay Methodology

A total of 18,361 drill core samples were analyzed for 45 elements by Inductively Coupled Plasma Mass Spectrometry (ICP-MS) after 4-acid digestion (Method MA200). The specifics of the analyses for copper, molybdenum and silver are given in Table 11-1. Samples with concentrations above the upper limit of detection and those with copper ≥ 8,000 ppm and molybdenum ≥ 3,200 ppm were systematically re-assayed by high-grade 4-acid digestion, Method MA370, and ICP-OES. The 8,000 and 3,200 ppm copper and molybdenum grade thresholds, respectively, were selected to maintain accuracy at grade levels within 20% of the upper limit of detection in Method MA200.

**TABLE 11-1: BUREAU VERITAS ASSAY SPECIFICATIONS**

| Element | Cu | Cu over limits | Cu Soluble | Mo | Mo over limits | Ag | Au |
|---|---|---|---|---|---|---|---|
| **Unit** | **ppm** | **%** | **%** | **ppm** | **%** | **ppm** | **ppm** |
| Lower Detection Limit | 0.1 | 0.001 | 0.001 | 0.1 | 0.001 | 0.1 | 0.005 |
| Upper Detection Limit | 10,000 | - | 10 | 4,000 | - | 200 | 10 |
| Digestion | 4 acids | 4 acids | Sulfuric acid at 5% | 4 acids | 4 acids | 4 acids | Fire assay |
| Instrumental Finish | ICP-MS | ICP-OES | AAS | ICP-MS | ICP-OES | ICP-MS | AAS |
| Method Code | MA200 | MA370 | GC921 | MA200 | MA370 | MA200 | FA430 |

To investigate the oxidation of primary copper sulfides, all drill core samples were analyzed for acid soluble copper ("ASCu") Method GC921 with an acid leach at room temperature using 5% sulfuric acid ($H_2SO_4$). Samples were agitated in a mechanical shaker for one hour, and then made up to volume with demineralised water. The solution was filtered and analyzed by Atomic Absorption Spectroscopy ("AAS").

Gold was analyzed in all samples from seven selected drill holes across the Rosemont deposit. A total 3,155 samples were analyzed for gold; which represents approximately 18% of Hudbay's 2014 drilling and sampling program. Gold was determined by lead-collection fire assay fusion, for total sample decomposition, and AAS instrumental finish ("Method FA430"). Fire assays were performed on 30 g subsample pulps to circumvent potential problems due to nugget effect.

As part of Hudbay's QA/QC program, QA/QC samples, shown in Table 11-2, were systematically introduced in the sample stream to assess adequate sub-sampling procedures, potential cross-contamination, precision, and accuracy. On average, the sampling program included 6% certified reference materials ("CRM"), 6% certified blanks, and 6% coarse duplicates. Blanks and CRMs were prepared by Ore Research and Exploration ("OREAS") in Australia. All QA/QC samples were analyzed following the same analytical procedures as those used for the drill core samples.

**TABLE 11-2: SUMMARY OF QA/QC SAMPLES**

| Category | No. of samples | Relative Frequency[1] | Type | Elements |
|---|---|---|---|---|
| OREAS 501b | 214 | 1.2% | CRM | Cu-Mo-Ag-Au |
| OREAS 502b | 211 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 503b | 210 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 504b | 198 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 902 | 55 | 0.3% | CRM | Cu-Mo-Ag-Cu Soluble |
| OREAS 930 | 179 | 1.0% | CRM | Cu-Mo-Ag |
| OREAS 22d | 534 | 3.0% | Certified Blank | Cu-Mo-Ag-Au |
| OREAS 26a | 554 | 3.0% | Certified Blank | Cu-Mo |
| Duplicates | 1,086 | 6.0% | Coarse Duplicate | Cu-Mo-Ag |
| Duplicates | 189 | 5.9%[2] | Coarse Duplicate | Cu-Mo-Ag-Au |

[1]Frequencies estimated relative to a sampling program comprising 18,361 samples

[2]Frequencies estimated relative to 3,155 samples analyzed for gold

### 11.1.9 Blanks

Certified OREAS blanks, shown in Table 11-3, were inserted into the sample stream approximately one every twenty samples to monitor potential cross-contamination.

**TABLE 11-3: OREAS CERTIFIED BLANKS**

| | Cu | Mo | Ag | Au |
|---|---|---|---|---|
| **Unit** | **ppm** | **ppm** | **ppm** | **ppb** |
| OREAS 22d | 9.23 | 2.36 | <0.1 | <1 |
| OREAS 26a | 50 | 1.50 | - | - |
| Certified Method | 4 acids | 4 acids | 4 acids | Fire assay |

Fine and coarse blanks were systematically inserted at the same rate for a total of 1,088 blanks representing 5.9% of the sampling program. OREAS 22d is a certified fine blank prepared from quartz sand. OREAS 26a is a certified coarse blank sourced from fresh and non-mineralized olivine basalt.

Fine blank OREAS 22d and coarse blank OREAS 26a were used to assess potential contamination during assaying and sample preparation, respectively. These blanks contain low trace level concentrations of copper, molybdenum, silver, and gold. Blank failure due to potential contamination issues is documented when the blank values exceed five times the lower limit of detection. For those blanks with concentration levels above the lower detection limits the failure thresholds are set to values that exceed the certified best value (“CBV”) plus three standard deviations. A summary of the blank performance is shown in Table 11-4.

**TABLE 11-4: SUMMARY OF BLANK PERFORMANCE**

| Fine Blank OREAS 22d | | | | | |
|---|---|---|---|---|---|
| **Element** | **No. of Blanks** | **Failed Blanks** | **Failure Rate** | **Maximum Contamination of CBV** | **Average Contamination of CBV** |
| Cu | 534 | 32 | 6.0% | 7.4 ppm | 1.7 ppm |
| Mo | 534 | 12 | 2.2% | 4.0 ppm | 1.2 ppm |
| Ag | 534 | 1 | 0.2% | 3.2 ppm | 3.2 ppm |
| Au | 89 | 0 | 0.0% | 0 | 0 |
| | | | | | |
| **Coarse Blank OREAS 26a** | | | | | |
| **Element** | **No. of Blanks** | **Failed Blanks** | **Failure Rate** | **Maximum Contamination of CBV** | **Average Contamination of CBV** |
| Cu | 554 | 39 | 7.0% | 294 ppm | 40 ppm |
| Mo | 554 | 8 | 1.4% | 69 ppm | 19 ppm |
| Ag | 554 | 1 | 0.2% | 0.1 ppm | 0.1 ppm |
| Au | 98 | 0 | 0% | 0 | 0 |

A total of 1,088 blank samples were systematically inserted along with the drill core samples and analyzed at Bureau Veritas. Contamination with copper and gold was insignificant. There are very few isolated cases of contamination at high-grade levels for silver and molybdenum. However, the overall blank failure rates are very low ranging from 0 to 7%.

The performance of blanks indicates no significant issues with contamination and therefore it is concluded that the results are acceptable and adequate for the resource estimation.

#### 11.1.10 Standards

OREAS certified reference materials, as shown in Table 11-5, were inserted one every twenty samples. In total, 1,067 CRMs were analyzed for a total insertion rate of 5.8%.

**TABLE 11-5: OREAS CERTIFIED REFERENCE MATERIAL**

| Unit | Cu % | Mo ppm | Ag ppm | Au ppm | Cu Soluble % |
|---|---|---|---|---|---|
| OREAS 501b | 0.260 | 99 | 0.778 | 0.248 | - |
| OREAS 502b | 0.773 | 238 | 2.090 | 0.495 | - |
| OREAS 503b | 0.531 | 319 | 1.540 | 0.695 | - |
| OREAS 504b | 1.110 | 499 | 3.070 | 1.610 | - |
| OREAS 902 | 0.301 | 12.2 | 0.343 | - | 0.111 |
| OREAS 930 | 2.520 | <1.5 | 9.0 | - | - |
| Certified Method | 4 acids | 4 acids | 4 acids | Fire assay | Sulfuric acid at 5% |

OREAS 501b to 504b represent a blend of porphyry copper-gold mineralization, barren gangue, and minor quantities of copper and molybdenum concentrate. Copper and gold mineralization occurs as stockwork quartz veins and disseminations associated with potassic alteration. Primary copper sulfides include bornite and chalcopyrite.

OREAS 902 was prepared from oxidized copper ore hosted in dolomitic, carbonaceous, and argillaceous sedimentary rocks. Copper oxides consist primarily of malachite, cuprite, chrysocolla, and chalcocite. Chalcopyrite is the primary copper sulfide.

OREAS 930 was prepared from a copper orebody hosted in carbonaceous siltstones and mudstones. Sulfides include chalcopyrite, bornite, pyrrhotite, pyrite, sphalerite, galena, and cubanite.

More than 30 and up to 214 samples were analyzed per CRM, as summarized in Table 11-6, 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, if the absolute relative bias is >10% the acceptance criteria is set relative to the CBV and standard deviation recommended by the CRM certificates.

**TABLE 11-6: SUMMARY OF CRM PERFORMANCE**

| Total Cu (ppm) | | | | | | |
|---|---|---|---|---|---|---|
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 214 | 0 | 0.0% | 2,600 | 2,581 | -0.7% |
| OREAS 502b | 211 | 0 | 0.0% | 7,730 | 7,532 | -2.6% |
| OREAS 503b | 210 | 0 | 0.0% | 5,310 | 5,241 | -1.3% |
| OREAS 504b | 198 | 2 | 1.0% | 11,100 | 11,000 | -0.9% |
| OREAS 902 | 55 | 1 | 1.8% | 3,010 | 3,028 | +0.6% |
| OREAS 930 | 179 | 4 | 2.2% | 25,200 | 25,480 | +1.1% |
| | | | | | | |
| **Mo (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 214 | 2 | 0.9% | 99 | 96 | -3.0% |
| OREAS 502b | 211 | 0 | 0.0% | 238 | 230 | -3.4% |
| OREAS 503b | 210 | 2 | 1.0% | 319 | 310 | -2.8% |
| OREAS 504b | 198 | 1 | 0.5% | 499 | 487 | -2.4% |
| OREAS 902 | 55 | 2 | 3.6% | 12.2 | 11.9 | -2.5% |
| | | | | | | |
| **Ag (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 214 | 4 | 1.9% | 0.778 | 0.78 | +0.3% |
| OREAS 502b | 211 | 2 | 0.9% | 2.09 | 2.17 | +3.8% |
| OREAS 503b | 210 | 0 | 0.0% | 1.54 | 1.61 | +4.5% |
| OREAS 504b | 198 | 3 | 1.5% | 3.07 | 3.28 | +6.8% |
| OREAS 902 | 55 | 9 | 16.4% | 0.343 | 0.38 | +10.8% |
| OREAS 930 | 179 | 25 | 14.0% | 9.00 | 9.98 | +10.9% |
| | | | | | | |
| **Au (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 37 | 0 | 0% | 0.248 | 0.252 | +1.6% |
| OREAS 502b | 35 | 0 | 0% | 0.495 | 0.496 | +0.2% |
| OREAS 503b | 32 | 0 | 0% | 0.695 | 0.697 | +0.3% |
| OREAS 504b | 34 | 0 | 0% | 1.610 | 1.596 | -0.9% |
| | | | | | | |
| **Soluble Copper (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 902 | 55 | 0 | 0% | 1,110 | 1,140 | +2.7% |

Accordingly, 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 and triggered re-analysis.

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

$$\text{Bias (\%)} = 100*[(\text{AVeo}/\text{CBV})-1]$$

AVeo represents the average of 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%.

The analytical bias of CRMs for total copper, molybdenum, soluble copper, and gold was good with values in the range between -3.4% and +2.7% (Table 11-6).

The analytical bias of silver in OREAS 501b to 504b (0.8 to 3 ppm Ag) was good to reasonable, with values between +0.3% and +6.8%. However, silver in OREAS 902 and 930 displayed a larger analytical bias, approximately +11% (Table 11-6).

The large bias of silver in OREAS 902 is attributed to the low silver content and large variance (0.343 ppm, SD = 0.043) of this CRM. The certified silver content of OREAS 902 is between three and four times the lower limit of detection (0.1 ppm), and the relative standard deviation (12%) of this CRM is larger than the analytical bias of +11%. Good performance is expected at values at least 10 times the lower limit of detection. Therefore, the performance of OREAS 902 is considered reasonable.

The certified best value for silver of OREAS 930 (9.00 ppm, SD = 1.09) is more than 10 times the lower detection limit. This CRM displayed a bias of 10.9% which is considered unacceptable (Table 11-6). However, the bias is lower than the relative standard deviation of this CRM which is 12%. To further investigate this issue, pulp duplicate re-analysis were resubmitted to Bureau Veritas laboratory for all failed OREAS 930 including eight drill core pulp samples centered on the failed CRMs. In total, 21 failed OREAS 930 and 135 drill core pulp samples with silver between 0.1 and 116 ppm were re-analyzed. After re-assaying, 86% of CRMs yielded silver values within the certified best value. Despite the poor reproducibility of OREAS 930, 90% of the re-assayed drill core pulps in the sample stream of OREAS 930 yielded similar results (silver 01.-117 ppm) evaluated for an absolute relative difference between pulp pairs equal to or smaller than 10%.

The CRM analysis indicates that the analytical accuracy for total copper, molybdenum, and soluble copper is of good quality for the resource estimation. The cause of the poor performance of silver in OREAS 930 is attributed to the large variability (SD = 1.09) of this CRM. It is also noted that OREAS 930 includes minerals such as sphalerite, galena, and cubanite, which are not found in significant quantities at Rosemont. Given the analysis discussed above, and good performance of silver in

OREAS 501b to 504b, it is concluded that the accuracy for silver is also adequate for resource estimation.

#### 11.1.11 Duplicates

Coarse duplicates, approximately one in every twenty samples, were requested to Bureau Veritas laboratory in order to monitor sub-sampling precision. Accordingly, after crushing to 10 mesh (2 mm), a 1,000 g coarse duplicate sub-sample was riffle split and pulverized to ≥85% passing through 200 mesh (75 µm). The duplicate sample was analyzed immediately after its paired sample. A total of 1,086 coarse duplicate samples were inserted for a total rate of 6%. Quarter-core twin sample duplicates and pulp duplicates were not analyzed during Hudbay's 2014 drilling program.

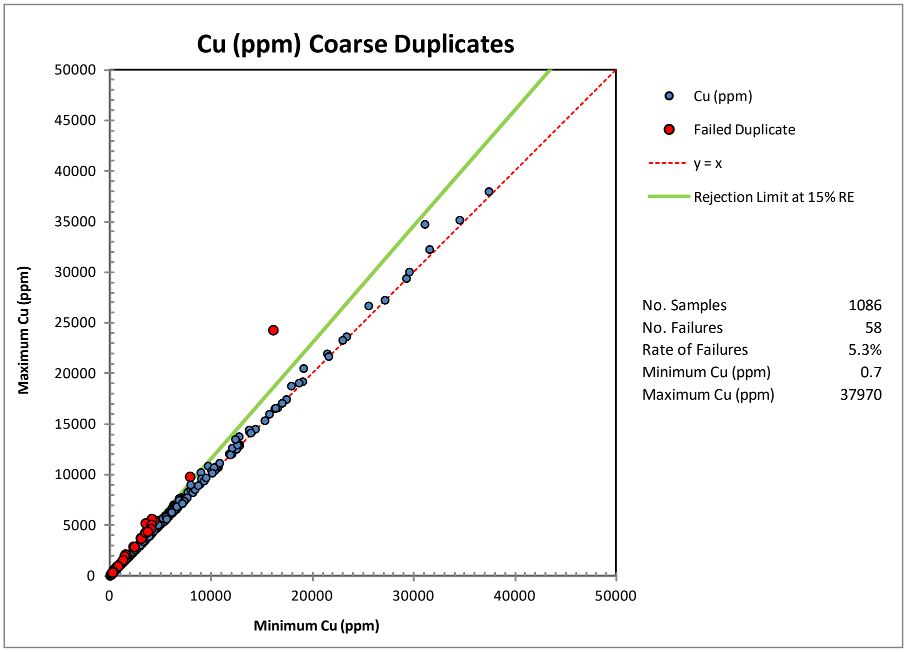
Coarse duplicates were reviewed using the hyperbolic method (Table 11-2) developed by AMEC (Simón, 2004). Minimum and maximum element concentrations of the sample pairs are plotted in the y and x axis, respectively. In the Minimum-Maximum diagrams, all samples plot along and above the x = y line and the failure boundary is given by the 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 15% absolute relative error ("RE"). The RE is calculated as the absolute value of the pair difference divided by the pair average and expressed in percentage. An acceptable level of sub-sampling variance is achieved when the failure rate does not exceed 10% of all sample pairs.

The failure rates of the duplicate pairs for total copper, molybdenum, silver, gold and soluble copper range between 4% to 6% based on the hyperbolic method for an absolute relative error of 15% (Table 11-7, Figure 11-1 to Figure 11-4). It is concluded that the sub-sampling procedures were adequate for all metals used in the resource model.

**TABLE 11-7: SUMMARY OF COARSE DUPLICATE ANALYSIS**

| Element | No. of Samples | No. of Failures | Failure Rate | Accepted Absolute RE |
|---|---|---|---|---|
| Cu | 1,086 | 58 | 5.3% | 15% |
| Mo | 1,086 | 66 | 6.1% | 15% |
| Ag | 1,086 | 66 | 6.1% | 15% |
| Au | 189 | 8 | 4.2% | 15% |
| Soluble Cu | 1,086 | 65 | 6.0% | 15% |

## FIGURE 11-1: COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT




## FIGURE 11-2: MOLYBDENUM COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

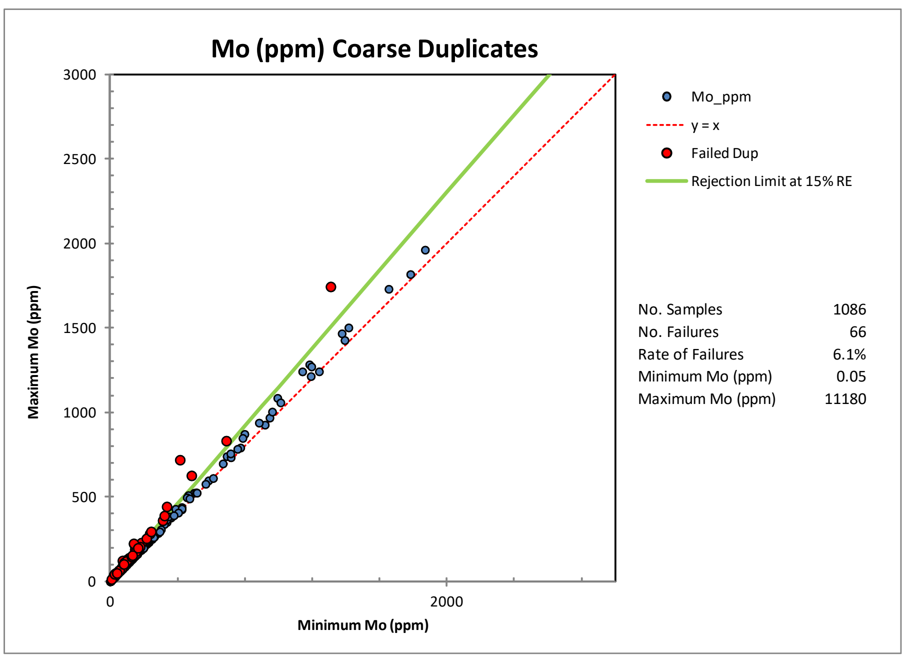


**FIGURE 11-3: SILVER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT**

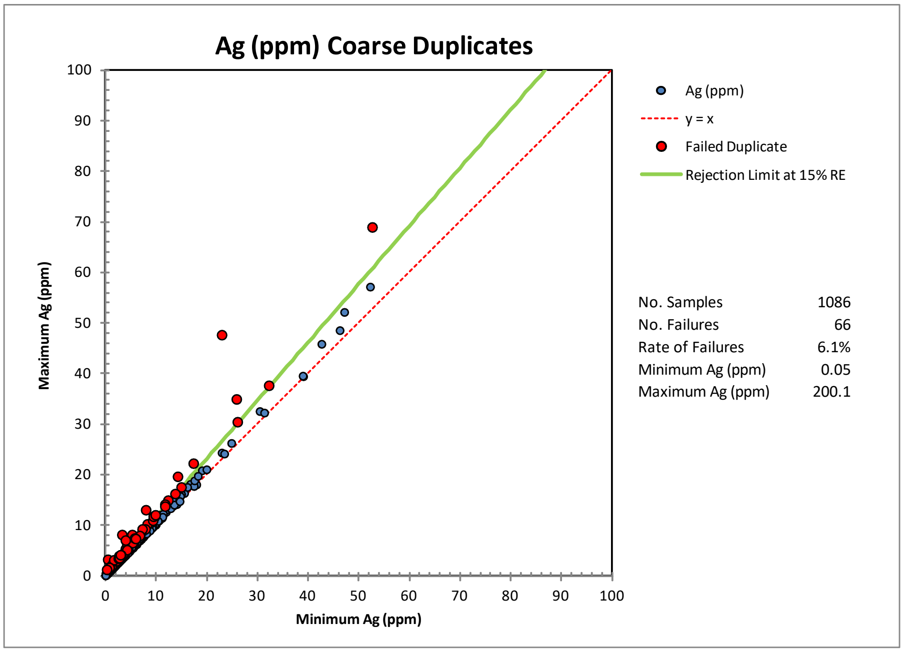
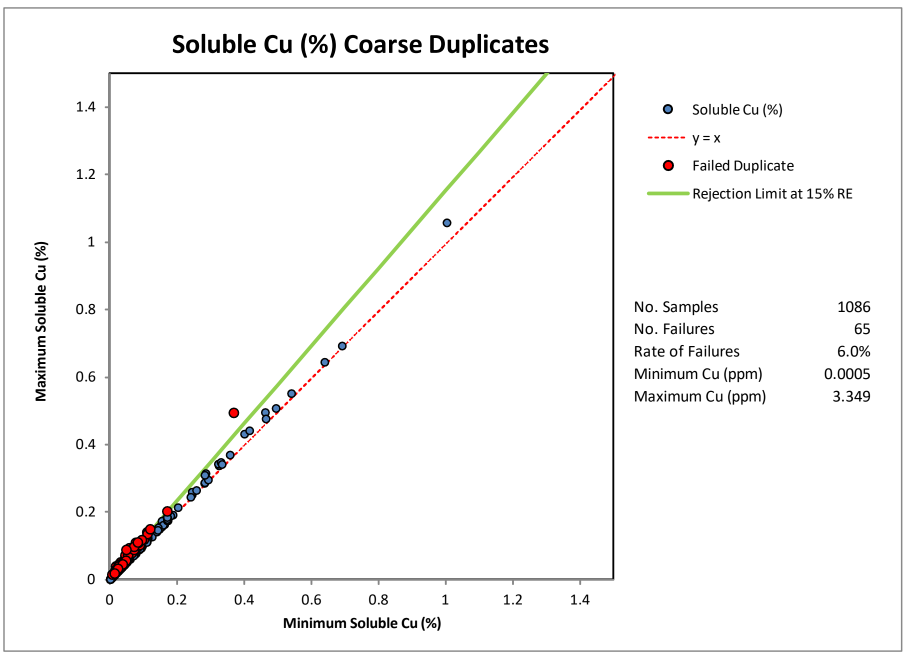



**FIGURE 11-4: SOLUBLE COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT**




### 11.1.12 Check Assaying

A total of 1,000 representative pulp samples (5.5%) were selected and re-analyzed at SGS Canada Inc. ("SGS") laboratory in Vancouver. This laboratory is independent from Hudbay and has a quality system that is compliant with the International Standards Organization ("ISO") 9001 Model for

Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories. Only samples with ≥500 ppm copper were submitted for re-analysis at the secondary laboratory.

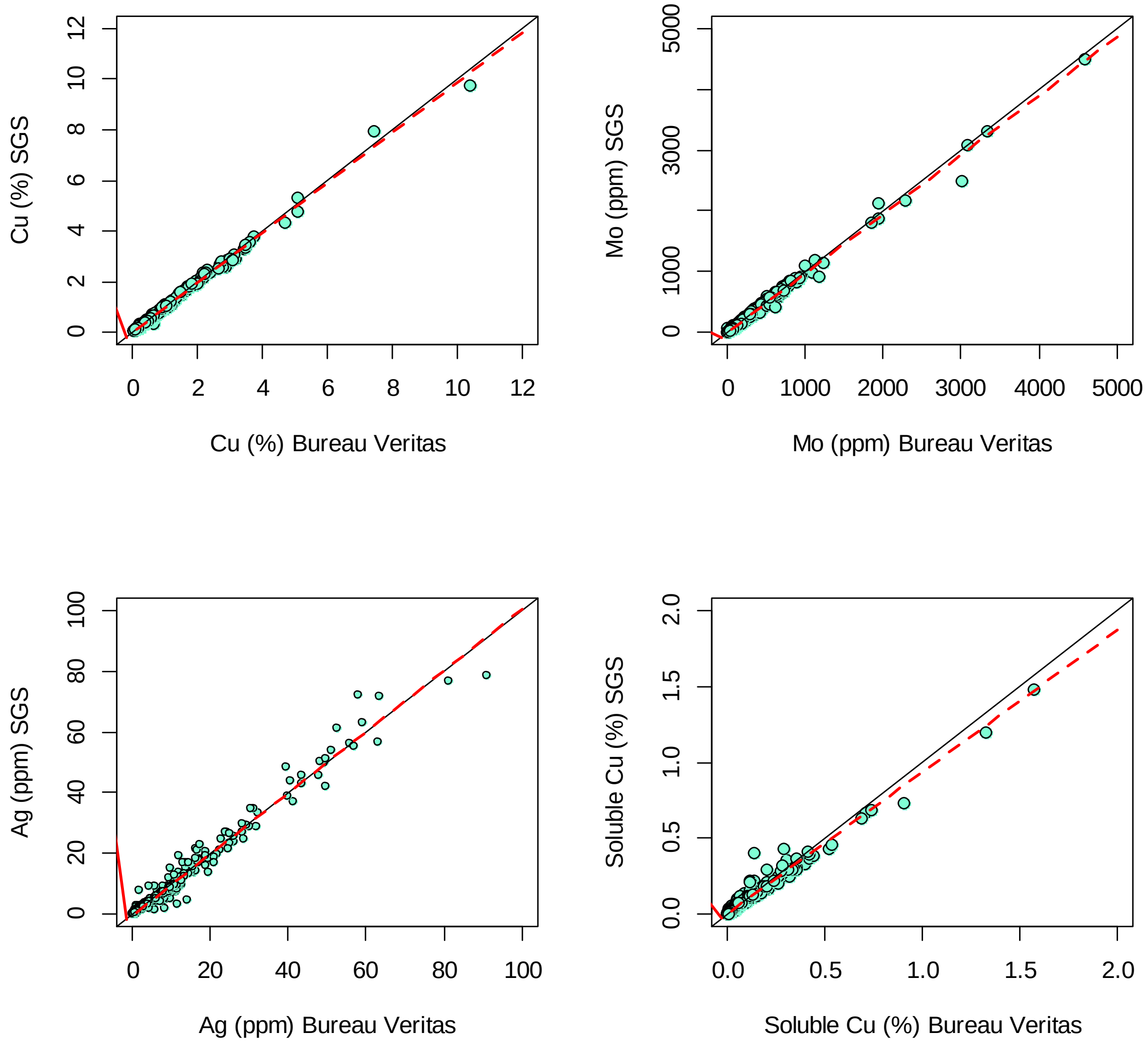
CRMs, certified blanks, and pulp duplicates were inserted along with the check samples following the same protocols used for monitoring Bureau Veritas. However, pulp duplicates, rather than coarse duplicates, were submitted to SGS. Duplicates and CRMs indicate that SGS achieved good levels of precision and accuracy. The overall bias deduced from the CRMs was -2.3% for copper, -4.5% for molybdenum, -2% for silver, and +2.1% for soluble copper. The analysis of blanks identified a few cases of economically insignificant copper contamination with average contamination of <30 ppm copper. It is concluded that the assay results from SGS are of good quality to evaluate the performance of Bureau Veritas.

A Reduced-to-Major-Axis regression (“RMA”) was used to evaluate the check samples (Kermack and Haldane, 1950). The RMA regression calculates an unbiased fit for values that are independent from each other. The coefficient of determination ($R^2$) is used to assess the variance explained by the linear relationship between the pairs. The bias, expressed as a percent, is calculated as Bias (%) = 1-RMAS in which RMAS is the slope of the RMA regression.

There is a good fit for copper ($R^2$ = 0.996), silver ($R^2$ = 0.976), molybdenum ($R^2$ = 0.993), soluble copper ($R^2$ = 0.970), and gold ($R^2$ = 0.987). The slope of the RMA regression for all metals ranges between 0.93 and 1.01 and all intercepts are below the practical limit of detection and approximate zero, as shown in Figure 11-5, where the black line represents the y = x line and the red dash line represents the RMA regression line.

The overall analytical bias of Bureau Veritas relative to SGS is +1.2% for copper, -1.0% for silver, +2.7% for molybdenum, +6.8% for soluble copper and -2.0% for gold. The overall bias estimated by the RMA regression analysis indicates that the accuracy achieved by Bureau Veritas for copper, molybdenum, silver, soluble copper is of good quality for resource estimation.

**FIGURE 11-5: XP PLOTS OF CHECK ASSAY DATA, COMPARING PRIMARY LABORATORY BUREAU VERITAS TO SECONDARY LABORATORY SGS**




## 11.2 Hudbay 2015

For consistency, the 2015 drilling campaign followed identical core logging, sample selection, core cutting, sample dispatching, and sample preparation protocols as those followed in 2014 as previously described in detail. The only notable exception in 2015 was the closure of Hidden Valley camp as all logging related activities took place at Rosemont camp which was expanded to accommodate up to 8 core logging geologists at a given time.

### 11.2.1 Bulk Density

A total of 755 samples in 46 drill holes were collected for specific gravity determinations at the Inspectorate preparation facility. The drill core samples, 4 to 6 inches (10 to 15 cm) long, were taken every 100 feet (30 meters) from half-split core and wax-coated following similar procedures to those used during the 2014 drilling program as previously described in detail.

#### 11.2.2 Assay Methodology

A total of 46 drill holes were sampled from top to bottom and assayed following the same digestion and analytical procedures used during 2014 and described in detail on Table 11-1, Section 11.1.

During 2015, 14,844 drill core samples were analyzed for 45 elements, including copper, molybdenum, and silver by ICP-MS after 4-acid digestion ("Method MA200"). All samples were also analyzed for ASCu ("Method GC921") with a 5% sulfuric acid ($H_2SO_4$) leach at room temperature. For gold assays, 5 drill holes were sampled from top to bottom with a total of 1,957 samples, 13% of Hudbay's 2015 program, assayed by lead-collection fire assay fusion (Table 11-1).

QA/QC protocols, duplicates, CRMs, and certified blanks, were similar to those used during the 2014 drilling program. However, OREAS 930 was replaced by OREAS 931 (Table 11-8). The QA/QC samples were systematically introduced to assess adequate sub-sampling procedures, potential cross-contamination, precision, and accuracy. The sampling program included 6% CRM, 6% certified blanks, and 6% coarse duplicates. Blanks and CRMs were prepared by OREAS in Australia. All QA/QC samples were analyzed following the same analytical procedures as those used for the drill core samples.

**TABLE 11-8: SUMMARY OF QA/QC SAMPLES**

| | No. of samples | Relative Frequency[1] | Type | Elements |
|---|---|---|---|---|
| OREAS 501b | 160 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 502b | 166 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 503b | 160 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 504b | 158 | 1.1% | CRM | Cu-Mo-Ag-Au |
| OREAS 902 | 91 | 0.6% | CRM | Cu-Mo-Ag-Cu Soluble |
| OREAS 931 | 159 | 1.1% | CRM | Cu-Mo-Ag |
| OREAS 22d | 436 | 2.9% | Certified Blank | Cu-Mo-Ag-Au |
| OREAS 26a | 438 | 2.9% | Certified Blank | Cu-Mo |
| Duplicates | 870 | 5.8% | Coarse Duplicate | Cu-Mo-Ag |
| Duplicates | 115 | 5.9%[2] | Coarse Duplicate | Cu-Mo-Ag-Au |

[1]Frequencies estimated relative to a sampling program comprising 14,868 samples
[2]Frequencies estimated relative to 1,957 samples analyzed for gold

#### 11.2.3 Blanks

Fine and coarse blanks were systematically inserted at the same rate for a total of 874 blanks representing 6% of the sampling program. OREAS 22d is a certified fine blank prepared from quartz sand. OREAS 26a is a certified coarse blank sourced from fresh and non-mineralized olivine basalt. The certified values for OREAS blanks are shown on Table 11-3.

OREAS 22d and OREAS 26a contain low trace level concentrations of copper, molybdenum, silver, and gold. Blank failure due to potential contamination issues is documented when the blank values exceed five times the lower limit of detection. For those blanks with concentration levels above the lower detection limits the failure thresholds are set to values that exceed the certified best value (CBV) plus three standard deviations. A summary of the blank performance is shown in Table 11-9.

**TABLE 11-9: SUMMARY OF BLANK PERFORMANCE**

| Fine Blank OREAS 22d | | | | | |
|---|---|---|---|---|---|
| **Element** | **No. of Blanks** | **Failed Blanks** | **Failure Rate** | **Maximum Contamination of CBV** | **Average Contamination of CBV** |
| Cu | 436 | 17 | 3.9% | 45 ppm | 7 ppm |
| Mo | 436 | 9 | 2.1% | 1 ppm | 1 ppm |
| Ag | 436 | 0 | 0 | 0 | 0 |
| Au | 57 | 0 | 0 | 0 | 0 |
| | | | | | |
| **Coarse Blank OREAS 26a** | | | | | |
| **Element** | **No. of Blanks** | **Failed Blanks** | **Failure Rate** | **Maximum Contamination of CBV** | **Average Contamination of CBV** |
| Cu | 438 | 13 | 3.0% | 40 ppm | 10 ppm |
| Mo | 438 | 1 | 0.2% | 12 ppm | 12 ppm |
| Ag | 438 | 0 | 0 | 0 | 0 |
| Au | 57 | 0 | 0 | 0 | 0 |

Contamination with copper, silver, and gold was insignificant (Table 11-9). There are very few cases of contamination at high-grade levels (12 ppm) of molybdenum. However, the overall blank failure rates are <3%.

The performance of blanks indicates no significant issues with contamination; therefore, it is concluded that the results are acceptable and adequate for the resource estimation.

### 11.2.4 Standards

OREAS certified reference materials (Table 11-10) were inserted one every twenty samples. In total, 894 CRMs were analyzed for a total insertion rate of 6.0%.

**TABLE 11-10: OREAS CERTIFIED REFERENCE MATERIAL**

| **Unit** | **Cu %** | **Mo Ppm** | **Ag ppm** | **Au ppm** | **Cu Soluble %** |
|---|---|---|---|---|---|
| OREAS 501b | 0.260 | 99 | 0.778 | 0.248 | - |
| OREAS 502b | 0.773 | 238 | 2.090 | 0.495 | - |
| OREAS 503b | 0.531 | 319 | 1.540 | 0.695 | - |
| OREAS 504b | 1.110 | 499 | 3.070 | 1.610 | - |
| OREAS 902 | 0.301 | 12.2 | 0.343 | - | 0.111 |
| OREAS 931 | 3.82 | - | 14.04 | - | - |
| Certified Method | 4 acids | 4 acids | 4 acids | Fire assay | Sulfuric acid at 5% |

The geological matrices for OREAS 501b to 504b and OREAS 902 are described in detail in Section 11. OREAS 931 is composed of a geological matrix, similar to OREAS 930, prepared from a copper ore body hosted in carbonaceous siltstones and mudstones mineralized with chalcopyrite, bornite, pyrrhotite, pyrite, sphalerite, galena, and cubanite.

Between 90 and 170 samples were analyzed per CRM (Table 11-8), which is sufficient information to set acceptance criteria relative to the average ("AV") and standard deviation ("SD") of the actual assay values of the CRMs. However, if the absolute relative bias is >10% the acceptance criteria is set relative to the CBV and standard deviation recommended by the CRM certificates.

The analytical bias for copper, molybdenum, and gold was good ranging between -2.5% and +1.1% (Table 11-11). The bias for soluble copper was reasonable at +6.3%.

The analytical bias of silver in OREAS 501b to 504b (0.8 to 3 ppm Ag) was good to reasonable with values between -1.1% and +7.7%. However, silver in OREAS 902 and 931 displayed larger analytical bias (+14.1% to +16%).

The large bias of silver in OREAS 902 and OREAS 931 is attributed to the low silver content of OREAS 902 and large variability of silver in these standards with relative standard deviations of 13% and 14%, respectively. The average silver value measured for these CRMs from the assays is within 1 and 2 SD of the CBV, respectively. Thus, the performance of silver in OREAS 902 and OREAS 931 is considered reasonable (Table 11-11).

Pulp duplicate re-analyses were requested to Bureau Veritas laboratory for all failed silver assays for OREAS 902 and OREAS 931 including six drill core pulp samples centred on the failed CRMs. In total, 21 failed CRMs and 126 drill core pulp samples with silver between 0.1 and 36.6 ppm were re-analyzed. After re-assaying, 90% of CRMs yielded silver values within the CBV and 92% of the re-assayed drill core pulps in the sample stream yielded similar results (0.1 - 27.8 ppm silver) evaluated for an absolute relative difference between pulp pairs equal to or smaller than 10%.

The CRM analysis indicates that the analytical accuracy for total copper, molybdenum, soluble copper, silver, and gold is of good quality for the resource estimation.

### TABLE 11-11: SUMMARY OF CRM PERFORMANCE

| Total Cu (ppm) | | | | | | |
|---|---|---|---|---|---|---|
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 160 | 2 | 1.3% | 2,600 | 2,589 | -0.4% |
| OREAS 502b | 166 | 3 | 1.8% | 7,730 | 7,539 | -2.5% |
| OREAS 503b | 160 | 1 | 1.3% | 5,310 | 5,233 | -1.4% |
| OREAS 504b | 158 | 1 | 0.6% | 11,100 | 10,971 | -1.2% |
| OREAS 902 | 91 | 2 | 2.2% | 3,010 | 3,003 | -0.2% |
| OREAS 931 | 159 | 1 | 0.6% | 38,200 | 38,488 | +0.8% |
| | | | | | | |
| **Mo (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 160 | 1 | 0.6% | 99 | 97 | -1.7% |
| OREAS 502b | 166 | 1 | 0.6% | 238 | 236 | -0.7% |
| OREAS 503b | 160 | 2 | 1.2% | 319 | 313 | -1.9% |
| OREAS 504b | 158 | 1 | 0.6% | 499 | 491 | -1.6% |
| OREAS 902 | 91 | 0 | 0% | 12.2 | 11.9 | -2.4% |
| | | | | | | |
| **Ag (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 160 | 2 | 1.3% | 0.778 | 0.77 | -1.1% |
| OREAS 502b | 166 | 3 | 1.8% | 2.09 | 2.16 | +3.3% |
| OREAS 503b | 160 | 2 | 1.3% | 1.54 | 1.59 | +3.5% |
| OREAS 504b | 158 | 0 | 0% | 3.07 | 3.31 | +7.7% |
| OREAS 902 | 91 | 14 | 15.4% | 0.343 | 0.40 | +16.0% |
| OREAS 931 | 159 | 13 | 8.2% | 14.04 | 16.01 | +14.1% |
| | | | | | | |
| **Au (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 501b | 23 | 0 | 0% | 0.248 | 0.251 | +1.1% |
| OREAS 502b | 22 | 0 | 0% | 0.495 | 0.493 | -0.4% |
| OREAS 503b | 20 | 0 | 0% | 0.695 | 0.693 | -0.2% |
| OREAS 504b | 21 | 0 | 0% | 1.610 | 1.600 | -0.7% |
| | | | | | | |
| **Soluble Copper (ppm)** | | | | | | |
| **Standard** | **No. of Samples** | **No. of Failures** | **Failure Rate** | **CRM Value (ppm)** | **Assay Average** | **Relative Bias** |
| OREAS 902 | 91 | 1 | 1.1% | 1,110 | 1,180 | +6.3% |

### 11.2.5 Duplicates

Coarse duplicates, approximately one in every twenty samples, were requested to Bureau Veritas laboratory in order to monitor sub-sampling precision (Section 11.1.11). Accordingly, after crushing to 10 mesh (2 mm), a 1,000 g coarse duplicate sub-sample was riffle split and pulverized to ≥85% passing through 200 mesh (75 μm). The duplicate sample was analyzed immediately after its paired sample.

A total of 870 coarse duplicate samples were inserted for a total rate of 6%. Quarter-core twin sample duplicates and pulp duplicates were not analyzed during Hudbay's 2015 drilling program. The coarse duplicates were evaluated using the hyperbolic method developed by AMEC (Simón, 2004) and explained in detail on Section 11.1.11.

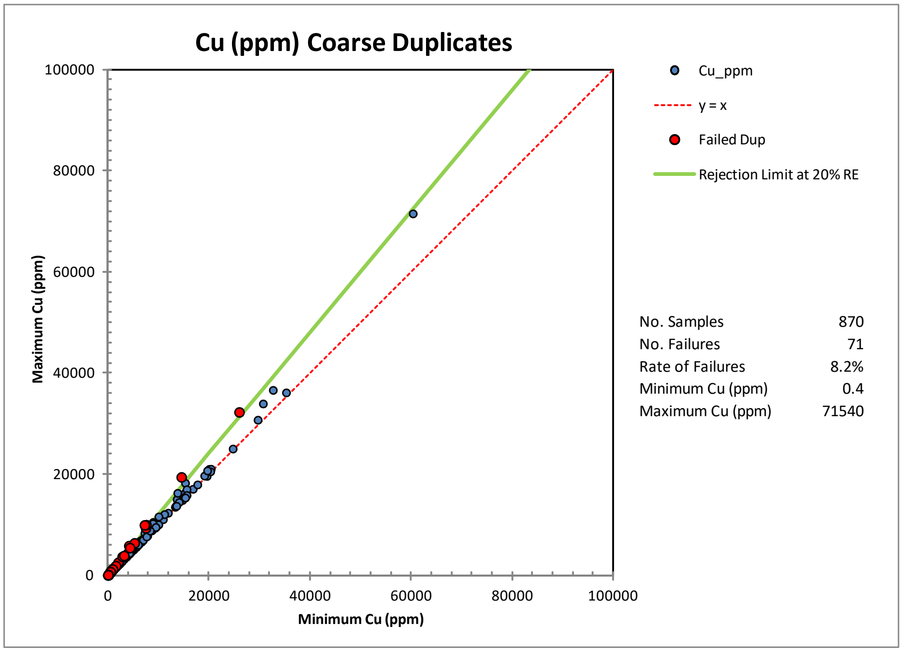
**TABLE 11-12: SUMMARY OF COARSE DUPLICATE ANALYSIS**

| Element | No. of Samples | No. of Failures | Failure Rate | Accepted Absolute RE |
|---|---|---|---|---|
| Cu | 870 | 71 | 8.2% | 20% |
| Mo | 870 | 72 | 8.3% | 20% |
| Ag | 870 | 25 | 2.9% | 20% |
| Au | 115 | 0 | 0% | 20% |
| Soluble Cu | 870 | 69 | 7.9% | 20% |

The results from the coarse duplicate analysis are presented on Table 11-12 and illustrated on Figure 11-6 to Figure 11-9. During 2015, an acceptable level of sub-sampling variance was achieved with, failure rates between 0% and 8.3%, for sample pairs evaluated for a maximum absolute relative error of 20% (Table 11-12). It is noteworthy that the sub-sampling variance achieved during 2015 was larger than the variance indicated by the coarse duplicates analysis during Hudbay's 2014 drilling program (Table 11-7). During 2014, an acceptable level of sub-sampling variance was achieved for an absolute relative error of 15%.

Despite the larger sub-sampling variance observed during the 2015 program, the duplicate failure rate for a maximum RE of 20% is acceptable for all metals used in the resource model.

## FIGURE 11-6: COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT




## FIGURE 11-7: MOLYBDENUM COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT

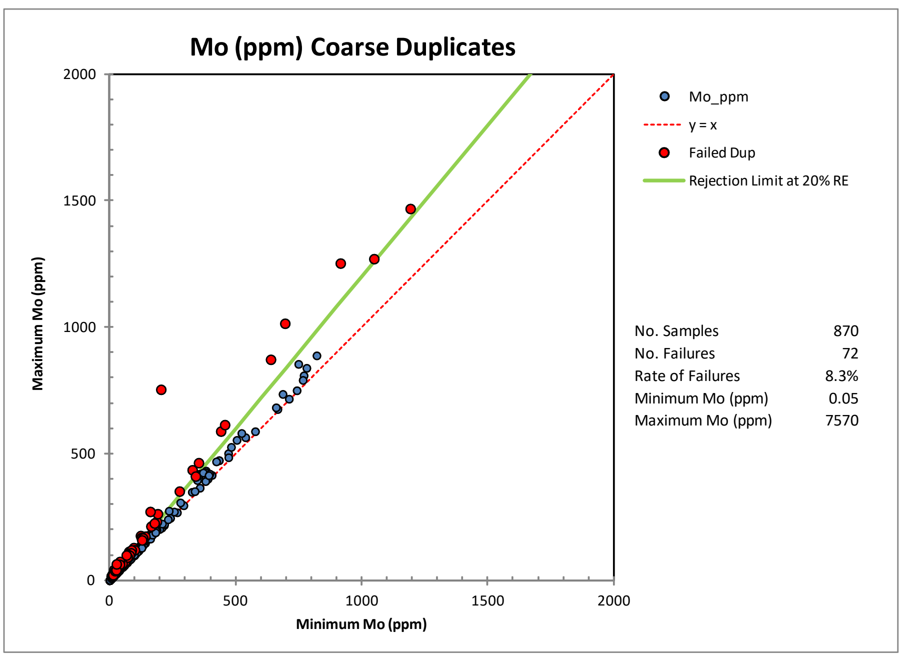


**FIGURE 11-8: SILVER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT**

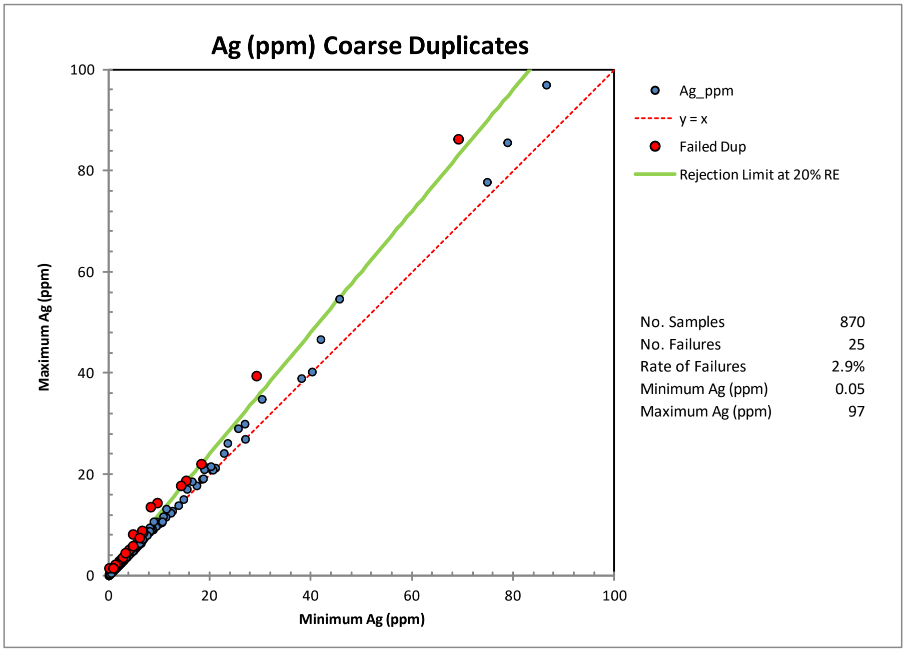
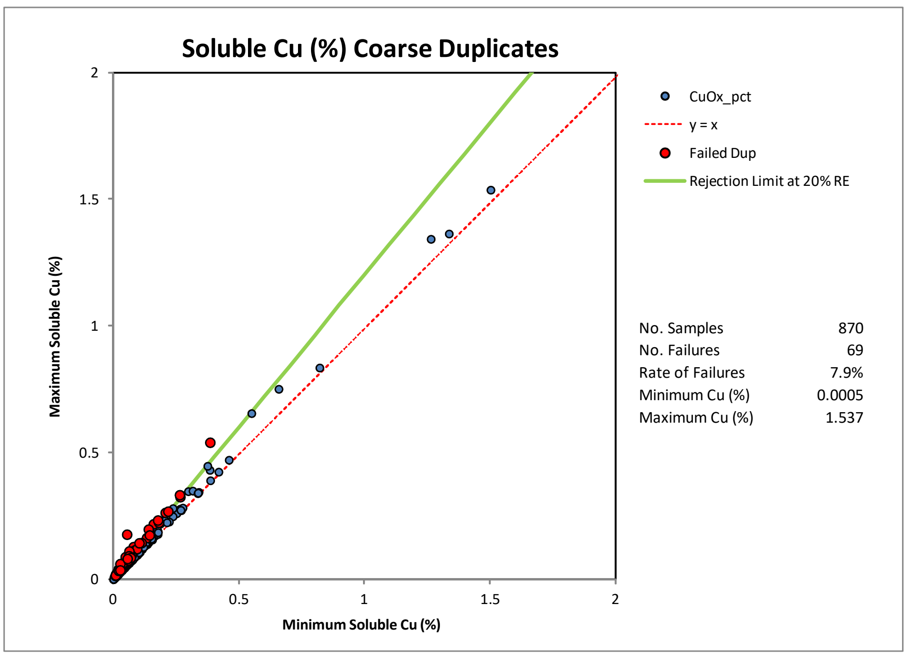



**FIGURE 11-9: SOLUBLE COPPER COARSE DUPLICATE MINIMUM AND MAXIMUM PLOT**



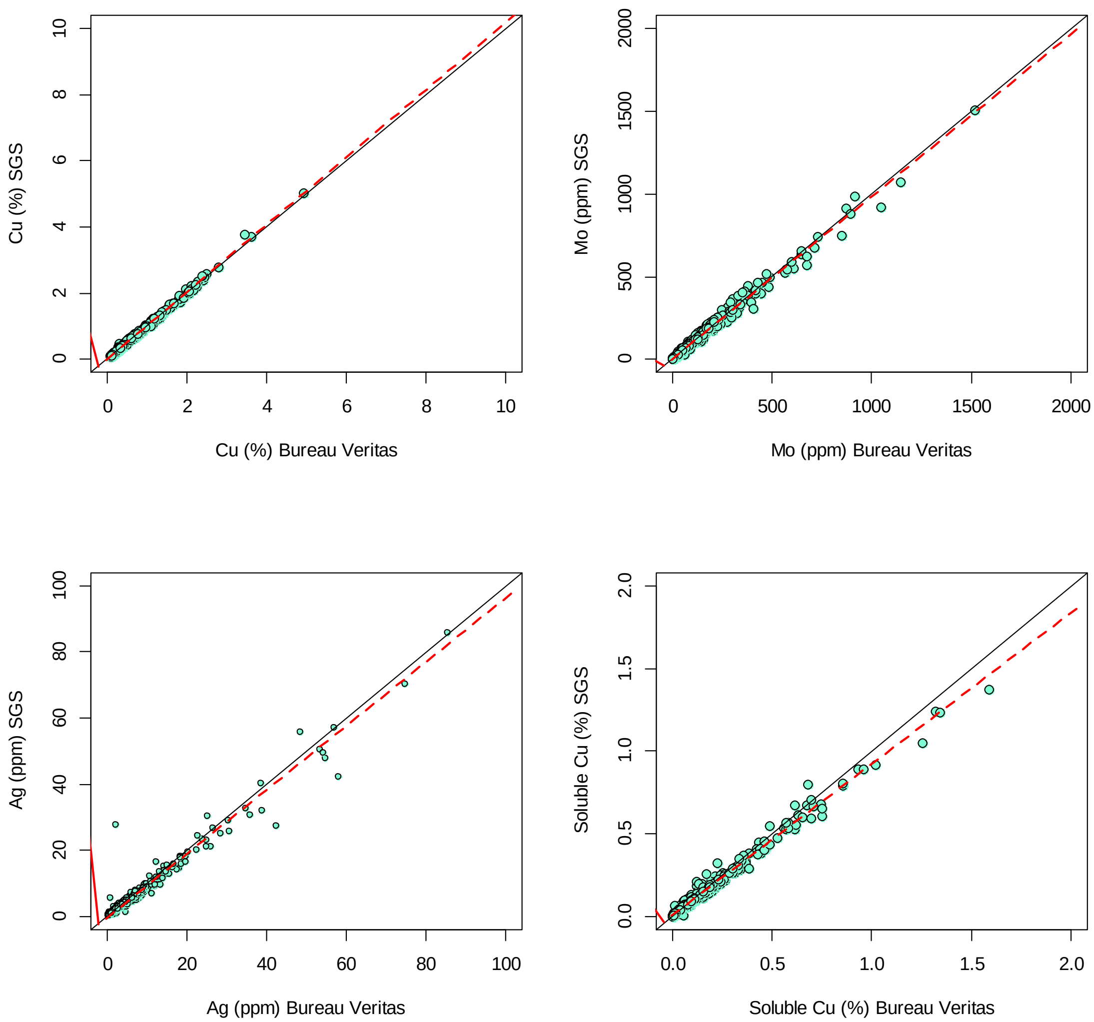

### 11.2.6 Check Assaying

A total of 742 representative pulp samples (5%) were selected and re-analyzed at SGS Canada Inc. ("SGS") laboratory in Vancouver. Only samples with ≥1000 ppm copper were submitted for re-analysis at the secondary laboratory.

CRMs, certified blanks, and pulp duplicates were inserted along with the check samples following the same protocols used for monitoring Bureau Veritas. However, pulp duplicates, rather than coarse duplicates, were submitted to SGS. Duplicates and CRMs indicate that SGS achieved good levels of precision and accuracy. The overall bias deduced from the CRMs was +2.7% for copper, -5.8% for molybdenum, +14% for silver, and +2% for soluble copper. The large bias of silver is explained by the large variance of silver in OREAS 931. The analysis of blanks indicates that there was no economically significant contamination. It is concluded that the assay results from SGS are of good quality to evaluate the performance of Bureau Veritas.

A Reduced-to-Major-Axis regression ("RMA") was used to evaluate the check samples (Kermack and Haldane, 1950). The RMA regression calculates an unbiased fit for values that are independent from each other (Section 11.1.12).

**FIGURE 11-10: XP PLOTS OF CHECK ASSAY DATA, COMPARING PRIMARY LABORATORY BUREAU VERTIAS TO SECONDARY LABORATORY SGS**



There is a good fit for copper ($R^2$ = 0.999), silver ($R^2$ = 0.979), molybdenum ($R^2$ = 0.990), soluble copper ($R^2$ = 0.989), and gold ($R^2$ = 0.977). The slope of the RMA regression for all metals ranges between 0.92 and 1.02 and all intercepts are below the practical limit of detection and approximate zero, as shown in Figure 11-5, where the black line represents the y = x line and the red dash line represents the RMA regression line.

The overall analytical bias of Bureau Veritas relative to SGS is -2.17% for copper, +3.32% for silver, +1.66% for molybdenum, +8.25% for soluble copper and +2.55% for gold. The overall bias estimated by the RMA regression analysis indicates that the accuracy achieved by Bureau Veritas for copper, molybdenum, silver, soluble copper, and gold is of good quality for resource estimation.

## 11.3 Augusta

A detailed description of sample preparation procedures and data verification processes conducted by Augusta are provided in several reports including two NI 43-101 technical reports prepared by M3 Engineering & Technology Corporation (2009, 2012). However, Hudbay conducted its own technical review and verification of the data; the results of which are summarized in this section.

### 11.3.1 Sample Preparation

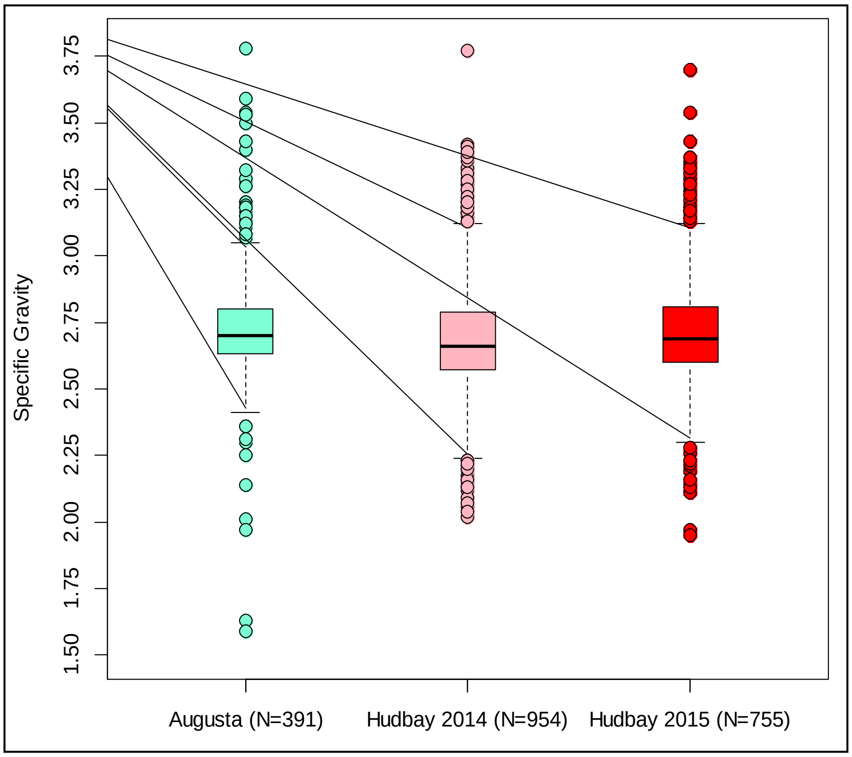
Overall, the documented protocols for handling diamond drill core, data security, drill core sampling, and sample custody by Augusta are acceptable and industry standard. All core drilled by Augusta was systematically logged for RQD, lithology, alteration, mineralization, and structures. Logging was paper-based and later recorded in spreadsheets. Geologists marked the logged core for cutting with diamond rock saws and sampling was conducted on half-split core. Samples were collected at fixed 5 feet (1.5 meters) intervals, assigned a unique sample number, and securely sealed in sample bags. The sample intervals were shortened to accommodate smaller zones with abrupt changes in copper and molybdenum mineralization, but typically the sampled intervals straddle geological boundaries.

### 11.3.2 Bulk Density

Augusta analyzed a total of 391 drill core samples across the Rosemont deposit for their specific gravity (“SG”) at Skyline Assayers & Laboratories (“Skyline”), Tucson, Arizona, USA. Skyline followed a protocol based on the differential weight of the sample in air and water. No paraffin coating was applied.

The SG results obtained by Augusta compare well with those measured by Hudbay at Inspectorate during its 2014 and 2015 drilling programs, as shown in Figure 11-11. The global average (2.74) and median (2.70) SG values of 391 samples measured at Skyline are comparable with the average (2.69) and median (2.66) of 954 samples analyzed by Hudbay during 2014, and 755 samples analyzed in 2015 (average= 2.72, median= 2.69). It is concluded that the SG reported by Augusta are of good quality and appropriate for resource estimation.

**FIGURE 11-11: BOXPLOTS OF SG MEASURED BY HUDBAY AND AUGUSTA AT INSPECTORATE AND SKYLINE LABORATORIES, RESPECTIVELY**




### 11.3.3 Assay Methodology

Augusta assayed drill core samples at Skyline's laboratory in Tucson, Arizona. Drill core samples were dried before being crushed using jaw crushers to produce a coarse fraction with ≥70% passing through 10 mesh (2 mm). The entire crushed sample was homogenized, riffle split, and a 300 to 400 g subsample split was pulverized to pass ≥95% through 150 mesh (105 µm) using Essa standard steel grinding bowls. Jaw crushers, preparation pans, and grinding bowls were cleaned with compressed air between samples. Coarse rejects and pulps were returned to Augusta.

Table 11-13 summarizes the assay methodologies and instrumental finishes conducted by Skyline.

**TABLE 11-13: ASSAY SPECIFICATIONS – SKYLINE**

| Element | Cu | Cu Soluble | Mo | Mo | Ag | Ag | Au |
|---|---|---|---|---|---|---|---|
| Lower Detection Limit | 100 ppm | 100 ppm | 50 ppm | 10 ppm | 0.4 pm | 0.1 ppm | 0.005ppm |
| Upper Detection Limit | 10% | 5% | 1% | 1% | 100 ppm | 100 ppm | 3 ppm |
| Digestion | 3 acids $HCl-HNO_3-HClO_4$ | Sulfuric acid at 10% $H_2SO_4-Na_2SO_3$ | 3 acids $HCl-HNO_3-HClO_4$ | 3 acids $HCl-HNO_3-HClO_4$ | Aqua Regia $HCl-HNO_3-$ | Aqua Regia $HCl-HNO_3-$ | Fire assay |
| Instrumental Finish | AAS | AAS | ICP-OES | ICP-OES | AAS | AAS | AAS |
| Method Code | MEA | Cu-AS | MEA | MEA | FA-O8 | FA-08 | FA-01 |
| Time Period | 2005-2012 | 2005-2012 | 2005-2006 | 2006-2012 | 2005 | 2005-2012 | 2005-2012 |

In total 21,197 samples were analyzed for total copper and 16,619 samples for molybdenum. Total copper and molybdenum were dissolved using a hot 3-acid digestion at 250°C and subsequently analyzed by AAS and ICP-OES, respectively. The lower detection limits for molybdenum are high relative to the average molybdenum grade of the Rosemont deposit (Table 11-13).

A total of 9,030 samples were analyzed for soluble copper using an acid leach at 10% sulfuric acid with sodium sulfite. The acid leach was conducted for an hour at room temperature and the solution was analyzed by AAS.

Silver, analyzed in 15,334 samples, was digested using an aqua regia leach in 0.25 g subsample pulp and analyzed by AAS. Two different lower limits of detection, 0.4 and 0.1 ppm, were used in 2005. The 0.4 ppm detection limit is high relative to the average silver grade of the deposit. However, the lower limit of detection was improved in the following years (Table 11-13).

A total of 4,932 samples were analyzed for gold by fire assay with an AAS finish.

Augusta conducted its own internal QA/QC program to independently evaluate the quality of the assays reported by Skyline. Standards and blanks were systematically inserted in the sample stream. Duplicates were not periodically inserted. The QA/QC program was initially provided by Geochemist, Kenneth A. Lovstrom (deceased). After 2006 the QA/QC program was managed by Geochemist, Shea Clark Smith of Minerals Exploration & Environmental Geochemistry.

Skyline is a certified laboratory accredited in accordance with the recognized International Standard ISO/IEC 17025:2005 General Requirements for the Competence of Testing and Calibration Laboratories. The sample preparation, analysis, and security procedures followed by Skyline are considered industry standard.

### 11.3.4 Blanks

In order to track potential contamination processes Augusta inserted non-certified blanks in the sample stream for an average insertion rate of 2%. Coarse barren marble and fine quartz sand were used as blanks in early drill programs through 2007, after which the marble blank was no longer used. The marble blank was used after high grade samples as a cleaner and to test for cross contamination, and this blank was excluded from statistical analyses because its contained metal content is unknown. The distinction between these two blanks was not documented by Augusta in the database and the results are evaluated as a combined single blank.

The assays results for copper, molybdenum, silver, and gold indicate that the blanks are barren relative to the metals of economic interest and appropriate to assess contamination. Blank failure due to potential contamination issues is triggered when the blank values exceed five times the lower limit of detection. A few cases of contamination at higher grade levels are documented for silver and molybdenum. However, all metals of economic interest have very low failure rates ranging from 0 to 6%, indicating that contamination is not a significant problem in the samples analyzed at Skyline as shown in Table 11-14.

**TABLE 11-14: SUMMARY OF BLANK PERFORMANCE AT SKYLINE**

| Element | Count | Failed Blanks | Failure Rate (%) | Maximum Contamination | Average Contamination |
|---|---|---|---|---|---|
| Cu | 553 | 5 | 1.0% | 690 ppm | 310 ppm |
| Ag | 552 | 33 | 6.0% | 7.6 ppm | 0.9 ppm |
| Mo | 440 | 7 | 1.6% | 120 ppm | 40 ppm |
| Au | 123 | 0 | 0.0% | 0 | 0 |

### 11.3.5 Standards

Augusta used 14 standard reference materials (“SRM”) inserted in the sample stream with an average insertion rate of 4.3%. The insertion rate is appropriate to assess the accuracy of the data.

Standards KM5, GRS3, GRS4, OC43, and OC48 were developed by Mr. Lovstrom. The R-series standards were prepared at MEG Labs in Carson City, Nevada. M3 Engineering & Technology Corporation (2012) has indicated that the MEG SRMs were prepared from mineralized rock collected from the Rosemont deposit with best values (“BV”) determined following a round robin program from a minimum of 25 samples analyzed by at least 5 different laboratories. The certificates of the SRMs used by Augusta are not available and details of the digestion protocols, sample matrix, and analytical finish are unknown.

Table 11-15 provides a summary of best values, standard deviations, and relative standard deviations (“RSD”) for the SRMs extracted from internal reports by Augusta. There are no records in the database indicating the use of SRM R4A. Therefore, the analysis presented here is based on the

13 remaining SRMs. Table 11-16 summarizes the analytical performance of the SRMs used by Augusta.

**TABLE 11-15: STANDARD REFERENCE MATERIALS – AUGUSTA**

| Total Cu (%) | | | |
|---|---|---|---|
| **SRM** | **Best Value** | **SD** | **RSD** |
| KM5 | 0.99 | 0.02 | 1.5% |
| GRS3 | 1.23 | 0.03 | 2.0% |
| GRS4 | 2.02 | 0.02 | 1.0% |
| R1 | 0.47 | 0.02 | 3.2% |
| R2 | 0.72 | 0.02 | 2.8% |
| R4A | 1.43 | 0.02 | 1.4% |
| R4B | 0.57 | 0.02 | 3.5% |
| R4C | 0.39 | 0.02 | 3.8% |
| R4D | 0.30 | 0.02 | 6.7% |
| R4E | 0.22 | 0.01 | 4.5% |
| R4F | 0.14 | 0.01 | 7.1% |
| R4G | 0.07 | 0.01 | 14.3% |
| | | | |
| **Mo (%)** | | | |
| **SRM** | **Best Value** | **SD** | **RSD** |
| OC43 | 0.035 | 0.001 | 2.9% |
| OC48 | 0.078 | 0.004 | 5.1% |
| R1 | 0.025 | 0.003 | 10.0% |
| R2 | 0.017 | 0.002 | 11.8% |
| R4A | 0.032 | 0.002 | 6.2% |
| R4B | 0.030 | 0.002 | 6.7% |
| R4C | 0.033 | 0.002 | 6.1% |
| R4D | 0.018 | 0.002 | 8.3% |
| R4E | 0.011 | 0.001 | 9.1% |
| R4F | 0.010 | 0.001 | 10.0% |
| R4G | 0.016 | 0.001 | 6.2% |
| | | | |
| **Ag (ppm)** | | | |
| **SRM** | **Best Value** | **SD** | **RSD** |
| R1 | 5.1 | 0.52 | 10.20% |
| R2 | 7.1 | 0.74 | 10.44% |
| R4A | 7.0 | 0.84 | 11.96% |
| R4B | 3.9 | 0.49 | 12.60% |
| R4C | 3.1 | 0.58 | 18.89% |
| R4D | 2.4 | 0.67 | 28.51% |
| R4E | 1.7 | 0.64 | 36.99% |
| R4F | 1.4 | 0.66 | 46.81% |
| R4G | 1.2 | 0.80 | 66.81% |

**TABLE 11-16: PERFORMANCE OF STANDARD REFERENCE MATERIALS AT SKYLINE**

| Total Cu (%) | | | | | | |
|---|---|---|---|---|---|---|
| **SRM** | **No. of Samples** | **No. of Failures** | **Failure Rate (%)** | **SRM Best Value** | **Skyline Average** | **Relative Bias** |
| KM5 | 59 | 0 | 0.0% | 0.99 | 1.01 | +2.0% |
| GRS3 | 18 | 0 | 0.0% | 1.23 | 1.12 | -8.9% |
| GRS4 | 20 | 1 | 5.0% | 2.02 | 1.90 | -5.9% |
| R1 | 417 | 15 | 3.6% | 0.47 | 0.47 | 0.0% |
| R2 | 233 | 2 | 0.9% | 0.72 | 0.71 | -1.4% |
| R4B | 33 | 1 | 3.0% | 0.57 | 0.57 | 0.0% |
| R4C | 151 | 1 | 0.7% | 0.39 | 0.40 | +2.6% |
| R4D | 74 | 2 | 2.7% | 0.30 | 0.30 | 0.0% |
| R4E | 90 | 1 | 1.1% | 0.22 | 0.21 | -4.5% |
| R4F | 93 | 3 | 3.2% | 0.14 | 0.14 | 0.0% |
| R4G | 23 | 1 | 4.3% | 0.07 | 0.07 | 0.0% |
| Total | 1,211 | | | | | |
| | | | | | | |
| **Mo (%)** | | | | | | |
| **SRM** | **No. of Samples** | **No. of Failures** | **Failure Rate (%)** | **SRM Best Value** | **Skyline Average** | **Relative Bias** |
| OC43 | 22 | 0 | 0.0% | 0.035 | 0.034 | -2.9% |
| OC48 | 21 | 1 | 4.8% | 0.078 | 0.073 | -6.4% |
| R1 | 233 | 6 | 2.6% | 0.025 | 0.025 | 0.0% |
| R2 | 225 | 1 | 0.4% | 0.017 | 0.018 | +5.9% |
| R4B | 33 | 0 | 0.0% | 0.030 | 0.029 | -3.3% |
| R4C | 141 | 1 | 0.7% | 0.033 | 0.031 | -6.1% |
| R4D | 51 | 1 | 2.0% | 0.018 | 0.018 | 0.0% |
| R4E | 81 | 2 | 2.5% | 0.011 | 0.009 | -18.2% |
| R4F | 74 | 2 | 2.7% | 0.010 | 0.009 | -10.0% |
| R4G | 23 | 1 | 4.3% | 0.016 | 0.014 | -12.5% |
| Total | 904 | | | | | |
| **Ag (ppm)** | | | | | | |
| **SRM** | **No. of Samples** | **No. of Failures** | **Failure Rate (%)** | **SRM Best Value** | **Skyline Average** | **Relative Bias** |
| R1 | 233 | 14 | 6.0% | 5.1 | 5.1 | 0.0% |
| R2 | 225 | 2 | 0.9% | 7.1 | 7.0 | -1.4% |
| R4B | 25 | 1 | 4.0% | 3.9 | 3.8 | -2.6% |
| R4C | 124 | 6 | 4.8% | 3.1 | 2.5 | -19.4% |
| R4D | 51 | 1 | 2.0% | 2.4 | 2.0 | -16.7% |
| R4E | 41 | 0 | 0.0% | 1.7 | 1.3 | -23.5% |
| R4F | 122 | 1 | 0.8% | 1.4 | 1.0 | -28.6% |
| R4G | 21 | 0 | 0.0% | 1.2 | 0.5 | -58.3% |
| Total | 842 | | | | | |

The analytical accuracy for copper was good to reasonable for all SRMs analyzed at Skyline with relative bias ranging from -9% to +3% (Table 11-16). Two SRMs (R4E and R4G) displayed poor accuracy for molybdenum with negative bias of less than -10%. However, 90% of the SRMs measured for molybdenum displayed relative bias between -10% and +6% indicating good to reasonable accuracies for molybdenum SRMs at grade levels of ≥ 100 ppm.

The results of the standards analyzed for silver indicate good to reasonable accuracies at grade levels ≥3 ppm. However, all SRMs with recommended best values <3 ppm silver displayed unacceptable negative bias between -58% and -19% (R4C to R4G; Table 11-16). The poor performance of these silver standards is attributed to several factors including imprecise characterization of the standards and silver grades close to the lower detection limit. For instance, the reported RSDs for standards R4C to R4G range from 19 to 67% indicating poor precision in the determination of the recommended best values (Table 11-15). The large RSDs diminish the value of these standards as reference materials.

It is concluded that the analytical accuracy achieved by Skyline for all metals of economic interest is good to reasonable and therefore adequate for resource estimation.

#### 11.3.6 Duplicates

Augusta did not insert duplicates periodically within the sample stream. On average, only 0.2% coarse duplicate (<50 samples) samples were analyzed. The insertion of duplicates is significantly below recommended rates of 2% to 6%. There are insufficient duplicate samples to correctly evaluate batch reproducibility.

#### 11.3.7 Check Assays

Augusta resubmitted sample pulps to Skyline for check assays. The significance of the check samples to independently estimate the confidence of Skyline is diminished given that a secondary laboratory was not used. Augusta submitted an average of 1.5% of samples for re-analysis of copper, molybdenum, silver, and soluble copper. The check assay rate is lower than a recommended rate of 4% to 5%.

A RMA was used to evaluate the check samples. The coefficient of determination ($R^2$) is used to assess the variance explained by the linear relationship between the pairs and is calculated as Bias (%) = 1-RMAS in which RMAS is the slope of the RMA regression.

Augusta re-assayed 373 samples for copper, 326 samples for silver, 326 samples for molybdenum, and 203 samples for soluble copper. The RMA regression for the sample pairs was calculated after removing extreme outliers that clearly represent switched samples. In total, there were three outliers for copper, four outliers for silver, two outliers for molybdenum, and one outlier for soluble copper.

There is a good fit for copper ($R^2$ = 0.999), silver ($R^2$ = 0.992), molybdenum ($R^2$ = 0.995), and soluble copper ($R^2$ = 0.968). The slope of the RMA regression for all metals ranges between 0.98 and 1.04 and all the intercepts approximate zero and are below the lower limit of detection for each element. The overall bias of the primary analysis relative to the check assays is +1.0% for copper, +2.0% for silver, +1.6% for molybdenum, and -3.7% for soluble copper. However, on a per sample basis, >10% relative error ("RE") is observed for silver grades ≤ 3 ppm and molybdenum grades ≤ 50 ppm. This is expected given the use of relatively high lower limits of detection for these metals (Table 11-13).

The overall bias estimated by the RMA regression analysis indicates that the global accuracy for copper, molybdenum, silver, and soluble copper is good for resource estimation.

## 11.4 Historic

Table 11-17 provides historical sample preparation procedures and data verification processes conducted by several property owners prior to 2005.

**TABLE 11-17: ROSEMONT DEPOSIT DRILLING SUMMARY**

| Company | Time Period | No. of Drill Holes | Metres Drilled |
|---|---|---|---|
| Banner | 1950s to 1963 | 3 | 1,311 |
| Anaconda | 1963 – 1973 | 113 | 41,708 |
| Anamax | 1973 – 1986 | 52 | 16,566 |
| ASARCO | 1988 – 2004 | 11 | 4,479 |
| **Total** | | **179** | **64,008** |

### 11.4.1 Sample Preparation

For over 50 years, significant diamond drilling, drill core sampling, and assaying programs were executed by several property owners preceding Hudbay and Augusta. Records are not available with details of sampling and security protocols used by these property owners.

Banner, Anaconda, and Anamax used similar methodologies for drill core logging and sampling. In general, lithology, alteration, structures, and mineralization were documented on paper logs. Drill core was half-split using mechanical splitters and sampled for assaying. Mineralized intervals were entirely sampled with sample length ranging from 1 to 5 feet (0.3 to 1.5 meters). Poorly mineralized intervals were sampled every 20 to 30 feet (6 to 10 meters) along 5 feet (1.5 meters) intervals.

Asarco logged drill core following the same methodology used by Banner, Anaconda, and Anamax. All geological information was captured on paper logs. The length of drill core samples was variable and subjected to the criteria of core logging geologists. Typical sampling intervals were approximately 10 feet (3 meters) in length.

Augusta data verification program included re-logging of the majority of available drill core mostly from Anaconda, Anamax, and Asarco following the procedures described in Section 11.3.1. The information was collected on paper logs and typed in spreadsheets. Augusta confirmed that historical drill core recoveries were better than 85% (M3 Engineering & Technology Corporation, 2009). Augusta also sampled intervals of historical drill core to fill-in missing analytical information and conducted a re-sampling program of 10 historical drill holes for data verification purposes.

### 11.4.2 Quality Control Evaluation

There are no available QA/QC records for sample preparation and assaying methodologies for Banner, Anaconda, and Anamax. These previous property owners regularly analyzed drill core samples for total copper and molybdenum. Silver was regularly analyzed by Anamax, but not commonly assayed by Banner and Anaconda. Visible oxidized zones were analyzed by soluble copper, but molybdenum was commonly not analyzed for oxide zone samples.

Copper, molybdenum, silver, and soluble copper were analyzed at Anaconda and Anamax in-house laboratories. The only existing record of digestion methodologies and analytical instruments used by these laboratories was provided by former Anaconda Chief Chemist, Mr. Dale Wood, in phone interviews on November 28, 2005, and January 21, 2006 (M3 Engineering & Technology Corporation 2009, 2012). Accordingly, copper and molybdenum were preliminary screened using x-ray fluorescence ("XRF"). Samples with > 0.2% Cu and > 200 pm Mo were then selected for wet chemical analysis. There is no documentation on the methods used to analyze silver and soluble copper.

Sample pulps for XRF analysis were placed on Mylar® film or prepared by adding cellulite and pressing it into a ring. There is no documentation on the type of XRF instrument, XRF technique, and internal laboratory QA/QC protocols.

Wet chemical analysis was a hot 3-acid digestion with hydrochloric, nitric, and perchloric acid, with a few drops of hydrofluoric acid. Copper and molybdenum were analyzed by colorimetry following phenolthylanaline titration for copper and iodine titration for molybdenum. There are no records on the model of colorimeters and internal laboratory QA/QC protocols.

#### 11.4.3 ASARCO

Asarco assayed drill core samples for total copper, molybdenum, and ASCu at Skyline; which is a certified laboratory accredited in accordance with the recognized International Standard ISO/IEC 17025:2005 General Requirements for the Competence of Testing and Calibration Laboratories. However, there are no records of the QA/QC practices followed by Asarco to independently monitor the quality performance of Skyline. Descriptions of digestion methodologies and analytical instruments are not available for the assay results conducted by Asarco.

#### 11.4.4 Augusta Re-Sampling Program

Augusta collected twin samples in 10 historical drill holes to verify the assay results reported by historical drilling and sampling programs. The remaining half-split core was sampled entirely along intervals similar to the original intervals sampled by previous property owners. Nine of these drill holes were drilled by Anaconda and Anamax and one (AH-4) by Asarco. Anaconda and Anamax used their in-house laboratories for assaying, whereas Asarco used Skyline. Copper and molybdenum were re-analyzed in all holes and only three drill holes were reanalyzed for silver. The twin sample re-assays were conducted at Skyline.

Duplicate analysis shows poor reproducibility of each individual Anaconda and Anamax assay relative to their twin sample analyzed by Augusta at Skyline. The poor reproducibility is attributed to factors including sample heterogeneity and poor analytical precision and accuracy of the Wet and XRF methods used by the Anaconda and Anamax laboratories.

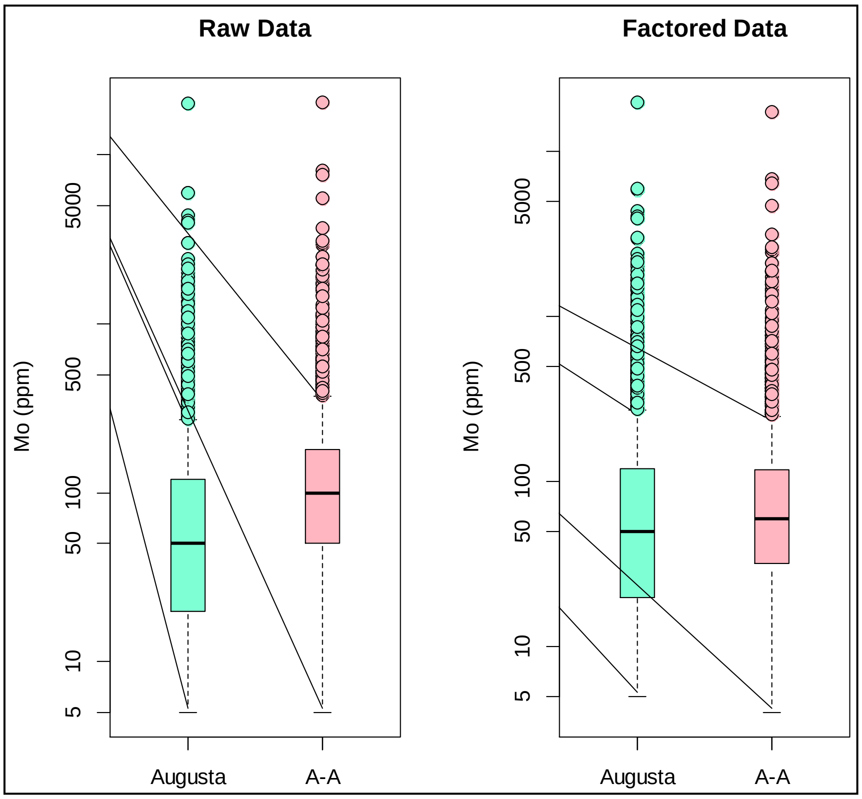
However, on a drill hole by drill hole comparison, the average historical copper assays display differences of less than 10% relative to the assays at Skyline, as shown in Table 11-18. Overall, the slightly lower copper content is deemed to result from the deterioration of these historical core samples over time (i.e. oxidation).

**TABLE 11-18: COMPARISON OF HISTORICAL ASSAY RESULTS AND TWIN HALF-SPLIT CORE SAMPLES ANALYZED BY AUGUSTA AT SKYLINE**

| Copper | | | | | |
|---|---|---|---|---|---|
| **Drill Hole** | **Cu (%) Historic** | **Cu (%) Re-Analyzed** | **No. of Samples** | **Bias (%)** | **Historic Laboratory** |
| A-804 | 0.45 | 0.43 | 315 | +4.7% | Anaconda-Anamax |
| A-813 | 0.51 | 0.51 | 239 | 0.0% | Anaconda-Anamax |
| A-821 | 0.57 | 0.53 | 302 | +7.5% | Anaconda-Anamax |
| A-834 | 0.48 | 0.45 | 282 | +6.7% | Anaconda-Anamax |
| A-858 | 0.35 | 0.34 | 199 | +2.9% | Anaconda-Anamax |
| 1485 | 0.43 | 0.39 | 239 | +10.3% | Anaconda-Anamax |
| 1508 | 0.94 | 0.90 | 207 | +4.4% | Anaconda-Anamax |
| 1916 | 0.39 | 0.39 | 256 | 0.0% | Anaconda-Anamax |
| 1917 | 0.24 | 0.25 | 63 | -4.0% | Anaconda-Anamax |
| AH-4 | 0.37 | 0.38 | 255 | -2.6% | Skyline |
| **Molybdenum** | | | | | |
| **Drill Hole** | **Mo (ppm) Historic** | **Mo (ppm) Re-Analyzed** | **No. of Samples** | **Bias (%)** | **Historic Laboratory** |
| A-804 | 70 | 50 | 244 | +40.0% | Anaconda-Anamax |
| A-813 | 330 | 280 | 178 | +17.9% | Anaconda-Anamax |
| A-821 | 380 | 290 | 230 | +31.0% | Anaconda-Anamax |
| A-834 | 180 | 180 | 244 | 0.0% | Anaconda-Anamax |
| A-858 | 180 | 160 | 199 | +12.5% | Anaconda-Anamax |
| 1485 | 170 | 60 | 239 | +183.3% | Anaconda-Anamax |
| 1508 | 240 | 230 | 207 | +4.3% | Anaconda-Anamax |
| 1916 | 260 | 160 | 257 | +62.5% | Anaconda-Anamax |
| 1917 | 230 | 130 | 65 | +76.9% | Anaconda-Anamax |
| AH-4 | 90 | 90 | 226 | 0.0% | Skyline |
| **Silver** | | | | | |
| **Drill Hole** | **Ag (ppm) Historic** | **Ag (ppm) Re-Analyzed** | **No. of Samples** | **Bias (%)** | **Historic Laboratory** |
| A-804 | 10.8 | 8.0 | 208 | +35.0% | Anaconda-Anamax |
| A-813 | 10.6 | 5.9 | | +79.7% | Anaconda-Anamax |
| A-821 | 10.2 | 4.6 | | +121.7% | Anaconda-Anamax |

Molybdenum reported by the Anaconda and Anamax laboratories (Wet and XRF) show significant positive bias of up to 183% on a drillhole by drillhole basis, relative to the twin samples analyzed at Skyline in Table 11-18. The average historical molybdenum over 2,136 samples is 195 ppm versus an average of 145 ppm in the twin samples, as shown in Figure 11-12. Relative to the twin samples, molybdenum reported as wet assays is 20% higher whereas XRF values are 130% higher. To circumvent problems related to the strong positive bias of the Anamax and Anaconda data, molybdenum grades reported by wet assays were multiplied by 0.85 and those reported by XRF by 0.45. After factoring, the average molybdenum in 2,136 assays by Wet and XRF is 147 ppm and compares well with an average of 145 ppm molybdenum in the twin samples, as shown in Figure 11-12. The factored molybdenum was used for resource estimation to minimize the impact of the large positive bias in the historical data.

**FIGURE 11-12: BOXPLOTS OF RAW MOLYBDENUM DATA AND FACTORED DATA REPORTED BY WET AND XRF (A-A = ANACONDA-ANAMAX AND Y-AXIS IN LOGARITHMIC SCALE)**




Silver reported by Anamax and Anaconda in-house laboratory also display a high bias of over 35% relative to the twin samples. However, due to the very small population of re-analyzed silver from only three holes and the Anaconda and Anamax representing approximately 12% of the entire dataset, the author decided not to impact the silver values until further investigation is complete.

Overall, the copper and molybdenum assays by Asarco (drill hole AH-4) compare well with the twin samples with very low bias. No silver was reported by Asarco.

In 2011, Augusta compared the historical drilling data to its more recent drilling results. Even though there were no twin holes drilled on the Property, six metallurgical holes located 13 to 29 feet from the historical holes were used for comparison purposes.

For most comparisons, copper grades show only minor variability with an average copper grade of 0.65% Cu for Augusta 0.65% Cu and 0.63% Cu for the historical data. Augusta concluded that the grade difference was linked to the natural variability of the skarn mineralization and the spacing between the Augusta and historical drilling.

More information can be found in the Rosemont feasibility study published by Augusta in 2012 and available on SEDAR website (http://www.sedar.com).

In the opinion of the author, the results from the re-assay program of Augusta and the comparison of metallurgical holes with closely located historical holes validated the use of historical copper and silver assays for resource estimation while Hudbay continues to perform confirmatory drilling. Thus far, this confirmation of drilling has confirmed resource quality and permitted the expansion of resource tonnage down the plunge of the deposit. The Molybdenum grade shows an over-estimation bias of approximately 15% in the historical drilling and this data was corrected prior to being used for resource estimation.

# 12 DATA VERIFICATION

## 12.1 Drill Hole Database

Hudbay built an entirely new drill hole database from all pre-Hudbay drilling and assaying information. Orix Geoscience Inc. was employed to digitally enter collar, downhole surveys and assay information from scanned drill logs and assay certificates for all holes drilled prior to Augusta.

The following subsections describe the process Hudbay used to build a completely new database of the drilling, assay values and the steps taken to verify the information. All pre-Augusta (prior to 2005) drill holes will hereby be referred to as “historical drill holes”.

The author’s opinion is that the data verification is adequate for the purposes used in the Technical Report.

### 12.1.1 Drill Hole Collars

Drill hole collar coordinates of historical drilling were reported in a local Anaconda grid system. The coordinates were converted to NAD83 UTM Zone 12N using MapInfo software by Augusta in 2005. The conversion was based on a best fit transformation using drill hole collars and corners from patented claim boundaries (approximately 6,000 points in total). This conversion is verified by plotting the converted coordinates against a drill hole collar compilation map prepared by Anamax Mining Co., in 1979 and the results are within acceptable margin of error (+/- 5 feet). Further verification was provided by Richard Darling, who located and surveyed 12 of these historical holes in UTM coordinates in 2006 and 2008.

Augusta drill hole collars were surveyed by J. Edmonson in 2006 and Darling Geomatics from 2006 to 2012 using a Trimble GLONASS (“Trimble”) sub-centimeter survey grade GPS. Darling Geomatics were also employed for surveying Hudbay drill hole collars using the same Trimble unit. All coordinates are reported in UTM Zone 12, NAD 83 horizontal datum in International Feet and NAVD 88 vertical datum in International Feet.

### 12.1.2 Downhole Surveys

Downhole survey files exist for 25 of the 181 historical drill holes, as shown in Table 12-1. The majority of the downhole surveys were conducted by Mollen-Hauer Surveying Company using a gyroscope that measured the drift angle and azimuth. The readings were generally recorded every 100 feet (30 meters). From the record sheets it cannot be determined if the azimuth recorded was adjusted for magnetic declaration, hence no further adjustments were made to these readings. However, of the 181 historical drill holes, 136 were drilled vertically.

**TABLE 12-1: DOWNHOLE SURVEYS OF HISTORICAL DRILLING**

| Company | Surveyed by | Instrument | Number of Holes | Year |
|---|---|---|---|---|
| Anaconda Mining | Eastco | Single Shot | 4 | 1966 |
| Anamax | Anamax | Gyroscope | 1 | 1974 |
| Anamax | Parsons Surveying Company | Gyroscope | 3 | 1974 |
| Anamax | Mollen-Hauer Surveying Company | Gyroscope | 17 | 1974-1983 |

Downhole surveys were conducted on all Augusta drill holes using a Reflex EZ-Shot which measures the dip and azimuth. The surveys were measured at 500 feet (152 meters) intervals by the drilling contractors of Layne-Christensen and Boart Longyear.

In 2014, Hudbay completed downhole surveys on either 200 feet (61 meters) intervals using a Reflex EZ Shot tool or on 50 feet (15 meters) intervals using a gyroscopic tool for their drill hole program, as shown in Table 12-2. Except for the single shot surveys measured using a Reflex EZ-Shot, all down hole data was digitally imported from the instrument into the database. Data was further assessed for reliability based on corresponding magnetic readings, subsequently discarding any readings above the threshold magnetic field.

In 2015, Reflex EZ shot tool was used to survey angled holes every 200 feet (61 meters) while drilling, and the gyroscopic tool was used to survey the holes every 50 feet (15 meters) at the conclusion of drilling (see Table 12-3). Single shot measurements were used to monitor the dip of the drill holes during its progress, while the gyroscopic readings are treated as the official downhole surveys with the exception of two holes for which no gyro survey was conducted. All data was digitally imported into the database from the instrument output files.

**TABLE 12-2: HUDBAY 2014 DOWNHOLE RESULTS**

| Surveyed by | Instrument | Number of Holes |
|---|---|---|
| Major Drilling | Reflex EZ-Shot | 5 |
| Layne-Christensen / Major Drilling | Reflex EZ-Trac | 14 |
| IDS Directional Surveys | Televiewer | 3 |
| Southwest Geophysics | GyroTracer Directional | 21 |

**TABLE 12-3: HUDBAY 2015 DOWNHOLE SURVEYS**

| Surveyed by | Instrument | Number of Holes |
|---|---|---|
| Layne-Christensen /National Drilling | Reflex EZ-Shot | 26 |
| Southwest Geophysics | GyroTracer Directional | 44 |

### 12.1.3 Historical and Augusta Assay Information

Hudbay acquired a compiled drill hole database from Augusta in 2014. In order to verify the data, Hudbay undertook the task of re-creating the historical assay database from the original paper certificates. The services of Orix Geosciences were employed to retrieve drill hole name, sample number, start and end depth of sample, assay values, and analytical methods from scanned copies of the historical paper certificates. Assay values were entered as reported on the paper logs including the lower than the detection limit values in the original reported units.

The newly compiled database was rechecked against the paper copies by Hudbay personnel to identify and fix any data entry errors and typos. Reoccurrences of sample identification for 590 samples from various sampling campaigns are present in the database, hence it was ruled out as the primary field for each sample. A unique key combining the drill hole name, along with start and end depth of the sample was created to adequately identify and index all of the samples in the database. Each assay field passed through several validation queries that flagged records outside of expected range, missing values and potential mismatch of characters.

In an effort to improve the density of analyses where core was only partially analyzed, Augusta performed a re-sampling program in conjunction with re-logging historic drill holes. Augusta also completely re-analyzed 10 historic drill holes as a validation of the quality of the historic analyses. In this process they collected over 1,800 samples (9,334 feet) of core. The re-analysis was completed at Skyline laboratories in Tucson.

Re-assay data was appended into the new Hudbay database as a separate column if the hole number matched perfectly with down-the-hole depth intervals. There were 10,056 samples that match this criterion with total footage of about 43,200 feet (13,200 meters). For approximately 1,000 samples, where the down-the-hole depth intervals did not match the original intervals Hudbay employed a weighted average method to import the re-assay information.

Augusta drilled 81 holes from 2005 to 2012 that were sampled and assayed. Laboratory assay certificates for all these holes were provided digitally by Skyline. Hudbay imported these digital certificates directly into their database.

For each original sample interval, every element assayed was ranked using the following criteria shown in Table 12-4. A separate field for each element was populated using the defined ranking with historical information given preference over re-assay, and for copper and molybdenum ranking Wet over XRF analysis. In instances where the original data is missing or reported as “Nil”, the next best ranked value was chosen. An associated data source field for every ranked assay documents the origin of the assay value in the database. Less than detection limits are reported as half the limit and assay unit measurements for the re-assay program were converted to historical units. All fields without any data to report are represented by a “-1” to distinguish them from a missing value.

**TABLE 12-4: DRILL HOLE ASSAY RANKING**

| Metal | Historical Information | | Re-assay at Skyline |
|---|---|---|---|
| | Wet | XRAY | |
| Copper | 1 | 2 | 3 |
| Molybdenum | 1 | 2 | 3 |
| Copper Soluble | 1 | | 2 |
| Silver | 1 | | 2 |
| Lead | 1 | | 2 |
| Zinc | 1 | | 2 |
| Gold | N/A | | 1 |

Augusta measured 391 SG samples from both the historical and their drill programs. This point data was merged into the assay database by matching corresponding sample interval and down-the-hole depth.

#### 12.1.4 Hudbay Assay Information

For the 2014 drilling campaign, Hudbay enlisted the services of an independent consultant to build a custom core logging database using the FileMaker database platform. This database was further enhanced and tested by Hudbay personnel. As the logging and sampling was completed at two different sites (Rosemont Camp and Hidden Valley Camp), separate clones of the main database were created and dispatched onto two laptop computers dedicated to data capture. Core logging and sampling was recorded on to a single database at each camp with up to seven geologists logging core simultaneously using tablets.

The second drilling campaign in 2015 eliminated the usage of Hidden Valley Camp and all core logging activities were centralized at Rosemont Camp. All logging was completed using tablets synced with a centralized FileMaker database hosted on a laptop.

Quality assurance protocols built into the core logging database prevented the loggers from duplicating sample numbers and entering out of range values for sample intervals. Sample types were restricted to a down-drop list in order to prevent several variations of a sample type. For quick data input and to prevent data entry errors, the sampling module was designed to predict the sample number, the interval length and the down-the-hole depths based on previous entered values, while allowing the users to edit and adjust the predicted values at their discretion. The sampling module was improved further in 2015 by pre-assigning sample numbers and sample types for each core logging geologist. This prevented incidents of incorrectly assigning sample types and typos in the sample numbers.

The sampling module portion of the FileMaker database generated color coded reports for each logged drill hole that listed hole name, depth interval, sample number and type of sample. These reports were visually examined by the lead geotechnician prior to core cutting and any sample overlaps or skipped samples were brought to the attention of the on-site database manager. Any discrepancies were fixed immediately in the database and the reports were reprinted.

The sample dispatch module of the FileMaker database was used to create assay requisition forms to submit samples to the assay laboratory, as well as export the samples included in each requisition to a PDF and CSV file. To maintain consistency, sampling dispatching was handled by the on-site database manager.

In order to minimize data loss, each of the databases was regularly saved. The FileMaker database created a backup of the data three hours on the host laptop. The database manager also saved a copy on a flash drive at the end of the day which was uploaded to a Google Drive. The updated copy of the database was merged into a centralized database at Hudbay's Toronto office and backed up on a different server on a daily basis.

The database manager in Toronto reviewed all the samples and logging from the previous day and communicated any edits or discrepancies that required amendments by the core loggers. Maintenance and updates to the field databases were carried out on bi-weekly basis.

Digital copies of the assay certificates received from the assay laboratory were imported into the main FileMaker database in Toronto by the database manager using custom-built importers. All fields were imported as they appeared on the certificates without substituting values for codes or special characters. All element fields in the database are named appropriately to include the element name, analytical method and units of measurement. Attribute fields which include hole name, down hole depths, sample type and requisition identification were populated using lookup functions for each sample number. A sample tracking module was created to track all jobs submitted to the assay laboratory along with analytical methods, number of samples and the date samples were sent and received.

SG samples were entered into the logging database by the geologist. The SG measured results were later imported into the assay table by matching the depth of the sample and assay interval.

An Open Database Connectivity was established between FileMaker database and Excel to import sample interval table and the sample assay table. A copy of this Excel workbook was placed in a secure location on the Hudbay server that was accessible to a small group of approved Hudbay personnel. These Excel files were imported into several different software (Minesight, Target for Geosoft, Leapfrog etc.) that allowed further validation of the data.

Overall quality of samples taken, recorded and submitted to the laboratory was excellent with few data entry errors that were identified and corrected. Of the 21,647 samples submitted from the Phase I campaign only one sample was lost. No samples were reportedly lost from the 17,485 samples submitted during the Phase II campaign. Importing the assay certificates directly into the database virtually eliminated any data entry errors.

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

#### 12.1.5 Data Security

The historical assay database and the Hudbay assay database are administered by the database manager with working copies kept on the local drive of a secure computer and backups placed on a secure location on a Hudbay server. Any requests for edits to the database are made to the database manager who updates all the copies. All paper copies of the historical assay certificates and logs are available on the Hudbay's internal Sharepoint website with restricted access.

Moving forward, all of the Rosemont historical and current data will be migrated to the AcQuire platform that provides robust data security and long-term data storage solutions.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Overview

Recorded metallurgical testwork on Rosemont ores comprises work beginning as early 1974 by Anamax Mining Company. Following Augusta's acquisition of the Rosemont group of properties in 2005, Augusta continued the work and concluded it with the publication of NI 43-101 technical reports, the first in 2007, followed by another dated August 28, 2012. These reports were authored by M3 Engineering & Technology QPs.

Following its acquisition of Rosemont in 3Q2014, Hudbay completed two drilling programs (the first commencing late 2014 and the second in late 2015) and initiated a series of phased metallurgical testing programs, each designed to advance its understanding of the deposit and metallurgical performance in response to treatment.

A mine planning effort was also initiated, beginning with an effort that utilized the two phases of drilling and associated metallurgical testwork programs that were conducted through 2014 and 2015. In early 2016, an updated block model was developed, and the mine plan was subsequently updated in mid-2016 to reflect the changes in the understanding of the deposit. The new mine plan drove some changes to pit phasing and mining sequences (see additional detail about the updated model and new mine plan in Sections 14 through 16).

The principal objectives of the phased metallurgical testing programs were to:

- Enhance the understanding of the Rosemont resource
- Confirm the quality of the prior metallurgical testwork
- Identify downstream processing methods, forecast recoveries and quality of final products
- Evaluate characteristics of tailing products
- Derive a required ore processing flowsheet and size process equipment

Three composite samples were prepared for metallurgical testing in 2015, and among these were a set of three samples that corresponded to ore that was projected (under the earlier mine plan) to report from the mine during the first five years of operation, the second five years, and a third sample for the balance of operations.

As reporting from the first phase of metallurgical testwork programs became available, a revised set of composites was prepared to further enhance the understanding of the orebody, more particularly as it related to metallurgical performance in the presence of clays, as well as process equipment sizing and selection (principally flotation and dewatering equipment). For this phase of the testwork, a subset of these composites was selected again on the basis of when the ore was scheduled to report from the mine (again, under the earlier draft of the mine plan), this time with date ranges chosen as production years 1 through 3, 4 through 7, and after year 7. Again, due to timing all of these composites were chosen using the production schedule from the original mine plan.

The period descriptors for these several composites may not be strictly accurate under the revised mine plan developed by Hudbay in 2016 since testing has shown copper recovery to be strongly correlated to the ratio of sulfide copper to total copper, regardless of ore type or location in the deposit. The balance of this report section will discuss:

- Historical metallurgical testwork programs
- Hudbay's phased metallurgical testwork program
- Principal conclusions and recommendations that result from this body of work

## 13.2 Historical Metallurgical Testwork Summary

The information presented in the following historical summary was important to Hudbay's early understanding of Rosemont mineralogy and Augusta's strategies for liberating metals of interest from the deposit. It was this early review and investigation of the prior work that drove the definition of the subsequent drilling and metallurgical testing program implemented immediately following Hudbay's acquisition of Rosemont. Subsequent drilling, sampling and metallurgical testing programs discussed in later paragraphs of this Section will provide an appreciation of the evolution of Hudbay's understanding of the deposit and final conclusions with respect to processing strategies, flowsheet development and forecasts of recoveries and product quality.

### 13.2.1 Early Work

The earliest reported testwork on Rosemont ores comprising preliminary grinding and flotation tests was completed by Anamax Mining Company in 1974. This early work was followed by a larger testwork campaign by Augusta in 2006-2007 to support the preparation of a feasibility study and technical report. Further testwork was then completed by Augusta between 2008 and 2013 to support engineering design and updates to the original technical report. The description of the Augusta work is provided in this Section and was part of the "Rosemont Copper Project, NI 43-101 Technical Report", dated August 28, 2012.

Historical metallurgical testwork programs were undertaken by Mountain State R&D International ("MSRDI"), SGS and G&T Metallurgical Services, with dewatering and rheology testing undertaken by Pocock, Outotec and FLSmidth. Early attempts to characterize the deposit were difficult due to the large differences in mineralogy and high degree of variability within the major lithologies. The testwork programs had previously isolated and tested different lithologies and period composites without successfully correlating metallurgical performance with specific ore types

The balance of this Section summarizes the results of the previous metallurgical test programs, those conducted prior to Hudbay's acquisition of Rosemont. These are more fully described in Augusta's technical report titled "Rosemont Copper Project, NI 43-101 Technical Report, Updated Feasibility Study" dated August 28, 2012.

### 13.2.2 Basis

The earliest existing records of metallurgical testing are from the period 1974 to 1975, at which time grinding and flotation tests were performed. In the first half of 2006, Augusta initiated test-work to provide a better understanding of the metallurgy of the Rosemont mineralization and establish criteria for the design of a process facility.

The program tested both composites and individual variability samples and are considered to be fairly representative of the variety of ore conditions within the deposit.

### 13.2.3 Mineralization & Ore Types

The ore contains three main copper sulfide minerals (in order of relative abundance): chalcopyrite, bornite, and chalcocite/covellite. The deposit was described as having three major and several minor lithological units, within which the various types of sulfide mineralization occur:

- Horquilla
- Earp
- Colina
- Other including Epitaph and Escabrosa

Two samples of ground Horquilla sulfide material were examined by detailed mineralogical modal analysis. The result of this analysis indicated a large difference in copper mineralogy within the Horquilla rock type and association of silver and gold with the copper sulfide minerals. Molybdenite, $MoS_2$, was the only molybdenum mineral identified.

The copper oxide minerals identified as primarily chrysocolla, tenorite, malachite, and azurite. Oxide resources are distributed in three major rock units as follows:

- Arkose
- Porphyry – Quartz Monzonite (“QMP”) or Quartz Latite (“QLP”)
- Andesite

### 13.2.4 Comminution Work

Grinding mill sizing parameters were provided to mill manufacturers for use in their mill sizing methods. The mill sizing parameters are shown in Table 13-1.

**TABLE 13-1: GRINDING MILL SIZING PARAMETERS**

| Parameter | Value |
|---|---|
| $CW_i$ | 4.90 |
| $RW_i$ | 12.40 |
| $BW_i$ | 11.40 |
| Tonnage | 3,400 tph |
| SAG Mill Feed Size | 150,000µ |
| Transfer Size | 3,000µ |
| Ball Mill Product Size | 105µ |

### 13.2.5 Flotation Testwork

Flotation test-work was performed during the years 1974 to 1975 and 2006 to 2008. The tests included bench-scale rougher-scavenger and cleaner tests, rougher variability tests, and rougher cleaner optimization tests. Based on the test results the flotation conditions were indicated to be as follows:

- Primary grinding to P80 = 105µ
- Rougher flotation pH= 9.7 to 10.8
- AP-238 and AX-343 collectors
- Regrind to P80 = 74µ
- One stage of cleaner flotation

The result of the variability tests indicated that the grind size has an effect on both copper recovery and rougher concentrate grade. The mineralogical modal analyses indicated that the chalcopyrite liberates at a coarser size, between 150 and 75µ, than bornite and chalcocite. The molybdenite began to liberate from the gangue between 150 and 75µ, but remained locked to a significant degree with gangue to about 22µ.

### 13.2.6 Molybdenum Testwork

During 2008, flotation tests were conducted at MSRDI on composite samples of five individual rock lithology samples and one composite sample representing the material expected to be processed during the first three years of process plant operation. The test program was designed to examine the process of producing molybdenite concentrate.

The bulk (copper-molybdenite) flotation concentrate from mineralized Horquilla produced a molybdenite concentrate grading 52.7% molybdenum with a 93% molybdenum recovery from bulk concentrate. The results of testing the other samples indicated lower molybdenite concentrate grades and with variable molybdenite recovery from the bulk concentrate with the procedure used. The results of the testing are presented in Table 13-2.

**TABLE 13-2: MOLYBDENITE FLOTATION**

| Molybdenite Flotation | | | | |
|---|---|---|---|---|
| Sample | Concentrate Assay % | | | Recovery % Mo |
| | Cu | Mo | Insol | |
| Horquilla | 0.44 | 52.7 | 1.8 | 93.0 |
| Colina | 0.70 | 26.5 | 16.9 | 96.5 |
| Earp | 0.50 | 42.8 | 6.5 | 93.0 |
| Epitaph | 0.30 | 39.3 | 17.5 | 55.7 |
| Escabrosa | 0.50 | 27.9 | 25.8 | 84.8 |
| 1 – 3 Year Composite | 0.06 | 41.6 | 13.5 | 96.5 |

### 13.2.7 2012 Metallurgical Test Program

In 2012, a metallurgical test program was designed by Augusta to prepare composite samples representing four periods of expected mine production and test them by bench scale test procedures. The test procedures followed the treatment methods proposed for the proposed process plant. The metallurgical test composite samples were prepared from half-core from six holes drilled in late 2011. The half-core drill segments were selected so that the composite samples were representative of grade, lithology, and spatial characteristics of material predicted to be produced during the mine operating periods of years 1 through 3, years 4 through 7, years 8 through 12, and years 13 through 21. The composition of the composite samples by lithology is shown in Table 13-3.

**TABLE 13-3: LITHOLOGY OF COMPOSITE SAMPLES**

| Lithology | Composite Samples Representing Expected Mine Production Years | | | |
|---|---|---|---|---|
| | 1 through 3 | 4 through 7 | 8 through 12 | 13 through 21 |
| Epitaph | | | 10% | 16% |
| Colina | | 11% | 17% | 25% |
| Earp | 16% | 28% | 23% | 16% |
| Horquilla | 84% | 61% | 50% | 43% |

The result of closed circuit flotation tests are summarized in Table 13-4:

**TABLE 13-4: 2012 CLOSED CIRCUIT FLOTATION RESULTS**

| Period | Copper Recovery | Molybdenum Recovery | Final Bulk Concentrate Grade | | |
|---|---|---|---|---|---|
| | | | Copper | Molybdenum | Silver |
| Yr 1-3 | 87.9% | 62% | 41% | 1.02% | 502ppm |
| Yr 4-7 | 81.2% | 2.5% | 44% | 0.047% | NA |
| Yr 8-12 | 92% | 84% | 28% | 1.22% | NA |
| * Yr 13-21 1st composite | 75.8% | 31.1% | 36% | 0.56% | NA |
| * Yr 13-21 2nd composite | 91.4% | 66% | 37% | 0.84% | NA |

* Note: Core submitted for the years 13 to 21 composite were found to have a higher content of oxidized material than was expected to be mined during that phase of mining and resulted in low overall metal recovery. For that reason, a second composite was assembled containing a lesser percentage of oxidized material and the flotation tests were repeated.

The anomalous value obtained for the molybdenum recovery (years 4-7) was checked by re-testing the same composite sample, resulting in improved rougher concentrate molybdenum recovery of 40% to 60%. Previous results from testing samples containing the Colina mineralization indicated that lower molybdenum recovery was to be expected, however the cause was not specifically known at the time.

Additional analysis of the concentrate produced in the testwork (years 8-12) showed that the concentrate contained low amounts of contaminants such as arsenic (<80 g/ton) and mercury (0.8 g/ton) and contained both gold (1.91 g/ton) and silver (294 g/ton).

### 13.2.8 Principal Observations

The significant conclusions that can be drawn from the prior metallurgical testing program are as follows:

- Rosemont ore responds well to proven and widely used mineral separation techniques.

- Concentrate grade vs. recovery and flotation feed grind size vs. recovery relationships were established and follow expected trends.
- Early process plant design criteria were developed on the basis of this work.

## 13.3 Hudbay Metallurgical Testing Programs

Following the acquisition of the Project in the third quarter of 2014, Hudbay undertook a series of additional drilling, sampling and metallurgical testwork programs. Drilling programs were undertaken in late 2014 and 2015, and are discussed in greater detail in Section 10 of this Report.

In 2014, Hudbay engaged XPS Consulting & Testwork Services ("XPS") to undertake mineral characterization and metallurgical testwork. Base Met Laboratory ("BML") was engaged in late 2015 to provide confirmation testwork of the XPS testwork and additional process optimization.

The Hudbay metallurgical testing programs are separated into the following phases:

- XPS Phase 1: Variability Test Program
- XPS Phase 2: Geometallurgical Variables
- XPS Phase 3: Copper / Moly Separation, flotation response to clays
- BML Confirmation Testing
- Production Period Testwork

## 13.4 XPS Phase 1

In late 2014, Hudbay initiated a variability test program (XPS Phase 1) that would improve the understanding of the mineral and lithological data and help define geo-metallurgical characteristics. The objective was to improve the correlation between mineralogy/geology and metallurgical variability observed in prior metallurgical testwork conducted by others.

The data presented in this section is taken from the 2015 SGS report on grindability characteristics of samples provided by XPS.

All of the samples in this program were analyzed by Inductively Coupled Plasma (ICP), X-Ray Diffraction ("XRD"), Quantitative Evaluation of Minerals by Scanning electron microscopy ("QEMSCAN"), Cation Exchange Capacity ("CEC") and Near-Infrared ("NIR"). ICP measured elemental assays while XRD measured mineral composition. CEC and NIR analysis determined the clay content and other alteration minerals content.

A total of 140 samples (Met1A, Met1B and Met2 samples) were sent to SGS Canada in Lakefield, Ontario for comminution testing, specifically for the JK drop-weight test, the SPI® test and the Bond ball mill grindability test at a closing size of 150 mesh (106 µm). JK drop-weight tests require coarser material and hence this testing was only possible on the 33 new full HQ core samples (Met2).

Statistics from the comminution test results are summarized in Table 13-5.

**TABLE 13-5: XPS PHASE 1 - COMMINUTION TEST STATISTICS***

| Statistic | DWT | | | SPI (min) | BWi (kWh/tonne) |
|---|---|---|---|---|---|
| | Relative Density* | A x b | $t_a$ | | |
| Average | 2.84 | 46.9 | 0.54 | 94.6 | 13.0 |
| Standard Deviation | 0.17 | 17.1 | 0.31 | 54.4 | 2.5 |
| Minimum* | 2.56 | 94.2 | 1.49 | 24.9 | 8.2 |
| Median | 2.83 | 45.6 | 0.49 | 82.1 | 13.0 |
| 75$^{th}$ Percentile | 2.94 | 37.0 | 0.39 | 117 | 14.4 |
| 90$^{th}$ Percentile | 3.06 | 25.1 | 0.22 | 151 | 16.0 |
| Maximum* | 3.28 | 18.6 | 0.14 | 401 | 21.7 |

* Reference: 5a-SGS-14816-001 FINAL Rpt Apr 17 2015.pdf

### 13.4.1 General Observations

Hudbay's technical services team initiated an effort to map geochemical characteristics of the various ore types, intending to utilize this data as predictors of recovery on the basis of ore type, indicators of clay distribution in the orebody, and other proxies. While this work has produced results, a stronger indicator of recovery is the sulfide component of total copper in the sample. Hudbay will continue to develop the geochemical database with the intent to leverage its value during subsequent project execution and operational phases.

Results from the XPS Phase 1 sample characterization and testwork program suggested the following:

- There was significant variability within the major lithologies with respect to copper oxide, clay content and ore hardness.
- All of the samples tested contained measurable amounts of swelling and/or magnesium-bearing clays, indicating widespread clay presence in the deposit and across all ore types.
- Swelling clay, as identified by its CEC content, varied widely from 4% to over 30%, averaging about 8.5%.
- Magnesium-bearing clays, including serpentine and talc, varied from 0% to over 35%, with an average of 1.8%.

