NI 43-101 Technical Report,

Constancia Mine Cuzco, Peru

March 29, 2018, effective as of December 31, 2017

HUDBAY

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CAUTIONARY NOTE REGARDING FORWARD-LOOKING INFORMATION

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

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

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

- the 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 availability of personnel for our exploration, development and operational projects and ongoing employee relations;
- the ability to secure required land rights to develop the Pampacancha deposit on schedule and as planned;
- maintaining good relations with the communities surrounding the Constancia;
- 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;
- no significant unanticipated litigation;
- certain tax matters, including, but not limited to current tax laws and regulations and the refund of certain value added taxes from the Peruvian government; 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 community protests and trespassers at our Constancia mine), 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, planned infrastructure improvements in Peru not being completed on schedule or as planned, 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.

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

This Technical Report has been prepared for Hudbay Minerals Inc. ("Hudbay") to support the public disclosure of mineral resources and mineral reserves at Constancia mine as of December 31, 2017. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

Hudbay is an integrated Canadian mining company with assets in North and South America, principally focused on the discovery, production, and marketing of base metals 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's operations at Constancia include the Constancia and Pampacancha pits, an ore processing plant, a waste rock facility, a tailings management facility and other ancillary facilities that support the operations.

As of the date of this report, the Constancia mine is in steady state production. The pre-stripping started in March 2014 and the concentrator ramp-up started in December 2014. Commercial production was achieved as of April 30, 2015 and the operations continue in alignment with management's expectations in terms of metal production and cost.

The Qualified Person (the "QP") who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo., Senior Vice President and Chief Operating Officer of Hudbay.

1.1 PROPERTY DESCRIPTION AND LOCATION

The Constancia mine is located approximately 600km southeast of Lima in the south-eastern Andes of Peru, in the Chamaca, Livitaca and Velille districts, province of Chumbivilcas, department of Cusco, at approximately longitude 71°47´ and latitude 14°27´ south.

Hudbay owns an indirect 100% interest in the property, which consists of 36 metallic mining concessions, all of them duly granted by the Peruvian State and recorded in the name of HudBay Peru S.A.C. ("Hudbay Peru"), a wholly-owned subsidiary of Hudbay. Figure 1-1, shows the location of Constancia.

The Constancia mine is accessible from Lima by air, via either Arequipa or Cusco, and then by vehicle over paved and gravel roads. The routes with approximate distances and driving times are shown in

Table 1-1.

The Constancia mine is located approximately 80km from Yauri by road. Power for the Constancia mine is supplied from the new 220kV transmission line from Tintaya to Constancia that was built for and is owned by Hudbay, but operated and maintained by a third party.

Figure 1-1 Constancia Mine Location

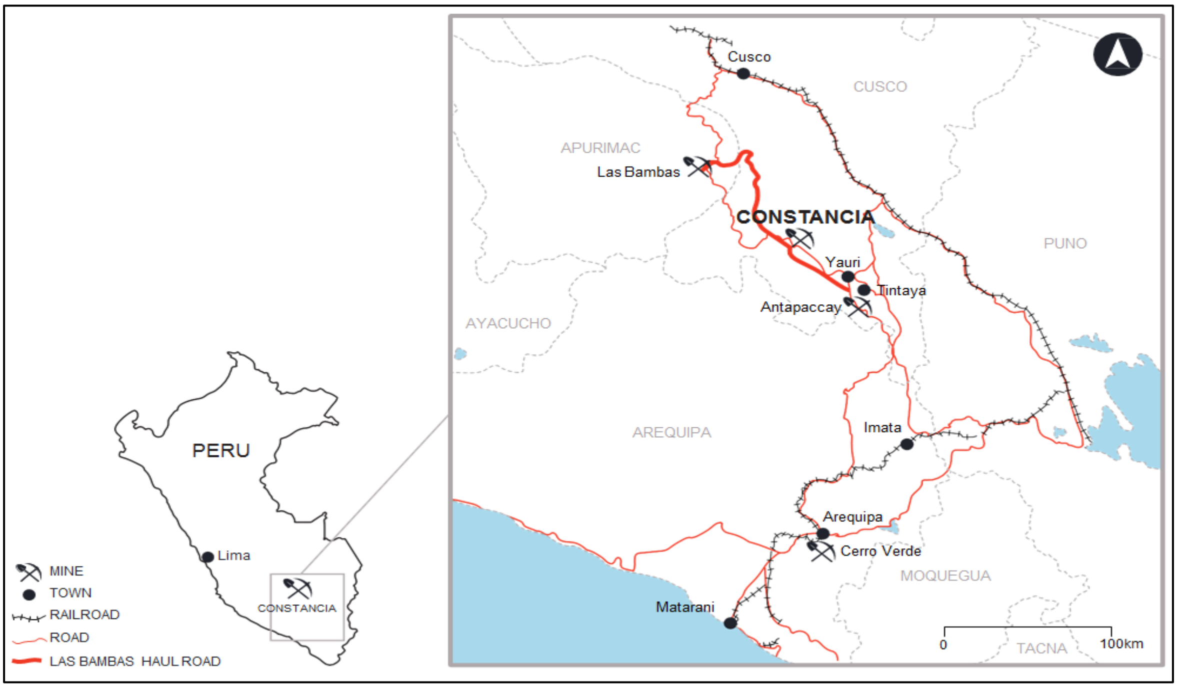


Table 1-1 Approximate Distances and Driving Times to Constancia

From	Leg	Distance (km)	Time (hours)
Arequipa	Canahuasi	80	1.25
	Imata	63	0.75
	Yauri/Espinar	103	2.50
	Uchucarcco turnoff	68	2.00
	Project Site	20	0.50
	Total	**334**	**7.00**
Cusco	Sicuani (paved road)	140	2.00
	El Descanso	40	1.00
	Yauri/Espinar	45	0.75
	Uchucarcco turnoff	68	2.00
	Project Site	20	0.50
	Total	**313**	**6.25**

1.2 GEOLOGICAL SETTING AND MINERALIZATION

CONSTANCIA

The Constancia deposit is a porphyry copper-molybdenum system which includes copper-bearing skarn mineralization. Multiple phases of monzonites and monzonite porphyry have intruded a sequence of sandstones, mudstones and micritic limestone of Cretaceous age.

The majority of the mineralization is associated with potassic alteration and quartz veining, occurring as chalcopyrite-(bornite)-molybdenite-pyrite mineralization in “A” and “B” type veinlets, and replacing ferromagnesian minerals or filling fractures. Copper grades are highest where fracture-filling style copper mineralization is superimposed on earlier disseminated copper mineralization. The higher-grade hypogene copper mineralization is hosted by a dense A-veinlet stockwork developed in an early porphyry phase. The pyrite/chalcopyrite ratio is typically below 2:1. Molybdenite commonly increases with depth in association to “B” veinlets. Bornite occurs sporadically especially at deeper levels, sometimes associated with some gold values.

Propylitic alteration is transitional to the potassic alteration and extends more than one kilometre from the porphyry intrusive contacts. The propylitic alteration mineral assemblage includes epidote-chlorite-calcite- pyrite-rhodochrosite. Subordinate chalcopyrite is also present, filling fractures or replacing mafic minerals. Sphalerite-galena veinlets and veins are distributed as a halo to the copper-molybdenum mineralization within the propylitic alteration zone up to 3 km away from the porphyry copper system.

Phyllic alteration forms a pervasive carapace surrounding and sometimes overprinting potassic alteration. The phyllic alteration accompanies almost complete destruction of primary rock textures; the mineral assemblage includes sericite-quartz-pyrite, limited amounts of chalcopyrite and associated occasional “D” veins and veinlets.

At the contact between intrusions and limestones, a magnetite garnet skarn develops, while a pyroxene–diopside (garnet–epidote) association is more common in calcareous sandstones and arkoses of the Chilloroya formation. Skarn mineralization is volumetrically much smaller, but grades are normally higher.

Structural deformation has played a significant role in concentrating the hydrothermal alteration and the copper-molybdenum-silver-gold mineralization, including skarn formation. Major inter and post mineral fracture systems in the deposit area strike northeast and include the Barite fault system. This is represented by a number of nearly parallel vein-faults carrying base metal sulphides and barite which have been exploited by artisanal workings throughout the property. A second important system strikes north-south. It appears to be more recent than the Barite system and controls part of the San José Pit mineralization and most of the silicified breccias (sometimes mineralized) in the system. This is the same direction as that of the post-mineral dykes and may have originated as tension gashes in the Barite direction.

PAMPACANCHA

The Pampacancha deposit is a porphyry Cu-Mo-Au related Skarn system. Oligocene unmineralized basement diorite is intruded by the diorite porphyry cited as the source for skarn mineralization. This in turn is cut by intra-mineral monzonite intrusions which provide minor local increases in Cu-Au and also locally replaces skarn Cu-Au mineralization which is most developed at the upper and lower margins of the limestone body. Magnetite-chalcopyrite-pyrite skarn ranges to marginal less well mineralized garnet and pyroxene skarn, locally overprinted by epidote-bearing retrograde skarn.

Epithermal mineralization as low sulphidation quartz-sulphide Au + Cu style accounts for common supergene enriched Au anomalies along with other features such as hydrothermal alteration and veins typical of near porphyry locations.

1.3 EXPLORATION

Surface Mapping and Sampling

From 2007 to 2011, 11,444 hectares were mapped in the Constancia project at several scales, including 1:1,000, 1:2,000 and 1:5,000. Of this, 8,905 hectares were mapped on Hudbay's mining concessions, which represent 39% of Hudbay's mining rights in the area. Additionally, 2,595 rock samples and 41 stream sediments samples were collected during this period.

Geophysics

An in-house interpretation of the geophysical data along with interpretation of available surface mapping and rock and stream sediment geochemistry helped identify several targets within the project area. The most important anomalies are associated with the Pampacancha deposit, the Chilloroya South prospect and the chargeability anomalies located in Uchucarco, 3.8 km northeast of the Constancia porphyry.

A Titan-24 DC-IP-MT survey was also completed in July 2011 to the south of the Constancia deposit.

Exploration Targets and Drilling

Several targets have been identified in the Constancia area, including the Chilloroya South prospect, located 5 km south of the Constancia porphyry. Evidence of porphyry-related Cu-Au-Mo mineralization and Cu-Au and Au-only bearing skarns occur in an area of about 3.5 km by 3.5 km coincident with several composite chargeability and magnetic anomalies at depth. Five primary targets were identified and preliminarily drill-tested from June 2010 to January 2011 (totalling 12,029 metres distributed in 35 holes).

In December 2017, Hudbay completed the acquisition of a large, contiguous block of mineral rights including the Caballito, Maria Reyna and Kusiorcco concessions to explore for mineable deposits within trucking distance of the Constancia processing facility. Hudbay is commencing permitting, community relations and technical activities required to access and conduct drilling activities on these properties.

1.4 DRILLING

Extensive drilling has been conducted at the Constancia mine and the Pampacancha deposit by several successive property owners. The most recent drilling was done by Hudbay, with prior drilling campaigns completed by Rio Tinto and Norsemont.

Table 1-2 summarizes the drill holes used to estimate the current mineral resource estimate, with regional exploration holes excluded.

Table 1-2 Drilling Programs by Year (in metres drilled)

Company	PQ	HQ	NQ	RC	HOLES	TOTAL
Rio Tinto (2003-2004)		7,124	359		24	7,483
NOM 2005		9,799			41	9,799
NOM 2006		20,026	377		66	20,403
NOM 2007		23,863	5,197		77	29,060
NOM 2008	3,380	39,502	7,374	12,792	219	63,048
NOM 2009		4,487	113	409	33	5,009
NOM 2010		16,604	1,933	7,694	93	26,231
HB 2011		28,090	1,866	984	186	30,940
HB 2012		5,045	130	464	46	5,639
HB 2014 - 2015		4,353			26	4,353
HB 2017		5,298			21	5,298
GRAND TOTAL	3,380	164,191	17,349	22,343	832	207,273

1.5 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Rio Tinto conducted its own internal Quality Control / Quality Assurance (QA/QC) program to independently evaluate the quality of the assays reported by ALS. Standards and blanks were systematically inserted in the sample stream.

The core from the Norsemont drilling programs from 2005 to 2010 were transported to SGS and ALS Laboratories in Lima, Peru for preparation and analysis. Drill core samples across the Constancia deposit were also measured for specific gravity at ALS Chemex, Lima, Peru. Norsemont conducted its own internal QA/QC program to independently evaluate the quality of the assays reported by SGS and ALS. Duplicates, standards and blanks were systematically inserted in the sample stream.

During the Hudbay 2011 campaign, the samples were transported to the SGS laboratory in Lima, Peru for preparation and analysis while in 2014, the samples were transported to the SGS Constancia laboratory at the mine site for preparation. Once samples were pulverized, a 250 g subsample pulp was collected and air freighted to the SGS laboratory in Lima, Peru, for analysis. In 2015, the samples were transported to the SGS Constancia laboratory at the mine site for preparation and analysis and in 2017, for the twin hole drilling program, blanks and standards were inserted at site, prior to dispatching the core boxes to the Certimin laboratory in Lima for sample preparation followed by analysis at the Certimin and SGS laboratories in Lima.

As part of Hudbay's QA/QC programs, samples were systematically introduced in the sample stream to assess adequate sub-sampling procedures, potential cross-contamination, precision, and accuracy.

1.6 MINERAL PROCESSING AND METALLURGICAL TESTING

The ore types that are currently being processed in the concentrator were established during the Definitive Feasibility Study (DFS) as the hypogene, supergene, skarn, mixed and high zinc types. The metallurgical responses of these ores are acceptable in terms of treatment rate, recovery and molybdenum and copper concentrate grades. The copper grade in the final concentrate is higher than 24%, with low levels of zinc, lead, iron, etc. The molybdenum concentrate produced is over 50% molybdenum with low contents of copper, lead, iron, etc. Metallurgical test work performed at laboratory and plant levels from hypogene, skarn, supergene, high zinc and mixed composites have enabled to optimize the reagents for each ore type.

The Pampacancha ore recovery and throughput assumptions when processed at the Constancia plant are based on prefeasibility level of engineering.

Table 1-3 shows the copper recoveries estimated for the 2018 to 2036 period. Copper recoveries for 2018 and 2019 are based on conservative assumptions calibrated on past operating practices. For the following years, improvements related to projects currently under development have been incorporated to incorporate planned increases in flotation recovery to both the coarse and fines fraction recovery.

Table 1-3 Copper Recovery

Year	Cu,%	Cu Recovery, %
2018	0.416	82.56%
2019	0.413	84.61%
2020	0.394	85.91%
2021	0.391	86.00%
2022	0.391	86.11%
2023	0.386	85.73%
2024	0.314	86.44%
2025	0.326	88.19%
2026	0.264	86.46%
2027	0.253	86.14%
2028	0.280	87.47%
2029	0.296	86.08%
2030	0.296	87.34%
2031	0.254	86.78%
2032	0.247	86.61%
2033	0.234	86.49%
2034	0.273	86.25%
2035	0.292	88.32%
2036	0.186	83.77%

1.7 MINERAL RESOURCE ESTIMATES

CONSTANCIA

The Constancia mineral resource estimate is effective as of December 31, 2017.

The initial mineral resource and mineral reserve estimate for Constancia was completed by AMEC in 2012, on behalf of Hudbay, and published by Hudbay in a technical report dated October 15, 2012 and titled “National Instrument 43-101 Technical Report, Constancia Project, Province of Chumbivilcas, Department of Cusco, Peru”.

Additional work performed by Hudbay between 2012 and 2016 led to mineral resource and mineral reserve estimate updates supporting a technical report filed on the property by Hudbay in November 2016 (the “2016 Technical Report”).

After carrying out additional work in 2017, Hudbay completed a new mineral resource and mineral reserve estimate for Constancia, with an effective date of December 31, 2017 and constitutes the basis for the updated mineral resource estimate presented in this report.

Resource modeling at Constancia is based on integrated geological and assay interpretations of information recorded from diamond core logging and assaying and is comprised of the following key steps: Exploratory Data Analysis, Modelling (Composites, Variography and Interpolation) and Validations. A total of 167,264 m (699 holes) have been drilled at the Constancia deposit.

The Constancia geological model was developed from an initial interpretation of six lithology domains, six alteration types and five oxidation zones. The oxidation model is based on sequential copper formulation and logging codes. This geological framework was then used to model two continuous estimation domains hosting the mineralization grading above 0.1% Cu respectively in the supergene and hypogene domains. Grade estimation was also conducted in a third domain solely for the purpose of estimating additional low-grade resources and is comprised of all the 'low grade' units including low-grade monzonite dykes, weakly mineralized monzonite, sediments and the oxidation/leaching cap.

For the two domains of mineralization, the previous criteria for resource classification based on a combination of estimation variance and drill spacing were also compared to the regression slope of the kriged estimates of the Cu grade. The classification criteria were validated and essentially maintained with some local adjustments to produce resource category domains that are smoother and more continuous. The criteria used for measured resources was thoroughly tested through reconciliation to confirm that quarterly tonnes, %Cu and metal content could all be predicted within +/-15% with a 90% level of confidence for the past 2.5 years of mining. The kriged regression slope for measured resources at Constancia is >80% and is between 60 and 80% for indicated resources. All resources estimated within the third 'low-grade' domain are categorized as inferred resources and exclude any material from the oxidized/leach cap zone.

The mineral resource estimates inclusive of mineral reserve estimates are reported in Table 1-4 at a 6.04 US$/tonne Net Smelter Return (NSR) cut-off and inside an economic pit shell demonstrating that they have reasonable prospects for economic extraction under the following assumptions: a copper price of US$3.00/lb, a molybdenum price of US$11.00/lb, copper recovery of 90.5% (copper primary zone), molybdenum recovery of 55%, and a processing cost of US$4.44/tonne milled and a mining cost of US$1.35/tonne mined.

The comparison between the December 31st, 2016 and December 31st, 2017 mineral resource estimates presented in Table 1-5 shows that the measured + indicated and inferred resource tonnage has been reduced in the 2017 estimates by respectively 78Mt and 84Mt due to the use of a more restrictive modelling envelope to prevent the smearing of high grade samples into large low grade domains. Conversely, the reduction of smoothing in the modelling process has enhanced the grade of the main metals of economic interest in all categories of resources further improving the potential for economic extraction of these mineral resource estimates. The copper grade improvement is compounded by the correction of a sampling bias identified in the supergene domain at Constancia.

Table 1-4 Constancia Mineral Resource Estimates Inclusive of Mineral Reserves as of December 31st, 2017

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	627,900,000	≥ $6.04	0.27	84	2.72	0.033
Indicated	253,700,000	≥ $6.04	0.21	61	2.38	0.034
Mea+Ind	**881,600,000**	≥ $6.04	**0.25**	**77**	**2.62**	**0.033**
Inferred	*54,100,000*	≥ $6.04	*0.24*	*43*	*1.71*	*0.018*

- Mineral resources that are not mineral reserves do not have demonstrated economic visibility. The above mineral resources include mineral reserves.
- Mineral resources are constrained within a computer generated pit using the Lerchs-Grossman algorithm. Metal prices of US$3.00/lb copper, US$11.00/lb molybdenum, US$18.00/troy oz silver and US$1260.00/troy oz gold. Metallurgical recoveries of 90.5% copper, 55% molybdenum, 72% silver and 60% gold were applied to hypogene material. Metallurgical recoveries of 88.4% copper, 55% molybdenum, 90% silver and 60% gold were applied to mixed and supergene material. A metallurgical recovery of 84.4% copper, 55% molybdenum, 52% silver and 60% gold for copper was applied to skarn and Hi Zinc material. NSR was calculated for every model block and is an estimate of the recovered economic value of copper, molybdenum, silver and gold 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 US$6.04/ton.
- The NSR cut-off was applied to a Selective Mining Unit (SMU) size of 10mx10mx15m above level 3,930m and to a 20mx20mx15m SMU size below this level assuming a less selective mining operation as the mine will progress at depth to lower grade and less variable hypogene mineralization.

Table 1-5 Comparison Between the December 31st, 2016 and December 31st, 2017 Mineral Resource Estimates for Constancia

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	987,993,000	0.24	72	2.4	0.032	2,322,000
B	2017 Production (Depletion)	28,700,000	0.52	126	3.9	0.040	150,000
C	A - B	959,293,000	0.23	70	2.4	0.032	2,172,000
D	Mineral Resource as of December 31st 2017	881,600,000	0.25	77	2.6	0.033	2,247,000
E	Gain/(Loss) - New block Model[2]	(77,693,000)					75,000

	Mineral Resource Reconciliation (Inferred)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	138,069,000	0.17	40	1.7	0.018	289,000
B	2017 Production (Depletion)						
C	A - B	138,069,000	0.17	40	1.7	0.018	233,000
D	Mineral Resource as of December 31st 2017	54,100,000	0.24	43	1.7	0.018	127,000
E	Gain/(Loss) - New block Model[2]	(83,969,000)					(106,000)

Notes:
[1]Totals may not add up correctly due to rounding
[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

PAMPACANCHA

Resource estimation at Pampacancha is based on a total of 140 holes (38,240 metres) with 28 of those being derived from reverse circulation drilling and the remaining 112 from HQ diameter diamond drilling. All holes were drilled from surface by Geotec. Core recovery is near 100% for all holes.

The drilling results were used to enable the preparation of a 3D geological interpretation and estimation of mineral resources. The skarn unit hosts the vast majority of the copper-gold-silver-molybdenum mineralization.

The Pampacancha geological model was developed from an initial interpretation of a high grade and a low grade skarn domain and further sub-divided into three sectors. This geological framework was further refined in 2017 by eliminating hard boundaries between sectors and high grade-low grade domains while using a smooth grade envelope defined at a 0.1% Cu cut-off to limit grade smearing into the host rocks. The most important change in the 2017 update of the resource model at Pampacancha is the recognition that a strong correlation exists between most metal grade and specific gravity. The proper weighting when compositing drill holes and interpolating grade by specific gravity has resulted in an increase in the copper as well as in most of the other metal content.

Similarly to Constancia, the regression slope values obtained from the kriging of the copper grade estimates were used as a basis for resource classification. 90% and 80% regression slope thresholds were used respectively to separate measured from indicated and indicated from inferred resources. Since the scale of the operation will be significantly smaller and no mining has yet occurred at Pampacancha, the threshold to report measured and indicated was set higher than at Constancia which used respectively 80% and 60% thresholds on the kriging regression slope.

The mineral resource estimates inclusive of mineral reserves are reported in Table 1-6 at a 6.04 US$/tonne Net Smelter Return (NSR) and inside a pit shell demonstrating that they have reasonable prospects for economic extraction under the following assumptions: a copper price of US$3.00/lb, a molybdenum price of US$11.00/lb, copper recovery of 90.5% (copper primary zone), molybdenum recovery of 55%, and a processing cost of US$4.44/tonne milled and mining cost of US$1.35/tonne mined.

As for Constancia, the comparison of the December 31st, 2016 and December 31st, 2017 mineral resource estimates shows a decrease in tonnage due to a more restrictive modelling of the mineralized envelope offset by a significant increase in grade (Table 1-7). The increase in grade has been further enhanced by the weighting of sample compositing and grade interpolation by specific gravity, which is strongly correlated to sulfide mineralization. Table 1-7 also shows the addition of 10Mt of inferred resources in the 2017 estimates which provide some options to extend the life of the mine after the reserves have been mined out.

Table 1-6 Pampacancha Mineral Resource Estimates Inclusive of Mineral Reserves as of December 31st, 2017

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	43,900,000	≥ $6.04	0.55	158	4.60	0.336
Indicated	13,400,000	≥ $6.04	0.50	133	5.49	0.307
Mea+Ind	**57,300,000**	≥ $6.04	**0.54**	**152**	**4.81**	**0.329**
Inferred	10,100,000	≥ $6.04	0.14	143	3.86	0.233

- Mineral resources that are not mineral reserves do not have demonstrated economic visibility. The above mineral resources include mineral reserves.
- Mineral resources are constrained within a computer generated pit using the Lerchs-Grossman algorithm. Metal prices of US$3.00/lb copper, US$11.00/lb molybdenum, US$18.00/troy oz silver and US$1260.00/troy oz gold. Metallurgical recoveries of 85% copper, 40% molybdenum, 70% silver, 70% gold, 41% lead and 30% zinc were applied to skarn material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, silver and gold 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 US$6.04/ton.
- The NSR cut-off was applied to a Selective Mining Unit (SMU) size of 20mx20mx15m.

Table 1-7 Comparison Between the December 31st, 2016 and December 31st, 2017 Mineral Resource Estimates for Pampacancha

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	65,746,000	0.40	129	3.9	0.249	289,000
B	2017 Production (Depletion)						
C	A - B	65,746,000	0.40	129	3.9	0.249	263,000
F	Mineral Resource as of December 31st 2017	57,309,200	0.54	152	4.8	0.329	307,000
E	Gain/(Loss) - New block Model[2]	(8,436,800)					44,000
	Mineral Resource Reconciliation (Inferred)	**Tonnes[1]**	**Cu%**	**Mo (g/t)**	**Ag (g/t)**	**Au (g/t)**	**Tonnes Cu**
A	2016 Mineral Resource as of December 31st 2016	191,000	0.14	92	3.2	0.189	300
B	2017 Production (Depletion)						
C	A - B	191,000	0.14	92	3.2	0.189	300
F	Mineral Resource as of December 31st 2017	10,100,000	0.14	143	3.9	0.233	14,400
E	Gain/(Loss) - New block Model[2]	9,909,000					14,100

Notes:
[1]Totals may not add up correctly due to rounding
[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

1.8 MINERAL RESERVE ESTIMATES

Mineral Reserves estimates for the Constancia and Pampacancha deposits, which are presented in this report, were prepared by Hudbay. The Qualified person who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo. Senior Vice-President and Chief Operating Officer. The mineral resource block model, as describe in Section 14 was used as the basis for the mineral reserve block model.

A copper grade equivalent optimization (Cu-eq.) model is simpler to implement than an NSR model but does not adequately represent the many variables used in the calculation of revenues. Hudbay has opted to use an NSR optimization model taking into account the Cu, Mo, Ag, and Au grades, mill recoveries, contained metal in concentrate, deductions and payable metal values, metal prices, freight costs, smelting and refining charges and royalty charges. Table 1-8 show the Metal prices used in the NSR calculation.

Table 1-8 Metal Prices

Revenue	Units	Constancia & Pampacancha
Metal Price		
Copper	$/lb	3.00
Molybdenum	$/lb	11.00
Silver	$/oz	18.00
Gold	$/oz	1,260.00

The 2017 Proven and Probable Reserve estimates at the Constancia mine total 568.6 million tonnes at a copper equivalent grade of 0.41% that support a 19 year mine life. The mine plan is based on the capacity of the process plant, which in turn relies on the grinding circuit throughput.

The plant has the capacity to process 31 Mtpy (90ktpd at 94% availability), with a mining rate of 70.4 Mtpy (ore plus waste).

Table 1-9 Constancia and Pampacancha Mineral Reserves as at December 31st, 2017

	Category	Mt	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)	CUEQ
Constancia	Proven	452.9	0.30	96	2.92	0.035	0.39
	Probable	72.8	0.23	72	3.09	0.035	0.31
	Total 2P	525.7	0.29	93	2.94	0.035	0.38
Pampacancha	Proven	32.4	0.59	178	4.48	0.37	0.92
	Probable	7.5	0.62	173	5.75	0.33	0.93
	Total 2P	39.9	0.60	177	4.72	0.360	0.92
Stockpile	Proven	3.1	0.4	98	4.68	0.068	0.53
	Proven	488.4	0.32	102	3.03	0.057	0.42
	Probable	80.3	0.27	82	3.33	0.062	0.37
Total		**568.6**	**0.32**	**99**	**3.07**	**0.058**	**0.41**

1. The mineral reserve estimates for the Constancia operations are based on a long range mine plan (LOM) with economic value calculation per block (NSR in $/t), mining, processing, and detailed engineering parametres.
2. The Constancia reserve pits (Constancia and Pampacancha) consist of operational pits of Proven and Probable reserves and are based on the following Long-term metal prices: copper US$3.00 per pound, molybdenum US$11.00 per pound, silver US$18.00 per ounce and gold US$1,260 per ounce; metallurgical revering applied by ore type (between 84.4% to 90.5%); and processing cost of US$4.44 per tonne, G&A costs of US$1.60 per tonne and mining costs of US$1.30 and US$1.35 per tonne (Waste and ore respectively).

A comparison of the December 31st 2016 and December 31st 2017 Mineral Reserve estimate is summarised in Table 1-10 for the Constancia pit and in

	Mineral Reserve Reconciliation (Proven & Probable)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2017 Mineral Reserve	541,200,000	0.28	88	2.8	0.037	1,538,000
B	2017 Production (Depletion)	28,700,000	0.52	126	3.9	0.040	150,000
C	(A - B)	512,500,000	0.27	86	2.7	0.037	1,388,000
G	Geology & Mine Planning (Gain/Loss)	16,200,000	-	-	-	-	170,000
H	2018 Mineral Reserve (C + G)	528,700,000	0.29	93	3.0	0.035	1,558,000

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
I	2017 Mineral Resource (Measured & Indicated)	449,500,000	0.18	52	2.0	0.028	797,000
J	2018 Mineral Resource (Measured & Indicated)	356,000,000	0.20	54	2.1	0.030	701,000
K	(J - I) Gain[2]/(Loss)	(93,500,000)	-	-	-	-	(96,000)

	Mineral Resource Reconciliation (Inferred)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
L	2017 Mineral Resource (Inferred)	138,100,000	0.17	40	1.7	0.018	233,000
M	2018 Mineral Resource (Inferred)	54,100,000	0.24	43	1.7	0.018	127,000
N	(M - L) Gain[2]/(Loss)	(84,000,000)	-	-	-	-	(106,000)

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

Table 1-11 for the Pampacancha pit. The December 31st 2017 Constancia Reserve estimate shows an increase of 12% after substracting the 2017 mining depletion in the copper contained in the Mineral Reserves estimates at Constancia and a 13% increase at Pampacancha, in both cases driven by a significant increase in Cu grade. This increase results from a combination of improvements in the resource modeling process at both Constancia and Pampacancha and the correction of a sampling bias identified in the Supergene Domain at Constancia.

Table 1-10 Comparison of the Mineral Reserve Estimates at Constancia between December 31st 2016 and December 31st, 2017

	Mineral Reserve Reconciliation (Proven & Probable)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2017 Mineral Reserve	541,200,000	0.28	88	2.8	0.037	1,538,000
B	2017 Production (Depletion)	28,700,000	0.52	126	3.9	0.040	150,000
C	(A - B)	512,500,000	0.27	86	2.7	0.037	1,388,000
G	Geology & Mine Planning (Gain/Loss)	16,200,000	-	-	-	-	170,000
H	2018 Mineral Reserve (C + G)	528,700,000	0.29	93	3.0	0.035	1,558,000

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
I	2017 Mineral Resource (Measured & Indicated)	449,500,000	0.18	52	2.0	0.028	797,000
J	2018 Mineral Resource (Measured & Indicated)	356,000,000	0.20	54	2.1	0.030	701,000
K	(J - I) Gain[2]/(Loss)	(93,500,000)	-	-	-	-	(96,000)

	Mineral Resource Reconciliation (Inferred)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
L	2017 Mineral Resource (Inferred)	138,100,000	0.17	40	1.7	0.018	233,000
M	2018 Mineral Resource (Inferred)	54,100,000	0.24	43	1.7	0.018	127,000
N	(M - L) Gain[2]/(Loss)	(84,000,000)	-	-	-	-	(106,000)

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

Table 1-11 Comparison of the Mineral Reserve Estimates at Pampacancha between December 31st, 2016 and December 31st, 2017

	Mineral Reserve Reconciliation (Proven & Probable)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2017 Mineral Reserve	43,000,000	0.49	156	4.2	0.276	210,000
B	2017 Production (Depletion)	-	-	-	-	-	-
C	(A - B)	43,000,000	0.49	156	4.2	0.276	210,000
G	Geology & Mine Planning (Gain/Loss)	(3,100,000)	-	-	-	-	28,000
H	2018 Mineral Reserve (C + G)	39,900,000	0.60	177	4.7	0.360	238,000

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
I	2017 Mineral Resource (Measured & Indicated)	22,700,000	0.23	79	3.3	0.198	53,000
J	2018 Mineral Resource (Measured & Indicated)	17,400,000	0.39	95	5.0	0.258	69,000
K	(J - I) Gain[2]/(Loss)	(5,300,000)	-	-	-	-	16,000

	Mineral Resource Reconciliation (Inferred)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
L	2017 Mineral Resource (Inferred)	-	-	-	-	-	-
M	2018 Mineral Resource (Inferred)	10,100,000	0.14	143	3.9	0.233	14,000
N	(M - L) Gain[2]/(Loss)	10,100,000	-	-	-	-	14,000

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

The updated mineral reserve estimates are supported by extensive reconciliation work conducted by Hudbay in 2017. This work is detailed in section 14 and shows a marked improvement in reconciliation results between the 2016 and 2017 reserve models.

1.9 MINING METHODS

The Constancia mine is a traditional open pit operation using conventional truck and shovel mining. The Constancia ultimate pit design will measure approximately 1.8 km east to west, 1.6 km north to south, and have a maximum depth of approximately 660 m. The Pampacancha ultimate pit design will measure approximately 0.6 km east to west, 1 km north to south, and have a maximum depth of approximately 300 m. A primary waste rock facility (WRF), which is located to the south and east of the Constancia pit, is intended to be used for both deposits.

The processing facility is located approximately 1 km west of the Constancia Pit. A non-acid generating (NAG) stockpile for waste material is located at the south side while the tailings management facility (TMF) is located 3.5 km to the southwest of the Constancia pit.

Mine Phases

Final pit limit designs have been created for Constancia and Pampacancha based on the selected pit shells from the pit optimization. Nine pit stages were developed for Constancia and two pit stages for Pampacancha Pit. The minimum mining width for each phase is 60 metres which allows for operation of a shovel, trucks (in two lines) and a drill. The phased development strategy consists of extracting the highest 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. The following
Figure 1-2 shows the footprint of the ultimate pits for Constancia and Pampacancha at the end of the mine life.

The mineral reserve estimates have been defined at a NSR Cut-off of 6.14 $/t both for Constancia and Pampacancha. The material with a marginal NSR value is treated as waste. The total in situ ore reserves in the final pit as of December 31st 2017 are estimated to be 568.6 million tonnes. Approximately 3.1 million tonnes of medium and low grade ore has been stocked in temporary stockpiles. At the end of the mine life the stockpiles will be reclaimed and processed.

Figure 1-2 Constancia and Pampacancha Ultimate Pit Design

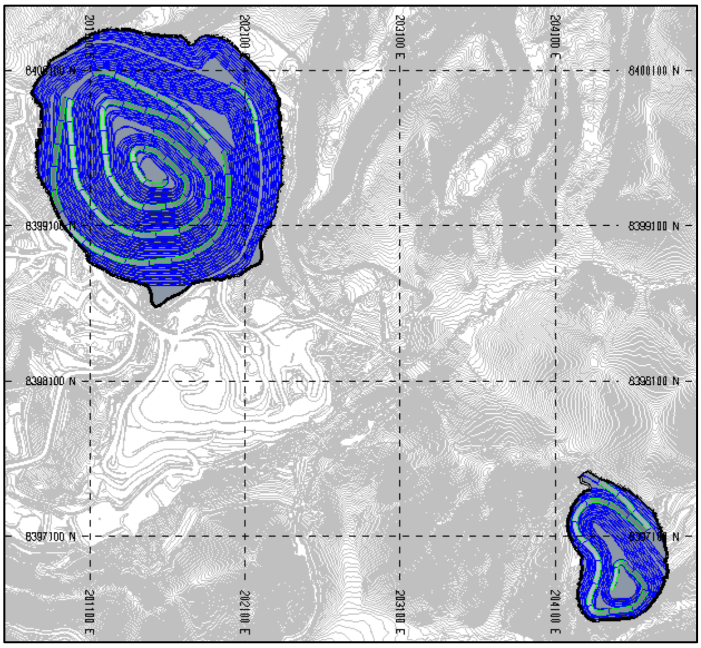


Table 1-12 Constancia and Pampacancha Breakdown by Phases as of December 31st, 2017

PIT	Phases	Ore, Mt	Cu%	Mo%	Auppm	Agppm	Waste, Mt
Constancia	01	6.1	0.35	0.014	0.029	3.08	2.4
	02	81.2	0.38	0.013	0.034	3.44	38.4
	04	102.1	0.30	0.011	0.037	2.97	61.7
	05	101.8	0.29	0.009	0.033	3.37	107.4
	06	66.9	0.30	0.009	0.043	2.54	79.5
	07						8.5
	08	67.4	0.23	0.007	0.031	2.75	153.8
	09	100.2	0.26	0.006	0.033	2.46	86.5
	Sub Total	**525.7**	**0.29**	**0.009**	**0.035**	**2.94**	**538.0**
Pampacancha	PC01	22.8	0.56	0.020	0.356	4.42	39.5
	PC02	17.1	0.64	0.014	0.365	5.12	40.5
	Sub Total	**39.9**	**0.60**	**0.018**	**0.360**	**4.72**	**79.9**
Stockpiles	I - V	**3.1**	**0.41**	**0.010**	**0.068**	**4.68**	
	TOTAL	**568.6**	**0.32**	**0.010**	**0.058**	**3.07**	**618.0**

Mine Production Schedule

The operating and scheduling criteria used to develop the mining sequence plans are summarized in Table 1-13 below.

Table 1-13 Mine Production Schedule Criteria

Annual Moved Production Base Rate	70.4 Mtonnes
Annual Ore Production Base Rate	**30.9 - 31.4 Mtonnes**
Daily Ore Production Base Rate	90 - 94 ktpd
Process Plant yearly availability	94%
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 are being scheduled around the clock. Allowances for down time and weather delays have been included in the mine equipment and manpower estimations.

An elevated cut-off grade strategy is implemented to bring forward the 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. Priority plant feed consists of higher grade material (NSR > 10$/t). The lower grade material is processed as needed or sent to long term ore stockpiles to be reclaimed at the end of mine life.

The mine production plan contains 618 Mt of waste and 569 Mt of ore (from pit and stockpiles), yielding a stripping ratio (waste / ore) of 1.1 to 1. An average yearly mining rate of 70.4 Mtpy is required to provide a nominal ore process feed rate of 31 Mtpy based on a variable throughput by pre type (90 to 94ktpd at 94% availability). The estimated plant feed schedule is summarized in Table 1-14.

Waste Rock Facility (WRF)

The final design geometry for this facility incorporates the same slope profile as the original WRF: 20 m high benches with 1.4H: 1V (36°) of bench slopes and 32 m wide catch benches. The overall slope of the stockpile will be about 3.0H: 1V (18°). The remaining capacity of the WRF is estimated at 498 Mt and as such there is sufficient capacity with minor design adjustments to accommodate the needs for waste rock disposal of the revised LOM plan.

Ore Stockpiles

Constancia Operations includes four ore stockpiles; these are distributed by type and ore grades. Lift height of these stockpiles is 12 or 15 metres. The ore stockpile capacities are as follows:

- Stockpile 01 : 600 Kt
- Stockpile 02 : 500 Kt
- Stockpile 03 : 600 Kt
- Stockpile 04 : 7,000 Kt

Table 1-14 Ore Processing Plan 2018 – 2036 (Total)

LOM		CP18Mar																			TOTAL
Year		2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
PRODUCTION	Tonnes to Cu Plant, kt	31,418	31,261	31,198	31,129	31,052	31,156	31,148	31,004	31,039	31,056	31,054	31,004	31,017	31,070	31,181	30,986	30,980	30,913	8,949	568,617
	Cu%	0.42	0.41	0.39	0.39	0.39	0.39	0.31	0.33	0.26	0.25	0.28	0.30	0.30	0.25	0.25	0.23	0.27	0.29	0.19	0.32
	Cu In Situ, Mlb	288	285	271	269	268	265	216	223	181	173	192	202	202	174	170	160	187	199	37	3,961
	Au (g/t)	0.04	0.07	0.13	0.14	0.15	0.10	0.04	0.03	0.03	0.03	0.03	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.04	0.06
	Ag (g/t)	4.08	3.67	3.05	3.53	3.55	3.80	3.38	2.74	3.00	2.91	2.70	3.19	2.65	2.74	2.85	2.46	2.62	2.30	3.40	3.07
	Tonnes to Mo Plant, kt	6,391	12,084	22,601	26,656	21,199	19,704	19,227	19,247	16,656	12,699	17,603	15,196	13,962	14,239	13,406	6,526	8,668	19,947		286,012
	Mo%	0.017	0.017	0.021	0.019	0.012	0.015	0.011	0.015	0.013	0.014	0.012	0.012	0.010	0.010	0.012	0.008	0.009	0.009		0.013
RECOVERIES	CuT Recovery, %	82.6%	84.6%	85.9%	86.0%	86.1%	85.7%	86.4%	88.2%	86.5%	86.1%	87.5%	86.1%	87.3%	86.8%	86.6%	86.5%	86.3%	88.3%	83.8%	86.1%
	Au (g/t) Recovery, %	52.0%	58.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	59.6%
	Ag (g/t) Recovery, %	66.5%	67.6%	67.7%	70.2%	70.6%	69.9%	69.3%	69.6%	69.3%	68.7%	68.3%	66.3%	69.5%	68.8%	68.2%	67.5%	67.0%	70.4%	64.7%	68.6%
	MoT Recovery, %	25.0%	35.0%	48.0%	53.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	52.0%
CONC. Cu	Conc. Tonnes	433,535	433,369	408,257	390,279	393,224	400,955	349,145	356,448	292,606	281,709	308,966	327,739	324,472	280,758	274,855	261,961	306,350	318,724	58,932	6,202,284
	Cu% into Conc.	24.9%	25.2%	25.9%	26.8%	26.6%	25.7%	24.2%	25.0%	24.3%	24.1%	24.6%	24.1%	24.7%	24.4%	24.3%	24.0%	23.8%	25.0%	23.6%	25.0%
	Au (g/t) into Conc.	1.6	2.8	6.0	6.7	7.2	4.5	1.9	1.7	2.1	2.2	2.0	2.4	2.4	2.7	2.5	2.2	1.9	1.9	3.3	3.2
	Ag (g/t) into Conc.	196.4	178.8	158.0	197.9	197.7	206.6	208.8	165.8	220.7	220.1	185.6	200.0	176.0	208.4	220.7	196.6	177.8	157.3	333.7	193.2
CONTAINED METAL IN CONC. Cu	Cu Pounds, Mlb	237.8	241.0	232.7	231.0	230.6	227.1	186.6	196.7	156.5	149.4	167.8	174.0	176.7	150.9	147.2	138.4	161.0	175.5	30.7	3,412
	Cu Fine, KTonnes	107.9	109.3	105.6	104.8	104.6	103.0	84.7	89.2	71.0	67.8	76.1	78.9	80.1	68.4	66.8	62.8	73.0	79.6	13.9	1,548
	Acum Cu Fine, KTonnes	108	217	323	428	532	635	720	809	880	948	1,024	1,103	1,183	1,251	1,318	1,381	1,454	1,534	1,548	
	Au Fine, KOunces	22	39	78	84	91	57	22	19	20	20	20	26	25	24	23	18	19	20	6	631
	Ag Fine, KOunces	2,738	2,492	2,074	2,483	2,500	2,663	2,343	1,900	2,076	1,994	1,844	2,108	1,836	1,881	1,951	1,656	1,751	1,612	632	38,534
CONC. Mo	Conc. Mo Tonnes	532	1,408	4,494	5,349	2,811	3,283	2,364	3,088	2,461	1,886	2,308	1,934	1,488	1,490	1,704	581	873	2,016		40,069
	Mo% into Conc. Mo	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%		50.0%
CONTAINED Mo	Mo Fine, KTonnes	0.3	0.7	2.2	2.7	1.4	1.6	1.2	1.5	1.2	0.9	1.2	1.0	0.7	0.7	0.9	0.3	0.4	1.0		20

Mine Equipment

There are several different rock types at the Constancia Operation but for production estimation purposes, the weighted average of all rock types was used. Major loading and haulage equipment is equipped with electronic monitors which will ensure maximum loading. All production is reported in dry metric tonnes which is consistent with the reserve model. A moisture content ranging between 3.5 and 4.5 percent was measured, 4.0 percent was used for haulage calculations.

Mine equipment requirements were developed based on the annual tonnage movements projected by the mine production schedule with bench heights of 15 metres, operating two twelve-hour shifts per day, 365 days per year. Productivity is based on current performance and material characteristics specific to the deposit. The fleet requirements also considered:

- production drilling;
- loading and hauling of sulfide ore to the primary crusher (located on the east side of the pit), to waste rock (WRF) and to tailings dam (TMF) areas;
- maintenance of mine haulage and access roads; and
- maintenance of the waste rock facility (WRF) areas, ore stockpiles, tailing management facility (TMF) and berms, and regrading of slopes and final surfaces.

Mine Engineering

In 2013, Knight Piesold developed a feasibility study for the Constancia pit slope design which was based on previous studies and additional geotechnical investigations. For Pampacancha, the study, “Constancia Expansion – Pampacancha Feasibility Study”, completed by TWP/Itasca (August, 2013), provided the pit slope design angles.

At the end of the updated mine life in 2035, the pit footprint at ground surface will be approximately 15% larger, and the ultimate pit floor will be approximately 30m deeper than the previous pit design (3750 m elevation versus 3780 m elevation). The predicted total dewatering rates for the Pit Expansion Design and the area of influence will be similar to the 180 L/s level approved in the Groundwater Use Licence. The design was partially driven by requirements for mine water supply but the main goal of the dewatering is to depressurize the pit wall to assure the overall pit slope stability.

1.10 RECOVERY METHODS

The Process Plant was designed, built and commissioned by Ausenco. the concentrator throughput capacity is 90 ktpd with a 94% physical availability. The Constancia Process Plant consists of the following areas: crushing, stockpiling, milling, flotation (rougher and cleaner), regrind, thickening, filtration, tailings and the molybdenum processing plant. Final products are copper and molybdenum concentrates which are sent via road from site to the port of Matarani for shipment to customers.

Power supply is provided by the south national energy system in the highland sector (New Tintaya sub-station). Water for the process plant is provided by recirculation from the tailings management facility (TMF), process water from thickening, and dewatering wells from the mine (fresh water).

For concentrator plant control, a DCS system is used for starting and stopping equipment, level monitoring, lubrication systems, flows, temperature, equipment protection alerts, among others.

1.11 PROJECT INFRASTRUCTURE

This section addresses the infrastructure facilities that support the current operation including the waste rock facility, the tailings management facility, water management, electric power supply and transmission, and improvements to the roads and the port.

The TMF is located on the south side of the Chilloroya River and has been designed to store 600 million tonnes (Mt) of tailings with a final design elevation of 4,186 masl. The TMF has two stages. The first designed by KP (engineering detail) to 4,160 masl and the second stage design by Golder (prefeasibility study) that rises from 4,160 to 4,186 masl without moving the previous footprint. The actual elevation of embankment is 4,097 masl, as approved by the authority for operations until 2019 with with two metres of freeboard. The design of the TMF includes a LLDPE geomembrane liner and underdrains for valleys.

Tailings have been deposited from designated off-take points from a distribution pipeline located along the upstream crest of the embankment at an elevation of 4,097m (spigotting system) and around the perimeter of the facility (discharge diffuser).

The Potentially Acid Generating and Non-Acid Generating Waste Rock Facilities (PAG & NAG WRF) are located in the Cunahuiri Valley east of the Constancia Pit and provide storage for 581 Mt. The facilities receive mine waste material from the operation of the Constancia pit and from the Pampacancha pit. Underdrain systems include groundwater underdrains to collect the water to WRF Retention pond.

The WRF Retention Pond is located downstream from the WRF in the Cunahuiri Valley and provides energy dissipation for surface water. The WRF Containment Pond is located downstream of the WRF Retention Pond to provide storage capacity for surface runoff, direct precipitation and seepage collected from the WRF and its contact diversion channels. Water management structures also include diversion channels, sediment ponds, water management ponds and main sediment pond. Hudbay Peru performed a site-wide process water balance for the final design phase of the Constancia mine based on the updated Mine Plan.

There are two camps on-site: Constancia is a permanent camp with a 2,930 people capacity and Fortunia is a pioneer camp and served as an overflow camp during the construction stages for 627 people. These camps have been designed with reference to the IFC standards for camp construction for mining activities.

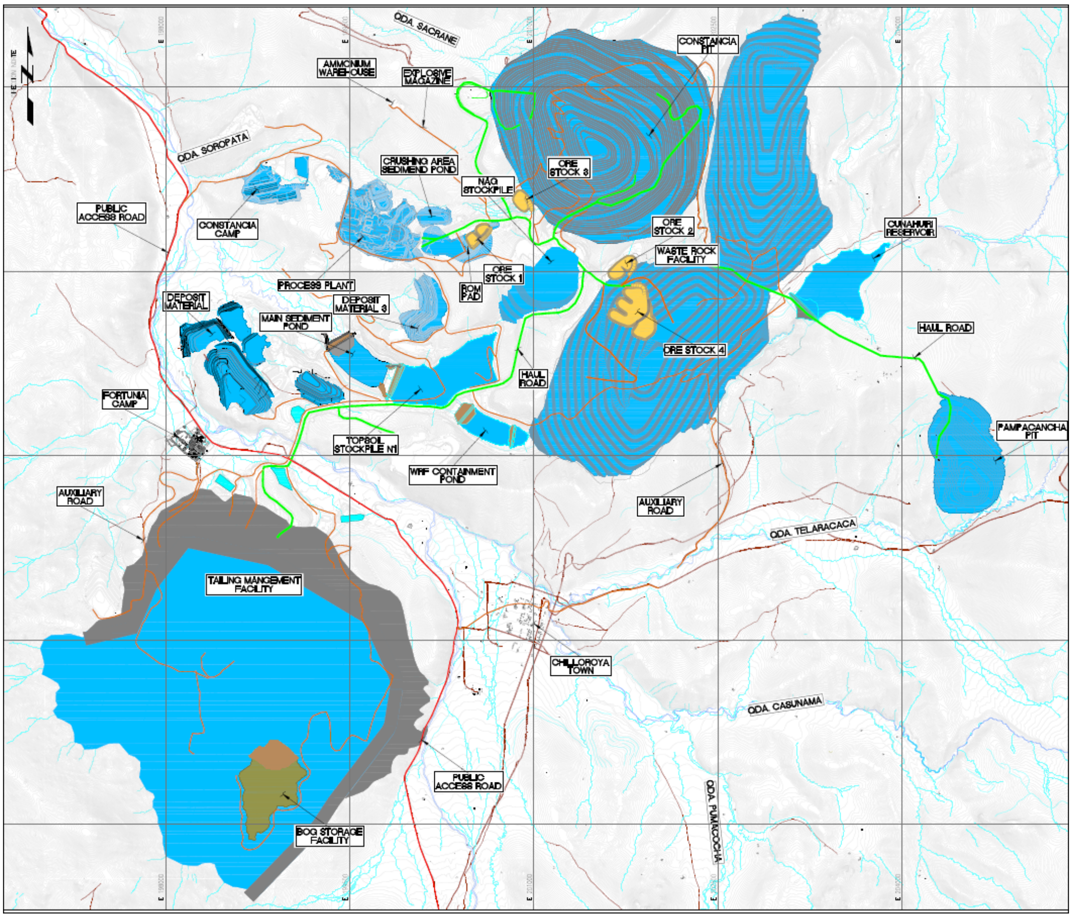
Power supply for the Constancia mine is brought from the new 220kV transmission line from Tintaya to Constancia that was built for and owned by Hudbay, but is operated and maintained by a third party. This transmission line is connected to the Constancia electric substation that transforms 220kV to 23 kV. The electric control room manages the 23kV system that provides energy to the entire mine. In case of emergency, there are three generators that provide energy in a semi-automated way.

The primary road to site, consists of a 65 km compacted dirt road (National Route PE-3SG) from Coporaque to the Puente Bailey/Constancia. These roads (and bridges) have been upgraded as necessary, to meet the needs of construction and life of mine use. In the meantime, in order to ensure good road conditions, Hudbay performs maintenance as necessary due to heavy truck traffic and the transportation of concentrate and consumables.

Copper concentrate is transported from the Constancia mine to the Matarani port by trucks (485km). These trucks are equipped with a hydraulically operated covered-box hinged at the rear, the front of which can be lifted to allow the concentrate to be deposited in the concentrate shed assigned to Hudbay by TISUR, the port operator. Pier C has been assigned to Hudbay and has a 75 Kt capacity. A chute from the shed feeds a tubular conveyor into a ship loader system with a nominal capacity of 1,500 tons per hour. The same conveyor and ship loading equipment is shared with other copper concentrate exporters. Hudbay is the primary customer for Pier C that was designed to take concentrates by truck and railroad.

Figure 1-3 provides an overall site layout showing the location of the various storage facilities relative to the Constancia and Pampacancha pits and the mill.

Figure 1-3 Overall Site Layout



1.12 MARKET STUDIES AND CONTRACTS

Constancia copper concentrate is a clean, medium grade concentrate containing small gold and silver by-product credits. It is a highly desirable feedstock for copper smelters in China which is the most geographically appropriate freight destination but is also suitable for processing by smelters in Europe, India and South America. Table 1-15 outlines an approximate analysis of the concentrate produced at the Constancia mine.

Constancia copper concentrate is sold directly to a variety of copper smelters in Asia, Europe and India as well as internationally recognized trading companies. Between 85 and 90% of sales are made pursuant to longer-term frame contracts which typically reference annual benchmark agreements between major concentrate producers and smelters for the purposes of fixing key terms such as treatment and refining charges. The balance of projected annual concentrate production has not been committed for sale in order to provide flexibility in the event of potential fluctuations in annual production and will be sold into the spot market each year at then-current market terms.

Table 1-15 Copper Concentrate Composition

Metal	Unit	Average	Range
Cu	%	25.0	22.0 – 28.0
Ag	g/t	120	90 – 200
Au	g/t	1.5	0.8 – 3.0
Zn	%	3.0	1.5 – 4.5
Pb	%	0.7	0.5 to 2.0
As	%	0.1	0.05 - 0.15

Production at Constancia is subject to a precious metals streaming agreement with Wheaton Precious Metals (Previously “Silver Wheaton (Caymans) Ltd.”) consisting of 50% of payable gold and 100% of payable silver. Hudbay will receive cash payments equal to the lessor of the market price and US$400 per ounce for gold and $5.90 per ounce of silver, subject to a 1% annual escalation starting three years after the completion date in 2016. Under the terms of the stream, gold recovery for the purposes of calculating payable gold will be fixed at 55% for gold mined from Constancia and 70% for gold mined from Pampacancha.

1.13 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Constancia Mine Environmental and Social Impact Assessment (ESIA) was approved by the Ministry of Energy and Mines (MINEM) in November 2010 and the first amendment to the ESIA (MOD I) was approved in August 2013 to increase the processing capacity and to match the Detailed Design Feasibility Study.

On April 2015, the second amendment to the ESIA (MOD II) was approved to increase reserves through the expansion of the Constancia Pit, inclusion of the Pampacancha deposit, and the waste rock facility (WRF) and the tailings management facility (TMF) expansion, among others.

Hudbay secured all necessary permits and authorizations on time to start construction activities and operation of the mine. A summary of these permits is detailed in Section 20.

Constancia’s closure plan was updated to incorporate recent studies and technological changes that will reduce costs and financial guarantees that Hudbay must provide annually to the Peruvian government. The updated closure plan was submitted in 2015 and approved by the MINEM in

June of 2016. This document provides updated measures of closure and post-closure of the mine, according to MOD I and MOD II.

The environmental and social impacts have been assessed and appropriate mitigation measures have been implemented. The ESIA and amendments comply with national regulation and have adopted the Equator Principles and International Finance Corporation's (IFC) Performance Standards. The Towards Sustainable Mining (TSM) standards of the Mining Association of Canada (MAC) are under implementation.

Hudbay has engaged an Independent Project Review Board (IPRB) to conduct periodic reviews of the major earth structures on the mine with regular visits to site, as well as for monitoring the progress of construction of the TMF. This activity was initiated during the construction stage and now continues during operation. Design reviews and site visits have been carried out at critical junctures throughout the course of construction and mine operations to observe the performance of the structure and development of subsequent raises to the dam and the impoundment.

The hydrogeological report (Golder 2013), which led to a more refined hydrogeological model and plant water balance, provided clarity to three principal areas: (1) unlikelyhood of hydrological connectivity between the open pit dewatering influence and neighboring distal reservoirs, (2) determined the impact of slope stability on the pit wall design; and (3) confirmed the availability of groundwater supply through the life of the mine.

The Hudbay water management strategy includes the following aspects:

- Zero water discharge to the environment from the plant, and PAG waste rock facility.
- Discharge from the TMF supernatant from 2020 only during the rainy season.
- Prioritization of water sources for the plant.
- Water available for compensation purpose for base flow of Chilloroya users and downstream community users.

The Constancia environmental monitoring program is summarized in the March 2016 Environmental Monitoring Plan (PLA-AMB-04: Plan de Monitoreo Ambiental). This plan covers monitoring of air, noise and vibration, surface water quality and flow, surface lake/reservoir levels, groundwater levels and quality, domestic water consumption and quality, effluent monitoring, topsoil, soil and sediment, biological (flora and fauna) monitoring and hydrobiological and meteorological data collection. During the operation phase, Hudbay is maintaining a joint environmental monitoring committee with the communities that are directly and indirectly influenced by the mine.

From a biodiversity point of view, Hudbay is working on specific management plans for wetlands, flora and fauna. Hudbay has developed a Biodiversity Action Plan (BAP), as generally anticipated and described in the ESIA and as required in the MAC TSM protocol and IFC performance standard N° 6.

Regarding social aspects, between February and April 2012 Hudbay reached agreements with the neighbouring communities of Uchucarco and Chilloroya for land required for the Constancia mine. These agreements were validated with support from two thirds of the community members. Both of these agreements have been recorded in the public land registry. A compensation plan was also prepared for the 36 landholding families from the lchuni area of Chílloroya. As a result of the Resettlement Action Plan (RAP) implementation, prepared in compliance with national laws and international provisions and standards in resettlement matters and in particular the IFC Performance Standards, the resettlement process was successfully completed in 2016.

Specific social programs were designed for the mitigation and prevention of the identified impacts of the resetelement program, considering infrastructure, social, economic and strategic development and claims and dispute resolution.

1.14 CAPITAL AND OPERATING COST

The LOM sustaining capex is estimated to be 748M (excluding capitalized stripping) at Constancia while the capex for the Pampacancha project is estimated to be 19M. All capex items are reported in real 2018 $USD.

The total includes capital required for major mining equipment acquisition, rebuilds, and major repair. The cost also includes site infrastructure expansion (tailings management facility, waste rock facility and others) and process plant infrastructure and the expenditures associated to the copper recovery improvement project, i.e. Eriez cells (see 2018-2019 plant other in Table 1-16). However they exclude all the cost related to mine closure.

Project capex associated with Pampacancha does not include acquiring surface rights but includes all other items. The capital costs for Constancia are developed and revised on an annual basis as part of the budget cycle. The 2018 LOM capital plan is shown in the Table 21-1.

Table 1-16 Sustaining CAPEX

Sustaining Capex		2018	2019	2020	2021	2022	2023-36	Total
Constancia								
Equipment - Purchase	US$'000s	2,800	-	3,250	-	-	16,250	**22,300**
Equipment - Major Repair	US$'000s	10,342	13,020	16,967	7,460	17,878	221,634	**287,301**
HCW - Tailings Dam	US$'000s	3,709	16,550	43,604	1,204	510	220,860	**286,437**
HCW - Waste rock facility	US$'000s	-	-	5,000	-	-	5,000	**10,000**
Mining - Other	US$'000s	8,138	6,611	2,500	3,000	3,000	26,600	**49,849**
Plant - Equipment & spares	US$'000s	1,120	1,000	1,600	1,600	1,600	13,000	**19,920**
Plant - Tailings pipeline	US$'000s	2,340	8,000	600	600	600	7,800	**19,940**
Plant - Other	US$'000s	945	34,900	1,000	1,000	1,000	13,000	**51,845**
Total (Before Capitalized Stripping)	**US$'000s**	**29,393**	**80,081**	**74,521**	**14,864**	**24,588**	**524,144**	**747,592**
Total (After Capitalized Stripping)	**US$'000s**	**54,849**	**88,385**	**89,594**	**35,708**	**34,961**	**737,878**	**1,041,375**
Pampacancha Project Capex		**2018**	**2019**	**2020**	**2021**	**2022**	**2023-36**	**Total**
Equipment - Purchase	US$'000s	-	3,000	-	-	-	-	**3,000**
HCW - General & other	US$'000s	-	7,738	1,000	1,000	-	-	**9,738**
Heavy CW - Pit dewatering	US$'000s	2,998	2,858	-	-	-	-	**5,856**
Total Pampacancha	**US$'000s**	**2,998**	**13,596**	**1,000**	**1,000**	**-**	**-**	**18,595**
Total Sustaining & Project Capex (Before Capitalized Stripping)	**US$'000s**	**32,392**	**93,677**	**75,521**	**15,864**	**24,588**	**524,144**	**766,186**
Total Sustaining & Project Capex (After Capitalized Stripping)	**US$'000s**	**57,847**	**101,981**	**90,594**	**36,708**	**34,961**	**737,878**	**1,059,969**

The operating costs at Constancia are developed annually as part of the site budget process. All operating costs are reported in real 2018 $USD.

The operating costs are divided in three centers of importance: mining, milling and G&A. The LOM operating costs are shown in Table 1-17.

Table 1-17 Operating Costs - OPEX

Operating Costs		2018	2019	2020	2021	2022	2023-36	LOM
Unit Costs								
Mining	(US$/tonne Milled)	3.04	2.80	2.93	2.89	2.83	2.78	2.81
Milling	(US$/tonne Milled)	4.11	4.21	4.32	4.36	4.32	4.25	4.25
G&A	(US$/tonne Milled)	1.68	1.66	1.57	1.53	1.53	1.35	1.41
Total Operating Costs (Before Capitalized Stripping)	(US$/tonne Milled)	**8.82**	**8.67**	**8.82**	**8.78**	**8.68**	**8.38**	**8.48**
Total Operating Costs (After Capitalized Stripping)	(US$/tonne Milled)	**8.01**	**8.41**	**8.34**	**8.11**	**8.34**	**7.86**	**7.96**

1.15 ECONOMIC ANALYSIS

Pursuant to NI 43-101, producing issuers may exclude the information required for Section 22 Economic Analysis on properties in production, unless the technical report includes a material expansion of current production. As Hudbay is a producing issuer, it has excluded information required by Item 22 of Form 43-101F1 as the Pampacancha expansion does not represent a material expansion of the current production facilities at Constancia.

1.16 CONCLUSIONS AND RECOMMENDATIONS

The Constancia mine has been in continuous operation since declaring commercial production on April 30, 2015. The mine has continually improved its performance indicators to a level supporting the projected throughput and forecasted results as outlined in this report with a processing plant capacity of 90,000 tpd producing both copper and molybdenum saleable concentrates. The resource and reserves stated for the Constancia and Pampacancha deposits are compliant to industry best practices as outlined in the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

The Pampacancha satellite deposit provides higher copper equivalent mill feed where mining is planned to start in 2019 Q1, provided an agreement with the local community can be finalized in time. Additional exploration concessions recently acquired by Hudbay within trucking distance of the processing plant offer additional opportunities to mine high grade skarn deposits in addition to their potential for finding a new large porphyry system.

The updated reserve estimates to mill reconciliation results indicate that the positive bias experienced with the old reserve model over the past 2 years has been explained. Thorough reconciliations from mineral reserve estimates to mill credited production will continue to be closely monitored in order to continue to validate the performance of the new reserve model. The updated mineral resource estimates show a decrease in total tonnage but an improvement in grade in all categories. This result is due to a correction of a sampling bias in the supergene domain and to a revision of the resource modeling method. The increase in the mineral resource estimates grade increases the potential for economic viability when the mineral reserve estimates will be depleted.

Other than the risks described in this report (including the need to acquire surface rights to mine the Pampacancha deposit), the general political and social risks associated with operating in Peru and the other risk factors described in Hudbay's most recent annual information form, there are no known significant risks and uncertainties that could reasonably be expected to materially affect the potential development of the mineral reserve and resource estimates in this report.Previous metallurgical testing has been updated by a recent study conducted in 2017 which has helped refine the forecast model for mill throughput and metals recovery. This work has also helped to identify opportunities for blending and plant /reagent modifications to further improve the plant performance going forward. Optimization studies will continue with a focus on improving recovery on end members of the grain size distibution, i.e. ultra fine and coarser fractions of the mineralization.

Further review is required regarding the operating practices and the performance of the molybdenum circuit to best understand what parametres and production can be improved.

2 INTRODUCTION

This Technical Report has been prepared for Hudbay to support the public disclosure of mineral resources and mineral reserves at Constancia mine at December 31, 2017.

Hudbay is an integrated Canadian 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's operations at Constancia include the Constancia and Pampacancha pits, an ore processing plant, a Waste Rock facility, a Tailings Management facility and other ancillary facilities that support the operations.

At the date of this report, the Constancia mine is in steady production. The pre-stripping started in March 2014 and the concentrator ramp-up started in December 2014. Commercial production was achieved as of April, 2015 and the operations continue with management's expectations in terms of ore, grade, recovery and cost.

2.1 QUALIFIED PERSON ("QP") AND SITE VISIT

This Technical Report has been prepared in accordance with NI Form 43-101 F1. The QP who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo., Senior Vice President and Chief Operating Officer of Hudbay.

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

Mr. Meagher has direct authority over the Constancia mine and was involved in early assessment of the project for Hudbay prior to its acquisition, during the construction and during the current operation. In his past role as Vice President South America Business Unit and in his current role, Mr. Meagher has made many visits and inspections to the site and has overseen the resource and reserve estimation process for the deposits. Mr. Meagher has acted as the Qualified Person for the overall report. Prior to publication Mr. Meagher's last site visit was in December, 2017.

2.2 SOURCE OF INFORMATION

Geology and Mineral Resources sources of information are: drilling log and sample data, blast hole sample data, in-pit geology mapping, as well as the relevant information from the previous Technical Report on the property prepared for Hudbay, entitled "National Instrument 43-101

Technical Report, Constancia Project, Province of Chumbivilcas, Department of Cusco, Peru", with an effective date of November 2016, which is publicly available on the system for Electronic Disclosure and Retrieval (SEDAR) at www.sedar.com, maintained by the Canadian Securities Administrators.

Mineral Reserve sources of information are the Mineral Resources, actual production and monitoring data (since 2014), budget projections, as well as the relevant (current) information from the previous Technical Report on the property.

Metallurgy, processing, and economic sources of information are the actual operating data acquired since copper production commenced in 2014, operating budget estimates, as well as the relevant (current) information from the previous Technical Reports on the property.

All other relevant information has been gathered from the previous Technical Report on the property and where necessary, updated with information or reports provided and translated by senior site personnel.

Multiple participants have worked on this report. Discussions were held with personnel from Hudbay Minerals Inc and Hudbay Peru:

- Javier Del Rio, Vice- President South America Business Unit
- Javier Toro, Director – Technical Services
- Marc-Andre Brulotte, Manager – Resource Evaluation
- Olivier Tavchandjian, Vice-President – Exploration and Geology
- Carlos Castro, Director – Corporate Affairs and Social Responsibility
- Hugo Granados, Manager – Process Plant
- Carlos Ojeda, Manager – Technical Services
- Julio Roncal, Manager – Environmental
- John Cederberg, Manager – Administration and Logistics
- Carlos Salazar, Manager – Exploration
- Richard Rodrigues, Superintendent – Mine Geology

Table 2-1 lists the participants to this Technical Report under Cashel Meagher as QP:

Table 2-1 Technical Report Participants

Section	DESCRIPTION	Participants	Responsible
1	Summary	Javier Del Rio - Javier Toro	Cashel Meagher
2	Introduction	Javier Del Rio - Javier Toro	Cashel Meagher
3	Reliance on Other Experts	Javier Del Rio - Javier Toro	Cashel Meagher
4	Property Description and Location	Carlos Castro	Cashel Meagher
5	Accessibility, Climate, Local Resources Infrastructure and Physiography	Julio Roncal	Cashel Meagher
6	History	Carlos Ojeda	Cashel Meagher
7	Geological Setting and Mineralization	Richard Rodrigues – Carlos Salazar	Cashel Meagher
8	Deposit Types	Richard Rodrigues – Carlos Salazar	Cashel Meagher
9	Exploration	Richard Rodrigues – Carlos Salazar	Cashel Meagher
10	Drilling	Richard Rodrigues – Carlos Salazar	Cashel Meagher
11	Sample Preparation Analyses and Security	Richard Rodrigues – Marc-Andre Brulotte	Cashel Meagher
12	Data Verification	Richard Rodrigues – Marc-Andre Brulotte	Cashel Meagher
13	Mineral Processing and Metallurgical Testing	Hugo Granados – Javier Toro	Cashel Meagher
14	Mineral Resource Estimates	Richard Rodrigues – Marc-Andre Brulotte – Olivier Tavchandjian	Cashel Meagher
15	Mineral Reserve Estimates	Javier Toro – Carlos Ojeda	Cashel Meagher
16	Mining Methods	Javier Toro – Carlos Ojeda	Cashel Meagher
17	Recovery Methods	Javier Del Rio - Hugo Granados	Cashel Meagher
18	Project Infrastructure	Javier Toro – Carlos Ojeda	Cashel Meagher
19	Market Studies and Contracts	Javier Del Rio - John Douglas	Cashel Meagher
20	Environmental Studies, Permitting, and Social or Community Impact	Carlos Castro - Julio Roncal	Cashel Meagher
21	Capital and Operating Costs	Javier Toro – Carlos Ojeda	Cashel Meagher
22	Economic Analysis	Javier Del Rio - Javier Toro	Cashel Meagher
23	Adjacent Properties	Olivier Tavchandjian – Carlos Salazar	Cashel Meagher
24	Other Relevant Data and Information	Carlos Ojeda	Cashel Meagher
25	Interpretation and Conclusions	Olivier Tavchandjian	Cashel Meagher
26	Recommendations	Olivier Tavchandjian	Cashel Meagher

2.3 UNIT OF MEASURE ABBREVIATIONS USED IN REPORT

- (%) - Percentage
- (g/t) - Grams per tonne
- (Mt) - Million tonnes
- µg/m^3 - Micrograms per cubic meter
- Ha - Hectare
- Km - Kilometres
- kV - Kilovolts
- kWh/m3 - Kilowatt – hours per cubic meter
- kWh/t - Kilowatt per tonne
- m - Meter
- m^3 - Cubic meter
- m^3/day - Cubic meter per day
- m^3/h - Cubic meter per hour
- m^3/year - Cubic meter per year
- Mt/yr ore - Million tonnes per year of ore
- Mtpa - Million tonnes per annum
- MW - Megawatt
- MW-h/y – Mega watt hours per year
- MWhr - Mega watt hour
- ppb - Parts per billion
- Ppm - Parts per million
- sq-km - Square kilometres
- t - Tonne
- t / m^3 - Tonne per cubic meter
- t/a - Tonnes per annum
- t/BCM - Tonnes per billion cubic metres
- t/Hole - Tonnes per hole
- t/m3. - Tonnes per cubic meter
- Tonne/Hr - Tonnes per hour
- Tpd - Tonnes per day
- US$/dmt - US dollars per dry metric tonne
- US$/oz - US dollars per ounce
- US$/wmt - US dollars per wet metric tonne
- (AAN) Andesite
- (AAS) Atomic Absorption Spectroscopy
- (AVRD) Absolute Value of the Relative Difference
- (BAP) Biodiversity Action Plan
- (CC-CR) Constancia Condemnation Drilling
- (CG) Constancia Geotechnical Drilling
- (CH) Constancia Hydrogeological Drilling
- (CM) Constancia Metallurgical Drilling
- (CO) Constancia Infill Drilling
- (CR) Reverse Circulation
- (CR) Reverse Circulation

2.4 ACRONYMS AND ABBREVIATIONS USED IN REPORT

- (CRMs) Certified Reference Materials
- (EIA) Environmental Impact Assessment
- (FEED) Front End Engineering and Design
- (GWI) Ground Water International Consulting Hydrogeologists
- (HA) Hectar
- (HG) High Grade Material
- (ICP-AES) Inductively Coupled Plasma – Atomic Emission Spectrometry
- (IP) Induced Polarisation
- (IRA) Inter-Ramp Angle
- (JV) Joint Venture
- (LG) Low Grade Material
- (LTEP) Long Term Equilibrium Prices
- (LTEP) Long Term Equilibrium Prices
- (MASL) Metres Above Sea Level
- (MCCs) Motor Control Center
- (MG) Medium Grade Material
- (MINEM) Peruvian Ministry of Energy and Mines
- (MMP)Micro Monzonite Porphyry
- (MP1) Monzonite Porphyry 1
- (MP2) Monzonite Porphyry 2
- (MR) Mining Royalty
- (Mt) Million tonnes
- (NAG) Non-Acid Generating Material
- (NI) National Instrument
- (PAG WRF) Potentially Acid Generating Waste Rock Facility
- (PAG) Potentially Acid Generating Material
- (PFS) Pre-Feasibility Study
- (PG) Pampacancha Geotechnical Drilling
- (PO, PR) Pampacancha Exploration-Infill Drilling
- (QC) The Quality Control
- (QMP) Quartz Monzonite Porphyry
- (RE) Relative Error
- (RMR) Rock Mass Rating
- (RQD) Rock Quality Designation
- (SD-EIA) Semi-Detailed Environmental Impact Assessment
- (SEIN) Peruvian National Interconnected Electric System
- (SMT) Special Mining Tax
- (SMU) Selective Mining Unit
- (SO, SR) Chilloroya South Exploration
- (SO, SR) Chilloroya South Exploration
- (TMF) Tailings Management Facility
- (UCS) Unconfined Compressive Strength
- (UEA) Unidad Economica Administrativa
- (UO) Uchucarco Exploration
- (WRF) Waste Rock Facility

3 RELIANCE ON OTHER EXPERTS

Hudbay has followed standard professional procedures in preparing the contents of this Technical Report. Data used in this report has been verified and the author has no reason to believe information has been withheld that would affect the conclusions made herein.

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

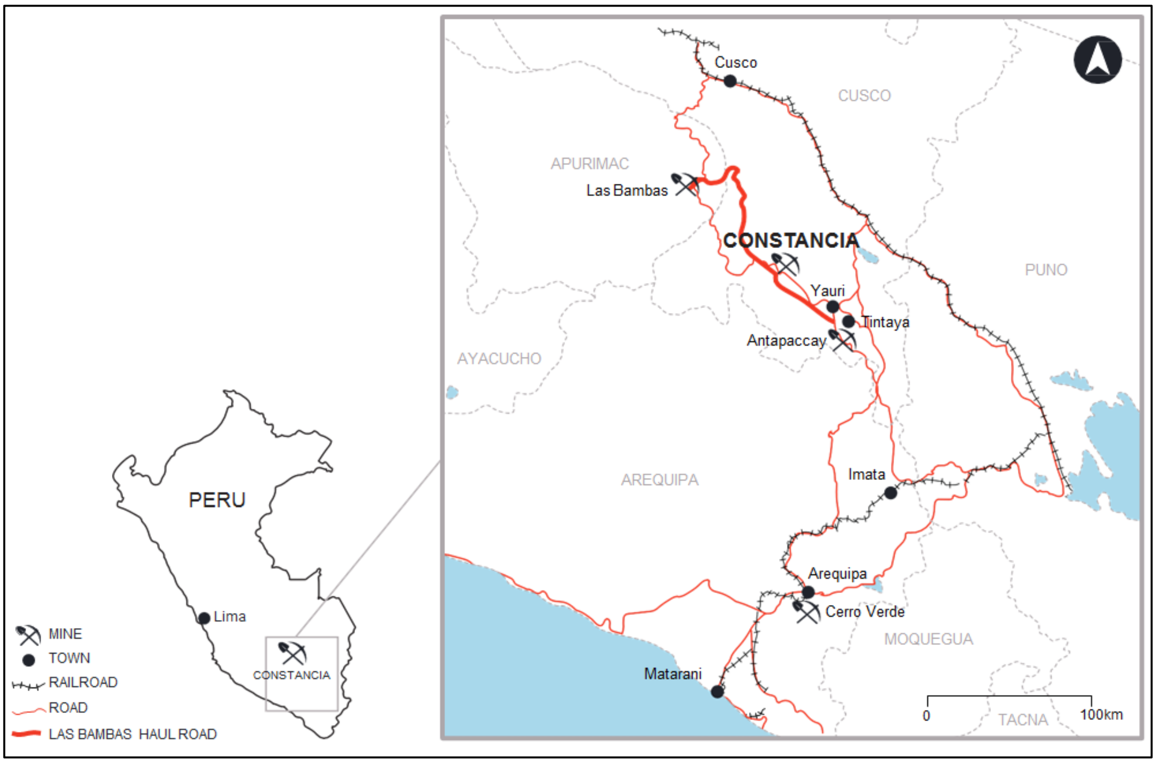
- Information available to Hudbay and the author at the time of preparation of this report, and
- Assumptions, conditions, and qualifications as set forth in this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 THE PROPERTY LOCATION

The Constancia mine is located in the south-eastern Andes of Peru, in the Chamaca, Livitaca and Velille districts, province of Chumbivilcas, department of Cusco. The property is approximately 600 km southeast of Lima at elevations of 4,000 to 4,500 masl. Figure 4-1 shows the general Constancia mine location. Road access to the property is from Arequipa (7 hours by road) or Cusco (6 hours by road). Geographic coordinates at the centre of the property are longitude 71°47' west and latitude 14°27' south.

Figure 4-1 Constancia Mine Location



4.2 MINERAL RIGHTS

4.2.1 PERUVIAN MINING LAW

The General Mining Law of Peru defines and regulates different categories of mining activities, from sampling and prospecting to commercialization, exploration, exploitation, general labour, processing and mining transportation (concessions granted by the Peruvian state are only required to conduct the latter five activities). Mining concessions (which grant the rights to explore and exploit mineral deposits) are granted using UTM[1] coordinates to define areas generally ranging from 100 ha to 1,000 ha in size.

Mining concessions are irrevocable and perpetual, as long as the titleholder complies with two main mining obligations: (i) pay an annual validity fee (“Derecho de Vigencia”); and, (ii) achieve an annual minimum production level on the concessions.

The “Derecho Vigencia” or validity fee is an annual maintenance fee of $3/ha (for metallic mineral concessions at general regimen) for each concession actually acquired, or for a pending application (petitorio), payable by June 30th of each year. Non-compliance with this obligation for two (2) consecutive years results in the cancellation of the respective mining concession.

The General Mining Law of Peru sets a time limit within which the holder of the mining concession must comply with obtaining the minimum production level. If the established production is not met by the deadline, then the holder of the mining concession must pay a penalty.

There are currently two systems for calculating the amount to be paid as a penalty. Each of these regimes will be applicable depending on whether the mining concession analyzed was granted until 2008 or later. Each of these regimes is explained below:

(i) Old Regime - Concessions granted until 2008: The minimum production that the holder of the mining concession must reach in these concessions is US$ 100 per hectare per year, for a term of six years from the date of granting the title. If the holder of the mining concession does not comply with said production, he or she must pay a penalty equivalent to US$ 6.00 per year per hectare. This penalty will be increased to US$ 20.00 per year per hectare if the non-compliance persists after year twelve.

    Under this regime, if the holder of the mining concession fails to pay the penalty for two consecutive years, this will result in the cancellation of the respective concession. It should also be noted that the holder of the mining concession may be released from payment of the aforementioned penalty if he or she provides proof to the mining authority that he or she has made investments in the concession in question for an amount equivalent to ten times the amount of the applicable penalty.

(ii) New Regime - Concessions granted from 2009: The minimum production to be reached by the holder of the mining concession in these concessions is one Tax Unit (UIT)[2] per hectare

[1] Universal Transverse Mercator Coordinate System.

[2] One current UIT is equivalent to US$ 1,280.00 approximately as of 2018.

per year for metal concessions and 10% of the Tax Unit (UIT) per hectare per year for non-metallic concessions, for which it will have a term of 10 years from the date of granting the title. On January 5, 2017, Legislative Decree No. 1320 that modified articles 40 and 41 of the General Mining Law (modified previously by Legislative Decrees No. 1010 and No. 1054), established new terms and amounts to be paid as penalties for failure to reach the minimum production of the New Regime, applicable from January 1, 2019, as detailed below:

(a) If the minimum production is not met by the tenth year after the mining concession was granted, a penalty of 2% of the value of the UIT per year and per hectare must be paid from the following year until the minimum production or minimum annual investment is reached.

(b) If the minimum production is not met by the fifteenth year after the mining concession was granted, a penalty of 5% of the UIT value per year and per hectare must be paid until the minimum production or minimum annual investment is reached from the following year onwards.

(c) If the minimum production is not met by the twentieth year after the mining concession was granted, a penalty of 10% of the UIT value per year and per hectare actually granted until the minimum production or minimum annual investment is reached from the following year onwards.

(d) If the minimum production is not met within thirty years of the mining concession being granted, the concession expires, with no exceptions.

(e) No penalty is required to be paid in each year if an investment of not less than ten times the applicable penalty amount is made.

As of January 1, 2019, the concessions of the new and old regimes that do not demonstrate compliance with the minimum production of the previous year, during the first semester of 2019 must pay the penalty of 2% of UIT per year and per hectare granted for each concession, continuing in case of non-compliance in subsequent years, with the terms, penalty amounts, and conditions detailed in the previous paragraph.

The holder of a mining concession is entitled to all the protection available to all holders of private property rights under the Peruvian Constitution, the Civil Code, and other applicable laws. A Peruvian mining concession is a property-related right; distinct and independent from the ownership of land on which it is located, even when both belong to the same person. The rights granted by a mining concession are defensible against third parties, are transferable and chargeable, and, in general, may be the subject of any transaction or contract.

To be enforceable, any and all transactions and contracts pertaining to a mining concession must be entered into a public deed and registered with the Public Mining Registry. The holder of a mining concession must develop and operate his/her concession in a progressive manner, in compliance with applicable safety and environmental regulations and with all necessary steps to avoid third-party damages. The concession holder must permit access to those mining authorities responsible for assessing that the concession holder is meeting all obligations established in the applicable regulation.

4.2.2 MINING CONCESSIONS

Excluding the concessions included in the Transfer Agreement and the Transfer Option and Mining Assignment Agreements, subscribed by Hudbay Peru S.A.C. with Panoro Apurimac S.A. and a private Peruvian company respectively, in December 2017, Hudbay has a total of sixty-six concessions in the province of Chumbivilcas, in the department of Cusco which belong to the Constancia Mine. These sixty-six concessions occupy an area of 43,536.7769 Ha.

Most of the known mineralization of Constancia mine is located in the following concessions: Katanga J, Katanga Q, Katanga K, Peta 7, Peta 6, Constancias 13 and Constancias 8.

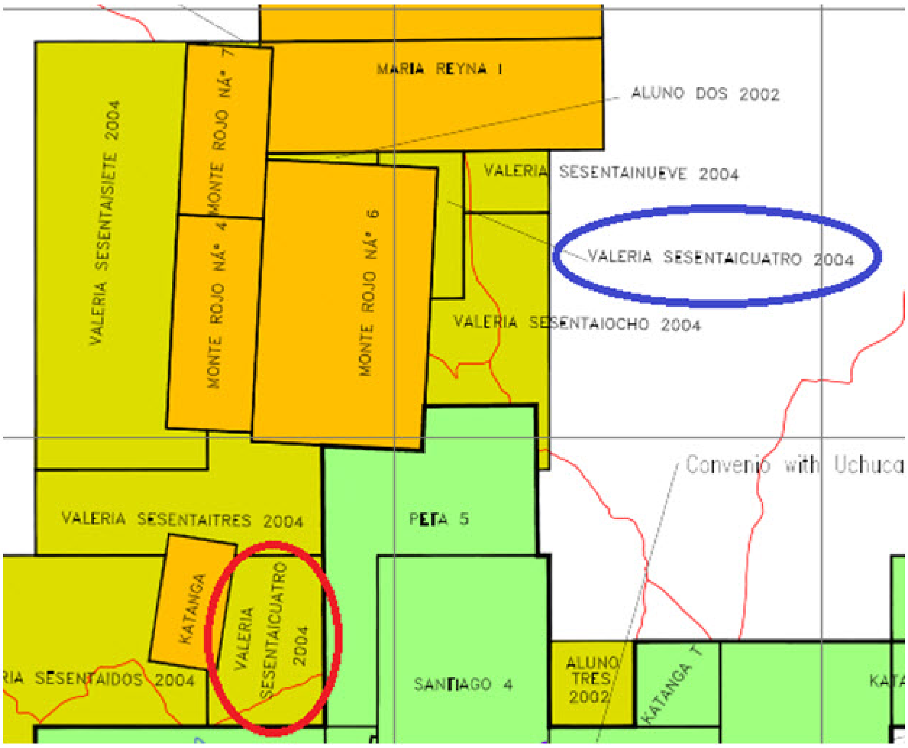
Figure 4-2 shows the concessions corresponding to the UEAs Constancia and Pampacancha and the concessions included in the Transfer Agreement and the Transfer Option and Mining Assignment Agreements.