Sample analysis results also showed that copper oxide varies widely, from 0% to 90%, averaging about 5.4% for these samples.

### 13.4.2 Comminution Results

Both JK drop-weight test ("DWT") and Bond ball mill work index ("BWI") results ranged from very soft to very hard while SAG Power Index ("SPI") test results ranged from soft to very hard. A 75$^{th}$ percentile DWT Axb of 37.0 was determined based on 33 samples, while a 75$^{th}$ percentile BWI of 14.4 kWh/ton was determined based on 140 samples. A 75$^{th}$ percentile parameters were chosen as the basis for design of the comminution circuit.

### 13.4.3 Flotation Results

The XPS Phase 1 flotation program consisted of rougher kinetic flotation tests for 107 samples (Met1A and Met1B). These tests followed a standardized flotation schedule drawn from previous work, employing the following conditions:

- PAX 50 g/tonne (0.1 lb/ton).
- Fuel Oil 30 g/tonne (0.06 lb/ton).
- MIBC 30 g/tonne (0.06 lb/ton).
- Lime to pH 9.5.
- Target grind P80 105 µm.

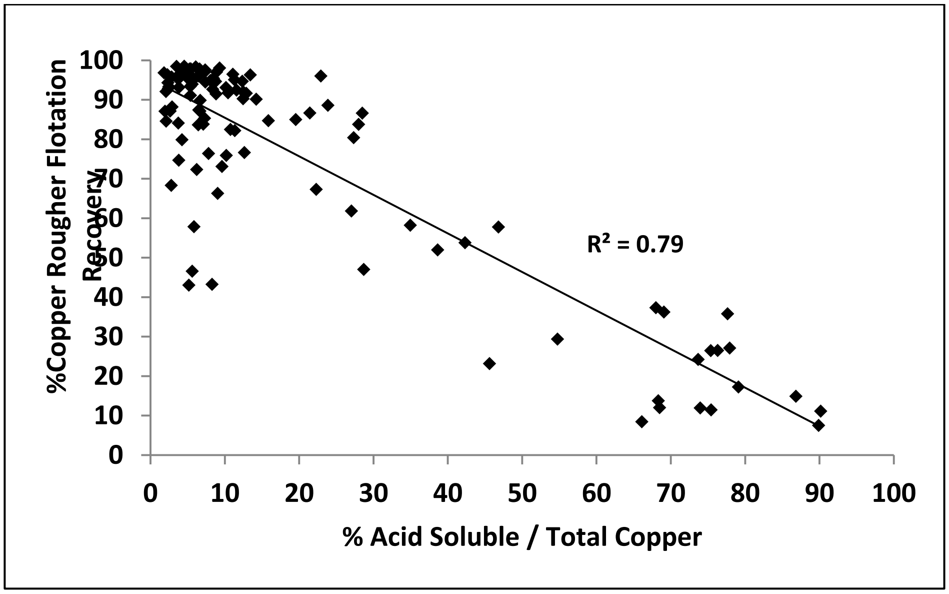
There was no attempt in Phase 1 to optimize flotation conditions. Grind determinations were recorded for each test. Lab results report the following average (rougher feed) grind sizes by sample type:

| Sample | $P_{80}$, µm | Std Deviation, µm |
|---|---|---|
| Met1A | 94 | 21 |
| Met1B | 107 | 31 |

Clay content, and specifically swelling (CEC) clay content, appears to have an impact on flotation recovery. However, the impact is not consistent across all of the tests. Samples with swelling clay content above about 12% typically yielded lower rougher recoveries.

Rougher flotation results showed a high level of variability, however, a strong correlation was determined between oxide copper (defined as the ratio of acid soluble copper to total copper) and rougher flotation recovery as shown in Figure 13-1.

**FIGURE 13-1: % COPPER ROUGHER FLOTATION RECOVERY VS % ACID SOLUBLE / TOTAL COPPER**




## 13.5 XPS Phase 2

The principal objective of the XPS Phase 2 testwork was to investigate the key geo-metallurgical variables identified in Phase 1, which are:

- Copper oxide content
- Swelling clays
- Magnesium clays
- Ore hardness

A series of composites were prepared, three according to production period criteria (production years 1 through 5, 6 through 10, and 11 through LOM), and four geometallurgical subtype composites and were tested under the Phase 2 testwork program as follows:

- Base 1: Main sulfide ore, Production Years 1 – 5
- Base 2: Main sulfide ore, Production Years 6 – 10
- Base 3: Main sulfide ore, Production Years > 10
- Sub 4: ~25% copper oxide ore
- Sub 5A: Swelling clay (CEC) rich ore >10%
- Sub 5B: Magnesium clay (Serpentine and talc) rich ore >10%
- Sub 6: Hard sulfide ore with BWi >14
- Note: "Production years" above refers to the preliminary, not final 2016 mine plan

The results of this phase of testwork are reported in the Geometallurgical Program Technical Review (XPS, 2015b).

### 13.5.1 Mineralogy

Phase 2 composites were submitted for mineralogical characterization using XRD with Rietveld Refinement, CEC analysis and QEMSCAN along with Electron Probe Micro-analysis ("EPMA") to validate phase compositions and copper deportments.

XRD analysis was completed as a cross-check to the QEMSCAN analyses and was found to correlate well. A summary of the QEMSCAN data is presented in Table 13-6. As QEMSCAN is limited in its ability to isolate and quantify both fine clays and swelling species, CEC analysis was used to define the swelling-clay content. These results are presented in Table 13-7.

**TABLE 13-6: XPS PHASE 2 - QEMSCAN ANALYSIS**

| Mineral | Base 1 | Base 2 | Base 3 | Sub 4 | Sub 5A | Sub 5B | Sub 6 |
|---|---|---|---|---|---|---|---|
| Serpentine Talc Clays | 2.2 | 3.4 | 6.4 | 1.3 | 2.6 | 18.7 | 0.9 |
| Muscovite | 0.5 | 0.9 | 1.0 | 0.8 | 3.4 | 0.5 | 2.4 |
| Biotite | 0.7 | 1.1 | 1.4 | 1.3 | 4.8 | 1.9 | 3.5 |
| Chlorite | 1.7 | 2.6 | 1.8 | 2.7 | 4.0 | 3.0 | 2.0 |
| Quartz | 23.3 | 15.4 | 7.1 | 25.5 | 24.5 | 0.3 | 19.1 |
| K-Feldspar | 7.0 | 8.4 | 3.1 | 9.2 | 13.6 | 0.4 | 21.7 |
| Garnet | 24.2 | 16.5 | 14.6 | 21.7 | 8.5 | 5.8 | 11.0 |
| Calcite | 17.9 | 26.9 | 39.5 | 23.3 | 14.6 | 40.5 | 6.4 |
| Pyrite | 0.4 | 0.5 | 0.2 | 0.1 | 0.3 | 0.7 | 0.3 |
| Chalcopyrite | 0.4 | 0.9 | 1.0 | 0.3 | 0.7 | 1.0 | 0.7 |
| Bornite | 0.2 | 0.2 | 0.1 | 0.1 | 0.1 | 0.3 | 0.3 |
| Chalcocite/Cov. | 0.1 | 0.2 | 0.1 | 0.2 | 0.3 | 0 | 0.1 |
| Cu Oxide-other | 0.1 | 0.3 | 0.1 | 0.8 | 0.1 | 0.1 | 0.1 |
| Other | 21.3 | 22.7 | 23.6 | 12.7 | 22.4 | 26.8 | 31.5 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

**TABLE 13-7: XPS PHASE 2 - CEC ANALYSIS**

| Test | Base 1 | Base 2 | Base 3 | Sub 4 | Sub 5A | Sub 5B | Sub 6 |
|---|---|---|---|---|---|---|---|
| CEC | 7.1 | 9.5 | 5.9 | 6.3 | 10.6 | 6.1 | 8.3 |

Based on a combination of the EPMA mineral composition data and the QEMSCAN modal abundance data, the elemental deportment by mineral can be calculated for copper as is summarized in Table 13-8.

**TABLE 13-8: XPS PHASE 2 - COPPER DEPORTMENT BY MINERAL SPECIES**

| Minerals | Base 1 | Base 2 | Base 3 | Sub 4 | Sub 5A | Sub 5B | Sub 6 |
|---|---|---|---|---|---|---|---|
| Chalcopyrite | 33.4 | 49.1 | 67.2 | 24.8 | 39.4 | 58.1 | 48.1 |
| Bornite | 31.5 | 18.2 | 16.2 | 7.6 | 11.1 | 32.7 | 28.9 |
| Chalcocite | 20.5 | 22.0 | 8.3 | 32.9 | 33.1 | 3.1 | 18.5 |
| Covellite | 6.1 | 1.3 | 0.1 | 1.3 | 4.8 | 0.0 | 0.1 |
| Other Sulfides | 0.4 | 0.3 | 0.2 | 0.4 | 4.3 | 0.3 | 0.1 |
| **Total Sulfide Copper** | **92.0** | **90.9** | **92.1** | **67.1** | **92.8** | **94.3** | **95.6** |
| Chrysocolla | 0.6 | 2.1 | 0.9 | 3.9 | 0.1 | 0.0 | 0.1 |
| Cu Chlorite | 3.6 | 3.6 | 3.3 | 5.3 | 5.1 | 4.1 | 3.0 |
| Goethite | 0.8 | 0.8 | 0.3 | 3.2 | 0.5 | 0.4 | 0.3 |
| Cu Oxide Other | 2.8 | 2.3 | 3.4 | 20.0 | 1.5 | 0.9 | 1.0 |

| Minerals | Base 1 | Base 2 | Base 3 | Sub 4 | Sub 5A | Sub 5B | Sub 6 |
|---|---|---|---|---|---|---|---|
| Other | 0.1 | 0.2 | 0.0 | 0.1 | 0.0 | 0.1 | 0.0 |
| **Total Oxide Copper** | **8.0** | **9.1** | **7.9** | **32.9** | **7.2** | **5.7** | **4.4** |
| **TOTAL** | **100** | **100** | **100** | **100** | **100** | **100** | **100** |
| Cu-Oxide by Assay | 5.1 | 7.7 | 7.1 | 24.7 | 3.8 | 5.7 | 3.0 |

In general terms, testwork conducted on the composited samples provide the following as indicators of variability in the orebody:

- Copper oxide content is variable and continues at depth.
- Copper deportment to chalcopyrite increases with depth while chalcocite and bornite contents are reduced.
- Base 2 composite has relatively high molybdenum levels when compared to others.
- Base 2 composite has elevated CEC clay content relative to Base 1 and Base 3.
- Serpentine and talc content increases with depth (Base 3 has three times the Mg clay content of Base 1)
- Fe and Si contents decrease with depth, while Ca and Mg contents increase.
- Calcite content increases from Base 1 to Base 3.

#### 13.5.2 Results - Phase 2 Flotation Testwork

XPS conducted over 104 flotation tests to investigate the effect of primary grind size, reagents, pH modifiers, dispersants and rougher and cleaner pulp densities in parallel with locked cycle testing. A primary grind size of 140 µm was selected for subsequent flotation testing on all composites.

Locked cycle tests were undertaken on Base 1, Base 2, Sub 4 and Sub 5A. The locked cycle flotation test results are given in Table 13-9.

### TABLE 13-9: XPS PHASE 2 - LOCKED CYCLE TEST RESULTS

| Sample | Test | Stream | Mass Percent | Grade - Percent | | Distribution - Percent | |
|---|---|---|---|---|---|---|---|
| | | | | Cu | Mo | Cu | Mo |
| Base 1 | Float 094 Cycles 3-5 | Feed | 100 | 0.58 | 0.012 | 100 | 100 |
| | | Cu Concentrate | 1.5 | 31.2 | 0.451 | 82.7 | 59.7 |
| | | Cu Cleaner Tail | 4.8 | 0.69 | 0.041 | 5.7 | 16.8 |
| | | Cu Rougher Tail | 93.7 | 0.07 | 0.003 | 11.6 | 23.6 |
| Base 2 | Float 114 Cycles 4-6 | Feed | 100 | 0.59 | 0.022 | 100 | 100 |
| | | Cu Concentrate | 1.7 | 22.1 | 0.394 | 64.0 | 30.1 |
| | | Cu Cleaner Tail | 11.0 | 0.88 | 0.070 | 16.5 | 34.7 |
| | | Cu Rougher Tail | 87.3 | 0.13 | 0.009 | 19.6 | 35.2 |
| Base 2 | Float 145 Cycles 4-6 | Feed | 100 | 0.59 | 0.022 | 100 | 100 |
| | | Cu Concentrate | 1.6 | 28.2 | 0.624 | 74.7 | 44.6 |
| | | Cu Cleaner Tail | 10.4 | 0.30 | 0.037 | 5.3 | 17.4 |
| | | Cu Rougher Tail | 88.0 | 0.14 | 0.010 | 20.0 | 38.0 |
| Sub 4 | Float 104 Cycles 4-6 | Feed | 100 | 0.54 | 0.005 | 100 | 100 |
| | | Cu Concentrate | 1.0 | 31.9 | 0.198 | 61.3 | 43.3 |
| | | Cu Cleaner Tail | 3.1 | 0.75 | 0.009 | 4.4 | 5.7 |
| | | Cu Rougher Tail | 95.8 | 0.19 | 0.003 | 34.3 | 51.0 |
| Sub 5A | Float 103 Cycles 4-6 | Feed | 100 | 0.46 | 0.010 | 100 | 100 |
| | | Cu Concentrate | 1.7 | 22.6 | 0.327 | 81.6 | 54.4 |

| Sample | Test | Stream | Mass Percent | Grade - Percent | | Distribution - Percent | |
|---|---|---|---|---|---|---|---|
| | | | | Cu | Mo | Cu | Mo |
| | | Cu Cleaner Tail | 6.9 | 0.44 | 0.033 | 6.6 | 22.6 |
| | | Cu Rougher Tail | 91.5 | 0.06 | 0.003 | 11.8 | 22.9 |

## 13.6 XPS Phase 3

The XPS Phase 3 of work focused on the separation of copper and molybdenum, investigating the flotation response of blended high-clay ore mixtures and the effect of calcium rich water on flotation behavior.

### 13.6.1 Copper-Molybdenum Separation Testwork

Copper-Molybdenum separation testwork was of necessity constrained by the limited sample available, a consequence of the relatively small quantity of Cu/Mo concentrate available for testwork. Subsequent paragraphs will describe general conclusions that may be drawn from the copper-moly testwork that has been completed, and suggests some additional work that may be required during project execution and operational phases of the Project.

Stored cleaner 1 concentrate from XPS Phase 2 tailings generation work was upgraded to produce a copper-molybdenum bulk concentrate suitable for copper-molybdenum separation. Application of three additional stages of cleaning raised the copper grade to 35.8%, 40.6% and 41.4% copper, respectively.

The copper cleaner 4 concentrate, or copper molybdenum bulk concentrate, was split for assays, mineralogical analysis, a scoping separation test and the remainder used for the copper molybdenum demonstration test.

To depress the copper minerals, the pH of the bulk concentrate was elevated to pH 12 with lime and then treated with sodium hydrosulfide (NaHS). Unfortunately for this phase of the testwork, there was not enough molybdenum rougher concentrate to do more than a single stage of molybdenum cleaning.

### 13.6.2 Observations & Discussion

While the results above suggest that additional stages of cleaning may be required to produce an acceptable bulk copper concentrate, subsequent analysis has demonstrated that acceptable CuMo concentrate grades can be achieved by applying staged flotation reactor (“SFR”) flotation technology in the bulk cleaner circuit flowsheet. SFR flotation cells have been proven to achieve recovery rates equivalent to or better than conventional flotation cells while realizing exceptionally high upgrade ratios. This performance is accomplished though separating the three phases of flotation (particle collection, bubble disengagement, and froth recovery) into different zones such that each phase can be optimized independent of the others. To further enhance upgrading performance, under-froth dilution/wash water can be applied in the froth recovery unit to significantly improve gangue rejection from the concentrate product. Accordingly, the incorporation of SFR flotation technology in the flotation circuit design supersedes the need for additional stages of cleaning.

### 13.6.3 Results – Phase 3 Copper – Moly Separation

The results from this test are shown in Table 13-10.

**TABLE 13-10: XPS PHASE 3 - MOLYBDENUM SEPARATION TEST**

| | Mass % | Grades | | | Recovery | | |
|---|---|---|---|---|---|---|---|
| | | Cu % | Mo % | MgO % | Cu % | Mo % | MgO % |
| Mo Cleaner Concentrate | 0.4 | 3.45 | 30.7 | 5.38 | 0.0 | 38.5 | 7.8 |
| Mo Rougher Concentrate | 1.5 | 19.4 | 18.0 | 4.48 | 0.7 | 79.6 | 22.8 |
| Mo Rougher Tails | 98.5 | 41.2 | 0.07 | 0.23 | 99.3 | 20.4 | 77.2 |
| Cleaner 4 Concentrate (Recalculated) | 100.0 | 40.9 | 0.34 | 0.29 | - | - | - |

Copper was depressed producing a molybdenum concentrate of 30.7% molybdenum upgraded from 18% and 0.34% molybdenum in the molybdenum rougher concentrate and bulk concentrate respectively. This corresponds to 79.6% molybdenum recovery from the bulk concentrate to the molybdenum rougher concentrate and 38.5% molybdenum recovery to the molybdenum cleaner. Additional cleaning stages were not pursued in this program due to the small sample size available.

### 13.6.4 Flotation of Composite Blends and Water Testwork

Blends were made up of clean ore (“Sub 6”), swelling-clay ore (“Sub 5A”) and magnesium-clay ore (“Sub 5B”) to determine if the blends behaved as the sum of the components. Additionally, one test was conducted with calcium saturated water. This water was used in all stages of grinding and flotation for Float 158, which was otherwise identical to the fresh water equivalent test in Float 150. The results are summarized in Table 13-11 and Table 13-12.

**TABLE 13-11: XPS PHASE 3 - FLOTATION BLENDS TEST RESULTS**

| Test | Sub 6 | Sub 5A | Sub 5B | Rougher | | Cleaner 1, 2 | | Overall Circuit | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | R%Cu | %Cu | R%Cu | %Cu | R%Cu | %Cu |
| Float 105 | 100 | - | - | 92.0 | 16.6 | 93.9 | 29.3 | 86 | 29.3 |
| Float 093 | - | 100 | - | 90.4 | 3.7 | 77.3 | 22.1 | 70 | 22.1 |
| Float 121 | - | - | 100 | 75.8 | 1.7 | 74.4 | 24.3 | 56 | 24.3 |
| Float 150 | 75 | 25 | - | 88.4 | 8.7 | 94.1 | 28.7 | 83 | 28.7 |
| Prediction | 75 | 25 | - | 91.6 | 9.4 | 90.2 | 27.6 | 83 | 27.6 |
| Float 149 | 75 | - | 25 | 88.8 | 4.7 | 70.5 | 24.5 | 63 | 24.5 |
| Prediction | 75 | - | 25 | 87.8 | 5.7 | 89.5 | 28.2 | 79 | 28.2 |
| Float 151 | 75 | 12.5 | 12.5 | 86.7 | 6.2 | 70.7 | 27.3 | 61 | 27.3 |
| Prediction | 75 | 12.5 | 12.5 | 89.7 | 7.1 | 89.9 | 27.9 | 81 | 27.9 |
| Float 152 | 50 | 25 | 25 | 85.4 | 3.5 | 75.2 | 25.6 | 64 | 25.6 |
| Prediction | 50 | 25 | 25 | 87.3 | 4.4 | 85.6 | 26.4 | 75 | 26.4 |

**TABLE 13-12: XPS PHASE 3 - FLOTATION WATER TEST RESULTS**

| | Sub 6 | Sub 5A | Sub 5B | Rougher | | Cleaner 1, 2 | | Overall Circuit | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | R%Cu | %Cu | R%Cu | %Cu | R%Cu | %Cu |
| Float 150 | 75 | 25 | - | 88.4 | 8.7 | 94.1 | 28.7 | 83 | 28.7 |
| Float 158* | 75 | 25 | - | 91.0 | 10.6 | 79.7 | 29.9 | 73 | 29.9 |

* High calcium water test.

The rougher results were close but generally slightly less than the mathematical weighted sum of the end member floats. However, cleaner results departed from predictions for the magnesium-clay blends as well as for the calcium-saturated float. Cleaner tests were conducted at low densities (ranging from 4 to 18% w/w solids) leaving reagent strategy, dispersants and physical set-up as possible mitigation factors.

## 13.7 BML Confirmation Testing

BML conducted a testwork program at their Kamloops laboratory in late 2015. The main objective was to confirm the flotation process parameters developed by XPS during the Phase 1 and Phase 2 testwork and to replicate the locked cycle results with the Base 1 and Base 2 samples. Results of this testwork are summarized in BML report BL065.

The test program was completed using two period composites, Base 1 and Base 2. The main difference between these replicate tests and previous XPS tests was the use of traditional manual froth scraping and shorter flotation times. More specifically, tests conducted at BML were performed with constant froth levels and a technician manually recovering froth, while the XPS program used a mechanical froth paddle system with level manipulation to achieve desired mass recoveries.

Further, BML locked cycle tests used three stages of cleaning and returned the cleaner scavenger concentrate and cleaner 2 tail to cleaner 1 rather than regrind as these streams should already be sufficiently reground.

The basic parameters of the XPS flowsheet were confirmed for Base 1, however, BML's Base 2 LCT grade-recovery performance was different, producing a higher copper concentrate grade of 34.5% (versus 28.2%) and lower copper recovery of 63.8% (versus 74.7%). Analysis of these results suggests that they are essentially different points on the same grade-recovery curve.

## 13.8 BML Production Period Testwork

In early 2016, production period samples, based on the 131-million tons/y mine plan developed for Hudbay by Independent Mining Consultants ("IMC") in November 2015, were bench tested for additional metallurgical and project engineering data. The purpose of this program was to add detail to the prior metallurgical testwork results in key areas in order to improve confidence in recovery and concentrate quality forecasts, as well as for sizing downstream process equipment, most notably the tailing dewatering and filtration equipment necessary to accomplish the drystack tailing deposition strategy. Results of this testing program are recorded in BML report BL076.

A program of batch and locked cycle testing was performed using the previously developed flowsheet and basic reagent scheme. In addition, SFR rougher pilot testing was done in parallel to the bench testing. Due to wall effects of the small SFR pilot units, the results of the pilot plant rougher tests fell short of bench test (conventional) results, thus SFR cells were not incorporated in the rougher flotation circuit design.

As noted previously, these "period composites" are not strictly accurate when referenced to the new 2016 Mine Plan; regardless, given the strong correlation of copper recovery to the sulfide copper component of the ore, moderate shifting of period composites in terms of production phase are not expected to materially impact the recovery conclusions.

**TABLE 13-13: PRODUCTION COMPOSITES**

| Composite | Years | Grade | | Acid Soluble Cu |
|---|---|---|---|---|
| | | %Cu | %Mo | ASCu/TCu, % |
| Period 1 | 1 – 3 | 0.54 | 0.013 | 9 |
| Period 2 | 4 – 6 | 0.48 | 0.013 | 13 |
| Period 3 | 7 - LOM | 0.72 | 0.018 | 13 |

### 13.8.1 Production Period Mineralogy

The feed sample mineral content was analyzed by SEM-EDX and XRD and the results presented in Table 13-14.

**TABLE 13-14: BML MINERAL CONTENT**

| | SEM-EDX | | | XRD | | |
|---|---|---|---|---|---|---|
| Mineral | Period 1 | Period 2 | Period 3 | Period 1 | Period 2 | Period 3 |
| Chalcopyrite | 0.39 | 0.44 | 0.59 | | | |
| Tetrahedrite | 0.01 | 0.01 | 0.02 | | | |
| Bornite | 0.21 | 0.29 | 0.61 | | | |
| Chalcocite/Covellite | 0.07 | 0.13 | 0.29 | | | |
| Other Copper | 0.01 | 0.00 | 0.05 | | | |
| Sphalerite | 0.04 | 0.05 | 0.07 | | | |
| Arsenopyrite | 0.00 | 0.01 | 0.00 | | | |
| Pyrite | 0.26 | 0.28 | 0.60 | | | |
| Molybdenite | 0.00 | 0.00 | 0.02 | | | |
| Fluorite | 0.1 | 0.1 | 0.1 | | | |
| Apatite | 0.1 | 0.2 | 0.1 | | | |
| Carbonates | 7.3 | 16 | 16 | 11 | 21 | 20 |
| Oxides | 0.4 | 0.5 | 0.5 | | | |
| Quartz | 25 | 12 | 12 | 18 | 9 | 10 |
| Feldspars | 16 | 18 | 15 | 18 | 17 | 17 |
| Mica | 1.5 | 2.1 | 1.8 | 0.0 | 2.5 | 1.5 |
| Pyroxene | 22 | 23 | 24 | 20 | 19 | 20 |
| Clays | 0.2 | 05 | 0.4 | 7.0 | 7.2 | 8.4 |
| Olivine | 0.0 | 0.1 | 0.1 | | | |
| Talc | 0.1 | 0.7 | 0.3 | | | |

| | SEM-EDX | | | XRD | | |
|---|---|---|---|---|---|---|
| **Mineral** | **Period 1** | **Period 2** | **Period 3** | **Period 1** | **Period 2** | **Period 3** |
| Epidote | 3.0 | 1.8 | 2.3 | 1.2 | 0.0 | 0.0 |
| Chlorite | 0.9 | 2.4 | 3.3 | 2.3 | 0.0 | 3.8 |
| Garnet | 19 | 10 | 16 | 22 | 13 | 16 |
| Amphibole | 1.0 | 1.7 | 1.6 | 0.0 | 1.6 | 1.8 |
| Serpentine | 0.0 | 4.0 | 2.0 | 0.0 | 4.0 | 0.8 |
| Other minerals | 2.0 | 5.5 | 2.6 | 1.2 | 6.3 | 1.6 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

The chemical, mineralogical and clay analyses were performed using the same laboratories as the previous Phase 2 test program. The swelling clay content as determined by CEC is given in Table 13-15.

**TABLE 13-15: BML CEC ANALYSIS**

| Test | Period 1 | Period 2 | Period 3 |
|---|---|---|---|
| CEC | 7.0 | 7.2 | 8.4 |

The composite samples were dominated by pyroxene, quartz, feldspars, garnet and carbonates. Previous testwork indicated several minerals that may affect flotation performance and include the serpentine group minerals, swelling clays (CEC), talc and fluorine bearing minerals (apatite, fluorite and micas). Period 1 had the lowest levels of these minerals while Period 2 had the highest.

As determined by SEM-EDX, sulfide content ranged from 1 to 2 percent of the feed, and were for the most part copper minerals with low levels of pyrite and trace levels of sphalerite, tetrahedrite and arsenopyrite.

The previously developed flowsheet and reagent scheme were used as a basis in the production period test program. The primary grind size was 140µm, and small changes in reagent additions and trial of gangue dispersants were investigated. Preliminary rougher and cleaner tests were performed prior to executing the locked cycle tests.

### 13.8.2 Results of Production Period Testwork

Rougher flotation tests indicated the addition of sodium hexametaphosphate (“SHMP”), a dispersant/chelating agent improved control of the non-sulfide gangue. The performance of Period composites 2 and 3 improved copper recovery by 8 and 3 units respectively. The batch cleaner tests showed shorter regrind time and reduced copper losses from cleaner tailings stream.

Based on BML results, the locked cycle tests averaged 97% recovery of the sulfide copper in the roughers and 93.7% copper recovery in the cleaners. Concentrate grade varied from 32% for Period 1, 34% for Period 2 and 38% for Period 3. The locked cycle test results for the Period composites together with the XPS replicate testing done previously at BML is given in Table 13-16.

**TABLE 13-16: BML LOCKED CYCLE TEST RESULTS**

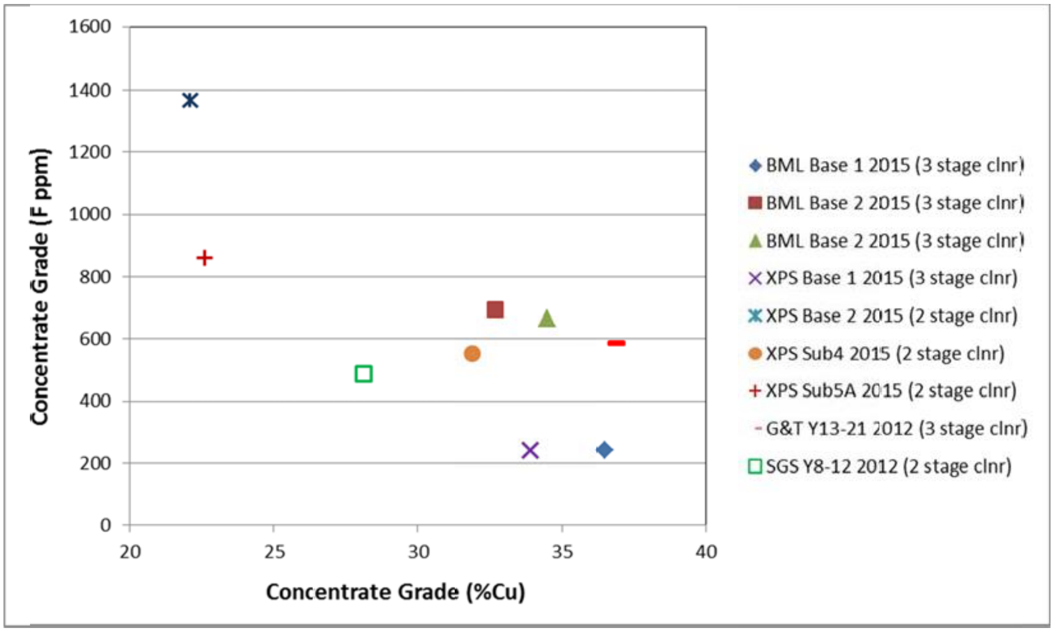
| Sample | Test | Stream | Mass Percent | Grade - Percent | | Distribution - Percent | |
|---|---|---|---|---|---|---|---|
| | | | | Cu | Mo | Cu | Mo |
| Base 1 | Float 011 Cycles D+E | Feed | 100 | 0.58 | 0.012 | 100 | 100 |
| | | Cu Concentrate | 1.2 | 36.5 | 0.298 | 78.5 | 32.2 |
| | | Cu Cleaner Tail | 7.1 | 0.79 | 0.046 | 9.7 | 28.0 |
| | | Cu Rougher Tail | 91.7 | 0.08 | 0.005 | 11.9 | 39.8 |
| Base 2 | Float 012 Cycles D+E | Feed | 100 | 0.61 | 0.022 | 100 | 100 |
| | | Cu Concentrate | 1.2 | 32.7 | 0.586 | 64.0 | 32.0 |
| | | Cu Cleaner Tail | 12.6 | 0.44 | 0.029 | 9.1 | 16.9 |
| | | Cu Rougher Tail | 86.2 | 0.19 | 0.013 | 26.9 | 51.0 |
| Base 2 | Float 014 Cycles D+E | Feed | 100 | 0.60 | 0.022 | 100 | 100 |
| | | Cu Concentrate | 1.1 | 34.5 | 0.558 | 63.8 | 28.0 |
| | | Cu Cleaner Tail | 10.3 | 0.56 | 0.035 | 9.6 | 16.1 |
| | | Cu Rougher Tail | 88.5 | 0.18 | 0.014 | 26.6 | 55.8 |
| Period 1 | Float Avg. 20,23,26 Cycles D+E | Feed | 100 | 0.53 | 0.011 | 100 | 100 |
| | | Cu Concentrate | 1.5 | 31.9 | 0.620 | 90.4 | 82.7 |
| | | Cu Cleaner Tail | 3.7 | 0.45 | 0.013 | 3.1 | 4.3 |
| | | Cu Rougher Tail | 94.8 | 0.037 | 0.002 | 5.9 | 13.0 |
| Period 2 | Float Avg. 21,24,27 Cycles D+E | Feed | 100 | 0.48 | 0.011 | 100 | 100 |
| | | Cu Concentrate | 1.05 | 33.9 | 0.497 | 73.1 | 48.8 |
| | | Cu Cleaner Tail | 4.3 | 1.03 | 8.7 | 6.6 | 9.5 |
| | | Cu Rougher Tail | 94.6 | 0.093 | 0.005 | 18.2 | 41.6 |
| Period 3 | Float Avg. 22,25,28 Cycles D+E | Feed | 100 | 0.73 | 0.016 | 100 | 100 |
| | | Cu Concentrate | 1.45 | 37.6 | 0.625 | 75.0 | 57.0 |
| | | Cu Cleaner Tail | 4.1 | 1.24 | 0.024 | 7.1 | 6.4 |
| | | Cu Rougher Tail | 94.4 | 0.14 | 0.006 | 18.0 | 36.6 |

## 13.9 Concentrate Quality

Fluorine is a deleterious element identified in Rosemont copper concentrates. Fluorine levels in copper concentrate above 350-400 ppm typically incur a penalty, with concentrates often rejected by smelters at fluorine levels greater than 900-1,000 ppm.

To mitigate the risk of copper concentrate rejection in the event of higher than normal fluorine levels, the maximum design fluorine grade in concentrate is 800 ppm. Cleaner flotation tests show that upgrading the concentrate rejects entrained fluorine. Available fluorine assays from locked cycle test of 8 concentrate samples are summarized in Figure 13-2. Four of these tests were carried out with only two stages of cleaning; the results suggest that additional cleaning tends to improve copper grade and decrease fluorine levels.

**FIGURE 13-2: LCT FINAL CONCENTRATE FLUORINE LEVELS**




The LCT results show that fluorine can be managed to acceptable levels (less than 800 ppm) with the use of two stage cleaning and froth washing. Good results were achieved with the Base 1 (Year 1-5) composite. Fluorine levels tended to be higher for later period composites.

Testing of concentrates also indicated that other common contaminants, such as arsenic and mercury, were found in concentrations sufficiently low to not warrant concern.

Given the presence of secondary copper sulfides and the need to produce a higher grade concentrate to reject fluorine, final concentrate grades may vary from 30-38% Cu. Nominal concentrate grades of 32% Cu for Years 1-5 and 33% Cu for Years 6-LOM were selected for equipment sizing. Operating assumptions included in the mill production model utilize concentrate grades of 32% Cu for operating years 1 through 3, 34% in year 4, and 35% in year 5 and beyond on the basis of expected increase in requirement to reject fluorine from concentrate.

## 13.10 Tailings Dewatering

Tailings samples were generated by XPS to be tested by Andritz, Bilfinger, FLSmidth (FLS), Outotec and Pocock for water separation and recovery from tailing prior to deposition in the DSTF. As expected, clay content and size distribution has a significant effect on tailings dewatering. The samples with lower clay content (Base 1 and Sub 4) generally achieved the highest thickener underflow densities. As expected, specific high clay samples (Sub 5A and Sub 5B) achieved lower densities across most tests.

On average, the high compression thickener tests achieved underflow densities 3%–4% higher than the high rate thickening tests. Generally, high rate thickeners could be expected to achieve an underflow density of 65% for lower clay content ore, while high compression thickeners could be expected to achieve an underflow density up to 65% even for higher clay content ore (Sub 5A and Sub 5B).

A solids loading rate of 0.6 ton/$m^2$/h was sufficient to achieve the target underflow density and samples achieved an overflow clarity lower than 200 ppm based on dynamic thickening testwork.

Tailings Proctor compaction testwork indicated the maximum moisture criteria to achieve compaction of 15.2% (equivalent to a dry-weight-basis moisture of 18%). The target moisture for tailings filtration was therefore deemed to be 15%.

A key outcome from the filtration testwork was that membrane filters can achieve lower moisture content at higher machine throughputs compared to the chamber, or recessed plate, filter press. The 15% moisture target was generally achieved after one minute with membrane filters. Increasing feed pressure and air blowing times generally improved the results.

The tailings material produced during the Period composite testing was sent for filtration and thickening tests. Together with the previous filtration results, the laboratory filtration rates were scaled to industrial sized filter criteria (pressure, mechanical time, filtering area, etc.) to determine the number of filters required in the engineering design. Filter sizing and counts considered the anticipated clay content as a component of mill feed according to the mine plan, and as determined by the resource clay proxy model.

## 13.11 Recovery Estimates

On the basis of the body of testwork that exists, including both the historical testwork, and the testing programs completed by Hudbay since the acquisition of Rosemont in 2014, forecasts of recovery, concentrate grade and quality, as well as characteristics of the resultant tailing product have been developed. The following paragraphs summarize the best estimate of these criteria.

### 13.11.1 Copper (Cu)

The results from the XPS Phase 1 and 2 and BML Replicate and Period testing as well as prior locked cycle tests (LCTs) confirmed there is a strong relationship between copper recovery and the content of oxide copper in the feed, as determined by acid soluble assay. Overall, rougher flotation recoveries of the copper sulfide component of the feed averaged 96.5%, and cleaner flotation recoveries averaged 93% of the copper sulfide component. The results from the various test programs were consolidated and modelled and resulted in the following equation to forecast recovery of copper as a function of total and ASCu in the feed:

Copper Recovery (%) = (1-ASCu:TCu) x 90

Overall copper recovery corresponding to the 2016 mine plan presented in this report is summarized in Table 13-17, expressed as percent of TCu:

**TABLE 13-17: PRODUCTION RECOVERY PROFILE**

| | Production Years | | | | |
|---|---|---|---|---|---|
| | 1−5 | 6−10 | 11−15 | 16−LOM | 1−LOM |
| Head Grade %Tcu | 0.53% | 0.53% | 0.39% | 0.28% | 0.45% |
| Ratio %AsCu | 10.0% | 8.6% | 12.0% | 15.2% | 10.6% |
| Copper Recovery | 81% | 82% | 79% | 71% | 80% |

Note: Based on Mine Plan RP16AUG Mine Plan

### 13.11.2 Molybdenum (Mo)

The ability to fully characterize molybdenum recoveries has been hampered because of limited sample availability on which to conduct testing. Nevertheless, a reasonable effort has been made to forecast molybdenum recoveries and molybdenum concentrate quality.

The limited XPS and BML copper-molybdenum separation testing provides the basis for the viability of producing separate molybdenum concentrate. In the production year composites, the Mo recovery into the Cu/Mo concentrate ranged as follows:

- Period 1 Composites: 83%
- Period 2 Composites: 49%
- Period 3 Composites: 62.8%

The Cu/Mo separation testing into Mo rougher concentrate reported Mo recoveries of 94% with 92% of the Cu reporting to the tails or Cu concentrate. A Mo separation circuit recovery factor of 90% was applied to the period sample bulk recoveries to estimate overall recovery of Molybdenum as follows;

| Production Period | Mo Recovery |
|---|---|
| Period 1 (Yrs. 1-3) | 74.4% |
| Period 2 (Yrs. 4-6) | 43.9% |
| Period 3 (Yrs. 7-LOM) | 51.3% |
| LOM Average | 53.4% |

In accordance with the RP16Aug Mine Plan and corresponding mill production schedule, the calculated Mo recovery figures were adjusted slightly to account for expected additional losses during the initial production ramp-up period.

Mo rougher concentrate grades were low (15%-22%) and additional testing is recommended, especially to improve performance in the presence of talc.

### 13.11.3 SILVER (Ag)

The head samples in XPS and BML testing ranged from 4.5-9.0 grams per ton, averaging approximately 6 grams per ton in the feed ore. The Ag assays in the copper concentrates ranged from 260 to 460 grams per ton averaging 323 grams per ton. There were no assays performed on the tails, the Ag recovery is estimated based on feed and concentrate mass and assays.

The 2016 production schedule developed by Hudbay processes ore with an average Ag content of 0.16 troy ounces/ton or 5.4 g/ton, similar to the test composite samples. Given an estimated Ag grade of the Cu concentrate of 323 g/ton (average test composite assays) the average recovery is:

Average Ag Recovery LOM = 74.4%

#### 13.11.4 GOLD (Au)

Similarly to the silver, gold recovery can be estimated based using ore feed and concentrate grades as there are no tails assays.

The head samples in the XPS and BML testing ranged from 0.03-0.05 grams per ton, averaging approximately 0.04 grams per ton in the ore feed. The Au assays in the copper concentrates ranged from 1.1 to 2.8 g/ton, averaging 2.1 g/ton. Similar to the Ag recovery, the gold recovery estimate is based on feed and concentrate assays;

Average Au Recovery LOM = 65.1%

However, gold had not been systematically assayed in all the drill holes and is therefore not part of the 2016 resource estimate. Nevertheless, 66 drill holes were assayed for gold, representing 17% of all the assaying conducted on the property. A geostatistical analysis was performed on the drill holes with gold results and has shown good similitude between the gold grade assayed in the drill holes and the gold grade assayed in the metallurgical tests heads.

## 13.12 Conclusions and Recommendations

The following paragraphs summarize, on the basis of the preceding narrative describing both the historical metallurgical testing programs, as well as the programs completed by Hudbay in the time since acquiring the Project. These recommendations will serve as the basis for the production phase recovery criteria that will drive inputs into the economic model for the Project.

Principal conclusions:

1) Despite the work of Augusta and the extensive work of Hudbay on matters of lithology and ore type, as well as to associate recoveries by ore type, the overwhelming evidence of the testwork, both past and present is that recovery is driven primarily by the component of soluble copper in the ore sample. When the grade of the sample is discounted by the amount of oxide ore in the sample, recovery of the remaining copper in sulfide minerals is 90%.
2) While lithology appears not to control to any significant degree the recovery of sulfide copper, it does appear to influence molybdenum recovery and more importantly grade of molybdenum concentrate.
3) Contaminants (Fluorine) may affect concentrate quality, but subsequent testwork, completed late in the period of time that Hudbay has been evaluating this project suggests that fluorine

can be controlled in copper concentrate through flotation equipment selection and operational strategies in a way that minimizes concerns with respect to concentrate quality over the life of the mine.

4) Clay in the ore with high concentrations was shown to have a detrimental effect on flotation performance under standard conditions, however, altering conditions (reducing pulp density, dispersing agent addition) was shown to counteract the negative effects. On average, clay content in the ore is expected to remain below concentrations that could result in reduced metal recovery, and when elevated clay contents are encountered, mitigating operational strategies in the process and blending strategies in the mine can be invoked if necessary to offset any negative effects.
5) The tailing properties have been sufficiently characterized as well as the dewatering performance of vendor equipment over the life of the operation to satisfy the estimated number, type and size of tailing filters for this Project. To be conservative, expansion space has been allocated for additional filtering equipment, to the extent that it may be necessary.

## 13.13 Discussion and Recommendations

While the production period composites used in the latter stages of the testwork campaign are no longer an exact match for the operating years they were intended to represent due to recent changes to the mine plan, the test results are nonetheless representative of changes in ore conditions as mining progresses deeper into the deposit.

It is recommended that efforts continue in seeking to utilize the extensive geometallurgical database to identify trends and metallurgical indicators that can inform and optimize production plans.

There is ample information in the database regarding serpentine group minerals, but not specifically for talc. There remains some uncertainty for Molybdenum production with respect to the occasional presence of talc in test samples and its potential to interfere with the production of saleable Mo concentrate. It is recommended to study the occurrence of talc in the deposit to better understand the potential effects on Mo production.

Although it has been concluded that fluorine can be readily rejected from copper concentrate, further study is recommended to develop understanding of ore conditions and indicators that trigger elevated fluorine content in concentrate.

# 14 MINERAL RESOURCE ESTIMATE

Hudbay prepared a 3D block model of the Rosemont deposit using MineSight® version 11.00-3, an industry standard commercial software that specializes in geologic modelling and mine planning. A Lerchs-Grossman ("LG") cone algorithm was applied to the block model to establish the component of the deposit that has a "reasonable prospect of economic extraction". The 3D block model and determination of the mineral resources at the Rosemont deposit were performed by Hudbay personnel following Hudbay procedures. The work was reviewed and approved by Cashel Meagher, P.Geo., Chief Operating Officer for Hudbay, Qualified Person and author of this Technical Report.

## 14.1 Key Assumptions of Model

As shown in Table 14-1, there are 356 drill holes totalling approximately 510,951 feet within the Rosemont database used to support the mineral resource estimation.

**TABLE 14-1: DRILLING DATA BY COMPANY**

| Company | Time Period | Number of Drill holes | Total Length (in feet) |
|---|---|---|---|
| Banner (Anaconda) | 1950 - 1963 | 3 | 4,300 |
| Anaconda | 1963 - 1971 | 113 | 136,838 |
| Anamax | 1973 -1983 | 52 | 54,350 |
| Asarco | 1988 - 1992 | 11 | 14,695 |
| Augusta | 2005 - 2012 | 87 | 132,483 |
| Hudbay | 2014 | 44 | 93,122 |
| Hudbay | 2015 | 46 | 75,164 |
| **Summary** | | **356** | **510,951** |

The drill hole database was provided in Microsoft Excel® format with a cut-off date for mineral resource estimate purposes of January 19th, 2016. The files were imported as collar, downhole survey and assay data into MineSight® Version 11.00-3.

The topographic surface is based on two LIDAR surveys performed in 2006 (10-feet contours) and 2008 (2-feet contours) by Cooper Aerial. Drill hole collars were compared to the topographic surface and only minor differences (98% of < 5 feet) in elevation between drill hole collars and the surveyed topography were found and corrected.

## 14.2 Wireframe Models and Mineralization

The Rosemont deposit trends approximately along an azimuth of N020° with a general dip of 50° to the east. The Backbone Fault forms the footwall contact along the entire length of the Rosemont deposit. Geologically, Rosemont is a skarn deposit. Higher grade mineralization correlates with the Horquilla, Earp (upper and lower) and, Epitaph lithologies and also with the intensity of skarn alteration. The Rosemont deposit is continuous along a strike length of 4,000 feet (1,200 m) in north-south direction, 3,000 feet (900 m) in an east-west direction and to a vertical depth of approximately 2,500 feet (750 m).

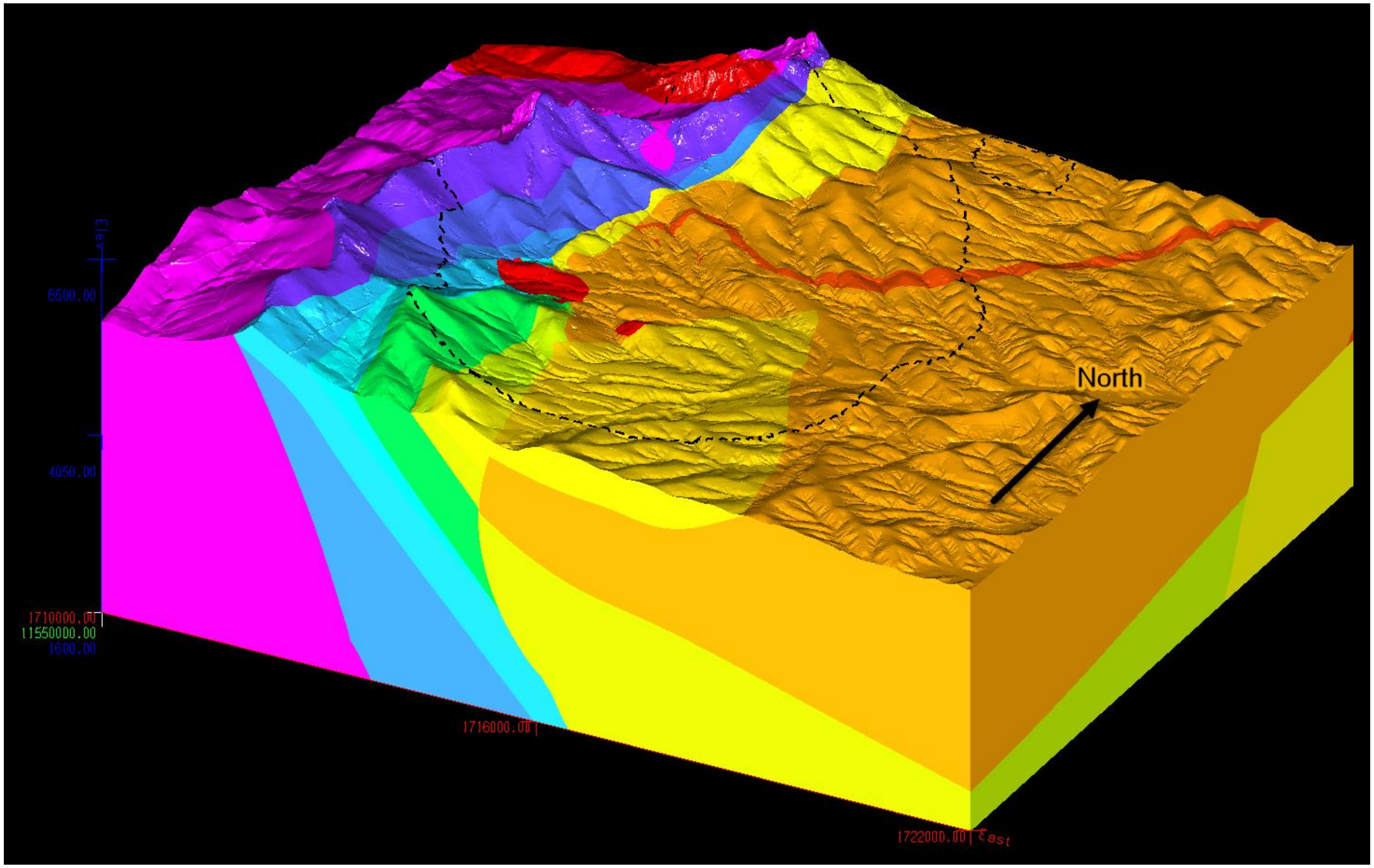
Three sets of structures were recognized, a north-northeast trending set, an east-west trending set and a gently east dipping set. The structures locally offset mineralization but some also appear to control mineralization, especially the oxidation.

Interpretations of the lithology, oxidation state, fault structures and ore types were built with Leapfrog® version 3.01 using the structural information, core angles and geochemical proxies developed by Hudbay. Figure 14-1 presents the Rosemont lithologies. Refer to Table 14-2 for the lithology legend. The details regarding geochemical proxy models developed by Hudbay to model the different lithologies and ore types can be found in section seven of this Technical Report.

Oxidation levels of the deposit have been modelled using the acid soluble copper to total copper ratio, where a ratio of ≥ 50% is defined as oxide, ≥30 and <50% as mixed and <30% as sulfides as shown in Figure 14-2.

As mentioned in section seven of this Technical Report, six different ore types, based on their level of oxidation, swelling and magnesium clays content, are found at Rosemont. Figure 14-3 presents the Rosemont ore types.

**FIGURE 14-1: 3D VIEW OF INTERPRETED LITHOLOGY WIREFRAMES, LOOKING NORTHWEST**

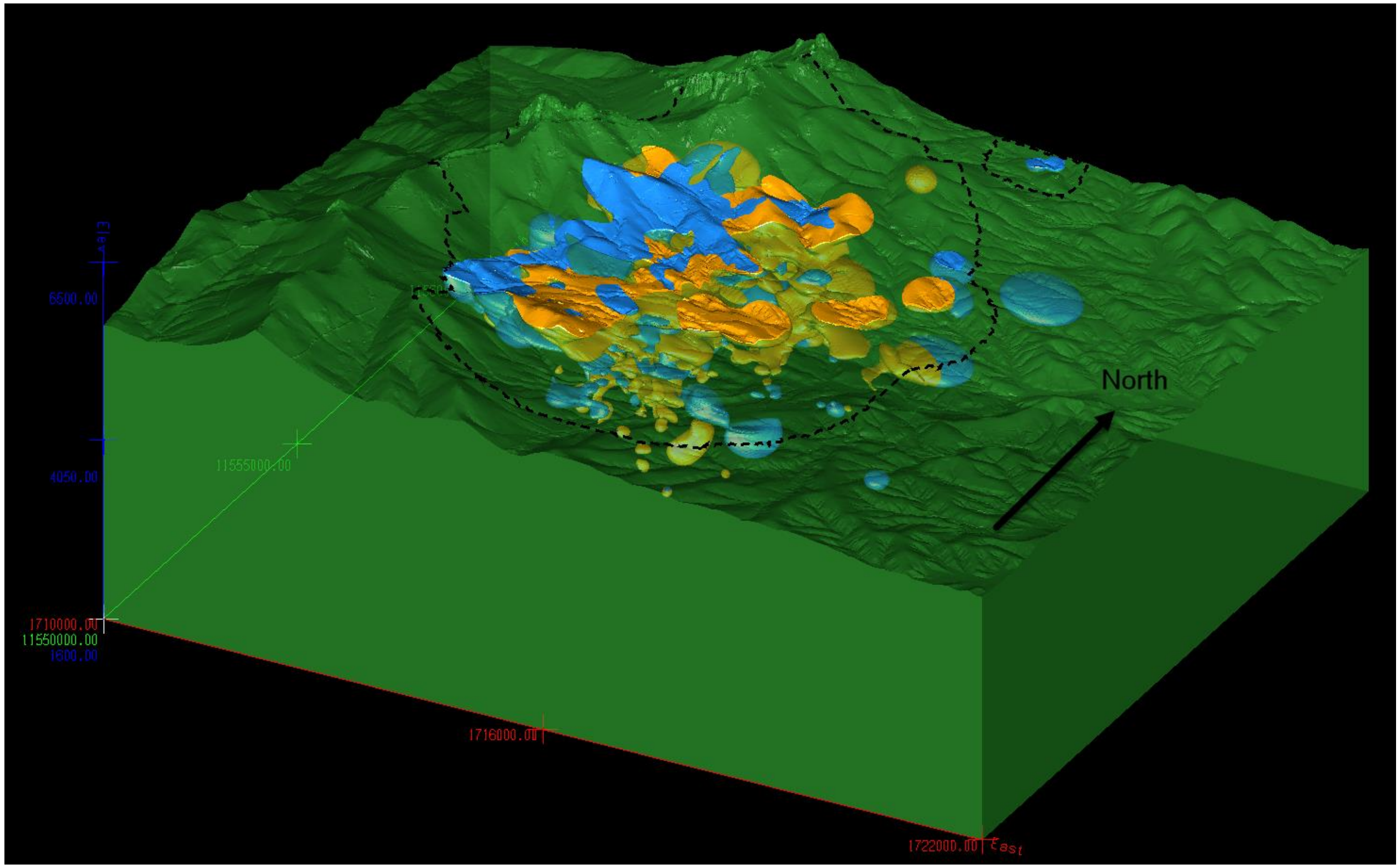



*Note: Lithology colour legend is approximately the same as that shown in Table 14-2. Resource pit is represented by the dashed black line. The resource pit is not indicative of the mine plan.*

**TABLE 14-2: LEGEND OF INTERPRETED WIREFRAMES**

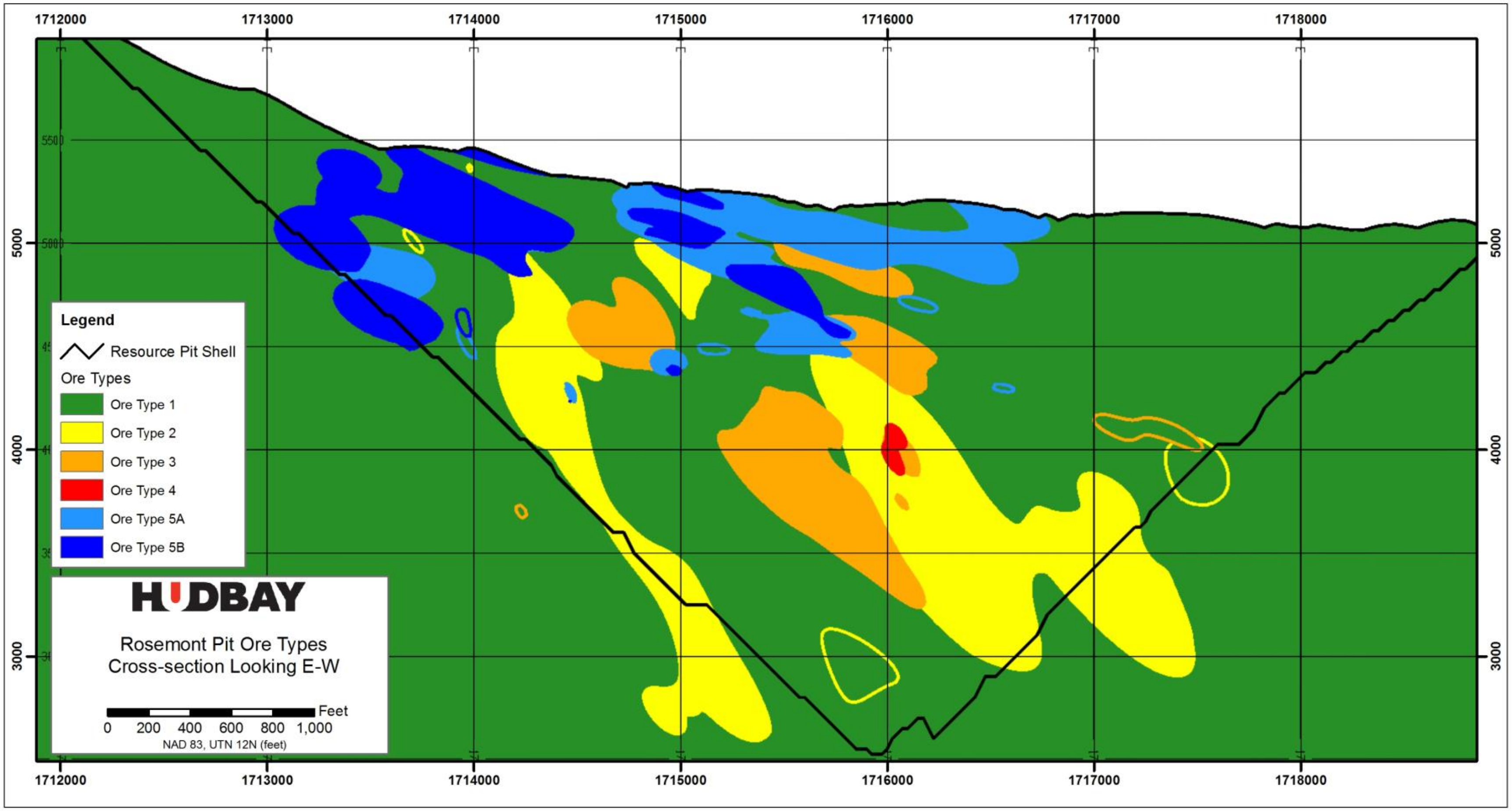
| Lithology | Code | Colour |
|---|---|---|
| Granodiorite | 0 | |
| Bolsa | 1 | |
| Abrigo | 2 | |
| Martin | 3 | |
| Escabrosa | 4 | |
| Horquilla | 5 | |
| Lower Earp | 6 | |
| Upper Earp | 7 | |
| Epitaph | 8 | |
| Scherrer | 9 | |
| Glance | 10 | |
| Gila | 11 | |
| Arkose | 12 | |
| Andesite | 13 | |
| QMP | 14 | |

**FIGURE 14-2: 3D VIEW OF INTERPRETED OXIDATION WIREFRAMES, LOOKING NORTHWEST**




*Note: Oxide in blue, Mixed in orange and Sulfides and un-altered in green. Resource pit is represented by the dashed black line. The resource pit is not indicative of the mine plan.*

**FIGURE 14-3: EW CROSS SECTION OF THE ORE TYPES WIREFRAMES**




*Note: Resource pit is represented by the black line. The resource pit is not indicative of the mine plan.*

## 14.3 Exploratory Data Analysis

Exploratory data analysis ("EDA") comprised basic statistical evaluation of the assays and composites for total copper, acid-soluble copper, molybdenum, silver and sample length.

## 14.4 Assays

The Table 14-3 presents the number of samples collected and total length anaylzed by element.

**TABLE 14-3: SAMPLES AND LENGTH ANALYZED**

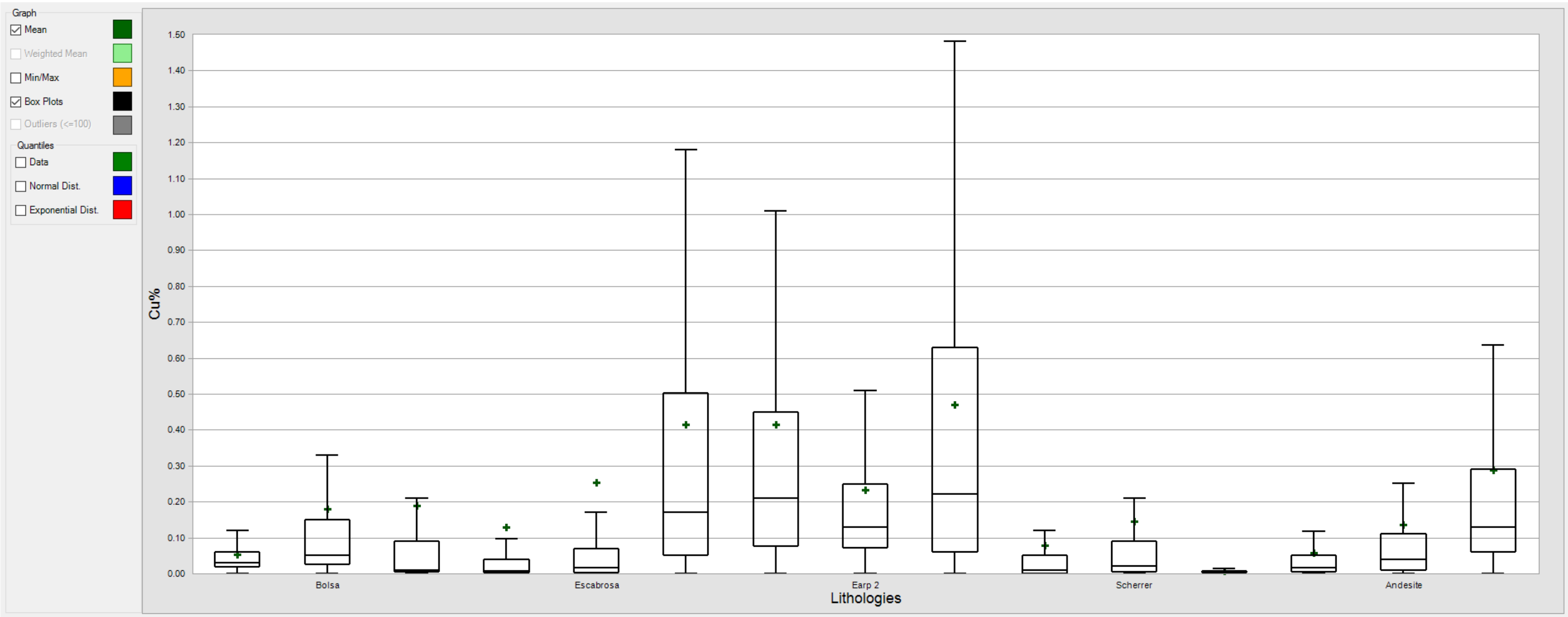
| | Number of Samples | Total Length in Feet |
|---|---|---|
| Cu% | 87,236 | 430,986 |
| SCu% | 56,497 | 283,055 |
| Mo% | 78,113 | 385,474 |
| Ag opt | 65,709 | 322,750 |
| SG g/cm3 | 2,066 | 10,380 |

### 14.4.1 Box Plots

Box plots of the basic statistics for TCu, ASCu, molybdenum (Mo) and silver (Ag), for each lithology within the sulfide portion of the deposit are displayed in Figure 14-4 to Figure 14-7.

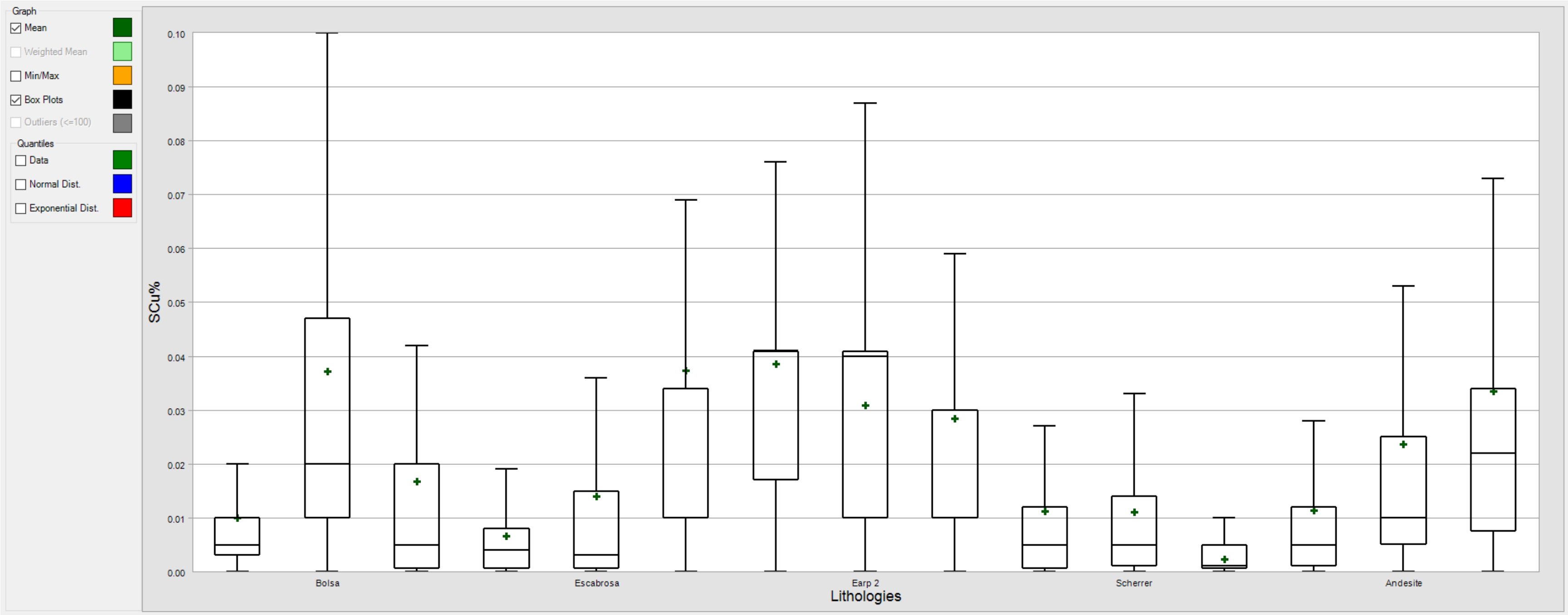
These box plots confirm that most of the mineralization of economic interest in sulfides will occur in the four main units of the lower plate group with some high copper grade also occurring in the Andesite, Arkose and QMP units. The minor units exhibit higher skewness in the copper sample statistics. Molybdenum and silver statistics display a high skewness in all the lithological units.

## FIGURE 14-4: BOX PLOTS OF TOTAL COPPER ASSAYS IN SULFIDES



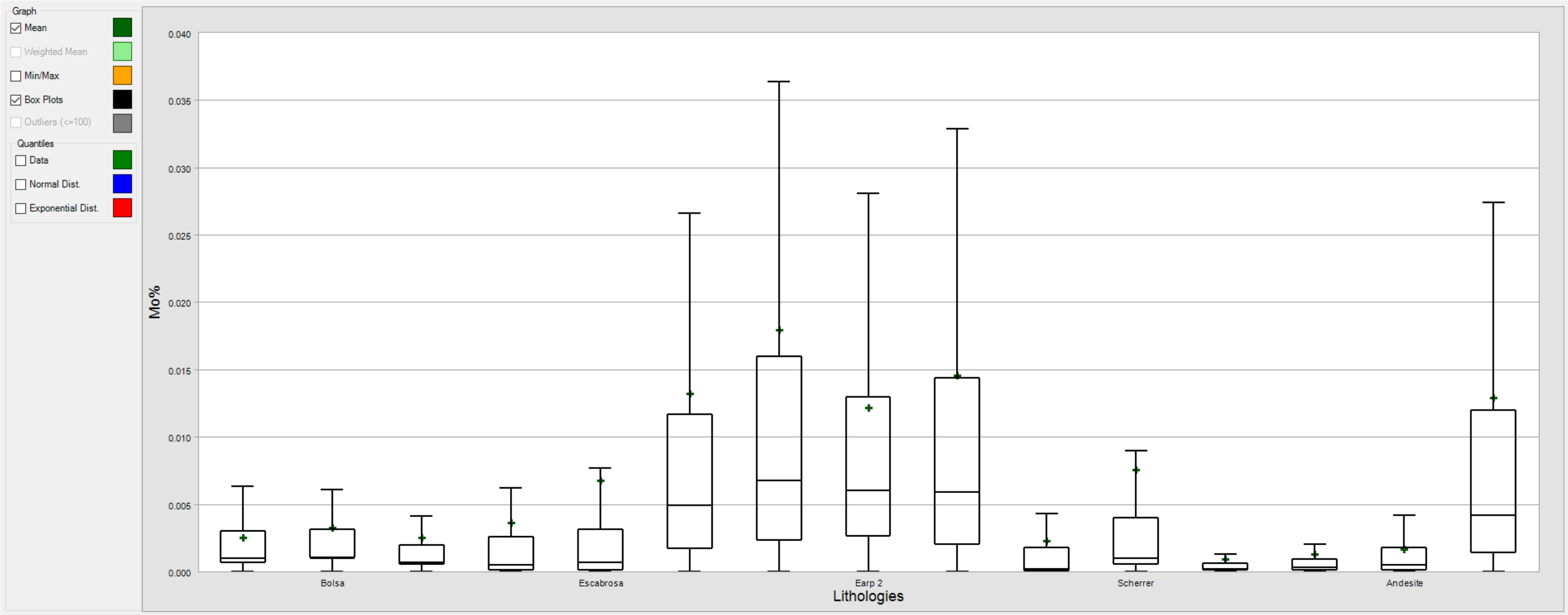

| | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp | Earp 2 | Epitaph | Glance | Scherrer | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Num Missing Samples | 49 | 24 | 38 | 71 | 87 | 123 | 37 | 30 | 175 | 154 | 121 | 144 | 1029 | 66 | 35 |
| Min | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00010 |
| Max | 0.67000 | 6.40000 | 5.25000 | 16.52000 | 24.70000 | 19.80600 | 10.51000 | 4.38000 | 24.51800 | 10.32300 | 5.26000 | 0.19000 | 9.19000 | 3.89000 | 15.88000 |
| Mean | 0.052765 | 0.180734 | 0.188366 | 0.129353 | 0.254158 | 0.415202 | 0.414981 | 0.233155 | 0.469877 | 0.079184 | 0.145023 | 0.007003 | 0.056520 | 0.135535 | 0.287213 |
| First quartile | 0.017710 | 0.025116 | 0.004508 | 0.001610 | 0.002898 | 0.050232 | 0.076314 | 0.070518 | 0.060214 | 0.000966 | 0.005152 | 0.002576 | 0.005152 | 0.010304 | 0.060214 |
| Median | 0.028819 | 0.050071 | 0.010143 | 0.006279 | 0.016261 | 0.170499 | 0.210105 | 0.129927 | 0.220087 | 0.009821 | 0.020125 | 0.004991 | 0.015617 | 0.040089 | 0.129283 |
| Third quartile | 0.058926 | 0.149730 | 0.089838 | 0.039928 | 0.069874 | 0.502320 | 0.449834 | 0.249872 | 0.629832 | 0.049910 | 0.088872 | 0.007084 | 0.049910 | 0.109802 | 0.291088 |
| SD | 0.070146 | 0.452218 | 0.553511 | 0.631118 | 1.385043 | 0.719605 | 0.654518 | 0.339519 | 0.705830 | 0.367530 | 0.396337 | 0.011470 | 0.191218 | 0.312070 | 0.614377 |
| Variance | 0.004920 | 0.204501 | 0.306374 | 0.398310 | 1.918344 | 0.517832 | 0.428393 | 0.115273 | 0.498196 | 0.135079 | 0.157083 | 0.000132 | 0.036564 | 0.097387 | 0.377460 |
| CV | 1.329406 | 2.502127 | 2.938492 | 4.879040 | 5.449535 | 1.733146 | 1.577223 | 1.456191 | 1.502160 | 4.641479 | 2.732923 | 1.637921 | 3.383179 | 2.302501 | 2.139098 |
| Skewness | 3.742939 | 7.845488 | 5.640790 | 15.542020 | 10.773900 | 6.747593 | 4.245255 | 4.710971 | 6.678458 | 17.376600 | 5.667892 | 8.765837 | 18.687200 | 5.334545 | 13.564440 |
| Kurtosis | 19.962670 | 83.196710 | 40.174550 | 336.243400 | 135.853500 | 95.419230 | 29.526270 | 31.362740 | 133.228100 | 401.450900 | 44.070930 | 112.497300 | 638.707600 | 36.843980 | 302.288300 |
| Min non-outlier | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00010 |
| Max non-outlier | 0.12000 | 0.33000 | 0.21000 | 0.09650 | 0.17010 | 1.18000 | 1.01000 | 0.51000 | 1.48200 | 0.12000 | 0.21000 | 0.01370 | 0.11660 | 0.25040 | 0.63660 |
| Num mild outliers | 31 | 29 | 17 | 66 | 125 | 856 | 208 | 254 | 639 | 98 | 60 | 113 | 503 | 149 | 83 |
| Num extreme outliers | 29 | 50 | 76 | 239 | 360 | 688 | 304 | 256 | 283 | 158 | 144 | 64 | 517 | 232 | 95 |

## FIGURE 14-5: BOX PLOTS OF ACID SOLUBLE COPPER IN SULFIDES



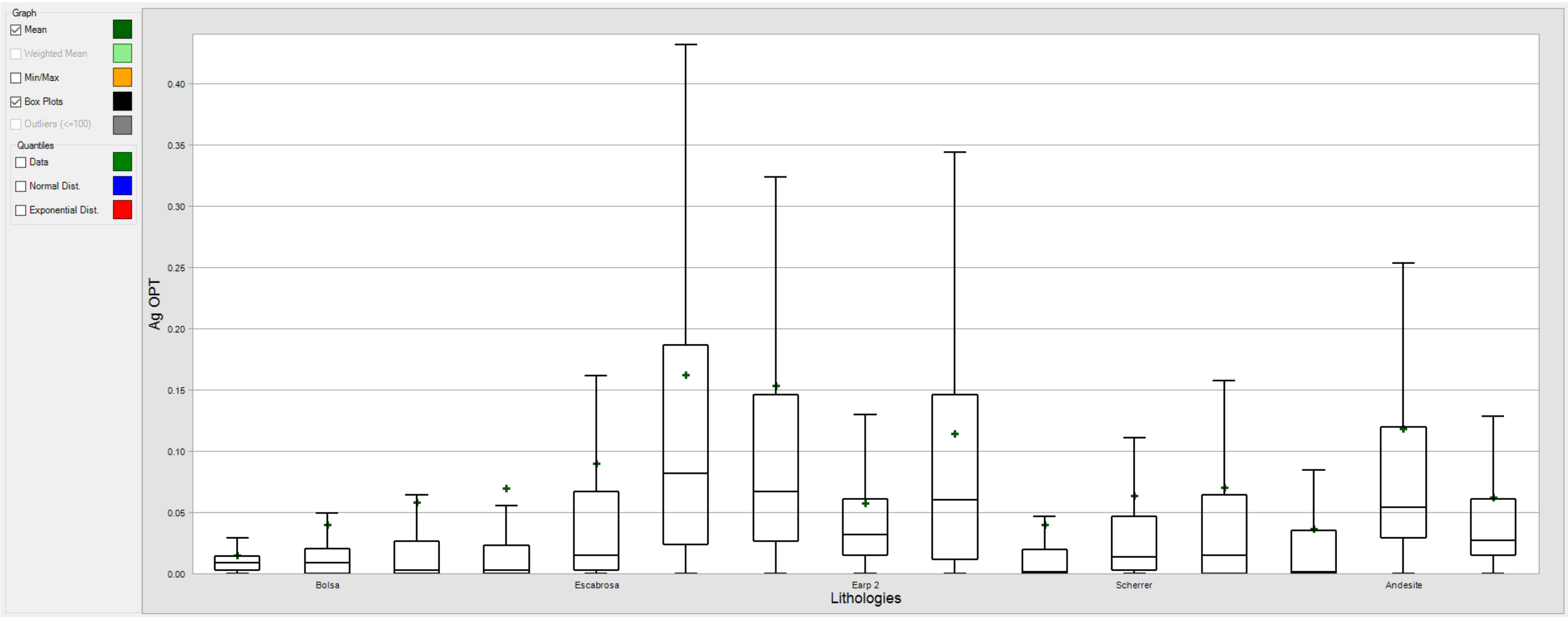

| | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp | Earp 2 | Epitaph | Glance | Scherrer | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Num Samples | 594 | 629 | 573 | 2014 | 3106 | 18385 | 5228 | 5249 | 13670 | 1967 | 1419 | 1639 | 9812 | 2981 | 1461 |
| Num Missing Samples | 49 | 24 | 38 | 71 | 87 | 123 | 37 | 30 | 175 | 154 | 121 | 144 | 1029 | 66 | 35 |
| Min | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max | 0.19200 | 1.13000 | 0.57000 | 0.57200 | 1.22000 | 4.69000 | 0.96000 | 1.93000 | 2.18000 | 0.93000 | 0.65000 | 0.05000 | 1.84700 | 3.44000 | 0.91800 |
| Mean | 0.010029 | 0.037245 | 0.016733 | 0.006610 | 0.013929 | 0.037255 | 0.038569 | 0.030799 | 0.028414 | 0.011251 | 0.010988 | 0.002286 | 0.011352 | 0.023599 | 0.033548 |
| First quartile | 0.003041 | 0.010028 | 0.000575 | 0.000575 | 0.000575 | 0.010028 | 0.017015 | 0.010028 | 0.010028 | 0.000575 | 0.001069 | 0.000575 | 0.001069 | 0.005014 | 0.007562 |
| Median | 0.004973 | 0.020016 | 0.004973 | 0.003987 | 0.003000 | 0.033990 | 0.040977 | 0.039990 | 0.029962 | 0.004973 | 0.004973 | 0.001028 | 0.004973 | 0.009987 | 0.021989 |
| Third quartile | 0.009946 | 0.046936 | 0.019975 | 0.007973 | 0.014960 | 0.033949 | 0.040936 | 0.040936 | 0.029921 | 0.011919 | 0.013974 | 0.004932 | 0.011919 | 0.024989 | 0.033949 |
| SD | 0.016248 | 0.083229 | 0.052319 | 0.022176 | 0.053517 | 0.079113 | 0.044192 | 0.041849 | 0.042247 | 0.040873 | 0.029922 | 0.002476 | 0.045050 | 0.083487 | 0.055888 |
| Variance | 0.000264 | 0.006927 | 0.002737 | 0.000492 | 0.002864 | 0.006259 | 0.001953 | 0.001751 | 0.001785 | 0.001671 | 0.000895 | 0.000006 | 0.002029 | 0.006970 | 0.003123 |
| CV | 1.620057 | 2.234638 | 3.126792 | 3.355149 | 3.841993 | 2.123527 | 1.145791 | 1.358788 | 1.486866 | 3.632983 | 2.723090 | 1.083093 | 3.968490 | 3.537761 | 1.665902 |
| Skewness | 5.178357 | 8.312076 | 7.897959 | 15.214790 | 12.034040 | 20.246620 | 6.942344 | 24.714650 | 18.306010 | 13.749080 | 12.424770 | 5.246796 | 22.884600 | 25.484810 | 7.307144 |
| Kurtosis | 38.496270 | 88.194260 | 70.299090 | 309.627400 | 188.594200 | 836.339200 | 82.567640 | 980.204400 | 661.506600 | 242.175900 | 203.766000 | 85.204660 | 689.621400 | 957.641400 | 81.829350 |
| Min non-outlier | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max non-outlier | 0.02000 | 0.10000 | 0.04200 | 0.01900 | 0.03600 | 0.06900 | 0.07600 | 0.08700 | 0.05900 | 0.02700 | 0.03300 | 0.01000 | 0.02800 | 0.05300 | 0.07300 |
| Num mild outliers | 24 | 11 | 7 | 42 | 74 | 853 | 192 | 57 | 417 | 69 | 27 | 4 | 305 | 59 | 64 |
| Num extreme outliers | 29 | 23 | 21 | 60 | 117 | 914 | 170 | 48 | 417 | 69 | 34 | 1 | 290 | 104 | 63 |

## FIGURE 14-6: BOX PLOTS OF MOLYBDENUM ASSAYS IN SULFIDES




| | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp | Earp 2 | Epitaph | Glance | Scherrer | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Num Samples | 584 | 612 | 566 | 1906 | 3072 | 18177 | 5158 | 5146 | 13196 | 1628 | 1268 | 1185 | 6985 | 2301 | 1444 |
| Num Missing Samples | 59 | 41 | 45 | 179 | 121 | 331 | 107 | 133 | 649 | 493 | 272 | 598 | 3856 | 746 | 52 |
| Min | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max | 0.04400 | 0.23900 | 0.15500 | 0.31700 | 0.76200 | 1.95000 | 2.66000 | 1.08380 | 5.96600 | 0.13360 | 1.09650 | 0.02180 | 0.19490 | 0.08200 | 1.73200 |
| Mean | 0.002511 | 0.003269 | 0.002508 | 0.003632 | 0.006748 | 0.013187 | 0.017966 | 0.012142 | 0.014559 | 0.002259 | 0.007586 | 0.000903 | 0.001297 | 0.001640 | 0.012878 |
| First quartile | 0.000656 | 0.001014 | 0.000537 | 0.000119 | 0.000119 | 0.001730 | 0.002327 | 0.002625 | 0.002028 | 0.000060 | 0.000537 | 0.000119 | 0.000119 | 0.000119 | 0.001432 |
| Median | 0.000984 | 0.000984 | 0.000686 | 0.000507 | 0.000686 | 0.004922 | 0.006771 | 0.005996 | 0.005877 | 0.000209 | 0.000984 | 0.000209 | 0.000328 | 0.000507 | 0.004206 |
| Third quartile | 0.002983 | 0.003162 | 0.001969 | 0.002565 | 0.003162 | 0.011693 | 0.015989 | 0.012946 | 0.014378 | 0.001790 | 0.003997 | 0.000597 | 0.000895 | 0.001790 | 0.011992 |
| SD | 0.003804 | 0.011207 | 0.008890 | 0.012736 | 0.031675 | 0.036238 | 0.069613 | 0.031655 | 0.069557 | 0.007116 | 0.039329 | 0.001895 | 0.004279 | 0.004138 | 0.051456 |
| Variance | 0.000014 | 0.000126 | 0.000079 | 0.000162 | 0.001003 | 0.001313 | 0.004846 | 0.001002 | 0.004838 | 0.000051 | 0.001547 | 0.000004 | 0.000018 | 0.000017 | 0.002648 |
| CV | 1.515020 | 3.428774 | 3.545238 | 3.506237 | 4.694022 | 2.748007 | 3.874685 | 2.607070 | 4.777594 | 3.150692 | 5.184125 | 2.098576 | 3.298666 | 2.523170 | 3.995736 |
| Skewness | 4.815528 | 16.558600 | 12.486510 | 11.809960 | 12.857600 | 20.557450 | 21.399950 | 18.734200 | 56.850020 | 9.340682 | 19.252380 | 4.973633 | 18.866280 | 10.538100 | 26.550310 |
| Kurtosis | 35.921980 | 328.976600 | 190.057400 | 220.429200 | 225.112800 | 829.044900 | 634.711400 | 493.691400 | 4,382.621000 | 120.770600 | 486.343400 | 35.155190 | 663.748100 | 157.244200 | 866.909400 |
| Min non-outlier | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max non-outlier | 0.00630 | 0.00610 | 0.00410 | 0.00620 | 0.00770 | 0.02660 | 0.03640 | 0.02810 | 0.03290 | 0.00430 | 0.00900 | 0.00130 | 0.00200 | 0.00420 | 0.02740 |
| Num mild outliers | 30 | 14 | 17 | 77 | 111 | 945 | 199 | 238 | 579 | 149 | 37 | 29 | 248 | 165 | 76 |
| Num extreme outliers | 25 | 27 | 40 | 142 | 262 | 1219 | 271 | 198 | 549 | 102 | 102 | 170 | 774 | 91 | 69 |

## FIGURE 14-7: BOX PLOTS OF SILVER ASSAYS IN SULFIDES




| | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp | Earp 2 | Epitaph | Glance | Scherrer | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Num Samples | 643 | 653 | 611 | 2085 | 3193 | 18508 | 5265 | 5279 | 13845 | 2121 | 1540 | 1783 | 10841 | 3047 | 1496 |
| Num Missing Samples | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Min | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max | 0.49580 | 2.18750 | 2.21380 | 9.80000 | 9.50000 | 6.70000 | 4.69000 | 2.15830 | 7.10000 | 5.83330 | 3.07160 | 2.64830 | 5.83630 | 2.74410 | 2.40000 |
| Mean | 0.014595 | 0.039873 | 0.057873 | 0.069736 | 0.089804 | 0.162475 | 0.153229 | 0.057671 | 0.114305 | 0.040165 | 0.063374 | 0.070448 | 0.036482 | 0.118195 | 0.062481 |
| First quartile | 0.002940 | 0.000098 | 0.000098 | 0.000098 | 0.002940 | 0.023324 | 0.026362 | 0.014602 | 0.011760 | 0.000098 | 0.002940 | 0.000098 | 0.000098 | 0.029204 | 0.014602 |
| Median | 0.008673 | 0.008673 | 0.002891 | 0.002891 | 0.014553 | 0.081683 | 0.067081 | 0.032095 | 0.060025 | 0.001519 | 0.013475 | 0.014553 | 0.001519 | 0.053753 | 0.026705 |
| Third quartile | 0.014504 | 0.020384 | 0.026264 | 0.023226 | 0.067032 | 0.186690 | 0.145726 | 0.061152 | 0.145726 | 0.019502 | 0.046648 | 0.064190 | 0.034986 | 0.119560 | 0.061152 |
| SD | 0.031589 | 0.146062 | 0.203742 | 0.421201 | 0.310570 | 0.283280 | 0.282110 | 0.097835 | 0.195600 | 0.204369 | 0.178869 | 0.136054 | 0.158686 | 0.217277 | 0.125406 |
| Variance | 0.000998 | 0.021334 | 0.041511 | 0.177411 | 0.096453 | 0.080248 | 0.079586 | 0.009572 | 0.038259 | 0.041767 | 0.031994 | 0.018511 | 0.025181 | 0.047209 | 0.015727 |
| CV | 2.164352 | 3.663170 | 3.520509 | 6.039968 | 3.458301 | 1.743526 | 1.841099 | 1.696429 | 1.711220 | 5.088203 | 2.822447 | 1.931281 | 4.349728 | 1.838285 | 2.007111 |
| Skewness | 8.748090 | 10.311040 | 6.859352 | 15.843350 | 13.962420 | 6.271017 | 4.798012 | 7.315504 | 10.517880 | 17.030830 | 8.231426 | 5.363384 | 25.684920 | 5.158853 | 8.570574 |
| Kurtosis | 107.632700 | 130.762600 | 55.002810 | 302.025500 | 316.708600 | 71.574230 | 34.381090 | 91.052380 | 242.849400 | 387.290700 | 97.686830 | 74.158210 | 870.394800 | 35.277160 | 118.244600 |
| Min non-outlier | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 | 0.00000 |
| Max non-outlier | 0.02920 | 0.04960 | 0.06420 | 0.05540 | 0.16180 | 0.43170 | 0.32380 | 0.13000 | 0.34420 | 0.04670 | 0.11080 | 0.15750 | 0.08460 | 0.25370 | 0.12830 |
| Num mild outliers | 32 | 15 | 25 | 81 | 178 | 704 | 236 | 236 | 627 | 74 | 64 | 91 | 522 | 147 | 81 |
| Num extreme outliers | 30 | 81 | 70 | 278 | 230 | 729 | 365 | 238 | 301 | 236 | 131 | 195 | 596 | 174 | 97 |

### 14.4.2 Grade Capping

Since most of the lithological units show a high skewness in the statistical distribution of the metal grade, length weighted, log-scaled probability plots and deciles analysis of the assays were used to define grade outliers for TCu, ASCu, molybdenum (Mo) and silver (Ag) within each of the separately evaluated domains. The capping thresholds are shown below in Table 14-4.

**TABLE 14-4: CAPPING THRESOLDS BY LITHOLOGY**

| LITHO | OXIDE | Copper (%) | Soluble Copper (%) | Molybdenum (%) | Silver (opt) |
|---|---|---|---|---|---|
| Granodiorite | Sulfides | 0.50 | 0.08 | 0.008 | 0.148 |
| Bolsa | | 1.75 | 0.43 | 0.030 | 0.480 |
| Abrigo | | 1.40 | 0.18 | 0.010 | 0.434 |
| Martin | | 1.60 | 0.46 | 0.050 | 0.435 |
| Escabrosa | | 5.40 | 0.24 | 0.060 | 0.625 |
| Horquilla | | 3.50 | 0.33 | 0.200 | 1.705 |
| Earp (lower & upper) | | 5.00 | 0.40 | 0.290 | 1.130 |
| Epitaph | | 4.00 | 0.30 | 0.160 | 0.769 |
| Scherrer | | 1.75 | 0.19 | 0.070 | 0.573 |
| Glance | | 1.00 | 0.23 | 0.027 | 0.588 |
| Gila | | 0.04 | n/a | n/a | 0.720 |
| Arkose | | 1.35 | 0.33 | 0.015 | 0.563 |
| Andesite | | 1.35 | 0.38 | 0.014 | 1.125 |
| QMP | | 1.60 | 0.25 | 0.110 | 0.486 |
| Granodiorite | Mixed | n/a | n/a | n/a | n/a |
| Bolsa | | 1.20 | 0.47 | 0.008 | 0.260 |
| Abrigo | | 1.60 | 0.23 | 0.043 | 0.790 |
| Martin | | 2.60 | 0.30 | 0.060 | 0.373 |
| Escabrosa | | 0.60 | 0.39 | 0.100 | 0.320 |
| Horquilla | | 1.80 | 0.89 | 0.076 | 0.850 |
| Earp (lower & upper) | | 1.95 | 0.85 | 0.060 | 0.500 |
| Epitaph | | 2.00 | 0.65 | 0.060 | 0.297 |
| Scherrer | | n/a | n/a | n/a | n/a |
| Glance | | 0.60 | 0.44 | n/a | 0.315 |
| Gila | | n/a | n/a | n/a | n/a |
| Arkose | | 0.40 | 0.29 | 0.019 | 0.500 |
| Andesite | | 1.10 | 0.73 | n/a | 0.200 |
| QMP | | 0.70 | 0.50 | 0.098 | 0.200 |
| Granodiorite | Oxide | 0.65 | n/a | n/a | n/a |
| Bolsa | | 1.35 | 0.81 | 0.025 | 0.552 |
| Abrigo | | 1.40 | 1.15 | 0.017 | 0.405 |
| Martin | | 0.60 | 0.47 | 0.017 | 0.170 |
| Escabrosa | | 1.30 | 1.00 | 0.080 | 0.300 |
| Horquilla | | 3.00 | 2.19 | 0.044 | 0.700 |
| Earp (lower & upper) | | 2.20 | 1.58 | 0.050 | 0.440 |
| Epitaph | | 1.60 | 1.16 | 0.028 | 0.200 |
| Scherrer | | 0.60 | 1.34 | 0.013 | 0.265 |
| Glance | | 0.70 | 0.60 | 0.028 | 0.400 |
| Gila | | n/a | n/a | 0.025 | 0.150 |
| Arkose | | 1.40 | 1.34 | 0.010 | 0.900 |
| Andesite | | 1.30 | 1.12 | 0.020 | 1.090 |
| QMP | | 1.20 | 0.95 | 0.040 | 0.390 |

### 14.4.3 Assay Statistics

Exploratory data analysis of assay statistics are summarized by uncapped and capped grades in Table 14-5, Table 14-6, Table 14-7 and Table 14-8. Since the earlier drill programs were mostly focused on copper, not all samples were analyzed for Mo and Ag and the exploratory data analysis shows fewer assays for these metals. There are a total of 9,174 samples with missing molybdenum assays and 21,578 missing silver assays. Capping was completed on the assays prior to compositing.

**TABLE 14-5: ASSAY STATISTICS FOR TOTAL COPPER BY LITHOLOGY IN SULFIDES**

| Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 581 | 622 | 552 | 1,913 | 2,987 | 18,293 | 5,217 | 5,238 | 13,483 | 1,807 | 1,345 | 1,525 | 9,461 | 2,934 | 1,461 |
| Missing Values | 62 | 31 | 59 | 172 | 206 | 215 | 48 | 41 | 362 | 314 | 195 | 258 | 1380 | 113 | 35 |
| Minimum | 0.0022 | 0.002 | 0.0002 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0008 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0001 | 0.0001 | 0.0001 |
| Maximum | 0.67 | 6.4 | 5.25 | 16.52 | 24.7 | 19.806 | 10.51 | 4.38 | 24.518 | 10.323 | 5.26 | 0.19 | 9.19 | 3.89 | 15.88 |
| Mean | 0.053946 | 0.18277 | 0.19553 | 0.13618 | 0.264284 | 0.41729 | 0.415856 | 0.233645 | 0.47639 | 0.086195 | 0.153 | 0.00753 | 0.05862 | 0.137706 | 0.28721 |
| 1st Quartile | 0.018354 | 0.02608 | 0.00515 | 0.00225 | 0.003864 | 0.050232 | 0.078568 | 0.071162 | 0.06021 | 0.00161 | 0.00644 | 0.0029 | 0.00515 | 0.012236 | 0.06021 |
| Median | 0.030107 | 0.05007 | 0.01014 | 0.00918 | 0.019481 | 0.175973 | 0.210105 | 0.129927 | 0.23007 | 0.010143 | 0.02013 | 0.00499 | 0.01884 | 0.040089 | 0.12928 |
| 3rd Quartile | 0.059892 | 0.15971 | 0.09982 | 0.03993 | 0.079856 | 0.509726 | 0.449834 | 0.249872 | 0.63981 | 0.058282 | 0.09692 | 0.00773 | 0.04991 | 0.109802 | 0.29109 |
| Std. Devn. | 0.070477 | 0.45435 | 0.56271 | 0.64685 | 1.411424 | 0.720808 | 0.65493 | 0.339707 | 0.70852 | 0.382676 | 0.4056 | 0.01172 | 0.19442 | 0.314084 | 0.61438 |
| Variance | 0.004967 | 0.20643 | 0.31665 | 0.41842 | 1.992119 | 0.519565 | 0.428933 | 0.115401 | 0.502 | 0.146441 | 0.16451 | 0.00014 | 0.0378 | 0.098649 | 0.37746 |
| Co. of Variation | 1.306439 | 2.48595 | 2.87787 | 4.74991 | 5.340569 | 1.727356 | 1.574895 | 1.453944 | 1.48726 | 4.439645 | 2.65094 | 1.55779 | 3.31673 | 2.280828 | 2.1391 |

| Capped Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 581 | 622 | 552 | 1,913 | 2,987 | 18,293 | 5,217 | 5,238 | 13,483 | 1,807 | 1,345 | 1,525 | 9,461 | 2,934 | 1,461 |
| Missing Values | 62 | 31 | 59 | 172 | 206 | 215 | 48 | 41 | 362 | 314 | 195 | 258 | 1380 | 113 | 35 |
| Minimum | 0.0022 | 0.002 | 0.0002 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0008 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0001 | 0.0001 | 0.0001 |
| Maximum | 0.5 | 1.75 | 1.4 | 1.6 | 5.4 | 3.5 | 5 | 4.38 | 4 | 1.75 | 1 | 0.04 | 1.35 | 1.35 | 1.6 |
| Mean | 0.053567 | 0.16253 | 0.15453 | 0.10306 | 0.193366 | 0.403324 | 0.412726 | 0.233645 | 0.46789 | 0.07415 | 0.12164 | 0.00702 | 0.05494 | 0.126311 | 0.25896 |
| 1st Quartile | 0.018306 | 0.02592 | 0.00502 | 0.00221 | 0.003726 | 0.050004 | 0.078408 | 0.07101 | 0.06005 | 0.00135 | 0.00643 | 0.00275 | 0.00502 | 0.012204 | 0.06005 |
| Median | 0.029997 | 0.04998 | 0.01002 | 0.00921 | 0.019521 | 0.175905 | 0.209979 | 0.130005 | 0.23001 | 0.010017 | 0.02001 | 0.005 | 0.01882 | 0.039987 | 0.1292 |
| 3rd Quartile | 0.059994 | 0.15995 | 0.09995 | 0.03996 | 0.079974 | 0.509976 | 0.449982 | 0.249966 | 0.63995 | 0.058482 | 0.09698 | 0.00788 | 0.04995 | 0.109998 | 0.29138 |
| Std. Devn. | 0.067634 | 0.28746 | 0.33382 | 0.28824 | 0.695424 | 0.590347 | 0.624917 | 0.339707 | 0.61632 | 0.198746 | 0.23963 | 0.00722 | 0.13142 | 0.23972 | 0.3435 |
| Variance | 0.004574 | 0.08263 | 0.11144 | 0.08309 | 0.483615 | 0.34851 | 0.390521 | 0.115401 | 0.37985 | 0.0395 | 0.05742 | 5.2E-05 | 0.01727 | 0.057466 | 0.11799 |
| Co. of Variation | 1.262598 | 1.76867 | 2.16024 | 2.79686 | 3.596423 | 1.463706 | 1.514119 | 1.453944 | 1.31722 | 2.680305 | 1.96999 | 1.02805 | 2.39209 | 1.897865 | 1.32647 |

**TABLE 14-6: ASSAY STATISTICS FOR ACID SOLUBLE COPPER BY LITHOLOGY IN SULFIDES**

| Soluble Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 581 | 621 | 552 | 1,911 | 2,968 | 18,057 | 5,197 | 5,182 | 13,359 | 1,754 | 1,340 | 1,490 | 9,200 | 2,844 | 1,455 |
| Missing Values | 62 | 32 | 59 | 174 | 225 | 451 | 68 | 97 | 486 | 367 | 200 | 293 | 1641 | 203 | 41 |
| Minimum | 0.0005 | 0.001 | 0.0002 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0005 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0001 | 0.0001 | 0.0001 |
| Maximum | 0.192 | 1.13 | 0.57 | 0.572 | 1.22 | 4.69 | 0.96 | 1.93 | 2.18 | 0.93 | 0.65 | 0.05 | 1.847 | 3.44 | 0.918 |
| Mean | 0.010254 | 0.03773 | 0.01737 | 0.00697 | 0.014577 | 0.037932 | 0.038799 | 0.031197 | 0.02908 | 0.012617 | 0.01164 | 0.00252 | 0.01211 | 0.024736 | 0.03369 |
| 1st Quartile | 0.003041 | 0.01003 | 0.00058 | 0.00058 | 0.000575 | 0.010028 | 0.018002 | 0.010028 | 0.01003 | 0.000575 | 0.00206 | 0.00058 | 0.00206 | 0.005014 | 0.00806 |
| Median | 0.004973 | 0.02002 | 0.00497 | 0.00497 | 0.003987 | 0.03399 | 0.040977 | 0.03999 | 0.02996 | 0.004973 | 0.00497 | 0.00201 | 0.00497 | 0.009987 | 0.02199 |
| 3rd Quartile | 0.009946 | 0.04694 | 0.01998 | 0.00797 | 0.01496 | 0.033949 | 0.040936 | 0.040936 | 0.02992 | 0.011919 | 0.01397 | 0.00493 | 0.01192 | 0.024989 | 0.03395 |
| Std. Devn. | 0.016359 | 0.08366 | 0.0532 | 0.02271 | 0.054661 | 0.079667 | 0.044223 | 0.041971 | 0.04251 | 0.043086 | 0.03067 | 0.00248 | 0.04643 | 0.08531 | 0.05596 |
| Variance | 0.000268 | 0.007 | 0.00283 | 0.00052 | 0.002988 | 0.006347 | 0.001956 | 0.001762 | 0.00181 | 0.001856 | 0.00094 | 6E-06 | 0.00216 | 0.007278 | 0.00313 |
| Co. of Variation | 1.595405 | 2.21754 | 3.06308 | 3.26043 | 3.74978 | 2.100257 | 1.139791 | 1.345354 | 1.4621 | 3.414925 | 2.63571 | 0.98769 | 3.83462 | 3.44888 | 1.66124 |

| Capped Soluble Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 581 | 621 | 552 | 1,911 | 2,968 | 18,057 | 5,197 | 5,182 | 13,359 | 1,754 | 1,340 | 1,490 | 9,200 | 2,844 | 1,455 |
| Missing Values | 62 | 32 | 59 | 174 | 225 | 451 | 68 | 97 | 486 | 367 | 200 | 293 | 1641 | 203 | 41 |
| Minimum | 0.0005 | 0.001 | 0.0002 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0005 | 0.0001 | 0.0001 | 0.0002 | 0.0005 | 0.0001 | 0.0001 | 0.0001 |
| Maximum | 0.08 | 0.43 | 0.18 | 0.46 | 0.24 | 0.33 | 0.4 | 0.4 | 0.3 | 0.19 | 0.23 | 0.04 | 0.33 | 0.38 | 0.25 |
| Mean | 0.009855 | 0.03486 | 0.01416 | 0.00691 | 0.012319 | 0.035349 | 0.038436 | 0.030697 | 0.02834 | 0.01103 | 0.01093 | 0.00251 | 0.01105 | 0.022301 | 0.03191 |
| 1st Quartile | 0.003 | 0.01001 | 0.0005 | 0.0005 | 0.000504 | 0.010008 | 0.018002 | 0.010008 | 0.01001 | 0.000504 | 0.00202 | 0.0005 | 0.00202 | 0.005015 | 0.00802 |
| Median | 0.005004 | 0.02001 | 0.005 | 0.005 | 0.003997 | 0.034 | 0.041008 | 0.04 | 0.02999 | 0.005004 | 0.005 | 0.002 | 0.005 | 0.009997 | 0.022 |
| 3rd Quartile | 0.009986 | 0.047 | 0.02 | 0.00799 | 0.01498 | 0.033989 | 0.040997 | 0.040997 | 0.02998 | 0.011979 | 0.01399 | 0.00499 | 0.01198 | 0.024988 | 0.03399 |
| Std. Devn. | 0.01352 | 0.05672 | 0.02785 | 0.02136 | 0.0295 | 0.045491 | 0.039808 | 0.027939 | 0.02948 | 0.023318 | 0.02109 | 0.00237 | 0.02469 | 0.043148 | 0.04053 |
| Variance | 0.000183 | 0.00322 | 0.00078 | 0.00046 | 0.00087 | 0.002069 | 0.001585 | 0.000781 | 0.00087 | 0.000544 | 0.00045 | 6E-06 | 0.00061 | 0.001862 | 0.00164 |
| Co. of Variation | 1.371967 | 1.62711 | 1.96642 | 3.09197 | 2.394714 | 1.286924 | 1.035706 | 0.91014 | 1.04036 | 2.114085 | 1.93034 | 0.94342 | 2.2346 | 1.934796 | 1.2704 |

## TABLE 14-7: ASSAY STATISTICS FOR MOLYBDENUM BY LITHOLOGY IN SULFIDES

| Molybdenum | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 351 | 474 | 259 | 785 | 1,393 | 14,847 | 4,456 | 4,688 | 11,064 | 526 | 701 | 220 | 1,536 | 757 | 1,165 |
| Missing Values | 292 | 179 | 352 | 1300 | 1800 | 3661 | 809 | 591 | 2781 | 1595 | 839 | 1563 | 9305 | 2290 | 331 |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum | 0.044 | 0.239 | 0.155 | 0.317 | 0.762 | 1.95 | 2.66 | 1.0838 | 5.966 | 0.1336 | 1.0965 | 0.0218 | 0.1949 | 0.082 | 1.732 |
| Mean | 0.003831 | 0.00407 | 0.00501 | 0.0084 | 0.014607 | 0.016052 | 0.020731 | 0.013279 | 0.01728 | 0.006608 | 0.01342 | 0.00376 | 0.00497 | 0.00442 | 0.01587 |
| 1st Quartile | 0.001253 | 0.00101 | 0.00101 | 0.00173 | 0.002028 | 0.003222 | 0.004057 | 0.003401 | 0.00364 | 0.001849 | 0.00185 | 0.00233 | 0.00185 | 0.001849 | 0.00292 |
| Median | 0.002178 | 0.002 | 0.002 | 0.00361 | 0.003609 | 0.006294 | 0.008084 | 0.006771 | 0.00761 | 0.004027 | 0.00361 | 0.00319 | 0.00337 | 0.002715 | 0.00629 |
| 3rd Quartile | 0.004773 | 0.004 | 0.004 | 0.00674 | 0.008591 | 0.014975 | 0.018495 | 0.01396 | 0.01694 | 0.006264 | 0.0071 | 0.004 | 0.00537 | 0.005847 | 0.01599 |
| Std. Devn. | 0.004438 | 0.01263 | 0.01271 | 0.01885 | 0.045829 | 0.039534 | 0.074522 | 0.032946 | 0.07566 | 0.011349 | 0.05219 | 0.00302 | 0.00811 | 0.00636 | 0.05689 |
| Variance | 0.00002 | 0.00016 | 0.00016 | 0.00036 | 0.0021 | 0.001563 | 0.005553 | 0.001085 | 0.00573 | 0.000129 | 0.00272 | 9E-06 | 6.6E-05 | 0.00004 | 0.00324 |
| Co. of Variation | 1.158504 | 3.10296 | 2.53489 | 2.2438 | 3.137481 | 2.462815 | 3.59477 | 2.481081 | 4.37903 | 1.717527 | 3.88926 | 0.80255 | 1.63074 | 1.439046 | 3.58515 |

| Capped Molybdenum | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 351 | 474 | 259 | 785 | 1,393 | 14,847 | 4,456 | 4,688 | 11,064 | 526 | 701 | 220 | 1,536 | 757 | 1,165 |
| Missing Values | 292 | 179 | 352 | 1300 | 1800 | 3661 | 809 | 591 | 2781 | 1595 | 839 | 1563 | 9305 | 2290 | 331 |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 |
| Maximum | 0.008 | 0.03 | 0.01 | 0.05 | 0.06 | 0.2 | 0.29 | 0.29 | 0.16 | 0.07 | 0.027 | 0.0218 | 0.015 | 0.014 | 0.11 |
| Mean | 0.003266 | 0.00337 | 0.00325 | 0.0071 | 0.009687 | 0.015191 | 0.018083 | 0.0127 | 0.01507 | 0.006397 | 0.00673 | 0.00376 | 0.00424 | 0.003875 | 0.01339 |
| 1st Quartile | 0.001203 | 0.001 | 0.001 | 0.0017 | 0.002002 | 0.003201 | 0.004004 | 0.003401 | 0.0036 | 0.001802 | 0.0018 | 0.0023 | 0.0018 | 0.001802 | 0.0029 |
| Median | 0.0022 | 0.002 | 0.002 | 0.0036 | 0.003598 | 0.006302 | 0.0081 | 0.006799 | 0.0076 | 0.004002 | 0.0036 | 0.0032 | 0.0034 | 0.002701 | 0.0063 |
| 3rd Quartile | 0.004799 | 0.004 | 0.004 | 0.0068 | 0.008599 | 0.014997 | 0.018499 | 0.013998 | 0.017 | 0.0063 | 0.0071 | 0.004 | 0.0054 | 0.005896 | 0.016 |
| Std. Devn. | 0.002374 | 0.00437 | 0.00289 | 0.01031 | 0.014869 | 0.025999 | 0.033329 | 0.021377 | 0.02245 | 0.009627 | 0.00777 | 0.00302 | 0.00334 | 0.002767 | 0.01902 |
| Variance | 0.000006 | 1.9E-05 | 8E-06 | 0.00011 | 0.000221 | 0.000676 | 0.001111 | 0.000457 | 0.0005 | 0.000093 | 0.00006 | 9E-06 | 1.1E-05 | 0.000008 | 0.00036 |
| Co. of Variation | 0.726977 | 1.29817 | 0.88945 | 1.4516 | 1.534944 | 1.711424 | 1.843102 | 1.683234 | 1.48966 | 1.505054 | 1.15462 | 0.80255 | 0.78849 | 0.713977 | 1.42092 |

## TABLE 14-8: ASSAY STATISTICS FOR SILVER BY LITHOLOGY IN SULFIDES

| Silver | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 525 | 468 | 421 | 1,541 | 3,102 | 18,365 | 5,171 | 5,154 | 12,896 | 1,326 | 1,420 | 980 | 5,609 | 2,981 | 1,453 |
| Missing Values | 118 | 185 | 190 | 544 | 91 | 143 | 94 | 125 | 949 | 795 | 120 | 803 | 5232 | 66 | 43 |
| Minimum | 0.0015 | 0.0015 | 0.0015 | 0.0015 | 0.0015 | 0.0011 | 0.0015 | 0.0015 | 0.0011 | 0.0015 | 0.0015 | 0.0058 | 0.0015 | 0.0015 | 0.0011 |
| Maximum | 0.4958 | 2.1875 | 2.2138 | 9.8 | 9.5 | 6.7 | 4.69 | 2.1583 | 7.1 | 5.8333 | 3.0716 | 2.6483 | 5.8363 | 2.7441 | 2.4 |
| Mean | 0.017876 | 0.05564 | 0.08399 | 0.09435 | 0.092439 | 0.16374 | 0.156015 | 0.05907 | 0.12272 | 0.064246 | 0.06873 | 0.12817 | 0.07051 | 0.120812 | 0.06433 |
| 1st Quartile | 0.00588 | 0.00588 | 0.00294 | 0.00157 | 0.00294 | 0.023324 | 0.026362 | 0.017542 | 0.02038 | 0.00294 | 0.00588 | 0.02332 | 0.0146 | 0.029302 | 0.0146 |
| Median | 0.008673 | 0.01171 | 0.01171 | 0.00867 | 0.017493 | 0.081683 | 0.070021 | 0.032095 | 0.06973 | 0.008771 | 0.01749 | 0.05542 | 0.03504 | 0.053753 | 0.02916 |
| 3rd Quartile | 0.017444 | 0.03783 | 0.04959 | 0.04665 | 0.069972 | 0.18669 | 0.148764 | 0.06419 | 0.15749 | 0.043708 | 0.04959 | 0.21286 | 0.06997 | 0.1225 | 0.06154 |
| Std. Devn. | 0.034115 | 0.17002 | 0.24102 | 0.4876 | 0.314707 | 0.284016 | 0.283899 | 0.098597 | 0.20011 | 0.255497 | 0.18529 | 0.16213 | 0.21512 | 0.218948 | 0.12678 |
| Variance | 0.001164 | 0.02891 | 0.05809 | 0.23776 | 0.09904 | 0.080665 | 0.080598 | 0.009721 | 0.04004 | 0.065279 | 0.03433 | 0.02629 | 0.04627 | 0.047938 | 0.01607 |
| Co. of Variation | 1.908463 | 3.05599 | 2.86958 | 5.16782 | 3.404496 | 1.734552 | 1.819693 | 1.669149 | 1.63065 | 3.97683 | 2.6959 | 1.26498 | 3.05077 | 1.812305 | 1.97079 |

| Capped Silver | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 525 | 468 | 421 | 1,541 | 3,102 | 18,365 | 5,171 | 5,154 | 12,896 | 1,326 | 1,420 | 980 | 5,609 | 2,981 | 1,453 |
| Missing Values | 118 | 185 | 190 | 544 | 91 | 143 | 94 | 125 | 949 | 795 | 120 | 803 | 5232 | 66 | 43 |
| Minimum | 0.0015 | 0.0015 | 0.0015 | 0.0015 | 0.0015 | 0.0011 | 0.0015 | 0.0015 | 0.0011 | 0.0015 | 0.0015 | 0.0058 | 0.0015 | 0.0015 | 0.0011 |
| Maximum | 0.148 | 0.48 | 0.434 | 0.435 | 0.625 | 1.705 | 1.13 | 1.13 | 0.769 | 0.573 | 0.588 | 0.72 | 0.563 | 1.125 | 0.486 |
| Mean | 0.016631 | 0.04512 | 0.0588 | 0.05214 | 0.070392 | 0.159409 | 0.145747 | 0.058626 | 0.11681 | 0.049221 | 0.05966 | 0.1262 | 0.06146 | 0.115227 | 0.05959 |
| 1st Quartile | 0.005814 | 0.00581 | 0.00292 | 0.0015 | 0.002916 | 0.023307 | 0.026308 | 0.01751 | 0.02031 | 0.002916 | 0.00581 | 0.02331 | 0.01461 | 0.029309 | 0.01461 |
| Median | 0.008704 | 0.01171 | 0.01171 | 0.0087 | 0.017502 | 0.081695 | 0.069999 | 0.032097 | 0.06969 | 0.008806 | 0.0175 | 0.0554 | 0.035 | 0.053801 | 0.0292 |
| 3rd Quartile | 0.017493 | 0.03789 | 0.0496 | 0.0467 | 0.06999 | 0.186698 | 0.148795 | 0.064193 | 0.15749 | 0.043784 | 0.0496 | 0.21289 | 0.06999 | 0.122487 | 0.06159 |
| Std. Devn. | 0.022872 | 0.08226 | 0.10636 | 0.09643 | 0.12674 | 0.242969 | 0.223328 | 0.092161 | 0.1416 | 0.099482 | 0.11468 | 0.14197 | 0.08599 | 0.180425 | 0.08525 |
| Variance | 0.000523 | 0.00677 | 0.01131 | 0.0093 | 0.016063 | 0.059034 | 0.049876 | 0.008494 | 0.02005 | 0.009897 | 0.01315 | 0.02015 | 0.0074 | 0.032553 | 0.00727 |
| Co. of Variation | 1.37529 | 1.82318 | 1.80881 | 1.84921 | 1.800493 | 1.524191 | 1.532304 | 1.572022 | 1.21225 | 2.021144 | 1.9221 | 1.12489 | 1.39923 | 1.565829 | 1.43069 |

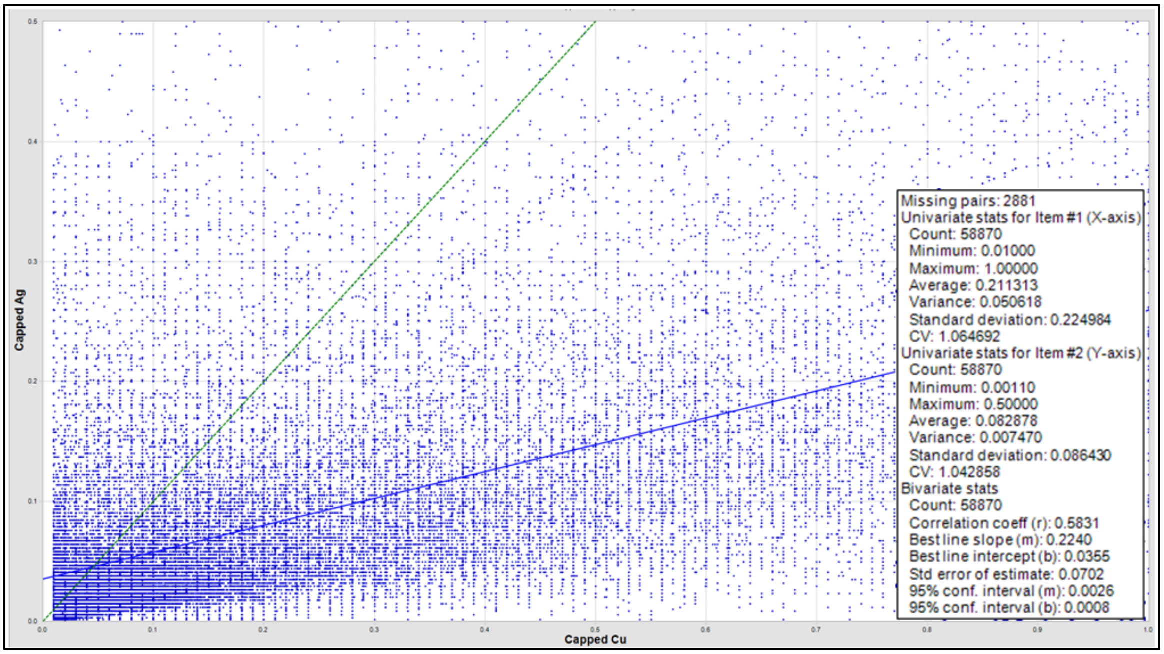
#### 14.4.4 Scatter Plots, Regression Analyses, Grade Adjustments and Analysis of Gold

Exploratory data analysis of assay scatter plots were examined between TCu, ASCu, molybdenum (Mo) and silver (Ag).

##### 14.4.4.1 Silver Adjustment

The scatter plot for silver against total copper is shown in Figure 14-8.

**FIGURE 14-8: SCATTER PLOT OF CAPPED SILVER AND CAPPED COPPER, ALL LITHOLOGY DOMAINS**




The scatter plot shows a relatively poor correlation (correlation coefficient of 0.58) between total copper and silver when looking at all the data. Better correlations were found when filtering by lithology and oxidation state. A reduction-to-major-axis (“RMA”) regression analyses was performed on silver against total copper for all the lithologies. Missing silver assays were assigned silver grades using the regression formula with copper grades when the correlation coefficient was above 0.7. The regression parameters used in the formula y = mx + b for assays within each lithology domain are tabulated in Table 14-9. Where the correlation coefficients were inferior to 0.7, missing silver values were replaced by zero as a conservative approach.

**TABLE 14-9: ASSAY RMA REGRESSION PARAMETERS, SILVER AGAINST COPPER**

| Hypogene | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Lower Earp | Upper Earp | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| m | | | | 0.00124 | | | 0.00192 | 0.00133 | | | | | | | |
| b | *Poor correlation* | | | 0.000126 | *Poor correlation* | | 0.000151 | 0.000047 | *Poor correlation* | | | | | | |
| r | | | | 0.82 | | | 0.71 | 0.72 | | | | | | | |

| Mix | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Lower Earp | Upper Earp | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| m | | | | | | | 0.00337 | 0.00102 | 0.0015 | | | | | | |
| b | *Poor correlation* | | | | | | -0.000403 | 0.000103 | -0.000133 | *Poor correlation* | | | | | |
| r | | | | | | | 0.96 | 0.83 | 0.86 | | | | | | |

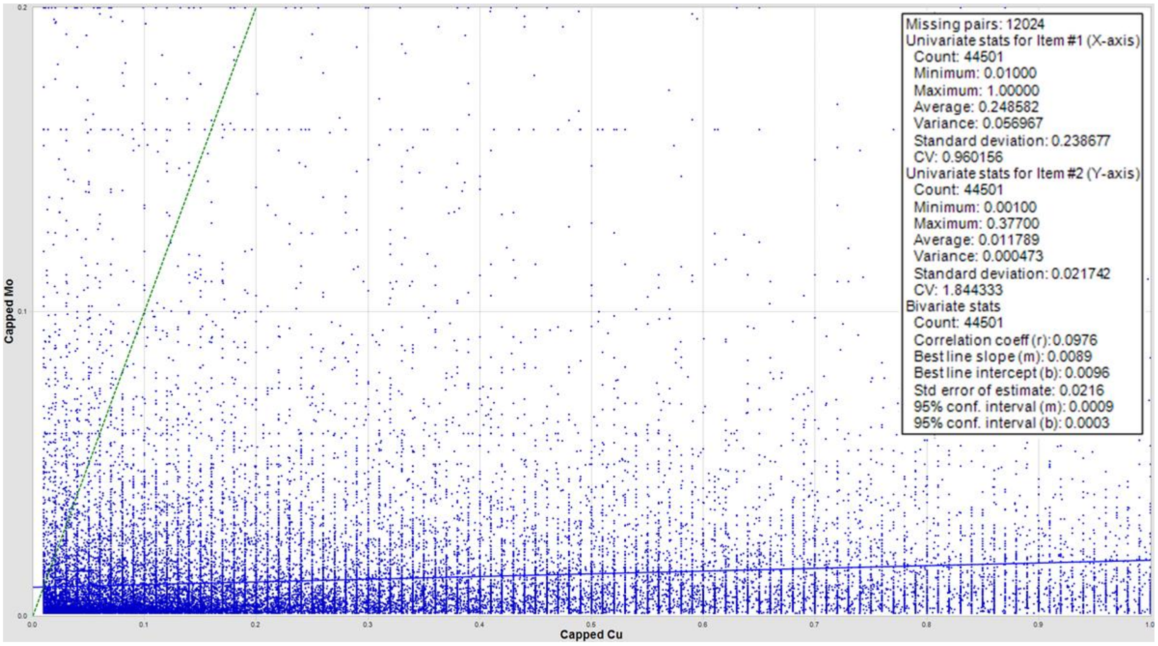
| Oxide | *Granodiorite* | Bolsa | Abrigo | *Martin* | *Escabrosa* | *Horquilla* | Lower Earp | Upper Earp | *Epitaph* | Scherrer | *Glance* | Gila | Arkose | *Andesite* | *QMP* |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| m | | | | | 0.00028 | | 0.000557 | | | 0.00199 | | | | | |
| b | *Poor correlation* | | | | 0.000131 | *Poor correlation* | 0.000206 | *Poor correlation* | | 0.000079 | *Poor correlation* | | | | |
| r | | | | | 0.98 | | 0.89 | | | 0.76 | | | | | |

Note: Regression parameters shown above using the formula y = mx + b. Slope is given by standard deviation silver / standard deviation of total copper.

### 14.4.4.2 Molybdenum Adjustment

In regards to the relationship between copper and molybdenum, the scatter plot of molybdenum against copper shows a poor correlation (correlation coefficient of 0.09) as shown in Figure 14-9. Given the poor correlation, no effort was made to calculate RMA equations.

**FIGURE 14-9: SCATTER PLOT OF CAPPED MOLYBDENUM AND CAPPED COPPER, ALL LITHOLOGY DOMAINS**



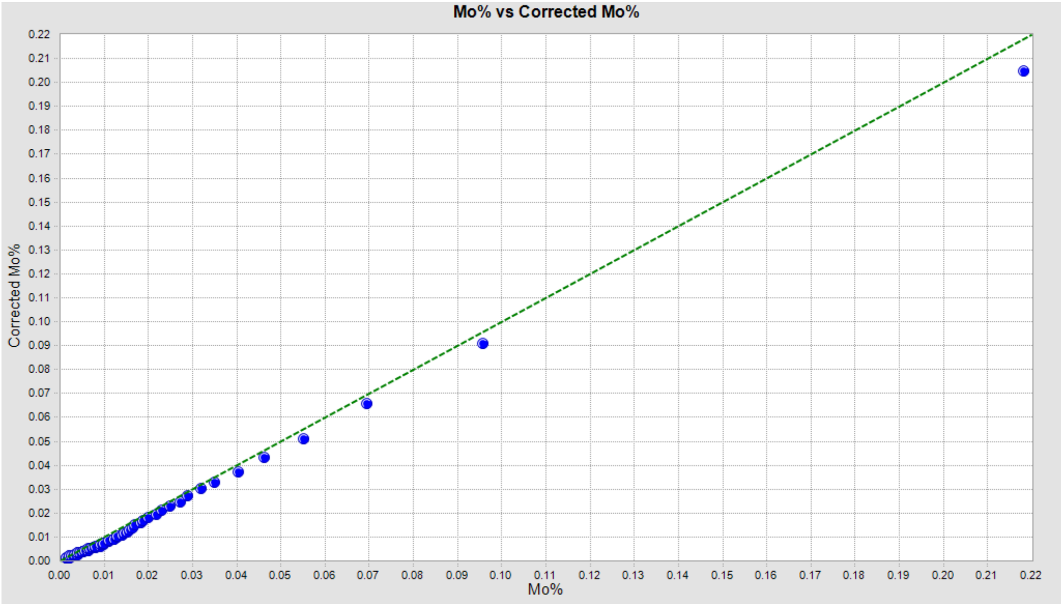

A bias in historical molybdenum assays analysed by Wet geochemical and X-ray analytical methods has been observed. As a result of the poor correlation between copper and molybdenum, no regression could be applied to the assays. Instead, two correction factors were applied to the affected historical assays, as described in Section 11 of this Technical Report. The formulas are given below:

Mo (corrected) = Mo (Wet) x 0.85

Mo (corrected) = Mo (X-ray) x 0.45

Figure 14-10 presents the QQ plot between the original and the corrected molybdenum grade.

**FIGURE 14-10: QQ PLOT OF ORIGINAL MOLYBDEMUM GRADE VERSUS CORRECTED MOLYBDENUM GRADE**



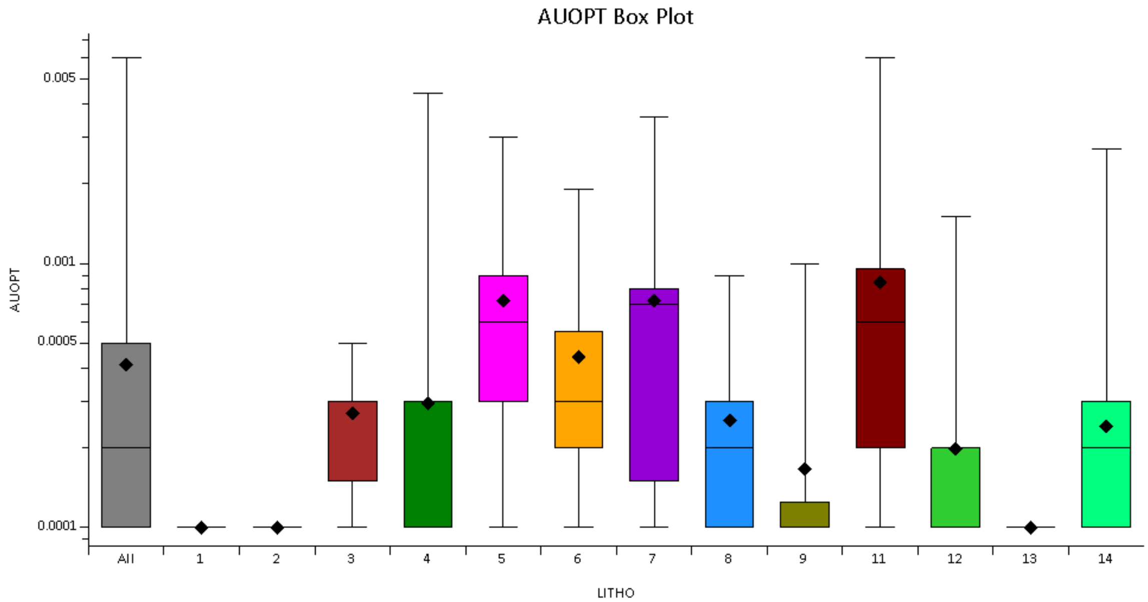

#### 14.4.4.3 Preliminary Analysis of Gold

Out of the 356 drill holes within the Rosemont database, only 66 drill holes were analyzed for gold, which representing only 17% of all the assaying conducted on the property. Table 14-10 presents the basic statistics of the drill data by company, while Figure 14-11 to Figure 14-13 present the box plots per lithology and oxidation state. Refer to Table 14-2 for the code equivalency.

**TABLE 14-10: DRILLING DATA BY COMPANY**

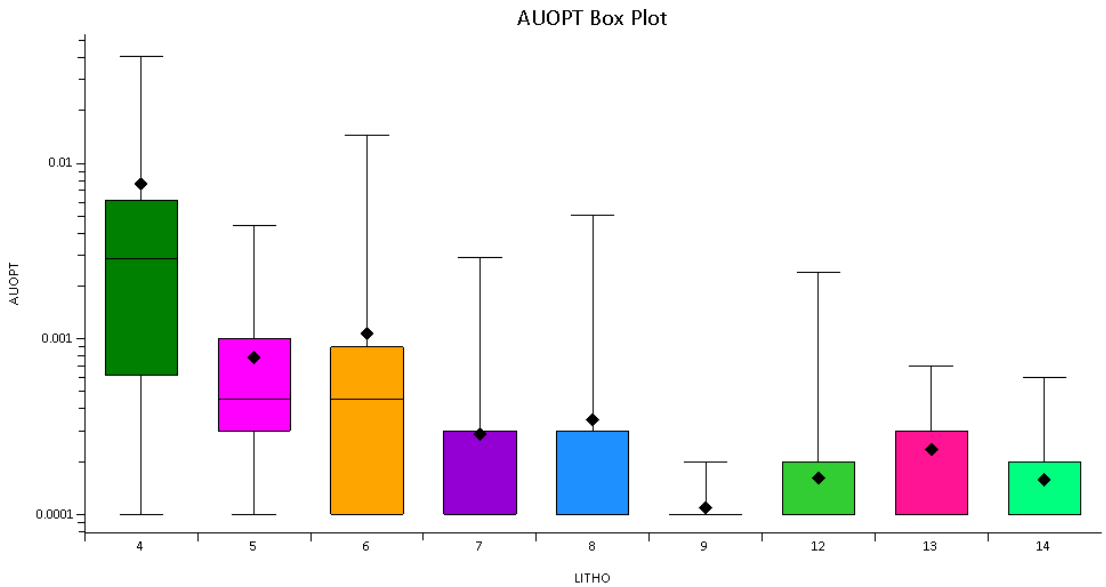
| | Anamax | Anaconda | Augusta | Hudbay |
|---|---|---|---|---|
| **Proportion** | **8.8%** | **20.1%** | **36.9%** | **34.2%** |
| # of DHs | 12 | 22 | 19 | 13 |
| Total # of Samples | 1,442 | 3,487 | 5,480 | 5,281 |
| Total Meters analyzed | 1,985 | 4,549 | 8,330 | 7,723 |
| Minimum | 0.0001 | 0.0001 | 0.0001 | 0.0001 |
| Maximum | 0.0063 | 0.0108 | 0.0384 | 0.0453 |
| Mean | 0.00112 | 0.00089 | 0.00057 | 0.00054 |
| Median | 0.0009 | 0.0006 | 0.0001 | 0.0002 |
| Mode | 0.0001 | 0.0001 | 0.0001 | 0.0001 |
| Standard Deviation | 0.00101 | 0.00114 | 0.00173 | 0.00144 |
| Sample Variance | 0.000001 | 0.000001 | 0.000003 | 0.000002 |
| Confidence Level (95.0%) | 0.000052 | 0.000038 | 0.000046 | 0.000039 |

## FIGURE 14-11: BOX PLOT OF GOLD IN OXIDE



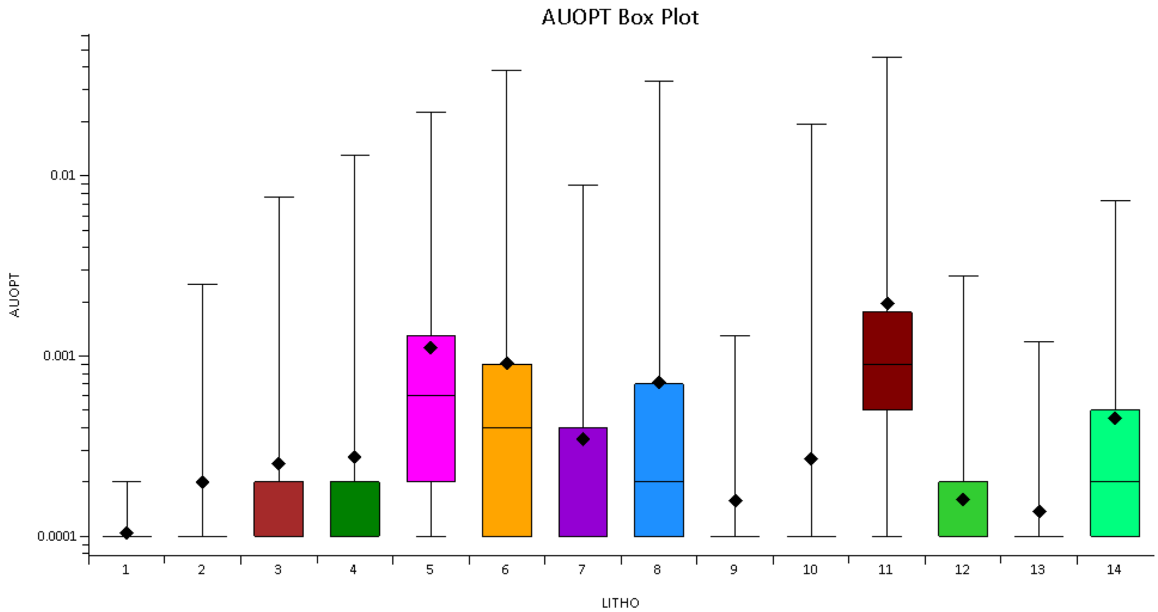

| Column | Domain Field | Domain | Count | Min | Max | Mean | Total | Variance | StDev | CV |
|---|---|---|---|---|---|---|---|---|---|---|
| AUOPT | LITHO | {all} | 1355 | 0.0001 | 0.006 | 0.000415 | 0.562 | 0 | 0.000539 | 1.3 |
| AUOPT | LITHO | 0 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 1 | 43 | 0.0001 | 0.0001 | 0.0001 | 0.0043 | 0 | 0 | 0 |
| AUOPT | LITHO | 2 | 17 | 0.0001 | 0.0001 | 0.0001 | 0.0017 | 0 | 0 | 0 |
| AUOPT | LITHO | 3 | 10 | 0.0001 | 0.0005 | 0.00027 | 0.0027 | 0 | 0.000134 | 0.5 |
| AUOPT | LITHO | 4 | 77 | 0.0001 | 0.0044 | 0.000296 | 0.0228 | 0 | 0.000575 | 1.94 |
| AUOPT | LITHO | 5 | 391 | 0.0001 | 0.003 | 0.000722 | 0.282 | 0 | 0.000638 | 0.88 |
| AUOPT | LITHO | 6 | 31 | 0.0001 | 0.0019 | 0.000442 | 0.0137 | 0 | 0.000401 | 0.91 |
| AUOPT | LITHO | 7 | 58 | 0.0001 | 0.0036 | 0.000722 | 0.0419 | 0 | 0.000768 | 1.06 |
| AUOPT | LITHO | 8 | 264 | 0.0001 | 0.0009 | 0.000254 | 0.0671 | 0 | 0.000165 | 0.65 |
| AUOPT | LITHO | 9 | 68 | 0.0001 | 0.001 | 0.000166 | 0.0113 | 0 | 0.000181 | 1.09 |
| AUOPT | LITHO | 10 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 11 | 47 | 0.0001 | 0.006 | 0.000847 | 0.0398 | 0 | 0.00111 | 1.31 |
| AUOPT | LITHO | 12 | 199 | 0.0001 | 0.0015 | 0.000198 | 0.0394 | 0 | 0.000175 | 0.88 |
| AUOPT | LITHO | 13 | 9 | 0.0001 | 0.0001 | 0.0001 | 0.0009 | 0 | 0 | 0 |
| AUOPT | LITHO | 14 | 141 | 0.0001 | 0.0027 | 0.000242 | 0.0341 | 0 | 0.000319 | 1.32 |

## FIGURE 14-12: BOX PLOT OF GOLD IN MIX




| Column | Domain Field | Domain | Count | Min | Max | Mean | Total | Variance | StDev | CV |
|---|---|---|---|---|---|---|---|---|---|---|
| AUOPT | LITHO | {all} | 1152 | 0.0001 | 0.0405 | 0.000484 | 0.557 | 0 | 0.00152 | 3.13 |
| AUOPT | LITHO | 0 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 1 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 2 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 3 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 4 | 8 | 0.0001 | 0.0405 | 0.00766 | 0.0613 | 0 | 0.0136 | 1.77 |
| AUOPT | LITHO | 5 | 286 | 0.0001 | 0.0044 | 0.000787 | 0.225 | 0 | 0.000813 | 1.03 |
| AUOPT | LITHO | 6 | 88 | 0.0001 | 0.0144 | 0.00107 | 0.0942 | 0 | 0.00241 | 2.25 |
| AUOPT | LITHO | 7 | 116 | 0.0001 | 0.0029 | 0.000286 | 0.0332 | 0 | 0.000471 | 1.64 |
| AUOPT | LITHO | 8 | 209 | 0.0001 | 0.0051 | 0.000347 | 0.0725 | 0 | 0.000662 | 1.91 |
| AUOPT | LITHO | 9 | 45 | 0.0001 | 0.0002 | 0.000109 | 0.0049 | 0 | 2.9E-05 | 0.26 |
| AUOPT | LITHO | 10 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 11 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 12 | 291 | 0.0001 | 0.0024 | 0.000162 | 0.047 | 0 | 0.000189 | 1.17 |
| AUOPT | LITHO | 13 | 23 | 0.0001 | 0.0007 | 0.000235 | 0.0054 | 0 | 0.000199 | 0.85 |
| AUOPT | LITHO | 14 | 86 | 0.0001 | 0.0006 | 0.000157 | 0.0135 | 0 | 0.000111 | 0.71 |

**FIGURE 14-13: BOX PLOT OF GOLD IN HYPOGENE**




| Column | Domain Field | Domain | Count | Min | Max | Mean | Total | Variance | StDev | CV |
|---|---|---|---|---|---|---|---|---|---|---|
| AUOPT | LITHO | {all} | 13182 | 0.0001 | 0.0453 | 0.000727 | 9.588 | 0 | 0.00153 | 2.1 |
| AUOPT | LITHO | 0 | 0 | | | | 0 | | | |
| AUOPT | LITHO | 1 | 24 | 0.0001 | 0.0002 | 0.000104 | 0.0025 | 0 | 2E-05 | 0.2 |
| AUOPT | LITHO | 2 | 88 | 0.0001 | 0.0025 | 0.0002 | 0.0176 | 0 | 0.000357 | 1.78 |
| AUOPT | LITHO | 3 | 266 | 0.0001 | 0.0076 | 0.000253 | 0.0672 | 0 | 0.000595 | 2.36 |
| AUOPT | LITHO | 4 | 622 | 0.0001 | 0.0131 | 0.000276 | 0.171 | 0 | 0.000675 | 2.45 |
| AUOPT | LITHO | 5 | 4215 | 0.0001 | 0.0227 | 0.00112 | 4.698 | 0 | 0.00168 | 1.51 |
| AUOPT | LITHO | 6 | 1645 | 0.0001 | 0.0384 | 0.000917 | 1.509 | 0 | 0.00182 | 1.98 |
| AUOPT | LITHO | 7 | 1302 | 0.0001 | 0.0089 | 0.000347 | 0.452 | 0 | 0.000592 | 1.7 |
| AUOPT | LITHO | 8 | 2748 | 0.0001 | 0.0334 | 0.000718 | 1.972 | 0 | 0.00157 | 2.19 |
| AUOPT | LITHO | 9 | 220 | 0.0001 | 0.0013 | 0.000158 | 0.0347 | 0 | 0.000169 | 1.07 |
| AUOPT | LITHO | 10 | 317 | 0.0001 | 0.0193 | 0.00027 | 0.0857 | 0 | 0.00124 | 4.57 |
| AUOPT | LITHO | 11 | 118 | 0.0001 | 0.0453 | 0.00196 | 0.231 | 0 | 0.00514 | 2.62 |
| AUOPT | LITHO | 12 | 1006 | 0.0001 | 0.0028 | 0.000159 | 0.16 | 0 | 0.000189 | 1.18 |
| AUOPT | LITHO | 13 | 287 | 0.0001 | 0.0012 | 0.000137 | 0.0393 | 0 | 0.000122 | 0.89 |
| AUOPT | LITHO | 14 | 324 | 0.0001 | 0.0073 | 0.000452 | 0.146 | 0 | 0.000775 | 1.72 |

In order to evaluate the global gold content of the Rosemont deposit, assay intervals were regularized by compositing drill hole data. The 25 ft intervals (+/- 12.5 feet of threshold) were composited using “honor geology” from the coded drill hole file. Once composited, oxidation levels were coded into the composite file.

For bias assessment purposes, assay intervals were also composited into 50 ft lengths (+/- 25 feet of threshold) using the same methodology. The 50 ft composites were used to estimate nearest neighbor models.

Down-the-hole and directional correlograms of gold were calculated using SAGE® software. However, given the limited number of pairs, the correlograms structures were found to be too erratic to produce meaningful variogram parameters. When directional correlograms were valid, the range continuity of the gold structures extended between 200 ft to 400 ft.

The preliminary gold grade estimation was completed on the 25ft length composites using Inverse-Distance-Squared ("IDW") grade interpolation method and three passes with increasing requirements using a composite and block matching system based on the lithology and oxidation codes.

The preliminary gold grade estimation was validated to ensure appropriate honoring of the input data. Nearest-neighbor (NN) from 50 ft composites was used to validate the IDW. Overall, no significant differences were observed.

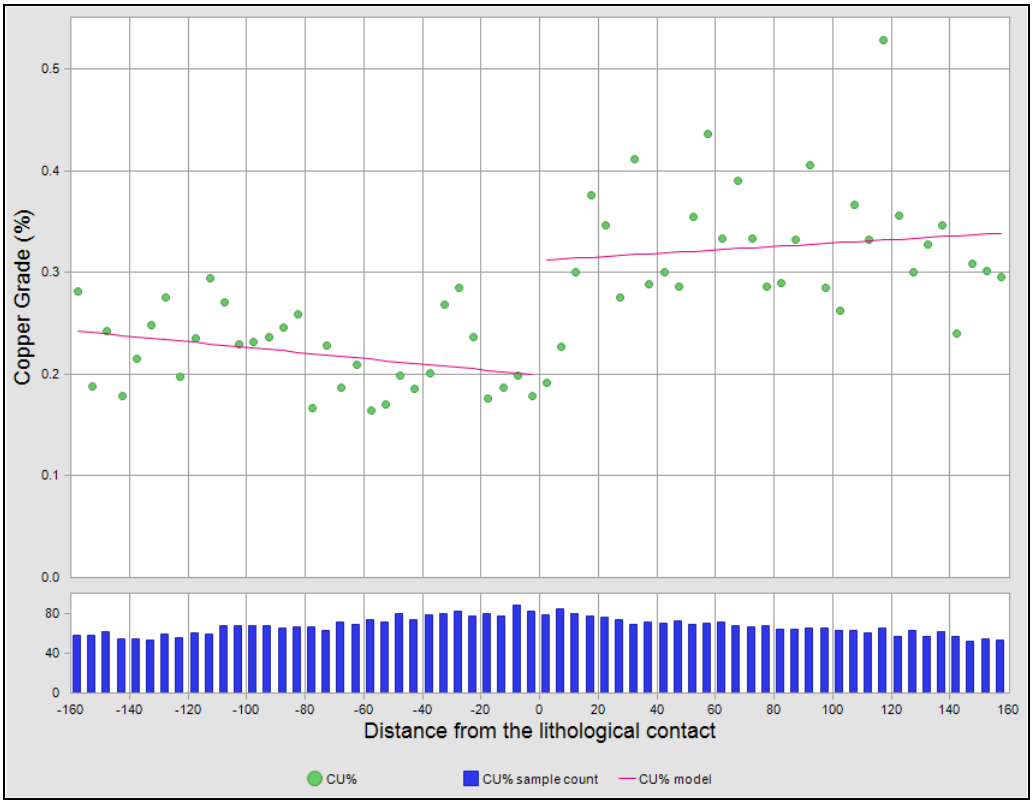
Even though the gold grade estimation is well-constrained by three-dimensional wireframes representing geologically realistic volumes of mineralization, the confidence level is considered "low" given the lack of data. Considering the fact that only a limited number of gold assay results are available, the gold mineralization for the Rosemont deposit cannot be classified under the 2014 CIM Definition Standards for Mineral Resources.

Nevertheless, the basic statistics of gold in the drill hole database and the interpolated blocks are showing similitude with the amount of gold measured from the head grade of the metallurgical tests (i.e. 0.03 to 0.05 g/ton or 0.0008 to 0.001 once per ton). Additional work should be undertaken to gain a better understanding of the gold distribution within the Rosemont deposit, both for the grade content and its spatial distribution.

#### 14.4.5 Contact Profiles

Exploratory data analysis ("EDA") of contact plots displaying average grades of Cu, ASCu, Mo and Ag by distance classes on either side of the contact between each lithology domain were created. The contact profiles show that there are sharp (hard), gradual (firm) and no (soft) changes in metal grade across the contacts. An example is shown in Figure 14-14 for the Earp lithologies. A matrix of boundary conditions for sulfide material is shown in Table 14-11, Table 14-12, Table 14-13 and Table 14-14.

**FIGURE 14-14: CONTACT PROFILE, UPPER AND LOWER EARP**



## TABLE 14-11: MATRIX OF BOUNDARY CONDITIONS, TOTAL COPPER

| Cu% | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp Lower | Earp Upper | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Granodiorite** | | | | | | | | | | | | | | | Soft |
| **Bolsa** | *Firm* | | | | | | | | | | | | | | |
| **Abrigo** | Sharp | *Firm* | | | | | | | | | | | | | |
| **Martin** | *Firm* | *Firm* | Sharp | | | | | | | | | | | | *Firm* |
| **Escabrosa** | | | | *Firm* | | *Firm* | | | | | | | | | *Firm* |
| **Horquilla** | | | | Sharp | *Firm* | | | | | | | | | | Sharp |
| **Earp - Lower** | | | | | | *Firm* | | | | | | | | | Sharp |
| **Earp - Upper** | | | | | | *Firm* | Sharp | | | | | | | | |
| **Epitaph** | | | | | | Sharp | *Firm* | *Firm* | | | | | | | *Firm* |
| **Scherrer** | | | | Sharp | | Sharp | Sharp | Sharp | Sharp | | Soft | | | *Firm* | *Firm* |
| **Glance** | | | | | | Sharp | Sharp | Sharp | Sharp | | | | | | *Firm* |
| **Gila** | | | | | | | | | Sharp | | | | Soft | | |
| **Arkose** | | | | | | | | | Sharp | Soft | Sharp | | | *Firm* | |
| **Andesite** | | | | | | | | Sharp | *Firm* | *Firm* | | | *Firm* | | |
| **QMP** | Soft | | | Soft | *Firm* | *Firm* | Sharp | | Soft | | *Firm* | | Soft | | |

## TABLE 14-12: MATRIX OF BOUNDARY CONDITIONS, ACID SOLUBLE COPPER

| ASCu% | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp Lower | Earp Upper | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Granodiorite** | | | | | | | | | | | | | | | Soft |
| **Bolsa** | Soft | | | | | | | | | | | | | | |
| **Abrigo** | Soft | *Firm* | | | | | | | | | | | | | |
| **Martin** | *Firm* | *Firm* | Soft | | | | | | | | | | | | Soft |
| **Escabrosa** | | | | *Firm* | | *Firm* | | | | | | | | | *Firm* |
| **Horquilla** | | | | Sharp | *Firm* | | | | | | | | | | *Firm* |
| **Earp - Lower** | | | | | | Soft | | | | | | | | | *Firm* |
| **Earp - Upper** | | | | | | *Firm* | Soft | | | | | | | | |
| **Epitaph** | | | | | | Sharp | *Firm* | *Firm* | | | | | | | Sharp |
| **Scherrer** | | | | *Firm* | | Sharp | Sharp | Sharp | *Firm* | | Soft | | | Sharp | Sharp |
| **Glance** | | | | | | Sharp | Sharp | Sharp | Sharp | | | | | | Sharp |
| **Gila** | | | | | | | | | Sharp | | | | Soft | | |
| **Arkose** | | | | | | | | | Sharp | *Firm* | *Firm* | | | *Firm* | |
| **Andesite** | | | | | | | | *Firm* | *Firm* | *Firm* | | | *Firm* | | |
| **QMP** | Sharp | | | Soft | *Firm* | *Firm* | Sharp | | *Firm* | | *Firm* | | Soft | | |

## TABLE 14-13: MATRIX OF BOUNDARY CONDITIONS, MOLYBDENUM

| Mo% | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp Lower | Earp Upper | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Granodiorite** | | | | | | | | | | | | | | | *Firm* |
| **Bolsa** | Sharp | | | | | | | | | | | | | | |
| **Abrigo** | Sharp | *Firm* | | | | | | | | | | | | | |
| **Martin** | Sharp | Sharp | *Firm* | | | | | | | | | | | | *Firm* |
| **Escabrosa** | | | | *Firm* | | *Firm* | | | | | | | | | *Firm* |
| **Horquilla** | | | | *Firm* | *Firm* | | | | | | | | | | *Firm* |
| **Earp - Lower** | | | | | | *Firm* | | | | | | | | | Sharp |
| **Earp - Upper** | | | | | | *Firm* | *Firm* | | | | | | | | |
| **Epitaph** | | | | | | Sharp | Soft | Soft | | | | | | | *Firm* |
| **Scherrer** | | | | *Firm* | | Sharp | *Firm* | Sharp | Sharp | | Soft | | | Soft | Sharp |
| **Glance** | | | | | | *Firm* | Sharp | Sharp | *Firm* | | | | | | Sharp |
| **Gila** | | | | | | | | | Sharp | | | | Soft | | |
| **Arkose** | | | | | | | | | Sharp | *Firm* | Sharp | | | *Firm* | |
| **Andesite** | | | | | | | | Sharp | Sharp | Soft | | | Soft | | |
| **QMP** | *Firm* | | | Soft | Sharp | Soft | *Firm* | | *Firm* | | Sharp | | *Firm* | | |

**TABLE 14-14: MATRIX OF BOUNDARY CONDITIONS, SILVER**

| Ag OPT | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp Lower | Earp Upper | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Granodiorite** | | | | | | | | | | | | | | | *Firm* |
| **Bolsa** | Sharp | | | | | | | | | | | | | | |
| **Abrigo** | Sharp | *Firm* | | | | | | | | | | | | | |
| **Martin** | Sharp | *Firm* | Sharp | | | | | | | | | | | | *Firm* |
| **Escabrosa** | | | | *Firm* | | *Firm* | | | | | | | | | Sharp |
| **Horquilla** | | | | Sharp | *Firm* | | | | | | | | | | *Firm* |
| **Earp - Lower** | | | | | | Soft | | | | | | | | | *Firm* |
| **Earp - Upper** | | | | | | *Firm* | Sharp | | | | | | | | |
| **Epitaph** | | | | | | Sharp | *Firm* | Soft | | | | | | | Soft |
| **Scherrer** | | | | Sharp | | Sharp | Sharp | *Firm* | Sharp | | *Firm* | | | Sharp | Soft |
| **Glance** | | | | | | *Firm* | Sharp | Soft | Soft | | | | | | *Firm* |
| **Gila** | | | | | | | | | Sharp | | | | Soft | | |
| **Arkose** | | | | | | | | | *Firm* | Soft | Sharp | | | Sharp | |
| **Andesite** | | | | | | | | Sharp | *Firm* | *Firm* | | | Sharp | | |
| **QMP** | *Firm* | | | Soft | Sharp | *Firm* | Sharp | | *Firm* | | *Firm* | | *Firm* | | |

## 14.5 Composites

In order to normalize the weight of influence for each sample, assay intervals were regularized by compositing drill hole data into 25-feet lengths using lithology boundaries to break composites. The 25-foot intervals (+/- 12.5 feet of threshold) were composited using "honor geology" from the coded drill hole file. For bias assessment purposes, assay intervals were also composited into 50-foot lengths (+/- 25 feet of threshold) using the same methodology. The 50-foot composites were used to estimate nearest neighbor models. Exploratory data analysis of the 25-foot composite statistics for TCu, ASCu, molybdenum (Mo) and silver (Ag) are shown in Table 14-15, Table 14-16, Table 14-17 and Table 14-18.

### TABLE 14-15: LENGTH WEIGHTED UNCAPPED AND CAPPED 25-FOOT COMPOSITE STATISTICS, COPPER IN SULFIDES

| Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 141 | 131 | 122 | 403 | 636 | 3,504 | 1,053 | 1,010 | 2,648 | 444 | 344 | 390 | 2,483 | 612 | 280 |
| Missing Values | 39 | 40 | 22 | 50 | 36 | 59 | 6 | 1 | 209 | 94 | 28 | 250 | 545 | 12 | 15 |
| Minimum | 0.0028 | 0.0074 | 0.0005 | 0.0001 | 0.0001 | 0.0006 | 0.0014 | 0.005 | 0.0004 | 0.0002 | 0.0003 | 0.0006 | 0.0001 | 0.0002 | 0.0002 |
| Maximum | 0.452 | 1.614 | 3.76 | 3.0866 | 13.0392 | 5.559 | 3.21 | 2.114 | 4.114 | 3.8458 | 1.9118 | 0.0713 | 2.854 | 1.612 | 6.0137 |
| Mean | 0.05228 | 0.17103 | 0.16779 | 0.11076 | 0.21366 | 0.40322 | 0.40405 | 0.22902 | 0.45703 | 0.07181 | 0.12005 | 0.00999 | 0.05153 | 0.12337 | 0.28154 |
| 1st Quartile | 0.01852 | 0.03012 | 0.00509 | 0.00404 | 0.00704 | 0.09206 | 0.11905 | 0.0961 | 0.08241 | 0.00509 | 0.00965 | 0.00417 | 0.00835 | 0.02008 | 0.07172 |
| Median | 0.03006 | 0.06604 | 0.01063 | 0.01519 | 0.0268 | 0.27506 | 0.25798 | 0.15484 | 0.28406 | 0.01858 | 0.02288 | 0.00619 | 0.02002 | 0.05444 | 0.13972 |
| 3rd Quartile | 0.05594 | 0.18398 | 0.06793 | 0.04994 | 0.08945 | 0.54791 | 0.46563 | 0.266 | 0.69108 | 0.06272 | 0.1034 | 0.01395 | 0.0519 | 0.136 | 0.35036 |
| Std. Devn. | 0.06141 | 0.26161 | 0.45447 | 0.30413 | 0.91044 | 0.45315 | 0.47334 | 0.23239 | 0.49761 | 0.23645 | 0.25483 | 0.00917 | 0.12281 | 0.19423 | 0.45782 |
| Variance | 0.00377 | 0.06844 | 0.20654 | 0.0925 | 0.82891 | 0.20534 | 0.22405 | 0.054 | 0.24761 | 0.05591 | 0.06494 | 0.00008 | 0.01508 | 0.03772 | 0.2096 |
| Co. of Variation | 1.17465 | 1.52958 | 2.70854 | 2.7458 | 4.26114 | 1.12382 | 1.17148 | 1.01468 | 1.08878 | 3.29256 | 2.12274 | 0.9184 | 2.38322 | 1.57431 | 1.62613 |

| Capped Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 141 | 131 | 122 | 403 | 636 | 3,504 | 1,053 | 1,010 | 2,648 | 444 | 344 | 390 | 2,483 | 612 | 280 |
| Missing Values | 39 | 40 | 22 | 50 | 36 | 59 | 6 | 1 | 209 | 94 | 28 | 250 | 545 | 12 | 15 |
| Minimum | 0.0028 | 0.0074 | 0.0005 | 0.0001 | 0.0001 | 0.0006 | 0.0014 | 0.005 | 0.0004 | 0.0002 | 0.0003 | 0.0006 | 0.0001 | 0.0002 | 0.0002 |
| Maximum | 0.434 | 0.908 | 1.4 | 1.3151 | 4.3584 | 3.118 | 3.1636 | 2.114 | 3.1256 | 1.2375 | 0.946 | 0.04 | 1.2165 | 1.132 | 1.452 |
| Mean | 0.05191 | 0.15427 | 0.13209 | 0.09098 | 0.16188 | 0.39348 | 0.40238 | 0.22902 | 0.45127 | 0.0617 | 0.09847 | 0.00969 | 0.04874 | 0.11498 | 0.25431 |
| 1st Quartile | 0.01844 | 0.03003 | 0.00501 | 0.00401 | 0.00702 | 0.09201 | 0.11903 | 0.09602 | 0.08233 | 0.00501 | 0.00963 | 0.00414 | 0.00832 | 0.02001 | 0.07174 |
| Median | 0.03001 | 0.06601 | 0.01061 | 0.01519 | 0.02678 | 0.27508 | 0.258 | 0.15479 | 0.28402 | 0.01859 | 0.0229 | 0.00621 | 0.01998 | 0.05441 | 0.13971 |
| 3rd Quartile | 0.05596 | 0.18397 | 0.06799 | 0.04999 | 0.08948 | 0.54397 | 0.4657 | 0.26599 | 0.68959 | 0.06276 | 0.10077 | 0.01399 | 0.052 | 0.13437 | 0.34758 |
| Std. Devn. | 0.06017 | 0.20332 | 0.28404 | 0.20446 | 0.46783 | 0.41137 | 0.46769 | 0.23239 | 0.47807 | 0.13086 | 0.17681 | 0.00822 | 0.09238 | 0.16273 | 0.27855 |
| Variance | 0.00362 | 0.04134 | 0.08068 | 0.0418 | 0.21886 | 0.16923 | 0.21873 | 0.054 | 0.22855 | 0.01712 | 0.03126 | 0.00007 | 0.00853 | 0.02648 | 0.07759 |
| Co. of Variation | 1.15914 | 1.31792 | 2.15034 | 2.24722 | 2.88988 | 1.04547 | 1.1623 | 1.01468 | 1.05939 | 2.12079 | 1.79565 | 0.84818 | 1.89527 | 1.41534 | 1.0953 |

### TABLE 14-16: LENGTH WEIGHTED UNCAPPED AND CAPPED 25-FOOT COMPOSITE STATISTICS, ACID SOLUBLE COPPER IN SULFIDES

| Soluble Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 141 | 131 | 122 | 402 | 636 | 3,500 | 1,053 | 1,009 | 2,630 | 426 | 343 | 339 | 2,360 | 602 | 280 |
| Missing Values | 39 | 40 | 22 | 51 | 36 | 63 | 6 | 2 | 227 | 112 | 29 | 301 | 668 | 22 | 15 |
| Minimum | 0.0005 | 0.0017 | 0.0004 | 0.0001 | 0.0001 | 0.0004 | 0.0005 | 0.0005 | 0.0004 | 0.0002 | 0.0003 | 0.0005 | 0.0001 | 0.0002 | 0.0002 |
| Maximum | 0.11 | 0.664 | 0.444 | 0.138 | 0.44 | 1.0444 | 0.3144 | 0.5544 | 0.5604 | 0.241 | 0.2012 | 0.01 | 1.1657 | 0.4816 | 0.5116 |
| Mean | 0.01139 | 0.04124 | 0.01535 | 0.00695 | 0.01407 | 0.03798 | 0.03917 | 0.03108 | 0.02936 | 0.01158 | 0.01144 | 0.00256 | 0.01202 | 0.0237 | 0.03826 |
| 1st Quartile | 0.00332 | 0.01001 | 0.00202 | 0.00062 | 0.00112 | 0.01481 | 0.02282 | 0.01331 | 0.01502 | 0.00171 | 0.0042 | 0.00091 | 0.00501 | 0.00602 | 0.00871 |
| Median | 0.00849 | 0.02001 | 0.00699 | 0.00499 | 0.00541 | 0.03321 | 0.0406 | 0.035 | 0.02999 | 0.00831 | 0.008 | 0.00201 | 0.00901 | 0.0162 | 0.028 |
| 3rd Quartile | 0.00999 | 0.0464 | 0.02 | 0.00799 | 0.01499 | 0.03499 | 0.04098 | 0.04098 | 0.02998 | 0.01198 | 0.01398 | 0.00498 | 0.01198 | 0.02498 | 0.03898 |
| Std. Devn. | 0.01658 | 0.07938 | 0.04165 | 0.01368 | 0.03628 | 0.05257 | 0.0313 | 0.02931 | 0.03321 | 0.02136 | 0.01773 | 0.0021 | 0.03228 | 0.04415 | 0.0559 |
| Variance | 0.00027 | 0.0063 | 0.00174 | 0.00019 | 0.00132 | 0.00276 | 0.00098 | 0.00086 | 0.0011 | 0.00046 | 0.00031 | 0 | 0.00104 | 0.00195 | 0.00312 |
| Co. of Variation | 1.45508 | 1.9249 | 2.71308 | 1.96806 | 2.57824 | 1.38395 | 0.79909 | 0.94297 | 1.13082 | 1.84407 | 1.5497 | 0.81924 | 2.68605 | 1.86296 | 1.46116 |

| Capped Soluble Copper | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 141 | 131 | 122 | 402 | 636 | 3,500 | 1,053 | 1,009 | 2,630 | 426 | 343 | 339 | 2,360 | 602 | 280 |
| Missing Values | 39 | 40 | 22 | 51 | 36 | 63 | 6 | 2 | 227 | 112 | 29 | 301 | 668 | 22 | 15 |
| Minimum | 0.0005 | 0.0017 | 0.0004 | 0.0001 | 0.0001 | 0.0004 | 0.0005 | 0.0005 | 0.0004 | 0.0002 | 0.0003 | 0.0005 | 0.0001 | 0.0002 | 0.0002 |
| Maximum | 0.11 | 0.372 | 0.18 | 0.138 | 0.2749 | 0.3948 | 0.2538 | 0.2796 | 0.4024 | 0.1695 | 0.14 | 0.01 | 0.4066 | 0.3212 | 0.3089 |
| Mean | 0.0111 | 0.03565 | 0.01317 | 0.0069 | 0.01223 | 0.03559 | 0.0388 | 0.03055 | 0.02871 | 0.01073 | 0.01088 | 0.00256 | 0.01139 | 0.0223 | 0.03538 |
| 1st Quartile | 0.0033 | 0.01001 | 0.002 | 0.0006 | 0.00111 | 0.01481 | 0.02281 | 0.01331 | 0.01502 | 0.00171 | 0.00422 | 0.00092 | 0.00501 | 0.00601 | 0.00871 |
| Median | 0.00849 | 0.02 | 0.00701 | 0.005 | 0.0054 | 0.0332 | 0.0406 | 0.035 | 0.03001 | 0.0083 | 0.00799 | 0.002 | 0.00899 | 0.0162 | 0.028 |
| 3rd Quartile | 0.00999 | 0.0464 | 0.01999 | 0.00798 | 0.015 | 0.03499 | 0.04099 | 0.04099 | 0.03 | 0.01199 | 0.014 | 0.00499 | 0.01199 | 0.02499 | 0.03898 |
| Std. Devn. | 0.01555 | 0.04984 | 0.02085 | 0.01322 | 0.02419 | 0.03619 | 0.02908 | 0.02307 | 0.02783 | 0.0156 | 0.01436 | 0.0021 | 0.01992 | 0.0337 | 0.04178 |
| Variance | 0.00024 | 0.00248 | 0.00043 | 0.00017 | 0.00059 | 0.00131 | 0.00085 | 0.00053 | 0.00077 | 0.00024 | 0.00021 | 0 | 0.0004 | 0.00114 | 0.00175 |
| Co. of Variation | 1.40048 | 1.3979 | 1.58295 | 1.91634 | 1.97839 | 1.01676 | 0.7494 | 0.75541 | 0.96935 | 1.45385 | 1.32007 | 0.81924 | 1.74805 | 1.51132 | 1.18081 |

## TABLE 14-17: LENGTH WEIGHTED UNCAPPED AND CAPPED 25-FOOT COMPOSITE STATISTICS, MOLYBDENUM IN SULFIDES

| Molybdenum | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 85 | 93 | 66 | 196 | 378 | 3,209 | 994 | 956 | 2,303 | 168 | 191 | 67 | 511 | 169 | 232 |
| Missing Values | 95 | 78 | 78 | 257 | 294 | 354 | 65 | 55 | 554 | 370 | 181 | 573 | 2517 | 455 | 63 |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.00102 | 0.001 | 0.001 | 0.001 | 0.00104 | 0.001 | 0.001 | 0.00101 |
| Maximum | 0.0136 | 0.0532 | 0.02608 | 0.07945 | 0.14421 | 0.464 | 0.5408 | 0.22924 | 1.3772 | 0.0392 | 0.15777 | 0.01448 | 0.05993 | 0.02308 | 0.49008 |
| Mean | 0.00359 | 0.00398 | 0.00401 | 0.00689 | 0.010548 | 0.014172 | 0.018947 | 0.012839 | 0.01559 | 0.004929 | 0.0097 | 0.00351 | 0.00439 | 0.003948 | 0.01483 |
| 1st Quartile | 0.00157 | 0.00121 | 0.0015 | 0.00175 | 0.002245 | 0.004118 | 0.005633 | 0.005013 | 0.00446 | 0.001845 | 0.0018 | 0.00209 | 0.0018 | 0.001832 | 0.0038 |
| Median | 0.002596 | 0.0024 | 0.00271 | 0.0034 | 0.003656 | 0.008394 | 0.010804 | 0.008793 | 0.00966 | 0.003794 | 0.004 | 0.0032 | 0.00329 | 0.002858 | 0.00868 |
| 3rd Quartile | 0.004848 | 0.00423 | 0.00405 | 0.00613 | 0.007809 | 0.016981 | 0.019391 | 0.015218 | 0.01812 | 0.005454 | 0.00788 | 0.00449 | 0.00541 | 0.005619 | 0.01712 |
| Std. Devn. | 0.002874 | 0.00657 | 0.00467 | 0.01044 | 0.020123 | 0.019812 | 0.037229 | 0.016743 | 0.04043 | 0.005345 | 0.01801 | 0.00221 | 0.00477 | 0.00335 | 0.03423 |
| Variance | 0.000008 | 4.3E-05 | 2.2E-05 | 0.00011 | 0.000405 | 0.000393 | 0.001386 | 0.00028 | 0.00163 | 0.000029 | 0.00032 | 5E-06 | 2.3E-05 | 0.000011 | 0.00117 |
| Co. of Variation | 0.800392 | 1.65175 | 1.16405 | 1.51569 | 1.907668 | 1.397944 | 1.964893 | 1.304049 | 2.59305 | 1.084332 | 1.85611 | 0.62871 | 1.08606 | 0.848361 | 2.30779 |

| Capped Molybdenum | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 85 | 93 | 66 | 196 | 378 | 3,209 | 994 | 956 | 2,303 | 168 | 191 | 67 | 511 | 169 | 232 |
| Missing Values | 95 | 78 | 78 | 257 | 294 | 354 | 65 | 55 | 554 | 370 | 181 | 573 | 2517 | 455 | 63 |
| Minimum | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.00102 | 0.001 | 0.001 | 0.001 | 0.00104 | 0.001 | 0.001 | 0.00101 |
| Maximum | 0.0078 | 0.0162 | 0.0068 | 0.04649 | 0.04477 | 0.1955 | 0.23576 | 0.12464 | 0.13739 | 0.0392 | 0.027 | 0.01448 | 0.01638 | 0.01336 | 0.09412 |
| Mean | 0.003155 | 0.00322 | 0.00278 | 0.00587 | 0.007226 | 0.013452 | 0.016642 | 0.012367 | 0.01361 | 0.004853 | 0.00561 | 0.00351 | 0.00385 | 0.003495 | 0.01238 |
| 1st Quartile | 0.00157 | 0.0012 | 0.0015 | 0.00174 | 0.002242 | 0.004112 | 0.005632 | 0.005 | 0.00446 | 0.001841 | 0.0018 | 0.00208 | 0.0018 | 0.001832 | 0.0038 |
| Median | 0.00258 | 0.0024 | 0.00264 | 0.00339 | 0.003651 | 0.008399 | 0.010799 | 0.0088 | 0.00966 | 0.003799 | 0.004 | 0.0032 | 0.00325 | 0.00284 | 0.00869 |
| 3rd Quartile | 0.00456 | 0.00397 | 0.00352 | 0.00613 | 0.007818 | 0.016979 | 0.019398 | 0.015228 | 0.01812 | 0.005458 | 0.0072 | 0.0045 | 0.0054 | 0.004958 | 0.01712 |
| Std. Devn. | 0.002019 | 0.00272 | 0.00145 | 0.00684 | 0.008699 | 0.015085 | 0.020973 | 0.012977 | 0.01359 | 0.005069 | 0.00498 | 0.00221 | 0.00255 | 0.002166 | 0.01265 |
| Variance | 0.000004 | 7E-06 | 2E-06 | 4.7E-05 | 0.000076 | 0.000228 | 0.00044 | 0.000168 | 0.00019 | 0.000026 | 2.5E-05 | 5E-06 | 6E-06 | 0.000005 | 0.00016 |
| Co. of Variation | 0.640075 | 0.84556 | 0.52127 | 1.16667 | 1.203853 | 1.121356 | 1.260259 | 1.049359 | 0.99813 | 1.044429 | 0.88826 | 0.62871 | 0.66173 | 0.619595 | 1.02177 |

## TABLE 14-18: LENGTH WEIGHTED UNCAPPED AND CAPPED 25-FOOT COMPOSITE STATISTICS, SILVER IN SULFIDES

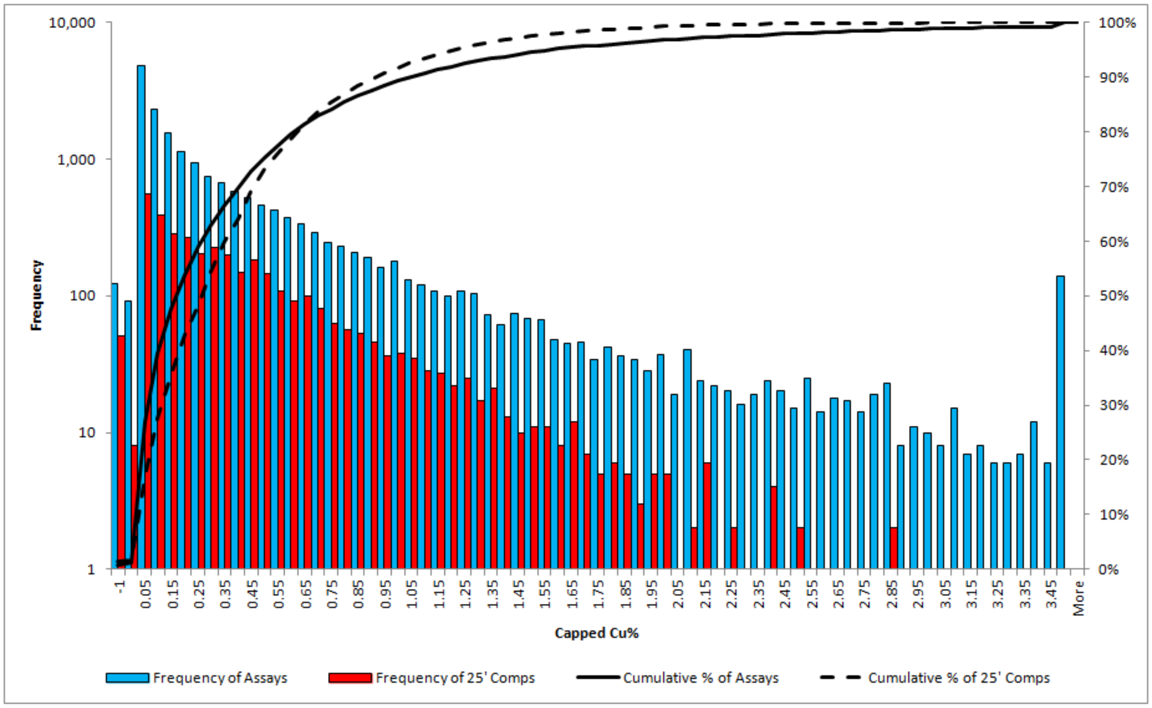
| Silver | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 106 | 101 | 84 | 315 | 654 | 3,511 | 1,050 | 1,004 | 2,555 | 306 | 356 | 193 | 1,221 | 615 | 279 |
| Missing Values | 74 | 70 | 60 | 138 | 18 | 52 | 9 | 7 | 302 | 232 | 16 | 447 | 1807 | 9 | 16 |
| Minimum | 0.0018 | 0.0017 | 0.0007 | 0.0003 | 0.0014 | 0.0001 | 0.0009 | 0.0002 | 0.0002 | 0.0005 | 0.0003 | 0.0029 | 0.0001 | 0.0014 | 0.0002 |
| Maximum | 0.1295 | 0.9322 | 1.6818 | 4.6521 | 2.6132 | 2.016 | 1.6118 | 0.6591 | 1.7803 | 1.5816 | 0.6636 | 0.645 | 1.8122 | 0.9607 | 1.2528 |
| Mean | 0.01636 | 0.05337 | 0.07605 | 0.08891 | 0.07711 | 0.15825 | 0.1477 | 0.057 | 0.1164 | 0.04923 | 0.04989 | 0.12425 | 0.06434 | 0.10508 | 0.06287 |
| 1st Quartile | 0.00642 | 0.00702 | 0.00433 | 0.00372 | 0.00572 | 0.04085 | 0.03703 | 0.02052 | 0.02363 | 0.00414 | 0.00391 | 0.02452 | 0.01503 | 0.02982 | 0.01754 |
| Median | 0.00989 | 0.01412 | 0.01631 | 0.01328 | 0.01849 | 0.10209 | 0.07981 | 0.03622 | 0.07641 | 0.0103 | 0.0144 | 0.0572 | 0.03501 | 0.06292 | 0.03301 |
| 3rd Quartile | 0.0181 | 0.04457 | 0.06499 | 0.06276 | 0.07378 | 0.20488 | 0.15278 | 0.06359 | 0.16389 | 0.03866 | 0.04499 | 0.19776 | 0.06876 | 0.12649 | 0.0709 |
| Std. Devn. | 0.01793 | 0.11684 | 0.20279 | 0.35631 | 0.18305 | 0.1882 | 0.20494 | 0.06951 | 0.13724 | 0.14076 | 0.09728 | 0.13899 | 0.11757 | 0.12885 | 0.09784 |
| Variance | 0.00032 | 0.01365 | 0.04112 | 0.12696 | 0.03351 | 0.03542 | 0.042 | 0.00483 | 0.01883 | 0.01981 | 0.00946 | 0.01932 | 0.01382 | 0.0166 | 0.00957 |
| Co. of Variation | 1.096 | 2.18935 | 2.66642 | 4.0074 | 2.37392 | 1.18927 | 1.38755 | 1.21947 | 1.17899 | 2.85914 | 1.94989 | 1.11865 | 1.82735 | 1.22626 | 1.55612 |

| Capped Silver | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 106 | 101 | 84 | 315 | 654 | 3,511 | 1,050 | 1,004 | 2,555 | 306 | 356 | 193 | 1,221 | 615 | 279 |
| Missing Values | 74 | 70 | 60 | 138 | 18 | 52 | 9 | 7 | 302 | 232 | 16 | 447 | 1807 | 9 | 16 |
| Minimum | 0.0018 | 0.0017 | 0.0007 | 0.0003 | 0.0014 | 0.0001 | 0.0009 | 0.0002 | 0.0002 | 0.0005 | 0.0003 | 0.0029 | 0.0001 | 0.0014 | 0.0002 |
| Maximum | 0.0837 | 0.4097 | 0.434 | 0.435 | 0.6142 | 1.4619 | 1.0331 | 0.6185 | 0.769 | 0.4154 | 0.472 | 0.5302 | 0.514 | 0.7945 | 0.4764 |
| Mean | 0.01548 | 0.04273 | 0.05369 | 0.04889 | 0.06077 | 0.15461 | 0.13912 | 0.05658 | 0.11134 | 0.03781 | 0.04474 | 0.12245 | 0.05651 | 0.10141 | 0.05808 |
| 1st Quartile | 0.0064 | 0.007 | 0.00431 | 0.00371 | 0.0057 | 0.0408 | 0.037 | 0.02051 | 0.02361 | 0.00411 | 0.0039 | 0.0245 | 0.01501 | 0.02981 | 0.01751 |
| Median | 0.0099 | 0.0141 | 0.01629 | 0.0133 | 0.0185 | 0.10211 | 0.0798 | 0.0362 | 0.07641 | 0.0103 | 0.01441 | 0.0572 | 0.03501 | 0.0629 | 0.033 |
| 3rd Quartile | 0.0181 | 0.04459 | 0.06159 | 0.06229 | 0.0738 | 0.2042 | 0.1528 | 0.06359 | 0.1634 | 0.0387 | 0.045 | 0.1978 | 0.0688 | 0.1265 | 0.0708 |
| Std. Devn. | 0.01461 | 0.06511 | 0.08392 | 0.07453 | 0.09241 | 0.17177 | 0.17415 | 0.06756 | 0.11475 | 0.06644 | 0.07776 | 0.13439 | 0.06618 | 0.11678 | 0.0666 |
| Variance | 0.00021 | 0.00424 | 0.00704 | 0.00555 | 0.00854 | 0.02951 | 0.03033 | 0.00456 | 0.01317 | 0.00441 | 0.00605 | 0.01806 | 0.00438 | 0.01364 | 0.00444 |
| Co. of Variation | 0.94379 | 1.52379 | 1.563 | 1.52444 | 1.52067 | 1.11101 | 1.25175 | 1.19404 | 1.03068 | 1.75716 | 1.73816 | 1.09749 | 1.17118 | 1.15157 | 1.14681 |

The length weighted mean grades of both 25-foot and 50-foot length composites are similar to those of the assays; therefore, providing confidence that the compositing process is working as intended. The appreciable amounts of sulfide mineralization, located within the Horquilla, Earp (lower and upper), and Epitaph lithologies, consist of low to moderate CV values for all metal types. These CV values suggest that no further domaining is warranted and that a linear interpolation method can be used. Linear interpolation was also used for the other lithological units given their minor contribution to the mineralization of economic interest. Applying non-linear interpolation methods and/or revisions of the wireframing criteria will be further investigated for these lithologies in future updates of the resource model.

Histogram and basic statistics for capped total copper within the Horquilla lithology unit are shown in Figure 14-15.

**FIGURE 14-15: HISTOGRAM, 25-FOOT COPPER COMPOSITES, HORQUILLA LITHOLOGY**



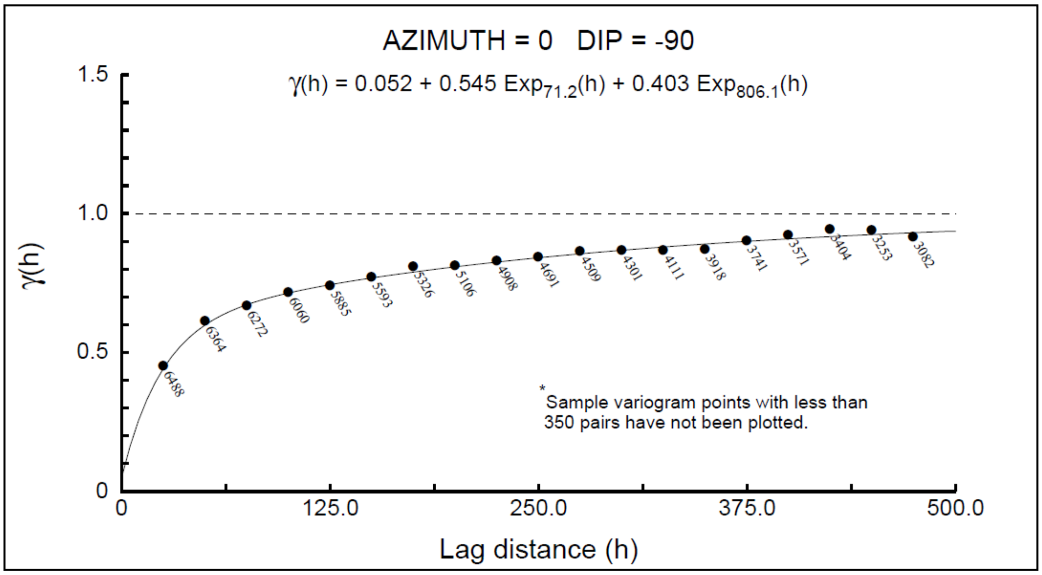

| | Assays | 25' Comps |
|---|---|---|
| Count | 18,385 | 3,576 |
| Minimum | 0 | 0 |
| Maximum | 3.5 | 3.12 |
| Range | 3.5 | 3.12 |
| Mean | 0.40 | 0.39 |
| Median | 0.1705 | 0.27 |
| Mode | 0.01 | 0.02 |
| Standard Deviation | 0.59 | 0.41 |
| Sample Variance | 0.35 | 0.17 |
| Standard Error | 0.00 | 0.01 |
| Confidence Level(95.0%) | 0.01 | 0.01 |

## 14.6 Variography

Down-hole and directional correlograms for total copper, acid-soluble copper, molybdenum and silver using three combined groups of lithologies were created using SAGE® software. The Footwall Group of lithologies lies to the west of the Backbone Fault and includes the Granodiorite, Bolsa, Abrigo, Martin and Escabrosa. The Lower Plate group of lithologies lies on the hanging wall of the Backbone Fault and includes Horquilla, Earp (lower and upper) and Epitaph. The Upper Plate group of lithologies lies above the Lower Plate group and includes Scherrer, Glance, Gila, Arkose, Andesite and the QMPs. Due to a limited number of pairs in the oxide and mixed zones, the analysis was conducted on oxidation state only rather than lithology.

The total copper variograms show very low to moderate nugget effects with values between 2% and 52% of the total variance for the sulfide mineralization. The ranges of correlation generally vary between 340 and 2,000 feet (103 and 609 meters). The downhole variogram for the lower plate group of lithologies is shown in Figure 14-16.

**FIGURE 14-16: DOWNHOLE VARIOGRAM COPPER, LOWER GROUP OF LITHOLOGIES IN SULFIDES**




A nugget and a nested exponential model were fitted to the experimental correlograms. An example of a variogram showing the anisotropy of the fitted model, together with the three principal directions is shown in Figure 14-17 and Figure 14-18. Correlogram model parameters for TCu, ASCu, molybdenum (Mo) and silver (Ag) are shown in Table 14-19.

**FIGURE 14-17: CORRELOGRAM OF THE MAIN STRUCTURE OF COPPER, LOWER GROUP OF LITHOLOGIES IN SULFIDES**

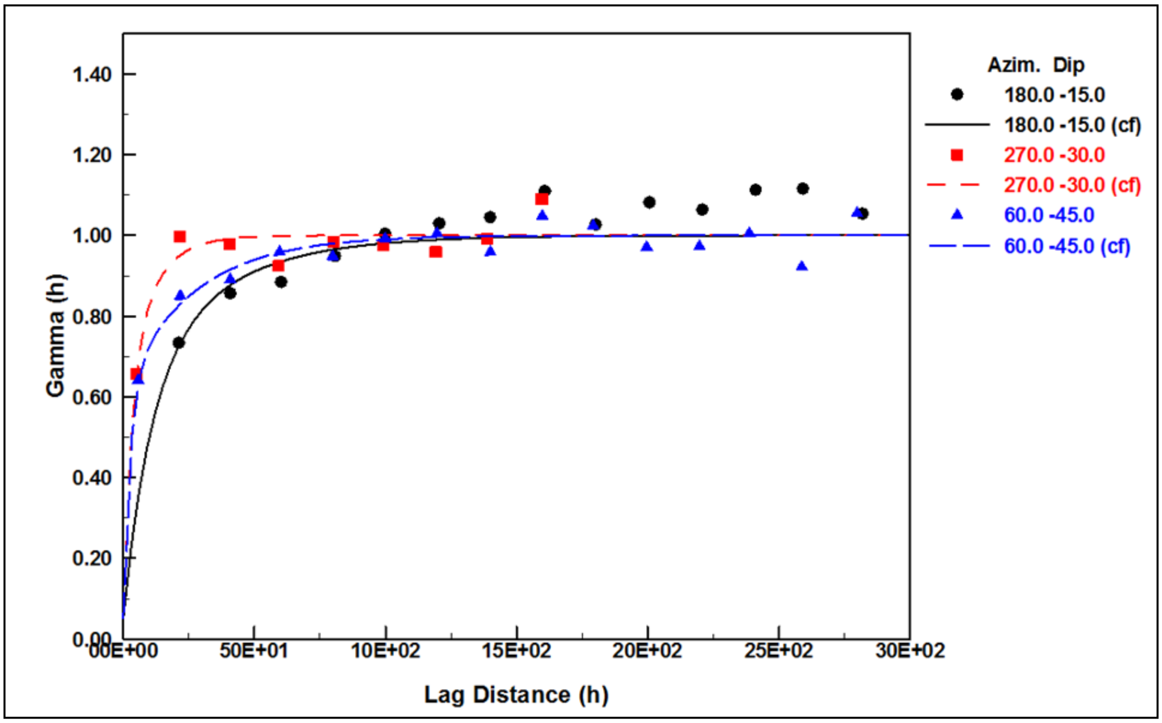
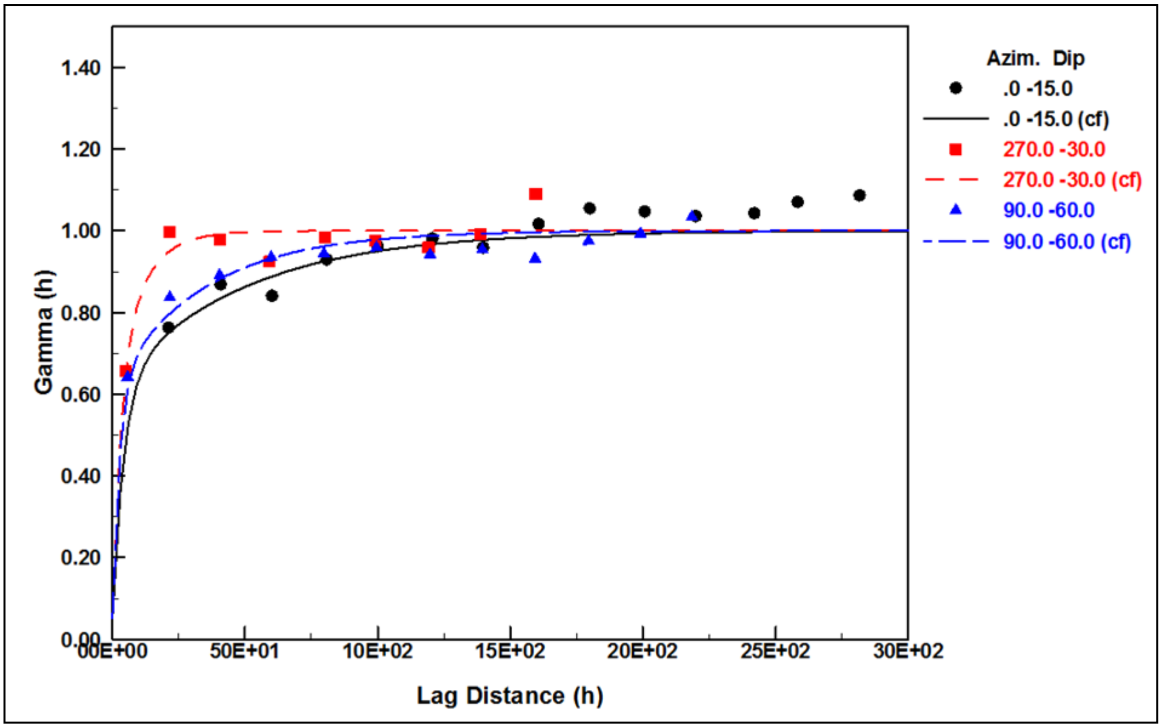



**FIGURE 14-18: CORRELOGRAM OF THE NESTED STRUCTURE OF COPPER, LOWER GROUP OF LITHOLOGIES IN SULFIDES**



## TABLE 14-19: VARIOGRAM MODELS AND ROTATION ANGLES

| | | | First Structure | | | | | | | Second Nested Structure | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Metal | Nugget | Sill 1 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 | Sill 2 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 |
| Oxide | Copper | 0.02 | 0.298 | 295 | 175 | 50 | -95 | 30 | 27 | 0.682 | 2260 | 1250 | 585 | -4 | 13 | -16 |
| | Soluble Copper | 0.147 | 0.306 | 365 | 920 | 585 | -20 | -27 | 21 | 0.547 | 2500 | 950 | 605 | -11 | 12 | -28 |
| | Molybdenum | 0.592 | 0.274 | 375 | 1000 | 525 | -18 | -52 | -3 | 0.135 | 165 | 345 | 1000 | 53 | 147 | -72 |
| | Silver | 0.102 | 0.707 | 735 | 155 | 90 | 5 | -13 | -82 | 0.191 | 1000 | 470 | 670 | -26 | 24 | 81 |

| | | | First Structure | | | | | | | Second Nested Structure | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Metal | Nugget | Sill 1 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 | Sill 2 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 |
| Mixed | Copper | 0.294 | 0.497 | 185 | 100 | 1000 | -37 | 0 | 10 | 0.209 | 1600 | 800 | 400 | 29 | 33 | -51 |
| | Soluble Copper | 0.001 | 0.591 | 190 | 95 | 50 | -113 | 71 | 80 | 0.408 | 2000 | 700 | 1415 | -45 | 15 | -23 |
| | Molybdenum | 0.47 | 0.002 | 500 | 250 | 125 | -64 | 99 | -140 | 0.528 | 1000 | 500 | 250 | -29 | 11 | 92 |
| | Silver | 0.124 | 0.681 | 220 | 630 | 55 | -57 | -43 | 9 | 0.195 | 1000 | 570 | 300 | 61 | 62 | -140 |

| | | | First Structure | | | | | | | Second Nested Structure | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Sulfides | Metal | Nugget | Sill 1 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 | Sill 2 | Major axis | Semi Major axis | Minor axis | ROT1 | ROT2 | ROT3 |
| Footwall Group | Copper | 0.029 | 0.642 | 220 | 370 | 90 | -32 | 81 | -57 | 0.329 | 2000 | 530 | 1765 | 2 | -6 | 68 |
| | Soluble Copper | 0.133 | 0.601 | 85 | 45 | 225 | 52 | -4 | 2 | 0.266 | 2010 | 755 | 2115 | -57 | 29 | 63 |
| | Molybdenum | 0.432 | 0.461 | 300 | 145 | 60 | -15 | -29 | 33 | 0.107 | 1250 | 715 | 500 | -10 | 5 | 72 |
| | Silver | 0.553 | 0.219 | 980 | 180 | 330 | -4 | 0 | 6 | 0.228 | 975 | 435 | 1000 | -10 | -10 | 49 |
| Lower Plate Group | Copper | 0.52 | 0.57 | 360 | 85 | 80 | 3 | 23 | 34 | 0.378 | 1420 | 340 | 1070 | -2 | -10 | 27 |
| | Soluble Copper | 0.063 | 0.659 | 165 | 65 | 55 | -33 | 89 | -9 | 0.278 | 1945 | 755 | 1775 | 27 | -24 | 62 |
| | Molybdenum | 0.432 | 0.461 | 300 | 145 | 60 | -15 | -29 | 33 | 0.107 | 1250 | 715 | 500 | -10 | 5 | 72 |
| | Silver | 0.551 | 0.154 | 485 | 340 | 100 | -2 | -13 | -107 | 0.295 | 925 | 350 | 500 | -23 | -4 | -2 |
| Upper Plate Group | Copper | 0.026 | 0.622 | 405 | 135 | 45 | -2 | 19 | -92 | 0.353 | 1550 | 520 | 1220 | -71 | 7 | 82 |
| | Soluble Copper | 0.133 | 0.601 | 85 | 45 | 225 | 52 | -4 | 2 | 0.266 | 2010 | 755 | 2115 | -57 | 29 | 63 |
| | Molybdenum | 0.432 | 0.461 | 300 | 145 | 60 | -15 | -29 | 33 | 0.107 | 1250 | 715 | 500 | -10 | 5 | 72 |
| | Silver | 0.553 | 0.219 | 980 | 180 | 330 | -4 | 0 | 6 | 0.228 | 975 | 435 | 1000 | -10 | -10 | 49 |

*Note: Ranges are in feet and search ellipse orientations are given using MEDS rotation convention.*

## 14.7 Estimation and Interpolation Methods

Lithology solids were used to code assay and composite intervals. The same solids were used to code blocks in the model based on a minimum 50% majority code threshold. Aside from the lithologies in the footwall of the backbone fault which have a limited number of composites, metal grade estimation used a composite and block matching system based on the lithology and oxidation codes. For example, in the case of the Horquilla lithology in the sulfides, only composites coded as Horquilla and sulfides were used to estimate block grades. In the case of swelling and magnesium clays, the ore types solids were used to code the assays, composites and blocks.

The block model consists of regular blocks (50 feet along strike x 50 feet across strike x 50 feet vertically). The block size was chosen such that geological contacts are reasonably well reflected and to support a large-scale open pit mining scenario.

The interpolation plan was completed on the uncapped and capped composites, 25 feet in length, using ordinary kriged (“OK”) grade interpolation method using three passes with increasing search distances.

The composite selection parameters for grade estimation in each domain (minimum, maximum, and maximum number of composites per hole) were selected to minimize bias. Table 14-20 and Table 14-21 show the search distances and search ellipse orientations for the estimation domains.

The first interpolation pass is restricted to a minimum of nine composites, a maximum of 12 composites (with a maximum of three composites per hole) and quadrant declustering. The second interpolation pass is restricted to a minimum of six composites, a maximum of 12 composites (with a maximum of three composites per hole) and quadrant declustering. Finally, the third interpolation pass is restricted to a minimum of four composites, a maximum of 12 composites (with a maximum of three composites per hole) without quadrant declustering.

Since the skarn mineralization and alteration system is driving the copper mineralization and the clays alteration at Rosemont, the copper interpolation plans were used to interpolate the magnesium and swelling clays content in every block using the ore types code matching between the composites and the blocks.

The swelling and magnesium clays were interpolated using the same multi pass system as describe above. At the end of the interpolation runs, some blocks located in small pods of ore type 2, 3, 4, 5A and 5B were left un-interpolated since they did not meet the interpolation requirements for the number of composites. In order to interpolate these isolated blocks, two additional interpolation pass had to be used. One used a minimum of two composites to interpolate the swelling and magnesium clays and the another one used a minimum of one composite.

The ordinary kriging (“OK”) interpolation results were validated against a NN model and an IDW model. The three models show similar values, hence giving confidence in the clays interpolation.

### TABLE 14-20: COPPER AND ACID SOLUBLE COPPER GRADE MODEL INTERPOLATION PLANS

| | | | | | Pass 1 | | | Pass 2 | | | Pass 3 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Copper | Oxidation | ROT1 2 | ROT2 2 | ROT3 2 | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis |
| Footwall Group | Oxide | -4 | 13 | -16 | 200 | 150 | 75 | 400 | 300 | 150 | 800 | 600 | 300 |
| | Mixed | 29 | 33 | -51 | 200 | 100 | 75 | 300 | 300 | 150 | 800 | 600 | 300 |
| | Sulfides | 2 | -6 | 68 | 200 | 75 | 150 | 150 | 150 | 400 | 200 | 530 | 800 |
| Lower Plate Group | Oxide | -4 | 13 | -16 | 200 | 150 | 75 | 400 | 300 | 150 | 800 | 600 | 300 |
| | Mixed | 29 | 33 | -51 | 200 | 100 | 75 | 400 | 200 | 150 | 800 | 600 | 300 |
| | Sulfides | -2 | -10 | 27 | 200 | 75 | 150 | 400 | 150 | 200 | 800 | 300 | 600 |
| Upper Plate Group | Oxide | -4 | 13 | -16 | 200 | 150 | 290 | 400 | 300 | 150 | 800 | 600 | 300 |
| | Mixed | 53 | 19 | -22 | 150 | 200 | 75 | 300 | 400 | 150 | 600 | 800 | 300 |
| | Sulfides | -71 | 7 | 82 | 200 | 75 | 150 | 400 | 150 | 300 | 800 | 300 | 600 |

| | | | | | Pass 1 | | | Pass 2 | | | Pass 3 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Soluble Copper | Oxidation | ROT1 2 | ROT2 2 | ROT3 2 | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis |
| Footwall Group | Oxide | -11 | 12 | -28 | 200 | 100 | 75 | 400 | 300 | 150 | 800 | 600 | 300 |
| | Mixed | -45 | 15 | -23 | 200 | 75 | 150 | 400 | 150 | 400 | 800 | 300 | 600 |
| | Sulfides | -57 | 29 | 63 | 200 | 75 | 200 | 400 | 150 | 400 | 800 | 300 | 800 |
| Lower Plate Group | Oxide | -11 | 12 | -28 | 200 | 100 | 75 | 400 | 200 | 100 | 800 | 600 | 300 |
| | Mixed | -45 | 15 | -23 | 200 | 75 | 150 | 400 | 200 | 400 | 800 | 300 | 600 |
| | Sulfides | 27 | -24 | 62 | 200 | 75 | 200 | 400 | 200 | 400 | 800 | 300 | 800 |
| Upper Plate Group | Oxide | -11 | 12 | -28 | 200 | 200 | 200 | 400 | 200 | 150 | 800 | 600 | 300 |
| | Mixed | -45 | 15 | -23 | 100 | 75 | 75 | 400 | 150 | 400 | 800 | 300 | 600 |
| | Sulfides | -57 | 29 | 63 | 75 | 150 | 200 | 400 | 150 | 400 | 800 | 300 | 800 |

*Note: Ranges are in feet and search ellipse orientations are given using MEDS rotation convention.*

## TABLE 14-21: MOLYBDENUM AND SILVER GRADE MODEL INTERPOLATION PLANS

| | | | | | Pass 1 | | | Pass 2 | | | Pass 3 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Molybdenum | Oxidation | ROT1 2 | ROT2 2 | ROT3 2 | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis |
| Footwall Group | Oxide | 53 | 147 | -72 | 75 | 100 | 200 | 150 | 300 | 400 | 165 | 345 | 800 |
| | Mixed | -29 | 11 | 92 | 200 | 100 | 75 | 400 | 300 | 150 | 800 | 500 | 250 |
| | Sulfides | -10 | 5 | 72 | 200 | 150 | 100 | 400 | 300 | 150 | 800 | 600 | 300 |
| Lower Plate Group | Oxide | 53 | 147 | -72 | 75 | 100 | 200 | 120 | 300 | 400 | 165 | 345 | 800 |
| | Mixed | -29 | 11 | 92 | 200 | 100 | 75 | 400 | 300 | 150 | 800 | 500 | 250 |
| | Sulfides | -10 | 5 | 72 | 200 | 150 | 100 | 400 | 300 | 150 | 800 | 600 | 500 |
| Upper Plate Group | Oxide | 53 | 147 | -72 | 75 | 100 | 200 | 120 | 400 | 400 | 165 | 345 | 800 |
| | Mixed | -29 | 11 | 92 | 200 | 100 | 75 | 300 | 300 | 300 | 800 | 500 | 250 |
| | Sulfides | -10 | 5 | 72 | 200 | 150 | 100 | 400 | 150 | 150 | 800 | 600 | 300 |

| | | | | | Pass 1 | | | Pass 2 | | | Pass 3 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Silver | Oxidation | ROT1 2 | ROT2 2 | ROT3 2 | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis | Major Axis | Semi Major Axis | Minor Axis |
| Footwall Group | Oxide | -26 | 24 | 81 | 200 | 100 | 150 | 400 | 150 | 300 | 800 | 300 | 600 |
| | Mixed | 61 | 62 | -140 | 200 | 150 | 100 | 400 | 300 | 150 | 800 | 570 | 300 |
| | Sulfides | -10 | -10 | 49 | 200 | 100 | 200 | 400 | 150 | 400 | 600 | 300 | 800 |
| Lower Plate Group | Oxide | -26 | 24 | 81 | 200 | 100 | 150 | 400 | 150 | 300 | 800 | 300 | 600 |
| | Mixed | 61 | 62 | -140 | 200 | 150 | 100 | 400 | 300 | 150 | 800 | 570 | 300 |
| | Sulfides | -23 | -4 | -2 | 200 | 100 | 200 | 400 | 150 | 300 | 800 | 300 | 500 |
| Upper Plate Group | Oxide | -26 | 24 | 81 | 200 | 100 | 150 | 400 | 400 | 400 | 800 | 300 | 300 |
| | Mixed | 61 | 62 | -140 | 200 | 150 | 100 | 150 | 300 | 150 | 800 | 570 | 300 |
| | Sulfides | -10 | -10 | 49 | 200 | 100 | 200 | 300 | 150 | 400 | 600 | 300 | 800 |

*Note: Ranges are in feet and search ellipse orientations are given using MEDS rotation convention.*

## 14.8 Tonnage Factor Assignment

There are a total of 2,486 specific gravity (SG) measurements in the drill hole database, including 2,066 Hudbay measurements with matching full geochemistry analysis. The pre-Hudbay specific gravity measurements were performed using a non-wax sealed immersion technique. The Hudbay measurements were performed by Inspectorate laboratory using a wax-sealed immersion technique to measure the weight of each sample in air and in water.

The lithologies identified at the Rosemont deposit display variable density contrast between the chemical sediments such as the limestones and dolostones, and the siliclastic sediments and crystalline rocks. In order to circumvent the relative low number of specific gravity measurements available, two linear regression models were developed based on the Hudbay specific gravity measurements and the geochemistry data. One linear regression model was fitted for the chemical sediments while the other model was adapted to the siliclastic sediments and crystalline rocks.

Table 14-22 presents the measured specific gravity measurements and the predicted specific measurements by decile and quantile.

**TABLE 14-22: MEASURED COMPARED TO CALCULATED SPECIFIC GRAVITY**

| Quantile | Measured SG | Predicted SG |
|---|---|---|
| 0% | 1.95 | 1.70 |
| 10% | 2.46 | 2.48 |
| 20% | 2.55 | 2.55 |
| 25% | 2.58 | 2.58 |
| 30% | 2.61 | 2.61 |
| 40% | 2.64 | 2.66 |
| 50% | 2.68 | 2.70 |
| 60% | 2.71 | 2.74 |
| 70% | 2.76 | 2.78 |
| 75% | 2.80 | 2.81 |
| 80% | 2.85 | 2.84 |
| 90% | 3.00 | 2.90 |
| 100% | 3.77 | 3.40 |

Missing SG measurements in the Hudbay drill holes were replaced by the calculated SG linear regression models. Table 14-23 shows the number of samples with density measurements and their basic statistics.

## TABLE 14-23: SPECIFIC GRAVITY MEASUREMENTS PER LITHOLOGY AND OXIDATION STATE

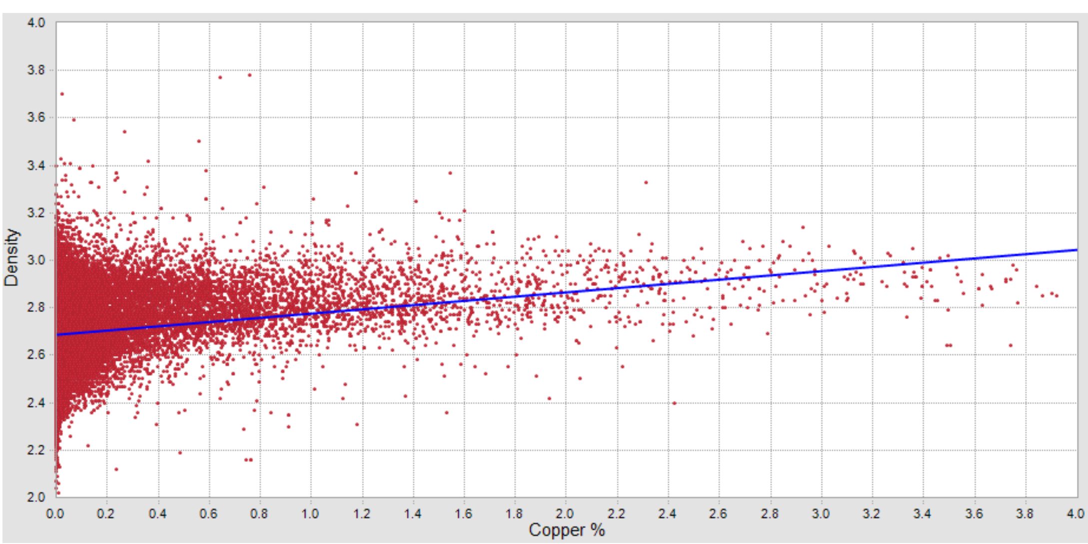
| Oxide | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 0 | 54 | 15 | 102 | 210 | 761 | 209 | 47 | 505 | 427 | 114 | 47 | 969 | 138 | 649 |
| Missing Values | 39 | 297 | 302 | 285 | 480 | 1635 | 223 | 164 | 394 | 858 | 227 | 10 | 1541 | 405 | 444 |
| Minimum | | 2.52 | 2.76 | 2.43 | 2.45 | 2.01 | 2.45 | 2.27 | 2.44 | 2.36 | 2.35 | 2.21 | 2.13 | 2.36 | 2.11 |
| Maximum | | 3.4 | 3 | 3.01 | 3.2 | 3.33 | 3.54 | 3.26 | 3.12 | 3.18 | 3.28 | 2.61 | 3.43 | 3.08 | 3.09 |
| Mean | | 2.637 | 2.851 | 2.713 | 2.712 | 2.778 | 2.844 | 2.738 | 2.766 | 2.787 | 2.689 | 2.387 | 2.599 | 2.625 | 2.562 |
| 1st Quartile | | 2.56 | 2.82 | 2.64 | 2.65 | 2.69 | 2.73 | 2.56 | 2.68 | 2.72 | 2.47 | 2.35 | 2.54 | 2.56 | 2.49 |
| Median | | 2.61 | 2.84 | 2.7 | 2.7 | 2.78 | 2.82 | 2.73 | 2.8 | 2.77 | 2.72 | 2.38 | 2.59 | 2.63 | 2.52 |
| 3rd Quartile | | 2.67 | 2.89 | 2.78 | 2.76 | 2.86 | 2.95 | 2.91 | 2.86 | 2.85 | 2.85 | 2.42 | 2.64 | 2.68 | 2.58 |
| Std. Devn. | | 0.16 | 0.066 | 0.098 | 0.09 | 0.121 | 0.172 | 0.228 | 0.135 | 0.13 | 0.206 | 0.059 | 0.124 | 0.095 | 0.135 |
| Variance | | 0.026 | 0.004 | 0.01 | 0.008 | 0.015 | 0.029 | 0.052 | 0.018 | 0.017 | 0.043 | 0.003 | 0.015 | 0.009 | 0.018 |
| Co. of Variation | | 0.061 | 0.023 | 0.036 | 0.033 | 0.044 | 0.06 | 0.083 | 0.049 | 0.047 | 0.077 | 0.025 | 0.048 | 0.036 | 0.053 |

| Mixed | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Valid | 7 | 72 | 32 | 96 | 16 | 284 | 203 | 176 | 254 | 359 | 25 | 10 | 1,701 | 139 | 219 |
| Rejected | 17 | 90 | 98 | 212 | 115 | 759 | 250 | 222 | 464 | 281 | 21 | 172 | 2665 | 216 | 335 |
| Minimum | 2.54 | 2.39 | 2.53 | 2.46 | 2.54 | 2.3 | 2.49 | 2.39 | 2.21 | 2.41 | 2.4 | 2.23 | 1.97 | 2.42 | 2.35 |
| Maximum | 2.59 | 2.86 | 2.87 | 2.96 | 2.91 | 3.26 | 3.32 | 3.07 | 2.98 | 3.19 | 2.95 | 2.8 | 3.16 | 2.95 | 2.87 |
| Mean | 2.55 | 2.579 | 2.691 | 2.711 | 2.736 | 2.811 | 2.879 | 2.646 | 2.689 | 2.709 | 2.679 | 2.375 | 2.578 | 2.67 | 2.543 |
| 1st Quartile | 2.54 | 2.53 | 2.62 | 2.64 | 2.68 | 2.72 | 2.77 | 2.55 | 2.59 | 2.64 | 2.44 | 2.26 | 2.51 | 2.6 | 2.48 |
| Median | 2.54 | 2.59 | 2.66 | 2.7 | 2.7 | 2.8 | 2.87 | 2.62 | 2.72 | 2.7 | 2.76 | 2.36 | 2.58 | 2.66 | 2.53 |
| 3rd Quartile | 2.55 | 2.62 | 2.71 | 2.79 | 2.78 | 2.9 | 2.98 | 2.74 | 2.8 | 2.76 | 2.83 | 2.39 | 2.64 | 2.73 | 2.57 |
| Std. Devn. | 0.018 | 0.078 | 0.101 | 0.11 | 0.096 | 0.14 | 0.145 | 0.146 | 0.14 | 0.116 | 0.194 | 0.162 | 0.102 | 0.084 | 0.095 |
| Variance | 0 | 0.006 | 0.01 | 0.012 | 0.009 | 0.02 | 0.021 | 0.021 | 0.02 | 0.013 | 0.038 | 0.026 | 0.01 | 0.007 | 0.009 |
| Co. of Variation | 0.007 | 0.03 | 0.037 | 0.041 | 0.035 | 0.05 | 0.05 | 0.055 | 0.052 | 0.043 | 0.072 | 0.068 | 0.04 | 0.031 | 0.037 |

| Sulfides | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Assays | 336 | 124 | 237 | 1,120 | 1,993 | 7,046 | 1,662 | 2,084 | 4,607 | 940 | 414 | 912 | 4,063 | 1,012 | 856 |
| Missing Values | 307 | 529 | 374 | 965 | 1200 | 11462 | 3603 | 3195 | 9238 | 1181 | 1126 | 871 | 6778 | 2035 | 640 |
| Minimum | 2.41 | 2.44 | 2.48 | 1.7 | 1.8 | 1.59 | 2.33 | 2.34 | 2.16 | 1.95 | 2.36 | 2.02 | 2.04 | 2.16 | 2.22 |
| Maximum | 2.85 | 2.92 | 3.2 | 3.37 | 3.43 | 3.7 | 3.78 | 3.17 | 3.59 | 3.4 | 3.24 | 2.74 | 3.77 | 3.09 | 3.17 |
| Mean | 2.583 | 2.609 | 2.745 | 2.723 | 2.719 | 2.817 | 2.83 | 2.674 | 2.735 | 2.747 | 2.708 | 2.379 | 2.545 | 2.698 | 2.622 |
| 1st Quartile | 2.55 | 2.54 | 2.65 | 2.65 | 2.64 | 2.73 | 2.73 | 2.61 | 2.67 | 2.67 | 2.57 | 2.33 | 2.46 | 2.65 | 2.5 |
| Median | 2.57 | 2.59 | 2.71 | 2.71 | 2.7 | 2.82 | 2.84 | 2.67 | 2.74 | 2.72 | 2.73 | 2.39 | 2.53 | 2.71 | 2.59 |
| 3rd Quartile | 2.61 | 2.65 | 2.87 | 2.79 | 2.78 | 2.9 | 2.93 | 2.73 | 2.81 | 2.81 | 2.83 | 2.42 | 2.61 | 2.76 | 2.73 |
| Std. Devn. | 0.062 | 0.09 | 0.132 | 0.132 | 0.134 | 0.127 | 0.146 | 0.106 | 0.123 | 0.136 | 0.174 | 0.084 | 0.117 | 0.104 | 0.164 |
| Variance | 0.004 | 0.008 | 0.017 | 0.017 | 0.018 | 0.016 | 0.021 | 0.011 | 0.015 | 0.019 | 0.03 | 0.007 | 0.014 | 0.011 | 0.027 |
| Co. of Variation | 0.024 | 0.034 | 0.048 | 0.049 | 0.049 | 0.045 | 0.051 | 0.04 | 0.045 | 0.05 | 0.064 | 0.035 | 0.046 | 0.039 | 0.063 |

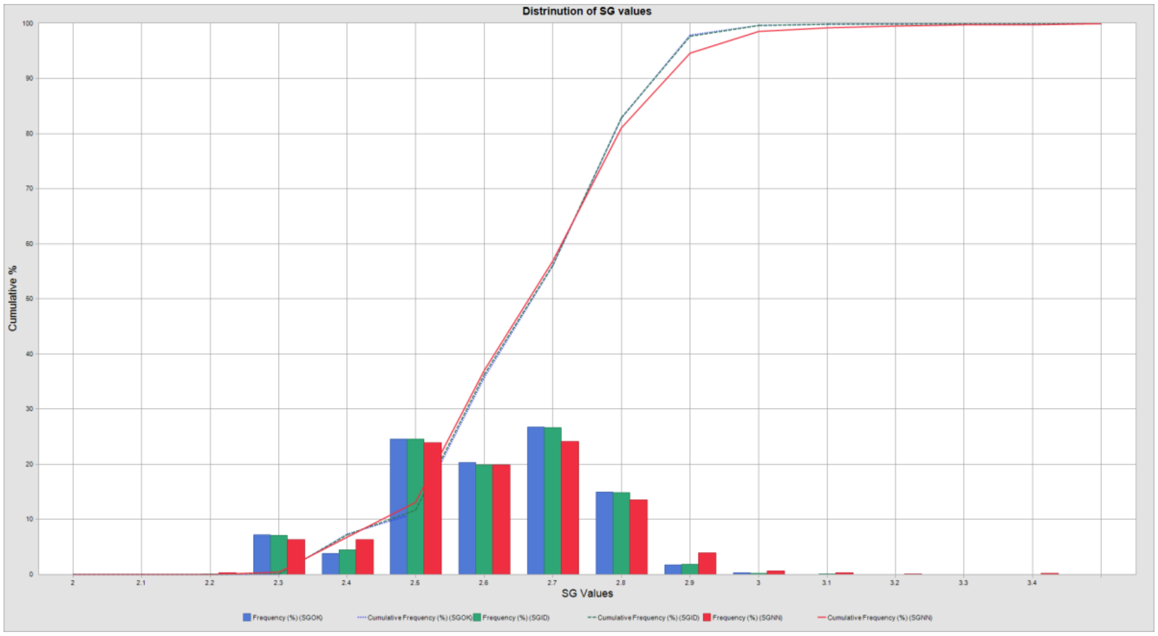
Since the skarn mineralization and alteration system was driving the copper mineralization and therefore the density, the copper interpolation plans were used to interpolate the specific density in every block using the lithology and oxidation code matching between the composites and the blocks. Figure 14-19 displays the relationship between the copper grade and the density values.

## FIGURE 14-19: SCATTERPLOT OF TOTAL COPPER AND SPECIFIC GRAVITY



The OK interpolation results were validated against a NN model and an IDW model. The three models show a similar distribution (Figure 14-20), confirming the absence of a global bias. The blocks that did not have a SG value after the interpolation were assigned an average SG value based on lithology and oxidation level (Table 14-24).

**FIGURE 14-20: OK, IDW AND NN SPECIFIC GRAVITY DISTRIBUTION**




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

**TABLE 14-24: SG BASELINE VALUES PER LITHOLOGY AND OXIDATION LEVEL**

| | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | 2.57 | 2.64 | 2.85 | 2.71 | 2.71 | 2.78 | 2.84 | 2.74 | 2.77 | 2.79 | 2.69 | 2.39 | 2.6 | 2.63 | 2.56 |
| Mixed | 2.55 | 2.58 | 2.69 | 2.71 | 2.74 | 2.81 | 2.88 | 2.65 | 2.69 | 2.71 | 2.68 | 2.38 | 2.58 | 2.67 | 2.54 |
| Sulfides | 2.58 | 2.61 | 2.75 | 2.72 | 2.72 | 2.82 | 2.83 | 2.67 | 2.74 | 2.75 | 2.71 | 2.38 | 2.55 | 2.7 | 2.62 |

Tonnage factors were calculated from the SG values using the formula TF = 2,000 / (SG * 62.42797).

The final tonnage factors are shown below in Table 14-25. The tonnage factors have been used directly as the dry bulk tonnage factors to report the tonnage estimates of the mineral resource.

## TABLE 14-25: TONNAGE FACTORS BY LITHOLOGY AND OXIDATION STATE

| Oxide | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Minimum | 12.00 | 9.59 | 9.56 | 10.86 | 10.50 | 10.54 | 10.61 | 11.69 | 10.97 | 10.90 | 10.54 | 13.40 | 10.24 | 10.75 | 11.32 |
| Maximum | 12.47 | 12.42 | 12.18 | 12.42 | 12.27 | 14.30 | 11.87 | 12.23 | 12.51 | 12.27 | 12.87 | 13.40 | 13.29 | 12.71 | 13.24 |
| Mean | 12.47 | 11.88 | 11.19 | 11.80 | 11.70 | 11.50 | 11.25 | 11.76 | 11.58 | 11.46 | 11.70 | 13.40 | 12.26 | 12.07 | 12.58 |

| Mixed | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Minimum | 12.09 | 11.48 | 11.44 | 11.78 | 11.69 | 10.97 | 10.68 | 11.16 | 11.09 | 10.57 | 11.95 | 13.46 | 11.16 | 11.61 | 12.00 |
| Maximum | 12.56 | 12.47 | 12.56 | 12.56 | 11.69 | 11.82 | 11.52 | 12.71 | 12.71 | 12.37 | 11.95 | 13.46 | 13.24 | 12.32 | 12.92 |
| Mean | 12.56 | 12.41 | 11.93 | 11.83 | 11.69 | 11.38 | 11.13 | 12.01 | 11.93 | 11.83 | 11.95 | 13.46 | 12.38 | 11.99 | 12.54 |

| Sulfides | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Minimum | 10.97 | 11.09 | 11.05 | 10.90 | 10.61 | 10.04 | 9.92 | 11.12 | 10.37 | 10.07 | 11.09 | 12.66 | 11.05 | 11.05 | 10.93 |
| Maximum | 12.66 | 12.66 | 12.42 | 12.71 | 12.71 | 13.29 | 12.81 | 12.87 | 12.97 | 13.69 | 12.87 | 14.18 | 13.40 | 12.87 | 13.35 |
| Mean | 12.40 | 12.25 | 11.66 | 11.80 | 11.67 | 11.32 | 11.35 | 11.95 | 11.77 | 11.65 | 11.78 | 13.46 | 12.54 | 11.81 | 12.21 |

## 14.9 Block Model Validation

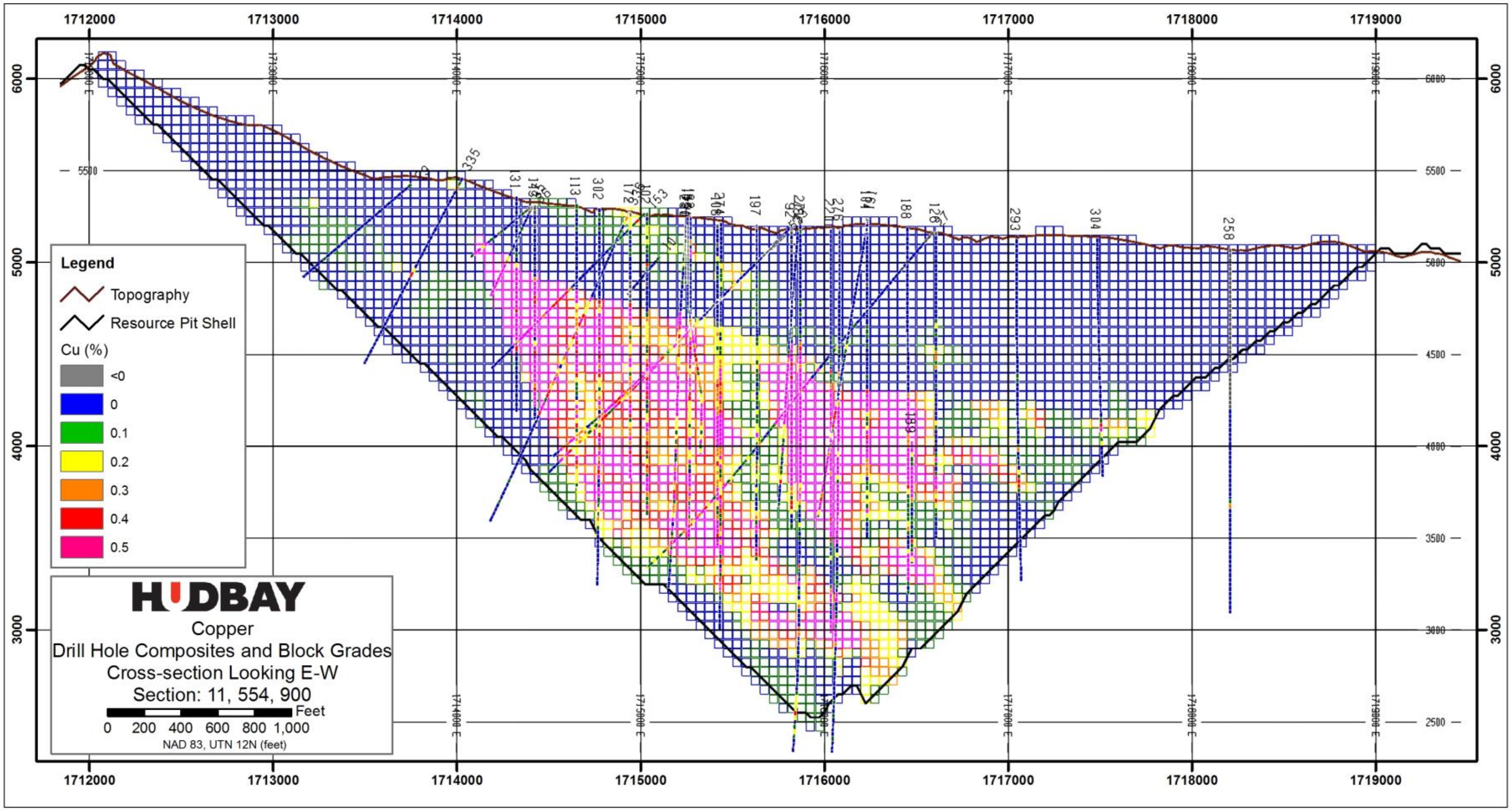
The Rosemont block model was validated to ensure appropriate honoring of the input data by the following methods:

- Visual inspection of the OK block model grades in plan and section views in comparison to composites grade
- Metal removed via grade capping methodology
- Comparison between the interpolation methods of NN using 50-foot composites and IDW were created 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 OK quality control parameters
- Comparison of the grade tonnage curves and statistics by estimation method

## 14.10 Visual Inspection

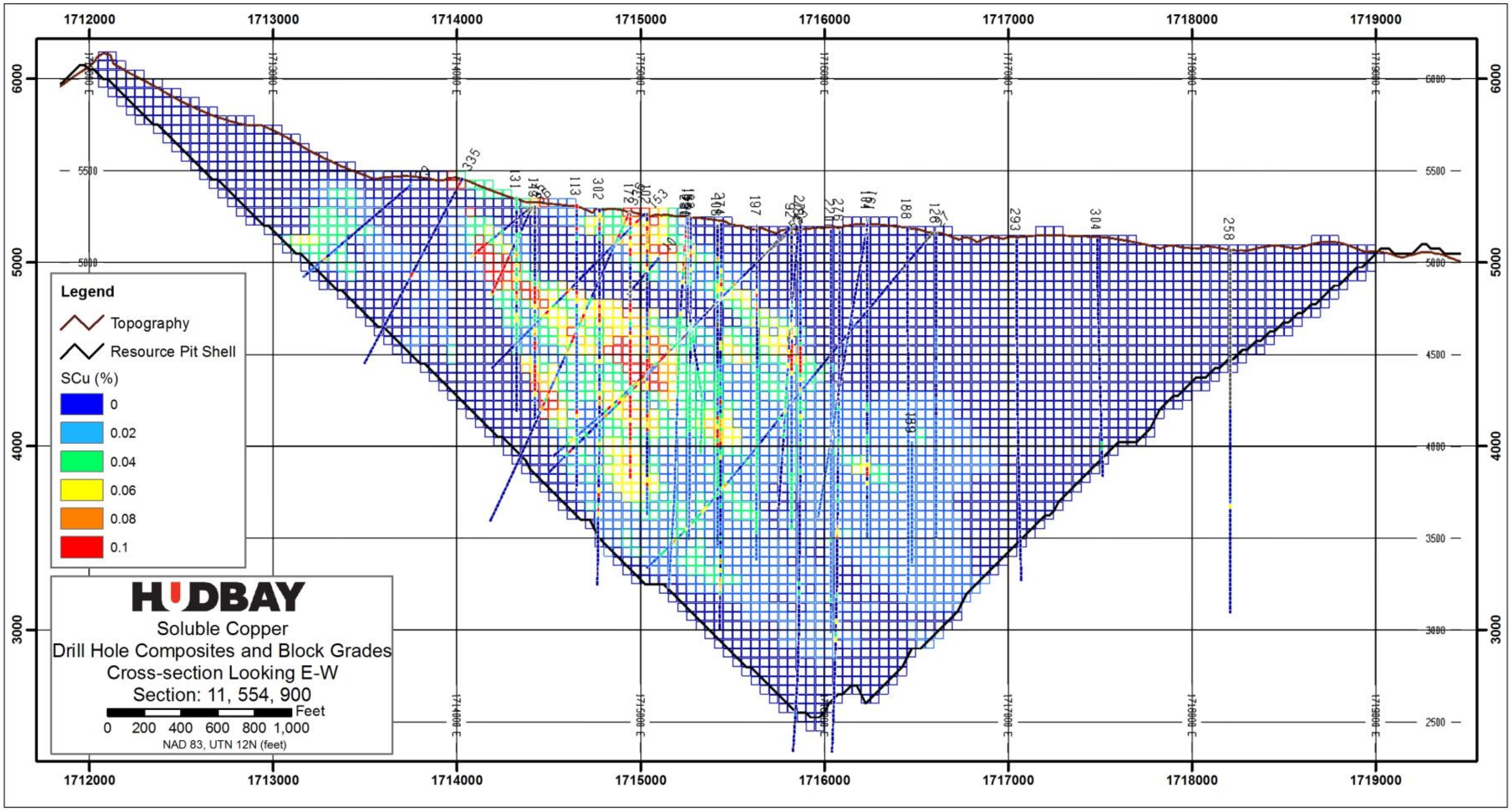
Visual inspection of block grade versus composited data was conducted in section and plan view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model. An east-west oriented cross-section is provided in Figure 14-21 to Figure 14-24.