<u>Administrative Economic Units (“Unidad Económica Administrativa” – “UEA”)</u>

Fourteen mining concessions out of the sixty-six concessions comprising the Constancia mine have been grouped together in an UEA called “Constancia”, while ten concessions have been grouped together in an UEA called Pampacancha.

UEAs are groups of two or more mining concessions of the same class and nature which allow the respective holder to work only in one or some of the concessions, instead of working simultaneously in all of those encompassing the UEA. However, the holder is obliged to obtain the minimum production levels corresponding to all the mining concessions comprising the UEA in those mining concessions which are exploited.

Figure 4-2 Constancia Mine Concession Boundaries



The UEA Constancia (code N° 01-00029-83-U) – previously called Katanga Este - was originally approved by Resolución Directoral No. 148/86-EM-DCM dated 1 September 1986 comprising only the "Katanga J" and "Katanga K" mining concessions. Through Resolución Jefatural N° 5404-2006-INACC/J issued on the 14th of December 2006, eleven mining concessions were added, bringing a total of 13 concessions to the UEA. On December 14, 2015, the name was changed to UEA Constancia. In December 2016, by Resolución de Presidencia No. 2207-2016-INGEMMET-PCD-PM one more concession was added to this UEA. The total of the area of this UEA is 6,146.6569 Ha.

Table 4-1 shows the names, codes, dates and areas of the concessions comprising the UEA Constancia. All the properties are located on the Livitaca (29-S) map sheet. The UEA Pampacancha (code 010002216U) was approved by Resolución de Presidencia No. 1899-2016-INGEMMET-PCD-PM on December 19, 2016. The total area of this UEA is 8,085.9011 Ha.

Table 4-2 shows the names, codes, dates and areas of the concessions comprising the UEA Pampacancha.

Table 4-1 Hudbay Concessions

Concession Name	Code	Concession Granted	Hectares
Peta 5	05006089X01	28-11-1989	**935**
Katanga J	05004406X01	29-03-1990	**400**
Katanga Q	05005529X01	9/5/1990	**150**
Katanga K	05004407X01	16-07-1990	**300**
Peta 6	05006090X01	29-10-1996	**1,000**
Santiago 4	10083495	23-12-1996	**34**
Santiago 3	10083695	25-03-1997	**701**
Santiago 5	10083295	30-04-1997	**602**
Katanga V	10248497	31-10-1997	**100**
Katanga T	10248397	15-11-1997	**100**
Santiago Apostol I	10229294	31-03-1998	**424**
Peta 17	0506198AX01	13-12-1999	**49**
Peta 7	05006198X01	13-12-1999	**352**
Constancias 7	10025507	19-11-2018	**1,000**

It is worth noting that all of the mining concessions involving the Constancia and Pampacancha UEAs have been titled and have also been recorded with the Public Registries in the name of Hudbay Peru S.A.C. Likewise, all of these concessions are currently in good standing and concession fees and applicable penalties have been paid for the calendar year 2017. All mining concessions are in good standing (i.e. the mining concessions are in full force and effect) as of 2017 and there are no foreseeable circumstances that could result in the mining concessions being cancelled by the Peruvian State.

Table 4-2 UEA Pampacancha

Concession Name	Code	Concession Granted	Hectares
Constancia 8	10025607	15-12-2008	**584**
Constancia 9	10025707	15-12-2008	**1,000**
Constancia 11	10025907	19-11-2008	**1,000**
Constancia 12	10026007	19-11-2008	**1,000**
Constancia 13	10026107	15-12-2008	**702**
Constancia 18	10614807	21-04-2008	**400**
Constancia 10	10025807	18-12-2008	**100**
Constancia 19	10614907	12-05-2008	**700**
Constancia 21	10615107	12-05-2008	**700**
Constancia 23	10615307	18-04-2008	**1,000**

Recently Acquired Mining Concessions

In December 2017, Hudbay Peru S.A.C. executed a Transfer Agreement with Panoro Apurimac S.A. and a Transfer Option and Mining Assignment Agreements with a private Peruvian company. Through these agreements, Hudbay Peru S.A.C. acquired title over 24 mining concessions.

The Transfer Agreement was subscribed by Hudbay Peru S.A.C. and Panoro S.A. in December 28, 2017. This contract was subsequently entered in the Public Registries. The total area of the concessions mentioned in this agreement is 3,961.7457 Ha. Table 4-3 shows the names, codes, dates and areas of the concessions transferred by the aforementioned Agreement:

It should be noted that the payment for the concepts of “Derecho de Vigencia” and the penalties corresponding to the years 2017 and 2018 of the mining concessions previously indicated in Table 4-3 are pending. Such payments should be made in a timely manner before June 30, 2018, as otherwise the concessions would expire.

The Transfer Option and Mining Assignment Agreement was subscribed by Hudbay Peru S.A.C. and a private Peruvian company on December 14, 2017. This contract was subsequently entered in the Public Registries. The total area of the concessions mentioned in this agreement is 5970.3258 Ha. Table 4-4 shows the names, codes, dates and areas of the concessions transferred by the aforementioned Agreement:

Table 4-3 Kusiorcco Concessions

Concession Name	Code	Hectares
Aluno Dos 2002	10170502	17
Aluno Tres 2002	10170802	100
Valeria Sesentaicinco 2004	10166804	561
Valeria Sesentaicuatro 2004	10166704	245
Valeria Sesentaidos 2004	10166504	725
Valeria Sesentainueve 2004	10167204	73
Valeria Sesentaiseis 2004	10166904	569
Valeria Sesetaisiete 2004	10167004	842
Valeria Sesentaitres 2004	10166604	370
Valeria Setentaicuatro 2004	10269204	71
Valeria Sesentaiocho 2004	10167104	288
PML1	10358414	100

Table 4-4 Caballito and Maria Reyna Concessions

Concession Name	Code	Hectares
Doris 7	05005966X01	840
Doris 8	05005967X01	840
Juana 4	0505971AX01	4
Katanga	05000317X01	120
Leviatan	05003143X01	2
Maria Reyna I	05006511X01	507
Maria Reyna II	05006512X01	1,000
Maria Reyna III	05006513X01	547
Monte Rojo N° 4	05002684X01	250
Monte Rojo N° 6	05003144X01	660
Monte Rojo N° 7	05003235X01	200
Peta	05005892X01	1,000

As of 2017, these concessions are up to date in the payment of the “Derecho de Vigencia” and penalties. The payment of the “Derecho de Vigencia” and penalties corresponding to 2018 is pending.

4.2.3 SURFACE LEGAL RIGHT ACCESS

The Constancia mine area is located within six lands owned by Hudbay. For rights purposes, Norsemont previously purchased the Fortunia property that covers most of the main resource area.

On March 19, 2012, Hudbay and the Uchucarco Community entered into an agreement granting Hudbay a right to use 256.50 hectares located in the so-called “Sayhualoma” sector through the life of the mine. Likewise, one significant plot of 750 ha in the so-called “Ichuni” sector has been purchased by Hudbay from the Chilloroya Community on April 12th, 2012. Hudbay has also purchased two pieces of land of 1361.87 ha and 7.73 ha, respectively called “Quinsachata Sub Lote 1” and “Quinsachata Sub Lote 2” for relocation purposes. Through these agreements, Hudbay has secured all surface land rights required for the construction and development of the Constancia Mine.

On May 9, 2012, Hudbay entered into an agreement with the Chilloroya Community in order to access and use for exploration purposes 6,148 hectares covering the Pampacancha deposit. This agreement was in force until 2015.

Hudbay is in the process of negotiating with the Chilloroya community to obtain surface rights over the Pampacancha area to allow exploitation activities therein.

Hudbay is also negotiating with the relevant communities’ surface rights over the recently acquired exploration properties. Below is an updated summary of the land rights held by Hudbay (Table 4-5).

Table 4-5 Surface Legal Access Rights Table

No	LOCATION	AREA (hectares)	RIGHT HELD BY HUDBAY	CONSTANCIA/ PAMPACANCHA
1	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Chilloroya Lote B, San Antonio, Area Afectable*"	499.64	Owner	Constancia
2	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Chilloroya Lote B, San Antonio, Area Inafectable*"	943	Owner	Constancia
3	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Fortunia, Area Afectable*".	974.91	Owner	Constancia
4	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Fortunia Area Inafectable*".	423.38	Owner	Constancia
5	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Arizona*".	885.35	Owner	Constancia
6	Rustic Property located in the District of Velille, Province of Chumbivilcas, Department of Cusco, named "*Arizona – Moroccota*".	340.71	Owner	Constancia
7	Rustic Property located in the District of Coporaque, Province of Espinar, Department of Cusco, named "*Quinsachata Sub Lote 1*".	1,361.8670	Owner	Constancia
8	Rustic Property located in the District of Coporaque, Province of Espinar, Department of Cusco, named "*Quinsachata Sub Lote 2*".	7,7319	Owner	Constancia
9	Rustic Property located in the District of Livitaca, Province of Chumbivilcas, Department of Cusco, named "*Ichuni*".	750	Owner	Constancia
10	Rustic Property located in the District of Chamaca, Province of Chumbivilcas, Department of Cusco, named "*Sayhualoma*".	256.5	Right of use to carry out all types of mining activities during the entire useful life of the Constancia Mine.	Constancia
11	Rustic Property located in the District of Livitaca, Province of Chumbivilcas, Department of Cusco, named "*Pampacancha*".	6148.70	Right of use to carry out all types of exploration activities during the 2012 – 2015	Pampacancha

4.3 THE TERMS OF ANY ROYALTY OR PAYMENTS

Hudbay's mining concessions are free and clear of recorded liens, encumbrances or agreements except for a 0.5% Net Smelter Return ("NSR Royalty") that has been established over the sale of the mineral products obtained from the following concessions: Katanga Q, Peta 5, Peta 6, Peta 7 and Peta 17. Such NSR Royalty has been granted in favor of Compañia Minera Katanga S.A. ("Katanga") and Minera Livitaca S.A. ("Livitaca") up to a maximum amount of US$ 10,000,000 (ten million and 00/100 Dollars of the United States of America). Once the aforesaid amount is reached, the NSR Royalty will automatically expire. To date, approximately US $3 million has been paid. As a result of a corporate reorganization, Katanga and Livitaca were merged with Docarb S.A., which took over all the assets of these companies. This NSR Royalty has been established as a consequence of the execution of the transfer of the 30% interest of the above-listed mining concessions granted by Katanga and Livitaca in favour of Rio Tinto Mining and Exploration S.A.C. (formerly Rio Tinto Mining and Exploration Limited, Sucursal del Peru, hereinafter "Rio Tinto"), formalized by public deed dated December 20, 2007 and duly recorded with the Public Registry.

By virtue of the Mining Concessions Transfer Agreement dated December 31, 2008, and its addendum, dated February 19, 2009, both executed by and between Norsemont and Rio Tinto, a legal mortgage up to the amount of US $500,000 has automatically been established (and is currently registered) over the following mining concessions in order to secure the timely payment of the additional or contingent payment.

The Transfer Agreement subscribed with Panoro S.A. in December 28, 2017 establishes a lien in the form of a 2% Net Smelter Return ("NSR Royalty") over the sale of the mineral products obtained from the following concessions: Aluno Dos 2002, Aluno Tres 2002, Valeria Sesentaicinco 2004, Valeria Sesentaicuatro 2004, Valeria Sesentaidos 2004, Valeria Sesentainueve 2004, Valeria Sesentaiseis 2004, Valeria Sesentaisiete 2004, Valeria Sesentaitres 2004, Valeria Sesentaiocho 2004, PML1.

The Transfer Option and Mining Assignment Agreement subscribed with a private Peruvian company in December 14, 2017 establishes that in case Hudbay Peru S.A.C. exercises the transfer option, it must sign a Royalty Agreement for the following concessions: Doris 7, Doris 8, Katanga, Juana 4, Leviatan, Maria Reyna I, Maria Reyna II, Maria Reyna III, Monte Rojo No4, Monte Rojo No6, Monte Rojo No7 and Peta.

Hudbay has LOM agreements with the two local communities, Chilloroya and Uchucarco. These agreements were part of the land access and purchase agreements and they outline a commitment to sustainable development in the respective communities.

4.4 ENVIRONMENTAL PROPERTY LIABILITIES

During the preparation of the environmental baseline for the ESIA, Knight Piesold Consultores S.A. determined the existence of 21 mining environmental liabilities within the Constancia mine in the areas called Sacsa Orcco, the San José pit, Chilloroya and the Yanaccocha Lake.

Since these environmental liabilities were generated by former operations and titleholders, Hudbay is not obligated to undertake any measure for its reclamation or closure.

Additionally, as a result of the preparation of the environmental baseline of the first amendment of the ESIA, an extra mining environmental liability was identified in the San José area. Hudbay Peru S.A.C. declared it before the competent authority. This area of the San José pit is part of the final closure plan of the Constancia mine.

4.5 RISK FACTORS PROPERTY

Hudbay has secured all legal rights on the mining concessions comprising the Constancia mine as well as on the lands required for the development of this mine. Considering that such rights have been registered before the Public Registries and are enforceable before the Peruvian state and third parties, Hudbay does not foresee any risk in losing its rights over the Constancia Mine, provided that the legal obligations applicable to keep such properties in full force and effect are duly complied with.

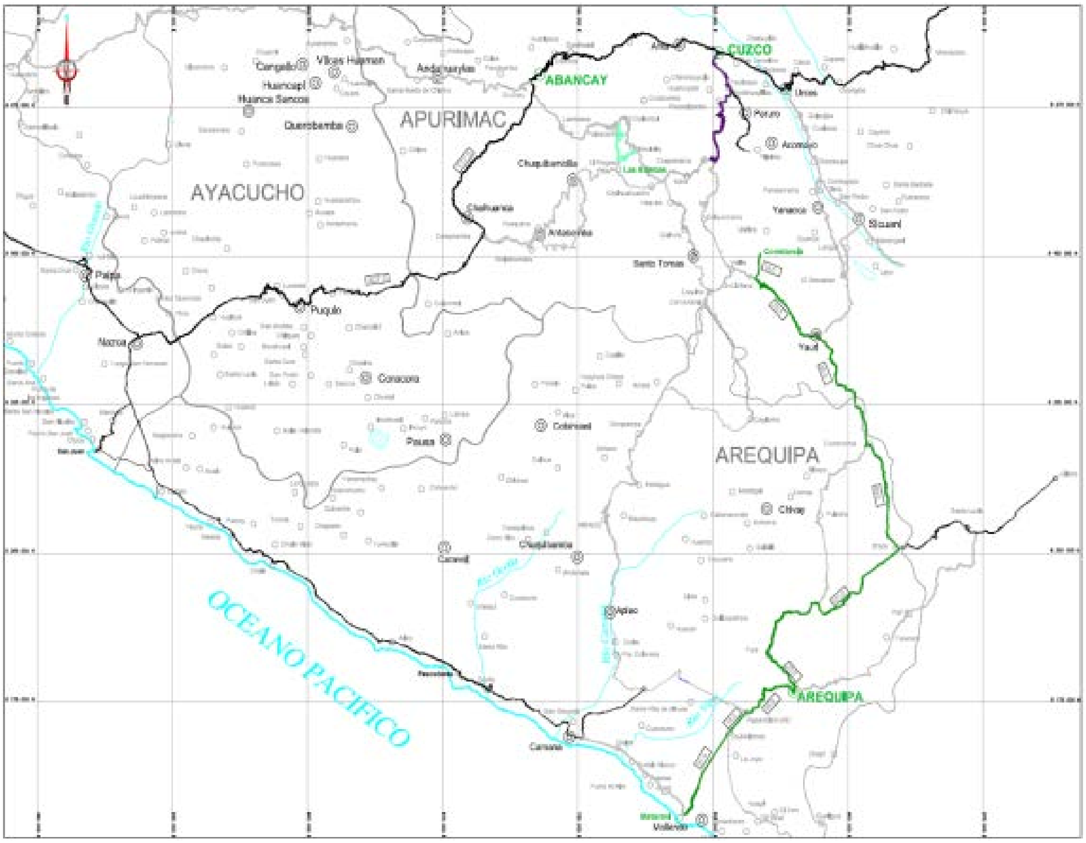
Regarding Pampacancha, although Hudbay has secured the necessary mining concessions to explore and exploit such deposit, Hudbay needs to enter into an agreement with the Chilloroya Community for use of the lands for construction and exploitation. Hudbay will need to secure the required surface rights before it can develop the Pampacancha deposit.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Constancia mine is located in Chamaca and Velille districts, Chumbivilcas province, Cusco region, an approximate distance of 634 km southeast from Lima, capital of Peru, with an altitude ranging between 4,000 and 4,500 metres. The mine is accessible from Lima by air, via either Arequipa or Cusco, and then by vehicle over paved and gravel roads. Figure 5-1 below shows the main route from Arequipa and Cusco to the mine.

Figure 5-1 Location of the Constancia Mine



The routes with approximate distances and driving times are shown in Table 5-1.

Table 5-1 Approximate Distances and Driving Times to Constancia

From	Leg	Distance (km)	Time (hours)
Arequipa	Canahuasi	80	1.25
	Imata	63	0.75
	Yauri/Espinar	103	2.50
	Uchucarcco turnoff	68	2.00
	Project Site	20	0.50
	Total	**334**	**7.00**
Cusco	Sicuani (paved road)	140	2.00
	El Descanso	40	1.00
	Yauri/Espinar	45	0.75
	Uchucarcco turnoff	68	2.00
	Project Site	20	0.50
	Total	**313**	**6.25**

5.2 CLIMATE

Climate at the Constancia site can be classified as humid and seasonably cool with well-defined, rainy, and drier seasons. The majority of the precipitation typically occurs over the period of October to April and the summer months. Elevation and physiograpy also influence the climate. Following is a list of important climatological data:

- Annual Precipitation:
  - Annual average precipitation: 1,000 mm
  - Annual maximum precipitation: 1,353 mm
  - Annual minimum precipitation: 590 mm
  - Average wet season (October to April) precipitation: 932 mm
  - Average dry season (May to September) precipitation: 72 mm
- Air Temperature:
  - Daily average maximum: between 13 and 16 °C
  - Daily average minimum: between (-11 and 0°C)
- Average Annual Evaporation:
  - Potential annual evaporation: 961 mm
  - Annual evaporation from existing ground: 480 mm
  - Evaporation from dry tailing: 455 mm
  - Evaporation from wet tailing surface: 865 mm

5.3 LOCAL RESOURCE

The main economic activity in the area is subsistence agriculture and cattle farming. Food and basic supplies can be obtained in Yauri/Espinar (62,000 people). Cusco (population 367,000 people) and Arequipa (population 864,000 people) are the nearest major centers, 6 and 7 hours drive from the mine, respectively.

Several permanently flowing streams which supply good flows are present in the area. More significant water sources include the Apurimac or Chilloroya rivers.

Near the community of Chilloroya, the local population is engaged in the informal mining of superficial gold. The company and members of the community have raised concerns with Peruvian authorities regarding environmental and labour conditions related to the informal mining activities. For example, levels of mercury found in sediments near a rudimentary gold processing area exceed international standards for the protection of aquatic life.

5.4 INFRASTRUCTURE

Arequipa and Cusco are the nearest major centres, and both are over 300 km from the Mine site by road. A public, but privately managed, landing strip is located in Espinar, about 2 hours from the mine. The air strip is paved and maintained by Glencore, and Hudbay has authorization to use the strip. Small commercial and/or charter flights can use the air strip for daytime operations.

The road to Yauri (in the general vicinity of Glencore's Antapaccay Mine) was upgraded to meet construction and life of mine transportation requirements. From Yauri, the road system is also currently used for supplies to several operating mines in the region. It is routed through the Arequipa area to the port at Matarani which is currently also used to ship copper concentrates from other mines in Southern Peru.

Power supply to Constancia is brought from a new 220kV transmission line 69 Km from Constancia that was built and is owned by Hudbay but operated and maintained by a third party.

Further details regarding the mine infrastructure can be found in Section 18 of this report.

5.5 PHYSIOGRAPHY

The Constancia region hosts fluvioglacial valleys, flood plains, hills and mountains with either moderate slopes or steep terrain.

Natural hazards in the area were identified through photo interpretation of the World View satellite image taken on June 19, 2012, with a spatial resolution of 0.5 m. The assessment identified six main geodynamic processes: karst geoshapes, rock falls, hillside erosion, river erosion, flood plains and soil saturation. In general, these processes occur locally; on the left and right margins of the Chilloroya river, in the Sacrane, Cunahuiri, Casanuma, Pumacocha, Huayllachane, Telaracaca, Ccatunhuaycco streams, and close to the lakes located northeast of the Mining Unit (Quesoccocha, Pincullune and Sillco).

According to the seismic hazard study conducted for the Constancia mine (Alva J. 2013), major earthquakes recorded intensities from VI to X on the Modified Mercalli scale (MM). Results of the probabilistic seismic analysis indicate that the value for the maximum horizontal acceleration for the OBE (Operating Basis Earthquake) is equal to 0.24 g, corresponding to a PGA (peak ground acceleration) for a return period of 475 years. The value of maximum horizontal acceleration for the MDE (Maximum Design Earthquake) is equal to 0.37 g, corresponding to a PGA for a return period of 2,475 years. The deterministic seismic analysis indicates that the largest acceleration for the Constancia Mining Unit area is 0.44 g, corresponding to the movement of the earth caused by the earthquake of intraplate deep subduction based on the average plus one standard deviation, called MCE (maximum credible earthquake).

6 HISTORY

6.1 EARLY HISTORY

Copper and gold were exploited at Katanga, located approximately 3km northwest of Constancia outside the Constancia property boundary from early last century to the early 1990s.

The Katanga deposit consists of narrow skarn bodies developed in the contact between marbles and monzonite stocks, with Cu, Ag and Au mineralization in hypogene sulphides.

6.2 PRIOR OWNERSHIP

The San José Prospect (now part of the Constancia Mine) was explored by Mitsui during the 1980s with a focus on locating more high-grade ore to add to the nearby Katanga mine operation. Exploration consisted of detailed mapping, soil sampling (1,949 samples), rock chip sampling (1,138 samples), ground magnetic and induced polarization (IP) surveys with several drilling campaigns. Drilling was mainly located in the western and southern sides of the prospect. Mitsui completed 24 drill holes (4,190.5m) and Minera Katanga completed 24 shallow close-spaced drill holes (1,239.8m) at San José.

In 1995, reconnaissance prospecting identified evidence for porphyry style mineralization exposed over an area 1.4 x 0.7 kilometres open in several directions, with some Cu enrichment below a widespread leach cap developed in both porphyry and skarn. Negotiations to acquire an interest in the property were unsuccessful at this time.

In May 2003, the area was revisited by Rio Tinto and the presence of a leached cap and potential for a significant copper porphyry deposit was confirmed. Negotiations with Mitsui, Minera Livitaca and Minera Katanga resulted in an agreement being signed on October 2003 with the underlying owners. The agreements included a joint venture (JV) option between Rio Tinto and Mitsui and Purchase Option Agreements with Minera Livitaca and Minera Katanga to acquire 100% interests in their property. Rio Tinto commenced exploration in December 2003.

The Rio Tinto exploration activities consisted of geological mapping, soil, and rock chip sampling, surface geophysics (magnetics and IP) and completed 24 diamond drill holes. In late 2004, Rio Tinto sought partners for the Constancia prospect, mostly because the property was thought to be too small.

Eventually, Norsemont entered into negotiations with Rio Tinto and these negotiations led to an agreement in early 2005. The first Norsemont geologists visited the property in June of 2005. By 2009, Norsemont had entered into a transfer agreement with Mitsui (November 2007) for the purchae of their 30% interest in the Constancia Project and had exercised the final option with Rio Tinto to acquire the remaining 19% interest in Constancia during 2008 providing 100% ownership of the Constancia Project.

During the period between 2005 and 2010 Norsemont's exploration activities were mainly focused on delineating the known Constancia and San José zones, although surface exploration of adjacent areas were also carried out, specifically focusing on the two new Cu-Au-Mo targets leading to the discovery of the Pampacancha porphyry-related skarn and Chilloroya South porphyry.

Norsemont got approval from the Environmental and Social Impact Assessment for the Constancia project by the Energy and Mines Ministry of Peru on November 24th, 2010.

6.3 HUDBAY OWNERSHIP AND PROJECT DEVELOPMENT

In March 2011, Hudbay acquired a 100% interest in the Constancia Project through its acquisition of all of the outstanding shares of Norsemont by way of a formal take-over bid.

Following the acquisition of the Constancia project in 2010, Hudbay's exploration activities focused on delineating the Pampacancha deposit.

From January to November 2011, there was a campaign of infill drilling and exploratory growth in Pampacancha, in order to define measured and indicated resources of Cu-Au-Mo. The most important was the identification at a north-western area of high grade skarn Cu-Au below post mineral rocks, which has meant a substantial increase in the amount of resources and the deposit grade.

On August 8, 2012, Hudbay's board of directors approved a US$1.5 billion investment to fund the development and construction of its Constancia project in Peru. Following substantial completion of the detailed engineering in the third quarter of 2013, the board approved a revised capital cost estimate for the project of US$1.7 billion.

Hudbay developed Environmental and Social Impact Assessment for the project that were approved by the Energy and Mines Ministry of Peru on August 20th, 2013 and April 15th, 2015.

Construction of the Mine-Milling complex of Constancia was completed on time and on budget at the end of 2014 with first ore mined in the fourth quarter of 2014.

6.4 PRODUCTION HISTORY

The start of 2015 brought substantial change to the Peru team with the transition from a construction project to operations. Constancia achieved commercial production in the second quarter of 2015, and subsequently ramped up to full production in the latter half of the year. Over the course of 2015, Constancia mine operations produced 105,897 tonnes of copper concentrate and 47,263 ounces of precious metals in concentrate.

In 2016 and 2017, Constancia mining operations and cost optimization continued as planned. In 2016 Constancia operations produced 527,296 DMT of Cu concentrate, 382 DMT of Mo concentrate and 26 koz of gold and 2.76 Moz of silver in concentrate. In 2017, the production of Cu concentrate reduced to 479,858 DMT due to a planned reduction in head grade. The Mo concentrate production increased to 914 DMT while 18 koz of gold and 2.37 Moz of silver where also produced in concentrate.

The port expansion at Matarani was completed at the end of June 2016, which improved access to Hudbay's designated pier and reduced port costs.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 DISTRICT GEOLOGY

The oldest rocks in the area (Figure 7-1) correspond to a sequence of white, red, violet or grey medium-grained sandstones with intercalations of reddish mudstones of the Lower Cretaceous Chilloroya Formation (also referred to as the Murco Formation). The Arcurquina Formation discordantly overlies the Chilloroya Formation and correlates with the Upper Cretaceous Ferrobamba Formation. These rocks are exposed in a north-south trend, 15 km long by 5 km wide, comprising a sequence of limestones, calc-arenites and lenses of conglomerates.

These sedimentary formations have been intruded by plutonic rocks belonging to the Andahuaylas-Yauri Batholith of the Oligocene Age. The batholith composition grades in composition varies from dioritic to granodioritic in composition, with plagioclase and orthoclase feldspar, quartz, hornblende, biotite, apatite, zircon and sphene being the main rock-forming minerals. Small seams, veins and lenses of massive magnetite skarn related to the emplacement of the batholiths are common.

Several monzonitic stocks, dykes or laccoliths intrude and cross-cut all the lithologies mentioned above. Where these rocks have intruded limestones, it is common to find mineralized skarns, some of which contain Cu-Au-Ag mineralization such as those at the Katanga mine. Some of the stocks have characteristics typical of porphyry copper deposits such as at Constancia.

7.1 PROPERTY GEOLOGY

7.2.1 CONSTANCIA

The Constancia porphyry copper prospect is located on the eastern margin of the Andahuaylas-Yauri Batholith, approximately 3 km southeast of the old Katanga mine (Figure 7-2).

Stratigraphy

The oldest stratigraphic unit recognized on the prospect comprises clastic sediments corresponding to the Chilloroya Formation, consisting of a sequence of white, red, violet and grey medium-grained sandstones with intercalations of reddish mudstones. Immediately overlying this basal unit are massive, grey micritic limestones with minor intercalations of shales, which outcrop sporadically around the prospect and near the contacts with monzonite, sometimes occurring as roof pendants.

Figure 7-1 Simplified Geology of the Andahuaylas-Yauri Area

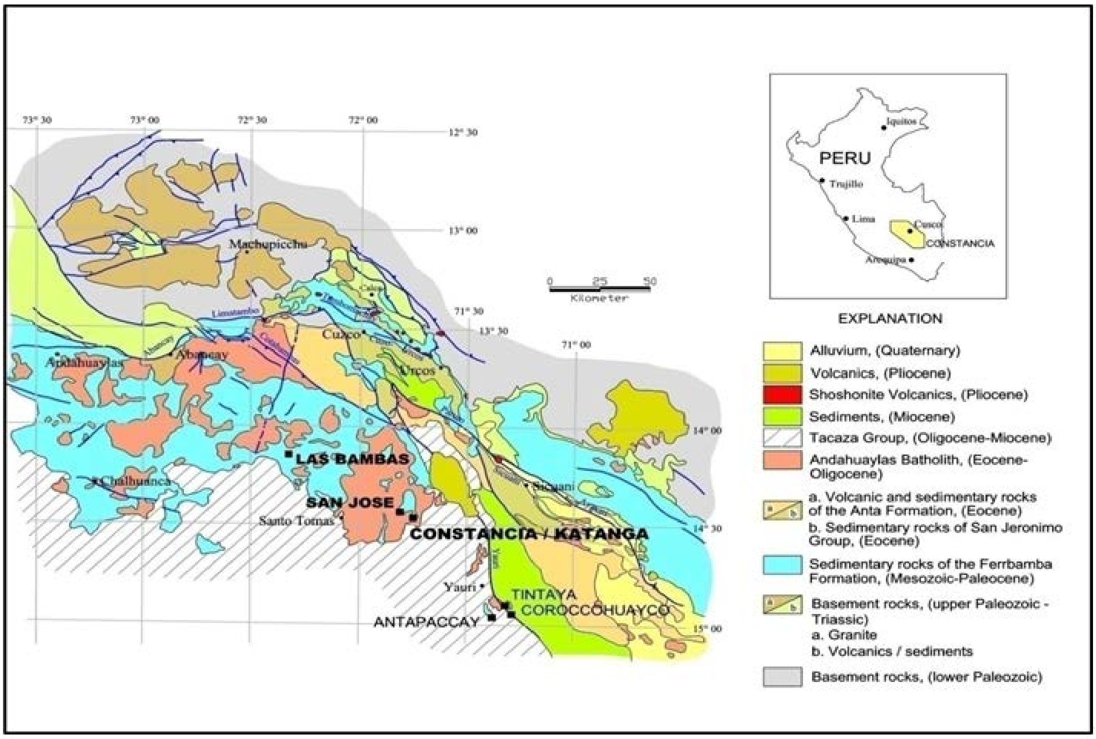
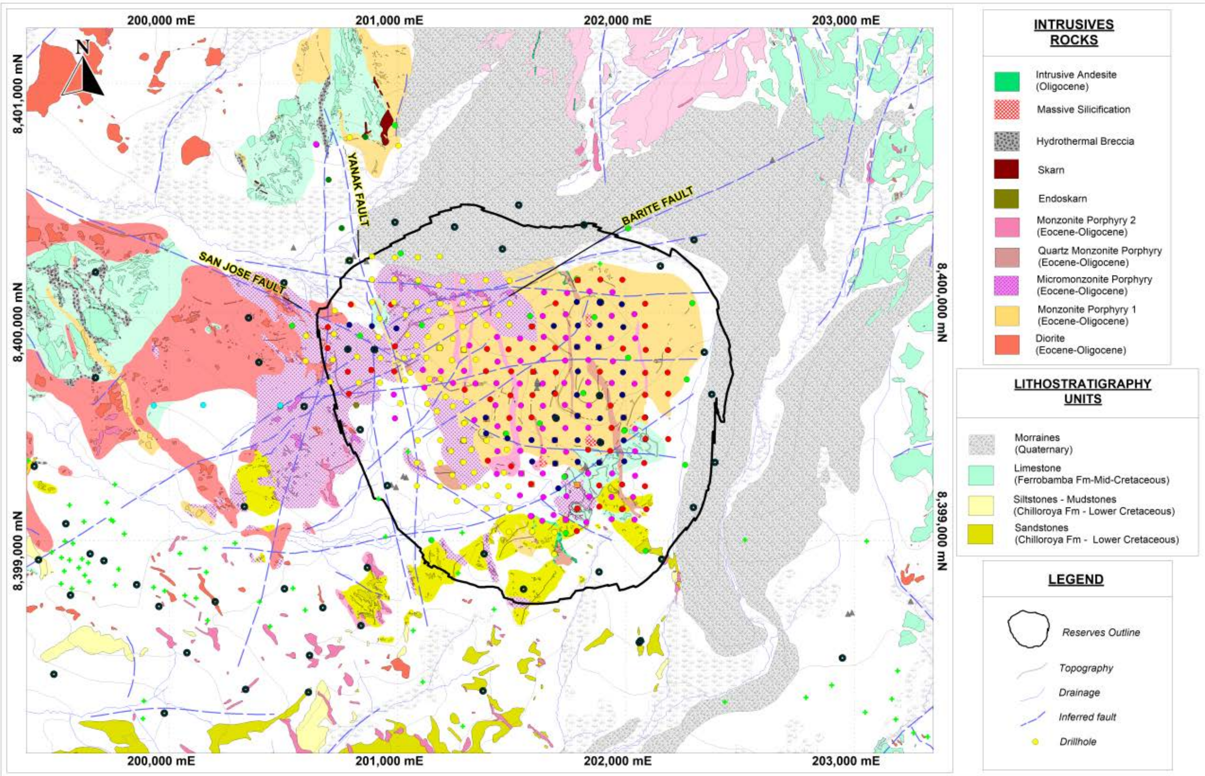


Figure 7-2 Geological Map of the Constancia Deposit



This unit corresponds to the Arcurquina Formation (locally known as Ferrobamba Formation). When in contact with intrusive rocks, these alter to marble or pyroxene diopside-garnet-magnetite-epidote skarn, with or without sulphides. The limestones and skarns dip gently south-east, away from the principal monzonite in the southern part of the Constancia mineralized zone. The overall thickness of the sedimentary package is unknown.

The determination of the base of this limestone unit may be of importance at Constancia, as this potential contact seems to correlate with the favourable skarn horizon at the ex Tintaya Mine and elsewhere in the region. This contact may be found at depth south and east of the presently drilled area, if not intruded by the monzonite.

The underlying clastic sediments of the Murco Formation, and possibly the upper parts of the further underlying Hualhuani Formation (locally known as Soraya Formation), which lithologically consist of sandstones and siltstones, with occasional calcareous and quartzite horizons, are known to host copper mineralization in several copper systems within the belt. At Mitsui's Quechua deposit near Tintaya, and also at the Antillas porphyry copper deposit near Antabamba (north of Constancia), these rocks are the main host for the copper mineralization. The same style of mineralization occurs at Haquira (10 km south of Las Bambas), where Murco sediments are the host for most of the copper oxide resource of the system.

At Constancia, these types of clastic sediments, especially the Murco or Chilloroya Formation which is the host for the copper oxide mineralization at Haquira, have been identified in the southern sector of the property, around the Chilloroya village, where recent surface exploration has identified evidence of porphyry-related copper-gold-molybdenum mineralization associated with the sediments and altered porphyry rocks. Common to these clastic sediments is their iron-stained colouration, which comes from oxidation of former disseminated pyrite.

Glacial moraines cover the northern and eastern margins of the Constancia deposit. To the east these moraines entirely cover potentially important extensions of copper mineralization along broad east-west structural zones.

Intrusions

Multiple phases of monzonite and monzonite porphyry characterize much of the surface area of the prospect, as well as dominating the rock types observed in the drilling to date.

At least four main phases of intrusion are recognized, with the second oldest being associated with the main mineralization event. From oldest to youngest they are:

Diorite: while not part of the intrusive event associated with mineralization, the Andahuaylas-Yauri Batholith forms the 'Intrusive Basement' to the Constancia deposit.

Monzonite Porphyry 1 (MP1): this unit outcrops as a large stock on the Constancia hill, extending west to San José. It hosts most of the porphyry-related mineralization. It is characterized by abundant plagioclase phenocrysts with hornblende in elongated crystals.

Micro Monzonite Porphyry (MMP): characterized by a fine-grained texture with plagioclase crystals, biotite and magnetite in the light gray matrix. This body outcrops as a stock in the south, but is more widespread west of the deposit, including the San José zone.

Quartz Monzonite Porphyry (QMP): this unit occurs mostly as wide, north- south to north-northwest trending dykes with dark, fine–grained chilled boundaries. Abundant plagioclase with tabular, well- formed hornblende crystals occur as phenocrysts in a greenish matrix. These vertical dykes are up to 60 m wide. There is no mineralization associated with the dykes.

Monzonite Porphyry 2 (MP2): this dyke-like monzonite porphyry outcrops between the Constancia and San José zones. It is characterized by abundant plagioclase phenocrysts sub-rounded in a whitish matrix with little magnetite and biotite. The monzonite porphyry occurs as dykes up to 150 m wide, which strike north-south with a steep easterly dip.

Andesite (AAN): dark-gray, aphanitic rock with plagioclase and hornblende phenocrysts. This occurs mainly as narrow dyke-like bodies, some of them close to the contacts with quartz monzonite porphyries.

Structural Geology

As with most porphyry copper complexes, structural activity at Constancia has played the most significant role in preparing and localizing the hydrothermal alteration and accompanying copper-molybdenum-silver-gold mineralization, including skarn formation.

Major inter-mineral and post-mineral fracture systems in the deposit area strike northeast. One of the best known is the ‘Barite Fault’, formed by a number of nearly parallel vein-faults carrying base metal sulphides and barite. These veins have been exploited by artisanal workings throughout the property. This system is clearly visible in the ground magnetic maps, controlling major features including topographic and shear fabric changes in the San José zone.

A second important system strikes north-south. It seems to be more recent than the Barite system, controlling part of the San José deposit and most of the silicified breccias (some of them mineralized) in the system. It shares the same direction as the post-mineral dykes, and may have originated as tension gashes to the Barite direction.

Latest in the sequence is the north-northwest to south-southeast oriented fault system, with the Yanak Fault as the main example. These faults generate wide areas of gouge and milled rock, some of which show high hydraulic gradients.

7.2.2 PAMPACANCHA

Stratigraphy

The main stratigraphic unit in the Pampacancha area is a massive, gray micritic limestone part of the Upper Cretaceous Ferrobamba Formation.

Intrusions

The sedimentary sequence described above is intruded by dioritic porphyry, which is the phase that generated the magnetite skarn that hosts the economic Cu-Au-Mo mineralization. Other intrusive rocks present can be assigned to the unmineralized Oligocene basement dioritic batholiths. This diorite is also intruded by the mineralizing dioritic porphyry (described above). The younger intrusive phases include intra-mineral monzonite intrusions which provide minor local increases in Cu-Au and also locally cut the skarn Cu-Au mineralization.

Diorite (DIO): this unit is characterized by a faneritic texture and in composition shows abundant plagioclase phenocrysts with also hornblende and biotite diorite and limestone constitute the host rock of deposit.

Diorite Porphyry (PDI): this rock is characterized by a porphyritic texture with hornblende phenocrysts and sulfides (pyrite-chalcopyrite-molybdenite) disseminated and veinlets with quartz inside an aphanitic light green matrix.

Monzonite Porphyry 2 (MP2): This rock in Pampacancha occurs as lopolith stock in the middle west of deposits cutting all rock; it is characterized by K-feldspar, plagioclase phenocrysts, hornblende, biotite phenocrysts and magnetite inside a medium gray aphanitic matrix.

Structural Geology

The structural geology of Pampacancha is a dominant factor for the emplacement of Cu-Mo porphyry-skarn and Au Veins mineralization. The occurrence of porphyry is associated to a NE-SW fault and shear zone system that contains three mineralized structural lineaments components NE-SW, NW-SE and NNE-SSW.

7.2 ALTERATION

7.3.1 CONSTANCIA

Potassic Alteration

The potassic alteration assemblage is characterized by secondary potassium-feldspar, and by variable amounts of hydrothermal biotite replacing earlier ferromagnesian minerals and rock matrix. Quartz veining is common; especially “A” and “B” type veinlets. Intensity of alteration is variable, ranging from weak to strong. Hydrothermal magnetite is also present as disseminations and associated with type veinlets in deeper sections. Anhydrite veinlets are also common. Within the potassic alteration zone, chalcopyrite-(bornite)-molybdenite-pyrite mineralization is present in A and B veinlets, and also replacing ferromagnesian minerals or filling fractures.

The high-grade (hypogene) copper mineralization is hosted by a dense A-veinlet stockwork developed in an early porphyry phase. Pyrite/chalcopyrite ratio is typically low and in the order of 1:1 to 2:1. Molybdenite also commonly increases with depth, related to “B” veinlets. Bornite occurs sporadically, especially at deeper levels, sometimes associated with gold values.

Propylitic Alteration

Propylitic alteration is transitional to the potassic alteration and extends more than one kilometre from the porphyry intrusive contacts. The propylitic alteration mineral assemblage includes epidote-chlorite-calcite-pyrite-rhodochrosite. Subordinate chalcopyrite is also present, filling fractures or replacing mafic minerals. Sphalerite-galena veinlets and veins are distributed as a halo to the copper-molybdenum mineralization within the propylitic alteration halo, occurring at distances of up to 3 km away from the porphyry copper system.

Phyllic Alteration

Phyllic alteration forms a pervasive carapace surrounding and sometimes overprinting potassic alteration. This alteration accompanies almost complete destruction of primary rock textures. The phyllic alteration mineral assemblage includes sericite-quartz-pyrite, limited amounts of chalcopyrite and associated occasional “D” veins and veinlets.

7.3.2 PAMPACANCHA

Skarn Alteration

Pro-grade proximal magnetite-chalcopyrite-pyrite skarn grades to distal less well-mineralized garnet and pyroxene skarn which is locally overprinted by epidote-bearing retrograde skarn.

Potassic Alteration

Porphyry diorite and diorite show potassic alteration assemblage characterized principally by quartz veining with rims of K-feldspar stockwork and hydrothermal secondary biotite replacing earlier ferromagnesian minerals; the intensity of alterations is variable, ranging from weak to strong, and the best development of potassic alteration is shown in deep diorite very close to the deposit.

7.3 MINERALIZATION

7.4.1 CONSTANCIA

The Constancia deposit is a porphyry Cu-Mo-Ag system which includes copper-bearing skarn mineralization. This type of mineralization is common in the Yauri-Andahuaylas metallogenic belt where several porphyry Cu-Mo-Au prospects have been described but not exploited.

Five distinct mineral associations are found within the Constancia Project area, namely:

1. Hypogene, porphyry-style mineralization including disseminated, quartz-vein stockwork and fracture-controlled chalcopyrite-molybdenite mineralization in the intrusive;
2. Hypogene chalcopyrite, rare bornite, galena and sphalerite mineralization in skarns;
3. Supergene digenite-covellite-chalcocite (rare native copper) mainly hosted by intrusive rocks lying below the leach cap;
4. Transitional (mixed) including secondary copper sulphides/chalcopyrite in the monzonite (overlap of 1 and 3, above); and
5. Oxide copper, usually cuprite and tenorite appears at this zone associated to malachite-chrysocolla and rarely azurite.

Of these, the hypogene mineralization (Type 1) constitutes the bulk of the deposit. Skarn (Type 2) is volumetrically smaller, but grades are normally higher, and mineralization occurs at or near the surface. At the contact between the intrusive and limestones, magnetite - garnet skarn develops, while the pyroxene–diopside (garnet–epidote) association is more common in calcareous sandstones and of the Chilloroya Formation. Supergene enrichment (Type 3) occurs immediately beneath, and occasionally as remnants within, the leached cap. The highest copper grades in the Constancia porphyry are typically associated with this and with the skarn zone. The transitional or mix zone (Type 4) corresponds to the zone where the supergene and hypogene mineralization co-exist. Finally, oxide copper mineralization (Type 5) occurs in shallow levels and deeper along fractures.

Two areas of porphyry-style mineralization are known within the project area, Constancia and San José. At Constancia, mineralization is deeper than that observed at San José which occurs at surface. The mineralized zone extends about 1,200 m in the north-south direction and 800 m in the east-west direction. At the present time, both areas are part of the Constancia Pit.

7.4.2 PAMPACANCHA

The Pampacancha mineralized body has been divided into three sectors, according to lithology and mineralization (

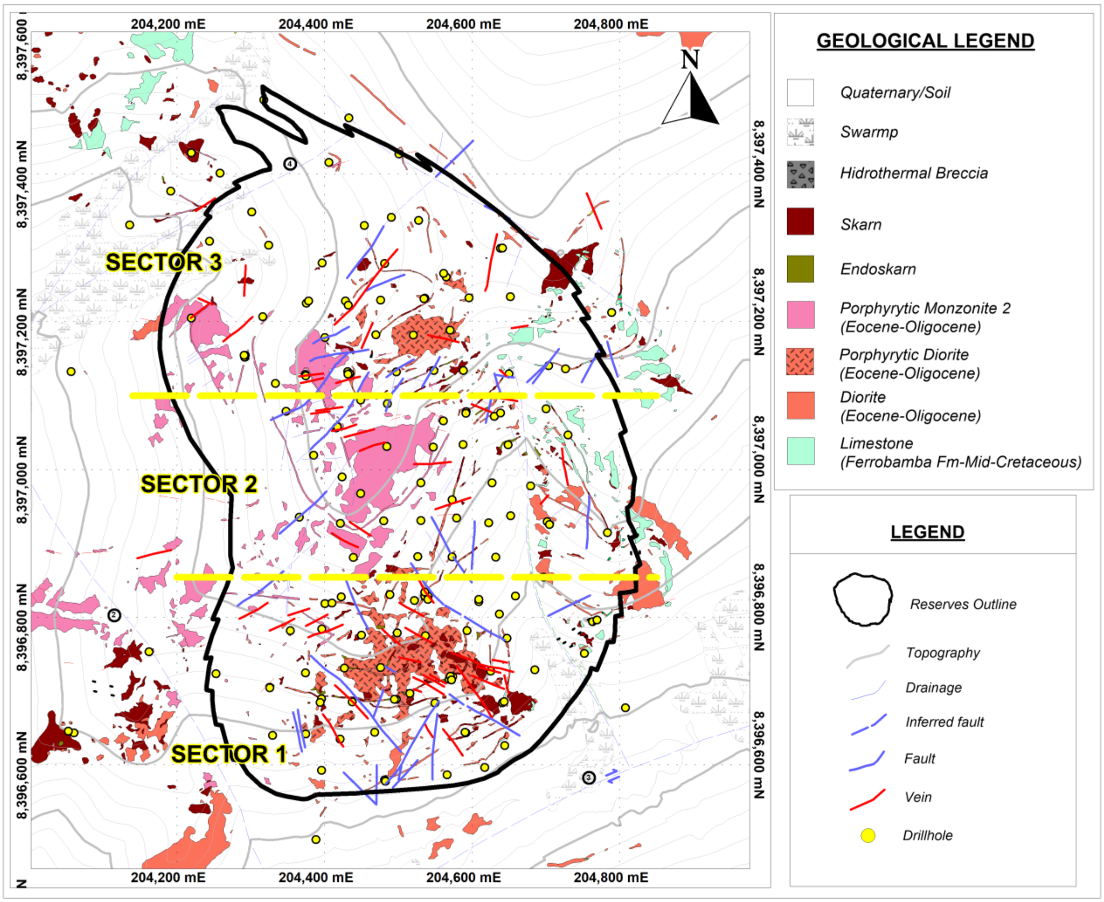
Figure 7-3).

Sector 1 is characterized by the predominant magnetite skarn over calc-silicate skarn. Mineralization in sector extends from surface down to about 200m (true depth) with occasional deeper lenses. The skarn is intruded by potassic altered dioritic porphyry and fine-to-medium grained diorite (also exhibiting potassic alteration).

Sector 2 is located immediately north of Sector 1 and is characterized by the presence of a limestone bracketed between two main skarn layers, vertically extending from 70 m to about 250m. The shallower segment of the skarn shows alternations with thin limestone layers, all disrupted by the emplacement of diorite porphyry dikes and endoskarn.

Sector 3 extends immediately NW of Sector 2 and has the same characteristics as Sector 2, with a limestone body intercalated between two skarn layers, which pinch out northward, eastward and westward.

Figure 7-3 Pampacancha Sectors



Epithermal mineralization of the low sulphidation quartz-sulphide Au + Cu style, accounts for common supergene enriched Au anomalies, along with other features such as hydrothermal alteration and veins typical of near porphyry settings.

8 DEPOSIT TYPES

The Constancia deposit is a porphyry Cu-Mo-Ag system which includes copper-bearing skarn mineralization. This type of mineralization is common in the Yauri - Andahuaylas metallogenic belt where several porphyry Cu-Mo-Au prospects have been described but not exploited.

The main porphyry deposits in the belt include Antapaccay (777 Mt @ 0.48% Cu, Glencore Resources and Reserves, as of Feb 2018 Report) and Quechua (~350 Mt @ 0.38% Cu, Mitsui Kinzoku press release, Nov 2015). Several other porphyry prospects are also being explored in the district.

Historically, the belt was better known for copper skarn deposits such as Las Bambas (copper skarn-porphyry) where MMG recently reported resources of 1,872 MT @ 0.62% Cu (MMG Mineral Resources and Ore Reserves Statement as of June 30, 2017).

The Pampacancha deposit is a Cu-Mo-Au porphyry related skarn system, with copper-bearing skarn mineralization. As mentioned above, this type of mineralization is common in the Yauri-Andahuaylas metallogenic belt.

The Constancia project area geology contains the classic characteristics typical of Andean porphyry copper deposits. The following generalized geological characteristics are the typical targets of exploration compilation efforts and support the classic genesis model of a copper porphyry:

- Associated to volcanic arc igneous assemblages.
- Ore bodies are associated with multiple intrusions and dikes of diorite to quartz monzonite composition with porphyritic textures.
- Breccia zones with angular or locally rounded fragments are commonly associated with the intrusives. The sulfide mineralization typically occurs between or within fragments.
- The deposits typically have an outer epidote - chlorite mineral alteration zone.
- Quartz - Sericite alteration zone typically occurs closer to the center and may overprint.
- A central potassic zone of secondary biotite and orthoclase alteration is commonly associated with most of the ore.
- Fractures are often filled or coated by sulfides, or by quartz veins with sulfides.
- Closely spaced fractures of several orientations are associated with the highest grade ore.
- The upper portions of porphyry copper deposits shows supergene enrichment with the metals in the upper portion being dissolved and carried down to below the water table, where they precipitate.

9 EXPLORATION

Copper and gold exploration in the Constancia area dates back to the early 1990s. The San José prospect (now part of the Constancia Mine) was explored by Mitsui during the 1980s with a focus on delineation of high-grade ore amenable to process at the Katanga facility. Exploration consisted of detailed mapping, soil sampling (1,949 samples), rock chip sampling (1,138 samples), ground magnetic and IP surveys with several drilling campaigns, mainly located in the western and southern sides of the prospect.

Mitsui completed 24 drillholes (4,190.5 m) and Minera Katanga completed 24 shallow, close-spaced drillholes at San José (1,239.8 m). Through a prospecting program carried out in 1995, an area of 1.4 km x 0.7 km (Constancia main) with porphyry-style mineralization was mapped and determined to be open in several directions.

Rio Tinto carried out exploration in the Constancia area between 2003 and 2004 including geological mapping, soil and rock chip sampling, and surface geophysics (magnetic; 20.3 lines-km and IP 12 lines-km). Rio Tinto completed 24 diamond drill holes for a total of 7,484.15 m.

Norsemont exploration activities between 2007 and 2011 in the district included the mapping of 11,444 hectares at the Constancia Project at several scales, including 1:1,000, 1:2,000 and 1:5,000. Of this, 8,905 hectares were mapped on Hudbay's mining concessions, which represent 39% of Hudbay's mining rights in the area. Additionally, 2,595 rock samples and 41 stream sediments samples were collected during this period. Ground magnetic and IP surveys cover all areas of Hudbay's concessions (magnetic; 281 lines-km and IP 319,6 lines-km).

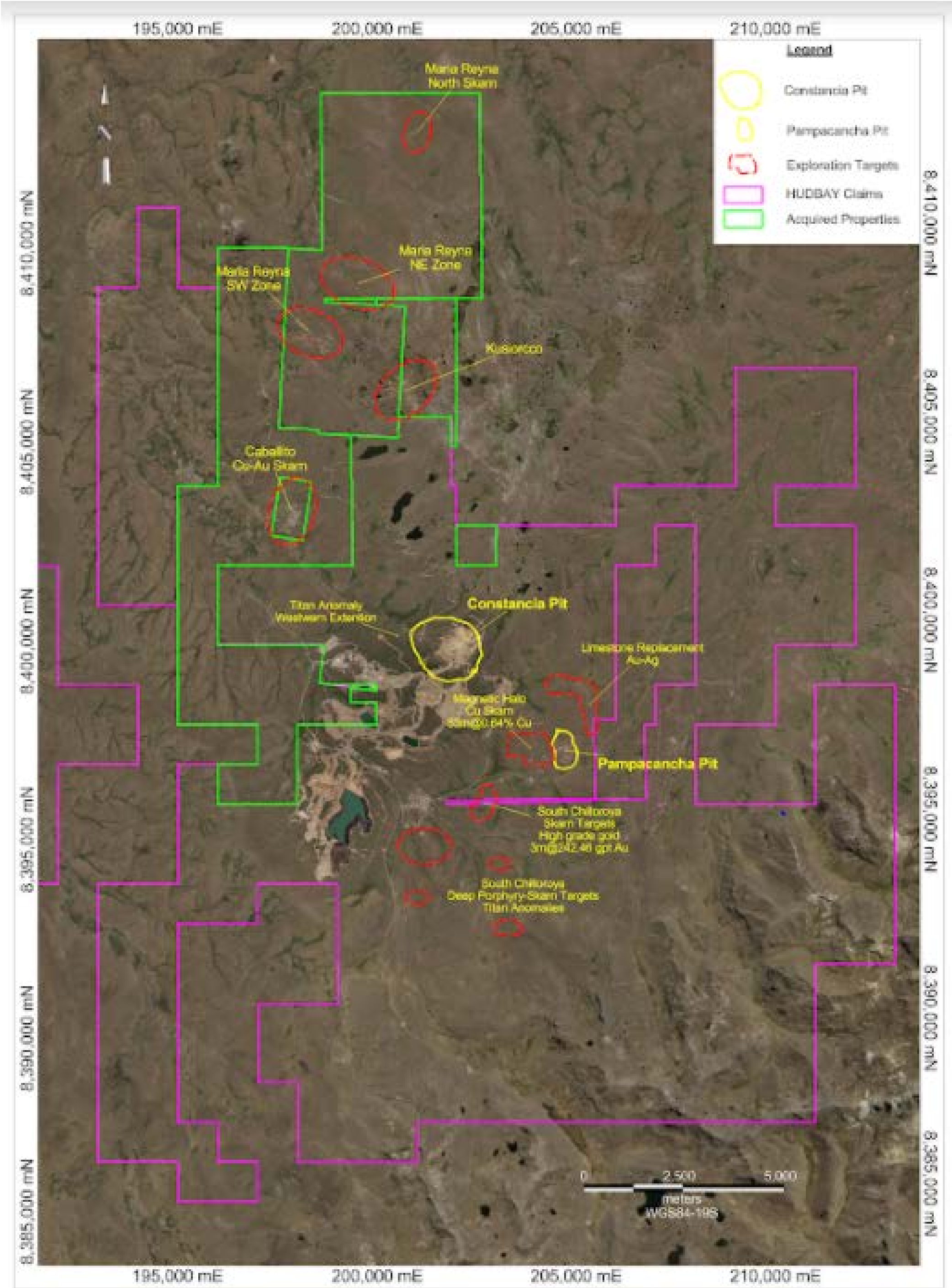
From 2005 to 2010, Norsemont drilled 153,556m for infill, condemnation, metallurgical, geotechnical, hydrogeological and exploration, drilled by core and reverse circulation.

Following its acquisition of the Constancia Project in March 2011, Hudbay's exploration activities focused on delineating the "new" Pampacancha deposit and understanding the district exploration upside. Since acquisition, Hudbay has completed over 45,000 m of drilling for resource definition, condemnation, metallurgical, geotechnical, hydrogeological and exploration drilling. These holes were drilled by core and reverse circulation. From 2012 to 2014 Hudbay also continued exploration through mapping of 10,703 hectares at 1:5,000 scale and collecting 756 rock samples and 124 stream sediments samples.

The author is of the opinion that the sampling methodologies and drill frequency is sufficient and that they are representative of the geology and deposits described.

In addition to the Constancia and Pampacancha prospects, other peripheral targets have been identified through prospecting, mapping, grab sampling and geophysics, including conventional IP and Titan 24 (38.4 km covering 8 lines, which was completed in 2011). The following Figure 9-1 shows a general view of the current Constancia district targets.

Figure 9-1 Near Mine Constancia Exploration Potential



In January 2018, Hudbay announced the entering into of the following agreements to acquire mining properties in southern Peru near its Constancia mine (Figure 9-1): (i) an option agreement with a private Peruvian consortium to earn a 100% interest in the Caballito (formerly Katanga) and Maria Reyna mining properties; and (ii) an agreement to acquire from Panoro Minerals Ltd. 100% of the Kusiorcco mining properties.

Caballito

The Caballito property, located approximately three kilometres northwest of Constancia, is a 120-hectare (297-acre) concession block and is the site of the former Katanga mine, which was operated by Mitsui Mining & Smelting Co., Ltd. and Minera Katanga at different times between the late 1970s and early 1990s. The deposit at Caballito consists of narrow skarn bodies developed in the contact between limestone and monzonite porphyries with copper, silver and gold mineralization in hypogene sulfides. Reliable available data over this area is limited to aeromagnetic and radiometric maps.

Maria Reyna

The Maria Reyna property, located within ten kilometres of Constancia, is a 5,850-hectare (14,456-acre) concession block. In 2010, diamond drilling by a previous optionee of the Maria Reyna property, intersected copper skarn, breccias and porphyry mineralization. Geophysical surveys and geological mapping have also been conducted on the property and Hudbay believes that the area remains very prospective for additional discoveries.

A summary of the historic drill results from Maria Reyna is contained in Table 9-1, however a qualified person has not independently verified this historical data or the quality assurance and quality control program that was applied during the execution of this drill program for Hudbay and, as such, Hudbay cautions that this information should not be relied upon by investors.

Kusiorcco

The Kusiorcco property is located within seven kilometres of Constancia and is nearby the Caballito and Maria Reyna properties. It consists of 10 concessions totalling 3,962 ha and hosts a large mineralized dacidic porphyry intrusive showing presence of quartz-sericite alteration, quartz stockwork and a leached cap. Geophysical data includes 24 kilometres of ground magnetic survey and 3 IP lines showing a coincident 1-km by 2-km induced polarization and resistivity anomalies overlying a 300m by 500m alteration zone.

Permitting and community relations work is ongoing to support exploration work on the Caballito, Maria Reyna and Kusiorcco properties.

Table 9-1: Historic Drill Results from the Maria Reyna Property

Vale Drill Intersections at 0.2% CuEq[1] Cut-off							
Hole ID	From (m)	To (m)	Ag (ppm)	Cu (%)	Mo (ppm)	CuEq %	Interval (m)
DH-001	206	256	1.5	0.20	113	0.27	50
DH-002	0	136	4.1	0.52	78	0.61	136
DH-003	226	256	1.7	0.24	122	0.31	30
	460	480	0.3	0.19	62	0.22	20
DH-004	10	240	3.0	0.26	124	0.35	230
	336	486	1.5	0.18	147	0.27	150
	502	522	0.8	0.19	87	0.24	20
DH-005	10	76	4.8	0.63	122	0.74	66
DH-006	0	114	4.0	0.32	112	0.41	114
DH-007	0	106	2.5	0.39	267	0.55	106
	176	216	1.7	0.25	280	0.41	40
	232	310	1.0	0.17	272	0.31	78
DH-008	256	394	1.4	0.28	130	0.36	138
	432	519.85	1.7	0.23	209	0.36	87.85
DH-009	18	90	1.7	0.28	335	0.47	72
	110	172	0.7	0.14	184	0.24	62
	196	256	0.9	0.18	106	0.24	60
DH-010	262	314	1.7	0.30	204	0.42	52
	344	406	2.1	0.34	641	0.68	62
DH-011	18	178	2.9	0.50	998	1.03	160
	374	406	1.1	0.14	175	0.24	32

Note: The intersections represent core length and are not representative of the width of the possible mineralised zone.
Note: For additional information, including drill hole locations and the data verification and quality assurance / quality control carried out by the prior owner, please refer to Management's Discussion and Analysis for Indico Resources Ltd. ("Indico") for the year ended May 31, 2014, as filed by Indico on SEDAR on September 29, 2014.
[1] Intervals were calculated with maximum of 10m of 0.1% CuEq internal dilution, 0.2% CuEq edge grade, minimum length of 15m. For CuEq calculations the following variables were used: $3.00/lb Cu, $15.00/lb Mo, $21.00/oz Ag; no allowances for metallurgical recoveries were made.

10 DRILLING

10.1 OVERVIEW

Extensive drilling has been conducted at Constancia since the early 2000's. The two most recent drilling programs were completed by Hudbay, with prior drilling campaigns performed by Rio Tinto and Norsemont Mining (Table 10-1).

Table 10-1 Drilling Programs by Year (in metres drilled)

Company	PQ	HQ	NQ	RC	HOLES	TOTAL
Rio Tinto (2003-2004)		7,124	359		24	7,483
NOM 2005		9,799			41	9,799
NOM 2006		20,026	377		66	20,403
NOM 2007		23,863	5,197		77	29,060
NOM 2008	3,380	39,502	7,374	12,792	219	63,048
NOM 2009		4,487	113	409	33	5,009
NOM 2010		16,604	1,933	7,694	93	26,231
HB 2011		28,090	1,866	984	186	30,940
HB 2012		5,045	130	464	46	5,639
HB 2014 - 2015		4,353			26	4,353
HB 2017		5,298			21	5,298
GRAND TOTAL	3,380	164,191	17349	22343	832	207,273

Exploration, infill, metallurgical, geotechnical, hydrogeological, condemnation and twin-hole drilling was conducted in order to adequately assess the deposit (Table 10-2).

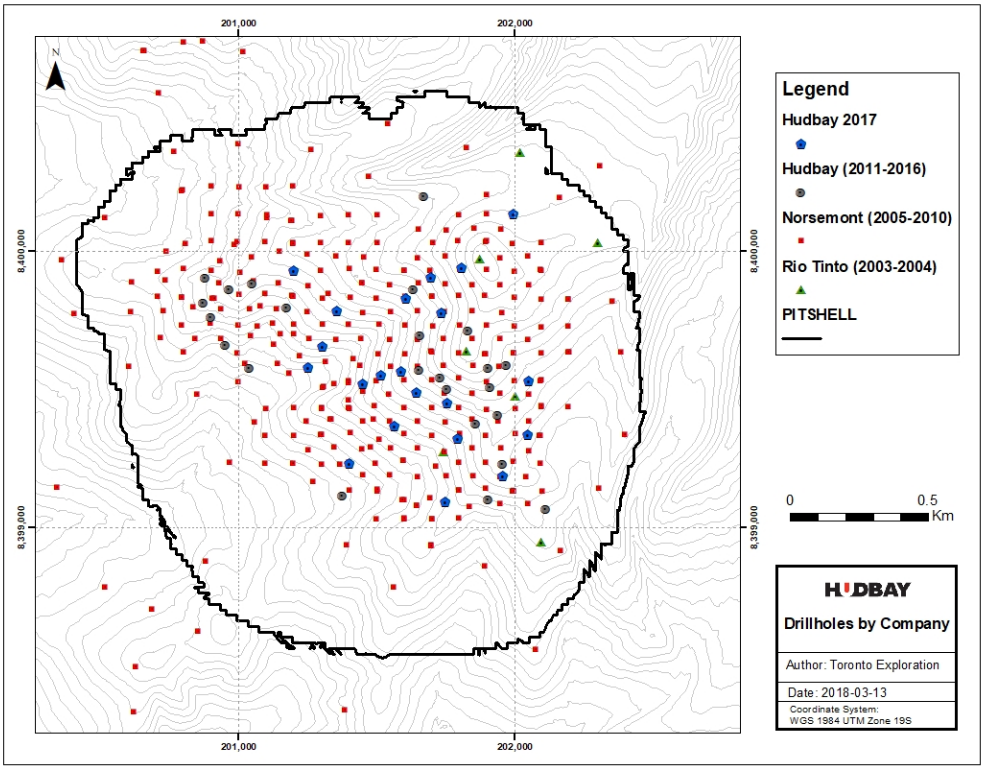
Table 10-2 Drilling Programs by Types

PROGRAM	COMPANY	METREAGE	HOLES
Constancia - Infill	Rio Tinto/Norsemont/Hudbay	328	112,376
Constancia - Condemnation	Norsemont/Hudbay	75	14,481
Constancia - Metallurgical	Norsemont/Hudbay	31	5,091
Constancia - Geotechnical	Norsemont/Hudbay	165	12,245
Pampacancha - Geotechnical	Hudbay	3	727
Pampacancha - Hydrogeological	Norsemont/Hudbay	27	3,816
Pampacancha - Infill	Norsemont/Hudbay	144	40,427
Constancia - Exploration	Norsemont	38	12,812
Constancia – Twin drilling	Hudbay	21	5,298
TOTAL		**832**	**207,273**

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 Constancia and Pampacancha deposits.

Figure 10-1 Map of Drill Hole Location



10.2 RIO TINTO (2003-2004)

Rio Tinto drilled 24 diamond drill holes for a combined length of 7,483 m, using an average azimuth of 110° and average dip of 70°. The average depth is 312 m and the drill spacing is 300 m. Information for recovery is not available. Collar locations were surveyed with a hand-held GPS and certified by Horizons Consulting of Lima, Peru. Down hole surveying is not available for this campaign.

10.3 NORSEMONT MINING (2005-2010)

Norsemont Mining drilled a total of 529 holes for a combined length of 153,550 m, using an average azimuth of 150° and average dip of 78°. The average depth is 287 m and the drill spacing is 70x70 m. The overall core recovery was coded as 95% in the original logging however further investigations of core pictures revealed that the recovery was measured on a length basis. Random checks conducted over several thousands of samples have shown that in reality half of the samples had a core recovery inferior to 85% when measured on a volumetric basis. This issue is quite common in the Peruvian porphyry deposits with intense alteration.

Collar locations were surveyed with a hand-held GPS and certified by Horizons Consulting of Lima, Peru. Down hole surveying was conducted on 30 m intervals (before 2007) and 50 m intervals (after 2007) using a combination of Flexit and Maxibor Survey instruments.

10.4 HUDBAY (2011-2017)

Hudbay acquired Constancia and conducted exploration, infill, metallurgical, geotechnical, hydrologeological, metallurgical and condemnation drilling between 2011 and 2015. A total of 262 drill holes were drilled using an average azimuth of 65° and average dip of 85°. The average drill spacing is 80x80 m. Holes were usually collared 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. For this drilling, it was also found that actual volumetric core recovery was inferior to the linear core recovery that had been recorded in the database.

In 2017, a twin-hole program of 21 holes was specifically conducted to assess the impact of material loss in previous drilling campaigns. Drilling was conducted with triple tube coring while using lubricants and minimizing fluid pressure. Volumetric core recovery significantly improved in this new campaign compared to the twinned historical Norsemont and Hudbay holes with the proportion of samples with core recovery over 85% increasing from 50 to more than 75%. The findings of the twin-hole study are discussed in more details in section 14.2.

The final collar locations were recorded with a hand-held GPS. All measurements were recorded using UTM coordinates based on the Provisional South America 1956 (PSAD56) datum. Collar locations were surveyed and certified by Hudbay Peru.

Before 2014, the drilling azimuths were set up by marking the front and back sights using a Brunton compass. For the latest drill campaigns, the drilling azimuths were set up with a Differential GPS Trible R8 model 4 with a precision of 3 mm. The inclinations were measured with the drill rig inclinometer (typically, a digital inclinometer). Comparisons between the designed and downhole survey measurements at the base of the casings have shown negligibale variations.

Down hole surveys were measured every 30 m intervals using a hand-held GPS which measured inclination/dip and azimuth direction. For the latest drill campaign, down hole surveys were conducted on 30 m intervals using a Gyro Survey instrument which measured inclination/dip and azimuth direction.

At Pampacancha, down hole surveys were measured every 3 m intervals using a Maxibor 2 instrument which measured inclination/dip and azimuth direction.

11 SAMPLE PREPARATION ANALYSES AND SECURITY

Sample preparation, analysis, and security procedures were reviewed by Cashel Meagher, P. Geo. and Qualified Person of this Technical Report. The sampling methodology, analyses and security measures have been documented in great details already in past published technical reports on the Constancia property and will only be summarised in this document. Unless specified in the report, all the laboratories used for samples prepararation, analysis and security are independent of Hudbay.

11.1 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 were inserted 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 being closed with a tightly fitted lid and being delivered to the core processing areas at the Constancia. Private 24-hour per day security guards or Hudbay personnel controlled site access and oversaw sample security at each camp and drill site.

Prior to measuring the core recovery parametres and rock quality data (RQD), visual checks were performed for incorrect placement and orientation of core fragments. Discrepancies caused by mislabelled or misplaced block markers were resolved by consulting the drilling contractors. The drill core was marked with cut lines designed to provide the most representative split.

Standard parametres for core recovery and RQD for each drill run were measured by either the trained geotechnicians or geologists. All core logging was performed by experienced 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.

Before 2014, all drill holes were logged on paper logs and later transferred into spreadsheets. Later, drill holes were logged directly on PC tablets. Drill cores are divided into sub-intervals based on the rock types observed by the geologists. Each interval was described for alteration, mineralization, and oxidation state of the primary sulfides. Any significant veins found were also logged along with identifiable structures.

11.2 SAMPLE SELECTION

Core samples for assaying were selected by the core logging geologist. The typical sample interval is 1 to 2 m long in mineralized rocks and 3 m in barren samples while being mindful of lithological contacts. Geologists were responsible for filling 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 (analyses generated from the same interval from a coarse reject or a quarter split), 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 (blanks, duplicates and standards), 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.

11.3 CORE PHOTOGRAPHS

Logged core boxes with the sample intervals marked and sample tags inserted were photographed using a digital camera mounted to a tripod in natural light. All photos taken were loaded on to a laptop computer and reviewed by the on-site database manager. Any photos deemed unacceptable were retaken.

11.4 CORE SPLITTING

Prior to splitting 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 to label sample bags. At the core splitting 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 quarter split, the second QA/QC sample tag was also placed in the same bag. Core was split along the cut line drawn by the geologist. In gouge and rubble intervals, an aluminium sampling scoop was used to separate the gouge into two halves in the core boxes. Completed sample bags were closed using the bag draw strings and secured at the neck using two zip ties. All saws and sampling buckets were rinsed with water after cutting each sample to prevent cross contamination.

11.5 SAMPLE DISPATCHING

A requisition form that listed the range of sample numbers, job order number, requested analytical codes and any special instructions was created. The requisition form were emailed to the preparation laboratory prior to sample shipment. QA/QC samples including blanks, duplicates 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.

11.6 SAMPLE PREPARATION

11.6.1 HISTORICAL

Table 11-1 presents the summary of the sample preparation method used for the drill campaigns conducted by the previous owners.

Table 11-1 Summary of the Sample Preparation Method by Companies

Company	Mitsui and Minera Katanga	Rio Tinto	Norsemont	Norsemont	Norsemont	Norsemont
Year	1970-1980	2003-2004	2005	2006-2007	2008-2009	2010
Core split	*unknown*	Yes	Yes	Yes	Yes	Yes
Laboratory	*unknown*	ALS Chemex, Lima	ALS Chemex, Lima	ALS Chemex, Lima	ALS Chemex/ SGS, Lima	SGS, Lima
ISO Certified	*unknown*	Yes	Yes	Yes	Yes	Yes
Drying	*unknown*	*Regular samples 110 °C*	*Regular samples 110 °C*	*Regular samples 110 °C*	*Regular samples 100 °C+/- 5°C*	*Regular samples 100 °C+/- 5°C*
Crushing	*unknown*	Jaw	Jaw	Jaw	Jaw	Jaw
Mesh size	*unknown*	-10 Mesh (2mm)	-10Mesh (2mm)	-10 Mesh (2mm)	-10 Mesh(2mm)	-10 Mesh (2mm)
Spitting	*unknown*	Riffle	Riffle	Riffle	Riffle	Riffle
Weight of sub-sample	*unknown*	250 g	250 g	250 g	250 g	250 g
Size of sub-sample	*unknown*	≥85% passing through -200 mesh (75 µm)	≥85% passing through -200 mesh (75 µm)	≥85% passing through -200 mesh (75 µm)	≥95% passing through -140 mesh (106 µm)	≥95% passing through -140 mesh (106 µm)
Grinding bowl	*unknown*	Steel /Chrome	Steel /Chrome	Steel /Chrome	Steel /Chrome	Steel /Chrome
Quartz wash	*unknown*	*unknown*	*Yes*	*Yes*	*Yes*	*Yes*
Assay charge	*unknown*	30 g	30 g	30 g	30 g	30 g
Storing of coarses and pulps rejects	*No*	*No*	*No*	*Yes*	*Yes*	*Yes*

11.6.2 HUDBAY

Drill core samples were picked up at the core processing facilities and transported via Sodexo from the camp site to Arequipa and ALS Chemex-SGS laboratories, and finally from Arequipa to Lima, Peru. Samples were weighed upon arrival, dried at 100°C, and crushed in jaw crushers to 90% passing through #10 mesh (2 mm). The entire crushed samples were homogenized, riffle split, and 250 g subsamples were pulverized to ≥95% passing through #140 mesh (106 μm) using steel/chrome 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. The remaining coarse and pulps rejects were sent for storage to Abil Corporation warehouse, located in Lima, Peru.

ALS Chemex and SGS have 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.

As for the 2017 twin hole program, drill core samples were picked up at the core processing facilities and transported page via Transportes ACOINSA from camp site to Certimin laboratory located in Lima, Peru. Samples were relabelled, weighed upon arrival and crushed in jaw crushers to 100% passing through #10 mesh (1.7 mm). The entire crushed samples (about 12 Kg) were homogenized and splitted via rotary splitter (about 500 gr). For every sample, one sub sample was kept while the other was pulverized to ≥85% passing through #200 mesh (74 μm) using Steel/Chrome 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. The remaining coarse and pulps rejects are stored temporarily at Certimin facilities located in Lima, Peru, and after finishing all processes (metallurgical), will be transported to Constancia mine for final storage.

Pulp samples prepared at Certimin were dispatched to SGS Lima for ICP-EOS analysis. Upon arrival, samples were weighted and 4% were randomly selected from individual batches for granulometry analysis. If the pulp sample's granulometry failed to achieve ≥95% passing through #140 mesh (106 μm), the samples in the batch were pulverized to ≥85% passing through #140 mesh (106 μm) using steel/chrome grinding bowls. Grinding bowls were cleaned by brush and compressed air between samples. Cleaning with a quartz wash was conducted between jobs. The remaining pulps rejects were stored temporarily at SGS facilities, located in Lima, Peru. Once the assay program was completed, all the pulps reject were transported back to the Constancia mine for final storage.

Certimin (Lima, Peru) 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.

SGS (Lima, Peru) 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.

11.7 BULK DENSITY

11.7.1 HISTORICAL

A total of 1,900 samples from 269 drill holes were collected for specific gravity determinations and sent to ALS Chemex and SGS labs in Lima, Peru. Density measurements were done using immersed wax coated core. The drill core samples, 8 to 10 cm long, were taken every 50 m from half-split core.

At the laboratory, samples were dried at 80°C overnight (12 hours) 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 0.01 mg, and 0.001 mg calibration weight. The sample is immersed in a pan containing molten paraffin, 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[3] is then used to calculate the specific gravity. In addition and following the same methodology, Norsemont re-analyzed 281 samples at SGS, Lima, Peru as part of its QC program. The results have shown that the SG values are accurate.

11.8 ASSAY METHODOLOGY

11.8.1 RIO TINTO

Drill core samples were analyzed for 27 elements by Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES), with four acids digestion. Selected samples above the

[3] Density = W1 / (V1 – (W2 – W1/Density Wax))

detection limit were analyzed by Atomic Absorption Spectroscopy (AAS). All the analysis was performed by ALS, Lima, Peru using the method ME-ICP61 and AA62 for Cu over-ranges.

Sequential copper data is not available. According to an exploration report from Rio Tinto Exploration (2004), limited Cu soluble analysis (CN leach) was performed over selected intervals when chalcocite was described as the main Cu-bearing mineral.

11.8.2 NORSEMONT

Drill core and RC samples were analysed by ALS Chemex for 27 elements using ME-ICP61 and by SGS for 41 elements using ICP40B with four acid digestions. Samples above detection limit were analysed by Cu_AA62 method.

11.8.3 HUDBAY

Constancia

Drill core samples were analyzed for 41 elements by Inductively Coupled Plasma - Optical Emission Spectroscopy (ICP-OES) with HNO3-HClO4-HF-HCl digestion and HCl Leach. Samples above detection limits were analyzed by AAS, while gold was analyzed by Fire Assay with AAS finish. Samples with copper concentrations above 2,000 ppm were systematically re-assayed by a sequential copper method (AAS73B).

As for the 2017 twin-hole program, drill core samples were analyzed for 41 elements by ICP-OES with HNO3-HClO4-HF-HCl digestion and HCl Leach and by AAS, while gold was analyzed by Fire Assay with AAS finish.

Pampacancha

Samples were analysed by the SGS laboratory through Inductively Coupled Plasma – Optical Emission Spectrometry (ICP-OES) after multi-acid digestion and gold was determined by fire assay with Atomic Absorption Spectroscopy (AAS). SGS is currently registered with ISO/IEC 17025 and ISO 9002 accreditation.

Samples were analyzed for 41 elements through ICP-OES (ICP40B Lab Code) after multi-acid digestion of a 0.20 g of sample. If silver, copper, iron, molybdenum, lead and zinc values were above upper reporting limit, then the element was re-assayed using AAS after multi- acid digestion of an 0.25 g of sample. Gold was analyzed by AAS after fire assay using 30 g of sample (FAA313 Lab Code) and values greater than 5000 ppb were re-assayed using a gravimetric finish. All analytical balances are certified annually by a third party. Check weights are used daily to verify calibration of balances. All metal standards used to make the calibration standards for the AAS and ICP are certified and traceable.

Samples with copper concentrations above 2,000 ppm were systematically re-assayed by a sequential copper method (AAS73B).

11.9 DATA VALIDATION

11.9.1 DRILL HOLE DATABASE

Hudbay built an entirely new drill hole database from all pre-Hudbay drilling and assaying information. The following subsections describe the process Hudbay used to build a completely new database of the drilling and assay values and the steps taken to verify the information. All non-Hudbay 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.

Drill Hole Collars

Drill hole collar coordinates of pre-Hudbay drilling were resurveyed and reported in a single grid system. The coordinates were converted to WGS 84 UTM Zone 19 using ArcGIS Software. The conversion was based on a best fit transformation using drill hole collars and corners from claim boundaries (approximately 800 points in total). This conversion was verified by plotting the converted coordinates against a drill hole collar compilation map prepared by Horizons South America in 2013, and the results are within an acceptable margin of error (+/- 0.07 m).

Downhole Surveys

Downhole survey files exist for 379 of the 463 historical drill holes. The majority of the downhole surveys were conducted by Geotec and Comprobe using survey methods that measured the drift angle and azimuth. The readings were generally recorded every 50 m for Flexit and every 3m for Maxibor. 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 463 historical drill holes, 17 were drilled vertically.

Hudbay field-checked the collar positions and visually verified plans and the database for correctness. Down-hole surveys were checked by examining coarse changes in the variables. Check runs were done at regular intervals to check the consistency of the drilling data.

A random revision from the lithology and mineralization log data and assay certificates from 27 drill holes was performed. Discrepancies were not identified between the log data and assay certificates and the drill hole database used for the mineral resource estimate.

An internal validation of the Pampacancha drillhole database against the original drill logs and assay certificate information was carried out by Hudbay. No significant discrepancies were observed within the database.

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 server.

11.10 QUALITY ASSURANCE AND QUALITY CONTROL PROGRAMS

11.10.1 HISTORICAL

Table 11-2 displays the standard reference material used while Table 11-3 presents overviews of the quality control used.

Table 11-2 Standard Reference Material

Company	Year	Reference material	Origin	Au ppm	Ag ppm	Cu ppm	Mo ppm	Zn ppm
NOM	2005	CDNCGS_ 6		0.266		3246	11	
NOM	2005	CDNCGS_7		0.963	3.59	10072	13	
NOM	2006	GQ601193	SGS			2452	86	
NOM	2006	GQ601194	SGS			5166	319	
NOM	2006	GQ601195	SGS			7384	65	
NOM	2006	GQ601196	SGS			12500	159	
NOM	2006	MV600011	SGS			2052	74	
NOM	2006	MV600013	SGS			5052	127	
NOM	2006	MV600014	SGS			7503	81	
NOM	2007	MV600015	SGS			24476	120	
NOM	2007	MV700038	SGS		5.9	1972	106	294
NOM	2007	MV700039	SGS		4	5039	76	751
NOM	2007	MV700040	SGS		3.3	7406	65	506
NOM	2007	MV700041	SGS		15.1	25013	121	2317
NOM	2009	NOM-STD-010	ACME LABS	0.019	1.3	2055	102	93
NOM	2009	NOM-STD-020	ACME LABS	0.066	2.4	5177	111	120
NOM	2009	NOM-STD-030	ACME LABS	0.079	5.8	9986	180	192
HB	2012	OREAS-501	ORE RESEARCH & EXPLORATION	0.204	0.84	2710	59.2	
HB	2012	OREAS-503	ORE RESEARCH & EXPLORATION	0.687	1.63	5660	390	
HB	2012	OREAS-504	ORE RESEARCH & EXPLORATION	1.48	3.13	11370	643	
HB	2017	HBSG-01	CHAPI			4560	1100	
HB	2017	BLANKS	ESDEL	0.02	0.3	50	20	50

11.10.2 NORSEMONT

The Quality Control (QC) protocol during the Constancia exploration campaigns from 2006 to 2010 included the insertion of blanks, duplicates and standards using an insertion rate of one of each every 20 samples.

The field duplicates, coarse duplicates, coarse blanks and Certified Reference Materials (CRMs) were inserted on the drill site prior to submission to the laboratory. Furthermore, Norsemont used the additional controls listed below:

- Pulp duplicates were routinely analyzed by the laboratory to monitor the precision of the laboratory. They were not submitted as “blind” to the laboratory, so therefore their value is diminished. These were inserted every 20 samples.
- Coarse duplicates (or preparation duplicates) were taken after crushing to provide information in regards to the sub-sampling variance. They were not submitted to the same laboratory so therefore, their value is diminished. These duplicates were submitted in a proportion of one in every 20 samples.
- Check samples (pulp rejects) were submitted to an umpire laboratory (ACME and Inspectorate). These samples were used to estimate the accuracy of the assay results, along with the standards. These pulps duplicates were submitted in a proportion of one in every 20 samples.