**FIGURE 14-21: VERTICAL E-W SECTION 11,554,900 SHOWING OK MODEL AND COMPOSITES - COPPER GRADE**



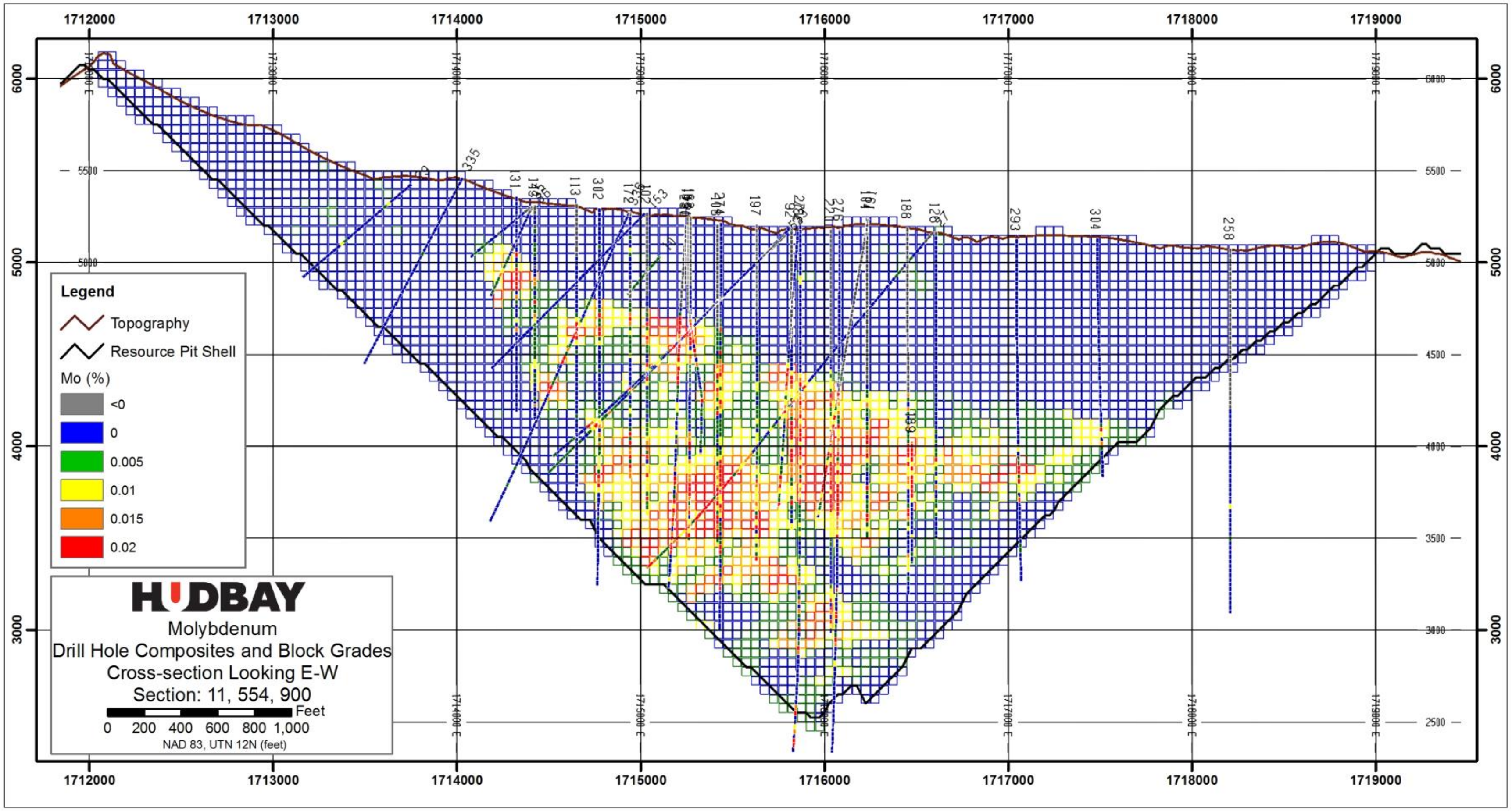

*Note: The resource pit is not indicative of the mine plan*

**FIGURE 14-22: VERTICAL E-W SECTION 11,554,900 SHOWING OK MODEL AND COMPOSITES – ACID SOLUBLE COPPER GRADE**



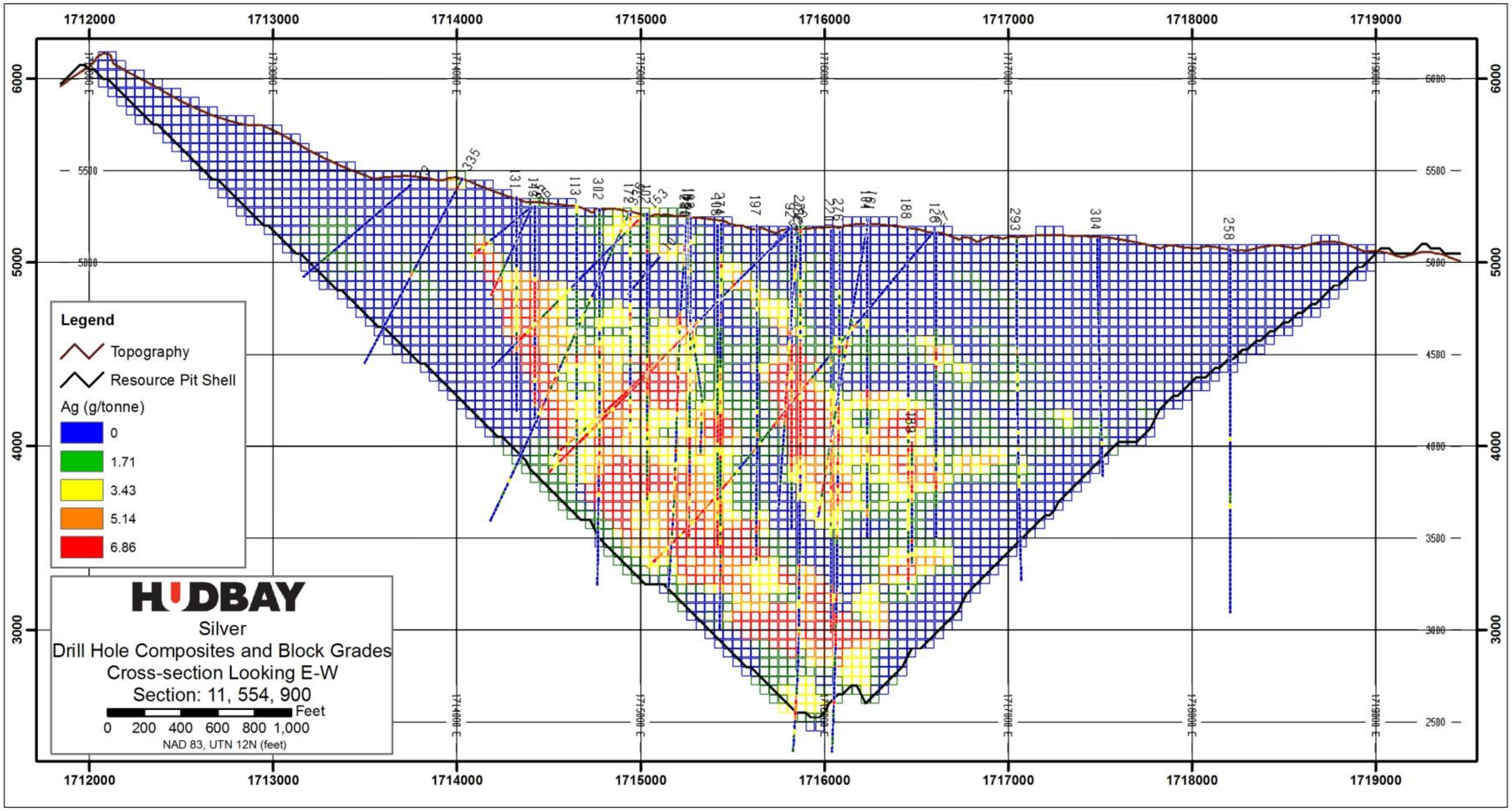

*Note: The resource pit is not indicative of the mine plan.*

**FIGURE 14-23: VERTICAL E-W SECTION 11,554,900 SHOWING OK MODEL AND COMPOSITES - MOLYBDENUM GRADE**




*Note: The resource pit is not indicative of the mine plan.*

**FIGURE 14-24: VERTICAL E-W SECTION 11,554,900 SHOWING OK MODEL AND COMPOSITES – SILVER GRADE**




*Note: The resource pit is not indicative of the mine plan.*

## 14.11 Metal Removed by Capping

The impact of 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 difference of the capped and uncapped assays. The percentages of metal removed by capping from the assays, NN, IDW and OK models in the blocks above $5.7/ton NSR contained within the resource pit are shown in Table 14-26 to Table 14-29. The amount of capping appears appropriate within the Horquilla, Earp (upper and lower) and Epitaph with a difference of approximately 0 to 3% for copper, 0 to 13% for ASCu, 4 to 22% for molybdenum and 1 to 8% for silver.

### TABLE 14-26: ASSAY, NN, IDW AND OK MODEL, COPPER REMOVED BY CAPPING IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Copper | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | Assays | -3% | -8% | -9% | -6% | -33% | -1% | -4% | -3% | -1% | -10% | -40% | | -3% | -4% | -4% |
| | NN | -8% | -9% | -8% | -7% | -21% | -1% | -3% | -1% | -1% | -13% | -70% | | -4% | -5% | -3% |
| | IDW | -11% | -12% | -9% | -5% | -21% | -2% | -4% | -1% | -1% | -11% | -63% | | -3% | -6% | -3% |
| | OK | -10% | -12% | -9% | -5% | -19% | -2% | -4% | -1% | -1% | -11% | -65% | | -3% | -6% | -2% |
| Mixed | Assays | 0% | -3% | 0% | 0% | | -8% | -15% | -29% | -2% | 0% | | | -4% | -28% | -8% |
| | NN | -2% | -7% | 0% | 0% | | -3% | -15% | -6% | -1% | 0% | | | -1% | -45% | -21% |
| | IDW | -4% | -8% | -3% | 0% | | -5% | -11% | -9% | -2% | 0% | | | -13% | -37% | -17% |
| | OK | -3% | -7% | -8% | 0% | | -5% | -10% | -10% | -3% | 0% | | | -11% | -37% | -13% |
| Sulfides | Assays | -1% | -11% | -21% | -24% | -27% | *-3%* | *-1%* | *0%* | *-2%* | -14% | -20% | | -6% | -8% | -10% |
| | NN | -13% | -9% | -8% | -21% | -15% | *-3%* | *0%* | *0%* | *-1%* | -20% | -14% | | -17% | -7% | -10% |
| | IDW | -17% | -9% | -12% | -26% | -15% | *-3%* | *0%* | *0%* | *-2%* | -19% | -17% | | -15% | -8% | -8% |
| | OK | -17% | -10% | -13% | -24% | -16% | *-3%* | *0%* | *0%* | *-2%* | -21% | -16% | | -14% | -8% | -9% |

### TABLE 14-27: ASSAY, NN, IDW AND OK MODEL, ACID SOLUBLE COPPER REMOVED BY CAPPING IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Soluble Copper | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | Assays | 0% | -10% | -28% | -2% | -4% | -9% | -8% | -4% | -14% | 0% | | | -20% | -26% | -18% |
| | NN | -8% | -5% | -12% | -9% | -28% | -2% | -2% | -2% | -2% | -7% | | | -4% | -8% | -5% |
| | IDW | -15% | -13% | -11% | -7% | -24% | -3% | -4% | -2% | -2% | -5% | | | -4% | -6% | -4% |
| | OK | -14% | -13% | -11% | -7% | -20% | -3% | -3% | -2% | -2% | -5% | | | -4% | -6% | -3% |
| Mixed | Assays | 0% | -30% | -21% | 0% | 0% | -16% | -12% | -7% | -9% | 0% | | | -27% | 0% | -8% |
| | NN | -17% | -22% | -8% | -2% | | -2% | -10% | -4% | 0% | 0% | | | -13% | -32% | 0% |
| | IDW | -13% | -15% | -9% | -1% | | -5% | -1% | -6% | -7% | 0% | | | -13% | -32% | -6% |
| | OK | -6% | -10% | -10% | -1% | | -8% | -1% | -7% | -7% | 0% | | | -12% | -35% | -8% |
| Sulfides | Assays | -14% | -17% | -32% | -15% | -33% | *-5%* | *-13%* | *-4%* | *-13%* | -3% | -49% | | -12% | -11% | -16% |
| | NN | -7% | -23% | -7% | -3% | -8% | *-4%* | *-1%* | *0%* | *-2%* | -15% | -3% | | -4% | -7% | -5% |
| | IDW | -10% | -14% | -18% | -10% | -8% | *-5%* | *-1%* | *-1%* | *-2%* | -13% | -5% | | -13% | -7% | -5% |
| | OK | -15% | -16% | -26% | -15% | -7% | *-6%* | *-1%* | *-1%* | *-2%* | -12% | -4% | | -11% | -7% | -6% |

## TABLE 14-28: ASSAY, NN, IDW AND OK MODEL, MOLYBDENUM REMOVED BY CAPPING IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Molybdenum | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Oxide** | Assays | 0% | -10% | -28% | -2% | -4% | -9% | -8% | -4% | -14% | 0% | 0% | | -20% | -26% | -18% |
| | NN | -4% | -10% | -15% | 0% | -7% | -7% | -3% | -11% | -6% | 0% | 0% | | -27% | -35% | -2% |
| | IDW | -4% | -11% | -17% | 0% | -5% | -8% | -4% | -7% | -7% | 0% | 0% | | -26% | -32% | -3% |
| | OK | -5% | -15% | -17% | 0% | -5% | -9% | -4% | -7% | -6% | 0% | 0% | | -25% | -31% | -3% |
| **Mixed** | Assays | 0% | -30% | -21% | 0% | | -16% | -12% | -7% | -9% | 0% | | | -27% | 0% | -8% |
| | NN | -29% | -40% | -25% | 0% | | -5% | -10% | -8% | -16% | 0% | | | 0% | 0% | -11% |
| | IDW | -42% | -44% | -32% | 0% | | -9% | -10% | -10% | -14% | 0% | | | 0% | 0% | -6% |
| | OK | -38% | -42% | -30% | 0% | | -10% | -10% | -10% | -12% | 0% | | | 0% | 0% | -5% |
| **Sulfides** | Assays | -14% | -17% | -32% | -15% | -33% | *-5%* | *-13%* | *-4%* | *-13%* | -3% | -49% | | -12% | -11% | -16% |
| | NN | -8% | -20% | -16% | -14% | -30% | *-5%* | *-21%* | *-9%* | *-13%* | -2% | -43% | | -32% | -7% | -18% |
| | IDW | -12% | -27% | -20% | -17% | -34% | *-4%* | *-22%* | *-7%* | *-15%* | -3% | -49% | | -31% | -7% | -18% |
| | OK | -12% | -30% | -19% | -17% | -36% | *-4%* | *-22%* | *-8%* | *-15%* | -3% | -49% | | -30% | -9% | -17% |

## TABLE 14-29: ASSAY, NN, IDW AND OK MODEL, SILVER REMOVED BY CAPPING IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Silver | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Oxide** | Assays | 0% | -4% | -10% | -9% | -27% | -10% | -22% | -4% | -9% | -10% | -20% | | -7% | -20% | -24% |
| | NN | -9% | -3% | -9% | -15% | -33% | -9% | -29% | 0% | -8% | -8% | -66% | | -5% | -19% | -6% |
| | IDW | -6% | -2% | -8% | -7% | -33% | -9% | -26% | -1% | -8% | -13% | -57% | | -5% | -16% | -4% |
| | OK | -6% | -3% | -9% | -7% | -34% | -11% | -20% | -4% | -7% | -13% | -60% | | -6% | -13% | -5% |
| **Mixed** | Assays | 0% | -35% | -7% | -36% | | -4% | -23% | 0% | -13% | 0% | | | -4% | -25% | -6% |
| | NN | -1% | -27% | -6% | 0% | | -1% | -30% | -2% | -3% | -17% | | | -6% | 0% | -16% |
| | IDW | -1% | -37% | -12% | -5% | | -2% | -24% | 0% | -11% | -15% | | | -28% | -1% | -10% |
| | OK | -2% | -43% | -19% | -4% | | -2% | -24% | -1% | -11% | -16% | | | -30% | -3% | -13% |
| **Sulfides** | Assays | -7% | -19% | -30% | -45% | -24% | *-3%* | *-7%* | *-1%* | *-5%* | -23% | -13% | | -13% | -5% | -7% |
| | NN | -47% | -23% | -34% | -47% | -14% | *-4%* | *-7%* | *-1%* | *-5%* | -31% | -4% | | -13% | -4% | -8% |
| | IDW | -54% | -19% | -32% | -48% | -17% | *-4%* | *-8%* | *-1%* | *-5%* | -32% | -5% | | -13% | -4% | -7% |
| | OK | -51% | -19% | -32% | -46% | -16% | *-4%* | *-8%* | *-1%* | *-5%* | -35% | -5% | | -13% | -4% | -8% |

## 14.12 Global Bias Checks

A comparison between the interpolation methods estimates was completed on all the blocks within the resource pit shell that have NSR values greater than $5.7/ton for global bias in the grade estimates. Differences between the NN, IDW and OK grades are acceptable in Horquilla, Earp (lower and upper) and Epitaph, with differences within 0% to 3% for copper, 0% to 3% for ASCu, 0% to 5% for molybdenum and 0% to 6% for silver. The differences are summarized in Table 14-30 to Table 14-33.

## TABLE 14-30: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR COPPER IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Copper | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | NN | 0.39 | 0.33 | 0.34 | 0.30 | 0.24 | 0.45 | 0.34 | 0.40 | 0.35 | 0.20 | 0.34 | | 0.22 | 0.24 | 0.23 |
| | IDW | 0.39 | 0.31 | 0.31 | 0.23 | 0.23 | 0.48 | 0.36 | 0.40 | 0.36 | 0.18 | 0.17 | | 0.22 | 0.25 | 0.24 |
| | OK | 0.42 | 0.33 | 0.32 | 0.25 | 0.24 | 0.47 | 0.36 | 0.41 | 0.35 | 0.18 | 0.18 | | 0.23 | 0.25 | 0.24 |
| | NN & OK % Difference | 6% | -1% | -7% | -16% | 0% | 3% | 5% | 1% | 2% | -7% | -45% | | 1% | 6% | 4% |
| | IDW & OK % Difference | 6% | 7% | 2% | 8% | 3% | -3% | -1% | 1% | -2% | 4% | 12% | | 2% | 3% | 2% |
| Mixed | NN | 0.61 | 0.43 | 0.25 | 0.29 | | 0.40 | 0.40 | 0.37 | 0.31 | 0.39 | | | 0.25 | 0.33 | 0.26 |
| | IDW | 0.52 | 0.43 | 0.29 | 0.28 | | 0.43 | 0.42 | 0.37 | 0.34 | 0.39 | | | 0.28 | 0.33 | 0.25 |
| | OK | 0.49 | 0.39 | 0.33 | 0.31 | | 0.44 | 0.41 | 0.38 | 0.33 | 0.34 | | | 0.26 | 0.33 | 0.25 |
| | NN & OK % Difference | -20% | -8% | 33% | 9% | | 10% | 2% | 4% | 7% | -13% | | | 4% | -1% | -5% |
| | IDW & OK % Difference | -6% | -9% | 12% | 11% | | 2% | -4% | 3% | -2% | -12% | | | -4% | -2% | -1% |
| Sulfides | NN | 0.36 | 0.29 | 0.36 | 0.28 | 0.27 | *0.36* | *0.34* | *0.25* | *0.40* | 0.22 | 0.22 | | 0.22 | 0.21 | 0.32 |
| | IDW | 0.30 | 0.29 | 0.37 | 0.28 | 0.30 | *0.36* | *0.35* | *0.24* | *0.41* | 0.19 | 0.24 | | 0.20 | 0.20 | 0.33 |
| | OK | 0.32 | 0.33 | 0.38 | 0.31 | 0.30 | *0.37* | *0.35* | *0.24* | *0.41* | 0.19 | 0.26 | | 0.21 | 0.20 | 0.33 |
| | NN & OK % Difference | -10% | 16% | 6% | 9% | 10% | *2%* | *3%* | *-1%* | *3%* | -13% | 16% | | -5% | -1% | 4% |
| | IDW & OK % Difference | 9% | 16% | 5% | 10% | 0% | *1%* | *0%* | *1%* | *0%* | -1% | 7% | | 3% | 4% | 0% |

## TABLE 14-31: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR ACID SOLUBLE COPPER IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Soluble Copper | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | NN | 0.262 | 0.171 | 0.222 | 0.178 | 0.149 | 0.304 | 0.223 | 0.260 | 0.207 | 0.119 | 0.273 | | 0.154 | 0.163 | 0.164 |
| | IDW | 0.253 | 0.171 | 0.203 | 0.134 | 0.148 | 0.327 | 0.234 | 0.259 | 0.219 | 0.099 | 0.134 | | 0.154 | 0.166 | 0.158 |
| | OK | 0.265 | 0.184 | 0.200 | 0.138 | 0.148 | 0.319 | 0.234 | 0.253 | 0.215 | 0.090 | 0.136 | | 0.152 | 0.169 | 0.158 |
| | NN & OK % Difference | 1% | 8% | -10% | -22% | -1% | 5% | 5% | -3% | 4% | -25% | -50% | | -1% | 4% | -4% |
| | IDW & OK % Difference | 5% | 8% | -1% | 4% | 1% | -3% | 0% | -3% | -2% | -9% | 2% | | -2% | 2% | 0% |
| Mixed | NN | 0.269 | 0.157 | 0.133 | 0.148 | | 0.138 | 0.060 | 0.191 | 0.042 | 0.208 | | | 0.174 | 0.195 | 0.059 |
| | IDW | 0.150 | 0.102 | 0.093 | 0.019 | | 0.127 | 0.062 | 0.123 | 0.065 | 0.166 | | | 0.152 | 0.203 | 0.061 |
| | OK | 0.132 | 0.088 | 0.098 | 0.022 | | 0.136 | 0.066 | 0.124 | 0.067 | 0.120 | | | 0.122 | 0.193 | 0.068 |
| | NN & OK % Difference | -51% | -44% | -26% | -85% | | -2% | 11% | -35% | 61% | -42% | | | -30% | -1% | 15% |
| | IDW & OK % Difference | -12% | -14% | 6% | 14% | | 7% | 7% | 1% | 4% | -27% | | | -20% | -5% | 13% |
| Sulfides | NN | 0.016 | 0.059 | 0.031 | 0.016 | 0.020 | *0.034* | *0.032* | *0.028* | *0.027* | 0.023 | 0.016 | | 0.032 | 0.036 | 0.039 |
| | IDW | 0.020 | 0.055 | 0.035 | 0.015 | 0.019 | *0.033* | *0.032* | *0.028* | *0.027* | 0.019 | 0.015 | | 0.021 | 0.031 | 0.041 |
| | OK | 0.022 | 0.053 | 0.045 | 0.017 | 0.019 | *0.034* | *0.032* | *0.028* | *0.027* | 0.019 | 0.015 | | 0.019 | 0.032 | 0.041 |
| | NN & OK % Difference | 44% | -11% | 44% | 4% | -6% | *1%* | *0%* | *1%* | *0%* | -15% | -1% | | -39% | -11% | 6% |
| | IDW & OK % Difference | 15% | -5% | 28% | 14% | -1% | *3%* | *1%* | *1%* | *1%* | 0% | 1% | | -5% | 3% | 1% |

## TABLE 14-32: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR MOLYBDENUM SILVER IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

| Molybdenum | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | NN | 0.005 | 0.003 | 0.005 | 0.002 | 0.005 | 0.005 | 0.003 | 0.005 | 0.004 | 0.004 | 0.003 | | 0.001 | 0.003 | 0.006 |
| | IDW | 0.002 | 0.002 | 0.004 | 0.002 | 0.005 | 0.005 | 0.003 | 0.002 | 0.004 | 0.002 | 0.003 | | 0.001 | 0.003 | 0.006 |
| | OK | 0.002 | 0.002 | 0.004 | 0.002 | 0.005 | 0.005 | 0.003 | 0.002 | 0.004 | 0.002 | 0.003 | | 0.001 | 0.003 | 0.006 |
| | NN & OK % Difference | -48% | -30% | -22% | -20% | -2% | -14% | -15% | -56% | -12% | -52% | 0% | | -10% | -16% | -1% |
| | IDW & OK % Difference | 0% | -1% | -2% | 1% | 0% | 0% | -1% | -1% | -2% | -8% | -7% | | -1% | -7% | -1% |
| Mixed | NN | 0.010 | 0.004 | 0.009 | 0.004 | | 0.009 | 0.011 | 0.008 | 0.013 | 0.015 | | | 0.001 | 0.007 | 0.021 |
| | IDW | 0.005 | 0.003 | 0.004 | 0.000 | | 0.009 | 0.013 | 0.009 | 0.010 | 0.011 | | | 0.001 | 0.005 | 0.020 |
| | OK | 0.005 | 0.003 | 0.004 | 0.000 | | 0.009 | 0.012 | 0.009 | 0.009 | 0.009 | | | 0.001 | 0.004 | 0.020 |
| | NN & OK % Difference | -54% | -29% | -62% | -93% | | -3% | 14% | 4% | -30% | -40% | | | 22% | -38% | -7% |
| | IDW & OK % Difference | -3% | -5% | 0% | -8% | | -1% | -2% | -1% | -6% | -24% | | | 14% | -14% | -3% |
| Sulfides | NN | 0.003 | 0.002 | 0.004 | 0.005 | 0.008 | *0.014* | *0.017* | *0.012* | *0.012* | 0.006 | 0.008 | | 0.002 | 0.002 | 0.012 |
| | IDW | 0.003 | 0.002 | 0.004 | 0.005 | 0.009 | *0.014* | *0.017* | *0.012* | *0.013* | 0.005 | 0.008 | | 0.002 | 0.002 | 0.013 |
| | OK | 0.003 | 0.002 | 0.004 | 0.005 | 0.009 | *0.014* | *0.016* | *0.012* | *0.013* | 0.005 | 0.008 | | 0.002 | 0.002 | 0.013 |
| | NN & OK % Difference | 9% | -11% | -2% | -9% | 9% | *0%* | *-5%* | *0%* | *3%* | -8% | 4% | | -22% | -15% | 5% |
| | IDW & OK % Difference | 9% | 7% | 2% | -3% | 1% | *0%* | *-2%* | *1%* | *0%* | 1% | 2% | | -2% | -4% | 2% |

## TABLE 14-33: NN, IDW AND OK MODEL STATISTICS MEAN BLOCK GRADE COMPARISONS FOR SILVER IN BLOCKS WITHIN THE RESOURCE PIT AND ABOVE $5.7/TON NSR

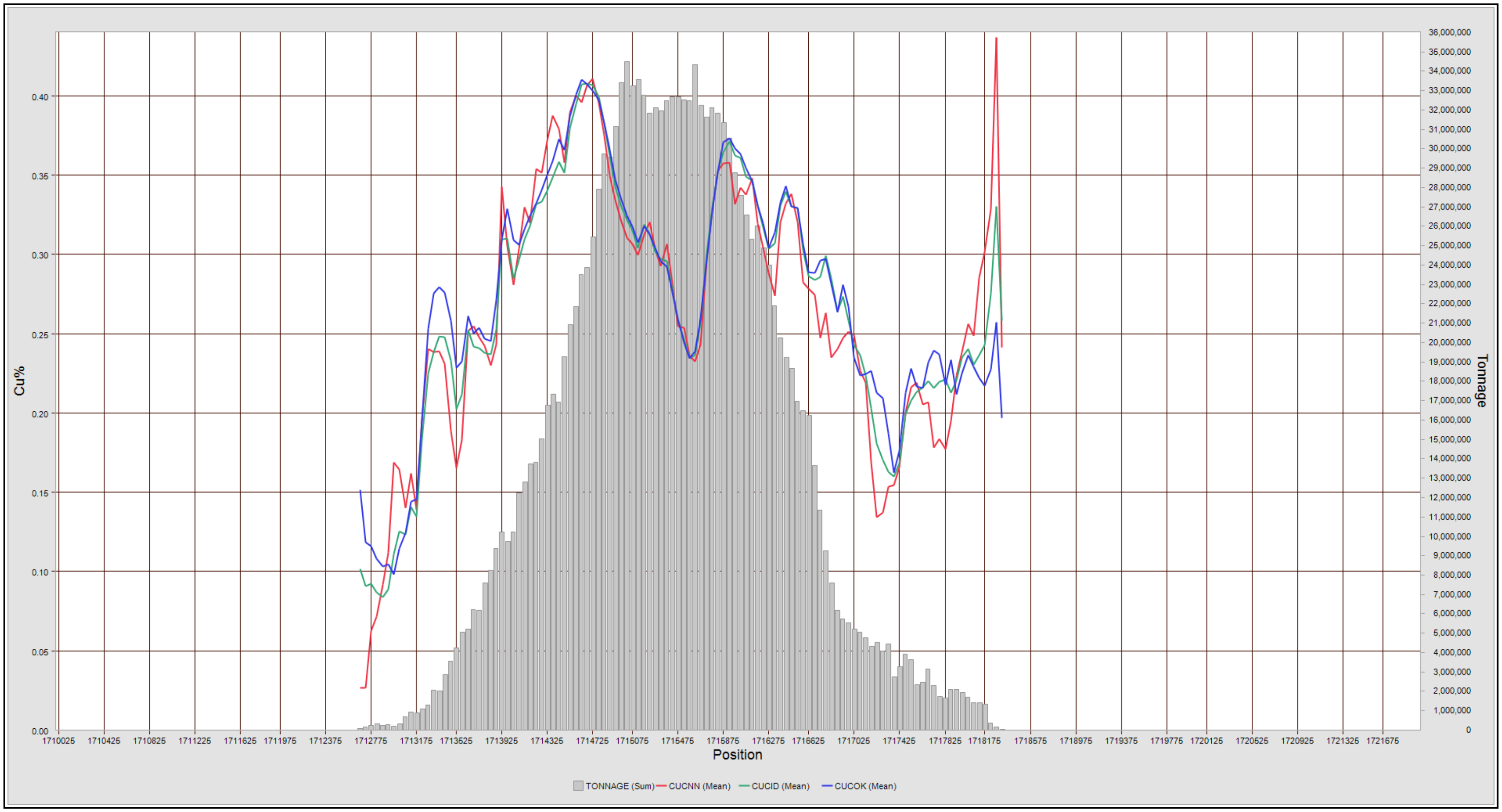
| Silver | | Granodiorite | Bolsa | Abrigo | Martin | Escabrosa | Horquilla | Earp (Lower) | Earp (Upper) | Epitaph | Scherrer | Glance | Gila | Arkose | Andesite | QMP |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Oxide | NN | 0.06 | 0.09 | 0.09 | 0.05 | 0.07 | 0.08 | 0.03 | 0.05 | 0.05 | 0.10 | 0.15 | | 0.13 | 0.06 | 0.04 |
| | IDW | 0.07 | 0.09 | 0.08 | 0.04 | 0.07 | 0.08 | 0.04 | 0.06 | 0.05 | 0.07 | 0.06 | | 0.14 | 0.08 | 0.03 |
| | OK | 0.09 | 0.09 | 0.07 | 0.04 | 0.06 | 0.08 | 0.04 | 0.06 | 0.04 | 0.06 | 0.09 | | 0.13 | 0.09 | 0.03 |
| | NN & OK % Difference | 54% | -2% | -15% | -18% | -7% | 2% | 22% | 9% | -4% | -33% | -40% | | 0% | 40% | -15% |
| | IDW & OK % Difference | 26% | -1% | -5% | 2% | -5% | 3% | -3% | -7% | -4% | -2% | 41% | | -1% | 4% | -2% |
| Mixed | NN | 0.12 | 0.07 | 0.09 | 0.07 | | 0.06 | 0.13 | 0.09 | 0.05 | 0.04 | | | 0.12 | 0.00 | 0.05 |
| | IDW | 0.05 | 0.03 | 0.05 | 0.04 | | 0.07 | 0.12 | 0.08 | 0.05 | 0.02 | | | 0.14 | 0.01 | 0.05 |
| | OK | 0.04 | 0.04 | 0.05 | 0.04 | | 0.07 | 0.12 | 0.08 | 0.05 | 0.02 | | | 0.14 | 0.01 | 0.05 |
| | NN & OK % Difference | -66% | -43% | -43% | -41% | | 29% | -7% | -8% | -2% | -44% | | | 11% | 209% | -10% |
| | IDW & OK % Difference | -13% | 19% | 12% | 18% | | 6% | 0% | 3% | -2% | 4% | | | -1% | 40% | -4% |
| Sulfides | NN | 0.07 | 0.04 | 0.08 | 0.09 | 0.11 | *0.14* | *0.12* | *0.06* | *0.09* | 0.06 | 0.07 | | 0.05 | 0.15 | 0.07 |
| | IDW | 0.06 | 0.04 | 0.08 | 0.08 | 0.09 | *0.14* | *0.12* | *0.06* | *0.09* | 0.06 | 0.08 | | 0.05 | 0.15 | 0.07 |
| | OK | 0.06 | 0.04 | 0.08 | 0.08 | 0.09 | *0.14* | *0.12* | *0.06* | *0.09* | 0.06 | 0.08 | | 0.05 | 0.14 | 0.07 |
| | NN & OK % Difference | -21% | 23% | 3% | -15% | -15% | *-1%* | *1%* | *-6%* | *0%* | -10% | 15% | | -13% | -7% | -3% |
| | IDW & OK % Difference | -2% | -1% | 2% | -2% | -3% | *0%* | *0%* | *0%* | *0%* | -6% | 3% | | -2% | -2% | -1% |

## 14.13 Local Bias Checks

A local bias check was performed by plotting the average total copper, acid soluble copper, molybdenum and silver of the NN, IDW and OK models in swath plots oriented along the model northing, easting and elevation.

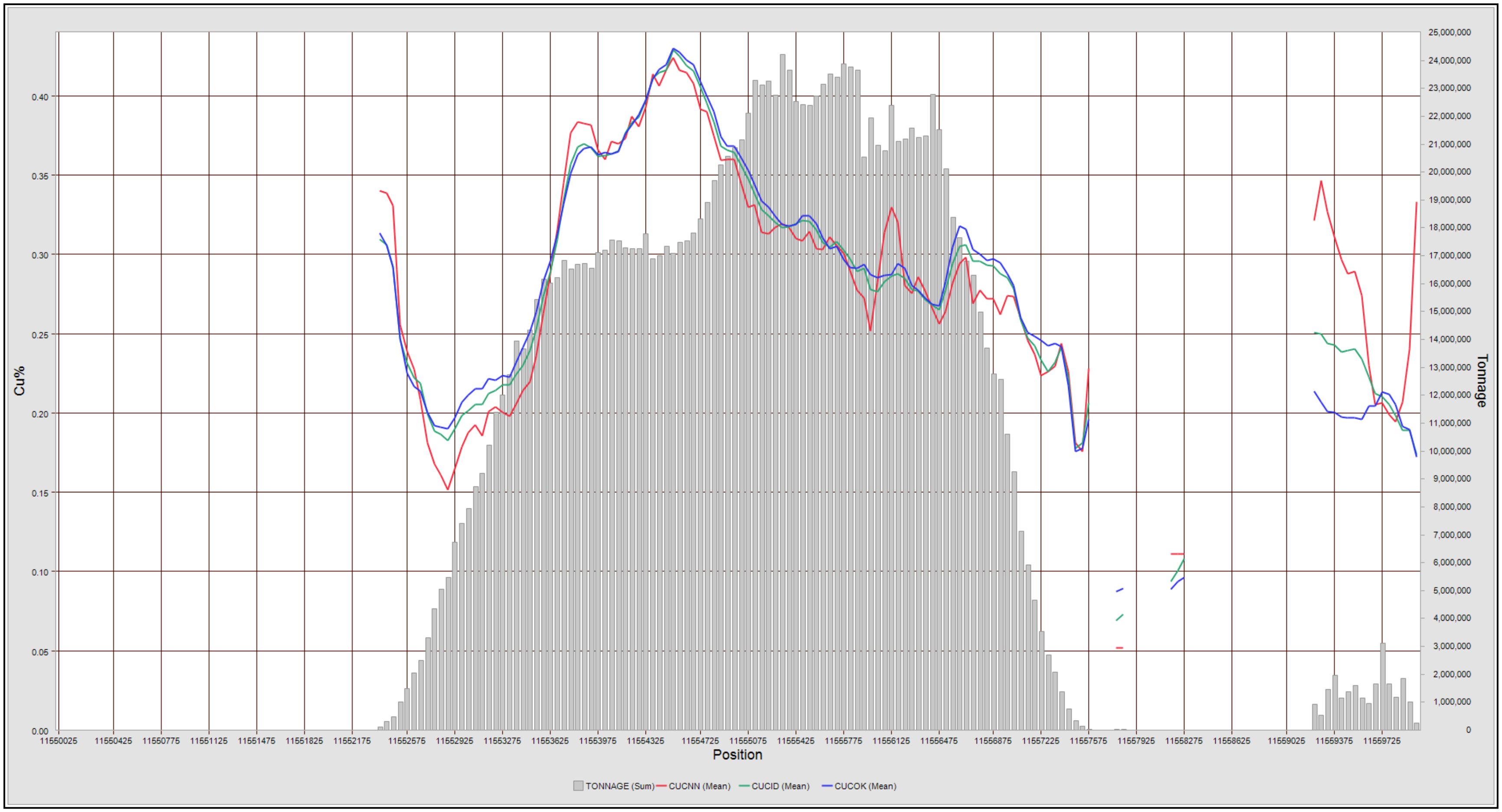
In reviewing the swath plots, only minor discrepancies were found between the different grade models. In areas where there is extrapolation beyond the drill holes, the swath plots indicate less agreement for all variables. The copper, acid soluble copper, molybdenum and silver swath plots for Measured and Indicated blocks are shown below in Figure 14-25 to Figure 14-36.

**FIGURE 14-25: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, COPPER SWATH PLOT BY EASTING**



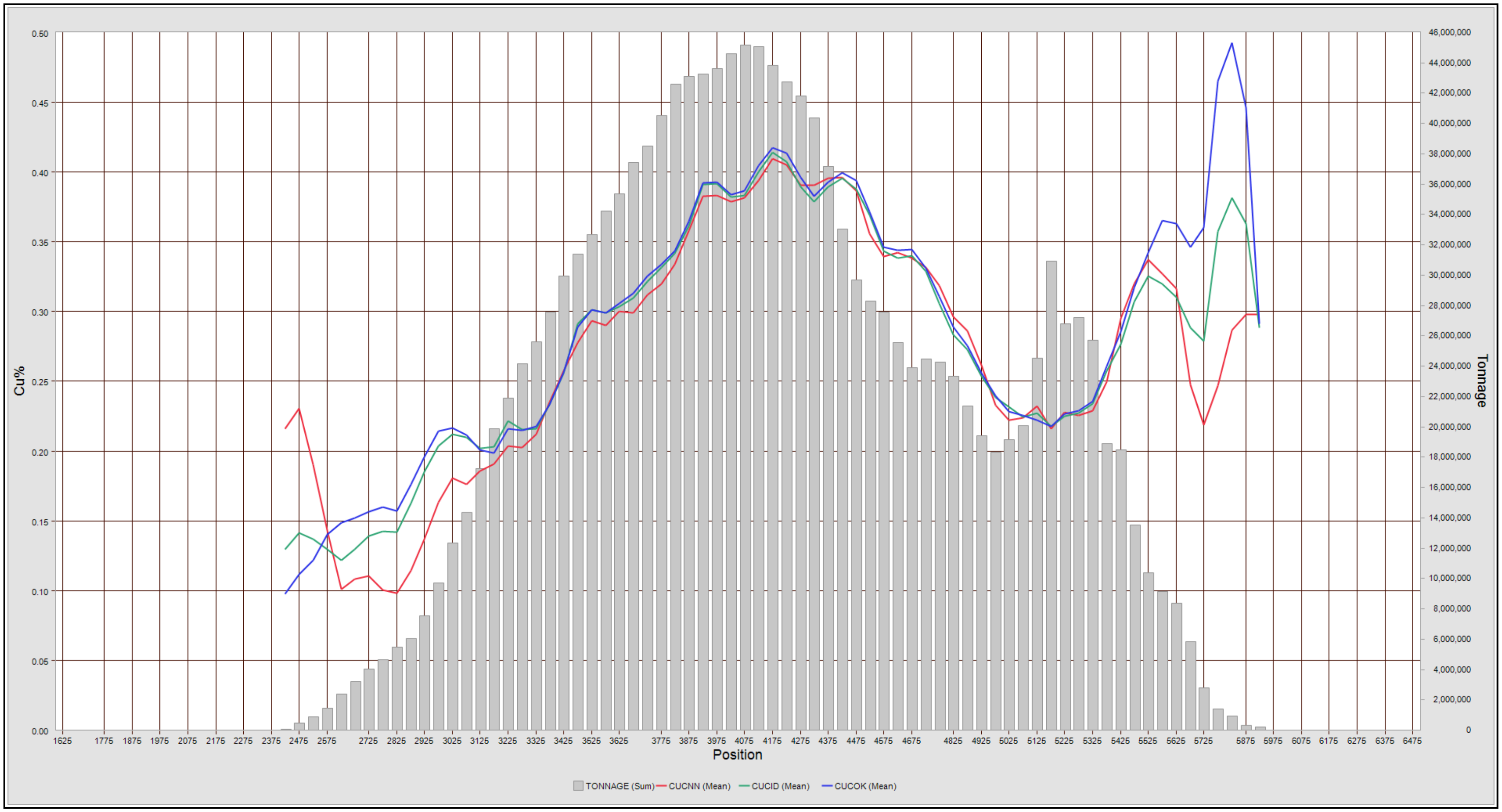

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-26: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, COPPER SWATH PLOT BY NORTHING**



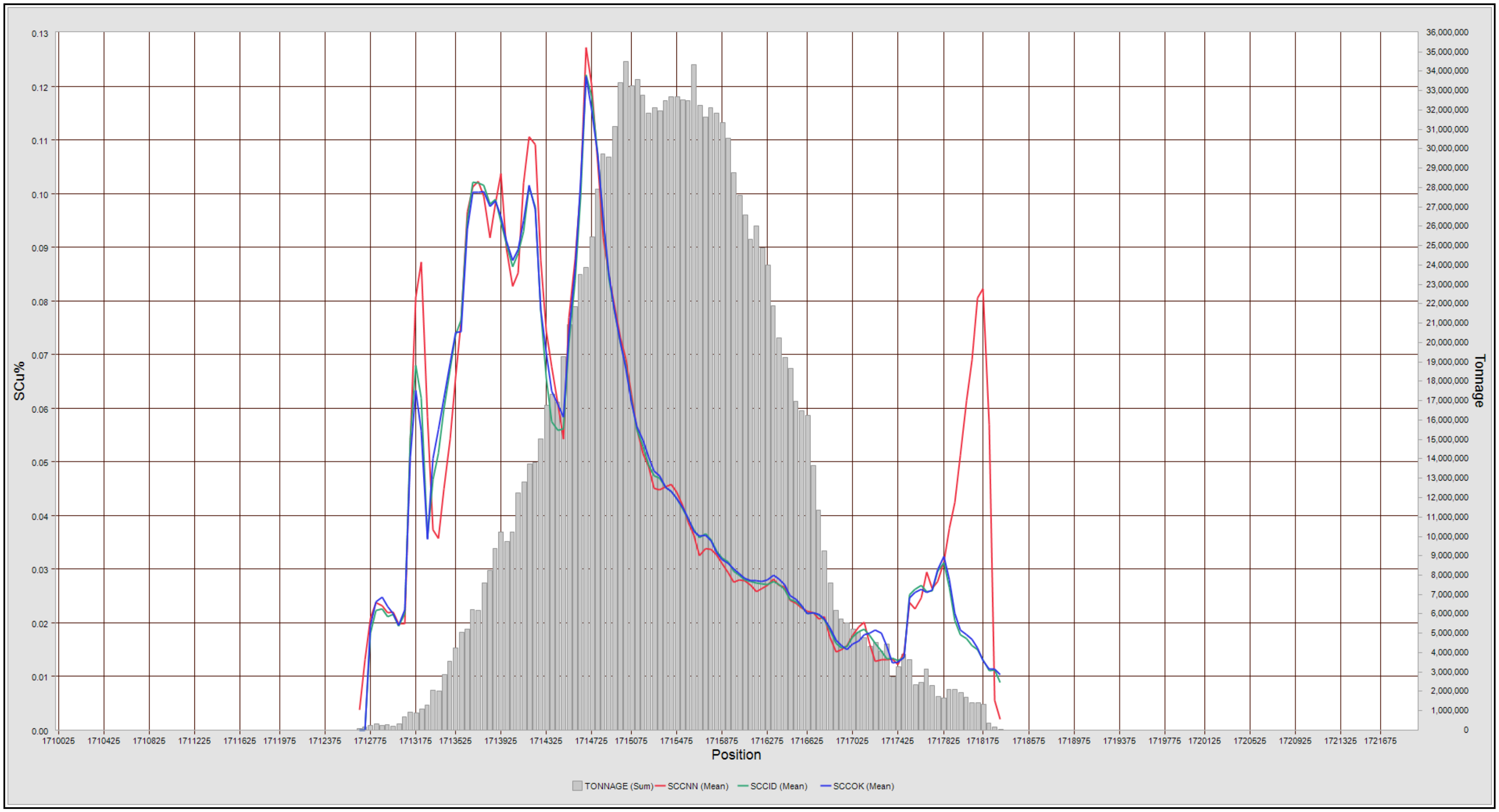

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-27: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, COPPER SWATH PLOT BY ELEVATION**



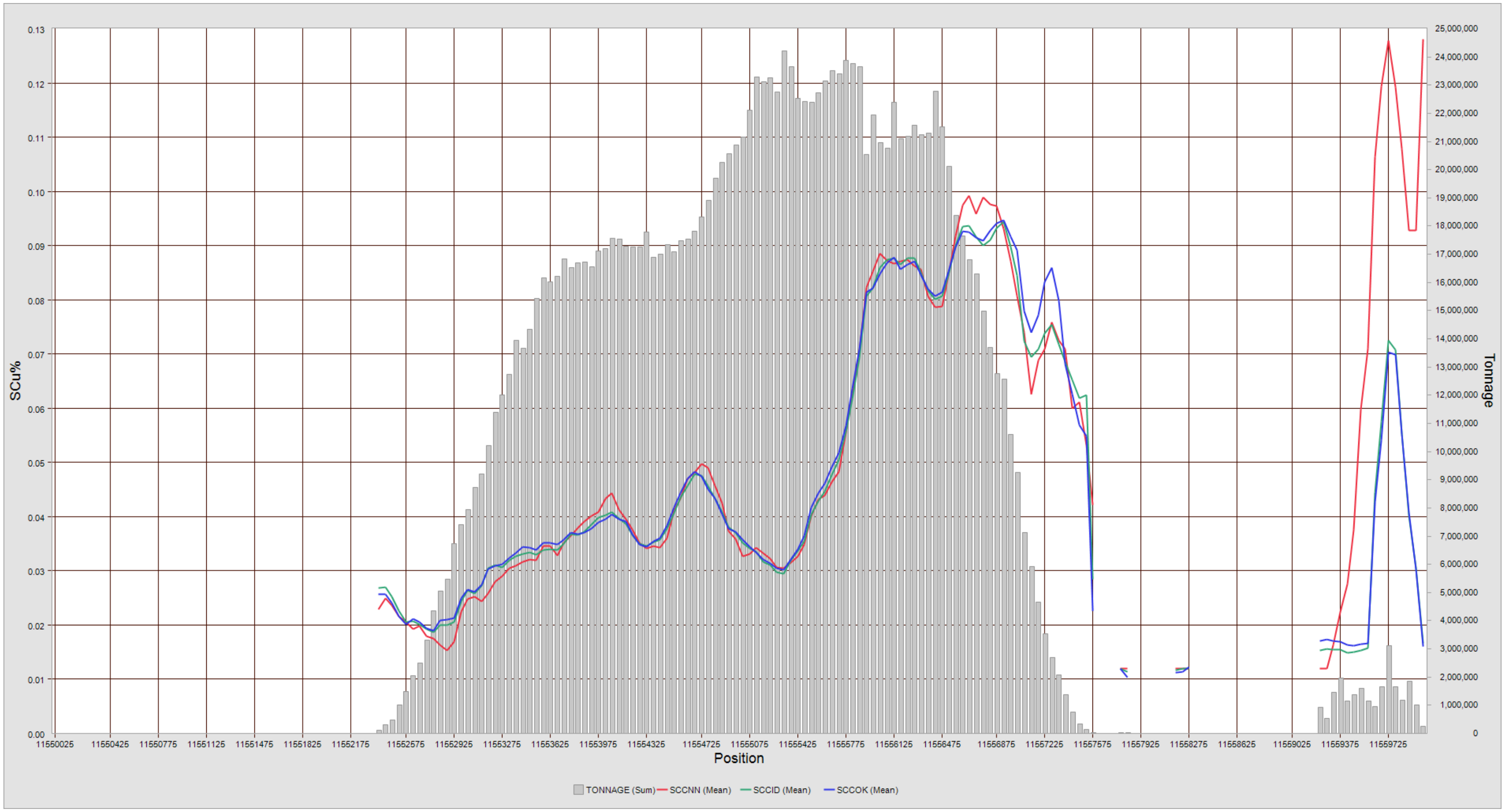

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-28: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, ACID SOLUBLE COPPER SWATH PLOT BY EASTING**



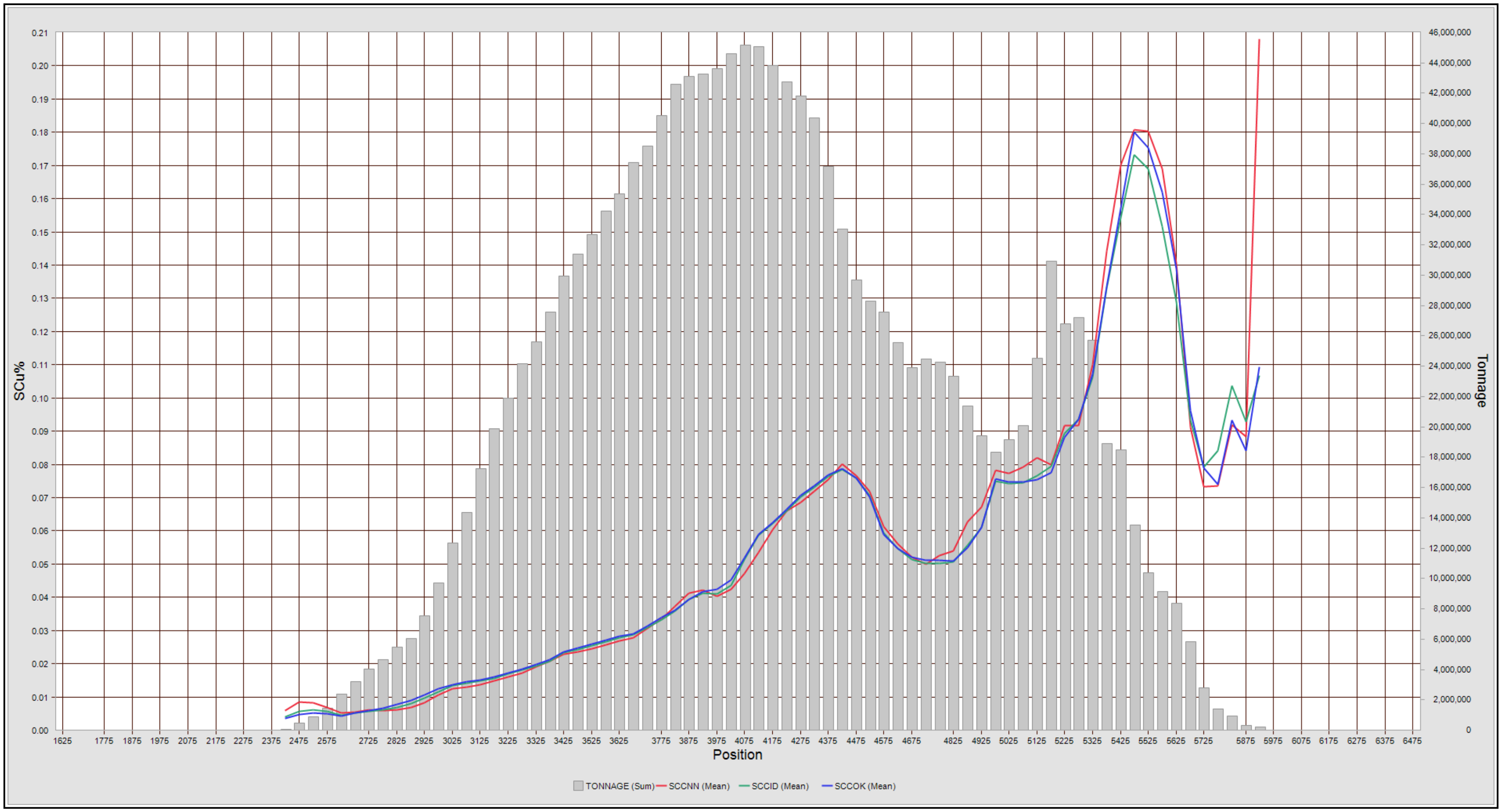

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-29: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, ACID SOLUBLE COPPER SWATH PLOT BY NORTHING**



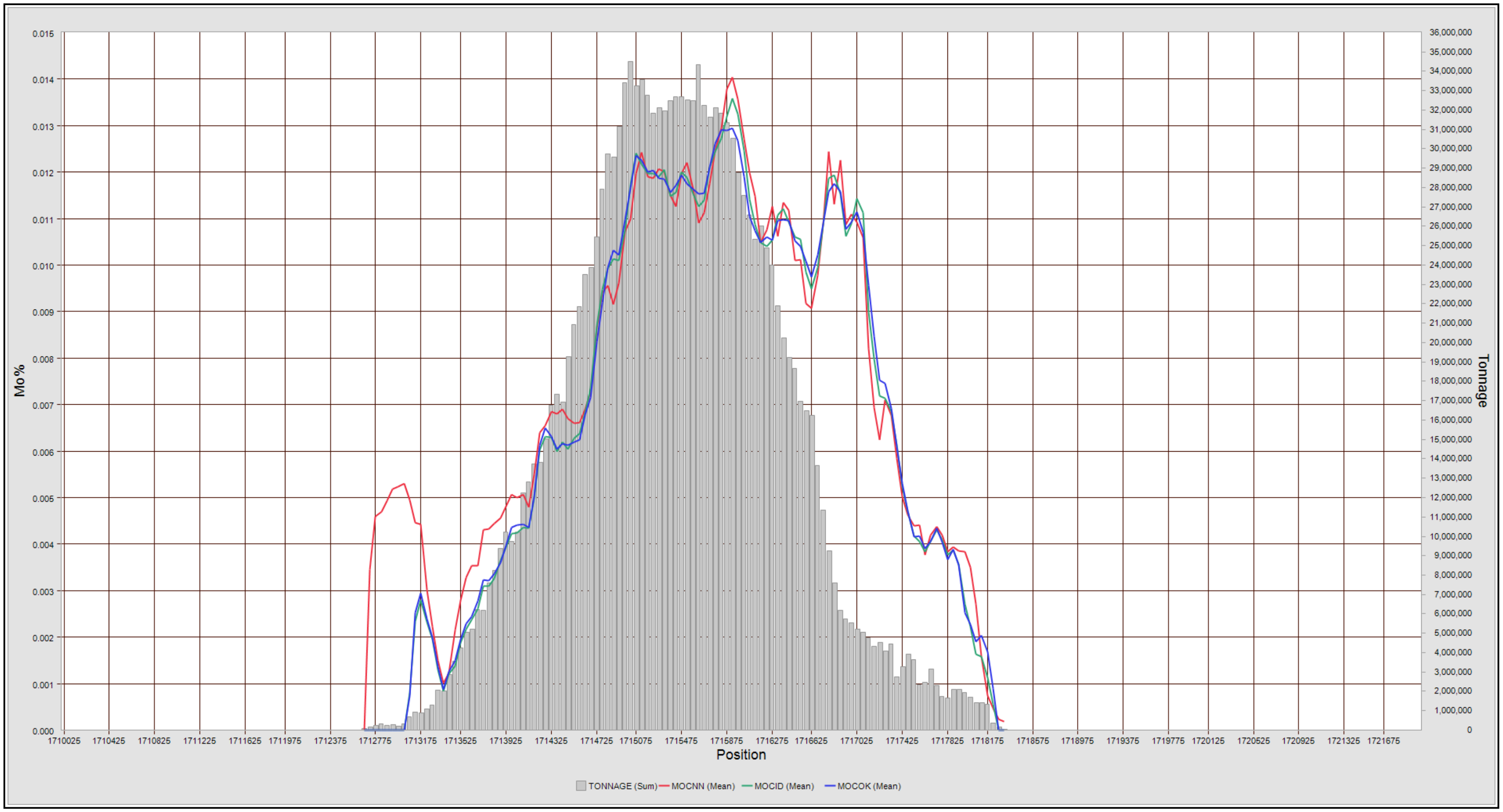

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-30: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, ACID SOLUBLE COPPER SWATH PLOT BY ELEVATION**



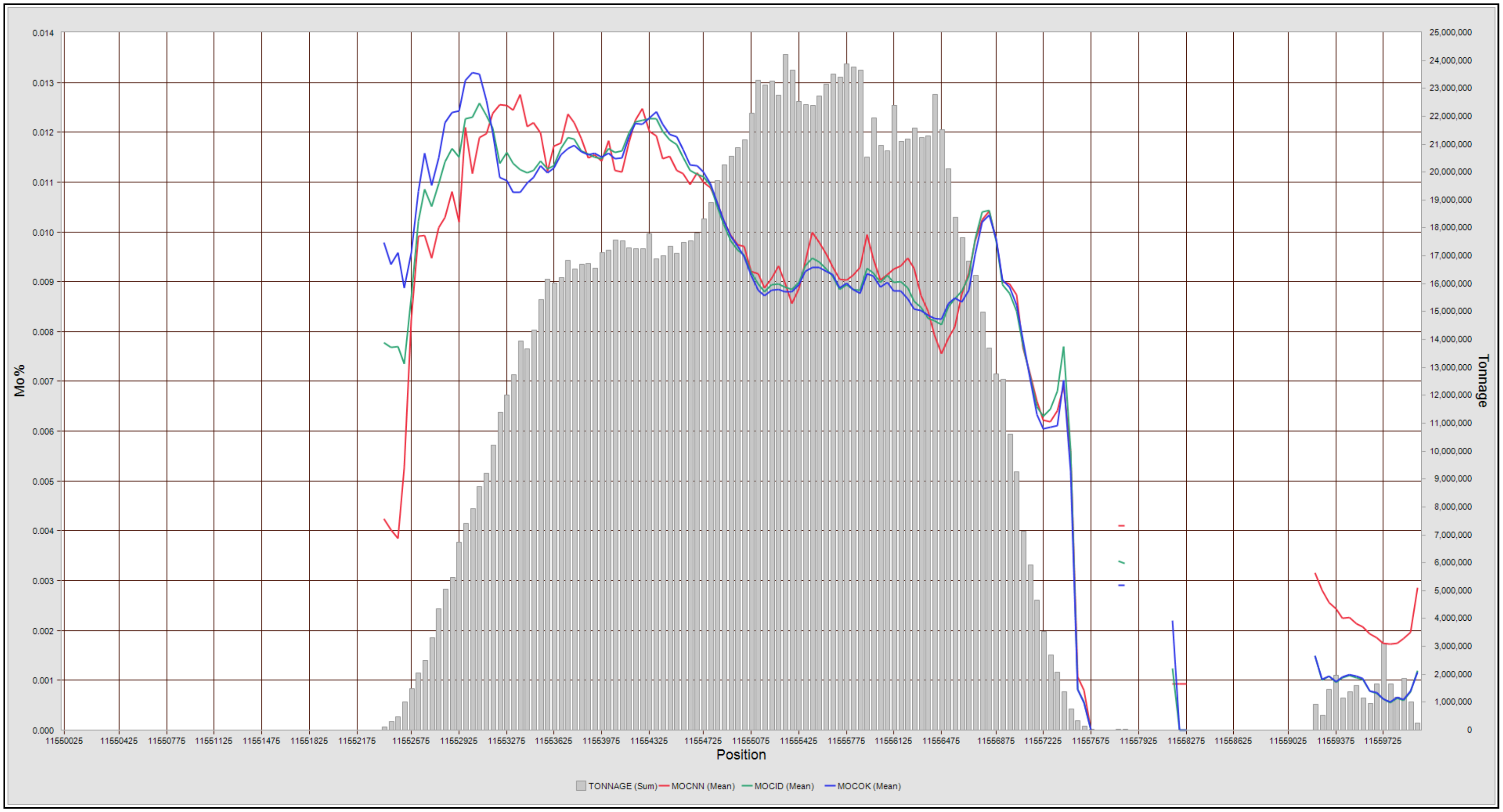

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-31: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, MOLYBDENUM SWATH PLOT BY EASTING**



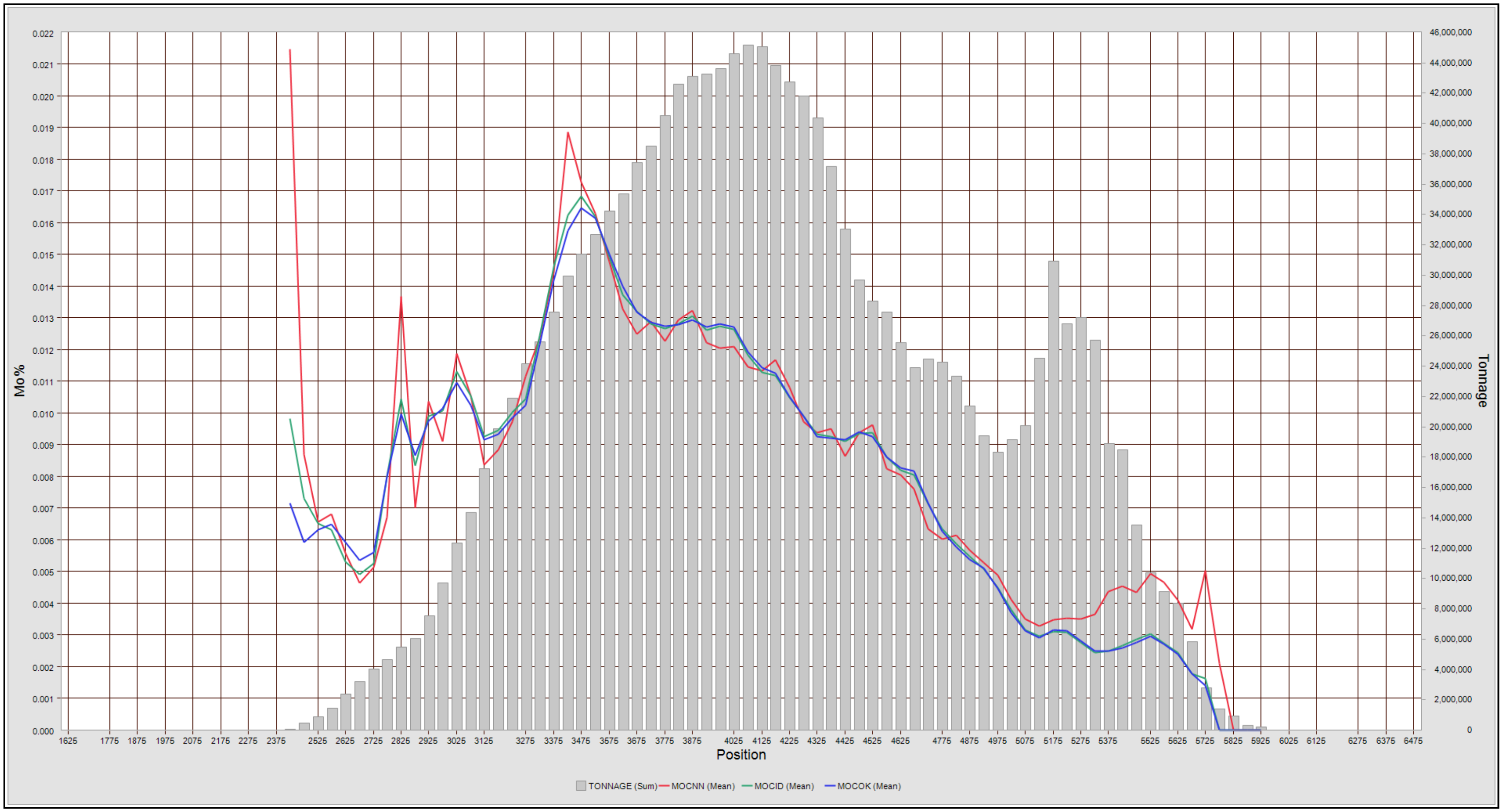

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-32: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, MOLYBDENUM SWATH PLOT BY NORTHING**



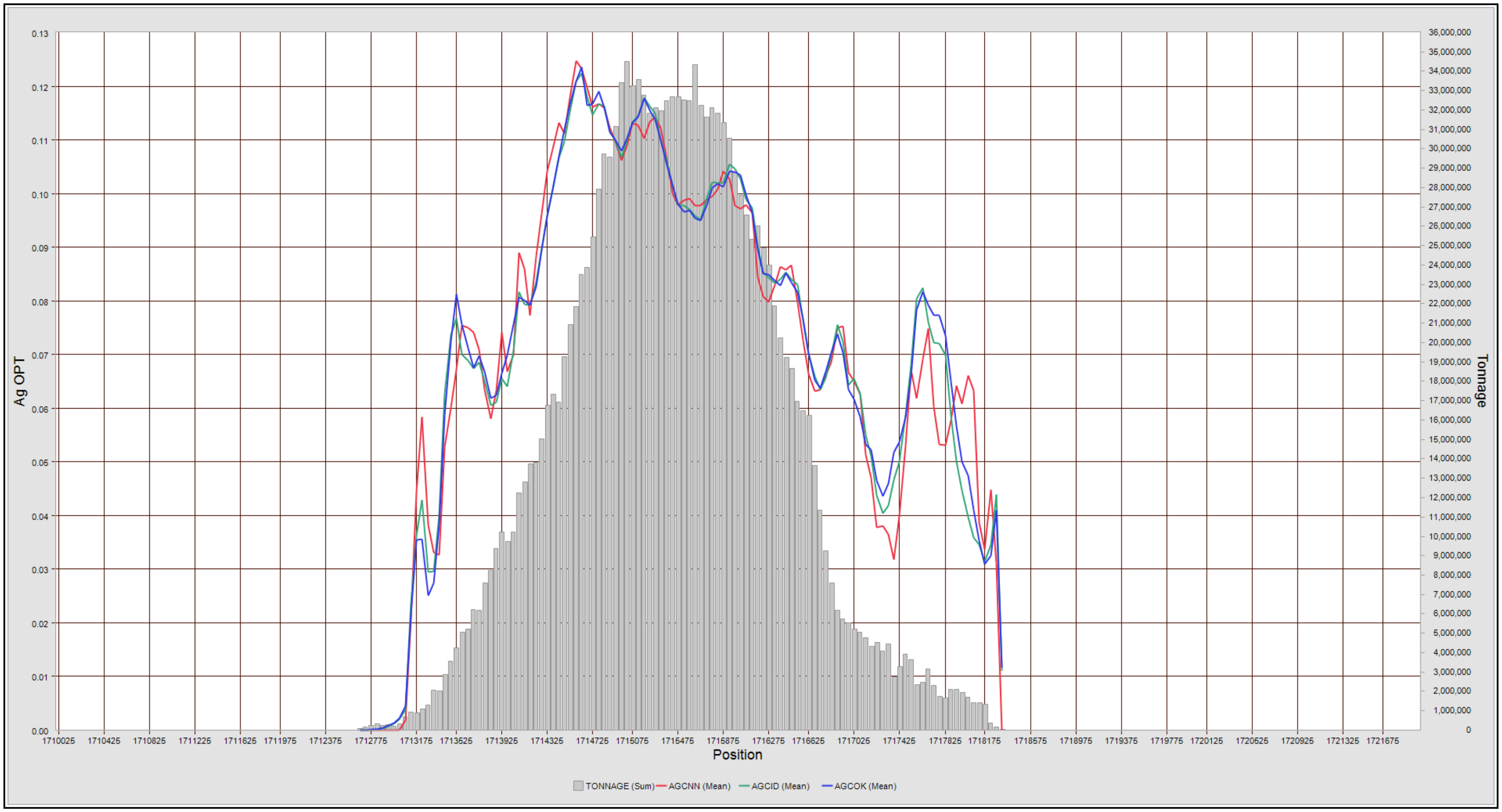

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-33: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, MOLYBDENUM SWATH PLOT BY ELEVATION**



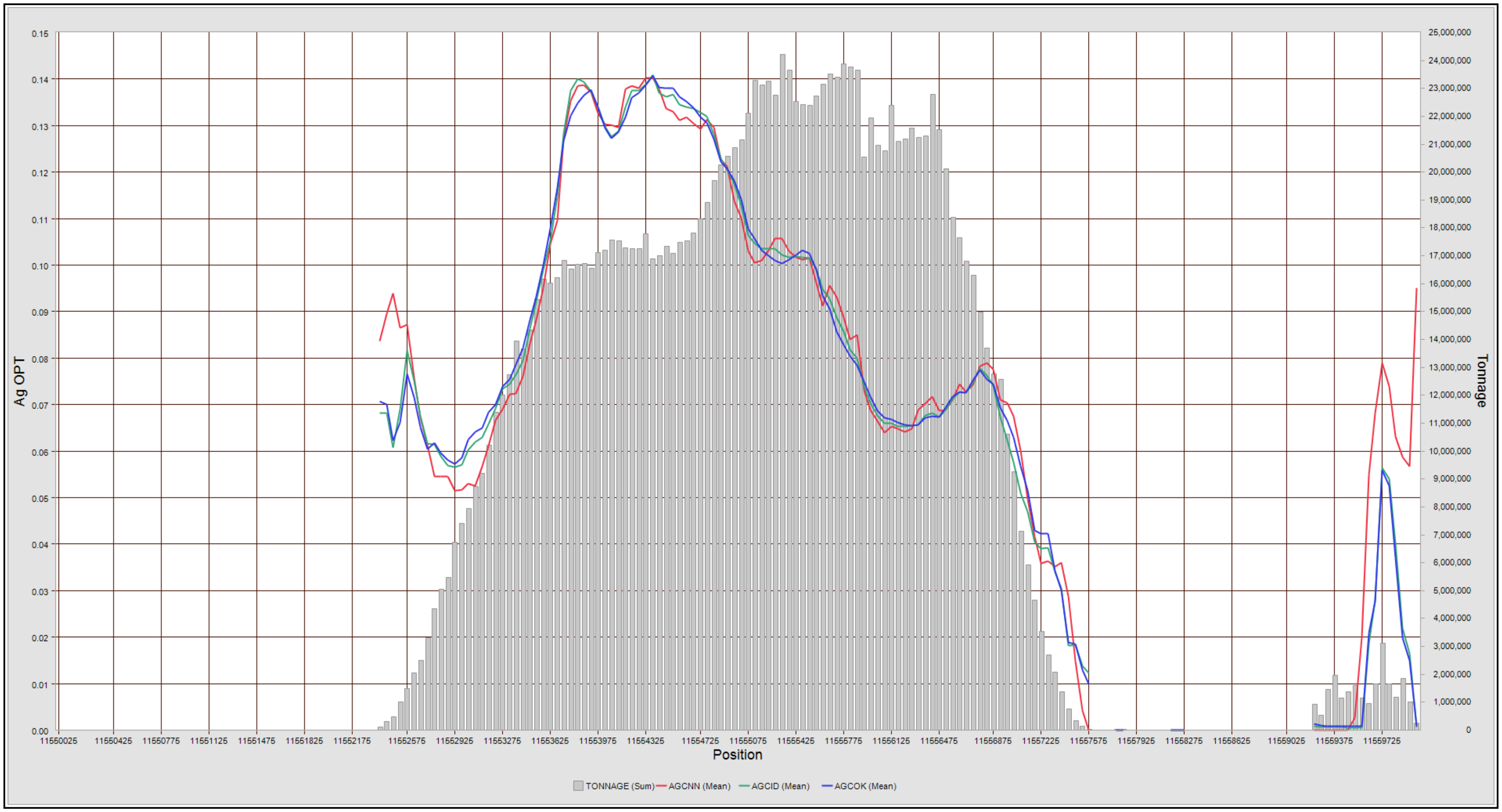

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-34: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, SILVER SWATH PLOT BY EASTING**



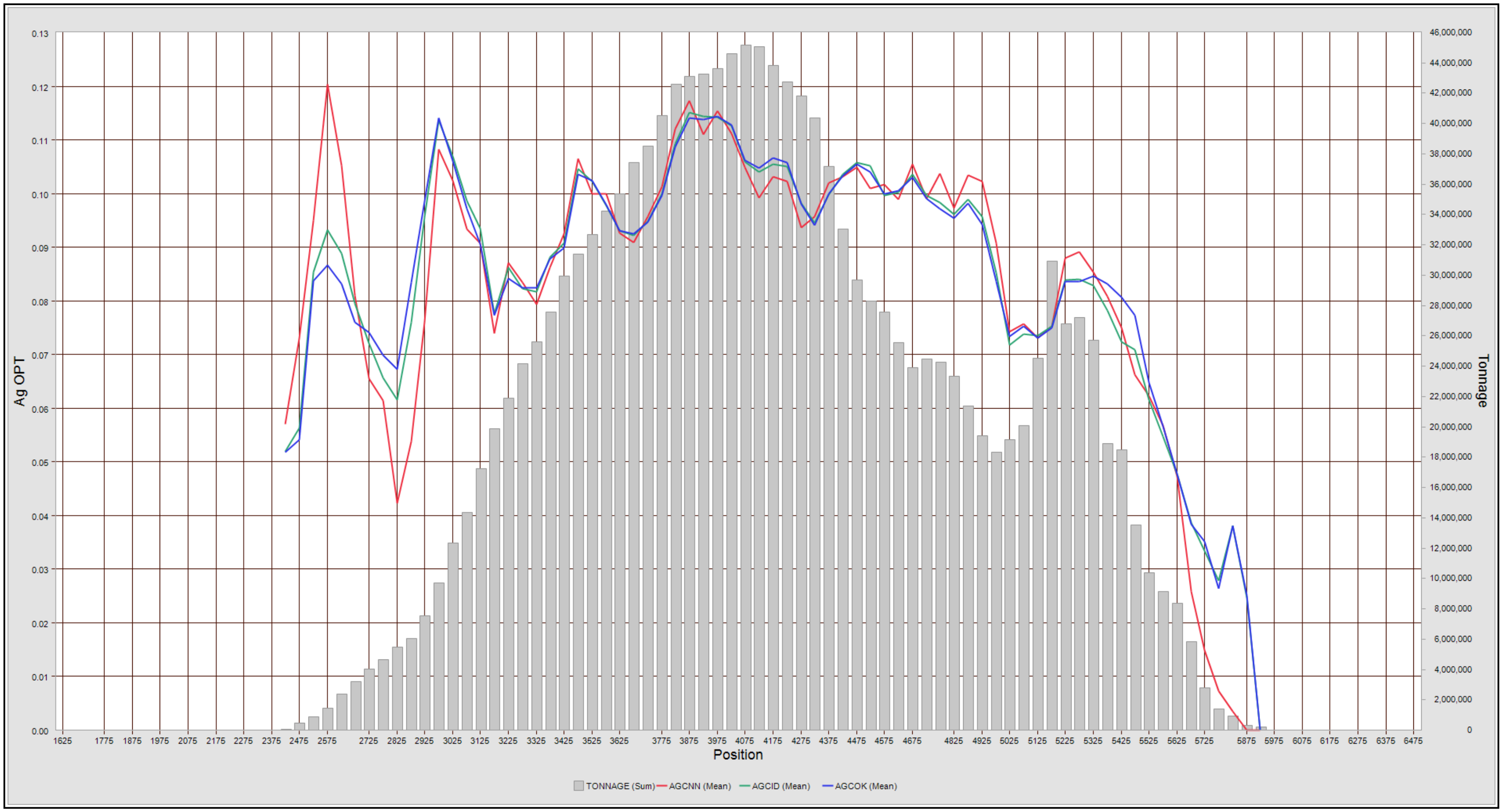

*Note: Line charts show the grades and histogram shows the tons. Green line represents IDW model. Red line represents NN model. Blue line represents OK model.*

**FIGURE 14-35: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, SILVER SWATH PLOT BY NORTHING**




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

**FIGURE 14-36: MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL, SILVER SWATH PLOT BY ELEVATION**




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

## 14.14 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"), the number of holes ("NHOLE"), the number of quadrants ("QUAD") used for the OK interpolation of copper were recorded in the block model.

The standard deviation of the kriging ("KSTD"), the regression slope ("RSLOP"), the local error ("LOCA"L), the relative variance ("RELVA") and the relative variance index ("RVI") were also recorded in the block model. Table 14-34 presents the quality control parameters recorded in the block model from the OK resource estimation.

**TABLE 14-34: QUALITY CONTROL STATISTICS OF THE COPPER INTERPOLATION IN MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL**

| Sulfides | | CDIST | MDIST | ADIST | NCOMP | NHOLE | QUAD | KSTD | RSLOP | LOCAL | RELVA | RVI |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Horquilla | Minimum | 2.5 | 76.1 | 63.6 | 4 | 2 | 0 | 0.32 | 0.25 | 0.00 | 0.05 | 0.16 |
| | Maximum | 584.2 | 800 | 693.1 | 12 | 8 | 4 | 1.11 | 0.96 | 1.07 | 524.29 | 20.51 |
| | Mean | 182.22 | 394.33 | 298.25 | 10 | 4 | 3 | 0.67 | 0.76 | 0.01 | 17.59 | 0.88 |
| Earp (Lower) | Minimum | 3.3 | 92.9 | 49.8 | 4 | 2 | 0 | 0.29 | 0.32 | 0.00 | 0.04 | 0.17 |
| | Maximum | 663.3 | 800 | 738.6 | 12 | 8 | 4 | 1.01 | 0.97 | 0.32 | 297.11 | 2.82 |
| | Mean | 184.67 | 427.66 | 316.42 | 10 | 4 | 2 | 0.67 | 0.75 | 0.02 | 13.50 | 0.89 |
| Earp (Upper) | Minimum | 3.6 | 116.9 | 75 | 4 | 2 | 0 | 0.32 | 0.34 | 0.00 | 0.20 | 0.24 |
| | Maximum | 614.2 | 800 | 703.3 | 12 | 7 | 4 | 1.00 | 0.96 | 0.12 | 256.33 | 2.44 |
| | Mean | 192.28 | 428.97 | 319.47 | 10 | 4 | 2 | 0.67 | 0.76 | 0.00 | 17.24 | 0.88 |
| Epitaph | Minimum | 7.5 | 89.6 | 55.8 | 4 | 2 | 0 | 0.28 | 0.33 | 0.00 | 0.04 | 0.09 |
| | Maximum | 757.3 | 800 | 777.2 | 12 | 8 | 4 | 1.02 | 0.96 | 0.61 | 524.29 | 3.92 |
| | Mean | 202.92 | 417.48 | 319.7 | 10 | 4 | 2 | 0.69 | 0.73 | 0.01 | 14.46 | 0.84 |

Overall, in the Horquilla, Earp (lower and upper) and Epitaph units of the sulfides, the average distance of the composites used to interpolate the grades in the measured and indicated blocks ranges from 298 to 319-foot, with an average of 10 composites used from more than 3 drill holes. The average standard deviation of the kriging for the bulk of the mineralization ranges from 0.67 to 0.69, indicating a high variability of the copper mineralization which is to be expected in a skarn deposit.

## 14.15 Grade-Tonnage Statistics

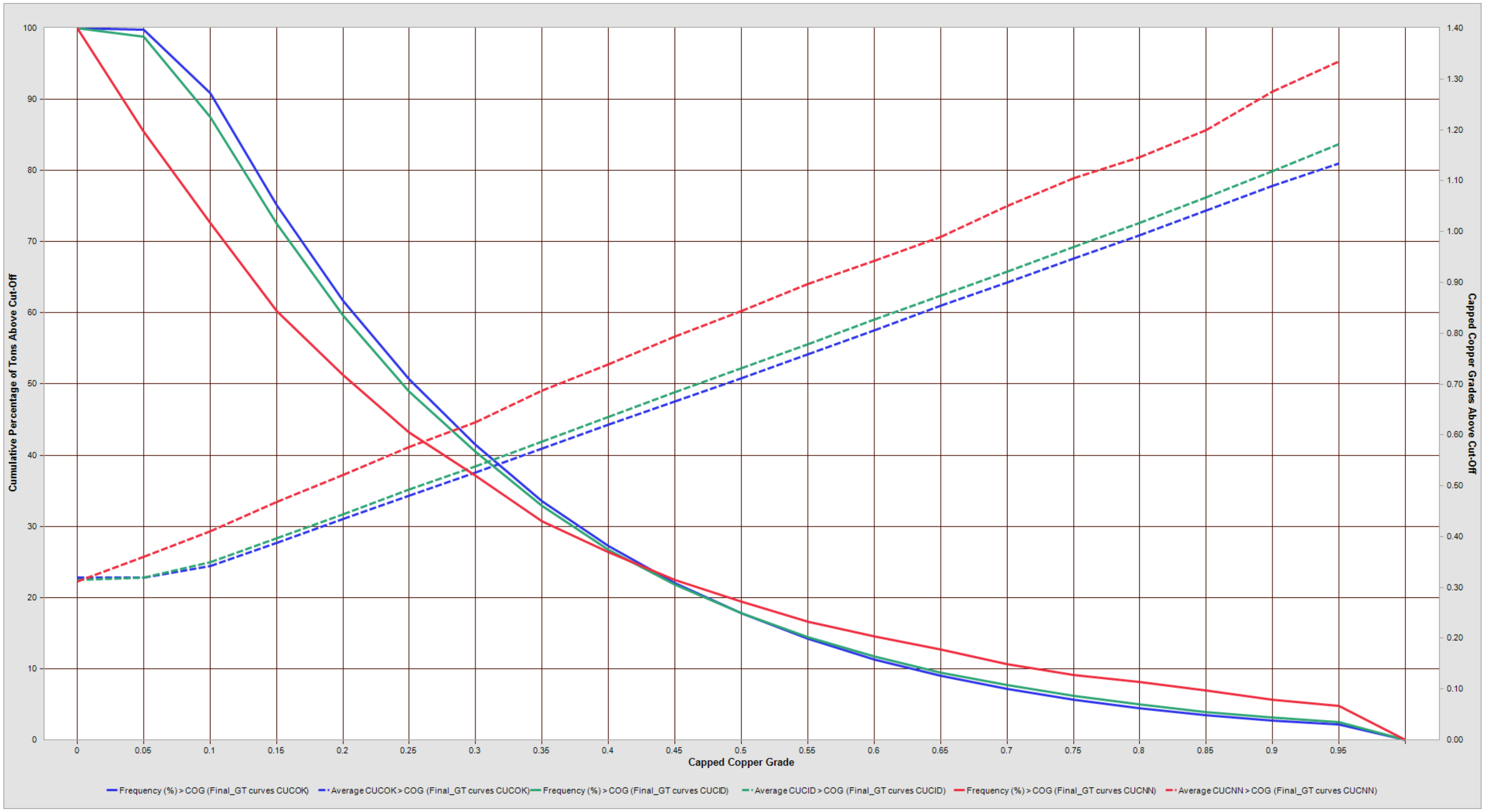
The 50 ft x 50 ft x 50 ft block size is considered suitable for a large scale open pit mining operation with production rates between 75,000 to 100,000 tons per day. Table 14-35 presents the grade-tonnage statistics of copper for each interpolation method at different cut-offs.

**TABLE 14-35: GRADE-TONNAGE STATISTICS, COPPER**

| | 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 | Percent of Tons Above Cut-Off | Mean Cu Grade | Percent of Tons Above Cut-Off | Mean Cu Grade | Percent of Tons Above Cut-Off | Mean Cu Grade | Percent of Tons Above Cut-Off | Mean Cu Grade | Percent of Tons Above Cut-Off | Mean Cu Grade |
| 0 | 100.00 | 0.31 | 100.00 | 0.32 | 100.00 | 0.32 | 0% | -1% | 0% | -2% | 0% | -1% |
| 0.05 | 85.41 | 0.36 | 98.76 | 0.32 | 99.79 | 0.32 | -14% | 13% | -14% | 13% | -1% | 0% |
| 0.1 | 72.68 | 0.41 | 87.50 | 0.35 | 90.92 | 0.34 | -17% | 17% | -20% | 20% | -4% | 2% |
| 0.15 | 60.29 | 0.47 | 72.50 | 0.40 | 75.15 | 0.39 | -17% | 18% | -20% | 21% | -4% | 2% |
| 0.2 | 51.21 | 0.52 | 59.61 | 0.44 | 61.71 | 0.43 | -14% | 17% | -17% | 20% | -3% | 2% |
| 0.25 | 43.17 | 0.58 | 48.97 | 0.49 | 50.68 | 0.48 | -12% | 17% | -15% | 20% | -3% | 2% |
| 0.3 | 37.13 | 0.62 | 40.53 | 0.54 | 41.50 | 0.53 | -8% | 16% | -11% | 19% | -2% | 2% |
| 0.35 | 30.76 | 0.69 | 32.88 | 0.59 | 33.57 | 0.57 | -6% | 17% | -8% | 20% | -2% | 2% |
| 0.4 | 26.37 | 0.74 | 26.72 | 0.64 | 27.21 | 0.62 | -1% | 16% | -3% | 19% | -2% | 3% |
| 0.45 | 22.51 | 0.79 | 21.78 | 0.68 | 22.06 | 0.67 | 3% | 16% | 2% | 19% | -1% | 3% |
| 0.5 | 19.40 | 0.84 | 17.76 | 0.73 | 17.81 | 0.71 | 9% | 15% | 9% | 19% | 0% | 3% |
| 0.55 | 16.63 | 0.90 | 14.49 | 0.78 | 14.23 | 0.76 | 15% | 15% | 17% | 18% | 2% | 3% |
| 0.6 | 14.55 | 0.94 | 11.75 | 0.83 | 11.34 | 0.81 | 24% | 14% | 28% | 17% | 4% | 3% |
| 0.65 | 12.68 | 0.99 | 9.49 | 0.87 | 8.99 | 0.85 | 34% | 13% | 41% | 16% | 6% | 2% |
| 0.7 | 10.63 | 1.05 | 7.71 | 0.92 | 7.13 | 0.90 | 38% | 14% | 49% | 17% | 8% | 2% |
| 0.75 | 9.11 | 1.11 | 6.17 | 0.97 | 5.64 | 0.95 | 48% | 14% | 62% | 17% | 9% | 2% |
| 0.8 | 8.09 | 1.15 | 4.98 | 1.02 | 4.46 | 0.99 | 62% | 13% | 81% | 15% | 12% | 2% |
| 0.85 | 6.96 | 1.20 | 3.94 | 1.07 | 3.46 | 1.04 | 77% | 12% | 101% | 15% | 14% | 2% |
| 0.9 | 5.62 | 1.28 | 3.11 | 1.12 | 2.70 | 1.09 | 81% | 14% | 108% | 17% | 15% | 3% |
| 0.95 | 4.81 | 1.34 | 2.45 | 1.17 | 2.12 | 1.13 | 96% | 14% | 127% | 18% | 16% | 3% |

The grade-tonnage curve for total copper is shown in Figure 14-37 as a way to present the overall assessment of the Measured and Indicated resources.

**FIGURE 14-37: NN, IDW AND OK COPPER GRADE-TONNAGE CURVES, ALL LITHOLOGIES IN MEASURED AND INDICATED BLOCKS ABOVE $5.7/TON NSR WITHIN THE RESOURCE PIT SHELL**




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

## 14.16 Classification of Mineral Resource

The resource category classification used for Rosemont relies on the relative difference between the kriged grade and the composites grades, and the Resource Classification Index (RCI) which uses the following formula[3]:

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

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

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

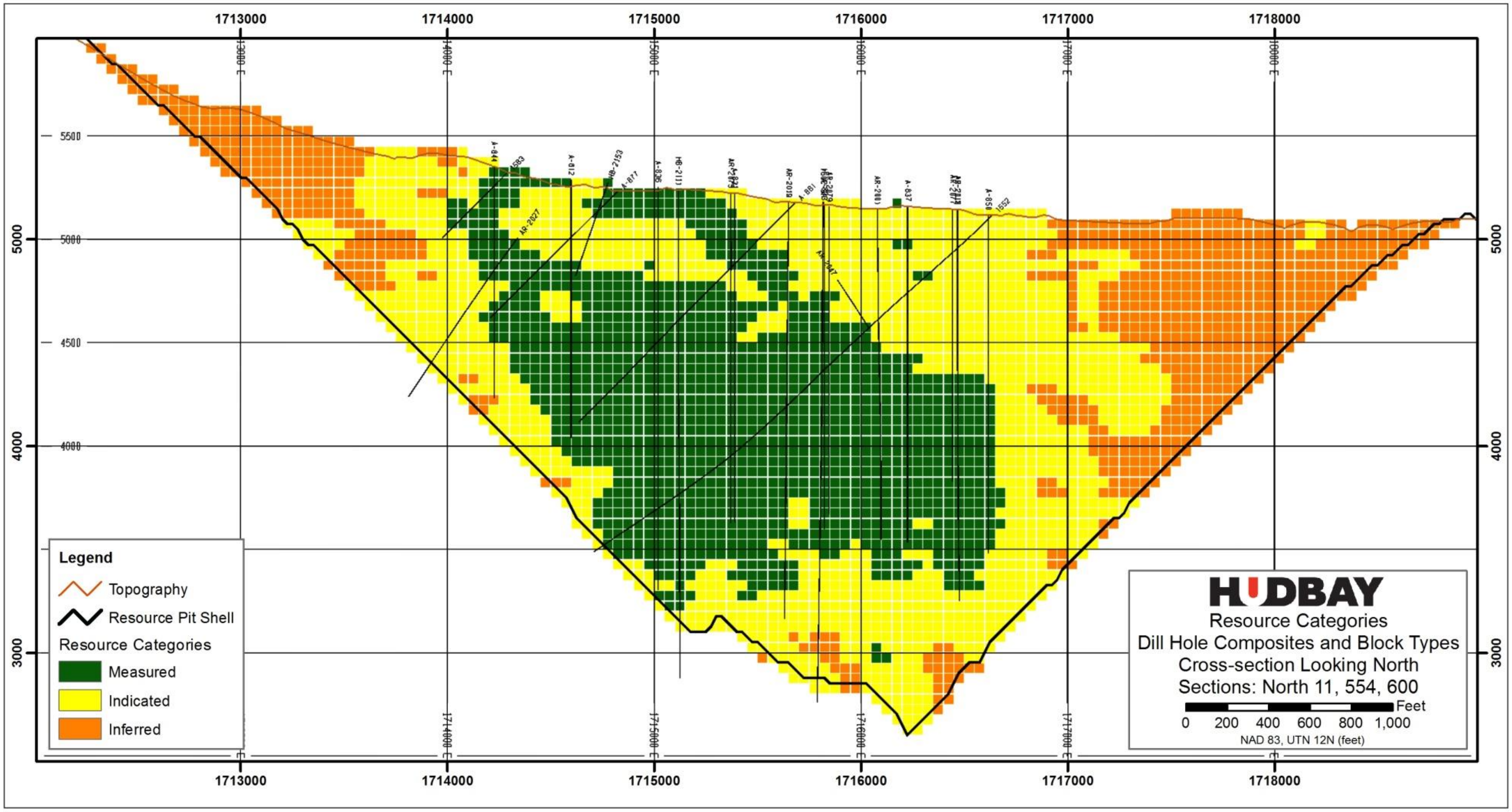
The RCI values corresponding to the 50th (0.216) and 95th (0.971) percentiles of the distribution of blocks with total copper grade above 0.1% contained within the resource pit were determined and used as thresholds for the Measured and Indicated resource categories, respectively.

Under this classification system, in order for a block to be considered as a Measured resource, the RCI value must be less than 0.216, the relative difference less than 0.15 and have a CDIST of less than 500 feet. In order for a block to be considered as an indicated resource, the RCI value must be less than 0.971, the relative difference less than 0.15 and have a CDIST of less than 500 feet. Blocks were classified as inferred resources when at least two drill holes were used 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. Proportions of Measured and Indicated category blocks were not changed significantly by this process. Figure 14-38 presents the resource categories for a typical cross section.

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

**FIGURE 14-38: VERTICAL E-W SECTION 11,554,600 SHOWING RESOURCE CLASSIFICATION AND DRILL HOLES**



## 14.17 Third Party Review

Hudbay requested Tim Maunula & Associates Consulting Inc. to perform an independent validation of the block model. The following minor issues were highlighted by the third party validation:

- Interpolation plan of the NN model was different than the IDW and OK models
- A small portion of blocks in the model estimated IDW values, but no estimated OK values and vice versa

These issues were corrected and the content of this Technical Report, including the tons and grades estimate, reflects these changes.

Based on the review, the third party has concluded that the mineral resources for the Project have been prepared using industry standard best practices and the methodology for the resource classification conforms to the requirements of the CIM Definition Standards.

## 14.18 Internal Peer Review

Hudbay Peru Technical Service group performed a full validation of the block model. The following minor issues were highlighted by the peer review:

- The Escabrosa formation should be interpolated with the lower plate units
- A low grade shell should be use to separate the high grade and low grade zones in Earp formation
- Resource classification assignation should include a distance component. Measured or Indicated blocks with CDIST > 500 feet should be reclassified as Inferred. Inferred blocks with ADIST < 300 feet should be reclassified as Indicated.

These recommendations were implemented and the content of this report, including the tons and grade estimate, reflects these changes.

## 14.19 Reasonable Prospects of Economic Extraction

The component of the mineralization within the block model that meets the requirements for reasonable prospects of economic extraction was based on the application of an LG cone pit algorithm. The mineral resources are therefore contained within a computer generated open pit geometry.

The following assumptions were applied to the determination of the mineral resources:

- Economic benefit was applied to Measured, Indicated and Inferred classified material within the resource cone.

- No effort was made to establish a pit with maximum return on investment; consequently, the mineral resource cone was the direct result of the following metal prices: \$3.15/lb copper, \$11.00/lb molybdenum, \$18.00/oz silver with a revenue ratio of 1.0, i.e. break-even logic.
- A constant 45-degree pit slope was used for the resource estimate.
- No haulage increment or bench discounting was applied to the resource estimate.

The LG cone input parameters are summarized in Table 14-36.

**TABLE 14-36: LERCHS-GROSSMAN CONE INPUTS**

| | | | Unit | Value |
|---|---|---|---|---|
| Mining | Mineralized Material | | \$/ton mined | \$1.00 |
| | Waste Material | | \$/ton mined | \$1.00 |
| Processing | Oxide | | \$/ton processed | \$4.00 |
| | Mixed and Sulfide | | \$/ton processed | \$5.00 |
| G&A | | | \$/ton processed | \$0.70 |
| Recovery | Copper | Oxide | % | 65 |
| | | Mixed | % | 40 |
| | | Sulfide | % | 85 |
| | Molybdenum | Oxide | % | 0 |
| | | Mixed | % | 30 |
| | | Sulfide | % | 60 |
| | Silver | Oxide | % | 0 |
| | | Mixed | % | 40 |
| | | Sulfide | % | 75 |
| Metal Price | Copper | | \$/lb | \$3.15 |
| | Molybdenum | | \$/lb | \$11.00 |
| | Silver | | \$/troy oz | \$18.00 |
| Slope Angle | | | Constant degrees | 45 |

Note: The recoveries in oxide present the recoverable portion of the sulfides by the process plan flotation. The cost and price inputs are considered approximation and were used to test the economic viability of the resource. The cost and price inputs differ from the ones used for the reserve, which used more accurate numbers.

The reporting of the mineral resource by NSR within the LG pit shell, reflects the combined benefit of producing copper, molybdenum and silver as per the following equations based on mineralized type, in addition to mine operating and processing costs:

| | | |
|---|---|---|
| NSR formula details | Copper Contribution | (Price of Copper -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) * unit conversion factor (% to ton) |
| | Molybdenum Contribution | (Price of Molybdenum -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) * unit conversion factor + (% to ton) |
| | Silver Contribution | (Price of Silver -(refining + freight & transport cost)) * recovery * payable * (100% - royalty) |
| Sulfide: | Copper Contribution | (\$3.15-\$0.4307) * 0.85 * 0.96 * 0.97 * 20 + |
| | Molybdenum Contribution | (\$11.00-\$1.50) * 0.60 * 1 * 0.97 * 20 + |
| | Silver Contribution | (\$18.00-\$0.50) * 0.75* 0.90 * 0.97 |
| Mixed: | Copper Contribution | (\$3.15-\$0.4307) * 0.40 * 0.96 * 0.97 * 20 + |
| | Molybdenum Contribution | (\$11.00-\$1.50) * 0.30 * 1 * 0.97 * 20 + |
| | Silver Contribution | (\$18.00-\$0.50) * 0.40 * 0.90 * 0.97 |
| Oxide: | Copper Contribution | (\$3.15-\$0) * 0.65 * 1 * 0.97 * 20 |
| | Molybdenum Contribution | None |
| | Silver Contribution | None |

The copper equivalency is calculated, using metal contributions and economic inputs as noted above, for each block using the following formula:

$$CuEq = Copper + (Contribution\ of\ Molybdenum) + (Contribution\ of\ Silver)$$

In oxide material, since molybdenum and silver are not considered, the copper equivalency value equals the copper value.

Table 14-37 presents the economic parameters used in addition to the LG cone inputs to calculate the NSR and CuEq formulas mentioned above.

**TABLE 14-37: ECONOMIC PARAMETERS**

| Freight | Copper | $/ton of Concentrate | $130.00 |
|---|---|---|---|
| Smelter Charges | Copper | $/ton of concentrate | $72.00 |
| | Molybdenum | $/lb of molybdenum includes freight | $1.50 |
| Refining Charges | Copper | $/lb of copper | $0.08 |
| | Silver | $/troy oz of silver | $0.50 |
| Smelting Terms | Copper | % concentrate | 30 |
| | | % payable | 96 |
| | Molybdenum | % concentrate grade | 50 |
| | | % payable | 99 |
| | Silver | % payable | 90 |
| Royalty | | % | 3 |

Note: The cost and price inputs are considered approximation and were used to test the economic viability of the resource. The cost and price inputs differ from the ones used for the reserve, which used more accurate numbers.

## 14.20 Mineral Resource Statement Inclusive of Mineral Reserve

Mineral resources for the Rosemont deposit were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves[4] by application of a NSR that reflects the combined benefit of producing copper, molybdenum and silver in addition to mine operating, processing and off-site costs.

The mineral resources, classified as Measured, Indicated and Inferred and are inclusive of the mineral reserves. Table 14-38 summarizes the resource estimate.

*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.*

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

**TABLE 14-38: RESOURCE BY CATEGORY, MINERALIZED ZONE AND NSR CUT-OFF (1)(2)(3)(4)(5)(6)(7)(8)(9)(10)**

| Measured | TONS | NSR Cut Off | CuEq (%) | Cu (%) | Mo (%) | Ag (opt) |
|---|---|---|---|---|---|---|
| Oxide | 111,800,000 | > = $5.70 | 0.38 | 0.38 | | |
| Mix | 18,800,000 | > = $5.70 | 0.45 | 0.40 | 0.009 | 0.069 |
| Hypogene | 566,800,000 | > = $5.70 | 0.53 | 0.45 | 0.013 | 0.140 |
| **Summary** | **697,400,000** | | **0.51** | **0.44** | **0.011** | **0.116** |
| | | | | | | |
| **Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 38,600,000 | > = $5.70 | 0.26 | 0.26 | | |
| Mix | 7,000,000 | > = $5.70 | 0.40 | 0.36 | 0.007 | 0.055 |
| Hypogene | 521,500,000 | > = $5.70 | 0.32 | 0.26 | 0.011 | 0.081 |
| **Summary** | **567,100,000** | | **0.32** | **0.26** | **0.010** | **0.076** |
| | | | | | | |
| **Measured + Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 150,400,000 | > = $5.70 | 0.35 | 0.35 | | |
| Mix | 25,800,000 | > = $5.70 | 0.43 | 0.39 | 0.008 | 0.065 |
| Hypogene | 1,088,400,000 | > = $5.70 | 0.43 | 0.36 | 0.012 | 0.112 |
| **Summary** | **1,264,600,000** | | **0.42** | **0.36** | **0.011** | **0.098** |
| | | | | | | |
| **Inferred** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 6,400,000 | > = $5.70 | 0.31 | 0.31 | | |
| Mix | 1,600,000 | > = $5.70 | 0.46 | 0.44 | 0.004 | 0.024 |
| Hypogene | 74,100,000 | > = $5.70 | 0.35 | 0.30 | 0.011 | 0.050 |
| **Summary** | **82,100,000** | | **0.35** | **0.30** | **0.010** | **0.045** |

Notes:

1. The above mineral resources include mineral reserves.
2. Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were based on core logging, structural and geochemical data.
3. Raw drill hole assays were composited to 25-foot lengths broken at lithology boundaries.
4. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
5. Block grades for copper, molybdenum and silver were estimated from the composites using OK interpolation into 50 ft x 50 ft x 50 ft blocks coded by domain.
6. Tonnage factors were interpolated by lithology and mineralized zone. Tonnage factors are based on 2,066 measurements collected by Hudbay and previous operators.
7. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
8. Mineral resources are constrained within a computer generated pit using the LG algorithm. Metal prices of $3.15/lb copper, $11.00/lb molybdenum and $18.00/troy oz silver. Metallurgical recoveries of 85% copper, 60% molybdenum and 75% silver were applied to sulfide material. Metallurgical recoveries of 40% copper, 30% molybdenum and 40% silver were applied to mixed material. A metallurgical recovery of 65% for copper was applied to oxide material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, and silver combined. Cut-off grades were set in terms of NSR based on current estimates of process recoveries, total process and G&A operating costs of $5.70/ton for oxide, mixed and sulfide material.
9. The oxide resource will be processed in the mill via flotation
10. Totals may not add up correctly due to rounding.

## 14.21 Sensitivity of the Mineral Resource

The sensitivity of the mineral resource was assessed for changes in copper, molybdenum and silver by reporting the estimation at lower and several higher NSR cut-offs, as shown in Table 14-39, Table 14-40, and Table 14-41. The results show that the mineral resource is not highly sensitive to small increases in NSR cut-offs (a proxy for changes in metal prices), therefore concluding that the mineral resource is robust with respect to the inputs used to estimate.

## TABLE 14-39: MEASURED RESOURCE BY MINERALIZED ZONE AND MULTIPLE NSR CUT-OFFS

| Mineralized Zone | Varioation of the Cut-Off NSR/Ton in % | Varioation of the Cut-Off NSR/Ton in $ | TONS | CuEq (%) | Cu (%) | Mo (%) | Ag (opt) |
|---|---|---|---|---|---|---|---|
| Oxide | -25% | $4.28 | 123,700,000 | 0.36 | 0.36 | | |
| | 0% | $5.70 | 111,800,000 | 0.38 | 0.38 | | |
| | 25% | $7.13 | 99,100,000 | 0.41 | 0.41 | | |
| | 50% | $8.55 | 88,400,000 | 0.43 | 0.43 | | |
| | 75% | $9.98 | 78,300,000 | 0.46 | 0.46 | | |
| Mixed | -25% | $4.28 | 27,900,000 | 0.38 | 0.33 | 0.007 | 0.08 |
| | 0% | $5.70 | 18,800,000 | 0.45 | 0.40 | 0.009 | 0.07 |
| | 25% | $7.13 | 13,400,000 | 0.50 | 0.45 | 0.009 | 0.08 |
| | 50% | $8.55 | 8,100,000 | 0.57 | 0.52 | 0.009 | 0.08 |
| | 75% | $9.98 | 5,000,000 | 0.64 | 0.58 | 0.009 | 0.09 |
| Sulfides | -25% | $4.28 | 591,900,000 | 0.52 | 0.43 | 0.013 | 0.14 |
| | 0% | $5.70 | 566,800,000 | 0.53 | 0.45 | 0.013 | 0.14 |
| | 25% | $7.13 | 543,800,000 | 0.55 | 0.46 | 0.013 | 0.14 |
| | 50% | $8.55 | 519,400,000 | 0.57 | 0.48 | 0.014 | 0.15 |
| | 75% | $9.98 | 491,500,000 | 0.59 | 0.50 | 0.014 | 0.15 |

*Note: Using a $5.70 per ton baseline NSR cut-off for oxide, mixed and sulfide material.*

## TABLE 14-40: INDICATED RESOURCE BY MINERALIZED ZONE AND MULTIPLE NSR CUT-OFFS

| Mineralized Zone | Varioation of the Cut-Off NSR/Ton in % | Varioation of the Cut-Off NSR/Ton in $ | TONS | CuEq (%) | Cu (%) | Mo (%) | Ag (opt) |
|---|---|---|---|---|---|---|---|
| Oxide | -25% | $4.28 | 51,600,000 | 0.23 | 0.23 | | |
| | 0% | $5.70 | 38,600,000 | 0.26 | 0.26 | | |
| | 25% | $7.13 | 29,200,000 | 0.29 | 0.29 | | |
| | 50% | $8.55 | 21,800,000 | 0.32 | 0.32 | | |
| | 75% | $9.98 | 15,600,000 | 0.36 | 0.36 | | |
| Mixed | -25% | $4.28 | 15,900,000 | 0.31 | 0.28 | 0.006 | 0.04 |
| | 0% | $5.70 | 7,000,000 | 0.40 | 0.36 | 0.007 | 0.05 |
| | 25% | $7.13 | 3,200,000 | 0.50 | 0.45 | 0.008 | 0.07 |
| | 50% | $8.55 | 1,900,000 | 0.57 | 0.52 | 0.008 | 0.09 |
| | 75% | $9.98 | 1,300,000 | 0.62 | 0.57 | 0.008 | 0.10 |
| Sulfides | -25% | $4.28 | 610,100,000 | 0.29 | 0.23 | 0.010 | 0.08 |
| | 0% | $5.70 | 521,500,000 | 0.32 | 0.26 | 0.011 | 0.08 |
| | 25% | $7.13 | 437,700,000 | 0.35 | 0.28 | 0.012 | 0.09 |
| | 50% | $8.55 | 367,000,000 | 0.38 | 0.31 | 0.013 | 0.09 |
| | 75% | $9.98 | 307,600,000 | 0.41 | 0.34 | 0.013 | 0.10 |

*Note: Using a $5.70 per ton baseline NSR cut-off for oxide, mixed and sulfide material.*

**TABLE 14-41: INFERRED RESOURCE BY MINERALIZED ZONE AND MULTIPLE NSR CUT-OFFS**

| Mineralized Zone | Variation of the Cut-Off NSR/Ton in % | Variation of the Cut-Off NSR/Ton in $ | TONS | CuEq (%) | Cu (%) | Mo (%) | Ag (opt) |
|---|---|---|---|---|---|---|---|
| Oxide | -25% | $4.28 | 7,800,000 | 0.28 | 0.28 | | |
| | 0% | $5.70 | 6,400,000 | 0.31 | 0.31 | | |
| | 25% | $7.13 | 5,300,000 | 0.34 | 0.34 | | |
| | 50% | $8.55 | 4,600,000 | 0.36 | 0.36 | | |
| | 75% | $9.98 | 3,600,000 | 0.40 | 0.40 | | |
| Mixed | -25% | $4.28 | 3,200,000 | 0.35 | 0.33 | 0.003 | 0.02 |
| | 0% | $5.70 | 1,600,000 | 0.46 | 0.44 | 0.004 | 0.02 |
| | 25% | $7.13 | 1,100,000 | 0.52 | 0.50 | 0.005 | 0.02 |
| | 50% | $8.55 | 800,000 | 0.58 | 0.55 | 0.006 | 0.02 |
| | 75% | $9.98 | 600,000 | 0.60 | 0.57 | 0.007 | 0.02 |
| Sulfides | -25% | $4.28 | 85,700,000 | 0.32 | 0.27 | 0.010 | 0.05 |
| | 0% | $5.70 | 74,100,000 | 0.35 | 0.30 | 0.011 | 0.05 |
| | 25% | $7.13 | 60,500,000 | 0.39 | 0.34 | 0.012 | 0.06 |
| | 50% | $8.55 | 53,600,000 | 0.42 | 0.36 | 0.012 | 0.06 |
| | 75% | $9.98 | 47,100,000 | 0.45 | 0.38 | 0.013 | 0.06 |

*Note: Using a $5.70 per ton baseline NSR cut-off for oxide, mixed and sulfide material.*

## 14.22 Comparison with the 2012 Resource Estimates

A review and comparison of 2017 Hudbay mineral resource and 2012 Augusta mineral resource was completed. The results (Table 14-42) of measured and indicated resources show that Hudbay reports a tonnage 29% higher, with copper grades 8% lower to those estimated in 2012. Molybdenum and silver grades are 17% and 4% lower than those reported in 2012. The 2017 oxide tonnage shows a difference of +137% with +106% copper grades.

**TABLE 14-42: MEASURED AND INDICATED, COMPARISON TO 2012 AUGUSTA ESTIMATE**

| Mineralized Zone | Hudbay Resource 2017 Model | | | | | Augusta Resource 2012 Model | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NSR ($/ton) | Tons | Cu (%) | Mo (%) | Ag (opt) | CuEq Cut-Off (%) | Tons | Cu (%) | Mo (%) | Ag (opt) |
| Oxide | > = $5.70 | 150,400,000 | 0.35 | | | >=0.10 | 63,400,000 | 0.17 | | |
| Mixed | > = $5.70 | 25,800,000 | 0.39 | 0.008 | 0.07 | >=0.30 | 49,900,000 | 0.53 | 0.007 | 0.05 |
| Sulfides | > = $5.70 | 1,088,400,000 | 0.36 | 0.012 | 0.11 | >=0.15 | 869,300,000 | 0.40 | 0.014 | 0.11 |

The difference in sulfide measured and indicated tonnage is partially a result of reinterpretation of the oxide blanket surface, as discussed in Section 7 of this report. Molybdenum grades are lower as a result of factoring of historical molybdenum assays, as discussed in Section 11 of this report. Silver grades are lower as a result of using regression against copper to assign values to samples with missing silver assays.

The significantly higher oxide tonnage and grade compared to the 2012 estimate was a result of lowering of the oxide blanket surface. The reduction of tons in the Inferred category is a direct result of the infill drilling completed by Hudbay in 2014 and 2015. The comparison is shown in Table 14-43.

**TABLE 14-43: INFERRED, COMPARISON TO 2012 AUGUSTA ESTIMATE**

| Mineralized Zone | Hudbay Resource 2017 Model | | | | | Augusta Resource 2012 Model | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | NSR ($/ton) | Tons | Cu (%) | Mo (%) | Ag (opt) | CuEq Cut-Off (%) | Tons | Cu (%) | Mo (%) | Ag (opt) |
| Oxide | > = $5.70 | 6,400,000 | 0.31 | | | >=0.10 | 1,100,000 | 0.15 | | |
| Mixed | > = $5.70 | 1,600,000 | 0.44 | 0.004 | 0.02 | >=0.30 | 10,100,000 | 0.39 | 0.006 | 0.02 |
| Sulfides | > = $5.70 | 74,100,000 | 0.30 | 0.011 | 0.05 | >=0.15 | 128,400,000 | 0.40 | 0.013 | 0.10 |

## 14.23 Factors That May Affect the Mineral Resource Estimate

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

- 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 testwork
- Changes to the tonnage and grade estimates may vary as a result of more drilling, new assay and tonnage factor information
- Assumptions as to the ability to maintain patented mining claims and surface rights, access to the site, obtain environmental and other regulatory permits and obtain social license to operate

## 14.24 Conclusions

The mineral resource estimation is well-constrained by three-dimensional wireframes representing geologically realistic volumes of mineralization. Exploratory data analysis conducted on assays and composites shows that the wireframes are suitable domains for mineral resource estimation. Grade estimation has been performed using an interpolation plan designed to minimize bias in the average grade.

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 (typically less than 3%) between the NN, IDW and the OK models.
- Checks for local bias (swath plots) indicate good agreement between the NN, IDW and OK for all variables.

- A review and comparison of the 2017 Hudbay estimate and 2012 Augusta estimate showed that the 2017 block model has greater tonnage and lower grades than previously reported. The oxide tonnage and grade are significantly higher in the 2017 model mainly due to lowering of the oxide blanket surface.

The impact of grade capping was evaluated by estimating uncapped and capped assay data. Generally, the amounts of metal removed by capping does not exceed 5%.

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 Project conform to the requirements of 2014 CIM Definition Standards – for Mineral Resources and Mineral Reserves and requirements in Form 43-101F1 of NI 43-101, Standards of Disclosure for Mineral Projects.

## 14.25 Recommendations

The author recommends that Hudbay further investigate the cause(s) of the differences in average molybdenum grade of the historical assays. Hudbay should also evaluate the application of non-linear interpolation or wireframing methods in the minor geological units.

It is also recommended that Hudbay further investigate change-of-support correction and alternative approach to resource classification taking into account the high production rate. This should be performed to ensure that the resource classification properly reflect the reduced risk when a large volume is mined and delivered to the mill on a quarterly and annual basis.

Finally, in order to better understand the distribution of gold with sufficient confidence, the following steps should be taken:

1. Select drill hole intervals located in zones that will be mine as ore and sent to the mill and perform gold analysis on the pulps to test the robustness of the proxy model.
2. Perform a variography analysis of the new dataset and re-interpolate the gold grade following the same method as described in this Technical Report.

# 15 MINERAL RESERVES ESTIMATE

The Mineral Reserves estimate for the Project are based on a LOM which uses the block model described in Section 14, Mineral Resource Estimates, with economic value calculation per block (NSR in $/ton) and mining, processing, and engineering detail parameters. The mineral reserve economics are described in Section 22.

This Mineral Reserves estimate has been determined and reported in accordance with NI 43-101 and the classifications adopted by CIM Council in November 2014. NI 43-101 defines Mineral Reserves as “the economically mineable part of measured and indicated mineral resources.”

The Mineral Reserves estimate for the Project, which is presented in this report, was prepared by Hudbay (Javier Toro - Director, Technical Services) and under the supervision of Cashel Meagher.

This Technical Report includes refinements of certain aspects of the Project’s mine plan. While consistency with issued and pending environmental permits and analysis related thereto has always been a key requirement for this effort, updates to the original mine plan will be necessary. To the extent that any regulatory agency concludes that the current plan requires additional environmental analysis or modification of an existing permit, the intent will be to work with that agency to either complete the required process or to adjust the current mine plan as necessary.

## 15.1 Pit Optimization

Revenue created from the Project will be generated from the sale of copper and molybdenum concentrates to smelters and roasters who will further refine the product. In addition, the copper concentrate contains payable silver quantities.

Pit optimization of multi-element revenue generating deposits like Rosemont can either be performed on the grade equivalent of all the revenue generating elements expressed in terms of the predominant metal (copper in Rosemont), or on the NSR.

A copper grade equivalent optimization model is simpler to implement than a NSR model but is not able to adequately represent the many variables used in the calculation of revenues as a NSR model can. Hudbay has therefore decided to use a NSR optimization model despite its additional complexity.

LG analyses were conducted using the Rosemont deposit model (described in Section 14) to determine the ultimate pit limits and best extraction sequence for open pit mine design (six pit phases were selected). Only mineral resources classified as Measured or Indicated were considered as potential ore in the LG analyses; all inferred resources were treated as waste.

### 15.1.1 Block model

The Block Model used for the Mineral Reserves estimation has the original Mineral Resources estimation described in Section 14 as a base, which has a Selective Mining Unit ("SMU") of 50x50x50 feet.

The optimized models, which were created to simulate the actual mining practice by utilizing the SMU block sizes, were considered undiluted models.

An economic subroutine was developed to compute a NSR value for each block in the deposit model. This computer algorithm incorporates block grades, expected smelting/refining contracts (i.e., payables and deductions), metallurgical recoveries and projected market prices for each metal (Cu, Mo and Ag) to yield a net revenue value expressed in terms of US Dollars per ton. The subroutine also applies to mining, ore processing and general/administration costs to calculate a net dollar value per block, which includes adjustments for surface topography. Concurrently, a NSR value in $ per ton is computed and stored in the block model.

### 15.1.2 Metallurgical Recoveries

Metal recoveries were derived from metallurgical testwork conducted by XPS. These tests included: grinding and flotation testwork. The metallurgical testwork is fully described in Section 13.

Based on results from this testwork, Table 15-1 presents the metallurgical recoveries used in the LG evaluations and subsequent mineral reserve estimation. Only the three primary metals, copper, molybdenum and silver were modelled and used in the revenue calculations. No recovery of molybdenum or silver from oxide ore was projected.

**TABLE 15-1: METALLURGICAL RECOVERIES USED IN LERCHS-GROSSMAN EVALUATIONS**

| Metal | Oxide Ore | Sulfide Ore | Mixed Ore |
|---|---|---|---|
| Copper[1] | 90.0 % | 90.0 % | 90.0 % |
| Molybdenum | - | 63.0 % | 30.0 % |
| Silver | - | 75.5 % | 38.0 % |

Note: 1. Expressed as recoveries of the quantity of copper contained in sulfides.

### 15.1.3 Economic Parameters

Table 15-2 summarizes the economic parameters and offsite costs used in the base-case LG evaluations of the Rosemont deposit.

**TABLE 15-2: BASE-CASE LERCHS-GROSSMAN ECONOMIC PARAMETERS**

| Parameter | Units | Value |
|---|---|---|
| **Revenue** | | |
| **Metal Price** | | |
| Copper | $/lb | 3.15 |
| Molybdenum | $/lb | 11.00 |
| Silver | $/oz | 18.00 |
| **Payable Contained Metal** | | |
| Copper | % | 96.5% |
| Molybdenum | % | 99.0% |
| Silver | % | 90.5% |
| **Concentrate grades** | | |
| Copper | % | 30% |
| *Moly concentrate grade* | | |
| Molybdenum | % | 45% |
| **Concentrate Moisture Content** | | |
| Copper concentrate | % | 8.0% |
| Moly concentrate | % | 8.0% |
| **Smelting Charges** | | |
| Smelting charges - Cu conc (dry) | $/dst Cu conc | 72.57 |
| Roasting charges - Mo conc (dry) | $/dst Mo conc | 1.50 |
| **Marketing Cost** | $/dst Cu conc | 5.08 |
| **Selling Cost (Freight)** | | |
| Transport Cu conc | $/dst conc | 137.55 |
| **Refining charges** | | |
| Cu | $/lb Cu | 0.08 |
| Ag | $/oz Ag | 0.50 |
| *S+T+R cost* | *$/lb Cu* | *0.4517* |
| **Royalties** | | |
| Royalties | % of NSR | 3.0% |
| | | |
| **Cost** | | |
| **Mining Cost** | | |
| Ore | $/tmined | 1.14 |
| Waste | $/tmined | 1.14 |
| **Incremental Cost by Bench** | | |
| Up | $/tmined | - |
| Down | $/tmined | 0.024 |
| **G&A Cost** | | |
| Ore | $/milled | 1.00 |
| **Process Cost** | | |
| Sulfide | $/milled | 5.00 |
| Mixed | $/milled | 5.00 |
| Oxide | $/milled | 5.00 |

The in-situ NSR value is first calculated and coded into each block in the model. This is to allow the pit optimization of the multi-element Rosemont deposit to be carried out on the in-situ NSR values. The following process is the procedure that was developed in order to achieve the NSR calculation:

- In-situ NSR is the net value of metals contained in a concentrate produced from an ore block after smelting and refining. Using the concentrator recovery of the metals with the concentrate and the grade of the concentrate produced, the mass pull of each block in the resource model expressed in terms of tons of concentrate per ton of ore processed is first estimated.

- The value of the payable metals in the concentrate is then calculated based on agreed payable metal content in the concentrate subject to deductions with smelters, refineries and roasters. In the case of the copper concentrate, the payable precious metal silver is added to the value of the payable copper. For the molybdenum concentrate, only the molybdenum metal is payable.

- From the value of the payable metals, the selling costs, which include marketing costs, transportation costs, port charges, insurance costs, shipping costs, and smelting charges expressed in $/dmt concentrate and other deductions like the refining charges and price participation (if applicable) expressed in $/payable metal are taken out to obtain the gross concentrate NSR value (before royalties).

- The applicable royalties are then deducted from the gross concentrate NSR value to obtain the net concentrate NSR value (after royalty). The concentrate NSR value calculations described above are applied for both the copper and molybdenum concentrates.

- The concentrate NSR value after royalty for the copper and molybdenum concentrates are then multiplied by their respective mass pull expressed in tons of concentrate produced per tonne of ore processed to obtain the contribution of each metal in the concentrate to the in-situ NSR value.

- The in-situ NSR of each block in the normalized resource model is the sum of the in-situ NSR value from the copper concentrate and the molybdenum concentrate.

Only Measured and Indicated Resource model block categories with NSR values greater than their processing costs are considered potential ore while blocks which have NSR values less than their processing costs are considered waste.

Process plant recoveries, throughput, operating costs, and concentrate grades vary by ore type. Consistent with ore reserve reporting guidelines, only Measured and Indicated resources are coded to generate revenues in the NSR model. Inferred resources are coded and reported as waste.

Processing metal recoveries for copper and silver are fixed numbers depending on metallurgical domain while molybdenum is calculated by formula linked to molybdenum and copper feed grades, and copper recovery. Copper and silver grades in the copper concentrate are calculated by formula. The grade of molybdenum in the molybdenum concentrate is a fixed number.

#### 15.1.4 NSR Input Parameters

The revenue, recovery and cost input parameters used for pit optimization are shown in Table 15-1 and Table 15-2.

### 15.1.5 Pit Slope Guidance

Overall slope angles used on the LG evaluations were derived from the geotechnical recommendations made by CNI & Hudbay for pit slope designs. The overall slopes were adjusted to accommodate the recommended slope angles and the anticipated placement of internal haulage ramps along the pit walls in certain design sectors to be used as berms (step outs). Hudbay assigned slopes angles for each block of resources model, and a slope code was assigned to the block representing each of the pit slopes. The slope codes and pit slopes are then read as input to the LG analysis. The plan view and the design parameters by sector are shown in Figure 15-1 and Table 15-3 respectively.

**FIGURE 15-1: PLAN VIEW CONTOURS OF SELECTED LERCHS-GROSSMAN PIT SHELL**

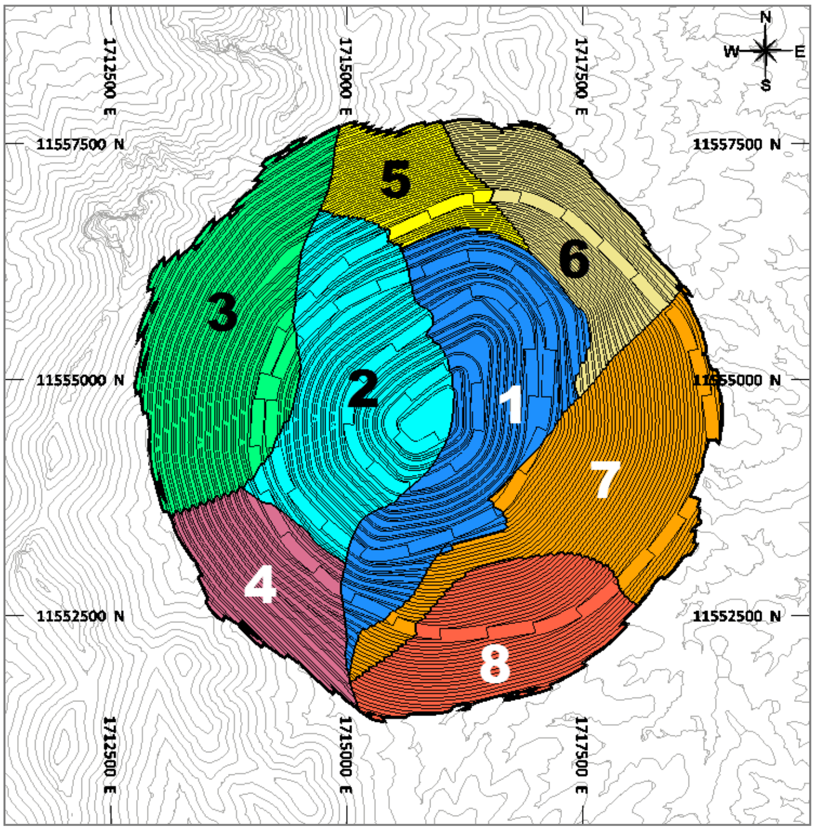



**TABLE 15-3: OVERALL SLOPE ANGLES USED IN LERCHS-GROSSMAN ANALYSIS**

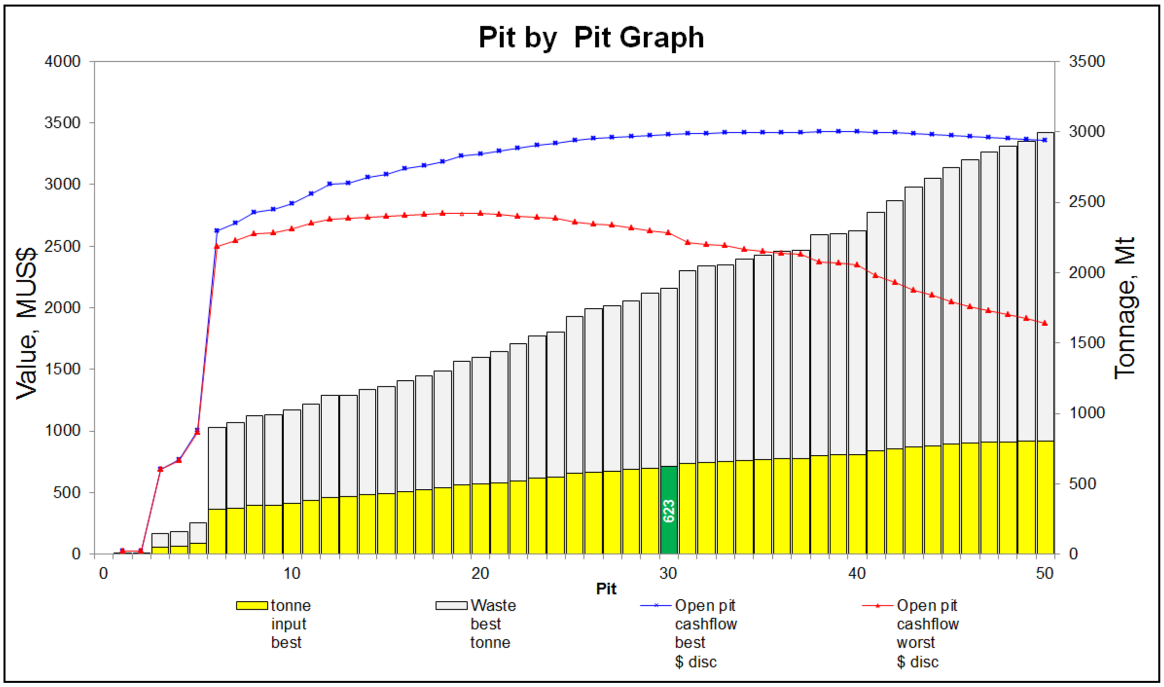
| Geotechnical Sector | Bench Height, feet | Bench Face Angle° | Inter-Ramp Slope Angle° | Catch Bench, feet | Overall Slope Angle° |
|---|---|---|---|---|---|
| 1 | 100 | 70 | 50 | 48 | 42 |
| 2 | 100 | 65 | 46 | 50 | 40 |
| 3 | 100 | 65 | 48 | 44 | 45 |
| 4 | 100 | 65 | 48 | 44 | 45 |
| 5 | 50 | 65 | 46 | 25 | 43 |
| 6 | 50 | 65 | 44 | 29 | 41 |
| 7 | 50 | 55 | 33 | 42 | 31 |
| 8 | 50 | 55 | 33 | 42 | 31 |

### 15.1.6 Lerchs-Grossman Analyses

All LG analyses were restricted to prevent the pit shells from crossing the topographic ridge immediately west of the deposit. This was done due to permit constraints.

The base-case LG pit shell 40 is defined by the recoveries and economic parameters listed in Table 15-1 and Table 15-2, respectively. This pit shell contains about 710 million tons of Measured and Indicated mineral resource above an internal NSR cut-off of $6.00/ton. The resulting stripping ratio is about 2.24:1 (tons waste per ton of ore). However, this is not the pit shell selected for pit design. Several economic analyses were developed for each nested pit. The purpose of this assessment was to evaluate free discounted cash flow, revenue, stripping ratio, development and sustaining capital. Figure 15-2 presents the results of the LG price and price sensitivity analyses, respectively.

**FIGURE 15-2: ROSEMONT WHITTLE RESULTS, REVENUE FACTOR SENSITIVITY**




Pit shell 30 was generated at a 0.80 revenue factor and contains approximately 622 M st of Ore and 1,269 M st of Waste. The pit shell captures about 99.3% of the Net Cash flow of the base revenue factor (RF) 1 pit shell 40. This pit generates a lower stripping ratio, better economics greater total revenue, and capital costs than other pits evaluated for the Project.

The selected LG pit shell 30 is shown in plan view in Figure 15-3 and in cross-section view in Figure 15-4.

**FIGURE 15-3: PLAN VIEW CONTOURS OF SELECTED LERCHS-GROSSMAN PIT SHELL**

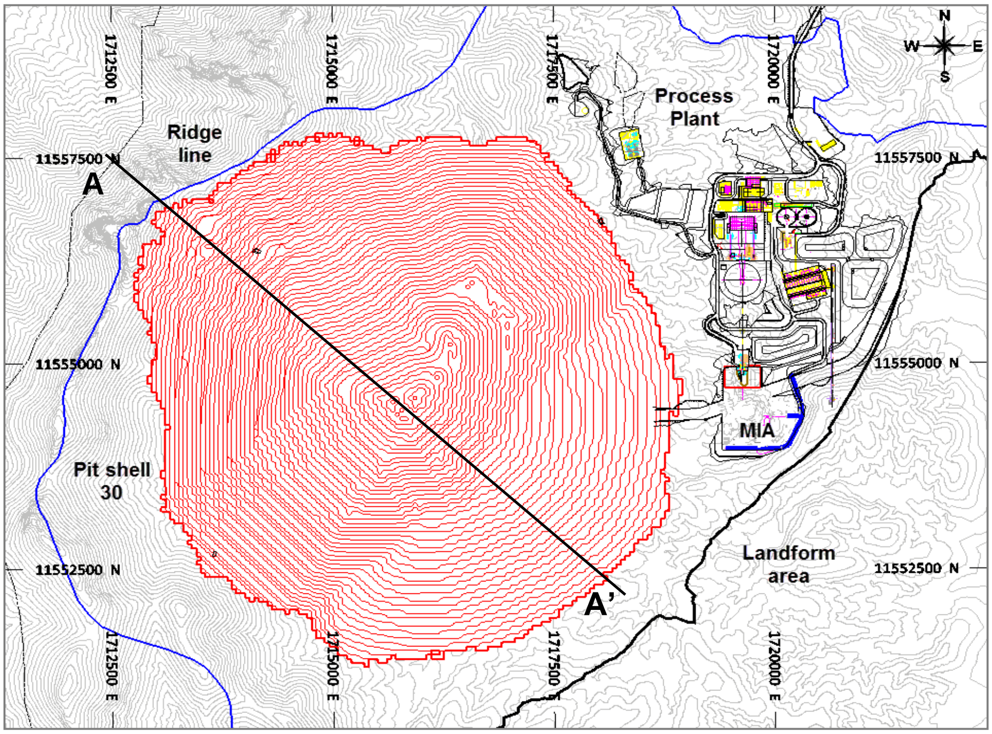
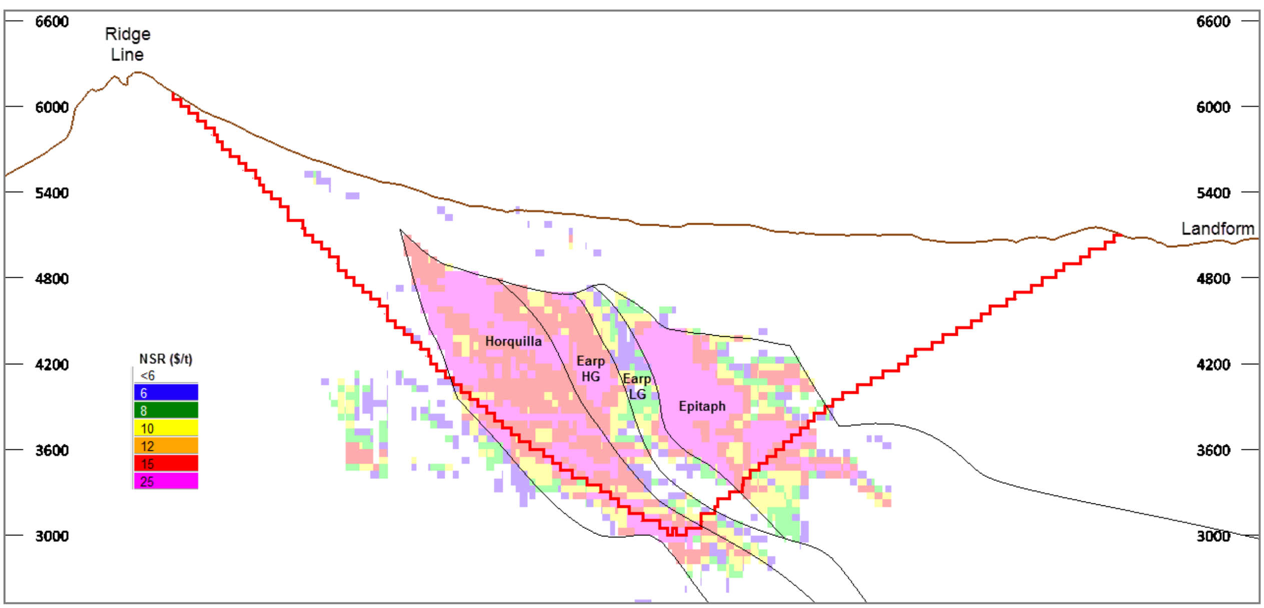



**FIGURE 15-4: AA' SECTION VIEW OF SELECTED LERCHS-GROSSMAN PIT SHELL**




### 15.1.7 Pit Design Criteria

Design criteria for final pit takes into consideration the geotechnical recommendations summarized earlier in Table 15-3, pushback (mine phase) width, phase sequence, haul roads and access, berms, ditches as well as other engineering considerations. Mine phase, or pushback, widths are typically

320 feet. The summary parameters used in the design of the ultimate pit are presented in Table 15-4.

**TABLE 15-4: PIT DESIGN PARAMETERS**

| Parameter | Value |
|---|---|
| Bench height<br>Bench face angle<br>Catch bench interval | 50 – 100 feet<br>55 – 70°<br>25 – 50 feet |
| Road width (including ditch & safety berm)<br>Nominal road gradient | 110 feet<br>10 % |
| Minimum pushback width | 320 feet |

## 15.2 Mineral Reserves

The Rosemont Mineral Reserves estimation is based on Measured and Indicated resources. Therefore, the potential exists for Inferred Mineral Resources within the ultimate pit to be included and reported as waste, as they currently do not meet the economic and mining requirements to be categorized as Mineral Reserves. It cannot be assumed that all or any part of Inferred mineral resources will ever be upgraded to a higher category.

The mining phase and ultimate pit designs were applied to the 3D resource block model of the deposit described in Section 14 to estimate contained tonnages and grades.

### 15.2.1 Ore Definition Parameters

The base-case price and operating cost estimates presented in Table 15-2 are used as the economic envelope to define ore in the mineral reserve estimates.

Mineralized oxide and mixed materials that are indicated to be economic (above an internal NSR cut-off of $6.00/ton) in the optimized pit analysis are included in the pit ore reserves for this study.

### 15.2.2 Material Densities

Bulk material densities, which vary by rock type, were read from values stored in the resource block model. These assignments are described in more detail in Section 14. Generally, rock tonnage factors range between 11.7 $ft^3$/ton and 12.4 $ft^3$/ton, with an average of 12.10 $ft^3$/ton for the rock contained within the ultimate pit.

### 15.2.3 Dilution

The Rosemont deposit is a well-disseminated polymetallic deposit that has large ore zones above the anticipated internal cut-off grade. With the planned bulk mining method, external ore dilution along the ore - waste contact edges is generally assessed to determine whether the feed grade from the run of mine production is adequately represented by those predicted from the resource block model.

The resource block model dimensions are 50x50x50 feet. The interpolated metal grade is averaged for the entire block. When the Project commences operations, ore feed will be delineated by implementing a detailed blasthole sampling program. Drill blast patterns will be smaller, 30 feet to 30 feet, than the resource block dimensions, thereby providing a better definition than from the resource model. This new definition will be provided by a new block model built by assays from blastholes projects, dynamic or short range block model, which is a common practice in Hudbay operations.

The author has confirmed that enough geological dilution is already incorporated in the resource model due to the smoothing effect of kriging. Based on experience in similar types of skarn deposits and scale of operation, it is reasonable to use the resource tonnes and grade from the individual 50 ft x 50 ft x 50 ft blocks from the resource model without any additional adjustment for ore losses or mining dilution.

#### 15.2.4 Mineral Resource and Mineral ReserveStatement

Proven and probable mineral reserves for the Rosemont deposit are summarized in Table 15-5. Proven and probable mineral reserves within the designed final pit total 592 million tons grading 0.45% Cu, 0.012% Mo and 0.13 oz Ag/ton. There are 1.25 billion tons of waste material, resulting in a stripping ratio of 2.1:1 (tons waste per ton of ore). Total material in the pit is 1.84 billion tons. Contained metal in proven and probable mineral reserves is estimated at 5.30 billion pounds of copper, 142 million pounds of molybdenum and 79 million ounces of silver.

Nearly 80% of the mineral reserves in the Rosemont ultimate pit are classified as proven with the remaining 20% identified as probable. The classifications are based on the exploration drilling in the Rosemont deposit. *All of the mineral reserves estimate reported are contained in the mineral resource estimates presented in Section 14.*

The Rosemont ultimate pit contains approximately 10 million tons of inferred mineral resources that are above the $6.00/ton NSR cut-off value for ore. *Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves.*

The mineral reserves estimate presented in this report is dependent on market prices for the contained metals, metallurgical recoveries and ore processing, mining and general/administration cost estimates. Mineral reserve estimates in subsequent evaluations of the Rosemont deposit may vary according to changes in these factors. As of the effective date of this report, there are no other known mining, metallurgical, infrastructure or other relevant factors that may materially affect the mineral reserve estimates.

Proven and Probable mineral reserves for the Rosemont deposit are summarized in Table 15-5 and classified by ore type in Table 15-6. Illustrations of the ultimate pit in plan and section view against economic shell 30 are shown in Figure 15-5 and Figure 15-6.

### TABLE 15-5: PROVEN AND PROBABLE MINERAL RESERVES IN ROSEMONT FINAL PIT

| | Short Tons | TCu[1] % | SCu[2] % | ASCu[3] % | Mo % | Ag opt | NSR $/t | CuEq[4] % |
|---|---|---|---|---|---|---|---|---|
| **Proven** | 469,708,117 | 0.48 | 0.43 | 0.05 | 0.012 | 0.14 | 22.1 | 0.56 |
| **Probable** | 122,324,813 | 0.31 | 0.28 | 0.03 | 0.010 | 0.09 | 14.7 | 0.38 |
| **Total** | **592,032,930** | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.57** | **0.53** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.

### TABLE 15-6: PROVEN AND PROBABLE MINERAL RESERVES IN ROSEMONT FINAL PIT BY ORE TYPE

| Ore Type | Short Tons | TCu[1] % | SCu[2] % | ASCu[3] % | Mo % | Ag opt | NSR $/t | CuEq[4] % |
|---|---|---|---|---|---|---|---|---|
| **Sulfide** | **542,969,276** | **0.45** | **0.41** | **0.04** | **0.013** | **0.14** | **21.5** | **0.53** |
| Proven | 431,620,325 | 0.49 | 0.45 | 0.04 | 0.013 | 0.15 | 23.1 | 0.57 |
| Probable | 111,348,950 | 0.31 | 0.29 | 0.03 | 0.011 | 0.09 | 15.1 | 0.38 |
| **Mixed** | **26,477,889** | **0.33** | **0.25** | **0.08** | **0.007** | **0.07** | **11.8** | **0.37** |
| Proven | 18,743,016 | 0.34 | 0.25 | 0.09 | 0.008 | 0.08 | 12.0 | 0.39 |
| Probable | 7,734,873 | 0.30 | 0.24 | 0.06 | 0.007 | 0.05 | 11.3 | 0.34 |
| **Oxide[5]** | **22,585,766** | **0.50** | **0.24** | **0.26** | **-** | **-** | **9.8** | **0.50** |
| Proven | 19,344,776 | 0.52 | 0.24 | 0.28 | - | - | 9.9 | 0.52 |
| Probable | 3,240,990 | 0.38 | 0.22 | 0.17 | - | - | 8.9 | 0.38 |
| **Total** | **592,032,930** | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.6** | **0.53** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.
5. Oxide ore refers only to the sulfide copper species.

**FIGURE 15-5: PLAN VIEW OF ROSEMONT FINAL PIT AND ECONOMIC SHELL 30**

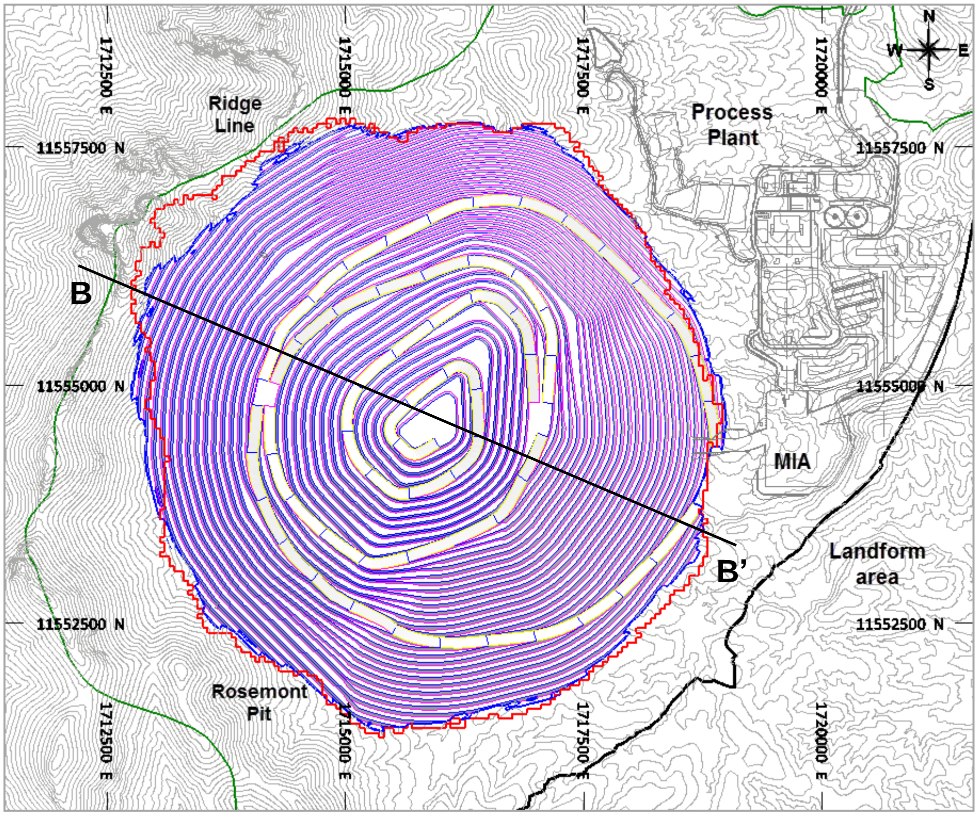



**FIGURE 15-6: SECTION VIEW BB' OF ROSEMONT FINAL PIT AND ECONOMIC SHELL 30**

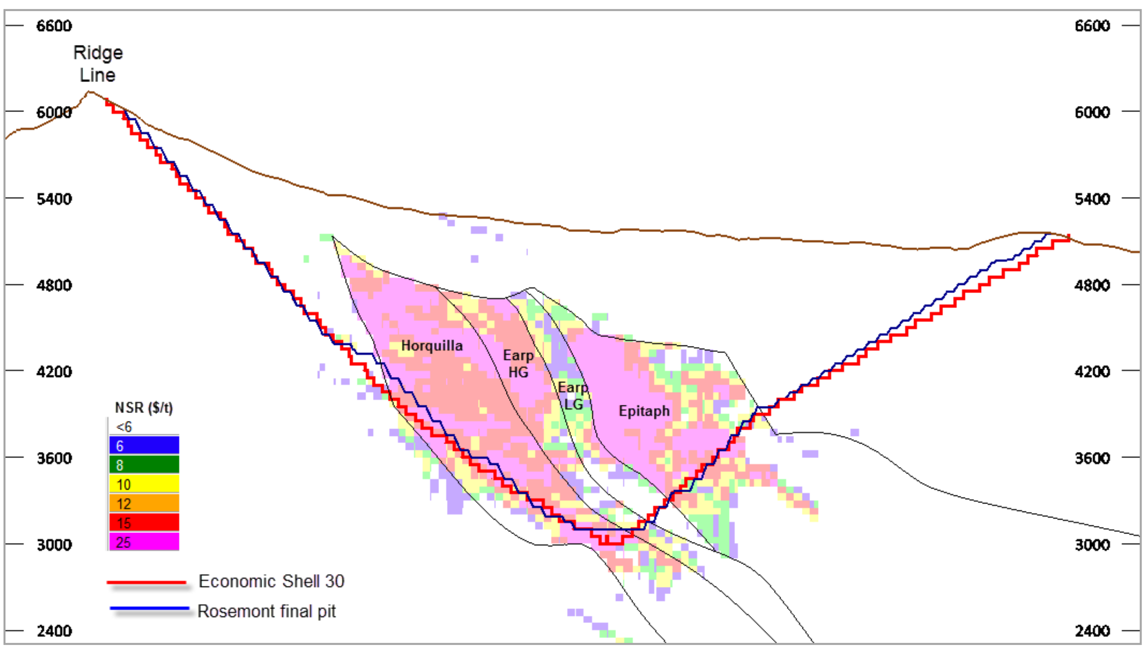



Table 15-7 presents the mineral resource estimates exclusive of the mineral reserve estimates, i.e. the mineral resources located inside the resource pit shell and outside of the pit design. It represents the portion of the mineral resources estimate with potential for economic extraction after the current mineral reserves estimate has been mined and processed.

**TABLE 15-7: ROSEMONT MINERAL EXCLUSIVE RESOURCE ESTIMATES**

| **Measured** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
|---|---|---|---|---|---|---|
| Oxide | 54,000,000 | > = $5.70 | 0.41 | 0.41 | | |
| Mix | 5,000,000 | > = $5.70 | 0.45 | 0.41 | 0.008 | 0.047 |
| Hypogene | 118,700,000 | > = $5.70 | 0.44 | 0.36 | 0.014 | 0.117 |
| **Summary** | **177,700,000** | | **0.43** | **0.38** | **0.009** | **0.079** |
| | | | | | | |
| **Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 18,600,000 | > = $5.70 | 0.27 | 0.27 | | |
| Mix | 2,600,000 | > = $5.70 | 0.36 | 0.34 | 0.005 | 0.037 |
| Hypogene | 392,000,000 | > = $5.70 | 0.31 | 0.25 | 0.012 | 0.080 |
| **Summary** | **413,200,000** | | **0.31** | **0.25** | **0.011** | **0.076** |
| | | | | | | |
| **Measured + Indicated** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 72,700,000 | > = $5.70 | 0.38 | 0.38 | | |
| Mix | 7,600,000 | > = $5.70 | 0.42 | 0.38 | 0.007 | 0.044 |
| Hypogene | 510,700,000 | > = $5.70 | 0.34 | 0.27 | 0.012 | 0.088 |
| **Summary** | **591,000,000** | | **0.35** | **0.29** | **0.011** | **0.077** |
| | | | | | | |
| **Inferred** | **TONS** | **NSR Cut Off** | **CuEq (%)** | **Cu (%)** | **Mo (%)** | **Ag (opt)** |
| Oxide | 3,500,000 | > = $5.70 | 0.33 | 0.33 | | |
| Mix | 1,300,000 | > = $5.70 | 0.47 | 0.45 | 0.004 | 0.019 |
| Hypogene | 63,900,000 | > = $5.70 | 0.35 | 0.29 | 0.011 | 0.049 |
| **Summary** | **68,700,000** | | **0.35** | **0.30** | **0.010** | **0.046** |

Notes:

1. Domains were modelled in 3D to separate mineralized rock types from surrounding waste rock. The domains were based on core logging, structural and geochemical data.
2. Raw drill hole assays were composited to 25-foot lengths broken at lithology boundaries.
3. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
4. Block grades for copper, molybdenum and silver were estimated from the composites using OK interpolation into 50 ft x 50 ft x 50 ft blocks coded by domain.
5. Tonnage factors were interpolated by lithology and mineralized zone. Tonnage factors are based on 2,066 measurements collected by Hudbay and previous operators.
6. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
7. Mineral resources are constrained within a computer generated pit using the LG algorithm. Metal prices of $3.15/lb copper, $11.00/lb molybdenum and $18.00/troy oz silver. Metallurgical recoveries of 85% copper, 60% molybdenum and 75% silver were applied to sulfide material. Metallurgical recoveries of 40% copper, 30% molybdenum and 40% silver were applied to mixed material. A metallurgical recovery of 65% for copper was applied to oxide material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, and silver combined. Cut-off grades were set in terms of NSR based on current estimates of process recoveries, total process and G&A operating costs of $5.70/ton for oxide, mixed and sulfide material.
8. The oxide resource will be processed in the mill via flotation.
9. Totals may not add up correctly due to rounding.

### 15.2.5 Factors That May Affect the Mineral Reserves Estimate

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

- 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 testwork.
- Changes to the tonnage and grade estimates may vary as a result of more drilling, new assay and tonnage factor information.

### 15.2.6 Comparison with the 2012 Mineral Reserves

A review and comparison of 2017 Hudbay mineral reserves and 2012 Augusta mineral reserves was completed. The results (Table 15-8) of proven and probable reserves show that Hudbay reports a tonnage 11% lower, with copper grades 2% higher. Molybdenum and silver grades are 17% lower and 11% higher, respectively, to those estimated in 2012.

**TABLE 15-8: PROVEN AND PROBABLE, COMPARISON TO 2012 AUGUSTA RESERVE ESTIMATE**

| Category | Hudbay Reserves 2017 | | | | Augusta Reserves 2012 Model | | | |
|---|---|---|---|---|---|---|---|---|
| | Tons | Cu (%) | Mo (%) | Ag (opt) | Tons | TCu (%) | Mo (%) | Ag (opt) |
| Proven | 469,708,117 | 0.48 | 0.012 | 0.14 | 308,075,000 | 0.46 | 0.015 | 0.12 |
| Probable | 122,324,813 | 0.31 | 0.010 | 0.09 | 359,131,000 | 0.42 | 0.014 | 0.12 |
| **TOTAL** | **592,032,930** | 0.45 | 0.012 | 0.13 | **667,206,000** | 0.44 | 0.014 | 0.12 |

The changes between 2012 and 2017 Reserves estimate can be mostly attributed to a revision of the mining, processing and general & administration cost assumptions, resulting in a marginally higher cut-off in 2017.

# 16 MINING METHODS

## 16.1 Mine Overview

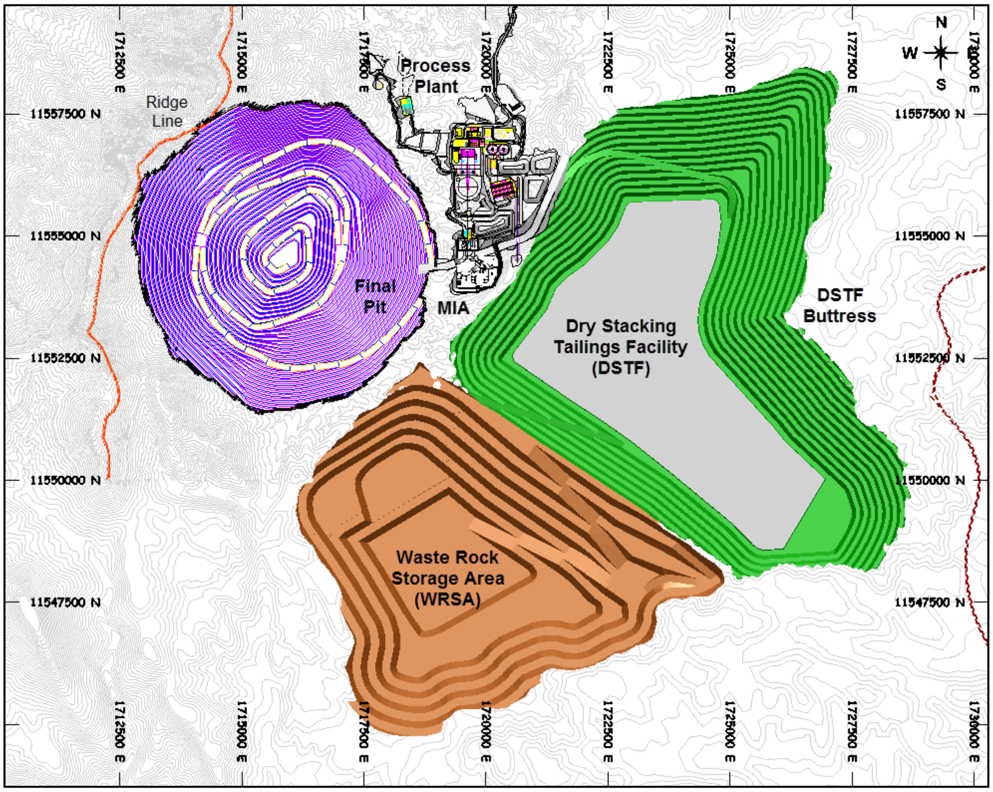
The Rosemont deposit is a large tonnage, skarn-hosted, porphyry-intruded, copper-molybdenum deposit located in close proximity to the surface. The Project will be a traditional open pit shovel/truck operation. The Project consists of open pit mining and flotation of sulfide minerals to produce commercial grade concentrates of copper and molybdenum. Payable silver will report to the copper concentrate.

The proposed pit operations will be conducted from 50-foot-high benches using large-scale mine equipment, including: 10-5/8-inch-diameter rotary blast hole drills, 60 $yd^3$ class electric mining shovels, 46 $yd^3$ class hydraulic shovel, 25 $yd^3$ front-end loader, and 260-ton capacity off-highway haul trucks.

The Rosemont final pit will measure approximately 6,000 feet east to west, 6,000 feet north to south, and have a total depth of approximately 2,900 feet down to 3,100 feet (AMSL). There is one primary WRSA, which is located 1,200 feet south east of the Rosemont final pit. The processing facility is located approximately 1,000 feet east of the final pit, while the dry stack tailings facility ("DSTF") is located 1,500 feet southeast of the Rosemont pit. The final pit and facilities can be seen in Figure 16-1.

The mine production plan contains 592 million tons of ore and approximately 1.25 billion tons of waste, yielding a life of mine waste to ore stripping ratio of 2.1 to 1. The mine has a 19-year life (including pre-stripping period but excluding initial haul road development), with ore to be delivered to the processing plant at a throughput of 90,000 tpd. Mine operations are scheduled for 24 hours per day, 365 days per year. A mining rate of 132 million tons per year through year 11 will be required to provide the assumed nominal process feed rate of 32.9 million tons of ore per year. From year 12 through year 18, the annual mining rate decreases due to lower stripping ratios, starting with an average of 50 million tons per year and ending with approximately 33 million tons in production year 18. Ore shortfall will be made up from stockpiled ore.

**FIGURE 16-1: ROSEMONT MINE PLAN SITE LAYOUT**




## 16.2 Mine Phases

### 16.2.1 Design Criteria

Mine phases and ultimate pit for the Project are designed for large-scale mining equipment (specifically, 60 $yd^3$ class electric shovels and 260-ton haulage trucks) and are derived from the selected LG pit shells described in the previous section. The design process included smoothing pit walls, eliminating or rounding significant noses and notches that may affect slope stability, and providing access to working faces by developing internal ramps (dual ramp for final pit). The summarized parameters used in the design of mine pit phases are presented in Table 16-1.

**TABLE 16-1: PIT DESIGN PARAMETERS**

| Parameter | Value |
|---|---|
| Bench height<br>Bench face angle<br>Catch bench interval | 50 – 100 feet<br>55 – 70°<br>25 – 50 feet |
| Road width (including ditch & safety berm)<br>Nominal road gradient | 110 feet<br>10 % |
| Minimum pushback width | 320 feet |

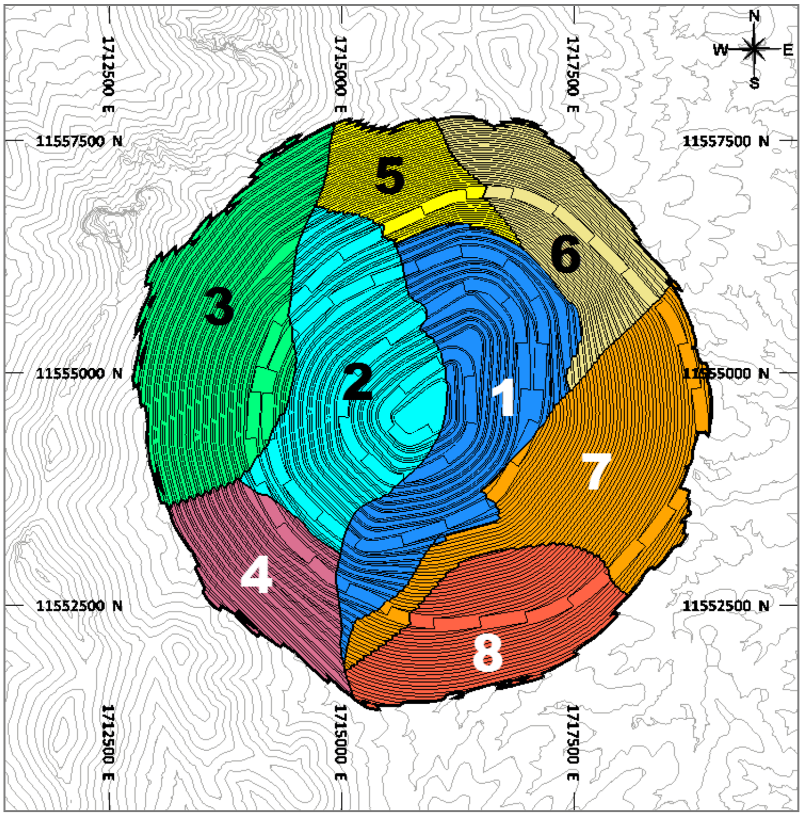
### 16.2.2 Pit Slopes Angles

For the pit design, the targeted minimum mining width is 320 feet and employs the wall slope design provided by CNI and Hudbay. Table 16-2 lists the configuration of the recommended pit slope configuration for each sector, and Figure 16-2 shows the Ultimate Pit Slope Design with the corresponding Geotechnical Sectors.

**TABLE 16-2: ROSEMONT SLOPE GUIDANCE**

| Geotechnical Sector | Bench Height, feet | Bench Face Angle° | Inter-Ramp Slope Angle° | Catch Bench, feet | Overall Slope Angle° |
|---|---|---|---|---|---|
| **1** | 100 | 70 | 50 | 48 | 42 |
| **2** | 100 | 65 | 46 | 50 | 40 |
| **3** | 100 | 65 | 48 | 44 | 45 |
| **4** | 100 | 65 | 48 | 44 | 45 |
| **5** | 50 | 65 | 46 | 25 | 43 |
| **6** | 50 | 65 | 44 | 29 | 41 |
| **7** | 50 | 55 | 33 | 42 | 31 |
| **8** | 50 | 55 | 33 | 42 | 31 |

**FIGURE 16-2: ROSEMONT GEOTECHNICAL SECTORS**




### 16.2.3 Mine Phases and Ultimate Pit

Six mining phases define the extraction sequence for the Rosemont deposit. The phase development strategy consists of extracting the higher metal grades along with minimum strip ratios during the initial years to maximize the economic benefits of the ore-body, while enabling smooth transitions in waste stripping throughout the life of the mine to ensure enough ore exposure for mill feed.

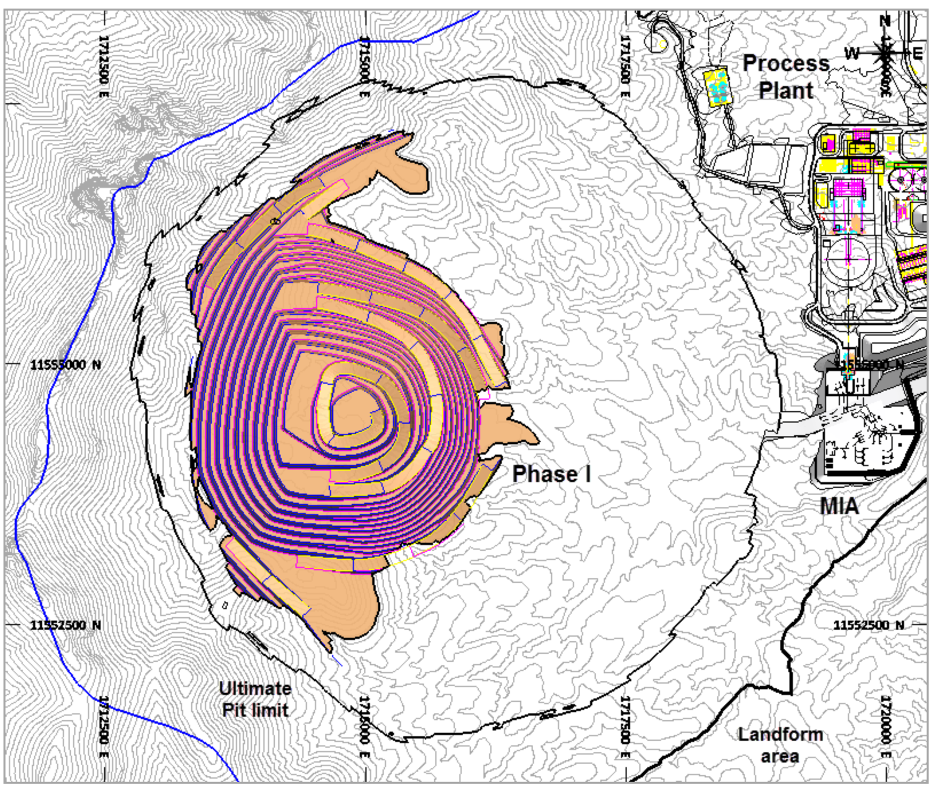
**Mine Phase 1**

The starter pit, Phase 1, is fit approximately to the LG pit shell defined by a $1.26/lb Cu price (equivalent to 40% of base metal price sensitivity case). This pit is located about 3,500 feet west of the primary crusher and ranges in elevation from 5,800 to 4,350 feet AMSL. The phase is approximately 3,000 feet wide east-west and 4,000 feet north-south. The upper benches will be dozed down until haul road access can be developed to the 5,700 feet elevation (AMSL). Phase 1 will develop approximately 85 million tons of ore at a stripping ratio of 2.2:1 (tons waste per ton of total ore). An illustration of the Phase 1 pit is shown in Figure 16-3.

Phase 1 material will be accessed via a haul road, 2C, which will be constructed from the pit exit eastward to the primary crusher. This road will also branch off towards the WRSA. These roads will be used for the life of the Project, and will also be extended to access the DSTF.

The pit entrance is at the 5,150 feet elevation (AMSL), and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,950, and 4,650 feet elevations (AMSL) before reversing to a counter clockwise direction to the bottom of the pit. All benches are accessed by a double lane width haul road.

**FIGURE 16-3: PLAN VIEW OF MINING PIT PHASE 1**

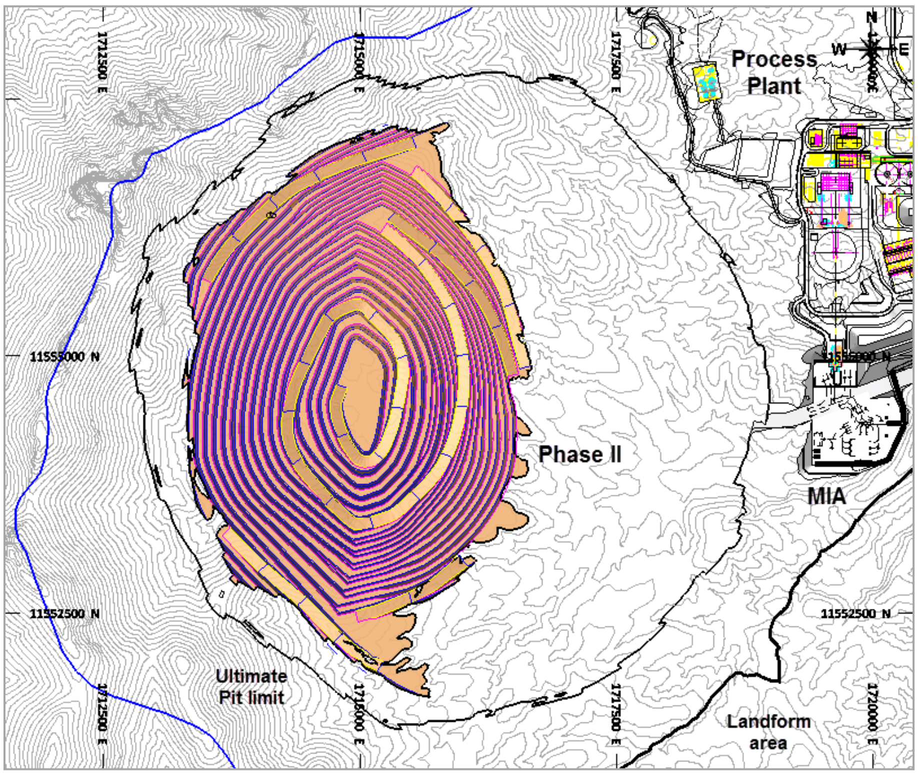


**Mine Phase 2**

Mining Phase 2 will expand the pit roughly 600 feet to the north, 400 feet to the east and 500 feet to the southeast. Bench toe elevations will range from 5,450 to 4,050 feet (AMSL). The phase is 2,500 feet wide east-west and 4,000 feet north-south. Phase 2 will supply over 88 million tons of ore. The average stripping ratio for this pushback is 1.3:1. An illustration of the Phase 2 pit is shown in Figure 16-4.

The pit entrance is at the 5,150 feet elevation (AMSL), and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,950, and 4,650 feet elevations (AMSL).

**FIGURE 16-4: PLAN VIEW OF MINING PIT PHASE 2**

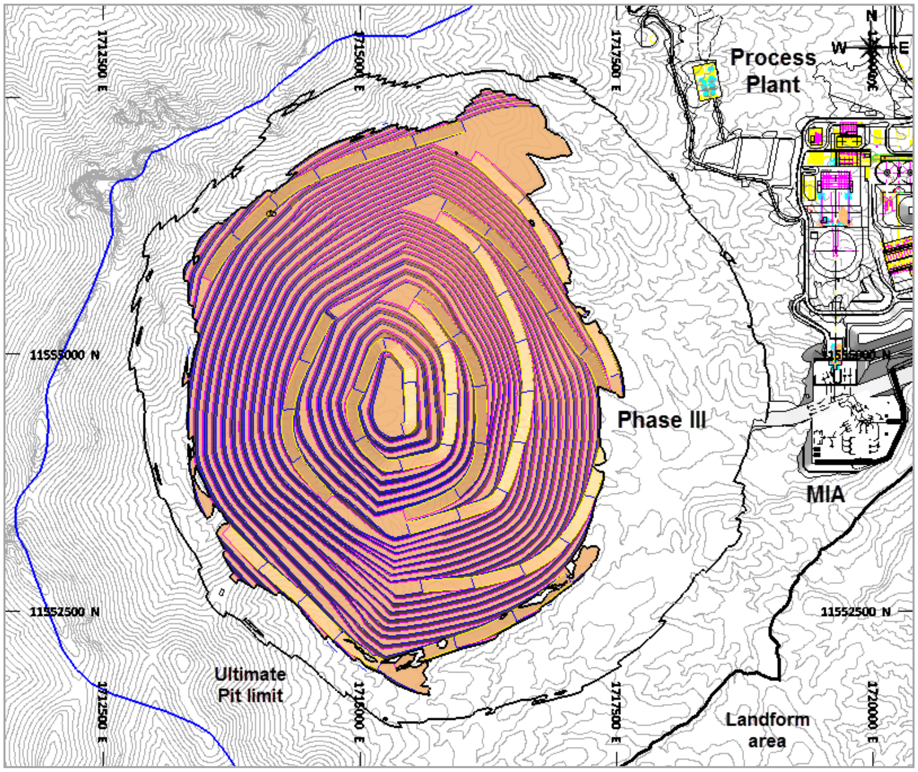



**Mine Phase 3**

The open pit is further expanded 500 to 600 feet to the east with the development of Phase 3. The easternmost limits of this pushback lie about 2,500 feet west of the primary crusher. Benches will range between 5,500 and 3,750 feet toe elevations (AMSL). The phase is 3,400 feet wide east-west and 5,000 feet long north-south. Over 75 million tons of ore will be generated by Phase 03 at an average stripping ratio of 2.4:1.

The pit entrance is at the 5,150 feet elevation (AMSL), and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,950 feet elevation (AMSL) to the bottom of the pit. An illustration of the Phase 3 pit is shown in Figure 16-5.

**FIGURE 16-5: PLAN VIEW OF MINING PIT PHASE 3**

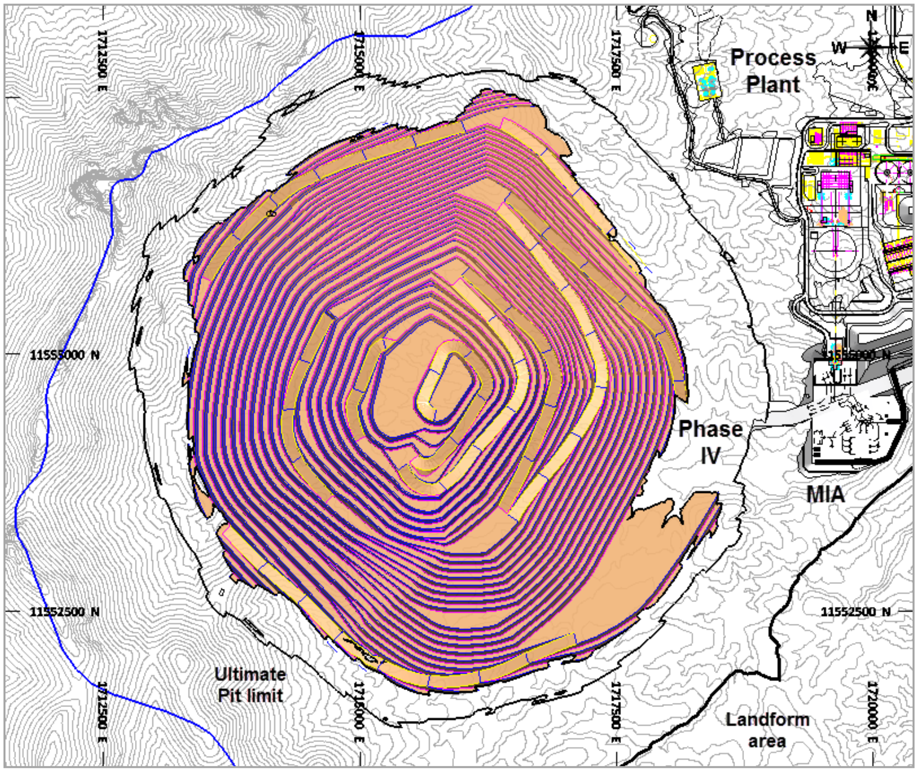



### Mine Phase 4

Phase 4 will expand the open pit about 600 feet to the east and 400 feet to the north. The easternmost limits of this pushback lie about 2,000 feet west of the primary crusher. Phase 4 benches range in elevation between 5,300 and 3,650 feet AMSL. The phase is 2,500 feet wide east-west and 5,500 feet north-south. Phase 4 will produce nearly 64 million tons of ore at a stripping ratio of 2.9:1. Phases 2, 3 and 4 fit approximately to the LG pit shell defined by a $1.30/lb Cu price (equivalent to 41% of base case metal price sensitivity). This expansion from the Phase 1 pit is split into 3 separate pushbacks, all in the same general direction. For each phase expansion, the ramp on the east side of the pit is re-developed. An illustration of the Phase 4 pit is shown in Figure 16-6.

The pit entrance is at the 5,100 feet elevation (AMSL), and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,850, 4,450, and 4,200 feet elevations (AMSL) before reversing to a clockwise direction to the bottom of the pit.

FIGURE 16-6: PLAN VIEW OF MINING PIT PHASE 4

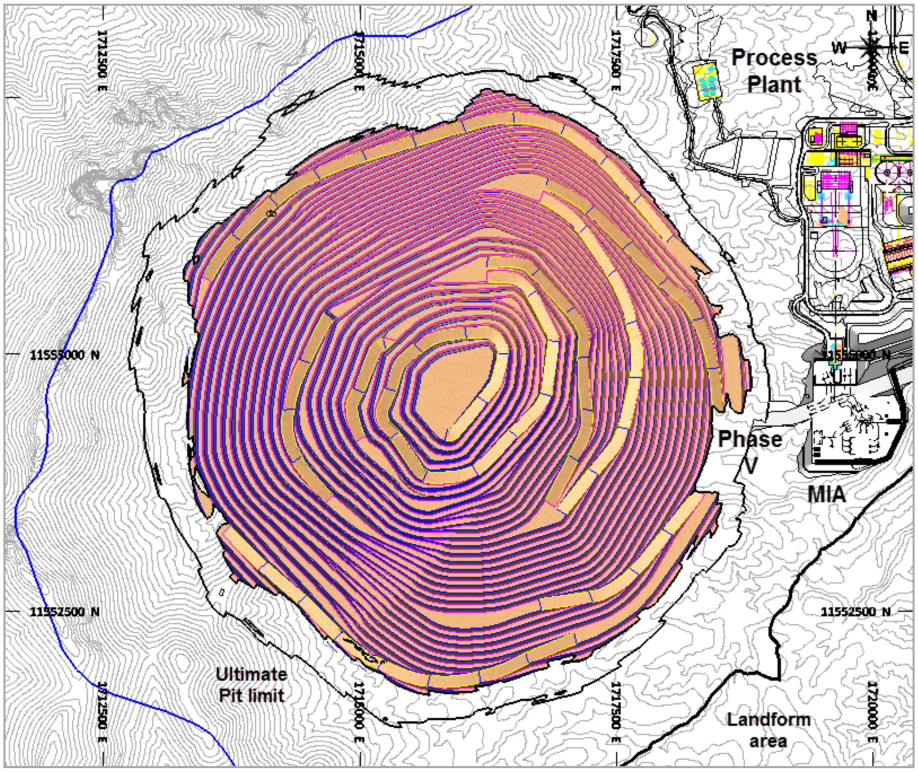



**Mine Phase 5**

Phase 5 is fit approximately to the LG pit shell defined by a $1.50/lb Cu price (equivalent to 47% of base case metal price value sensitivity). Mining Phase 5 expands the pit approximately 300 feet to the north and 600 feet to the east. The easternmost limits of this pushback lie about 1,200 feet west of the primary crusher. Phase 5 bench elevations range between 5,300 and 3,450 feet (AMSL). The phase is 3,000 feet wide east-west and 5,000 feet north-south. Phase 5 will produce nearly 60 million tons of ore at a stripping ratio of 2.5:1. The ramp on the east side of the pit is developed for this phase. An illustration of the Phase 5 pit is shown in Figure 16-7.

The pit entrance is at the 5,050 feet elevation (AMSL), and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at the 4,800, 4,450, 4,200 and 3,900-feet elevations (AMSL) before reversing to a counter clockwise direction to the bottom of the pit.

**FIGURE 16-7: PLAN VIEW OF MINING PIT PHASE 5**

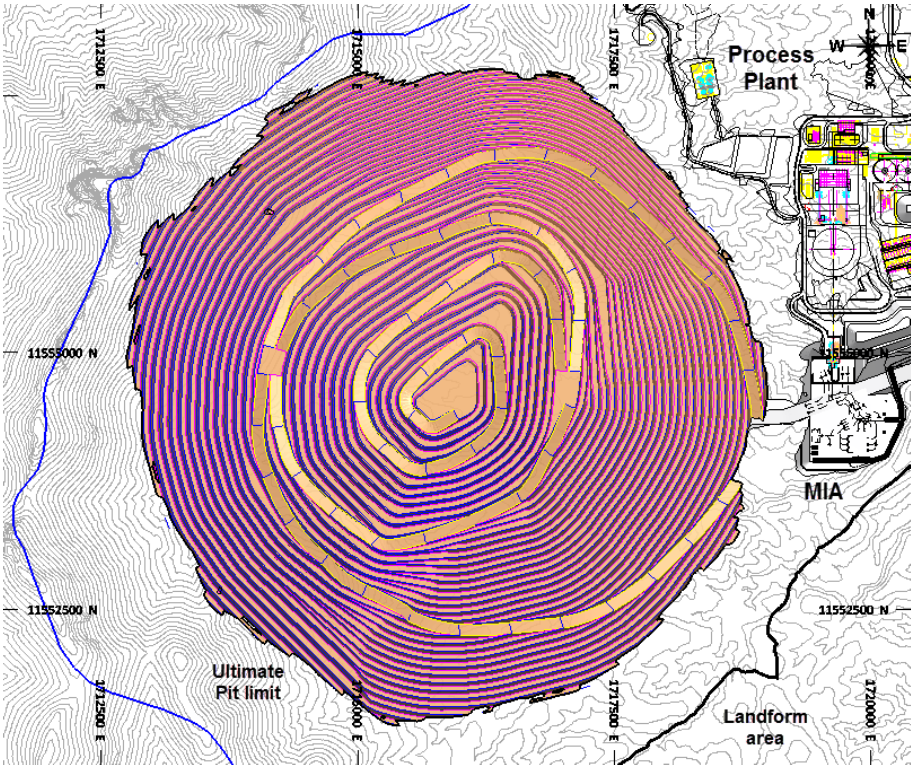



**Mine Phase 6 and Ultimate Pit**

The final pushback, Phase 6, extends the open pit from 300 to 600 feet along the east side to its ultimate limits and down to its maximum depth at the 3,100 feet elevation (AMSL). The ultimate pit will be about 6,000 feet wide east-west and 6,500 feet wide north-south. Phase 6 is fit approximately to the LG pit shell defined by a $2.52/lb Cu price (equivalent to 80% of base case metal price value sensitivity). Phase 6 will generate nearly 220 million tons of ore at a stripping ratio of 2.0:1. An illustration of the Phase 6 pit, or final pit, is shown in Figure 16-8.

Total ore reserves extracted from the six mining phases are estimated to be 592 million tons and will generate 1.25 billion tons of waste material. Approximately 55 million tons of medium and low grade oxide, mixed and sulfide ore will be stockpiled.

**FIGURE 16-8: PLAN VIEW OF MINING PIT PHASE 6 (ULTIMATE PIT)**




Final configuration of mine phases is presented in plan view in Figure 16-9 and in cross section in Figure 16-10. Mineral reserves for the Rosemont deposit by mine phase are summarized in Table 16-3 and classified by ore type in Table 16-4.

**FIGURE 16-9: PLAN VIEW OF ROSEMONT MINE PHASES**

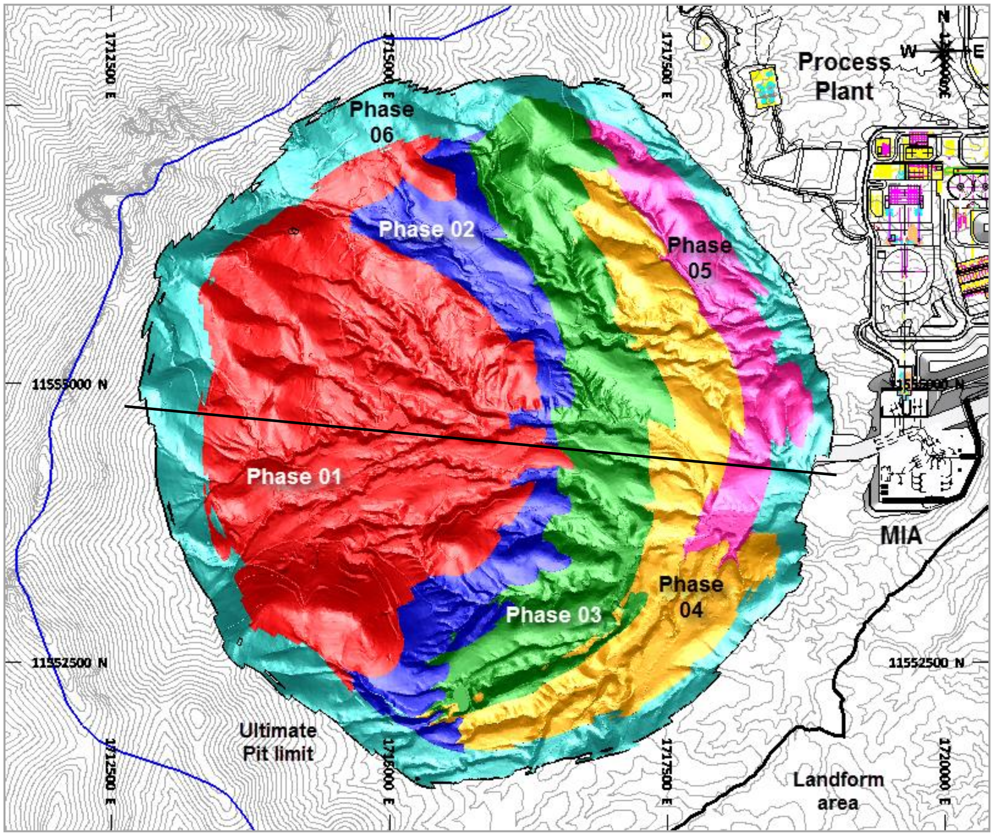
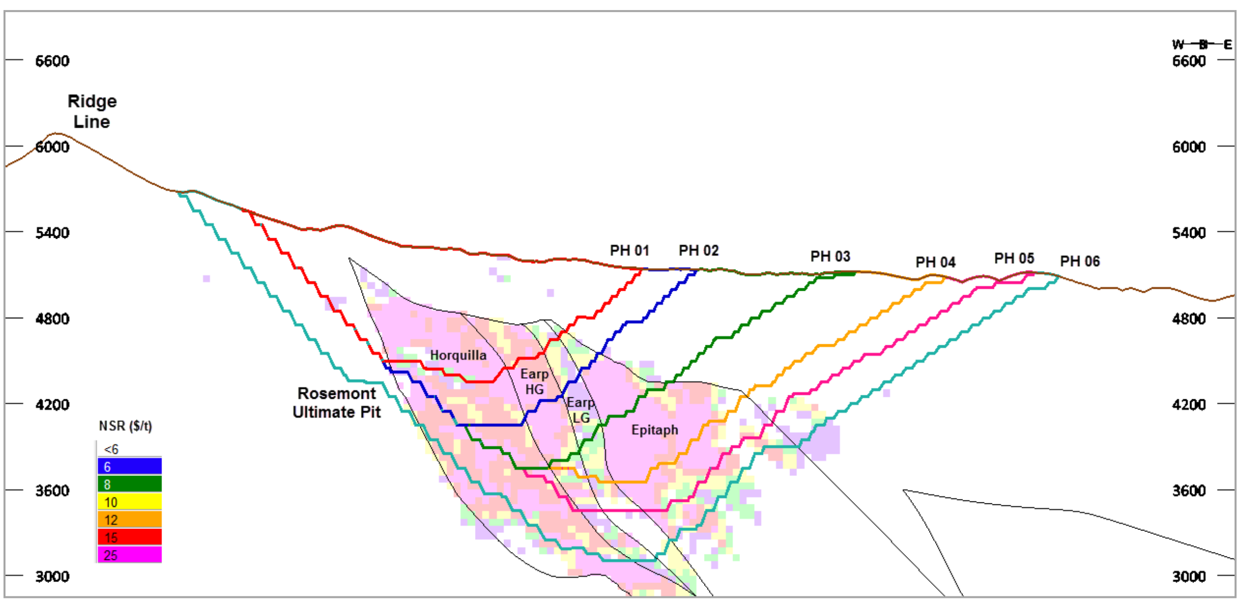



**FIGURE 16-10: AA' SECTION VIEW OF ROSEMONT MINE PHASES**



### TABLE 16-3: ROSEMONT MINE PHASES MINERAL RESERVES

| | Ore M Tons | TCu % | SCu % | ASCu % | Mo % | Ag opt | NSR $/t | CuEq % | Waste M Tons | Total M Tons | S.R. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **PH01** | 84.8 | 0.49 | 0.43 | 0.06 | 0.011 | 0.16 | 21.80 | 0.57 | 190.3 | 275.1 | 2.24 |
| **PH02** | 88.3 | 0.43 | 0.38 | 0.05 | 0.010 | 0.15 | 19.77 | 0.51 | 115.6 | 203.9 | 1.31 |
| **PH03** | 74.8 | 0.50 | 0.45 | 0.04 | 0.012 | 0.15 | 23.18 | 0.58 | 177.9 | 252.7 | 2.38 |
| **PH04** | 63.5 | 0.53 | 0.50 | 0.03 | 0.014 | 0.13 | 25.26 | 0.62 | 182.5 | 246.0 | 2.87 |
| **PH05** | 59.4 | 0.47 | 0.44 | 0.03 | 0.014 | 0.12 | 22.65 | 0.56 | 150.3 | 209.8 | 2.53 |
| **PH06** | 221.2 | 0.39 | 0.34 | 0.05 | 0.012 | 0.12 | 17.64 | 0.46 | 431.9 | 653.1 | 1.95 |
| **Total** | **592.0** | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.57** | **0.53** | **1,248.6** | **1,840.6** | **2.11** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.

### TABLE 16-4: ROSEMONT MINE PHASES, MINERAL RESERVES BY ORE TYPE

| | Ore M Tons | TCu % | SCu % | ASCu % | Mo % | Ag opt | NSR $/t | CuEq % |
|---|---|---|---|---|---|---|---|---|
| **PH01** | **84.8** | **0.49** | **0.43** | **0.06** | **0.011** | **0.16** | **21.80** | **0.57** |
| Sulfide | 74.8 | 0.50 | 0.45 | 0.05 | 0.011 | 0.16 | 23.27 | **0.58** |
| Mixed | 4.2 | 0.28 | 0.21 | 0.08 | 0.008 | 0.09 | 10.15 | **0.34** |
| Oxide | 5.8 | 0.56 | 0.28 | 0.28 | 0.006 | 0.13 | 11.33 | **0.56** |
| **PH02** | **88.3** | **0.43** | **0.38** | **0.05** | **0.010** | **0.15** | **19.77** | **0.51** |
| Sulfide | 78.8 | 0.44 | 0.40 | 0.04 | 0.010 | 0.16 | 20.84 | **0.52** |
| Mixed | 7.5 | 0.30 | 0.23 | 0.07 | 0.007 | 0.09 | 11.34 | **0.35** |
| Oxide | 2.0 | 0.43 | 0.22 | 0.20 | 0.004 | 0.13 | 9.23 | **0.43** |
| **PH03** | **74.8** | **0.50** | **0.45** | **0.04** | **0.012** | **0.15** | **23.18** | **0.58** |
| Sulfide | 70.7 | 0.50 | 0.47 | 0.04 | 0.012 | 0.15 | 23.86 | **0.59** |
| Mixed | 2.7 | 0.32 | 0.26 | 0.06 | 0.007 | 0.06 | 12.29 | **0.36** |
| Oxide | 1.4 | 0.41 | 0.25 | 0.16 | 0.005 | 0.08 | 10.18 | **0.41** |
| **PH04** | **63.5** | **0.53** | **0.50** | **0.03** | **0.014** | **0.13** | **25.26** | **0.62** |
| Sulfide | 62.3 | 0.54 | 0.50 | 0.03 | 0.015 | 0.13 | 25.55 | **0.62** |
| Mixed | 1.1 | 0.30 | 0.22 | 0.08 | 0.008 | 0.04 | 10.69 | **0.34** |
| Oxide | 0.1 | 0.36 | 0.21 | 0.15 | 0.003 | 0.10 | 8.62 | **0.36** |
| **PH05** | **59.4** | **0.47** | **0.44** | **0.03** | **0.014** | **0.12** | **22.65** | **0.56** |
| Sulfide | 58.1 | 0.48 | 0.45 | 0.03 | 0.015 | 0.12 | 22.90 | **0.56** |
| Mixed | 1.2 | 0.29 | 0.24 | 0.05 | 0.011 | 0.05 | 11.53 | **0.34** |
| Oxide | 0.1 | 0.37 | 0.27 | 0.10 | 0.002 | 0.09 | 11.02 | **0.37** |
| **PH06** | **221.2** | **0.39** | **0.34** | **0.05** | **0.012** | **0.12** | **17.64** | **0.46** |
| Sulfide | 198.3 | 0.38 | 0.35 | 0.03 | 0.013 | 0.12 | 18.44 | **0.46** |
| Mixed | 9.8 | 0.38 | 0.27 | 0.10 | 0.007 | 0.05 | 12.86 | **0.42** |
| Oxide | 13.2 | 0.49 | 0.22 | 0.27 | 0.004 | 0.09 | 9.10 | **0.49** |
| **Grand Total** | 592.0 | **0.45** | **0.40** | **0.05** | **0.012** | **0.13** | **20.57** | **0.53** |

Notes:

1. TCu % corresponds to the total copper grade.
2. SCu % grade corresponds to the sulfide copper in the Ore. As per formula SCU = TCU – ASCu
3. ASCu % grade corresponds to the soluble copper.
4. CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.
5. Oxide ore refers only to the sulfide copper species.

# 16.3 Mine Schedule and Production Plan

## 16.3.1 Production Scheduling Criteria

The operating and scheduling criteria used to develop the mining sequence plans are summarized in Table 16-5 below.

**TABLE 16-5: MINE PRODUCTION SCHEDULE CRITERIA**

| Parameter | Value |
|---|---|
| Annual Ore Production Base Rate | 32,850,000 tons |
| Daily Ore Production Base Rate | 90,000 tons |
| Operating Hours per Shift | 12 |
| Operating Shifts per Day | 2 |
| Operating Days per Week | 7 |
| Scheduled Operating Days per Year | 365 |
| Number of Mine Crews | 4 |

Pit and mine maintenance operations will be scheduled around-the-clock. Allowances for downtime and weather delays have been included in the mine equipment and manpower estimations.

A mill ramp up period for concentrator start-up has been considered. Provisions are included to reach full and steady production (throughput) by the end of the sixth month of year one of operation. The mill production targets schedule is presented in Table 16-6. The author believes this period is attainable, considering Hudbay's recent experience in building a similar project in Peru (ramp-up to full production was also approximately 6 months).

**TABLE 16-6: MILL RAMP-UP SCHEDULE**

| Month | | Days | Efficiency | Design tph | Ramp Up Factor | Mill Ore 000 tpm | Mill Ore 000 tpd |
|---|---|---|---|---|---|---|---|
| **1** | Q1 | 31 | 92.1% | 4,073 | 30% | 837 | **27.0** |
| **2** | | 28 | 92.1% | 4,073 | 40% | 1,008 | **36.0** |
| **3** | | 31 | 92.1% | 4,073 | 75% | 2,093 | **67.5** |
| **4** | Q2 | 30 | 92.1% | 4,073 | 87% | 2,349 | **78.3** |
| **5** | | 31 | 92.1% | 4,073 | 95% | 2,651 | **85.5** |
| **6** | | 30 | 92.1% | 4,073 | 100% | 2,700 | **90.0** |

## 16.3.2 Mill Feed and Cut-Off Grade Strategy

An elevated cut-off grade strategy has been implemented to bring forward the higher grade ore from the pit into the early part of the ore production schedule. Delivering higher grade ore to the mill in the early years will improve the net present value and internal rate of return of the Project.

NSR values are calculated for each block in the resource model to represent the net Cu, Mo, and Ag metal values. The pit reserves are estimated based on a cut-off with an NSR value of $6.00/ton. This is the minimum value of mineralized material that will cover the processing and G&A costs, and is therefore reserved for mill feed.

Priority plant feed will consist of high grade material (NSR above $12.00/ton). The medium and low grade material (NSR between $6.00 and $12.00/ton) will be fed as needed and will otherwise be stockpiled.

Mill feed strategy considers that high grade ore stockpiled during the pre-stripping period will be processed during the first year of plant production, utilizing a dynamic stockpile located near the primary crushing facility.

#### 16.3.3 Overburden Stripping Requirements

Mineral reserve tabulations by bench and by phase, and a mine production scheduling program (MSSO, a module from MineSight® software) were used to analyze long-term stripping requirements for the Project. Elevation and phase order dependencies and sinking rate controls were used in conjunction with mill ore production targets and an internal NSR cut-off of $6.00/ton to simulate open pit mining. The program, through successive iterations, allows the user to examine waste stripping rates over the life of the mine and their impact on ore exposure and mill head grades.

The stripping analysis determined that a minimum preproduction stripping of approximately 94 million tons of waste was required. Approximately 11 million tons of ore will also be mined and stockpiled during this period. The estimated Year 1 waste stripping total is 100 million tons, followed by 87 million tons for Year 2. The estimated waste stripping from Year 3 through Year 11 will average about 95 million tons per year to maintain a minimum of six months of ore exposure levels for uninterrupted ore deliveries to the mill. Waste stripping rates will decline to an annual average of 32 million tons for the next 3-year period, and then drop to an average of 5 million tons for the last 3 production years as the final mining phase approaches the pit bottom.

Preproduction stripping is planned to be conducted over a 12-month timeframe and will ramp up according to the delivery of mining equipment (particularly electric shovels) and the hiring and training of work crews. The long-term and peak mining rates suggest the use of at least two large (60 $yd^3$ class) electric shovels, one large (25 $yd^3$) front-end loader and a hydraulic shovel (46 $yd^3$). Ramp-up for the mine is a part of the mine schedule target for total movement capacity by period.

#### 16.3.4 Mine Plan

Mining sequence plans have been developed on a quarterly basis from preproduction through to the end of year 5, and on an annual basis through to year 19. The preproduction period consists of four quarters, or 12 months.

A mine life of approximately 19 years of production is projected by this development plan. Peak mining rates of 367,000 tpd of total material are planned in year 1 until year 11. Average mining rates during years 12-14 are planned to be 180,000 tpd of total material, and then reduce to an average of 105,000 tpd from years 15 – 17 as the strip ratio drops.

During the pre-production period before the first ore is delivered to the mill, the pit will be pre-stripped of waste to expose ore and develop the upper benches for subsequent pushbacks. Specifically, pre-stripping will occur in pit Phase 1 for ore exposure and in Phase 2 to 6 for development. By the end of pre-production, the Phase 1 pit will be down to the 5,150-foot bench. At the end of this period, the ore will be exposed to deliver uninterrupted ore to the mill. Phase 2 will be stripped sufficiently ahead to ensure a supply of ore for mill feed by Year 2.

The mine schedule drawings for life of the mine are shown in Figure 16-11 to Figure 16-31. Mine schedule details regarding total annual movement, stripping ratios, mill feed by lithology and by ore type are presented in Figure 16-32 to Figure 16-34. The estimated mine production schedule is summarized in Table 16-7.

The development of the WRSA (Section 16.4.1) and dry stack tailing facility buttress (Section 16.4.2) was designed to allow concurrent reclamation of the facilities. Materials are placed at final slope angles required by the reclamation and closure plan conceptually approved by the USFS. Revegetation will start once the next lift is completed, this provides the opportunity for early bond release in some areas of the facility minimizing closure requirements at the end of the life of the facility. All elevations shown are in feet (AMSL).