Table 11-3 Quality Control by Drilling Campaigns

Company	Mitsui and Minera Katanga	Rio Tinto	Norsemont	Norsemont	Norsemont	Norsemont
Year	1970-1980	2003-2004	2005	2006-2007	2008-2009	2010
Sample type	Core	Core	Core	Core	Core	Core
Number of holes	48	24	41	143	192	86
Number of samples	*unknown*	3,413	5,460	27,650	33,701	*11,430*
Total meters drilled	5430m	7,483.9m	9,799.05m	49,465.5m	59,625,95m	24,536.9m
Collar survey	unknown	Hand GPS survey at the time of drilling, checked by outside contractor in 2006. Not all holes were found in 2006 by Norsemont staff	T&S	T&S	Geosurvey	Geosurvey
Down-hole survey azimuth	*unknown*	*unknown*	GEOTEC	GEOTEC	Bornav-Maxibor	Bornav-Maxibor
Down-hole survey dip	*unknown*	Eastman single shot at the time of the drilling. Not all holes were surveyed	GEOTEC	GEOTEC	Bornav-Maxibor	Bornav-Maxibor
Assay Lab	*unknown*	ALS Chemex Lima	ALS Chemex Lima	ALS Chemex Lima	ALS Chemex Lima - SGS Lima	SGS Lima
QA/QC program	*unknown*	5% of all samples submitted for assaying	5% of all samples submitted for assaying	5% of all samples submitted for assaying	5% of all samples submitted for assaying	5% of all samples submitted for assaying
Supervisor QA/QC	*unknown*	*unknown*	*unknown*	*unknown*	Gaston Loyola	Gaston Loyola
Core recovery	*unknown*	97.3%	*unknown*	*unknown*	98%	98%
Blanks	*unknown*	1.5 % of the all samples submitted for assaying	4.1% of all samples submitted for assaying	3.9% of all samples submitted for assaying	2.7% of all samples submitted for assaying	2.8% of all samples submitted for assaying
Duplicates	*unknown*	*unknown*	4.1% of all samples submitted for assaying	3.6% of all samples submitted for assaying	2.7% of all samples submitted for assaying	2.5% of all samples submitted for assaying
Standards	*unknown*	3.5 % of all the samples submitted for assaying	2.4% of all samples submitted for assaying	3.2% of all samples submitted for assaying	2.7% of all samples submitted for assaying	2.6% of all samples submitted for assaying
Total QA/QC	*unknown*	189 samples	575 samples	2,944 samples	2,726 samples	903 samples
Umpire lab	*unknown*	*unknown*	*unknown*	*unknown*	*unknown*	*unknown*
Used in resource model	No	Yes	Yes	Yes	Yes	Yes
Comments	Data excluded from the resource estimate	Data for these holes are recorded in the Norsemont database. Drill cores were not relogged by Norsemont.				

11.10.3 HUDBAY

CONSTANCIA

Blanks

Blank material consisting of barren quartz certified by INS-1078 (Inspectorate) was introduced into the sampling stream (one every 30 samples) to monitor cross-contamination and samples swaps.

Standards

Two standard reference materials from Ore Research & Exploration (Table 11-4) were introduced into the sampling stream (one every 30 samples) to monitor the accuracy of the assay results.

Table 11-4 Ore Research & Exploration Standard Reference Materials

Reference material	Au ppm	Ag ppm	Cu ppm	Mo ppm	Zn ppm
OREAS-501	0.204	0.84	2710	59.2	-
OREAS-503	0.687	1.63	5660	390	-

Duplicates

Duplicate samples consisting of 29 samples were introduced into the sampling stream (one every 30 samples) to monitor the precision of the assay results and to assess the homogeneity of the mineralization.

Check Assay

No check assays (i.e. pulp reruns) were analyzed through an umpire lab.

PAMPACANCHA

Blanks

Blank material consisting of barren quartz certified by three laboratories (INSPEC, CIMM, and SGS) was introduced into the sampling stream (one every 30 samples) to monitor cross contamination and samples swaps.

Standards

Three matrix match standard references consisting of rock from the Pampacancha deposit and certified by Acme labs, were introduced into the sampling stream (one every 30 samples) to monitor the accuracy of the assay results (Table 11-5).

Table 11-5 Matrix Match Standard Reference Materials

Reference material	Au ppm	Ag ppm	Cu ppm	Mo ppm	Zn ppm
MV700041		15.1	25013	121	2317
NOM-STD-030	0.079	5.8	9986	180	192
OREAS-501	0.204	0.84	2710	59.2	
OREAS-503	0.687	1.63	5660	390	
OREAS-504	1.480	3.13	11370	643	

Duplicates

Duplicate samples consisting of quartered splits were introduced into the sampling stream (one every 30 samples) to monitor the precision of the assay results and to assess the homogeneity of the mineralization.

11.11 SITE VISITS

Hudbay geologists have visited the Constancia and Pampacancha deposits to conduct site inspections to become familiar with conditions on the property, to observe the geology and mineralization, to perform core review and to verify the work completed on the property as part of the mineral resource estimation and technical report process since 2012.

12 DATA VALIDATION

Data verification was conducted under the supervision of Cashel Meagher (Qualified Person pursuant to NI 43-101) and it is the opinion of the author that the quality of the data is suitable for use in resource and reserve calculations and that sampling to date is representative of the deposit. Data validation protocols and results already documented in details in previous Technical Reports issued by Hudbay for Constancia are summarised in this section together with details on new activities conducted in 2017.

12.1 CONSTANCIA

12.1.1 NORSEMONT QUALITY CONTROL PROTOCOL AND RESULTS

The Quality Control (QC) protocol during the Constancia exploration campaigns from 2005 to 2011 included the following insertions prior to dispatching samples to the laboratories:

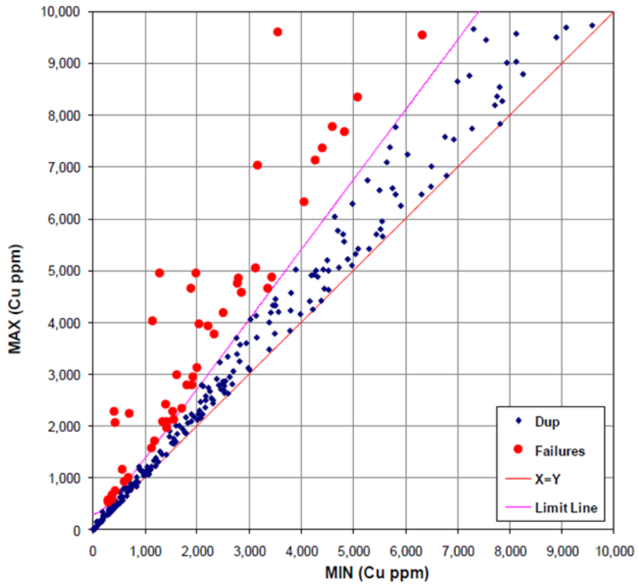
- Duplicate coarse and pulp samples: one in 20 samples;
- CRMs: one in 20 samples; four CRMs were inserted in alternate order; and
- Blanks: one in 20 samples.

Duplicate Samples

1,952 duplicate samples obtained by splitting half core samples, were inserted in the samples stream between 2005 and 2011. The sampling procedure is considered to be acceptable if the proportion of failures using the hyperbolic method does not exceed 10% using as a failure limit, the $y^2=m^2x^2+b^2$ hyperbola, evaluated for a 30% relative error[1] (RE). The failure rates for Cu, Mo, Zn and Ag ranged between 7% and 16%.

Figure 12-1 represents the duplicate analysis scatter plot. This figure shows that several outliers are evidently occuring from sample mix-up which when excluded provide an acceptable rate of failure. Norsemont also inserted a total of 51 field duplicates from RC holes. The failure rates for Cu, Mo, Zn and Ag ranged between 0% and 8% which indicates that the sampling precision was reasonable.

Figure 12-1 Cu in Duplicate Samples



[1] Relative error: calculated as the absolute value of the pair difference divided by the pair average, and expressed in percentage; also known as Absolute Value of the Relative Difference, or AVRD.

CRMs

Certified Reference Materials (CRMs) were prepared by SGS and ACME Laboratories from coarse rejects from previous drilling campaigns at Constancia. The CRM dataset includes 1,807 assays from 18 different CRMs.

The analytical bias was calculates as follow:

$$\text{Bias (\%)} = (AV_{eo} / BV) - 1$$

Where AV_{eo} represents the average recalculated after the exclusion of the outliers (AV±3*SD), and BV is the Best Value calculated as a result of round-robin tests.

For eleven out of twelve CRMs, biases were between -3.9% and 4.5% with low proportions of outliers. The twelveth CRM shows high biases of 9% and 6% respectively for Cu and Mo.

The author concludes that the Ag accuracy at the ALS Chemex during the 2005 to 2011 campaigns were very good.

Blanks

A total of 2,119 coarse blanks, made from barren material, were inserted in the samples streams. Possible contamination issues were investigated if the blank value exceeded five times the detection limit for the element, and/or if a definite, positive rapport was observed between the blank grade and the grade of the preceding sample.

The sample preparation process at ALS Chemex produced no significant Cu, Mo, Zn, and Ag contamination in most of the blanks evaluated. Only one blank showed failure rates which exceeded 10% for Cu and Mo. However, it is deemed that the failure rate was due to an inadequate sample blank preparation.

Check Samples

388 samples submitted for external control to ACME in 2007, which acted as a secondary laboratory for the 2006 and part of the 2007 campaigns were analyzed.

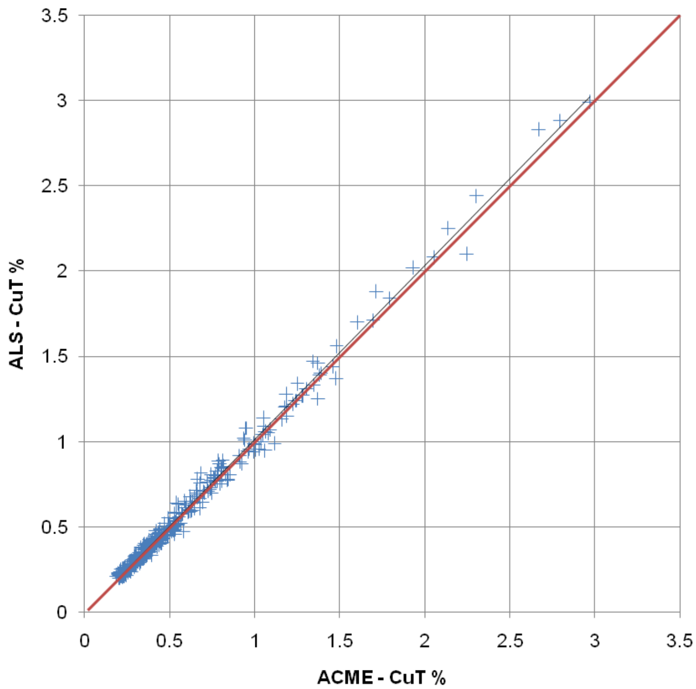
Reduced-to-Major-Axis (RMA) plots we used to assess the check samples. The RMA method offers an unbiased fit for two sets of pair values (original samples and checks sample) that are considered independent from each other. In this case, the coefficient of determination R2 between the two laboratories is determined, and the bias of the primary laboratory for each element as compared to the secondary laboratory is calculated as:

$$\text{Bias (\%)} = 1 - \text{RMAS}$$

Where RMAS is the slope of the RMA regression line of the secondary laboratory values versus the primary laboratory values for each element.

The RMA plot indicates good fits for Cu (0.988 R2), Mo (0.974 R2), Zn (0.994 R2) and Ag (0.92 R2) after the exclusion of four outliers for Mo (1.0%), two outliers for Zn (0.5%), and eight outliers for Ag (3.8%). Figure 12-2 represents the results.

Figure 12-2 Cu in Check Samples – ACME vs ALS



12.1.3 HUDBAY QUALITY CONTROL PROTOCOL AND RESULTS

The Quality Control (QC) protocol during the 2014-2015 campaign included the insertion of the following control samples in the sample batches:

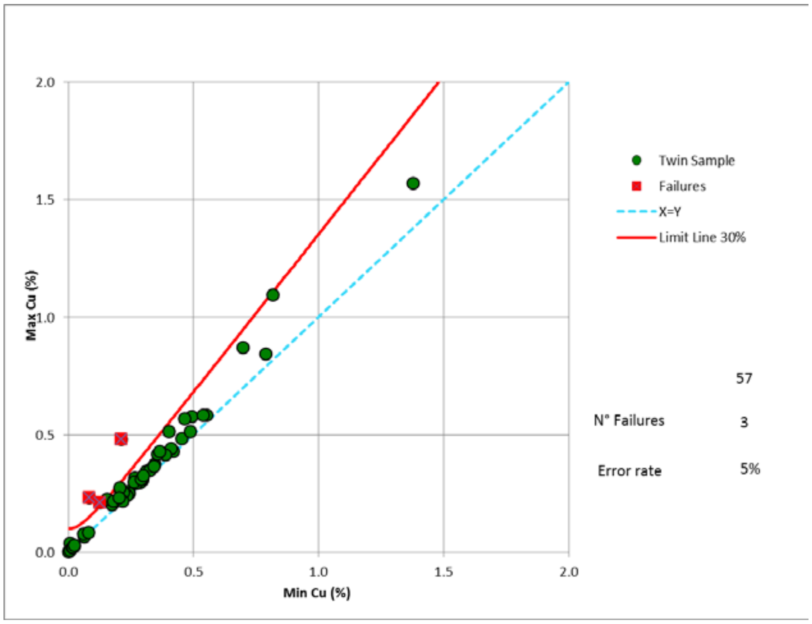
- Duplicate samples (quarter core): one in 20 samples;
- CRMs: one in 20 samples, where four CRMs were inserted in alternate order; and
- Blanks: one in 20 samples.

The duplicate samples, coarse duplicates, coarse blanks and Certified Reference Materials (CRMs) were inserted at site prior to submission to the laboratory.

Duplicates Samples

Hudbay inserted 57 duplicate samples during the drill campaign in 2015. This data was processed using the hyperbolic method. Figure 12-3 represents the precision evaluation for copper and confirms the absence of bias or any significant issue with sample swap.

Figure 12-3 Precision Evaluation CuT – Duplicates Samples



CRMs

Hudbay used Certified Reference Materials (CRMs) prepared by Target Rocks (6 laboratories, round robin) using material coarse reject from the mine.

To evaluate the accuracy of assaying using CRMs, Hudbay excluded the outliers (out from the mean +/- 2 times the standard deviation calculated from the actual assay values of the inserted standards), and calculates the analytical bias.

These standards were prepared using porphyry material. All elements present a bias (Bias (%) = (Mean / Best Value)-1) lower than 10%. Only one standard displayed a higher bias for %Mo.

Blanks

A total of 63 samples, prepared using barren material from the mine, were regularly inserted in the submission batches. Their analysis indicated that the percentage of failures in was acceptable for all the elements.

12.1.4 2017 DRILL PROGRAM

A total of 2,132 core samples were sent for assaying at Certmin Lima and SGS Lima laboratories. Blanks and standards had an insertion rate of 25% and 6% respectively. Samples preparation, AAS and metallurgical test work were performed at ICP-ES done as an additional check on the analytical method at SGS. The coarse and pulp duplicates, the blanks and standards analyzed by Certimin were identified but were "blind" for SGS and for the umpire laboratory (Bureau Veritas Lima).

Certimin

Duplicate samples results have shown that all elements were inside a 20% accepted relative error, implying that the results from the laboratory are precise. Standards presented very low failure rates (mean+ 2SD) for all elements including copper (2.7%), implying that the results from the laboratory are accurate. Performance of the blanks using 10 times the detection limit as performance gates also demonstrated that there were no cross contamination or samples swaps issues at the laboratory.

SGS

All elements were within the performance gates of the standards (mean + 2SD) with low failure rates for copper (<3%), implying that the results from the laboratory are accurate. The performance of the blanks using 10 times the detection limit as a threshold were also excellent implying no cross contamination or samples swaps at the laboratory.

Bureau Veritas Umpire Lab

Bureau Veritas was used as an umpire lab in order to ensure that the results from Certimin and SGS were precise and accurate. 10% of the total samples analyzed by Certimin and SGS were randomly selected. Half of the samples were re-dispatched to their original labs and the other half was dispatched to Bureau Veritas for ICP-ES. A total of 30% quality control samples (coarse and pulp duplicates, blanks and standards) were inserted into the samples stream.

Duplicate samples results have shown that all elements were inside of 20% accepted relative error. All elements were within the performance gates of the standards (mean + 2SD) with low failure rates for copper (<3%). The performance of the blanks using 10 times the detection limit also did not show any issue with respect to sample swaps or cross-contamination.

12.1.5 CORE RECOVERY

Prior to the 2017 DDH campaign, core recovery was measured via a linear method and the average core recovery was 95%. Further investigations and random checks of the core box pictures revealed that a significant proportion of these core samples had a volumetric recovery inferior to 85%. During the 2017 twin-hole program, core recovery was measured and compared via the volumetric method. For the 17 pairs of holes, the proportion of 2m composites with a volumetric core recovery higher than 85% increased from less than 50% to more than 75%.

12.2 PAMPACANCHA

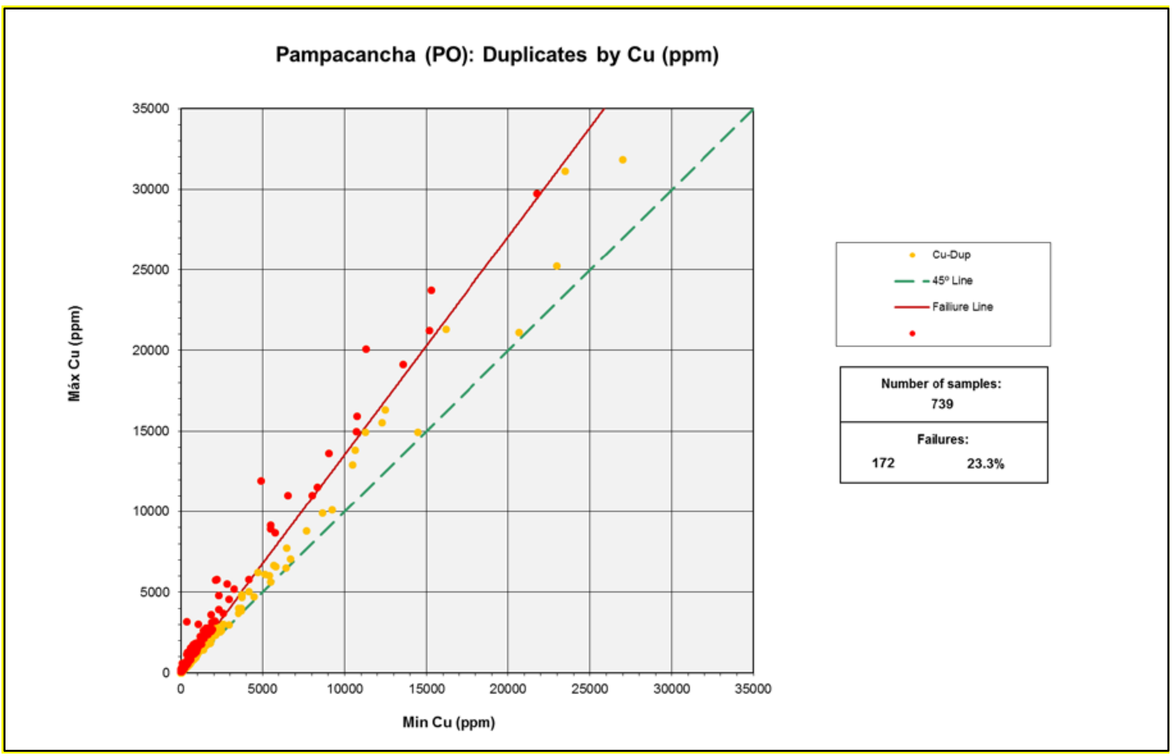
12.2.1 QUALITY ASSURANCE / QUALITY CONTROL

Hudbay submitted a total of 15,932 samples from 110 drill holes to the SGS's laboratory for analysis. In addition to these samples, 739 duplicate samples, 471 coarse blanks, 336 reference standards were also submitted. The author's opinion is that the results of the QAQC program at Pampacancha validate the use of the assaying information for resource modeling.

Duplicate Samples

The rate of failure observed on Figure 12-4 Precision Cu (ppm) – Duplicate Samples is deemed to be related to the fact that quartered core samples were used as duplicates. Given the high variability of the mineralization, it is recommended that future duplicates from core samples be taken from crushed, homogenized and split samples, rather than using quartered split core.

Figure 12-4 Precision Cu (ppm) – Duplicate Samples



Standards

A total of 548 copper standards were submitted as a mean to monitor the accuracy of the laboratory results. Out of these, 113 of the assays fell outside the average +/- two standard deviations. However the bias in all cases is acceptable: 36 failures from 548 in copper (7% of failures), 31 from 548 in silver (6% of failures), 17 from 548 in gold (3%), 18 from 548 in molybdenum (3% of failures) and 11 from 308 in zinc (2% of failures).

Blanks

A total of 720 barren samples prepared from mined rocks were inserted in the submission batches in order to monitor cross-contamination or samples swaps. Results are acceptable for all blanks except for Molybdenum. Failures in Molybdenum indicate a high level of contamination. While this is a concern, it is not considered a material impact to the resultant resource statement.

Drilling Database

An internal validation of the drillhole database against the original drill logs and assay certificate information was carried out by Hudbay. The validation included 100% of the assay values from the Pampacancha drilling. No significant discrepancies existed within the database and it is believed to be accurate and suitable for mineral resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Constancia mill is now an operating plant running at steady state and as a result, several of the initial metallurgical testing and assumptions have been revised to reflect the operating experience and performance of the plant over the past 2.5 years of operation in processing the ore produced from the Constancia pit.

This section will first briefly describe some of the optimization tests that have been conducted while processing the Constancia ore types to adjust their milling recovery formulae followed by a recap in more details of the metallurgical test work done during the pre-feasibility study for the Pampacancha deposit that is planned to start feeding the mill in 2019Q1.

Table 13-1, summarizes the annual average copper recoveries estimated for the life of the mine between 2018 and 2036 for the combined feed from the two deposits.

Table 13-1 Copper Recovery

Year	Cu,%	Cu Recovery, %
2018	0.416	82.56%
2019	0.413	84.61%
2020	0.394	85.91%
2021	0.391	86.00%
2022	0.391	86.11%
2023	0.386	85.73%
2024	0.314	86.44%
2025	0.326	88.19%
2026	0.264	86.46%
2027	0.253	86.14%
2028	0.280	87.47%
2029	0.296	86.08%
2030	0.296	87.34%
2031	0.254	86.78%
2032	0.247	86.61%
2033	0.234	86.49%
2034	0.273	86.25%
2035	0.292	88.32%
2036	0.186	83.77%

Additional testwork was conducted in 2017 to confirm the copper recovery formulation assuming no changes to the plant flowsheet. This work is summarized in section 13.1. While the feasibility testing remains relevant, the onsite operating team accountable to mineral recovery at Constancia has implemented its own process improvement program to address forecasting of recovery. The forecasted recoveries utilized in this technical report were calculated based on current performance and knowledge with specific improvements to be conducted on minor modifications to the circuit. This practice of business improvement will continue and validate the future implemented improvement initiatives.

In Table 13-1 post-2019, the copper recovery increases based on the utilization of the Proflote technology for the recovery of particles finer than 25 microns and the use of Eriez cells to recover particles larger than 150 microns.

13.1 CONSTANCIA PLANT OPTIMIZATION AND UPDATED RECOVERY FORMULATION

The Constancia ore types currently being processed in the Plant were established during the Feasibility Study and include: hypogene, supergene, skarn, mixed and high zinc. Their metallurgical response in terms of treatment, recovery and molybdenum and copper concentrate grades were deemed satisfactorily during all historical tests. Copper grade in the final concentrate are higher than 24%, with low levels of zinc, lead and iron while the molybdenum concentrate produced grades over 50% Mo with low contents of copper, lead and iron.

For the commissioning of the Plant, the design parametres, size of feed P80, dosing and type of reagent, % of solids, PH, etc. were taken into account. Simultaneously, several metallurgical tests were performed to check and optimize the treatment of the different types of ore fed to the Plant.

First, tests were performed using different reagents (collectors) in order to increase recoveries of Copper, Silver and Gold; the F500x (Oxiquim) was selected as primary collector and D-101 (Mathiesen) as secondary collector for different types of ore processed at the Plant.

Testing was also performed with different frothers, chosing the RE-100 from the rest.

Different types of flocculants were also tested for the tailings thickener: Orifloc-5310, 5320 and A-130. The test work provideded good results with the Orifloc 5320 replacing the Orifloc 3118 selected in the original design. Using Orifloc 5320, solids discharge increased in the tailings thickener to 58%, while flocculant usage was also reduced from 30g/t to 15g/t.

As a result of the changes mentioned above, the plant has reached steady operating performance with the CuOx/CuT and Zn/CuT ratios not exceeding 10%. A close coordination between the process plant team and the geology department has been implemented to ensure that the ore fed to the Plant is within these values.

Through the test work it was also established that the treatment for each different type of ore should be done as independently as possible. For example, soluble copper from supergene ores activates pyrite within a hypogene-supergene mix having a detrimental effect on the concentrate grade.

13.1.1 FLOTATION ASSAYS BY ORE TYPE

Flotation tests were conducted by ore type in order to better predict the metallurgical behavior and to make the necessary adjustments regarding reagents dosing and other control variables such as: P80, solids percentage and pH.

Constancia Pit Mineralogy and Copper Recovery

The Constancia deposit is a Cu-Mo-Ag porphyry with some of the copper occurring as mineralized skarn. For metallurgical modeling purposes, three zones were identified based on their dominat ore types:

- Hypogene zone with primary mineralization of porphyry type. It represents the largest portion of the deposit, including disseminated, quartz veins in stock work and fractures containing chalcopyrite-molybdenite that run below the 3,900 masl. Molybdenum is present as molybdenite and is mostly un-oxidized.
- Mineralized skarn zone, smaller in volume, but with higher grades of chalcopyrite, hypogene, rare bornite, galena and sphalerite with mineralization occurring close to the surface. The skarn contains lower levels of molybdenum. The zinc as sphalerite concentrates higher in skarn and in the hypogene adjacent to skarn. Lead is present as galena in close relation to sphalerite.
- Supergene enrichment zone which occurs lower, made up by secondary ores of digenite copper, covellite, chalcocite (rare native copper) deposited under a leached layer.

Based on the results of these tests, the following recovery formulae were developed for each zone:

Hypogene

$$\text{Recovery Cu T } (\%) =$$

$$83.24 + 11.75 * (\text{Cu T Feed fresh}) - 43.59 * (\text{Zn Feed fresh}) - 0.61 * (\text{Fe Feed fresh}) - 0.59 *$$

$$(\text{CuSS Feed fresh})$$

Where: CuSS, is copper soluble in acid.

$$\text{Cu grade } (\%) = 22.84 + 9.40 * (\text{Cu T Feed fresh}) - 17.48 * (\text{Zn Feed fresh}) - 0.30 * (\text{Fe Feed fresh})$$

Restrictions:

$$0.28 \leq \text{Cu T}(\%) \leq 0.88\,;\, 0.003 \leq \text{CuSS} \leq 0.09;\, 0.02 \leq \text{Zn } (\%) \leq 0.14;\, 2.17 \leq \text{Fe} \leq 7.39$$

In the recovery of hypogene type the ratio of Zn/ Cu T is critical and can, negatively influence the recovery.

Supergene

$$\text{Recovery Cu T }(\%) =$$
$$78.64 + 15.18 * (\text{Cu T Feed fresh}) - 18.65 * (\text{Zn Feed fresh}) - 8.07 * (\text{CuSS Feed fresh})$$

Where: CuSS, is copper soluble in acid.

$$\text{Cu grade }(\%) = 24.35 + 5.05 * (\text{Cu T Feed fresh}) - 8.45 * (\text{Zn Feed fresh}) - 0.42 * (\text{Fe Feed fresh})$$

Restrictions:

$$0.33 \leq \text{Cu T}(\%) \leq 1.08 \,;\, 0.035 \leq \text{Cu SS }(\%) \leq 0.20 \,;\, 0.01 \leq \text{Zn }(\%) \leq 0.14 ; 2.06 \leq \text{Fe} \leq 7.14$$

The recovery of supergene type ore is subject to the percentage of acid soluble copper and zinc in ore, both will negatively influence the recovery.

Skarn

$$\text{Recovery Cu T }(\%) =$$
$$79.45 + 14.29 * (\text{Cu T Feed fresh}) - 17.56 * (\text{Zn Feed fresh}) - 0.19 * (\text{Fe Feed fresh}) - 70.68 * (\text{CuSS Feed fresh})$$

Where: CuSS, is copper soluble in acid.

$$\text{Cu grade }(\%) = 23.06 + 4.60 * (\text{Cu T Feed fresh}) - 5.64 * (\text{Zn Feed fresh}) - 0.17 * (\text{Fe Feed fresh})$$

Restrictions:

$$0.36 \leq \text{Cu T}(\%) \leq 1.12 \,;\, 0.02 \leq \text{Cu SS }(\%) \leq 0.06 \,;\, 0.04 \leq \text{Zn }(\%) \leq 0.42 ; 2.92 \leq \text{Fe }(\%) \leq 12.34$$

The recovery of skarn type ore is subject to the percentage of iron in ore, which will negatively influence the recovery.

13.2 PAMPACANCHA PFS TESTWORK

Pampacancha is a resource that was delineated during the Constancia feasibility study phase but with the metallurgical test work at a pre-feasibility study stage of engineering. It was planned that the Constancia plant, process and reagents would be used to treat the Pampacancha ore. However, the mineralization and rock types at Pampacancha are different and consequently, it was necessary to develop a different regime for this deposit.

13.2.1 PHASE 1 TESTING

Preliminary metallurgical tests on the Pampacancha samples were performed by SGS in 2012. The objective of this Phase 1 testwork was to provide comminution and flotation design information for the Pampacancha resource. Phase 1 included mineralogical analysis through SGS Chile and comminution and flotation tests at SGS Peru. Phase 1 involved head assay, specific gravity, natural pH determination, preliminary rougher flotation reagent screening and pH on flotation performance.

Locked cycle tests were also conducted using the standard Constancia regime with some optimization tests.

The core samples selected for the Pampacancha metallurgical testwork were fresh quarter core samples collected from diamond drill holes drilled from 2008 to 2011. Samples with skarn and hypogene lithologies were considered representative of the Pampacancha deposit. There was some concern about the age of the samples as oxidation of the samples may impact flotation results. Three master composites for comminution testing and two additional master composites for flotation testing were produced.

Comminution

Preliminary comminution test, SAG Power Index (SPI), Abrasion Index (Ai) and Bond Ball Work Index (BWi), were conducted by SGS Peru, results are shown in
Table 13-2.

Table 13-2 Phase 1: Pampacancha Comminution Results

Comminution Parametres	Unit	Value
SAG Power Index	Min	59.3 - 93.6
Bond Abrasion Index	g	0.0772 – 0.1160
Bond Ball Work Index	kW-h/t	12.1 – 13.0

Locked Cycle Test

Revised conditions for two composites at a primary grind of P80 of 106 µm, used a collector dosage of 30 g/t A-3302 and 10 g/t A-404 at a pH of 9.5 for a rougher flotation time of 10 min, a regrind P80 of 25 µm and three stages of cleaning at a pH of 11. Locked cycle test results are shown in

Table 13-3. Concentrate grade from 19.6 to 21.6% Cu with a recovery from 78.6 to 79.2% Cu were achieved.

Table 13-3 Phase 1: Pampacancha Locked Cycle Test Results – Revised Conditions

Composite	Grade									
	Cu	Fe	Au	Ag	S	Cu	Fe	Au	Ag	S
	%	%	g/t	g/t	%	%	%	%	%	%
PMC01	20	25	10	174	23	79	2.4	61	62	56
PMC02	22	24	18	207	25	79	2.9	73	68	61

Using a single collector dosage of 30 g/t A-3302 at a pH of 10 with rougher flotation time of 10 min, a regrind P80 of 25 µm and three stages of cleaning at a pH of 11.5 improved results (Table 13-4).

Table 13-4 Phase 1: Pampacancha Locked Cycle Test Results – Constancia Project Conditions

Composite	Grade					% Recovery				
	Cu	Fe	Au	Ag	S	Cu	Fe	Au	Ag	S
PMC01	27	27	13	223	26	78	2	60	62	47
PMC02	22	22	22	203	23	80	2.6	77	63	53

Flotation Variability Test

Using the optimal rougher conditions, fifteen samples were floated to determine the effect of the variability on rougher flotation. The head grades ranged from 0.36 to 1.89% Cu. The variability flotation test was a kinetic flotation for 0, 1, 2, 4, 8, 12 and 18 min. The results of the variability test are shown in (

Table 13-5). Rougher concentrate recoveries were in the range from 81.2 to 93.5% Cu, 7.85 to 21.1% Fe, 75.8 to 93% Au and 65.9 to 88.4% Ag.

Table 13-5 Phase 1: Rougher Variability Testing Results

ID Sample	Head Calculated				Rougher Grade				Rougher Recovery				Mass Pull%
	Cu %	Au g/t	Ag g/t	Fe %	Cu %	Au g/t	Ag g/t	Fe %	Cu %	Au g/t	Ag g/t	Fe %	
PMS-01	1.19	0.53	4.4	35.7	5.91	2.71	19.7	26.8	84.7	86.7	76.7	12.8	17.1
PMS-03	0.68	0.25	3.92	11.3	2.96	1.13	14.1	11.8	88.2	90.2	72.7	21.1	20.1
PMS-04	0.88	0.33	8.91	33.7	3.71	1.37	34.3	25.6	91.1	88.3	83.2	16.4	21.6
PMS-05	0.49	0.42	8.2	12.8	2.13	1.73	33	11.2	90.1	84.8	83.5	18.2	20.7
PMS-06	0.72	1.03	7.93	35.6	4.07	6.03	40.7	26.3	89.8	93	81.3	11.7	15.8
PMS-07	1.05	0.22	6.23	39	4.44	0.83	22	29.5	88	78.1	73.7	15.7	20.8
PMS-08	1.89	0.89	6.2	41.4	9.38	4.05	23.9	30.5	90.9	83.2	70.7	13.5	18.3
PMS-09	0.93	0.34	2.65	48.6	7.18	2.42	15	32.9	89.5	81.7	65.9	7.85	11.6
PMS-10	0.93	0.3	13.1	36.1	5.56	1.49	70.5	28.1	92.4	76.5	83.4	12.1	15.5
PMS-11	0.36	0.26	2.98	12.2	1.58	1.1	10.9	10.6	81.2	77.5	68.3	16.3	18.7
PMS-13	0.83	1.1	21.2	21.8	3.42	4.71	83	19.4	89.6	92.8	85.2	19.4	21.7
PMS-15	0.62	0.57	9.26	17.4	2.48	2.29	35.3	15.5	86	86.2	81.6	19.1	21.4
PMS-16	0.84	0.42	4.36	32	4.64	2.19	20.5	20.8	93.5	88.2	80	11	17
PMS-17	0.92	0.39	6.47	24	3.63	1.35	20.5	21.6	87.3	75.8	70	19.9	22.1
PMS-18	0.72	0.46	5.29	17.9	3.37	1.98	23.8	15.4	91.2	84.6	88.4	16.9	19.6
AVERAGE	**0.87**	**0.5**	**7.4**	**28**	**4.3**	**2.36**	**31.2**	**21.7**	**88.9**	**84.5**	**77.6**	**15.5**	**18.8**

Mineralogy

A PMA QemSCAN, analysis was conducted on the Locked Cycle Test (LCT) head stream for the two composites to determine the mineral assemblage. The copper occurrence in both cases was dominated by chalcopyrite (70-73%), bornite (22-28%) with minor chalcocite (1%) and enargite - tennantite (2%). Table 13-6 reports the QemSCAN modal analysis results.

Table 13-6 Phase 1: Mineralogical analysis (Mineral Mass %)

Minerals	PMC-01	PMC-02
Chalcopyrite	11.2	10.29
Chalcocite/Covellite	0.23	0.48
Bornite	2.81	1.98
Enargite/Tennantite	0.11	0.29
Pyrite	4.66	3.64
Pyrrhotite	0.16	0.08
Molybdenite	0.03	0.69
Sphalerite	1.23	1.41
Galena	0.35	0.45
Quartz	9.75	13.32
Plagioclase	0.73	1
K-Feldspar	3.39	3.91
Biotite	4.61	2.99
Amphibole	3.11	4.24
Pyroxene	1.39	1.26
Tourmaline	0.88	1.15
Other Silicates	0.09	0.08
Muscovite/Sericite	0.86	1.32
Chlorite	10.31	5.04
Albite	0.06	0.23

Minerals	PMC-01	PMC-02
Epidote	0.16	0.19
Clays	0.26	0.59
Calcite	15.75	16.24
Ankerite	6.64	10.59
Anhydrite/Gypsum	1.74	1.51
Rutile	0.13	0.21
Ilmenite	0.01	0.02
Fe Oxides	18.88	15.2
Sphene	0.07	0.07
Siderite	0.08	1.19
Apatite	0.23	0.21
Others	0.11	0.13
Total	100	100

13.2.2 PHASE 2

During 2012, the SGS Laboratory in Lima, Peru conducted a test program using drill core rejects from the 2011 program. The main objective of Phase 2 was to obtain the metallurgical response of mineralized material from the Pampacancha deposit to the Constancia mill flotation conditions. Phase 2 involved head assay, specific gravity, natural pH determination, a locked cycle test to approximate plant metallurgical results and variability flotation conditions. Samples from the skarn and hypogene lithologies were considered representative for the Pampacancha deposit. Three master composites were produced and designated for each sector of the Pampacancha deposit.

Locked Cycle Test

The Constancia project flotation conditions for the locked cycle tests are shown in Table 13-7.

Table 13-7 Pampacancha Locked Cycle Test Conditions

Stage	Reagents, g/t			Grind	Time, min	Time, min	
	CaO	A-3302	AF-65	P80, µm	Cond.	Flot.	pH
Grinding	Req	15		106			
Conditioning	Req	15	Req				10
Rougher						10	
Regrinding				25	2		
Cleaner 1	Req					4	11.5
Conditioning							
Scavenger						8	11.5
Cleaner 2	Req					3	11.5
Cleaner 3	Req					3	11.5

Locked cycle test results in

Table 13-8 show that a concentrate grade from to 14.9 to 25.3% Cu with a recovery from 67.8 to 80.6% Cu was achieved.

Table 13-8 Pampacancha Phase 2: Locked Cycle Test Results

Composite	Grade				Recovery			
	Cu, %	Fe, %	Au, g/t	Ag, g/t	Cu, %	Fe, %	Au, %	Ag, %
PMC-03	21.1	22.4	9.7	188.5	76.3	2.2	62.2	63.1
PMC-04	25.3	24.7	11.3	151.1	67.8	1.7	53.8	56.5
PMC-05	14.9	21.1	6.7	165.8	80.6	2.4	61.3	70.3

Flotation Variability Test

A variability test was performed on 30 skarn and hypogene samples (Table 13-9). The sample head grades were in the range from 0.21 to 1.29% Cu. The variability flotation test was a kinetic flotation for 0, 1, 2, 4, 8, 12 and 18 min. The results of the variability test including, cumulative recovery and grade until flotation completion (18 min), are shown in Table 13-9. Rougher concentrate recoveries obtained were in the range of 75.3 to 95.7% Cu, 8.6 to 36.5% Fe, 54.7 to 91.8% Au and 54.1 to 95.1% Ag.

Table 13-9 Phase 2: Rougher Variability Testing Results

Samples	Head Calculated				Rougher Grade				Rougher Recovery				Mass %
	Cu %	Fe %	Au g/t	Ag g/t	Cu %	Fe %	Au g/t	Ag g/t	Cu %	Fe %	Au %	Ag %	
PRS-01	0.31	19.9	0.13	0.76	1.37	12.2	0.55	2.54	90	12.5	89.8	68.3	20.3
PRS-02	1.26	2.6	0.55	6.81	5.47	26.7	2.27	28.3	90	13	85.1	86.3	20.8
PRS-03	0.43	32.1	0.14	6.58	1.78	22	0.55	23	89.9	14.8	82.9	75.3	21.5
PRS-04	1.3	33.3	0.62	6.8	4.45	28.4	1.63	19.9	90.3	22.4	69	77.1	26.3
PRS-07	0.31	28.1	0.13	1.85	1.03	20.2	0.43	5.67	86.9	18.5	84.2	79.1	25.8
PRS-11	0.22	3.76	0.11	2.79	0.55	3.96	0.23	6.18	87.6	36.5	74.3	76.6	34.7
PRS-12	0.6	26.9	0.26	3.73	1.92	20.2	0.81	11.2	85.2	19.8	82.1	79.1	26.4
PRS-15	0.38	13.4	0.12	1.51	1.38	11.5	0.43	5.3	86.6	20.7	88.2	84.4	24
PRS-16	0.46	30.2	0.17	3.36	2.25	21.3	0.84	15.4	89	13	91.8	84.7	18.5
PRS-18	0.39	24.8	0.34	10.5	1.2	20.1	0.9	28.7 72.7	84.2	22	72.7	74.4	27.2
PRS-19	1.24	41.1	0.63	11.2	4.9	32.8	3.12	41.2	85.5	17.3	72.8	79.9	21.7
PRS-20	1	21.1	0.69	29	2.62	18.4	1.59	71.5	88.7	29.3	77.6	83	33.7
PRS-21	0.64	25.2	0.19	5.02	2.05	17.5	0.59	14.7	88.7	19.3	84.1	81.1	27.8
PRS-22	0.35	41.2	0.19	5.23	1.51	29.9	0.09	18.6	89.2	14.8	86.9	72.3	20.3
PRS-23	0.33	14.3	0.26	5.13	0.97	12.1	0.68	13.4	83.8	24.5	74.8	75.5	28.9
PRS-24	0.27	11.5	0.21	5.65	0.73	12.4	0.47	9.24	90.2	35.6	73.6	54.1	33.1
PRS-27	0.37	15.5	0.16	2.38	1.02	12.3	0.4	6.25	84.5	24.7	80.5	81.5	31
PRS-30	0.6	43	0.18	2.9	2.86	29.7	0.78	10.4	90.6	13.1	83.1	68.1	18.9
PRS-31	0.64	18.2	0.17	6.14	1.78	15.4	0.47	15.5	93.2	28.3	89.3	84.6	33.5
PRS-32	1.08	39.6	0.42	7.01	3.82	29.6	1.24	16.1	75.3	16	63.6	68.9	21.4
PRS-34	0.46	13.2	0.6	9.84	1.45	11.3	1.6	29.8	89.7	24.4	75.8	86.5	28.6
PRS-35	0.74	7.55	0.62	23.5	1.77	7.61	2.59	90.7	82	23.4	85.9	79	20.5
PRS-36	0.65	15.1	0.098	14.4	2.17	14	0.31	47.9	95.7	26.6	89.1	95.1	28.7
PRS-37	0.48	28.8	0.31	6.59	2.46	19.2	1.41	32.2	84.9	10.9	75.5	80.4	16.5
PRS-39	0.52	16.7	0.16	13.1	2.19	18.9	0.52	52.3	92.1	24.8	69.9	87.8	22
PRS-40	0.49	31.9	0.13	2.61	2.45	21.2	0.61	10.5	88.3	11.8	84.1	71.6	17.8
PRS-42	0.5	22.2	0.19	11.3	1.68	16.9	0.52	35.7	84.9	19	67.5	78.9	25
PRS-43	0.37	14.5	0.16	6.42	1.26	11.9	0.48	21.6	88.7	21.6	81.5	88.3	26.2
PRS-46	0.8	40.1	0.32	6.33	4.04	24	1.61	31.4	91.8	10.8	90.5	89.6	18.1
PRS-49	0.45	29.3	0.17	6.35	2.75	16.7	0.61	34.4	92.9	8.56	54.7	81.5	15.1

13.2.3 CONCLUSION

The Phase 1 locked cycle test with revised conditions yielded a final concentrate grade at a 95 per cent confidence level of 19.6 ± 2.7% Cu for PMC01 and 21.6 ± 1.8% Cu for PMC02 with a recovery of 78.6 ± 0.5% Cu for PMC01 and 79.2 ± 2.2% Cu for PMC02. Locked cycle test with Constancia mill conditions yielded a final concentrate grade at 95 per cent confidence level of 26.6 ± 0.7% Cu for PMC01 and 21.9 ± 0.8% Cu for PMC02 with a recovery of 77.5 ± 0.9% Cu for PMC01 and 79.9 ± 2.7% Cu for PMC02 at the 95% level of confidence.

Approximately 80% of the Phase 1 copper sulphide (chalcopyrite, bornite, chalcocite, covellite, digenite) were liberated for both composite PMC01 and PMC02. Phase 2 locked cycle test with revised conditions yielded a final concentrate grade at 95% confidence level of 21.1 ± 1.7% Cu for PMC03, 25.3 ± 0.9% Cu for PMC04 and 14.9 ± 4.4% Cu for PMC05 with a recovery of 76.3 ± 0.9% Cu for PMC03, 67.8 ± 2.8% Cu for PMC04 and 80.6 ± 1.9% Cu for PMC05.

13.2.4 DISCUSSION

During the Phase 1 locked cycle testing with revised conditions, the copper grade for composite PMC02 was below industry commercial copper grade. Chemical characterization of the cleaner concentrate reported values that may be close to penalty levels for As (0.26%), Sb (0.19%), and Pb (1.5%). In the locked cycle testing under the Constancia mill conditions, the copper grade for composite PMC02 was also below commercial copper grade. Chemical characterization of the cleaner concentrate reported values that may be close to penalty levels for As (0.25%), Sb (0.19%), and Pb (1.02%). Lime consumption was from 865 to 1,245 g/t.

During the Phase 2 locked cycle test under the Constancia mill conditions, the copper grade for composites PMC03 and PMC05 was below commercial copper grade. The chemical characterization for the cleaner concentrate reported values that might be close to penalty levels for Sb (0.15%) and Zn (3.48%) for PMC03 and Zn (3.33%) for PMC04. Lime consumption was from 1,595 to 2,053 g/t.

The low concentrate grades from the locked cycle test, and the low recoveries in cleaning are accompanied with low sulphur grades in the concentrates. Preliminary mineralogy suggests that this may be due to small copper sulphide inclusions in hard silicates. The QemSCAN mineralogy showed that over 90% of the copper sulphides were moderately liberated or better, which was also evident in the rougher flotation variability testwork. However, 10 to 16% of the copper sulphides were associated with silicates which explains the difficulty in getting adequate cleaning recovery and final concentrate grade in the locked cycle testwork.

13.2.5 RECOMMENDATIONS

In Phase 1, average rougher concentrate recoveries of 88.9% Cu, and 77.6% Ag were obtained during the variability tests. Therefore, recoveries of 85% Cu and 65% Ag were used for pre-feasibility study level projections, assuming typical losses in cleaning. The locked cycle testing had much higher losses in cleaning, but it is assumed at this stage that further investigations will resolve this issue.

While the concentrate in the locked cycle tests had low copper grades, they were not diluted by pyrite, lead, zinc or other metals that are typically difficult to remove in cleaning. Therefore it was estimated that the silicate dilution of concentrate will be resolved in the next round of testwork. As a result, 25 to 28% Cu concentrate has been used for the pre-feasibility study purposes.

It is planned to treat the Pampacancha resource in the Constancia plant, blending it with Constancia ores. It is recommended that further testwork be conducted on mining blends of these two ore types to ensure that they do not adversely affect each other. On the other hand, this also may be an opportunity as they may produce better results than either ore type by itself.

As with the Constancia supergene ore, the minor chalcocite-diginite components in Pampacancha feed are expected to activate sphalerite in the high zinc Constancia ore type, so the two should be kept separate. Current mine planning will allow for segregation of the high zinc Constancia ore should separate treatment prove to be necessary.

The next phase should use fresh drill cuttings to avoid sample oxidation from impacting flotation recovery. Confirmation of the comminution results is also required, in case oxidation may have weakened the samples.

Further testwork on comminution should include JK Drop Weight Tests, SMC testing and a repeat of Bond work index and abrasion indexes to confirm the assumptions made at this stage.

Detailed diagnostic mineralogy is required on rougher concentrates and final concentrate and cleaner scavenger tailings samples in order to target regime changes and improve cleaner recovery and final concentrate grades.

14 MINERAL RESOURCE ESTIMATES

The documentation of the resource modelling work supporting the mineral resource estimates is presented separately for the two deposits: Constancia and Pampacancha.

Hudbay prepared an update of the Constancia mine and Pampacancha project 3D block models using MineSight® version 12.60, industry standard commercial software that specializes in geologic modelling and mine planning. The 3D block models and determination of the updated mineral resources were performed by Hudbay personnel in Toronto with peer reviews at each stage of the process by the mine's personel in Peru. The Qualified Person (the "QP") who supervised the preparation of this Technical Report is Cashel Meagher, P.Geo., Senior Vice President and Chief Operating Officer of Hudbay.

The topographic surface is the same as used for previous resource models and is based on photogrammetric data surveys performed in 2013 (2 metres contours) by Horizons South America S.A.C. This surface was used as the base to construct all the related topographic surfaces in MineSight.

14.1 CONSTANCIA

Resource Modeling Database

As shown in

Table 14-1, there are 418 assayed drill-holes totalling approximately 128,241 metres within the Constancia database used to support the mineral resource estimate. These 418 drill-holes include 17 twin-holes drilled in 2017, for a total of 4,167 metres. The other holes listed in Table 10-1 were either considered too distant from the mineralized zone or were drilled for other purposes, i.e. hydrogeology, geotechnical, geometallurgical and condemnation.

Table 14-1 Resource Modeling Database

Total number of Drill Holes	418
Total Drilled Meters	128,241

DH drilled in 2005	41
Total meters drilled in 2005	9,799
DH drilled in 2006	66
Total meters drilled in 2006	21,232
DH drilled in 2007	77
Total meters drilled in 2007	28,726
DH drilled in 2008	167
Total meters drilled in 2008	52,212
DH drilled in 2017	17
Total meters drilled in 2017	4,167

The main drillhole database is managed through MineSight Torque®. All the drillhole data was validated prior to being imported in MineSight Torque® through the use of Datashed®, commercial data management software. The drillhole data was validated after been loaded into the software, by randomly checking laboratory certificates, logs, assay intervals and survey data against their values in the database. No error or discrepancies were found in the database.

The drillhole database was exported from MineSight Torque® and provided in Microsoft Excel® format with a cut-off date for mineral resource estimate purposes of December 20th, 2017. The files were imported into MineSight and the drillhole traces were desurveyed using collar coordinates and downhole survey data and then merged with assays and their logging codes for lithology, oxidation and alteration.

All drillholes have full chemistry obtained from ICP analysis while approximately 30% were also analysed for sequential copper, i.e. sulphuric (CuSS) and cyanide (CuCN) soluble copper and less than 2% were measured for SG (Table 14-2). The ICP was considered the most reliable measure of total copper and whenever the sum of CuSS+CuCN was higher than Cu, the CuSS and CuCN values were rescaled so as to not change their relative proportion while ensuring that their sum did not exceed total copper measured from ICP analysis.

Table 14-2 Samples and Length Analyzed

Elements	Meters Assayed	Overall %
Cu%	204,200	99.9%
MoPPM	169,508	82.9%
AgPPM	204,138	99.8%
AuPPM	203,472	99.5%
Pb%	204,200	99.9%
Zn%	204,200	99.9%
Fe%	204,200	99.9%
Ca%	201,022	98.3%
AsPPM	204,200	99.9%
S%	204,200	99.9%
CuSS	65,764	32.2%
CuCN	65,764	32.2%
SG	2,786	1.4%
Total meters of samples		204,491

14.1.1 WIREFRAME MODEL AND MINERALIZATION

The Constancia deposit trends approximately along an azimuth of N150° with a general dip of 70° to the East. Geologically, Constancia is a Cu-Mo porphyry type deposit with small bodies of Cu-Au skarn located at the contact between the porphyry intrusions and the sediments. The Constancia deposit is continuous along a strike length of 3.0 km east-west, 3.0 km north-south and to a vertical depth of approximately 900 m.

Constancia presents the typical porphyry alteration zonation: argillic in the upper part, phyllic and potassic in the middle, with the propylitic alteration surrounding the deposit. Higher grade mineralization correlates with the phyllic and potassic alteration zones.

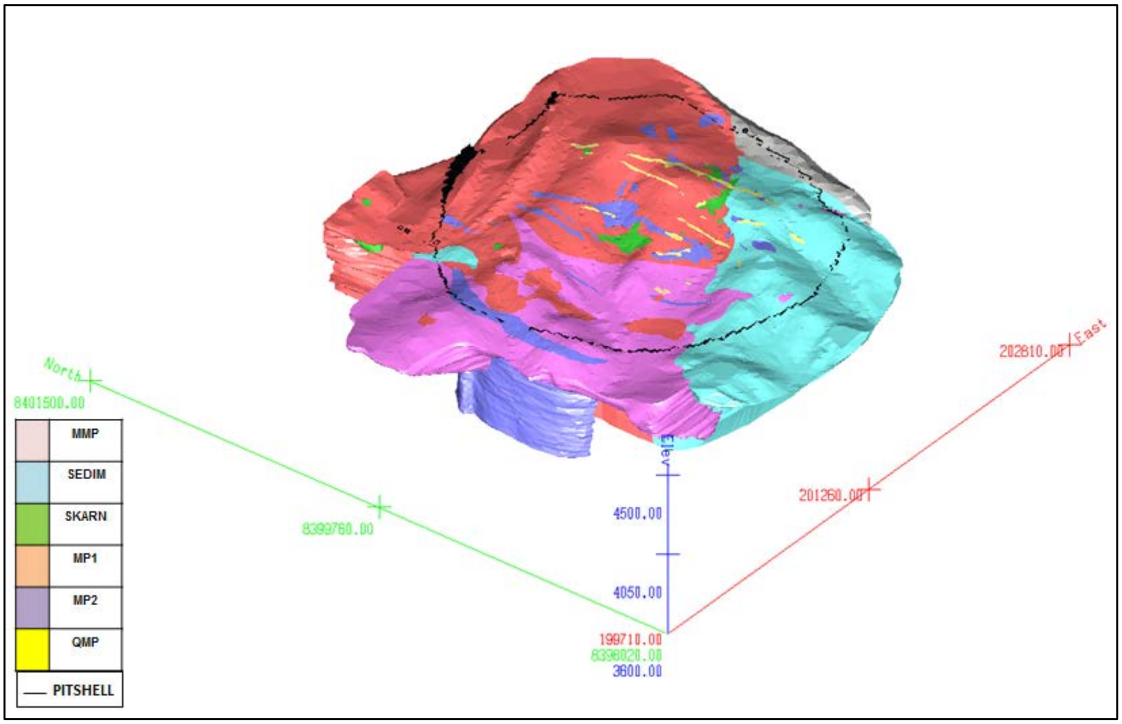
Oxidation levels of the deposit have been modelled using the total copper to acid soluble copper (sulphuric and cyanide) ratio following the formulas presented in Table 14-3.

Table 14-3 Oxidation Zone Formulation

Oxidation Levels	Criteria
Oxide	(CuSS / CuT) > 40%
Supergene	(CuSS + CuCN / CuT) > 70%
Mix	70 ≥ (CuSS + CuCN / CuT) > 30%
Hypogene	(CuSS + CuCN / CuT) ≤ 30%

Wireframes were initially constructed from 50 m cross sectional interpretations for each lithological, alteration and oxidation unit and then further refined in 3D to create solids. The resulting models are presented in Figure 14-1 (lithologies), Figure 14-2 (alteration) and Figure 14-3 (oxidation).

Figure 14-1 3D View of Interpreted Lithology Wireframes, Looking North East



In the 2016 resource model, grade interpolation was conducted within 9 different domains for Cu grade consisting of a complex combination of lithology and mineralization types. Visual checks of these wireframes in longitudinal and cross section revealed local complexity and tortuous shapes which required for many of the domains to 'back tag' the drillhole composites for grade estimation implicitly acknowledging the difficulty to model these bodies as individual solids (Figure 14-4). In addition, some evidence of undue high grade smearing was observed on the margins of the deposit in areas poorly drilled and generally reported as inferred resources in the 2016 Technical Report.

Figure 14-2 3D View of Interpreted Alteration Wireframes, Looking North East

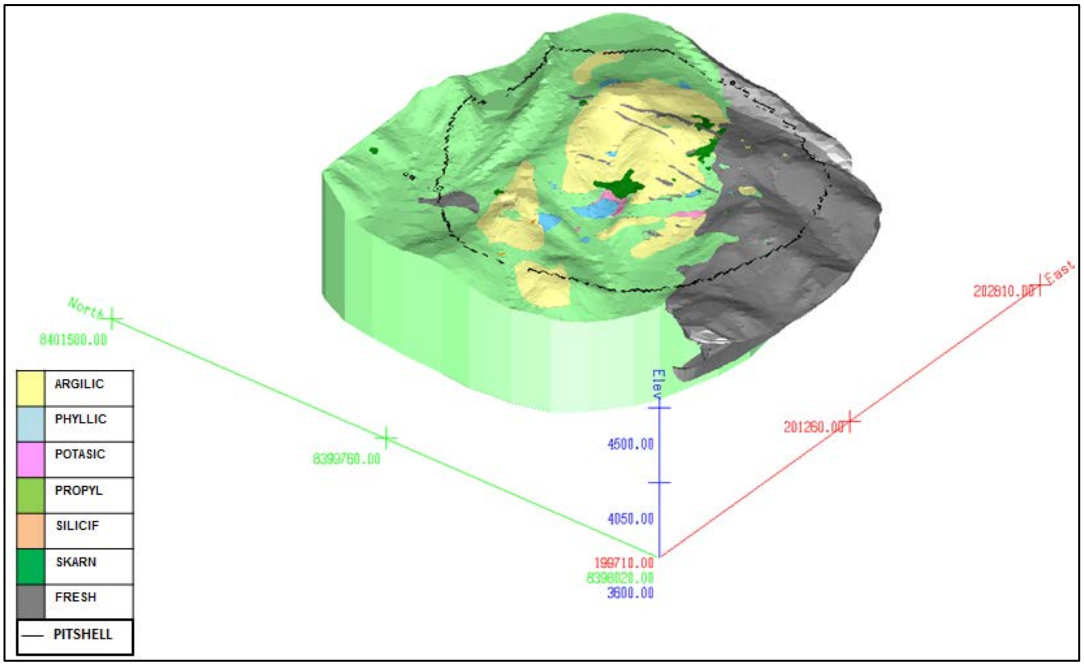
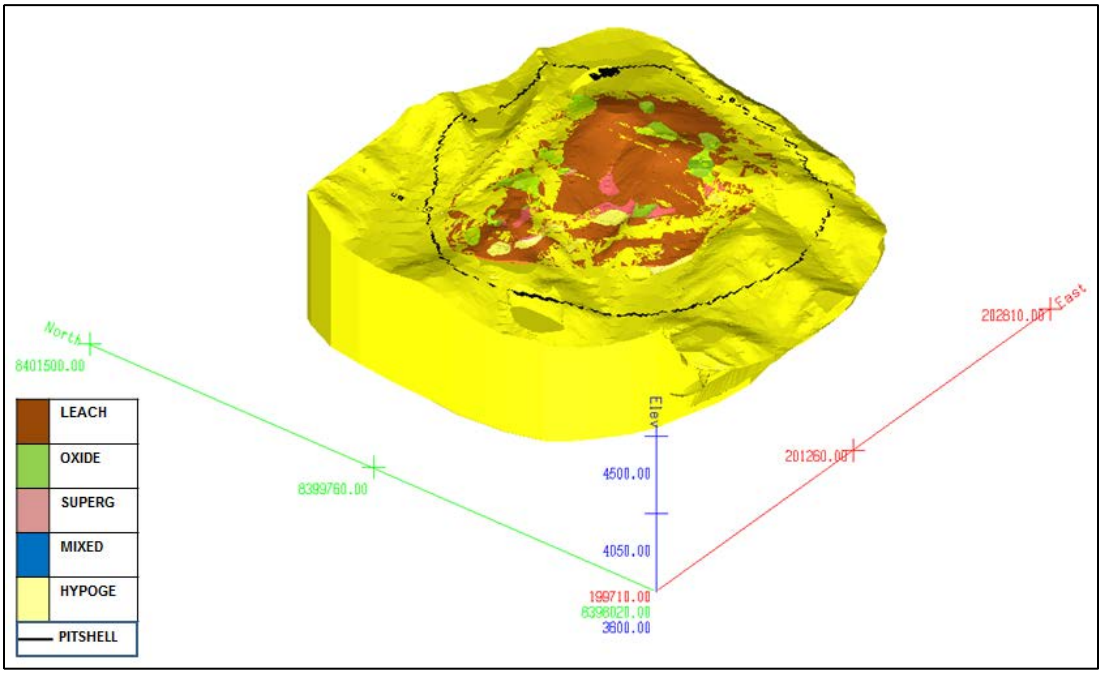


Figure 14-3 3D View of Interpreted Mineralization Wireframes, Looking North East



Following discussions with the mine geologists, the review of bench mapping from blast hole information and based on 2.5 years of operating experience, a simplified geological framework was discussed and adopted for the 2017 model. This simplified approach is illustrated on Figure 14-4 and Figure 14-5 and involves the grouping of lithology and mineralization types as follows:

- Domain 1: a hypogene envelope dominated by primary sulphides (mostly chalcopyrite) and also including most of the skarn mineralization
- Domain 2: a supergene envelope dominated by secondary sulphides (mostly chalcocite) including what was previously coded as supergene and mixed mineralization; and
- Domain 3: a low grade envelope that includes sediments, stocks and intrusive dykes.

The original interpretation was used as a starting point but with an attempt to limit the amount of mineralization grading below 0.1% Cu while ensuring spatial continuity and consistency of the two mineralized domains.

Figure 14-4 Comparison of the 2016 and 2017 Resource Modeling Framework for Grade Estimation

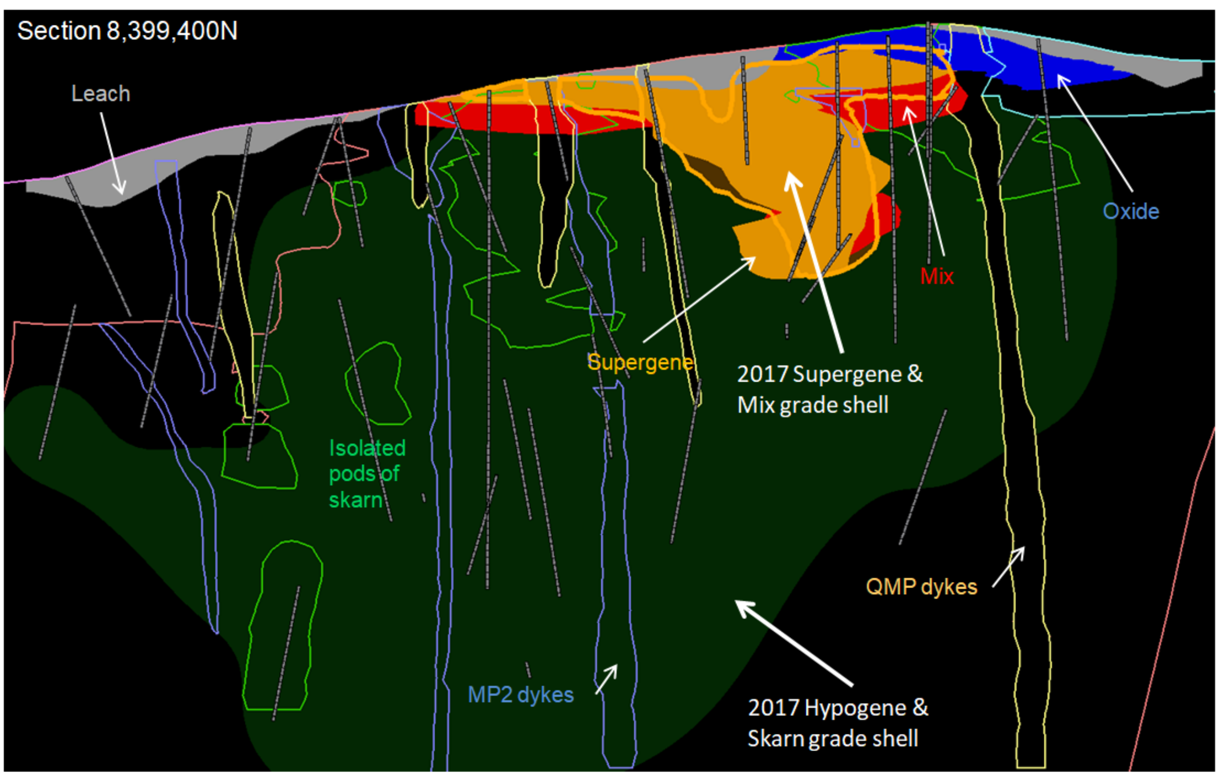
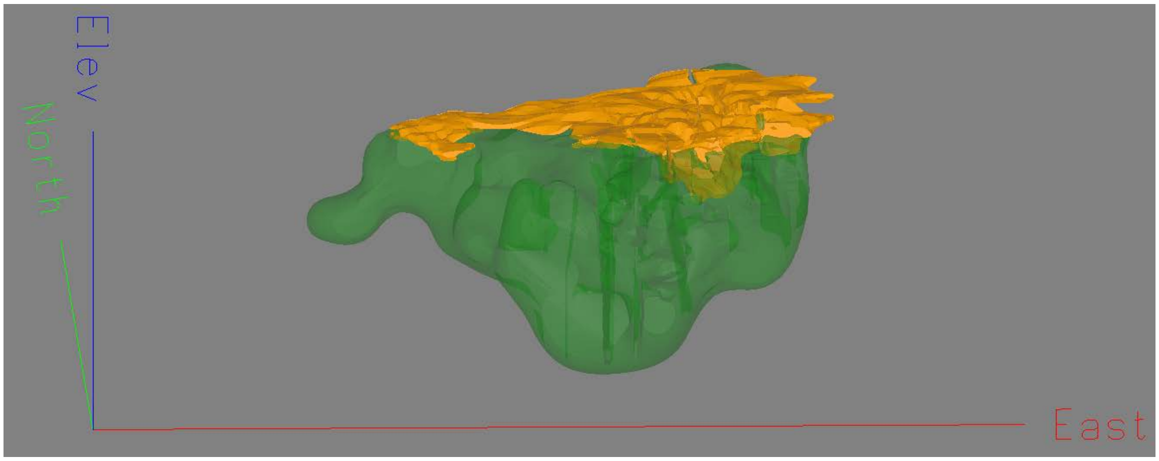
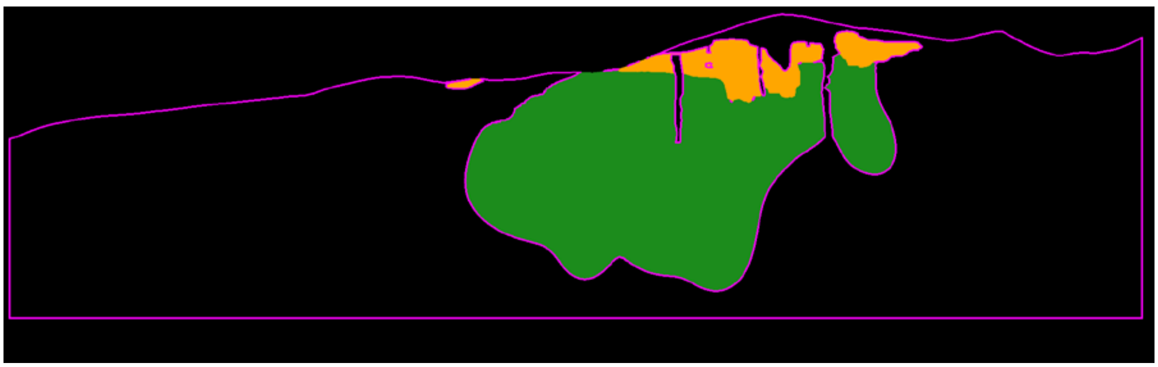


Figure 14-5 Mineral Domains Supporting the 2017 Long-Term Resource Model





Note: Supergene + Mix in orange and hypogene + skarn in green. Low grade domain in pink outline

A third domain coded in the resource model groups all the low-grade mineralization rocks (Figure 14-5). This domain includes the oxide-leach cap at the top of the deposit, low-grade monzonite dykes cross-cutting the mineralization and porphyry mineralization grading below 0.1% Cu located on the periphery of the two mineralized domains. Only inferred mineral resource estimates are reported from domain 3 from either the low-grade dykes or low-grade porphyry mineralization since the oxidized-leach cap mineralization cannot be processed by the plant.

14.1.2 2017 TWIN-HOLE PROGRAM: CU GRADE BIAS ANALYSIS AND CORRECTION

Over the past two years of operation, the Constancia mine and mill have consistently reported a 'positive grade reconciliation' to the mineral reserve estimates. Although some of this bias in grade was also offset partially by a negative reconciliation on tonnes, the overall metal content was still being significantly under-estimated in the 2016 mineral reserve model as illustrated in Figure 14-6.

While some of the under-estimation of grade could be associated to the over-estimation of tonnage and corrected by reducing the level of smoothing in the resource modelling process, a systematic reexamination of core boxes and core pictures revealed that core recovery may not have been as good as recorded during logging and could have introduced a sampling bias.

As an example, Figure 14-7 illustrates a core box where core recovery would have been initially recorded as greater than 95% on a linear basis in the database but clearly volumetric recovery is significantly lower. This is not an uncommon problem in deposits with high alteration intensity where core drilling in poorly consolidated material can result in loss of the finer material and as a result potentially in biased metal grade measurements.

Figure 14-6 2016 Reserve Model versus Mill Credit Reconciliation an a Rolling Quarterly Basis

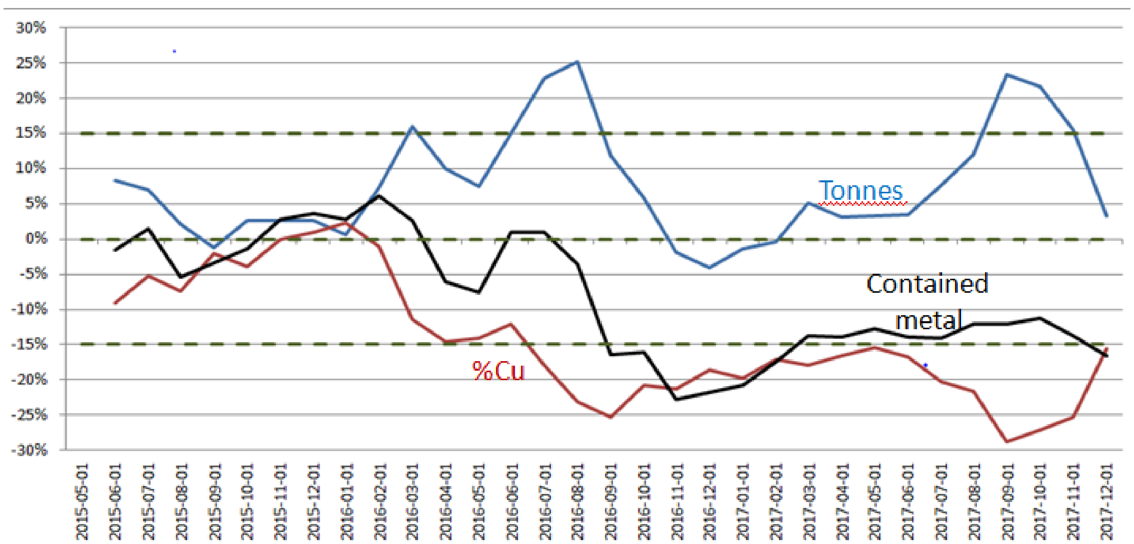


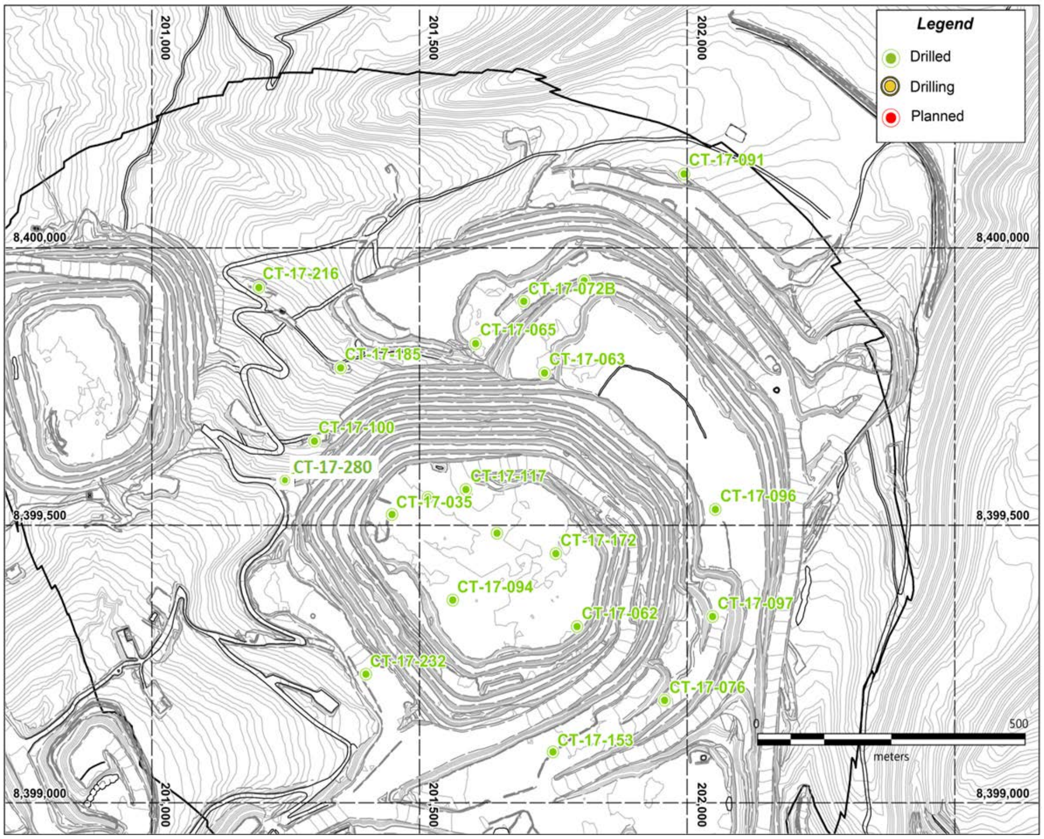
Figure 14-7 Example of Material Originally Incorrectly Recorded as >95% Core Recovery

A twin-hole program of 5,298 m was designed to confirm the existence and magnitude of a potential bias and also to conduct additional MET testing to update the throughput and recovery models. 17 holes intersecting the various enrichment and alteration units over the entire volume of the 2016 reserve pit were selected to be twined within 3 m of their original location (Figure 14-8). The drilling program was conducted during the second half of 2017 with geotechnical supervision and drilling quality control provided by Golder Associates and verified by Hudbay Minerals geologists.

Drilling was conducted with the greatest level of care (1 m per hr) using triple tubing and lubricants to minimize loss of material during the program. Overall the data quality obtained from the twin-hole program is significantly improved with the proportion of composites with an estimated volumetric core recovery higher than 85% increasing from 50% in the original 17 holes to more than 75% for their twins.

All samples were prepared by Certimin and assayed by AA and ICP methods with a thorough QA/QC program including independent checks from an empire lab and check assays at the Constancia lab.

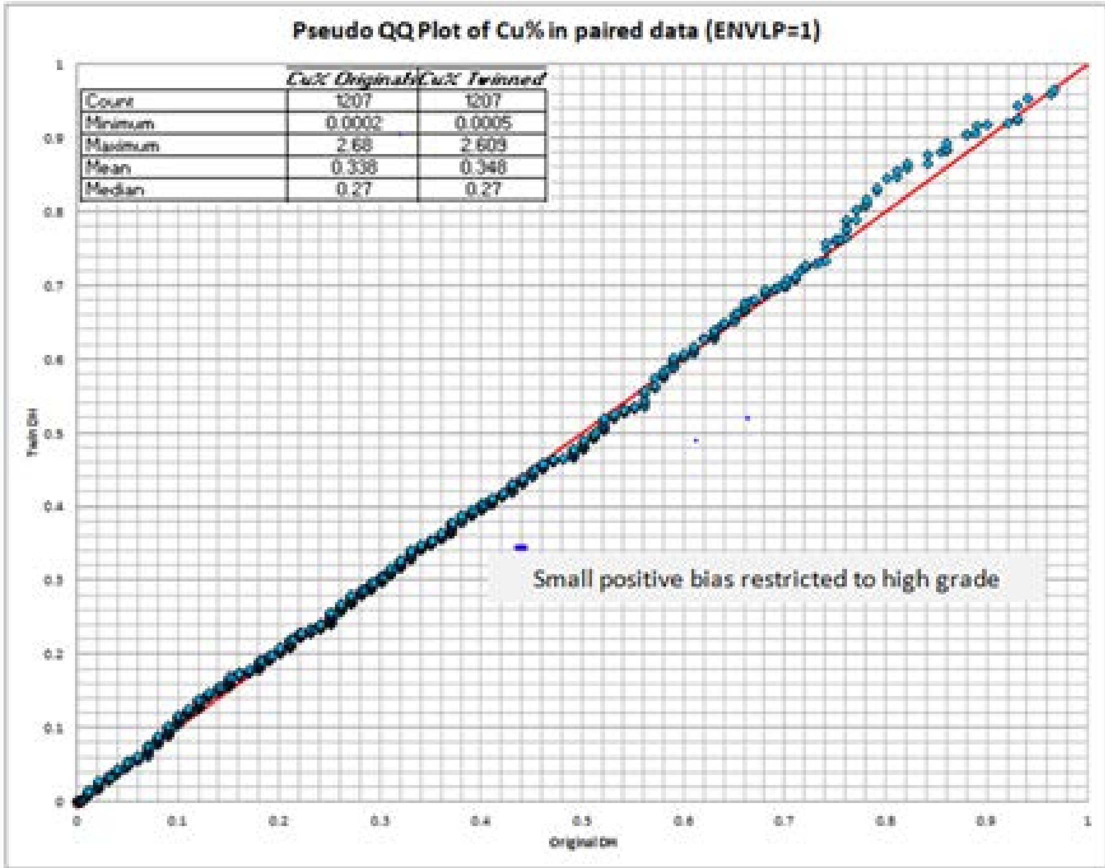
Figure 14-8 Example Twin-Drilling Program



Results from the twin-hole program focused on a comparative analysis of the %Cu grade and the analysis was conducted by mineralization domain. Domain 3 was excluded from this analysis.

In the Hypogene Domain (Domain 1), from 1,207 pairs of collocated samples of 2m, no significant bias could be observed as illustrated in Figure 14-9 below. A small bias is still possible in the high grade portion of this mineralization but the difference is not significant enough to support any data adjustment. Since material losses were also noted from the review of core pictures in the Hypogene Domain, a likely possibility is that the finer material not recovered does not have a material difference in its copper content. Sieving tests are currently underway to obtain the chemistry by grain size to confirm this assumption.

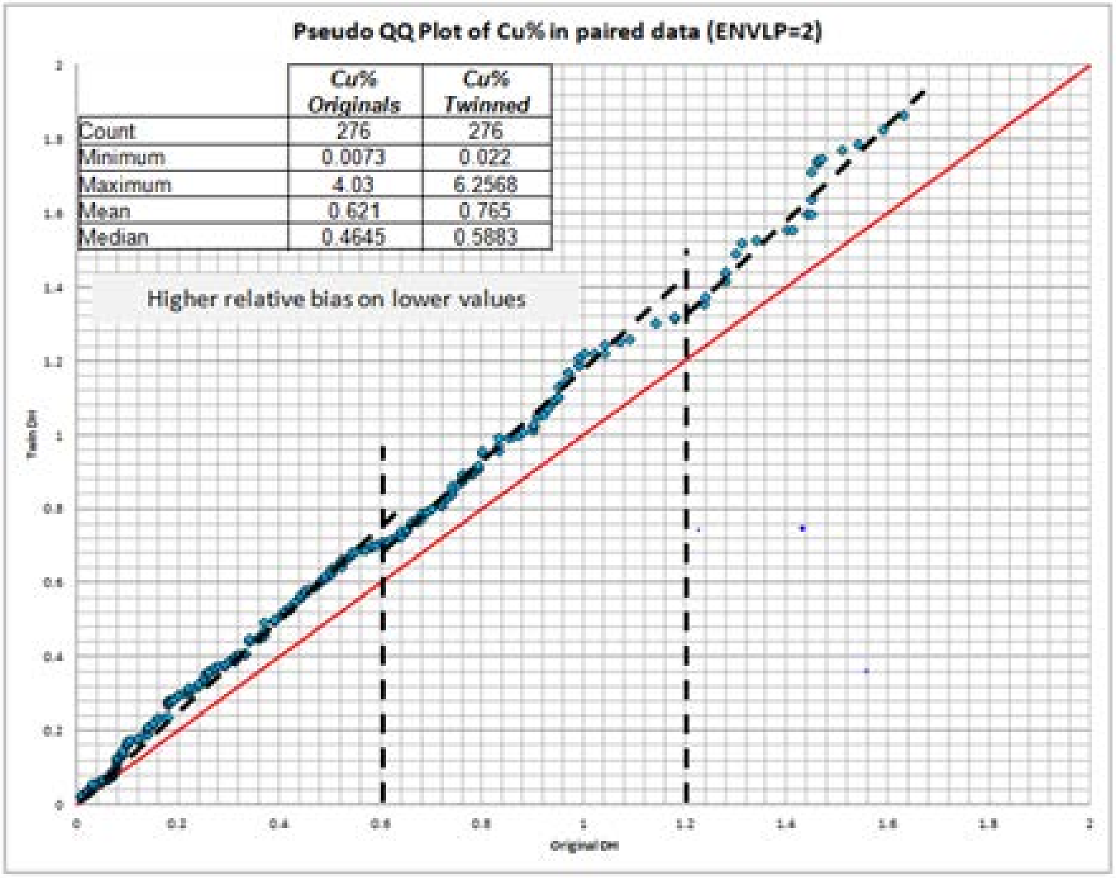
Figure 14-9 QQ Plot of %Cu From Twin-Hole Results – Hypogene Domain



In the Supergene Domain (Domain 2) however, the 276 pairs show a clear bias resulting in a global difference of circa 20% between the average grade of the original and the new samples (

Figure 14-10). The magnitude of the bias is well aligned with a similar bias that has been observed in the material mined from July 2016 until today between the old DDH data and their collocated blasthole composites as illustrated on Figure 14-11. The volumes of supergene mineralization mined since July 2016 and the remaining supergene mineralization to be mined and covered by the twin-hole program are spatially contiguous. These results confirm that the bias evidenced through this twin-hole drilling program is responsible for a large part of the positive grade bias observed since July 2016 between the reserve estimates and the mill credited production and would continue if not corrected in the resource modeling database.

Figure 14-10 QQ Plot of %Cu from Twin-Hole Results – Supergene Domain



A linear correction was applied to the modeling drillhole database by grade ranges only in the Supergene Domain as follows:

For %Cu < 0.6:	New %Cu = Old %Cu x 0.25
For 0.6 < %Cu < 1.2:	New %Cu = Old %Cu x 0.18
For %Cu > 1.2:	New %Cu = Old %Cu x 0.15

Figure 14-12 shows that after applying this grade correction, the QQ plot of the twinned holes show that the grade bias has effectively been removed in the old data with still a small conservative tendency.

For reconciliation purposes, the correction was not applied to the samples located in the volume mined during period 2 and in a new small zone only recently recognized as supergene and for which no twin drilling information is available. For the volume mined during period 1, the correction was only applied to the samples grading 1.2% Cu as per the comparison between blast holes and diamond drillholes illustrated in Figure 14-11.

Figure 14-11 QQ Plot Of %Cu In Twinned Blast Holes and Ddh– Supergene Domain

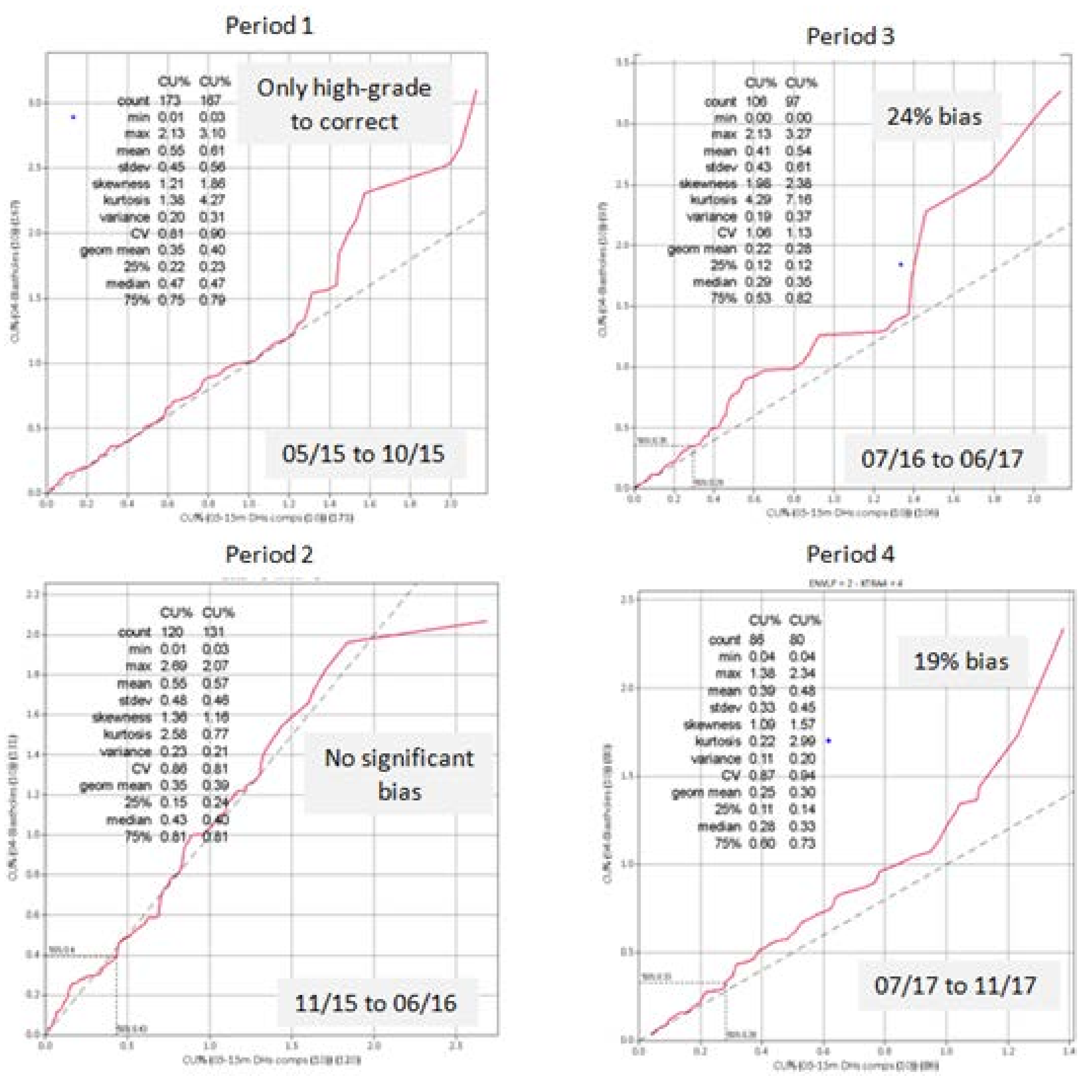
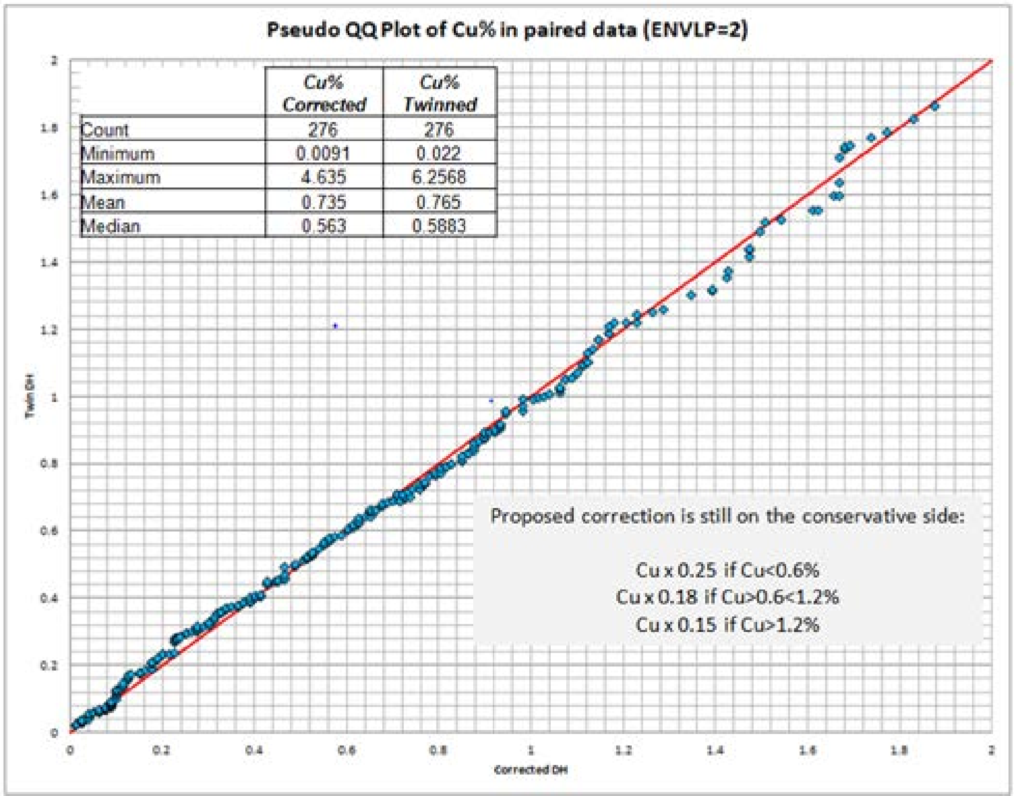


Figure 14-12 QQ Plot of %Cu After Grade Bias Correction – Supergene Domain



14.1.3 SPECIFIC GRAVITY

Specific Gravity (SG) was only measured for 2,786 samples and these actual measurements were used to derive multi-linear correlations between SG and the geochemistry in order to assign a calculated SG to the other samples used in the construction of the resource model.

The following multi-linear regressions were developed:

$$SG\ regression\ in\ ENVLP\ 1\ =\ CU\% * -0.05207 + ASPPM * 0.00058 + FE\% * 0.03354 + 2.46773$$

$$SG\ regression\ in\ ENVLP\ 2\ =\ CU\% * -0.08065 + FE\% * -0.02785 + S\% * 0.04476 + 2.39208$$

$$SG\ regression\ in\ ENVLP\ 3\ = AGPPM * -0.0151 + ZN\% * 0.1462 + ASPPM * 0.00068 + FE\% * 0.0199 + 2.516$$

When some of these metal grades were not available, the following baseline values were used:

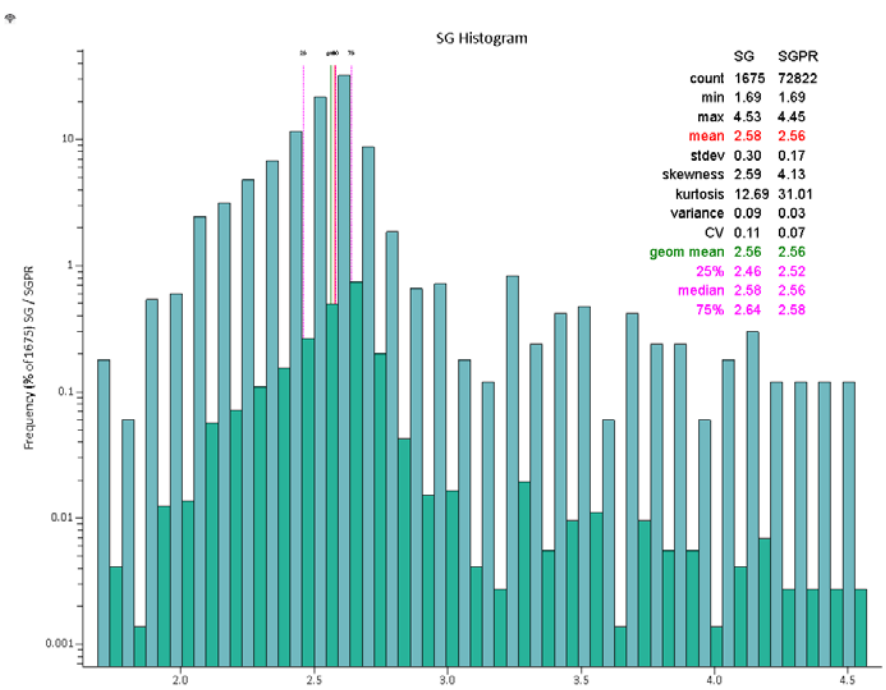
	Hypogene & Skarn within the 0.1% Cu grade shell					
Alteration	Argilic	Phyllic	Potassic	Propylitic	Silicification	Skarc
Baseline SG value	2.48	2.52	2.57	2.6	2.59	2.95

	Supergene & Mix within the 0.1% Cu grade shell				
Alteration	Argilic	Phyllic	Potassic	Propylitic	Skarn
Baseline SG value	2.29	2.34	2.43	2.46	2.35

	Outside the 0.1% Cu grade shell							
Alteration	Argilic	Phyllic	Potassic	Propylitic	Silicification	Skarc	Freh	Top soil
Baseline SG value	2.32	2.44	2.59	2.6	2.62	2.76	2.64	2.61

The predicted results were validated against the measured results which are showing similar distribution and statistics (Figure 14-13).

Figure 14-13 Comparison of Measured and Calculated Specific Gravity Distribution



Note: Measured density in dark green and predicted density in light green.

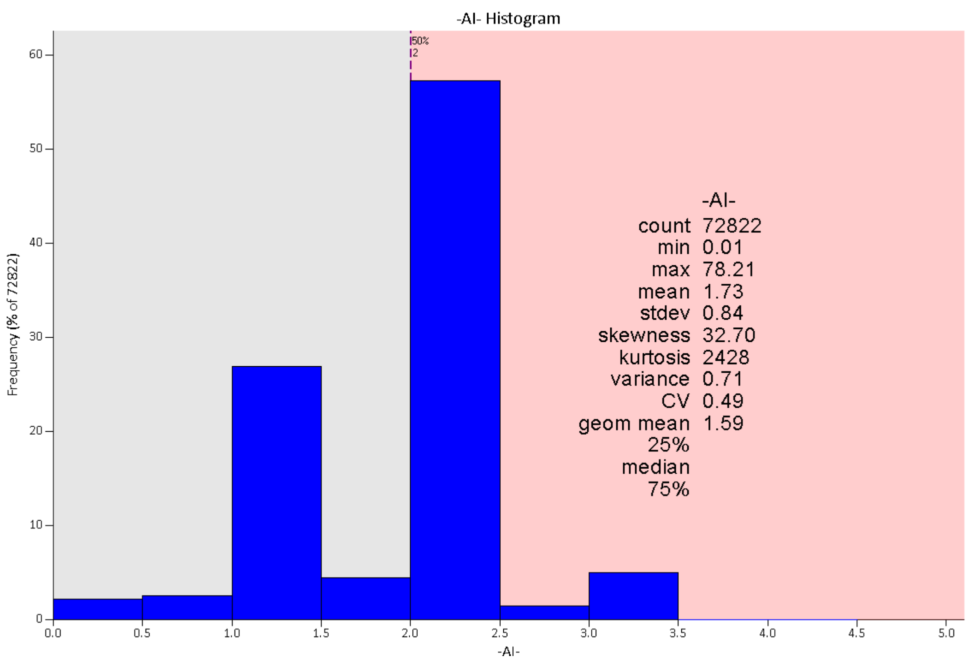
14.1.5 DRILLHOLE COMPOSITING, EXPLORATORY ANALYSIS AND GRADE CAPPING

In order to normalize the weight of influence for each sample, assay intervals were regularized by compositing the drillhole data into equal length intervals using domain boundaries to break composites. Although nearly 80% of the samples were assayed over a 2 metres-length (Figure 14-14), the compositing length was set to 7.5 m when considering the block height of 15 m. The exploratory analysis was conducted by element and by domain and used as a basis to establish whether weighting the interpolation by SG could have a significant impact and if grade capping was required. No detailed exploratory analysis was conducted for the Domain 3 composites due to the low grade and heterogenous nature of the mineralization included in this envelope which is only used for reporting inferred mineral resource estimates.

Compositing over a longer interval than the original assay results in a loss of details and resolution in the downhole grade distribution but does not impact the quality of the block estimates with a 15 m height. Composites with a length lower than 3.75 m were discarded. Compositing was weighted by specific gravity which has a material impact only for iron, zinc and arsenic grade. Assay intervals were also composited into 15 m lengths (+/- 7.5 m of threshold) using the same methodology. The 15 m composites were simply used to estimate nearest neighbour models for global bias check.

The exploratory analysis was conducted by element and by domain and used as a basis to establish whether weighting the interpolation by SG could have a significant impact and if grade capping was required. No detailed exploratory analysis was conducted for the Domain 3 composites due to the low grade and heterogenous nature of the mineralization included in this envelope which is only used for reporting inferred mineral resource estimates.

Figure 14-14 Interval Length Histograms



14.1.6 EXPLORATORY ANALYSIS IN THE SUPERGENE DOMAIN

Figure 14-15 to Figure 14-17 illustrate the main statistical properties of some of the key metals considered in the Supergene Domain (Domain 2). After compositing and with the exception of specific gravity, all the metals present a skewed histogram, reflecting the non-selective nature of the mineral envelope. In the Supergene Domain, weighting by density has a negligible impact on the statistical properties for most of the elements except for Zn and As.

Figure 14-15 Exploratory Analysis of %Cu in Supergene

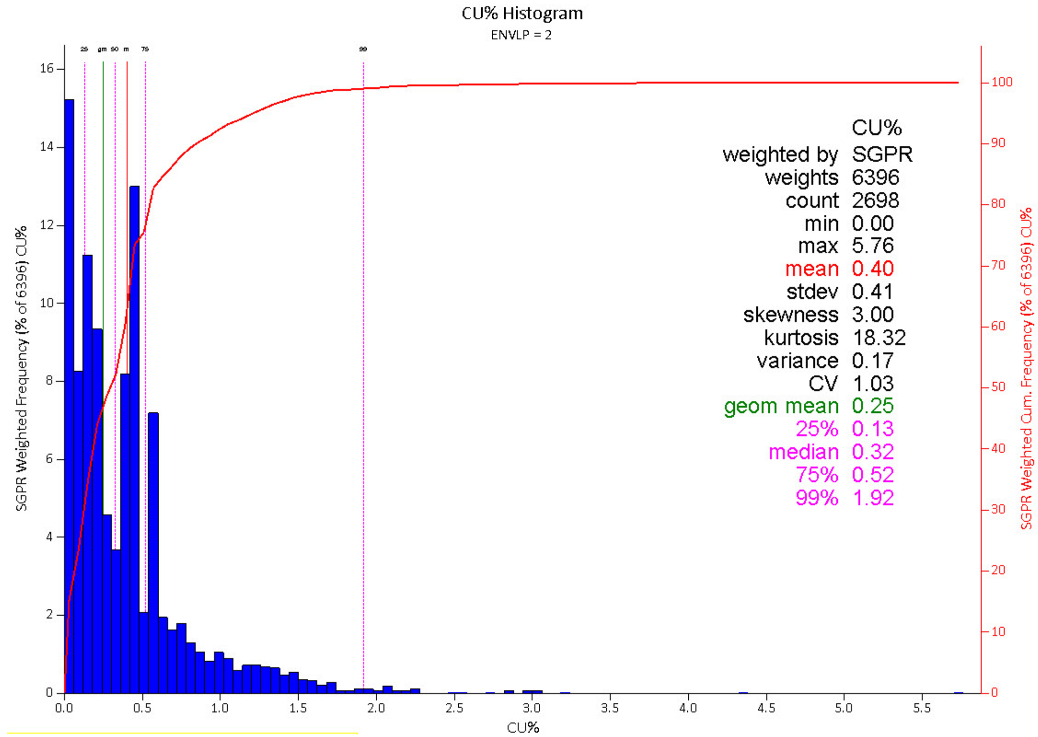


Figure 14-16 Exploratory Analysis of %Zn in Supergene

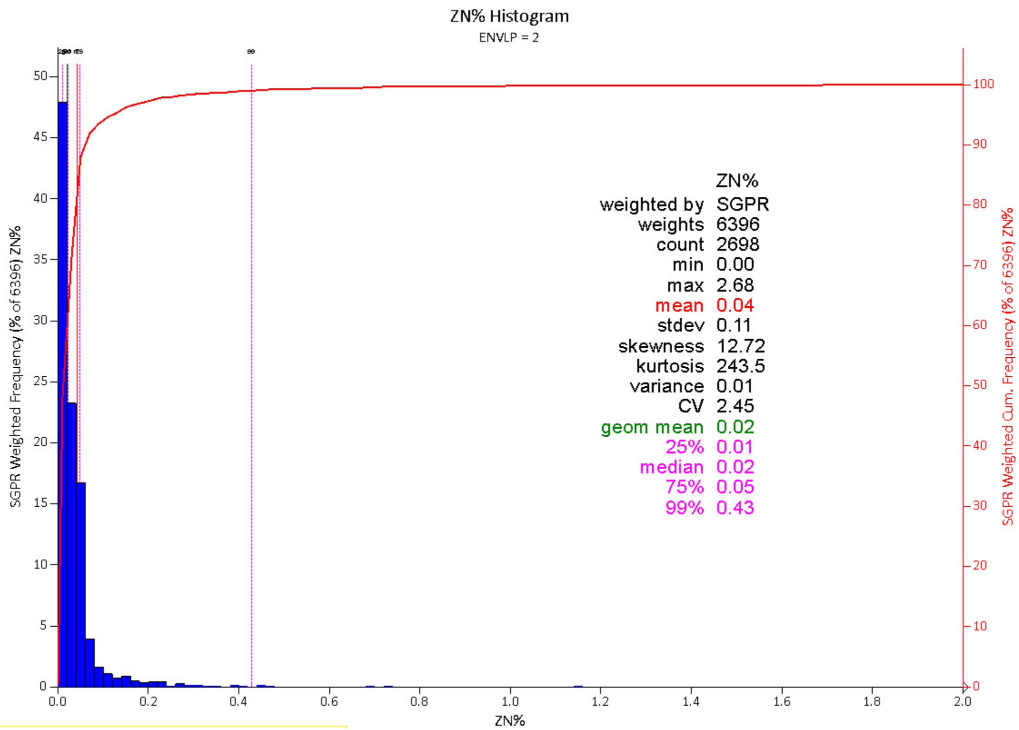
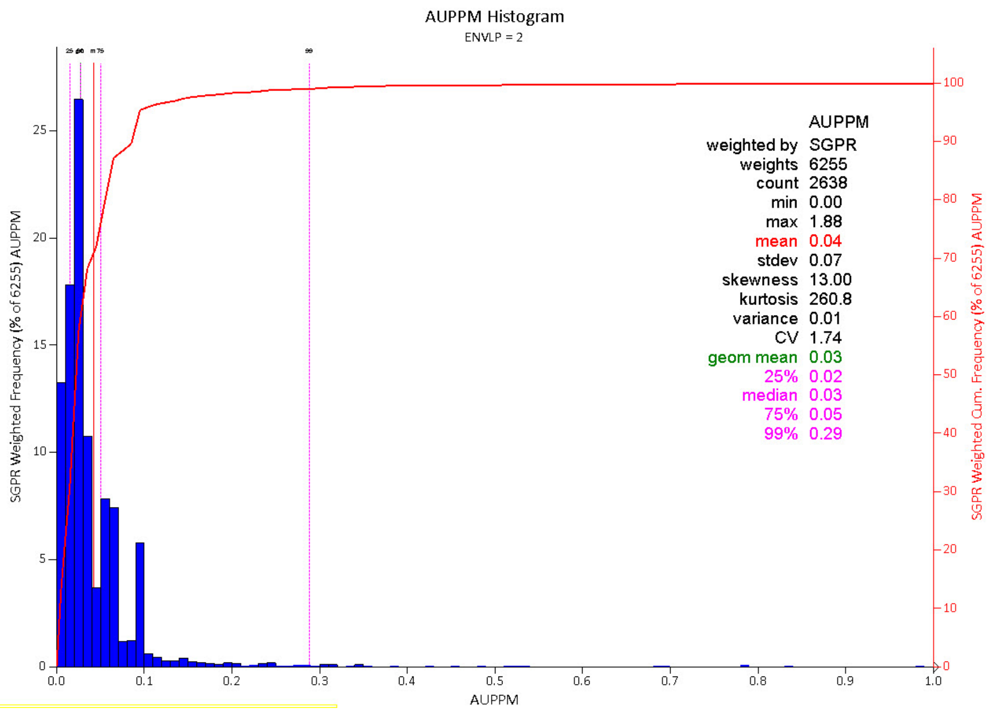


Figure 14-17 Exploratory Analysis of G/T Au in Supergene



Capping was considered due to the skewed nature of the frequency distributions in the supergene envelope for most metals. The approach to define an appropriate amount of metal at risk was based on defining a population of outliers as the highest 1% for each metal grade.

For these outliers, the mean and median values were compared and the natural breaks in the cumulative histogram of the outliers were also checked to identify natural slope changes. When the outliers represent less than 10% of the total metal content, the metal at risk was deemed low and no capping is required.

Zinc, lead and arsenic present skewed distribution and a large portion of the metal content occurs in the outliers. However, as these metals represent deleterious elements for the mill and are not a source of revenue, as a prudent approach, no capping was applied.

For Au and Ag, the outliers represent respectively 12 and 10% of the total metal content and capping was further investigated. The difference between the mean and the median grade of the population of outliers was measured as a fair representation of the quantity of metal at risk. The median is considered a more reliable measure of the central tendency of this grade population, while the mean is biased high due to the influence of a few extreme high values.

Capping thresholds were tested for values corresponding to natural breaks in the cumulative histogram of grade and the estimated metal loss was compared to the reduction obtained by replacing the mean of the outliers by their median grade. Capping was finally applied to natural breaks on the cumulative histograms best matching the metal reduction obtained from the mean-median comparison.

The metal reductions resulting from the application of capping are respectively 5% and 3% for Au and Ag on the composited samples.

14.1.7 EXPLORATORY ANALYSIS IN THE HYPOGENE DOMAIN

Figure 14-18 to Figure 14-20 illustrate the main statistical properties of some of the key metals considered in the Hypogene Domain (Domain 1).

As for the Supergene Domain, after compositing, all the metals present a skewed histogram. In the hypogene zone, weighting by density has some limited impact of a few percent on the statistical properties for several of the metals including Cu, S, Au and Ag and a more significant impact again on Fe, Zn and As.

Zinc, lead and arsenic present skewed distribution with a large portion of the metal content occurring in the outlier's population but these metals were not capped to maintain a prudent modeling approach for deleterious elements in the mill feed.

In the hypogene zone, capping was again applied only for Au and Ag based on the difference between the median and mean grade of the high-grade outlier's population.

20% of the total CuSS and CuCN content is also contained in only 1% of the composites in the hypogene zone but these elements are solely used for the purpose of ore type classification and as a result were not capped.

The metal reductions resulting from the application of capping on Au and Ag are 9% and 3% respectively of the composited samples.

Figure 14-18 Exploratory Analysis of %Cu in Hypogene

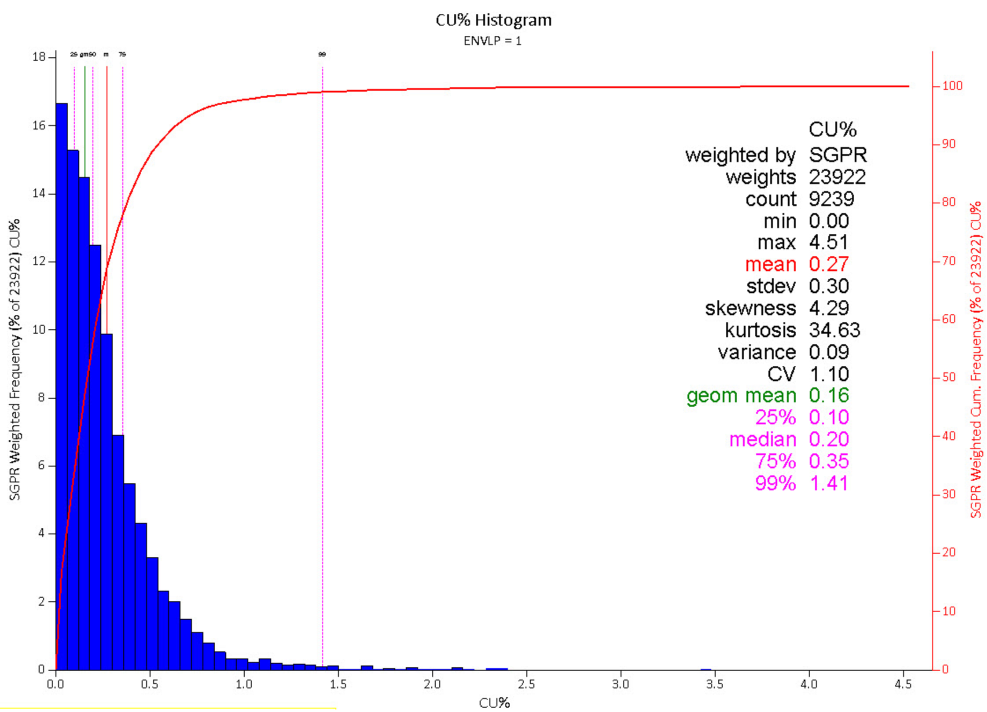


Figure 14-19 Exploratory Analysis of %Zn in Hypogene

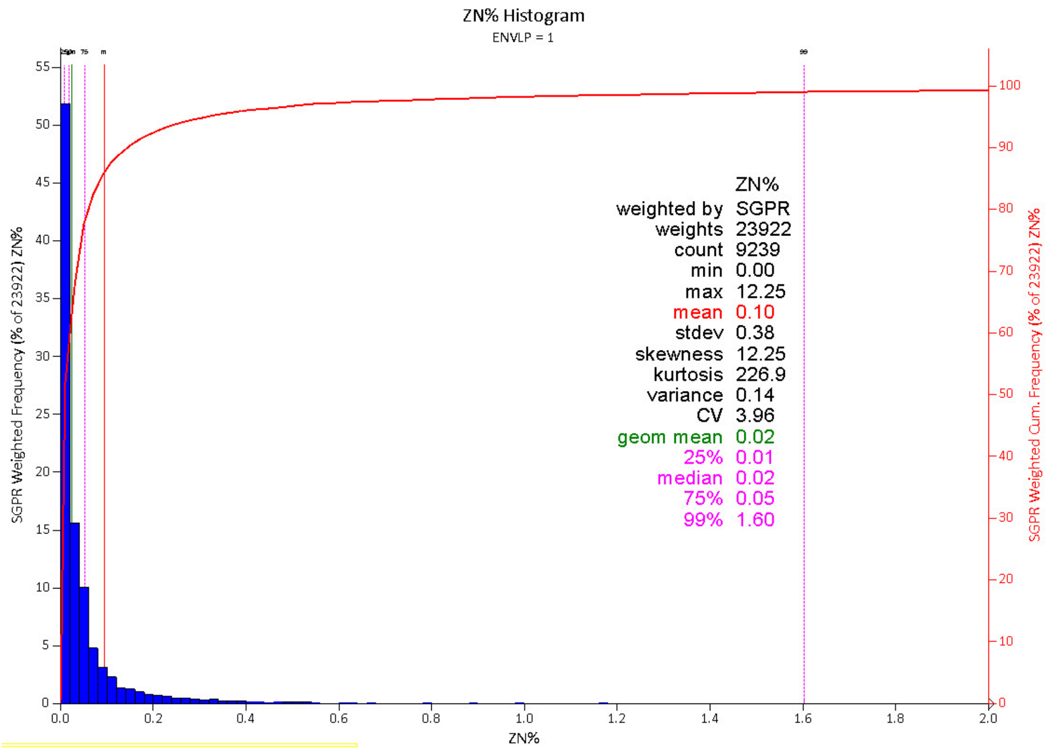
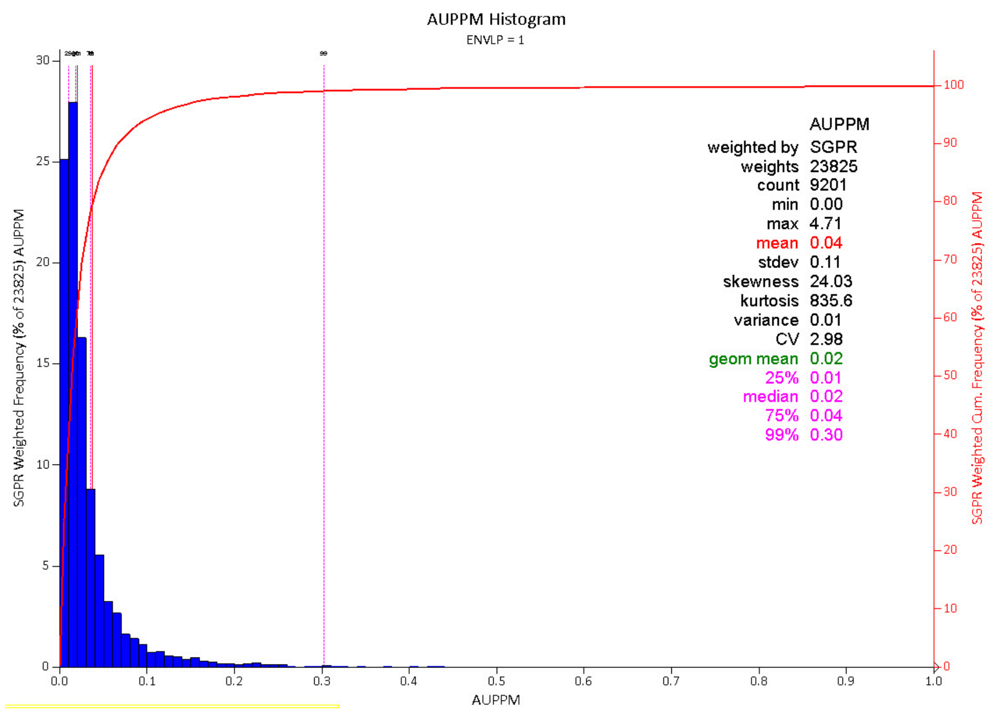


Figure 14-20 Exploratory Analysis of G/T Au in Hypogene



14.1.8 VARIOGRAPHY

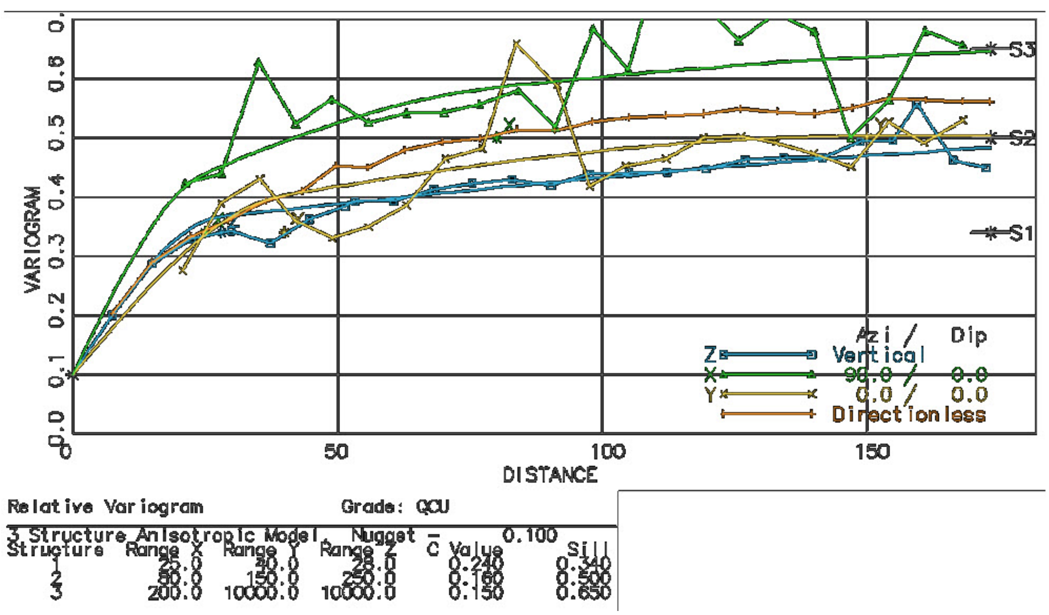
Grade variograms were calculated for each element by domain. No variograms were calculated for the Domain 3 composites due to the heterogeneity and low-grade nature of the mineralization within this envelope which is only used for reporting inferred mineral resource estimates.

First, blast hole composites were used to calculate the Cu variograms in order to define the short-range variability in the horizontal plane. This data cannot be used with confidence in the vertical direction as it represents only a few benches and the blast hole is composited over a 15m length. However, the tight drill spacing provides a useful guide to initiate variogram modeling in the XY plane.

In a second step, pairwise relative variograms were calculated from the DDH composites only for the Cu grade. A linear combination of a nugget effect and two spherical structures were fitted starting from the variogram model tested on the blast hole data. A third spherical structure was added to model a zonal anisotropy in the E-W direction in the Hypogene Domain only corresponding to the general orientation of the alteration zoning. Several rotations were tested in the horizontal and vertical plane but none was found significant enough and in the end an unrotated variogram model was adopted.

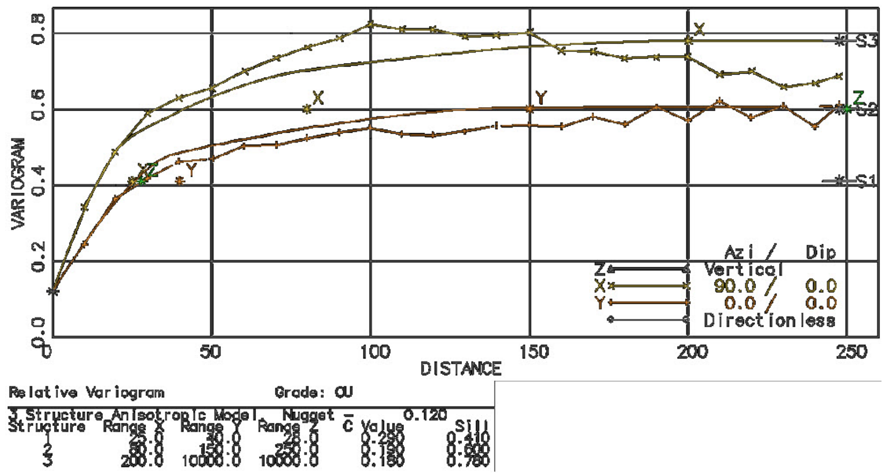
Figure 14-21 illustrates the Cu variogram with the best direction of continuity being in the vertical direction followed by the N-S direction with the E-W direction showing the most variability.

Figure 14-21 Cu Variogram in the Hypogene Domain



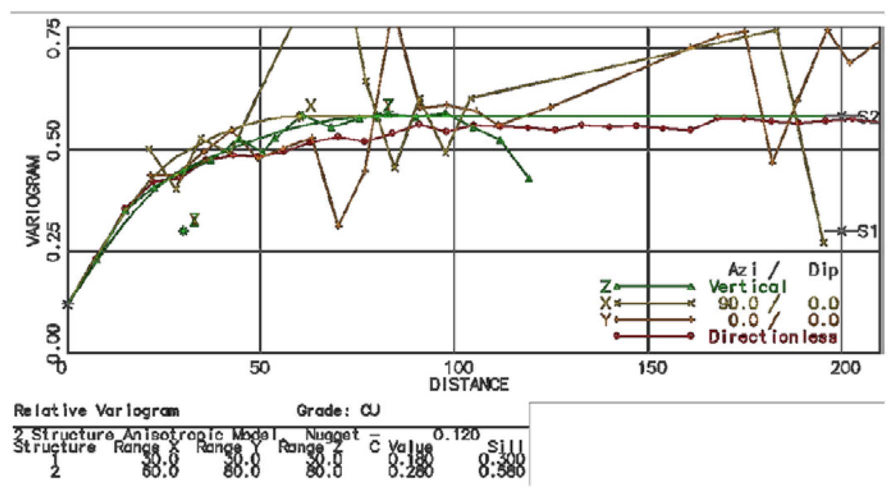
The model adjusted to the pairwise relative variogram of Cu from DDH composites was then checked back against a pairwise relative variogram calculated from the blast hole data and found to be in good agreement considering the differences in the length of the composites (Figure 14-22).

Figure 14-22 Cu Variogram Model Check Against Blast Hole Data - Hypogene Domain



The same approach was applied to the Supergene domain. In this zone, no zonal anisotropy was identified which is to be expected as this domain hosts secondary mineralization without significant zoning in alteration patterns (Figure 14-23). In the supergene zone, no geometric anisotropy was identified in the horizontal plane and only a slightly lower vertical range was adopted.

Figure 14-23 Cu Variogram - Supergene Domain



The variogram models fitted to the experimental variograms of %Cu were then tested after some re-scaling of the sill to the pairwise variograms calculated for the other metals in each domain. When required, the ranges of the model were also adjusted.

In the Hypogene Domain, the zonal anisotropy observed for the Cu grade in the Hypogene Domain could not be reproduced for the other elements. The Cu variogram model was significantly changed for Zn, Pb, Au, As, CuSS and CUNN (similar model for these last two elements).

In the Supergene Domain, the Cu variogram model was significantly changed only for SG, Pb and Au. The variogram models adopted for all the metals interpolated in the resource model are summarised in Table 14-4 and Table 14-5 for each domain.

14.1.9 ESTIMATION AND INTERPOLATION METHODS

A block model prototype with cells 20 m x 20 m x 15 m was created in order to entirely fill the Domain 1 and Domain 2 envelopes. The rest of the prototype model is coded as Domain 3. The block size was chosen such that geological contacts are reasonably well reflected and to support a large-scale mining scenario while taking into consideration the DDH spacing.

Table 14-4 Variogram models in the Hypogene Domain

Grade	Nugget	Spherical model 1				Spherical model 2				Spherical model 3			
		X Range	Y Range	Z Range	Sill	X Range	Y Range	Z Range	Sill	X Range	Y Range	Z Range	Sill
Cu, Ag	0.1	25	40	28	0.24	80	150	250	0.16	200	10000	10000	0.15
Fe, S	0.03	25	40	35	0.1	80	150	250	0.08				
SG	0.00036	25	40	35	0.0012	80	150	250	0.00096				
Zn	0.18	25	25	22	0.22	80	80	250	0.16				
Pb	0.28	25	25	25	0.22	150	150	250	0.14				
CUSS	0.08	45	45	45	0.18	120	120	250	0.15	250	10000	10000	0.15
CUCN	0.1	35	35	35	0.15	120	120	250	0.12	250	10000	10000	0.15
Au	0.1	40	40	25	0.2	150	150	250	0.25				
As	0.1	32	32	32	0.13	60	60	250	0.13				
Mo	0.16	25	40	35	0.3	80	150	250	0.32				

Table 14-5 Variogram Models in the Supergene Domain

Grade	Nugget	Spherical model 1				Spherical model 2				Spherical model 3			
		X Range	Y Range	Z Range	Sill	X Range	Y Range	Z Range	Sill	X Range	Y Range	Z Range	Sill
Cu, Fe, S, Zn, CuSS, CuCn, Ag, Mo, As, Ca	0.12	30	30	30	0.18	60	80	80	0.28				
SG	0.0003	45	45	17	0.00155	90	90	125	0.00125				
Pb	0.2	30	30	25	0.25	60	80	120	0.15	10000	10000	150	0.1
Au	0.1	25	25	25	0.11	80	120	120	0.165				

There is no cell splitting in the model as it would provide an artificial level of details which is not available from the current drill spacing and is not necessary for long-term mine planning. The interpolation plan was completed on the composites, 7.5 m in length, via ordinary kriging (OK) inside Domains 1 and 2. Grade interpolation in Domain 3 was conducted by inverse square distance using a search strategy similar to the one used for the Hypogene Domain.

All the OK and inverse distance grade estimates were weighted by specific gravity. Grade estimation used a composite and block matching scheme based on the envelope code, i.e. the envelopes were treated as hard boundaries for grade interpolation purposes.

The interpolation process used a nested search strategy with three passes of increasing ellipsoid radius. The composite selection parametres for grade estimation in each domain (minimum, maximum and, maximum number of composites per hole) were selected to minimize bias.

Table 14-6 displays the search distances and search ellipse orientations used for copper, soluble copper (sulphuric acid and cyanide), molybdenum, silver, gold, zinc, iron, calcium, arsenic, sulphur and SG.

Table 14-6 Search Ellipse Parametres

	Field	X radius	Y radius	Z radius
Hypogene initial search radius	Cu, Fe, SG, S, Ag, Mo	25	70	110
	Zn, Pb, Au, As	50	50	110
	CUSS, CUCN	40	60	110

	Field	X radius	Y radius	Z radius
Supergene initial search radius	Cu, Fe, S, Zn, CuSS, CuCn, Ag, Au, Mo, As	60	80	80
	SG	90	90	110
	Pb	110	110	70

	Min # of Comps	Max # of Comps	Max # of Comps per hole	Declustering	Max Comps per quadrant	Min number of Quadrant	Search ellipse
Pass #1	1	32	6	no	8	1	3x the initial
Pass #2	16	32	6	yes	8	3	2x the initial
Pass #3	16	32	6	yes	8	3	initial

The first interpolation pass is restricted to a minimum of one composite, and a maximum of 32 composites (with a maximum of three composites per hole), without quadrant declustering.

The second and third interpolation passes are restricted to a minimum of 16 composites, and a maximum of 32 composites (with a maximum of six composites per hole) with quadrant declustering.

The maximum of 6 composites per hole with a composite length of half the block size still allow enough samples to be considered from the 'bench above' and 'bench below'.

As block grade interpolation is conducted with more stringent criteria, the previous estimate is replaced with the new and more robust one.

14.1.10 RESOURCE MODEL VALIDATION

The final block model was validated to ensure:

- appropriate honouring of the input data but acknowledging that some natural smoothing should occur between samples and that the grade of a sample in the middle of a block is not the average grade of the block;
- absence of global bias by comparing the mean grade estimated by kriging to the original composite average grade and to a declustered grade obtained from a nearest neighbour interpolation; and
- assessment of the level of smoothing in the model for the selected selective mining unit (SMU) dimensions considered for reserve reporting.

The validation of a resource model can only be fully satisfactorily if it can be demonstrated that the reserve estimates reconcile well with actual production reported by the mill. Of course this is not possible where production data is not available or not reliable. At Constancia however, after 2.5 years of operation and with sound grade control methods in place, Hudbay believes that this exercise constitutes the ultimate validation of the performance of the resource model.

Honouring the DDH Data

Systematic visual inspection of block grade versus composited data was conducted in section and plan view for all three domains. This check confirmed a good reproduction of the data by the model. Examples of east-west and south-north oriented cross sections are provided in Figure 14-24 and Figure 14-25.

Figure 14-24 Example of Visual Validation of Grade Interpolation on E-W Cross-Section

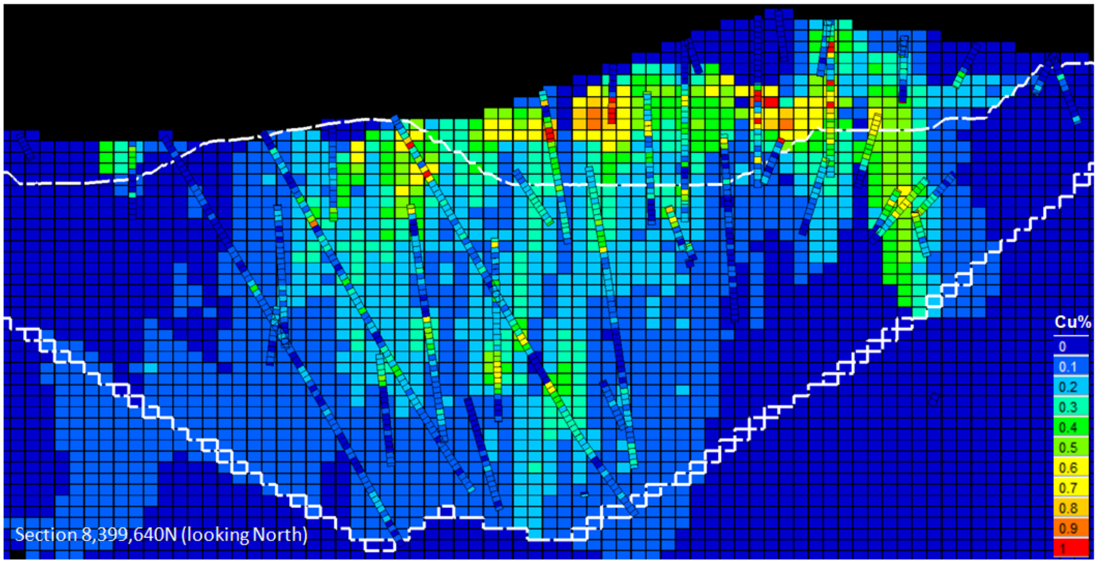
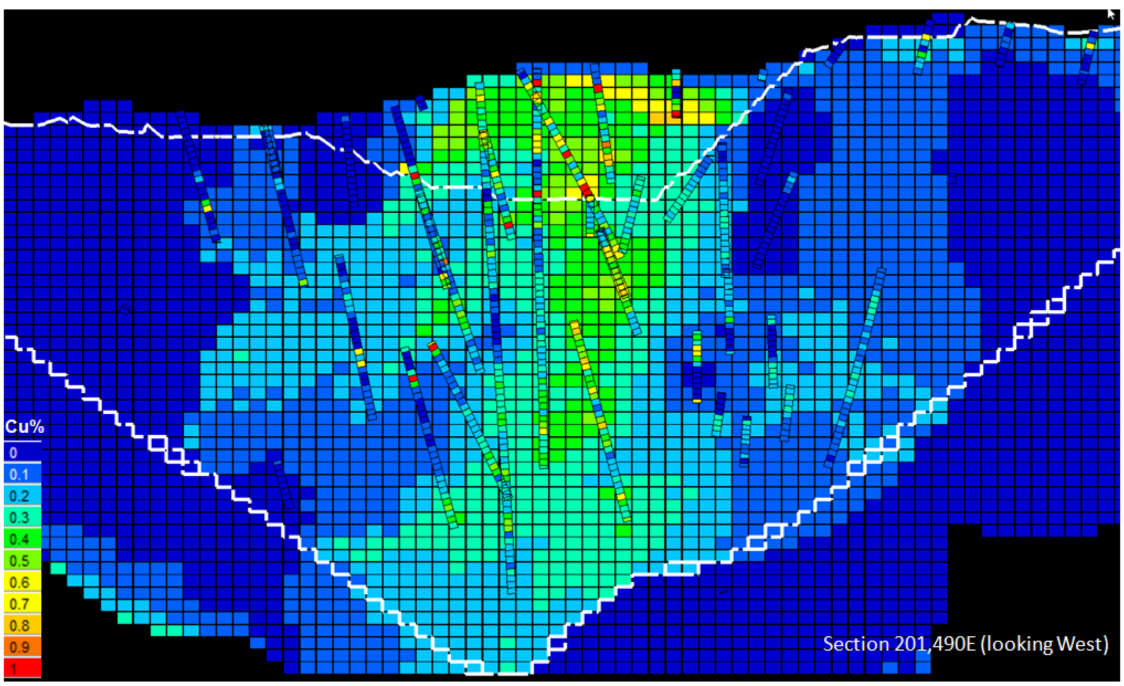


Figure 14-25 Example of Visual Validation of Grade Interpolation on S-N Cross-Section



Check for Absence of Global Bias

This validation step consisted of comparing the global average grade of each element (after capping for gold and silver) between the original composites, the kriged block estimates and the nearest neighbour estimates.

A nearest neighbour interpolation is equivalent to the declustered statistics of the composites based on weighting each composite by its polygon of influence. The average grade obtained from this method is a useful benchmark but not a perfect one as it fails to incorporate the nugget effect measured by the variogram. The higher the nugget effect, the closer the average grade should be to the mean of the composites. Ordinary kriging is in fact the best method of declustering.

A valid global check is performed to verify that the kriged mean block estimate was located between the mean of the composites and the mean of the nearest neighbour model.

Table 14-6 and Table 14-7 present the results of these global checks for each element for Domains 1 &2 - revealing no significant issues for any of the interpolated variables. The impact of declustering has a significant impact on the mean grade for most of the economic metals associated with the sulphide mineralization but also for CuSS and CuCN. This is a fairly common result as higher grade zones tend to be over-sampled compared to lower grade zones. Similar comparisons for Domain 3 are less meaningful due to the discontinuous and heterogeneous nature of the mineralization zones combined in this domain. Validation in Domain 3 was limited to visual and global bias checks (Table 14-7). All the statistics presented in Table 14-7, Table 14-8 and Table 14-9 are weighted by specific gravity.

Table 14-7 Check for Global Bias in the Hypogene Domain

Grade	Composites	OK Model	NN Model
Cu	0.272	0.238	0.227
Mo	92	74	73
Ag capped	2.6	2.4	2.3
Au capped	0.035	0.033	0.032
Pb	2.90	0.03	0.04
Zn	0.10	0.10	0.09
As	14	14	14
S	1.95	1.81	1.78
Fe	4.10	3.93	3.83
CuSS	0.012	0.012	0.011
CuCN	0.031	0.032	0.029

Table 14-8 Check for Global Bias in the Supergene Domain

Grade	Composites	OK Model	NN Model
Cu	0.445	0.474	0.478
Mo	84	84	83
Ag capped	4.8	4.1	4.0
Au capped	0.040	0.039	0.039
Pb	3.35	0.05	0.05
Zn	0.04	0.05	0.06
As	19	15	15
S	2.25	1.71	1.75
Fe	3.23	3.35	3.35
CuSS	0.065	0.094	0.096
CuCN	0.271	0.331	0.326

Table 14-9 Check for Global Bias in the Low Grade Domain

Grade	Composites	ID Model	NN Model
Cu	0.071	0.042	0.038
Mo	27	16	15
Ag capped	1.9	1.4	1.4
Au capped	0.022	0.018	0.018
Pb	0.04	0.03	0.03
Zn	0.10	0.06	0.06
As	15	14	14
S	1.33	1.50	1.40
Fe	3.44	3.37	3.32
CuSS	0.036	0.014	0.011
CuCN	0.024	0.020	0.018

Smoothing Assessment

The visual validation conducted in section and plan views (see examples on Figure 14-24 and Figure 14-25) confirmed that the block grade interpolation was consistent with the supporting composite data after capping. The larger number of composites used for grade estimation in the new model significantly improves the individual block grade estimates but, at the same time results in a much smoother model requiring a careful assessment.

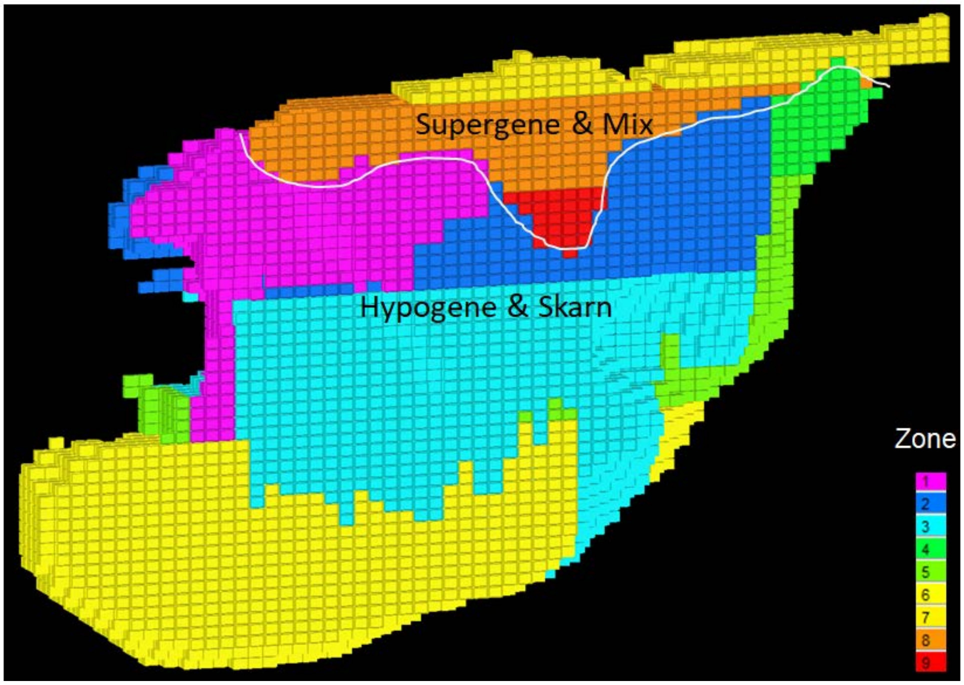
The extent of grade 'over-smoothing' in the model was investigated separately in 6 separate zones within the Hypogene Domain and in 3 separate zones within the Supergene Domain, in order to take into consideration domains with material differences in grade distribution and/or drilling density as illustrated in Figure 14-26. No smoothing assessment was conducted in Domain 3 as no reliable variogram could be derived from this zone where the large spacing between samples is less likely to have resulted in a significant over-smoothing.

The mean and variance of the kriged estimates were compared in each zone with the variance of the composites after declustering (obtained from a nearest neighbour interpolation). The expected true variance between SMUs can be calculated from the variogram models summarized in Table 14-3 and Table 14-4.

14.1.9 ESTIMATION AND INTERPOLATION METHODS

Two SMU dimensions were tested corresponding to two possible levels of mining selectivity envisaged by the mining operations. A SMU of 10mx10mx15m corresponds to the level of selectivity applied today while a larger SMU size of 20mx20mx15m was preferred when the average grade of the mine and grade variability will decrease and the emphasis will move towards reducing operating costs.

Figure 14-26 Domains Used for Smoothing Assessment and Correction



The results of the smoothing assessment are illustrated in Table 14-10 for Cu. For example, the variance between blocks 20mx20mx15m should be 0.101 or approximately 65% of the total declustered variance of the composites in zone 1 according to the variogram model. The OK model actually produced estimates with a variance of 0.0337 or only 17% of the nearest neighbour variance. This implies that the true variance between blocks should be almost three times larger in this zone (smoothing ratio of 3.00) than the one obtained from the OK estimates.

Over-smoothing is a normal outcome of a sound interpolation method when the drill spacing is not sufficient to address the short-range variability in the metal grade distribution. Smoothing will gradually reduce as additional infill drilling is performed during the pre-production period. As expected the over-smoothing is more severe for a smaller selective mining unit.

This exercise was repeated for all the elements used for mine planning including Cu, Au, Ag, Mo, Fe, Zn, Pb, As, CuSS and CuCN.

Table 14-10 Summary of the Smoothing Assessment by Zone for %Cu

Zone	NN model variance	OK model variance	Theoritical varaince between 20m x 20m x15m blocks	Over-smoothing ratio for 20m x 20m x15m blocks	Theoritical varaince between 10m x 10m x15m blocks	Over-smoothing ratio for 10m x 10m x15m blocks
1	0.1948	0.0337	0.101	3.00	0.122	3.62
2	0.0350	0.0115	0.018	1.57	0.022	1.90
3	0.0403	0.0093	0.021	2.25	0.025	2.72
4	0.0349	0.0120	0.018	1.51	0.022	1.83
5	0.0348	0.0073	0.018	2.49	0.022	3.01
6	0.0332	0.0119	0.017	1.45	0.021	1.75
7	0.2347	0.0708	0.118	1.66	0.143	2.02
8	0.2743	0.0890	0.138	1.55	0.167	1.88
9	0.5239	0.0824	0.263	3.19	0.319	3.87

Smoothing Correction for Resource and Reserve Reporting

Using the smooth kriged estimate would result in an erroneous grade-tonnage curve and reporting resources or reserves at a cut-off grade different than 0% would result in biased estimates, usually over-estimating tonnes and under-estimating grade.

An indirect log-normal correction was used to perform a change of support on the kriged model in order to obtain unbiased grade tonnage curves within the future pit. This correction is only valid globally and provides poorer local estimates than the smoothed kriged model but does not materially alter the global average grade within each zone and provide the correct grade-tonnage curve for the variogram models fitted on the drillhole data. It is an appropriate method to predict the recoverable tonnage and grade such as the volume mined over three months of production which should be a realistic aim for a long-term reserve model based on exploration drilling.

For some of the elements, the correction did not fully attain the targeted variance reflecting that the log-normal model does not perfectly fit these elements. However, for Cu which is the main element of economic interest, the targeted variance was reached within a very close limit, as illustrated in Table 14-11.

Table 14-11 Summary of the Smoothing Correction Applied to the Cu Grade Estimates

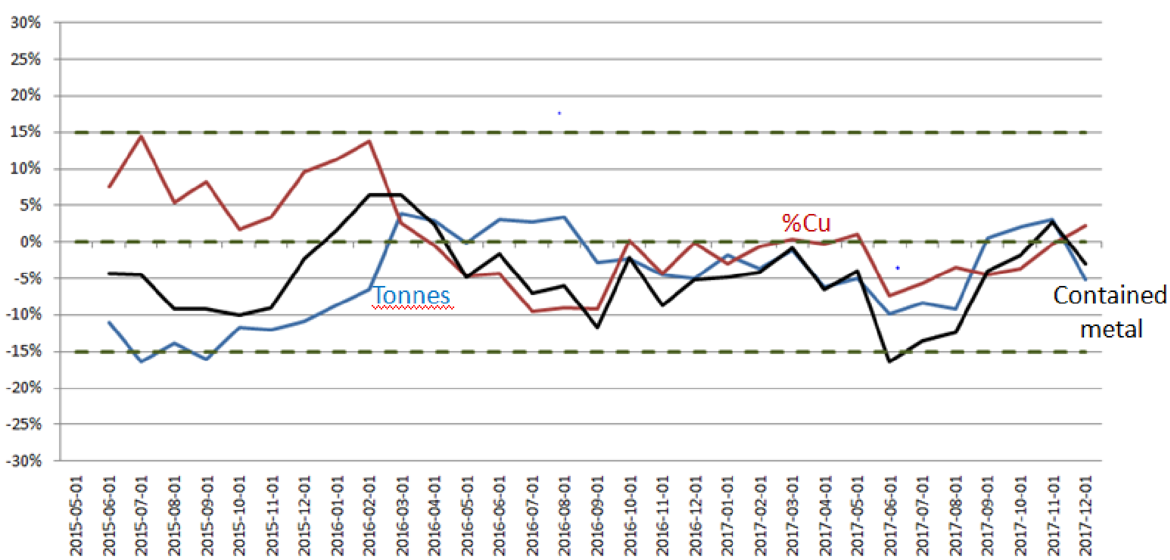
Zone	NN model variance	OK model variance	Theoritical varaince between 20m x 20m x15m blocks	Corrected model variance for 20m x 20m x15m blocks	Theoritical varaince between 10m x 10m x15m blocks	Corrected model variance for 10m x 10m x15m blocks
1	0.1948	0.0337	0.101	0.101	0.122	0.121
2	0.0350	0.0115	0.018	0.018	0.022	0.022
3	0.0403	0.0093	0.021	0.020	0.025	0.024
4	0.0349	0.0120	0.018	0.018	0.022	0.021
5	0.0348	0.0073	0.018	0.021	0.022	0.026
6	0.0332	0.0119	0.017	0.017	0.021	0.020
7	0.2347	0.0708	0.118	0.111	0.143	0.131
8	0.2743	0.0890	0.138	0.127	0.167	0.148
9	0.5239	0.0824	0.263	0.305	0.319	0.366

The corrected grade values must then be used for resource and reserve reporting.

Reconciliation Between the Updated Resource Model and Past Mine Production

Reconciliation conducted for the 2016-2017 period using the 2017 resource model (Figure 14-27) shows a significant improvement over the reconciliation results obtained from the 2016 model. The quarterly production is almost always predicted within +/-15% on a quarterly basis both for tonnes, grade and metal content. The resource model remains slightly on the conservative side but by less than 5% overall. The SMU dimensions adopted for this reconciliation work are 10mx10mx15m as per the current operating practice.

Figure 14-27 3 Months Rolling Average Reconciliation between the 2017 Resource Model and Mill Credited Production of the Mine



The initial variability observed in 2015 is related to the fact that during this period the mine was operating at a higher cut-off grade than at present resulting in a slight over-estimation of the grade and under-estimation of the tonnage in the model. But at the current operating cut-off, Figure 14-27 shows that the 2017 resource model is a reliable predictor of the quarterly production with a minimal conservative bias and constitute a significant improvement over the 2016 resource model. The improvement in these results can partly be attributed to the correction of the sampling bias in the Supergene Domain and to the correction of the over-smoothing naturally occuring in grade interpolation during the kriging process.

14.1.11 RESOURCE CLASSIFICATION

In Domains 1 & 2, the search ellipsoid used, as well as a number of other quality control parametres were recorded for each block in the model including: the number of samples and the number of holes used in the interpolation, the distance to the nearest hole and the average distance to all the composites used in the interpolation, the number of quadrants where samples were found, the kriging variance as well as the regression slope for each individual block estimate.

The resource classification index (RCI) used previously at Constancia was based on the relative difference between the kriged grade and the composites grades and uses the following formula:

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

Where 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:

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

RCI values corresponding to the 50th (0.123) and 90th (0.323) percentiles of the distribution of blocks with total copper grade above 0.05% were used as thresholds for the measured and indicated resource categories, respectively. Blocks were classified as inferred resources when their RCI was higher than 0.323.

For the present update of the resource model, this method was compared to the regression slope values obtained from the kriging of the copper grade estimates and also to the interpolation pass required. Results indicated a reasonable support for the RCI method, hence validating the previous classification system.

Table 14-12 Comparison of the Category Obtained from the Regression Slope and the Interpolation Pass for Cu Grade Interpolation

Category	Regression Slope value	Interpolation Pass
Measured	> 0.8	3rd pass (i.e. most restrictive)
Indicated	> 0.6 & < 0.8	2nd pass (i.e. intermediate)
Inferred	< 0.6	1st pass (i.e. less restrictive)

The first pass of resource classification at Constancia using the RCI is illustrated in Figure 14-28 showing blocks classified as measured in green, as indicated in yellow and as inferred in orange. In a second pass, 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. The final resource classification coded in the block model uses smooth and homogenous digitized solids of the Measured, Indicated and Inferred categories. Figure 14-29 represents only the solids of the Measured and Indicated resources. An Inferred envelope of all the remaining blocks was also created but is not shown on this figure. Proportions of measured and indicated category blocks were not changed significantly through this smoothing process.

Figure 14-28 First Pass Resource Classification Process

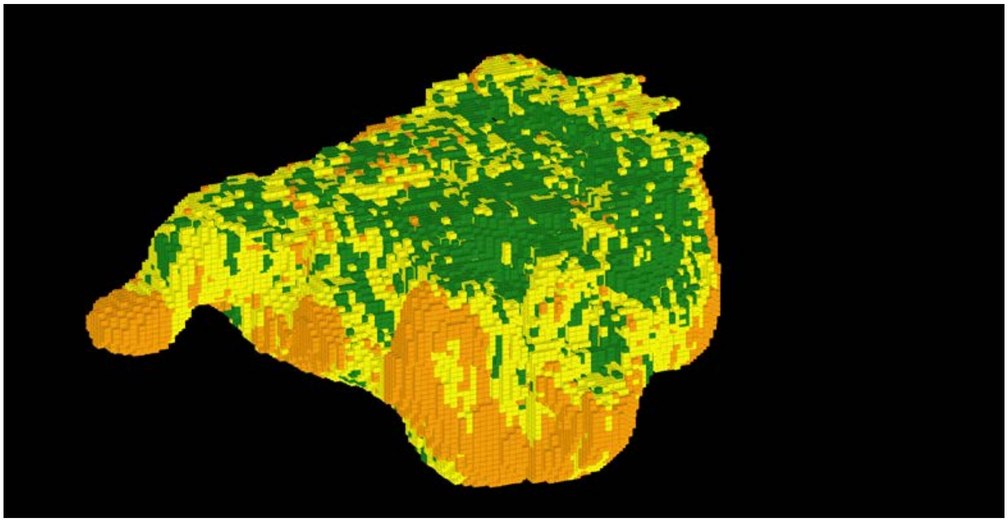
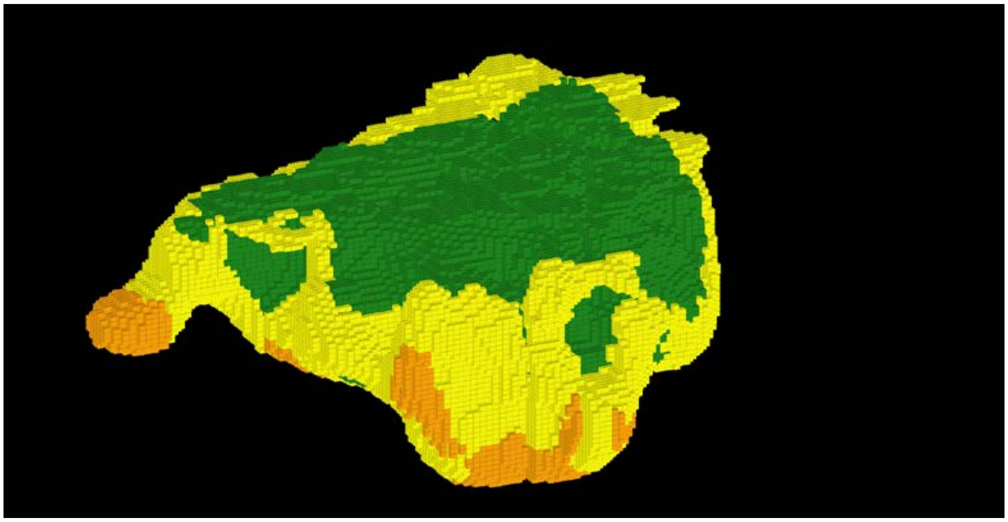


Figure 14-29   Final Resource Classification



The reconciliation results obtained from the rolling quarterly estimates confirm the validity of the classification scheme used by Hudbay at Constancia. The error on metal, tonnes and grade is within the +/-15% range on a quarterly basis with a 90% level of confidence as a mining industry benchmark to report measured resources. Measured resources constitute the vast majority of the material mined during the period represented in Table 14-27.

All resource estimates reported from Domain 3 were classified as inferred due to the large spacing, low grade and discontinuous nature of the mineralization.

Resource Reporting

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 the 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:

1. Economic benefit was applied to measured, indicated and inferred classified material within the resource cone.
2. A constant 45 degree pit slope (40 degree in the south west corner) was used for the resource estimate.
3. A haulage increment or bench discounting was applied to the blocks above (+ $0.004) and below (+ $0.01) RL 4185.

The Lerchs-Grossman cone is presented in Figure 14-30 and the input parametres are summarized in Table 14-13 to Table 14-16. The reporting of the mineral resource by Net Smelter Return (NSR) within the LG pit shell reflect the combined benefit of producing copper, molybdenum, silver and gold as per the following equations based on ore grades, recoveries, grade of concentrate, payable metals, deductions and royalties, in addition to mine operating and processing costs:

Table 14-13 Costs Inputs Pit Shell Construction

Cost		Units	Constancia
Mining Cost			
	Ore	$/tmined	1.35
	Waste	$/tmined	1.30
Variable Cost			
by Bench	Up	$/tmined	0.004
		Exit bench	4185
	Down	$/tmined	0.010
G&A Cost			
	Ore	$/ttrat	0.88
	Waste	$/tmined	-
Process Cost			
	Ore	$/ttrat	4.02

Table 14-14 Recovery by Material Type

Recovery	Supergene	Mixed	Hypogene	Skarn	HiZinc
Cu %	88.4%	88.4%	90.5%	84.4%	84.4%
Ag %	90.0%	90.0%	72.0%	52.0%	52.0%
Au %	60.0%	60.0%	60.0%	60.0%	60.0%
Mo %	55.0%	55.0%	55.0%	55.0%	55.0%
Pb %	41.4%	41.4%	41.4%	41.4%	41.4%
Zn %	80.0%	30.0%	30.0%	30.0%	30.0%

Table 14-15 Metal Price Assumption

Revenue	Units	Constancia
Metal Price		
Copper	$/lb	3.0
Molybdenum	$/lb	11.0
Silver	$/oz	18.0
Gold	$/oz	1,260

Figure 14-30 3D View of Resource Pit Shell, Looking North West

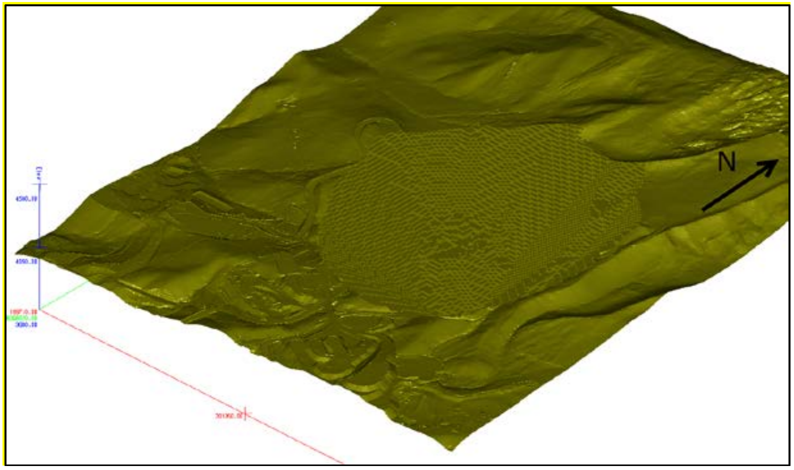


Table 14-16 Other Economical Considerations

Parameter	Units	Value
Payable Contained Metal		
Copper	%	96.5%
Molybdenum	%	100.0%
Silver	%	90.0%
Gold	%	90.0%
Concentrate grades		
Copper concentrate grade		
Copper	%	Cu lbs/(Cu lbs/0.30 + Zn lbs/0.55 + Pb lbs/0.70)
Silver	g/t	Ag ozs*31.1034/conc tonnes
Gold	g/t	Au ozs*31.1034/conc tonnes
Zinc	%	Pb lbs/2204.62262/conc tonnes
Lead	%	Zn lbs/2204.62262/conc tonnes
Moly concentrate grade		
Molybdenum	%	50.0%
Deductions		
Copper deduction	Per unit of pay Cu	1.00
Moly deduction	%Mo in Moly conc	1.00
Concentrate Moisture Cont		
Copper concentrate	%	8%
Moly concentrate	%	15%
Selling Cost		
Transport Cu conc	$/wmt conc	41.03
Transport Mo conc	$/wmt conc	175.00
Port charges conc	$/wmt conc	26.00
Insurance	$/wmt conc	5.00
Shipping Cu conc	$/wmt conc	25.00
Smelting Charges		
Smelting charges - Cu conc (dry)	$/dmt Cu conc	77.83
Roasting charges - Mo conc (dry)	$/dmt Mo conc	1,653
Refining charges		
Cu	$/lb Cu	0.078
Mo	$/lb Mo	1.300
Ag	$/oz Ag	0.450
Au	$/oz Au	4.500
Royalties		
Royalties	% of NSR	0.4%

14.1.12 MINERAL RESOURCE STATEMENT

Mineral resources for the Constancia 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, are summarized in Table 14-17 and Table 14-18. The Qualified Person for the mineral resource estimate is Cashel Meagher, P. Geo. Chief Operating Officer of Hudbay Minerals. The mineral resource estimates have an effective date of December 31st 2017.

Table 14-17 Constancia Mineral Resources Inclusive of Reserves(1),(2),(3),(4),(5),(6),(7),(8)

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	627,900,000	≥ $6.04	0.27	84	2.72	0.033
Indicated	253,700,000	≥ $6.04	0.21	61	2.38	0.034
Mea+Ind	**881,600,000**	≥ $6.04	**0.25**	**77**	**2.62**	**0.033**
Inferred	*54,100,000*	≥ $6.04	*0.24*	*43*	*1.71*	*0.018*

Table 14-18 Constancia Mineral Resources Exclusive of Reserves(1),(2),(3),(4),(5),(6),(7),(8)

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	175,000,000	≥ $6.04	0.20	51	2.19	0.028
Indicated	180,900,000	≥ $6.04	0.20	56	2.09	0.033
Mea+Ind	**355,900,000**	≥ $6.04	**0.20**	**54**	**2.14**	**0.030**
Inferred	54,100,000	≥ $6.04	0.24	43	1.71	0.018

Notes:

1. Raw drill hole assays were composited to 7.5 metres lengths broken at lithology boundaries.
2. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
3. Block grades for copper and the rest of elements were estimated from the composites using ordinary kriging interpolation 20 m x 20 m x 15m.
4. The interpolated block grade were adjusted to represent the level f selectivity corresponding to selective mining units of 10mx10mx15m above 3,930m and to selective mining units of 20mx20mx15m below 3,930m
5. Bulk Density was assigned by lithology, alteration zone and bench. Bulk density based on 1,247 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 Lerchs-Grossman algorithm. Metal prices of US$3.00/lb copper, US$11.00/lb molybdenum, US$18.00/troy oz silver and US$1260.00/troy oz gold. Metallurgical recoveries of 90.5% copper, 55% molybdenum, 72% silver and 60% gold were applied to sulfide material. Metallurgical recoveries of 88.4% copper, 55% molybdenum, 90% silver and 60% gold were applied to mixed and supergene material. A metallurgical recovery of 84.4% copper, 55% molybdenum, 52% silver and 60% gold for copper was applied to skarn and Hi Zinc material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, silver and gold 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 US$6.04/ton.
8. Totals may not add up correctly due to rounding.

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

Table 14-19 presents the comparison between the 2016 and the 2017 resource estimate for Constancia. The 2017 mineral resource estimates show a reduction in total tonnage and an increase in grade for copper and most of the other metals of economic interest compared to the 2016 mineral resource estimates. The relative reduction in tonnage is more pronounced in the inferred category and is due to the modelling process used in 2017 which was aimed at isolating large weakly mineralized areas from the main domain of hypogene mineralization. This approach has conversely resulted in an improvement in the average grade for all categories of resource estimates which enhance their potential for economic extraction after the mineral reserve estimates have been extracted. The improvement in the resource estimate grade was further compounded in the Supergene Domain by the correction of a sampling bias that negatively impacted the copper grade in previous mineral resource estimates.

Table 14-19 Comparison between the December 31st, 2016 and December 31st, 2017 Mineral Resource Estimates for Constancia

	Mineral Resource Reconciliation (Measured & Indicated)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	987,993,000	0.24	72	2.4	0.032	2,322,000
B	2017 Production (Depletion)	28,700,000	0.52	126	3.9	0.040	150,000
C	A - B	959,293,000	0.23	70	2.4	0.032	2,172,000
D	Mineral Resource as of December 31st 2017	881,600,000	0.25	77	2.6	0.033	2,247,000
E	Gain/(Loss) - New block Model[2]	(77,693,000)					75,000

	Mineral Resource Reconciliation (Inferred)	Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	138,069,000	0.17	40	1.7	0.018	289,000
B	2017 Production (Depletion)						
C	A - B	138,069,000	0.17	40	1.7	0.018	233,000
D	Mineral Resource as of December 31st 2017	54,100,000	0.24	43	1.7	0.018	127,000
E	Gain/(Loss) - New block Model[2]	(83,969,000)					(106,000)

Notes:
[1]Totals may not add up correctly due to rounding
[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

14.2 PAMPACANCHA

The previous Pampacancha mineral resource model was developed by Hudbay's Technical Group in Peru in 2013. Overall this previous model was found to constitute a conservative basis for reporting the reserve estimates. A review of the modeling methodology was undertaken and the following changes were adopted to update the model in 2017:

- revise the mineral envelopes to ensure a smooth and continuous modeling of the domain affected by the skarn mineralization without attempting to separate high grade and low-grade zones by hard contacts;

- use density weighting for both drillhole compositing and grade interpolation as there is a strong correlation between most metal grade and density;
- use more composites during grade interpolation resulting in more reliable but smoother block grade estimates; and
- implement a change of support to obtain unbiased grade-tonnage curves at the selected mining unit size and cut-off grade.

14.2.1 RESOURCE MODELING DATABASE

The database used for the 2017 update of the resource model at Pampacancha is the same as used in previous versions and includes 140 drill holes totalling approximately 38,240 metres (Table 14-20). The drillhole database was provided in Microsoft Access® format. The files were imported as collar, downhole survey and assay data into MineSight®.

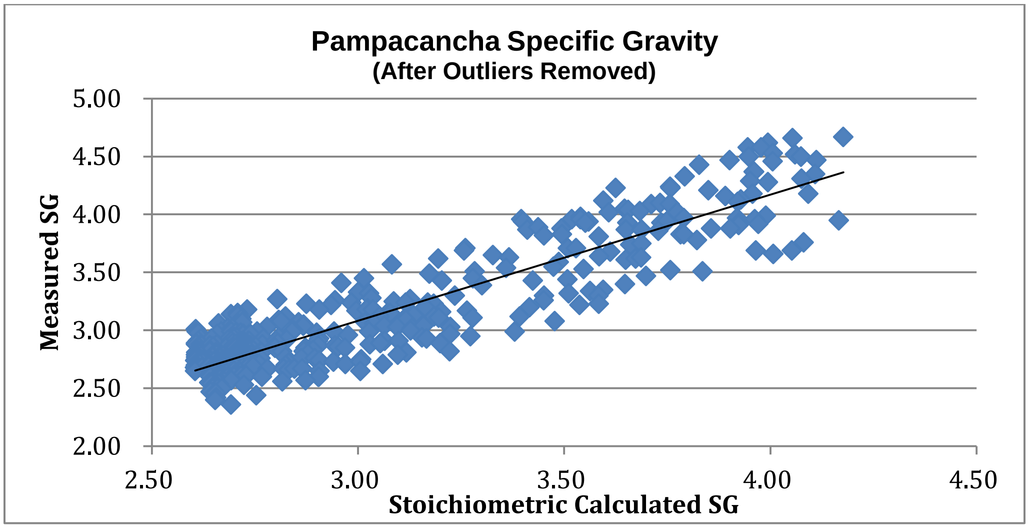
Table 14-20 Drilling Data Used for Pampacancha Resource Modeling

Company	Time Period	Number of Drill Holes	Total Length (in metres)
Norsemont	2008	6	1,652
Norsemont	2009	9	2,679
Norsemont	2010	35	8,403
Hudbay	2011	57	17,310
Hudbay	2012	32	7,999
Hudbay	2013	1	197
Summary		**140**	**38,240**

All drill holes have full chemistry obtained from ICP analysis while approximately 14% were also analysed for sequential copper, i.e. sulphuric (CuSS) and cyanide (CuCN) soluble copper. The ICP was considered as the most reliable measure of total copper and whenever the sum of CuSS+CuCN was higher than Cu, the CuSS and CuCN values were rescaled so as to not change their relative proportion while ensuring that their sum did not exceed total copper measured from ICP analysis.

Specific Gravity (SG) was measured for 633 samples or approximately 2% of the total sampled length. A stochiometric formula used to assign SG to the samples 2013 resource model was checked and validated against the actual SG measurements in the skarn mineralized domain, which is the only zone hosting significant mineralization (Figure 14-31).

Figure 14-31 Comparison of Stoichiometric Calculated and Measured Sg Values



The stoichiometric method used in 2013 calculates the percentage of chalcopyrite, sphalerite, galena, magnetite, pyrite and arsenopyrite from the chemical composition of each sample. Stoichiometric (calculated) density proportions of each metal by mineral type are shown in

Table 14-21. The percentage of elemental copper, lead, iron, sulphur and arsenic is added where the total for each sample must add to 100% so the remainder (one, minus total metals measured) was assumed to be gangue material.

Table 14-21 Proportion of Metal in Minerals

Mineral	Density	Formula	Cu	Zn	Pb	Fe	S	As
Sphalerite	4.00	ZnS		0.671			0.329	
Pyrite	5.00	FeS2				0.466	0.534	
Chalcopyrite	4.20	CuFeS2	0.346			0.304	0.35	
Pyrrhotite	4.60	Fe1-xS				0.635	0.365	
Galena	7.50	PbS			0.866		0.134	
Arsenopyrite	6.00	FeAsS				0.343	0.197	0.46
Gange	2.60	Qtz, feldspar						
Magnetite	5.17	Fe3O4				0.724	0.276	
Andradite	3.90	Ca3Fe2Si3O12				0.22		
Bornite	5.10	Cu5FeS4	0.633			0.111	0.256	

The SG calculated from the stochiometric formula presents a very strong linear correlation with %Fe and a simple linear regression from the iron grade, which can be easily be used to ensure a SG value was assigned to all the samples located within the mineralized envelope used for grade interpolation.

14.2.2 WIREFRAME MODEL AND MINERALIZATION

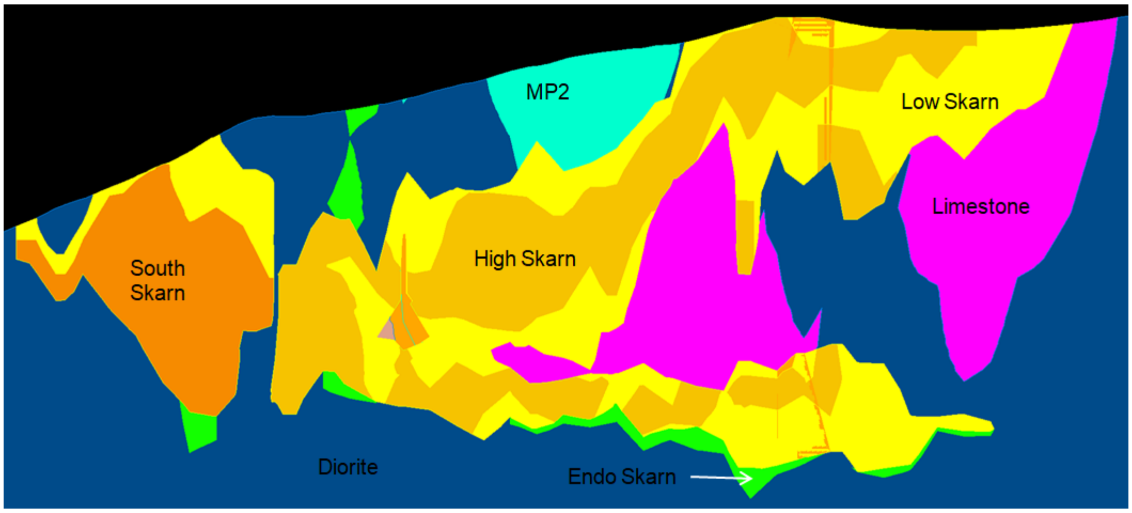
At Pampacancha, the skarn is the most important lithology and hosts the majority of the copper mineralization. A magnetite alteration model was investigated due to the presence of high grade copper often associated with magnetite. However, the relationship between copper and magnetite was found to be discontinuous.

The skarn mineralization is located at the contact between a limestone, a semi-circle shaped diorite and a monzonite porphyry (MP2) and strikes approximately N160. Porphyry dDiorite dikes are interpreted as the source of the copper mineralization. For this revised geological model, all occurrences of diorite were considered as the same rock type.

In the 2013 model, the interpretation of irregular zones of low grade and high-grade skarn mineralization on cross-sections created envelopes with inconsistent shapes in plan and longitudinal views (Figure 14-32). As for Constancia, the necessity to back-tag samples was an implicit acknowledgement of the difficulty to model these bodies as discrete solids.