**FIGURE 16-11: MINE PLAN END OF PERIOD PRE-PRODUCTION**

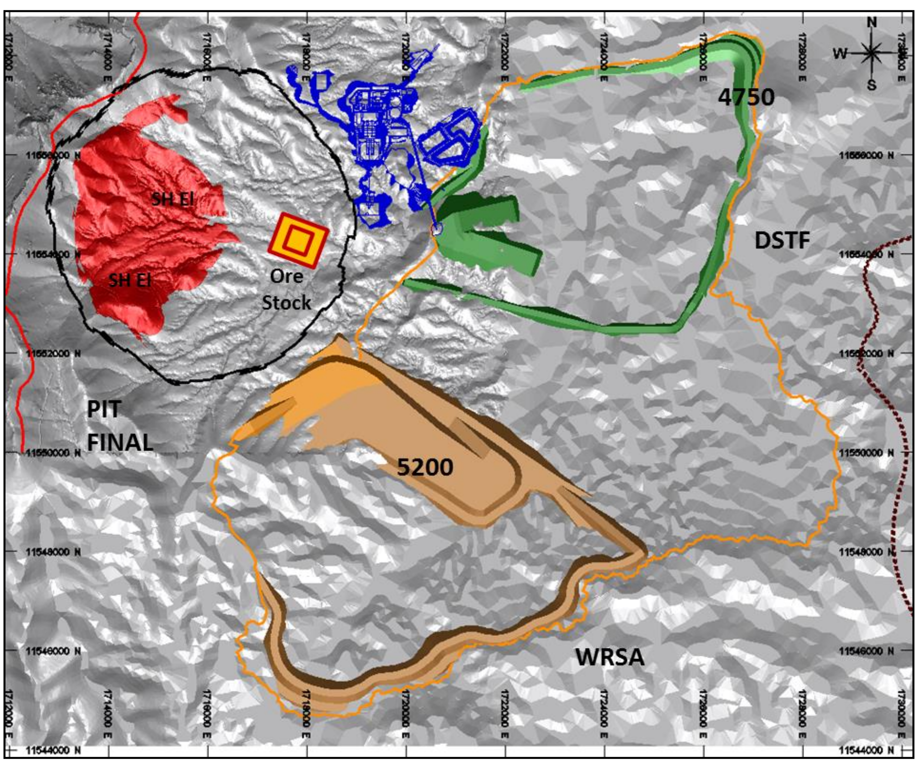


**FIGURE 16-12: MINE PLAN END OF PERIOD YEAR 1**

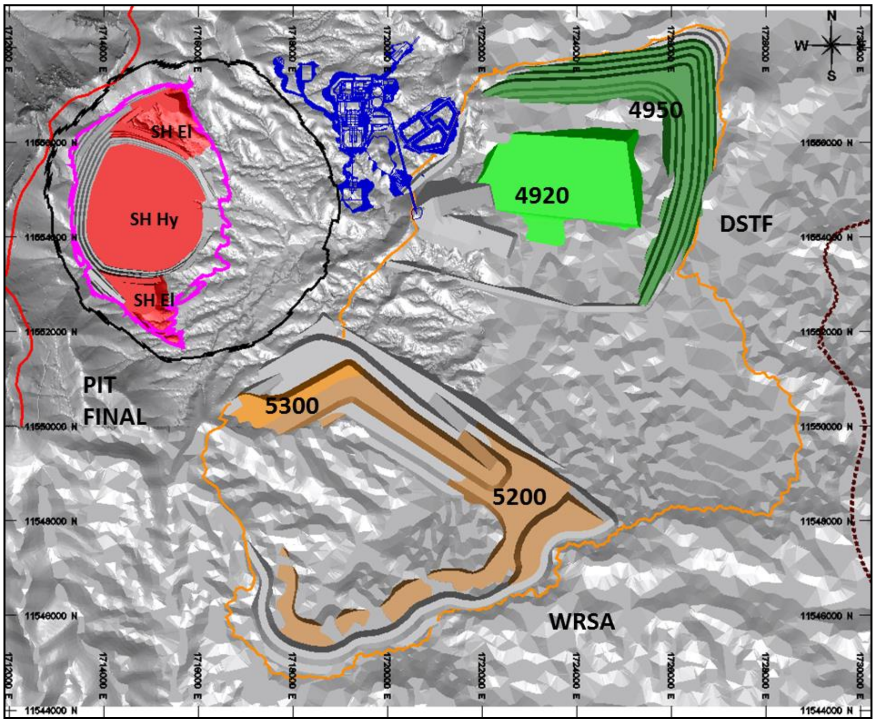



**FIGURE 16-13: MINE PLAN END OF PERIOD YEAR 2**

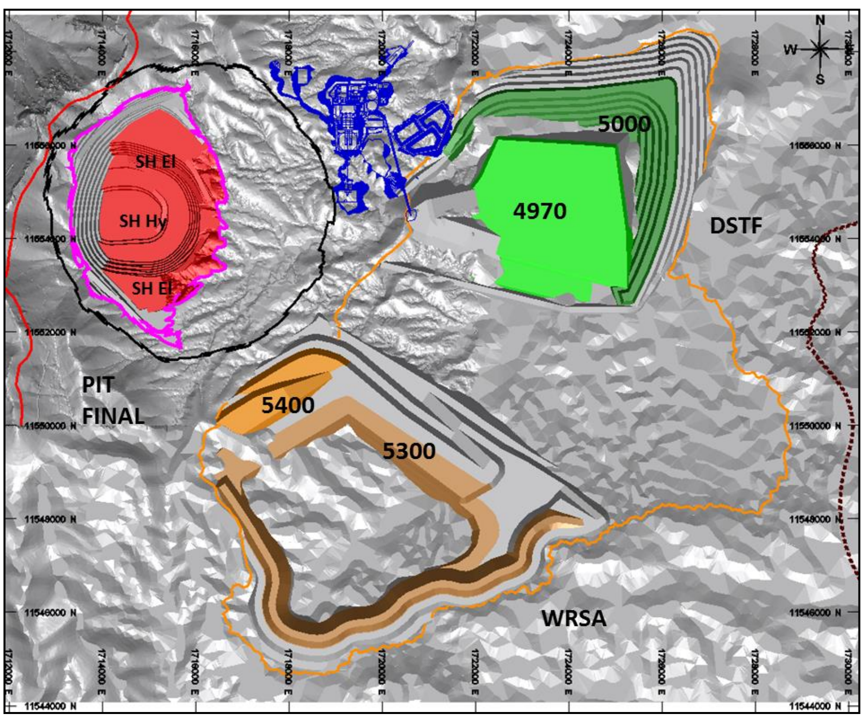


**FIGURE 16-14: MINE PLAN END OF PERIOD YEAR 3**

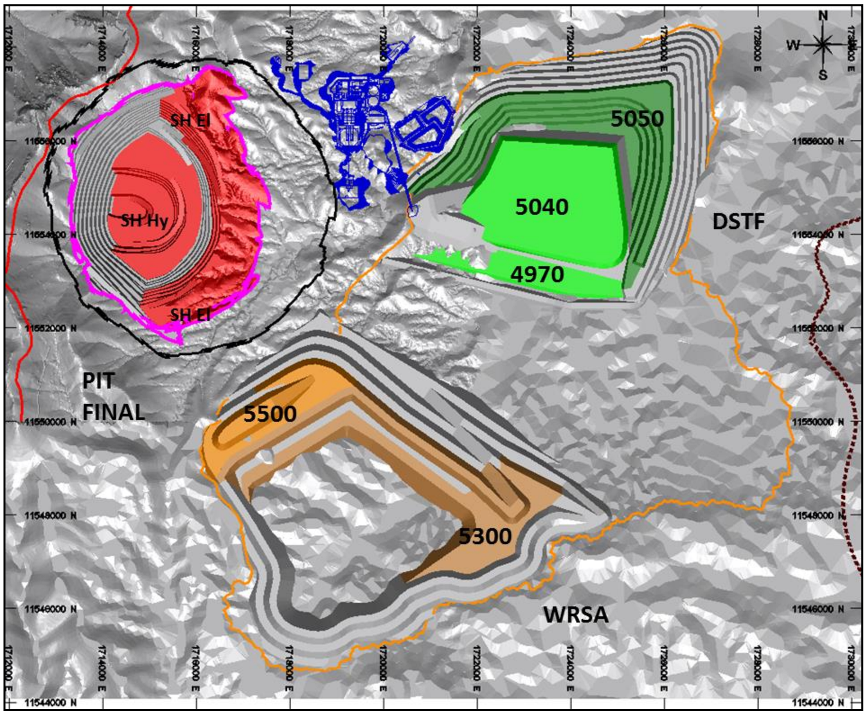



**FIGURE 16-15: MINE PLAN END OF PERIOD YEAR 4**

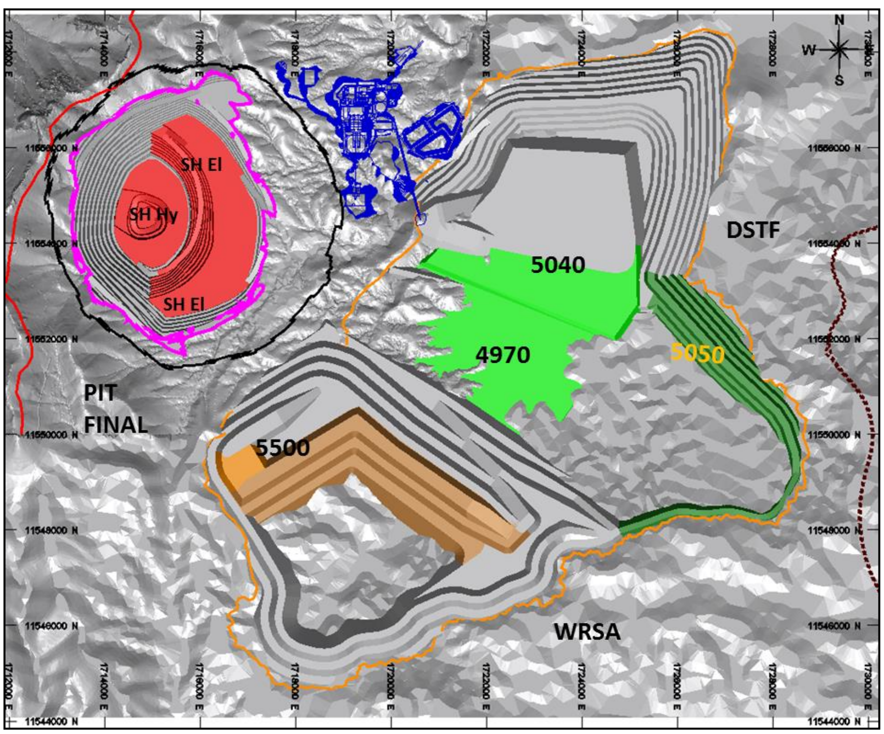


**FIGURE 16-16: MINE PLAN END OF PERIOD YEAR 5**

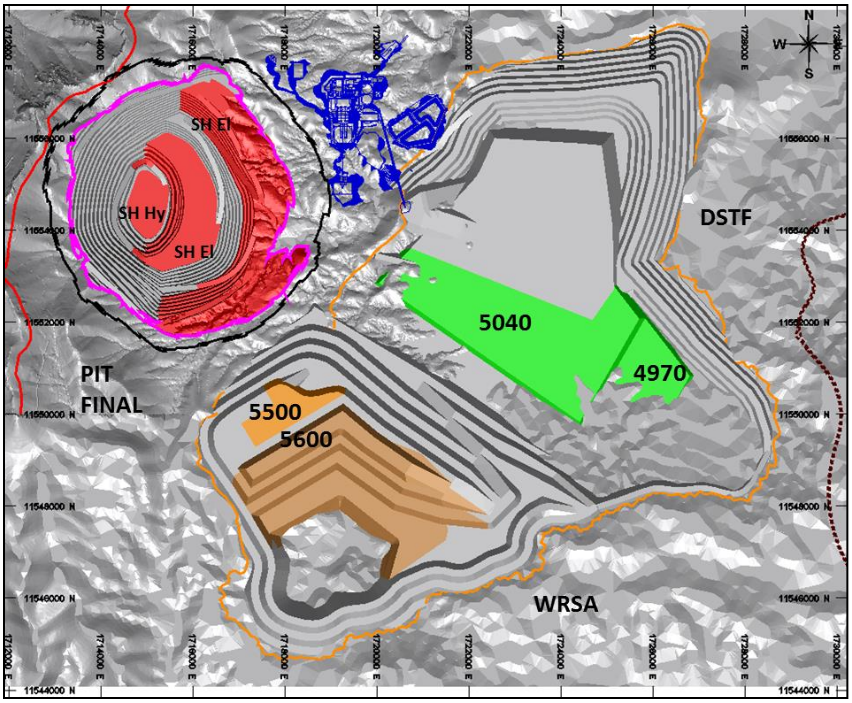



**FIGURE 16-17: MINE PLAN END OF PERIOD YEAR 6**

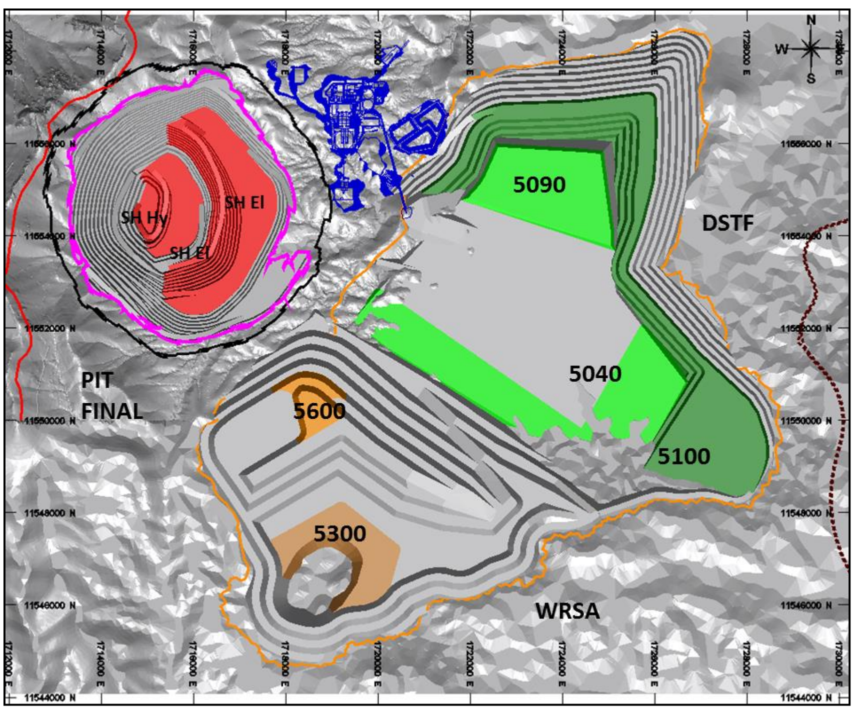


**FIGURE 16-18: MINE PLAN END OF PERIOD YEAR 7**

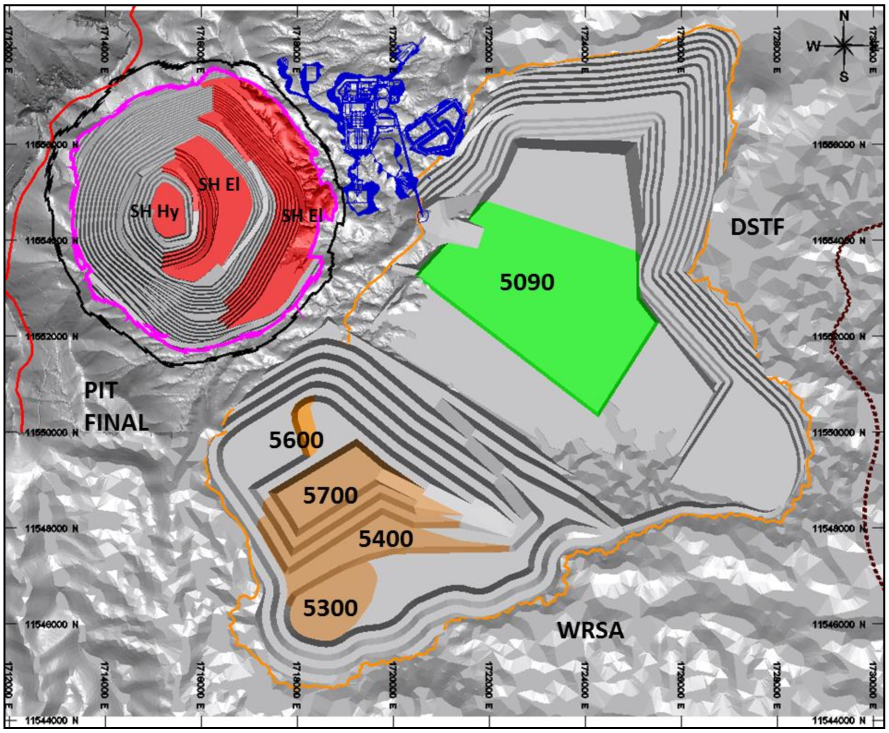



**FIGURE 16-19: MINE PLAN END OF PERIOD YEAR 8**

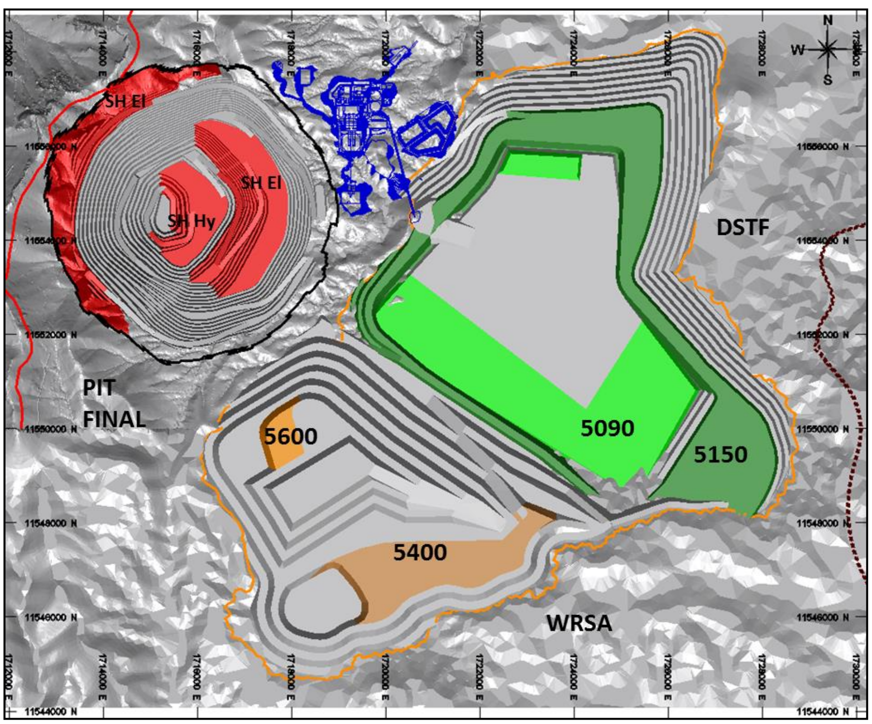


**FIGURE 16-20: MINE PLAN END OF PERIOD YEAR 9**

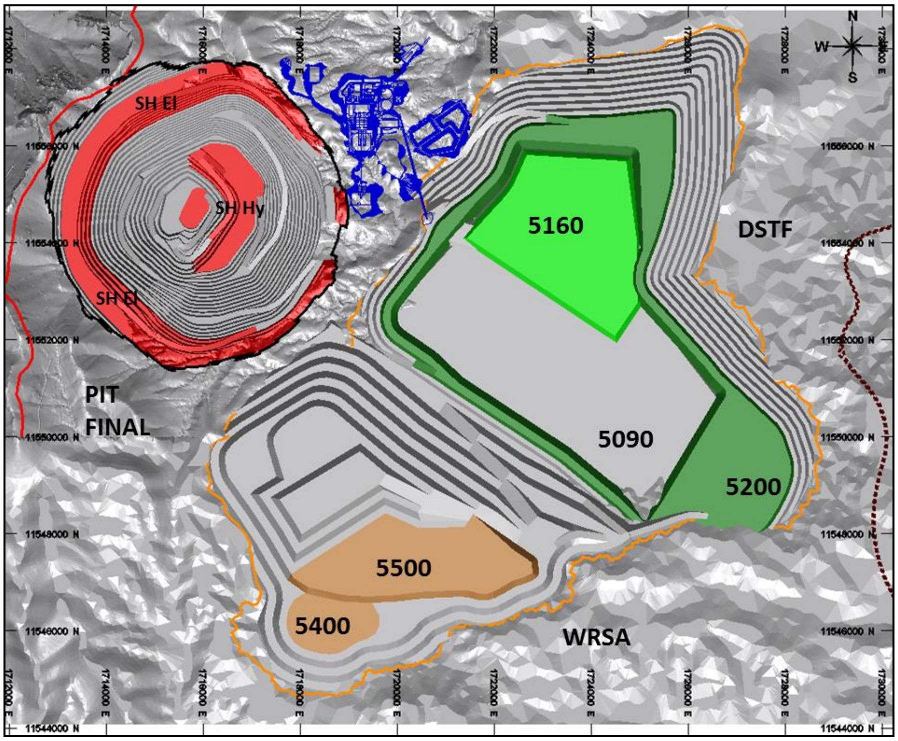



**FIGURE 16-21: MINE PLAN END OF PERIOD YEAR 10**

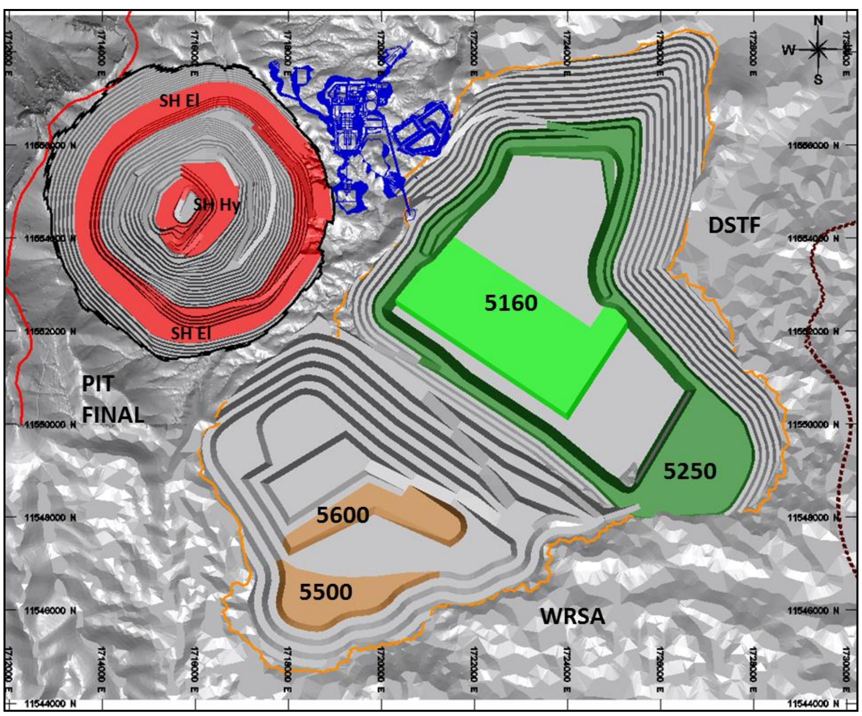


**FIGURE 16-22: MINE PLAN END OF PERIOD YEAR 11**

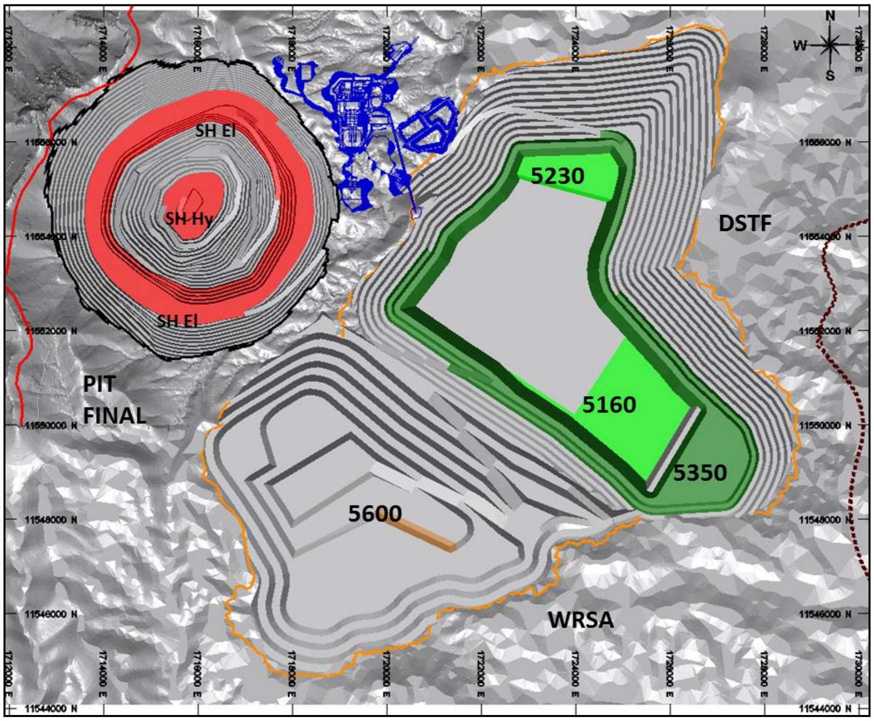



**FIGURE 16-23: MINE PLAN END OF PERIOD YEAR 12**

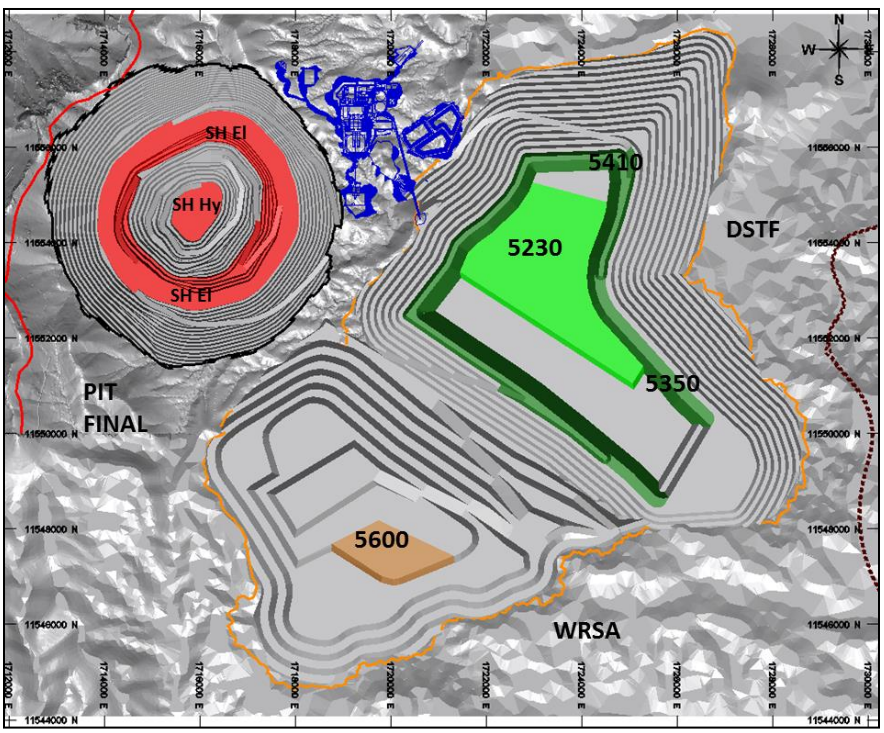


**FIGURE 16-24: MINE PLAN END OF PERIOD YEAR 13**

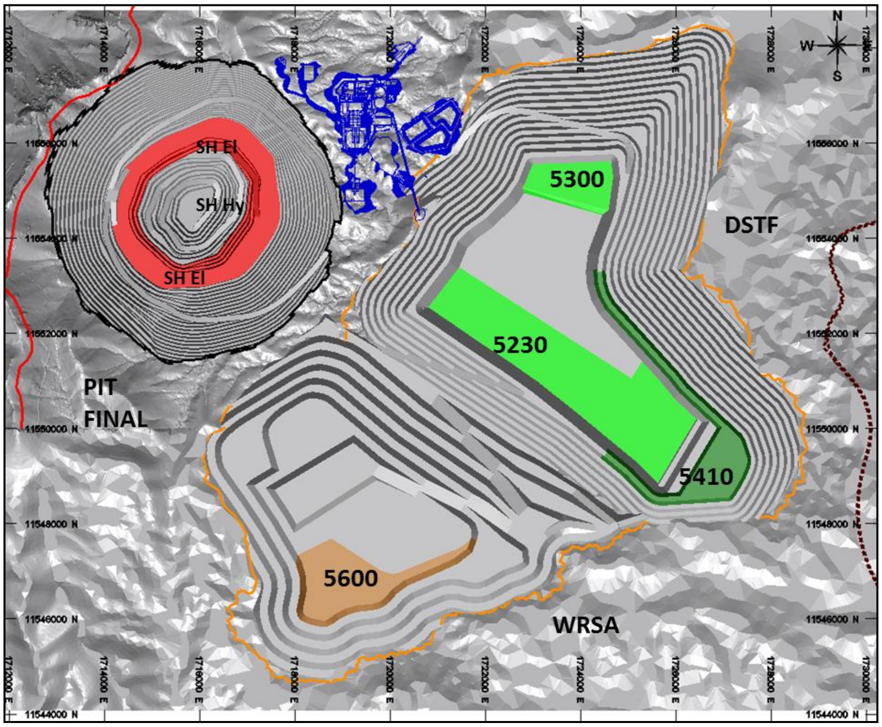



**FIGURE 16-25: MINE PLAN END OF PERIOD YEAR 14**

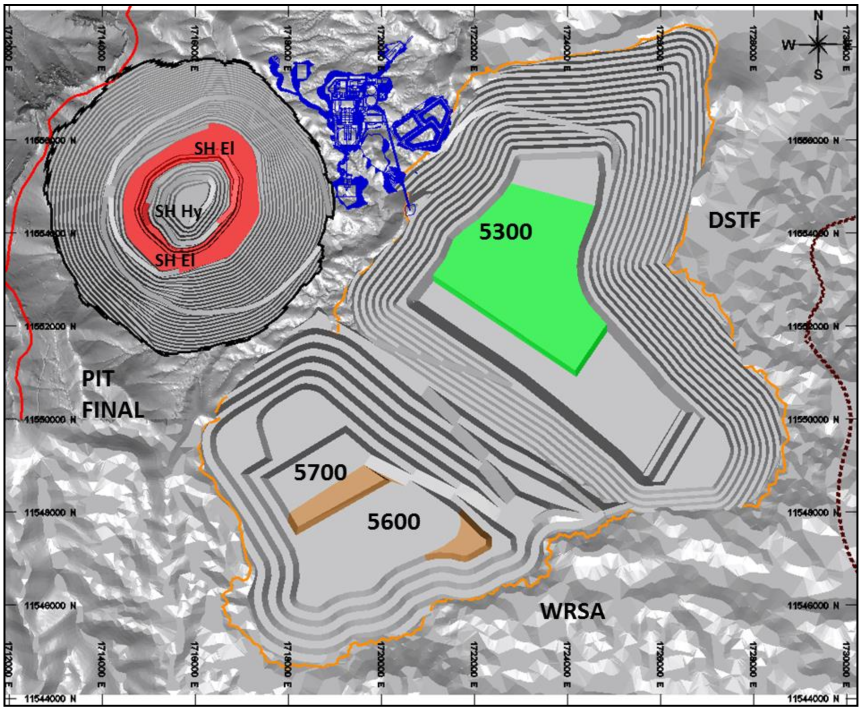


**FIGURE 16-26: MINE PLAN END OF PERIOD YEAR 15**

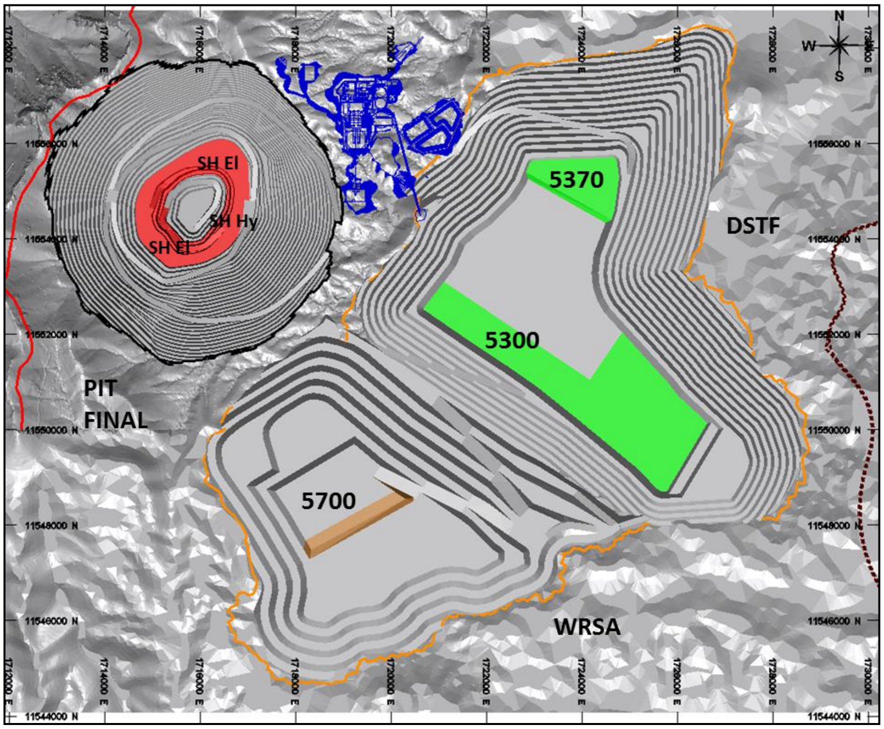



**FIGURE 16-27: MINE PLAN END OF PERIOD YEAR 16**

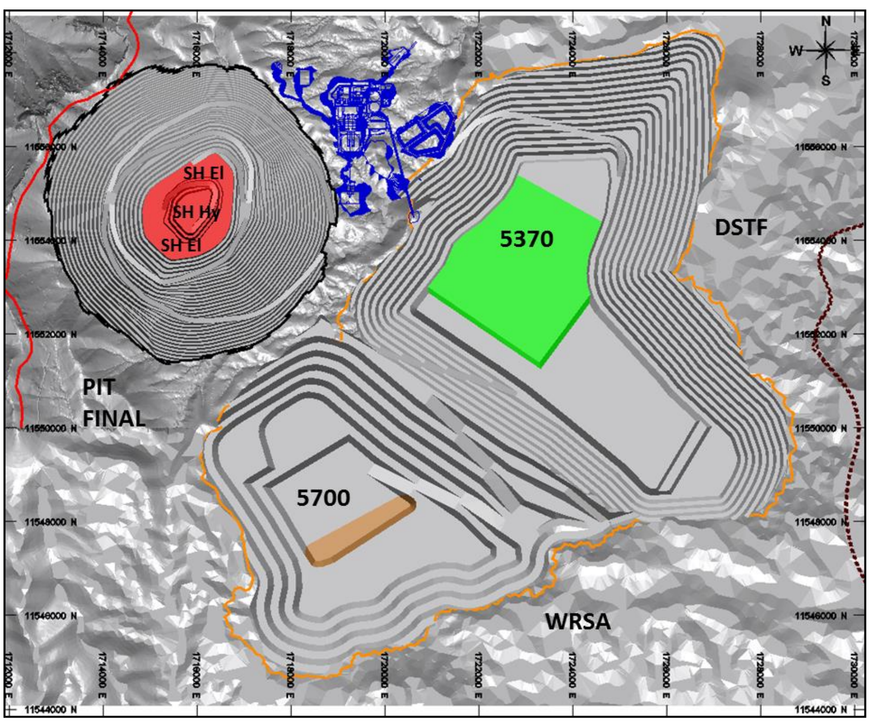


**FIGURE 16-28: MINE PLAN END OF PERIOD YEAR 17**

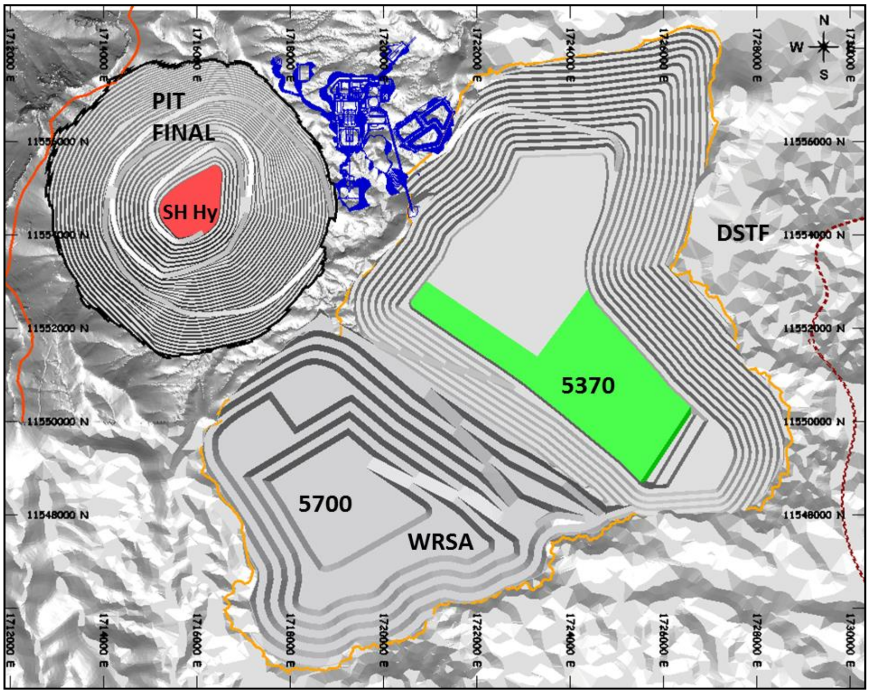



**FIGURE 16-29: MINE PLAN END OF PERIOD YEAR 18**

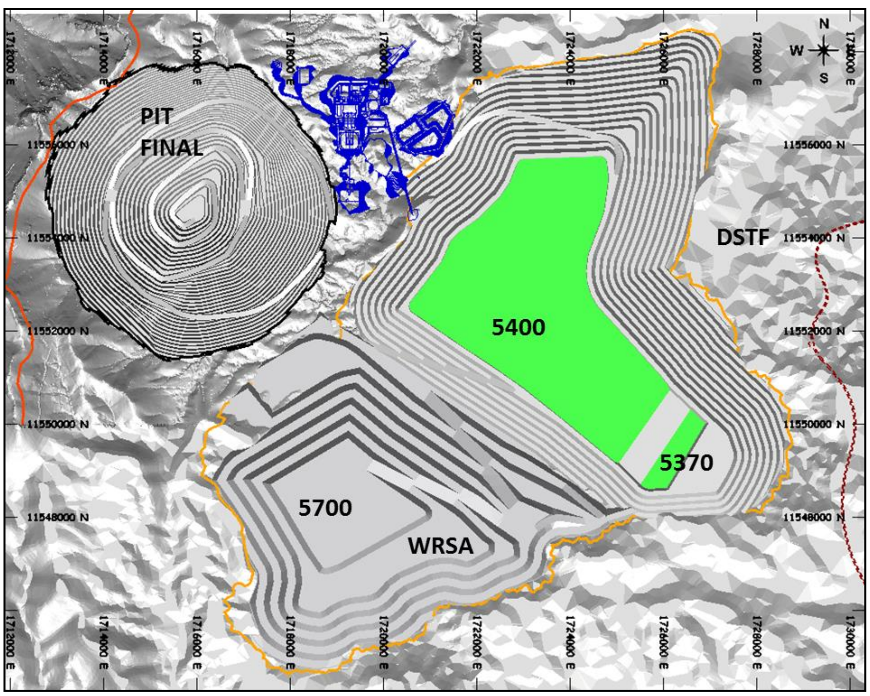


**FIGURE 16-30: MINE PLAN END OF PERIOD YEAR 19**

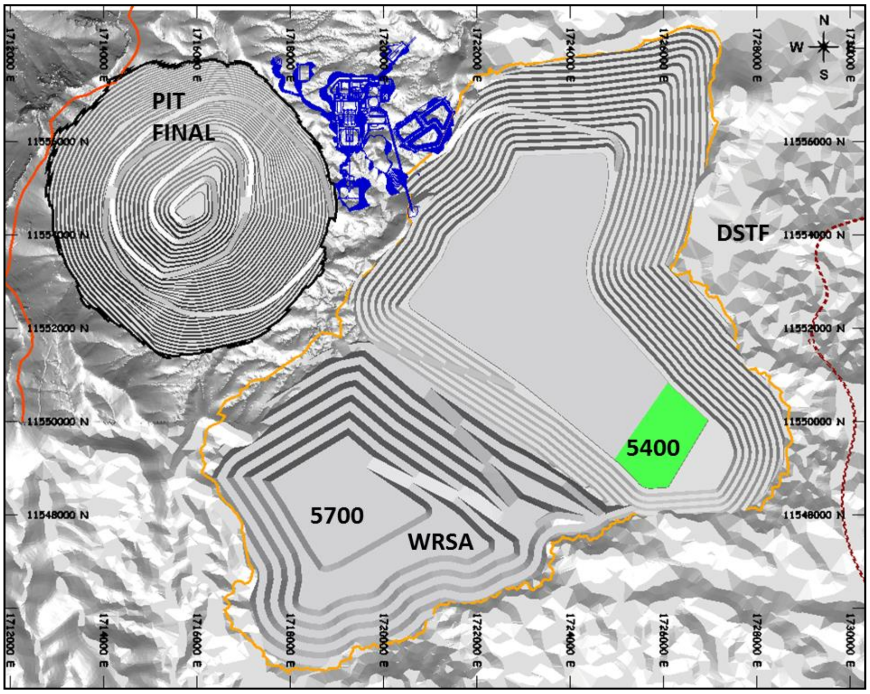



**FIGURE 16-31: MINE PLAN, FINAL TOPOGRAPHY**

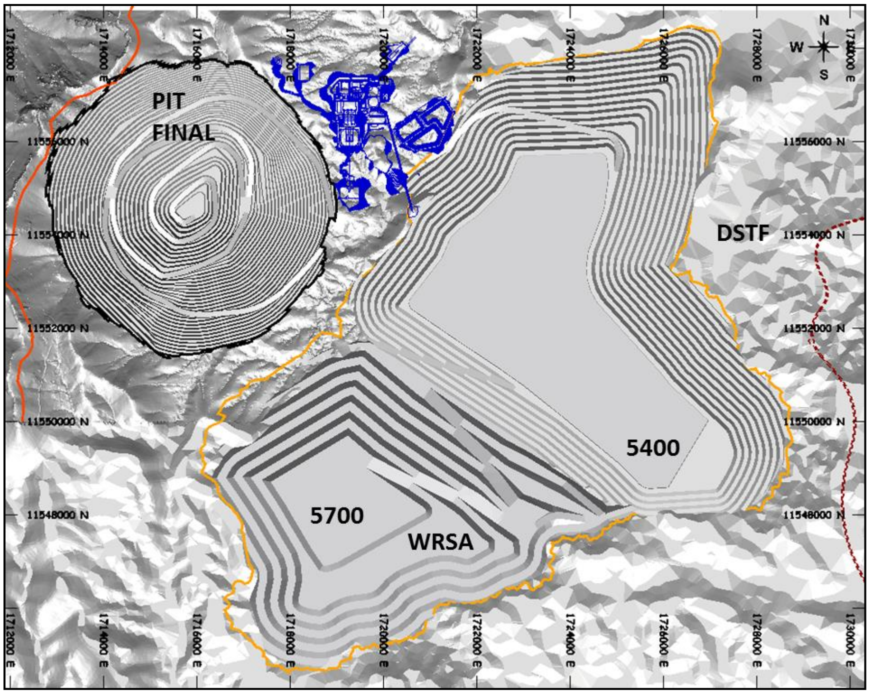


**TABLE 16-7: MINE PRODUCTION SCHEDULE – LOM RP16AUG**

| | | Yr -1 | Yr1 | Yr2 | Yr3 | Yr4 | Yr5 | Yr6 | Yr7 | Yr8 | Yr9 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 | Yr15 | Yr16 | Yr17 | Yr18 | Yr19 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mill** | M Tons | - | 28.1 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 32.8 | 32.9 | 32.9 | 32.8 | 32.9 | 32.9 | 32.9 | 32.9 | 32.9 | 5.5 | **592.0** |
| | SCu % | - | 0.43 | 0.50 | 0.50 | 0.44 | 0.51 | 0.55 | 0.48 | 0.55 | 0.39 | 0.46 | 0.32 | 0.30 | 0.34 | 0.37 | 0.39 | 0.36 | 0.24 | 0.14 | 0.12 | **0.40** |
| | TCu % | - | 0.51 | 0.55 | 0.54 | 0.49 | 0.55 | 0.59 | 0.51 | 0.61 | 0.44 | 0.49 | 0.37 | 0.36 | 0.40 | 0.40 | 0.42 | 0.39 | 0.28 | 0.19 | 0.18 | **0.45** |
| | ASCu% | - | 0.08 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.04 | 0.06 | 0.05 | 0.04 | 0.05 | 0.06 | 0.06 | 0.04 | 0.03 | 0.03 | 0.04 | 0.06 | 0.06 | **0.05** |
| | Ox% | - | 16% | 9% | 9% | 12% | 10% | 9% | 9% | 11% | 14% | 10% | 14% | 15% | 15% | 11% | 9% | 9% | 14% | 24% | 24% | **12%** |
| | Mo % | - | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.02 | 0.02 | 0.01 | 0.01 | 0.00 | **0.01** |
| | Ag opt | - | 0.16 | 0.18 | 0.16 | 0.17 | 0.17 | 0.18 | 0.13 | 0.14 | 0.10 | 0.13 | 0.10 | 0.10 | 0.12 | 0.12 | 0.14 | 0.14 | 0.11 | 0.07 | 0.07 | **0.13** |
| | CuEq % | - | 0.59 | 0.64 | 0.63 | 0.58 | 0.64 | 0.69 | 0.60 | 0.70 | 0.50 | 0.58 | 0.43 | 0.43 | 0.48 | 0.48 | 0.52 | 0.49 | 0.34 | 0.23 | 0.21 | **0.53** |
| | NSR $/t | - | 21.8 | 25.6 | 25.3 | 22.6 | 25.7 | 28.0 | 24.2 | 27.8 | 19.3 | 23.3 | 16.4 | 15.6 | 17.9 | 19.1 | 20.6 | 19.6 | 12.9 | 7.2 | 6.3 | **20.6** |
| | SWCL % | - | 6.0 | 7.1 | 8.3 | 8.4 | 8.2 | 8.9 | 8.9 | 8.7 | 7.9 | 7.0 | 7.1 | 7.6 | 8.5 | 7.7 | 6.5 | 6.3 | 6.9 | 8.5 | 7.9 | **7.7** |
| | MGCL % | - | 1.4 | 2.6 | 3.2 | 2.5 | 4.6 | 4.8 | 4.4 | 5.7 | 2.6 | 4.5 | 3.5 | 3.8 | 3.0 | 3.9 | 3.7 | 4.2 | 2.3 | 1.5 | 1.2 | **3.4** |
| | BWI Kw-Hr/t | - | 13.6 | 13.2 | 12.9 | 12.8 | 12.3 | 13.3 | 12.6 | 13.1 | 12.7 | 12.7 | 13.1 | 12.9 | 12.2 | 11.7 | 12.2 | 12.6 | 13.2 | 13.3 | 12.4 | **12.8** |
| **SPM-** | M Tons | - | 0.1 | - | - | - | - | - | - | - | - | - | 0.4 | - | - | - | - | - | 1.7 | 4.6 | 1.2 | **8.0** |
| **SPM+** | M Tons | 1.7 | 1.2 | 0.9 | 2.2 | 0.4 | 1.0 | 0.3 | 0.1 | 0.2 | - | - | - | - | - | - | - | - | - | - | - | **8.0** |
| **SPO-** | M Tons | - | 0.3 | - | - | - | - | - | - | - | - | - | 0.2 | - | - | - | - | - | 1.0 | 3.7 | 0.8 | **5.9** |
| **SPO+** | M Tons | 1.9 | 0.7 | 0.6 | 0.6 | 0.3 | 0.7 | 0.1 | - | 1.0 | - | - | - | - | - | - | - | - | - | - | - | **5.9** |
| **SPS-** | M Tons | - | 3.2 | - | - | - | - | - | - | - | - | - | 2.9 | - | - | - | - | - | 14.0 | 24.6 | 3.5 | **48.3** |
| **SPS+** | M Tons | 7.7 | 2.3 | 10.9 | 10.4 | 2.1 | 4.2 | 6.1 | 1.7 | 3.0 | - | - | - | - | - | - | - | - | - | - | - | **48.3** |
| **WRSA** | M Tons | 78.2 | 45.5 | 58.5 | 47.3 | 75.0 | 93.3 | 23.0 | 97.3 | 23.7 | 47.2 | 27.1 | 1.0 | 9.9 | 15.8 | 8.4 | 3.6 | 5.8 | 5.7 | - | - | **666.3** |
| **DSTF** | M Tons | 15.6 | 54.1 | 28.2 | 38.7 | 21.3 | - | 69.6 | - | 71.4 | 51.9 | 72.1 | 98.1 | 51.7 | 9.5 | - | - | - | - | - | - | **582.3** |
| **Total** | M Tons | **105.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **132.0** | **94.0** | **58.0** | **41.0** | **36.0** | **39.0** | **39.0** | **33.0** | **5.0** | **1,903.0** |

Notes: CuEq% is calculated based on metal prices of $3.15/lb Cu, $11.00/lb Mo and $18.00/oz Ag.
Ox%: Oxide Ratio, between Soluble Copper and total copper, as per formula: ASCu% / TCu%.
SWCL%: Swelling clays grade.
MGCL%: Magnesium clays grade.
BWI: Bond Work Index.
SPM-: Mixed Ore Stockpile (Out).
SPM+: Mixed Ore Stockpile (In).
SPO-: Oxide Ore Stockpile (Out).
SPO+: Oxide Ore Stockpile (In).
SPS-: Sulfide Ore Stockpile (Out).
SPS+: Sulfide Ore Stockpile (In).
WRSA: Waste rock facility destination.
DSTF: Dry stack tailings facility destination.

**FIGURE 16-32: ROSEMONT MINE SCHEDULE, MATERIAL MOVEMENT**

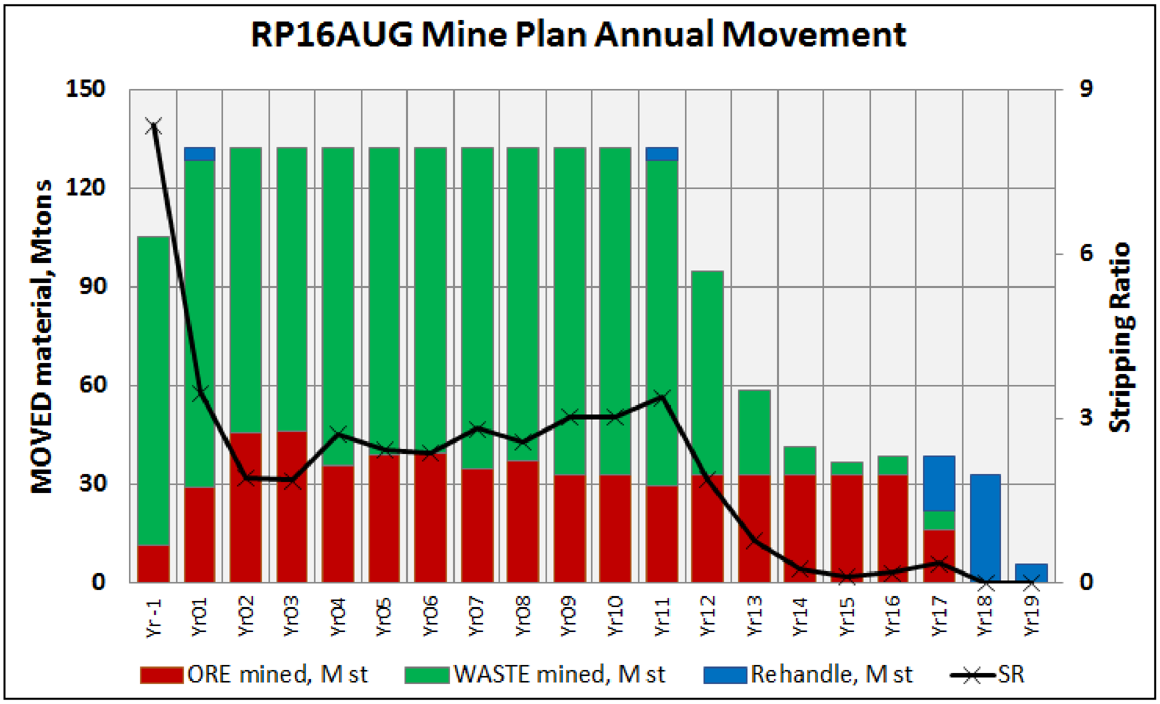
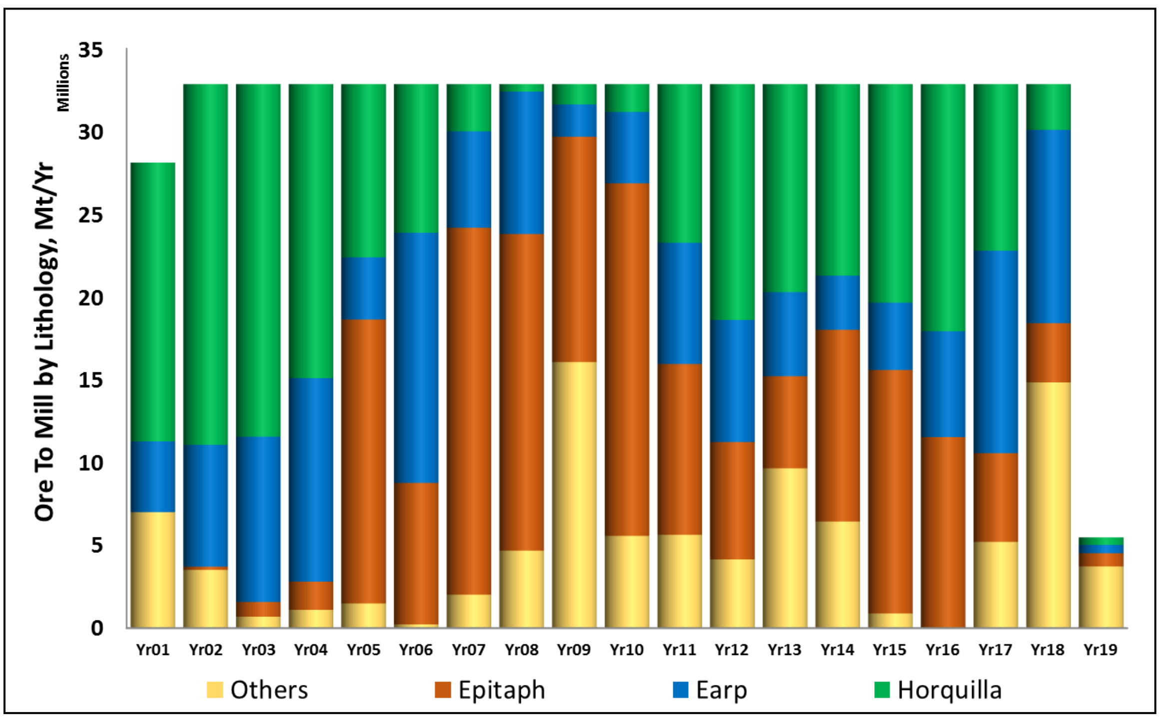
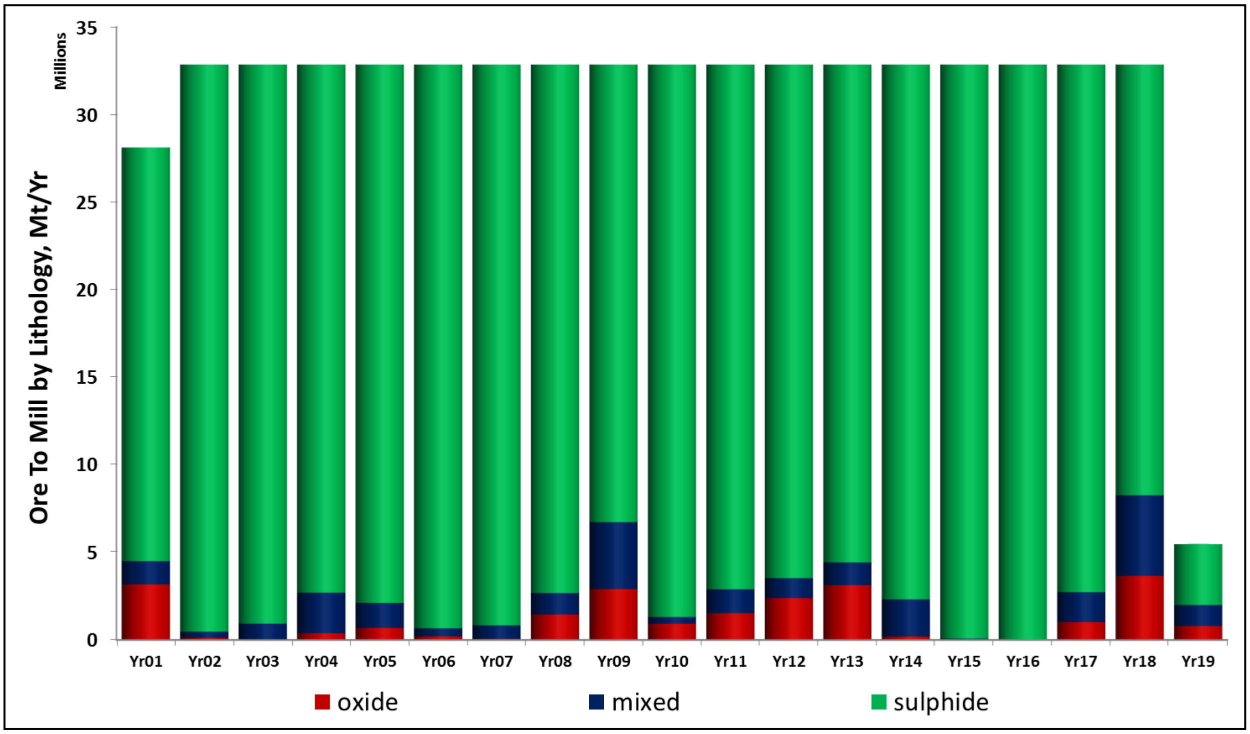



**FIGURE 16-33: ROSEMONT MINE SCHEDULE, MILL FEED ORE BY LITHOLOGY**




Note: Ore Mill in others lithology are Abrigo 9.6Mt@0.42%Cu, Andesite 16Mt@0.23%Cu, Arkose 13.5Mt@0.21%Cu, Bolsa 6.3Mt@0.38%Cu, Escabrosa 9.8Mt@0.54%Cu, Glance 4.7Mt@0.22%Cu, Granodiorite 2Mt@0.62%Cu, Martin 5.7Mt@0.31%Cu, QMP 17.7Mt@0.40%Cu and Scherrer 7.5Mt@0.35%Cu, which represents 16% of the total Ore mill.

**FIGURE 16-34: ROSEMONT MINE SCHEDULE, MILL FEED ORE BY ORE TYPE**




## 16.4 Mine Facilities

### 16.4.1 WRSA

Overburden and other waste rock encountered in the course of mining will be placed into the WRSA located to the south and southeast of the planned open pit and into landform area. The design criteria for the WRSA area and associated haul roads are summarized in Table 16-8 below. The general mine site layout is shown in Figure 16-1.

**TABLE 16-8: WRSA DESIGN CRITERIA**

| Parameter | Value |
|---|---|
| Angle of Repose | 37° |
| Average Tonnage Factor (with swell) | 16.02 $ft^3$/ton |
| Overall Slope Angle | 3.5H:1V |
| Total Height, feet | 600 |
| Haul Road, feet | 120 |
| Max Elevation, feet (AMSL) | 5700 |

One of the objectives in the early years of operation (specifically, Years 1 to 5) is to construct a series of buttresses and berms around the eastern and southern perimeters of the DSTF and WRSA respectively, for permit commitment. These buttresses and berms will also allow re-grading and re-vegetation of the facilities side slopes at much earlier time periods than with traditional mine waste rock closure plans.

The WRSA berms and internal loading plan are designed to facilitate subsequent re-grading and concurrent reclamation. Side slopes in the WRSA will be re-graded to a maximum of 3:1 (horizontal: vertical) slopes. The WRSA loading plan will consist of haul trucks end-dumping waste rock in 100-foot lifts at the angle of repose (approximately 37°). The WRSA crests will be set back to allow simple dozing of the crests down to meet the target re-graded slope angles to support concurrent reclamation.

### 16.4.2 DSTF Buttress

Dry stack tailing resulting from processing mine ore will be placed behind the buttresses constructed from mine waste rock. Any acid generating waste will be disposed in the DSTF buttress. The DSTF is north of the WRSA and east-northeast of the pit. The design criteria for the DSTF and associated haul roads are summarized in Table 16-9 below. The general mine site layout is shown in Figure 16-1.

**TABLE 16-9: DSTF BUTTRESS ROCK STORAGE DESIGN CRITERIA**

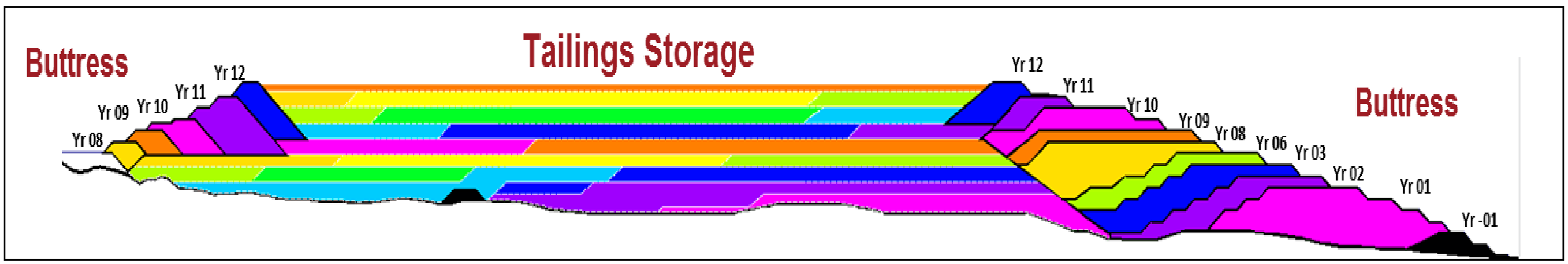
| Description | Unit |
|---|---|
| Angle of Repose | 37° |
| Average Tonnage Factor (with swell) | 16.02 $ft^3$/ton |
| Overall Slope Angle | 3.5H:1V |
| Total Height, feet | 700 |
| Haul Road, feet | 120 |
| Max Elevation, feet (AMSL) | 5,490 |

The DSTF and WRSA are described in more detail in Section 18.4 of this report. As the mine matures, waste rock generation declines which forces maximum utilization during many of the early years to construct the buttress in the DSTF and berms in the WRSA. The DSTF buttress construction is planned to be finished by the end of year 13 with sufficient capacity to store the remainder of the tailing material generated during the 19-year mine life. Table 16-10 summarizes LOM Waste distribution for WRSA and DSTF. A cross section N-S view of DSTF buttress by year is shown in Figure 16-35.

**TABLE 16-10: LOM WASTE ROCK DISTRIBUTION AND LANDFORMING STORAGE PLAN**

| | Un | PP | Yr1 | Yr2 | Yr3 | Yr4 | Yr5 | Yr6 | Yr7 | Yr8 | Yr9 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 | Yr15 | Yr16 | Yr17 | Yr18 | Yr19 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Waste Rock (Plan)** | Mt | **94** | **100** | **87** | **86** | **96** | **93** | **93** | **97** | **95** | **99** | **99** | **99** | **62** | **25** | **8** | **4** | **6** | **6** | **0** | **0** | **1,249** |
| **To DSTF Buttress** | Mt | 16 | 54 | 28 | 39 | 21 | 0 | 70 | 0 | 71 | 52 | 72 | 98 | 52 | 10 | 0 | 0 | 0 | 0 | 0 | 0 | **582** |
| *Accum tons in DSTF* | Mt | *16* | *70* | *98* | *137* | *158* | *158* | *228* | *228* | *299* | *351* | *423* | *521* | *573* | *582* | *582* | *582* | *582* | *582* | *582* | *582* | |
| **To WRSA** | Mt | 78 | 45 | 58 | 47 | 75 | 93 | 23 | 97 | 24 | 47 | 27 | 1 | 10 | 16 | 8 | 4 | 6 | 6 | 0 | 0 | **666** |
| *Accum tons in WRSA* | Mt | *78* | *124* | *182* | *229* | *304* | *398* | *421* | *518* | *542* | *589* | *616* | *617* | *627* | *643* | *651* | *655* | *661* | *666* | *666* | *666* | |
| **Tailings** | Mt | 0 | 28 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 33 | 5 | **592** |
| *Accum tailings* | Mt | 0 | 28 | 61 | 94 | 127 | 160 | 192 | 225 | 258 | 291 | 324 | 357 | 389 | 422 | 455 | 488 | 521 | 554 | 587 | 592 | |
| **Ore to Stockpile** | Mt | 8 | 4 | 12 | 13 | 3 | 6 | 6 | 2 | 4 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | **59** |
| **Ore Stockpile** | Mt | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 4 | 0 | 0 | 0 | 0 | 0 | 17 | 33 | 5 | **59** |
| *Accum ore in stock* | Mt | 8 | 12 | 24 | 38 | 40 | 46 | 53 | 54 | 59 | 59 | 59 | 55 | 55 | 55 | 55 | 55 | 55 | 38 | 5 | 0 | |
| **Material to landform** | Mt | 101 | 104 | 99 | 99 | 99 | 99 | 99 | 99 | 99 | 99 | 99 | 96 | 62 | 25 | 8 | 4 | 6 | -11 | -33 | -5 | |
| *Accum in landform* | Mt | **101** | **205** | **304** | **404** | **503** | **602** | **701** | **800** | **899** | **998** | **1,098** | **1,193** | **1,255** | **1,280** | **1,289** | **1,292** | **1,298** | **1,287** | **1,254** | **1,249** | |

**FIGURE 16-35: DSTF NS SECTION VIEW, LOM BUTTRESS BY YEAR**



## 16.5 Mine Equipment

### 16.5.1 Equipment Operating Parameter

Mine equipment was selected based on the production requirements shown in Table 16-7. During the first quarter of preproduction, a 46 $yd^3$ hydraulic excavator and a 25 $yd^3$ loader will be matched with 260-ton-class haul trucks; supported with dozers, graders and water trucks to develop the initial mine area. At the end of the second quarter of preproduction, the first 60 $yd^3$ class electric shovel will come on line followed by one more in preproduction third quarter.

The mine will operate two 12-hour shifts per day, for 365 days a year. No significant weather delays are expected and the mine will not be shut down for holidays. The craft work schedule will consist of a standard four crew rotation.

Material characteristics used to determine productivity calculations are listed in Table 16-11. Although there are several different rock types at Rosemont, the weighted average of all rock types was used for production estimation. Major loading and haulage equipment will be equipped with electronic load monitors, which will ensure optimum loading. All equipment production is reported in dry short tons, which is consistent with the reserve model. Moisture content is expected to range between 3 and 4 percent; for haulage calculations 3.5 percent was used.

**TABLE 16-11: MATERIAL CHARACTERISTICS**

| Parameter | Value |
|---|---|
| In Situ Bulk Density | 11.85 cubic feet per ton |
| Material Swell | 40 Percent |
| Loose Density | 16.02 cubic feet per ton |
| Moisture Content | 3.5 Percent |

### 16.5.2 Mine Equipment Calculation

Mine equipment requirements were developed based on the annual tonnage movement projected by the mine production schedule in Table 16-7, bench heights of 50 feet, two twelve hour shifts per day, 365 days per year operation, with manufacturer machine specifications and material characteristics specific to the deposit.

Specific manufacturer's models used in this study are only intended to represent the size and class of equipment selected. The final equipment manufacturer selection will be done as required to meet delivery dates and current need of the operation.

A summary of fleet requirements by time period for major mine equipment is shown in Table 16-12. Furthermore, Table 16-13 lists equipment KPI's, Availability and Utilization, and equipment productivity used to dimension equipment fleets for the different mine operations.

This represents equipment necessary to perform the following mine tasks:

- Mine site clearing and topsoil salvage and stockpiling
- Construction of the main haul roads
- Production and pre-split drilling
- Loading and hauling of sulfide ore to the primary crusher (located on the east side of the pit), and waste rock to WRSA and DSTF areas
- Maintaining mine haulage and access roads
- Maintaining WRSA, DSTF, berms, and re-grading of slopes and final surfaces

## TABLE 16-12: MAJOR FLEET REQUIREMENTS FOR LOM

| | PP | Yr1 | Yr2 | Yr3 | Yr4 | Yr5 | Yr6 | Yr7 | Yr8 | Yr9 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 | Yr15 | Yr16 | Yr17 | Yr18 | Yr19 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Pit Viper Drill** | 2 | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | - | - |
| **Cable Shovel** | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | 1 |
| **Hydraulic Shovel** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | - |
| **Front End Loader** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | - |
| **793F Haul Truck** | 23 | 25 | 25 | 28 | 33 | 34 | 38 | 38 | 38 | 36 | 33 | 33 | 33 | 18 | 18 | 14 | 14 | 14 | 6 | 6 |
| **D10T Track Dozer** | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 4 | 4 | 2 | 2 | 2 | 2 | 1 | 1 |
| **834K Wheel Dozer** | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | - |
| **14M Grader** | 2 | 3 | 3 | 3 | 3 | 3 | **3** | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 1 | 1 | 1 |
| **777G Water Truck** | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 3 | 2 |
| **988K FEL** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| **CS78 Compactor** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - |
| **352 Excavator** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| **TOTAL** | **43** | **49** | **50** | **53** | **58** | **59** | **63** | **63** | **62** | **60** | **57** | **57** | **53** | **37** | **33** | **29** | **29** | **27** | **16** | **13** |

## TABLE 16-13: MAJOR EQUIPMENT KPI AND PRODUCTIVITY

| | PP | Yr1 | Yr2 | Yr3 | Yr4 | Yr5 | Yr6 | Yr7 | Yr8 | Yr9 | Yr10 | Yr11 | Yr12 | Yr13 | Yr14 | Yr15 | Yr16 | Yr17 | Yr18 | Yr19 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Drills Fleet** | 2 | 2 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | - | - |
| Availability % | 90 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | 85 | | |
| Utilization % | 73 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | 69 | | |
| Productivity (ft/hr) | 140 | 136 | 140 | 145 | 145 | 145 | 145 | 145 | 145 | 140 | 140 | 140 | 140 | 140 | 136 | 125 | 125 | 125 | - | - |
| **Cable Shovel Fleet** | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 | 1 |
| Availability % | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 |
| Utilization % | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 |
| Productivity (st/hr) | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 | 6,808 |
| **Hydraulic Shovel Fleet** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | - |
| Availability % | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | - | - | - | - | - | - | - | - |
| Utilization % | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | 73 | - | - | - | - | - | - | - | - |
| Productivity (st/hr) | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | 5,406 | - | - | - | - | - | - | - | - |
| **Wheel Loader Fleet** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - | - | - | - | - |
| Availability % | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | - | - | - | - | - | - | - | - |
| Utilization % | 73 | 61 | 61 | 61 | 61 | 61 | 61 | 61 | 61 | 61 | 61 | 61 | - | - | - | - | - | - | - | - |
| Productivity (st/hr) | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | 2,965 | - | - | - | - | - | - | - | - |
| **Haul Truck Fleet** | 23 | 25 | 25 | 28 | 33 | 34 | 38 | 38 | 38 | 36 | 33 | 33 | 33 | 18 | 18 | 14 | 14 | 14 | 6 | 6 |
| Availability % | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 90 | 88 | 88 | 88 | 88 | 88 | 88 | 88 | 88 | 88 | 88 | 88 |
| Utilization % | 73 | 73 | 73 | 72 | 72 | 72 | 72 | 72 | 72 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 70 | 67 | 63 | 63 |
| Productivity (st/hr) | 888 | 872 | 840 | 760 | 655 | 629 | 558 | 559 | 551 | 636 | 651 | 655 | 465 | 524 | 374 | 452 | 448 | 529 | 1,382 | 1,382 |

## 16.6 Mine Operations

### 16.6.1 Drilling and Blasting

Production drilling will be done using 10-5/8 inch holes on a 30-foot by 30-foot pattern for ore and 33-foot by 33-foot pattern for waste. Blast hole depth will be 50 feet with 5 feet of sub-drilling. Subgrade drilling in limestones and skarns may be increased if hard toe conditions are encountered.

Drilling speed rates will vary between 129-153 feet per hour depending on the rock type and mineralization. The penetration rates are consistent with rates being used by other mines like Constancia which is currently in operation and has characteristics similar to Rosemont.

Powder factors varied between 0.29 – 0.43 pounds per ton depending on rock type and mineralization. Ammonium nitrate and fuel oil (“ANFO”) blasting agents will be loaded in dry holes, while wet holes will be pumped dry and sleeved before loading with ANFO. If this cannot be accomplished, emulsion will be used as a wet hole explosive.

Drills will be outfitted with GPS and electronic sensing units to allow recording of penetration rates in drill holes to assist in determining decking requirements for individual holes. Drill productivities are expected to range between 8,500 and 10,500 tons per hour (“tph”), depending on rock type.

### 16.6.2 Loading

Major loading equipment consists of two 60 $yd^3$ class electric shovels, one 46 $yd^3$ hydraulic excavator and a 25 $yd^3$ front-end loader. On average, 71% of total material movement will be handled by the electric shovels, 22% by the hydraulic shovel and 7% by the front end loader.

The equipment was selected to work a 50-foot bench height and load 260 ton-class trucks. For this study, the 260-ton-class trucks were chosen based on economics, but the loading fleet is sized for the larger trucks to give the operator flexibility in fleet selection at a later date.

Loading 260-ton trucks with a 60 $yd^3$ class shovel requires three passes at 35 seconds per cycle, 30 second spot and queuing for a total load time of 2.30 minutes per truck. Loading the 260-ton trucks with 46 $yd^3$ hydraulic excavator requires four passes at 35 seconds per pass, a 30-second spot time and queuing time, for total load time of 2.8 minutes. Finally, 260-ton trucks loaded with a 25 $yd^3$ FEL require seven passes at 40 seconds per pass, a 30-second spot time and queuing time, for total load time of 5.2 minutes.

Loading equipment production rates vary during equipment start up, and according to operator training and experience. After reaching a steady state, the 60 $yd^3$ class shovel productivity will be 6,800 tph, hydraulic shovel will be 5,400 tph and the loader productivity will be 2,800 tph. Loading productivity is directly related to how well the shovel-loader/trucks match, the material being loaded and the haulage profile.

#### 16.6.3 Hauling

The 260-ton class truck was chosen based on an economic evaluation and as a result of the support in the region. Main factors influencing the study were fuel burn, tire costs and repair costs. Truck fleet requirements vary from 23 units at the start of pre-production to 38 by year 6. The fleet remains constant from year 6 until year 8, when the waste volumes start to decrease and only 18 units are required. In year 18, the truck requirements decrease to 6 units. An average load factor of 260 tons was used for production calculations for haulage trucks.

#### 16.6.4 Support Equipment

Major support equipment includes mine equipment that is not directly responsible for production, but which is scheduled on a regular basis to maintain in-pit and ex-pit haul roads, pit benches, WRSA and DSTF and to perform miscellaneous construction work as needed. Equipment operating requirements were estimated for this equipment based on the major mine equipment support requirements and WRSA slope re-grading schedules. Equipment in the mine support fleet includes:

- Crawler dozers, D10T2 class
- Rubber-tired dozers, 834K class
- Motor graders, 14H class
- Water trucks, 777G class

In general, the rubber-tired 834K-class dozers will be used in the pit to clean up around the primary loading units, with the track dozers used for haul road construction, pit development, WRSA and DSTF management, and final re-grading requirements. The graders and water trucks will be used to maintain roads and control dust.

## 16.7 Mine Engineering

#### 16.7.1 Geotechnical and Mine Planning

CNI was contracted by Hudbay to provide an update of their geotechnical recommendations for slope angles for the open pit development of the Rosemont deposit. The current and previous work included geologic and geotechnical mapping, drilling, rock strength testing and slope stability analysis to determine pit slope design criteria that is consistent with industry norms for safety and cost effectiveness. CNI provided a report in May 2016 - Feasibility-Level Geotechnical Study for The Rosemont Deposit.

Based on the CNI report, Hudbay worked to find the best strategy to combine geotechnical engineering, pit design, mine planning and operational point of view. With regards to geotechnical engineering and pit design, the following considerations have been made:

- Concave pit design which is more stable that a convex pit
- Height of ultimate pit is more than 2,900 feet deep at an elevation of 3,100 feet (AMSL)
- A two ramp ingress/egress system

- Drainage system following the main haulage ramps
- Mining sequence, by phases and periods:
  - Final pit wall will be established during year 08 (Figure 16-19)
  - On-going evaluation of new data resulting from actual pit development

With respect to the general mine development sequence, Hudbay has developed the following strategy:

- As part of the pit dewatering plan, three pumping wells will be installed close to the Phase I development area. As currently planned, these holes will be core drilled using PQ diameter to obtain additional geological, geotechnical, and hydrogeological information. During year 5, one additional pumping well will be developed with the same strategy (multifunction hole).
- Pre-stripping will expose several geological faults identified during the geotechnical study, allowing for better definition, exact location, geotechnical properties and behavior.
- The strategy will remain the same as the mine progresses and other faults are encountered. Mine development will include specific design parameters to minimize the unintended structural issues, specifically:
  - Inter ramp angle controls and review for optimization (wall phases)
  - Bench face angles
  - Control wall damage with blasting analyses
  - Blasting control (VPP)
  - Ground control (survey, water level)
  - Slope monitoring system

RQD (%) and the hardness block model have been developed to support a geotechnical strategy for mine design to be implemented for mine planning and operations. Figure 16-36 to Figure 16-39 show plan and section views of the block model and final Rosemont pit.

**FIGURE 16-36: ROSEMONT GEOTECHNICAL SECTORS**

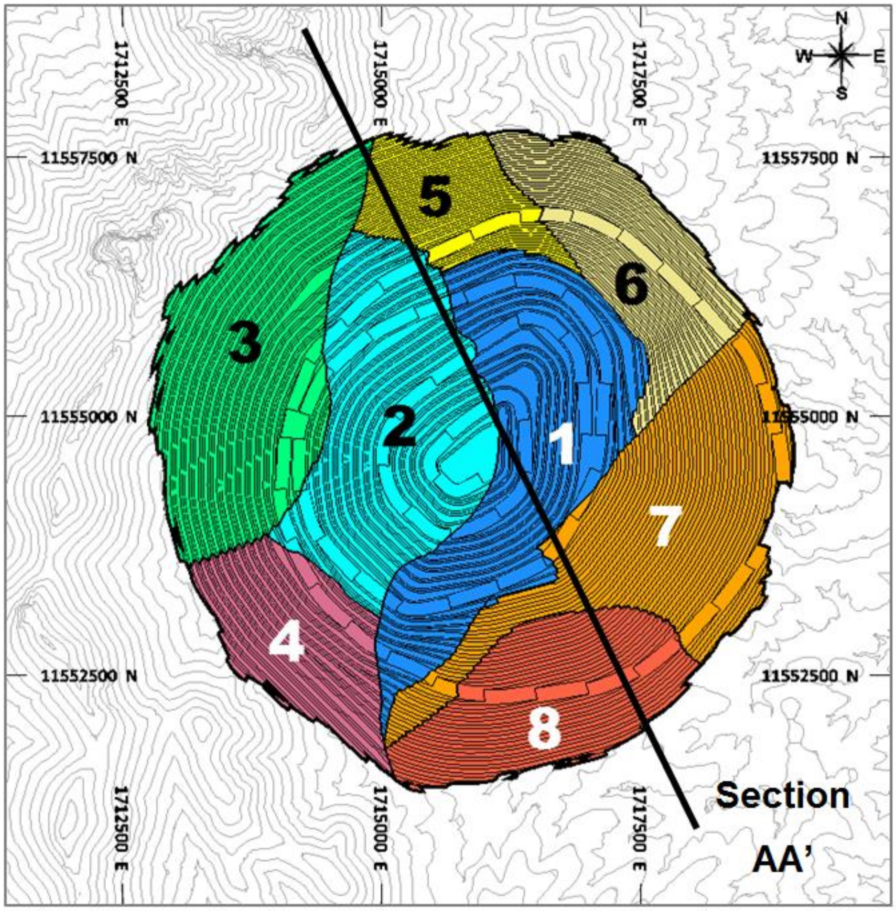



**FIGURE 16-37: SECTION AA' SHOWING RQD VALUES IN FINAL ROSEMONT PIT**

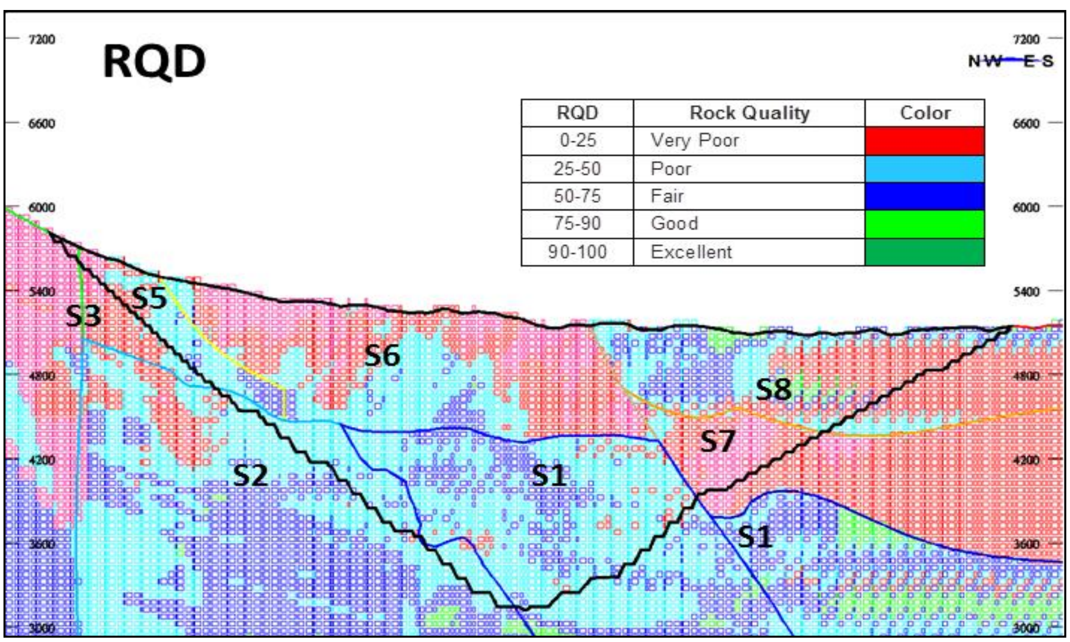


**FIGURE 16-38: ROSEMONT FINAL PIT, LITHOLOGY IN FINAL WALL**

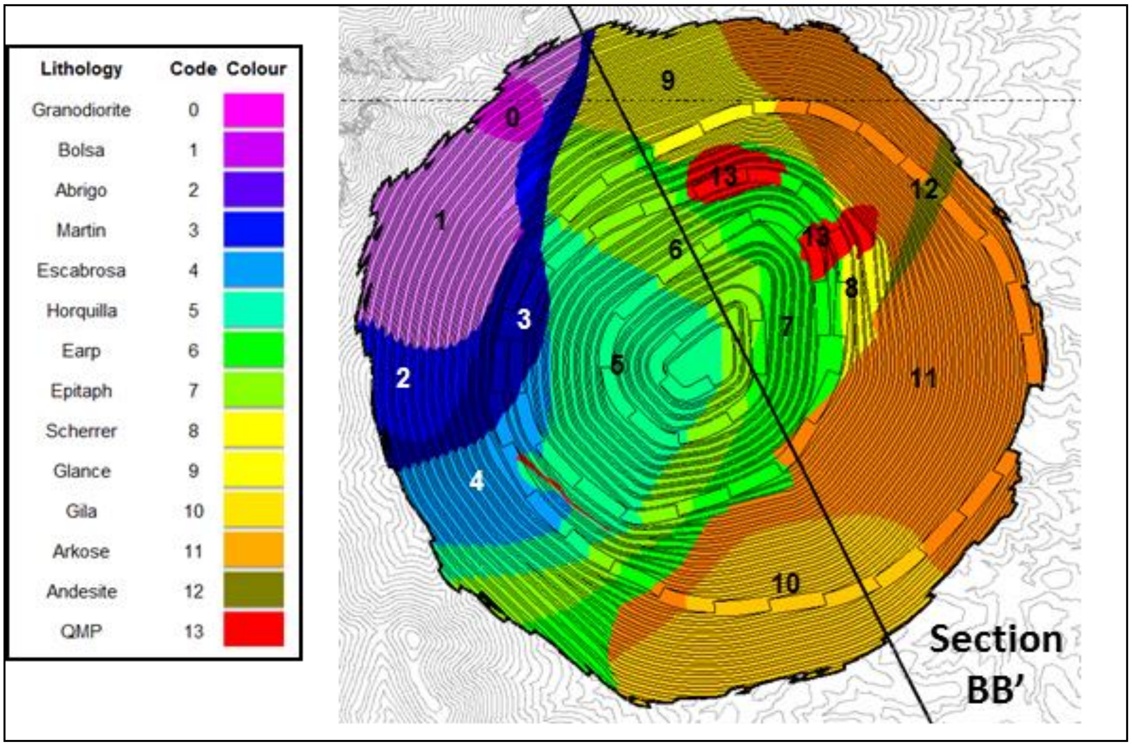
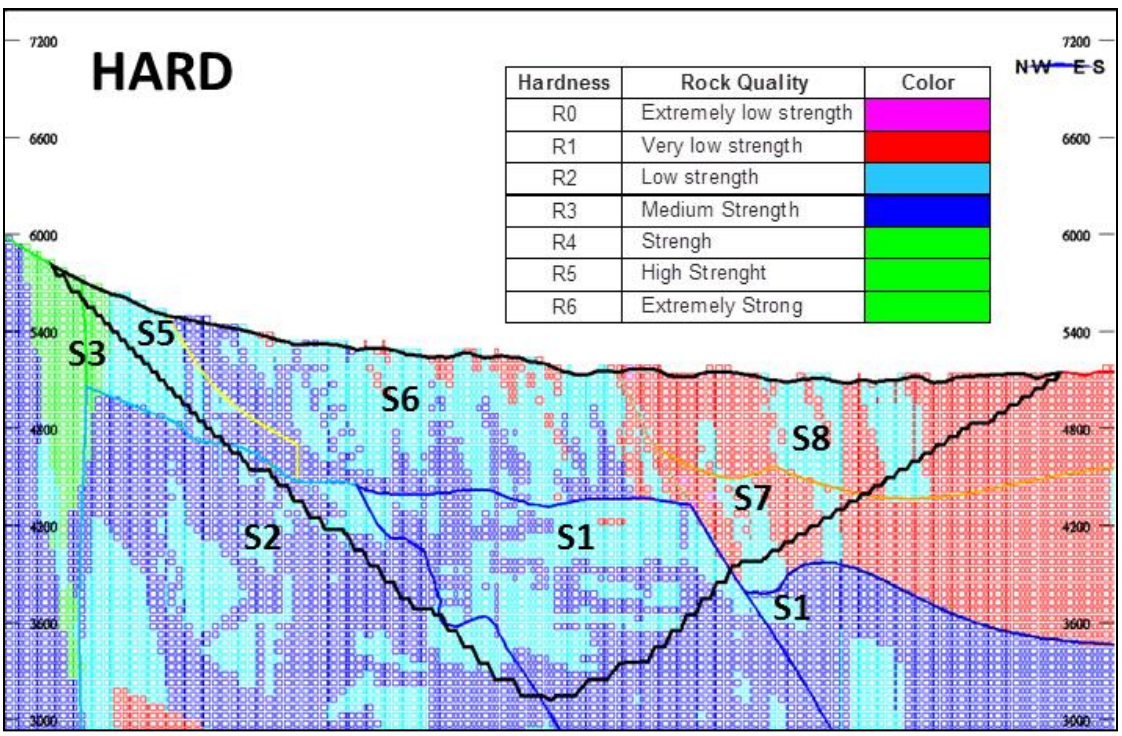



**FIGURE 16-39: SECTION BB' SHOWING HARD VALUES IN FINAL ROSEMONT PIT**



### 16.7.2 Hydrogeology and Mine Planning

Neirbo Hydrogeology was contracted by Hudbay to provide a hydrogeological study. Based on a refined and localized version of the 2010 Regional Groundwater Model prepared for the Environmental Impact Statement (EIS) process, Neirbo provided a report in May 2016 – Hydrogeological Study for The Rosemont deposit. Based on the Neirbo report, Hudbay worked to find the best strategy to combine: pit dewatering, pit design, mine planning and operational objectives.

The following general strategy has been considered:

- Starting the drilling and pumping before pre-stripping and continuing during the pre-stripping
- Dynamic updating of the hydrogeological parameters and model for each well
- Monitoring wells focused on dewatering
- Active and passive depressurization verification according to mining advance
- Updating the areas indicating high and low conductivity
- Establishing an operational correlation between the geological, geotechnical and hydrogeological parameters

The pit dewatering plan consists of:

- 10 wells during pre-production (14,850 feet)
- 10 additional wells between year 1 to year 5 (10,800 feet)
- 10 additional wells after year 5
- Annual horizontal drain sustaining capital cost ($ 3M) was considered for the arkose material as it will be mined every year in the mine's life
- Limited by pre-production pumping described in the EIS as 18,500 acre-feet of water

The pit opens up in the central-western zone; away from the final walls which is expected to provide an opportunity to:

- Pre-mining
    - To control the inter ramp angle ("IRA") and the bench face angle ("BFA")
    - To manage the water with wells and superficial water management
    - To install and monitor the impact of 14 pumping wells
- Year 1
    - As the mine expands through Phase 1, monitoring of the effects of the pumping wells will continue. This will include water captured via the in-pit ponds
    - Phase 2 will begin in year one and monitoring of the pumping wells and surface ponds will continue.
- Year 2

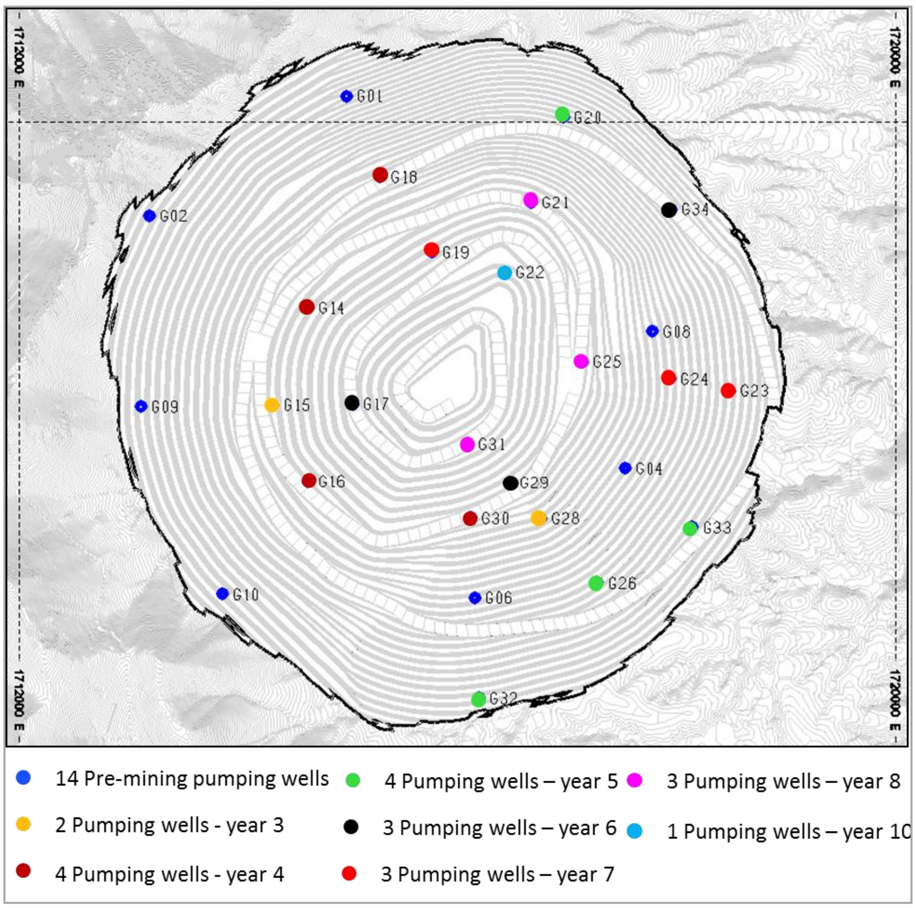
- o Phases 1 and 2 will remain the active mining areas supported by the original pumping wells and ponds.

- Year 3

  - o Beginning in Year 3 and continuing for the remainder of the mine's life, additional wells will be drilled to achieve the drawdown and depressurization requirements necessary to safely advance the ore extraction sequence.

Figure 16-40 presents LOM well holes in the final Rosemont pit, and Table 16-13 summarizes LOM well holes for the pre-production and operating stages.

**FIGURE 16-40: SECTION AA' SHOWING HARD VALUES IN FINAL ROSEMONT PIT**




## 16.8 Manpower Requirements

### 16.8.1 Mine Operations Manpower

Mine supervision, technical staff, mine maintenance, workshop personnel and equipment operator requirements over the life of the mine are based on the mine plan. During the Pre-Production period, direct (workshop and operators) and indirect (Staff, supervision and technicians) requirements total 337, building up to 424 in the steady production (132 M ton per year) and is shown in Table 16-14.

Hourly mine operation personnel requirements are calculated based on equipment operating hour requirements, fleet estimation, shown in Table 16-12. Maintenance personnel are calculated based on estimated maintenance repair time for mining equipment. The percent of maintenance to total hourly personnel averages 32% throughout the mine life.

The schedule assumed that the hourly personnel would be hired two months prior to the date they were actually required on-site to facilitate training requirements for MSHA, Safety, Environmental and other required training which is captured in the mining costs. Operation training is captured in Operational Readiness estimates.

Mine staff manpower employees and salaries were developed for Mine Administration, Mine Geology, Mine Operations, and Mine Maintenance. Salaries were a composite of information provided by Hudbay which was calibrated against local mine salaries. Salary information includes wages, burden and bonus for staff employees.

**TABLE 16-14: LABOR ESTIMATION FOR ROSEMONT MINE OPERATIONS**

| Item | Category | Mine Operations Labor Requirements (Steady Production) |
|---|---|---|
| Mine Operations | Direct Labor | **227** (Operators for shovels, trucks, drills and auxiliary equipment) |
| Workshop Personnel | Direct Labor | **90** (Mechanics, Welder, Electrician and Helpers) |
| General Mine Staff | Indirect Labor | **34** (Management, Supervision & Laborer) |
| Technical Services | Indirect Labor | **26** (Mine Engineer, Geologist, Geotechnical and Surveyor) |
| Mine Maintenance | Indirect Labor | **47** (Management, Supervision and Technicians) |
| **TOTAL** | | **424** |

### 16.8.2 Process Operations Manpower

Hourly process plant operation personnel requirements are estimated based on hourly equipment operating criterion, and fleet estimation as shown in Table 16-15. Maintenance personnel are determined based on estimated maintenance repair time for equipment. The percent of maintenance to total hourly personnel averages 32% throughout for the mine life.

Operating labor costs are based on staffing levels developed by the Project for a copper molybdenum concentrator with a DSTF.

Staff positions were grouped into four broad categories:

1. Mill Management
2. Mill Operations
3. Mill Maintenance
4. Mill Technical Services

Staff numbers are based on 24-hour operating coverage with personnel working 12-hour shifts on a four crew roster system. Labor positions categorized are summarized in Table 16-15.

No labor has been included for an analytical laboratory because there will be no on-site facility for this purpose. Mine, plant, and concentrate quality samples will transported off site to a contract laboratory.

Besides staff labor, allowances have been made for contract labor for major periodic tasks. This is to free up staff maintenance personnel and allow them to perform required/preventative maintenance tasks while the affected circuits are down.

**TABLE 16-15: LABOR ESTIMATION FOR ROSEMONT PROCESS OPERATIONS**

| Item | Mine Operations Labor Requirements (Steady Production) |
|---|---|
| Mill Management | **22** (Management and Supervision) |
| Mill Operations | **76** (Operations Technicians) |
| Mill Maintenance | **38** (Trade Workers and Technicians) |
| Mill Technical Services | **15** (Engineers and Technicians) |
| **TOTAL** | **151** |

### 16.8.3 General and Administration (G&A)

General and Administration has been derived from each area within the G&A group. The estimate for each area was built up using project manpower inputs and industry standards values.

The G&A group consist of the following departments and are showed in Table 16-16:

1. General Administration - includes general office, legal and corporate affairs areas.
2. Procurement - included in this area are administration and logistic.
3. Safety
4. IT- Information and Technology and support
5. Accounting – includes controller, payroll, taxes and finance staff.
6. Environmental – included in this area are permitting, monitoring, and mitigation personnel.
7. Human Resources

**TABLE 16-16: LABOR ESTIMATION FOR ROSEMONT G&A OPERATIONS**

| Item | G&A Labor Requirements (Steady Production) |
|---|---|
| General Administration | **16** (Management, Legal, and Corporate Affairs) |
| Procurement | **17** (Administration and Logistic) |
| Safety | **8** |
| IT | **8** |
| Accounting | **12** (Controller, Payroll, Taxes, Finance) |
| Environmental | **22** (Monitors, Technicians) |
| Human Resources | **5** |
| **TOTAL** | **88** |

# 17 RECOVERY METHODS

## 17.1 Introduction

The Project process plant is a conventional copper-molybdenum concentrator and process design is typical of concentrators treating low sulfur copper porphyry-skarn style ores. The process involves crushing, grinding, flotation, molybdenum separation, concentrate dewatering, and tailings dewatering.

With minor modifications, the plant is designed to process on average 90,000 ton/d (32.8 million ton/y) of ore. The Project details included in this section were specifically designed and evaluated to fall within the permitted facility constraints included in the EIS and State of Arizona permits while optimizing production, minimizing costs and ramping up production as quickly as technically and environmentally as possible. State of Arizona Department of Environmental Quality (ADEQ) permits that were issued based on early designs will be amended to include designs included in the EIS and this section; these amendments are customary in the state.

This Technical Report includes refinements of certain aspects of the Project's mine plan. While consistency with issued and pending environmental permits and analysis related thereto has always been a key requirement for this effort, updates to the original mine plan will be necessary. To the extent that any regulatory agency concludes that the current plan requires additional environmental analysis or modification of an existing permit, the intent will be to work with that agency to either complete the required process or to adjust the current mine plan as necessary.

### 17.1.1 Facility Layout and Location

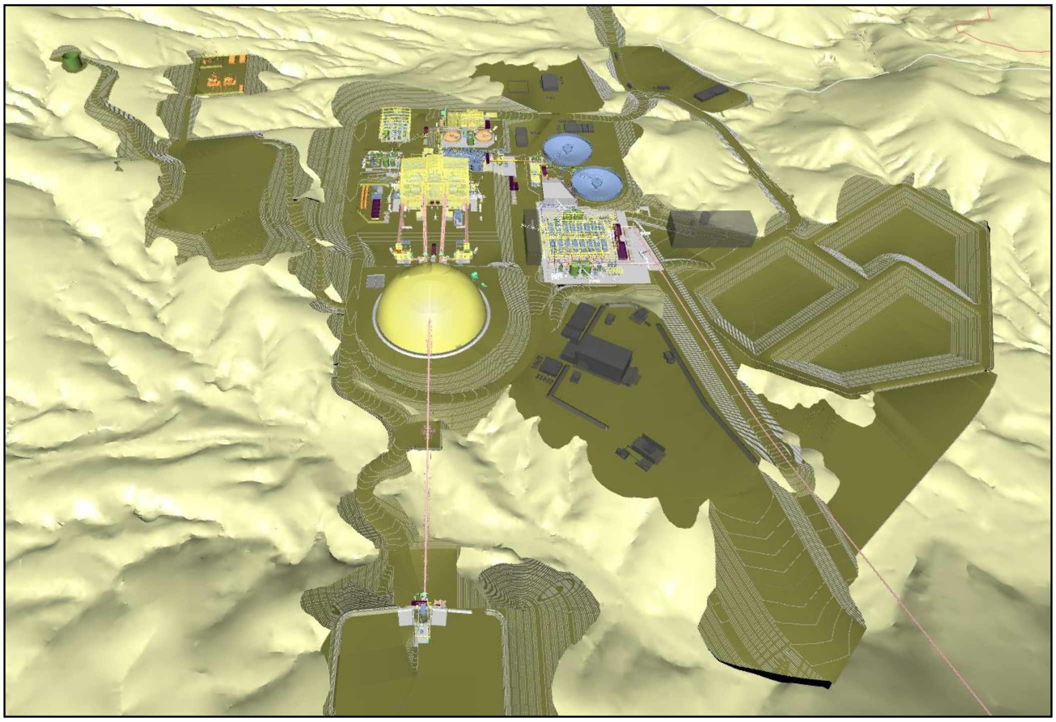
The plant will be located east of the open pit and has been arranged in a north-south orientation. Ore flow is from south to north with the ROM stockpile and dump pad located to the south and concentrate filtration and load-out located at the north end of the facility.

ROM ore will be transported from the mine to the primary crusher by off-highway haulage trucks. After crushing the ore, it will be conveyed to the Coarse Ore Stockpile to then be conveyed to the concentrator facility.

Copper concentrate produced at the concentrator facility will be loaded onto highway haul trucks for transportation to smelting and refining facilities. Molybdenum concentrate will be bagged and loaded onto trucks for shipment to market.

An overall view of the processing facilities is given in Figure 17-1.

**FIGURE 17-1: OVERALL VIEW OF PROCESS PLANT LOOKING NORTH**



### 17.1.2 Facility Description

The process plant is modelled after the Constancia Copper Project design with changes made in certain areas. In some cases, ore characteristics, local conditions and permitting constraints have dictated that changes be made such as the filter plant for dry stack tailings.

With minor modifications, the plant is designed to process 90,000 tpd (32,850,000 tpy). Production during the first year of operation is expected to fall short of full capacity (28,114,000 tons) to account for a staggered commissioning schedule and brief ramp-up and optimization period. Annual concentrate production is expected to reach an average of 344,000 tons.

The coarse ore stockpile, grinding areas, pebble crushers, conveyors, molybdenum plant, and filter plant process areas will be covered and painted to match the natural landscape. Design reviews were held to ensure that safety, environmental, permitting, technical, quality, constructability, maintainability and operability considerations had been correctly addressed.

Key facility design criteria are summarized in Table 17-1.

**TABLE 17-1: KEY FACILITY DESIGN CRITERIA**

| Parameter | Units | Value |
|---|---|---|
| Plant capacity | Mton/yr | 32.9 |
| Copper feed grade | Avg % | 0.48 |
| Copper feed grade | Max % | 0.66 |
| Molybdenum feed grade | Avg % | 0.014 |
| Molybdenum feed grade | Max % | 0.023 |
| Copper concentrate grade | % | 32 |
| Molybdenum concentrate grade | % | 45 |

The design grades were selected on the basis of the prevailing mine plan in order to properly size flotation cells, pumps and pump boxes, pipelines, tanks, and concentrate de-watering and handling equipment to ensure adequate unit operation capacities under the vast majority of circumstances. While it is possible for feed grade to the plant to fall outside of range of these parameters, these excursions are expected to be both rare and brief, and can be managed with relative ease. Average and monthly maximum copper head grades in the current mine plan are 0.447% and 0.670%, respectively.

## 17.2 Buildings

The facility will include buildings for the following:

- Administration Facilities comprised of the Administration Building, Medical Clinic/Emergency Response Building, vehicular staging pads and dedicated vehicular parking areas.
- Security Gatehouse and Weighbridge (truck scale) are located at the entrance to the plant.
- Plant Maintenance Building where all routine plant maintenance activities are conducted. Additional ancillary space includes a secure tool crib, parts store and electrical and instrument shop.
- Metallurgical Lab which includes space for sample preparation and testing.
- Plant Change House that provides space for plant workers to shower, change and use the restroom.
- Plant Production & Maintenance Office for plant and maintenance staff.
- Plant Control Room, a modular building housing the plant control operators.
- Plant Warehouse Building with Plant Warehouse Office and external warehouse yard.

## 17.3 Processing Plant

The process plant design is based on a combination of metallurgical testwork, Project production plan and in-house information. Benchmarking has been used to define and support the design parameters. This includes the copper-molybdenum separation circuit where testwork has been limited to a few tests. This is due to the relatively large sample mass required for a more detailed molybdenum testwork program and analysis.

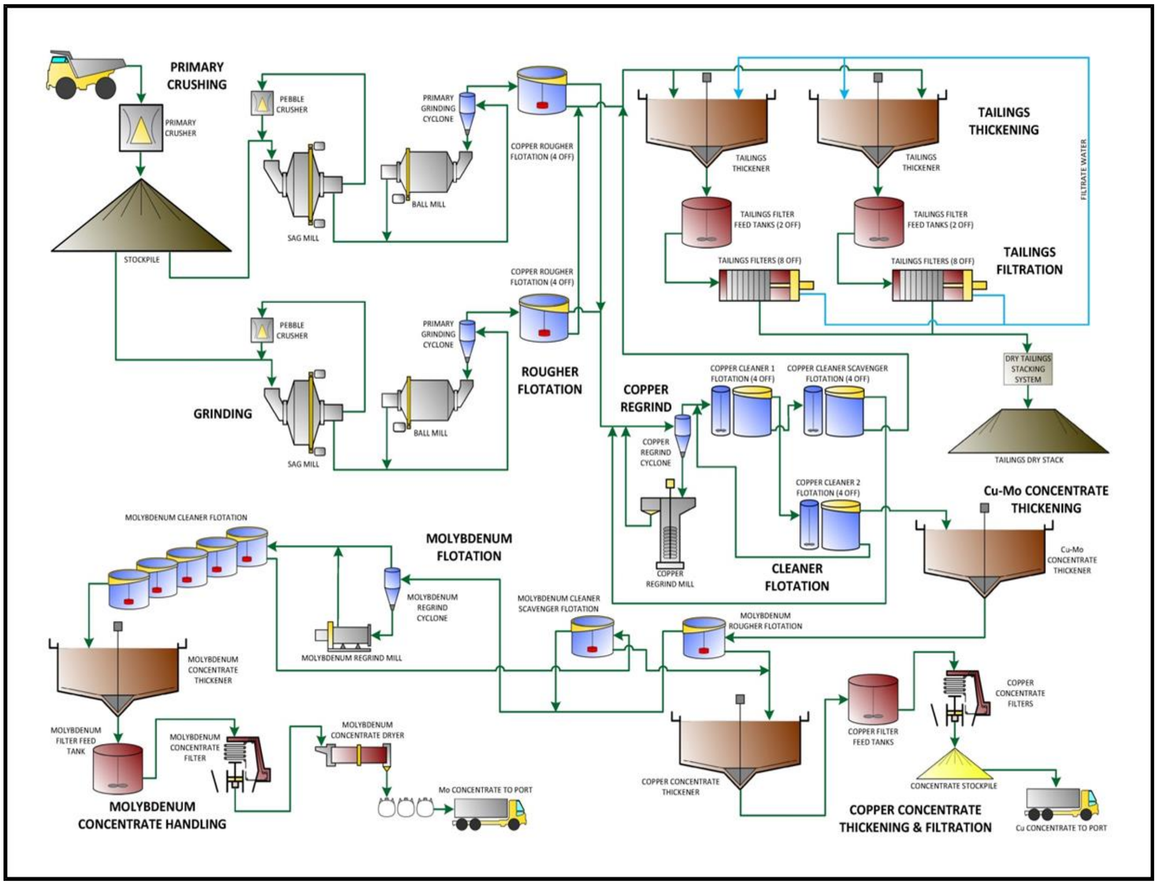
The molybdenum plant design is based primarily on projected mass flows, grades and densities as well as the recent Constancia plant design.

The flowsheet has been developed from previous feasibility study work, value engineering studies and recent testwork. The Rosemont process plant includes the following unit processes and facilities:

- Primary crushing.
- Crushed ore stockpile and reclaim.
- Parallel SABC grinding lines with pebble crushing.
- Copper flotation comprising rougher flotation, concentrate regrind, and two stage cleaning.
- Cu-Mo concentrate thickening.
- Molybdenum flotation including roughing, concentrate regrind and five stages of cleaning.
- Molybdenum concentrate thickening, filtration and drying.
- Copper concentrate thickening and filtration.
- Copper concentrate load out and storage.
- Tailings thickening, filtration and dry stacking.
- Reagents storage and distribution (including lime slaking, flotation reagents, water treatment and flocculant).
- Grinding media storage and addition.
- Water services (including fresh water, fire water, gland water, cooling water and process water).
- Potable water treatment and distribution.
- Air services (including high pressure air and low pressure process air).
- Plant control rooms.

A generalized process flow diagram (PFD) is provided in Figure 17-2.

**FIGURE 17-2: PROCESS PLANT PROCESS FLOW DIAGRAM**




The flowsheet consists of primary crushing, followed by two parallel SAG, ball milling and pebble crushing (SABC) circuits, copper flotation with regrinding ahead of cleaning, a moly separation circuit, concentrate thickening and filtering and tailings thickening, filtering and dry stacking.

With minor modifications, the process plant is designed to treat on average 90,000 tons/d (or 32.8 million ton/y).

Key design criteria used in the plant design are summarized in Table 17-2.

**TABLE 17-2: KEY DESIGN CRITERIA**

| Parameter | Units | Value |
|---|---|---|
| Plant capacity | tons/day | 90,000 |
| Flotation feed size, $P_{80}$ | µm | 140 |
| Flotation feed density, nominal | % solids (w/w) | 34 |
| Flotation feed density, minimum[1] | % solids (w/w) | 28 |
| Tailings thickener underflow density | % solids (w/w) | 65 |
| Tailings filter cake moisture | % | 15 |

Notes: 1. Minimum density at which design rougher minimum residence time can be achieved.

The process plant capacity was determined by a combination of ore characterization testwork, environmental factors and regulatory constraints, mine planning, engineering estimation, and financial analysis to define best economic return for the Project. Flotation feed size was selected on the basis of the best balance of moderating energy input in the grinding circuit and achievement of recovery targets. Flotation feed density design values are based on the results of numerous flotation tests and typical industry practice. Tailings dewatering targets are selected on the basis of thickening and filtration testwork, capacity requirements, and environmental compliance limitations. Equipment and unit operations throughout the plant have been designed to meet these requirements on a routine basis.

The overall annual plant operating schedule is 8,059 hours (92% of available hours). Operating availability is summarized in Table 17-3.

**TABLE 17-3: PLANT UTILIZATION SUMMARY**

| Description | Units | Value |
|---|---|---|
| Crusher Utilization | % | 75 |
| Grinding and Flotation Availability | % | 93.0 |
| Concentrate Filter Utilization | % | 84 |
| Tailings Filtration and Dry Stack Utilization[1] | % | 98.6 |
| Overall Concentrator Asset Efficiency[2] | % | 92.1 |

[1]factored on a mill runtime basis [2]assumes production ceases when ore flow to the SAG mill is interrupted

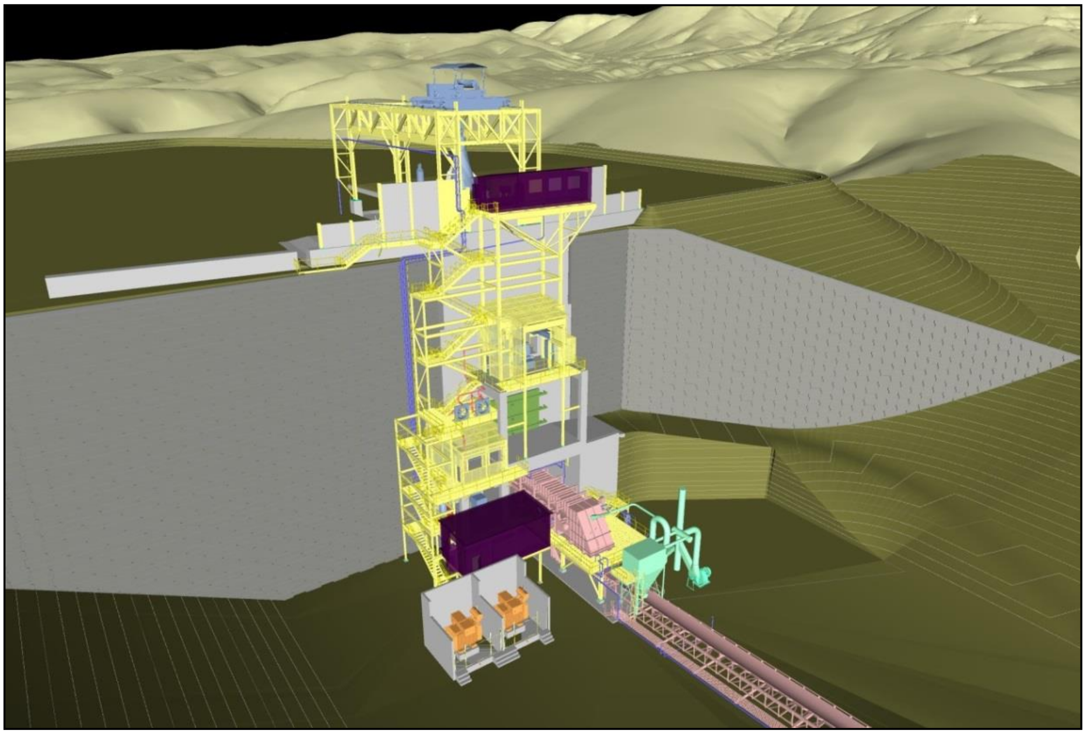
Availability estimates are based on typical industry experience for plants of similar size and configuration and utilizing typical maintenance and operating practices. A 99% utilization factor is applied to the availability to derive the asset efficiency factor, which accounts for non-productive time unrelated to mechanical maintenance or failure such as shut-down (grind-out) and start-up, lack of ore, or other upstream/downstream constraints.

## 17.4 Crushing

### 17.4.1 Primary Crushing

The Primary Crusher is a 60 x 113 size Gyratory Crusher fitted with manganese steel concave and mantle liners. The primary crusher was selected based on the required mill feed rate, expected ROM feed size distribution, ore bulk density, crushing work index, stockpile capacity, and SAG feed size. The design crusher feed rate is 6,000 tons/h with a capacity of 90,000 tons/d, based on a crusher availability of 75% and a 20% catch-up capacity factor. Design parameters including the expected range of feed size distributions and crushing work indices were provided to vendors to confirm throughput performance and motor power requirements. The primary crusher dump pocket is designed to allow two trucks to dump simultaneously, one from each side. It has the capacity to hold two truckloads, approximately 520 tons in total.

**FIGURE 17-3: PRIMARY CRUSHER**



A modular Crusher Control Room is located above the ROM wall to provide a direct line of sight to the dump pocket as well as the Stockpile Feed Conveyor. The Control Room includes space for crushing plant operators and two mine fleet controllers who manage the truck fleet. It is fitted with washroom and break facilities. A water spray system is fitted at each corner of the dump pocket for dust control when trucks are dumping. Dust generated in the transfer point between the Feeder and the Stockpile Feed Conveyor is captured by a dedicated Cartridge Dust Collector. Dust generated in the crusher vault is vented to the dump hopper and controlled by the water spray system.

### 17.4.2 Stockpile Feed Conveyor

Crushed ore is transferred from the Primary Crusher to the Crushed Ore Stockpile by a Stockpile Feed Conveyor. The conveyor belt is 72 inches wide and 1,014 feet in length with a 211-foot lift and has capacity to convey 6,600 tons/h of crushed feed to the stockpile (i.e. crusher capacity +10%). The conveyor is covered outside the Stockpile Dome to minimize dust emissions.

The conveyor is driven by two 1,200 HP drives (one mounted on each side of the drive pulley) complete with high speed disc brake and variable speed drive for controlled start-up. The drives are located adjacent to the gravity take-up.

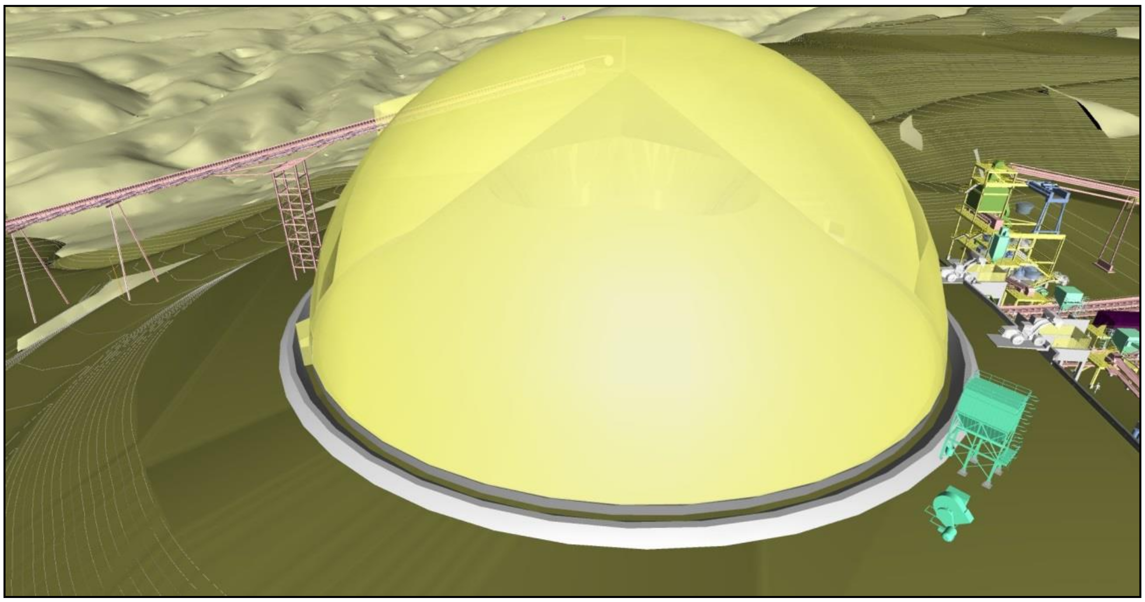
### 17.4.3 Coarse Ore Stockpile and Reclaim

Crushed ore for both grinding lines is stored in a single Conical Ore Stockpile. The stockpile is enclosed by a dome structure with an impervious colored fabric cover (approx. 380 feet diameter and 164 feet high) as shown in Figure 17-4.

The Stockpile Cover consists of a structural steel multi-arch frame pinned to a circular ring beam at the center of the dome and pinned at ground level to a concrete ring beam. A 20-foot-wide path around the stockpile perimeter provides access for dozer and equipment travel inside the cover. Two sets of access doors are included to allow machinery access to the stockpile area.

Live capacity of the stockpile will range between 23,000 to 51,000 tons (6 to 12 hours at full production rate and dependent on prevailing ore characteristics), representing 14% to 28% of the total stockpile capacity of 198,000 tons. Stockpile ore in dead storage can be reclaimed by heavy equipment (dozer and/or excavator) to allow for up to two additional days' interruption of feed from the primary crusher.

**FIGURE 17-4: STOCKPILE FABRIC COVER**



Coarse ore is reclaimed from the stockpile by four 72-inch wide Apron Feeders, two for each grinding line. Each Apron Feeder is fitted with a variable-speed drive and has the capacity to provide 100% of the full tonnage rate to its respective SAG mill.

## 17.5 Grinding

### 17.5.1 SAG and Ball Mill Grinding

The selected grinding circuit consists of two parallel SABC grinding lines, each comprising one SAG mill in closed circuit with a sizing screen and pebble crusher followed by a ball mill in closed circuit with hydrocyclones. The selected grinding mills are summarized as follows:

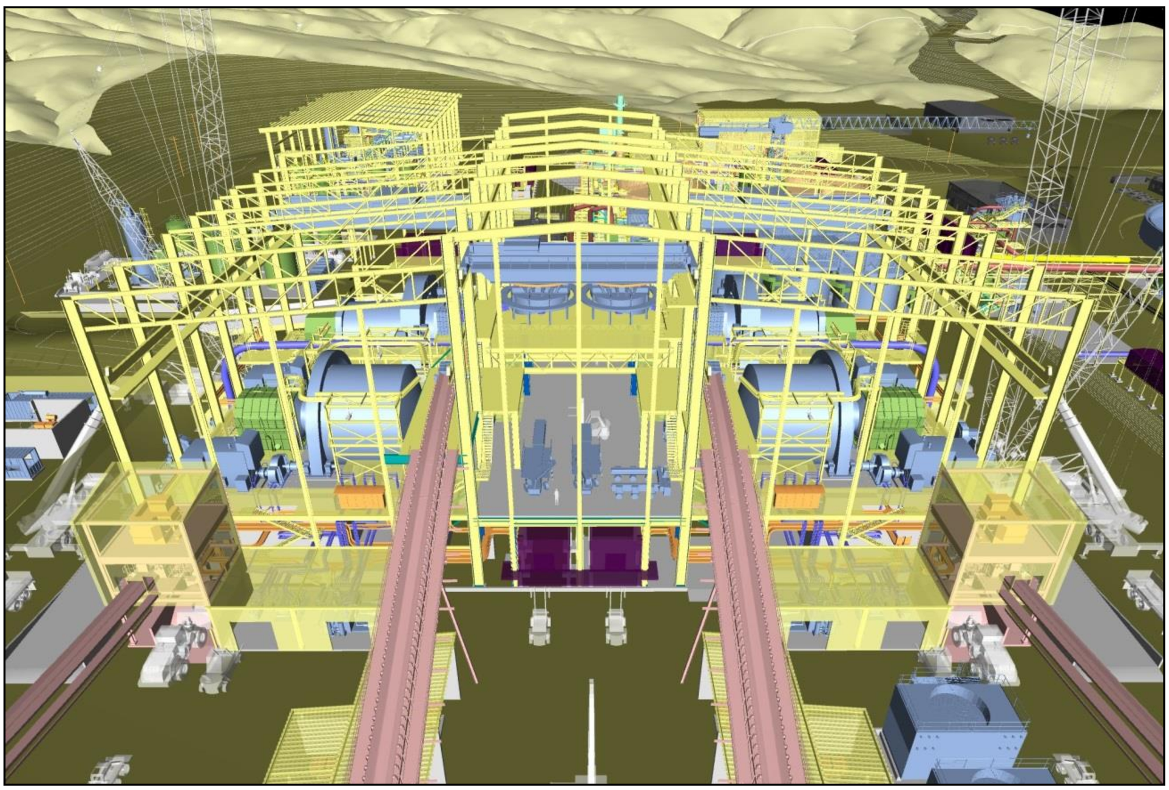
- SAG mills – 36 feet diameter (inside shell) x 24 feet effective grinding length (EGL) with 18 MW twin pinion drives (9 MW per pinion). Motors are controlled via an SER / hyper synchronous variable speed drive.

- Ball mills – 26 feet diameter (inside shell) x 40.5 feet effective grinding length (EGL) with 16.4 MW GMD.

Mill size and power requirements were determined via Ausgrind, Ausenco's proprietary power-based comminution calculation program using the 75th percentile values of ore parameters (ore competency and hardness). RQD was also used to adjust SAG feed size based on a correlation identified between RQD and sample depth. A +10% design factor was also added to the SAG motor specification to ensure mill power limitations would not be a significant factor for achieving throughput targets.

The mills are positioned at right angles to the Feed Conveyor as shown Figure 17-5 to minimize footprint. A full-width platform is located in between the Grinding Mills to provide sufficient space for Mill relining activities.

**FIGURE 17-5: GRINDING BUILDING FROM STOCKPILE LOOKING NORTH (ROOF AND WALLS REMOVED)**

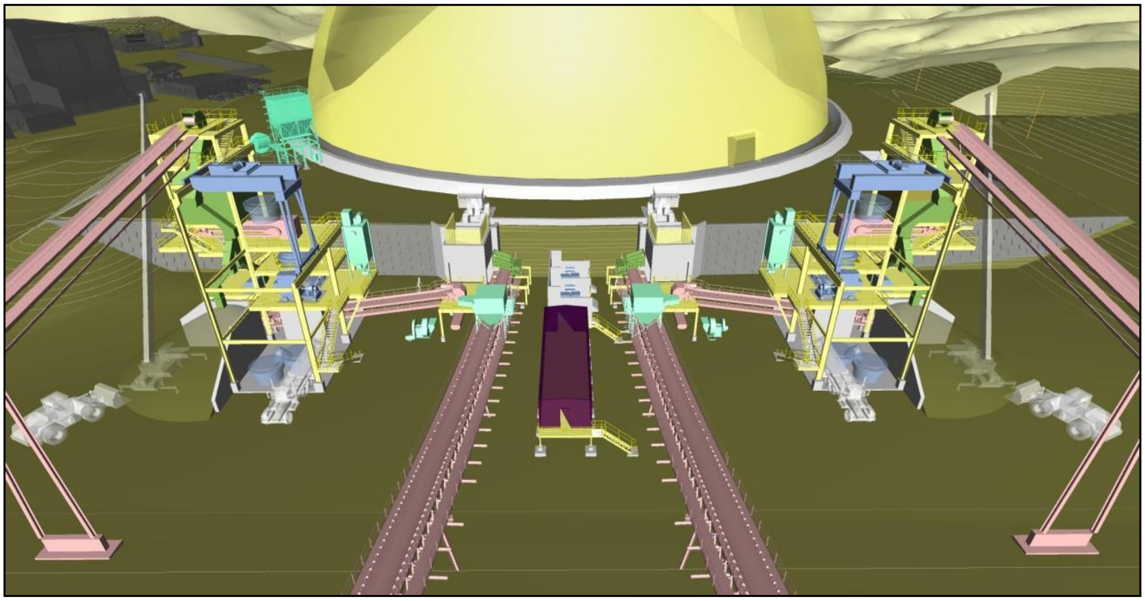


### 17.5.2 Pebble Crushing and Conveyor Systems

Measured ore competency shows that an SABC grinding circuit with a pebble crusher is required. Pebble crushers were selected based on crusher feed rate, ore characteristics and competency factors, pebble top size, and crusher product size. Pebble rate is expected to vary from 510 tph (15% to 25% of new SAG mill feed nominally), up to 700 tph (30% with worn SAG mill grates).

Cone crushers (one per grinding line) were selected for pebble crushing duty and will have sufficient capacity of 495 tph to over 1000 tph depending on liner profile and gap setting.

**FIGURE 17-6: PEBBLE CRUSHING LOOKING SOUTH FROM THE GRINDING AREA**



Pebbles from each SAG Mill are transferred to the Pebble Crusher Bin via the 54-inch Pebble Conveyors. Tramp metal is captured by two cross-belt self-cleaning magnets arranged in series for each crusher.

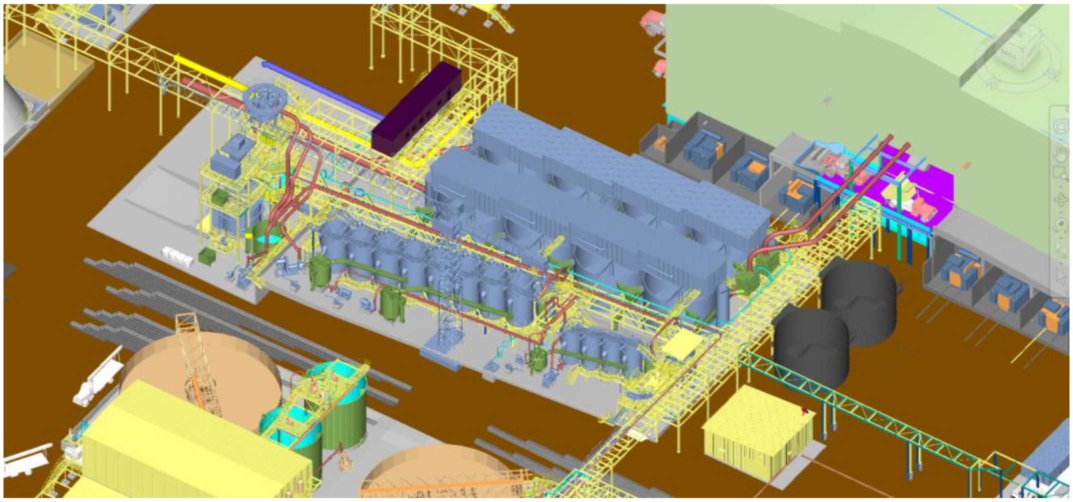
The Pebble Conveyor discharges onto a diverter gate which directs the material to the Pebble Crushing Bin or allows bypass to the Pebble Conveyor Bypass Bunker or the Pebble Crusher Product Conveyor.

## 17.6 Copper Flotation

The Copper Flotation Area is positioned perpendicular to the Grinding Area taking advantage of the ground sloping west to east as shown in Figure 17-7.

The circuit consists of two parallel trains of Rougher Flotation Cells. A third train of Cells includes the first and second stage Cleaner Cells as well as the Cleaner Scavenger Cells and is arranged parallel to the Rougher Cells.

**FIGURE 17-7: COPPER FLOTATION**



The copper flotation circuit consists of two parallel trains of four rougher flotation cells followed by rougher concentrate regrind and two stages of cleaner flotation. Flotation feed, from primary cyclone overflow, reports to two rougher flotation trains each consisting of four forced air mechanical flotation tank cells. The Copper Rougher Flotation Cells are conventional 630 $m^3$ (22,000 $ft^3$) tank cells with 650 HP direct drive arrangement and are fed low pressure air by blowers. The two lines provide a total of 34 minutes' residence time at the nominal feed density (34% solids) and 28 minutes' residence time at 28% solids feed density at a potential 90,000 tpd.

The rougher flotation cells produce a low grade copper-molybdenum (Cu-Mo) concentrate that requires further liberation and upgrading. Copper rougher concentrate is combined in the copper regrind feed hopper and pumped to the regrind cyclones for classification.

The copper rougher tailings stream from each train are combined and gravitate to the flotation tailings thickener feed distributor via a cross-cut sampler. The cleaner scavenger concentrate can also be directed to the regrind feed hopper if required. The underflow from the copper regrind cyclone cluster reports to the regrind mill, which overflows back into the copper regrind feed hopper.

The overflow from the regrind mill cyclone cluster is pumped directly to the copper cleaner flotation circuit. The copper cleaner circuit consists of two stages of cleaning and one bank of cleaner scavenger cells.

The first cleaner consists of four forced-air mechanical staged flotation reactor ("SFR") flotation cells complete with froth wash water system to minimize non-sulfide gangue entrainment. The second cleaners also comprise four forced-air mechanical SFR cells with froth wash water systems.

The Copper Cleaner 1, Copper Cleaner 2 and Cleaner Scavenger flotation cells will be SFRs. The Copper Cleaner 1 and Cleaner Scavenger cells will have approximately 3,900 $feet^3$ of volumetric capacity, while the Copper Cleaner 2 will have 1600 $feet^3$ of volumetric capacity.

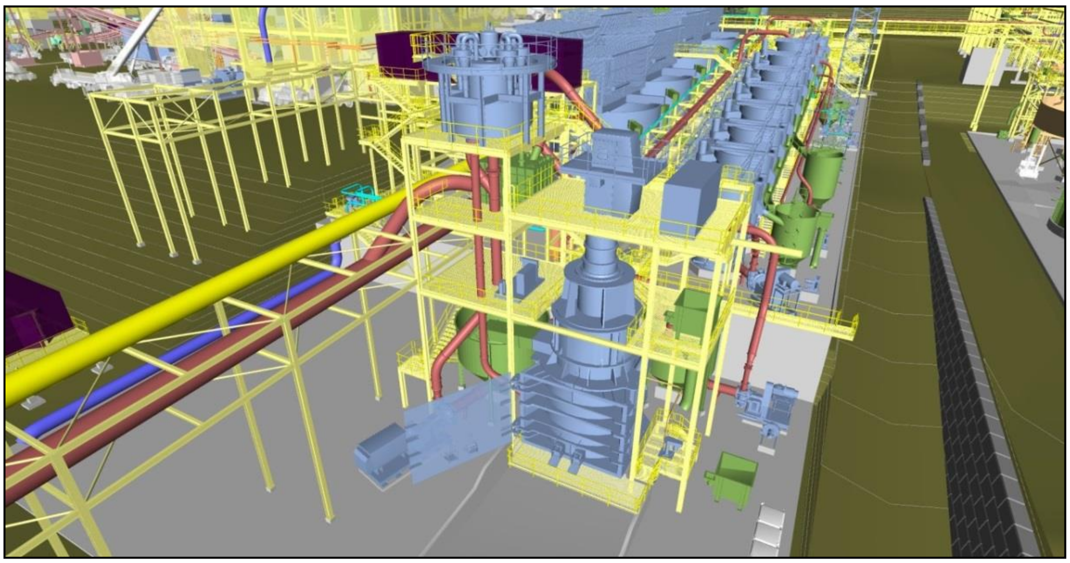
SFR flotation cells were selected for the cleaner duty due to their ability to achieve high upgrade ratios with a relatively small footprint and reduced air and power consumption.

The online stream analyzer ("OSA") is located in a dedicated area on the west side of the facility. Major concentrate and tailings streams are pumped to the OSA to allow optimization of reagent additions and flotation performance. Samples are collected using Gravity Samplers or Pressure Samplers at the pump discharges and transferred to the OSA using peristaltic pumps. The samples are sorted by a multiplexer and are returned to the process by horizontal centrifugal pumps after analysis.

### 17.6.1 Copper Regrind

The Copper Regrind circuit consists of a Copper Regrind Mill Feed Hopper and a Copper Regrind Cyclone Cluster in a closed circuit with the Copper Regrind Mill, as shown in Figure 17-8.

**FIGURE 17-8: COPPER REGRIND AREA LOOKING WEST**



The concentrate regrind mill was selected based on the expected range of concentrate feed rate (rougher mass pull), estimated feed size and product size (40 µm). Regrind product size was selected based on mineralogy studies and locked cycle flotation tests.

A typical regrind cyclone partition curve was used to estimate cyclone underflow size distribution and cyclone mass split. The estimated nominal recirculating load based on simulated size distributions was 170% with regrind mill feed size $F_{80}$ (cyclone underflow) approximating 70-75 µm.

Regrind mill power requirements were estimated based on the overall flotation circuit mass balance, grind-size target as determine by testwork programs (40 µm), and Ausenco's in-house regrind specific energy data set calculations. A VertiMill with 3000 kWh motor was selected for regrinding duty.

### 17.6.2 Copper-Molybdenum Concentrate Thickening

The bulk copper-molybdenum (Cu-Mo) concentrate is pumped from the copper flotation circuit to the Cu-Mo concentrate thickener via a trash screen. The trash screen removes coarse oversize that can damage or block downstream equipment, e.g. copper pressure filter ports and metallurgical samplers. Trash reports to a collection box via a chute. Trash screen undersize gravitates to the copper-molybdenum concentrate thickener via a thickener feed box.

Cu-Mo concentrate is thickened to reduce the volume of residual copper flotation reagents in the molybdenum flotation feed.

Thickener overflow from the Cu-Mo concentrate thickener is pumped to the tailings thickener where it is recovered as process water. Cu-Mo concentrate thickener underflow is pumped to the molybdenum flotation circuit by centrifugal pumps.

The Cu-Mo concentrate thickener and molybdenum flotation circuit can be bypassed if required by diverting the bulk Cu-Mo concentrate (from copper flotation) direct to the copper concentrate thickener.

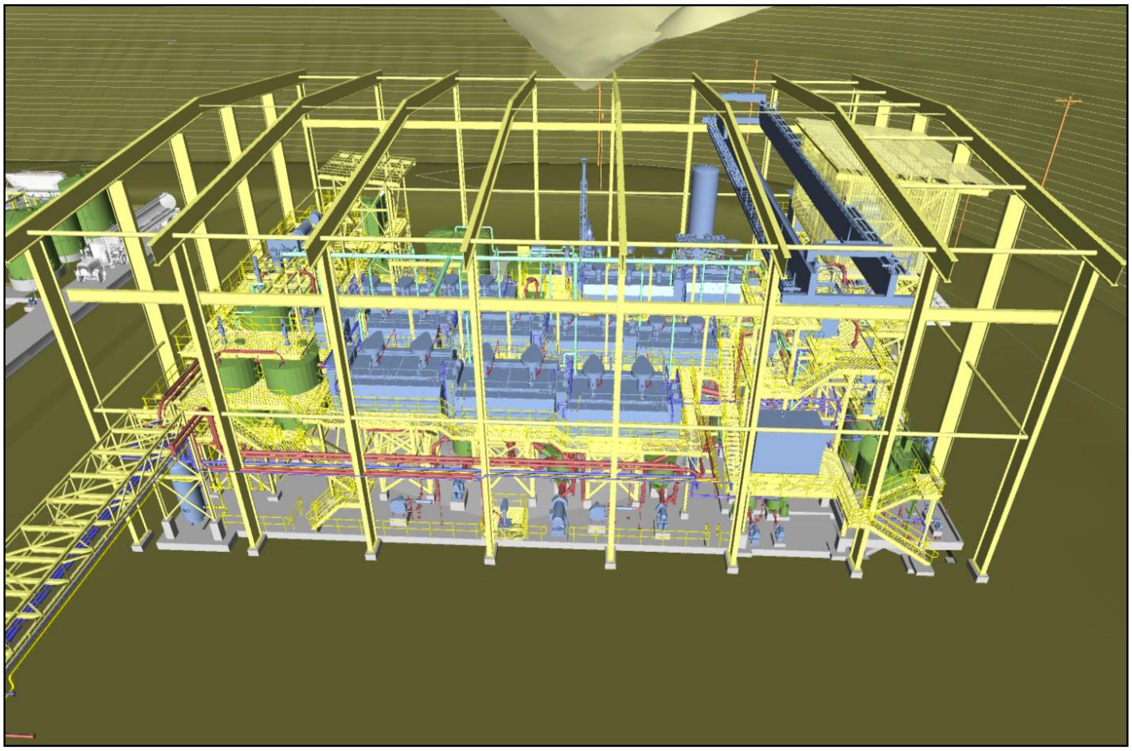
The area is bunded and can contain the entire volume of a Filter Feed Tank combined with the Copper Area bund. Any further spillage overflows into the site drainage system and reports to the primary settling basin.

## 17.7 Copper-Molybdenum Separation

The molybdenum separation circuit consists of rougher flotation cells followed by a regrind circuit, five stages of cleaner flotation and one stage of cleaner scavenger flotation. All flotation cells, with the exception of the fifth cleaner cell (Jameson cell), are self-aspirated mechanical flotation cells fitted with covers. All self-aspirated cells are driven by a v-belt drive.

The Molybdenum Plant is shown in Figure 17-9.

**FIGURE 17-9: MOLYBDENUM PLANT LOOKING WEST (ROOF AND WALLS REMOVED)**



Flotation cells selected for the molybdenum separation circuit are summarized as follows:

- Rougher – six 988 $ft^3$ (28 $m^3$) self-aspirated inert gas cells in a 2-2-2 configuration.
- Cleaner 1 – five 495 $ft^3$ (14 $m^3$) self-aspirated inert gas cells in a 1-2-2 configuration.
- Cleaner scavenger – four 495 $ft^3$ (14 $m^3$) self-aspirated inert-gas cells in a 2-2 configuration.
- Cleaner 2 – four 495 $ft^3$ (14 $m^3$) self-aspirated inert gas-cells in a 2-2 configuration.
- Cleaner 3 – five 300 $ft^3$ (8.5 $m^3$) self-aspirated inert-gas cells in a 2-3 configuration.
- Cleaner 4 – four 152 $ft^3$ (4.3 $m^3$) self-aspirated inert-gas cells in a 2-2 configuration.
- Cleaner 5 – single Jameson flotation cell.

The layout takes advantage of gravity used from Molybdenum Cleaner 5 through to Molybdenum Cleaner 2 with only the concentrate flows being pumped. Gravity transfer is used from Molybdenum Cleaner 1 to the Molybdenum Cleaner Scavenger and the rougher cells tails.

The molybdenum flotation circuit separates molybdenum and copper minerals as separate concentrates from a bulk Cu-Mo concentrate.

An OSA is used to monitor metal contents and solids concentrations in the feed, final concentrate, cleaner scavenger tailings and rougher tailings streams and allow operators to optimize reagent additions and flotation performance.

### 17.7.1 Molybdenum Scrubbing System

The scrubbing system comprises of a Molybdenum Scrubber and uses caustic soda. The scrubbing system is mounted between the NaHS and plant diesel area and includes redundancy of all components for safety reasons. The scrubber removes gas from all the flotation cells, hopper and tanks via an elevated pipe network, converts gases to NaHS by using caustic soda addition (through intermediate bulk containers “IBCs”) and discharges clean air through the main stack.

## 17.8 Molybdenum Concentrate Thickening, Filtration and Drying

The molybdenum concentrate handling circuit consists of a small concentrate thickener, pressure filter, dryer and concentrate bag loading system. Molybdenum concentrate gravitates from the molybdenum plant OSA to the molybdenum concentrate thickener via a thickener feed box.

The concentrate thickener overflow reports to the tailings thickener. Molybdenum concentrate solids settle for collection at the underflow cone at a density of 60% w/w solids. The thickener underflow stream is pumped to an agitated filter feed tank by peristaltic pumps. A trash screen is located prior to the filter feed tank. The trash screen removes coarse oversize that may damage or block the filter.

High pressure air for the concentrate filter is supplied by a dedicated air compressor. High pressure air for drying is stored in a dedicated air receiver. Filter membrane pressing is supplied by the filter pressing water pump. Molybdenum filter cake is discharged from the filter and directed to a concentrate dryer via a screw feeder. The molybdenum concentrate dryer is a Holoflite dryer with a thermal oil heater and off-gas scrubber. The Holoflite drier reduces concentrate moisture to approximately 5% w/w.

Dried concentrate is stored in a bin ready to be bagged in the bagging station. The storage bin is sized to allow bagging of concentrate on dayshift only. Bagged concentrate is weighed and labelled prior to being loaded onto trucks by fork lift.

## 17.9 Copper Concentrate Dewatering and Storage

### 17.9.1 Copper Concentrate Dewatering

Concentrate thickeners were sized using benchmarked typical unit settling rates for copper concentrates with comparable size distribution and mineral composition. Thickener feed rates were based on maximum design copper head grades at 90,000 tpd. Copper-molybdenum and final copper concentrate thickeners with diameters of 79 feet (24 meters) were selected based on a unit settling rate of 2 $ft^2$/tpd (0.2 tonne/$m^2$/h).

Two pressure filters were selected for the copper concentrate filter duty. The nominated filters are expected to operate with sufficient design margin; however, if additional filter capacity is required, each filter can be expanded from 1,162 $ft^2$ (108 $m^2$) to 1,550 $ft^2$ (144 $m^2$) with the installation of additional plates.

### 17.9.2 Copper Concentrate Storage and Load Out

Copper concentrate filter cake is discharged by gravity to a covered stockpile. A front-end loader (FEL) is used to maximize concentrate storage within the covered building. The covered building provides storage capacity for up to 5,000 tons at average production rates.

Copper concentrate is loaded into containers which are then covered and sealed. These containers sit on top of trucks licensed for use on the highway for transport from the mine site to a storage terminal.

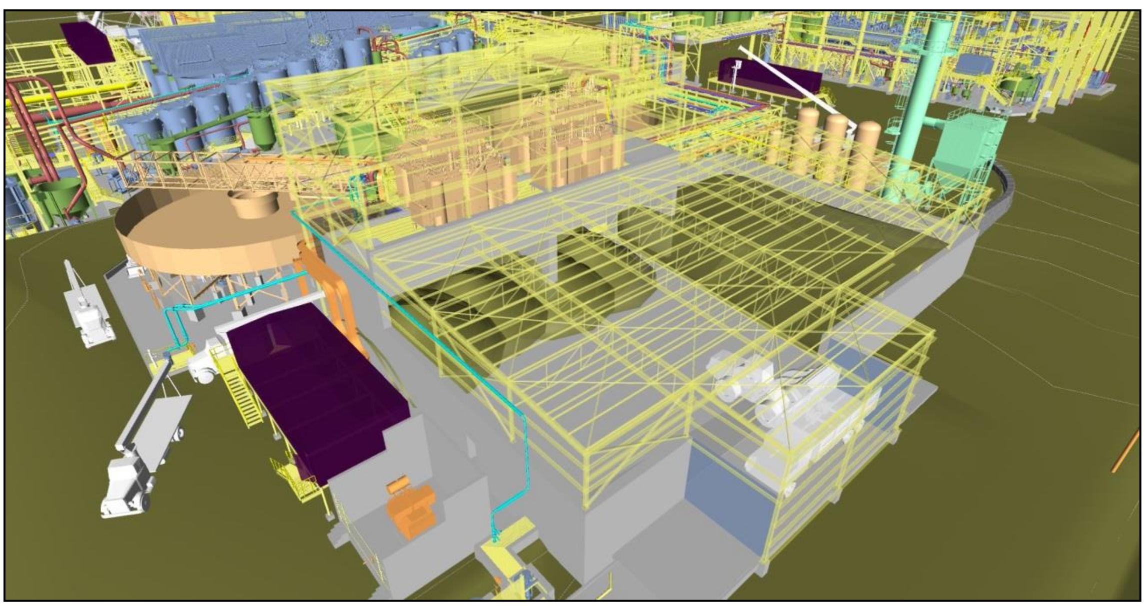
An automated truck wash washes concentrate from the road trucks as they leave the concentrate storage building. Wash water and solids are recovered in a sump and pumped to the copper concentrate thickener.