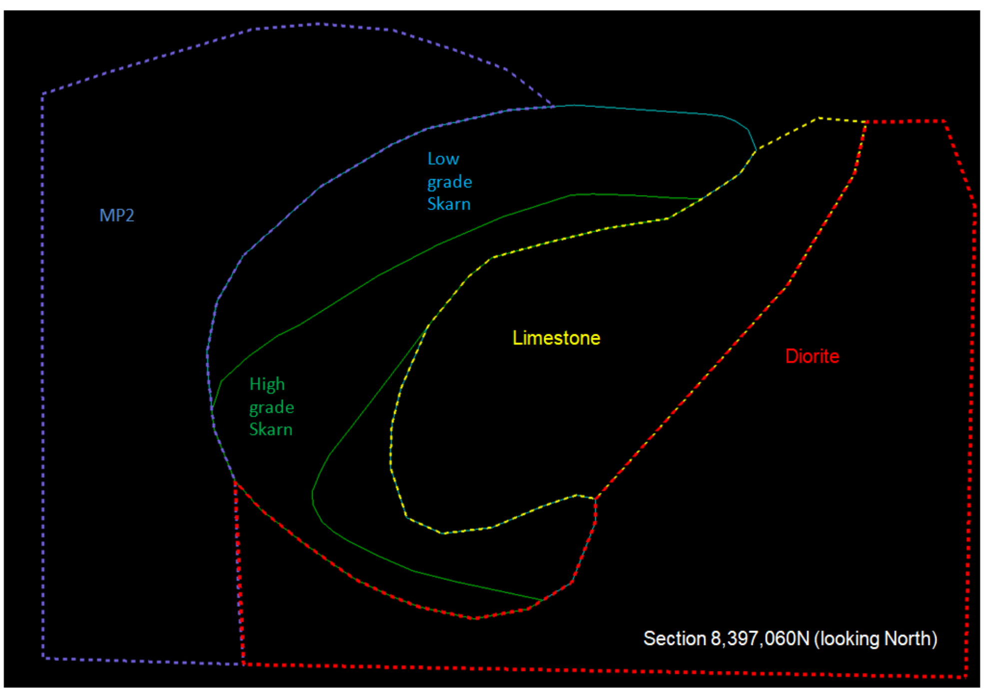
For the 2017 revision of the model, the geological framework for grade interpolation was simplified and made more robust by drawing a smoother and continuous envelope of the skarn mineralization generally grading above 0.1% Cu. The original interpretation of the diorite and limestone bodies was used as a guide to limit the skarn mineral envelope on all sides (Figure 14-33). A high-grade skarn envelope was also modelled but was used only for the smoothing correction, i.e. change of support, and was not applied as a hard boundary during grade interpolation.

Figure 14-32 Longitudinal Section through the 2013 Pampacancha Model



Small areas generally grading above 0.1% Cu but located outside of the skarn mineral envelopes were also interpolated but any resource reported from these small isolated volumes is classified as inferred and requires further exploration. These inferred resources outside of the skarn mineralization (mostly in diorite) represent a very small proportion of the total resource estimates at Pampacancha.

Figure 14-33 Simplified Geological Framework for the 2017 Long-Term Model



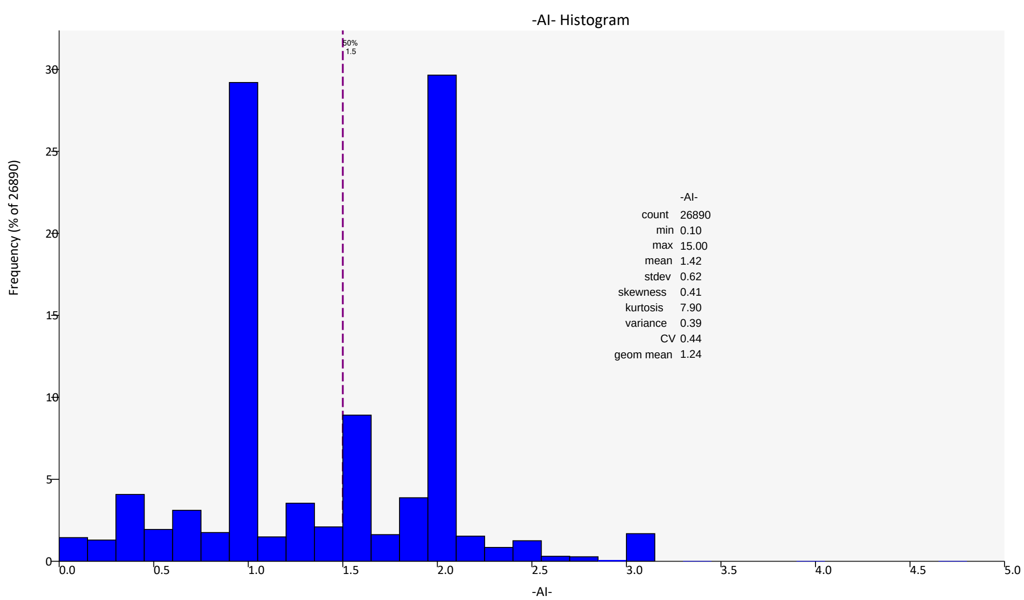
14.2.3 DRILL HOLE COMPOSITING, EXPLORATORY DATA ANALYSIS AND GRADE CAPPING

In order to normalize the weight of influence for each sample, assay intervals were regularized by compositing the drill hole data into equal length intervals using domain boundaries to break composites. Although the samples were assayed over a shorter length (Figure 14-34), the compositing length was set to 7.5m when considering the envisaged mining block height of 15 m.

Compositing over a longer interval than the original assay results in a loss of detail and resolution in the downhole grade distribution but does not impact the quality of the block estimates with a 15m height. Composites with a length lower than 3.75m were discarded. Compositing was weighted by specific gravity which has a material impact for most of the elements in the skarn mineralized envelope.

Assay intervals were also composited into 15 m lengths (+/- 7.5 m of threshold) using the same methodology. The 15 m composites were used to estimate nearest neighbour models for a global bias check and smoothing assessment.

Figure 14-34 Frequency Distribution of the Original Assay Intervals



The exploratory analysis was conducted by element within a single mineralized skarn domain. This analysis was then used as a basis to establish whether weighting the interpolation by SG could have a significant impact and if grade capping was required.

Figure 14-35 to Figure 14-37 illustrate the main statistical properties of some of the key metals considered in the skarn mineral envelope. With the exception of specific gravity, all the metals present a skewed histogram, reflecting the non-selective nature of the mineral envelope.

Weighting by specific gravity has a very significant impact on the Cu, Fe and As grade with a reduced impact on most of the other elements.

Capping was considered due to the skewed nature of the frequency distributions. The approach to define an appropriate amount of metal at risk was based on Constancia, on defining a population of outliers as the highest grading of 1% for each metal. For these outliers, the mean and median values were compared and the natural breaks in the cumulative histogram of the outliers were checked to identify natural slope changes.

For all the elements, the outliers represent less than 10% of the total metal content and the metal at risk was deemed low and no capping was applied for the modeling of Pampacancha.

Figure 14-35 Exploratory Analysis of %Cu

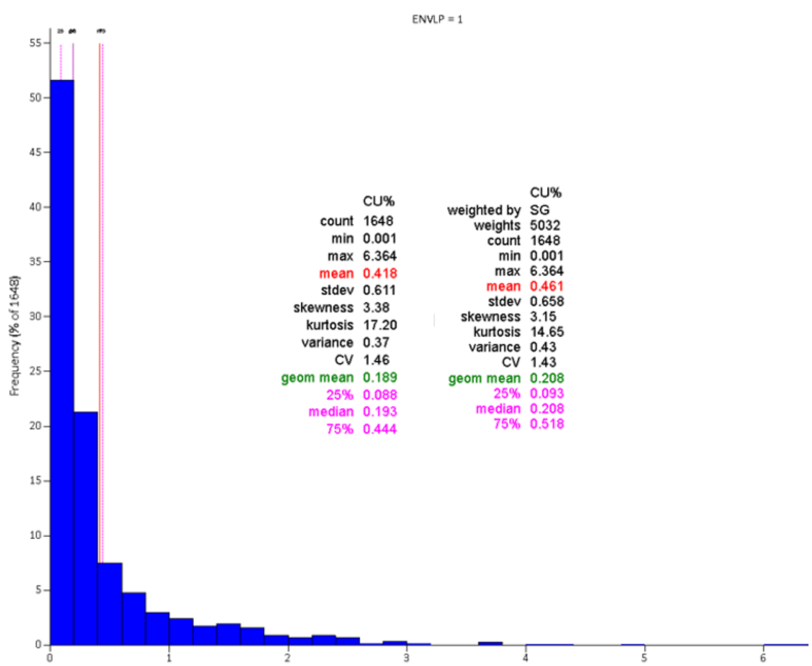


Figure 14-36 Exploratory Analysis of %Zn

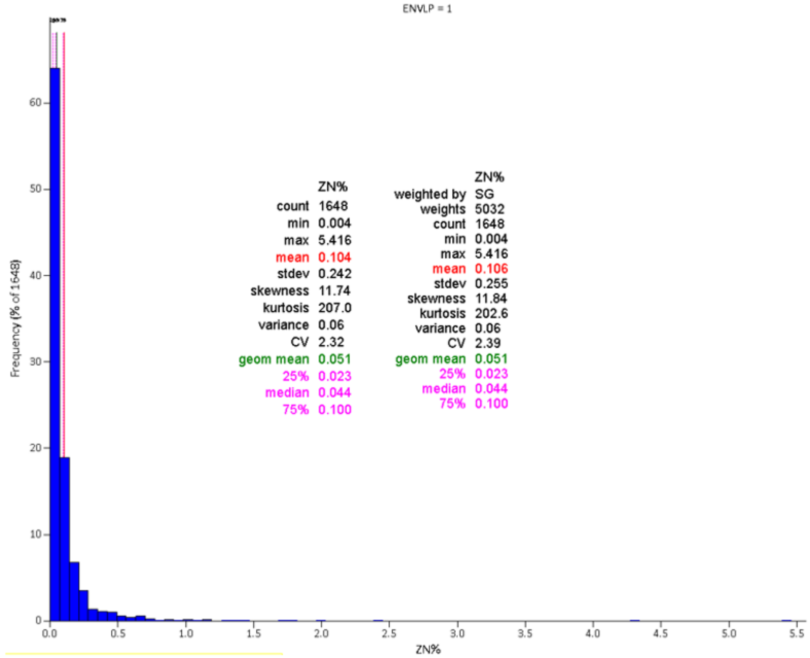
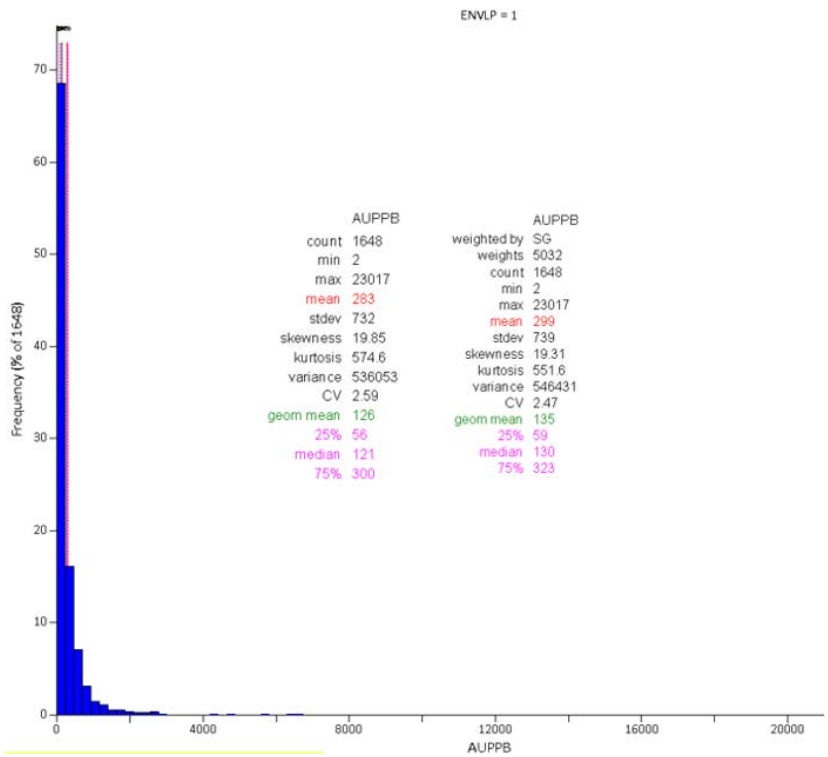


Figure 14-37 Exploratory Analysis of G/T Au



14.2.5 VARIOGRAPHY

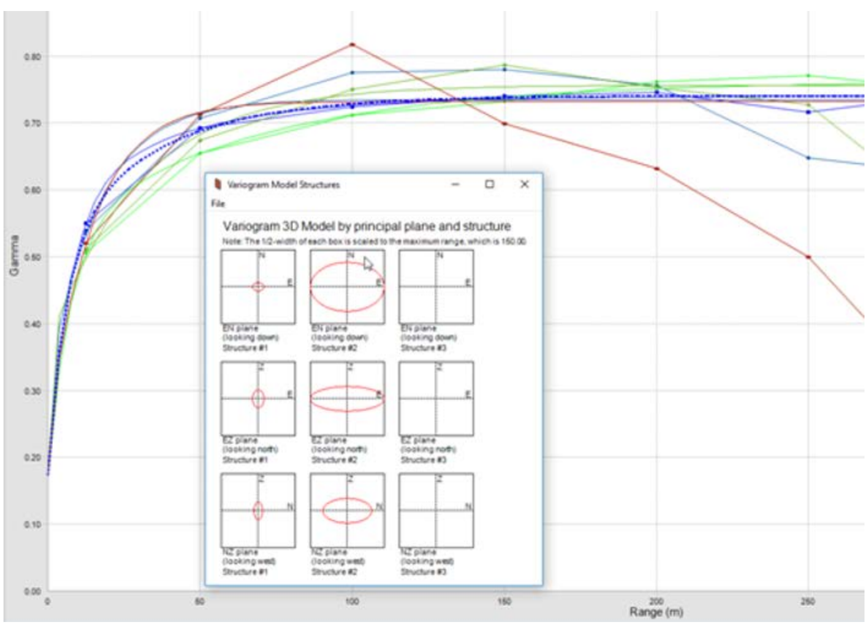
Pairwise relative variograms were calculated for each element within the mineral envelope. A linear combination of a nugget effect and two exponential (spherical, in the case of Fe) structures were fitted. The variogram models adopted for all the metals interpolated in the resource model are summarised in Table 14-22.

Table 14-22 Variogram Models for Pampacancha

			Structure #1				Structure #2			
	Model Type	Nugget	Major Axis	Semi Major Axis	Minor Axis	Sill S1	Major Axis	Semi Major Axis	Minor Axis	Sill S2
Cu	Exp	0.1732	23	17	35	0.3317	150	100	50	0.2358
CuSS	Exp	0.1136	40	20	15	0.3061	150	125	75	0.3768
CuCN	Exp	0.04053	65	60	20	0.429	200	110	50	0.2488
Mo	Exp	0.3366	38	15	48	0.4247	200	150	100	0.3357
Ag	Exp	0.1605	40	30	20	0.3048	200	150	100	0.1916
Au	Exp	0.2093	30	10	20	0.2549	120	80	60	0.2364
As	Exp	0.1863	30	30	20	0.3015	120	100	50	0.1792
Pb	Exp	0.2897	40	35	20	0.3901	200	100	60	0.09412
Zn	Exp	0.1576	30	20	25	0.3408	120	120	60	0.07938
Fe	Sph	0.1453	30	20	26	0.2283	110	110	100	0.1305
SG	*Exp*	*0.001524*	*50*	*26*	*52*	*0.007759*	*90*	*90*	*50*	*0.003316*

Figure 14-38 illustrates the Cu variogram with the best direction of continuity being in the vertical direction followed by the N-S direction and with the E-W direction showing the most variability.

Figure 14-38 Cu Variogram at Pampacancha



14.2.6 GRADE ESTIMATION

A block model prototype with cells 20 m x 20 m x 15 m was created in order to entirely fill the mineralized domain. The block size was chosen such that geological contacts are reasonably well reflected and to support a large-scale mining scenario while taking into consideration the DDH spacing.

There is no cell splitting in the model as it would provide an artificial level of details which is not available from the current drill spacing and is not necessary for long-term mine planning. The interpolation plan was completed on the composites, 7.5 m in length, via ordinary kriging (OK).

All the OK grade estimates were weighted by specific gravity. Grade estimation used a composite and block matching scheme based on the envelope code which is treated as a hard boundary for grade interpolation purposes.

The interpolation process used a nested search strategy with three passes of increasing ellipsoid radius. The composite selection parametres for grade estimation in each domain (minimum, maximum and, maximum number of composites per hole) were selected to minimize bias. Table 14-23 Initial Search Ellipse Parametres displays the initial search distances and search ellipse orientations used for all the interpolated elements.

Table 14-23 Initial Search Ellipse Parametres

	CU%	MOPPM	AGPPM	AUPPB	SG	ZN%	FE%	PB%	ASPPM	CUSS	CUCN
West-East	75	100	100	60	45	60	55	100	60	75	100
North-East	50	75	75	40	45	60	55	50	50	62.5	55
Vertical	25	50	50	30	25	30	50	30	25	37.5	25

This first interpolation pass is restricted to a minimum of 16 composites and a maximum of 32 composites (without a maximum of six composites per hole) with quadrant declustering. For the blocks that did not meet this criteria, a second interpolation pass is restricted to a minimum of 16 composites and a maximum of 32 composites (with a maximum of six composites per hole) and quadrant declustering but with a search radius increased by 50% in all directions. Finally, in a third pass, the blocks not yet estimated were interpolated inside a search ellipsoid with a radius doubled from the second search (3 x the initial search) with a minimum of one composite and a maximum of 32 composites (with a maximum of six composites per hole) without quadrant declustering.

14.2.7 BLOCK MODEL VALIDATION

The final block model was validated to ensure:

- appropriate honouring of the input data but acknowledging that some natural smoothing should occur between samples and that the grade of a sample in the middle of a block is not the average grade of the block;
- absence of global bias by comparing the mean grade estimated by kriging to the original composite average grade and a declustered grade obtained from a nearest neighbour interpolation; and
- assessment of the level of smoothing in the model for the selected Selective Mining Unit (SMU) dimensions considered for reserve reporting.

Honouring the DDH Data

Systematic visual inspection of block grade versus composited data was conducted in section and plan view. This check confirmed a good reproduction of the data by the model. Examples of east-west and south-north oriented cross section are provided in Figure 14-39 and Figure 14-40.

Check for Absence of Global Bias

This validation step consists of comparing the global average grade of each element (after capping for gold and silver) between the original composites, the kriged block estimates and the nearest neighbour estimates.

Figure 14-39 Example of Visual Validation of Grade Interpolation on E-W Cross-Section

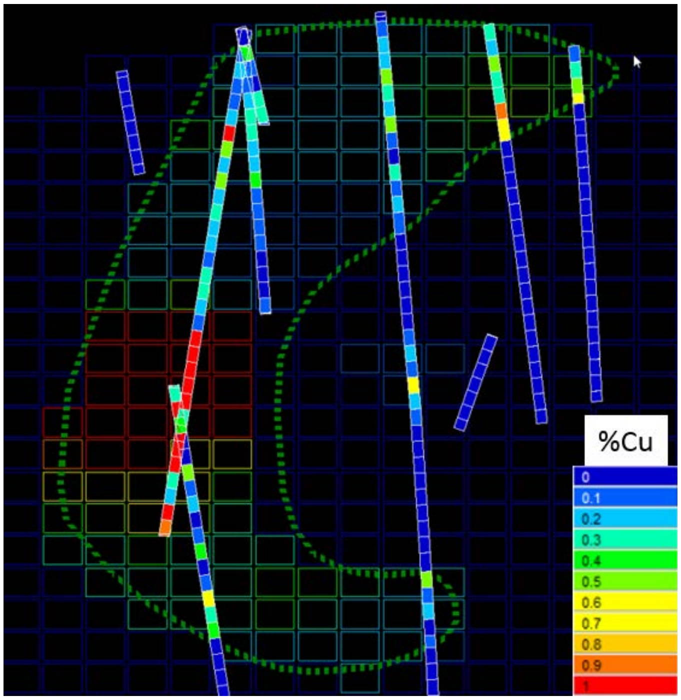
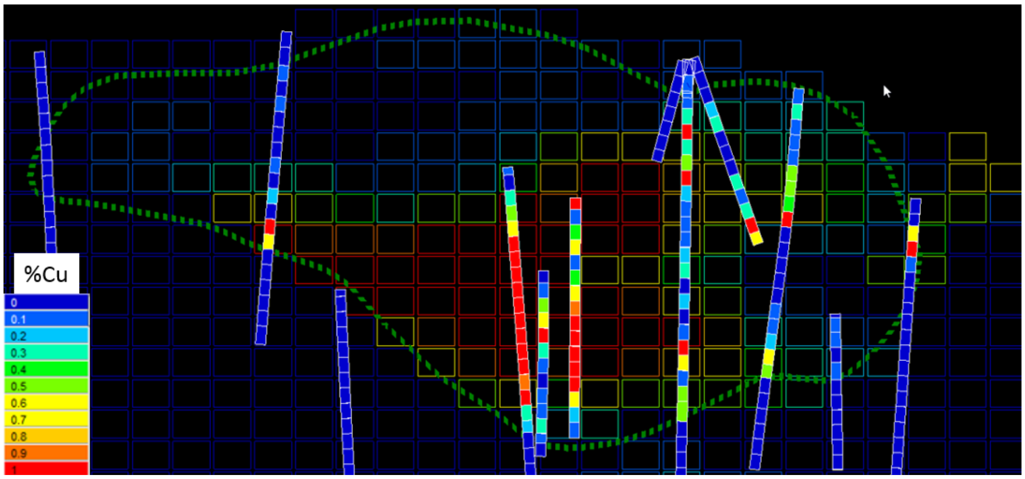


Figure 14-40 Example of Visual Validation of Grade Interpolation on S-N Cross-Section



A nearest neighbour interpolation is equivalent to the declustered statistics of the composites based on weighting each composite by its polygon of influence. The average grade obtained from this method is a useful benchmark but not a perfect one, as it fails to incorporate the nugget effect measured by the variogram. The higher the nugget effect, the closer the average grade should be to the mean of the composites. Ordinary kriging is in fact the best method of declustering.

A valid global check consists therefore in verifying that the kriged mean block estimate was located between the mean of the composites and the mean of the nearest neighbour model.

Table 14-24 represents the results of these global checks for each element revealing no significant issues for any of the interpolated variables. The impact of declustering has a significant impact on the mean grade for most of the economic metals associated with the sulphide mineralization and also for CuSS and CuCN. This is a fairly common result as higher grade zones tend to be over sampled compared to lower grade zones. All the statistics presented in Table 14-24 are weighted by specific gravity.

Table 14-24 Check for Global Bias

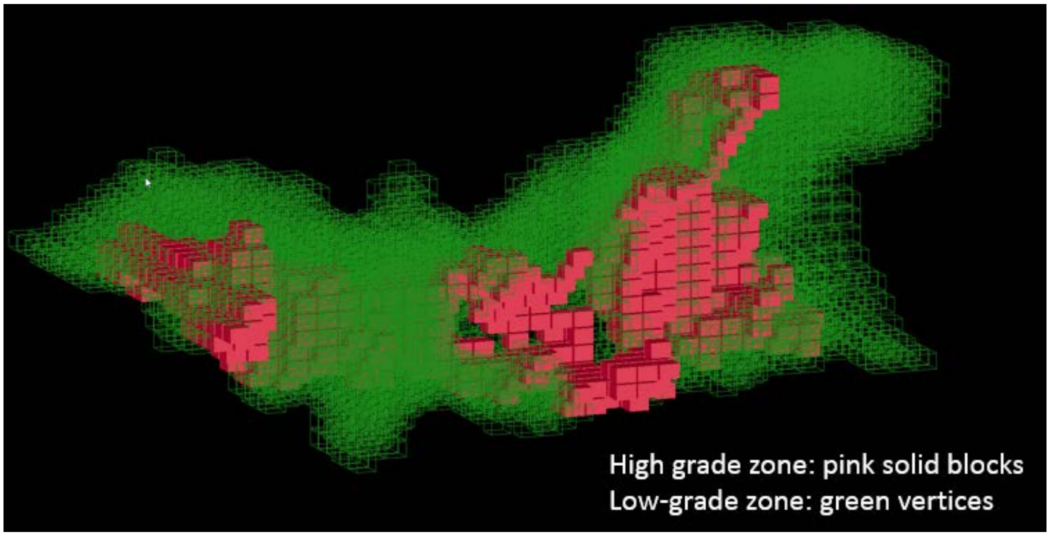
Grade	Composites	OK Model	NN Model
Cu	0.461	0.464	0.449
CuSS	0.077	0.065	0.058
CuCN	0.169	0.148	0.144
Mo	131	138	144.655
Ag	4.51	4.67	4.6
Au	299	301	301.6
Pb	0.032	0.034	0.032
Zn	0.106	0.112	0.111
As	41	44.8	45
Fe	17.654	17.925	17.66
SG	3.05	3.09	3.06

Smoothing Assessment

The visual validation conducted in section and plan views (see examples on Figure 14-41 and Figure 14-42) confirmed that the block grade interpolation was consistent with the supporting composite data after capping. The larger number of composites used for grade estimation in the new model significantly improves the individual block grade estimates but at the same time results in a much smoother model requiring a careful assessment.

The extent of grade 'over-smoothing' in the model was investigated separately in two separate zones in order to take into consideration domains with material differences in grade distribution as illustrated in Figure 14-40.

Figure 14-41 Domains Used for Smoothing Assessment And Correction



The mean and variance of the kriged estimates were compared separately in each zone with the variance of the composites after declustering (obtained from a nearest neighbour interpolation). The expected true variance between SMUs can be calculated from the variogram models summarized in Table 14-22.

The results are illustrated in Table 14-25 for %Cu. The expected variance between blocks 20mx20mx15m is 0.102 or approximately 54% of the total declustered variance of the composites in the low-grade zone. The OK model actually produced estimates with a variance of 0.061 or only 28% of the nearest neighbour variance. This implies that the true variance between blocks should be approximately 1.6 times larger in this zone than the one obtained from the OK estimates. The over-smoothing is slightly higher in the high-grade zone. This smoothing is a normal outcome of a sound interpolation method when the drill spacing is not sufficient to address the short-range variability in the metal grade distribution. Smoothing will gradually reduce as additional infill drilling is performed during the pre-production period.

This exercise was repeated for all the elements used for mine planning including Cu, Au, Ag, Mo, Fe, Zn, Pb, As, CuSS and CuCN.

Table 14-25 Summary of the Smoothing Assessment by Zone for %Cu

Zone	NN Model variance	OK Model variance	Theoretical variance between 20mx20mx15m blocks	Over-smoothing ratio for 20mx20mx15m blocks
Low-grade	0.188	0.061	0.102	1.66
High-grade	0.618	0.175	0.334	1.90

Smoothing Correction for Resource and Reserve Reporting

Obviously, using the smooth kriged estimate will result in an erroneous grade-tonnage curve and reporting resources or reserves at a cut-off grade different than 0% would result in biased estimates, usually over-estimating tonnes and under-estimating grade.

An indirect log-normal correction was used to perform a change of support on the kriged model in order to obtain unbiased grade tonnage curves within the future pit. This correction is only valid globally and provides poorer local estimates than the smoothed kriged model but does not materially alter the global average grade within each zone and provides the correct grade-tonnage curve for the variogram models fitted on the drillhole data.

For some of the elements, the correction did not fully attain the targeted variance reflecting that the log-normal model does not perfectly fit these elements. However, for Cu which is the main element of economic interest, the targeted variance was reached within a very close limit as illustrated in Table 14-26.

Table 14-26 Summary of the Smoothing Correction Applied to the Cu Grade Estimates

Zone	NN Model variance	OK Model variance	Theoretical variance between 20mx20mx15m blocks	Corrected OK model variance between 20mx20mx15m blocks
Low-grade	0.188	0.061	0.102	0.094
High-grade	0.618	0.175	0.334	0.319

The corrected grade values must then be used for resource and reserve reporting.

14.2.8 MINERAL RESOURCE CLASSIFICATION

The search ellipsoid used, as well as a number of other quality control parametres were recorded for each block in the model including: the number of samples and the number of holes used in the interpolation, the distance to the nearest hole and the average distance to all the composites used in the interpolation, the number of quadrants where samples were found, the kriging variance as well as the regression slope for each individual block estimate.

Similarly to Constancia, the regression slope values obtained from the kriging of the copper grade estimates were used as a basis for resource classification. 90% and 80% regression slope thresholds were used respectively to separate measured from indicated and indicated from inferred resources. Since no mining has yet occurred at Pampacancha, the criteria for defining measured and indicated resources is higher than at Constancia which used 80% and 60% thresholds respectively on the kriging regression slope.

The first pass of resource classification at Pampacanca is illustrated on Figure 14-42 showing blocks classified as Measured in green, as Indicated in yellow and as Inferred in orange. In a second pass, 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.

Figure 14-42 First Pass Resource Classification

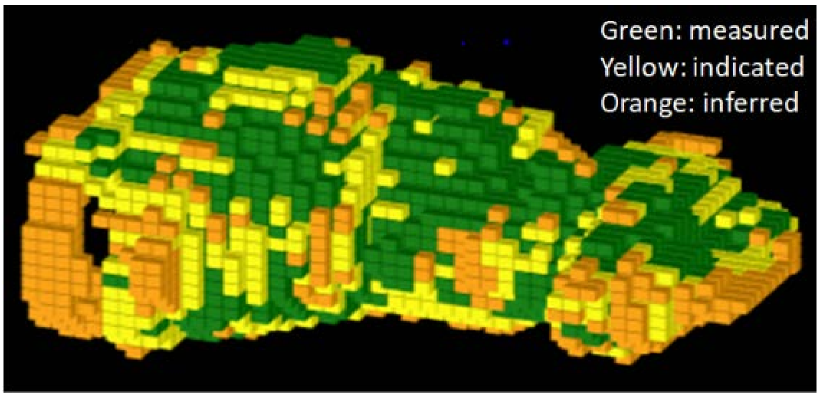
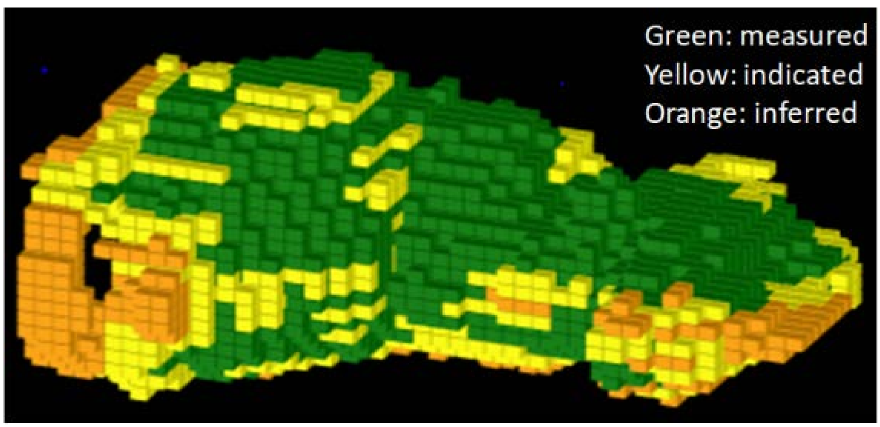


Figure 14-43 represents only the solid measured and indicated resources simply for clarity purposes. An inferred solid englobing all the remaining blocks was also created but is not shown in this figure. Proportions of measured and indicated category blocks were not changed significantly by this smoothing process.

Figure 14-43 Final Resource Classification



14.2.9 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 the 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:

1. Economic benefit was applied to measured, indicated and inferred classified material within the resource cone.
2. A constant 45 degree pit slope was used for the resource estimate.
3. A haulage increment or bench discounting was applied to the blocks above (+ \$0.004) and below (+ \$0.01) RL 4305.
4. The resource estimate was not limited by any property or permit constraints.

The Lerchs-Grossman cone is presented in Figure 14-44 and the input parametres as those used at Constancia for metal price assumptions and for the parametres used in the construction of the pipt shell. However, the cost and recovery assumptions summarized in Table 14-27 and Table 14-28, and are specific for Pampacancha. The reporting of the mineral resource by Net Smelter Return (NSR) within the LG pit shell reflect the combined benefit of producing copper, molybdenum, silver and gold as per the following equations based on ore grades, recoveries, grade of concentrate, payable metals, deductions and royalties, in addition to mine operating and processing costs.

Figure 14-44 3D View of Resource Pit Shell, Looking North West

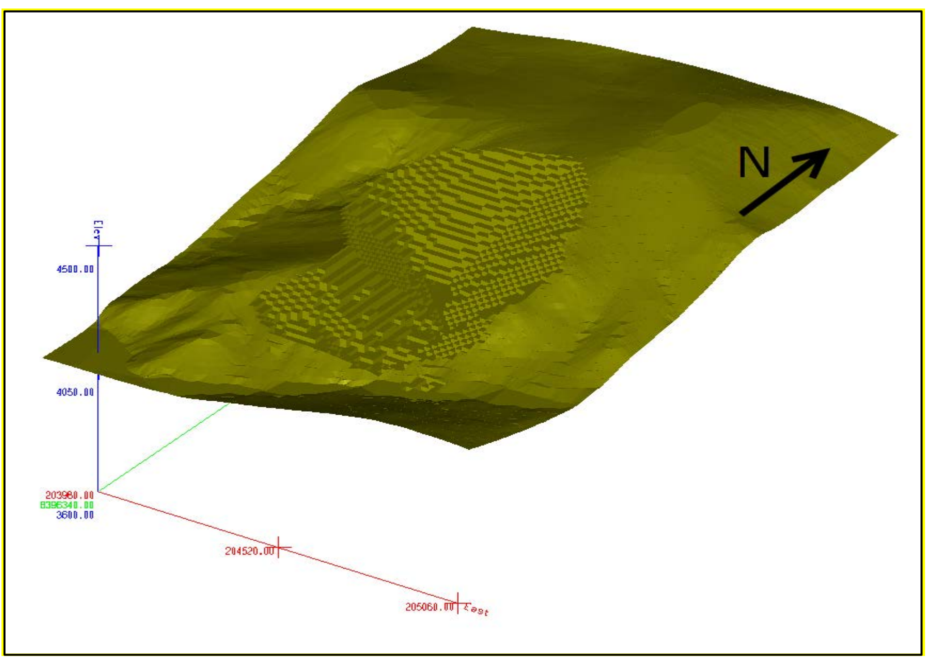


Table 14-27 Cost Inputs for Pit Shell Construction

Cost		Units	Pampacancha
Mining Cost			
	Ore	$/tmined	1.85
	Waste	$/tmined	1.55
Variable Cost			
by Bench	Up	$/tmined	0.004
		Exit bench	4305
	Down	$/tmined	0.010
G&A Cost			
	Ore	$/ttrat	0.88
	Waste	$/tmined	-
Process Cost			
	Ore	$/ttrat	4.02

Table 14-28 Recovery vs. Material Assumed for Pit Shell Construction

Recovery	Skarn PC
Cu %	85.0%
Ag %	70.0%
Au %	70.0%
Mo %	40.0%
Pb %	41.0%
Zn %	30.0%

14.2.10 MINERAL RESOURCE STATEMENT

Mineral resources for the Pampacancha deposit were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves[5] 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 are summarized in Table 14-29 and Table 14-30. The Qualified Person for the mineral resource estimate is Cashel Meagher, P. Geo, Chief Operating Officer of Hudbay Minerals. The mineral resources are reported using the long-term metal price assumptions and have an effective date of December 31st, 2017.

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

Table 14-29 Pampacancha Mineral Resource inclusive of Reserves(1),(2),(3),(4),(5),(6),(7)

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	43,900,000	≥ $6.04	0.55	158	4.60	0.336
Indicated	13,400,000	≥ $6.04	0.50	133	5.49	0.307
Mea+Ind	**57,300,000**	≥ $6.04	**0.54**	**152**	**4.81**	**0.329**
Inferred	10,100,000	≥ $6.04	0.14	143	3.86	0.233

Table 14-30 Pampacancha Mineral Resource exclusive of Reserves(1),(2),(3),(4),(5),(6),(7)

Category	Tonnes	NSR Cut-Off	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)
Measured	11,500,000	≥ $6.04	0.42	101	4.95	0.245
Indicated	5,900,000	≥ $6.04	0.35	84	5.15	0.284
Mea+Ind	**17,400,000**	≥ $6.04	**0.39**	**95**	**5.02**	**0.258**
Inferred	10,100,000	≥ $6.04	0.14	143	3.86	0.233

Notes:

1. Raw drill hole assays were composited to 7.5 metres lengths broken at lithology boundaries.
2. Capping of high grades was considered necessary and was completed for each domain on assays prior to compositing.
3. Block grades for copper and the rest of elements were estimated from the composites using ordinary kriging interpolation into 20 m x 20 m x 15m.
4. Bulk Density was assigned by lithology, alteration zone and bench. Bulk density based on 583 measurements collected by Hudbay and previous operators.
5. Blocks were classified as Measured, Indicated or Inferred in accordance with CIM Definition Standards 2014.
6. Mineral resources are constrained within a computer generated pit using the Lerchs-Grossman algorithm. Metal prices of US$3.00/lb copper, US$11.00/lb molybdenum, US$18.00/troy oz silver and US$1260.00/troy oz gold. Metallurgical recoveries of 85% copper, 40% molybdenum, 70% silver, 70% gold, 41% lead and 30% zinc were applied to skarn material. NSR was calculated for every model block and is an estimate of recovered economic value of copper, molybdenum, silver and gold 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 US$6.04/ton.
7. Totals may not add up correctly due to rounding.

Table 14-31 presents the comparion between the 2016 and the 2017 resource estimate for Pampacancha. The 2017 Pampacancha mineral resource estimates show a small reduction in inferred resource tonnage but are largely offset by a significant increase in grade for copper and for most of the other metals of economic interest compared to the 2016 mineral resource estimates. The reduction in tonnage is due to the modelling process used in 2017 which was aimed at reducing the smearing of isolated high grade samples into low grade areas. This approach has conversely resulted in an improvement in the average grade for all categories of resource estimates also enhanced by the application of proper specific gravity weighting during the drillhole compositing and grade interpolation stages of the modelling process.

Table 14-31 Comparison between the December 31st, 2016 and December 31st, 2017 Mineral Resource Estimates for Pampacancha

Mineral Resource Reconciliation (Measured & Indicated)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	65,746,000	0.40	129	3.9	0.249	289,000
B	2017 Production (Depletion)						
C	A - B	65,746,000	0.40	129	3.9	0.249	263,000
F	Mineral Resource as of December 31st 2017	57,309,200	0.54	152	4.8	0.329	307,000
E	Gain/(Loss) - New block Model[2]	(8,436,800)					44,000

Mineral Resource Reconciliation (Inferred)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2016 Mineral Resource as of December 31st 2016	191,000	0.14	92	3.2	0.189	300
B	2017 Production (Depletion)						
C	A - B	191,000	0.14	92	3.2	0.189	300
F	Mineral Resource as of December 31st 2017	10,100,000	0.14	143	3.9	0.233	14,400
E	Gain/(Loss) - New block Model[2]	9,909,000					14,100

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

15 MINERAL RESERVE ESTIMATES

The Mineral Reserves are contained within two open pit deposits (Constancia and Pampacancha) and surface stockpiles. Proven and Probable Mineral Reserves for the two deposits as of December 31st, 2017 are estimated to be 568.6 Mt grading 0.41% CuEq with a 19 year mine life. A complete mine planning study has been developed in order to define the updated mineral reserves.

The Mineral Reserve estimates for Constancia Operations are based on the resource models described in Section 14 (Mineral Resource Estimates), Net Smelter Return, and the detailed Pit Designs.

Mining, processing, and economic parametres have been used to develop the Mineral Reserve estimates for the Constancia Operations.

This Mineral Reserve estimates have been determined and reported in accordance with NI 43-101 and the classifications adopted by CIM (2014). NI 43-101 defines a Mineral Reserve as “the economically mineable part of a Measured and/or Indicated Mineral Resource”.

15.1 PIT OPTIMIZATION

Pit optimization of multi-element revenues at Constancia and Pampacancha can either be performed on a grade equivalent basis of all the revenue generating elements expressed in terms of the predominant metal (copper in the case of Constancia and Pampacancha) or on the Net Smelter Return (NSR in dollars).

A copper grade equivalent optimization (Cu-eq.) model is simpler to implement than a NSR model but does not adequately represent the many variables used in the calculation of revenues as a NSR model can.

Ultimate and staged pit designs were developed by Hudbay from optimization shells. Nine pit stages were selected for the Constancia pit (San José is the third Stage) and two pit stages for the Pampacancha pit.

Resource Models

The resource models used for the Mineral Reserve Estimation are described in details Section 14, with a Selective Mining Unit (SMU) ranging from 10x10x15 metres in the upper part of the deposit to 20x20x15 metres below 3,930m where the mine plans to operate in a less selective manner due to the more consistent and uniform nature of the mineralization.

Mine Design and Reserve estimation for Constancia and Pampacancha pits used an NSR value calculation stored in each block in the resource models. The NSR calculation takes into account the Cu, Mo, Ag, and Au grades, mill recoveries, contained metal in concentrate, deductions and payable metal values, metal prices, freight costs, smelting, refining and royalty charges.

Net Smelter Return

Prior to importing the resource model into the pit optimization software Whittle, the in-situ NSR value was calculated and stored into each block in the model. The following points detail the NSR calculation:

- In-situ NSR is the net value of metals (in dollars) contained in a concentrate produced from an ore block after smelting and refining. Using the concentrator recovery of the metals into the concentrate and the grade of the concentrate produced, the mass pull of each block in the resource model expressed in terms of tonnes of concentrate per tonne 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 and roasters. In the case of the copper concentrate, the payable precious metals silver and gold are 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 the marketing costs, transport costs, port charges, insurance costs, shipping costs, and smelting charges expressed in $/dmt concentrate and other deductions like the refining charges and price participation expressed in $/payable metal are taken out to obtain the gross concentrate NSR value (before royalties).
- The royalties applicable are then deducted from the gross concentrate NSR value to obtain the net concentrate NSR value (after royalties). The concentrate NSR value calculations described above are applied for both the copper and molybdenum concentrates.
- The concentrate NSR value after royalties for the copper and molybdenum concentrates are then multiplied by their respective mass pull expressed in tonnes of concentrate produced per tonne of ore processed to obtain the contribution of each concentrate to the in-situ NSR value.
- The in-situ NSR of each block in the resource model is the sum of the in-situ NSR value from the copper concentrate and the molybdenum concentrate.
- Measured and Indicated resource blocks with NSR values greater than their processing

costs are considered to be potential ore, while blocks which have NSR values less than their processing costs are considered to be waste.

- Process plant recoveries, throughputs, 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, silver, gold, zinc, and lead are fixed numbers depending on the metallurgical domain. Copper, silver, gold, molybdenum, zinc, and lead grades in the copper concentrate and in the molybdenum concentrate are estimated based on the test work detailed in Section 13.

NSR Input Parametres

The metal price, mineral marketing parametres, operating cost, and metal recoveries used for pit optimization are showed in Table 15-1 to Table 15-4.

Table 15-1 Metal Prices

Revenue	Units	Constancia & Pampacancha
Metal Price		
Copper	$/lb	3.00
Molybdenum	$/lb	11.00
Silver	$/oz	18.00
Gold	$/oz	1,260.00

Table 15-2 Operating Cost

Cost		Units	Constancia	Pampacancha
Mining Cost				
	Ore	$/tmined	1.35	1.85
	Waste	$/tmined	1.30	1.55
Variable Cost				
by Bench	Up	$/tmined	0.004	0.004
		Exit bench	4185	4305
	Down	$/tmined	0.01	0.01
G&A Cost				
	Ore	$/ttrat	1.60	1.60
	Waste	$/tmined	0.00	0.00
Process Cost				
	Ore	$/ttrat	4.54	4.54

Table 15-3 Metal Recoveries

Recovery	Supergene	Mixed	Hypogene	Skarn	HiZinc	Skarn
Cu %	88.4%	88.4%	90.5%	84.4%	84.4%	85.0%
Ag %	90.0%	90.0%	72.0%	52.0%	52.0%	70.0%
Au %	60.0%	60.0%	60.0%	60.0%	60.0%	70.0%
Mo %	55.0%	55.0%	55.0%	55.0%	55.0%	40.0%
Pb %	41.4%	41.4%	41.4%	41.4%	41.4%	41.0%
Zn %	80.0%	30.0%	30.0%	30.0%	30.0%	30.0%

Table 15-4 Mineral Marketing Parametres

Parameter	Units	Value
Payable Contained Metal		
Copper	%	96.5%
Molybdenum	%	100.0%
Silver	%	90.0%
Gold	%	90.0%
Concentrate grades		
Copper concentrate grade		
Copper	%	Cu lbs/(Cu lbs/0.30 + Zn lbs/0.55 + Pb lbs/0.70)
Silver	g/t	Ag ozs*31.1034/conc tonnes
Gold	g/t	Au ozs*31.1034/conc tonnes
Zinc	%	Pb lbs/2204.62262/conc tonnes
Lead	%	Zn lbs/2204.62262/conc tonnes
Moly concentrate grade		
Molybdenum	%	50.0%
Deductions		
Copper deduction	Per unit of pay Cu	1.00
Concentrate Moisture Cont		
Copper concentrate	%	8%
Moly concentrate	%	15%
Selling Cost		
Transport Cu conc	$/wmt conc	41.03
Transport Mo conc	$/wmt conc	175.00
Port charges conc	$/wmt conc	26.00
Insurance	$/wmt conc	5.00
Shipping Cu conc	$/wmt conc	25.00
Smelting Charges		
Smelting charges - Cu conc (dry)	$/dmt Cu conc	77.83
Roasting charges - Mo conc (dry)	$/dmt Mo conc	1,653
Refining charges		
Cu	$/lb Cu	0.078
Mo	$/lb Mo	1.300
Ag	$/oz Ag	0.450
Au	$/oz Au	4.500
Royalties		
Royalties	% of NSR	0.4%

Pit Slope Guidance

The pit slope angles used for pit optimization of the Constancia and Pampacancha pits are based on engineering studies conducted at pre-feasibility or higher level of engineering. The pit slope design was developed by Knight Piésold, in January 2013 for Constancia and by TWP/Itasca in August 2013 for Pampacancha. The plan view and the design parametres for the two pits are shown by sector in and in Table 15-6. Figure 15-1, Figure 15-2, Table 15-5 and in Table 15-6.

Figure 15-1 Constancia Pit Slope Sectors

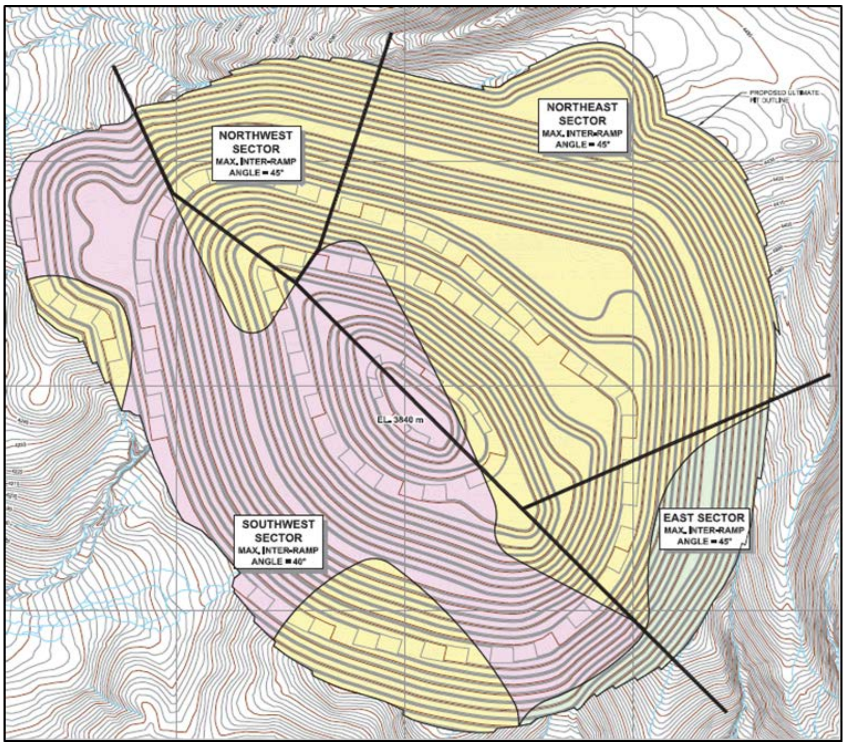


Table 15-5 Constancia, Recommended Pit Slope Angles

Pit Design Sector	Geotechnical Domain	Bench Face Angle (º)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (º)	Max. Inter-ramp Slope Height (m)
Southwest	B-II: Heavily Faulted intrusive rock	60	15	9	**40**	200
Northwest	B-I: Slightly Faulted intrusive rock	65	15	8	**45**	200
Norteast	B-I: Slightly Faulted intrusive rock	65	15	8	**45**	200
Southeast	A: Sedimentary Rock	65	15	8	**45**	200

Figure 15-2 Pampacancha Pit Slope Sectors

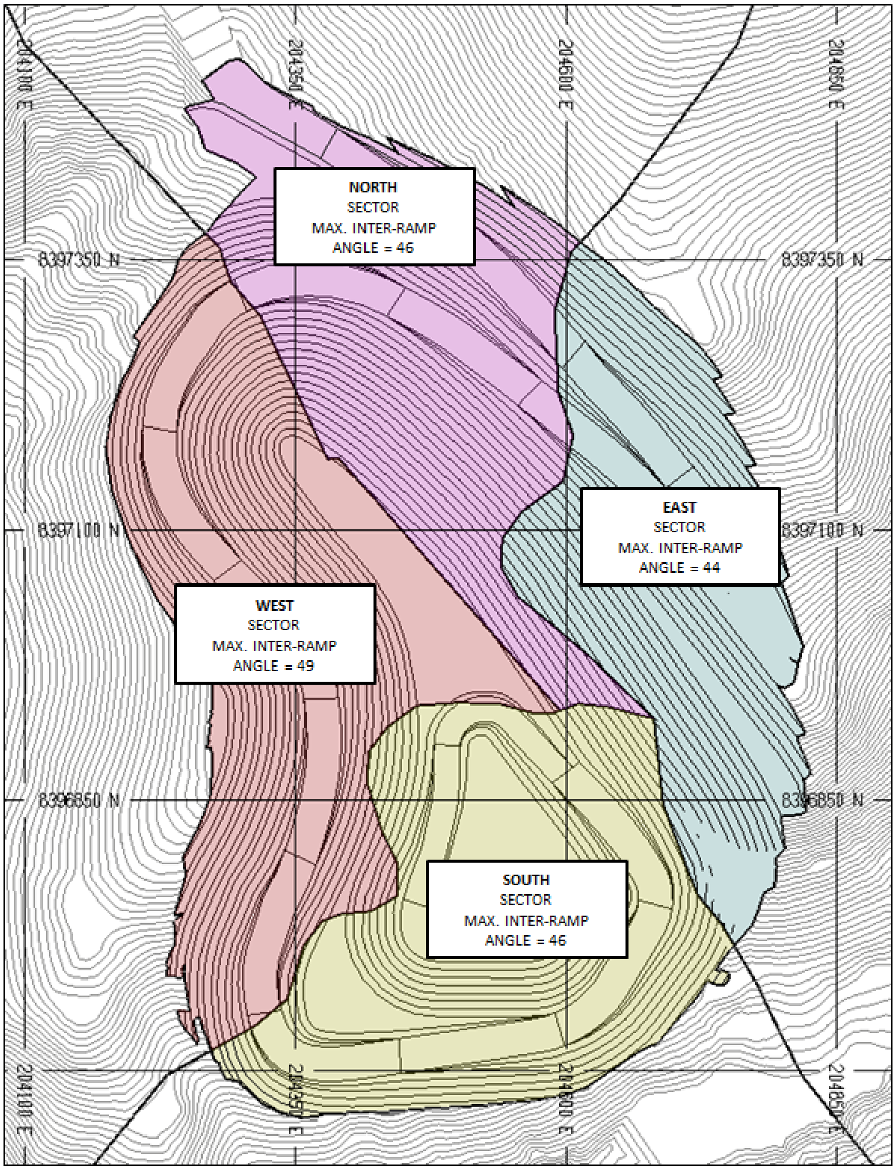


Table 15-6 Pampacancha, Recommended Pit Slope Angles

Pit Design Sector	Geological domain	Bench Face Angle (º)	Bench Height	Bench Width	Inter-ramp Angle	Max. Inter-ramp Slope Height
			(m)	(m)	(º)	(m)
North	Domain II	65	15	7.5	**46**	42
East	Domain I	65	15	8.5	**44**	40
South	Domain III	65	15	7.5	**46**	40
West	Domain II	70	15	7.5	**49**	44

Optimization Results

The Whittle software was used to create incremental economic pit-shells using the Lerchs-Grossman (LG) algorithm to calculate the optimum pit limit. This optimization relies on the blocks NSR value in order to find each mining block that can be mined with a profit. The optimum economic shell generated by Whittle is then use as a guide for mine design.

Pit shell 32 for Constancia was generated at a 0.80 revenue factor and contains approximately 536 Mt of Ore and 582 Mt of Waste. The pit shell captures close to 99.0% of the Net Cash flow of the 1.00 revenue factor pit shell.

Pit shell 32 for Pampacancha was generated at a 0.85 revenue factor and contains approximately 42 Mt of ore and 75 Mt of waste. The pit shell captures about 100% of the net cash flow of the 1.00 revenue factor pit shell.

Figure 15-3 and Figure 15-4 respectively present the results of the Whittle analysis for the Constancia and Pampacancha pits.

Figure 15-3 Constancia Whittle Results

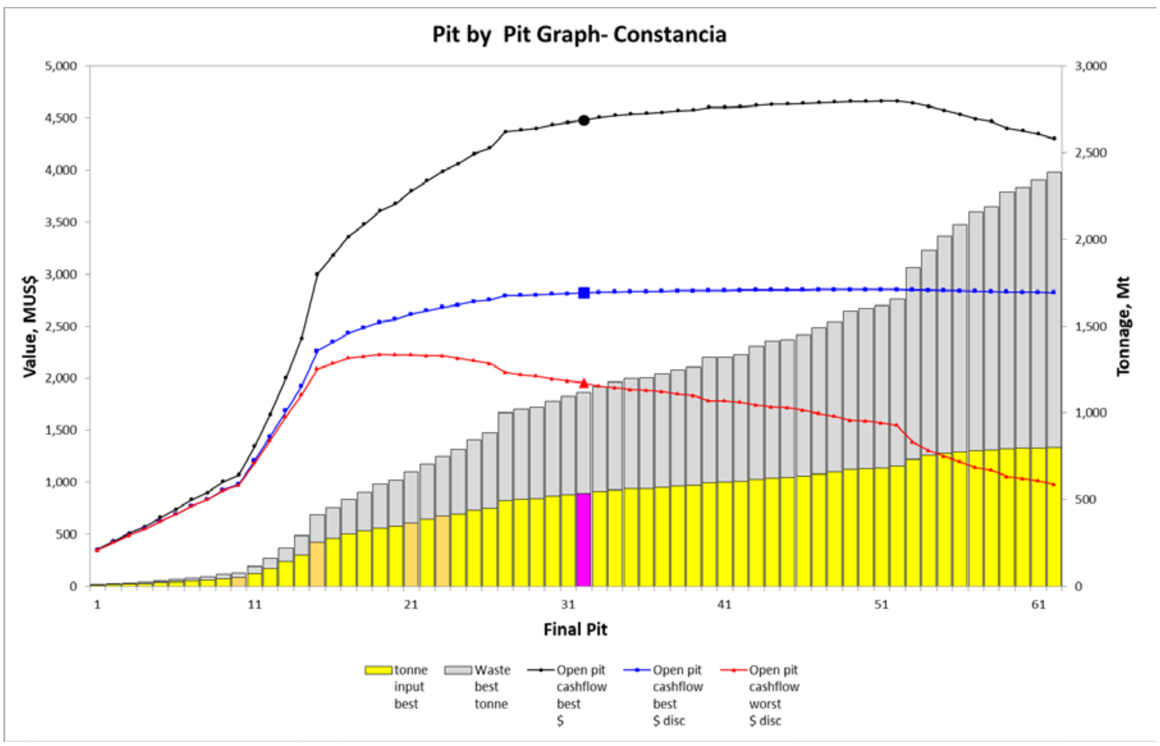
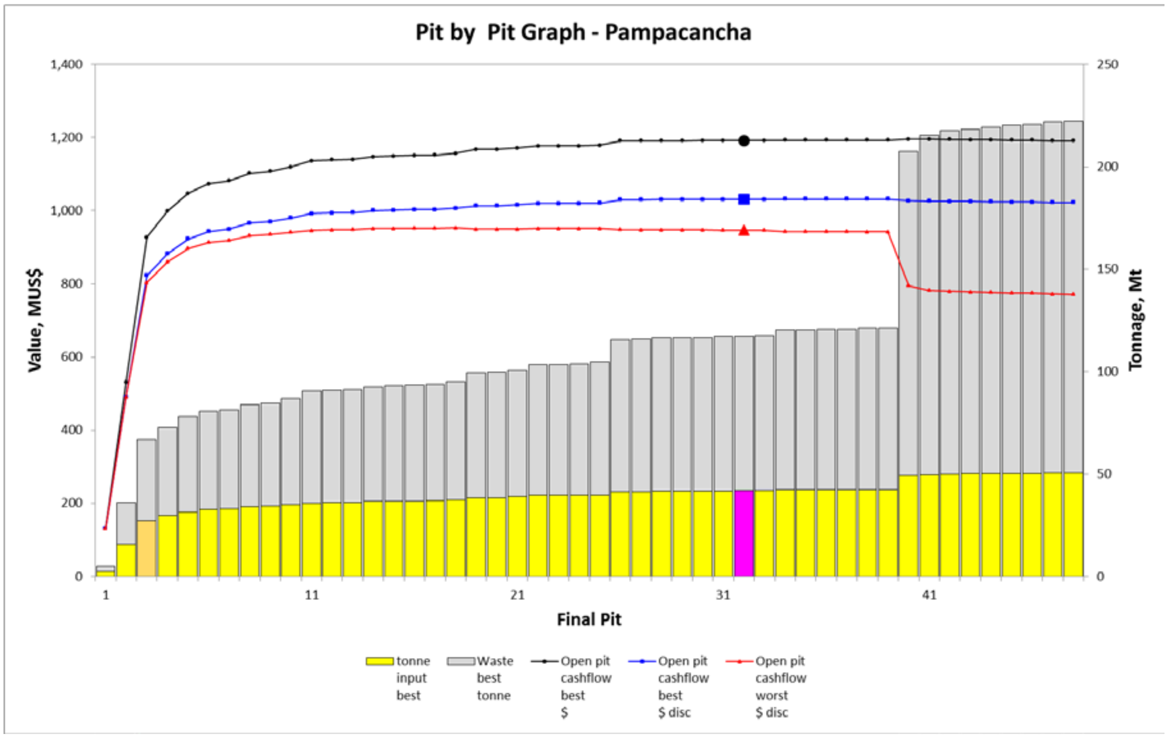


Figure 15-4 Pampacancha Whittle Results



Pit Design Criteria

The pit design parametres for Constancia and Pampacancha are summarised in Table 15-7 and the final pit extents are illustrated on Figure 15-5 and Figure 15-6.

Table 15-7 Pit Design Parametres

Description	Constancia	Pampacancha
Slopes angles	2013 KP Slopes	2013 TWP/Itasca
Road width	32 m	32 m
Catch bench	8 to 9 m (depending on IRAS)	7.5 to 8.5 (depending on IRAS)
Bench height	15 m single bench	15 m single bench
Ramp design grade	10%	10%
Other	--	Last 2 benches as a single lane (24 m with 10% ramp)

Figure 15-5 Constancia Ultimate Pit Design

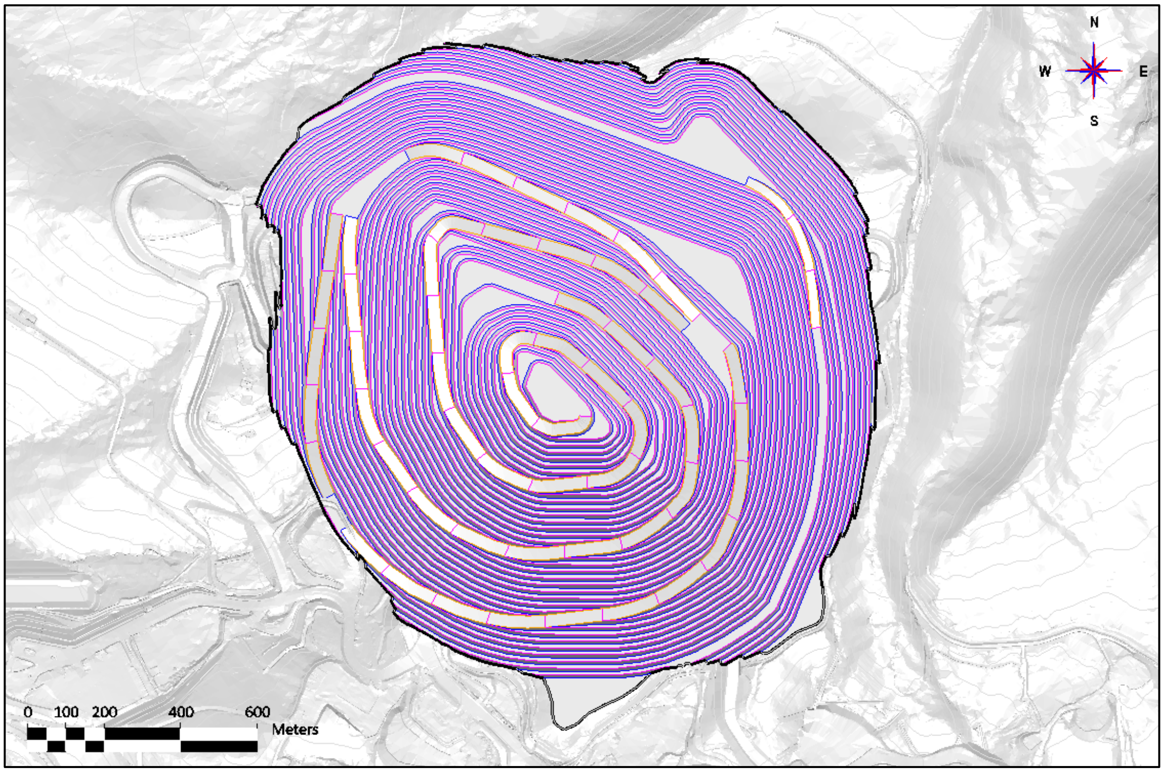
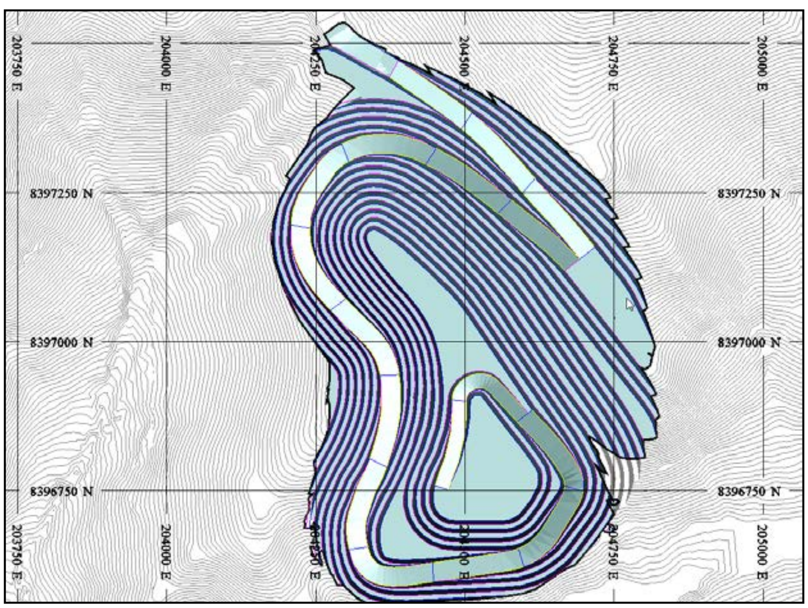


Figure 15-6 Pampacancha Ultimate Pit Design



15.2 MINERAL RESERVES

Proven and Probable Reserves at the Constancia mine now stand at 568.6 million tonnes at a copper equivalent grade of 0.41% that support a 19 year mine life.The mine plan is based on the capacity of the process plant, which in turn relies on the grinding circuit throughput.

The plant has the capacity to process 31 Mtpy (90ktpd at 94% availability), with a mining rate of 70.4 Mtpy (ore plus waste). Table 15-8 presents Constancia and Pampacancha Mineral Reserves at December 31st, 2017.

Table 15-8 Constancia and Pampacancha Mineral Reserve estimates as at December 31st, 2017

	Category	Mt	Cu (%)	Mo (g/t)	Ag (g/t)	Au (g/t)	CUEQ
Constancia	Proven	452.9	0.30	96	2.92	0.035	0.39
	Probable	72.8	0.23	72	3.09	0.035	0.31
	Total 2P	525.7	0.29	93	2.94	0.035	0.38
Pampacancha	Proven	32.4	0.59	178	4.48	0.37	0.92
	Probable	7.5	0.62	173	5.75	0.33	0.93
	Total 2P	39.9	0.60	177	4.72	0.360	0.92
Stockpile	Proven	3.1	0.4	98	4.68	0.068	0.53
	Proven	488.4	0.32	102	3.03	0.057	0.42
	Probable	80.3	0.27	82	3.33	0.062	0.37
Total		**568.6**	**0.32**	**99**	**3.07**	**0.058**	**0.41**

A comparison of the December 31st 2016 and December 31st 2017 Mineral Reserve estimate is summarised in Table 15-9 for the Constancia pit and in Table 15-10 for the Pampacancha pit. The December 31st 2017 Constancia Reserve estimate shows an increase of 12% after substracting the 2017 mining depletion in the copper contained in the Mineral Reserves estimates at Constancia and a 13% increase at Pampacancha, in both cases driven by a significant increase in Cu grade. This increase results from a combination of improvements in the resource modeling process at both Constancia and Pampacancha and of the correction of a sampling bias identified in the Supergene Domain at Constancia.

Table 15-9 Comparison of the Mineral Reserve Estimates at Constancia as at December 31st, 2016 and as at December 31st, 2017

Mineral Reserve Reconciliation (Proven & Probable)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2017 Mineral Reserve	541,200,000	0.28	88	2.8	0.037	1,538,000
B	2017 Production (Depletion)	28,700,000	0.52	126	3.9	0.040	150,000
C	(A - B)	512,500,000	0.27	86	2.7	0.037	1,388,000
G	Geology & Mine Planning (Gain/Loss)	16,200,000	-	-	-	-	170,000
H	2018 Mineral Reserve (C + G)	528,700,000	0.29	93	3.0	0.035	1,558,000

Mineral Resource Reconciliation (Measured & Indicated)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
I	2017 Mineral Resource (Measured & Indicated)	449,500,000	0.18	52	2.0	0.028	797,000
J	2018 Mineral Resource (Measured & Indicated)	356,000,000	0.20	54	2.1	0.030	701,000
K	(J - I) Gain[2]/(Loss)	(93,500,000)	-	-	-	-	(96,000)

Mineral Resource Reconciliation (Inferred)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
L	2017 Mineral Resource (Inferred)	138,100,000	0.17	40	1.7	0.018	233,000
M	2018 Mineral Resource (Inferred)	54,100,000	0.24	43	1.7	0.018	127,000
N	(M - L) Gain[2]/(Loss)	(84,000,000)	-	-	-	-	(106,000)

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

Table 15-10 Comparison of the Mineral Reserve Estimates at Pampacancha as at December 31st, 2016 and as at December 31st, 2017

Mineral Reserve Reconciliation (Proven & Probable)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
A	2017 Mineral Reserve	43,000,000	0.49	156	4.2	0.276	210,000
B	2017 Production (Depletion)	-	-	-	-	-	-
C	(A - B)	43,000,000	0.49	156	4.2	0.276	210,000
G	Geology & Mine Planning (Gain/Loss)	(3,100,000)	-	-	-	-	28,000
H	2018 Mineral Reserve (C + G)	39,900,000	0.60	177	4.7	0.360	238,000

Mineral Resource Reconciliation (Measured & Indicated)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
I	2017 Mineral Resource (Measured & Indicated)	22,700,000	0.23	79	3.3	0.198	53,000
J	2018 Mineral Resource (Measured & Indicated)	17,400,000	0.39	95	5.0	0.258	69,000
K	(J - I) Gain[2]/(Loss)	(5,300,000)	-	-	-	-	16,000

Mineral Resource Reconciliation (Inferred)		Tonnes[1]	Cu%	Mo (g/t)	Ag (g/t)	Au (g/t)	Tonnes Cu
L	2017 Mineral Resource (Inferred)	-	-	-	-	-	-
M	2018 Mineral Resource (Inferred)	10,100,000	0.14	143	3.9	0.233	14,000
N	(M - L) Gain[2]/(Loss)	10,100,000	-	-	-	-	14,000

Notes:

[1]Totals may not add up correctly due to rounding

[2]Since they are due to a combination of factors, grades are not provided in "the Gains or Loss"

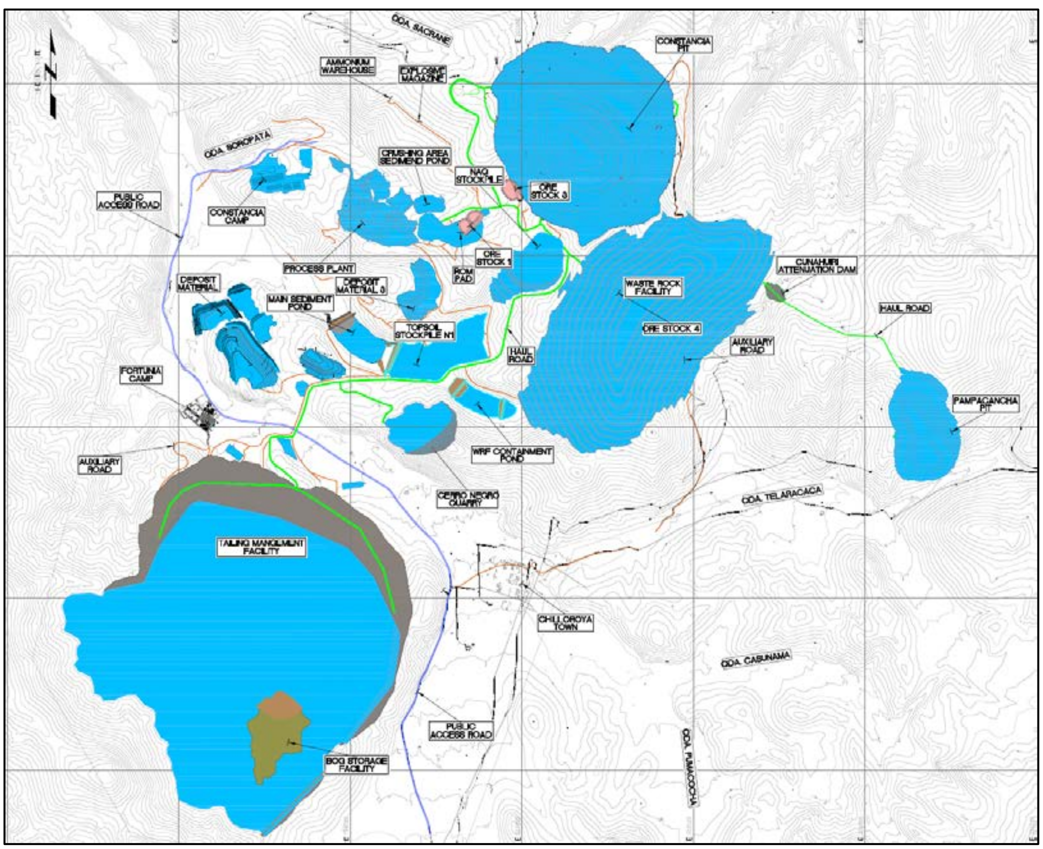
16 MINING METHODS

16.1 MINE OVERVIEW

The Constancia mine is an open pit mining operation relying on conventional trucks and shovels. The Constancia ultimate pit design will measure approximately 1.8 km east to west, 1.6 km north to south, and have a maximum depth of approximately 660 m. The Pampacancha ultimate pit design will measure approximately 0.6 km east to west, 1 km north to south, and have a maximum depth of approximately 300 m. A primary waste rock facility (WRF), which is located to the south and east of the Constancia pit, is intended to be used for both deposits.

The processing facility is located approximately 1 km west of the Constancia Pit. The NAG waste rock is deposited south of the Constancia pit while the tailings management facility (TMF) is located 3.5 km southwest of the Constancia pit. The general layout of the Constancia Operations site is shown in Figure 16-1.

Figure 16-1 General Site Layout



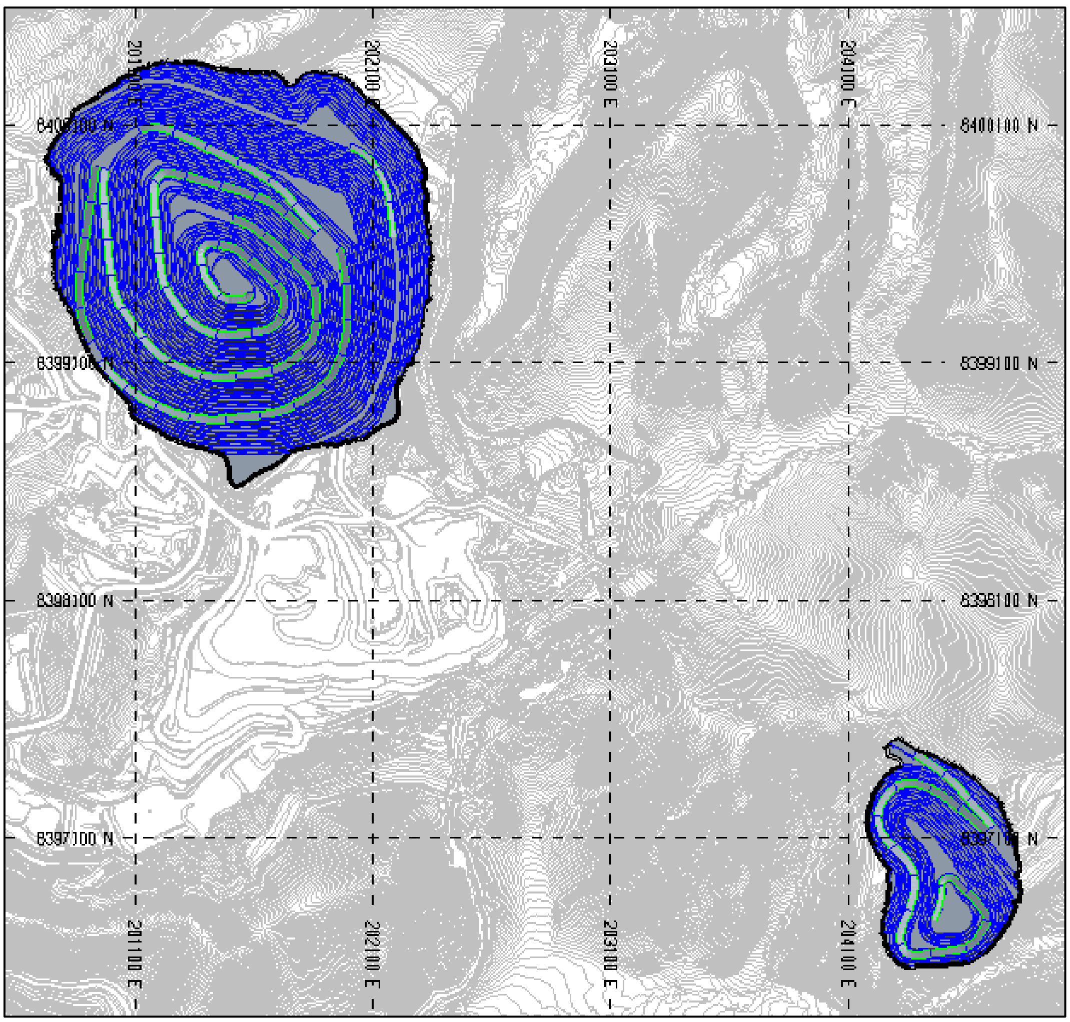
16.2 MINE PHASES

16.2.1 Design Criteria

Final pit limit designs have been created for Constancia and Pampacancha based on the optimized pit shells. Constancia is mined in nine stages and a two stage pit is planned at Pampacancha. The minimum mining width for each phase is 60 metres; this will allow a shovel, trucks (in two lines) and a drill to work safely and simultaneously.

The haul roads at Constancia and Pampacancha have been designed with a 10% grade and 32m width for double lane haul roads and 24m for single lane haul roads. Figure 16-2 presents the ultimate pit footprint for Constancia and Pampacancha.

Figure 16-2 Constancia and Pampacancha Ultimate Pit Design



16.2.2 Pit Slopes Angles

In 2013, Knight Piesold presented a feasibility study for the Constancia Pit Slope Design which was based on previous geotechnical studies. The designed pit was approximately 580 m deep.

During 2013, new reserves were considered, resulting in new limits for the ultimate pit. As a consequence, a new study to support the planned expansion for the ultimate pit design was required. The design of the expanded total pit was approximately 650 m deep (70 m deeper than the previous pit design).

In 2014, Golder Associates carried out a Pre-Feasibility Study for pit slope design based on the ultimate pit design of the Constancia pit (expansion shell). Stability analyses were conducted for the overall pit walls considering the potential for non-circular failure mechanisms that could involve multiple benches. The results from this study indicated that due to relatively low rock mass strength, the inter-ramp angles obtained from the kinematic analyses had to be adjusted to ensure safe operation. Table 16-1 and Figure 16-3 present the recommended pit slope design for Constancia.

At Pampacancha, the study, “Constancia Expansion – Pampacancha Feasibility Study”, developed by TWP/Itasca (August, 2013), provide the geotechnical information about the pit slope design angles as pit optimization input. Table 16-2 and Figure 16-4 show the recommendations for open pit slope design for Pampacancha.

Table 16-1 Constancia, Recommended Pit Slope Angles (Golder, 2014)

Geotechnical Domain	Location	BFA[1] (°)	Vertical Bench Separation (m)	Berm Width (m)	Max. Uninterrupted IRA Height (m)	Max. IRA[1] (°)	Max. Overall Angle (°)
A: Sedimentary Rock	Southeast	65	15	9	195	43	36[2]
B-I: Slightly Faulted Intrusive Rock	North	65	15	8	195	45	39
	Northwest	65	15	8	195	45	40
B-II: Heavily Faulted Intrusive Rock	Southwest	60	15	9	195	40[3]	37

Table 16-2 Pampacancha Pit Slope Sectors (Itasca, 2013)

Pit Design Sector	Geological domain	Bench Face Angle (º)	Bench Height (m)	Bench Width (m)	Inter-ramp Angle (º)	Max. Inter-ramp Slope Height (m)
North	Domain II	65	15	7.5	**46**	42
East	Domain I	65	15	8.5	**44**	40
South	Domain III	65	15	7.5	**46**	40
West	Domain II	70	15	7.5	**49**	44

Figure 16-3 Constancia Pit Slope Sectors (Golder, 2014)

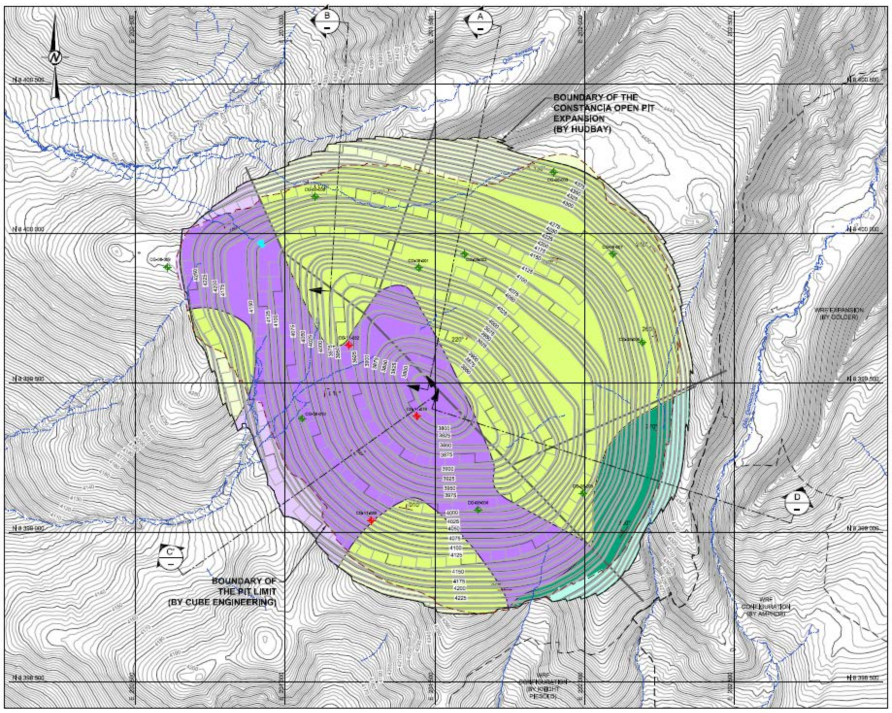
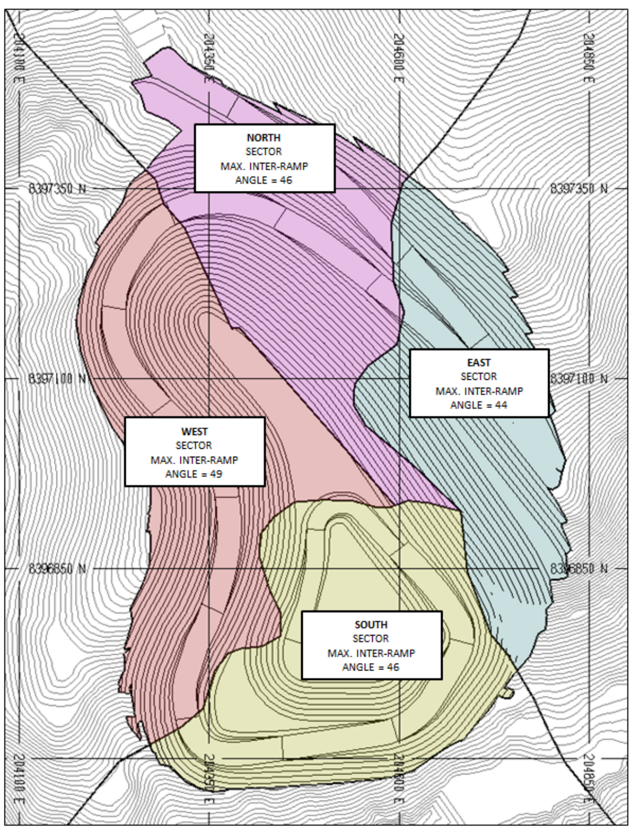


Figure 16-4 Pampacancha Pit Slope Sectors (Itasca, 2013)



16.2.3 Mine Phases and Ultimate Pit

The extraction sequence is defined by nine mining phases at Constancia and two mining phases at Pampacancha. In parallel, eight pushbacks are planned for ore exposure purposes and optimal sequencing at Constancia with one pushback for waste and construction purposes. The phase development strategy consists of extracting the highest metal grades along with minimum strip ratios during the initial years to maximize the economic benefits, while enabling smooth transitions in waste-stripping throughout the life of the mine to ensure sufficient ore exposure.

The NSR cut-off grade to report Mineral Reserves has been defined at a 6.14 $/t cut-off for both Constancia and Pampacancha. Marginal material is treated as waste. The optimum pit shell and extraction sequence by phase are illustrated for Constancia in Figure 16-5 andFigure 16-6 and for Pampacancha in Figure 16-7 and Figure 16-8.

Figure 16-5 Pit by Pit Graph - Constancia

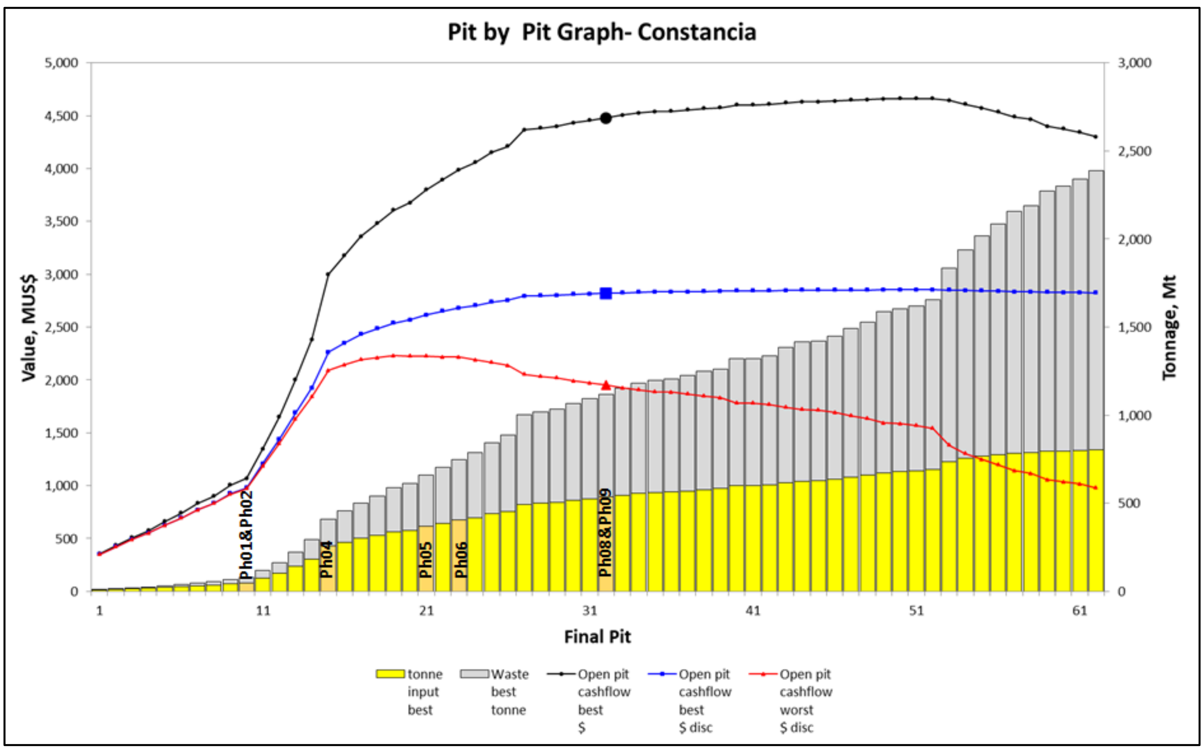


Figure 16-6 Constancia Phase Design, Section View EW

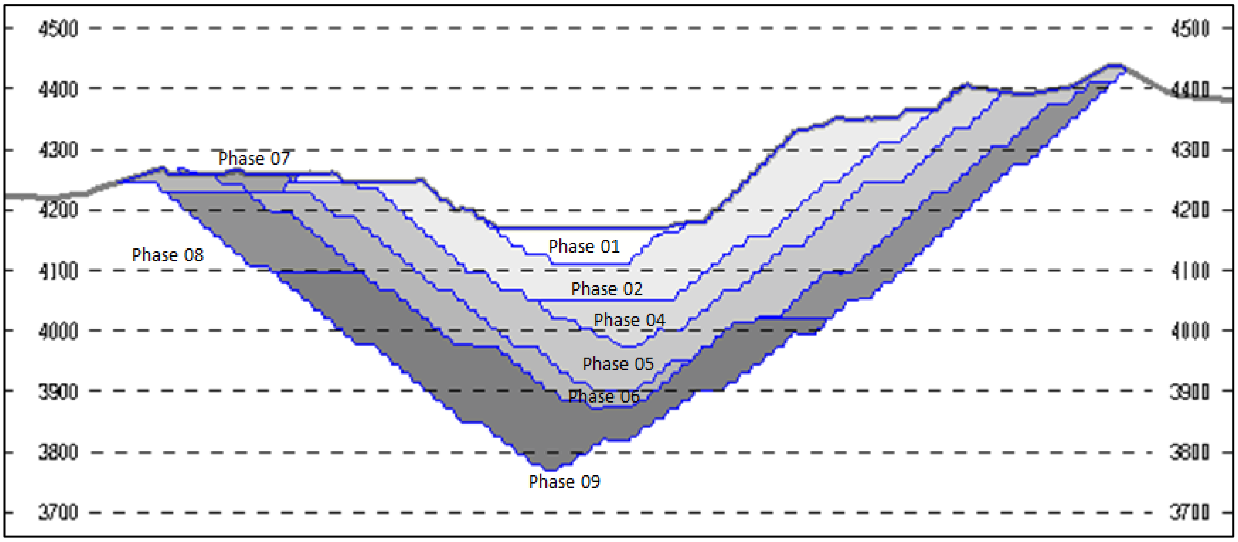


Figure 16-7 Pit by Pit Graph - Pampacancha

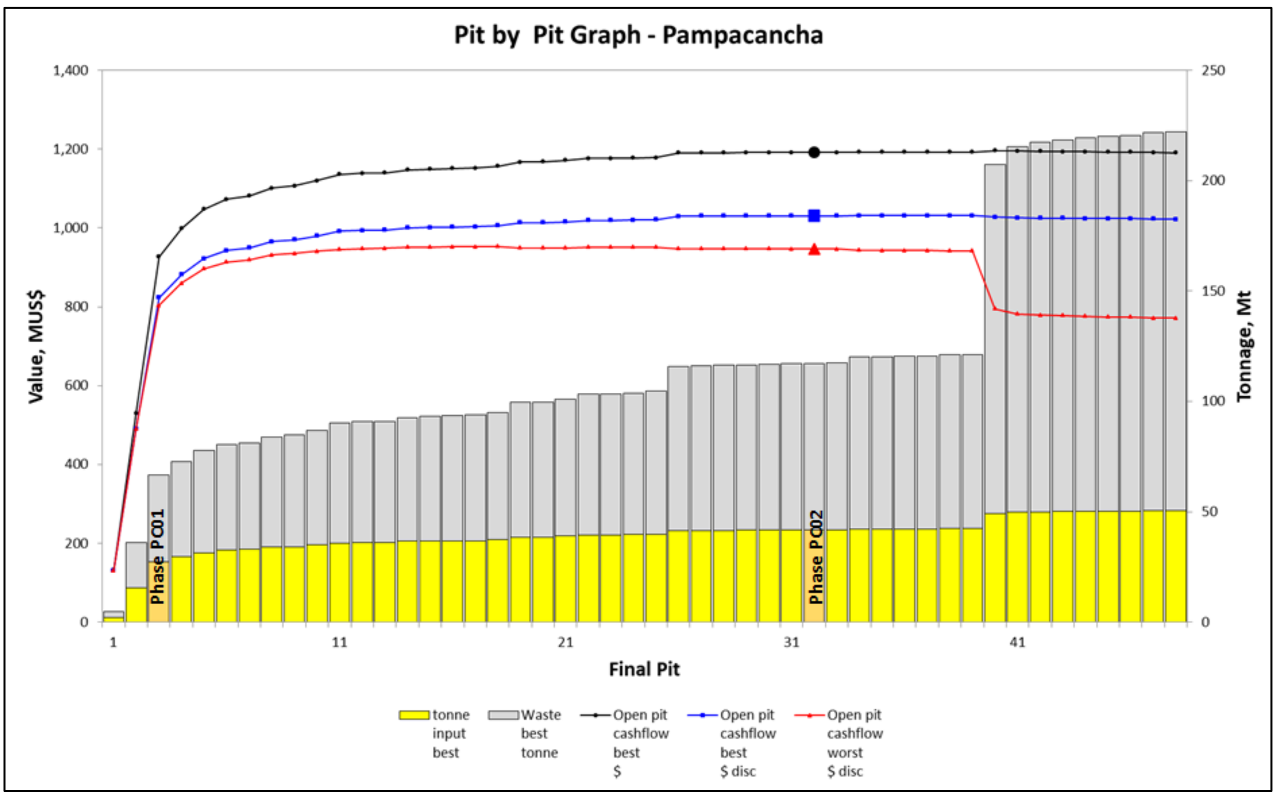
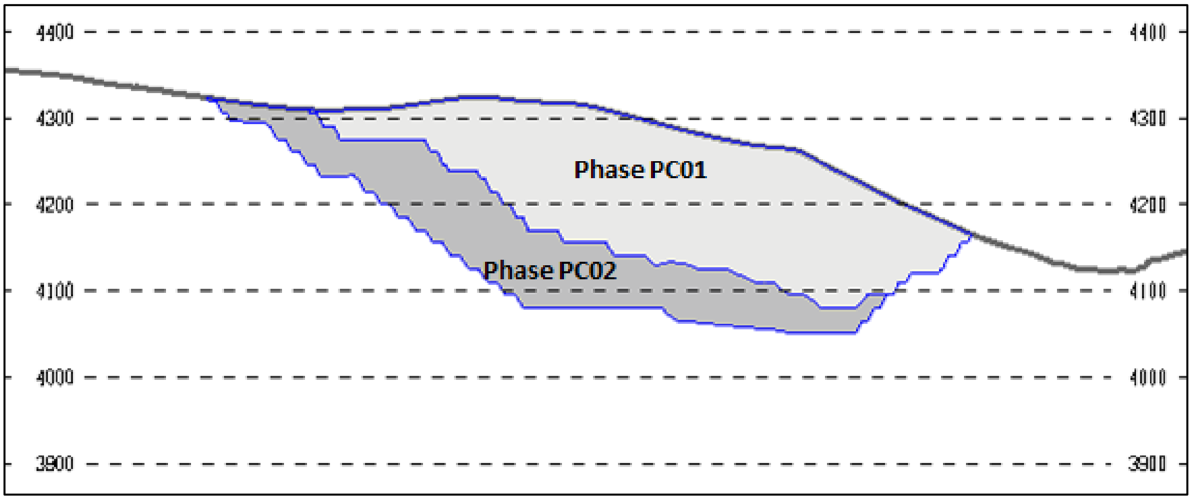


Figure 16-8 Pampacancha Phase Design, Section View NS



Constancia – Mine Phase 1

The starter pit, Phase 1 mined since 2014, correspond approximately to the Lerchs-Grossman pit shell defined by a $1.14/lb Cu price (38% of base metal price sensitivity case). This pit is located about 1,750 metres east of the primary crusher and ranges in elevation from 4,170 to 4,110 masl. The phase is approximately 460 metres wide east-west and 400 metres north-south. Phase 1 produces approximately 8.7 million tonnes of ore at a stripping ratio of 0.4:1 (tonnes of waste per tonnes of total ore). An illustration of the Phase 1 pit is shown in Figure 16-9.

Material from Phase 1 is hauled via haul road 3. This road branches off to the primary crusher, the waste rock facility (WRF) and tailings management facility (TMF) areas. These roads will be used for the life of the operation.

The pit entrance is at 4,245 masl, and the ramp enters the pit in a counter clockwise direction. The ramp switches back at 4,170 masl before reversing to a clockwise direction to the bottom of the pit.

Constancia - Mine Phase 2

Phase 2, also mined since 2014, corresponds approximately to the Lerchs-Grossman pit shell defined by a $1.14/lb Cu price (38% of base metal price sensitivity case). This pit is located about 2,000 metres east of the primary crusher and ranges in elevation from 4,440 to 4,050 masl. The phase is approximately 1,100 metres wide east-west and 1,170 metres north-south. Phase 2 will produce approximately 81 million tonnes of ore at a stripping ratio of 0.49:1 (tonnes of waste per tonnes of total ore). Figure 16-10 presents Phase 2.

Material from Phase 2 is hauled by haul roads 3 and 4. This road also branches off towards the primary crusher, the waste rock facility (WRF) and tailings management facility (TMF) areas. These roads will be used during the life of operations.

Ramp 2 starts from the pit entrance, at an elevation of 4,245 masl, and enters the pit in counter-clockwise and clockwise directions. The ramp switches back at the 4,155 masl before reversing to a clockwise direction to the bottom of the pit.

Constancia - Mine Phase 3

Phase 3 of mining has been completed in 2016 and is not discussed in this document as it does not impact the future sequence of extraction.

Constancia - Mine Phase 4

Mining in Phase 4 starts in 2018. This phase corresponds approximately to the Lerchs-Grossman pit shell defined by a $1.44/lb Cu price (48% of base metal price sensitivity case). This pit ranges from 4,410 to 3,960 masl in elevation. The phase is approximately 1,250 metres wide east-west and 1,100 metres north-south. Phase 4 will generate approximately 102 million tonnes of ore at a stripping ratio of 0.60:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 4 pit is shown in Figure 16-11.

Constancia - Mine Phase 5

Mining in Phase 5 will start in 2021 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $1.80/lb Cu price (60% of base metal price sensitivity case). This pit will range from 4,440 to 3,900 masl elevation. The phase is approximately 1,450 metres wide east-west and 1,550 metres north-south. Phase 5 will produce approximately 102 million tonnes of ore at a stripping ratio of 1.07:1 (tonnes waste per tonnes of total ore). Figure 16-12 display Phase 5.

Constancia - Mine Phase 6

Mining in Phase 6 will start in 2024 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $1.92/lb Cu price (64% of base metal price sensitivity case). This pit will range from 4,275 to 3,855 masl in elevation. The phase is approximately 1,300 metres wide east-west and 1,550 metres north-south. Phase 6 will produce approximately 67 million tonnes of ore at a stripping ratio of 1.19:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 6 pit is shown in Figure 16-13.

Constancia - Mine Phase 7

Phase 7 will finish mining in 2021, the waste material from this phase will be primarily used for the Tailing Management Facility (TMF). The phase is approximately 800 metres wide east-west and 400 metres north-south. Phase 7 will produce approximately 7 million tonnes of waste. An illustration of the Phase 7 pit is displayed in Figure 16-14. The pit entrance is at the 4,230 masl.

Constancia - Mine Phase 8

Mining in Phase 8 will start in 2026 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $2.40/lb Cu price (80% of base metal price sensitivity case). This pit will range in elevation from 4,440 to 4,020 masl. The phase is approximately 1,650 metres wide east-west and 1,650 metres north-south. Phase 8 will generate approximately 67 million tonnes of ore at a stripping ratio of 2.28:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 8 pit is presented in Figure 16-15.

Constancia - Mine Phase 9

Mining the final pushback, Phase 9 will start in 2032 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $2.40/lb Cu price (80% of base metal price sensitivity case). This pit elevation will range from 4,185 to 3,750 masl. The phase is approximately 1,160 metres wide east-west and 1,200 metres north-south. Phase 9 will produce approximately 100 million tonnes of ore at a stripping ratio of 0.86:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 9 pit is shown in Figure 16-16.

Figure 16-9 Constancia Pit Phase 1

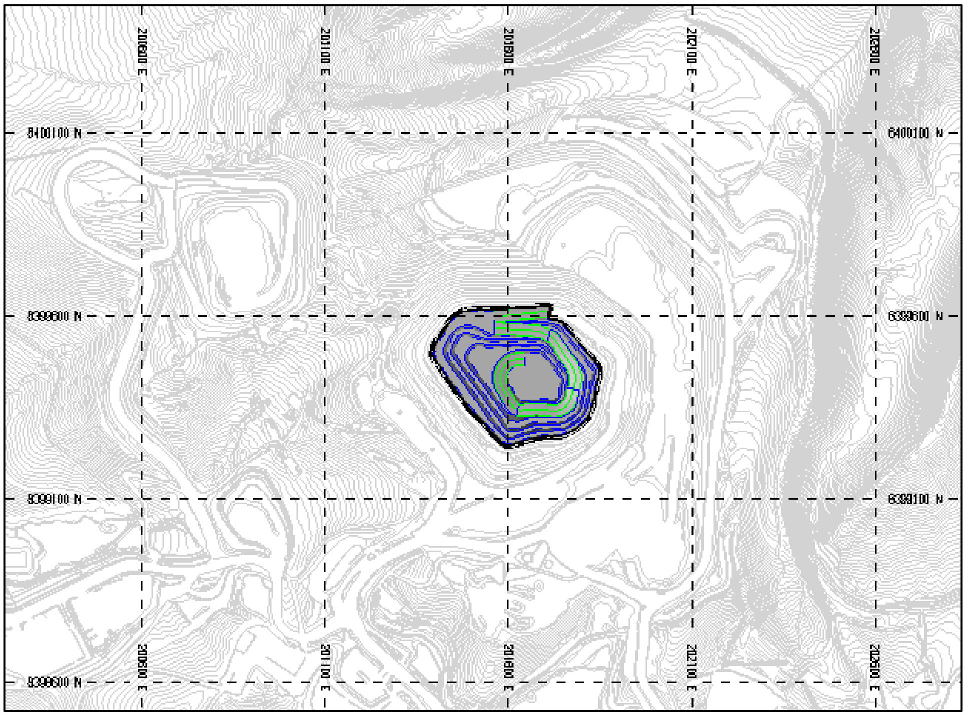


Figure 16-10 Constancia Pit Phase 2

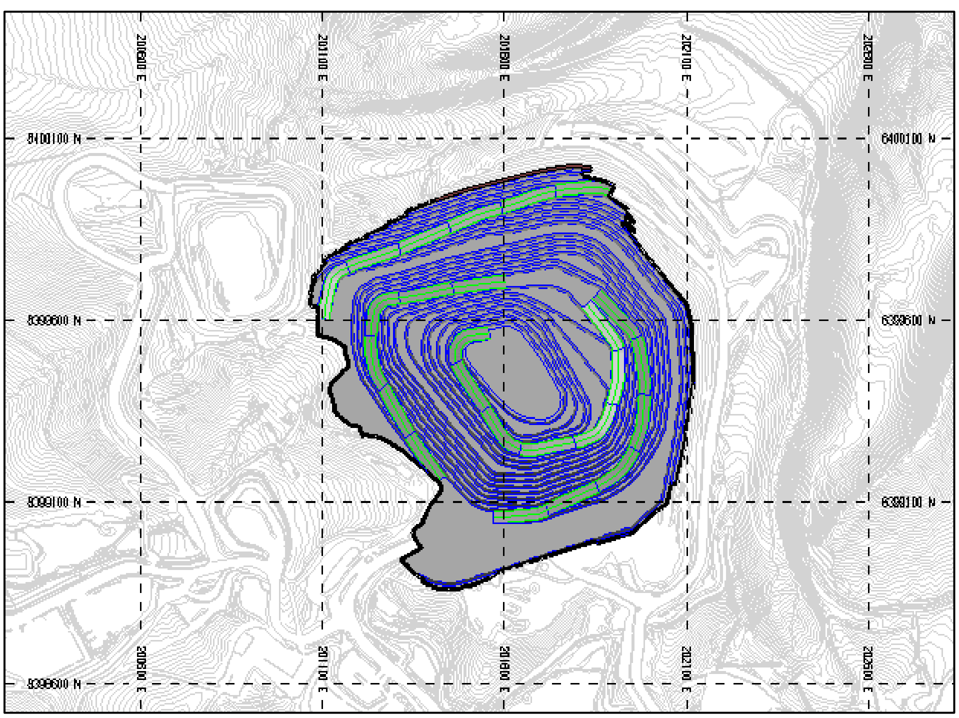


Figure 16-11 Constancia Pit Phase 4

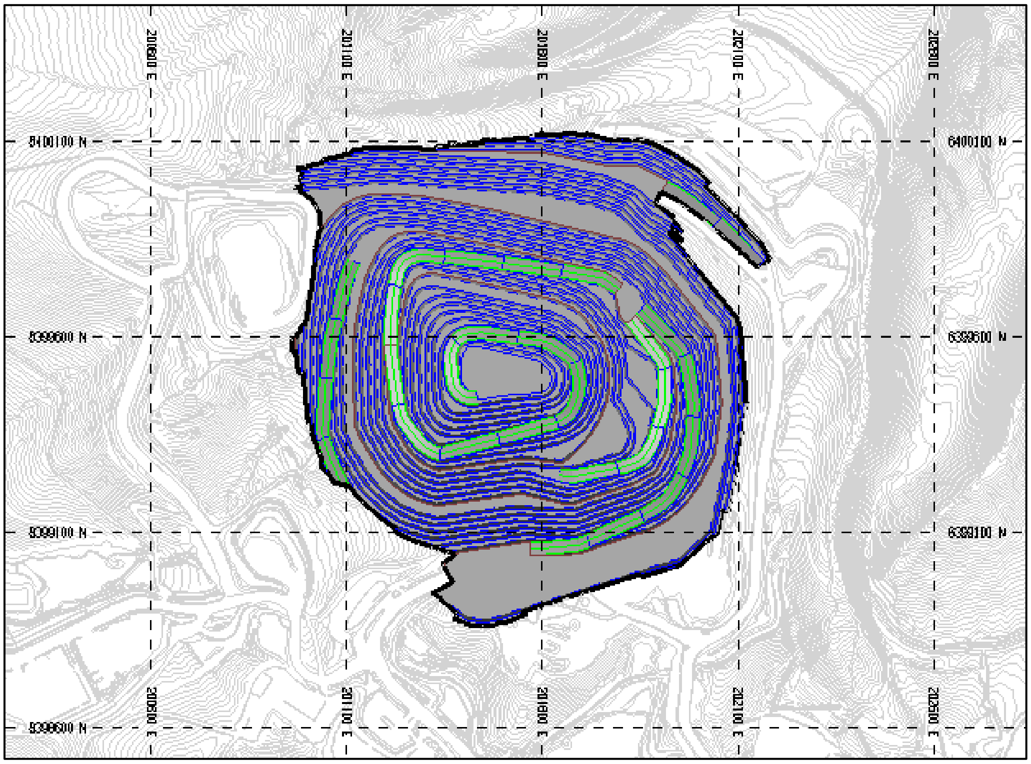


Figure 16-12 Constancia Pit Phase 5

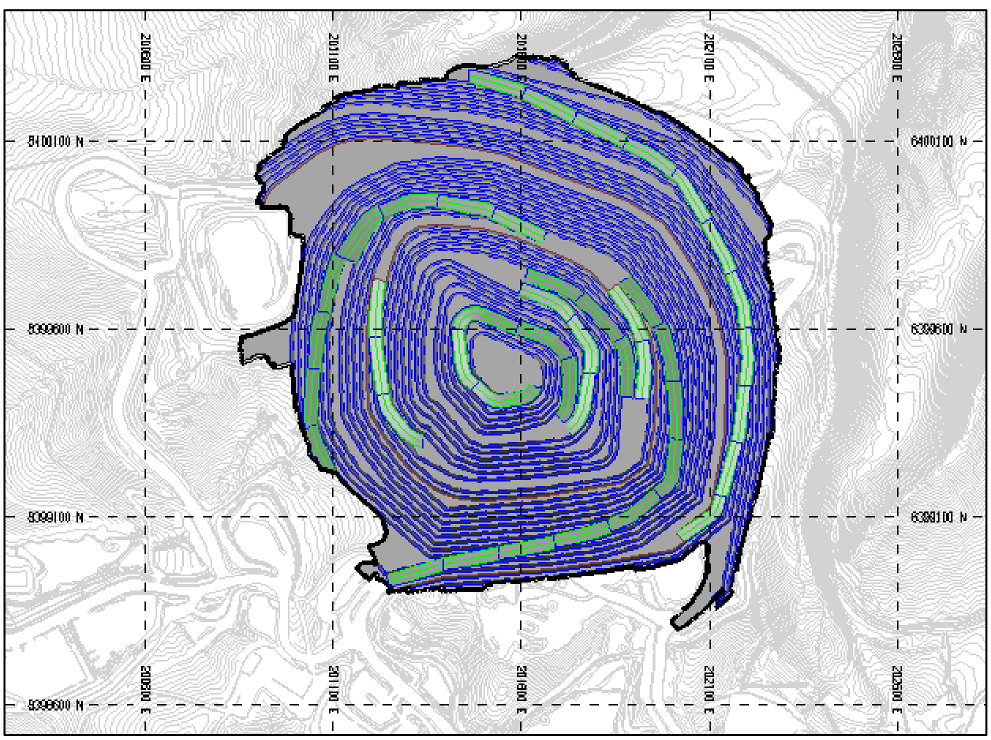


Figure 16-13 Constancia Pit Phase 6

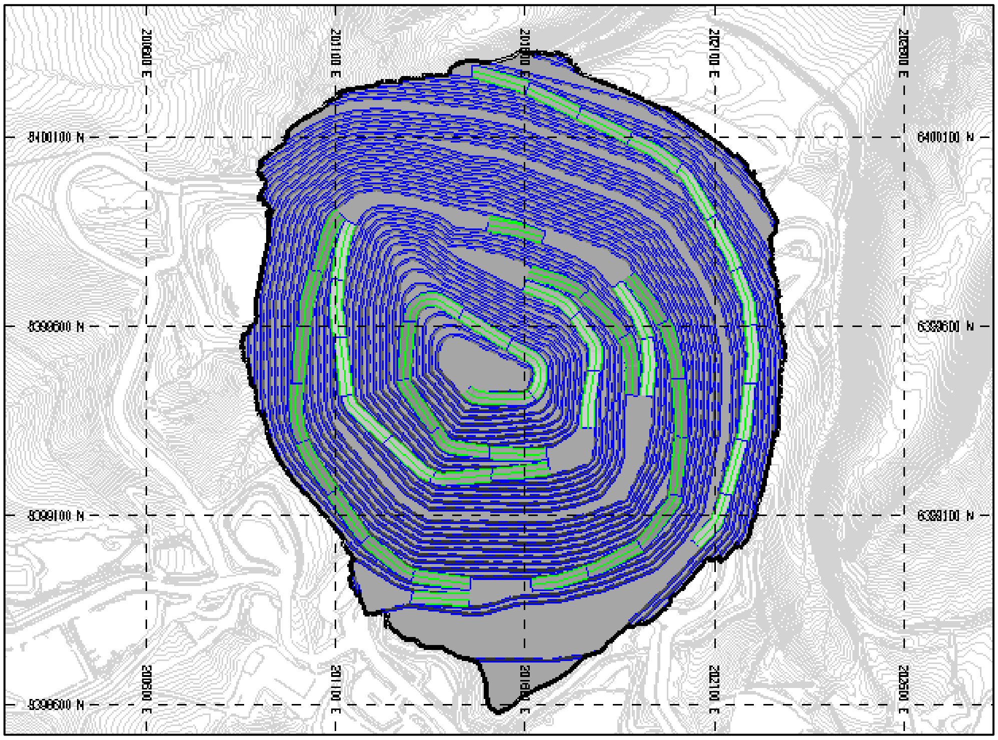


Figure 16-14 Constancia Pit Phase 7

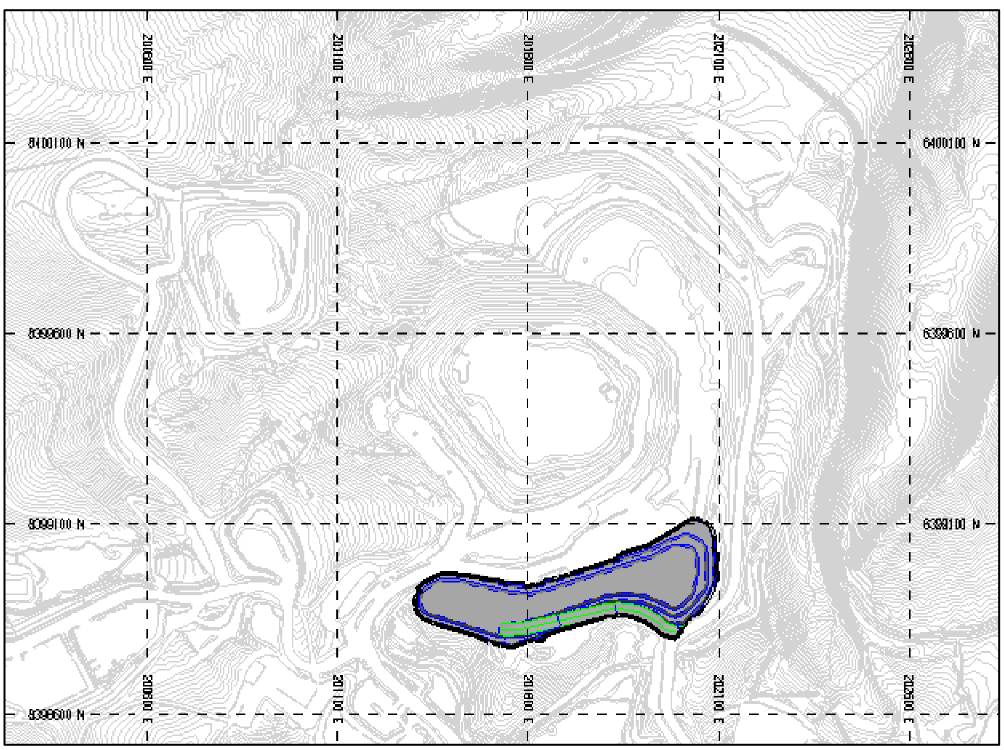


Figure 16-15 Constancia Pit Phase 8

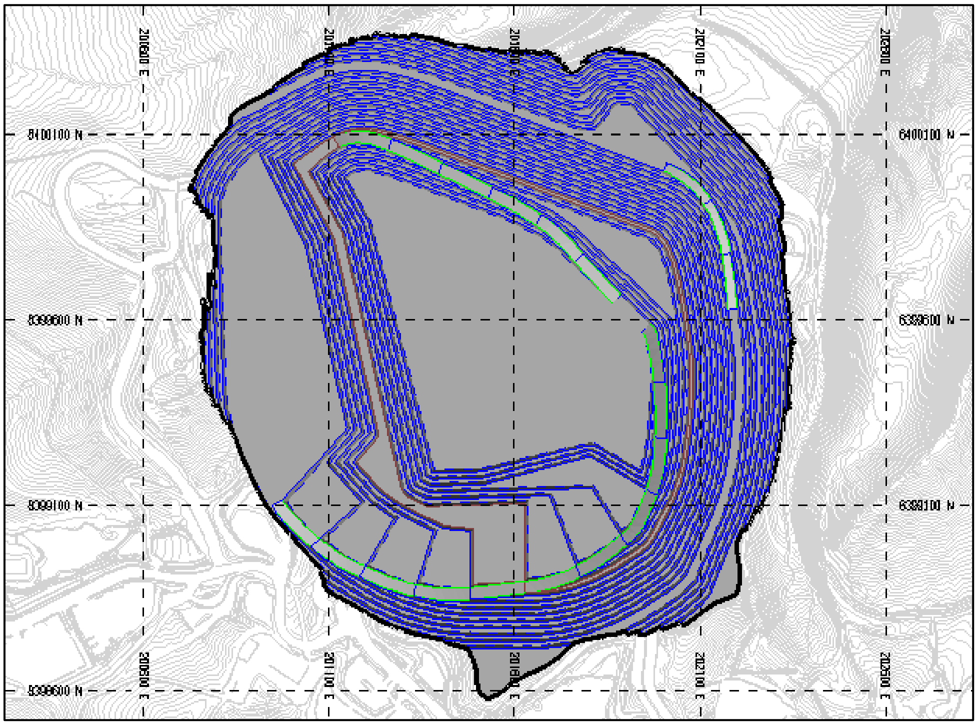
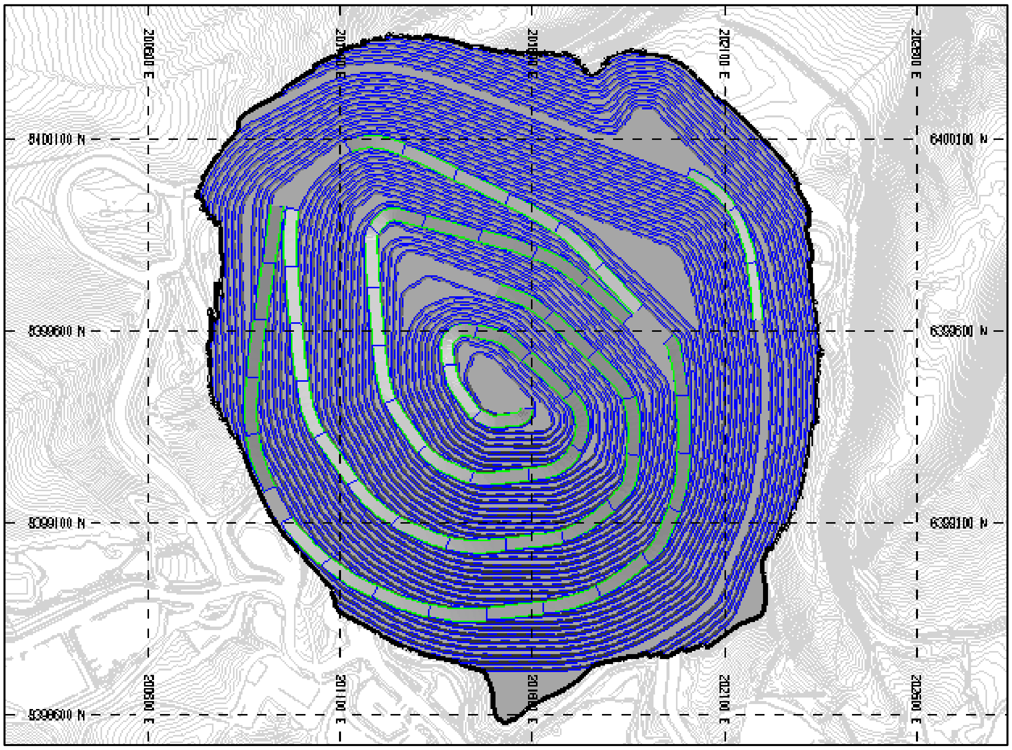


Figure 16-16 Constancia Pit Phase 9



Pampacancha - Mine Phase 1

Mining in Phase 1 will start in 2019 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $0.24/lb Cu price (24% of base metal price sensitivity case). This pit is located about 5,700 metres east of the primary crusher and the elevation ranges from 4,320 to 4,080 masl. The phase is approximately 570 metres wide east-west and 800 metres north-south. Phase PC01 will produce approximately 23 million tonnes of ore at a stripping ratio of 1.74:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 1 pit is shown in Figure 16-17.

Phase 1 material will hauled via haul road 06, which will be constructed from the pit exit and will branch off towards the primary crusher and the waste rock facility (WRF). This road will be used for the life of the operation.

The pit entrance is at 4,290 masl, and a ramp from that location enters the pit in a counter clockwise direction. The ramp switches back at 4,260 and 4,200 masl. All benches are accessible by a double lane haul road.

Pampacancha - Mine Phase 2

The mining of the final pushback during Phase 2, will start in 2021 and will correspond approximately to the Lerchs-Grossman pit shell defined by a $2.55/lb Cu price (85% of base metal price sensitivity case). This phase will range in elevation from 4,350 to 4,050 masl. The phase is approximately 650 metres wide east-west and 1000 metres north-south. Phase 2 will produce approximately 17 million tonnes of ore at a stripping ratio of 2.36:1 (tonnes waste per tonnes of total ore). An illustration of the Phase 2 pit is shown in Figure 16-18.

Figure 16-17 Constancia Pit Phase I

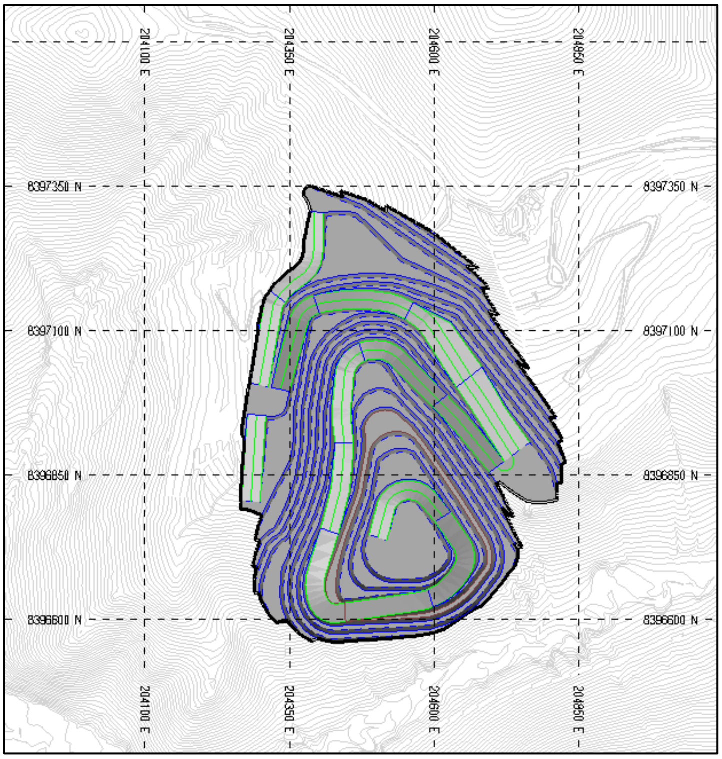
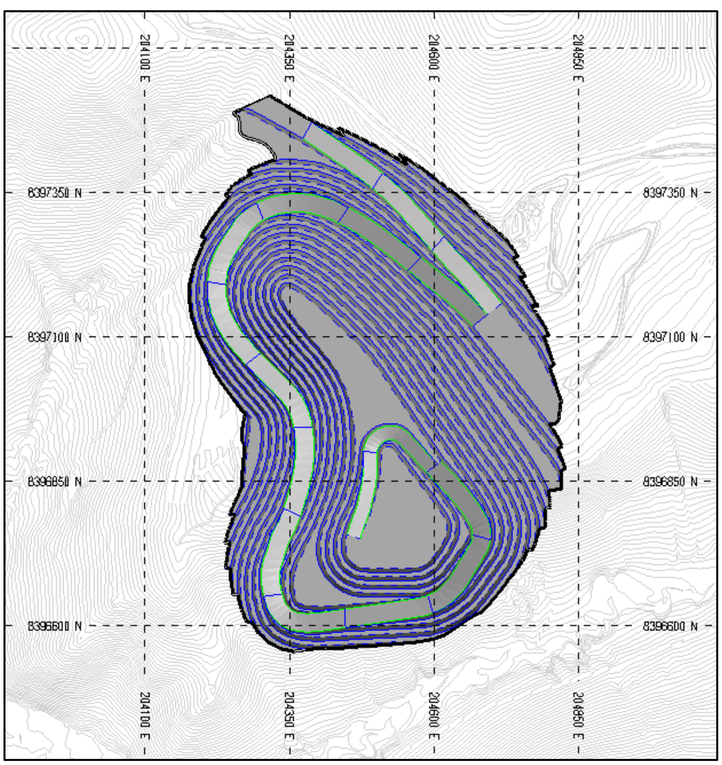


Figure 16-18 Pampacancha Pit Phase PC02 (Grid 250m x 250m)



Total ore reserves included in the final two pits are estimated to be 568.6 million tonnes and 618 million tonnes of waste material. Approximately 3.1 million tonnes of medium and low grade ore have been hauled to temporary stockpiles. At the end of the life of the mine, the ore from these stockpiles will be reclaimed and processed.

Table 16-3 presents the Mineral Reserves estimates breakdown by Phases.

Table 16-3 Constancia and Pampacancha Breakdown by Phases as at December 31st, 2017

PIT	Phases	Ore, Mt	Cu%	Mo%	Auppm	Agppm	Waste, Mt
Constancia	01	6.1	0.35	0.014	0.029	3.08	2.4
	02	81.2	0.38	0.013	0.034	3.44	38.4
	04	102.1	0.30	0.011	0.037	2.97	61.7
	05	101.8	0.29	0.009	0.033	3.37	107.4
	06	66.9	0.30	0.009	0.043	2.54	79.5
	07						8.5
	08	67.4	0.23	0.007	0.031	2.75	153.8
	09	100.2	0.26	0.006	0.033	2.46	86.5
	Sub Total	**525.7**	**0.29**	**0.009**	**0.035**	**2.94**	**538.0**
Pampacancha	PC01	22.8	0.56	0.020	0.356	4.42	39.5
	PC02	17.1	0.64	0.014	0.365	5.12	40.5
	Sub Total	**39.9**	**0.60**	**0.018**	**0.360**	**4.72**	**79.9**
Stockpiles	I - V	**3.1**	**0.41**	**0.010**	**0.068**	**4.68**	
	TOTAL	**568.6**	**0.32**	**0.010**	**0.058**	**3.07**	**618.0**

16.3 MINE PRODUCTION SCHEDULE

16.3.1 PRODUCTION SCHEDULING CRITERIA

The operating and scheduling criteria used to develop the mining sequence plans are summarized in Table 16-4. Pit and mine maintenance operations are scheduled around the clock. Allowances for down time and weather delays have been included in the mine equipment and manpower estimations.

Table 16-4 Mine Production Schedule Criteria

Annual Moved Production Base Rate	**70.4 Mtonnes**
Annual Ore Production Base Rate	**30.9 - 31.4 Mtonnes**
Daily Ore Production Base Rate	90 - 94 ktpd
Process Plant yearly availability	94%
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	3

16.3.2 MILL FEED AND CUT-OFF GRADE STRATEGY

The mine plan criterias are summarized in Table 16-5.

Table 16-5 Mine Plan Criteria

SOURCE	INFO	COMMENT
1. Planning	Mine Plan	Yearly plan from 2018 to 2036. Ore restrictions: Hy + Skrn : CuOx% < 20% & Zn%/Cu% < 25% & Fe ≤10.0%. Sp + Mx : CuOx% < 20% & Zn%/Cu% < 25% & Fe ≤10.0%.
	Survey	Actual Topography from Jan 1st, 2018
	CutOff	Constancia: NSR > 6.14 $/t Pampacancha: NSR > 6.14 $/t
	Phases / Designs	Constancia: PH01, PH02, PH04, PH05, PH06, PH07, PH08 & PH09 (Update 2017) Pampacancha: PC01 & PC02 (Update 2017)
	Productivity	EX5600 shovel: 3,700 tph 994 loader: 1,800 tph
2. Maintenance	Equipment	# Shovel: 03 (Availability by Maintenance) # Loader: 01 - Only Stocks (Availability by Maintenance) # Trucks: 22-28 Trucks (Availability by Maintenance) # Dozer: 04
3. Geotech	Pit geometric parameters	Slope angle: 57 - 66 degrees IRA angle: 38 - 46 degrees (according with last Geotechnical Design)
	Dump geometric parameters	WRF Design (Update 2017)
		According to Geotech recommendations.
4. Geology	Geological model	Constancia Model: LTP CS Model 2018_v6_20x20x15 Pampacancha Model: LTP PC Model 2017_v4_20x20x15
	Recoveries	Metallurgical recoveries through Formula (Feb 2018) supplied by the process team
	Grades	Cu, Mo, Au, Ag, etc
5. Metallurgy	Metallurgical parameters	Keep grades (range 0.3% - 0.8%) Concentrate constraints (Pb<1.5%, Zn<3.0% and Pb+Zn<4.0%)
	Tonnage to Plant	be considered a production of 90,000 t/d & mechanical availability of 94% (84,600 t/d)
6. Production	Cu Fine Tonnes	Between 100Kt - 110kt according corporate targets (from 2018 to 2023)
7. Previous	Estimated LRP	Parameters updated and based on NI-43101 (2016)

An elevated cut-off grade strategy was implemented in order to bring forward the higher grade ore from the pit. Delivering higher grade ore to the mill in the early years will improve the net present value and internal rate of return of the operations.

NSR values are calculated for each block in the resource model to represent the net Cu, Mo, Au and Ag metal values. The pit reserves are estimated using an NSR cut-off value of 6.14 $/t.

This is the minimum value for mineralized material to cover the processing and G&A costs.

Priority plant feed consists of high grade material (NSR grade above 10 $/t). The lower grade material (NSR below 10 $/t) is being fed as needed or otherwise sent to the long term stockpile to be reclaimed at the end of mine life.

16.3.3 OVERBURDEN STRIPPING REQUIREMENTS

Mineral reserve tabulations by bench, phase and mine production scheduling program (MSSO and MSReserves, modules from MineSight® software) are used to analyze long-term stripping requirements for the Constancia Operations. Elevation and phase order dependencies and sinking rate controls are used in conjunction with mill ore production targets and an internal NSR cutoff of 6.14 $/t 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 analysis determined a minimum production stripping requirement of approximately 40 million tonnes of waste and 31 million tonnes of ore by year. Approximately 34 million tonnes of ore will also be mined and stockpiled during this period and will be reclaimed when the plant feed requirements exceeds the daily mine production.

16.3.4 MINE PLAN

The mine production plan includes 618 Mt of waste and 565.5 Mt of ore (from both pits and stockpiles), yielding a stripping ratio (waste / ore) of 1.1. An average yearly mining rate of 70.4 Mtpy, with a maximum of 73 Mtpy, is required to provide nominal ore process feed rate of 31 Mtpy based on a variable throughput by ore type (90-94 ktpd and 94% availability). The ore production schedule for the operation presents two trends: from 2018 to 2023, the Cu grade is 0.40 % and from 2024 to 2036 Cu grade is 0.27%. This implies a LOM average grades of 0.32% Cu, 0.010% Mo, 0.058 g/t Au and 3.07 g/t Ag, for the remaining 19 years of the life of mine.

The destination flowsheet of Constancia operations is shown in Figure 16-19 while Table 16-6 presents the distances in km, between Mine Phases and the main facilities. The mine schedule drawings for life of mine are shown in
Figure 16-20 to Figure 16-28.

Figure 16-19 Destination Flowsheet

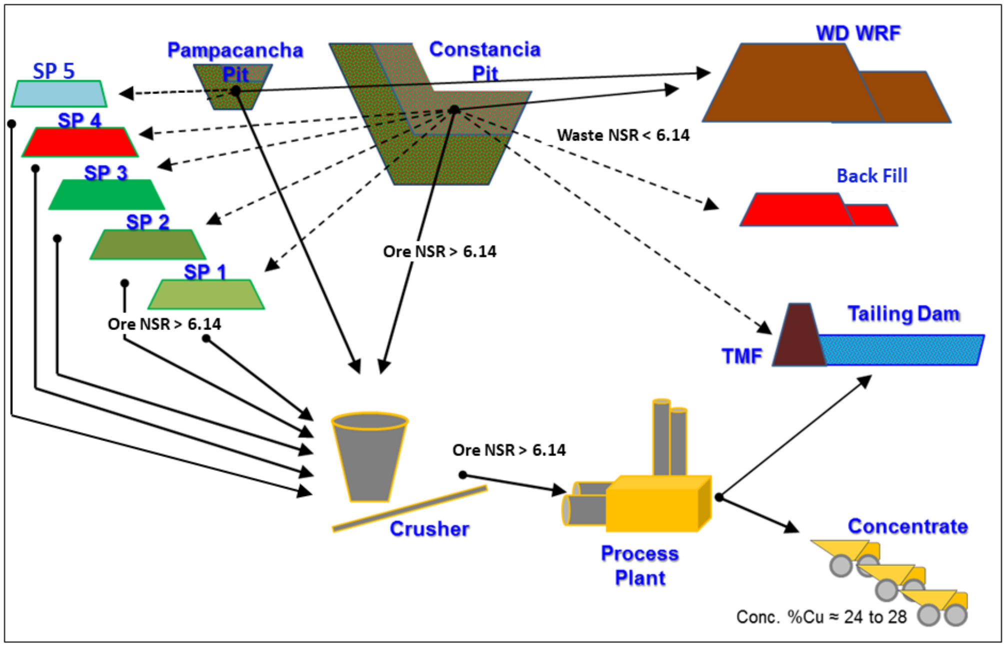


Table 16-6 Distances between Phases and Facilities

Phase	Destination	Distances(km)	Phase	Destination	Distances(km)
Ph01	Crusher	3.5	Ph06	Crusher	4.3
	TMF	7.3		BFPC	6.3
	WRF	4.5		TMF	8.0
	Stock 01	3.0		WRF	3.8
	Stock 02	2.9		Stock 02	3.6
	Stock 04	3.5		Stock 04	4.6
	Stock 05	3.6	Ph07	TMF	7.1
Ph02	Crusher	3.4		WRF	3.7
	TMF	7.6	Ph08	Crusher	3.0
	WRF	4.5		Stock 02	3.2
	Stock 01	2.7		TMF	8.5
	Stock 02	2.9		WRF	4.2
	Stock 04	3.1		BFPC	7.3
	Stock 05	3.4	Ph09	Crusher	4.2
Ph04	Crusher	3.6		TMF	8.4
	TMF	7.5		WRF	5.2
	WRF	4.0		Stock 04	4.9
	BFPC	8.7	PhP1	Crusher	6.8
	Stock 01	2.7		WRF	4.5
	Stock 02	3.1		Stock 03	4.5
	Stock 04	3.8	PhP2	Crusher	7.1
	Stock 05	3.4		WRF	4.5
Ph05	Crusher	3.7		Stock 01	7.2
	TMF	8.2	Stock 01	Crusher	0.6
	WRF	4.2	Stock 02	Crusher	1.1
	BFPC	7.6	Stock 03	Crusher	2.1
	Stock 01	3.6	Stock 04	Crusher	2.1
	Stock 02	3.4	Stock 05	Crusher	2.4
	Stock 04	3.0			
	Stock 05	2.9			

Figure 16-20 Mine Plan of Periods 2018 and 2019

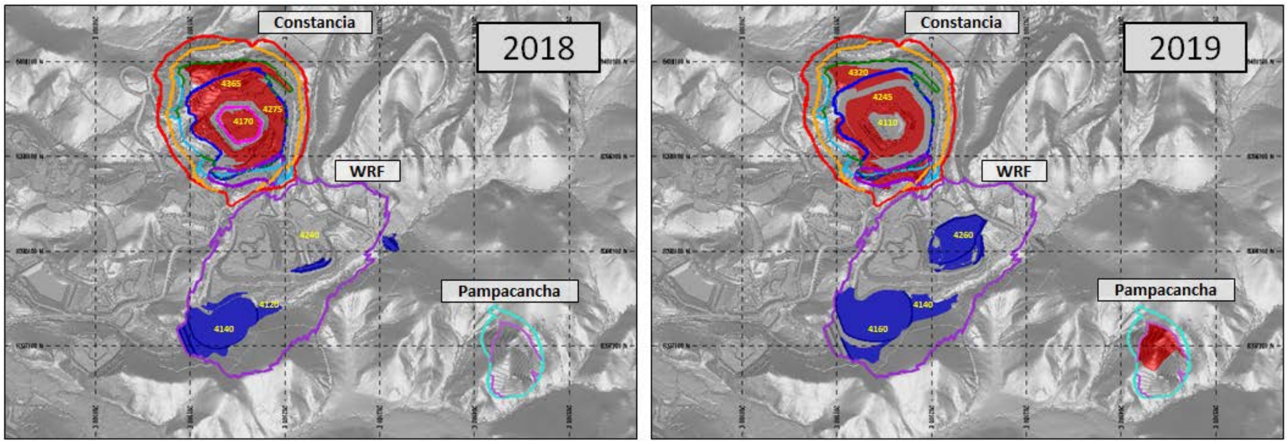


Figure 16-21 Mine Plan of Periods 2020 and 2021

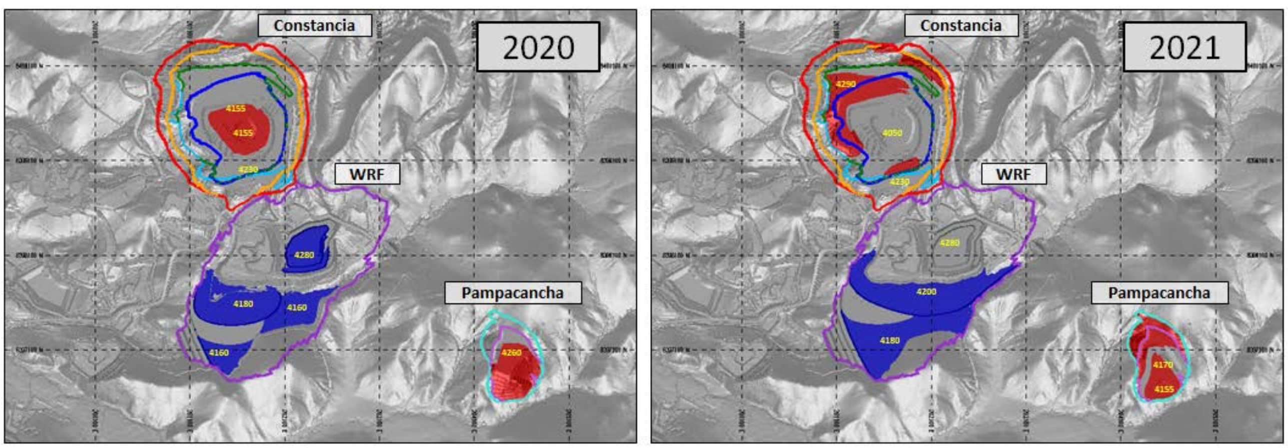


Figure 16-22 Mine Plan of Periods 2022 and 2023

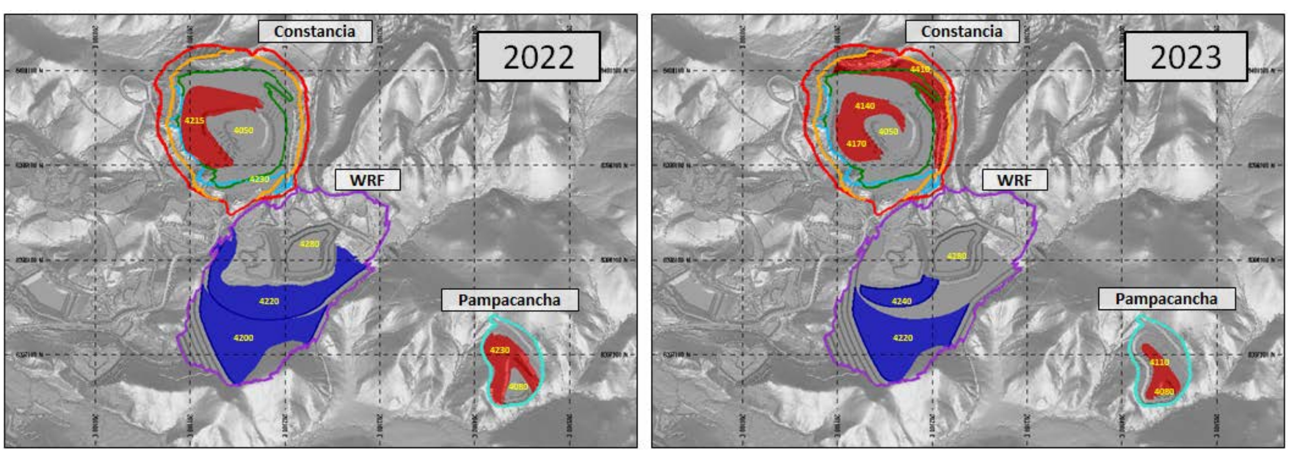


Figure 16-23 Mine Plan of Periods 2024 and 2025

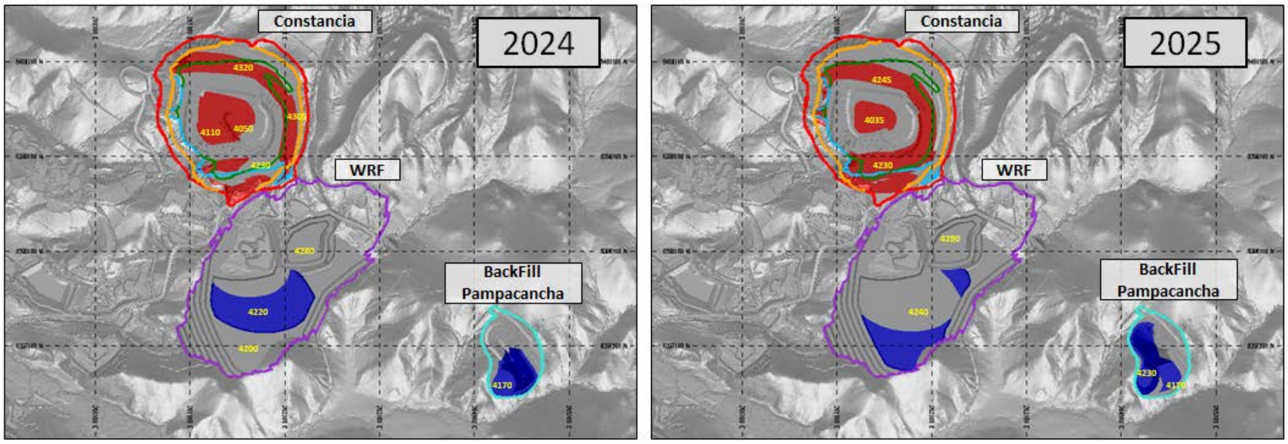


Figure 16-24 Mine Plan of Periods 2026 and 2027

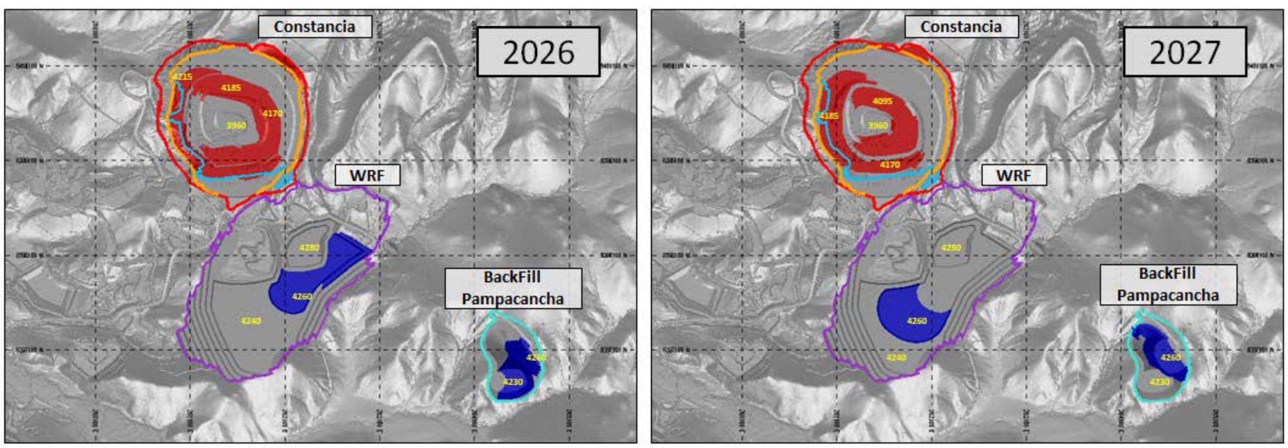


Figure 16-25 Mine Plan of Periods 2028 and 2029

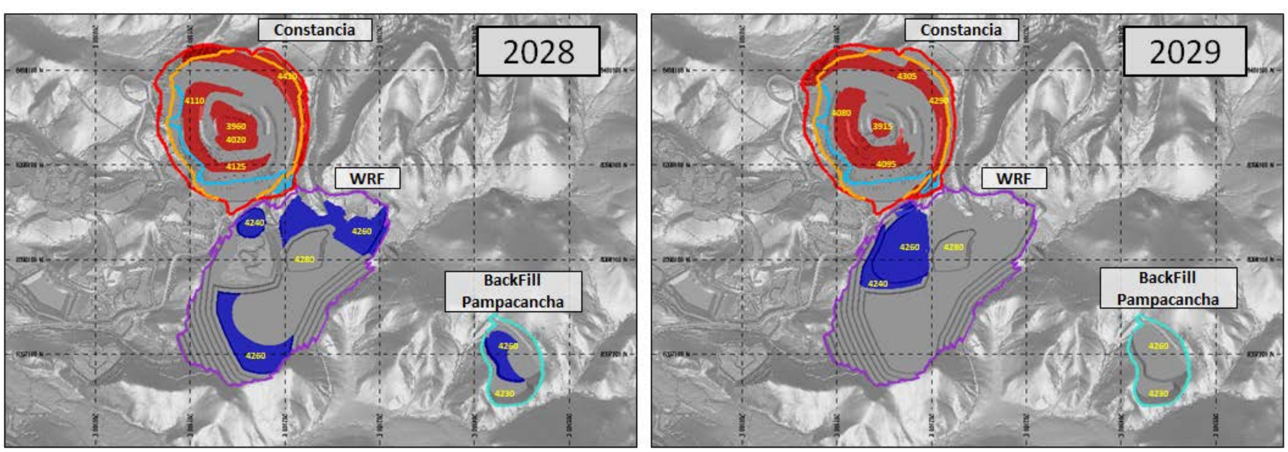


Figure 16-26 Mine Plan of Periods 2030 and 2031

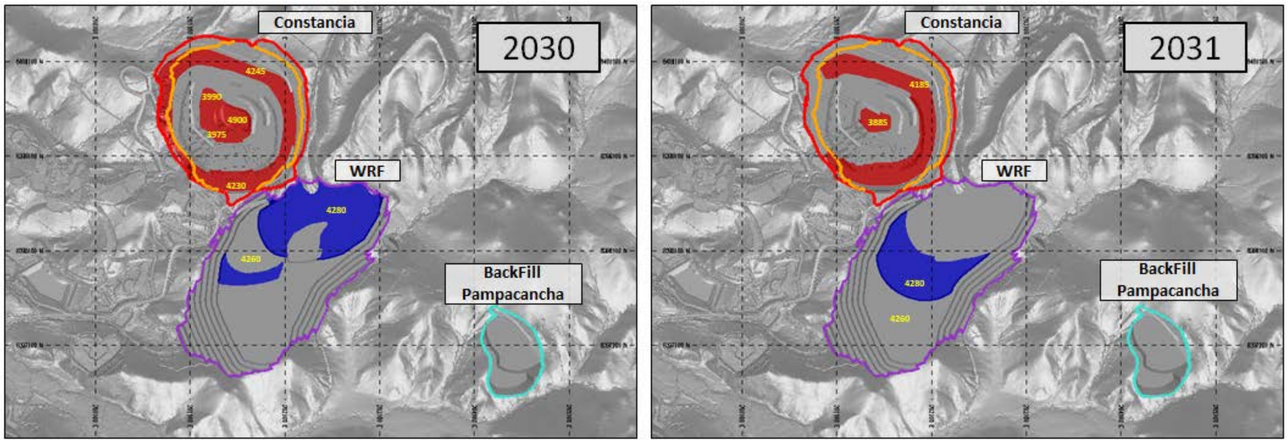


Figure 16-27 Mine Plan of Periods 2032 and 2033

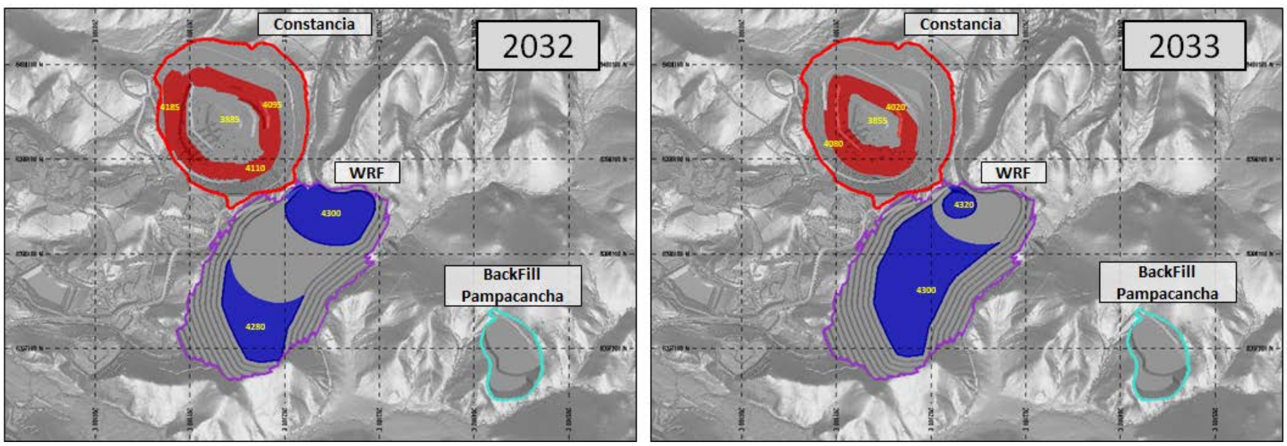
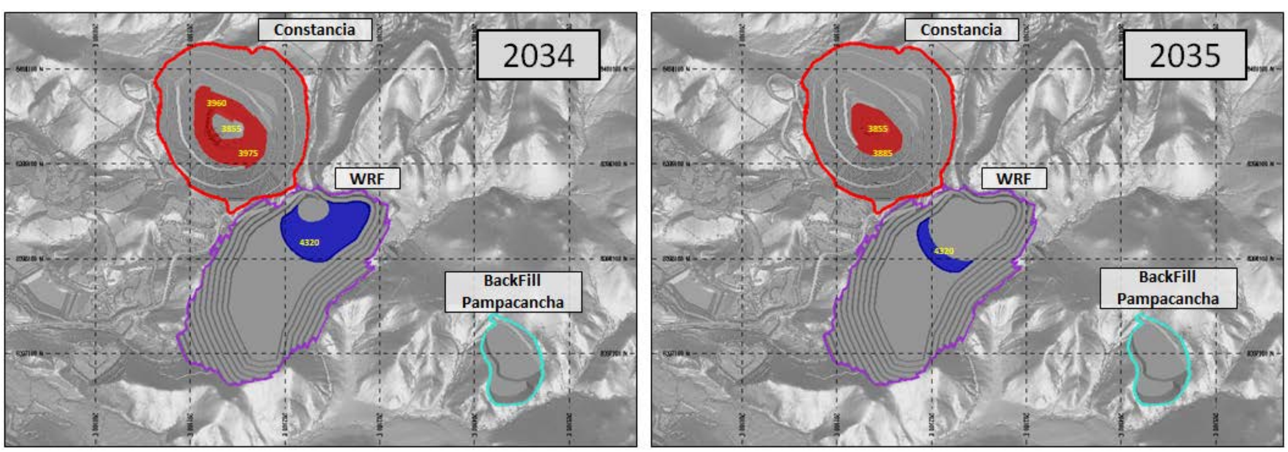


Figure 16-28 Mine Plan of Periods 2034 and 2035



The estimated mine production schedule is summarized in Table 16-7. Mine schedule details for the total annual movement by destination and by phases, stripping ratios, mill feed by ore type and by phases are presented in Figure 16-29 to Figure 16-33.

Table 16-7 Constancia Operations Mine Plan at December, 2017

Year	Days	Pits / Zones	Moved, Mt	Mined, Mt	Waste, Mt	Ore Pit to Stocks, Mt	Ore Pit to MILL, Mt	Ore Stock to MILL, Mt	Ore MILLED, Mt	Cu%	Mo%	Au ppm	Ag ppm	Fe%	Ratio CuOx	Ratio Zn/Cu	NSR, $/t
2018	365	Constancia	69.9	69.9	35.5	5.2	29.2		29.2	0.42	0.012	0.041	4.03	4%	9%	19%	23.9
		Stockpiles	2.2					2.2	2.2	0.41	0.010	0.067	4.64	7%	5%	50%	23.0
		SubTotal	**72.1**	**69.9**	**35.5**	**5.2**	**29.2**	**2.2**	**31.4**	**0.42**	**0.012**	**0.042**	**4.08**	**4%**	**9%**	**21%**	**23.9**
2019	365	Constancia	57.4	57.4	23.8	8.0	25.6		25.6	0.43	0.013	0.038	3.67	5%	5%	20%	24.5
		Pampacancha	12.7	12.7	8.7	0.0	4.0		4.0	0.28	0.018	0.247	3.54	18%	15%	21%	21.9
		Stockpiles	1.6					1.6	1.6	0.50	0.017	0.061	3.95	4%	4%	20%	29.1
		SubTotal	**71.8**	**70.1**	**32.5**	**8.0**	**29.7**	**1.6**	**31.3**	**0.41**	**0.013**	**0.066**	**3.67**	**6%**	**6%**	**20%**	**24.4**
2020	366	Constancia	32.3	32.3	6.6	3.8	21.9		21.9	0.33	0.018	0.023	3.02	4%	5%	28%	19.7
		Pampacancha	33.7	33.7	25.4	0.3	8.0		8.0	0.56	0.014	0.437	3.15	19%	14%	14%	39.6
		Stockpiles	1.3					1.3	1.3	0.37	0.014	0.043	3.08	4%	3%	19%	21.9
		SubTotal	**67.3**	**66.0**	**32.0**	**4.1**	**29.9**	**1.3**	**31.2**	**0.39**	**0.017**	**0.130**	**3.05**	**8%**	**8%**	**23%**	**24.9**
2021	365	Constancia	32.2	32.2	17.8		14.4		14.4	0.24	0.010	0.022	2.33	3%	4%	14%	14.1
		Pampacancha	33.5	33.5	20.2		13.2		13.2	0.58	0.026	0.292	5.07	22%	10%	22%	38.3
		Stockpiles	3.6					3.6	3.6	0.31	0.015	0.048	2.69	4%	4%	20%	19.1
		SubTotal	**69.2**	**65.7**	**38.1**	**0.0**	**27.6**	**3.6**	**31.1**	**0.39**	**0.017**	**0.139**	**3.53**	**11%**	**8%**	**20%**	**24.9**
2022	365	Constancia	39.7	39.7	18.9	0.8	20.0		20.0	0.27	0.010	0.038	2.98	4%	3%	15%	16.3
		Pampacancha	28.4	28.4	20.6		7.8		7.8	0.79	0.010	0.500	5.83	22%	10%	11%	52.7
		Stockpiles	3.2					3.2	3.2	0.16	0.009	0.010	1.51	2%	4%	15%	9.7
		SubTotal	**71.3**	**68.1**	**39.5**	**0.8**	**27.9**	**3.2**	**31.1**	**0.39**	**0.010**	**0.152**	**3.55**	**8%**	**7%**	**13%**	**24.8**
2023	365	Constancia	57.4	57.4	30.4	3.1	24.0		24.0	0.31	0.011	0.039	3.37	5%	4%	21%	18.7
		Pampacancha	11.5	11.5	5.0		6.5		6.5	0.67	0.016	0.310	5.44	21%	12%	15%	42.3
		Stockpiles	0.6					0.6	0.6	0.18	0.008	0.026	3.53	3%	18%	17%	11.7
		SubTotal	**69.6**	**68.9**	**35.4**	**3.1**	**30.5**	**0.6**	**31.2**	**0.39**	**0.012**	**0.095**	**3.81**	**8%**	**7%**	**19%**	**23.5**
2024	366	Constancia	66.8	66.8	31.8	4.1	30.9		30.9	0.31	0.010	0.035	3.35	4%	4%	22%	18.6
		Stockpiles	0.2					0.2	0.2	0.28	0.007	0.196	7.26	4%	4%	23%	21.2
		SubTotal	**67.0**	**66.8**	**31.8**	**4.1**	**30.9**	**0.2**	**31.1**	**0.31**	**0.010**	**0.036**	**3.38**	**4%**	**4%**	**22%**	**18.6**
2025	365	Constancia	67.1	67.1	35.5	0.6	31.0		31.0	0.33	0.011	0.032	2.74	3%	4%	17%	19.0
		SubTotal	**67.1**	**67.1**	**35.5**	**0.6**	**31.0**		**31.0**	**0.33**	**0.011**	**0.032**	**2.74**	**3%**	**4%**	**17%**	**19.0**
2026	365	Constancia	67.6	67.6	36.0	0.6	31.0		31.0	0.26	0.010	0.033	3.00	3%	5%	25%	15.9
		SubTotal	**67.6**	**67.6**	**36.0**	**0.6**	**31.0**		**31.0**	**0.26**	**0.010**	**0.033**	**3.00**	**3%**	**5%**	**25%**	**15.9**
2027	365	Constancia	68.4	68.4	38.6		29.7		29.7	0.26	0.009	0.034	2.96	4%	5%	27%	15.3
		Stockpiles	1.3					1.3	1.3	0.16	0.007	0.020	1.70	3%	4%	20%	9.7
		SubTotal	**69.7**	**68.4**	**38.6**		**29.7**	**1.3**	**31.1**	**0.25**	**0.008**	**0.034**	**2.91**	**4%**	**5%**	**26%**	**15.1**
2028	366	Constancia	73.0	73.0	40.3	1.6	31.1		31.1	0.28	0.009	0.033	2.70	3%	3%	20%	16.4
		SubTotal	**73.0**	**73.0**	**40.3**	**1.6**	**31.1**		**31.1**	**0.28**	**0.009**	**0.033**	**2.70**	**3%**	**3%**	**20%**	**16.4**
2029	365	Constancia	72.8	72.8	41.5	0.3	31.0		31.0	0.30	0.008	0.043	3.19	5%	3%	30%	17.4
		SubTotal	**72.8**	**72.8**	**41.5**	**0.3**	**31.0**		**31.0**	**0.30**	**0.008**	**0.043**	**3.19**	**5%**	**3%**	**30%**	**17.4**
2030	365	Constancia	71.8	71.8	40.3	0.5	31.0		31.0	0.30	0.007	0.042	2.65	4%	3%	18%	17.3
		SubTotal	**71.8**	**71.8**	**40.3**	**0.5**	**31.0**		**31.0**	**0.30**	**0.007**	**0.042**	**2.65**	**4%**	**3%**	**18%**	**17.3**
2031	365	Constancia	66.7	66.7	39.1		27.6		27.6	0.25	0.007	0.035	2.50	3%	5%	18%	15.0
		Stockpiles	3.5					3.5	3.5	0.26	0.007	0.085	4.55	5%	4%	65%	15.9
		SubTotal	**70.2**	**66.7**	**39.1**		**27.6**	**3.5**	**31.1**	**0.25**	**0.007**	**0.040**	**2.74**	**3%**	**5%**	**23%**	**15.1**
2032	366	Constancia	66.3	66.3	43.5		22.8		22.8	0.27	0.009	0.038	2.82	3%	6%	17%	16.0
		Stockpiles	8.4					8.4	8.4	0.19	0.006	0.035	2.93	4%	7%	47%	11.6
		SubTotal	**74.7**	**66.3**	**43.5**		**22.8**	**8.4**	**31.2**	**0.25**	**0.008**	**0.037**	**2.85**	**4%**	**6%**	**23%**	**14.8**
2033	365	Constancia	72.0	72.0	40.5	0.5	31.0		31.0	0.23	0.006	0.031	2.46	4%	3%	23%	13.7
		SubTotal	**72.0**	**72.0**	**40.5**	**0.5**	**31.0**		**31.0**	**0.23**	**0.006**	**0.031**	**2.46**	**4%**	**3%**	**23%**	**13.7**
2034	365	Constancia	48.9	48.9	14.3	3.6	31.0		31.0	0.27	0.005	0.031	2.62	5%	3%	23%	15.5
		SubTotal	**48.9**	**48.9**	**14.3**	**3.6**	**31.0**		**31.0**	**0.27**	**0.005**	**0.031**	**2.62**	**5%**	**3%**	**23%**	**15.5**
2035	365	Constancia	33.4	33.4	3.6		29.8		29.8	0.30	0.008	0.033	2.28	3%	2%	11%	17.0
		Stockpiles	1.1					1.1	1.1	0.20	0.004	0.042	2.96	6%	4%	125%	10.9
		SubTotal	**34.5**	**33.4**	**3.6**		**29.8**	**1.1**	**30.9**	**0.29**	**0.007**	**0.033**	**2.31**	**3%**	**2%**	**13%**	**16.8**
2036	366	Stockpiles	8.9					8.9	8.9	0.19	0.006	0.036	3.40	4%	11%	57%	11.4
		SubTotal	**8.9**					**8.9**	**8.9**	**0.19**	**0.006**	**0.036**	**3.40**	**4%**	**11%**	**57%**	**11.4**
TOTAL			**1,220**	**1,184**	**618**	**32.9**	**532.6**	**36.0**	**568.6**	**0.32**	**0.010**	**0.058**	**3.08**	**5%**	**5%**	**21%**	**18.9**

Figure 16-29 Total Movement by Destination

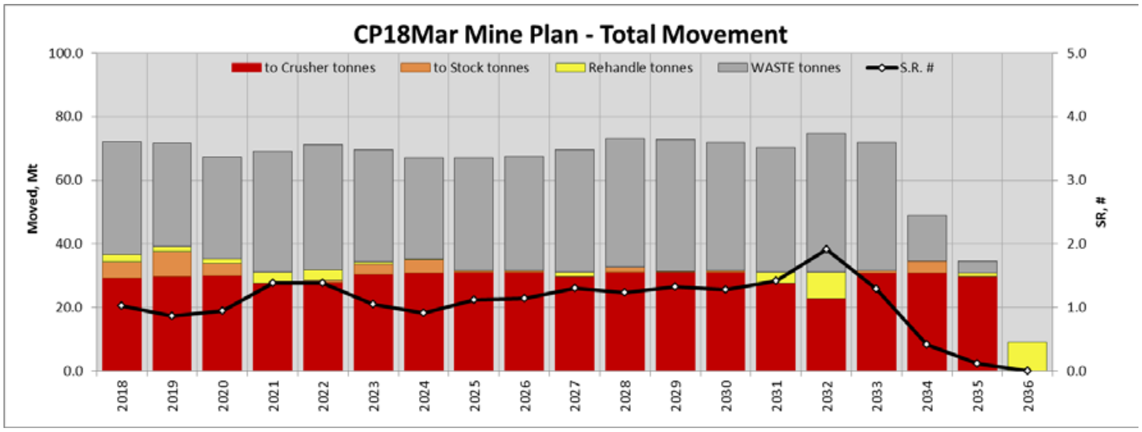


Figure 16-30 Total Movement by Phase

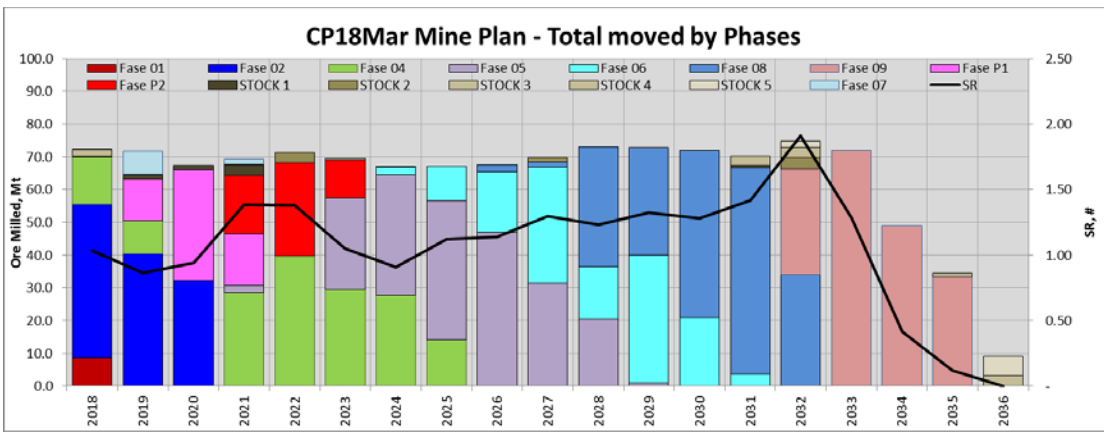


Figure 16-31 Mill Feed by Ore type

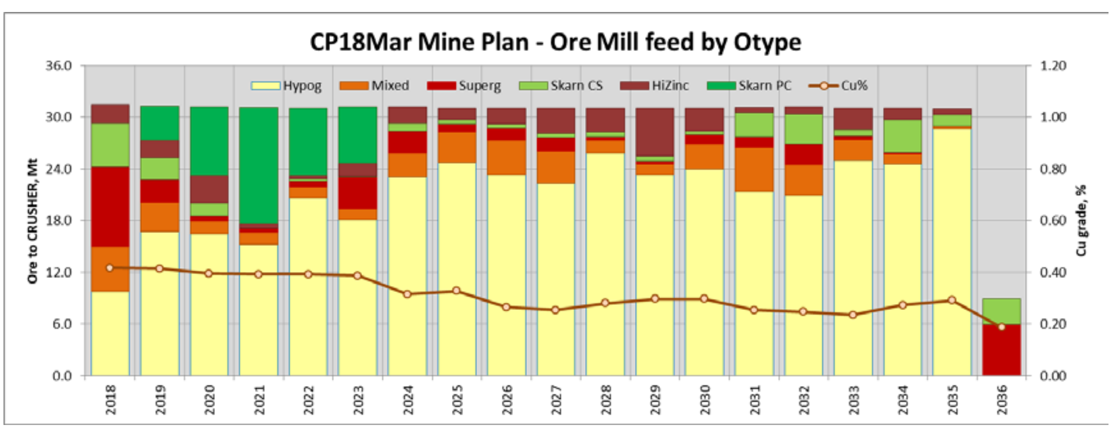


Figure 16-32 Ore Milled % by Ore Type

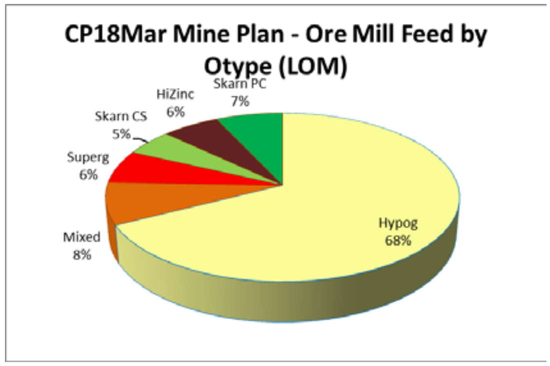
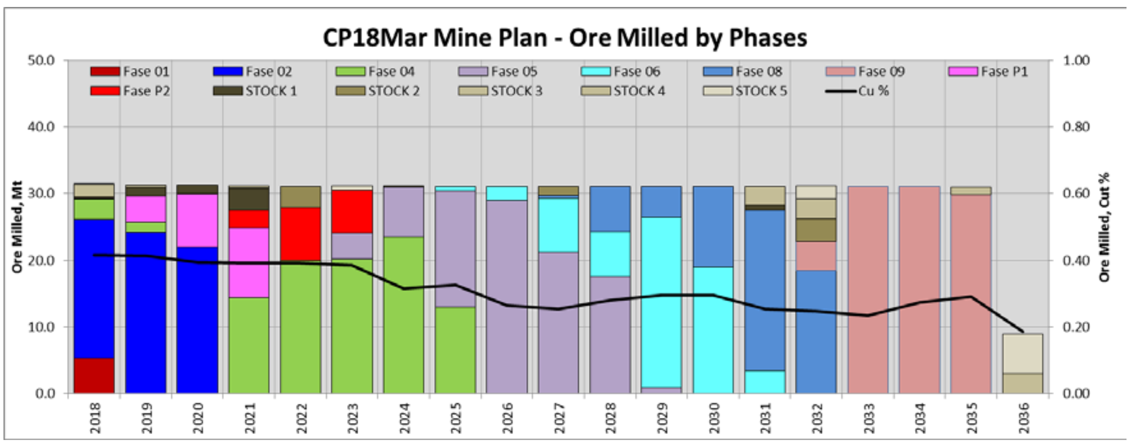


Figure 16-33 Ore Tonnage Milled by Phase



The processing schedule corresponding to the combined LOM plans of Constancia and Pampacancha, is illustrated in Figure 16-34 and in Table 16-8 to Table 16-10.