Road trucks are weighed on a weighbridge located at the main security gate prior to leaving the mine site.

The Copper Concentrate area consists of the Concentrate Filter building and storage shed. Compressed air services are located to the west and the Electrical Rooms are located on the east side.

The Concentrate Filter Building, shown in Figure 17-10**,** houses filters and ancillary equipment such as control panels and mufflers.

**FIGURE 17-10: COPPER CONCENTRATE FILTRATION, STORAGE AND LOAD OUT LOOKING SOUTH**

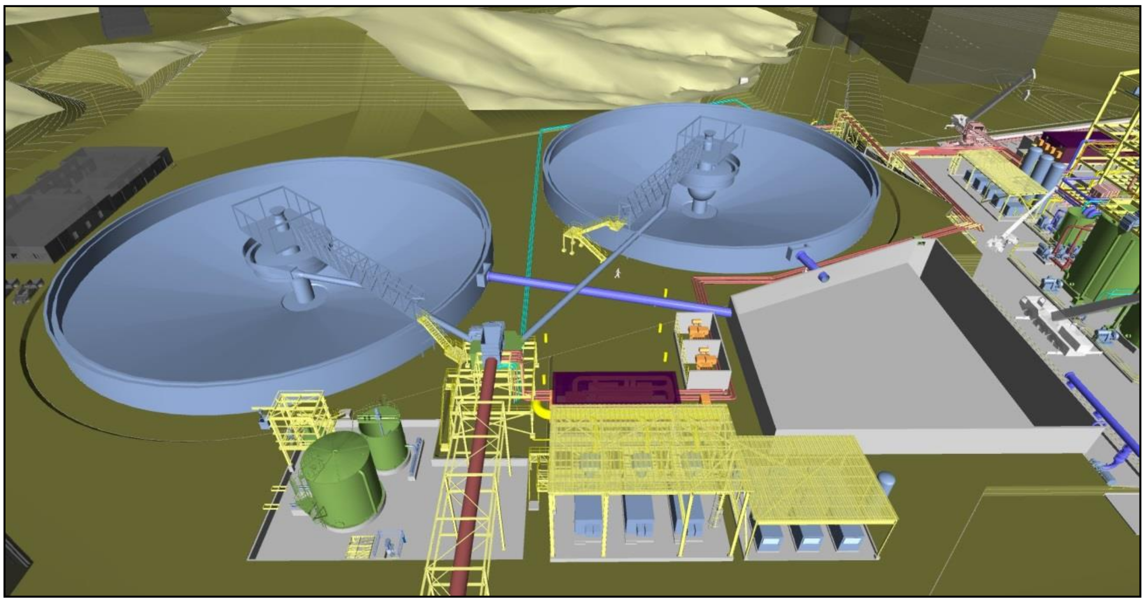


## 17.10 Tailings Thickening

The copper rougher tailings streams are combined with the copper cleaner scavenger tailings and the concentrate thickener overflow streams and gravitate to the tailings thickener feed distributor via a metallurgical cross-cut sampler. Tailings slurry is split into two uniform feed streams and directed to each thickener. Flocculant is added to the thickener feed streams to enhance settling.

The tailings thickening circuit consists of a tailings feed distribution box and two 213 feet (65 meters) diameter high compression thickeners to thicken flotation tailings to 65% w/w solids and recover process water to the process water tank.

**FIGURE 17-11: TAILING THICKENERS LOOKING EAST**

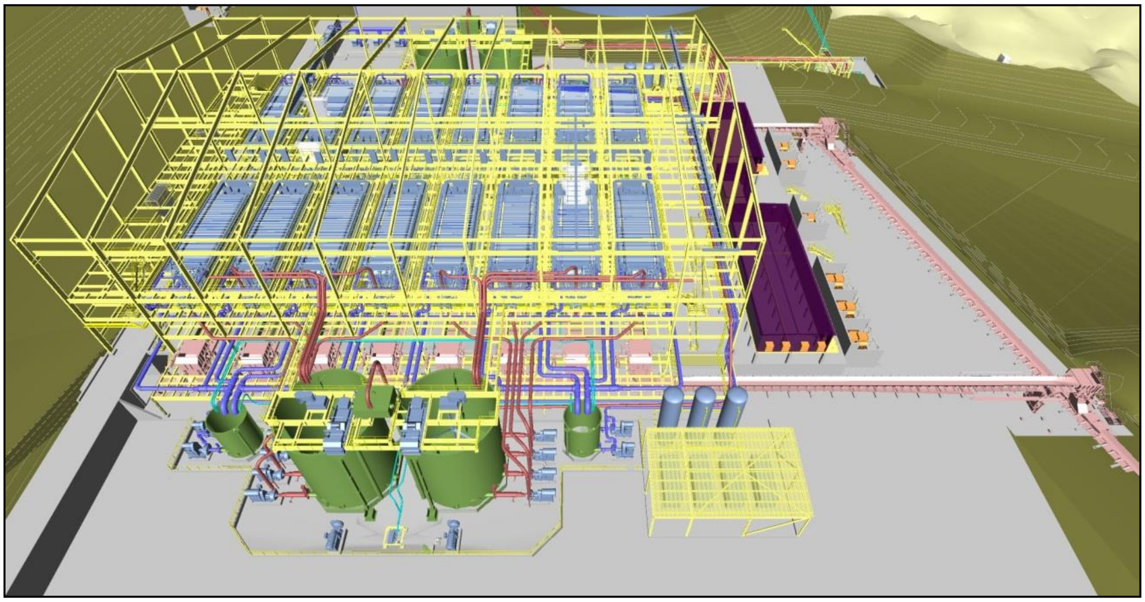


Tailings thickeners were selected as part of an overall tailings dewatering strategy that involved optimising underflow density to maximize downstream filtration rates, and were sized based on settling rate data derived from testwork conducted by FLSmidth, Outotec, Bilfinger, and Pocock Industrial. Optimum underflow density targets a narrow range below the point where the slurry yield stress inhibits handling by centrifugal pumps (without shear thinning systems).

## 17.11 Tailings Filtration Plant

The tailings filter area as shown in Figure 17-12 is located on the east side of the plant. The design consists of two identical parallel trains of filter feed tanks, and utilities such as compressed air and water.

**FIGURE 17-12: TAILINGS FILTER PLANT (ROOF AND WALLS REMOVED)**



Thickened underflow is pumped from the tailings thickeners to two lines of agitated filter feed tanks. Filter feed tank design has a 4.5-hour residence time.

The design allows for a total of 20 filters to be installed in order to process ore requirements over the life of mine. Nevertheless, as a risk mitigation strategy, additional space has been allowed for the installation of 4 additional filters, in the event that they may be required. Filter cake from the filters discharges to a single belt feeder via a set of 'bomb-bay' doors. The belt feeder operates continuously but at a low rate to deliver filter cake to the downstream conveyor continuously over the full cycle time of the filter.

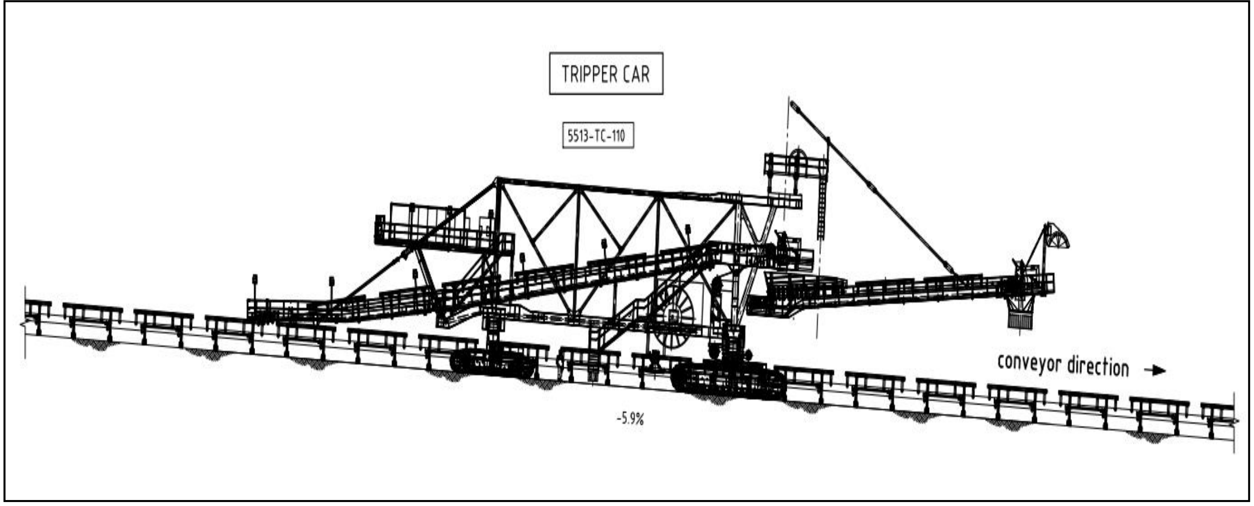
The overland conveyor to the discharge point for the tailings stacking system is 2,171 feet long with 211-foot lift. The drive is installed at the head end, as is the gravity take-up system. The conveyor is driven by two 1200 HP drives mounted one on each side of the drive pulley.

The control room for this area is mounted on the upper deck central to the building on the west end. This bay includes a drive-through for trucks and other mobile equipment. The east end includes a drop down zone for filter components loading onto a truck.

### 17.11.1 Tailings Stacking

Dried tailings from each tailing transfer conveyor is discharged to a single overland tailings conveyor. This conveyor delivers tailings via a bifurcated chute to either the primary or secondary stacking system, which comprise the material handling system for the DSTF.

**FIGURE 17-13: TAILINGS SHIFTABLE CONVEYOR/MOBILE TRIPPER**




The detailed design and selection philosophy for the stacking equipment considers the following drivers:

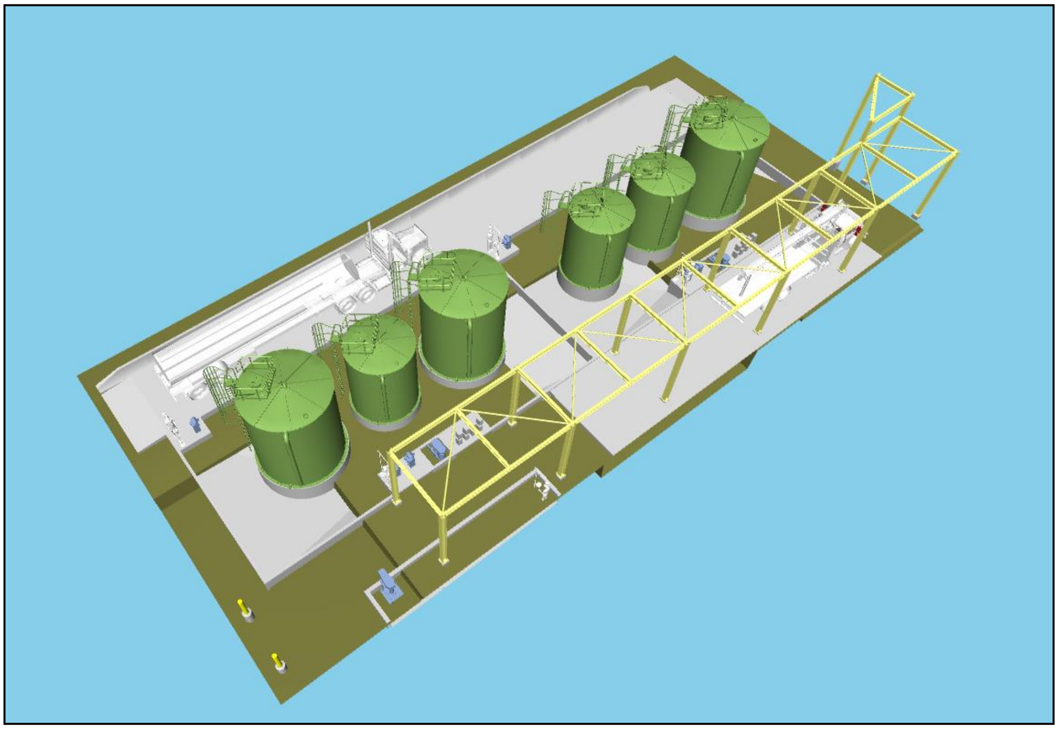
- Minimize stacker downtime for changeovers between lifts and sweeps.
- Maintain stacker progression rates within practical limitations.
- Maximize evaporative drying of the stacked tailings.
- Minimize stacker and conveyor costs.
- Equipment should be mechanically robust to minimize unplanned downtime.
- Stack material in a way which addresses geotechnical considerations, including equipment setback distances
- Permit constraints.

Dry Tailings Stacking Mobile Conveyors operate in series and transport tailings from the Mobile Tripper to the Extendable Mobile Dry Tailings Stacker. During operation, the number of these conveyors required is dependent on the final tailings deposition location relative to the position of the Shiftable Conveyor and Mobile Tripper.

## 17.12 Reagents and Consumables

The main reagent area is located at the west end of the copper flotation area, with the flocculant area being located next to the tailings thickeners. All of the reagents required for the molybdenum plant are handled inside the molybdenum building.

**FIGURE 17-14: REAGENTS AREA**



Major process reagents and consumables are received and stored on site as either dry product or bulk liquids. Where required, dedicated mixing, storage and dosing facilities are provided for each reagent.

- Lime is used to increase slurry pH and subsequently depress minor pyrite in copper flotation.
- Collector (SIBX) - Sodium isobutyl xanthate ("SIBX") is used as the collector in the bulk copper-molybdenum flotation circuit.
- Promoter-Cytec AP3894 promoter is added as a secondary collector in the bulk copper-molybdenum flotation circuit.
- Frother-Methyl isobutyl carbinol ("MIBC") is used to provide a stable froth in the copper flotation cells.
- Diesel fuel oil is used as a collector/promoter to assist flotation of molybdenum minerals in the copper and molybdenum flotation circuits.
- Sodium hydrosulfide ("NaHS") is used in the molybdenum flotation circuit as a depressant for non-molybdenum minerals.
- Sodium silicate can be used in the molybdenum flotation circuit as a dispersant for fine non-sulfide gangue and clay-bearing minerals.
- Flocculant is used as a settling aid in the concentrate and flotation tailings thickeners.
- Carbon dioxide is used to control pH to 9.0-10.0 in the molybdenum rougher and cleaner conditioning tanks.

- Nitrogen is used in the molybdenum flotation cells to minimize oxygen entrainment and maximize depression of copper. A small pressure swing adsorption (“PSA”) plant supplies nitrogen to the molybdenum flotation circuit.
- Additional miscellaneous reagents are required in the plant; however, these are expected to be used in relatively small quantities.

## 17.13 Plant Services

### 17.13.1 Process Water Services

Fresh water is sourced from wells located on the western side of the Santa Rita Mountains and is pumped through a series of booster tanks and pumps to the fresh water tank located above the plant site.

- Fire water.
- Potable water including a water treatment plant.
- Gland seal water.
- Cooling water including chillers and closed loop cooling system.

Process water for ‘general use’ is sourced from the tailings thickener overflow (including concentrate thickener overflow and tailings filter filtrate water) and the fresh water tank as required. Supplementary water sources used for process water make-up include:

- Plant site run-off collected in the primary settling basin.
- Crusher stormwater pond.
- Mine pit sump and pit storm water pond.
- Mine ground water dewatering wells.

Process water is stored in the process water pond. Process water pond pumps transfer water from the storage pond to the process water tank. Excess water in the process water tank overflows back to the process water pond.

The tailings thickener overflow streams report directly to the process water tank for immediate distribution and use. Process water pumps distribute process water to the grinding mills, copper flotation, regrind circuits and lime slaking plant. Cloth wash water pumps distribute process water to the tailings filters for automated cloth washing and manifold flushing.

### 17.13.2 Air Services

Three separate plant air compressors provide air service throughout the plant. Due to its remote location, the primary crusher is serviced by a dedicated air compressor with an air dryer and filter system.

## 17.14 Process Control Strategy

### 17.14.1 Process Plant PCS

The process control system ("PCS") is an integrated plant-wide design, enabling the start-up, monitoring and control and shutdown of equipment from the plant control rooms.

The process plant is monitored and controlled from three separate control rooms:

- Crusher control room.
- Main control room (located in the flotation area).
- Tailings control room (located in the tailings filtration building).

Operators can control the plant via PC-based human machine interface ("HMI") stations. Each HMI station provides dynamic graphical representation of the plant operation; equipment control functions; alarm displays; event logging; trending; data collection and reporting to assist in analysis of plant operations.

Where specific equipment forms part of an approved vendor package and drives are controlled from a vendor control panel, a communications interface is used to enable remote control and monitoring from the PCS. This includes digital and analogue signals for alarms; faults; instrumentation and monitoring; motor and valve control; process variables and interlock controls.

The crusher control room contains a single HMI station with two monitors. The HMI station provides dedicated control of the crushing plant area.

The main control room contains three HMI stations, each with two monitors. The main control room provides dedicated control of the main plant areas. Control of the crusher and tailings areas is also possible from the main control room. An engineering development workstation is also located in the main control room building.

The tailings control room contains one HMI station with two monitors. The HMI station provides dedicated control of the tailings filtration and dry stack areas.

### 17.14.2 On-Stream Analysers

Plant instrumentation includes OSA that are used to continuously monitor copper, molybdenum, iron and density in key process streams and assist with optimizing concentrate grade and recovery.

Dedicated OSA systems are provided in the copper and molybdenum flotation circuits. Each OSA unit is centrally located in the respective plant and elevated to allow gravity discharge of samples to sample return pumps. Each analyzer has two 6-channel multiplexers. Sub-samples for shift composites are collected automatically.

A single PSA is installed to continuously measure the particle size of the copper regrind cyclone overflow.

#### 17.14.3 Closed Circuit Television (CCTV) Systems

A closed-circuit television (“CCTV”) system is used to assist control room operators in monitoring the operation of plant and equipment.

The CCTV system provides real-time monitoring with archived recording for a nominal period. Camera types include fixed cameras and cameras with remote pan-tilt and/or zoom functions accessible by the control room operators.

# 18 PROJECT INFRASTRUCTURE

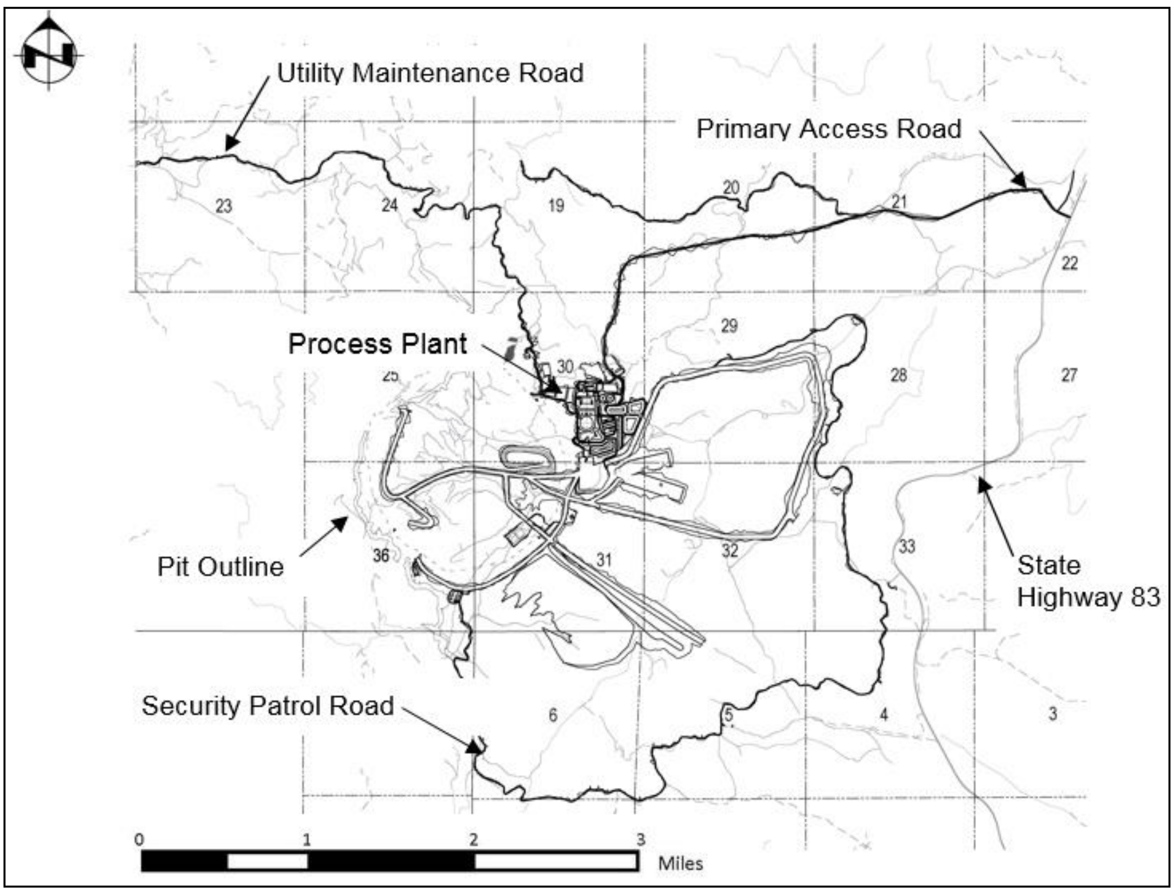
This section addresses the infrastructure facilities that will support the Rosemont mine and processing facilities. The infrastructure facilities include the access roads into the plant site, source of electrical power and power distribution, source of fresh water and water distribution, DSTF, WRSA, transportation and shipping, communications, and mobile equipment.

This Technical Report includes refinements of certain aspects of the Project's mine plan. While consistency with issued and pending environmental permits and analysis related thereto has always been a key requirement for this effort, updates to the original mine plan will be necessary. To the extent that any regulatory agency concludes that the current plan requires additional environmental analysis or modification of an existing permit, the intent will be to work with that agency to either complete the required process or to adjust the current mine plan as necessary.

## 18.1 Access Roads, Plant Roads and Haul Roads

Access and plant roads consist of an access road into the plant from State Highway 83, in-plant roads, haul roads and a security patrol road around the toe of the WRSA and DSTF. The plant and access roads are shown in Figure 18-1.

**FIGURE 18-1: PLANT AND ACCESS ROAD**



The access road to the property starts at State Highway 83 at a point between mile markers 46 and 47 and ends at the main guard house at the entrance to the plant. The intersection of the access road with State Highway 83 will be modified to provide safe ingress and egress from the access road in compliance with ADOT and AASHTO standards. Modifications will include a northbound acceleration lane, northbound left turn lane and a southbound right turn lane.

In-plant roads extend from the plant entrance both through and around the perimeter of the process facilities. Secondary roads, such as the utility maintenance road, leave this perimeter road to serve the main substation, water storage tank, and access the utility corridor. As per the State of Arizona Air Quality Control Permit and the EIS analysis, specific in-plant roads will be paved to reduce dust emissions.

Haul roads used for access to and construction of perimeter waste rock buttresses shall be a minimum of 150 feet wide to allow trucks to turn around with the roadway surface.

A security patrol road will be provided around the toe of the WRSA and the DSTF along the security fence line for security to monitor the plant boundaries and provide maintenance access to the WRSA and DSTF.

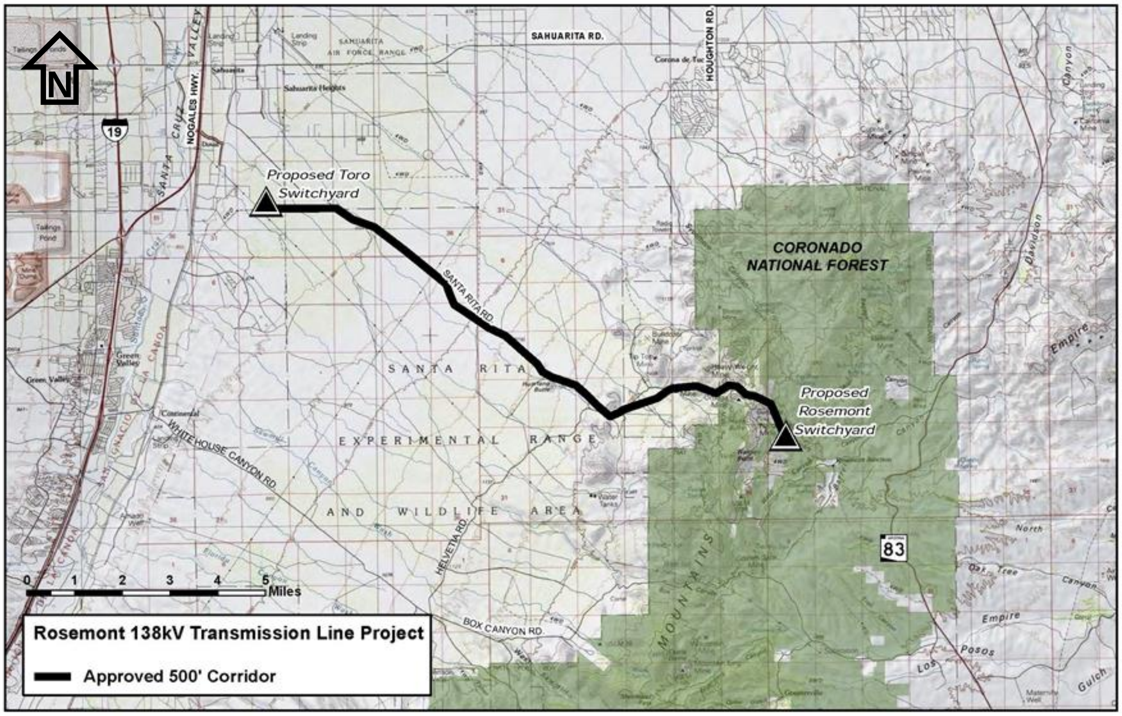
## 18.2 Power Supply and Distribution

Pursuant to the Certificate of Environmental Compatibility (“CEC”) issued by the Arizona Corporation Commission (ACC) on June 12, 2012, TEP will provide the electrical power supply for the Rosemont mine and process facilities. The total connected load for the Rosemont mine and process facilities is estimated to be approximately 183 MVA and will require a transmission voltage of 138 kV.

The proposed Toro Switchyard, located approximately 3 miles south of Sahuarita Road and 3.5 miles east of I-19 near the Country Club Road and Corto Road alignments will tap into the existing 138kV transmission line that extends from the South Substation to the Green Valley Substation. The transmission line follows a 13.2-mile-long route originating at the Toro Switchyard and terminating on Rosemont private property at the Rosemont Switchyard, Figure 18-2.

Distribution power tapping from the Rosemont Switchyard to the substation will provide power to the process plant and the mine.

**FIGURE 18-2: CEC APPROVED UTILITY CORRIDOR FOR 138KV TRANSMISSION LINE**



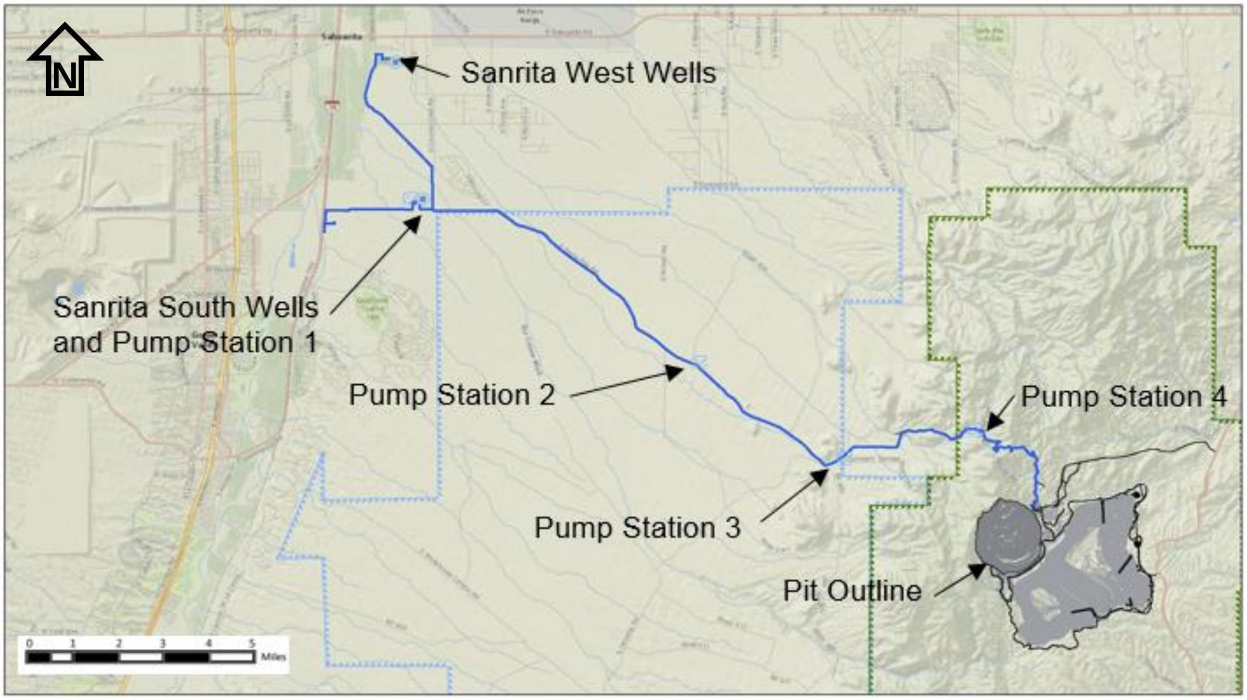

## 18.3 Water Supply and Distribution

The fresh water design requirement for the Rosemont facilities is 3,500 gallons per minute (“gpm”) and peak of 5,000 gpm. The delivery requirements are based on the draft overall site water balance developed by Ausenco which takes into consideration dust control, process make-up water, process fresh water requirements, and potable water. The source of water supply identified for the Project is groundwater in the basin-fill deposits of the upper Santa Cruz basin, which lies west of the Project and the Santa Rita Mountains. The Project has a permit to withdraw groundwater for Mineral Extraction and Metallurgical Processing in the amount of 6,000 acre-feet per year for 20 years.

Rosemont Copper has acquired a 53-acre land parcel near Santa Rita and Davis Roads (Sanrita West), and a 20-acre parcel near Santa Rita Road and Country Club Drive (Sanrita South, or Station No. 1 site), for the purpose of constructing and operating a production well field for the Rosemont water supply.

The wells will deliver water to pump station no. 1 located at Sanrita South. There are three (3) other pump stations located strategically along the alignment of the water pipeline to pump the necessary water to the storage tank located at the mine site, Figure 18-3.

**FIGURE 18-3: UTILITY CORRIDOR FOR WATER LINE**




The pipeline will discharge to the Rosemont fresh/fire water tank, which serves to provide storage and reserve for the operations. The lower portion of the tank, with an approximate capacity of 300,000 gallons, will be reserved for the fire water system. Flow of fire water and fresh water is provided by gravity.

Water will be provided to a potable water system, fresh water system, process water system, and fire water system.

The potable water system consists of a potable water treatment package, potable water tank and a distribution network delivering potable water by gravity to all ancillary buildings, process facilities, restrooms, and safety showers. The fresh water system consists of the gravity distribution network from the fresh water storage tank to the process facilities requiring fresh water. The fresh water usage is for gland water pump seals, fresh water make-up to the mills, flotation plant make-up, and reagent make-up. The process water system consists of a process water pond that collects process water from the concentrate and tailings de-watering equipment for recycling back into the circuit. The fire water system consists of a gravity distribution network from the fresh water / fire water storage tank to a system of hydrants around the ancillary buildings and process facilities.

Rosemont has voluntarily committed to recharging 105% of the groundwater used during operations. Thus far, 45,000 acre-feet of water resulting in nearly 42,600 acre-feet of storage credits have been recharged back into the Tucson Active Management Area, the area of planned withdrawal. Additionally, in an effort to reduce water usage, Rosemont is committed to use dry stack tailings instead of conventional tailings. The tailings dewatering system is expected to recycle about 15,000,000 gallons of water per day.

## 18.4 Tailings Management

The Rosemont DSTF has been designed to receive dewatered tailings from the processing plant at a nominal rate of 90,000 dry tpd. This material will be stacked behind large containment buttresses constructed from pit run waste rock.

The deposition of dewatered tailings, waste rock and overburden will be initiated with a series of perimeter buttresses and berms. The staging of these buttresses will also allow reclamation to begin early in the operation. Soil will be salvaged from pit and WRSA and DSTF for use as a vegetation growth medium. The dewatered tailings deposition will incorporate staged waste rock buttresses for visual screening and to improve mechanical and erosional stability of the tailings.

### 18.4.1 DSTF Location and Design

Design criteria and objectives for the original dry stacked tailings facility included:

- Provision of secure long-term storage of a minimum 500 million tons (Mt), which is sufficient for the ore to be mined and processed during the Project life;
- Location within the immediate general area of the mine (approximately five-mile radius from the proposed mine pit);
- Prevention of airborne release of tailings solids to the environment by provision of dust suppression measures;
- Compliance with all applicable regulations including Arizona Best Available Demonstrated Control Technology (“BADCT”) standards;
- Creation of a site-specific design that accounts for local factors including climate, geology, hydrogeology, seismicity and vegetation; and
- Establishment of an effective and efficient reclamation program, with a focus on concurrent reclamation.

Advantages of the dry stacked tailings over a conventional tailings impoundment is that it eliminates the need for an engineered embankment and seepage containment system, maximizes water conservation and minimizes water makeup requirements, results in a very compact site limiting disturbance to a single drainage, and allows opportunities for concurrent reclamation and provisions for dust control.

The selected site is located just east of the proposed mill site in Barrel Canyon. The DSTF site is characterized by terrain sloping generally east from the plant area to the Barrel Canyon, which generally runs north south in this area.

The design was developed based on hydrological and geotechnical studies that included review of regional climate data, drilling and testing programs, and laboratory characterization of subsurface and tailings samples.

### 18.4.2 DSTF Stability

The DSTF is designed as a low hazard facility with waste rock placed as buttressing material. The filtered tailings lifts will densify under successive controlled conveyor lift placement and will result in an increase in the lower lift fill strength over time. The DSTF stability analyses considered the maximum ultimate height at the maximum section through the facility for downstream and upstream stability.

The tailings will be placed in a dewatered state for acceptable handleability during conveyance and trafficability of the tailing surface, which will limit susceptibility to liquefaction under dynamic loading. However, limited higher moisture zones within the tailings mass created by meteoric water may occur. This condition was considered in the stability modelling by applying reduced shear strength to thin layers within the tailings mass at various levels to simulate these higher moisture zones and to evaluate the subsequent earthquake resistance of the facility.

Thus, adequate factors of safety for static and pseudo-static were obtained from the stability analyses based on the selected parameters and proposed facility. The use of dry stack tailings as oppose to conventional tailing impoundments eliminates the danger of dam failure typically seen with tailings ponds.

The slope stability analyses performed on the outer slope indicate the dry stack tailings operations can be constructed with stable 3H:1V inter-bench slopes and an overall stable slope of approximately 3.5H:1V.

### 18.4.3 Hydrologic Modeling

Modelling calculations performed for the EIS indicate that infiltration of rain into the DSTF did not develop. The resistance to infiltration is a function of the fine-grained nature of the crushed and ground tailings material and the compaction that will occur with placement and facility construction.

Much of the DSTF will ultimately lie above the ultimate groundwater capture zone predicted by the groundwater models. Within this zone, any seepage that may occur would ultimately flow via groundwater to the open pit. The portions of the facility not included in this capture zone will generate seepage and is permitted under the Aquifer Protection Permit program in the state of Arizona. Water entrained in the DSTF that comprises drain-down has been chemically analyzed and modeled and is not expected to exceed AWQS at compliance locations, located around the perimeter of the facilities. The ADEQ evaluated the potential for seepage and issued permits based on their analysis. Modelling and analytical results indicate that seepage constituent concentrations will be below the AWQS for regulated constituents. This modeling is supported by 174 geochemical samples of waste rock evaluated during the EIS process as well as ten tailings samples specifically reviewed by the Arizona Department of Environmental Quality during the Aquifer Protection Permit process. Samples were subject to combinations of testing to determine their acid generating potential, whole rock analysis, synthetic precipitation leach procedure testing, meteoric water mobility testing, and humidity cell testwork. The tests performed on the waste rock showed a net neutralization potential of 225. This is considered highly buffering (zero or less being acid generating) ensuring that the buttress materials will not generate acid rock drainage (ARD) The tailings testing resulted in analysis of seepage that supported the ADEQ determination.

Later work continues to support this finding and is consistent with prior testwork. Mineralogical analysis of 107 drill core composite samples (Section 7.6) indicating that Rosemont ore will contain less than 3% by weight sulfide (potentially acid generating) minerals, most of which will be recovered as valuable concentrates in the process. The analysis also indicates that the ore will contain approximately 20% carbonates (calcite, dolomite) which are alkaline and will serve to neutralize any acidic species that could be generated by decomposition residual sulfide minerals in the process plant tailings. Pit-run waste rock will consist largely of limestone and skarn rock types, with some andesite, quartz monzonite porphyry, and arkose. The presence of substantial quantities of limestone and skarn (97%) along with low-sulfide content, supports the analysis in the EIS that determined there is a large buffering capacity within the buttress materials which will minimize the potential generation of ARD.

#### 18.4.4 Surface Water Control

Once the perimeter buttresses/berms are placed across the drainages and washes, stormwater run-on will be limited by ponding stormwater upstream of the dry stack areas. Stormwater runoff sediments from the waste rock buttresses will be captured in sediment basins located downstream of the tailings facility. During operations, the tailings surface will be sloped away from the waste rock buttresses to limit potential water impoundment against the buttresses. Perimeter ditches will be constructed at the upgradient outer edges of the tailings surface to retain and evaporate water.

#### 18.4.5 DSTF Operations

Dewatered tailings will be delivered by conveyor and placed with a radial stacker. A dozer might be used to spread the dry tailings to provide a suitable surface for the conveyor and stacker as needed.

An initial starter buttress will be constructed with waste rock. Concurrent tailings placement and buttress construction using waste rock placement will occur throughout the life of the tailings facility. Waste rock will be advanced ahead of the tailings level in successive lifts using the upstream construction method. The waste rock buttresses will accommodate haul traffic and outer slopes generally of 3H:1V with benches to achieve an overall sloped facility of 3.5H:1V.

## 18.5 Communications

There are requirements for accounting, purchasing, maintenance, and general office business as well as specialized requirements for control systems.

The two most common options are to design separate data networking and telecommunication systems or to integrate the two into a common infrastructure. For this Project, the proposed approach is integration.

A voice over I/P (VoIP) phone system will be a part of the office network and VoIP handsets will be used for voice communication.

The office ethernet network will support accounting, payroll, maintenance, and other servers as well as individual user computers. High bandwidth routers and switches will be used to logically segment the system and to provide the ability to monitor and control traffic over the network.

The control system ethernet network will support the screen, historian, and alarm servers and connect to the Control Room computers as well as the Programmable Logic Controllers and other control systems provided with ethernet communication capabilities. This system will incorporate redundancy and will be designed to minimize traffic and latencies. No phone or user computer will be connected to this system.

A security system will also be incorporated into the plant network. Using a dedicated video server and monitors, I/P cameras utilizing power over ethernet connections will be plugged into dedicated switches. Security cameras are typically located in storerooms, parking lots, visitor lobbies, warehouses, and areas where sensitive materials are kept.

Mobile radios will also be used by the mine and plant operation personnel for daily control and communications while outside the offices.

# 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. The Project's copper concentrates are expected to be a clean, high grade concentrate containing small gold and silver by-product credits which will be suitable as a feedstock for smelters globally. Approximately 50% of the copper concentrate production has been contracted under long term sales contracts.

Table 19-1 below summarizes the key assumptions for the sale of Rosemont's copper concentrate.

**TABLE 19-1: COPPER CONCENTRATE**

| | Units | LOM Total / Average |
|---|---|---|
| Copper Concentrate Base Treatment Charge | $ / dry short ton con | $73 |
| Copper Refining Charge | $ / lb Cu | $0.08 |
| Silver Refining Charge | $ / oz Ag | $0.50 |
| | | |
| Copper Concentrate Transport & Freight | $ / wet short ton con | $127 |
| LOM Copper Grade in Copper Concentrate | % Total Cu | 34.3% |
| Moisture Content of Copper Concentrate | % $H_2O$ | 8.0% |

No deleterious elements are expected to be produced in quantities which would result in material selling penalties.

A precious metals stream agreement with Silver Wheaton Corporation for 100% of payable gold and silver from the Project was entered into on February 11, 2010. Under the agreement, Hudbay will receive payments equal to the lesser of the market price and $450 per ounce for gold and $3.90 per ounce for silver, subject to 1% annual escalation after three years.

Rosemont is expected to produce a marketable 45% molybdenum concentrate. Table 19-2 summarizes the key assumptions for the sale of Rosemont's molybdenum concentrate.

**TABLE 19-2: MOLYBDENUM CONCENTRATE**

| | Units | LOM Total / Average |
|---|---|---|
| Molybdenum Concentrate Base Treatment Charge | $ / lb Mo | $1.50 |
| Molybdenum Concentrate Transport & Freight | $ / wet short ton con | $124 |
| LOM Molybdenum Grade in Molybdenum Concentrate | % Mo | 45.0% |
| Moisture Content of Molybdenum Concentrate | % $H_2O$ | 8.0% |

# 20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Project permitting status is well advanced and continues to progress since July 2007. The final approvals required include the Final Record of Decision (“ROD”) from the U.S. Forest Service (“USFS”) and the 404 Permit from the U.S. Army Corps of Engineers (“USACE”). There have been over 450 days of public comment associated with this Project that have culminated in over 43,500 comments. All comments have been reviewed, categorized, and either incorporated or answered by the various State and Federal agencies.

Since issuing the Final Environmental Impact Statement (“FEIS”), the USFS has issued two Supplemental Information Reports (“SIR”) and a Supplemental Biological Assessment (“SBA”), both of which were required after the Draft ROD was issued in December 2013. The SIRs considered whether new information or changed circumstances remained within the scope of the effects disclosed in the FEIS. The second SIR was issued to summarize on-going analysis and reporting completed since the first SIR was produced in 2015. Nothing disclosed to date would indicate that the information in either SIR falls outside the effects considered in the FEIS, which would require the FEIS to be supplemented.

The SBA evaluated the sighting of a jaguar and ocelot and resulted in the reinitiation of consultation with the U.S. Fish and Wildlife Service (“USFWS”). This consultation process included species such as the ocelot, the yellow-billed cuckoo, the Mexican garter snake, and other aquatic species. The reinitiated consultation was completed in April 2016 and culminated in an Amended Final Reinitiated Biological and Conference Opinion (“BO”) similar to the one produced in 2013. Both BO’s stated that the Project would not jeopardize the existence of any endangered species. The next step will be to issue a Final ROD. The Final ROD will trigger a requirement for an updated operating plan ( MPO) that will include measures to be taken so the Project will meet the requirements of the ROD, including measures to mitigate adverse environmental impacts.

The USACE Division Offices are evaluating the 404 permit application, the record, and the mitigation package that will go into making their permit decision. The mitigation package is designed to mitigate impacts to a total of 68.8 acres, which consists of 40.4 acres of ephemeral channels on the Project site plus the 28.4 acres of off-site indirect impacts. This mitigation incorporates the restoration of a floodplain that was impacted by agriculture; mitigation for two sites impacts by grazing, poor roadway maintenance, and other activities; as well as preservation of sites near to the Project site. Once the USACE evaluation is complete, and a positive permit decision is made, the terms and conditions of the permit and appropriate financial assurance will be negotiated.

At this time, State of Arizona environmental permits and approvals have been issued for the Project, and these permits remain in force and are current. Two of the permits (air and groundwater protection) will need to be amended to match the applicable Federal permits. The Project continues to comply with these current State permit terms and conditions.

As stated above, the Project details included in this document were specifically designed and evaluated to fall within the permit constraints included in the EIS and State of Arizona permits with amendments. This Technical Report includes refinements of certain aspects of the Project's mine plan. While consistency with issued and pending environmental permits and analysis related thereto has always been a key requirement for this effort, updates to the original mine plan will be necessary. To the extent that any regulatory agency concludes that the current plan requires additional environmental analysis or modification of an existing permit, the intent will be to work with that agency to either complete the required process or to adjust the current mine plan as necessary.

Certain permits issued for the site have specific design and monitoring requirements built into the permits. In particular, the Project meets (and in some design elements exceeds) the Arizona Department of Environmental Quality (ADEQ) Best Available Demonstrate Control Technology (BADCT) requirements. BADCT covers specific requirements for any "discharging facility" and includes items such as specific liner requirements for ponds, design requirements for WRSA and DSTF including seismic design requirements as well as geochemical characterization requirements for possible discharges. In addition, the aquifer protection permit (APP) issued by ADEQ has specific groundwater monitoring requirements that requires quarterly monitoring for various parameters in specific wells.

The USFS has incorporated the permit requirements required by ADEQ, as well as other agencies, into their Mitigation Measures listed in the FEIS (Appendix B of the FEIS). FEIS mitigation measures also cover mitigation measures requirements to cover areas of interest/concern to the USFS. These requirements will be incorporated into the Final ROD and into the updated MPO. Mitigation Measures in the FEIS include categories such as:

- Geology, Minerals, and Paleontology
- Soils and Revegetation
- Groundwater Quality and Quantity
- Surface Water Quantity and Quality
- Seeps, Springs, and Riparian
- Geology, Minerals, and Paleontology
- Biological Resources
- Landownership and Boundary Management
- Dark Skies
- Visual Resources
- Recreation and Wilderness
- Hazardous Materials
- Transportation/Access
- Noise
- Public Health and Safety
- Cultural Resources
- Air Quality and Climate Change
- Fire and Fuels

- Power Use
- Community Programs

Certain permits issued require financial assurance to ensure the success of mitigation, while others are solely to ensure that adequate funds are available at closure. The requisite bonds for the Project are expected to be obtained from the surety market with an estimated annual bond fee of 2% of the bond's notional value.

Currently the Project has bonds in place to cover $40,000 in surface reclamation costs for the Arizona State Mine Inspector (AMSI), and $4,300,000 for the ADEQ APP permit specifically for closure of discharging facilities. It is expected that the bond for ADEQ will be reduced once the permit amendment is completed. Additionally, the reclamation bond with ASMI is expected to be incorporated into the USFS bonding.

In addition, a USFS bond will be required to cover reclamation and closure costs and will be updated in 3-year increments throughout the life of the Project. The required bonding for the initial 3-year period will be negotiated with the USFS during their review of the MPO. Hudbay has estimated and included fees for a $65 million USFS bond through construction and operations with some curtailment in the final four years of operations as certain reclamation activities are completed. Although the final amount of the USFS bond remains to be negotiated, the notional bond value is not expected to differ materially from this estimate.

The USACE bond will be a performance bond with a long-term management component to ensure the mitigation proposed is successful. The overall USACE bonding has been estimated at approximately $50 million; however, this bond package will need to be negotiated and the estimate includes items that may be eliminated in the final negotiation. Initial bond amounts have been estimated at $35 million with $15 million long-term bonding added during the first ten years of operation. The USACE bond is not expected to be required after the tenth year of operations.

It is also expected that the bond for ADEQ will be reduced once the permit amendment is completed. Additionally, the reclamation bond with ASMI, currently at $40,000, is expected to increase to $4 million once mine construction commences. However, Hudbay expects the requirements related to this bond to be covered by the USFS bonding pending future negotiation with ASMI and USFS. At this time, Hudbay has assumed that a $4M ASMI bond will stay in place during the construction and operation of the mine.

With regard to community outreach and other social commitments, the following provides a summary of costs associated with those items:

- $650,000 for the relocation of the Arizona Trail;
- $6,500,000 to $7,500,000 to repair the pavement, install bus pull outs, and replace guard rails on a 12-mile stretch of State Route 83, the main roadway connecting I-10 to the entrance road into the Project;

- $25 million endowment fund for conservation purposes. The development of this endowment trust will fund priority community projects including recreation, cultural, and environmental conservation projects. Currently proposals under consideration by the Forest Service set annual funding of up to $1.5 million depending on annual production and metal prices and is subject to a cap of $25 million. By agreement, 10% of this endowment is available to the Mescalero Apache tribe for grants for specific cultural and educational projects.
- $500,000 in annual community donations.

In addition, there are specific mitigation and data recovery obligations related to archaeological (cultural) sites associated with the Project. These specific requirements are a culmination of negotiations between the USFS and the Tribes with input from various state agencies, other co-operators, and Hudbay.

Details on permit status and authorizations for current project activities are included in Appendix A3-1.

## 20.1 Reclamation and Closure Plan

The Reclamation and Closure Plan is based on several key components, referred herein as initiatives. These initiatives provide the physical and philosophical foundation that will remain constant throughout the operation of the facility. As related to this Plan, some of these initiatives include:

- Beginning with the end in mind. The placement of materials in the various storage areas is based on the final closure configuration. For example, the overall out-slopes of the Landform will generally be 3.5H:1V (H: Horizontal, V: Vertical) with inter-bench slopes at 3.0H:1V. The placement of waste rock that comprises the outer shell of the Landform, i.e., the outer slopes of the waste rock buttresses/berms, will incorporate setbacks to facilitate efficient regrading to achieve the final design slopes.
- Constructing outer facility berms/buttresses. During the initial years of mine development, waste rock from stripping operations will be preferentially placed along portions of the outer footprint of the WRSA (berms) and the DSTF (buttresses). In addition to defining the outer footprint, placement of these outer berms/buttresses will be used to help screen active mining operations from vantage points in the area, such as from SR 83, and will serve to isolate mining activities and protect down-gradient surface water quality. Regarding the DSTF area, an outer waste rock buttress will serve to stabilize the dry tailings stacks and prevent the erosion of tailings into the down-gradient drainages.

- Concurrently reclaiming the outer surfaces of the WRSA and DSTF. Reclamation of the Landform will not be deferred to the end of the Project. Concurrent reclamation, as practicable, is planned for the outer shell encompassing the WRSA and DSTF. Growth media (soil) and woody debris will be salvaged from the facility footprints and placed directly on reclaimed areas or stockpiled for future use. Reclamation of the outer shell of the Landform also includes the construction of stormwater management features, such as channels and drop structures, in order to divert as much stormwater down-gradient as practicable. The final reclaimed surfaces will have a soil (growth media) cover or a combination of rock/soil cover and will be revegetated using a seed mixture appropriate for the Project area.
- Using modern technology to minimize the generation of impacted water. The Rosemont operation will include milling operations for sulfide ores. Conventional slurry line and settling pond technology for tailings disposal will not be used at Rosemont. Tailings will instead be mechanically filtered to 18% (or less) moisture content by weight and stacked behind a waste rock buttress. The formation and migration of seepage from this tailings disposal system is negligible. The dry stack tailings disposal method also facilitates the ability to concurrently reclaim the facility as described above.

To the maximum extent practicable, the final landform is graded to route as much stormwater runoff off the reclaimed surface and into the down-gradient flow system. Bench channels and drop chutes are constructed on the surface to direct stormwater down-gradient toward lower Barrel Canyon drainage. Building facilities within the Plant Site are removed and the area regraded. The Plant Site area will be also regraded with the intent to route as much stormwater down-gradient as practicable – in this case to the McCleary Canyon drainage which feeds the lower Barrel Canyon drainage. Reclaimed areas are covered with growth media (soil salvaged from the facility footprints) and revegetated.

The reclamation and closure of the Utility Corridor includes the removal of facilities (such as the water and power lines and pump stations) and the regrading and revegetation of disturbed areas.

# 21 CAPITAL AND OPERATING COSTS

## 21.1 Introduction

Capital costs are estimated in constant 2016 US dollars.

## 21.2 Capital Costs

The total initial capital required to construct the processing plant, purchase mining equipment and pre-strip the pit is estimated to be $1,921 million including 15% contingency on all items as shown in Table 21-1.

**TABLE 21-1: INITIAL CAPITAL COST SUMMARY**

| **Initial Capital** | **000 US $** |
|---|---|
| Site Wide | 42,433 |
| Mining | 474,070 |
| Process Plant | 670,525 |
| Site Services and Utilities | 21,802 |
| Internal Infrastructure | 127,300 |
| External Infrastructure | 113,954 |
| Common Construction Facilities | 50,914 |
| EPCM Services | 107,009 |
| Owner’s Cost | 312,895 |
| **Total Initial Capital** | **1,920,903** |

The initial capital investment represents the total project cost; including facility costs, infrastructure costs and Owner’s Costs. The estimate is based on inputs from various organizations as follows:

- Ausenco: Process Plant
- Ausenco and M3: Plant Infrastructure
- Ausenco and M3: DSTF
- Knight Piésold: Engineering quantities relating to Heavy Civil Works (“HCW”): geotechnical, haul roads, buttresses, WRSA and diversion channels
- Stantec: Water Infrastructure
- Tucson Electric Power: Power Infrastructure
- Hudbay: Owner’s Costs

The estimate was produced using Prism Project Estimator based on a bottom-up approach. The overall capital cost estimate meets the Association for Advancement of Cost Engineers (“AACE”) Class 3 requirement of an accuracy range between -10% and +20% of the final project cost (excluding contingency). It has a base date of end of March 2016 with no allowance for escalation. This assessment is based on:

- Project maturity 10 – 40%
- Engineering progress on the processing plant was approximately 38% complete

- Engineering progress on the off-site infrastructure was approximately 50% complete

Process plant costs were estimated by Ausenco with input from various consulting firms including Knight Piésold. Construction labor rates and productivities were developed on a discipline by discipline basis with input from major industrial contractors in the Southwest U.S. Labor rates, supply rates and productivities were benchmarked against projects of similar size and scope.

External infrastructure for the water supply system was designed and estimated by Stantec. External infrastructure for the power supply was designed and estimated by costs were provided by Tucson Electric Power.

Direct cost estimate quantities were derived from engineering lists, material take-offs, consultant databases (previous projects) and vendor input as shown in the table below:

<table>
<tr><th>Description</th><th>WBS</th><th>Earthworks</th><th>Concrete</th><th>Structural</th><th>Platework</th><th>Piping</th><th>E & I</th></tr>
<tr><td>Geology and Mine Design /Mining (infrastructure only)</td><td>1000+2000</td><td>Preliminary Design</td><td>Preliminary Design</td><td>Preliminary Design</td><td>Preliminary Design</td><td>Factor</td><td>Factor</td></tr>
<tr><td>Process Plant</td><td>3000</td><td rowspan="18">Preliminary Design</td><td rowspan="12">Preliminary Design</td><td rowspan="2">Preliminary Design</td><td rowspan="2">Preliminary Design</td><td rowspan="18">Preliminary Design and Previous Project</td><td rowspan="18">Preliminary Design</td></tr>
<tr><td>Primary Crusher</td><td>3110</td></tr>
<tr><td>Stockpile Feed Conveyor</td><td>3120</td><td>Vendor</td><td>Vendor</td></tr>
<tr><td>Stockpile and Reclaim</td><td>3210</td><td rowspan="2">Preliminary Design</td><td rowspan="3">Preliminary Design</td></tr>
<tr><td>Grinding</td><td>3220</td></tr>
<tr><td>Pebble Crushing</td><td>3230</td><td>Previous Project</td></tr>
<tr><td>Pebble Crushing Conveying</td><td>3240</td><td>Vendor</td><td>Vendor</td></tr>
<tr><td>Copper Flotation</td><td>3250</td><td rowspan="5">Preliminary Design</td><td rowspan="5">Preliminary Design</td></tr>
<tr><td>Copper Regrind</td><td>3260</td></tr>
<tr><td>Copper Concentrate Thickening</td><td>3270</td></tr>
<tr><td>Copper Concentrate Filtration and Loadout</td><td>3280</td></tr>
<tr><td>Tailings Thickening</td><td>3290</td></tr>
<tr><td>Molybdenum Plant</td><td>3300</td><td>Previous Project</td><td>Preliminary Design and Previous Project</td><td>Previous Project</td></tr>
<tr><td>Reagents</td><td>3400</td><td>Preliminary Design</td><td rowspan="5">Preliminary Design</td><td rowspan="3">Preliminary Design</td></tr>
<tr><td>Plant Services</td><td>3500</td><td>Previous Project</td></tr>
<tr><td>Tailings Filter Plant</td><td>3600</td><td>Preliminary Design</td></tr>
<tr><td>Site Services and Utilities</td><td>4000</td><td>Preliminary Design and Previous Project</td><td>Previous Project</td></tr>
<tr><td>Site Stormwater Ponds</td><td>4100</td><td>Preliminary</td><td>N/A</td></tr>
</table>

| Description | WBS | Earthworks | Concrete | Structural | Platework | Piping | E & I |
|---|---|---|---|---|---|---|---|
| Plant Fuel Storage and Distribution | 4200 | | Design | N/A | | | |
| Sewerage and Waste Management | 4300 | | Factor | | | | |
| Communications and IT | 4400 | | N/A | | | N/A | N/A |
| Site Infrastructure | 5000 | | Preliminary Design | Preliminary Design | | | Preliminary Design |
| Tailings Management | 5500 | | N/A | Vendor | Vendor | Vendor | Vendor |
| External Infrastructure | 6000 | | Preliminary Design | Preliminary Design | N/A | Preliminary Design | Preliminary Design |

Project indirect costs were developed based on construction sequencing plans and quotes for EPCM service providers. Owner's Costs were developed by Rosemont Copper based on quoted equipment costs, third-party technical experts and in-house estimates.

A contingency component of $251M or 15% is included in the initial capital cost. No contingency is included in sustaining capital. All sunk costs are excluded from the capital cost estimates.

The estimate was reviewed against Constancia's actual costs by the Hudbay project team. An independent review of the capital cost estimate was performed by M3 Engineering and Technology; who has experience with mining projects in the US and benchmarked the estimate against other local projects of similar size.

The LOM sustaining capital costs are estimated to be $387 million excluding capitalized stripping and $1,168 million including capitalized stripping. Sustaining capital costs associated with mining include new mine equipment purchases and major rebuilds, ongoing haul road construction and expansions to the truck shop and heavy truck fuel facility. Sustaining capital for the processing plant includes three tailings stacking expansions, upgrades to the regrind mill, and the installation of a SHMP mixing facility. Sustaining capital also includes the installation of buttress drop structures.

**TABLE 21-2: SUSTAINING CAPITAL**

| **Sustaining Capital** | **000 $** |
|---|---|
| Mining – Equipment and Rebuilds | 138,898 |
| Mining – Equipment Major Repair | 170,123 |
| Mining – Heavy Civil Works | 19,540 |
| Mining – Facilities | 6,445 |
| Plant – Tailing Stacking Expansions | 23,855 |
| Plant – Upgrades | 3,417 |
| Plant – Ramp-up Support | 4,970 |
| Buttress Drop Structures | 12,515 |
| Light Vehicles & Misc | 7,100 |
| **Total Sustaining Capital (Excluding Capitalized Stripping)** | **386,865** |
| | |
| Capitalized Stripping | 780,897 |
| **Total Sustaining Capital Expenditures (Including Capitalized Stripping)** | **1,167,762** |

## 21.3 Operating Costs

Operating costs were developed by Hudbay based on a bottom-up approach and utilizing budget quotes from local suppliers, Arizona operations experience, and labor costs within the region. Site visits were conducted to other facilities currently utilizing dry stack technology to better understand the operations and maintenance requirements. Mining operating costs were validated against actual costs at Constancia.

The total LOM operating costs, including off-site costs (transport, TCRCs, etc.) are estimated at $12.34/ton milled (before deducting capitalized stripping) and $11.02/ton milled (after deducting capitalized stripping) as shown in Table 21-3. The mining costs below do not include pre-stripping costs as these are included in the development capital cost.

**TABLE 21-3: OPERATING COST SUMMARY**

| | Before Deducting Capitalized Stripping | | After Deducting Capitalized Stripping | |
|---|---|---|---|---|
| | **Unit Cost ($/ton milled)** | **Annual Average 19 Year LOM (000 $)** | **Unit Cost ($/ton milled)** | **Annual Average 19 Year LOM (000 $)** |
| Mining ($/ton moved) | $1.08 | $101,997 | $0.64 | $60,897 |
| | | | | |
| Mining | $3.27 | $101,997 | $1.95 | $60,897 |
| Processing | $4.71 | $146,723 | $4.71 | $146,723 |
| G&A, Other | $1.26 | $39,245 | $1.26 | $39,245 |
| **Total Mine / Mill / G&A** | **$9.24** | **$287,965** | **$7.92** | **$246,865** |
| | | | | |
| Off-Site Costs | $3.10 | $96,993 | $3.10 | $96,993 |
| | | | | |
| **Total** | **$12.34** | **$384,958** | **$11.02** | **$343,858** |

Reclamation costs of approximately $25M (which are incurred over the 19 year LOM) and closure costs of approximately $9M are included in operating G&A costs.

The total C1 cash costs and sustaining cash costs (net of by-product credits at stream prices) over the LOM and over the first 10 years are shown in Table 21-4. C1 cash costs include mining, milling, G&A and offsite costs. Sustaining cash costs include C1 costs plus royalties and sustaining capital.

**TABLE 21-4: CASH COSTS (NET OF BY-PRODUCT CREDITS AT STREAM PRICES)**

| **Cash Costs (Net of By-Product Credits at Stream Prices)** | **Units** | **Before Deducting Capitalized Stripping** | **After Deducting Capitalized Stripping** |
|---|---|---|---|
| Years 1-10 Average C1 Cash Costs | $ / lb Cu in con | $1.40 | $1.14 |
| Years 1-10 Average C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.59 | $1.59 |
| LOM C1 Cash Costs | $ / lb Cu in con | $1.47 | $1.29 |
| LOM C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.65 | $1.65 |

## 21.4 Working Capital Costs

Working capital for accounts receivable and accounts payable will vary over the mine life based on revenue, operating costs and capital costs. The working capital estimate is based on 33% of revenue as accounts receivable and 33% of cost of goods sold and capital costs as accounts payable in a given quarter. This is equivalent to an approximate 30-day delay in converting revenue to cash and an approximate 30-day delay in paying cash for capital and operating costs. All of the working capital is assumed to be recaptured by the end of the mine life and the closing value of the account is zero.

# 22 ECONOMIC ANALYSIS

## 22.1 Key Model Assumptions

All figures in the economic analysis are shown on an unlevered 100% Project basis (except where indicated in Section 22.5) and include the impact of the existing precious metals stream with Silver Wheaton.

### 22.1.1 Metal Prices

The economic viability of the Project has been evaluated using the metal prices outlined in Table 22-1. The metal prices used in the economic analysis are based on a blend of consensus metal price forecasts from over 30 well known financial institutions and Wood Mackenzie.

**TABLE 22-1: METAL PRICE ASSUMPTIONS**

| Metal | Unit | Price |
|---|---|---|
| Spot Copper | $/lb | $3.00 |
| Spot Molybdenum | $/lb | $11.00 |
| Spot Silver | $/oz | $18.00 |
| Streamed Silver[1] | $/oz | $3.90 |

1. Subject to a 1% annual inflation adjustment

The terms of the existing precious metals streaming agreement with Silver Wheaton were included in the analysis. Silver Wheaton will make upfront cash payments totaling $230 million to fund initial development capital in exchange for 100% of the silver and gold production from Rosemont. Silver Wheaton will make ongoing payments of $3.90 per ounce of silver and $450 per ounce of gold subject to a 1% inflation adjustment starting on the third anniversary of production.

Although gold is not part of the current reserve estimate, metallurgical testing has demonstrated economic concentrations of gold in copper concentrate as outlined in Section 13. Over the LOM, approximately 309 thousand ounces of gold are expected to be recovered in copper concentrate (although the financial impact has not been included).

At the effective realized prices including the impact of the stream, the revenue breakdown at Rosemont is approximately 92% copper, 6% molybdenum, and 2% silver.

### 22.1.2 Life of Mine Model Summary

The key mine and mill operating assumptions used in the cash flow model are outlined in Table 22-2.

**TABLE 22-2: MINE AND MILL OPERATING ASSUMPTIONS USED IN THE FINANCIAL MODEL (100% PROJECT BASIS)**

| | Units | LOM Total / Average |
|---|---|---|
| **Mine Plan** | | |
| Ore Mined | M short ton | 592 |
| Waste Mined (Excluding Pre-Strip)[1] | M short ton | 1,155 |
| Strip Ratio (Excluding Pre-Strip)[1] | Waste/Ore | 2.0 |
| Total Copper Grade Milled | % TCu | 0.45% |
| Sulfide Copper Grade Milled | % Sulfide Cu | 0.40% |
| Molybdenum Grade Milled | % Mo | 0.012% |
| Silver Grade Milled | oz/short ton Ag | 0.133 |
| **LOM Metallurgical Recoveries** | | |
| Sulfide Copper Recovery | % Sulfide Cu | 90.0% |
| Effective Total Copper Recovery | % TCu | 80.4% |
| Molybdenum Recovery | % Mo | 53.4% |
| Silver Recovery | % Ag | 74.4% |
| **LOM Concentrate Specifications** | | |
| Copper Grade in Copper Concentrate | % TCu | 34.3% |
| Molybdenum Grade in Molybdenum Concentrate | % Mo | 45.0% |
| Moisture Content of Copper & Molybdenum Concentrates | % $H_2O$ | 8.0% |
| **Production & Mill Throughput** | | |
| Design Mill Throughput | k short ton / day | 90 |
| Mine Life (Including Processed Stockpiles) | Years | 19 |
| Years 1-10 Average Annual Copper Production | k short ton Cu in con | 140 |
| LOM Average Annual Copper Production | k short ton Cu in con | 112 |
| LOM Average Annual Molybdenum Production | k short ton Mo in con | 2.0 |
| LOM Average Annual Silver Production | k oz Ag in con | 3,095 |

1. Pre-stripping costs included as part of the development capital costs

Key capital and operating cost assumptions used in the cash flow model are outlined in Table 22-3.

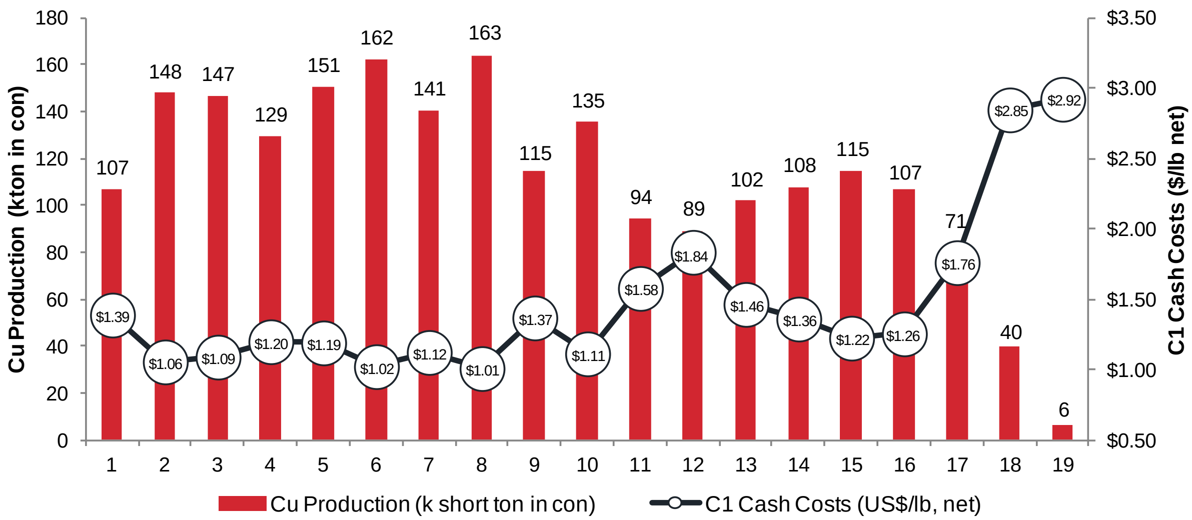
## TABLE 22-3: CAPITAL AND OPERATING COST ASSUMPTIONS USED IN THE FINANCIAL MODEL (100% PROJECT BASIS)

| | Units | LOM Total / Average |
|---|---|---|
| **Capital Costs** | | |
| Development Capital (Including Pre-Strip) [1] | $M | $1,921 |
| Development Capital Less Upfront Stream Proceeds | $M | $1,691 |
| LOM Sustaining Capital (Excluding Capitalized Stripping) | $M | $387 |
| Capitalized Stripping | $M | $781 |
| LOM Sustaining Capital (Including Capitalized Stripping) | $M | $1,168 |
| **Onsite Operating Costs (Before Deducting Capitalized Stripping)** | | |
| Mining (Before Deducting of Capitalized Stripping) | $ / short ton moved | $1.08 |
| Mining (Before Deducting of Capitalized Stripping) | $ / short ton milled | $3.27 |
| Milling | $ / short ton milled | $4.71 |
| On-Site G&A [2] | $ / short ton milled | $1.26 |
| Total Onsite Operating Costs | $ / short ton milled | $9.24 |
| **Onsite Operating Costs (After Deducting Capitalized Stripping)** | | |
| Mining (After Deducting Capitalized Stripping) | $ / short ton moved | $0.64 |
| Mining (After Deducting Capitalized Stripping) | $ / short ton milled | $1.95 |
| Milling | $ / short ton milled | $4.71 |
| On-Site G&A [2] | $ / short ton milled | $1.26 |
| Total Onsite Operating Costs | $ / short ton milled | $7.92 |
| **Offsite Operating Costs** | | |
| Copper Concentrate Base Treatment Charge | $ / dry short ton con | $73 |
| Copper Refining Charge | $ / lb Cu | $0.08 |
| Silver Refining Charge | $ / oz Ag | $0.50 |
| Copper Concentrate Transport & Freight | $ / wet short ton con | $127 |
| Molybdenum Concentrate Base Treatment Charge | $ / lb Mo | $1.50 |
| Molybdenum Concentrate Transport & Freight | $ / wet short ton con | $124 |
| **Cash Costs (Net of By-Product Credits at Stream Prices) Before Deducting Capitalized Stripping** | | |
| Years 1-10 Average C1 Cash Costs | $ / lb Cu in con | $1.40 |
| Years 1-10 Average C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.59 |
| LOM C1 Cash Costs | $ / lb Cu in con | $1.47 |
| LOM C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.65 |
| **Cash Costs (Net of By-Product Credits at Stream Prices) After Deducting Capitalized Stripping** | | |
| Years 1-10 Average C1 Cash Costs | $ / lb Cu in con | $1.14 |
| Years 1-10 Average C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.59 |
| LOM C1 Cash Costs | $ / lb Cu in con | $1.29 |
| LOM C1 Cash Costs + Royalties + Sustaining Capex | $ / lb Cu in con | $1.65 |

1. 15% contingency is included.
2. G&A also includes property tax, reclamation and closure costs.

Rosemont's annual copper production (contained copper in concentrate) and C1 cash costs (net of by-products at stream prices after deducting capitalized stripping) are shown below in Figure 22-1. Over the first 10 years, annual production is expected to average 140 thousand tons of copper at an average C1 cash cost of $1.14/lb. Over the 19 year LOM, annual production is expected to average 112 thousand tons of copper at an average C1 cash cost of $1.29/lb.

**FIGURE 22-1: ROSEMONT ANNUAL COPPER PRODUCTION AND C1 CASH COSTS**




### 22.1.3 Taxes and Royalties

#### 22.1.3.1 Applicable Tax Rates

The Project will be subject to a federal income tax rate of 35% (effectively reduced to 32% by the Section 199 deduction, a manufacturing and production credit available in the U.S.) and an alternative federal minimum tax rate of 20% (effectively reduced to 17% by the Section 199 deduction). A state income tax rate of 4.9% is also applicable to the Project. The amount of state tax payable in a given period reduces the amount of taxable income that is subject to federal income tax.

A depletion allowance of 15% has been utilized to reduce taxable income. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. The gross income from the property is defined as metal revenue less downstream costs (smelting, refining and transportation). Taxable income is defined as gross income minus operating expenses, overhead expenses, depreciation and state taxes.

A 2.5% severance tax is imposed in Arizona in lieu of sales tax on mining minerals. The net severance base is 50% of the difference between gross value of production and the production cost. The amount of tax is calculated by multiplying the net severance base by 2.5%. Severance tax is considered an income tax and is included in cash income taxes.

Rosemont has accrued approximately $123 million in net operating losses and approximately $333 million in depreciable capital that can be used to offset future income taxes.

The combined effective tax rate (excluding property tax which is included as part of G&A costs) varies in any given year but amounts to approximately 37% of taxable income over the life of mine.

### 22.1.4 Depreciation

Tax depreciation was applied to the development capital costs depending on the classification of capital as detailed in Table 22-4. 70% of the development capital associated with the pre-strip is expensed in the year it occurs and the remaining 30% is depreciated on a 5-year straight line basis.

**TABLE 22-4: TAX DEPRECIATION FOR DEVELOPMENT CAPITAL**

| | Development Capital (Excluding Infrastructure & Pre-Strip) | Pre-Strip Development Capital | Infrastructure Development Capital |
|---|---|---|---|
| Year 1 | 10.71% | 20.00% | 5.00% |
| Year 2 | 19.13% | 20.00% | 9.50% |
| Year 3 | 15.03% | 20.00% | 8.55% |
| Year 4 | 12.25% | 20.00% | 7.70% |
| Year 5 | 12.25% | 20.00% | 6.93% |
| Year 6 | 12.25% | - | 6.23% |
| Year 7 | 12.25% | - | 5.90% |
| Year 8 | 6.13% | - | 5.90% |
| Year 9 | - | - | 5.91% |
| Year 10 | - | - | 5.90% |
| Year 11 | - | - | 5.91% |
| Year 12 | - | - | 5.91% |
| Year 13 | - | - | 5.90% |
| Year 14 | - | - | 5.91% |
| Year 15 | - | - | 5.90% |
| Year 16 | - | - | 2.95% |
| **Total** | **100%** | **100%** | **100%** |

Sustaining capital was depreciated on 7-year straight-line basis for tax purposes (14.29% annually).

#### 22.1.4.1 Royalties

A 3% NSR royalty exists on the Project and is included in the economic analysis.

## 22.2 Annual Cash Flow Model

A summary of the annual cash flow model is outlined in Table 22-5.

# TABLE 22-5: ANNUAL CASH FLOW MODEL

| | | LOM Total/Avg. | Year -3 | Year -2 | Year -1 | Year 1 | Year 2 | Year 3 |
|---|---|---|---|---|---|---|---|---|
| **Mine & Mill Plan** | | | | | | | | |
| Ore Mined Directly to Mill | 000 short tons | 529,848 | - | - | - | 24,511 | 32,850 | 32,850 |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.43% | - | - | - | 0.44% | 0.50% | 0.50% |
| Copper Grade | % Total Cu | 0.47% | - | - | - | 0.52% | 0.55% | 0.54% |
| Molybdenum Grade | % Mo | 0.013% | - | - | - | 0.011% | 0.011% | 0.013% |
| Silver Grade | oz/short ton Ag | 0.140 | - | - | - | 0.161 | 0.181 | 0.161 |
| | | | | | | | | |
| Ore Mined to Stockpile | 000 short tons | 62,185 | - | - | 11,227 | 4,264 | 12,459 | 13,188 |
| Copper Grade | % Total Cu | 0.22% | - | - | 0.31% | 0.21% | 0.20% | 0.21% |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.16% | - | - | 0.23% | 0.13% | 0.16% | 0.16% |
| Molybdenum Grade | % Mo | 0.006% | - | - | 0.005% | 0.004% | 0.005% | 0.008% |
| Silver Grade | oz/short ton Ag | 0.079 | - | - | 0.108 | 0.087 | 0.087 | 0.077 |
| | | | | | | | | |
| Stockpile to Mill | 000 short tons | 62,185 | - | - | - | 3,602 | - | - |
| Copper Grade | % Total Cu | 0.22% | - | - | - | 0.43% | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.16% | - | - | - | 0.37% | - | - |
| Molybdenum Grade | % Mo | 0.006% | - | - | - | 0.007% | - | - |
| Silver Grade | oz/short ton Ag | 0.079 | - | - | - | 0.165 | - | - |
| | | | | | | | | |
| Waste Mined (Excluding Pre-Strip) | 000 short tons | 1,154,780 | - | - | - | 99,622 | 86,691 | 85,962 |
| Strip Ratio (Excluding Pre-Strip) | waste/ore | 2.0 | - | - | - | 3.5 | 1.9 | 1.9 |
| | | | | | | | | |
| Ore Milled | 000 short tons | 592,033 | - | - | - | 28,114 | 32,850 | 32,850 |
| Copper Grade | % Total Cu | 0.45% | - | - | - | 0.51% | 0.55% | 0.54% |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.40% | - | - | - | 0.43% | 0.50% | 0.50% |
| Molybdenum Grade | % Mo | 0.012% | - | - | - | 0.011% | 0.011% | 0.013% |
| Silver Grade | oz/short ton Ag | 0.133 | - | - | - | 0.161 | 0.181 | 0.161 |
| | | | | | | | | |
| **Metallurgical Recoveries** | | | | | | | | |
| Copper | % Total Cu | 80.4% | - | - | - | 72.0% | 82.4% | 82.4% |
| Molybdenum | % Mo | 53.4% | - | - | - | 63.9% | 74.4% | 74.4% |
| Silver | % Ag | 74.4% | - | - | - | 71.5% | 74.5% | 74.5% |
| | | | | | | | | |
| **Cocentrate Produced** | | | | | | | | |
| Copper Concentrate Produced | 000 short dry tons | 6,212 | - | - | - | 344 | 462 | 458 |
| Contained Copper | 000 short tons | 2,129 | - | - | - | 107 | 148 | 147 |
| Contained Copper | 000 metric tonnes | 1,932 | - | - | - | 97 | 134 | 133 |
| Contained Silver | 000 oz | 58,809 | - | - | - | 3,315 | 4,430 | 3,939 |
| Payable Copper | 000 short tons | 2,055 | - | - | - | 103 | 143 | 141 |
| Payable Copper | 000 metric tonnes | 1,864 | - | - | - | 94 | 129 | 128 |
| Payable Silver | 000 oz | 53,370 | - | - | - | 3,009 | 4,020 | 3,575 |
| | | | | | | | | |
| Molybdenum Concentrate Produced | 000 short dry tons | 84.4 | - | - | - | 4.7 | 6.2 | 6.8 |
| Contained Molybedenum | 000 short tons | 37.9 | - | - | - | 2.1 | 2.8 | 3.1 |
| Payable Molybedenum | 000 short tons | 37.6 | - | - | - | 2.1 | 2.8 | 3.0 |
| | | | | | | | | |
| **Metal Prices** | | | | | | | | |
| Spot Copper Price | US$/lb | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 |
| Spot Molybdenum Price | US$/lb | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 |
| Spot Silver Price | US$/oz | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 |
| Stream Silver Price | US$/oz | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 | $3.90 |
| | | | | | | | | |
| **Revenue** | | | | | | | | |
| Gross Revenue (at Stream Prices) | US$ 000 | $13,376,617 | - | - | - | $676,374 | $931,875 | $929,955 |
| TCRCs & Offsite Costs | US$ 000 | ($1,837,216) | - | - | - | ($100,050) | ($134,879) | ($134,562) |
| **Net Revenue** | **US$ 000** | **$11,539,402** | **-** | **-** | **-** | **$576,324** | **$796,997** | **$795,393** |
| | | | | | | | | |
| **Operating Costs** | | | | | | | | |
| Mining (After Deducting Capitalized Stripping) | US$ 000 | ($1,157,050) | - | - | - | ($75,284) | ($66,256) | ($62,278) |
| Milling | US$ 000 | ($2,787,733) | - | - | - | ($137,699) | ($152,913) | ($153,367) |
| On-Site G&A (Including Closure Costs) | US$ 000 | ($745,933) | - | - | - | ($41,193) | ($34,238) | ($50,457) |
| **Total Onsite Costs** | **US$ 000** | **($4,690,716)** | **-** | **-** | **-** | **($254,176)** | **($253,408)** | **($266,101)** |
| | | | | | | | | |
| Royalties | US$ 000 | ($368,372) | - | - | - | ($18,562) | ($25,610) | ($25,374) |
| **Total Operating Costs** | **US$ 000** | **($5,059,088)** | **-** | **-** | **-** | **($272,738)** | **($279,018)** | **($291,475)** |
| | | | | | | | | |
| **Cash Costs (Net of By-Products at Stream Prices)** | | | | | | | | |
| C1 (After Deducting Capitalized Stripping) | US$/lb | $1.29 | - | - | - | $1.39 | $1.06 | $1.09 |
| C1 + Royalties + Sustaining Capex | US$/lb | $1.65 | - | - | - | $1.74 | $1.45 | $1.52 |
| | | | | | | | | |
| **Capital Costs** | | | | | | | | |
| Development Capital | US$ 000 | ($1,920,903) | ($143,780) | ($861,141) | ($768,411) | ($147,572) | - | - |
| Sustaining Capital | US$ 000 | ($386,865) | - | - | - | ($16,902) | ($39,569) | ($37,274) |
| Capitalized Stripping | US$ 000 | ($780,897) | - | - | - | ($40,056) | ($52,086) | ($62,134) |
| **Total Capital Costs** | **US$ 000** | **($3,088,665)** | **($143,780)** | **($861,141)** | **($768,411)** | **($204,530)** | **($91,655)** | **($99,408)** |
| | | | | | | | | |
| Change in Working Capital | US$ 000 | ($0) | $34,732 | $40,693 | ($25,796) | ($75,773) | $2,424 | ($6,548) |
| Stream Upfront Payments | US$ 000 | $230,000 | $98,421 | $131,579 | - | - | - | - |
| **Unlevered Cash Flow Before Tax (100% Basis)** | **US$ 000** | **$3,621,649** | **($10,627)** | **($688,869)** | **($794,207)** | **$23,282** | **$428,747** | **$397,963** |
| | | | | | | | | |
| Cash Income Taxes | US$ 000 | ($718,192) | - | - | - | ($8,419) | ($17,945) | ($26,243) |
| **Unlevered Free Cash Flow After Tax (100% Basis)** | **US$ 000** | **$2,903,457** | **($10,627)** | **($688,869)** | **($794,207)** | **$14,863** | **$410,802** | **$371,719** |
| | | | | | | | | |
| Exisiting JV Loan Repayment | US$ 000 | $20,000 | $20,000 | - | - | - | - | - |
| Joint Venture Earn-in Payments | US$ 000 | $106,000 | $45,359 | $60,641 | - | - | - | - |
| | | | | | | | | |
| **Free Cash Flow to Hudbay** | **US$ 000** | **$2,430,563** | **$50,660** | **($502,459)** | **($635,365)** | **$11,891** | **$328,642** | **$297,376** |

| | | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 |
|---|---|---|---|---|---|---|---|---|
| **Mine & Mill Plan** | | | | | | | | |
| Ore Mined Directly to Mill | 000 short tons | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.44% | 0.51% | 0.55% | 0.48% | 0.55% | 0.39% | 0.46% |
| Copper Grade | % Total Cu | 0.49% | 0.55% | 0.59% | 0.51% | 0.61% | 0.44% | 0.49% |
| Molybdenum Grade | % Mo | 0.011% | 0.010% | 0.013% | 0.013% | 0.015% | 0.010% | 0.013% |
| Silver Grade | oz/short ton Ag | 0.171 | 0.169 | 0.184 | 0.132 | 0.138 | 0.101 | 0.130 |
| Ore Mined to Stockpile | 000 short tons | 2,820 | 5,809 | 6,494 | 1,815 | 4,108 | - | - |
| Copper Grade | % Total Cu | 0.18% | 0.21% | 0.17% | 0.15% | 0.24% | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.13% | 0.16% | 0.14% | 0.12% | 0.14% | - | - |
| Molybdenum Grade | % Mo | 0.005% | 0.007% | 0.008% | 0.008% | 0.007% | - | - |
| Silver Grade | oz/short ton Ag | 0.077 | 0.055 | 0.061 | 0.037 | 0.061 | - | - |
| Stockpile to Mill | 000 short tons | - | - | - | - | - | - | - |
| Copper Grade | % Total Cu | - | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | - | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | - | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | - | - | - | - | - | - | - |
| Waste Mined (Excluding Pre-Strip) | 000 short tons | 96,330 | 93,341 | 92,656 | 97,335 | 95,042 | 99,150 | 99,150 |
| Strip Ratio (Excluding Pre-Strip) | waste/ore | 2.7 | 2.4 | 2.4 | 2.8 | 2.6 | 3.0 | 3.0 |
| Ore Milled | 000 short tons | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 |
| Copper Grade | % Total Cu | 0.49% | 0.55% | 0.59% | 0.51% | 0.61% | 0.44% | 0.49% |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.44% | 0.51% | 0.55% | 0.48% | 0.55% | 0.39% | 0.46% |
| Molybdenum Grade | % Mo | 0.011% | 0.010% | 0.013% | 0.013% | 0.015% | 0.010% | 0.013% |
| Silver Grade | oz/short ton Ag | 0.171 | 0.169 | 0.184 | 0.132 | 0.138 | 0.101 | 0.130 |
| **Metallurgical Recoveries** | | | | | | | | |
| Copper | % Total Cu | 80.8% | 82.3% | 83.2% | 83.7% | 81.6% | 79.2% | 83.3% |
| Molybdenum | % Mo | 43.9% | 43.9% | 43.9% | 51.3% | 51.3% | 51.3% | 51.3% |
| Silver | % Ag | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% |
| **Cocentrate Produced** | | | | | | | | |
| Copper Concentrate Produced | 000 short dry tons | 381 | 430 | 463 | 402 | 467 | 329 | 387 |
| Contained Copper | 000 short tons | 129 | 151 | 162 | 141 | 163 | 115 | 135 |
| Contained Copper | 000 metric tonnes | 117 | 137 | 147 | 127 | 148 | 104 | 123 |
| Contained Silver | 000 oz | 4,180 | 4,130 | 4,509 | 3,224 | 3,366 | 2,475 | 3,186 |
| Payable Copper | 000 short tons | 125 | 145 | 156 | 136 | 158 | 111 | 131 |
| Payable Copper | 000 metric tonnes | 113 | 132 | 142 | 123 | 143 | 101 | 119 |
| Payable Silver | 000 oz | 3,794 | 3,748 | 4,092 | 2,926 | 3,055 | 2,246 | 2,891 |
| Molybdenum Concentrate Produced | 000 short dry tons | 3.7 | 3.3 | 4.2 | 5.0 | 5.8 | 3.7 | 4.8 |
| Contained Molybedenum | 000 short tons | 1.7 | 1.5 | 1.9 | 2.3 | 2.6 | 1.7 | 2.2 |
| Payable Molybedenum | 000 short tons | 1.6 | 1.5 | 1.9 | 2.2 | 2.6 | 1.7 | 2.2 |
| **Metal Prices** | | | | | | | | |
| Spot Copper Price | US$/lb | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 |
| Spot Molybdenum Price | US$/lb | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 |
| Spot Silver Price | US$/oz | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 |
| Stream Silver Price | US$/oz | $3.94 | $3.98 | $4.02 | $4.06 | $4.10 | $4.14 | $4.18 |
| **Revenue** | | | | | | | | |
| Gross Revenue (at Stream Prices) | US$ 000 | $800,032 | $919,049 | $996,221 | $874,631 | $1,015,427 | $711,447 | $843,783 |
| TCRCs & Offsite Costs | US$ 000 | ($110,575) | ($123,994) | ($134,489) | ($118,140) | ($137,058) | ($95,976) | ($113,935) |
| **Net Revenue** | **US$ 000** | **$689,457** | **$795,055** | **$861,732** | **$756,491** | **$878,369** | **$615,471** | **$729,848** |
| **Operating Costs** | | | | | | | | |
| Mining (After Deducting Capitalized Stripping) | US$ 000 | ($47,300) | ($77,520) | ($50,443) | ($56,542) | ($60,499) | ($66,972) | ($50,246) |
| Milling | US$ 000 | ($155,083) | ($155,590) | ($157,075) | ($156,528) | ($157,333) | ($155,629) | ($156,350) |
| On-Site G&A (Including Closure Costs) | US$ 000 | ($48,287) | ($48,945) | ($46,851) | ($45,432) | ($44,071) | ($41,839) | ($40,546) |
| **Total Onsite Costs** | **US$ 000** | **($250,670)** | **($282,054)** | **($254,369)** | **($258,502)** | **($261,903)** | **($264,440)** | **($247,142)** |
| Royalties | US$ 000 | ($22,284) | ($25,428) | ($27,568) | ($23,919) | ($27,625) | ($19,398) | ($23,094) |
| **Total Operating Costs** | **US$ 000** | **($272,954)** | **($307,483)** | **($281,937)** | **($282,420)** | **($289,528)** | **($283,838)** | **($270,236)** |
| **Cash Costs (Net of By-Products at Stream Prices)** | | | | | | | | |
| C1 (After Deducting Capitalized Stripping) | US$/lb | $1.20 | $1.19 | $1.02 | $1.12 | $1.01 | $1.37 | $1.11 |
| C1 + Royalties + Sustaining Capex | US$/lb | $1.74 | $1.65 | $1.50 | $1.57 | $1.40 | $1.85 | $1.59 |
| **Capital Costs** | | | | | | | | |
| Development Capital | US$ 000 | - | - | - | - | - | - | - |
| Sustaining Capital | US$ 000 | ($34,210) | ($53,890) | ($34,146) | ($16,187) | ($17,463) | ($22,847) | ($21,351) |
| Capitalized Stripping | US$ 000 | ($83,679) | ($57,946) | ($91,635) | ($85,942) | ($83,788) | ($68,601) | ($83,675) |
| **Total Capital Costs** | **US$ 000** | **($117,889)** | **($111,836)** | **($125,781)** | **($102,129)** | **($101,250)** | **($91,448)** | **($105,027)** |
| Change in Working Capital | US$ 000 | $1,166 | ($12,705) | $8,416 | $3,680 | ($8,875) | $19,348 | ($7,702) |
| Stream Upfront Payments | US$ 000 | - | - | - | - | - | - | - |
| **Unlevered Cash Flow Before Tax (100% Basis)** | **US$ 000** | **$299,780** | **$363,031** | **$462,429** | **$375,622** | **$478,716** | **$259,533** | **$346,883** |
| Cash Income Taxes | US$ 000 | ($18,752) | ($34,510) | ($51,805) | ($28,890) | ($75,965) | ($40,822) | ($70,418) |
| **Unlevered Free Cash Flow After Tax (100% Basis)** | **US$ 000** | **$281,028** | **$328,520** | **$410,624** | **$346,732** | **$402,751** | **$218,711** | **$276,465** |
| Exisiting JV Loan Repayment | US$ 000 | - | - | - | - | - | - | - |
| Joint Venture Earn-in Payments | US$ 000 | - | - | - | - | - | - | - |
| **Free Cash Flow to Hudbay** | **US$ 000** | **$224,823** | **$262,816** | **$328,500** | **$277,386** | **$322,201** | **$174,969** | **$221,172** |

| | | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 | Year 16 | Year 17 |
|---|---|---|---|---|---|---|---|---|
| **Mine & Mill Plan** | | | | | | | | |
| Ore Mined Directly to Mill | 000 short tons | 29,329 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 16,108 |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.33% | 0.30% | 0.34% | 0.37% | 0.39% | 0.36% | 0.30% |
| Copper Grade | % Total Cu | 0.38% | 0.36% | 0.40% | 0.40% | 0.42% | 0.39% | 0.33% |
| Molybdenum Grade | % Mo | 0.011% | 0.011% | 0.013% | 0.013% | 0.015% | 0.017% | 0.012% |
| Silver Grade | oz/short ton Ag | 0.095 | 0.097 | 0.116 | 0.124 | 0.144 | 0.138 | 0.135 |
| | | | | | | | | |
| Ore Mined to Stockpile | 000 short tons | - | - | - | - | - | - | - |
| Copper Grade | % Total Cu | - | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | - | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | - | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | - | - | - | - | - | - | - |
| | | | | | | | | |
| Stockpile to Mill | 000 short tons | 3,521 | - | - | - | - | - | 16,742 |
| Copper Grade | % Total Cu | 0.26% | - | - | - | - | - | 0.23% |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.21% | - | - | - | - | - | 0.18% |
| Molybdenum Grade | % Mo | 0.009% | - | - | - | - | - | 0.008% |
| Silver Grade | oz/short ton Ag | 0.098 | - | - | - | - | - | 0.081 |
| | | | | | | | | |
| Waste Mined (Excluding Pre-Strip) | 000 short tons | 99,150 | 61,572 | 25,317 | 8,418 | 3,602 | 5,773 | 5,671 |
| Strip Ratio (Excluding Pre-Strip) | waste/ore | 3.4 | 1.9 | 0.8 | 0.3 | 0.1 | 0.2 | 0.4 |
| | | | | | | | | |
| Ore Milled | 000 short tons | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 | 32,850 |
| Copper Grade | % Total Cu | 0.37% | 0.36% | 0.40% | 0.40% | 0.42% | 0.39% | 0.28% |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.32% | 0.30% | 0.34% | 0.37% | 0.39% | 0.36% | 0.24% |
| Molybdenum Grade | % Mo | 0.010% | 0.011% | 0.013% | 0.013% | 0.015% | 0.017% | 0.010% |
| Silver Grade | oz/short ton Ag | 0.095 | 0.097 | 0.116 | 0.124 | 0.144 | 0.138 | 0.107 |
| | | | | | | | | |
| **Metallurgical Recoveries** | | | | | | | | |
| Copper | % Total Cu | 78.2% | 74.9% | 77.6% | 81.7% | 82.8% | 83.0% | 77.9% |
| Molybdenum | % Mo | 51.3% | 51.3% | 51.3% | 51.3% | 51.3% | 51.3% | 51.3% |
| Silver | % Ag | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% | 74.5% |
| | | | | | | | | |
| **Cocentrate Produced** | | | | | | | | |
| Copper Concentrate Produced | 000 short dry tor | 270 | 253 | 291 | 308 | 328 | 305 | 202 |
| Contained Copper | 000 short tons | 94 | 89 | 102 | 108 | 115 | 107 | 71 |
| Contained Copper | 000 metric tonne | 86 | 80 | 92 | 98 | 104 | 97 | 64 |
| Contained Silver | 000 oz | 2,327 | 2,374 | 2,838 | 3,040 | 3,516 | 3,370 | 2,628 |
| Payable Copper | 000 short tons | 91 | 86 | 98 | 104 | 111 | 103 | 68 |
| Payable Copper | 000 metric tonne | 83 | 78 | 89 | 94 | 100 | 94 | 62 |
| Payable Silver | 000 oz | 2,112 | 2,154 | 2,575 | 2,759 | 3,191 | 3,058 | 2,385 |
| | | | | | | | | |
| Molybdenum Concentrate Produced | 000 short dry tor | 3.9 | 4.3 | 5.0 | 5.0 | 5.6 | 6.3 | 3.7 |
| Contained Molybedenum | 000 short tons | 1.7 | 1.9 | 2.3 | 2.3 | 2.5 | 2.8 | 1.6 |
| Payable Molybedenum | 000 short tons | 1.7 | 1.9 | 2.2 | 2.2 | 2.5 | 2.8 | 1.6 |
| | | | | | | | | |
| **Metal Prices** | | | | | | | | |
| Spot Copper Price | US$/lb | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 |
| Spot Molybdenum Price | US$/lb | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 |
| Spot Silver Price | US$/oz | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 |
| Stream Silver Price | US$/oz | $4.22 | $4.27 | $4.31 | $4.35 | $4.39 | $4.44 | $4.48 |
| | | | | | | | | |
| **Revenue** | | | | | | | | |
| Gross Revenue (at Stream Prices) | US$ 000 | $593,844 | $564,431 | $649,772 | $686,144 | $733,298 | $693,935 | $456,365 |
| TCRCs & Offsite Costs | US$ 000 | ($80,272) | ($76,416) | ($87,996) | ($92,858) | ($99,301) | ($94,135) | ($61,871) |
| **Net Revenue** | **US$ 000** | **$513,571** | **$488,014** | **$561,777** | **$593,286** | **$633,996** | **$599,800** | **$394,494** |
| | | | | | | | | |
| **Operating Costs** | | | | | | | | |
| Mining (After Deducting Capitalized Stripping) | US$ 000 | ($74,004) | ($109,317) | ($80,444) | ($72,736) | ($62,290) | ($62,679) | ($50,659) |
| Milling | US$ 000 | ($154,086) | ($153,882) | ($154,343) | ($154,560) | ($154,800) | ($154,522) | ($153,237) |
| On-Site G&A (Including Closure Costs) | US$ 000 | ($36,707) | ($36,819) | ($35,833) | ($34,787) | ($33,535) | ($32,131) | ($30,129) |
| **Total Onsite Costs** | **US$ 000** | **($264,797)** | **($300,019)** | **($270,620)** | **($262,083)** | **($250,626)** | **($249,333)** | **($234,026)** |
| | | | | | | | | |
| Royalties | US$ 000 | ($16,280) | ($15,528) | ($17,911) | ($18,928) | ($20,322) | ($19,238) | ($12,802) |
| **Total Operating Costs** | **US$ 000** | **($281,077)** | **($315,547)** | **($288,531)** | **($281,012)** | **($270,948)** | **($268,571)** | **($246,828)** |
| | | | | | | | | |
| **Cash Costs (Net of By-Products at Stream Prices)** | | | | | | | | |
| C1 (After Deducting Capitalized Stripping) | US$/lb | $1.58 | $1.84 | $1.46 | $1.36 | $1.22 | $1.26 | $1.76 |
| C1 + Royalties + Sustaining Capex | US$/lb | $2.12 | $2.10 | $1.61 | $1.51 | $1.36 | $1.39 | $1.86 |
| | | | | | | | | |
| **Capital Costs** | | | | | | | | |
| Development Capital | US$ 000 | - | - | - | - | - | - | - |
| Sustaining Capital | US$ 000 | ($26,756) | ($19,776) | ($11,733) | ($13,040) | ($11,613) | ($9,064) | ($521) |
| Capitalized Stripping | US$ 000 | ($59,965) | ($11,389) | - | - | - | - | - |
| **Total Capital Costs** | **US$ 000** | **($86,721)** | **($31,165)** | **($11,733)** | **($13,040)** | **($11,613)** | **($9,064)** | **($521)** |
| | | | | | | | | |
| Change in Working Capital | US$ 000 | $19,132 | ($1,358) | ($12,434) | ($2,796) | ($4,647) | $3,068 | $13,808 |
| Stream Upfront Payments | US$ 000 | - | - | - | - | - | - | - |
| **Unlevered Cash Flow Before Tax (100% Basis)** | **US$ 000** | **$164,905** | **$139,944** | **$249,079** | **$296,438** | **$346,789** | **$325,233** | **$160,954** |
| | | | | | | | | |
| Cash Income Taxes | US$ 000 | ($24,886) | ($25,451) | ($48,833) | ($58,549) | ($70,621) | ($82,203) | ($33,810) |
| **Unlevered Free Cash Flow After Tax (100% Basis)** | **US$ 000** | **$140,019** | **$114,494** | **$200,246** | **$237,889** | **$276,168** | **$243,029** | **$127,144** |
| | | | | | | | | |
| Exisiting JV Loan Repayment | US$ 000 | - | - | - | - | - | - | - |
| Joint Venture Earn-in Payments | US$ 000 | - | - | - | - | - | - | - |
| | | | | | | | | |
| **Free Cash Flow to Hudbay** | **US$ 000** | **$112,015** | **$91,595** | **$160,197** | **$190,311** | **$220,934** | **$194,424** | **$101,715** |

| | | Year 18 | Year 19 | Year 20 | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 |
|---|---|---|---|---|---|---|---|---|---|
| **Mine & Mill Plan** | | | | | | | | | |
| Ore Mined Directly to Mill | 000 short tons | - | - | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | - | - | - | - | - | - | - | - |
| Copper Grade | % Total Cu | - | - | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | - | - | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | - | - | - | - | - | - | - | - |
| | | | | | | | | | |
| Ore Mined to Stockpile | 000 short tons | - | - | - | - | - | - | - | - |
| Copper Grade | % Total Cu | - | - | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | - | - | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | - | - | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | - | - | - | - | - | - | - | - |
| | | | | | | | | | |
| Stockpile to Mill | 000 short tons | 32,850 | 5,469 | - | - | - | - | - | - |
| Copper Grade | % Total Cu | 0.19% | 0.18% | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.14% | 0.12% | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | 0.006% | 0.004% | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | 0.069 | 0.069 | - | - | - | - | - | - |
| | | | | | | | | | |
| Waste Mined (Excluding Pre-Strip) | 000 short tons | - | - | - | | | | | |
| Strip Ratio (Excluding Pre-Strip) | waste/ore | - | - | - | | | | | |
| | | | | | | | | | |
| Ore Milled | 000 short tons | 32,850 | 5,469 | - | - | - | - | - | - |
| Copper Grade | % Total Cu | 0.19% | 0.18% | - | - | - | - | - | - |
| Copper Grade (Sulfide) | % Sulfide Cu | 0.14% | 0.12% | - | - | - | - | - | - |
| Molybdenum Grade | % Mo | 0.006% | 0.004% | - | - | - | - | - | - |
| Silver Grade | oz/short ton Ag | 0.069 | 0.069 | - | - | - | - | - | - |
| | | | | | | | | | |
| **Metallurgical Recoveries** | | | | | | | | | |
| Copper | % Total Cu | 63.1% | 60.9% | - | - | - | - | - | - |
| Molybdenum | % Mo | 51.3% | 51.3% | - | - | - | - | - | - |
| Silver | % Ag | 74.5% | 74.5% | - | - | - | - | - | - |
| | | | | | | | | | |
| **Cocentrate Produced** | | | | | | | | | |
| Copper Concentrate Produced | 000 short dry tor | 115 | 17 | - | - | - | - | - | - |
| Contained Copper | 000 short tons | 40 | 6 | - | - | - | - | - | - |
| Contained Copper | 000 metric tonne | 36 | 6 | - | - | - | - | - | - |
| Contained Silver | 000 oz | 1,681 | 282 | - | - | - | - | - | - |
| Payable Copper | 000 short tons | 39 | 6 | - | - | - | - | - | - |
| Payable Copper | 000 metric tonne | 35 | 5 | - | - | - | - | - | - |
| Payable Silver | 000 oz | 1,525 | 256 | - | - | - | - | - | - |
| | | | | | | | | | |
| Molybdenum Concentrate Produced | 000 short dry tor | 2.2 | 0.2 | - | - | - | - | - | - |
| Contained Molybedenum | 000 short tons | 1.0 | 0.1 | - | - | - | - | - | - |
| Payable Molybedenum | 000 short tons | 1.0 | 0.1 | - | - | - | - | - | - |
| | | | | | | | | | |
| **Metal Prices** | | | | | | | | | |
| Spot Copper Price | US$/lb | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 | $3.00 |
| Spot Molybdenum Price | US$/lb | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 | $11.00 |
| Spot Silver Price | US$/oz | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 | $18.00 |
| Stream Silver Price | US$/oz | $4.53 | $4.57 | $4.62 | $4.66 | $4.71 | $4.76 | $4.81 | $4.85 |
| | | | | | | | | | |
| **Revenue** | | | | | | | | | |
| Gross Revenue (at Stream Prices) | US$ 000 | $261,171 | $38,864 | - | - | - | - | - | - |
| TCRCs & Offsite Costs | US$ 000 | ($35,446) | ($5,262) | - | - | - | - | - | - |
| **Net Revenue** | **US$ 000** | **$225,725** | **$33,602** | **-** | **-** | **-** | **-** | **-** | **-** |
| | | | | | | | | | |
| **Operating Costs** | | | | | | | | | |
| Mining (After Deducting Capitalized Stripping) | US$ 000 | ($27,373) | ($4,207) | - | - | - | - | - | - |
| Milling | US$ 000 | ($148,715) | ($22,020) | - | - | - | - | - | - |
| On-Site G&A (Including Closure Costs) | US$ 000 | ($45,780) | ($7,711) | ($3,355) | ($2,645) | ($1,614) | ($1,514) | ($1,514) | - |
| **Total Onsite Costs** | **US$ 000** | **($221,868)** | **($33,938)** | **($3,355)** | **($2,645)** | **($1,614)** | **($1,514)** | **($1,514)** | **-** |
| | | | | | | | | | |
| Royalties | US$ 000 | ($7,388) | ($1,111) | - | - | - | - | - | - |
| **Total Operating Costs** | **US$ 000** | **($229,256)** | **($35,049)** | **($3,355)** | **($2,645)** | **($1,614)** | **($1,514)** | **($1,514)** | **-** |
| | | | | | | | | | |
| **Cash Costs (Net of By-Products at Stream Prices)** | | | | | | | | | |
| C1 (After Deducting Capitalized Stripping) | US$/lb | $2.85 | $2.92 | - | - | - | - | - | - |
| C1 + Royalties + Sustaining Capex | US$/lb | $2.95 | $3.01 | - | - | - | - | - | - |
| | | | | | | | | | |
| **Capital Costs** | | | | | | | | | |
| Development Capital | US$ 000 | - | - | - | - | - | - | - | - |
| Sustaining Capital | US$ 000 | ($521) | - | - | - | - | - | - | - |
| Capitalized Stripping | US$ 000 | - | - | - | - | - | - | - | - |
| **Total Capital Costs** | **US$ 000** | **($521)** | **-** | **-** | **-** | **-** | **-** | **-** | **-** |
| | | | | | | | | | |
| Change in Working Capital | US$ 000 | $12,497 | ($260) | $204 | ($97) | ($47) | ($8) | - | ($125) |
| Stream Upfront Payments | US$ 000 | - | - | - | - | - | - | - | - |
| **Unlevered Cash Flow Before Tax (100% Basis)** | **US$ 000** | **$8,445** | **($1,707)** | **($3,150)** | **($2,742)** | **($1,661)** | **($1,522)** | **($1,514)** | **($125)** |
| | | | | | | | | | |
| Cash Income Taxes | US$ 000 | ($65) | ($5) | - | - | - | - | - | - |
| **Unlevered Free Cash Flow After Tax (100% Basis)** | **US$ 000** | **$8,379** | **($1,711)** | **($3,150)** | **($2,742)** | **($1,661)** | **($1,522)** | **($1,514)** | **($125)** |
| | | | | | | | | | |
| Exisiting JV Loan Repayment | US$ 000 | - | - | - | - | - | - | - | - |
| Joint Venture Earn-in Payments | US$ 000 | - | - | - | - | - | - | - | - |
| | | | | | | | | | |
| **Free Cash Flow to Hudbay** | **US$ 000** | **$6,704** | **($1,369)** | **($2,520)** | **($2,194)** | **($1,328)** | **($1,218)** | **($1,211)** | **($100)** |

## 22.3 Financial Analysis (100% Project Basis)

Rosemont (on a 100% basis) has an unlevered after-tax NPV8% of $769 million and a 15.5% after-tax IRR at a copper price of $3.00/lb as summarized in Table 22-6. The Project NPV and IRR are calculated using end of period quarterly discounting in the quarter before initial development capital is spent.

**TABLE 22-6: LIFE OF MINE FINANCIAL METRICS (100% PROJECT BASIS)**

| | Units | LOM Total |
|---|---|---|
| Gross Revenue (Stream Prices) | $M | $13,377 |
| TCRCs | $M | ($1,837) |
| On-Site Operating Costs<br>(after deducting capitalized stripping) | $M | ($4,691) |
| Royalties | $M | ($368) |
| **Operating Margin** | **$M** | **$6,480** |
| Development Capital | $M | ($1,921) |
| Stream Upfront Payment | $M | $230 |
| Sustaining Capital (excludes capitalized stripping) | $M | ($387) |
| Capitalized Stripping | $M | ($781) |
| **Pre-Tax Cash Flow** | **$M** | **$3,622** |
| Cash Income Taxes | $M | ($718) |
| **After-Tax Free Cash Flow** | **$M** | **$2,903** |
| After-Tax NPV8% | $M | $769 |
| After-Tax NPV10% | $M | $496 |
| After-Tax IRR | % | 15.5% |
| After-Tax Payback Period | Years | 5.2 |

## 22.4 Sensitivity Analysis (100% Project Basis)

The NPV8% (100% Project basis) was sensitized based on percentage changes in various input assumptions above or below the base case. Each input assumption change was assumed to occur independently from changes in other inputs. The sensitivity analysis is summarized in Figure 22-2. The Project is most sensitive to the copper price followed by initial capital costs, on-site operating costs, and the molybdenum price.

**FIGURE 22-2: NPV8% SENSITIVITY (100% BASIS)**

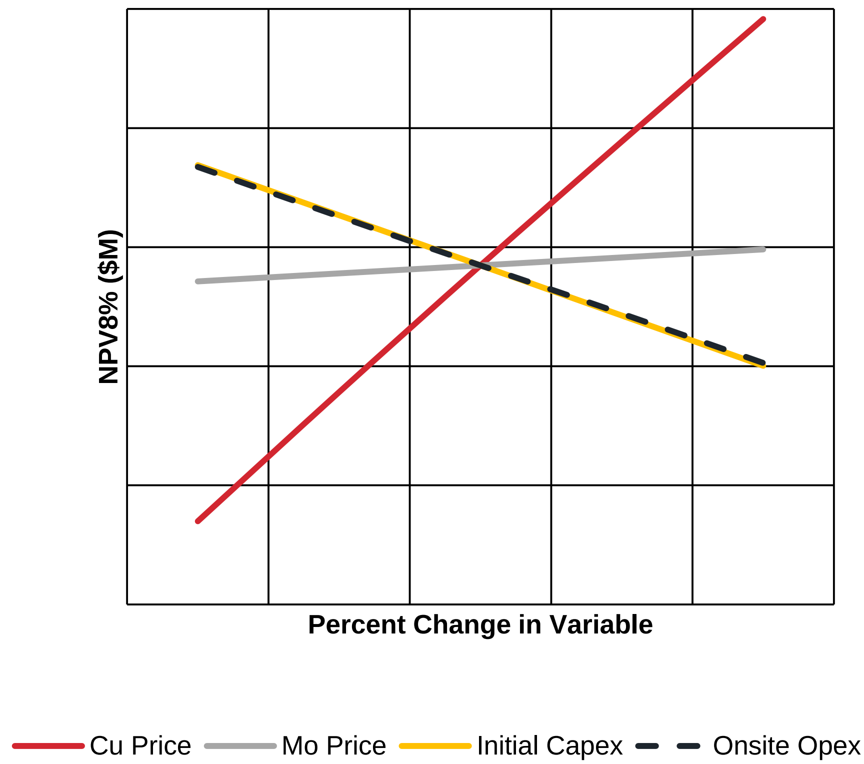



Table 22-7 below reports the after-tax NPV8%, NPV10%, IRR and payback of the Project (on a 100% basis) at various flat copper prices assuming all other inputs remain constant.

**TABLE 22-7: AFTER-TAX NPV8%, NPV10% AND IRR SENSITIVITY AT VARIOUS FLAT COPPER PRICES (100% BASIS)**

| | Flat Copper Price ($/lb) | | | | |
|---|---|---|---|---|---|
| | $2.50 | $2.75 | **$3.00** | $3.25 | $3.50 |
| After-Tax NPV8% ($M) | $45 | $412 | **$769** | $1,115 | $1,448 |
| After-Tax NPV10% ($M) | ($122) | $192 | **$496** | $792 | $1,076 |
| After-Tax IRR (%) | 8.5% | 12.2% | **15.5%** | 18.5% | 21.2% |
| After-Tax Payback (years) | 6.9 | 5.9 | **5.2** | 4.4 | 4.3 |

## 22.5 Project Ownership Impact on Valuation

The existing Joint Venture Agreement requires cash payments from UCM totaling $106 million to the JV in order for UCM to complete its earn-in for 20% ownership of the Project. The payments will be made on an installment basis to fund the initial development capital, and payments will commence once certain milestones are achieved. The NPV attributable to Hudbay is improved beyond 80% of the standalone Project NPV due to the JV payments, and the IRR attributable to Hudbay is improved beyond the standalone Project IRR as a result of the reduced time period between development capital spending and positive Project cash flow. Table 22-8 shows the adjusted key financial metrics attributable to Hudbay.

**TABLE 22-8: KEY FINANCIAL METRICS ATTRIBUTABLE TO HUDBAY**

| | Units | LOM Total |
|---|---|---|
| Development Capital (100% Basis) | $M | $1,921 |
| Stream Upfront Payment | $M | ($230) |
| Joint Venture Earn-in Payment | $M | ($106) |
| JV Share of Remaining Capital (20%) | $M | ($317) |
| JV Loan Repayment to Hudbay[1] | $M | ($20) |
| **Hudbay's Share of Development Capital** | **$M** | **$1,248** |
| After-Tax NPV8% to Hudbay | $M | $719 |
| After-Tax NPV10% to Hudbay | $M | $499 |
| After-Tax IRR to Hudbay | % | 17.7% |
| After-Tax Payback Period to Hudbay | Years | 4.9 |

1. Hudbay is funding the JV's share of project expenditures until receipt of material permits and approximately $20M in principal and accrued interest is due to Hudbay.

# 23 ADJACENT PROPERTIES

The author is not aware of any relevant work on properties immediately adjacent to the Project.

# 24 OTHER RELEVANT DATA AND INFORMATION

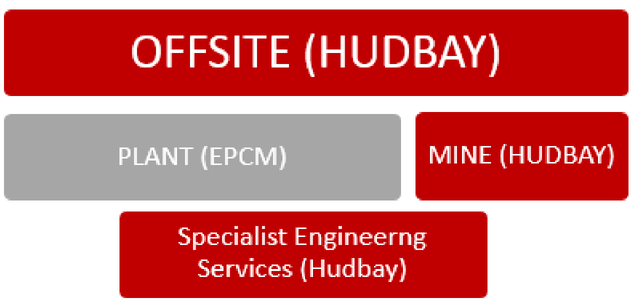
A unpublished draft feasibility study was completed for the Project which included information on the basis of design, infrastructure, design strategies, Project Execution Strategy, risks assessments and recommendations. The EPCM team has also completed a draft construction execution plan.