Figure 16-34 Total Copper Production (2018 – 2036) in Concentrate

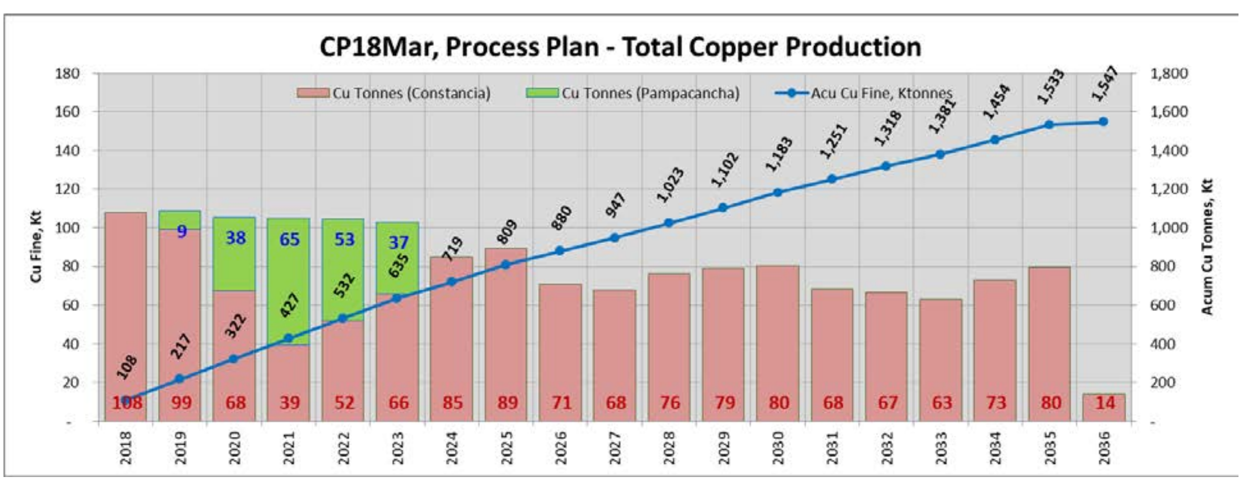


Table 16-8 Constancia Ore Processing Plan 2018 – 2036

LOM		CP18Mar - CONSTANCIA																			TOTAL
Year		2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
PRODUCTION	Tonnes to Cu Plant, kt	31,418	27,258	23,197	17,631	23,212	24,623	31,148	31,004	31,039	31,056	31,054	31,004	31,017	31,070	31,181	30,986	30,980	30,913	8,949	528,742
	Cu%	0.42	0.43	0.34	0.25	0.26	0.31	0.31	0.33	0.26	0.25	0.28	0.30	0.30	0.25	0.25	0.23	0.27	0.29	0.19	0.29
	Cu In Situ, Mlb	288	260	172	99	131	169	216	223	181	173	192	202	202	174	170	160	187	199	37	3,435
	Au (g/t)	0.04	0.04	0.02	0.02	0.03	0.04	0.04	0.03	0.03	0.03	0.03	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.04	0.04
	Ag (g/t)	4.08	3.69	3.02	2.41	2.78	3.37	3.38	2.74	3.00	2.91	2.70	3.19	2.65	2.74	2.85	2.46	2.62	2.30	3.40	2.95
	Tonnes to Mo Plant, kt	6,391	12,084	17,155	13,990	17,155	14,716	19,227	19,247	16,656	12,699	17,603	15,196	13,962	14,239	13,406	6,526	8,668	19,947		258,868
	Mo%	0.017	0.017	0.021	0.012	0.011	0.014	0.011	0.015	0.013	0.014	0.012	0.012	0.010	0.010	0.012	0.008	0.009	0.009		0.013
RECOVERIES	CuT Recovery, %	82.6%	84.6%	86.4%	87.7%	87.3%	86.1%	86.4%	88.2%	86.5%	86.1%	87.5%	86.1%	87.3%	86.8%	86.6%	86.5%	86.3%	88.3%	83.8%	86.3%
	Au (g/t) Recovery, %	52.0%	58.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	59.3%
	Ag (g/t) Recovery, %	66.5%	67.3%	66.8%	70.6%	70.9%	69.8%	69.3%	69.6%	69.3%	68.7%	68.3%	66.3%	69.5%	68.8%	68.2%	67.5%	67.0%	70.4%	64.7%	68.4%
	MoT Recovery, %	25.0%	35.0%	48.0%	53.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%		51.9%
CONC. Cu	Conc. Tonnes	433,535	400,181	276,884	159,622	213,094	272,634	349,145	356,448	292,606	281,709	308,966	327,739	324,472	280,758	274,855	261,961	306,350	318,724	58,932	5,498,613
	Cu% into Conc.	24.9%	25.0%	24.4%	24.6%	24.4%	24.2%	24.2%	25.0%	24.3%	24.1%	24.6%	24.1%	24.7%	24.4%	24.3%	24.0%	23.8%	25.0%	23.6%	24.5%
	Au (g/t) into Conc.	1.6	1.6	1.2	1.6	2.2	2.1	1.9	1.7	2.1	2.2	2.0	2.4	2.4	2.7	2.5	2.2	1.9	1.9	3.3	2.0
	Ag (g/t) into Conc.	196.4	168.9	169.2	188.0	214.8	212.6	208.8	165.8	220.7	220.1	185.6	200.0	176.0	208.4	220.7	196.6	177.8	157.3	333.7	194.0
CONTAINED METAL IN CONC. Cu	Cu Pounds, Mlb	237.8	220.2	148.8	86.7	114.6	145.5	186.6	196.7	156.5	149.4	167.8	174.0	176.7	150.9	147.2	138.4	161.0	175.5	30.7	2,965
	Cu Fine, KTonnes	107.9	99.9	67.5	39.3	52.0	66.0	84.7	89.2	71.0	67.8	76.1	78.9	80.1	68.4	66.8	62.8	73.0	79.6	13.9	1,345
	Acum Cu Fine, KTonnes	108	208	275	315	367	433	517	606	677	745	821	900	980	1,049	1,116	1,178	1,251	1,331	1,345	
	Au Fine, KOunces	22	20	11	8	15	18	22	19	20	20	20	26	25	24	23	18	19	20	6	355
	Ag Fine, KOunces	2,738	2,173	1,507	965	1,471	1,864	2,343	1,900	2,076	1,994	1,844	2,108	1,836	1,881	1,951	1,656	1,751	1,612	632	34,302
CONC. Mo	Conc. Mo Tonnes	532	1,408	3,501	1,732	2,102	2,195	2,364	3,088	2,461	1,886	2,308	1,934	1,488	1,490	1,704	581	873	2,016		33,662
	Mo% into Conc. Mo	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%		50.0%
CONTAINED Mo	Mo Fine, KTonnes	0.3	0.7	1.8	0.9	1.1	1.1	1.2	1.5	1.2	0.9	1.2	1.0	0.7	0.7	0.9	0.3	0.4	1.0		17

Table 16-9 Pampacancha Ore Processing Plan 2019 – 2023

LOM		CP18Mar - PAMPACANCHA																			TOTAL
Year		2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
PRODUCTION	Tonnes to Cu Plant, kt		4,003	8,001	13,497	7,840	6,533														39,875
	Cu%		0.28	0.56	0.57	0.79	0.67														0.60
	Cu In Situ, Mlb		25	99	170	137	96														526
	Au (g/t)		0.25	0.44	0.29	0.50	0.31														0.36
	Ag (g/t)		3.54	3.15	5.00	5.83	5.44														4.72
	Tonnes to Mo Plant, kt			5,446	12,665	4,044	4,988														27,144
	Mo%			0.019	0.027	0.016	0.020														0.022
RECOVERIES	CuT Recovery, %		85.0%	85.0%	85.0%	85.0%	85.0%														85.0%
	Au (g/t) Recovery, %		58.0%	60.0%	60.0%	60.0%	60.0%														59.9%
	Ag (g/t) Recovery, %		70.0%	70.0%	70.0%	70.0%	70.0%														70.0%
	MoT Recovery, %			48.0%	53.0%	55.0%	55.0%														52.7%
CONC. Cu	Conc. Tonnes		33,189	131,374	230,658	180,130	128,320														703,671
	Cu% into Conc.		28.5%	29.0%	28.4%	29.2%	28.9%														28.8%
	Au (g/t) into Conc.		17.3	16.0	10.2	13.1	9.5														12.2
	Ag (g/t) into Conc.		298.5	134.2	204.8	177.6	193.8														187.1
CONTAINED METAL IN CONC. Cu	Cu Pounds, Mlb		20.9	83.9	144.3	116.0	81.6														447
	Cu Fine, KTonnes		9.5	38.1	65.4	52.6	37.0														203
	Acum Cu Fine, KTonnes		9	48	113	166	203														
	Au Fine, KOunces		18	67	76	76	39														276
	Ag Fine, KOunces		319	567	1,518	1,029	800														4,232
CONC. Mo	Conc. Mo Tonnes			993	3,617	709	1,088														6,407
	Mo% into Conc. Mo			50.0%	50.0%	50.0%	50.0%														50.0%
CONTAINED Mo	Mo Fine, KTonnes			0.5	1.8	0.4	0.5														3

Table 16-10 Ore Processing Plan 2018 – 2036 (Total)

LOM		CP18Mar																			TOTAL
Year		2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	
PRODUCTION	Tonnes to Cu Plant, kt	31,418	31,261	31,198	31,129	31,052	31,156	31,148	31,004	31,039	31,056	31,054	31,004	31,017	31,070	31,181	30,986	30,980	30,913	8,949	568,617
	Cu%	0.42	0.41	0.39	0.39	0.39	0.39	0.31	0.33	0.26	0.25	0.28	0.30	0.30	0.25	0.25	0.23	0.27	0.29	0.19	0.32
	Cu In Situ, Mlb	288	285	271	269	268	265	216	223	181	173	192	202	202	174	170	160	187	199	37	3,961
	Au (g/t)	0.04	0.07	0.13	0.14	0.15	0.10	0.04	0.03	0.03	0.03	0.03	0.04	0.04	0.04	0.04	0.03	0.03	0.03	0.04	0.06
	Ag (g/t)	4.08	3.67	3.05	3.53	3.55	3.80	3.38	2.74	3.00	2.91	2.70	3.19	2.65	2.74	2.85	2.46	2.62	2.30	3.40	3.07
	Tonnes to Mo Plant, kt	6,391	12,084	22,601	26,656	21,199	19,704	19,227	19,247	16,656	12,699	17,603	15,196	13,962	14,239	13,406	6,526	8,668	19,947		286,012
	Mo%	0.017	0.017	0.021	0.019	0.012	0.015	0.011	0.015	0.013	0.014	0.012	0.012	0.010	0.010	0.012	0.008	0.009	0.009		0.013
RECOVERIES	CuT Recovery, %	82.6%	84.6%	85.9%	86.0%	86.1%	85.7%	86.4%	88.2%	86.5%	86.1%	87.5%	86.1%	87.3%	86.8%	86.6%	86.5%	86.3%	88.3%	83.8%	86.1%
	Au (g/t) Recovery, %	52.0%	58.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	59.6%
	Ag (g/t) Recovery, %	66.5%	67.6%	67.7%	70.2%	70.6%	69.9%	69.3%	69.6%	69.3%	68.7%	68.3%	66.3%	69.5%	68.8%	68.2%	67.5%	67.0%	70.4%	64.7%	68.6%
	MoT Recovery, %	25.0%	35.0%	48.0%	53.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	55.0%	52.0%
CONC. Cu	Conc. Tonnes	433,535	433,369	408,257	390,279	393,224	400,955	349,145	356,448	292,606	281,709	308,966	327,739	324,472	280,758	274,855	261,961	306,350	318,724	58,932	6,202,284
	Cu% into Conc.	24.9%	25.2%	25.9%	26.8%	26.6%	25.7%	24.2%	25.0%	24.3%	24.1%	24.6%	24.1%	24.7%	24.4%	24.3%	24.0%	23.8%	25.0%	23.6%	25.0%
	Au (g/t) into Conc.	1.6	2.8	6.0	6.7	7.2	4.5	1.9	1.7	2.1	2.2	2.0	2.4	2.4	2.7	2.5	2.2	1.9	1.9	3.3	3.2
	Ag (g/t) into Conc.	196.4	178.8	158.0	197.9	197.7	206.6	208.8	165.8	220.7	220.1	185.6	200.0	176.0	208.4	220.7	196.6	177.8	157.3	333.7	193.2
CONTAINED METAL IN CONC. Cu	Cu Pounds, Mlb	237.8	241.0	232.7	231.0	230.6	227.1	186.6	196.7	156.5	149.4	167.8	174.0	176.7	150.9	147.2	138.4	161.0	175.5	30.7	3,412
	Cu Fine, KTonnes	107.9	109.3	105.6	104.8	104.6	103.0	84.7	89.2	71.0	67.8	76.1	78.9	80.1	68.4	66.8	62.8	73.0	79.6	13.9	1,548
	Acum Cu Fine, KTonnes	108	217	323	428	532	635	720	809	880	948	1,024	1,103	1,183	1,251	1,318	1,381	1,454	1,534	1,548	
	Au Fine, KOunces	22	39	78	84	91	57	22	19	20	20	20	26	25	24	23	18	19	20	6	631
	Ag Fine, KOunces	2,738	2,492	2,074	2,483	2,500	2,663	2,343	1,900	2,076	1,994	1,844	2,108	1,836	1,881	1,951	1,656	1,751	1,612	632	38,534
CONC. Mo	Conc. Mo Tonnes	532	1,408	4,494	5,349	2,811	3,283	2,364	3,088	2,461	1,886	2,308	1,934	1,488	1,490	1,704	581	873	2,016		40,069
	Mo% into Conc. Mo	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%		50.0%
CONTAINED Mo	Mo Fine, KTonnes	0.3	0.7	2.2	2.7	1.4	1.6	1.2	1.5	1.2	0.9	1.2	1.0	0.7	0.7	0.9	0.3	0.4	1.0		20

16.4 MINE FACILITIES

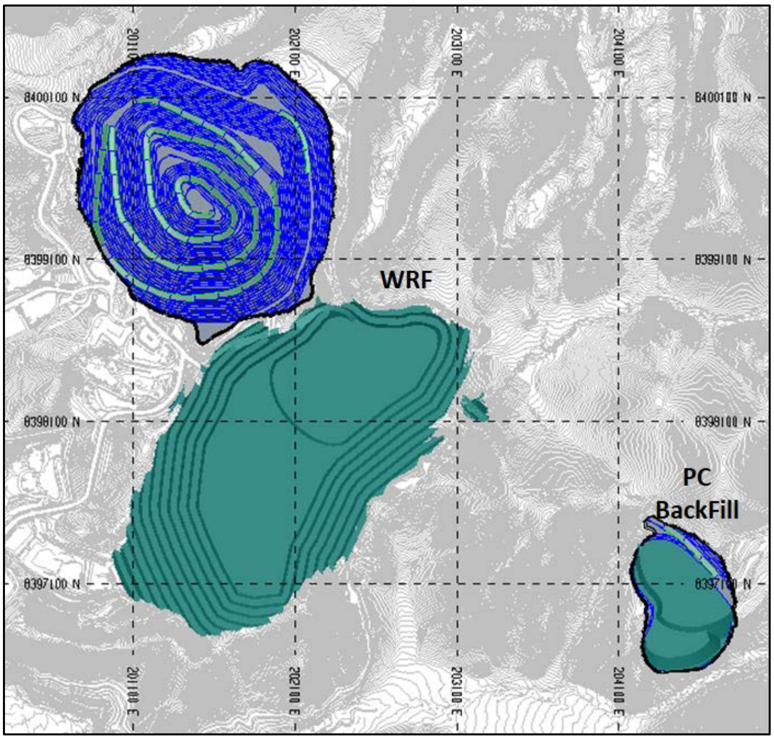
16.4.1 WASTE ROCK FACILITY (WRF)

The final design geometry for this facility incorporates the same slope profile as the original WRF: 20 m high benches with 1.4H:1V (36°) of bench slopes and 32 m wide benches. The overall slope of the stockpile will be about 3.0H:1V (18°). The current remaining capacity of the WRF is estimated at 498 Mt. The waste material balance is shown in Table 16-11 and the Waste Rock Facility is represented on Figure 16-35.

Table 16-11 Constancia, Waste Material Balance

Waste	Mt
Total Waste, Mt	**618**
Waste to TMF, Mt	150
Waste to BackFill PC, Mt	66
Waste to WRF, Mt	**402**
WRF remain capacity, Mt	**498**
Difference, Mt	**96**

Figure 16-35 Waste Rock Facility

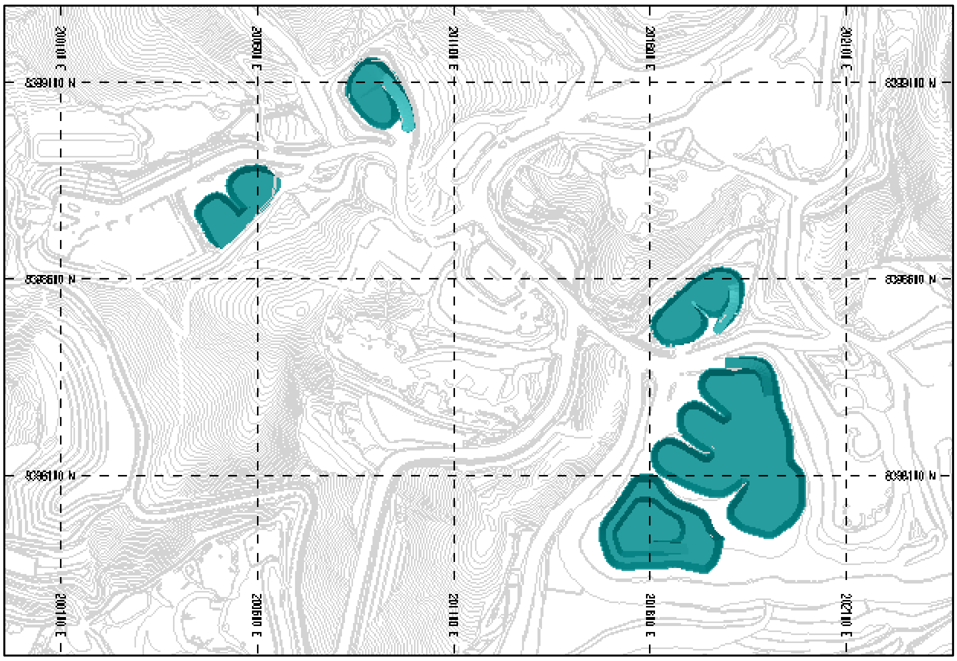


16.4.2 Ore Stockpiles

The Constancia mine is planned to have four operational stockpiles (Figure 16-36) which will segregate the ROM material by ore type and grade range. The lift height of these stockpiles ranges between 12 and 15 metres. The planned maximum stockpiles capacities are:

- Stockpile 01 : 600 Kt
- Stockpile 02 : 500 Kt
- Stockpile 03 : 600 Kt
- Stockpile 04 : 7,000 Kt

Figure 16-36 Location of Ore Stockpiles



16.4.3 Tailings Management Facility (TMF)

The Tailings Management Facility (TMF) area is where mill tailings are placed behind waste rock containment buttresses. The material for the buttresses will come throughout the life of the mine, from mining operation and quarries. The TMF area is located southwest of the mining facilities area. The design for the TMF includes ramps access to the lower and upper levels for the regular mining fleet (240 tonnes trucks) as well as a smaller fleet called HCW which has been used since October 2015 to send NAG material from the Constancia pit to the TMF area.

At the end of the impoundment, the final dam crest elevations will be:

- Tailings Dam equal to 4,190 masl

- Saddle Dam equal to 4,190 masl, where the height of the saddle dam will be 12 m, with 4 m of minimum freeboard.
- The average dry density of deposited tailings for this expansion was considered as 1.45 gr/cm^3 and will be revaluated at the next level of study.

The planning related to this mine operation facility was developed by Hudbay. The TMF Stacking Plan is illustrated in Table 16-12 and the final TMF design is presented in Figure 16-37.

Figure 16-37 TMF - Constancia

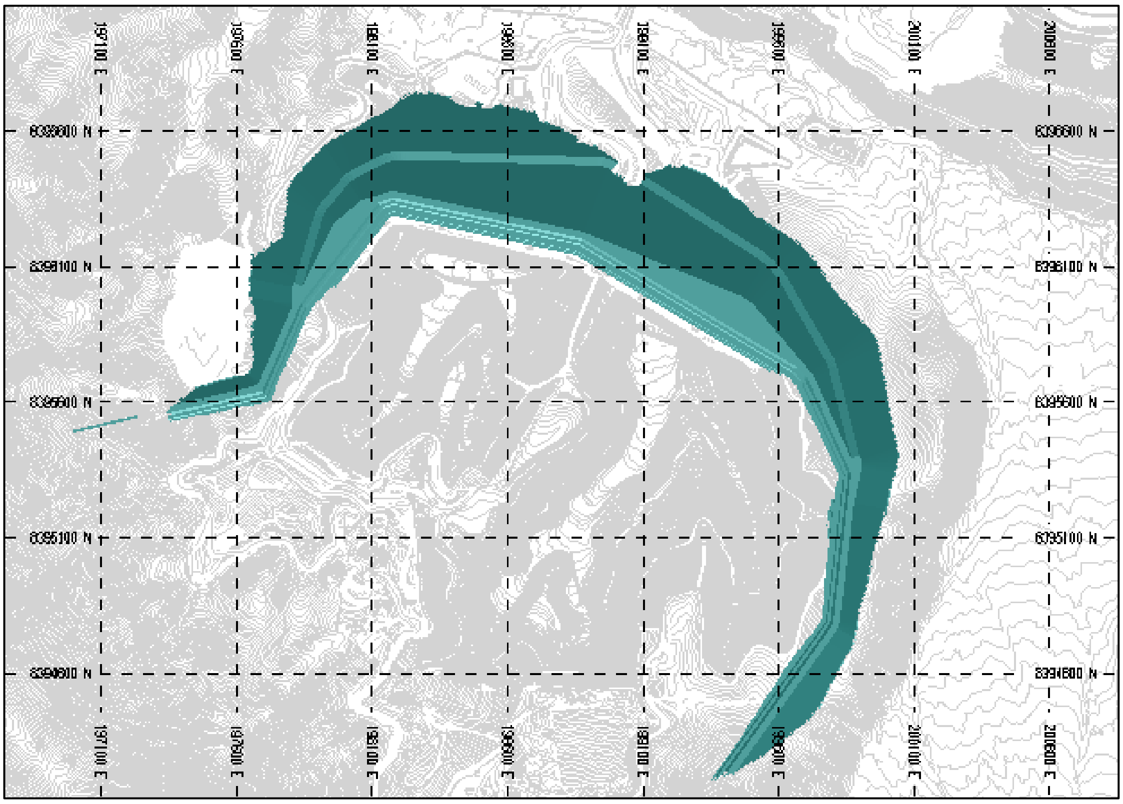


Table 16-12 Tailing Management Facility (TMF) Stacking Plan

TMF Stacking plan		Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	Total
Mining Fleet	Down stream	Mtonnes	12.0	5.3	0.8	3.8	1.7	8.5	0.8	6.2	8.4	10.1	12.1	13.9	14.3	17.2	1.7	
	Up stream	Mtonnes	0.0	1.9	3.6	0.0	0.0	2.6	0.0	3.1	0.0	3.3	0.0	3.2	0.0	4.8	4.6	
	Sub total	Mtonnes	12.0	7.2	4.3	3.8	1.7	11.1	0.8	9.3	8.4	13.4	12.1	17.1	14.3	22.0	6.2	143.6
HCW	Core, Trans / Filter	Mtonnes	0.0	0.3	0.7	0.0	0.0	0.5	0.0	0.5	0.0	0.8	0.0	0.8	0.0	1.3	1.6	
	D1/D2	Mtonnes	0.0	0.0	0.8	0.0	0.0	0.0	0.0	1.2	0.0	0.0	0.0	1.5	0.0	0.0	3.0	
	Sub total	Mtonnes	0.0	0.3	1.5	0.0	0.0	0.5	0.0	1.7	0.0	0.8	0.0	2.4	0.0	1.3	4.5	12.9
TOTAL		**Mtonnes**	**12.0**	**7.5**	**5.8**	**3.8**	**1.7**	**11.5**	**0.8**	**11.0**	**8.4**	**14.1**	**12.1**	**19.4**	**14.3**	**23.3**	**10.8**	**156.5**

16.5 MINE EQUIPMENT

Equipment Operating Parameter

Mine equipment was selected based on the production requirements as shown in Table 16-14 and Table 16-15. At the mine site, three crews operate with two twelve-hour shifts per day, 365 days of the year. No significant weather delays are expected and the mine will not be shut down for holidays.

The parametres used to determine productivity are listed in Table 16-13. There are several different rock types at Constancia, however, for production estimation the weighted average of all rock types was used. Major loading and haulage equipment is equipped with electronic monitors which optimize loading time. The production is reported in dry metric tonnes which is consistent with the reserve model. Moisture content is expected to range between 3.5 and 4.5 percent. Haulage calculations relied on a 4.0% moisture content.

Table 16-13 Material Characteristics

In Situ Bulk Density	2.4 cubic meter per tonnes
Material Swell	25 Percent
Loose Density	1.9 cubic meter per tonnes
Moisture Content	4.0 Percent

Mine Equipment Calculation

Mine equipment requirements were developed based on annual tonnage movements from the mine production schedule based on 15 metre bench heights, two twelve hour shifts per day, 365 days per year operation, manufacture machine specifications and material characteristics specific to the deposit (Figure 16-38 Truck Study Results).

A summary of fleet requirements by time period is shown in Table 16-14 and Table 16-15. This equipment is considered necessary to perform the following tasks:

- production drilling;
- loading and hauling ore to the primary crusher (located on the east side of the pit), waste rock facility (WRF) and to the tailings dam (TMF);
- maintaining mine haulage and access roads;and
- maintaining the waste rock facility (WRF) areas, ore stockpiles, tailing management facility (TMF) and berms and for the regrading of slopes and final surfaces.

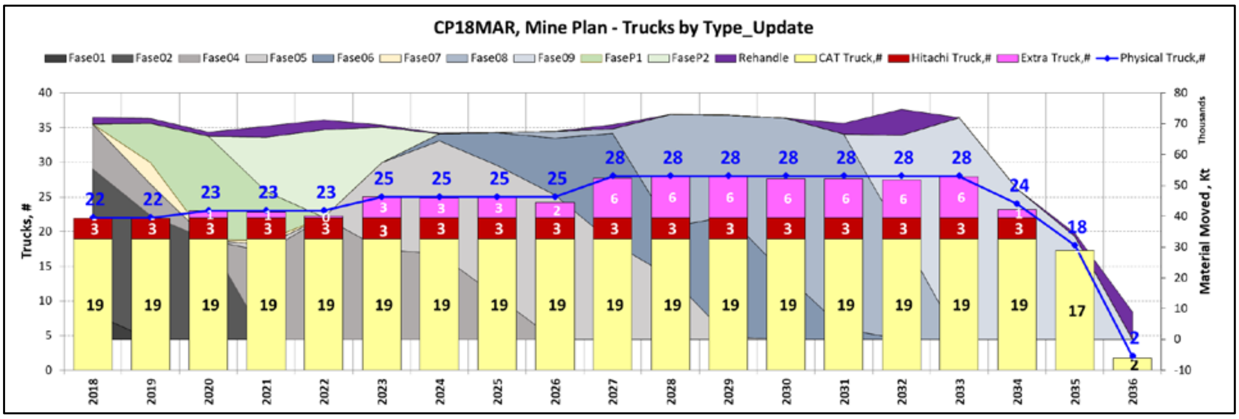
Table 16-14 Mine Equipment 2018 – 2027

Year		2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
MOVED	Mtonnes	72.1	71.8	67.3	69.2	71.3	69.6	67.0	67.1	67.6	69.7
Waste	Mtonnes	35.5	32.5	32.0	38.1	39.5	35.4	31.8	35.5	36.0	38.6
Ore	Mtonnes	34.4	37.7	34.0	27.6	28.6	33.6	35.0	31.6	31.6	29.7
Rehandle	Mtonnes	2.2	1.6	1.3	3.6	3.2	0.6	0.2	0.0	0.0	1.3
Shovels	#	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**
Loaders	#	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Trucks	#	**22**	**22**	**23**	**23**	**23**	**25**	**25**	**25**	**25**	**28**
Drills	#	3	3	3	3	3	3	3	3	3	3
Bulldozers	#	5	5	4	4	4	4	4	4	4	4
Whelldozers	#	2	2	2	2	2	2	2	2	2	2
Graders	#	3	3	3	3	3	3	3	3	3	3
Water trucks	#	2	2	2	2	2	2	2	2	2	2

Table 16-15 Mine Equipment 2028 – 2036

Year		2028	2029	2030	2031	2032	2033	2034	2035	2036
MOVED	Mtonnes	73.0	72.8	71.8	70.2	74.7	72.0	48.9	34.5	8.9
Waste	Mtonnes	40.3	41.5	40.3	39.1	43.5	40.5	14.3	3.6	0.0
Ore	Mtonnes	32.7	31.3	31.5	27.6	22.8	31.5	34.6	29.8	0.0
Rehandle	Mtonnes	0.0	0.0	0.0	3.5	8.4	0.0	0.0	1.1	8.9
Shovels	#	**3**	**3**	**3**	**3**	**3**	**3**	**2**	**1**	**1**
Loaders	#	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Trucks	#	**28**	**28**	**28**	**28**	**28**	**28**	**24**	**18**	**2**
Drills	#	3	3	3	3	3	3	2	1	-
Bulldozers	#	5	5	5	5	5	5	3	2	2
Whelldozers	#	2	2	2	2	2	2	2	2	1
Graders	#	3	3	3	3	3	3	3	3	2
Water trucks	#	2	2	2	2	2	2	2	2	1

Figure 16-38 Truck Study Results



16.6 MINE OPERATIONS

Open pit mining at the Constancia operation is based on conventional open pit mining techniques. The Constancia operation consists of two pits, Constancia and Pampacancha. The Constancia pit operation started in 2014 while Pampacancha will start in 2019.

The mine production plan contains 618 Mt of waste and 568.6 Mt of ore (from pit and stockpiles), yielding a stripping ratio (waste / ore) of 1.1 to 1. An average yearly mining rate of 70.4 Mtpy, with a maximum of 73 Mtpy, will be required to provide a nominal ore process feed rate of 31 Mtpy based on a variable throughput by ore type (90 to 94ktpd and 94% available). LOM average grades are 0.32% Cu, 0.010% Mo, 0.058 g/t Au and 3.07 g/t Ag, where the life of the mine is 19 years.

The priority to feed the process plant will involve optimizing the net value based on NSR ($/t), where high value material (HG) is fed first. The low value material (LG) will be fed as needed, sent to stockpiles, or will otherwise be sent to the waste rock facility (WRF).

16.6.1 DRILLING

For primary drilling production, single – pass 270 mm diameter (or 10 5/8”) drills are used. After the evaluation of equipment dimensioning, three drills PV-271 were selected in order to achieve the production rate. The penetration ratio of the drill is approximately 43 m/h.

Drill Patterns and Parametres

Drill patterns were designed only for ore and waste material, based on geomechanical characterization. Table 16-16 presents the parametres for the drilling production.

Table 16-16 Parametres for Drilling Production

Parameter	Unit	Ore	Waste
Burden	m	9	10
Spacing	m	10	10
Bench Height	m	15	15
Over Drilling	m	0.5	0.5
Density	**t/BCM**	**2.5**	**2.4**
Tonnage to remove	**t/hole**	**3,375**	**3,600**

16.6.2 BLASTING

For wet and dry holes, Heavy ANFO 73 (70% of Emulsion and 30% of ANFO) is used. Based on the evaluation of productivity, powder factors are estimated to be about 0.18 Kg/t for ore and 0.16 Kg/t for waste. The estimated blast design, explosive type, and powder factors for the Constancia operation are based on the mine operation results. The overview of blasting practices include the supply and transportation of explosives to the hole, loading, stemming, tie-in, evacuation of work and blast areas, and the ignition of the blast.

16.6.3 LOADING

Two 27 m^3 (Hitachi EX5600-6) shovels and a 19 m^3 (CAT 994H) loader are used to excavate blasted materials. The loader provide flexibility for blending purposes. Pampacancha is expected to enter into production in 2019, and will require a loader at the start of the mining activities.

All phases have been designed to achieve high productivity, taking advantage of double-side loading and working at faces around ~60 metres in width.

16.6.4 HAULING

For life of mine, ore and waste will be transported by 240-tonne capacity haul trucks (CAT 793F). The use of this class of trucks minimizes road congestion, labor requirements, and operating costs. 240-tonne trucks require a minimum haulage road width of 32 metres. Haulage routes were designed to improve productivity and operating hours required per year.

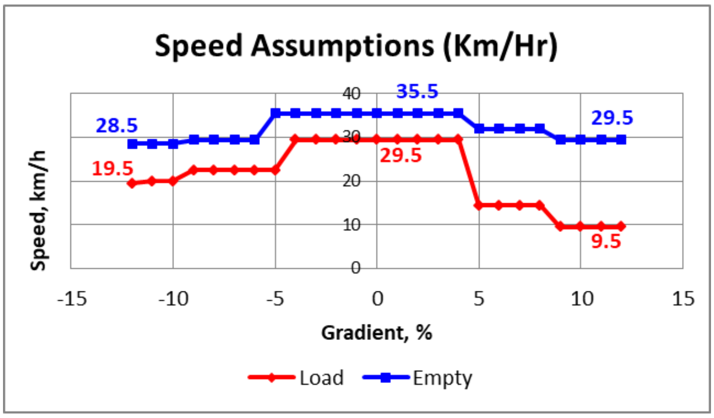
Table 16-17 and Figure 16-39 show parametres used in the truck & shovel study.

Table 16-17 Truck Fixed Time

Load (min)	
Shovel	5.6
Loader	6.2

Dump (min)	
Crusher	2.3
WRF	1.3
TMF-AB	1.9
TMF-AR	1.9

Figure 16-39 Truck Speed by Gradient



16.6.6 SUPPORT EQUIPMENT

Major support equipment includes mine equipment that is not directly responsible for production, but which is used on a regular basis to maintain in pit and ex pit haul roads, pit benches, WRF and TMF and to perform miscellaneous construction work as needed. Mine support fleet equipment includes:

- Crawler dozers, D10T2 class;
- Rubber-tired dozers, 824K class;
- Motor graders, 14H class; and
- Water trucks, 777G class.

In general, the rubber-tired 824K-class dozers is used in the pit to clean up around the primary loading units, while the track dozers are used for haul road construction, pit development, WRF and TMF management, and final re-grading requirements. The graders and water trucks are used respectively to maintain roads and control dust.

16.7 MINE ENGINEERING

16.7.1 GEOTECHNICAL PARAMETRES AND MINE PLANNING

The same geotechnical parametres were used as already described in section 16.2.

Pit dewatering is required to achieve the planned pit slope angles. In some areas, the requirement of depressurizing the walls has been paramount in order to maintain slope stability for the slopes excavated in the Heavily Faulted Intrusive Rock - Upper (south sector) and Sedimentary Rock - Upper (SE sector) domains.

In early 2016, Anddes Asociados developed an Updated Study of Pit Slope Design for Phases 1, 2 and 3. This study was mainly carried out for engineering support during the current operations in Phases 1, 2 and 3. The study was based on previous studies, additional geotechnical investigations and the current alteration model.

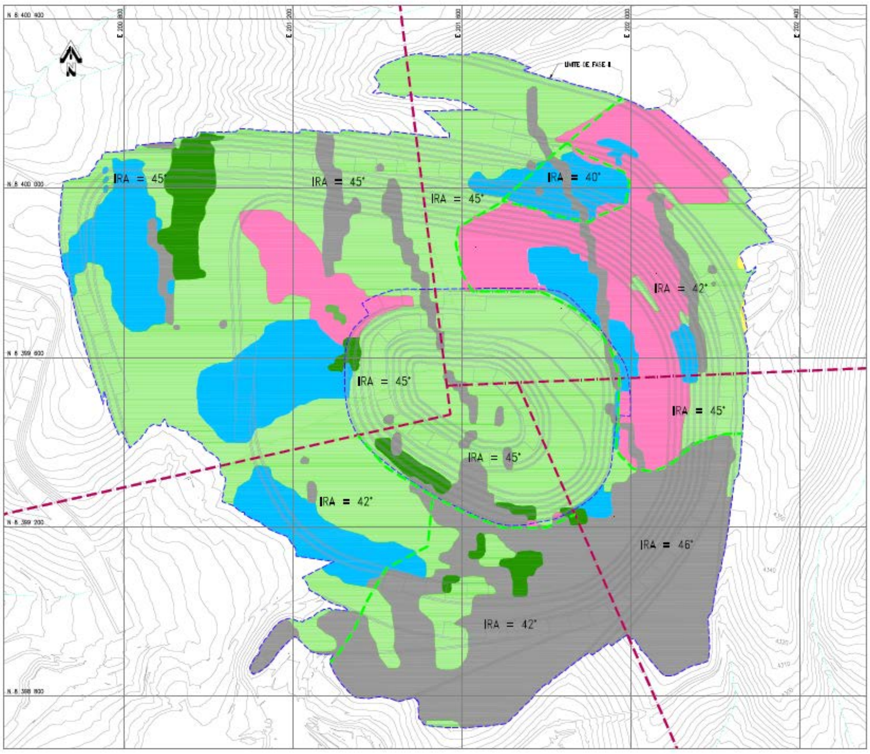
Stability analyses were conducted for phases and pit walls considering the potential for non-circular failure mechanisms that could involve multiple benches.

Table 16-18 and Figure 16-40 show the recommendations for open pit slope design for Constancia Phases 1, 2 and 3.

Table 16-18 Constancia, Recommended Pit Slope Angles (Anddes, 2016)

Design Sector	Alteration Type	Bench Height (m)	Catch Bench Width (m)	Bench Face Angle (°)	Design Inter-Ramp Angle	Maximum Inter-Ramp Height
Southeast (Domain I)	Phyllic	15	8	65	45	105
	Unaltered	15	8	66	46	180
	Potassic	15	8	65	45	165
	Propylitic	15	8	65	45	180
Northeast (Domain II)	Phyllic	15	8	57	40	105
		15	8	59	41	75
		15	8	60	42	60
	Potassic	15	8	60	42	165
	Propylitic	15	8	65	45	180
Northwest (Domain III)	Phyllic	15	8	65	45	105
	Potassic	15	8	65	45	165
	Propylitic	15	8	65	45	180
	Skarn	15	8	65	45	180
Southwest (Domain IV)	Phyllic	15	8	60	42	105
	Unaltered	15	8	60	42	180
	Propylitic	15	8	60	42	180
	Skarn	15	8	60	42	180

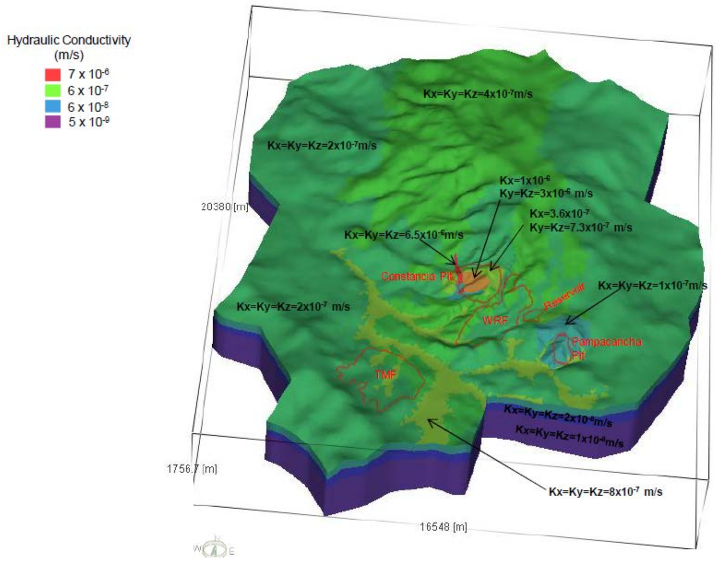
Figure 16-40 Constancia Pit Slope Sectors (Anddes, 2016)



16.7.2 HYDROGEOLOGICAL PARAMETRES

A regional map of the estimated hydraulic conductivities covering both the Constancia and Pampacancha pits is presented on Figure 16-44.

Figure 16-41 Predicted Hydraulic Conductivities in Constancia and Pampacancha Pits



Constancia Pit

The Constancia pit will operate from 2014 to 2035. At the end of mine life, the pit footprint at surface will be approximately 15% larger, and the ultimate pit floor will be approximately 30 m deeper than the previous pit design (3,750 m elevation versus 3780 m elevation).

Predicted total dewatering rates for the Pit Expansion Design and the area of influence should be close to 180 L/s as approved in the Groundwater Use Licence. The design was partially driven by the requirement for mine water supply. However, the main goal of dewatering is to depressurize the pit wall to assure the overall pit slope stability. Although the pit is slightly larger in the expanded design, the depressurization targets are expected to be less than the 100 m distance. Assuming the bedrock hydraulic conductivity at depths below 250 m is greater than 1 x 10-8 m/s, pit dewatering using a combination of vertical dewatering wells and/or horizontal drain holes should be feasible. Since limited hydraulic conductivity tests have been conducted below this depth, vertical wells and horizontal drain holes could be less effective if the hydraulic conductivities are inferior to10-8 m/s.

If horizontal drain holes are used to achieve depressurization in targeted sections of the pit, it is expected that the drain hole spacing will range between 10 and 15 m (assuming a bedrock hydraulic conductivity of 1 x 10-8 m/s to 1 x 10-7 m/s). In the upper weak zones where higher hydraulic conductivities were observed, the spacing could be somewhat larger (~30 m) and the installation of vertical dewatering wells could be more applicable. The final spacing and length of the drain holes would be established and refined during operation by phased installation of the drain hole system.

The number of vertical dewatering wells required to achieve depressurization would likely range between 15 and 50, based on the 2013 hydrogeological assessment, and the subsequently recognition of reduced depressurization requirements. The actual number of wells will depend on the amount of depressurization required for slope stability and the hydraulic conductivity of the bedrock below 250 m depth. It is assumed that horizontal drain holes will likely be required below depths of 250 m. If in-pit wells are used, depending on the mine plan, wells may need to be progressively moved and replaced as the pit is widened and deepened. As mining progresses, optimization of the dewatering system will likely be necessary based on hydrogeological data collected from dewatering wells and the network of piezometres.

In practice, a staged approach to dewatering is applied where geological, structural, and hydrogeological information that is gathered at each stage of mine development is used to refine and optimize the dewatering activities. The staged approach focuses dewatering efforts in areas of identified concern (i.e., poorly draining slopes with low factors of safety), and selecting the most appropriate method to mitigate high pressures, if present. From the hydrogeological perspective this staged approach requires detailed monitoring of the dewatering progress, dewatering well pumping rates, and pit inflows.

Dewatering progress is typically assessed by observations gathered from multi-level piezometres and monitoring wells that are installed at critical locations behind the pit walls. Monitoring of water levels in pumping wells is typically not sufficient for monitoring dewatering progress since well inefficiencies can cause the water level in the pumping well to be different from true groundwater elevations near these installations.

Pampacancha Pit

The hydrogeological model enabled the planning of the pit drainage strategy. As a result, the construction of five perimeter wells, which are to be implemented prior to the start of mining, is required. These wells will be adapted to the requirements and geometry of the pit design.

In addition, in the first year of mining (2019), the two existing wells, constructed during the Feasibility Study and located inside the pit will be used, which will contribute to the start of the draining process.

The expected flow rate will be increased as mining and pit exploitation progresses, until a maximum rate of approximately 30 litres per second (L/s) is achieved. Based on the modeling undertaken, it is estimated that each well will have approximate flow rates varying between 5 – 12 L/s, depending on the local geology of each well.

To monitor the advancement of pit dewatering activities, it is recommended that three additional piezometres be installed to complement the network of existing piezometres already within the pit area.

17 RECOVERY METHODS

17.1 INTRODUCTION

17.1.1 GENERAL LAYOUT AND FACILITY DESCRIPTION

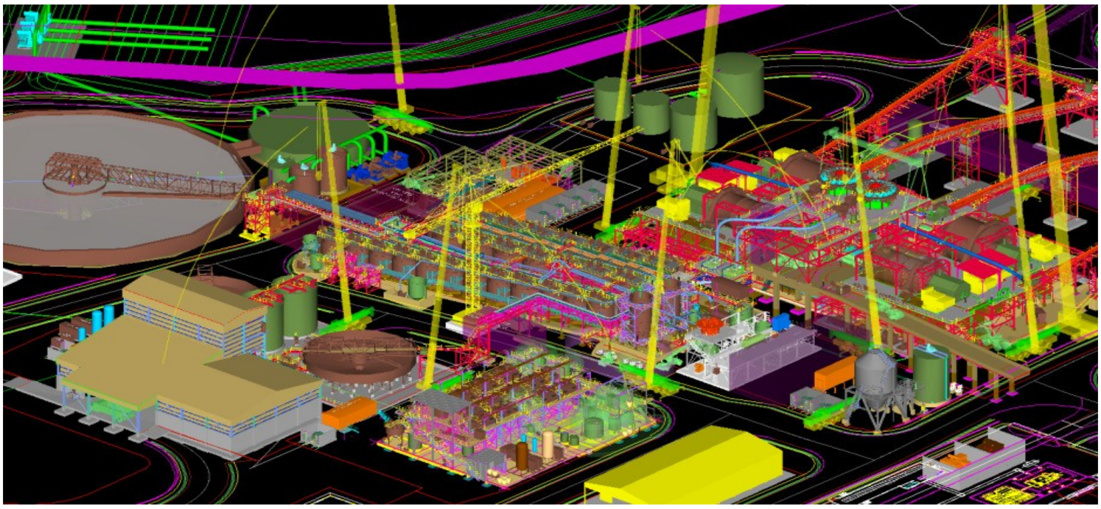
The process plant and facilities have been designed by Ausenco (Australia, Canada and Peru regional offices). The design was based on the GRDMinproc DFS design, with one phase of optimization of the comminution circuit, followed by front end engineering and design (FEED), and detailed engineering as part of the EPCM contract.

The mine is located on mountainous terrain. The plant site was selected as being the closest site available to the mine that is both economical and practical to develop, but significant earthworks were required to establish a multi-tiered pad. The crushing facility is located on the hillside in order to minimize expensive ROM pad construction. The layout allows the crushing facility to be free-draining, hence limiting the amount of ground work maintenance during the rainy season.

The primary crusher, belt conveyors, thickeners, tanks, flotation cells, mills and other equipments are located outdoors without buildings or enclosures. To facilitate the appropriate level of operation and maintenance, the molybdenum concentrate bagging plant, copper concentrate filters and concentrate storage are housed in clad steel buildings

A 3D view of the plant in 3D is shown on Figure 17-1.

Figure 17-1 3D View of the Concentrator



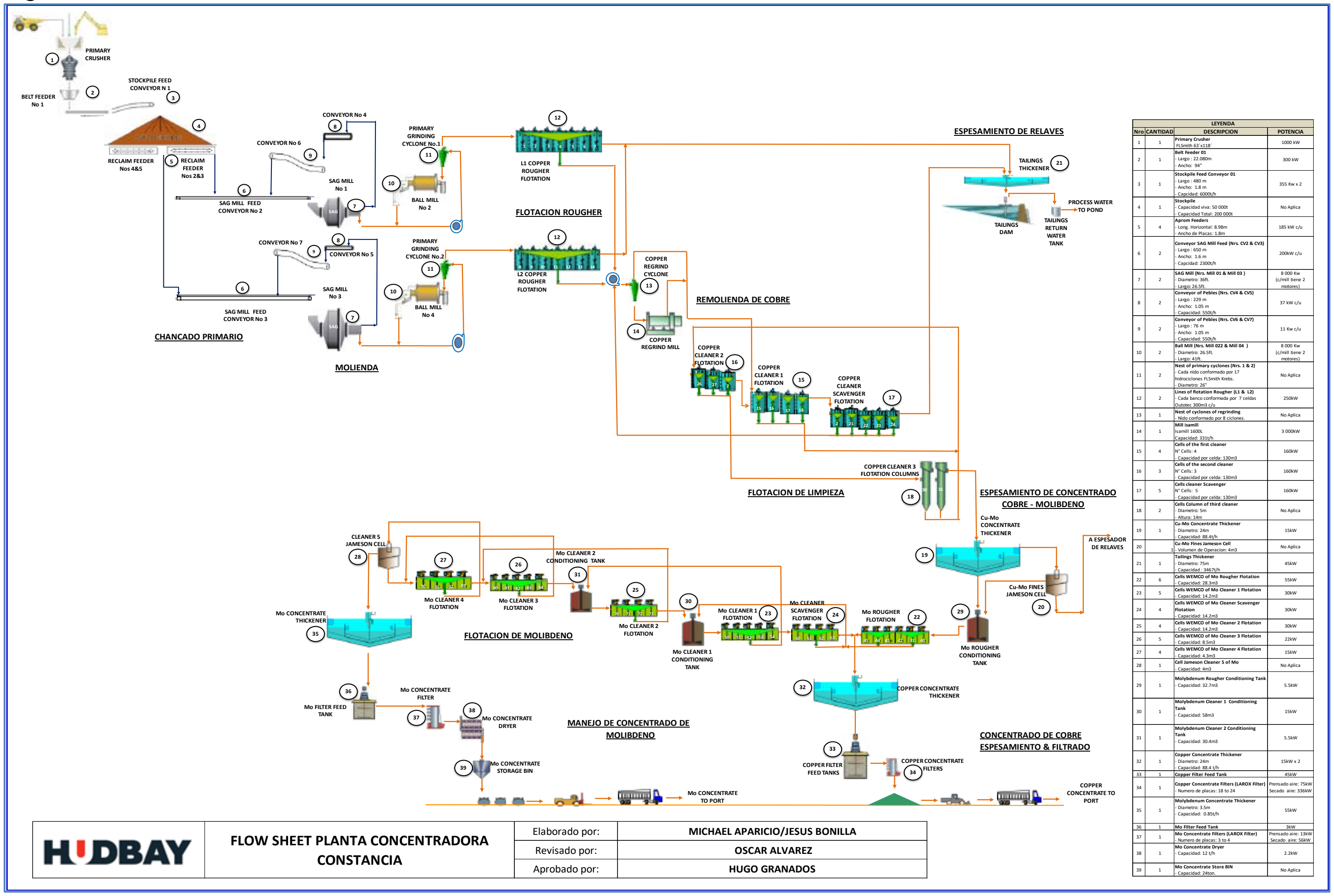
Ausenco designed the processing facilities according to the process design criteria presented in Table 17-1.

Table 17-1 Key Process Design Criteria

Plant Design capacity	Mt/a	30.8
Copper feed grade	Max, Cu %	0.79
Copper feed grade	Average, Cu %	0.39
Molybdenum feed grade	Max, Mo g/t	202
Molybdenum feed grade	Average, Mo g/t	105
Primary crushing availability	%	75
Grinding availability	%	94
Design Bond Ball Work Index	kW-h/t	15.9
Primary crusher size	Inches	63 x 118
	kW	600
SAG mill size	Feet	36 x 26.5
	kW	16,000 x 2
Ball mill size	Feet	26 x 41
	kW	16,000 x 2
Rougher flotation cells	m^3, nominal	300
	Number	14
Flotation cleaning column	Stages	2 Column Cells
Bulk concentrate thickener diameter	m	24
Copper concentrate thickener diameter	m	24
Molybdenum concentrate thickener diameter	m	4
Tailings thickener diameter	m	24

The process flowsheet for the Constancia recovery process, as represented in Figure 17-2, is similar to other major Copper-Molybdenum plants in Latin America.

Figure 17-2 Constancia Process Flowsheet



17.1.2 THROUGHPUT OPTIMIZATION

Since 2012, considerable efforts were made by Ausenco and Hudbay to re-evaluate the actual throughput capacity of the processing facilities. The plant has now demonstrated its capacity to process 90 ktpd of ore (31 Mtpa at 94% plant availability) from the Constancia and Pampacancha pit. Its processing capacity is higher than the original Ausenco design based on the following:

(i) While the process design criteria was 76,000 t/d at 91.3% availability, it has been demonstrated that 94% availability can be achieved with good maintenance and operating practices.

(ii) The SAG mill throughput was initially constrained by the hardness of the hypogene ore, however further reviews showed that most of this domain had very low rock quality (RQD). Therefore SAG milling requires less energy per tonne, thus increasing the overall throughput.

(iii) The grind size was increased for the hypogene ore, from P80 106 to 130 µm. There has been an impact on the copper recovery; however, this will be mitigated with the following projects:

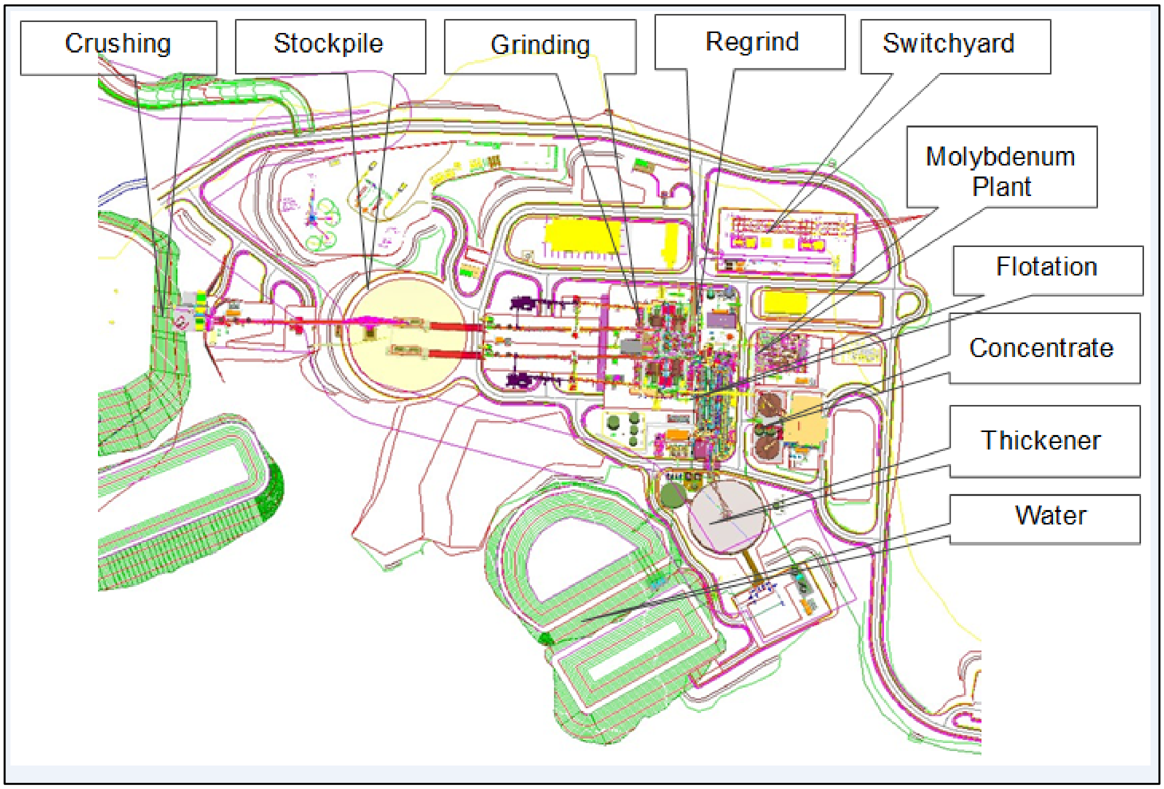
- Utilization of Proflote Technology (recovery of fine particles lower than 25 micron, to be implemented in 2018. It will allow a 1.5% increase of copper recovery.
- Application of Eriez cells technologies (Hidrofloat, recovery of coarse particles larger than 150 microns) will allow a 3.25% increase copper recovery.

(iv) There was no extra capital expense required to increase the throughput as it was already within the considered design factors.

17.2 PROCESSING PLANT - GENERAL

The general layout of the processing plant is shown in Figure 17-3.

Figure 17-3 Processing Plan Layout



The processing plant has been laid out in accordance with established industry best practices for traditional grinding and flotation plants. The major objective was to make the best possible use of the natural ground contours to minimize pumping requirements by using gravity flows and to reduce the height of steel structures.

To optimize the cost of the major footings for the SAG mill, the height of the SAG mill above grade was minimized by situating the cyclone feed sump as low as possible. The mill cyclones have been located so that the cyclone overflow can gravitate into the rougher conditioning tank.

At the tailing end of the flotation bank, the copper tailings thickener has also been situated to facilitate gravity flow and eliminate the requirement for another large set of pumps.

Due consideration has been given to the layout of the molybdenum plant to facilitate good housekeeping and occupational health and safety requirements. The potentially hazardous nature of the molybdenum bearing froth very often results in difficult housekeeping conditions within the work area. To this effect, the use of sodium hydrogen sulphide (NaHS) for copper depression and separation from molybdenum concentrate has the potential to generate hydrogen sulphide (H2S) gas.

The copper and molybdenum plants are independent of each other and separated by a reasonable distance. Furthermore, the molybdenum concentrator is located downwind of major high occupancy plant areas (prevailing wind on site is from the north). Similarly and given the fact that reagents can release hazardous gases, the storage facility is also located downwind of high occupancy plant areas, but close enough to the flotation area to avoid long runs of piping.

Due to these considerations the molybdenum plant is fenced out to strictly control personnel access. Similarly, a thorough training program has been put in place to educate anyone entering the plant about the hazards and safety procedures.

Rainwater runoff and general spillages from the processing plant are fully contained and directed to the process plant sediment pond. Again, the natural topography of the site is used to direct the flow to the pond, located at the western end of the site to avoid the need for pumps.

Wherever possible, the major substations and motor control centre (MCCs) have been located as close as possible to the electrical drives to minimize cable run lengths.

17.3 BUILDINGS

The following buildings were constructed as part of the facilities:

1. Office building – originally the EPCM offices on site. These were converted for general administrative offices.
2. Warehouse and workshop – the building shell was constructed as part of the original construction with services running to the building; however, completion was deferred to a later date. Process plant maintenance is working out of the construction shops originally set up for the SMP contractor. Warehousing is managed within a fenced platform near the process plant using construction structures put in place for commissioning.
3. A plant office building with a control room, server room, a DCS engineering station and a shift boss office.

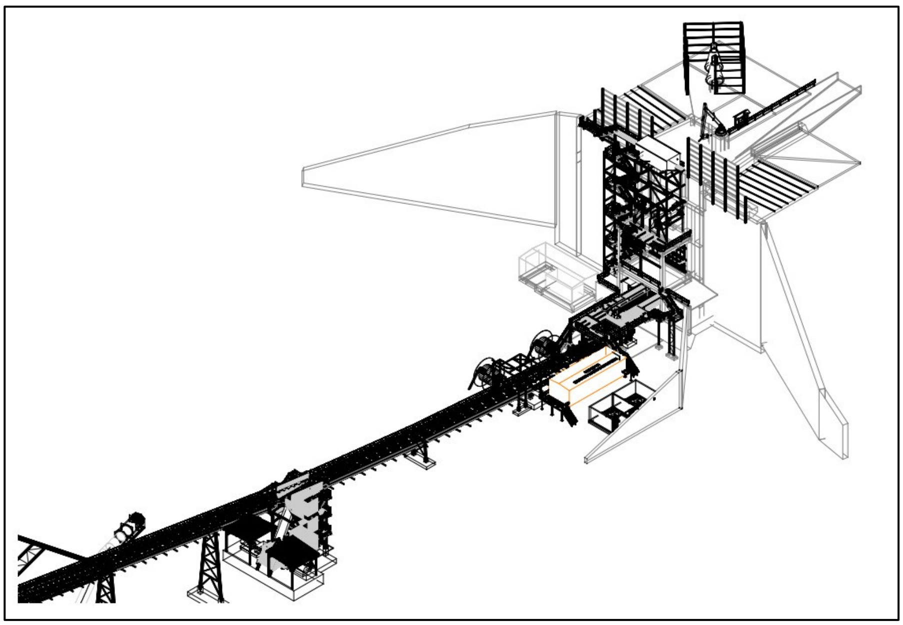
4. A combined chemical and metallurgical laboratory building.
5. A crusher control room.
6. A combined copper concentrate filtration and storage building.
7. A molybdenum concentrate packing plant shed.

17.4 CRUSHING

17.4.1 CRUSHING

A 63” x 118” gyratory crusher receives ROM of up to 1 m in size and reduces it to less than 125 mm (Figure 17-4). A variable speed belt feeder delivers ore from the crusher chamber to the coarse ore stockpile feed conveyor. The site mobile crane is used for major crusher maintenance. A drive-in dump pocket makes it possible to clear the crusher with a small FEL or bobcat.

Figure 17-4 Primary Crusher Isometric View

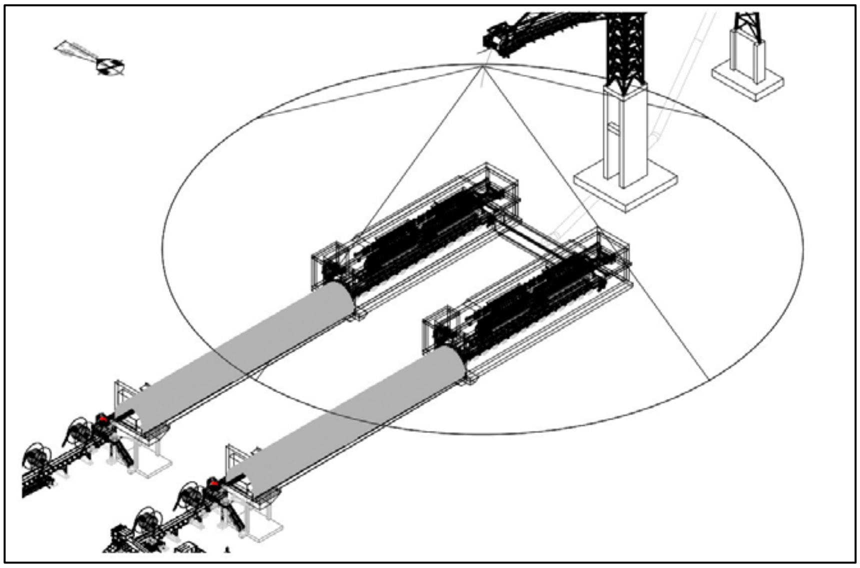


The crusher is an FLSmidth 63” x 118” gyratory crusher with a 1000 kW motor. This is likely to be under-utilized for most of the highly fractured ore, but will ensure throughput when processing an area of competent rock. The ROM bin and crusher surge pocket has the volume to hold 1.6 truck loads

17.4.2 CRUSHED ORE STOCKPILE AND RECLAIM

The grinding circuit requires two SAG mill feed conveyors (FIGURE 17-5). Dual reclaim chambers (near- parallel) are used to house a total of four apron feeders that draw ore from the crushed ore stockpile to the SAG mill feed conveyors. A secondary egress or emergency tunnel is provided through the connection of the two reclaim chambers.

Figure 17-5 Coarse Ore Stockpile Isometric View



17.5 GRINDING

The grinding circuit consists of two trains of grinding mills, each train identical, treating 1875 t/h (FIGURE 17-6). Each train consists of a single FLSmidth semi-autogenous grinding (SAG) mill, a ball mill and a potential addition of a pebble crusher (SABC). The ball mills are operated in closed circuit with a FLSmidth (Krebbs) cyclone cluster to produce a product from the grinding circuit of 80% passing until 130 µm.

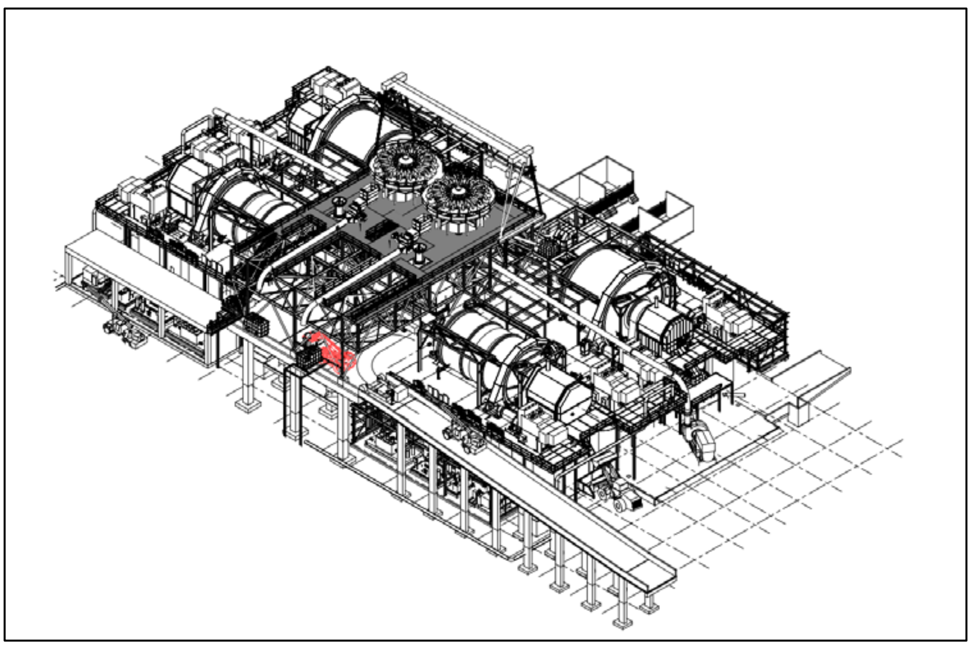
The SAG mills are fitted with a dual pinion drive system and are driven by a variable-speed drive comprised of a SER hyper-synchronous drive. The SAG mills are 11 m (36’) dia, 7.3 m EGL (26.5’). Each SAG mill is driven by two 8 MW motors. The 16 MW, twin pinion, SAG mills were selected over the equivalent GMD to reduce both capital and commissioning costs.

The 16 MW ball mills selected are fitted with twin pinion drives operated at fixed speed. The mills are 7.9 m (26.5’) dia, 12.4 m EGL (41’). Like the SAG mills, each ball mill is driven by two, 8 MW motors. Each of the eight mill motors are interchangeable, and started using conventional liquid resistant starter circuits (although once started, the SAG mill SER system takes over control enabling limited variable speed control). The commonality of motors reduces the number of maintenance spares required to be held on site.

The installation of the pebble crushers has been deferred and their future need is still being determined. Each of the two pebble crushers will be fitted with a 600 kW drive. Hydrocone crushers were selected for their capability of adjusting the closed side setting depending on power demand. This increases the flexibility of the machines and reduces the need for pebble storage prior to the crusher.

Each grinding train has a single (duty only), 1.5 MW Warman 650 MCR M200 variable speed cyclone feed pump. A full spare assembly is held on site and changed out when required. The layout allows for the flexibility to recycle cyclone underflow to the SAG mills; a single “drive-on” grinding floor; a simple cyclone tower; clear access to the wet end pumps; and clear access to scats clean-up from the ball mill.

Figure 17-6 Grinding Circuit Isometric View



The capital cost was reduced by not installing standby cyclone feed pumps. Spare units can be installed in approximately six hours. Easy access by mobile cranes is provided and it is possible to only change the wet end of the pump which minimized downtime.

17.6 COPPER-MOLYBDENUM FLOTATION

The function of the copper flotation area (Figure 17-7) is to recover copper and molybdenum into a copper/molybdenum concentrate, reject zinc to the cleaner scavenger tailings stream and reject coarse non-sulphide gangue to the rougher tailings. Each grinding circuit line is conditioned with slaked lime to ensure the circuit pH is maintained at its set value.

Cyclone overflow from each of the primary grinding hydrocyclones reports via gravity to the corresponding bulk rougher circuit. FLSmidth Krebs gMAX 26 cyclones are used. The bulk flotation consists of two copper rougher flotation banks each consisting of seven 300 m^3 mechanical flotation cells, operated at a pH of 10. These produce a bulk concentrate containing copper, silver, gold and molybdenum which reports to the regrind cyclone feed sump. The bulk tailings flow by gravity to the tailings thickener for eventual disposal at the TMF. Spillage and water are removed by two rougher flotation area sump pumps.

The bulk rougher concentrate is pumped to a cyclone cluster. Finer material, which does not require regrinding, would report to the cyclone overflow. The coarse material reports to the cyclone underflow and is pumped to the bulk cleaner regrind mill. A 3 MW Xstrata M10.000 IsaMill is used.

The regrind mill discharge is combined with the regrind mill cyclone cluster overflow, and is pumped to the first bulk cleaner flotation cell. In the bulk cleaner circuit the pH is increased to 11.0 with slaked lime.

The copper cleaner circuit consists of three stages of cleaning and one bank of cleaner scavenger flotation cells.

The first bulk cleaning circuit consists of four OK-130 130 m^3 flotation cells. Concentrate containing copper and molybdenum is recovered and reports to the 2nd stage cleaner feed sump and the 2nd stage cleaner flotation cells for further upgrading. The tails report to the five cleaner scavenger flotation cells. An additional launder was installed during operation to allow the concentrate from the first cell to be re-directed to the final concentrate when the quality is sufficient. This allows for removing some of the load from the cleaner circuit during high head grade conditions.

The 2nd stage copper cleaner flotation stage consists of three OK-130 130 m^3 mechanical flotation cells. The concentrate reports to the 3rd copper cleaner flotation stage for further upgrading. The tails from the 2nd copper cleaner report back to the copper cleaner conditioning tank.

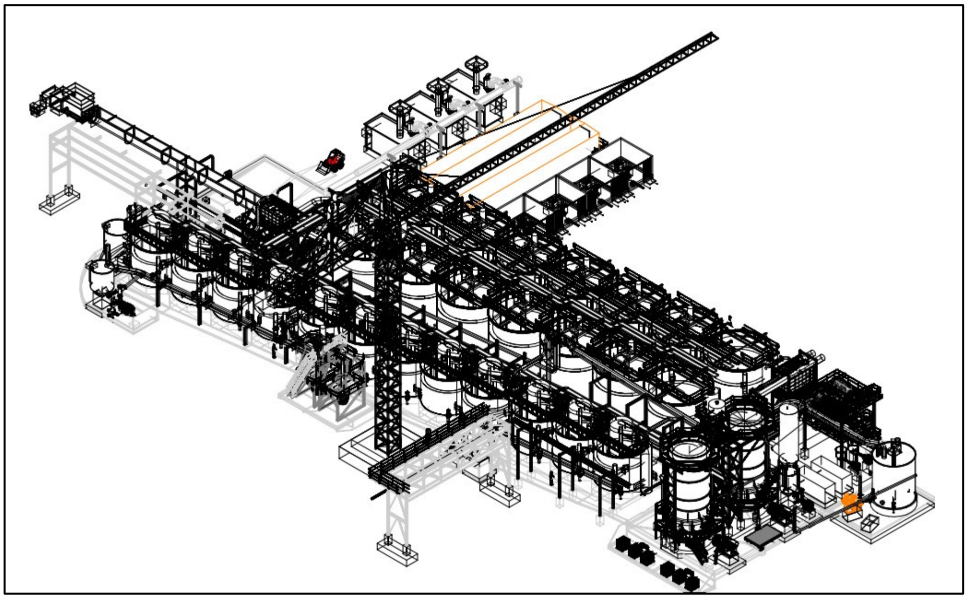
The 3rd copper cleaner flotation stage consists of two ERIEZ (CPT) flotation columns, 4.88 m diameter x 12 m high. The columns produce a bulk copper/molybdenum concentrate which reports to the molybdenum feed thickener in the molybdenum flotation circuit for further treatment. The concentrate can be bypassed directly to the final concentrate thickener when the molybdenum plant is not in operation. The tails stream reports to the head of the 2nd stage copper cleaner flotation stage via the feed sump. The flotation columns are supplied with compressed air from dedicated copper flotation column compressors.

The copper cleaner scavenger stage consists of six OK-130 130 m^3 mechanical flotation cells. Low grade concentrate is recovered and reports back to the copper cleaner conditioning tank. The tails are fed to the tailings thickener.

The flotation cells, other than the flotation columns, are supplied with low pressure air from dedicated low pressure air blowers.

The copper/molybdenum concentrate from the copper 3rd stage cleaner flotation column reports to a 24.0 m diameter copper/molybdenum feed thickener. The main function of the thickener is to reduce carryover of reagents from copper flotation into the molybdenum flotation circuit. Thickener overflow is returned to the copper circuit.

Figure 17-7 Cu-Mo Flotation Circuit Isometric View



17.6.1 LAYOUT

The cleaning circuit, consisting of 1st, 2nd and scavenger flotation stages is arranged in a counter current arrangement, whereby the largest flow, the tails, flows by gravity from one stage to the next. All cells are the same size. The concentrate launders and pumps are all arranged on the outside of the plant to allow for easy maintenance access. This arrangement also allows for the number of cells in each stage to be modified, with some minor pipework changes, during operations.

17.6.2 BULK FLOTATION AREA SUMPS

To reduce capital costs and improve the overall plant, the volume held in each sump has been minimized.

Under normal operating conditions, the slurry that is collected under the flotation areas is composed of concentrate overflowing from the top of the cells. This is collected and pumped from the sump pump area back into the process.

During routine shutdowns or power failures, the mechanical cells can stop and do not need to be drained. For maintenance, it will be occasionally necessary to dump the contents of the flotation cells.

Since the value of the slurry in each rougher cell is low, the sump is not designed to contain an entire cell. In a typical maintenance shutdown the slurry dumped from the rougher cells is expected to be pumped to tails by the rougher area sump pump. If this pump fails or if there is no power, then the rougher slurry will overflow the sump and runs along a channel to the plant run off pond. This would be an exceptional case, such as in an emergency.

In the case of the 3rd cleaner columns, the sump has enough volume to hold one of the two columns. When the plant is down for more than three to four hours then the columns will need to be drained. If the sump pump is down then the sump will over flow into the 2nd cleaner sump.

17.7 MOLYBDENUM SEPARATION

The function of the molybdenum flotation area (Figure 17-8) is to recover molybdenum into a molybdenum concentrate and reject copper to the tailings as a copper concentrate. The stages of molybdenum flotation are: rougher, 1st to 5th stage cleaners and cleaner scavengers.

Underflow from the copper-molybdenum concentrate thickener is pumped to a 38 m^3 agitated molybdenum rougher conditioning tank. NaHS is added to the conditioning tank to inhibit the flotation of copper, with promoter (diesel emulsion) also added to enhance the flotation of molybdenum. From there the overflow flows to the ten 10 m^3 molybdenum rougher flotation cells. The tailings from the rougher cells form a portion of the copper concentrate and are pumped, with the molybdenum cleaner scavenger tailings, to the copper concentrate thickener. The molybdenum rougher concentrate reports to the molybdenum 1st cleaner flotation cells.

All of the molybdenum cells, other than the final cleaner, will be WEMCO type self aspirating cells from FLSmidth.

The molybdenum cleaner circuit consists of five stages of cleaning and one bank of cleaner scavenger flotation cells. The molybdenum 1st stage cleaner consists of six 28.3 m^3 mechanical flotation cells. From these, a concentrate containing molybdenum values is recovered and flows to the molybdenum 2nd stage cleaner flotation cells for further upgrading. The tailings from the molybdenum 1st stage cleaner flow to the four 14.2 m^3 molybdenum cleaner scavenger flotation cells.

The molybdenum 2nd stage cleaner flotation consists of four 14.2 m^3 flotation cells. From the flotation cells the concentrate reports to the molybdenum 3rd stage cleaner flotation cells for further upgrading.

The tailings from the molybdenum 2nd stage cleaner flotation cells reports to the molybdenum 1st stage cleaner flotation cells.

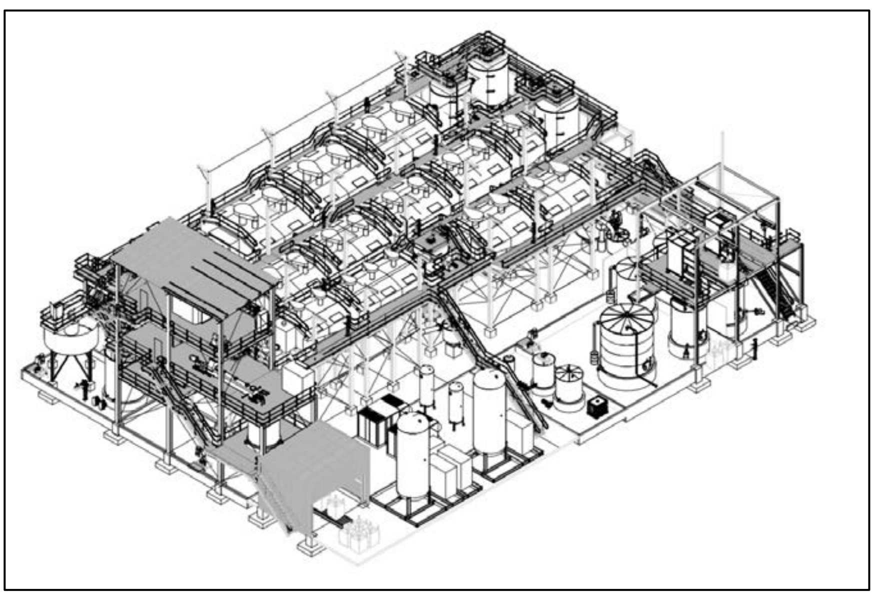
The molybdenum 3rd stage cleaner flotation consists of five 8.5 m^3 flotation cells. From the cells, the concentrate reports to the molybdenum 4th stage cleaner flotation cells for further upgrading. The molybdenum 3rd stage cleaner tailings report to the molybdenum 2nd stage cleaner flotation cells.

The molybdenum 4th stage cleaner flotation consists of four 4.3 m^3 flotation cells. The cells produce molybdenum concentrate which reports to the molybdenum 5th stage cleaner flotation cells. The tailings from the molybdenum 4th stage cleaner cells reports to the molybdenum 3rd stage cleaner flotation cells.

The molybdenum 5th stage cleaner flotation consists of a single 4 m^3 Xstrata Jameson flotation cell. The cell produces the final molybdenum concentrate. The tailings from the molybdenum 5th stage cleaner cell reports to the molybdenum 4th stage cleaner flotation cell.

The molybdenum cleaner scavenger consists of four 14.2 m^3 mechanical flotation cells. A low grade concentrate is recovered and is pumped back to the copper/molybdenum feed conditioner tank. The tailings from the molybdenum cleaner scavenger are pumped to the copper concentrate thickener. Spillage and water are removed by a sump pump in the flotation building.

Figure 17-8 Cu-Mo Separation Circuit Isometric View



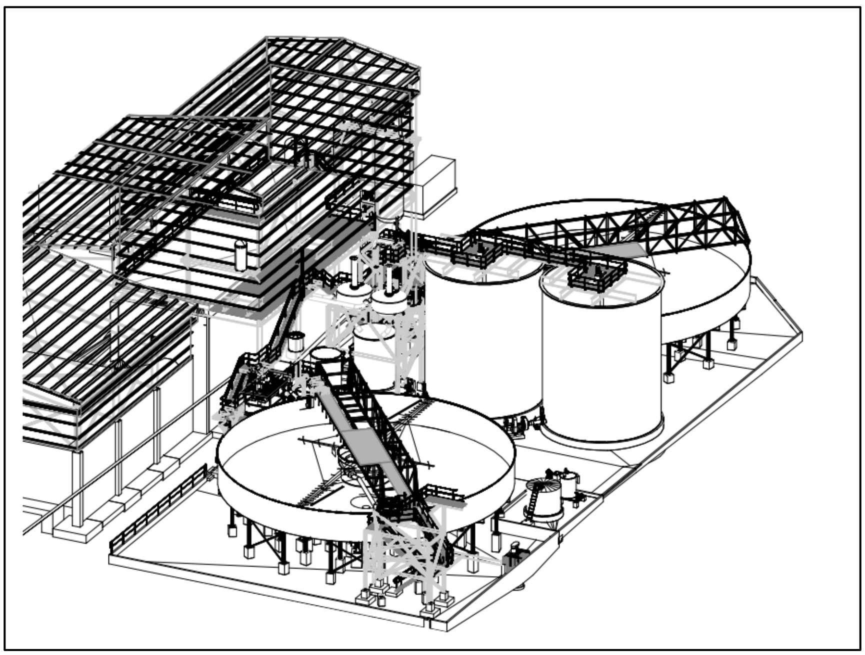
17.8 COPPER CONCENTRATE THICKENING AND FILTRATION

The tailings from the molybdenum flotation roughers and molybdenum cleaner scavengers report to a 24m diameter copper concentrate thickener (

Figure 17-9), via a static screen to remove tramp material. Flocculent is added to enhance settling. The clarified thickener overflow reports to the process water pond, to be pumped to the process water tank. The thickener underflow is removed at 60% solids via duty/standby peristaltic pumps.

The thickened and de-tramped slurry is stored in two 860 m^3 agitated tanks. These provide a 24 hour surge capacity, allowing filter maintenance to be conducted without affecting mill throughput. The filter feed is pumped to one of two LAROX horizontal plate pressure filters with a combined maximum filtration rate of 91.2 t/h. The filters operate with a total cycle time of 9.8 min and are designed to produce a filter cake of less than 8.5% moisture. The filter cake is dropped to the floor and then transferred to the concentrate storage pile by a front end loader. Filtrate, cloth wash, and flushing water are returned to the copper concentrate thickener.

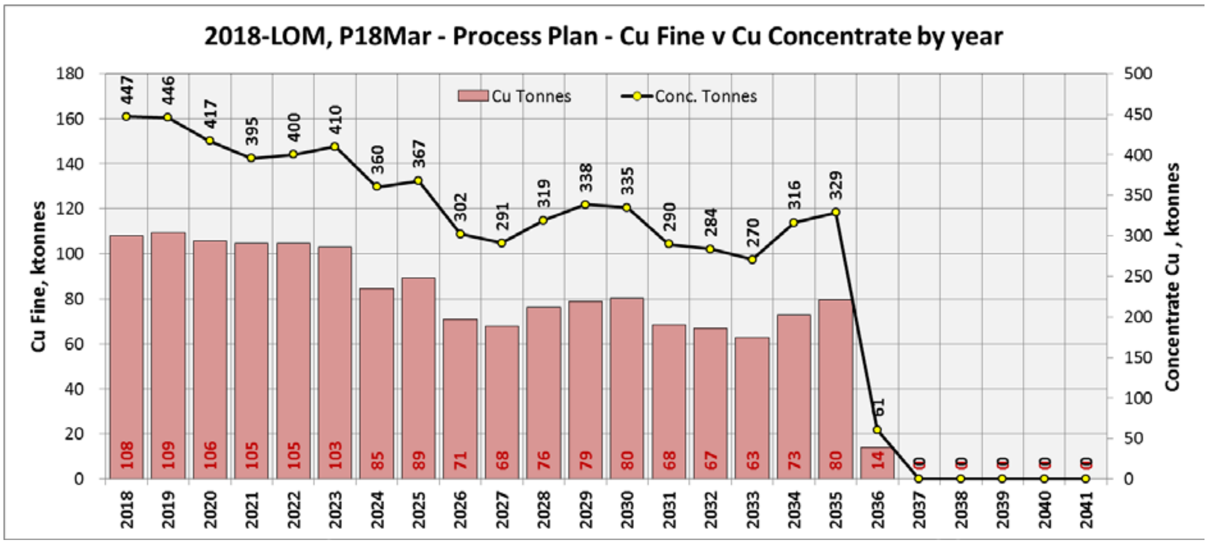
Figure 17-9 Cu Concentrate Thickening and Filtration Area Isometric View



17.9 CONCENTRATE STORAGE AND LOADOUT

Annual concentrate production rates showed at Figure 17-10.

Figure 17-10 LOM Copper Concentrate Production



After filtering, the copper concentrate is transferred to a copper concentrate product storage stockpiles by front end loader inside the copper concentrate and load-out building. The building contains a live stockpile with capacity for seven days production (12,000 t).

A fleet of approximately 140 tractor trailers, each with 36 t hauling capacity, are used to transport the concentrate to the port. A single front end loader is used to collect concentrate as it falls from the filters and either load it directly onto the trucks or temporarily store it in the same building. The trailers are fully enclosed units with pneumatically operated lids. Locks and security seals are applied at site. A truck scale weighs each truck before leaving site and upon arrival in Matarani. One primary contractor is used for the base load with surge capacity being made up with several smaller contractors.

The trucks have a two day round trip from site to the port and back, depending upon road conditions. Rigorous control procedures are applied during transport including separate escorts, GPS tracking and monitoring, controlled resting periods for the drivers, and secure stopping points en route. The position and velocity of every convoy is monitored continuously from the security control center in Constancia.

17.10 MOLYBDENUM CONCENTRATE THICKENING AND FILTRATION

The molybdenum concentrate gravitates to a 4 m diameter thickener where it is thickened to 60% solids. The thickener overflow reports to the molybdenum process water tank, where it is used as process water in the molybdenum flotation circuit. The thickened concentrate is pumped to an agitated 48 m^3 molybdenum surge tank, which has a residence time of 24 hours and acts as surge capacity for filtration.

The slurry is then pumped to a pressure filter to produce a filter cake of approximately 15% moisture, the next stage is the drying stage where we can obtain concentrate of 5% moisture. Filter cake is conveyed by a bobcat to the molybdenum bulk bagging plant. Molybdenum concentrate is bagged into 1 m^3 bulk bags. The bulk bags are loaded onto trucks using forklifts.

The filtrate from the filter reports to the tailings thickener. The thickener overflow flows by gravity to the process water tank.

17.11 TAILINGS DISPOSAL

The copper tailings flow by gravity from the two copper rougher tailings lines, and by pumps from the cleaner scavenger tailings line, to a 75 m diameter tailings thickener, where it is thickened to 58% solids. Thickened underflow is then pumped to the tailings dam. The thickener overflow gravitates to the process water tank where it is used in the grinding and copper roughing circuits.

The tailings pumping system consists of five centrifugal pumps in series with an HDPE lined steel pipe that crosses the Chilloroya valley. The pumping station at the plant site is designed to take tailings thickener underflow and dilution water and pump the mixture to the tailings dam on the other side of the valley. The pipeline follows the haul road route to save costs, increase maintenance access and reduce land disturbance. The road is built at low slope angles, typically less than 10%. The tailings pipeline has a maximum inclination of 12%. This is less than the angle of repose of the settled solids in the slurry, which is 15°. Consequently, if there is an un-programmed shutdown, for example a power outage, the slurry can sit in the line without having to be discharged or flushed. To reduce the risk of plugs during startup, all of the pumps in the pump station are fitted with variable speed drives.