## 24.1 Project Implementation

A draft project execution plan was delivered to Hudbay as part of the draft DFS. The Project will be executed following a classical “EPCM” (Engineering, Procurement and Construction Management) model, which is an industry standard for projects of the magnitude and complexity of the Project. Figure 24-1 describes the Project delivery strategy. The project execution strategy will remain preliminary until the Project is funded and approved to begin after which Hudbay’s Project Team will be developed, roles defined, positions filled and strategies confirmed or modified.

Hudbay’s Project Technical Services is self-delivering the mine development scope for the Project and Hudbay’s Project Team is responsible for the offsite facilities and for the management of the specialist engineering and consulting services providers. The EPCM services provider is responsible for the process plant and related facilities, as well as for the overall project construction management and Health Safety Environmental Community (“HSEC”).

**FIGURE 24-1: OVERALL CONSTRUCTION MANAGEMENT & HSEC (EPCM)**




The general intent of Hudbay’s Project Team is to provide managerial and technical resources to safely and responsibly manage and control scope, cost, schedule and the quality of the work in the field. The Hudbay Project Team has been assigned responsibility for the management of all aspects of engineering, procurement and construction, with oversight in compliance from the Arizona Business Unit leadership. All communications, coordination and control of services providers and contractors are directed through Hudbay’s Project Team.

The roles and responsibilities for the Hudbay’s Project Management group are detailed below.

### 24.1.1 Project Director

The Project Director has the overall responsibility for meeting the Owner’s requirements and completing the Project within budget and on schedule.

#### 24.1.2 Area Managers

There are four Area Managers reporting to the Project Director who are accountable for the Project Management of specific scope including:

- Process Plant and Infrastructure
- Mining
- Heavy Civil Works
- Offsite Infrastructure

#### 24.1.3 Functional Managers

The Project Director and Area Managers are supported by functional managers and teams in the areas of:

- Project Services including cost control, administration and document control
- Project Planning and Scheduling
- Commercial including Risk, Procurement and Contracts
- Engineering
- Systems Integration
- Operational Readiness

#### 24.1.4 Hudbay's Rosemont Project Internal Stakeholders

Hudbay's Project internal stakeholders include the Project Services group, Business Development, Social Responsibility, Environmental, Finance, Legal, Internal Audit and Technical Services Teams.

#### 24.1.5 Project Sponsors Team

The purpose of the Sponsors team is to remove barriers to success, provide guidance and direction and facilitate the alignment of the integrated team, providing a culture driven to meet the project requirement and to celebrate the project successes.

#### 24.1.6 EPCM Services Provider

EPCM scope is principally the detail engineering of the Process Plant and related facilities, from the dump pocket of the primary crusher to the discharge of tailing conveying systems at the tailing management facility. The scope also includes all water piping and pumping systems, as well as infrastructure such as distribution power and communication. On the concentrate side, the scope extends to the discharge of the concentrate filters, from which point the concentrate will be loaded onto trucks using front-end loaders for shipment.

Procurement scope includes all of the major equipment packages plus supply of major bulk commodities such as structural steel, pipe electrical cable and cable ladder. These will be free-issued to the Contractors for installation. Contract packages prepared by EPCM service provider will detail the scope of construction work by package, as well as applicable interfaces and battery limits including responsibility for traffic and logistics, receipt, storage and issue of provided materials and equipment.

The EPCM service provider has responsibility for construction management for the overall Project. Other service providers will provide services in general support of the CM effort. An example is a specialty civil engineering firm who will provide field quality assurance services for Heavy Civil Works ("HCW") projects.

Through the process of advancing the Project to its current state, the EPCM scope also includes bringing in management systems and oversight to ensure delivery of a project that meets Project business objectives and commitments. During the course of the work, Hudbay's Project Team will monitor and manage EPCM's performance in this respect.

#### 24.1.7 Project Execution Plan

Various plans have been preliminarily developed to be included in the overall Project Execution Plan such as:

- Project Delivery Strategy
- Health, Safety, Environmental and Community Plans – to include implementation of systems, orientations, security, badging, transportation, emergency response, solid waste removal, compliance inspections, site specific programs, drug/alcohol testing,
- Construction Management Environmental Plan – The plan is based on compliance with the Arizona Business Unit's Environmental Plan and details implementation of management systems.
- Engineering Execution Plan – Details strategy, processes and standards for delivery of the Engineering deliverables to ensure HSEC, schedule, cost and quality objectives.
- Procurement Management Plan – The main objective is to purchase the equipment and bulk materials required for the Project and manage risks. It will detail out sole source tender procedures, bidding, supplier relations and supply chain management, packaging plan, bulks, and procurement management software. Included in this plan is a sub-plan for materials and equipment management.
- Contracting Management Plan – The Contracting Strategy has been developed considering the strengths of local Contractors and their ability to provide skilled labor, engineered and bulk materials such as earthworks and concrete and in some instances equipment such as concrete batch plants, as an example. In general, horizontal packages favoring Contractor's strengths have been preferred over vertical packages where a single Contractor coordinates

the work of numerous subcontractors or self performs work that is not a proven core capability.

- Construction Execution Plan – The Construction Execution Plan is developed to safely deliver a quality finished product in a cost effective and timely manner. The Project will be carried out in accordance with EPCM and Hudbay's Rosemont Copper Construction Management and HSEC Management Policies to maximise construction efficiency, do no harm to the health and safety of the Project team and minimize impact on the environment and surrounding community.

## 24.2 Risk Assessments

The Project has undergone various risk assessments and workshops during the years. These risk assessments were mainly isolated to project specific scopes, therefore, facilitated and maintained by Hudbay's engineering consultant firms.

**TABLE 24-1: RISK ASSESSMENTS**

| Document | Date | Authoring Entity |
|---|---|---|
| Risk Model Workshop Results | 25 June 2015 | M3/Amec |
| Project Risk Assessment from Geological Point of View, Rev 1 | 16 July 2015 | Knight Piésold |
| Hazard Identification Review Results | 3-4 Nov. 2015 | Ausenco |
| Project Risk Register | 11 March 2016 | Ausenco |

During June 2016, the Project team took ownership of the risk assessment and conducted an internal facilitated workshop. The facilitator took into consideration the previous assessments, interviews with Hudbay personnel, and a review of documentation (draft DFS, schedule, etc.) to compile a list of underlying assumptions. This list served as the basis for potential risk which were discussed, quantified/measured, and to some extent mitigated.

Following up quarterly with the risk assessments, a second session occurred in November 2016.

# 25 INTERPRETATION AND CONCLUSIONS

The purpose of this Technical Report is to present Hudbay's estimate of the mineral reserves and mineral resources for the Project based on the current mine plan, the current state of metallurgical testing, operating cost and capital cost estimates. This Technical Report includes refinements of certain aspects of the Project's mine plan. While consistency with issued and pending environmental permits and analysis related thereto has always been a key requirement for this effort, updates to the original mine plan will be necessary. To the extent that any regulatory agency concludes that the current plan requires additional environmental analysis or modification of an existing permit, the intent will be to work with that agency to either complete the required process or to adjust the current mine plan as necessary.

The results of "feasibility study" level work conducted partly by external contractors and partly internally by Hudbay, completion of drill program and bench marking including Hudbay's Constancia mine has resulted in the following fundamental conclusions:

- The Rosemont deposit consists of copper-molybdenum-silver-gold mineralization primarily hosted in skarn formed on a chemical/siliciclastic sedimentary sequence after the intrusion of Laramide quartz monzonite porphyry intrusions.
- A new geological model was built based on chemostratigraphy and lithogeochemistry using ICP multi-element assays (4 acid digestion) from 33,000 samples covering the full footage of the 2014 and 2015 Hudbay drilling programs. Geochemical based geological model reduces uncertainties in the formational, lithological and alteration logging. The updated geological model incorporated a revised structural framework based on a surface and downhole structural review.
- The resource economic parameters utilized are slightly different than those supporting the mineral reserve statement. The cost and price inputs for the mineral resource economic parameters are considered to represent reasonable estimates and were used to test the economic viability of the resource. Although these cost and prices differ from the ones used for the mineral reserves, it is the opinion of the QP that changing the resource parameters would not materially change the output of the reserve.
- A proven and probable reserve of 592 million tons has been identified and its economic viability demonstrated. An additional 591 million tons of measured and indicated resources and 69 million tons of inferred resources have been identified and outlined as having further potential for economic extraction once the mineral reserves have been extracted.
- Mining equipment performance and maintenance requirements and costs are benchmarked from Constancia's actual operating information and are robust.
- Metallurgical testwork has confirmed that the Rosemont ores respond well to proven and widely used sulfide mineral processing techniques.
- The tailing properties have been sufficiently characterized as well as the dewatering performance of vendor equipment over the life of the operation to satisfy the estimated number, type and size of tailing filters for this Project. To be conservative, expansion space has been allocated for additional filtering equipment, to the extent that it may be necessary.

- The Project process plant design has been modelled after the operating Constancia processing plant's flow sheet and is sufficiently robust to routinely achieve key production targets such as throughput, recovery, and concentrate grade as stated in the production schedule, based on the metallurgical testing conducted.
- Flexibility exists within the mine plan to optimize plant recovery and performance through the management of feed types including clays, oxides and hardness.
- The Project is one of many large projects scheduled to be constructed. The author believes that schedule slip will be the principal pressure on cost should the Project experience construction delays. In the opinion of the author, the contingency allotted to construction capital cost should mitigate most cost risk. At the time of publication of this Technical Report, committed and spent dollars would raise this contingency to approximately 15% of the required project capital as estimated in the dDFS.
- Related to the capital costs, M3 Engineering and Amec Foster Wheeler (acting under a Joint Venture agreement) and Ausenco each completed a value engineering phase and independently produced capital estimates for the project which were within 5% of each other. Since then, Ausenco has also completed a feasibility quality capital estimate, benchmarked their findings with Constancia (and other similar projects), and engaged third party construction contractors and various consulting firms to provide additional input into the estimate. Hudbay has also completed an independent third party review of the feasibility study estimate.
- The net present value of the project is most sensitive to the price of copper. The resulting project NPV8% ($769 million) and IRR (15.5%) utilizing the current Hudbay long term view on metal prices, TCRC's and other economic assumptions, in the opinion of the author, support the declaration of mineral reserves as outlined in the CIM guidelines.
- The project execution plan is modelled after the Constancia delivery method. Many key personnel who developed the Constancia mine are members of the Project who will be involved in the development of the Project and therefore considered a robust project execution plan.

The Project is uniquely located in a copper mining jurisdiction that has sustained economic copper production for close to 140 years. Since it is located approximately 30 miles from Tucson, it is expected to have a significant impact on employment and economic gain for the region. The proposed mining, processing, and logistics plan provides a step forward in innovation and sustainability. The dry stack tailings deposition proposed would be among the largest in size and address industry and stakeholder concerns regarding the use of water and the stability of tailings impoundment facilities. The proposed design and operating practice that will be applied in respect of the Project is expected to set a new standard by which other large mining projects are judged with respect to their impact on stakeholders, the ecology and the environment.

In recognition of the scarcity of world economic copper reserves in an environment of ever increasing consumption of the metal, Hudbay has carefully considered the ecological, environmental, and ethical extraction methods to be applied to the Project in an effort to set it apart from others in the world. This Project located in a first world leading nation, where extraction and production is governed by laws with due process and human rights fundamental to the consumer.

This Technical Report also concludes that the estimated mineral reserves and mineral resources for the Project conform to the requirements of 2014 CIM Definition Standards – for Mineral Resources and Mineral Reserves and requirements in Form 43-101F1 of NI 43-101, Standards of Disclosure.

# 26 RECOMMENDATIONS

The Author recommends the following:

- The author recommends that Hudbay further investigate the cause(s) of the differences in average molybdenum grade from the historical assays. Hudbay should also evaluate the application of non-linear interpolation or wireframing methods in the minor geological units.
- Consideration of a program of drill hole twinning of the pre-Augusta drilling to bolster the confidence of the re-assaying campaign as conducted by Augusta.
- Further investigate change-of-support correction and alternative approaches to resource classification taking into account the high production rate. This should be performed to ensure that the resource classification properly reflects the reduced risk when a large volume is mined and delivered to the mill on a quarterly and annual basis.
- It is recommended by the author that 5% of the samples should be sent for check assay in future drill hole campaigns.
- Future drilling campaigns should consider the by-product credit contribution of gold noted contained in copper concentrate produced through testing and increase the confidence in geological continuity such that it can be included in the reserve statement.
- Metallurgical testwork has confirmed that marketable copper concentrates can be produced. Mine and mill production sequencing and planning will require care to manage clay content that can adversely affect flotation and tailings filtration.
- A geometallurgical program is recommended as a component of the operating plan to further refine, monitor and optimize the mine to mill performance. It is concluded that fluorine can be readily rejected from copper concentrate, and further study is recommended to develop understanding of ore conditions and indicators that trigger elevated fluorine content in concentrate.
- Ongoing optimization of pit slope designs should be conducted during operation accounting for more detailed mapping of local alterations, jointing, and faulting. These observations and compilation can then provide the basis for revised slope geometry and pushback (mining phase) configurations that have the potential to increase mineral reserves and reduce overall stripping requirements.

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# 28 SIGNATURE PAGE

This NI 43-101 technical report titled “Mineral Resource Estimate, Rosemont Project, Pima County, Arizona, USA”, dated and effective as of March 30, 2017 was prepared and signed by the following author:

Dated this 30th day of March, 2017.

(signed) Cashel Meagher

_________________________Signature of Qualified Person

Cashel Meagher, P.Geo. Senior Vice President & Chief Operating Officer, Hudbay

# 29 CERTIFICATES OF QUALIFIED PERSONS

## CASHEL MEAGHER

## CERTIFICATE OF QUALIFICATION

## Re: Rosemont Project Technical Report, March 30, 2017

I, Cashel Meagher, B. Sc., P. Geo, of Toronto, Canada, do hereby certify that:

1. I am currently employed as Senior Vice President and Chief Operating Officer, with Hudbay Minerals Inc., 25 York St, Suite 800, Toronto Ontario.
2. I graduated from Saint Francis Xavier University with a Joint Advanced major in Geology and Chemistry in 1994.
3. I am a member in good standing with the Association of Professional Geoscientists of Ontario, member #1056.
4. I have practiced my profession continuously over 20 years and have been involved in mineral exploration, project evaluation, resource and reserve evaluation, and mine operations in underground and open pit mines for base metal and precious metal deposits in North and South America.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education and affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purpose of NI 43-101.
6. I have reviewed and approved the Summary of the Technical Report and I am responsible for the preparation of this Technical Report titled “NI 43-101 Technical Report, Rosemont Project, Pima County, Arizona, USA” (the “Technical Report”), dated March 30, 2017.
7. I last visited the property on April 21, 2016. I also visited it several times prior to that date.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. 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 supervision of an independent qualified person”.
10. I have been directly involved with the Rosemont Project property, which is the subject of the Technical Report, continuously since January, 2016.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument and form.
12. 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 as of March 30, 2017.

*Original signed by:*

Cashel Meagher, P.Geo.

Senior Vice President & Chief Operating Officer, Hudbay

# 30 A1-1 LAND TENURE

The following tables identify the patented claims, unpatented claims, and fee owned associated lands.

# A1-2 Rosemont Project Patented Claims

| COUNT | PARCEL NO. | PROPERTY NAME | SECTION-TOWNSHIP-RANGE | ASSESSED ACRES | 2015 FEES |
|---|---|---|---|---|---|
| 1 | 305540020 | BLACK BESS | 13-18-15 | 13.540 | $12.21 |
| 2 | 305540030 | FLYING DUTCHMAN | 13-18-15 | 20.380 | $12.21 |
| 3 | 305540040 | WISCONSIN | 13-18-15 | 20.660 | $12.21 |
| 4 | 305540050 | EXCHANGE | 13-18-15 | 20.660 | $12.21 |
| 5 | 305540060 | EXCHANGE NO. 2 | 13-18-15 | 6.590 | $12.21 |
| 6 | 305540070 | COPPER WORLD | 13-18-15 | 20.660 | $12.21 |
| 7 | 305540080 | OWOSKO | 13-18-15 | 20.660 | $12.21 |
| 8 | 305540090 | BLACK HORSE | 13-18-15 | 13.810 | $12.21 |
| 9 | 305540100 | BRUNSWICK | 13-18-15 | 18.660 | $12.21 |
| 10 | 305540110 | ANTELOPE | 13-18-15 | 17.360 | $12.21 |
| 11 | 305550010 | NEWMAN | 14-18-15 | 16.500 | $12.21 |
| 12 | 305550040 | CHANCE | 14-18-15 | 20.160 | $12.21 |
| 13 | 305550050 | BLACK HAWK | 14-18-15 | 11.360 | $12.21 |
| 14 | 305550060 | TELEMETER | 14-18-15 | 8.150 | $12.21 |
| 15 | 305550070 | WEST END | 14-18-15 | 19.530 | $12.21 |
| 16 | 305550080 | HATTIE | 14-18-15 | 12.190 | $12.21 |
| 17 | 305550090 | SILVER SPUR | 14-18-15 | 8.610 | $12.21 |
| 18 | 305550100 | SLIDE | 14-18-15 | 12.880 | $12.21 |
| 19 | 305550110 | BACK BONE | 14-18-15 | 19.070 | $12.21 |
| 20 | 305550130 | BUZZARD | 14-18-15 | 20.660 | $12.21 |
| 21 | 305550140 | HEAVY WEIGHT | 14-18-15 | 20.660 | $12.21 |
| 22 | 305550150 | LIGHT WEIGHT | 14-18-15 | 20.660 | $12.21 |
| 23 | 305560040 | PEACH | 15-18-15 | 18.070 | $12.21 |
| 24 | 305560050 | SOUTH END | 15-18-15 | 17.810 | $12.21 |
| 25 | 305560060 | MONITOR | 15-18-15 | 13.320 | $12.21 |
| 26 | 305560070 | GAP | 15-18-15 | 16.250 | $12.21 |
| 27 | 305580080 | WATER WISH | 23-18-15 | 20.660 | $12.21 |
| 28 | 305580090 | NEW MEXICO | 23-18-15 | 15.130 | $12.21 |
| 29 | 305580100 | GRIZZLY | 23-18-15 | 20.660 | $12.21 |
| 30 | 305580110 | OLD DICK | 23-18-15 | 20.130 | $12.21 |
| 31 | 305580120 | AMERICAN | 23-18-15 | 20.100 | $12.21 |
| 32 | 305580130 | RECORDER | 23-18-15 | 6.700 | $12.21 |
| 33 | 305580140 | MOHAWK | 23-18-15 | 13.550 | $12.21 |
| 34 | 305580150 | WEDGE | 23-18-15 | 19.310 | $12.21 |
| 35 | 305580160 | DAN | 23-18-15 | 2.480 | $12.21 |
| 36 | 305580170 | GENERAL | 23-18-15 | 9.170 | $12.21 |
| 37 | 305580180 | ELGIN | 23-18-15 | 14.000 | $12.21 |
| 38 | 305580190 | SUNSETE | 23-18-15 | 0.667 | $12.21 |
| 39 | 305580200 | TELEPHONE | 23-18-15 | 18.660 | $12.21 |
| 40 | 305580220 | ELGIN MILLSITE | 23-18-15 | 4.994 | $12.21 |
| 41 | 305580250 | DAN MILLSITE | 23-18-15 | 2.856 | $12.21 |
| 42 | 305580260 | WEDGE MILLSITE | 23-18-15 | 4.987 | $12.21 |
| 43 | 305580270 | OLD DICK MILLSITE | 23-18-15 | 2.196 | $12.21 |
| 44 | 305590060 | ARCOLA | 24-18-15 | 20.660 | $12.21 |
| 45 | 305590070 | BONNIE BLUE | 24-18-15 | 20.660 | $12.21 |
| 46 | 305590080 | KING | 24-18-15 | 20.660 | $12.21 |
| 47 | 305590090 | EXILE | 24-18-15 | 16.020 | $12.21 |
| 48 | 305590100 | VULTURE | 24-18-15 | 15.730 | $12.21 |
| 49 | 305590110 | ISLE ROYAL | 24-18-15 | 20.660 | $12.21 |
| 50 | 305590120 | INDIAN CLUB | 24-18-15 | 19.200 | $12.21 |
| 51 | 305590130 | A.O.T. | 24-18-15 | 14.200 | $12.21 |
| 52 | 305590140 | BALTIMORE | 24-18-15 | 9.620 | $12.21 |
| 53 | 305590150 | PILOT | 24-18-15 | 14.700 | $12.21 |
| 54 | 305590160 | LITTLE DAVE | 24-18-15 | 20.660 | $12.21 |
| 55 | 305590170 | COPPER FEND | 24-18-15 | 20.660 | $12.21 |

| COUNT | PARCEL NO. | PROPERTY NAME | SECTION-TOWNSHIP-RANGE | ASSESSED ACRES | 2015 FEES |
|---|---|---|---|---|---|
| 56 | 305590180 | TALLY HO | 24-18-15 | 20.380 | $12.21 |
| 57 | 305590190 | LEADER | 24-18-15 | 20.660 | $12.21 |
| 58 | 305590200 | OMEGA | 24-18-15 | 20.660 | $12.21 |
| 59 | 305590220 | ECLIPSE COPPER | 24-18-15 | 20.660 | $12.21 |
| 60 | 305590230 | SCHWAB | 24-18-15 | 9.261 | $12.21 |
| 61 | 305590240 | NARRAGANSETT BAY | 24-18-15 | 12.428 | $12.21 |
| 62 | 305590250 | LANDOR | 24-18-15 | 15.870 | $12.21 |
| 63 | 305590260 | WARD | 24-18-15 | 17.693 | $12.21 |
| 64 | 305590270 | ALTA COPPER | 24-18-15 | 18.180 | $12.21 |
| 65 | 305590280 | BROAD TOP | 24-18-15 | 17.150 | $12.21 |
| 66 | 305590290 | MALACHITE | 24-18-15 | 20.660 | $12.21 |
| 67 | 305600040 | YORK | 25-18-15 | 13.380 | $12.21 |
| 68 | 305600050 | OLCOTT | 25-18-15 | 5.485 | $12.21 |
| 69 | 305600060 | HILO CONSOLIDATED | 25-18-15 | 12.190 | $12.21 |
| 70 | 305600070 | ELDON | 25-18-15 | 18.984 | $12.21 |
| 71 | 305600080 | RAINBOW | 25-18-15 | 7.765 | $12.21 |
| 72 | 305600090 | AJAX CONSOLIDATED | 25-18-15 | 13.980 | $12.21 |
| 73 | 305600100 | CUBA | 25-18-15 | 12.030 | $12.21 |
| 74 | 305600110 | FALLS | 25-18-15 | 16.340 | $12.21 |
| 75 | 305600130 | OLD PUT CON | 25-18-15 | 20.650 | $12.21 |
| 76 | 305600140 | FRANKLIN | 25-18-15 | 20.540 | $12.21 |
| 77 | 305600150 | CUSHING | 25-18-15 | 15.040 | $12.21 |
| 78 | 305600160 | CENTRAL | 25-18-15 | 17.860 | $12.21 |
| 79 | 305600170 | POTOMAC | 25-18-15 | 20.620 | $12.21 |
| 80 | 305610010 | MARION | 36-18-15 | 20.660 | $12.21 |
| 81 | 305610030 | EXCELSIOR | 36-18-15 | 20.575 | $12.21 |
| 82 | 305610040 | EMPIRE | 36-18-15 | 10.210 | $12.21 |
| 83 | 305610050 | ALTAMONT | 36-18-15 | 20.610 | $12.21 |
| 84 | 305610060 | ERIE | 36-18-15 | 19.610 | $12.21 |
| 85 | 305610080 | CHICAGO | 36-18-15 | 16.660 | $12.21 |
| 86 | 305610090 | COCONINO | 36-18-15 | 14.100 | $12.21 |
| 87 | 305630020 | OLUSTEE | 19-18-16 | 20.520 | $12.00 |
| 88 | 305630040 | AMOLE | 19-18-16 | 17.921 | $12.00 |
| 89 | 305640020 | CHICAGO MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 90 | 305640030 | COCONINO MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 91 | 305640040 | OLD PUT MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 92 | 305640050 | OREGON MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 93 | 305640060 | OLD PAP MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 94 | 305640070 | AJAX CONSOLIDATED MILLSITE | 29-18-16 | 5.000 | $13.70 |
| 95 | 305650020 | R. G. INGERSOLL | 30-18-16 | 20.620 | $12.00 |
| 96 | 305650040 | PATRICK HENRY | 30-18-16 | 19.050 | $12.00 |
| 97 | 305660050 | MOHAWK SILVER | 01-19-15 | 19.760 | $7.89 |
| 98 | 305660060 | TREMONT | 01-19-15 | 12.860 | $7.89 |
| 99 | 30554012A | BLUE POINT | 13-18-15 | 19.288 | $12.21 |
| 100 | 30555012A | HEAVY WEIGHT MILLSITE | 14-18-15 | 5.000 | $12.21 |
| 101A | 30558021A | TELEPHONE MILLSITE | 23-18-15 | 4.610 | $12.21 |
| 102 | 30558023A | RECORDER MILLSITE | 23-18-15 | 2.640 | $12.21 |
| 101B | 30558023B | TELEPHONE, RECORDER & AMERICAN MILLSITE | 23-18-15 | 3.830 | $12.21 |
| 103 | 30558024A | AMERICAN MILLSITE | 23-18-15 | 4.540 | $12.21 |
| 104 | 30559021A | OMEGA FIRST EXTENSION SOUTH | 24-18-15 | 20.660 | $12.21 |
| 105A | 30560003A | DAYLIGHT | 25-18-15 | 13.210 | $12.21 |
| 105B | 30560003B | DAYLIGHT | 30-18-16 | 5.960 | $12.21 |
| 106 | 30560012A | OLD PAP COPPER | 25-18-15 | 20.650 | $12.21 |
| 107 | 30560012D | FALLS NO. 2 | 25-18-15 | 7.320 | $12.21 |
| 108 | 30560012F | WEDGE NO. 2 | 25-18-15 | 1.280 | $12.21 |

| COUNT | PARCEL NO. | PROPERTY NAME | SECTION-TOWNSHIP-RANGE | ASSESSED ACRES | 2015 FEES |
|---|---|---|---|---|---|
| 109 | 30560012G | WEDGE | 25-18-15 | 6.600 | $12.21 |
| 110 | 30560012H | SANTA RITA FRACTION | 25-18-15 | 0.980 | $12.21 |
| 111A | 30560012J | SANTA RITA #13 | 25-18-15 | 10.520 | $12.21 |
| 112 | 30561007A | OREGON COPPER | 36-18-15 | 16.080 | $12.21 |
| 113A | 30561007D | SANTA RITA #15 | 36-18-15 | 13.590 | $12.21 |
| 114 | 30561007E | SANTA RITA #14 | 36-18-15 | 19.160 | $12.21 |
| 115 | 30561007F | SANTA RITA #12 | 36-18-15 | 19.620 | $12.21 |
| 116 | 30561007G | LAST CHANCE NO. 1 | 36-18-15 | 15.600 | $12.21 |
| 117 | 30561007H | LAST CHANCE NO. 2 | 36-18-15 | 18.270 | $12.21 |
| 118 | 30561007J | SANTA RITA #26 | 36-18-15 | 20.030 | $12.21 |
| 119 | 30561007K | SANTA RITA #27 | 36-18-15 | 18.760 | $12.21 |
| 120A | 30561007L | SANTA RITA #28 | 36-18-15 | 18.570 | $12.21 |
| 121 | 30562034C | SANTA RITA #16 | 31-18-16 | 18.920 | $12.00 |
| 113B | 30562034D | SANTA RITA #15 | 31-18-16 | 6.440 | $12.00 |
| 120B | 30562034E | SANTA RITA #28 | 31-18-16 | 2.010 | $12.00 |
| 111B | 30562034F | SANTA RITA #13 | 31-18-16 | 7.510 | $12.00 |
| 122 | 30563003A | CUPRITE | 19-18-16 | 20.660 | $12.00 |
| 123 | 30564008A | FRANKLIN MILLSITE | 29-18-16 | 5.000 | $12.00 |
| 124 | 30565003A | LA FAYETTE | 30-18-16 | 13.950 | $12.00 |
| 125 | 30565003D | SANTA RITA #4 | 30-18-16 | 19.000 | $12.00 |
| 126 | 30565003E | SANTA RITA #5 | 30-18-16 | 19.020 | $12.00 |
| 127 | 30565003F | SANTA RITA #6 | 30-18-16 | 18.990 | $12.00 |
| 128A | 30565003G | SANTA RITA #8A | 25-18-15 | 3.660 | $12.00 |
| 129A | 30565003H | SANTA RITA #9 | 30-18-16, 31-18-16 | 19.580 | $12.00 |
| 130 | 30565003J | SANTA RITA #10 | 30-18-16, 31-18-16 | 20.560 | $12.00 |
| 131 | 30565003K | SANTA RITA #11 | 31-18-16 | 20.560 | $12.00 |
| 128B | 30565003L | SANTA RITA #8A | 25-18-15,(S/B 30-18-16) | 10.750 | $12.00 |
| 129B | 30565003M | SANTA RITA #9 | 25-18-15 | 1.020 | $12.00 |
| 132A | 30565005A | DAN WEBSTER | 30-18-16 | 15.190 | $12.00 |
| 132B | 30565005B | DAN WEBSTER | 25-18-15 | 3.770 | $12.00 |
| **PATENTED CLAIM TOTALS** | | | | **2003.520** | **$1,705.08** |

# A1-3 Rosemont Project Unpatented Claims

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 1 | York Fraction | AMC2198 | $155.00 |
| 2 | Travis #1 | AMC2199 | $155.00 |
| 3 | Jim | AMC2200 | $155.00 |
| 4 | Isle Royal Fraction | AMC2201 | $155.00 |
| 5 | Indian Club Fraction | AMC2202 | $155.00 |
| 6 | Pilot Fraction | AMC2203 | $155.00 |
| 7 | A.O.T. Fraction | AMC2204 | $155.00 |
| 8 | Malachite Fraction | AMC2211 | $155.00 |
| 9 | MAX 121 B/Relocation | AMC13284 | $155.00 |
| 10 | MAX 123 B/Relocation | AMC13286 | $155.00 |
| 11 | MAX 125 B/Relocation | AMC13288 | $155.00 |
| 12 | MAX 126 B/Relocation | AMC13289 | $155.00 |
| 13 | MAX 127 B/Relocation | AMC13290 | $155.00 |
| 14 | MAX 128 B/Relocation | AMC13291 | $155.00 |
| 15 | MAX 129 B/Relocation | AMC13292 | $155.00 |
| 16 | MAX 130 B/Relocation | AMC13293 | $155.00 |
| 17 | MAX 131 B/Relocation | AMC13294 | $155.00 |
| 18 | MAX 132 B/Relocation | AMC13295 | $155.00 |
| 19 | MAX 133 B/Relocation | AMC13296 | $155.00 |
| 20 | MAX 134 B/Relocation | AMC13297 | $155.00 |
| 21 | MAX 135 B/Relocation | AMC13298 | $155.00 |
| 22 | MAX 136 B/Relocation | AMC13299 | $155.00 |
| 23 | MAX 137 B/Relocation | AMC13300 | $155.00 |
| 24 | MAX 138 B/Relocation | AMC13301 | $155.00 |
| 25 | MAX 139 B/Relocation | AMC13302 | $155.00 |
| 26 | MAX 140 B/Relocation | AMC13303 | $155.00 |
| 27 | MAX 141 B/Relocation | AMC13304 | $155.00 |
| 28 | MAX 142 B/Relocation | AMC13305 | $155.00 |
| 29 | MAX 143 B/Relocation | AMC13306 | $155.00 |
| 30 | MAX 144 B/Relocation | AMC13307 | $155.00 |
| 31 | MAX 145 B/Relocation | AMC13308 | $155.00 |
| 32 | MAX 146 B/Relocation | AMC13309 | $155.00 |
| 33 | MAX 147 B/Relocation | AMC13310 | $155.00 |
| 34 | MAX 148 B/Relocation | AMC13311 | $155.00 |
| 35 | MAX 149 B/Relocation | AMC13312 | $155.00 |
| 36 | MAX 150 B/Relocation | AMC13313 | $155.00 |
| 37 | MAX 151 B/Relocation | AMC13314 | $155.00 |
| 38 | MAX 152 B/Relocation | AMC13315 | $155.00 |
| 39 | MAX 153 B/Relocation | AMC13316 | $155.00 |
| 40 | MAX 154 B/Relocation | AMC13317 | $155.00 |
| 41 | MAX 155 B/Relocation | AMC13318 | $155.00 |
| 42 | MAX 156 B/Relocation | AMC13319 | $155.00 |
| 43 | Rosaland | AMC14972 | $155.00 |
| 44 | Michael M | AMC14973 | $155.00 |
| 45 | Lydia J | AMC14974 | $155.00 |
| 46 | Ida D | AMC14975 | $155.00 |
| 47 | D & D #1 | AMC14976 | $155.00 |
| 48 | D & D II | AMC14977 | $155.00 |
| 49 | Frijole | AMC14978 | $155.00 |
| 50 | Frijole II | AMC14979 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 51 | Frijole III | AMC14980 | $155.00 |
| 52 | Frijole IV | AMC14981 | $155.00 |
| 53 | Frijole V | AMC14982 | $155.00 |
| 54 | Frijole VII | AMC14984 | $155.00 |
| 55 | Frijole VIII | AMC14985 | $155.00 |
| 56 | Frijole IX | AMC14986 | $155.00 |
| 57 | Frijole X | AMC14987 | $155.00 |
| 58 | Frijole XI | AMC14988 | $155.00 |
| 59 | Frijole XI Extension | AMC14989 | $155.00 |
| 60 | Deering Springs No. 2 A/Relocation | AMC15002 | $155.00 |
| 61 | Deering Springs No. 4 A/Relocation | AMC15003 | $155.00 |
| 62 | Deering Springs No. 6 A/Relocation | AMC15004 | $155.00 |
| 63 | Deering Springs No. 8 A/Relocation | AMC15005 | $155.00 |
| 64 | Deering Springs No. 10 A/Relocation | AMC15006 | $155.00 |
| 65 | Deering Springs No. 12 A/Relocation | AMC15007 | $155.00 |
| 66 | Deering Springs No. 14 A/Relocation | AMC15008 | $155.00 |
| 67 | Deering Springs No. 15 A/Relocation | AMC15009 | $155.00 |
| 68 | Deering Springs No. 16 A/Relocation | AMC15010 | $155.00 |
| 69 | Deering Springs No. 17 A/Relocation | AMC15011 | $155.00 |
| 70 | Deering Springs No. 21 A/Relocation | AMC15012 | $155.00 |
| 71 | Deering Springs No. 22 A/Relocation | AMC15013 | $155.00 |
| 72 | Deering Springs No. 23 A/Relocation | AMC15014 | $155.00 |
| 73 | Deering Springs No. 24 A/Relocation | AMC15015 | $155.00 |
| 74 | Deering Springs No. 25 A/Relocation | AMC15016 | $155.00 |
| 75 | Deering Springs No. 26 A/Relocation | AMC15017 | $155.00 |
| 76 | Deering Springs No. 27 A/Relocation | AMC15018 | $155.00 |
| 77 | Deering Springs No. 28 A/Relocation | AMC15019 | $155.00 |
| 78 | Deering Springs No. 29 A/Relocation | AMC15020 | $155.00 |
| 79 | Deering Springs No. 30 A/Relocation | AMC15021 | $155.00 |
| 80 | Deering Springs No. 31 A/Relocation | AMC15022 | $155.00 |
| 81 | Deering Springs No. 32 A/Relocation | AMC15023 | $155.00 |
| 82 | Deering Springs No. 33 A/Relocation | AMC15024 | $155.00 |
| 83 | Deering Springs No. 34 A/Relocation | AMC15025 | $155.00 |
| 84 | Deering Springs No. 35 A/Relocation | AMC15026 | $155.00 |
| 85 | Deering Springs No. 36 A/Relocation | AMC15027 | $155.00 |
| 86 | Deering Springs No. 37 A/Relocation | AMC15028 | $155.00 |
| 87 | Deering Springs No. 38 A/Relocation | AMC15029 | $155.00 |
| 88 | Deering Springs No. 39 A/Relocation | AMC15030 | $155.00 |
| 89 | Deering Springs No. 42 A/Relocation | AMC15031 | $155.00 |
| 90 | Deering Springs No. 51 A/Relocation | AMC15032 | $155.00 |
| 91 | Deering Springs No. 52 A/Relocation | AMC15033 | $155.00 |
| 92 | Kid 1 | AMC25210 | $155.00 |
| 93 | Kid 2 | AMC25211 | $155.00 |
| 94 | Kid 3 | AMC25212 | $155.00 |
| 95 | Kid 4 | AMC25213 | $155.00 |
| 96 | Kid 5 | AMC25214 | $155.00 |
| 97 | Kid 6 | AMC25215 | $155.00 |
| 98 | Kid 7 | AMC25216 | $155.00 |
| 99 | Kid 8 | AMC25217 | $155.00 |
| 100 | Kid 9 | AMC25218 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 101 | Kid 10 | AMC25219 | $155.00 |
| 102 | Kid 11 | AMC25220 | $155.00 |
| 103 | Kid 12 | AMC25221 | $155.00 |
| 104 | Kid 13 | AMC25222 | $155.00 |
| 105 | Kid 14 | AMC25223 | $155.00 |
| 106 | Kid 15 | AMC25224 | $155.00 |
| 107 | Kid 16 | AMC25225 | $155.00 |
| 108 | Kid 17 | AMC25226 | $155.00 |
| 109 | Kid 18 | AMC25227 | $155.00 |
| 110 | Kid 19 | AMC25228 | $155.00 |
| 111 | Kid 20 | AMC25229 | $155.00 |
| 112 | Kid 21 | AMC25230 | $155.00 |
| 113 | Kid 22 | AMC25231 | $155.00 |
| 114 | Kid 23 | AMC25232 | $155.00 |
| 115 | Kid 24 | AMC25233 | $155.00 |
| 116 | Kid 25 | AMC25234 | $155.00 |
| 117 | Kid 26 | AMC25235 | $155.00 |
| 118 | Kid 27 | AMC25236 | $155.00 |
| 119 | Kid 28 | AMC25237 | $155.00 |
| 120 | Kid 29 | AMC25238 | $155.00 |
| 121 | Kid 34 | AMC25243 | $155.00 |
| 122 | Kid 35 | AMC25244 | $155.00 |
| 123 | Kid 36 | AMC25245 | $155.00 |
| 124 | Kid 37 | AMC25246 | $155.00 |
| 125 | Kid 38 | AMC25247 | $155.00 |
| 126 | Kid 39 | AMC25248 | $155.00 |
| 127 | Kid 40 | AMC25249 | $155.00 |
| 128 | Kid 41 | AMC25250 | $155.00 |
| 129 | Kid 42 | AMC25251 | $155.00 |
| 130 | Kid 43 | AMC25252 | $155.00 |
| 131 | Kid 44 | AMC25253 | $155.00 |
| 132 | Kid 45 | AMC25254 | $155.00 |
| 133 | Kid 46 | AMC25255 | $155.00 |
| 134 | Kid 47 | AMC25256 | $155.00 |
| 135 | Wasp 52 | AMC25257 | $155.00 |
| 136 | Wasp 53 | AMC25258 | $155.00 |
| 137 | Wasp 54 | AMC25259 | $155.00 |
| 138 | Wasp 55 | AMC25260 | $155.00 |
| 139 | Wasp 56 | AMC25261 | $155.00 |
| 140 | Wasp 57 | AMC25262 | $155.00 |
| 141 | Wasp 58 | AMC25263 | $155.00 |
| 142 | Wasp 60 | AMC25264 | $155.00 |
| 143 | Wasp 61 | AMC25265 | $155.00 |
| 144 | Wasp 101 | AMC25268 | $155.00 |
| 145 | Wasp 102 | AMC25269 | $155.00 |
| 146 | Wasp 103 | AMC25270 | $155.00 |
| 147 | Wasp 104 | AMC25271 | $155.00 |
| 148 | Wasp 105 | AMC25272 | $155.00 |
| 149 | Wasp 106 | AMC25273 | $155.00 |
| 150 | Wasp 107 | AMC25274 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 151 | Wasp 111 | AMC25275 | $155.00 |
| 152 | Wasp 112 | AMC25276 | $155.00 |
| 153 | Wasp 113 | AMC25277 | $155.00 |
| 154 | Wasp 114 | AMC25278 | $155.00 |
| 155 | Wasp 115 | AMC25279 | $155.00 |
| 156 | Wasp 116 | AMC25280 | $155.00 |
| 157 | Wasp 117 | AMC25281 | $155.00 |
| 158 | Wasp 118 | AMC25282 | $155.00 |
| 159 | Wasp 119 | AMC25283 | $155.00 |
| 160 | Wasp 120 | AMC25284 | $155.00 |
| 161 | Wasp 121 | AMC25285 | $155.00 |
| 162 | Wasp 122 | AMC25286 | $155.00 |
| 163 | Wasp 123 | AMC25287 | $155.00 |
| 164 | Wasp 124 | AMC25288 | $155.00 |
| 165 | Wasp 125 | AMC25289 | $155.00 |
| 166 | Wasp 126 | AMC25290 | $155.00 |
| 167 | Wasp 127 | AMC25291 | $155.00 |
| 168 | Wasp 128 | AMC25292 | $155.00 |
| 169 | Wasp 129 | AMC25293 | $155.00 |
| 170 | Wasp 130 | AMC25294 | $155.00 |
| 171 | Wasp 201 | AMC25295 | $155.00 |
| 172 | Wasp 202 | AMC25296 | $155.00 |
| 173 | Wasp 203 | AMC25297 | $155.00 |
| 174 | Wasp 204 | AMC25298 | $155.00 |
| 175 | Wasp 205 | AMC25299 | $155.00 |
| 176 | Wasp 206 | AMC25300 | $155.00 |
| 177 | Wasp 207 | AMC25301 | $155.00 |
| 178 | Wasp 208 | AMC25302 | $155.00 |
| 179 | Wasp 209 | AMC25303 | $155.00 |
| 180 | Wasp 210 | AMC25304 | $155.00 |
| 181 | Wasp 211 | AMC25305 | $155.00 |
| 182 | Wasp 212 | AMC25306 | $155.00 |
| 183 | Wasp 213 | AMC25307 | $155.00 |
| 184 | Wasp 214 | AMC25308 | $155.00 |
| 185 | Wasp 215 | AMC25309 | $155.00 |
| 186 | Wasp 216 | AMC25310 | $155.00 |
| 187 | Wasp 217 | AMC25311 | $155.00 |
| 188 | Wasp 218 | AMC25312 | $155.00 |
| 189 | Wasp 313 | AMC25349 | $155.00 |
| 190 | Wasp 315 | AMC25351 | $155.00 |
| 191 | Wasp 317 | AMC25353 | $155.00 |
| 192 | Wasp 319 | AMC25355 | $155.00 |
| 193 | Wasp 321 | AMC25357 | $155.00 |
| 194 | Wasp 323 | AMC25359 | $155.00 |
| 195 | Wasp 325 | AMC25361 | $155.00 |
| 196 | Wasp 327 | AMC25363 | $155.00 |
| 197 | Wasp 329 | AMC25365 | $155.00 |
| 198 | Wasp 331 | AMC25367 | $155.00 |
| 199 | Wasp 333 | AMC25369 | $155.00 |
| 200 | Wasp 335 | AMC25371 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 201 | Wasp 337 | AMC25373 | $155.00 |
| 202 | Wasp 339 | AMC25375 | $155.00 |
| 203 | Wasp 341 | AMC25377 | $155.00 |
| 204 | Wasp 343 | AMC25379 | $155.00 |
| 205 | Wasp 344 | AMC25380 | $155.00 |
| 206 | Wasp 345 | AMC25381 | $155.00 |
| 207 | Wasp 346 | AMC25382 | $155.00 |
| 208 | Wasp 347 | AMC25383 | $155.00 |
| 209 | Wasp 348 | AMC25384 | $155.00 |
| 210 | Wasp 349 | AMC25385 | $155.00 |
| 211 | Wasp 350 | AMC25386 | $155.00 |
| 212 | Wasp 351 | AMC25387 | $155.00 |
| 213 | Wasp 352 | AMC25388 | $155.00 |
| 214 | Wasp 353 | AMC25389 | $155.00 |
| 215 | Wasp 354 | AMC25390 | $155.00 |
| 216 | Max 41 | AMC25662 | $155.00 |
| 217 | Max 43 | AMC25664 | $155.00 |
| 218 | Max 45 | AMC25666 | $155.00 |
| 219 | Max 47 | AMC25668 | $155.00 |
| 220 | Max 49 | AMC25670 | $155.00 |
| 221 | Max 71 | AMC25692 | $155.00 |
| 222 | Max 72 | AMC25693 | $155.00 |
| 223 | Max 73 | AMC25694 | $155.00 |
| 224 | Max 74 | AMC25695 | $155.00 |
| 225 | Max 75 | AMC25696 | $155.00 |
| 226 | Max 76 | AMC25697 | $155.00 |
| 227 | Max 77 | AMC25698 | $155.00 |
| 228 | Max 78 | AMC25699 | $155.00 |
| 229 | Max 79 | AMC25700 | $155.00 |
| 230 | Max 80 | AMC25701 | $155.00 |
| 231 | Max 81 | AMC25702 | $155.00 |
| 232 | Max 82 | AMC25703 | $155.00 |
| 233 | Max 83 | AMC25704 | $155.00 |
| 234 | Max 84 | AMC25705 | $155.00 |
| 235 | Max 85 | AMC25706 | $155.00 |
| 236 | Max 86 | AMC25707 | $155.00 |
| 237 | Max 87 | AMC25708 | $155.00 |
| 238 | Max 88 | AMC25709 | $155.00 |
| 239 | Max 89 | AMC25710 | $155.00 |
| 240 | Max 90 | AMC25711 | $155.00 |
| 241 | Max 91 | AMC25712 | $155.00 |
| 242 | Max 93 | AMC25714 | $155.00 |
| 243 | Max 95 | AMC25716 | $155.00 |
| 244 | Max 97 | AMC25718 | $155.00 |
| 245 | Max 99 | AMC25720 | $155.00 |
| 246 | Max 101 | AMC25722 | $155.00 |
| 247 | Max 102 | AMC25723 | $155.00 |
| 248 | Max 103 | AMC25724 | $155.00 |
| 249 | Max 104 | AMC25725 | $155.00 |
| 250 | Max 105 | AMC25726 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 251 | Max 106 | AMC25727 | $155.00 |
| 252 | Max 107 | AMC25728 | $155.00 |
| 253 | Max 108 | AMC25729 | $155.00 |
| 254 | Max 109 | AMC25730 | $155.00 |
| 255 | Max 110 | AMC25731 | $155.00 |
| 256 | Max 111 | AMC25732 | $155.00 |
| 257 | Max 112 | AMC25733 | $155.00 |
| 258 | Max 113 | AMC25734 | $155.00 |
| 259 | Max 114 | AMC25735 | $155.00 |
| 260 | Max 115 | AMC25736 | $155.00 |
| 261 | Max 116 | AMC25737 | $155.00 |
| 262 | Max 117 | AMC25738 | $155.00 |
| 263 | Max 118 | AMC25739 | $155.00 |
| 264 | Max 119 | AMC25740 | $155.00 |
| 265 | Max 120 | AMC25741 | $155.00 |
| 266 | Elk 1 | AMC27423 | $155.00 |
| 267 | Elk 2 | AMC27424 | $155.00 |
| 268 | Elk 3 | AMC27425 | $155.00 |
| 269 | Elk 4 | AMC27426 | $155.00 |
| 270 | Elk 5 | AMC27427 | $155.00 |
| 271 | Elk 6 | AMC27428 | $155.00 |
| 272 | Elk 35 | AMC27451 | $155.00 |
| 273 | Elk 36 | AMC27452 | $155.00 |
| 274 | Elk 37 | AMC27453 | $155.00 |
| 275 | Elk 39 | AMC27455 | $155.00 |
| 276 | Elk 41 | AMC27457 | $155.00 |
| 277 | Elk 43 | AMC27459 | $155.00 |
| 278 | Elk 45 | AMC27461 | $155.00 |
| 279 | Elk 70 | AMC27465 | $155.00 |
| 280 | Elk 71 | AMC27466 | $155.00 |
| 281 | Elk 72 | AMC27467 | $155.00 |
| 282 | Elk 73 | AMC27468 | $155.00 |
| 283 | Elk 74 | AMC27469 | $155.00 |
| 284 | Elk 75 | AMC27470 | $155.00 |
| 285 | Elk 76 | AMC27471 | $155.00 |
| 286 | Elk 77 | AMC27472 | $155.00 |
| 287 | Elk 78 | AMC27473 | $155.00 |
| 288 | Elk 79 | AMC27474 | $155.00 |
| 289 | Elk 80 | AMC27475 | $155.00 |
| 290 | Elk 81 | AMC27476 | $155.00 |
| 291 | Elk 83 | AMC27478 | $155.00 |
| 292 | Elk 85 | AMC27480 | $155.00 |
| 293 | Elk 87 | AMC27482 | $155.00 |
| 294 | Alpine #5 | AMC27513 | $155.00 |
| 295 | Alpine #6 | AMC27514 | $155.00 |
| 296 | Alpine #7 | AMC27515 | $155.00 |
| 297 | Alpine #8 | AMC27516 | $155.00 |
| 298 | Alpine #9 | AMC27517 | $155.00 |
| 299 | Alpine #10 | AMC27518 | $155.00 |
| 300 | Alpine #11 | AMC27519 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 301 | Alpine #12 | AMC27520 | $155.00 |
| 302 | Alpine #13 | AMC27521 | $155.00 |
| 303 | Alpine #14 | AMC27522 | $155.00 |
| 304 | Alpine #15 | AMC27523 | $155.00 |
| 305 | Alpine #16 | AMC27524 | $155.00 |
| 306 | Alpine #17 | AMC27525 | $155.00 |
| 307 | Alpine #18 | AMC27526 | $155.00 |
| 308 | Alpine #19 | AMC27527 | $155.00 |
| 309 | Alpine #20 | AMC27528 | $155.00 |
| 310 | Alpine #21 | AMC27529 | $155.00 |
| 311 | Alpine #22 | AMC27530 | $155.00 |
| 312 | Alpine #23 | AMC27531 | $155.00 |
| 313 | Alpine #24 | AMC27532 | $155.00 |
| 314 | Santa Rita Wedge | AMC28871 | $155.00 |
| 315 | Buzzard No. 5 | AMC36021 | $155.00 |
| 316 | Shadow #4 | AMC36025 | $155.00 |
| 317 | John 1 | AMC36026 | $155.00 |
| 318 | John 2 | AMC36027 | $155.00 |
| 319 | Flying Dutchman No. 2 | AMC36028 | $155.00 |
| 320 | Flying Dutchman No. 3 | AMC36029 | $155.00 |
| 321 | Flying Dutchman No. 4 | AMC36030 | $155.00 |
| 322 | Flying Dutchman No. 5 | AMC36031 | $155.00 |
| 323 | Flying Dutchman No. 6 | AMC36032 | $155.00 |
| 324 | Black Bess No. 2 | AMC36034 | $155.00 |
| 325 | K.W.L. | AMC36036 | $155.00 |
| 326 | G.E.J. | AMC36037 | $155.00 |
| 327 | R.F.E. | AMC36038 | $155.00 |
| 328 | R.C.M. | AMC36039 | $155.00 |
| 329 | Sycamore #1 | AMC36040 | $155.00 |
| 330 | Sycamore #2 | AMC36041 | $155.00 |
| 331 | Sycamore #3 | AMC36042 | $155.00 |
| 332 | Sycamore #4 | AMC36043 | $155.00 |
| 333 | Sycamore #5 | AMC36044 | $155.00 |
| 334 | Sycamore #6 | AMC36045 | $155.00 |
| 335 | Sycamore #7 | AMC36046 | $155.00 |
| 336 | Sycamore #8 | AMC36047 | $155.00 |
| 337 | Sycamore #9 | AMC36048 | $155.00 |
| 338 | Sycamore #10 | AMC36049 | $155.00 |
| 339 | Sycamore #11 | AMC36050 | $155.00 |
| 340 | Sycamore #12 | AMC36051 | $155.00 |
| 341 | Naragansett Extension #1 | AMC36052 | $155.00 |
| 342 | Naragansett Ext. #2 | AMC36053 | $155.00 |
| 343 | Naragansett Extension #3 | AMC36054 | $155.00 |
| 344 | Naragansett Extension #4 | AMC36055 | $155.00 |
| 345 | Naragansett Extension #5 | AMC36056 | $155.00 |
| 346 | Naragansett Extension #6 | AMC36057 | $155.00 |
| 347 | Naragansett Extension #7 | AMC36058 | $155.00 |
| 348 | Naragansett Extension #8 | AMC36059 | $155.00 |
| 349 | Narragansett Ext. No. 9 | AMC36060 | $155.00 |
| 350 | Schwab Extension #1 North West | AMC36061 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 351 | Rocky 1 | AMC36062 | $155.00 |
| 352 | Amole No. 2 | AMC36063 | $155.00 |
| 353 | Falls No. 3 | AMC36065 | $155.00 |
| 354 | Falls No. 4 | AMC36066 | $155.00 |
| 355 | Perry No. 1 | AMC36067 | $155.00 |
| 356 | Perry #2 | AMC36068 | $155.00 |
| 357 | Perry #3 | AMC36069 | $155.00 |
| 358 | Perry #4 | AMC36070 | $155.00 |
| 359 | Perry #7 | AMC36073 | $155.00 |
| 360 | Perry #8 | AMC36074 | $155.00 |
| 361 | Perry #9 | AMC36075 | $155.00 |
| 362 | Perry #10 | AMC36076 | $155.00 |
| 363 | Perry #11 | AMC36077 | $155.00 |
| 364 | Perry #12 | AMC36078 | $155.00 |
| 365 | Perry #15 | AMC36081 | $155.00 |
| 366 | Perry #16 | AMC36082 | $155.00 |
| 367 | Perry #17 | AMC36083 | $155.00 |
| 368 | Perry #18 | AMC36084 | $155.00 |
| 369 | Gunsite 1-A | AMC36086 | $155.00 |
| 370 | Gunsite No. 2 | AMC36087 | $155.00 |
| 371 | Gunsite No. 3 | AMC36088 | $155.00 |
| 372 | Gunsite No. 4 | AMC36089 | $155.00 |
| 373 | Gunsite 5A | AMC36090 | $155.00 |
| 374 | Gunsite 6-B | AMC36091 | $155.00 |
| 375 | Gunsite No. 7 | AMC36092 | $155.00 |
| 376 | Gunsite 7A | AMC36093 | $155.00 |
| 377 | Gunsite No. 8 | AMC36094 | $155.00 |
| 378 | Gunsite No. 9 | AMC36095 | $155.00 |
| 379 | Gunsite No. 10 | AMC36096 | $155.00 |
| 380 | Gunsite No. 11 | AMC36097 | $155.00 |
| 381 | Gunsite No. 12 | AMC36098 | $155.00 |
| 382 | Gunsite No. 13 | AMC36099 | $155.00 |
| 383 | Gunsite No. 14 | AMC36100 | $155.00 |
| 384 | Gunsite No. 15 | AMC36101 | $155.00 |
| 385 | Gunsite No. 16 | AMC36102 | $155.00 |
| 386 | Gunsite No. 17 | AMC36103 | $155.00 |
| 387 | Gunsite No. 18 | AMC36104 | $155.00 |
| 388 | Gunsite No. 19 | AMC36105 | $155.00 |
| 389 | Gunsite No. 20 | AMC36106 | $155.00 |
| 390 | Gunsite No. 21 | AMC36107 | $155.00 |
| 391 | Gunsite No. 22 | AMC36108 | $155.00 |
| 392 | Gunsight No. 23 | AMC36109 | $155.00 |
| 393 | Gunsite No. 24 | AMC36110 | $155.00 |
| 394 | Gunsite No. 25 | AMC36111 | $155.00 |
| 395 | Gunsite No. 26 | AMC36112 | $155.00 |
| 396 | Gunsite No. 27 | AMC36113 | $155.00 |
| 397 | Gunsight No. 28 | AMC36114 | $155.00 |
| 398 | Gunsight No. 29 | AMC36115 | $155.00 |
| 399 | Gunsight No. 30 | AMC36116 | $155.00 |
| 400 | Gunsight No. 31 | AMC36117 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 401 | Gunsight No. 32 | AMC36118 | $155.00 |
| 402 | Gunsight No. 33 | AMC36119 | $155.00 |
| 403 | Gunsight No. 35 | AMC36121 | $155.00 |
| 404 | Gunsight No. 36 | AMC36122 | $155.00 |
| 405 | Gunsight No. 37 | AMC36123 | $155.00 |
| 406 | Gunsight No. 38 | AMC36124 | $155.00 |
| 407 | Gunsight No. 39 | AMC36125 | $155.00 |
| 408 | Gunsight No. 40 | AMC36126 | $155.00 |
| 409 | Gunsight No. 41 | AMC36127 | $155.00 |
| 410 | Gunsight No. 42 | AMC36128 | $155.00 |
| 411 | Gunsight No. 43 | AMC36129 | $155.00 |
| 412 | Gunsight 44 | AMC36130 | $155.00 |
| 413 | Gunsight #45 | AMC36131 | $155.00 |
| 414 | Gunsight #46 | AMC36132 | $155.00 |
| 415 | Gunsight #47 | AMC36133 | $155.00 |
| 416 | Gunsight #48 | AMC36134 | $155.00 |
| 417 | Gunsight #49 | AMC36135 | $155.00 |
| 418 | Gunsight #50 | AMC36136 | $155.00 |
| 419 | Williams Folly | AMC36137 | $155.00 |
| 420 | Williams Folly #2 | AMC36138 | $155.00 |
| 421 | Santa Rita #1 | AMC46740 | $155.00 |
| 422 | Santa Rita #2 | AMC46741 | $155.00 |
| 423 | Santa Rita #3 | AMC46742 | $155.00 |
| 424 | Santa Rita #7 | AMC46746 | $155.00 |
| 425 | Santa Rita #17 | AMC46756 | $155.00 |
| 426 | Santa Rita #18 | AMC46757 | $155.00 |
| 427 | Santa Rita #19 | AMC46758 | $155.00 |
| 428 | Santa Rita #20 | AMC46759 | $155.00 |
| 429 | Santa Rita #21 | AMC46760 | $155.00 |
| 430 | Santa Rita #22 | AMC46761 | $155.00 |
| 431 | Santa Rita #23 | AMC46762 | $155.00 |
| 432 | Santa Rita #24 | AMC46763 | $155.00 |
| 433 | Santa Rita #25 | AMC46764 | $155.00 |
| 434 | Santa Rita #29 | AMC46768 | $155.00 |
| 435 | Santa Rita #30 | AMC46769 | $155.00 |
| 436 | Santa Rita #31 | AMC46770 | $155.00 |
| 437 | Catalina #1 | AMC46771 | $155.00 |
| 438 | Catalina #2 | AMC46772 | $155.00 |
| 439 | Catalina #3 | AMC46773 | $155.00 |
| 440 | Catalina #4 | AMC46774 | $155.00 |
| 441 | Catalina #5A | AMC46775 | $155.00 |
| 442 | Catalina #6A | AMC46776 | $155.00 |
| 443 | Catalina #7 | AMC46777 | $155.00 |
| 444 | Catalina #8 | AMC46778 | $155.00 |
| 445 | Fred Bennett | AMC46779 | $155.00 |
| 446 | Fred Bennett | AMC46780 | $155.00 |
| 447 | Rosemont #9 | AMC46781 | $155.00 |
| 448 | Rosemont #11 | AMC46782 | $155.00 |
| 449 | Rosemont 11-A | AMC46783 | $155.00 |
| 450 | Rosemont #12 | AMC46784 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 451 | Rosemont #13 | AMC46785 | $155.00 |
| 452 | Rosemont #15 | AMC46786 | $155.00 |
| 453 | Rosemont #16 | AMC46787 | $155.00 |
| 454 | Rosemont #17 | AMC46788 | $155.00 |
| 455 | Rosemont #18 | AMC46789 | $155.00 |
| 456 | Rosemont 21 | AMC46790 | $155.00 |
| 457 | Fred Bennett Fraction | AMC46791 | $155.00 |
| 458 | Last Chance No. 3/Relocation | AMC46794 | $155.00 |
| 459 | Cave | AMC46796 | $155.00 |
| 460 | Strip | AMC46800 | $155.00 |
| 461 | Cuba Fraction | AMC46801 | $155.00 |
| 462 | Patrick Henry Fraction/Relocation | AMC46802 | $155.00 |
| 463 | R. G. Ingersoll Fraction | AMC46803 | $155.00 |
| 464 | Daylight Fraction | AMC46804 | $155.00 |
| 465 | Travis #2 | AMC46805 | $155.00 |
| 466 | Travis #3 | AMC46806 | $155.00 |
| 467 | Travis #4 | AMC46807 | $155.00 |
| 468 | Travis #5 | AMC46808 | $155.00 |
| 469 | Travis #6 | AMC46809 | $155.00 |
| 470 | Art | AMC46810 | $155.00 |
| 471 | Al | AMC46811 | $155.00 |
| 472 | Sam | AMC46812 | $155.00 |
| 473 | Fred | AMC46813 | $155.00 |
| 474 | Bert | AMC46814 | $155.00 |
| 475 | Bob | AMC46815 | $155.00 |
| 476 | Canyon No. 34 | AMC47482 | $155.00 |
| 477 | Canyon No. 35 | AMC47483 | $155.00 |
| 478 | Canyon No. 36 | AMC47484 | $155.00 |
| 479 | Canyon No. 37 | AMC47485 | $155.00 |
| 480 | Canyon No. 38 | AMC47486 | $155.00 |
| 481 | Canyon No. 39 | AMC47487 | $155.00 |
| 482 | Canyon No. 40 | AMC47488 | $155.00 |
| 483 | Canyon No. 41 | AMC47489 | $155.00 |
| 484 | Canyon No. 42 | AMC47490 | $155.00 |
| 485 | Canyon No. 43 | AMC47491 | $155.00 |
| 486 | Canyon No. 64 | AMC47512 | $155.00 |
| 487 | Canyon No. 65 | AMC47513 | $155.00 |
| 488 | Canyon No. 66 | AMC47514 | $155.00 |
| 489 | Canyon No. 67 | AMC47515 | $155.00 |
| 490 | Canyon No. 68 | AMC47516 | $155.00 |
| 491 | Canyon No. 69 | AMC47517 | $155.00 |
| 492 | Canyon No. 70 | AMC47518 | $155.00 |
| 493 | Canyon No. 71 | AMC47519 | $155.00 |
| 494 | Canyon No. 72 | AMC47520 | $155.00 |
| 495 | Canyon No. 73 | AMC47521 | $155.00 |
| 496 | Canyon No. 74 | AMC47522 | $155.00 |
| 497 | Canyon No. 75 | AMC47523 | $155.00 |
| 498 | Canyon No. 76 | AMC47524 | $155.00 |
| 499 | Canyon No. 77 | AMC47525 | $155.00 |
| 500 | Canyon No. 78 | AMC47526 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 501 | Canyon No. 79 | AMC47527 | $155.00 |
| 502 | Telemeter Fraction | AMC62785 | $155.00 |
| 503 | West End Fraction | AMC62786 | $155.00 |
| 504 | Hattie Fraction | AMC62787 | $155.00 |
| 505 | Cactus | AMC64123 | $155.00 |
| 506 | Travis #7 | AMC64124 | $155.00 |
| 507 | Fox #1 | AMC64125 | $155.00 |
| 508 | Fox #2 | AMC64126 | $155.00 |
| 509 | Fox #7 | AMC64131 | $155.00 |
| 510 | Fox #13 | AMC64133 | $155.00 |
| 511 | Cloud Rest | AMC64134 | $155.00 |
| 512 | Big Windy | AMC64135 | $155.00 |
| 513 | Big Windy Fraction | AMC64136 | $155.00 |
| 514 | Blue Wing | AMC64137 | $155.00 |
| 515 | Cloud Rest No. 1 | AMC64138 | $155.00 |
| 516 | Kent #1 Long John | AMC66835 | $155.00 |
| 517 | Kent #2 Patricia C. | AMC66836 | $155.00 |
| 518 | Kent #3 Little Joe | AMC66837 | $155.00 |
| 519 | Belle of Rosemont | AMC66838 | $155.00 |
| 520 | John | AMC74390 | $155.00 |
| 521 | Joe | AMC74391 | $155.00 |
| 522 | Ben | AMC74392 | $155.00 |
| 523 | Pete | AMC74393 | $155.00 |
| 524 | Adolph Lewisohn | AMC74394 | $155.00 |
| 525 | Adolph Lewisohn | AMC74395 | $155.00 |
| 526 | Rosemont | AMC74396 | $155.00 |
| 527 | Rosemont | AMC74397 | $155.00 |
| 528 | Albert Steinfeld | AMC74398 | $155.00 |
| 529 | Albert Steinfeld | AMC74399 | $155.00 |
| 530 | Hugh Young | AMC74400 | $155.00 |
| 531 | Hugh Young | AMC74401 | $155.00 |
| 532 | Ethel | AMC74402 | $155.00 |
| 533 | Albert | AMC74403 | $155.00 |
| 534 | Rosemont #1 | AMC74404 | $155.00 |
| 535 | Rosemont #2 | AMC74405 | $155.00 |
| 536 | Rosemont #3 | AMC74406 | $155.00 |
| 537 | Rosemont #4 | AMC74407 | $155.00 |
| 538 | Rosemont #7 | AMC74408 | $155.00 |
| 539 | Rosemont #8 | AMC74409 | $155.00 |
| 540 | Rosemont #14 | AMC74410 | $155.00 |
| 541 | Rosemont #19 | AMC74411 | $155.00 |
| 542 | Rosemont #20 | AMC74412 | $155.00 |
| 543 | Rosemont #20 | AMC74413 | $155.00 |
| 544 | Rosemont #22 | AMC74414 | $155.00 |
| 545 | Rosemont #23 | AMC74415 | $155.00 |
| 546 | Rosemont #24 | AMC74416 | $155.00 |
| 547 | Rosemont #25 | AMC74417 | $155.00 |
| 548 | RX | AMC74418 | $155.00 |
| 549 | Flying Dutchman #7A | AMC75181 | $155.00 |
| 550 | Blue Point No. 2A | AMC75182 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 551 | Alpine #1A | AMC75183 | $155.00 |
| 552 | Alpine #2A | AMC75184 | $155.00 |
| 553 | Alpine #3A | AMC75185 | $155.00 |
| 554 | Alpine #4A | AMC75186 | $155.00 |
| 555 | Frijole VI A | AMC95315 | $155.00 |
| 556 | Falcon 1A | AMC99789 | $155.00 |
| 557 | Falcon 2A | AMC99790 | $155.00 |
| 558 | Falcon 3A | AMC99791 | $155.00 |
| 559 | Falcon 4A | AMC99792 | $155.00 |
| 560 | Falcon 5A | AMC99793 | $155.00 |
| 561 | Falcon 6A | AMC99794 | $155.00 |
| 562 | Falcon 7A | AMC99795 | $155.00 |
| 563 | Falcon 8A | AMC99796 | $155.00 |
| 564 | Falcon 9A | AMC99797 | $155.00 |
| 565 | Falcon 10A | AMC99798 | $155.00 |
| 566 | Falcon 11A | AMC99799 | $155.00 |
| 567 | Falcon 12A | AMC99800 | $155.00 |
| 568 | Falcon 13A | AMC99801 | $155.00 |
| 569 | Falcon 14A | AMC99802 | $155.00 |
| 570 | Falcon 15A | AMC99803 | $155.00 |
| 571 | Falcon 16A | AMC99804 | $155.00 |
| 572 | Falcon 17A | AMC99805 | $155.00 |
| 573 | Falcon 18A | AMC99806 | $155.00 |
| 574 | Falcon 19A | AMC99807 | $155.00 |
| 575 | Falcon 20A | AMC99808 | $155.00 |
| 576 | Falcon 21A | AMC99809 | $155.00 |
| 577 | Falcon 22A | AMC99810 | $155.00 |
| 578 | Falcon 27A | AMC99811 | $155.00 |
| 579 | Falcon 28A | AMC99812 | $155.00 |
| 580 | Falcon 29A | AMC99813 | $155.00 |
| 581 | Falcon 30A | AMC99814 | $155.00 |
| 582 | Falcon 31A | AMC99815 | $155.00 |
| 583 | Falcon 32A | AMC99816 | $155.00 |
| 584 | Wasp 62A | AMC99817 | $155.00 |
| 585 | Wasp 63A | AMC99818 | $155.00 |
| 586 | Wasp 219A | AMC99819 | $155.00 |
| 587 | Wasp 220A | AMC99820 | $155.00 |
| 588 | Wasp 221A | AMC99821 | $155.00 |
| 589 | Wasp 222A | AMC99822 | $155.00 |
| 590 | Tecky | AMC99823 | $155.00 |
| 591 | MIA 1A | AMC117293 | $155.00 |
| 592 | MIA 2A | AMC117294 | $155.00 |
| 593 | MIA 3A | AMC117295 | $155.00 |
| 594 | MIA 4A | AMC117296 | $155.00 |
| 595 | MIA 5A | AMC117297 | $155.00 |
| 596 | MIA 6A | AMC117298 | $155.00 |
| 597 | MIA 7A | AMC117299 | $155.00 |
| 598 | MIA 8A | AMC117300 | $155.00 |
| 599 | MIA 9A | AMC117301 | $155.00 |
| 600 | MIA 12A | AMC117304 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 601 | MIA 13A | AMC117305 | $155.00 |
| 602 | MIA 14A | AMC117306 | $155.00 |
| 603 | BILLY C. | AMC129394 | $155.00 |
| 604 | Hope-1 | AMC303950 | $155.00 |
| 605 | Hope 2 | AMC303951 | $155.00 |
| 606 | Hope-3 | AMC303952 | $155.00 |
| 607 | Hope-4 | AMC303953 | $155.00 |
| 608 | Hope-5 | AMC303954 | $155.00 |
| 609 | Hope-6 | AMC303955 | $155.00 |
| 610 | Hope-7 | AMC303956 | $155.00 |
| 611 | Hope 8 | AMC303957 | $155.00 |
| 612 | Hope-9 | AMC303958 | $155.00 |
| 613 | Hope 10 | AMC303959 | $155.00 |
| 614 | Hope-10A | AMC303960 | $155.00 |
| 615 | Hope-11 | AMC303961 | $155.00 |
| 616 | Hope-12 | AMC303962 | $155.00 |
| 617 | Hope-13 | AMC303963 | $155.00 |
| 618 | Hope-14 | AMC303964 | $155.00 |
| 619 | Hope-15 | AMC303965 | $155.00 |
| 620 | Hope-16 | AMC303966 | $155.00 |
| 621 | Hope-17 | AMC303967 | $155.00 |
| 622 | Hope-18 | AMC303968 | $155.00 |
| 623 | Hope-19 | AMC303969 | $155.00 |
| 624 | Hope-20 | AMC303970 | $155.00 |
| 625 | Hope-21 | AMC303971 | $155.00 |
| 626 | Hope-22 | AMC303972 | $155.00 |
| 627 | Hope 23 | AMC303973 | $155.00 |
| 628 | Hope-24 | AMC303974 | $155.00 |
| 629 | Hope-25 | AMC303975 | $155.00 |
| 630 | Hope-26 | AMC303976 | $155.00 |
| 631 | Hope-27 | AMC303977 | $155.00 |
| 632 | Hope-28 | AMC303978 | $155.00 |
| 633 | H-29 | AMC303979 | $155.00 |
| 634 | Hope-30 | AMC303980 | $155.00 |
| 635 | Hope-31 | AMC303981 | $155.00 |
| 636 | Hope 32 | AMC303982 | $155.00 |
| 637 | Hope-33 | AMC303983 | $155.00 |
| 638 | Hope-34 | AMC303984 | $155.00 |
| 639 | Hope-35 | AMC303985 | $155.00 |
| 640 | Hope-36 | AMC303986 | $155.00 |
| 641 | Hope-37 | AMC303987 | $155.00 |
| 642 | H-38A | AMC313532 | $155.00 |
| 643 | H-39A | AMC313533 | $155.00 |
| 644 | H-40A | AMC313534 | $155.00 |
| 645 | H-41A | AMC313535 | $155.00 |
| 646 | H-42A | AMC313536 | $155.00 |
| 647 | H-43A | AMC313537 | $155.00 |
| 648 | H-44A | AMC313538 | $155.00 |
| 649 | H-45A | AMC313539 | $155.00 |
| 650 | H-46A | AMC313540 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 651 | H-47A | AMC313541 | $155.00 |
| 652 | H-48A | AMC313542 | $155.00 |
| 653 | H-49A | AMC313543 | $155.00 |
| 654 | H-50A | AMC313544 | $155.00 |
| 655 | H-51A | AMC313545 | $155.00 |
| 656 | H-52A | AMC313546 | $155.00 |
| 657 | H-53A | AMC313547 | $155.00 |
| 658 | H-54A | AMC313548 | $155.00 |
| 659 | H-55A | AMC313549 | $155.00 |
| 660 | H-56A | AMC313550 | $155.00 |
| 661 | H-57A | AMC313551 | $155.00 |
| 662 | H-58A | AMC313552 | $155.00 |
| 663 | H-59A | AMC313553 | $155.00 |
| 664 | H-60A | AMC313554 | $155.00 |
| 665 | H-61A | AMC313555 | $155.00 |
| 666 | H-62A | AMC313556 | $155.00 |
| 667 | H-63A | AMC313557 | $155.00 |
| 668 | H-64A | AMC313558 | $155.00 |
| 669 | H-65A | AMC313559 | $155.00 |
| 670 | H-66A | AMC313560 | $155.00 |
| 671 | H-67A | AMC313561 | $155.00 |
| 672 | H-68A | AMC313562 | $155.00 |
| 673 | H-69A | AMC313563 | $155.00 |
| 674 | H-70A | AMC313564 | $155.00 |
| 675 | H-71A | AMC313565 | $155.00 |
| 676 | H-72A | AMC313566 | $155.00 |
| 677 | H-73A | AMC313567 | $155.00 |
| 678 | H-74A | AMC313568 | $155.00 |
| 679 | H-75A | AMC313569 | $155.00 |
| 680 | H-76A | AMC313570 | $155.00 |
| 681 | H-77A | AMC313571 | $155.00 |
| 682 | H-78A | AMC313572 | $155.00 |
| 683 | H-79A | AMC313573 | $155.00 |
| 684 | H-80A | AMC313574 | $155.00 |
| 685 | H-81A | AMC313575 | $155.00 |
| 686 | H-82A | AMC313576 | $155.00 |
| 687 | H-83A | AMC313577 | $155.00 |
| 688 | H-84A | AMC313578 | $155.00 |
| 689 | H-85A | AMC313579 | $155.00 |
| 690 | H-86A | AMC313580 | $155.00 |
| 691 | H-87A | AMC313581 | $155.00 |
| 692 | H-88A | AMC313582 | $155.00 |
| 693 | H-89A | AMC313583 | $155.00 |
| 694 | H-90A | AMC313584 | $155.00 |
| 695 | H-91A | AMC313585 | $155.00 |
| 696 | H-92A | AMC313586 | $155.00 |
| 697 | H-93A | AMC313587 | $155.00 |
| 698 | H-94A | AMC313588 | $155.00 |
| 699 | H-95A | AMC313589 | $155.00 |
| 700 | H-96A | AMC313590 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 701 | H-97A | AMC313591 | $155.00 |
| 702 | H-98A | AMC313592 | $155.00 |
| 703 | H-99A | AMC313593 | $155.00 |
| 704 | H-100A | AMC313594 | $155.00 |
| 705 | H-101A | AMC313595 | $155.00 |
| 706 | H-102A | AMC313596 | $155.00 |
| 707 | H-103A | AMC313597 | $155.00 |
| 708 | H-104A | AMC313598 | $155.00 |
| 709 | H-105A | AMC313599 | $155.00 |
| 710 | H-106A | AMC313600 | $155.00 |
| 711 | H-107A | AMC313601 | $155.00 |
| 712 | H-108A | AMC313602 | $155.00 |
| 713 | H-109A | AMC313603 | $155.00 |
| 714 | H-110A | AMC313604 | $155.00 |
| 715 | H-111A | AMC313605 | $155.00 |
| 716 | H-112A | AMC313606 | $155.00 |
| 717 | H-113A | AMC313607 | $155.00 |
| 718 | H-114A | AMC313608 | $155.00 |
| 719 | H-115A | AMC313609 | $155.00 |
| 720 | H-116A | AMC313610 | $155.00 |
| 721 | H-117A | AMC313611 | $155.00 |
| 722 | H-118A | AMC313612 | $155.00 |
| 723 | H-119A | AMC313613 | $155.00 |
| 724 | H-120A | AMC313614 | $155.00 |
| 725 | H-121A | AMC313615 | $155.00 |
| 726 | H-122A | AMC313616 | $155.00 |
| 727 | H-123A | AMC313617 | $155.00 |
| 728 | H-124A | AMC313618 | $155.00 |
| 729 | H-125A | AMC313619 | $155.00 |
| 730 | H-126A | AMC313620 | $155.00 |
| 731 | H-127A | AMC313621 | $155.00 |
| 732 | H-128A | AMC313622 | $155.00 |
| 733 | H-129A | AMC313623 | $155.00 |
| 734 | H-130A | AMC313624 | $155.00 |
| 735 | H-131A | AMC313625 | $155.00 |
| 736 | H-132A | AMC313626 | $155.00 |
| 737 | H-133A | AMC313627 | $155.00 |
| 738 | H-134A | AMC313628 | $155.00 |
| 739 | H-135A | AMC313629 | $155.00 |
| 740 | H-136A | AMC313630 | $155.00 |
| 741 | H-137A | AMC313631 | $155.00 |
| 742 | H-138A | AMC313632 | $155.00 |
| 743 | H-139A | AMC313633 | $155.00 |
| 744 | H-140A | AMC313634 | $155.00 |
| 745 | H-141A | AMC313635 | $155.00 |
| 746 | H-142A | AMC313636 | $155.00 |
| 747 | H-143A | AMC313637 | $155.00 |
| 748 | H-144A | AMC313638 | $155.00 |
| 749 | H-145A | AMC313639 | $155.00 |
| 750 | H-146A | AMC313640 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 751 | H-147A | AMC313641 | $155.00 |
| 752 | H-148A | AMC313642 | $155.00 |
| 753 | H-149A | AMC313643 | $155.00 |
| 754 | H-150A | AMC313644 | $155.00 |
| 755 | H-151A | AMC313645 | $155.00 |
| 756 | H-152A | AMC313646 | $155.00 |
| 757 | H-153A | AMC313647 | $155.00 |
| 758 | H-154A | AMC313648 | $155.00 |
| 759 | H-155A | AMC313649 | $155.00 |
| 760 | H-156A | AMC313650 | $155.00 |
| 761 | H-157A | AMC313651 | $155.00 |
| 762 | H-158A | AMC313652 | $155.00 |
| 763 | H-159A | AMC313653 | $155.00 |
| 764 | H-160A | AMC313654 | $155.00 |
| 765 | H-161A | AMC313655 | $155.00 |
| 766 | H-162A | AMC313656 | $155.00 |
| 767 | H-163A | AMC313657 | $155.00 |
| 768 | H-164A | AMC313658 | $155.00 |
| 769 | H-165A | AMC313659 | $155.00 |
| 770 | H-166A | AMC313660 | $155.00 |
| 771 | H-167A | AMC313661 | $155.00 |
| 772 | H-168A | AMC313662 | $155.00 |
| 773 | H-169A | AMC313663 | $155.00 |
| 774 | H-170A | AMC313664 | $155.00 |
| 775 | H-171A | AMC313665 | $155.00 |
| 776 | H-177A | AMC313671 | $155.00 |
| 777 | H-178A | AMC313672 | $155.00 |
| 778 | H-179A | AMC313673 | $155.00 |
| 779 | H-180A | AMC313674 | $155.00 |
| 780 | H-181A | AMC313675 | $155.00 |
| 781 | H-182A | AMC313676 | $155.00 |
| 782 | H-183A | AMC313677 | $155.00 |
| 783 | H-187A | AMC313678 | $155.00 |
| 784 | H-188A | AMC313679 | $155.00 |
| 785 | H-189A | AMC313680 | $155.00 |
| 786 | H-190A | AMC313681 | $155.00 |
| 787 | H-191A | AMC313682 | $155.00 |
| 788 | H-192A | AMC313683 | $155.00 |
| 789 | H-194A | AMC313684 | $155.00 |
| 790 | H-195A | AMC313685 | $155.00 |
| 791 | H-196A | AMC313686 | $155.00 |
| 792 | H-197A | AMC313687 | $155.00 |
| 793 | H-198A | AMC313688 | $155.00 |
| 794 | H-199A | AMC313689 | $155.00 |
| 795 | Hope No. 201 | AMC330891 | $155.00 |
| 796 | Hope 201A | AMC330892 | $155.00 |
| 797 | Hope No. 202 | AMC330893 | $155.00 |
| 798 | Hope No. 203 | AMC330894 | $155.00 |
| 799 | Hope No. 204 | AMC330895 | $155.00 |
| 800 | Hope No. 205 | AMC330896 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 801 | Hope No. 206 | AMC330897 | $155.00 |
| 802 | Hope No. 207 | AMC330898 | $155.00 |
| 803 | Hope No. 208 | AMC330899 | $155.00 |
| 804 | Hope No. 209 | AMC330900 | $155.00 |
| 805 | Hope No. 210 | AMC330901 | $155.00 |
| 806 | Hope No. 211 | AMC330902 | $155.00 |
| 807 | Hope No. 212 | AMC330903 | $155.00 |
| 808 | Hope No. 213 | AMC330904 | $155.00 |
| 809 | Hope No. 214 | AMC330905 | $155.00 |
| 810 | Hope No. 215 | AMC330906 | $155.00 |
| 811 | Hope No. 216 | AMC330907 | $155.00 |
| 812 | Hope No. 222 | AMC330910 | $155.00 |
| 813 | Hope No. 223 | AMC330911 | $155.00 |
| 814 | Hope No. 224 | AMC330912 | $155.00 |
| 815 | Hope No. 225 | AMC330913 | $155.00 |
| 816 | Hope 226A | AMC330914 | $155.00 |
| 817 | Hope 227A | AMC330915 | $155.00 |
| 818 | Hope 228A | AMC330916 | $155.00 |
| 819 | Hope 229A | AMC330917 | $155.00 |
| 820 | Hope No. 230 | AMC330918 | $155.00 |
| 821 | Hope No. 231 | AMC330919 | $155.00 |
| 822 | Hope No. 232 | AMC330920 | $155.00 |
| 823 | Hope No. 233 | AMC330921 | $155.00 |
| 824 | Hope No. 234 | AMC330922 | $155.00 |
| 825 | Hope No. 235 | AMC330923 | $155.00 |
| 826 | Hope No. 236 | AMC330924 | $155.00 |
| 827 | Hope No. 237 | AMC330925 | $155.00 |
| 828 | Hope No. 238 | AMC330926 | $155.00 |
| 829 | Hope No. 239 | AMC330927 | $155.00 |
| 830 | Hope No. 240 | AMC330928 | $155.00 |
| 831 | Hope No. 241 | AMC330929 | $155.00 |
| 832 | Hope No. 242 | AMC330930 | $155.00 |
| 833 | Hope No. 243 | AMC330931 | $155.00 |
| 834 | Hope No. 244 | AMC330932 | $155.00 |
| 835 | Hope No. 245 | AMC330933 | $155.00 |
| 836 | Hope No. 246 | AMC330934 | $155.00 |
| 837 | Hope No. 250 | AMC330935 | $155.00 |
| 838 | Hope No. 251 | AMC330936 | $155.00 |
| 839 | Hope No. 252 | AMC330937 | $155.00 |
| 840 | Hope No. 253 | AMC330938 | $155.00 |
| 841 | Hope No. 254 | AMC330939 | $155.00 |
| 842 | Hope No. 255 | AMC330940 | $155.00 |
| 843 | Hope No. 256 | AMC330941 | $155.00 |
| 844 | Hope No. 257 | AMC330942 | $155.00 |
| 845 | Elk 47/Relocation | AMC330943 | $155.00 |
| 846 | H-172 B/Relocation | AMC331308 | $155.00 |
| 847 | H-173 B/Relocation | AMC331309 | $155.00 |
| 848 | H-174 B/Relocation | AMC331310 | $155.00 |
| 849 | H-175 B/Relocation | AMC331311 | $155.00 |
| 850 | H-176 B/Relocation | AMC331312 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 851 | MMRE | AMC367652 | $155.00 |
| 852 | HV 1 | AMC380250 | $155.00 |
| 853 | HV 2 | AMC380251 | $155.00 |
| 854 | HV 3 | AMC380252 | $155.00 |
| 855 | HV 4 | AMC380253 | $155.00 |
| 856 | ROSE 1 | AMC385174 | $155.00 |
| 857 | ROSE 2 | AMC385175 | $155.00 |
| 858 | ROSE 3 | AMC385176 | $155.00 |
| 859 | ROSE 4 | AMC385177 | $155.00 |
| 860 | ROSE 5 | AMC385178 | $155.00 |
| 861 | ROSE 6 | AMC385179 | $155.00 |
| 862 | ROSE 7 | AMC385180 | $155.00 |
| 863 | ROSE 8 | AMC385181 | $155.00 |
| 864 | ROSE 9 | AMC385182 | $155.00 |
| 865 | HV 6 | AMC387231 | $155.00 |
| 866 | HV 7 | AMC387232 | $155.00 |
| 867 | HV 8 | AMC387233 | $155.00 |
| 868 | HV 9 | AMC387234 | $155.00 |
| 869 | HV 10 | AMC387235 | $155.00 |
| 870 | HV 11 | AMC387236 | $155.00 |
| 871 | HV 12 | AMC387237 | $155.00 |
| 872 | HV 13 | AMC387238 | $155.00 |
| 873 | HV 23 | AMC387241 | $155.00 |
| 874 | HV 24 | AMC387242 | $155.00 |
| 875 | HV 25 | AMC387243 | $155.00 |
| 876 | HV 16 | AMC390077 | $155.00 |
| 877 | HV 17 | AMC390078 | $155.00 |
| 878 | HV 18 | AMC390079 | $155.00 |
| 879 | HV 19 | AMC390080 | $155.00 |
| 880 | HV 20 | AMC390081 | $155.00 |
| 881 | HV 21 | AMC390082 | $155.00 |
| 882 | HV 22 | AMC390083 | $155.00 |
| 883 | WAIT-1 | AMC390084 | $155.00 |
| 884 | WAIT-2 | AMC390085 | $155.00 |
| 885 | WAIT-3 | AMC390086 | $155.00 |
| 886 | WAIT-4 | AMC390087 | $155.00 |
| 887 | WAIT-5 | AMC390088 | $155.00 |
| 888 | WAIT-6 | AMC390089 | $155.00 |
| 889 | WAIT-7 | AMC390090 | $155.00 |
| 890 | WAIT-8 | AMC390091 | $155.00 |
| 891 | WAIT-9 | AMC390092 | $155.00 |
| 892 | WAIT-10 | AMC390093 | $155.00 |
| 893 | WAIT-11 | AMC390094 | $155.00 |
| 894 | WAIT-12 | AMC390095 | $155.00 |
| 895 | WAIT-13 | AMC390096 | $155.00 |
| 896 | WAIT-14 | AMC390097 | $155.00 |
| 897 | WAIT-15 | AMC390098 | $155.00 |
| 898 | WAIT-16 | AMC390099 | $155.00 |
| 899 | WAIT-17 | AMC390100 | $155.00 |
| 900 | WAIT-18 | AMC390101 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 901 | WAIT-19 | AMC390102 | $155.00 |
| 902 | WAIT-20 | AMC390103 | $155.00 |
| 903 | WAIT-21 | AMC390104 | $155.00 |
| 904 | WAIT-22 | AMC390105 | $155.00 |
| 905 | WAIT-23 | AMC390106 | $155.00 |
| 906 | WAIT-24 | AMC390107 | $155.00 |
| 907 | WAIT-25 | AMC390108 | $155.00 |
| 908 | WAIT-26 | AMC390109 | $155.00 |
| 909 | WAIT-27 | AMC390110 | $155.00 |
| 910 | WAIT-28 | AMC390111 | $155.00 |
| 911 | WAIT-29 | AMC390112 | $155.00 |
| 912 | WAIT-30 | AMC390113 | $155.00 |
| 913 | WAIT-31 | AMC390114 | $155.00 |
| 914 | WAIT-32 | AMC390115 | $155.00 |
| 915 | FALLS FRACTION | AMC391154 | $155.00 |
| 916 | H-69B | AMC391155 | $155.00 |
| 917 | NO CHANCE No. 3 | AMC391156 | $155.00 |
| 918 | SCHWAB FRACTION | AMC391157 | $155.00 |
| 919 | H FRAC. 1 | AMC392445 | $155.00 |
| 920 | H FRAC. 2 | AMC392446 | $155.00 |
| 921 | H FRAC. 3 | AMC392447 | $155.00 |
| 922 | H FRAC. 4 | AMC392448 | $155.00 |
| 923 | H FRAC. 5 | AMC392449 | $155.00 |
| 924 | H FRAC. 6 | AMC392450 | $155.00 |
| 925 | H FRAC. 7 | AMC392451 | $155.00 |
| 926 | H FRAC. 8 | AMC392452 | $155.00 |
| 927 | BILLY FRAC. | AMC393532 | $155.00 |
| 928 | DSM 1 | AMC393533 | $155.00 |
| 929 | DSM 2 | AMC393534 | $155.00 |
| 930 | DSM 3 | AMC393535 | $155.00 |
| 931 | DSM 4 | AMC393536 | $155.00 |
| 932 | DSM 5 | AMC393537 | $155.00 |
| 933 | DSM 6 | AMC393538 | $155.00 |
| 934 | DSM 7 | AMC393539 | $155.00 |
| 935 | DSM 8 | AMC393540 | $155.00 |
| 936 | DSM 9 | AMC393541 | $155.00 |
| 937 | DSM 10 | AMC393542 | $155.00 |
| 938 | HV5 A | AMC393543 | $155.00 |
| 939 | MIA FRAC 1 | AMC393544 | $155.00 |
| 940 | MIA FRAC 2 | AMC393545 | $155.00 |
| 941 | SON OF GUN 34 | AMC394006 | $155.00 |
| 942 | RMT FRAC 1 | AMC394561 | $155.00 |
| 943 | RMT FRAC 2 | AMC394562 | $155.00 |
| 944 | RMT FRAC 3 | AMC394563 | $155.00 |
| 945 | RMT FRAC 4 | AMC394564 | $155.00 |
| 946 | NC-CF | AMC396422 | $155.00 |
| 947 | Thankful | AMC404128 | $155.00 |
| 948 | RCC-1 | AMC411964 | $155.00 |
| 949 | RCC-2 | AMC411965 | $155.00 |
| 950 | RCC-3 | AMC411966 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 951 | RCC-4 | AMC411967 | $155.00 |
| 952 | RCC-5 | AMC411968 | $155.00 |
| 953 | RCC-6 | AMC411969 | $155.00 |
| 954 | RCC-7 | AMC411970 | $155.00 |
| 955 | RCC-8 | AMC411971 | $155.00 |
| 956 | RCC-9 | AMC411972 | $155.00 |
| 957 | RCC-10 | AMC411973 | $155.00 |
| 958 | RCC-11 | AMC411974 | $155.00 |
| 959 | RCC-12 | AMC411975 | $155.00 |
| 960 | RCC-13 | AMC411976 | $155.00 |
| 961 | RCC-14 | AMC411977 | $155.00 |
| 962 | RCC-15 | AMC411978 | $155.00 |
| 963 | RCC-16 | AMC411979 | $155.00 |
| 964 | RCC-17 | AMC411980 | $155.00 |
| 965 | RCC-18 | AMC411981 | $155.00 |
| 966 | RCC-19 | AMC411982 | $155.00 |
| 967 | RCC-20 | AMC411983 | $155.00 |
| 968 | RCC-21 | AMC411984 | $155.00 |
| 969 | RCC-22 | AMC411985 | $155.00 |
| 970 | RCC-23 | AMC411986 | $155.00 |
| 971 | RCC-24 | AMC411987 | $155.00 |
| 972 | RCC-25 | AMC411988 | $155.00 |
| 973 | RCC-26 | AMC411989 | $155.00 |
| 974 | RCC-27 | AMC411990 | $155.00 |
| 975 | RCC-28 | AMC411991 | $155.00 |
| 976 | RCC-29 | AMC411992 | $155.00 |
| 977 | RCC-30 | AMC411993 | $155.00 |
| 978 | RCC-31 | AMC411994 | $155.00 |
| 979 | RCC-32 | AMC411995 | $155.00 |
| 980 | RCC-33 | AMC411996 | $155.00 |
| 981 | RCC-34 | AMC411997 | $155.00 |
| 982 | RCC-35 | AMC411998 | $155.00 |
| 983 | RCC-36 | AMC411999 | $155.00 |
| 984 | RCC-37 | AMC412000 | $155.00 |
| 985 | RCC-38 | AMC412001 | $155.00 |
| 986 | RCC-39 | AMC412002 | $155.00 |
| 987 | RCC-40 | AMC412003 | $155.00 |
| 988 | RCC-41 | AMC412004 | $155.00 |
| 989 | RCC-42 | AMC412005 | $155.00 |
| 990 | RCC-43 | AMC412006 | $155.00 |
| 991 | RCC-44 | AMC412007 | $155.00 |
| 992 | RCC-45 | AMC412008 | $155.00 |
| 993 | RCC-46 | AMC412009 | $155.00 |
| 994 | RCC-47 | AMC412010 | $155.00 |
| 995 | RCC-48 | AMC412011 | $155.00 |
| 996 | RCC-49 | AMC412012 | $155.00 |
| 997 | RCC-50 | AMC412013 | $155.00 |
| 998 | RCC-51 | AMC412014 | $155.00 |
| 999 | RCC-52 | AMC412015 | $155.00 |
| 1000 | RCC-53 | AMC412016 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 1001 | RCC-54 | AMC412017 | $155.00 |
| 1002 | RCC-55 | AMC412018 | $155.00 |
| 1003 | RCC-56 | AMC412019 | $155.00 |
| 1004 | RCC-57 | AMC412020 | $155.00 |
| 1005 | RCC-58 | AMC412021 | $155.00 |
| 1006 | RCC-59 | AMC412022 | $155.00 |
| 1007 | RCC-60 | AMC412023 | $155.00 |
| 1008 | RCC-61 | AMC412024 | $155.00 |
| 1009 | RCC-62 | AMC412025 | $155.00 |
| 1010 | RCC-63 | AMC412026 | $155.00 |
| 1011 | RCC-64 | AMC412027 | $155.00 |
| 1012 | RCC-65 | AMC412028 | $155.00 |
| 1013 | RCC-66 | AMC412029 | $155.00 |
| 1014 | RCC-67 | AMC412030 | $155.00 |
| 1015 | RCC-68 | AMC412031 | $155.00 |
| 1016 | RCC-69 | AMC412032 | $155.00 |
| 1017 | RCC-70 | AMC412033 | $155.00 |
| 1018 | RCC-71 | AMC412034 | $155.00 |
| 1019 | RCC-72 | AMC412035 | $155.00 |
| 1020 | RCC-73 | AMC412036 | $155.00 |
| 1021 | RCC-74 | AMC412037 | $155.00 |
| 1022 | RCC-75 | AMC412038 | $155.00 |
| 1023 | RCC-76 | AMC412039 | $155.00 |
| 1024 | RCC-77 | AMC412040 | $155.00 |
| 1025 | RCC-78 | AMC412041 | $155.00 |
| 1026 | RCC-79 | AMC412042 | $155.00 |
| 1027 | RCC-80 | AMC412043 | $155.00 |
| 1028 | RCC-81 | AMC412044 | $155.00 |
| 1029 | RCC-82 | AMC412045 | $155.00 |
| 1030 | RCC-83 | AMC412046 | $155.00 |
| 1031 | RCC-84 | AMC412047 | $155.00 |
| 1032 | RCC-85 | AMC412048 | $155.00 |
| 1033 | RCC-86 | AMC412049 | $155.00 |
| 1034 | RCC-87 | AMC412050 | $155.00 |
| 1035 | RCC-88 | AMC412051 | $155.00 |
| 1036 | RCC-89 | AMC412052 | $155.00 |
| 1037 | RCC-90 | AMC412053 | $155.00 |
| 1038 | RCC-91 | AMC412054 | $155.00 |
| 1039 | RCC-92 | AMC412055 | $155.00 |
| 1040 | RCC-93 | AMC412056 | $155.00 |
| 1041 | RCC-94 | AMC412057 | $155.00 |
| 1042 | RCC-95 | AMC412058 | $155.00 |
| 1043 | RCC-96 | AMC412059 | $155.00 |
| 1044 | RCC-97 | AMC412060 | $155.00 |
| 1045 | RCC-98 | AMC412061 | $155.00 |
| 1046 | RCC-99 | AMC412062 | $155.00 |
| 1047 | RCC-100 | AMC412063 | $155.00 |
| 1048 | AGAVE-1 | AMC412064 | $155.00 |
| 1049 | AGAVE-2 | AMC412065 | $155.00 |
| 1050 | AGAVE-3 | AMC412066 | $155.00 |

| COUNT | UNPATENTED CLAIM NAME | BLM SERIAL NUMBER | ANNUAL FEE |
|---|---|---|---|
| 1051 | AGAVE-4 | AMC412067 | $155.00 |
| 1052 | AGAVE-5 | AMC412068 | $155.00 |
| 1053 | AGAVE-6 | AMC412069 | $155.00 |
| 1054 | CONTINENTAL-1 | AMC412070 | $155.00 |
| 1055 | CONTINENTAL-2 | AMC412071 | $155.00 |
| 1056 | CONTINENTAL-3 | AMC412072 | $155.00 |
| 1057 | CONTINENTAL-4 | AMC412073 | $155.00 |
| 1058 | CONTINENTAL-5 | AMC412074 | $155.00 |
| 1059 | CONTINENTAL-6 | AMC412075 | $155.00 |
| 1060 | TAILOR | AMC423213 | $155.00 |
| 1061 | AGAVE-7 | AMC429429 | $155.00 |
| 1062 | AGAVE-8 | AMC429430 | $155.00 |
| 1063 | AGAVE-9 | AMC429431 | $155.00 |
| 1064 | RECORDER FRACTION | AMC429432 | $155.00 |
| **UNPATENTED CLAIM TOTALS** | | | **$164,920.00** |

# A1-4 Rosemont Project Fee Owned (Associated) Lands

| COUNT | PARCEL NO. | PROPERTY NAME | SECTION-TOWNSHIP-RANGE | ASSESSED ACRES | 2015 FEES |
|---|---|---|---|---|---|
| 1 | 305580280 | HELVETIA RANCH (KILGORE/ANDERSEN) | 23-18-15 | 10.080 | $460.00 |
| 2 | 305580330 | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT | 23-18-15 | 40.000 | $20.53 |
| 3 | 305580350 | HELVETIA RANCH ANNEX (DE LA OSSA) | 23-18-15 | 10.000 | $76.72 |
| 4 | 305580360 | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 23-18-15 | 10.000 | $12.21 |
| 5 | 305580370 | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 23-18-15 | 20.000 | $12.21 |
| 6 | 305580420 | HELVETIA RANCH ANNEX (MAATR) | 23-18-15 | 40.000 | $20.53 |
| 7 | 30553002D | HELVETIA RANCH ANNEX NORTH (TERRA BELLA) | 10-18-15 | 20.000 | $12.21 |
| 8 | 30553002F | HELVETIA RANCH ANNEX NORTH (TERRA BELLA) | 10-18-15 | 120.000 | $61.69 |
| 9 | 30553002G | HELVETIA RANCH ANNEX NORTH (TERRA BELLA) | 10-18-15 | 310.000 | $159.51 |
| 10 | 30553002H | HELVETIA RANCH ANNEX NORTH (AVRA VALLEY/LEBRECHT) | 10-18-15 | 108.420 | $55.73 |
| 11 | 30553004D | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 27-18-15 | 40.000 | $20.53 |
| 12 | 30553004H | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 27-18-15 | 40.000 | $20.53 |
| 13 | 30556001B | HELVETIA RANCH ANNEX NORTH (AVRA VALLEY/LEBRECHT | 15-18-15 | 313.110 | $161.04 |
| 14 | 30556001C | HELVETIA RANCH ANNEX NORTH (AVRA VALLEY/LEBRECHT) | 15-18-15 | 67.800 | $869.00 |
| 15 | 30557004B | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 22-18-15 | 5.000 | $12.21 |
| 16 | 30557004C | HELVETIA RANCH ANNEX (ADC/CALICA) | 22-18-15 | 52.480 | $12.81 |
| 17 | 30557004D | HELVETIA RANCH (KILGORE/ANDERSEN) | 22-18-15 | 10.000 | $87.44 |
| 18 | 30557005B | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 22-18-15 | 20.000 | $12.21 |
| 19 | 30557013B | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 22-18-15 | 35.000 | $18.09 |
| 20 | 30557013C | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 22-18-15 | 40.000 | $20.53 |
| 21 | 30557013D | HELVETIA RANCH ANNEX (SUTTLES) | 22-18-15 | 20.000 | $341.75 |
| 22 | 30557013E | HELVETIA RANCH ANNEX (AVRA VALLEY/LEBRECHT) | 22-18-15 | 40.000 | $20.53 |
| 23 | 30557022C | HELVETIA RANCH ANNEX (MAATR) | 22-18-15 | 40.000 | $20.53 |
| 24 | 30558034C | HELVETIA RANCH ANNEX (PIPELINE TRIANGLE) | 23-18-15 | 2.190 | $12.21 |
| 25 | 30562006B | ROSEMONT RANCH | 14-18-16 | 34.120 | $17.31 |
| 26 | 30562007D | ROSEMONT RANCH | 15-18-16 | 40.000 | $20.17 |
| 27 | 30562007F | ROSEMONT RANCH | 15-18-16 | 40.000 | $20.17 |
| 28 | 30562007G | ROSEMONT RANCH | 15-18-16 | 70.590 | $35.64 |
| 29 | 30562007H | ROSEMONT RANCH | 15-18-16 | 160.000 | $80.97 |
| 30 | 30562008C | ROSEMONT RANCH (HIDDEN VALLEY) | 21-18-16 | 60.150 | $23.35 |
| 31 | 30562008F | ROSEMONT RANCH (HIDDEN VALLEY) | 21-18-16 | 35.060 | $380.24 |
| 32 | 30562008G | ROSEMONT RANCH (HIDDEN VALLEY) | 21-18-16 | 5.010 | $472.95 |
| 33 | 30562008H | ROSEMONT RANCH (HIDDEN VALLEY | 21-18-16 | 24.880 | $12.99 |
| 34 | 30562008J | ROSEMONT RANCH (HIDDEN VALLEY) | 21-18-16 | 35.270 | $13.70 |
| 35 | 30562009A | ROSEMONT RANCH | 23-18-16 | 160.000 | $92.38 |
| 36 | 30562011A | ROSEMONT RANCH | 27-18-16 | 40.000 | $23.01 |
| 37 | 30562012A | ROSEMONT RANCH | 32-18-16 | 20.000 | $13.70 |
| 38 | 30562012C | ROSEMONT RANCH | 32-18-16 | 180.000 | $3,602.68 |
| **FEE OWNED (ASSOCIATED) TOTALS** | | | | **2319.160** | **$7,330.01** |

# A2-1 Permits and Authorizations

Details on permit status and authorizations for current project activities.

| PERMIT AND AUTHORIZATIONS TABLE | | | | | |
|---|---|---|---|---|---|
| **Permit** | **Lead Agency and Description** | **Submittal Date** | **Status** | **Issue Date** | **Term** |
| **Federal Permits and Authorizations Issued** | | | | | |
| MSHA ID Number | Mine Safety and Health Administration | Paperwork filed for the Rosemont Copper Project | Issued<br>02-03256 | July 21, 2010 | |
| Hazardous Waste Identification Number | Environmental Protection Agency (EPA) – Issued for hazardous waste can be generated and transported off site in quantities in excess of 100 pounds. | Submitted<br>Requires a contingency plan. | Received<br>RCRA EPA ID Number: AZR000509976 | Sept 14, 2011 | Life of the facility |
| **State Permits and Authorizations Issued** | | | | | |
| Groundwater withdrawal permits | ADWR – Groundwater withdrawal rights | Mineral Extraction Right | Issued<br>Permit No. 59-215979.0000 | Jan 18, 2008 | 20 years |
| Dam Safety Permit | ADWR – Regulates the construction and operation of large containment structures | Storage Analysis submitted Feb 17, 2012 for review by State Engineer | No permit necessary as designed | | |
| Well Drilling Permit | Arizona Department of Water Resources (ADWR) – Issued anytime drilling may intercept water table | On-going submittals for mineral exploration, geotechnical, and hydrologic investigation activities | Issued for current, still needed for future activities<br>Issued for current wells<br>55-225120<br>55-225121<br>55-225122<br>55-225123<br>55-225124<br>55-225125<br>55-225126<br>55-225127 | | Until well or borehole closed |
| Water Right for Hydrologic testing | ADWR | Issued for production water well testing | Expired – June 25, 2016<br>55-225120<br>55-225121 | June 16, 2015 | One year |

| PERMIT AND AUTHORIZATIONS TABLE | | | | | |
|---|---|---|---|---|---|
| **Permit** | **Lead Agency and Description** | **Submittal Date** | **Status** | **Issue Date** | **Term** |
| Class II Air Permit | ADEQ | Application submittal on Nov 23, 2011,<br>Activities must follow dust control plan | Issued<br>Permit no. 55223 | Jan 31, 2013 | Five years |
| Aquifer Protection Permit | Arizona Department of Environmental Quality (ADEQ) – Groundwater discharge permit (includes Landfill) | Application submitted to ADEQ February 27, 2009, Administratively complete on May 21, 2009 | Issued<br>Permit No. P-106100<br>Place ID 135845<br>LTF 49639 | April 3, 2012 | Life of facility once issued |
| Type 2.02 General Permit | ADEQ – Intermediate Stockpile Permit | Submitted for three stockpiles | Issued<br>Inventory 106100<br>LTF(s) 54136, 54138 and 59138 | Feb 8, 2012 and<br>Jun 12, 2013<br>All revised May 26, 2016 | 7 years<br>May 2, 2018 and<br>May 14, 2020 |
| Type 3.03 General Permit | ADEQ – Vehicle/Equipment Wash | Submitted for Southwest Energy Vehicle Wash 2nd Quarter 2011 | Issued<br>Inventory 106100<br>USAS No 509976-03<br>LTF 64358<br>Place ID 19883 | Sep 12, 2011<br>Renewal form Jul 11, 2016 | 5 years<br>Sept 12, 2021 |
| Stormwater permit | ADEQ – Regulate stormwater discharge quality | AZPDES MSGP for all onsite activities | Covered<br>AZMSG 2010-003 AZMSG-74939 | Feb 7, 2013 | Five years (2015 or until new general permit issued) |
| Construction Stormwater General Permit | Arizona Department of Environmental Quality (ADEQ) – Issued for construction activities | The Notice of Intent (NOI) was received by ADEQ on July 10, 2015 – inactive and unstaffed status | Covered<br>AZCON 86646 | Jul 10, 2015 | Dec 31, 2018 |
| 401 Certification | ADEQ | Application submitted Jan 12, 2012<br>Surface Water Mitigation Plan accepted Dec 2014 | Issued<br>Application NO SPL-2008-00816-MB<br>ADEQ LTF 55425 | Feb 3, 2015 | No expiration |

| PERMIT AND AUTHORIZATIONS TABLE | | | | | |
|---|---|---|---|---|---|
| **Permit** | **Lead Agency and Description** | **Submittal Date** | **Status** | **Issue Date** | **Term** |
| Arizona Mined Land Reclamation Permit | Arizona State Mine Inspector – Permit for reclamation activities at a site. | Submitted planned May 2008 | Complete | July 10, 2009 | Life of facility – annual updates |
| Start-up Notice for Mine Operations | Arizona State Mine Inspector | Registers mine with Arizona State Mine Inspector | Filed | Sept 9, 2009 | |
| Agricultural Land Clearing Permit | Arizona Department of Agriculture – Permit to clear land | Submittal prior to construction of facilities or disturbance of state-protected native plants | As needed | | |
| Certificate of Environmental Compatibility | Arizona Corporation Commission and the Line Siting Committee | Application for power line route submitted by Tucson Electric Power on November 2 | Issued (to TEP)<br>Docket No.<br>L-00000C-11-0400-00164<br>Case No. 164 | CEC signed Dec 19, 2011<br>Approved - Mar 21, 2012<br>Amended - Jun 12, 2012 | Life of facility (Certificate expires in 7 years unless line carries power.) |
| **County Permits and Authorizations Issued** | | | | | |
| Pima County Flood Control District Permit | Pima County Flood Control | Submitted planned Nov 2013 | Issued and renewed<br>FPUP 13-640 (original)<br>FPUP 15-170RP (renewal) | May 2, 2016 | One Year from date of issue |
| **Town Permits and Authorizations Issued** | | | | | |
| Right of Way Encroachment | Town of Sahuarita | This is a 35' encroachment is for a waterline along Santa Rita Road | Issued<br>License Contract No.<br>CO13-0029 | June 24, 2013 | none |

| PERMIT AND AUTHORIZATIONS TABLE | | | | | |
|---|---|---|---|---|---|
| **Permit** | **Lead Agency and Description** | **Submittal Date** | **Status** | **Issue Date** | **Term** |
| **Federal Permits and Authorizations – NOT Issued** | | | | | |
| Mine Plan of Operation | Forest Service – This permit is needed for mining operations on public lands.<br><br>Because all actions are connected to EIS produced for this the MPO is needed to start ground disturbing activities | Plan was submitted in sections:<br>▪ Mine Plan of Operations and supporting documents – July 11, 2007<br>▪ Reclamation Plan – August 7, 2007<br>▪ Infrastructure Plan – July 25, 2007 | Sufficient for NEPA Analysis,<br>NOI issued in FR Mar 13, 2008<br>Final EIS issue Nov 2013<br>Draft ROD issued Dec 2013<br>Supplemental BO issued Apr 22, 2016 | | Life of mine |
| 404 permit | Army Corps of Engineers – Allow operations in Waters of the U.S. There is no formal Corps delineation to date<br>Needed for powerline, waterline, access road and discharge activities | Preliminary Jurisdictional Delineation submitted in May 2009<br>Application for 404 permit re-submitted October 11, 2011<br>Habitat Mitigation and Management Plans submitted – April to Aug 2014 | Pubic Notice published Dec 5, 2011 | | |
| Hazardous Materials Transportation Permit | Department of Transportation (DOT) – Permit needed to transport or received “hazardous materials” under the DOT definitions. | Registration and plan only - will be dependent upon construction schedule but number should be in place prior to construction start. | | | 1-3 years dependent upon permit |
| Radio Licenses | Federal Communications Commission (FCC) | License already exists for current on-site use, additional uses will require additional licenses that will be managed by the contractor (Empire) | Issued for current use – need larger project use license | | |
| Blasting License | Bureau of Alcohol Tobacco and Firearms | Submittal will be dependent upon development of on-site facilities and blasting personnel | | | |

| PERMIT AND AUTHORIZATIONS TABLE | | | | | |
|---|---|---|---|---|---|
| **Permit** | **Lead Agency and Description** | **Submittal Date** | **Status** | **Issue Date** | **Term** |
| **State Permits and Authorizations – NOT Issued** | | | | | |
| State Land Right of Way | Arizona State Land Department | Applications Submitted in Nov 8, 2010 All studies, surveys, and appraisals complete | 30-day notice letters issued in August 2012 awaiting auction | | |
| Encroachment Permit | Arizona Department of Transportation | Traffic Impact Assessment was submitted in July 2010 and the Encroachment Permit application submitted Feb 28, 2011 | Permit in draft, issuance keyed on ROD | | |
| AZPDES DeMinimis Storm water Permit | ADEQ | Submitted as needed for project – well development, drilling, etc. | Issued as necessary and closed when not in use | | |
| Septic System APP | ADEQ – Onsite Wastewater Permit | Awaiting design and percolation testing | | | Life of facility (will be combined with facility-wide APP) |
| Hazardous Waste Identification Number | ADEQ – Issued by EPA/ADEQ so hazardous waste can be generated and transported off site in quantities in excess of 100 pounds. | RCRA EPA ID Number: AZR000509976<br>Requires a contingency plan. | Issued – need to register number with State when activities start | | Life of the facility |
| **County Permits and Authorizations – NOT Issued** | | | | | |
| Hazardous Waste Management | PCDEQ – registration with PCDEQ for all EPA ID Nos. | RCRA EPA ID Number: AZR000509976 | Issued – need to register number with County when activities start | | Life of facility |
| Drinking Water System Registration | PCDEQ – system plans need to be approved prior to installation, registration for all non-community non-transient drinking water systems | Submittal prior to construction of system.<br>Sampling and emergency plans required. | | | |