The tailings pumping train can operate with as few as three pumps in operation allowing for extended maintenance times and coordinating with mill liner changes to minimize downtime. Under a normal scheduled stoppage, process water will be used to flush the solids from the line into the tailings dam.

17.12 REAGENTS

The reagents area is located to the west of the grinding and flotation areas.

Coarse lime and flocculent are delivered as solids to the reagents area by road trucks and stored in the reagents building.

A packaged flocculent mixing plant prepares flocculent for storage in an agitated tank for distribution. Other powdered reagents are mixed in agitated tanks before storage in tanks for distribution.

Liquid flotation collector and frother are received in one cubic meter iso-containers that are stored in the reagent store and taken by forklift to the reagent dosing area as required. The use of iso-containers also allows for placement in the grinding or flotation areas for flexibility in reagent dosing and plant scale testing.

Reagents are distributed by duty/standby pumps or multiple peristaltic metering pumps along flexible small diameter tubing (or by gravity in the case of temporary dosage by placing iso-containers within the grinding and flotation areas). Spillage and water are removed by the reagents area sump pump to tailings.

A diesel/fuel oil emulsion preparation facilities allows addition of small doses of this reagent for molybdenum promotion while assuring even dosing in the slurry.

Lime silo and attrition mills prepare milk of lime for pH adjustment. Lime is distributed by a lime ring to the dosing points. Spillage and water are removed by a lime slaking area sump pump.

Nitrogen for blanketing the molybdenum separation cells is supplied by a pressure swing adsorption plant on site.

Carbon dioxide that will be used for pH regulation in the molybdenum plant is supplied in tanks.

NaHS is delivered as a concentrated liquid by tanker. It is diluted with water onsite for dosing. The decision to not have solid NaHS preparation facilities onsite was made to reduce CAPEX, reduce safety risks and reduce the working capital and management issues of storing solid NaHS.

Hydrogen sulphide gas that comes from the NaHS is extremely toxic. Sensors and suitable engineering controls have been designed in order to use NaHS safely in the plant.

17.13 OPERATING PARAMETRES

The current level of reagent, power, water and grinding media consumption rates are shown in Table 17-2 Operating Parametres

Table 17-2 Operating Parametres

Item	Section	Item	Value	Unit
Reagent Addition	Grinding	Quicklime	1000	g/t fresh feed.
	Cu Flotation Cu	Primary Collector (F500X) Secondary Collector (D101) Frother (RE-100) Depressant	17 12 30	g/t fresh feed. g/t fresh feed. g/t fresh feed.
	Flotation Mo	(NaHS) Promoter (Diesel Emulsion) Carbon Dioxide	6500 10 36	g/t circuit feed. g/t circuit feed. g/t circuit feed.
	ConcentrateThickeners	Flocculant (Superfloc A130)	10	g/t thickener feed
	Tailings Thickener	Flocculant (Ixfloc 5320)	18	g/t thickener feed
Power Consumption	Plant	Electrical Load	636,000	MW-h/y
Water	Raw Water	Annual Average Flow	0.88	Mm3
	Requirement	Planned Average Flow	98	m3/h
		Instantaneous Flow	107	m3/h
Grinding Media	Grinding	SAG mill media	315	g/t
		Ball mill media	500	g/t
		Regrind media	63	g/t

17.14 METALLURGICAL ACCOUNTING AND CONTROL

The plant normally runs in automatic mode whereby all major units are monitored and maintained by the DCS system logic.

The plant operations are monitored and controlled from a central control room that is located between the grinding, flotation and regrinding areas. This control room has CCTV views of different plant areas and DCS views of all the plant equipment, valves and instrumentation. DCS interfaces have also been placed in the crusher control room, molybdenum plant control room, and the flotation deck.

Plant operations are facilitated by two online stream analysers (Outotec Couriers with stream multiplexers). These units constantly measure the principal streams for elements of interest and transmit the results to the operations and metallurgical staff for plant control.

Metallurgical accounting is achieved by shift composite sample analysis in the site laboratory. The samples are taken from the cyclone overflow, rougher tailings, cleaner scavenger tailings and overall tailings.

Flows are accounted for by weightometres on the SAG mill feed conveyors and flow metres on the concentrate and tailings flows.

Final concentrate tonnage is controlled by weigh bridges that weigh trucks as they leave site and arrive at the port. A weightometer on the ship loading conveyor measures the concentrate weight loaded onto the ships.

17.15 PAMPACANCHA

The metallurgical performance of the plant when fed with ore from Pampacancha is based on pre-feasibility study level of engineering.

The comminution results indicated that the rock is typically less competent and softer than the Constancia hypogene ore, so the grinding circuit is expected to be able to sustain the same throughput as the Constancia ore.

Pampacancha contains a significant amount of magnetite, so the ore has a higher density than Constancia. As such, the primary crusher, feeders and stockpiles are not expected to be bottle necks as the volumetric flowrate will be lower.

The flotation tests show that similar retention times and mass pulls are expected to those designed for Constancia.

The tailings thickener is assumed to have similar or higher capacity for Pampacancha than for Constancia due to the high magnetite content. Additional test work is planned to confirm this assumption.

The concentrate filters have been designed for similar copper minerals and higher concentrate tonnages than Pampacancha, so they are not expected to be bottle necks.

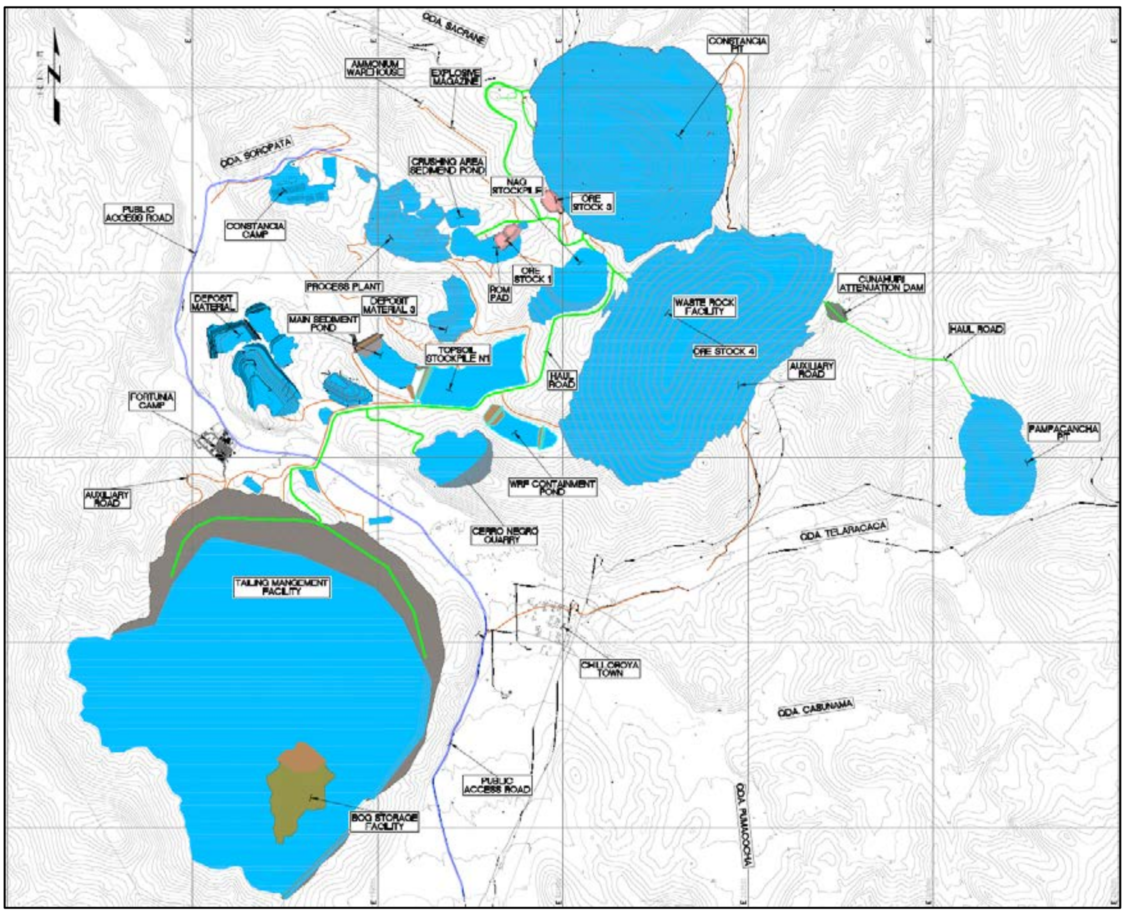
The tailings pipeline has been designed for solids with a lower specific gravity. A higher minimum velocity may be required for Pampacancha tailings. The current design allows for higher flowrates, but further testwork and engineering will be required to confirm that the pipeline will be suitable for Pampacancha tailings.

18 PROJECT INFRASTRUCTURE

The project infrastructure facilities include the Waste Rock Facility (WRF), Tailings Management Facility (TMF), as well as the water management facility, electric power supply and transmission, and improvements to the roads and the port. Hudbay is in charge of the planning and operation of these facilities (excepting the roads and port) and is working with different consultants for the engineering, construction and quality control / quality assurance (QA/QC) for their expansion.

Figure 18-1 provides an overall site layout showing the location of the various storage facilities relative to the Constancia and Pampacancha mines and the mill.

Figure 18-1 Overall Site Layout



18.1 TAILINGS, WASTE AND WATER STORAGE

Hudbay is managing the planning, construction and operation of the Tailings Management Facility (TMF) and the Waste Rock Facility (WRF) with design and engineering support from Knight Piésold and Co (KP) and Golder Associates (Golder).

18.1.1 TAILINGS MANAGEMENT FACILITY (TMF)

The TMF has been developed behind an east to west aligned, zoned landfill and rockfill cross-valley embankment dam that is expected to be raised in stages over the life of the mine and has sufficient capacity to accommodate the tailings generated over the current LOM plan. The tailings are expected to be impounded behind the south side of the embankment in a basin that contains two natural valleys running north to south that are separated by a central ridge. The TMF is located on the south side of the Chilloroya River and has a final design elevation of 4,186 masl. The TMF has two stages. The first designed by KP (engineering detail) at 4,160 masl and the second stage design by Golder (prefeasibility study) that rises from 4,160 to 4,186 masl without moving the previous footprint.

Tailings have been deposited from designated off-take points from a distribution pipeline located along the upstream crest of the embankment (spigotting system) and around the perimeter of the facility (discharge diffuser), and are delivered through drop-bar pipes running down the upstream face of the embankment and down the valley perimeter slopes into the TMF. The points of active deposition have been frequently rotated to form a thin layered, drained, and well-consolidated beach that will slope away from the embankment towards the south side of the TMF basin. Initially, the surface pond was located against the embankment in the east valley, but it has been displaced progressively upward and to the south by the development of the tailings beach, such that within the first three years of operations the beach is expected to become well-developed against the embankment. The surface pond will vary in size throughout the life of mine depending on the season, precipitation, and operational requirements.

During the construction stage and for the first two years of operations, the embankment was constructed by the downstream method to raise it to the elevation of 4,090 masl, and the Rockfill Platform forming the foundation for the upstream portion of the first centerline raise was completed in 2017 on the upstream side on the tailings. Conventional centerline construction started in 2017 at elevation 4,090 masl and will continue until the end of the operating life of the TMF. The centerline raises will follow a common construction method with the core, filter drain, and transition zones raised vertically above, at an elevation of 4,090 masl. These zones are confined between upstream and downstream structural rock fill zones. The TMF has already achieved at 4097 masl, which is sufficient for operations until 2019 with two metres of freeboard.

The design of the TMF includes a LLDPE geomembrane liner and underdrains. The liner is intended to cover the base of the entire eastern valley and a majority of the western valley to provide containment in areas where the surface water pond will be in contact with the base at any time over the life of mine. The liner will also be extended up the upstream face of the embankment to the top of the downstream constructed raise. Underdrain systems include groundwater underdrains beneath the liner, tailings underdrains, and an embankment toe drain.

Water collected by the underdrains is conveyed to sumps located immediately downstream of the embankment in both the east and west valleys. The groundwater underdrain and tailings underdrain outlet pipes in each valley were installed in reinforced concrete encasements under the embankment. At the sumps, monitoring and control systems allow for automated water quality and flow rate determinations to be made prior to pumping back to the surface water pond.

Basin roads will be constructed to support basin grading activities, liner installation and anchorage, and tailings pipeline construction and access. The roads will generally be developed within cut, but may have localized areas of fill to maintain continuity of the basin grading.

Instrumentation was installed in the TMF to monitor the structure throughout the life of the facility. Settlement and deformation monuments have been installed on the downstream embankment face and on the embankment crest during each major stage of construction. Additionally, vibrating wire piezometres have been installed within the embankment. During 2017 a vista data vision system (VDV) was installed to monitor the main instruments on-line and in real time.

According to the LOM plan of the TMF, and per the water balance, the construction periods will be 2019, 2020, 2023, 2025, 2027, 2029, 2031 and 2033. Optimization of the LOM to ensure sufficient quality rock from the pit for the TMF construction has been completed, such that the mining fleet is utilized for hauling, placement and compaction of structural fill over the downstream and upstream zone of the embankment.

Risks and potential mitigating measures were identified as follows:

- Stability of Modified Centerline Raises - Risk is mainly associated with liquefaction of the tailings deposit in the event of a significant earthquake which could affect slope stability of the structure.

Mitigation strategies for this are to alter the cross-section of the embankment. Sophisticated geotechnical analyses were completed to confirm the final geometry of the modified centerline raise and the widths of the various zones that are proposed.

- Availability and Suitability of Borrow Materials - Mitigation strategies include evaluation of alternate borrow sources.
- Delays to Construction Schedule and Sequencing - Careful construction management strategies were implemented and correctly timed.
- Poor Tailings Water Quality - A careful tailings management plan has been implemented so that the tailings are not left exposed for an extended period of time. Also, current planning for the facility is for it to operate as a "zero discharge" facility.

Figure 18-2 General View of TMF from East to West Side

Figure 18-3 General View of TMF from West to East Side

18.1.2 WASTE ROCK FACILITY (WRF), TOPSOIL AND PONDS

The Potentially Acid Generating and Non Acid Generating Waste Rock Facilities (PAG & NAG WRF) are located in the Cunahuiri Valley east of the Constancia Pit and provide storage for the PAG and NAG waste based on the updated mine plan developed in 2017. The facilities receive mine waste material from the operation of the Constancia pit and from the Pampacancha pit. Underdrain systems include groundwater underdrains to collect the water to WRF Retention

The Waste Rock Facility Retention Pond is located downstream from the WRF in the Cunahuiri Valley. The pond provides energy dissipation for surface water reporting to the downstream WRF Containment Pond via overland flows and the contact diversion channels and protects the WRF Containment Pond from possible falling waste rock and sedimentation. It has a design capacity of approximately 24,000 m^3, i.e. until 2021.

The Waste Rock Facility Containment Pond is located downstream of the WRF Retention Pond in Cunahuiri Valley. The pond provides storage capacity for surface runoff, direct precipitation and seepage collected from the WRF and its contact diversion channels. The pond also stores Acid Rock

Drainage (ARD) water, which is pumped to a lime neutralization facility at the process plant prior to its eventual use in the process plant. A foundation grout curtain will be incorporated into the embankment foundation to minimize seepage. The pond design capacity is approximately 600,000 m^3 and sufficient to operate until 2021, in order to accommodate the maximum operational volume and additional storage to accommodate the 100-yr/24-hr storm event. The capacity of these ponds may be reassessed during the last years of the operation due to the potential expansion of the WRF.

Non-Contact Channels have been sized to accommodate the 200-year/24-hour (200-yr/24-hr) storm event with an additional freeboard for a 500-yr/24-hr storm event with no allowance for a freeboard. There are three Non-Contact Channels on operations: NC1, NC2 and NC3.

Topsoil Stockpile No.1 is located in Cunahuiri Valley downstream of the WRF Containment Pond and has a design storage capacity of 4.7 milion m^3. This stockpile stores topsoil excavated during construction activities. Topsoil is used throughout operations for progressive closure of various facilities.

The Non-PAG Waste Rock Facility has a design storage capacity of 27 Mt (14 Mm^3 at 1.9 t/m^3). Primarily, Non-PAG waste rock is to be used for the construction of various construction roads, access roads, and haul roads as well as the Tailings Management Facility (TMF) embankment. Excess Non-PAG waste may be placed within a temporary stockpile located 1 km southwest of the Constancia Pit.

The pumping system between the WRF Containment Pond and the treatment plant is designed to pump ARD water from within the pond to the lime neutralization facility. This pumping rate is such that the WRF Containment Pond water level will be maintained at or below the maximum operational water level. The operation pumping rate may be impacted from the results or findings of the continuous hydrogeological study review.

18.1.3 WATER MANAGEMENT

Hydrologic and hydraulic analyses, including a detailed process water balance, were completed to determine the frequencies and quantities of flows as well as the types of flow, location, and resulting sizes of the water management structures required to contain or convey them. From the site assessment of the water management within the mine site, the water was divided into categories of process water (water that is associated with the process will be used in the process operation and/or will be transferred throughout the mine site via pipelines) and non-process water (water that is not associated with process and typically consists of storm water runoff from areas within the mine site). The non-process water was further divided into contact water (potentially acid generating (PAG) runoff water and non-PAG runoff water) and non-contact water.

Structures directly associated with the site water management include the following:

- Diversion Channels – These structures divert both contact and non-contact water around the mine site to the sediment or water management ponds, or to natural drainages down-gradient of the planned mine facilities (Constancia pit, WRF, etc.).
- Sediment Ponds – These structures typically accept diverted contact water and control the release of sediments down-gradient of the mine site by settling the sediments out of the discharged water. The sediment ponds for the Constancia site are identified as Main Sediment Pond, Road Sediment Pond No.1, Road Sediment Pond No.2, TMF Sediment Pond No.1, TMF Sediment Pond No. 2, Plant Construction Sediment Ponds (which work as sediment ponds just during construction phase), and Crusher Sediment Pond (construction phase as well).
- Water Management Ponds – These structures typically deal with temporary water storage, and may include both process and non-process water structures. The water management ponds for the Constancia mine are identified as the TMF Ponds (east and west), Plant Process Pond, Plant Contact Pond, Crusher Pond, Main Sediment Pond, and WRF Containment Pond.
- Water Balance – Hudbay Peru performed a site-wide process water balance for the final design phase of the Constancia mine based on the current Mine Plan with the support of Piteau on the basis of the original KP design. Based on the analysis of the water balance model, enough water exists in the system for average climatological conditions to operate at the proposed mining and milling rates and no additional water sources are required for operation.

The Main Sediment Pond is a source of makeup for operations, especially for dry conditions. As required, water accumulating in the sediment pond can be sent to the TMF for use in the process. This water is stored in the TMF for use during the dry season, as needed. During the dry season, the flow rates are minimal as little water accumulates in the main sediment pond.

18.2 ON SITE CAMPS

Constancia is a remote site with restricted road access from the nearest significant town 1 ½ hours away. Currently there are two camps; Constancia; a permanent camp and Fortunia that was a pioneer camp and served as an overflow camp during the construction, Currently Fortunia is for practical purposes mainly being used for construction and specific contractors such as Heavy Civil Works, Security and Community-Relations. The camp facilities have been designed with reference to the IFC standards for camp construction for mining activities.

18.2.1 CONSTANCIA CAMP

This camp was the previous construction camp. It´s located at the north side of the mine. This permanent camp was developed to provide accommodations for operating personnel and necessary support personnel. The camp is divided into three pads (Figure 18-4). The first has the dining room, laundry, parking zone, warehouse and administrative offices. The second and the third pads have the housing modules with capacity for 2,930 people. The camp has a potable water treatment plant with 365 m^3 per day capacity, a residual water treatment plant with 600 m^3 per day capacity, a power house with three (standby) gen-sets of 500 kW each for emergency power and a gas storage capacity of 20,000 gallons. The main medical attention center is across this camp.

18.2.2 FORTUNIA CAMP

The Fortunia camp is used during the Heavy Civil Works activities, and has a dining room, laundry, parking zone, warehouse and offices (Figure 18-5). It is located at the ex-hacienda Fortunia. The main house is used as offices and dormitory. Since its expansion, the dormitory has a housing capacity of 627 persons. It has a power house with two (standby) gen-sets of 500 kW each for emergency, a potable water treatment plant of 120m^3 per day capacity, a waste water plant with 170 m^3 per day capacity and a gas storage tank with a capacity of 4,000 gallons. This camp is also used as a contingency for major plant shutdowns.

Figure 18-4 Constancia Camp General Layout

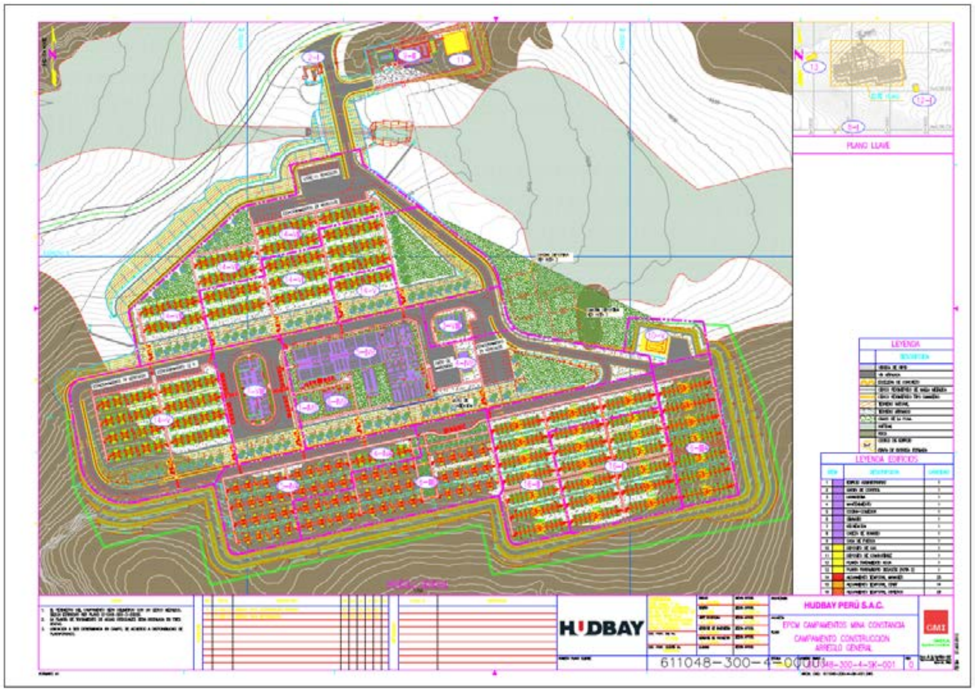
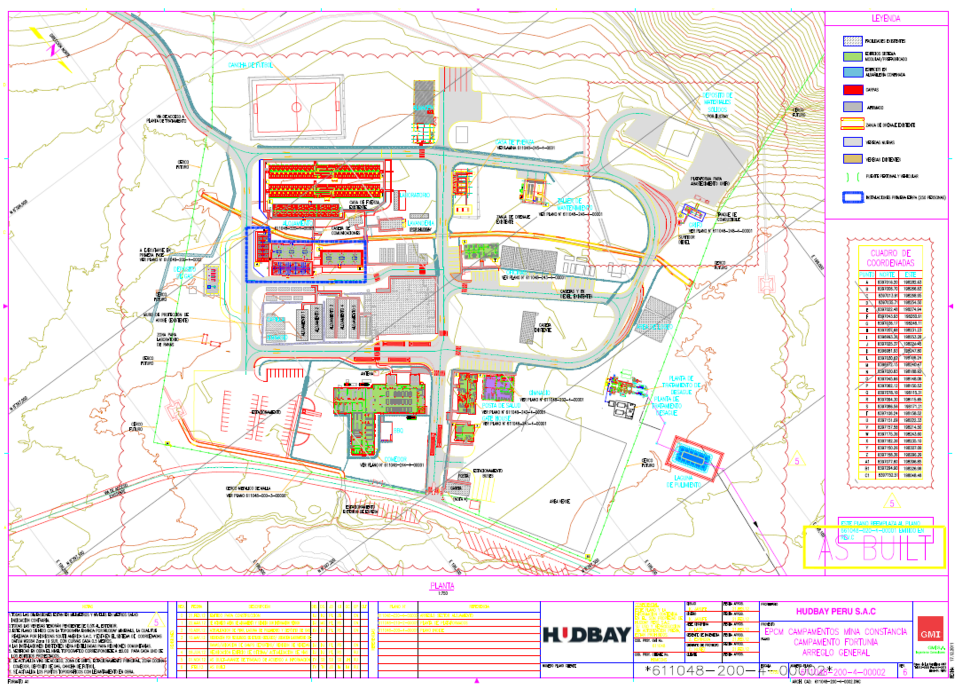


Figure 18-5 Fortunia Camp General Layout



18.3 ELECTRIC POWER SUPPLY AND TRANSMISSION

18.3.1 POWER SUPPLY

Power supply for the Constancia mine is brought from the new 220kV transmission line from Tintaya to Constancia that was built for and owned by Hudbay but is operated and maintained by a third party.

18.3.2 POWER TRANSMISSION

This transmission line is connected to the Constancia electric substation that transforms 220kV to 23 kV. The switching area (220kV / 23kV) with its transmission line arrival cells, harmonic filters, power transformers, and an electrical control room are located within this substation.

The electric control room manages the 23kV system that provides energy to the entire mine. In case of emergency, there are three generators that provide energy in a semi-automatic way. The diesel powered engines generate 1,500 kW of electricity at 23 kV and are connected to 23 kV system.

18.3.3 CURRENT STATUS

While the Constancia Substation and the 220Kv power line was built, and is maintained and operated by a contractor, this infrastructure and the land on which it was built will remain under the ownership of Hudbay and not become part of the electrical transmission company’s concession. The historical average and peak power consumptions are shown in Figure 18-6 and Figure 18-7.

Figure 18-6 Average Power Consumption (MW)

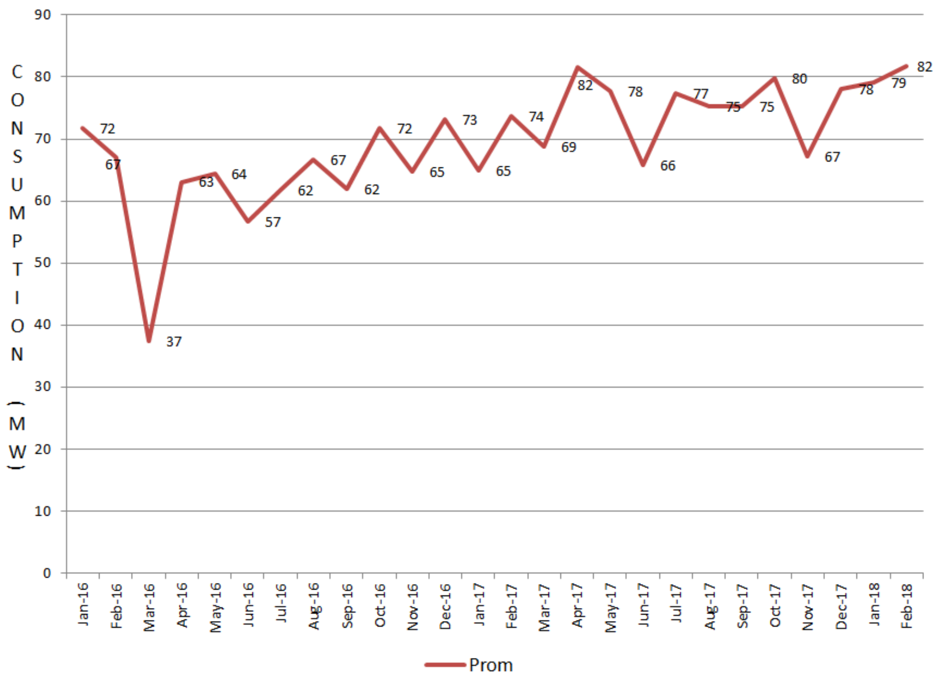
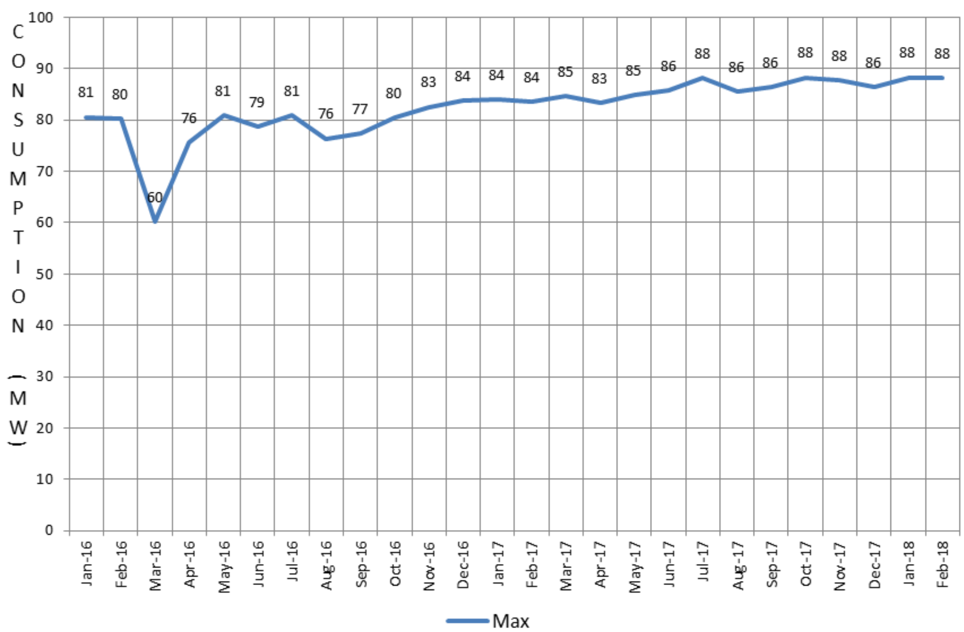


Figure 18-7 Power Consumption (MW) Peak Value



18.4 ROADS AND PORT

18.4.1 ROADS

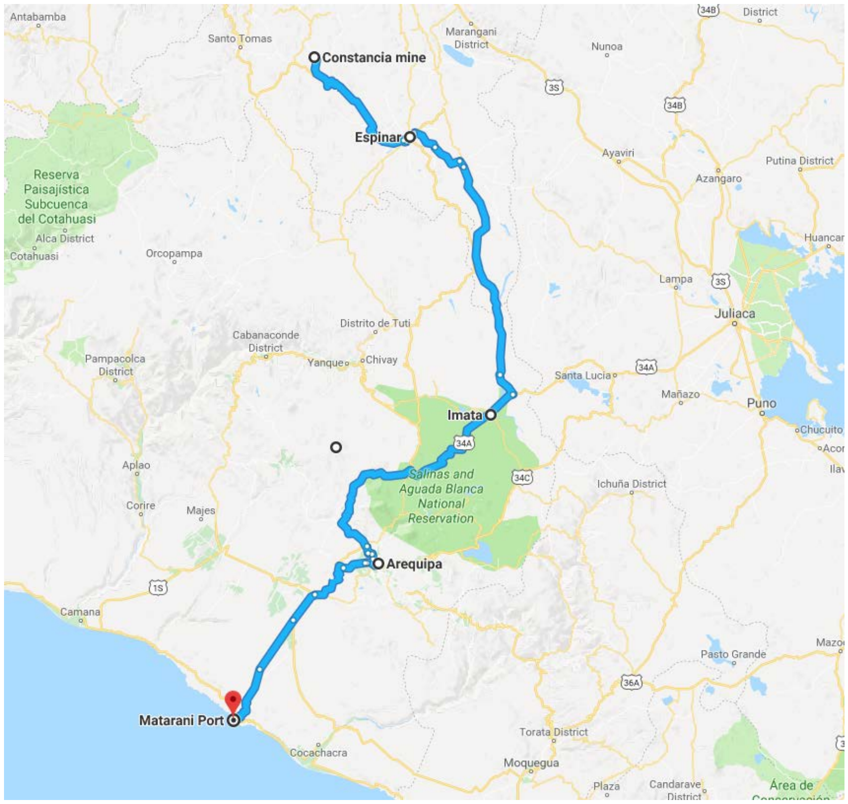
The primary road to the site, consists of a 65 km compacted dirt road (National Route PE-3SG) from Coporaque to the Puente Bailey/Constancia (Figure 18-8). These roads (and bridges) have been upgraded, as necessary, to meet the needs of construction and life of mine use.

The access road from Coporaque to Constancia consists of:

1. Coporaque District Road running from Coporaque to Puente Kero (4.97 Km)
2. Provias National Road running from Puente Kero to Desvio Livitaca (48.7 Km)
3. Velille District Road running from Desvio Livitaca to Constancia (10.83 Km)

The section of road between Yauri and the Mine is classified as a National Class 3 road (less than 400 vehicles per day). In order to maintain the access road in good condition, it is necessary for Hudbay to intervene and maintain the road as necessary due to the heavy truck traffic – concentrate & consumables transport. The most difficult section is the National road passing over Huallyapacheta pass at an elevation of 4,689 masl. Due to heavy rains and traffic counts, the bitumen surface quickly failed soon after the start of operations and the road was converted into a compacted dirt roadway. This was done to improve transitability and lower costs to Hudbay whom has assumed 100% of the routine maintenance. Hudbay has engineered an alternative 7.4 km road that will connect Constancia traffic to the Las Bambas HHR within 20 km of Constancia. This will allow for safer passage of the Constancia heavy haul traffic on a roadway designed & constructed in anticipation of this type of traffic. The plan is for Hudbay to continue maintaining this current roadway until the connection to the Las Bambas HHR is permitted & constructed. Hudbay has a checkpoint at Yauri for control of trucks travelling to and from site.

Figure 18-8 Peru – Constancia Site to Port Access Roads



18.4.2 PORT

Copper concentrate is shipped from the Constancia mine via road (~485km) and arrives at the Matarani port by trucks (Figure 18-9 Matarani Port (Aerial View)). These trucks are equipped with a hydraulically operated covered-box hinged at the rear, the front of which can be lifted to allow the concentrate to be deposited in the concentrate shed assigned to Hudbay by TISUR, the port operator.

Pier C has been assigned to Hudbay and has a 75 Kt capacity. A chute from the shed feeds a tubular conveyor into a ship loader system. The same conveyor and ship loading equipment is shared with other copper concentrate exporters.

The ship loading feeder conveyor is tubular to avoid concentrate losses and to comply with Peruvian regulations. The design, installation and operation of the port is by TISUR. Hudbay pays for its use on a per metric tonne basis.

Pier F was completed by TISUR in 2016, and is for the exclusive use of Cerro Verde (Freeport), Antapaccay (Glencore) and Las Bambas (MMG). Hudbay is the primary customer for Pier C. Both piers are designed to take concentrates by truck and by railroad.

Figure 18-9 Matarani Port (Aerial View)

19 MARKET STUDIES AND CONTRACTS

19.1 CONCENTRATE MARKETING

Copper Concentrates

Constancia copper concentrate is a clean, medium grade concentrate containing small gold and silver by-product credits. It is a highly desirable feedstock for copper smelters in China which is the most geographically appropriate freight destination but is also suitable for processing by smelters in Europe, India and South America. Table 19-1 outlines an approximate analysis of the concentrate produced at the Constancia mine.

Constancia copper concentrate is sold directly to a variety of copper smelters in Asia, Europe and India as well as to internationally recognized trading companies. Between 85 and 90% of sales are made pursuant to longer term frame contracts, which typically reference annual benchmark agreements between major concentrate producers and smelters for the purposes of fixing key terms such as treatment and refining charges. The balance of projected annual concentrate production has not been committed for sale in order to provide flexibility in the event of potential fluctuations in annual production and will be sold into the spot market each year at then-current market terms.

Constancia copper concentrate is trucked from the Constancia mine to the Matarani port facility in Southern Peru where it is loaded onto ocean vessels in lots of 10,000 or 20,000 mt for delivery to customers.

Table 19-1 Copper Concentrate Composition

Metal	Unit	Average	Range
Cu	%	25.0	22.0 – 28.0
Ag	g/t	120	90 – 200
Au	g/t	1.5	0.8 – 3.0
Zn	%	3.0	1.5 – 4.5
Pb	%	0.7	0.5 to 2.0
As	%	0.1	0.05 - 0.15

Molybdenum Concentrate

The Constancia metallurgical facility is equipped with a molybdenum flotation plant which produces a 50% Mo concentrate containing <1.0% Cu and trace amounts of Ca, Mg and Pb. Mo concentrates will be shipped to customers in South America, Asia or the United States in 1.5 mt bags in lots of approximately 20 mt. Marketing costs are estimated to be approximately 20% of the contained value of Molybdenum in concentrate

19.2 CONTRACTS

Production at Constancia is subject to a precious metals streaming agreement with Wheaton consisting of 50% of payable gold and 100% of payable silver. Hudbay will receive cash payments equal to the lessor of the market price and US$400 per ounce for gold and $5.90 per ounce of silver, subject to a 1% annual escalation starting 3 years after the completion date in 2016. Under the terms of the stream, gold recovery for the purposes of calculating payable gold will be fixed at 55% for gold mined from Constancia and 70% for gold mined from Pampacancha.

For operations and the assured supply of goods and services, standard contracts are place as well as policies and procedures for contracts execution and closure.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL

20.1.1 LEGAL FRAMEWORK

The objective of current mining legislation in Peru is to ensure that mining operations incorporate measures to prevent or mitigate foreseeable impacts to the surrounding environment, and that discharges meet the applicable effluent and emissions limits.

The mine was designed to comply with all legislation applicable to the development of mining projects in Peru including mines, roads, port and transmission lines. In addition, the mine complies with regulation regarding archaeological areas of significance, endangered and protected species as well as community relations and public disclosure programs. The main applicable regulation and international standards are detailed below:

- Peruvian Political Constitution, 1993
- Environmental General Law, 2005
- Private Investment Growth Law, 1991
- Environmental Impact for Works and Activities Evaluation Law, 1997
- Sustainable Use of Natural Resources Organic Law, 1997
- Environmental Impact Assessment National System Law, 2001
- Environmental Protection and Management Regulation for mining, Benefit, General Labor, Transportation and Mining Storage Activities, 2014
- Water Resources Law, 2009
- Water Quality National Standards, 2017
- Air Quality National Standards, 2017
- Noise Quality National Standards, 2003
- Soil Environmental Quality Standards, 2017
- Health Law, 1997
- Environmental Management National System Law, 2004
- Cultural Heritage Protection Law, 2004
- Solid Waste Law, 2016
- Forestry and Wildlife Protection Law, 2011
- Mines Closure Law, 2003
- Public Participation Regulations, 2008
- Maximum Permissible Levels for Liquid Effluents from Mining – Mineral Processing Activities, 2010

- IFC/World Bank Social and Environmental Performance Standards.
- Towards Sustainable Mining (TSM) Protocols of the Mining Association of Canada (MAC)

20.1.2 LOCAL AND REGIONAL CONTEXT

The Constancia mine is located in the districts of Chamaca, Livitaca, and Velille – all of which are in the Province of Chumbivilcas, Region of Cusco. It includes 5,452.41 ha of land purchased from private owners and the community of Chilloroya (District of Velille), as well as 256.50 ha of surface land granted in use by the community of Uchucarco (Chamaca District).

The area has a mining history that dates back 50 years to when Mitsui operated a copper-zinc mine in Katanga near the community of Uchucarco. Mitsui sold the mine to a Peruvian company which abandoned the mine in the late 1970s, leaving a legacy of environmental contamination. Tailings were discharged into the river basin, and the open pits and waste dumps were never reclaimed.

Near the community of Chilloroya, the local population has been traditionally engaged in artisanal and informal mining of superficial gold. Concerns have been raised by both Hudbay and the community to Peruvian authorities regarding environmental and labour conditions related to this activity and levels of mercury found in nearby water bodies exceeding international standards for the protection of aquatic life.

20.1.3 ENVIRONMENTAL SETTING

The surface water quality of the Chilloroya River and its different tributaries (micro watersheds) are generally within the established national standards for water quality. However, due to natural factors (mineralization of the area, water erosion, water runoff) and anthropogenic factors (environmental liabilities, artisanal mining and community activities) some values are above national standards. These were identified during the Environmental Baseline.

The pH of the Chilloroya River and its main tributaries varies in the wet and dry season; during the wet season the pH varies from neutral to slightly alkaline (6.5 to 8.5) and in the dry season the pH varies from slightly alkaline to alkaline (8.0 to 9.5). In most of the monitoring stations, pH showed characteristics with alkaline tendency, even exceeding national standards. There were also some isolated values with slightly acidic characteristics, below the lower pH limit.

The total metals concentrations in ravines and the Chilloroya River are normally below the national standards However, some total metal concentrations exceeded national standards, these exceedances usually happen during the wet season. Total metals that registered high concentrations were copper, lead, manganese, molybdenum and zinc. On the other hand, isolated values of aluminium, mercury and iron also exceeded national standards. These exceedances would be associated with the sediments dragging, environmental liabilities and several activities of informal mineral extraction.

The microbiological parametres (Escherichia coli, fecal and total coliforms) along the Chilloroya River increased compared with the Environmental Baseline, reporting values above the national standards. This increase is directly related to livestock activities and non-controlled discharges of domestic wastewater treatment systems from Juan Velasco Alvarado village and the Uchucarcco community.

Regarding air quality, maximum concentrations of particles (PM-10 and PM-2.5), were found in the towns of Uchucarco, Juan Velasco Alvarado and Urazana. All values were below national standards for 24 hours. Also, maximum concentrations of lead, arsenic and copper were recorded below national standards. Gas concentrations are mainly due to their own local emissions from light and heavy vehicle traffic surrounding the monitoring stations. None of the recorded values for gases exceeded the national standards. In general, air quality parametres, have been consistent with seasonality except for gases. Results showed higher concentrations during the dry season (August) than during the wet season (February and April).

Regarding flora, 342 different species of flora and 7 vegetation types were identified within the mining unit area (grassy brush, rock vegetation, very wet meadow, bog, rushes, bushes and high land vegetation). In total, there are 13 species that are within the conservation concern category in Peruvian legislation and 17 endemic species from Peru (4.97%) were recorded.

Regarding fauna, 256 species were recorded: 21 species of mammals, 59 species of birds, 4 species of amphibians, 2 species reptiles and 170 species of arthropods. Also, from the total amount of recorded species, 15 are of conservation concern: four mammal species, 9 species of birds, and 2 species of amphibians. From this, there is only one endemic species, the mammal calomys sorellus (reddish evening mouse).

A detailed hydrobiological evaluation of aquatic environments located within the mine area, was conducted. 6 species of fish were identified, 1 exotic (trout) and 5 native fish belonging to the "catfish" family, from these the trichomycterus rivolatus is in the category of "near threatened" as determined by the IUCN (2013).

An archaeological survey was completed in the areas directly affected by Constancia mine facilities and activities. A total of 46 archaeological sites were identified in the area of the mine. The certificate of non-existence of archaeological remains of significance (CIRA) was approved by the MC (Ministry of Culture) in September 2011. Hudbay has obtained a CIRA for (i) all the area comprised by the Constancia mine (excluding the archaeological sites as established in that same Certificate); (ii) for the route covered by the power transmission line (excluding the identify archaeological sites); and (iii) for an area for agricultural and forestry training. Later, as part of investigations into the Pampacancha sector, three archaeological sites and five cultural isolated elements were identified on the surface of Pampacancha. Following that, the corresponding Certificate of Nonexistence of Archaeological Remains was obtained in December 2015.

20.1.4 ENVIRONMENTAL INSTRUMENTS

Constancia Mine Environmental Impact Assessment (ESIA) was approved by the Ministry of Energy and Mines (MINEM) on November 24, 2010, by Directoral Resolution (DR) No 390-2010-MEM-AAM, to conduct mining activities, with an average ore processing capacity of 55,000 tons per day for a period of 15 years.

The first amendment to the ESIA (EISA MOD I) was approved by MINEM on August 20, 2013, through DR. 309-2013-MEM/AAM, to increase the processing capacity to 81,900 tonnes per day, and to reduce the useful mine life to 11 years. The purpose of this amendment was to present changes stemming from engineering development. Additionally, Hudbay obtained the approval of an Environmental Technical Report by DR. No. 454-2013-MEM-AAM for the inclusion of auxiliary components necessary to complete construction activities on schedule.

The second amendment to the ESIA (EISA MOD II) was approved by MINEM on April 17, 2015, through DR. 168-2015-MEM-DGAAM, to increase reserves through the expansion of the Constancia pit, inclusion of the Pampacancha deposit, and the Waste Rock Facility (WRF) and the Tailings Management Facility (TMF) expansion, among others. A 17-year useful life of the mining operations was approved through this amendment.

Additionally, in 2015 and 2016, Hudbay obtained the approval of two Environmental Technical Reports for the MOD II. The first one, approved by DR. No. 516-2015-MEM-AAM, included the relocation of auxiliary components. The second one, approved by DR. N° 63-2016-SENACE/DCA, included the modification, replacement and addition of auxiliary components, considering the optimization and usage change of existing components, and the use of areas enabled during mine construction.

Hudbay is currently working on the elaboration of the third amendment to the EISA (EISA MOD III) to include an early discharge from the TMF supernatant (as a contingency), optimize the water balance and water management plan, include an alternative access road for concentrate transportation, change the TMF dike design criteria, among others. The specific required permits and the correspondent mine closure plan amendment are going to be submitted once the EISA MOD III is approved.

20.1.5 ENVIRONMENTAL MANAGEMENT

Environmental and social impacts have been assessed and appropriate mitigation measures have been implemented. The EISA and amendments comply with national regulation and have adopted the Equator Principles and International Finance Corporation's (IFC) Performance Standards. The Towards Sustainable Mining (TSM) Standards of the Mining Association of Canada (MAC) are currently being implemented.

The hydrogeological report (Golder 2013), which led to a more refined hydrogeological model and plant water balance, addressed three principal concerns related to the detailed design of the mine and plant water systems: (1) to confirm there is not full connectivity between the pit dewatering area and reservoirs to the north of the pit; (2) determine the impact of slope stability on the pit wall design; and (3) to confirm availability of groundwater supply through life of mine.

Hudbay has engaged an Independent Project Review Board (IPRB) to conduct periodical reviews of the major earth structures on the mine, with regular visits to the site, as well as, for monitoring the progress of construction of the TMF. The plan for such review was developed during the construction stage, and is now in operation. Design reviews and site visits have been carried out at critical junctures throughout the course of construction and mine operations to observe the performance of the structure and development of subsequent raises to the dam and the impoundment.

The water management strategy includes the following:

- Zero water discharge to the environment from the plant, and PAG waste rock facility.
- Discharge from the TMF supernatant from 2021 and only during the rainy season, where early discharge will be required only as a contingency.
- Our prioritization of water sources for the plant is as follows:

1) Use of process water or water in contact with PAG infrastructures as pit sinkwater, WRF drainage water and TMF supernatant water in the process plant.
2) Use of water in contact with Non-PAG Infrastructure main sediment pond (MSP), and four other small sediment ponds. This type of water with sediments can be discharged in compliance with the Peruvian maximum allowable limit (LMP).
3) Use of pit dewatering groundwater for the process plant and dust control.

- The MSP and the NC2 and NC4 channels are used for water flows mitigation purposes, in particular they use water supply to mitigate any reduced river flow or water supply potential impacts of pit dewatering during dry season (26L/s up to 26.7 on 2020), and the 10 L/s based on an agreement with the Chilloroya community.

The Constancia environmental monitoring program is summarized in the March 2016 Environmental Monitoring Plan (PLA-AMB-04: Plan de Monitoreo Ambiental). This plan covers monitoring of air, noise and vibration, surface water quality and flow, surface lake/reservoir levels, groundwater levels and quality, domestic water consumption and quality, effluent monitoring, topsoil, soil and sediment, biological (flora and fauna) monitoring, hydrobiological and meteorological data collection.

During the operation phase, Hudbay is maintaining a joint environmental monitoring committee with the communities of direct and indirect influence.

From a biodiversity point of view, Hudbay is working on specific management plans for wetlands, flora and fauna; Hudbay has also developed a Biodiversity Action Plan (BAP), as generally anticipated and described in the EISA and as required in the MAC TSM protocol and IFC performance standard N° 6. The BAP considered the consolidation of commitments and effort regarding biodiversity. During 2017, Hudbay obtained the certification of the ISO 14001: 2004 and OHSAS 18001: 2007 standards. On March 2017 the certification audit process was carried out in compliance with the International Organization for Standardization (ISO) ISO 14001: 2004 standard and the OHSAS 18001: 2007 standard, resulting in zero non-conformances and 15 observations that have already been corrected. In 2018, the External Audit for migration process to ISO 14001: 2015 and the first Follow-up Audit of compliance with OHSAS 18001: 2007 will be carried out.

20.1.6 MINE CLOSURE PLAN

The Constancia closure plan, describing the closure activities for the mine components aligned with the EISA, was approved through Resolution No. 286-2012-MEM-AAM dated September 4th, 2012.

Constancia's closure plan was updated to incorporate recent studies and technological changes that will reduce costs and provide financial guarantees that Hudbay must provide annually to the Peruvian government. The updated plan was approved by the Ministry of Energy and Mines in June of 2015 by Directorial Resolution No. 255-2015-MEM-AAM. This document updated measures of closure and post closure of the mine, according to the EISA MOD I and EISA MOD II.

The approved updated plan estimates a closure cost of $206.5M at the end of mine life. The cost guarantees provided to the Peruvian Government from Hudbay Peru SAC account for the discounted cost as calculated per Peruvian legislation for the reclamation of actual disturbed mining activity.

20.2 PERMITTING

20.2.1 CONSTANCIA

Hudbay has obtained several permits for the construction and operation of the Constancia mine. All of these permits were supported by the environmental background provided by the ESIA and amendments.

The Constancia mine has secured all necessary permits and authorizations to conduct construction and operation activities at the mine. A summary of these permits is detailed below:

- <u>ESIA</u>. - Besides being an environmental background and requirement for other permits, ESIA allows the construction and operation of certain facilities that do not have a specific construction and operation permit, for instance, camps and stockpiles. As mentioned in previous sections, an ESIA for the Constancia mine was approved in 2010, and to date it has two amendments and three Environmental Technical Reports.
- <u>Certificate of Non-Existence of Archeological Remains (CIRA)</u>.- Although not a permit, it is highly recommended to obtain a CIRA from the Peruvian Ministry of Culture, in order to provide evidence that no archeological sites or remains are impacted as a consequence of the mine. On October 6th, 2011, Hudbay obtained CIRA 2011-366-MC for the area comprised by the Constancia mine, excluding identified archeological sites. In 2014 and 2015, Hudbay obtained five more CIRA certificates related to the mine: CIRA 2011-098; CIRA 2014-153: CIRA 2014-309, CIRA 2015-423; and CIRA 424.
- <u>Beneficiation concession (construction stage)</u>. - A beneficiation concession allows its titleholder to process, purify and refine minerals by using chemical and/or physical procedures in process plants. The procedure to obtain a beneficiation concession is mainly divided in two stages: (i) the construction stage; and, (ii) the beneficiation concession granting stage (i.e. granting of the operation license), which is requested after the process plant construction is finished. Through Resolution No. 189-2012-MEM-DGM/V dated June 8th, 2012, Hudbay obtained the authorization to construct the process plant, including the TMF and water collection pond. This Authorization was amended in 2014 by Resolution No. 274-2014-MEM-DGM/.
- Beneficiation Concession (Operation Stage). –
  - After the construction of the process plant, Hudbay obtained the corresponding operation licence which was granted by Directorial Resolution No. 315-2014-MEM-DGM. This resolution

  allows Hudbay to operate Line 2 of the process plant up to a capacity of 40,000 MT/day of ore, and operation of the TMF to 4050 metres (east dam).
  - In 2015, Hudbay obtained an authorization to operate Line 1 of the Process Plant and Molybdenum Plant to reach a treatment capacity of 81,900 MT / day, by Resolution No. 392-2015-MEM-DGM.
  - In 2016, Hudbay obtained the Authorization to operate the TMF initial dam, to level 4,070 by Resolution No. 0006-2016-MEM-DGM/V; the Authorization to operate the west tailing dam to level 4,065 by Resolution No. 0070-2016-MEM-DGM/V; and finally, the amendment of the TMF construction schedule was approved by Resolution No. 0618-2016-MEM-DGM/V.
  - In 2017, Hudbay obtained the Authorization to operate the TMF to level 4,080 by Resolution No. 0047-2017-MEM-DGM/V.
  - In 2018, we have recently obtained the authorization to operate the TMF to level 4,097 by Resolution No. 148-2018-MEM-DGMV. This permit included the TMF dam construction using the central line raising technique from the crest elevation level 4,090.
- Underground Water License (pit dewatering). - Through Administrative Resolution No. 512-2014-ANA-ALA AAV Hudbay obtained the underground water license. This permit authorizes the use of groundwater from wells installed for the pit dewatering for mining purposes, for an annual volume of 2,522,880 m3. This license was later modified by Directorial Resolution No. 603-3015-ANA-AAA.XI-P.A., which authorizes the use of 5,676,000 m3 of groundwater from the pit dewatering.
- Surface Water License. - Through Administrative Resolution No. 46-2015-ANA-AAA.XI.PA, Hudbay obtained a surface water license, which authorizes the use of surface water in the area of the mine for mining purposes, for an annual volume of up to 22,780,000 m3.
- Mine Permit. - On December 28th, 2012, Hudbay obtained authorization to start mining exploitation activities through Directorial Resolution No. 279-2012-MEM/DGM. This Resolution approves the mining plan and authorizes the exploitation activities in the Constancia and San Jose pits. In December 24th, 2013, the Ministry of Energy and Mines approved the first amendment to this authorization, by Directorial Resolution No. 477-2013-MEM-DGM/V. This resolution granted the authorization for the development and preparation activities of the waste rock facility and auxiliary services. Two more permits related to the mine permit are Resolution 189-2014-MEM-DGM; and Resolution No. 451-2014-MEM-DGM/V. The first authorizes operation of the ion of waste rock facility in its first stage (up to 4180 metres); while the second authorizes the operation of the waste rock facility (NAG) up to 4250 metres.

PAMPACANCHA

As mentioned above, Hudbay has secured the ESIA MOD II and the mine closure plan, which are the environmental certifications required to support mine activities in Pampacancha.

In addition, Hudbay has started a negotiation process with the Chilloroya community to acquire rights over the Pampacancha surface. This is required to continue with the permitting process..Once land title is secured, Hudbay will apply for the following permits: (i) a three-stage underground water license for pit dewatering activities; (ii) modification of the current Constancia surface water license; (iii) authorization to start mine development and exploitation; and (iv) explosives permits.

20.3 SOCIAL

Between February and April 2012, Hudbay reached agreements with the neighboring communities of Uchucarco and Chilloroya for the land required for the Constancia mine. These agreements were validated in a meeting with support from two thirds of the community members. Both of these agreements have been recorded in the public land registry.

A compensation plan was prepared for the 36 landholding families from the lchuni area of Chilloroya, taking into account the needs of each family with respect to restructuring their working activities and adapting to relocation. The purpose of the plan was to cover all aspects a family needs to live a sustainable life. To this end, an approach was developed where the compensation plan established general criteria that should be adjusted in the individual negotiation process with each family, to reach an agreement that satisfied both parties. As the impact on each family was different, the Resettlement Action Plan (RAP) sought to ensure that each affected family was compensated fairly. This criterion was prepared in compliance with national laws and international provisions and standards in resettlement matters and in particular the IFC performance standards.

As a result of the RAP implementation, the resettlement process was successfully completed in 2016.

A Community Relations Plan has been developed taking into account the following requirements:

- Final determination of the Direct and Indirect Areas of Influence of the mine, based on the outcomes of the Social Impact Analysis as mentioned above.
- The needs in the construction and operation phases, as determined in the Social Impact Analysis.
- The State's requirements, as expressed in the Community Relations Guidelines of the MINEM and the Prior Commitment Act (DS-042-2003-EM).
- The requirements of international financial institutions, taking into account as main references the Equator Principles, IFC Performance Standards, United Nations Environment Programme's APELL (Awareness and Preparedness for Emergencies at Local Level) for Mining, and, supplementary social management standards.

Specific social programs were designed for the mitigation and prevention of identified impacts. Generally, these programs addressed the following key topics:

- Communication and consultation
- Participatory monitoring

- Infrastructure
- Social development
- Economic development
- Strategic development
- Claims and dispute resolution.

Stakeholder mapping has been undertaken and is periodically updated, identifying the two principal stakeholder groups in the direct area of influence as the communities of Uchucarco and Chilloroya. There are three well-defined groups in each of the communities, namely:

- Community assembly: people from the community.
- Artisanal miners: from the community and who also possess land in the community (but do not possess title to the land).
- Youth Groups: (in the case of Chilloroya and Uchucarco, represented through the Youth Association) have higher education than the rest of the local residents and the experience of having lived in other cities; however, they own no land and possess no economic capital.

Benefits from the artisanal mining activities in Chilloroya are not distributed evenly across the population, but are primarily collected by those members of the population who possess the land where the illegal mining activities occur.

Uchucarco has a greater number and variety of organisations inside its community structure; some of which have duties that are independent from the Governing Board, these include: JAAS, the Committee of Water Users ("Comité de Regantes"), among others. In addition, Uchucarco is the most active community in the District of Chamaca, and has more relationships with district and private institutions.

In Chilloroya and Uchucarco, the political strength of the Governing Board is solid and the different associations within the community are not well organised. Although there is an array of organisations, many are focused on specific tasks and have little political presence, as in the case of women's and parents' organisations.

The main economic activity in the area is agriculture and cattle farming. A labour force with basic mining knowledge is present in Espinar. Food and basic supplies can be obtained in Espinar.

Arequipa and Cusco are the nearest major centres, and both are over 300 km from the mine site by road.

20.4 HEALTH AND SAFETY

Table 20-1 represents fourteen high-risk activities related to mining operations identified and their specific controls implemented.

Table 20-1 Identified Risks and Controls

Risk	Controls
Work with Hazardous Energy Sources (mechanical, pneumatic, hydraulic, thermal, atmospheric, kinetic electromagnetic, residual or stored)	Qualified, authorized and accredited staff. Communicate and obtain permits for locking out circuits to intervene. Isolate and lock out the main power source of equipment, or circuits to intervene. Use tags and personal locks. Drain stored energy if necessary.
Work in Confined Spaces.	Accredited staff and qualified watch/look-out person. Prepare the High Risk Work Permit (PETAR). Monitor atmosphere in work environment. Identify, isolate and lock out all energy sources entering and leaving that confined space. Keep updated record of entry and exit to that confined space. Ventilated confined space. Verify that the emergency response service is alerted.
Openings in Floors or Removal of Railings (open hole, horizontal opening, vertical opening)	Trained and accredited personnel. Have Safe Work permit and "Open Hole" permit. Verify fall protection systems. Review protection barriers and verify they are signaled. Check that all platforms, floors and railings removed are secured.
Lifting or Suspended Loads	Trained and accredited operator and rigger. Pre-operational inspection of equipment, accessories and lifting devices, before the maneuver. Inspect the work area. Verify that the area for maneuvers is demarcated and signaled. Review permit for Critical Lifting (where applicable). Maintain effective communication between operator and rigger.
Work at Heights (Any work above 1.80 metres, measured from the platform where feet are located)	Trained and accredited personnel. Verify Permit for Work at Heights (PETAR). Verify that the fall protection systems (harness, anchor line, anchor points, etc.) are certified, inspected and installed properly. Verify that portable ladders have inspection and maintenance record. Pre-operational inspection of equipment. Verify signaling at lower levels.
Excavations and Trenches.	Trained and accredited personnel. Excavation permit. Verify study of soil mechanics for excavations deeper than 2 metres. Have a support design for excavations deeper than 1.20 metres. Excavations deeper than 1.50 metres require Confined Space permit. Material from the excavation should be stacked 1 meter away from the edge. Verify safe entrances and exits. Verify demarcation and signs.

Hot Works (welding, cutting sparks, grinding and other related work)	Trained and accredited personnel. Inspect the area where hot works will be conducted. Have the procedure for Hot Works, written permit for High Risk Works (PETAR). Verify that staff have P.P.E. (Personal Protection Equipment) for hot works. Have enough fire extinguishers depending on the type of combustible material. Check grounding cables. Verify the monitoring of atmospheres in tanks, containers or piping systems containing flammable liquids or gases. Check the area 30 minutes after completing the work.
Operation of Heavy, Light and Mobile Equipment	Equipment operator or driver shall have a valid Internal License and/or accreditation. Pre-operational inspection of equipment. Proper use of seat belts. Respect minimum distances between equipment. Verify height of the safety wall, it should not be less than 3/4 of the diameter of the tire of the largest vehicle traveling in the area.
Work with Energized Circuits	Isolate, lockout and tag-out (ABE) all potential sources of energy. Tag circuit breakers. Verify that electrician personnel are accredited. Check that PPE is in good condition.
Works with or near Power or Moving Parts	Identify pinch points, cut, abrasion or projection. Verify that protective guards are implemented and in good condition. Signal to identify sources of energy. Check emergency stop devices.
Works with or near Chemicals (H2S)	Review the Material Safety Data Sheet (MSDS). Check labeling and storage according to MSDS. Check that the P.P.E. (Personal Protection Equipment) is appropriate according to MSDS. Verify fixed monitors or portable monitors for gases. Verify knowledge and authorization of staff using and transporting the chemical. Check whether the chemical requires an antidote in case of emergency.
Works with Explosives	Trained and accredited personnel. Verify that the transport of explosives is performed only in authorized vehicles. The storage of explosives is performed only in authorized magazines. Check blasting lookout people or watches. Verify demarcation of blasting area. Have a blueprint identifying the blasting influence area. For loaded blasting projects, personnel is evacuated 500 metres away from the loaded face.
Works with HDPE Pipes	Trained and accredited personnel. Verify Safe Work permit and permit for Safe Handling of HDPE pipes. Check the presence of a watch/ look-out person. Verify that equipment, components, tools and/or accessories are approved for transportation. Check the maximum height for stacking HDPE pipes. Establish safe zones for unloading and handling pipes.

Work in Water Bodies and Tailings	Specific procedures for working in water bodies and tailings. Trained and accredited personnel, including the following controls: training, entrance permits for water bodies and tailings, specific availability of equipment (life jackets and lines) and ongoing monitoring.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The LOM sustaining capex is estimated to be 748M (excluding capitalized stripping) at Constancia while the capex for the Pampacancha project is estimated to be 19M. All capex items are reported in real 2018$ USD.

The total includes capital required for major mining equipment acquisition, rebuilds, and major repair. The cost also includes site infrastructure expansion (tailings management facility, waste rock facility and others) and process plant infrastructure however they exclude all the cost related to mine closure. Project capex associated with Pampacancha does not include acquiring surface rights but includes all other items.

The capital costs for Constancia are developed and revised on an annual basis as part of the budget cycle. The LOM capital plan is shown in Table 21-1.

Table 21-1 Sustaining Capex

Sustaining Capex		2018	2019	2020	2021	2022	2023-36	Total
Constancia								
Equipment - Purchase	US$'000s	2,800	-	3,250	-	-	16,250	**22,300**
Equipment - Major Repair	US$'000s	10,342	13,020	16,967	7,460	17,878	221,634	**287,301**
HCW - Tailings Dam	US$'000s	3,709	16,550	43,604	1,204	510	220,860	**286,437**
HCW - Waste rock facility	US$'000s	-	-	5,000	-	-	5,000	**10,000**
Mining - Other	US$'000s	8,138	6,611	2,500	3,000	3,000	26,600	**49,849**
Plant - Equipment & spares	US$'000s	1,120	1,000	1,600	1,600	1,600	13,000	**19,920**
Plant - Tailings pipeline	US$'000s	2,340	8,000	600	600	600	7,800	**19,940**
Plant - Other	US$'000s	945	34,900	1,000	1,000	1,000	13,000	**51,845**
Total (Before Capitalized Stripping)	**US$'000s**	**29,393**	**80,081**	**74,521**	**14,864**	**24,588**	**524,144**	**747,592**
Total (After Capitalized Stripping)	**US$'000s**	**54,849**	**88,385**	**89,594**	**35,708**	**34,961**	**737,878**	**1,041,375**

Pampacancha Project Capex		2018	2019	2020	2021	2022	2023-36	Total
Equipment - Purchase	US$'000s	-	3,000	-	-	-	-	**3,000**
HCW - General & other	US$'000s	-	7,738	1,000	1,000	-	-	**9,738**
Heavy CW - Pit dewatering	US$'000s	2,998	2,858	-	-	-	-	**5,856**
Total Pampacancha	**US$'000s**	**2,998**	**13,596**	**1,000**	**1,000**	**-**	**-**	**18,595**
Total Sustaining & Project Capex (Before Capitalized Stripping)	**US$'000s**	**32,392**	**93,677**	**75,521**	**15,864**	**24,588**	**524,144**	**766,186**
Total Sustaining & Project Capex (After Capitalized Stripping)	**US$'000s**	**57,847**	**101,981**	**90,594**	**36,708**	**34,961**	**737,878**	**1,059,969**

21.2 OPERATING COSTS

The operating costs at Constancia are developed annually as part of the site budget process. All operating costs are reported in real 2018$ USD. The operating costs are divided in three categories: mining, milling and G&A. The LOM operating costs are shown in Table 21-2.

Table 21-2 On-Site Operating Costs

Operating Costs		2018	2019	2020	2021	2022	2023-36	LOM
Unit Costs								
Mining	(US$/tonne Milled)	3.04	2.80	2.93	2.89	2.83	2.78	2.81
Milling	(US$/tonne Milled)	4.11	4.21	4.32	4.36	4.32	4.25	4.25
G&A	(US$/tonne Milled)	1.68	1.66	1.57	1.53	1.53	1.35	1.41
Total Operating Costs (Before Capitalized Stripping)	(US$/tonne Milled)	**8.82**	**8.67**	**8.82**	**8.78**	**8.68**	**8.38**	**8.48**
Total Operating Costs (After Capitalized Stripping)	(US$/tonne Milled)	**8.01**	**8.41**	**8.34**	**8.11**	**8.34**	**7.86**	**7.96**

The mining operating costs are estimated for each activity, i.e. drilling, blasting, loading, haulage, auxiliary and indirect cost with the haulage activity having the highest impact on the total mining operation cost. For each activity, the volume of consumables and manpower requirements are calculated from the details of the life of mine plan. A breakdown of the mining operating costs by activity and component are shown in Table 21-3 and Table 21-4.

Table 21-3 Operating Costs – OPEX – Mining/Activity

erating Costs (Unit Cost)		2018	2019	2020	2021	2022	2023-36	LOM
ning	**(US$/tonne Moved)**	**1.32**	**1.22**	**1.36**	**1.30**	**1.23**	**1.32**	**1.31**
Drilling	(US$/tonne Moved)	0.04	0.04	0.04	0.04	0.04	0.04	0.04
Blasting	(US$/tonne Moved)	0.15	0.15	0.15	0.15	0.15	0.15	0.15
Loading	(US$/tonne Moved)	0.21	0.22	0.21	0.23	0.20	0.22	0.22
Haulage	(US$/tonne Moved)	0.46	0.37	0.45	0.43	0.41	0.47	0.46
Auxiliary	(US$/tonne Moved)	0.19	0.18	0.21	0.18	0.18	0.19	0.19
Indirect	(US$/tonne Moved)	0.27	0.27	0.29	0.27	0.26	0.26	0.26
tal Operating Costs	(US$/tonne Moved)	**1.32**	**1.22**	**1.36**	**1.30**	**1.23**	**1.32**	**1.31**

Table 21-4 Operating Costs – OPEX – Mining/Components

Operating Costs (Unit Cost)		2018	2019	2020	2021	2022	2023-36	LOM
Mining	**(US$/tonne Moved)**	**1.32**	**1.22**	**1.36**	**1.30**	**1.23**	**1.32**	**1.31**
Direct	(US$/tonne Moved)	1.05	0.95	1.07	1.03	0.98	1.06	1.05
Labour	(US$/tonne Moved)	0.14	0.12	0.15	0.14	0.14	0.15	0.15
Diesel	(US$/tonne Moved)	0.38	0.38	0.42	0.41	0.40	0.46	0.44
Maintenance	(US$/tonne Moved)	0.39	0.30	0.36	0.33	0.30	0.31	0.32
Explosives	(US$/tonne Moved)	0.14	0.15	0.15	0.14	0.14	0.14	0.14
Indirect	(US$/tonne Moved)	0.27	0.27	0.29	0.27	0.26	0.26	0.26
Total Operating Costs	(US$/tonne Moved)	**1.32**	**1.22**	**1.36**	**1.30**	**1.23**	**1.32**	**1.31**

The operating costs in the process plant are divided into seven main components, with power having the greatest impact on the process plant cost. The G&A cost includes all administrative areas at site as well the main office in Lima. A breakdown of milling and G&A costs are show in Table 21-5.

Table 21-5 Operating Costs – OPEX – Milling and G&A

Operating Costs (Unit Cost)		2018	2019	2020	2021	2022	2023-36	LOM
Milling	**(US$/tonne Milled)**	**4.11**	**4.21**	**4.32**	**4.36**	**4.32**	**4.25**	**4.25**
Labour	(US$/tonne Milled)	0.28	0.27	0.30	0.30	0.30	0.29	0.29
Services	(US$/tonne Milled)	0.06	0.06	0.06	0.06	0.06	0.05	0.06
Misc materials	(US$/tonne Milled)	0.02	0.02	0.02	0.02	0.02	0.02	0.02
Travel & accomodation	(US$/tonne Milled)	0.05	0.06	0.06	0.06	0.06	0.05	0.05
Maintenance (services & spares)	(US$/tonne Milled)	0.62	0.62	0.62	0.62	0.62	0.61	0.62
Chemical lab	(US$/tonne Milled)	0.03	0.03	0.03	0.03	0.03	0.02	0.02
Reagents	(US$/tonne Milled)	0.43	0.48	0.56	0.59	0.55	0.50	0.51
Fuel (diesel)	(US$/tonne Milled)	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Liners & other consumables	(US$/tonne Milled)	0.32	0.33	0.33	0.33	0.33	0.33	0.33
Steel (balls)	(US$/tonne Milled)	0.85	0.86	0.86	0.86	0.86	0.86	0.86
Power	(US$/tonne Milled)	1.45	1.48	1.48	1.48	1.48	1.48	1.48
G&A	**(US$/tonne Milled)**	**1.68**	**1.66**	**1.57**	**1.52**	**1.53**	**1.35**	**1.41**
Total Milling & G&A	(US$/tonne Milled)	**5.79**	**5.87**	**5.89**	**5.87**	**5.84**	**5.59**	**5.66**

Waste stripping operating costs that provide future economic benefits are capitalized when strip ratios in a given year are above the average strip ratio for a total pit phase (capitalized stripping). Operating costs that are expected to be capitalized as stripping are summarized in Table 21-6.

Table 21-6 Capitalized Stripping (Total Estimate and Unit Costs)

Capitalized Stripping Costs		2018	2019	2020	2021	2022	2023-36	LOM
Capitalized Stripping	US$ 000	25,455	8,304	15,073	20,844	10,373	213,734	293,783
Capitalized Stripping	(US$/tonne Moved)	0.35	0.12	0.22	0.30	0.15	0.25	0.24
Capitalized Stripping	(US$/tonne Milled)	0.81	0.27	0.48	0.67	0.33	0.52	0.52

Cash costs and sustaining cash costs per pound of copper are summarized in Table 21-7. Cash costs include mining, milling, G&A, off-site costs and TCRCs. Sustaining cash costs also include sustaining capital and royalties (but exclude Pampacancha project capex). Both cash costs and sustaining costs include the impact of capitalized stripping are reported net of by-product credits (calculated at reserves prices) including the impact of the precious metal streams.

Table 21-7 Operating Costs and Sustaining Costs per Pound

Operating Costs (Unit Cost)		2018	2019	2020	2021	2022	2023-36	LOM
Cash Cost	US$/lb Cu	1.35	1.29	1.05	0.94	1.06	1.59	1.44
Sustaining Cash Cost	US$/lb Cu	1.60	1.66	1.44	1.11	1.22	1.93	1.75

22 ECONOMIC ANALYSIS

Pursuant to NI 43-101, producing issuers may exclude the information required for Section 22 Economic Analysis on properties in production, unless the technical report includes a material expansion of current production.

As Hudbay is a producing issuer, it has excluded information required by Item 22 of Form 43-101F1 as the Pampacancha expansion does not represent a material expansion of current production at the Constancia mine.

23 ADJACENT PROPERTIES

There are no relevant adjacent properties to the Constancia and Pampacancha deposits.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information material to the Constancia mine that is necessary to make this Technical Report not misleading.

25 CONCLUSIONS AND RECOMMENDATIONS

The Constancia mine operation has been mining ore and waste since March 2014 and began commercial production in April 2015. Since that time the mine has run relatively uninterrupted and consistently achieved improved performance with regards to metallurgical recovery and throughput. The deposit represents what is considered by industry standards a low grade copper porphyry with the principal mineralized ore type being hypogene in nature. The plant has proven to produce a marketable clean concentrates for both molybdenum and copper.

As of December 31st, 2017 the mineral resource estimate is distributed in two open pit shells, Constancia and Pampacancha. The Hudbay internal peer reviews and validation processes including a thorough reconciliation between the reserve model and the past 2.5 years of operations have confirmed that the models are constructed utilizing industry best practices and sound QAQC principles.

The economic and design parametres modelled, assumed and stated provide for economic extraction of reserves from the stated resource.

The resource and reserve estimate meet with the criteria outlined in industry best practices as outlined in Canadian institute of Mining, Metallurgy and Petroleum Definition Standards for mineral Resources and Mineral Reserves (May2014).

The production and compilation of this technical report was performed by the capable and professional management and staff of the corporate office of Hudbay Minerals corporate and at the Constancia mine. The supervision, revision and approval of the assembly of this report is by the QP Cashel Meagher, PGeo SVP and COO of Hudbay Minerals Inc.

25.1 CONCLUSIONS

- The resource economic parametres utilized are slightly different than those to support the statement of mineral reserve. Although the parametres differ it is the opinion of the QP that changing the resource parametres that outline the resource pit shell will not materially change the output of the reserve. That is the reserve lies wholly within the resource as outlined.
- The resulting mine plans account for feed restrictions as stipulated by the processing plant with regards to feed grade and ratios of zinc, iron, oxide and mineral type.

- The current performance of the operation supports the position that the plant is capable of 90,000 tonnes per day average production, including consideration for maintenance.
- Ongoing sustaining capital reported, sufficiently outlines the material capital requirements for mine fleet, plant maintenance and tailings expansion.
- Separate project capital outlined adequately accounts for development of the Pampacancha satellite deposit.
- Relevant mining permits and land title are in good standing excluding the required surface rights for the Pampacancha deposit exploitation.
- The metallurgical test work supporting the metallurgical recovery in section 13 are restated from the work compiled in the Technical Report published in 2016.
- The Pampacancha pit provides for higher grade copper equivalent grades than what remains available in the Constancia open pit.
- Other than the risks described in this report (including the need to acquire surface rights to mine the Pampacancha deposit), the general political and social risks associated with operating in Peru and the other risk factors described in Hudbay's most recent annual information form, there are no known significant risks and uncertainties that could reasonably be expected to materially affect the potential development of the mineral reserve and resource estimates in this report.

2.5.2 RECOMMENDATIONS

Thorough reconciliations from mineral reserve estimates to mill credited production will continue to be closely monitored in order to continue to validate the performance of the new reserve model.

Recent metallurgical testing has helped refine the forecast model for mill throughput and metals recovery. This work has also helped to identify opportunities for blending and plant /reagent modifications to further improve the plant performance going forward. Optimization studies will continue with a focus on improving recovery on end members of the grain size distibution, i.e. ultra fine and coarser fractions of the mineralization.

Further review is required regarding the operating practices and the performance of the molybdenum circuit to best understand what parametres and production can be improved.

26 REFERENCES

- Andes, Actualización del Estudio Geotécnico del Tajo Comnstancia (Spanish), Technical Report, Enero 2016
- AMEC, Amec QC Evaluation, NI 43 101 Technical Report, Constancia Project, August 2011.
- AMEC, Checking Samples, NI 43 101 Technical Report, Constancia Project, From 2006 to 2007.
- AMEC, Constancia Mineral Resources Estimate, NI 43 101 Technical Report, Constancia Project, August 2011.
- Amphos 21, Estudio Hidrogeológio Tajo Pampacancha Proyecto Expansion Constancia Estudio de Factyibilidad de Pampacancha (Spanish), (Spanish), Technical Report, September 2013
- Amphos 21, Actualización del Manejo de Aguas. Proyecto Expansion Constancia Estudio de Factyibilidad de Pampacancha (Spanish), Technical Report, September 2013
- Amphos 21, Estudio Hidrogeológico de la Ampliación del WRF. Proyecto Expansion Constancia Estudio de Factyibilidad de Pampacancha (Spanish), Technical Report, September 2013
- Ausenco PSI, Constancia Tailings Transport System Detail Engineering, Lab Report Samples AdoB420 and AdoB421 (PSP5605-2000-INF-HI-006), April 2012.
- C.H. Plenge & Cia, Metallurgical Investigation No.6447-6449 Norsemont Peru S.A.C. Mina Constancia Composites HY, SG, SK, March 2007.
- C.H. Plenge &Cia, Metallurgical Investigation No.6637-6639 Norsemont Peru S.A.C. Mina Constancia Composites PSG, PHY, PSK, July 2008.
- Golder Associates. Prefeasibility Hydrogeological Assessment Pampacancha Pit, Final Report, Constancia Project, August 2012.
- Golder Associates. Pre-feasibility geotechnical study for Constancia Pit Expantion. Tehnical Report. March 2014.
- Golder Associates. Pre –feasibility study for expanded capacity of the Tailings management Facility – Constancia Projetc. Technnical Report, March 2014.

- Golder Associates. Pre-Feasibility study for expande capacity of waste dump (WRF) - Constancia Projet. Technical report, March 2014
- JKTech Pty Ltd, SMC Test Report on Fifty Samples from Constancia Project Tested at A.R. MacPherson Consultants Ltd., SGS Lakefield Research Chile S.A., Santiago, Chile for Norsemont (JKTech Job No. 08004), May-June 2008
- JKTech Pty Ltd, SMC Test Report on Two Samples from Constancia Project Tested at A.R. MacPherson Consultants Ltd., SGS Lakefield Research Chile S.A., Santiago, Chile for Norsemont (JKTech Job No. 08004) , June 2008
- Knight Piesold, "Constancia Project Detailed Engineering Climatological Data Analysis Report", December 8, 2011, Prepared for Hudbay Perú S.A.C. by Knight Piésold Consultores S.A.
- Knight Piésold, Open Pit Slope Design, Technical Report, Constancia Project, January 2013.
- Knight Piésold, Prefeasibility Pit Slope Design for the Pampacancha Pit, Technical Report, Constancia Project, July 2012.
- Knight Piésold, Pampacancha Preliminary Geochemical Assessment, Technical Report, Constancia Project, May 2012.
- Itasca, Estudio Geotécnico del Tajo Pampacancha (spanish), Technical Report, september 2013.
- Norsemont, Data Verification (Norsemont QC Protocol), NI 43 101 Technical Report, Constancia Project, From 2006 to 2012.
- Pampacancha: SGS, Sample Preparation and Assaying, NI 43 101 Technical Report, Constancia Project, For Pampacancha 2011
- Pocock Industrial, Sample Characterization & PSA, Flocculant, Screening, Gravity Sedimentation and Pulp Rheology Studies, April 2012
- SGS Canada Vancouver, An investigation into Flotation Behavior and QEMScan Characteristics of Constancia Samples, April 2005.
- SGS Lakefield Research Chile, An investigation by QEMScan into the Mineralogical Characteristics of the Composite Samples from the Constancia Project (Project 4169-Q77), April 2009
- SGS Lakefield Research Chile, Final Report: Third Report (Third Flotation Stage) Metallurgical Test Program (Project Code 4169) , September 2009.

- SGS Lakefield Research Chile, June 2008. An investigation by QEMScan into the Mineralogical Characteristics of the Heads Products from the Constancia Project (Project 4169-Q33)
- SGS Lakefield Research Chile, Pruebas de Molienda y Medición de Densidad, (Proyecto OL-4081/Informe final), August 2007
- SGS Lakefield Research Chile, Report: Molybdenum Separation Flotation Test Program (Project Code 4169) , September 2009
- SGS Mineral Services Chile, Programa de Ensayos Piloto de Flotación con Mineral de Constancia (Proyecto 4169 - Informe #2), March 2009
- "Norsemont Perú S.A.C. Proyecto Constancia Estudio de Impacto Social y Ambiental" (Spanish), March 2010, Prepared for Norsemont Perú S.A.C. by Knight Piésold Consultores S.A.
- "Cuarta Modificación Del Estudio De Impacto Ambiental Semi-Detallado (Categoria II) Proyecto De Exploración Constancia" (Spanish), April 2012, Prepared for Hudbay Perú S.A.C. by Insideo S.A.C.
- "Norsemont Perú S.A.C. Proyecto Constancia Plan de Cierre" (Closure Plan in Spanish), November 2011, Prepared for Norsemont Perú S.A.C. by Knight Piésold Consultores S.A.
- "Norsemont Perú S.A.C. Proyecto de Exploración Constancia Tercera Modificación del Estudio de Impacto Ambiental Semidetallado Categoría II" (Spanish), December 2010, Prepared for Norsemont Perú S.A.C. by Knight Piésold Consultores S.A.
- "Hudbay Peru S.A.C. Modificación del Estudio de Impacto Social y Ambiental del Proyecto Constancia" (Spanish), October 2012, Prepared for Hudbay Perú S.A.C. by Insideo S.A.C.
- "Hudbay Peru S.A.C. Informe Técnico Sustentatorio de la Ampliación de Componentes del EIA del Proyecto Constancia" (Spanish), September 2013, Prepared for Hudbay Perú S.A.C. by Golder Associates S.A.C.
- "Hudbay Peru S.A.C. Segunda Modificación del Estudio de Impacto Social y Ambiental del Proyecto Constancia – Ampliación Pampacancha" (Spanish), December 2013, Prepared for Hudbay Perú S.A.C. by Golder Associates S.A.C.
- "Hudbay Peru S.A.C. Informe Técnico Sustentatorio para la Inclusión de Componentes Auxiliares en la U.M. Constancia" (Spanish), August 2015, Prepared for Hudbay Perú S.A.C. by Golder Associates S.A.C.

- “Hudbay Peru S.A.C. Modificación del Plan de Cierre de Minas de la U.M. Constancia” (Spanish), February 2015, Prepared for Hudbay Perú S.A.C. by Golder Associates S.A.C.
- “Hudbay Peru S.A.C. Segundo Informe Técnico Sustentatorio: Modificación, Sustitución y Adición de Componentes Auxiliares” (Spanish), Julio 2016, Prepared for Hudbay Perú S.A.C. by Golder Associates S.A.C.
- “Titan-24 DC/IP/MT Survey Geophysical Report”, August 2011, Prepared for Norsemont Perú S.A.C by Quantec Geoscience Ltd.

27 SIGNATURE PAGE

This Technical Report titled "NI 43-101 Technical Report, Constancia Mine Cuzco, Peru", dated March 29, 2018 and effective as of December 31, 2017, was prepared under the supervision and signed by the following author:

Dated this 29th day of March, 2018

(signed) Cashel Meagher

__"Cashel Meagher"________
Signature of Qualified Person

Cashel Meagher, P. Geo.
Senior Vice President and Chief Operating Officer
Hudbay

28 CERTIFICATES OF QUALIFIED PERSON

CASHEL MEAGHER

CERTIFICATE OF QUALIFICATION

Re: Constancia Project Technical Report, March 29, 2018

I, Cashel Meagher, B. Sc., P. Geo, of Toronto, Ontario, do hereby certify that:

1. I am currently employed as Senior Vice President and Chief Operating Officer, with HudBay Minerals Inc., 25 York Street, Suite 800, Toronto, Ontario, Canada M5J 2V5.
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 fulfill 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, Constancia Mine Cuzco, Peru” (the “Technical Report”), dated March 29th, 2018 and effective as of December 31st, 2017.
7. I last visited the property on December 18st, 2017 for duration of 2 days. 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 involved with the Constancia Project property, which is the subject of the Technical Report, continuously since March, 2011.

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 29, 2018.

Original signed by:

"Cashel Meagher"

Cashel Meagher, B. Sc., P.Geo.